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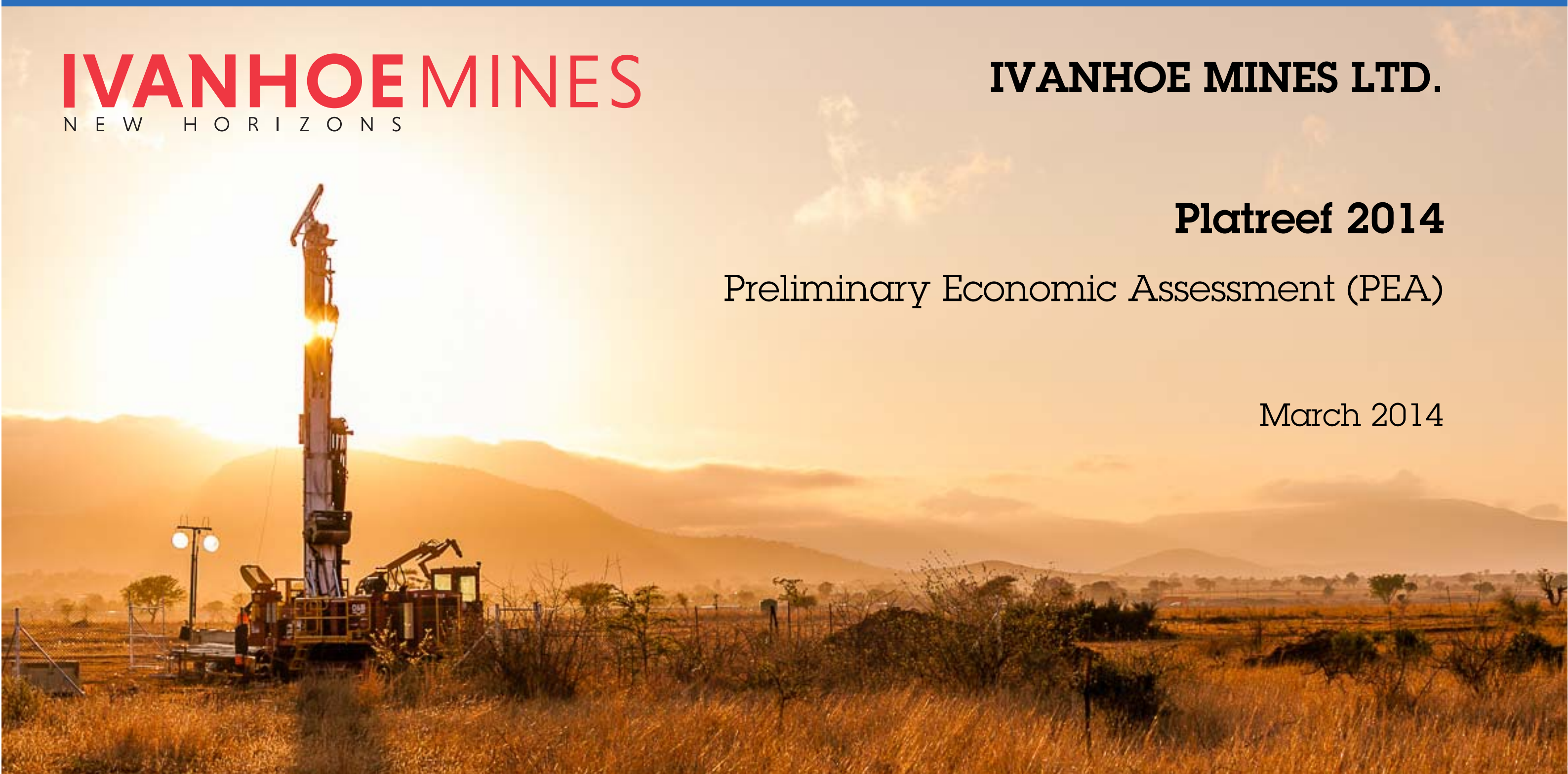

IVANHOE MINES LTD.

Platreef 2014

Preliminary Economic Assessment (PEA)

March 2014

IMPORTANT NOTICE

This notice is an integral component of the Platreef 2014 Preliminary Economic Assessment (Platreef 2014 PEA) and should be read in its entirety and must accompany every copy made of the Platreef 2014 PEA. The Platreef 2014 PEA has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The Platreef 2014 PEA has been prepared for Ivanhoe Mines Ltd. (Ivanhoe) by OreWin Pty Ltd (OreWin), AMEC E&C Services Inc (AMEC), SRK Consulting Inc (SRK), Stantec Consulting International LLC (Stantec), Metallicon Process Consulting (Pty) Ltd (Metallicon), and Geo Tail (Pty) Limited (Geo Tail) as the Report Contributors. The Platreef 2014 PEA 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 2014 PEA is intended to be used by Ivanhoe, subject to the terms and conditions of its contract with the Report Contributors. Recognizing that Ivanhoe has legal and regulatory obligations, the Report Contributors have consented to the filing of the Platreef 2014 PEA with Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval (SEDAR).

This report is a Preliminary Economic Assessment (PEA) 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 categorized as Mineral Reserves, and there is no certainty that the results will be realized. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the PEA 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 PEA. 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 2014 PEA are to the accuracy stated in the Platreef 2014 PEA only and rely on assumptions stated in the Platreef 2014 PEA. The results of further work may indicate that the conclusions, estimates and assumptions in the Platreef 2014 PEA 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 2014 PEA. 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 2014 PEA will be prone to fluctuations with time and changing industry circumstances.

The Platreef 2014 PEA should be construed in light of the methodology, procedures and techniques used to prepare the Platreef 2014 PEA. Sections or parts of the Platreef 2014 PEA should not be read or removed from their original context.

Title Page

Project Name: Platreef Project

Title: Platreef 2014 PEA

Location: Limpopo Province.
Republic of South Africa

Effective Date of Technical Report: 25 March 2014

Effective Date of Mineral Resources:

Mineral Resource Amenable To Open Pit Mining Methods: 31 March 2011.

Mineral Resource Amenable To Underground Mass Mining Methods: 13 March 2013.

Mineral Resource Amenable To Selective Underground Mining Methods: 3 April 2013.

Bikkuri Mineral Resource Amenable To Selective Underground Mining Methods: 8 May 2013.

Supply of Ongoing Drill Programme Information: 18 February 2014.

Qualified Persons:

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining was responsible for: Sections 1.1 to 1.5, 1.19, 1.20, 1.21.1, 1.22.1 ; Sections 2; Section 3; Section 15; Section 16.2.2.1; Section 19; Section 20; Section 21.1, 21.2, 21.3, 21.10; Section 22; Section 23; Section 24; Section 25.1; Section 26.1 and 26.6 and Section 27.
- Dr Harry Parker, SME Registered Member (2460450), Technical Director, AMEC E&C Services Inc was responsible for: Sections 1.6 to 1.14, 1.21.2, and 1.22.2; Section 2; Section 3.1, 3.2, 3.3; Section 4; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.8, 10.9, 10.10.1; Section 11; Section 12; Section 14; Sections 25.2 and 25.3; Section 26.2 and Section 27.
- Timothy Kuhl, SME Registered Member (1802300), Principal Geologist, AMEC E&C Services Inc was responsible for: Sections 1.6 to 1.14, 1.21.2, and 1.22.2; Section 2; Section 3.1, 3.2, 3.3; Section 4; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.8, 10.9, 10.10.1; Section 11; Section 12; Section 14; Sections 25.2 and 25.3; Section 26.2 and Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting Inc., as Corporate Consultant, was responsible for: Sections 2, Section 3; Section 16.1.
- Mel Lawson, B. Eng. (Mining), SME Registered Member (1859650), Mining Principal, Stantec Consulting International LLC was responsible for: Section 1.16, Section 1.21.3, Section 1.22.3, Section 2, Section 3, Section 16.2.1, Section 16.2.2, Section 16.2.2.2, Section 16.2.2.3, Section 16.2.3 to 16.2.7, Section 21.4, Section 21.7, Section 25.4, Section 26.3, Section 27.
- Guillaume de Swardt, P. Eng., Eng Council Sth Africa (950429) Director, Geo Tail (Pty) Limited was responsible for: Sections 2, Section 3; Sections 18.16; Section 21.6 and 21.9.
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Signature Page

Project Name:	Platreef Project
Title:	Platreef 2014 PEA
Location:	Limpopo Province. Republic of South Africa
Effective Date of Technical Report:	25 March 2014

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1.1 Introduction

The Platreef 2014 PEA is a Preliminary Economic Assessment (PEA) with an effective date of 25 March 2014 that has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43 101). The Platreef 2014 PEA 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 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, whose principal properties are 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.

Ivanhoe has undertaken scoping study work on the Platreef Project that has formed the basis of the Platreef 2014 PEA which summarises the current Ivanhoe development strategy for Platreef.

Figure 1.1 Platreef Project Location

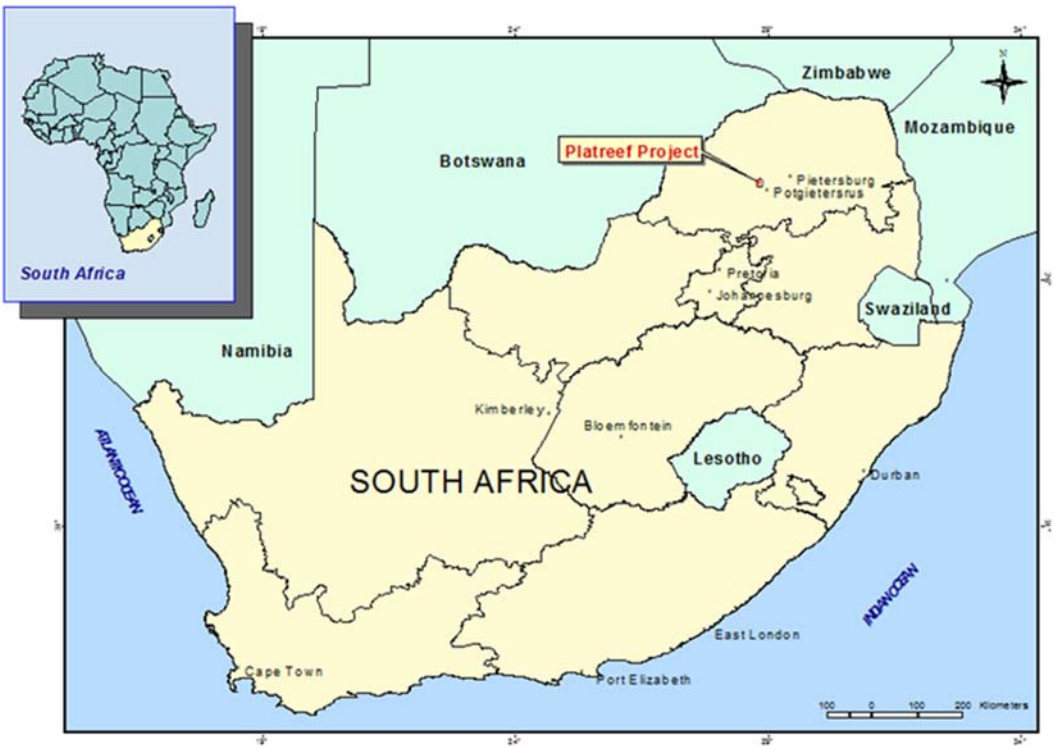


Figure supplied by Ivanhoe, 2013.

Ivanhoe holds a 90% interest in South African Prospecting Right LP30/5/1/1/2/872PR, while a 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) (the Japanese Consortium) holds the remaining 10% interest. 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 (the Platreef Project). Holdings in the Platreef Project are through South African subsidiary, Platreef Resources (Pty) Limited (Platreef Resources).

In 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 will be modified to include a Broad-Based Black Economic Empowerment (B-BBEE) partner. The B-BBEE partner will acquire a 26% interest in the Platreef Project through B-BBEE Special Purpose Vehicle (SPV) a private company incorporated in South Africa that will represent the interests of local communities and employees. It is proposed that Ivanhoe will retain a 49% interest in B-BBEE SPV.

A Mining Right 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. Upon receipt of the Mining Right, Ivanhoe will own 64% of the Platreef Project, B-BBEE SPV will own 26% and the Japanese Consortium will own 10%. 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.

Highlights of the Platreef 204 PEA are:

- Development of a large, mechanized, underground mine is planned through a phased approach.
- Three run-of-mine production scenarios were examined: 4 Mtpa; a base case of 8 Mtpa; and 12 Mtpa.
- An initial 4 Mtpa scenario would establish an operating platform.
- Options available to accelerate expansions, to the base-case 8 Mtpa and also the 12 Mtpa scenarios, as the market dictates.
- Opportunities exist for additional phases of development beyond 12 Mtpa, subject to further study.

Key features of the 8 Mtpa base-case scenario include:

- Base-case annual production target of 785 koz of platinum, palladium, rhodium and gold (3PE+Au). (At an expanded operating scenario of 12 million tonnes per year, the annual production target would be 1.1 Moz 3PE+Au).
- Estimated pre-production capital requirement of approximately US$1.7 billion, including US$381 million in contingencies.
- After-tax Net Present Value (NPV) of US$1.6 billion, at an 8% discount rate.
- After-tax internal rate of return (IRR) of 14.3%.

1.2 Platreef Development Scenarios

Ivanhoe has identified significant Mineral Resources on the Platreef Project. The Platreef 2014 PEA analyses part of the selectively-mineable Mineral Resources within and adjacent to Turfspruit Cyclic Unit (TCU) mineralized zones. There remains substantial additional Mineral Resources on the Platreef Project to be the subject of further study. Ivanhoe's development plan for Platreef Project considers three phases of underground mining and concentrator expansion. The three phases are:

- Phase 1 Concentrator 4 Mtpa.
- Phase 2 Concentrator 8 Mtpa (Base Case).
- Phase 3 Concentrator 12 Mtpa.

The base case for the Platreef 2014 PEA analysis is Phase 2 the 8 Mtpa concentrator case. The development scenarios and additional options for the Platreef Project are shown in Figure 1.2. 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.

Figure 1.2 Platreef Project Development Scenarios

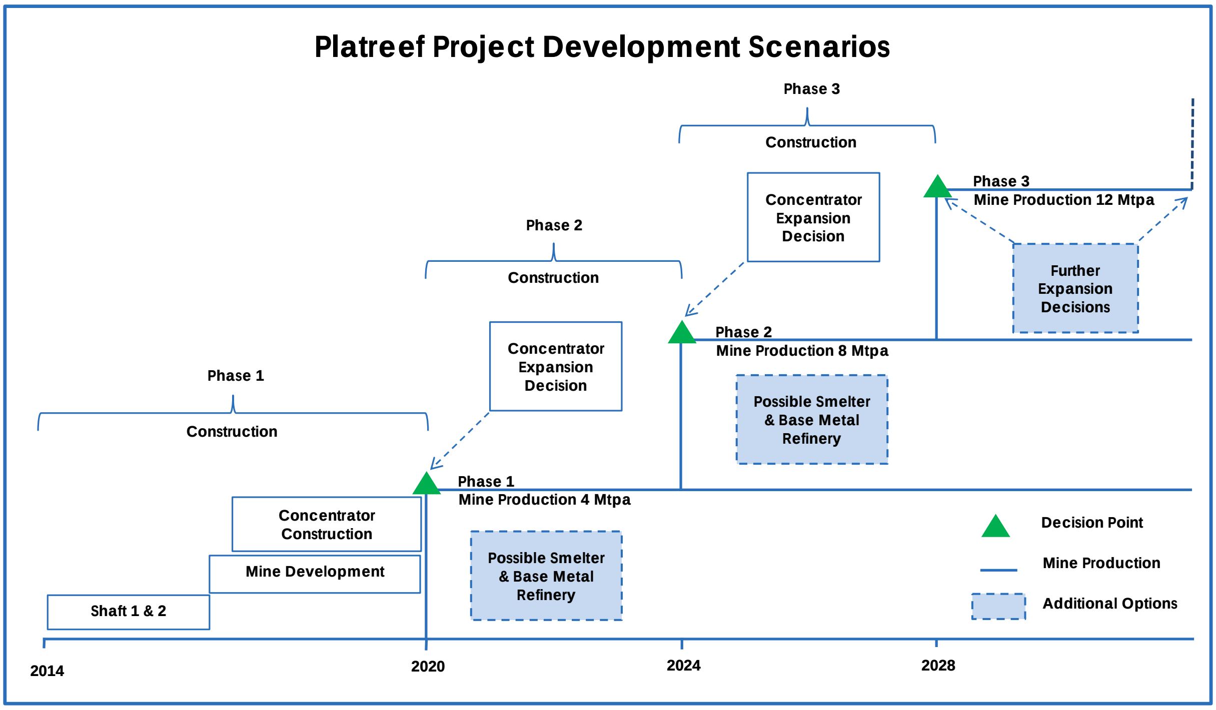


Figure by OreWin 2014.

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 started with the sinking of an exploration shaft to provide underground access to the Platreef deposit for the purpose of obtaining a bulk sample. A Bulk Sample Application was lodged with the Department of Mineral Resources (DMR) in Polokwane in September 2012, and was granted in late 2013.

The three phases have been costed and the economic analyses have been 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 Net Present Value at 8% Discount Rate (NPV8) and Internal Rate of Return (IRR) of the Platreef 2014 PEA is shown in Table 1.1.

The Platreef 2014 PEA is a Preliminary Economic Assessment as defined in National Instrument 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 categorized as Mineral Reserves, and there is no certainty that the results will be realized. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.

Table 1.1 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
Nickel	%	0.34	0.35	0.34
Platinum	g/t	1.84	1.70	1.71
Palladium	g/t	1.93	1.78	1.77
Copper	%	0.16	0.16	0.16
Gold	g/t	0.27	0.27	0.27
Rhodium	g/t	0.13	0.12	0.12
Concentrate	kt	4,665	8,771	12,396
Nickel	%	5.8	6.0	5.8
Platinum	g/t	40.5	37.0	37.3
Palladium	g/t	42.4	38.7	38.4
Copper	%	3.6	3.6	3.5
Gold	g/t	5.3	5.2	5.2
Rhodium	g/t	2.8	2.6	2.6
Nickel (Saleable Metal)	Mlb	599	1,160	1,582
Platinum (Saleable Metal)	koz	6,075	10,444	14,864
Palladium (Saleable Metal)	koz	6,362	10,915	15,294
Copper (Saleable Metal)	Mlb	368	695	957
Gold (Saleable Metal)	koz	789	1,462	2,065
Rhodium (Saleable Metal)	koz	413	742	1,050
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

Notes:

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 categorized as Mineral Reserves, and there is no certainty that the results will be realized.
2. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.
3. Price assumptions of US$8.35/lb nickel, US$1,700/oz platinum, US$820/oz palladium, US$1,300/oz gold, US$3.00/lb for copper, and US$1,700/oz rhodium.
4. Each Phase has a 30 year life.
5. 3PE+Au is the sum of the grades for Pt, Pd, Rh, and Au.

1.3 Summary of Financial Results

The economic analysis uses price assumptions of US$8.35/lb nickel, US$1,700/oz platinum, US$820/oz palladium, US$1,300/oz gold, US$3.00/lb for copper and US$1,700/oz rhodium. 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.

Comparison of the results of the financial analysis for each phase shows that there is a progressive increase in Net Present Value (NPV) for the three phases. The After Tax NPV at an 8% discount rate (NPV8), Internal Rate of Return (IRR) and Project payback period for each phase are shown in Table 1.2 and Figure 1.3. 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 1.2 After Tax Financial Results

		Phase 1 4 Mtpa	Phase 2 8 Mtpa	Phase 3 12 Mtpa
Net Present Value (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 1.3 After Tax NPV @ 8%

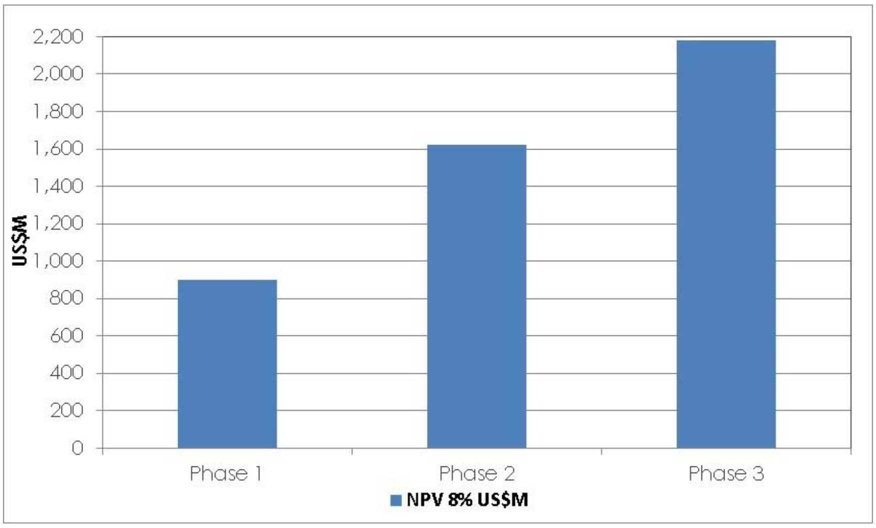


Figure by OreWin 2014.

Cash flow increases with each concentrator expansion. There is an increase in IRR for from Phase 1 through to Phase 3 and the payback period also increases with each phase. The cumulative cashflow after tax for each phase over the years can be seen in Figure 1.4 and Figure 1.5.

Figure 1.4 Cumulative Cashflow After Tax

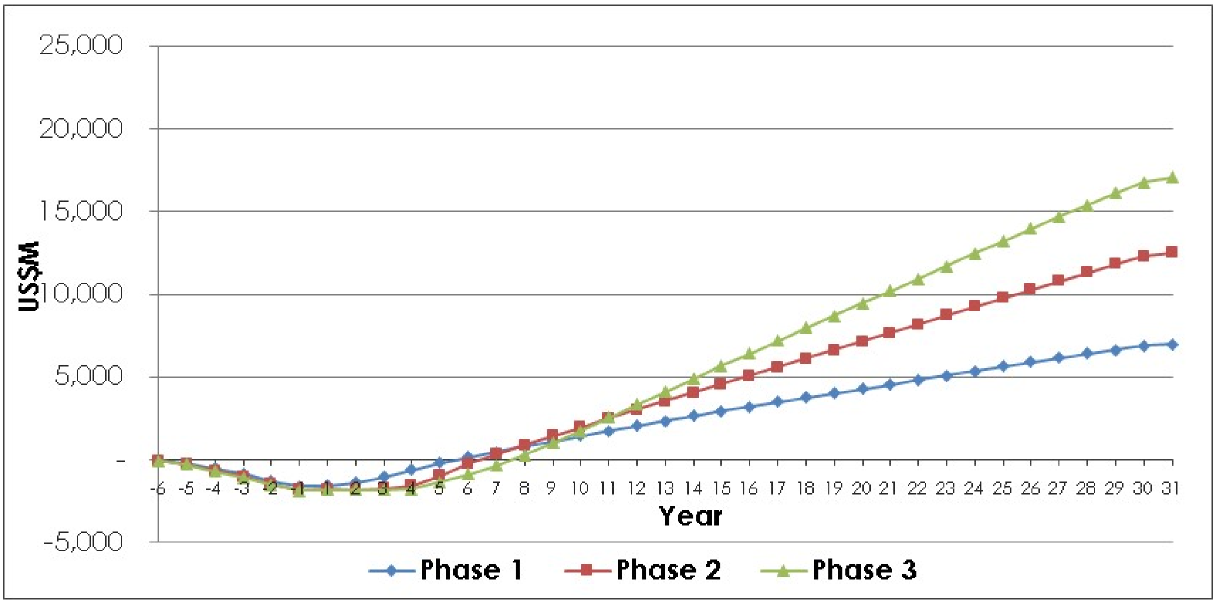


Figure by OreWin 2014.

Figure 1.5 Cumulative Cashflow After Tax (Initial Years)

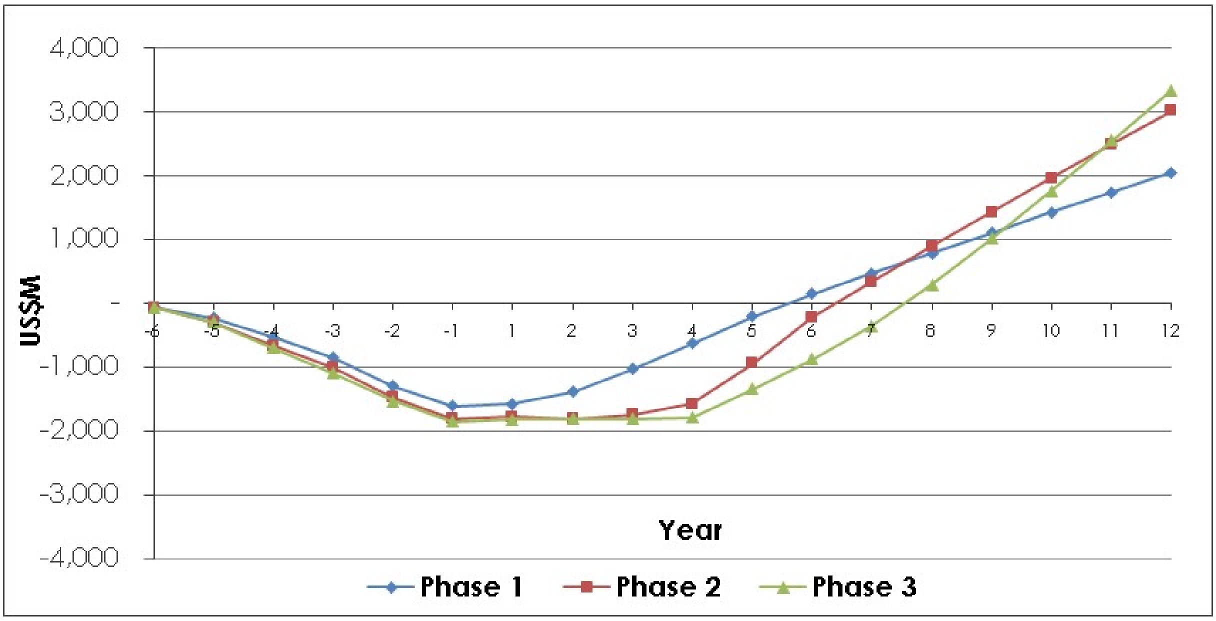


Figure by OreWin 2014.

1.4 Production Summary

The key average annual production results over the 30 year mine life are shown in Table 1.3.

Table 1.3 Production Summary

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

1.5 Capital and Operating Cost Summary

The pre-production capital cost, including contingency, for each phase is in Table 1.4. The sustaining and expansion and total capital costs are shown in Table 1.5 and Table 1.6. The operating costs are summarised in Table 1.7 to Table 1.9. 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 1.4 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
Capitalized 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 & Studies	–	19	19
Mining: Indirects	55	58	58
Mining: EPCM	80	93	97
Processing & Infrastructure: EPCM	37	37	37
Subtotal	**172**	**207**	**211**
Owners Cost			
Capitalized G&A	26	26	26
Mining	60	79	79
Processing & Infrastructure	17	18	17
Subtotal	**103**	**123**	**122**
Capital Expenditure Before Contingency	**1,185**	**1,338**	**1,376**
Mining Contingency	221	259	272
Processing & Infrastructure Contingency	120	122	120
Capital Expenditure After Contingency	**1,525**	**1,719**	**1,769**

Table 1.5 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
Capitalized 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 & Studies	–	–	19
Mining: Indirects	–	–	–
Mining: EPCM	–	–	–
Processing & Infrastructure: EPCM	4	33	63
Subtotal	**4**	**33**	**81**
Owners Cost			
Capitalized G&A			
Mining	–	–	–
Processing & Infrastructure	2	14	29
Subtotal	**2**	**14**	**29**
Capital Expenditure Before Contingency	**814**	**2,029**	**3,282**
Mining Contingency	124	354	572
Processing & Infrastructure Contingency	36	146	266
Capital Expenditure After Contingency	**974**	**2,528**	**4,120**

Table 1.6 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
Capitalized 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 & Studies	–	19	38
Mining: Indirects	55	58	58
Mining: EPCM	80	93	97
Processing & Infrastructure: EPCM	41	70	99
Subtotal	**177**	**240**	**292**
Owners Cost			
Capitalized 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 & Infrastructure Contingency	156	268	386
Capital Expenditure After Contingency	**2,499**	**4,247**	**5,888**

Table 1.7 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
Realization Cost	402	416	413
Total Cash Costs Before Credits	**814**	**840**	**854**
Nickel Credits	-367	-411	-397
Copper Credits	-81	-89	-86
Total Cash Costs After Credits	**367**	**341**	**371**

Table 1.8 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: Realization Costs			
Transport	334	628	887
Refining Charges	4,207	7,540	10,577
Government Royalty	938	1,628	2,261
Total Realization 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 & 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 1.9 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: Realization 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 Realization 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**

1.6 Mineral Tenure and Surface Rights

1.6.1 Macalacaskop and Turfspruit

Platreef Resources (Pty) Limited, registration number 1988/000334/07, holds exclusive prospecting rights to prospect for base and precious metals on farms Turfspruit 241-KR and Macalacaskop 243-KR under a 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. This prospecting right is valid to 31 May 2014. Macalacaskop contains 4,281 ha of land. Turfspruit contains 3,561 ha of land. The combined total is 7,842 ha.

On 6 June 2013, Ivanhoe applied for a mining right in respect of platinum group metals and all associated metals and minerals mined out of necessity and convenience together the with the platinum group metals, over the farms Macalacaskop 243-KR and Turfspruit 241-KR. The Mining Right Application was accepted by the DMR on 17 July 2013. At the Report effective date, the application is still being processed and has yet to be granted.

1.6.2 Reitfontein

Atlatsa Resources Corporation (Atlatsa; formerly Anooraq Resources Corporation) through its South African subsidiary, Plateau Resources Limited, holds exclusive prospecting rights to prospect for base and precious metals on the farm Rietfontein 2 KS. Rietfontein Farm has an area of 2,878 ha. The mineral lease is identified as Prospecting Right No. MPT 76/2007 PR. The prospecting right was valid for a five-year period, and was to expire on 27 November, 2011. Prior to the expiry date, on 22 August 2011, Plateau Resources lodged an application to renew the prospecting right for a three-year extension of term. At the effective date of the Report, the renewal was still pending. Legal opinion provided to AMEC notes that under the terms of section 18(3) of the MPRDA a Prospecting Right in respect of which an application for renewal has been lodged shall, despite its expiry date, remain in force until such time as the application has been granted or refused.

1.6.3 Surface Rights

Surface rights over the Macalacaskop and Turfspruit farms are 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, Mountian View, and Michelle Communities are the lawful occupiers of the Macalacaskop Farm, and the Tshamahansi (Hlongwane, Baloyi and Matjeke), Kgobudi, Masodi, and Magongoa Communities are the lawful occupiers of the Turfspruit Farm (see Figure 4.4). Rights to prospect and mine the land are granted by the State.

Ivanhoe has advised that it had undertaken extensive consultation with the communities who are the lawful occupiers of the Macalacaskop and Turfspruit areas, and surface use and co-operation agreements regulating among other things the compensation for losses and damages were entered into with four local communities during 2010.

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 in order to start negotiations for a long-term surface lease.

1.7 History and Exploration

Early exploration on the Platreef mineralization 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. None of this historical exploration information was available to AMEC.

Ivanhoe acquired a prospecting licence for both Turfspruit and Macalacaskop farms in February 1998 and subsequently entered into a JV with Atlatsa over the Rietfontein farm in 2001. The JV agreement was updated in 2009.

The initial exploration focus was on delineation of mineralization that could support open pit mining. Ivanhoe engaged a series of consultants to provide various studies involving concentrator/smelter options (Hatch Ltd in 2003), metallurgical testwork (Mineral Development Services Ltd. in 2003), and conceptual mining studies to assess reasonable prospects of developing an open pit operation (African Minerals and AMEC in 2004). Mining cost assumptions were updated towards the end of 2006, and capital and operating costs were updated in 2007 to support mineral resource assessments.

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 on the Project to date includes geological mapping, airborne and ground geophysical surveys, limited trenching, percussion drilling over the Platreef sub-crop, diamond core drilling, petrography, density determinations, metallurgical testwork, geotechnical and hydrological investigations, preliminary mineralogical studies, and Mineral Resource estimation.

1.8 Geological Setting and Mineralization

The Platreef mineralization 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 of local country rocks.

Within the Project area, five major cyclic units have been recognized 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 mineralized 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 mineralized reefs. The TCU is laterally continuous across large parts of the Project area. Mineralization 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 UG2. Contamination of the UCZ units by assimilation of Transvaal Supergroup metasedimentary rocks can occur within any of the stratigraphic horizons; however, it is predominantly confined to the units below the TCU. To date, no evidence of the existence of Lower Critical Zone (LCZ) lithologies have been found within the Turfspruit area, although LCZ mafic to ultramafic rocks have been intersected in many deep holes within the Project area.

Within the TCU, high-grade PGE–Ni–Cu mineralization is consistently hosted within an unconformable, non-cumulate, pegmatoidal, mafic to ultramafic sequence, commonly bound by chromitite stringers and containing coarse-grained sulphides; this is known as T2. The T2 pegmatoid is subdivided into an upper pyroxenitic unit (T2 Upper) and a lower olivine-bearing pyroxenitic or harzburgitic unit (T2 Lower). Overlying this pegmatoidal package is a barren, non-pegmatoidal, feldspathic pyroxenite unit of variable thickness, termed T1.

A second mineralized zone, called T1m, of disseminated, medium- to coarse-grained sulphides, is perched near the top of the T1 feldspathic pyroxenite.

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

- A double reef package informally termed the Bikkuri Reef, wherein an upper pyroxenite-dominated and mineralized sequence (the Bikkuri Reef) is separated from a thicker, mixed-lithology sequence by Main Zone and metasedimentary lithologies.
- Three different areas where TCU lithologies show significant thickening into what appear to be large depressions (pothole depressions) controlled by existing pre-Bushveld fold structures.
- Presence of a flat-lying portion of the TCU (Flatreef) that is related to structural controls.

A unique feature recognized within Zone 1 is the Flatreef portion of the Platreef, initially recognized as being approximately flat-lying compared to the steeper-dipping reefs within the proposed open pit (Zone 4) area. The subhorizontal geometry of the TCU within the Flatreef appears to be broadly controlled by faults active after the deposition of the layered rocks of the UCZ and TCU, which may be reactivated older faults. The Flatreef in essence appears to contain better-mineralized T2 mafic to ultramafic units compared to the surrounding areas, where the T1 and T2 reefs occur in closer proximity to each other.

The most noticeable structural feature recognized within the TCU is a large depression that occurs within the eastern part of Zone 1, where significant thickening of the NC1 and the upper (T1) stratigraphic layer of the TCU occurs. This depression contains a distinct thickening of the T1 feldspathic pyroxenite. A similar depression (only partly drilled) is present towards the north-western edge of Zone 1. However, in this case, thickening of both the TCU as well as its footwall units appears to have occurred. A third depression occurs mainly in the Zone 2 area in the northern part of Macalacaskop farm. These depressions are interpreted to represent potholes analogous to those known in the Merensky Reef.

Detailed drilling in Zone 1 suggests that, at the current ±100 m scale of drilling resolution, perturbations in TCU structure contours appear to have resulted from potholes and faults. This interpretation was used for the March 2013 model, but is under review by Ivanhoe geologists as part of ongoing re-interpretative studies.

Pyrrhotite, pentlandite and chalcopyrite occur as interstitial sulphides in the TCU lithologies. Platinum group minerals are mainly present as PGE–sulphides, 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.

Geological investigations at the Platreef Project are ongoing. Re-logging of core in the footwall of the TCU is under way to decipher the stratigraphy and better understand the geological controls of the mineralization. Where space permits, Critical Zone cyclical units (NC2) are observed to form below the T2. This can extend to the development of the UG2.

1.9 Drilling

Drilling on the Project has been undertaken in two major phases; the first from 2001 to 2003 is termed the open pit programme, and tested Platreef mineralization at Turfspruit (ATS) and Macalacaskop (AMK). The second phase commenced in 2007 and is ongoing. This second drill phase is termed the underground programme, testing Platreef mineralization, designated UMT, including Bikkuri. From the 954 core drillholes (excluding re-drilled mother holes and all deflections) a total of 624,248 m were drilled and completed by 26 October 2012, the date the database was closed for resource estimation purposes; 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 18 February 2014, a total of 64 drillholes (44,712 m) and 59 deflections (55,886 m) have been completed at Platreef. The additional drillholes have been completed for geotechnical data, metallurgical samples, and geology/resource drilling. Nine drillholes were in progress on 18 February 2014.

Standardised geological core 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.

Collar surveys were conducted by a licensed land surveyor on all completed holes. The majority of drillholes are downhole surveyed. All unsurveyed drillholes in the area that may potentially have Mineral Resources amenable to open pit mining are vertical and range in depth from 7–583 m. All drillholes in the area that may potentially be amenable to underground mining have been downhole surveyed by either gyroscopic (gyro) and/or electronic multi-shot (EMS) instruments.

1.10 Sample Preparation, Analyses and Security

Over the duration of Ivanhoe's work programmes, sample preparation and analyses were performed by accredited independent laboratories, including Set Point Laboratories (Set Point) in Johannesburg, Lakefield Laboratory (Lakefield; now part of the SGS Group) in Johannesburg, UltraTrace (UltraTrace) Laboratory in Perth, Genalysis Laboratories, Perth and Johannesburg (Genalysis), and SGS Metallurgical Services (SGS) in South Africa, Acme in Vancouver, and ALS Chemex in Vancouver.

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 PGE–Au–Ni–Cu deposits. Drill programmes included insertion of blank, duplicate and standard reference material (SRM) samples. The quality assurance and quality control (QA/QC) programme results do not indicate any problems with the analytical programmes 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 sample preparation facility.

1.11 Data Verification

AMEC reviewed the sample chain of custody, quality assurance and control procedures, and qualifications of analytical laboratories. The procedures and QA/QC control are acceptable to support Mineral Resource estimation. AMEC also audited the assay database, core logging, and geological interpretations. Based on this review, these data are considered to be acceptable to support Mineral Resource estimation.

1.12 Mineral Resource Estimates

Since 2001, four mutually-exclusive Mineral Resource models have been constructed, each of which reflected the focus of planned development at the time.

The 2014 PEA mine plan and financial analysis are based on the estimate prepared using the underground selectively-mineable model.

Mineral Resources amenable to selective mining methods occur below the 650 m elevation (approximately 500 m depth) and near the top of the Platreef. Mechanized drift-and-fill, bench-and-fill and large scale sublevel open stoping are being contemplated. Components of the TCU and adjacent material were modelled deterministically. Two main mineralized zones were modelled with three internal grade shells with nominal cut-off grades of 1 g/t, 2 g/t, and 3 g/t 2PE+Au (Pt + Pd + Au). Rhodium was not used to define the grade shells because assays were incomplete at the time of modelling. Three sets of faults were interpreted using regional structure as a guide to orientation and observed discontinuities in structure contour maps as to dip and dip direction. The lithological units and grade shells were hung on an artificial horizontal plane. Interpolation of nickel, copper, platinum, palladium, gold and rhodium was performed using ID3 methods, with validation in the 100 m drill-spaced areas by kriging. This Mineral Resource model was completed in March 2013.

The estimate presented in Table 1.10 is for the Mineral Resources amenable to underground mining methods (UMT).

Table 1.10 Mineral Resource Statement for Mineral Resources Amenable to Underground Selective Mining Methods Within and Adjacent to the TCU

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)	Ni (%)	Cu (%)
3 g/t	137	2.27	2.31	0.35	0.15	5.09	0.38	0.18
2 g/t	214	1.83	1.89	0.29	0.12	4.13	0.34	0.17
1 g/t	387	1.28	1.34	0.21	0.09	2.92	0.28	0.14
Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Ni (Mlbs)	Cu (Mlbs)
3 g/t	–	10.0	10.2	1.53	0.67	22.4	1,133	558
2 g/t	–	12.6	13.0	2.00	0.85	28.5	1,610	794
1 g/t	–	15.9	16.7	2.67	1.09	36.3	2,408	1,189
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)	Ni (%)	Cu (%)
3 g/t	211	2.09	2.06	0.34	0.14	4.63	0.38	0.18
2 g/t	415	1.57	1.59	0.27	0.11	3.54	0.33	0.16
1 g/t	1,054	0.96	1.02	0.18	0.07	2.23	0.26	0.13
Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Ni (Mlbs)	Cu (Mlbs)
3 g/t	–	14.2	14.0	2.29	0.97	31.5	1,764	855
2 g/t	–	20.9	21.3	3.58	1.44	47.2	3,032	1,490
1 g/t	–	32.7	34.7	5.95	2.32	75.7	5,934	3,035

Notes:

1. Mineral Resources have an effective date of 3 April 2013. The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME.
2. Mineral Resources estimated assuming underground selective mining methods within and adjacent to the TCU are exclusive of the Mineral Resources estimated assuming mass-mining methods. The 2 g/t 3PE+Au cut-off is considered the base case estimate. (Highlighted); the 3 g/t 3PE+Au cut-off is also being considered.
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 m x 400 m (locally to 400 m x 200 m and 200 m x 200 m) spacing.
4. Reasonable prospects for economic extraction were determined using the following assumptions. Assumed commodity prices are Ni: $8.81/lb, Cu: $2.73/lb, Pt: $1,699/oz, Pd: $667/oz, Au: $1,315/oz, and Rh: $2,065/oz. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $40/t) and process, G&A, and concentrate transport costs (average $12.50/t of mill feed for a 4 Mtpa operation) would be covered. The process recoveries vary with block grade but typically would be 85–90% for Pt, Pd and Rh; 75% for Au and 70% for Ni and 85% for Cu.
5. Totals may not sum due to rounding.

1.13 Additional Mutually Exclusive Mineral Resource Estimates

Outside the selectively-mineable model, three other mutually-exclusive Mineral Resource models have been constructed, since 2001, each of which reflected the focus of planned development at the time. These are:

- Mineral Resources amenable to underground mass-mining methods occur below the 650 m elevation (approximately -500 m depth) and are located below the TCU and adjacent material (within 75 m). Block-caving and sublevel caving are contemplated. Nickel, copper, platinum, palladium gold and rhodium were interpolated using inverse distance cubed (ID3) methods within lithological units such as norites, pyroxenites, harzburgites and mixtures of harzburgites and pyroxenites. These lithologies were interpreted to be stratiform. Stratabound occurrences of contaminated (with floor rocks) and disturbed zones were also modelled and estimated. The Mineral Resource model was last updated in March 2011.
- Mineral Resources amenable to open pit mining methods occur above the 650 m elevation, which is approximately 500 m below the topographic surface. The Platreef has been modelled as a series of dipping layers of norites and pyroxenites, serpentinites (harzburgites), and xenoliths (rafts of hornfels). Nickel, copper, platinum, palladium, and gold were all interpolated in each layer using inverse distance interpolation methods. These Mineral Resource models were completed in 2003.
- Bikkuri area Mineral Resources are amenable to underground selective mining methods. This consists of material within and adjacent to grade shells in the Bikkuri Reef. This Mineral Resource has been estimated using revised geological interpretations and incorporation of additional drilling in Zone 1 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 March 2013.

The estimate using mass mining method assumptions is presented in Table 1.11; that for the estimate assuming open pit mining methods in Table 1.12, and the Bikkuri estimate in Table 1.13.

1.13.1 Comments on Mineral Resource Estimates

This section covers all four of the mutually-exclusive resource models and resource statements.

Mineral Resources are reported on a 100% ownership basis; attributable ownership is discussed in detail in Section 4.4. Dr Parker and Mr Kuhl are of the opinion that the Mineral Resources for the Project, which have been estimated using core-drill data, have been performed to industry best practices (CIM, 2003), and conform to the requirements of CIM Definition Standards, 2010.

Factors which 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, assumptions used to generate the conceptual data for consideration of reasonable prospects of economic extraction, and the pending renewal of the Rietfontein prospecting licence (which would affect the open pit mineral resource estimate only).

Table 1.11 Inferred Mineral Resources Assuming Underground Mass Mining Methods

Tonnage and Grades							
Property	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**2PE+Au (g/t)**	**Ni (%)**	**Cu (%)**
Turfspruit	1,870	0.40	0.49	0.09	0.98	0.21	0.13
Macalacaskop	40	0.28	0.39	0.09	0.76	0.21	0.14
Total 2PE + Au	**1,910**	**0.40**	**0.49**	**0.09**	**0.98**	**0.21**	**0.13**
Contained Metal							
Property	–	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**2PE+Au (Moz)**	**Ni (Mlbs)**	**Cu (Mlbs)**
Turfspruit	–	24.0	29.4	5.5	58.9	8,740	5,520
Macalacaskop	–	0.4	0.5	0.1	1.0	190	120
Total	**–**	**24.4**	**29.9**	**5.6**	**59.9**	**8,930**	**5,650**

Notes:

1. Mineral Resources have an effective date of 13 March 2013. The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME.
2. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from the 650 m elevation (-500 m depth) downward to approximately -400 m elevation (-1,550 m depth). The 2011 block model has been trimmed to exclude the 2013 block model for Mineral Resources amenable to selective mining methods. Inferred Mineral Resources are based on an area drilled on approximately 400 m x 400 m (locally 400 m x 200 m and 200 m x 200 m) spacing.
3. The estimate is reported at a cut-off grade of 0.15% Ni. Mineral Resources at the 0.15% Ni cut-off grade occur in continuous zones; there are a relatively minor number of blocks inside these zones that are below cut-off and have been excluded.
4. The cut-off grade (0.15% Ni) used to report the base case for reasonable prospects of economic extraction assumes commodity prices of Ni: $8.81/lb, Cu: $2.73/lb, Pt: $1,699/oz, Pd: $667/oz, Au: $1,315/oz. It has been assumed that payable metals would be 82% from smelter/refinery and that a mix of block cave and sub-level mining costs (averaging $20/t, and ranging from $9/t to $35/t), and process, G&A, and concentrate transport costs (average of $12/t of mill feed) would be covered for a conceptual 10 Mtpa operation. Process recoveries are taken from metal-specific equations for serpentinite. Nickel is presented as an example where nickel recovery = ((9.3 *Ln (Ni head grade) + 84.9).
5. Totals may not sum due to rounding.
6. In this table 2 PE + Au = Pt + Pd + Au.

Table 1.12 Indicated and Inferred Mineral Resources that are Amenable to Open Pit Mining Methods

Property/Deposit	Mt	Ni Sulphide (%)	Cu Sulphide (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	2PE+Au (g/t)
ATS – Indicated							
Turfspruit 241 KR	470	0.20	0.14	0.34	0.45	0.09	0.87
Rietfontein 2 KS	40	0.21	0.17	0.28	0.41	0.09	0.78
Total ATS Indicated	**520**	**0.20**	**0.14**	**0.33**	**0.44**	**0.09**	**0.86**
Contained Metal		**Ni Sulphide (Mlb)**	**Cu Sulphide (Mlb)**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**2PE+Au (Moz)**
Turfspruit 241 KR		2,038	1,484	5.1	6.8	1.3	13.20
Rietfontein 2 KS		202	161	0.4	0.6	0.1	1.10
Total ATS Indicated		**2,240**	**1,645**	**5.5**	**7.4**	**1.4**	**14.3**
ATS – Inferred							
	Mt	**Ni Sulphide (%)**	**Cu Sulphide (%)**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**2PE+Au (g/t)**
Turfspruit 241 KR	260	0.16	0.10	0.41	0.47	0.10	0.97
Rietfontein 2 KS	0	0.00	0.00	0.00	0.00	0.00	0.00
Total ATS Inferred	**260**	**0.16**	**0.10**	**0.41**	**0.47**	**0.10**	**0.97**
Contained Metal		**Ni Sulphide (Mlb)**	**Cu Sulphide (Mlb)**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**2PE+Au (Moz)**
Turfspruit 241 KR		899	589	3.4	3.9	0.8	8.10
Rietfontein 2 KS		0	0	0.0	0.0	0.0	0.00
Total ATS Inferred		**899**	**589**	**3.4**	**3.9**	**0.8**	**8.1**
AMK – Inferred							
	Mt	**Ni Sulphide (%)**	**Cu Sulphide (%)**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**2PE+Au (g/t)**
Macalacaskop 243 KR	250	0.17	0.11	0.52	0.55	0.10	1.18
Contained Metal		**Ni Sulphide (Mlb)**	**Cu Sulphide (Mlb)**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**2PE+Au (Moz)**
Macalacaskop 243 KR		913	581	4.2	4.5	0.8	9.5
Total – Open pit (AMK + ATS)							
	Mt	**% Ni Sulphide**	**% Cu Sulphide**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**2PE+Au (g/t)**
Indicated	520	0.20	0.14	0.33	0.44	0.09	0.86
Inferred	510	0.16	0.10	0.46	0.51	0.10	1.07
Contained Metal		**Ni Sulphide (Mlb)**	**Cu Sulphide (Mlb)**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**2PE+Au (Moz)**
Indicated		2,240	1,645	5.5	7.4	1.4	14.30
Inferred		1,812	1,171	7.6	8.3	1.6	17.5

Notes:

1. Mineral Resources have an effective date of 31 March 2011. The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME.
2. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from 650 m elevation to surface (approximately 500 m depth extent). A selective mining unit (SMU) of 15 m x 15 m x 10 m has been assumed. External dilution has not been applied. Indicated Mineral Resources are based on an area drilled on approximately 75 m x 100 m spacings. Inferred Mineral Resources are based on an area drilled on approximately 120 m x 140 m spacings.
3. Mineral Resources are reported assuming a 0.1% sulphide nickel cut-off grade. At a 0.1% sulphide nickel cut-off grade, the mineralization is continuous.
4. The 0.1% sulphide Ni cut-off grade is based on assumed costs and metal prices for the purposes of assessing reasonable prospects of economic extraction. Commodity prices were assumed to be Ni: \$9.20/lb, Cu: \$3.00/lb, Pt: \$1,785/oz, Pd: \$650/oz, Au: \$1,265/oz. Concentrator, G&A and concentrate transport costs are estimated to average \$11/t of mill feed for a conceptual 10 Mt/a operation. Mining costs are estimated at an average of \$5/t.
5. Totals may not sum due to rounding.
6. In this table 2PE + Au = Pt + Pd + Au.

Table 1.13 Mineral Resource Estimates for the Bikkuri Reef Within and Adjacent to Grade Shells Assuming Selective Underground Mining Methods

Indicated Mineral Resources Tonnage and Grades								
Grade Shells	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**Rh (g/t)**	**3PE+Au (g/t)**	**Ni (%)**	**Cu (%)**
3 g/t	2.3	1.67	1.45	0.36	0.09	3.57	0.40	0.22
2 + 3 g/t	5.6	1.34	1.20	0.30	0.08	2.92	0.36	0.20
1 + 2 + 3 g/t	17.2	0.83	0.79	0.19	0.05	1.86	0.27	0.15
Contained Metal								
Grade Shells	–	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Ni (Mlbs)**	**Cu (Mlbs)**
3 g/t	–	0.12	0.11	0.03	0.01	0.26	20.2	11.1
2 + 3 g/t	–	0.24	0.22	0.05	0.01	0.52	44.0	24.4
1 + 2 + 3 g/t	–	0.46	0.44	0.10	0.03	1.03	101.7	58.2
Inferred Mineral Resources Tonnage and Grades								
Grade Shells	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**Rh (g/t)**	**3PE+Au (g/t)**	**Ni (%)**	**Cu (%)**
3 g/t	0.9	1.57	1.41	0.37	0.08	3.42	0.40	0.21
2 + 3 g/t	2.3	1.30	1.16	0.31	0.07	2.84	0.34	0.18
1 + 2 + 3 g/t	10.0	0.75	0.73	0.18	0.04	1.70	0.25	0.14
Contained Metal								
Grade Shells	–	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Ni (Mlbs)**	**Cu (Mlbs)**
3 g/t	–	0.05	0.04	0.01	0.00	0.10	7.8	4.0
2 + 3 g/t	–	0.10	0.09	0.02	0.01	0.21	17.4	9.1
1 + 2 + 3 g/t	–	0.24	0.23	0.06	0.01	0.55	55.0	30.6

Note:

1. Mineral Resources have an effective date of 8 May, 2013. The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME.
2. Mineral Resources estimated assuming underground selective mining methods for the Bikkuri Reef are exclusive of the Mineral Resources estimated assuming mass-mining methods and the Mineral Resources estimated within and adjacent to the TCU. The grade shell rows are also not additive. The 3 g/t 2PE + Au shell is included in the 2 + 3 g/t 2PE + Au shell, which is in turn included in the 1 + 2 + 3 g/t 2PE + Au shell.
3. The 2 + 3 g/t 2PE + Au shell is considered the base case estimate. (Highlighted)
4. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately 400 m to 800 m elevation. Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 m x 400 m (locally to 400 m x 200 m and 200 m x 200 m) spacing.
5. Reasonable prospects of economic extraction were determined using assumed commodity prices of Ni: $8.81/lb, Cu: $2.73/lb, Pt: $1,699/oz, Pd: $667/oz, Au: $1,315/oz, Rh: $2.065/oz. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $40/t) and process, G&A, and concentrate transport costs (average $12.50/t of mill feed for a 4 Mtpa operation) would be covered. The process recoveries vary with block grade but typically would be 85–90% for Pt, Pd and Rh; 75% for Au and 70% for Ni and 85% for Cu. The Mineral Resources within the 1+2+3, 2+3 or 3 g/t 2PE + Au grade shells have been estimated (at nominal cut-off grades of 1, 2, and 3 g/t 2PE + Au respectively) to show sensitivity to cut-off grade and to provide multiple options for consideration in future mining studies. No allowances for mining recovery and external dilution have been applied.
6. Totals may not sum due to rounding.
7. In this table 3PE + Au = Pt + Pd + Rh + Au. 2PE + Au = Pt + Pd + Au.

1.14 Exploration Targets

Beyond the current Mineral Resources, mineralization is open to expansion to the south and west. Two exploration targets have been identified.

Target 1 is based on results from 14 wide-spaced step-out drillholes completed between 26 October 2012 and 18 February 2014. Target 1 could contain up-an additional 115–235 Mt grading 3.1 to 4.5 g/t 3PE + Au, (comprising 1.2–1.7 g/t Pt, 1.7–2.3 g/t Pd, 0.06–0.14 g/t Rh, and 0.17–0.26 g/t Au), 0.23% – 0.28% Ni and 0.11% – 0.14% Cu over an area of 3.7 km^2. The tonnage and grade ranges are based on intersections of 2 g/t 3PE + Au mineralization in drillholes completed in Target 1.

Target 2, which surrounds the Mineral Resource estimates in Zones 1 and 2, could contain an estimated additional 260-450 Mt grading 3.4 to 4.5 g/t 3PE + Au (comprising 1.7–2.4 g/t Pt, 1.2–1.6 g/t Pd, 0.14–0.20 g/t Rh, 0.26–0.33 g/t Au), 0.30%–0.35% Ni and 0.15%–0.18% Cu over an area of 7.6 km^2. The tonnage and grade ranges are based on 2 g/t 3PE + Au intersections of mineralization in 19 widely-spaced drillholes completed in Target 2 and adjacent drillholes within the Inferred Mineral Resources. These drillholes were also completed between 26 October 2012, and 18 February 2014.

AMEC 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.

Beyond these exploration target areas is approximately 37 km^2 of unexplored ground on the property under which the Platreef is projected to lie. It is not possible to estimate a range of tonnages and grades for this ground. There is excellent potential for mineralization to significantly increase with further step-out drilling to the south-west.

1.15 Metallurgical Test Work Overview

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

Up until 2006, metallurgical test work was carried out mainly on lower grade shallow material from the potentially large open pit area. Flotation recoveries and concentrate grades were generally low, resulting in the necessity for further processing on site via combinations of smelting, converting and magnetic separation; hydrometallurgical treatment was also considered.

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

A flotation test work programme on high-grade 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, T2 Upper (T2U), and T2 Lower (T2L).

Previous comminution tests indicated that the plant feed is competent with respect to SAG milling and that a crusher and ball mill circuit will be the preferred option. The Platreef material is classified as hard to very hard. The flotation test work 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 platinum group metals (PGM) population locked in gangue at sizes of 10 µm or finer and amounting to approximately 10% – 15% of the contained PGMs.

Although this phase of the test work is preliminary it did indicate that an effective flow sheet will involve several stages of cleaner flotation with recycling of the stage tailings. All of the three geo-metallurgical units and the two blends produced acceptable smelter-grade final concentrates at acceptable recoveries.

The processing plant consists of a relatively standard flotation concentrator targeted at producing a saleable concentrate.

The design approach currently entails a concentrator able to accommodate Phases 1 to 3 beginning at a concentrator feed rate of 4 Mtpa followed by expansions to 8 Mtpa (Phase 2) and 12 Mtpa (Phase 3). A smelter and BMR could be added to the plant as options to Phase 2 and or Phase 3. 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 the smelter and or a 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.

1.16 Mining

Stantec Consulting (Stantec) was requested by Platreef Resources to prepare a scoping level evaluation for underground mining of part of the Mineral Resources Within and Adjacent to the TCU (Table 1.10). The study has been developed using the updated Platreef Resource Model dated March 2013 and evaluates the options for mining the Indicated and Inferred Mineral Resources in the Platreef Project based on three different production rate scenarios of Phase 1 (4 Mtpa), Phase 2 (8 Mtpa) and Phase 3 (12 Mtpa), using long-hole stoping and drift-and-fill extraction methods followed with either cemented paste, cemented rock fill (CRF), or waste rock backfill where applicable.

The mine plans have been developed for a total project life of 36 years including pre-production (total production life of 30 years after mill start-up).

Figure 1.6 is an elevated view showing the mining areas and preliminary layouts for Phase 3 (12 Mtpa).

Figure 1.6 Phase 3 (12 Mtpa) Development (Looking South-east)

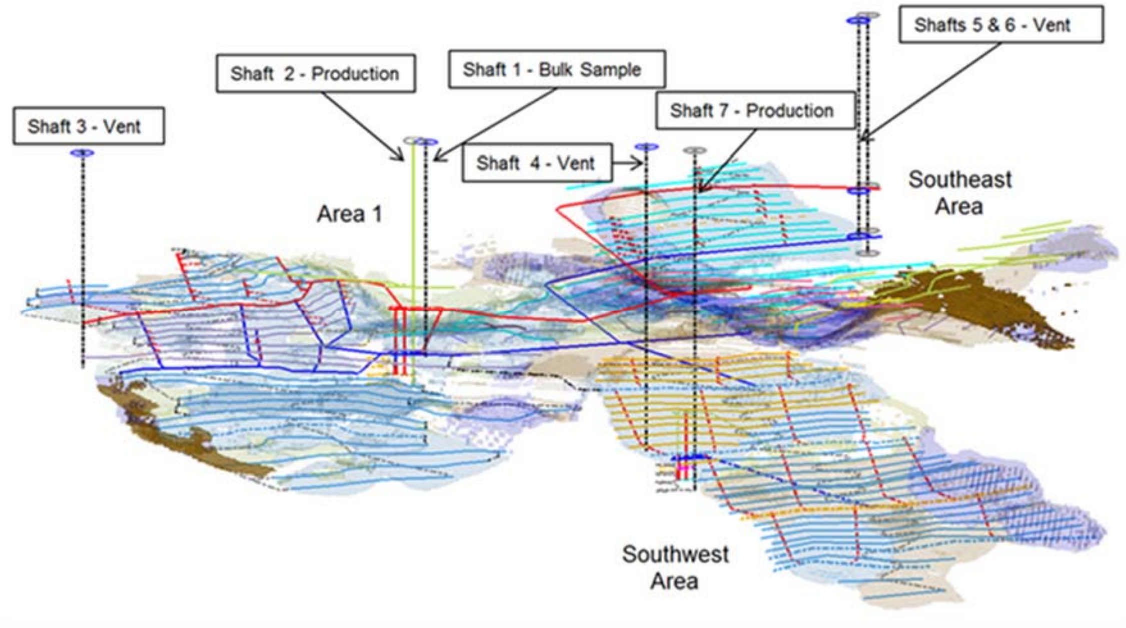


Figure by Stantec. Note: Colors are for presentation only; they have no technical significance.

Figure 1.7 is a similar view that roughly illustrates the relative mine expansions for Phase 1 (4 Mtpa), Phase 2 (8 Mtpa) and Phase 3 (12 Mtpa).

Figure 1.7 Phase 1 to 3 Mine Expansions (Looking South-east)

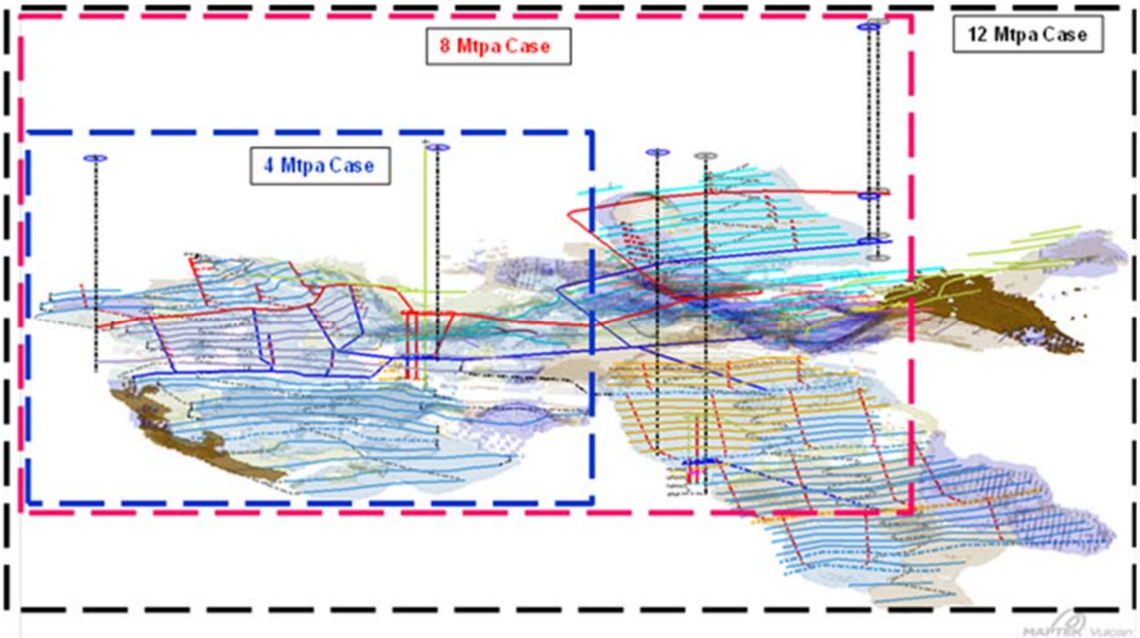


Figure by Stantec. Note: Colors are for presentation only; they have no technical significance.

1.17 Recovery Methods

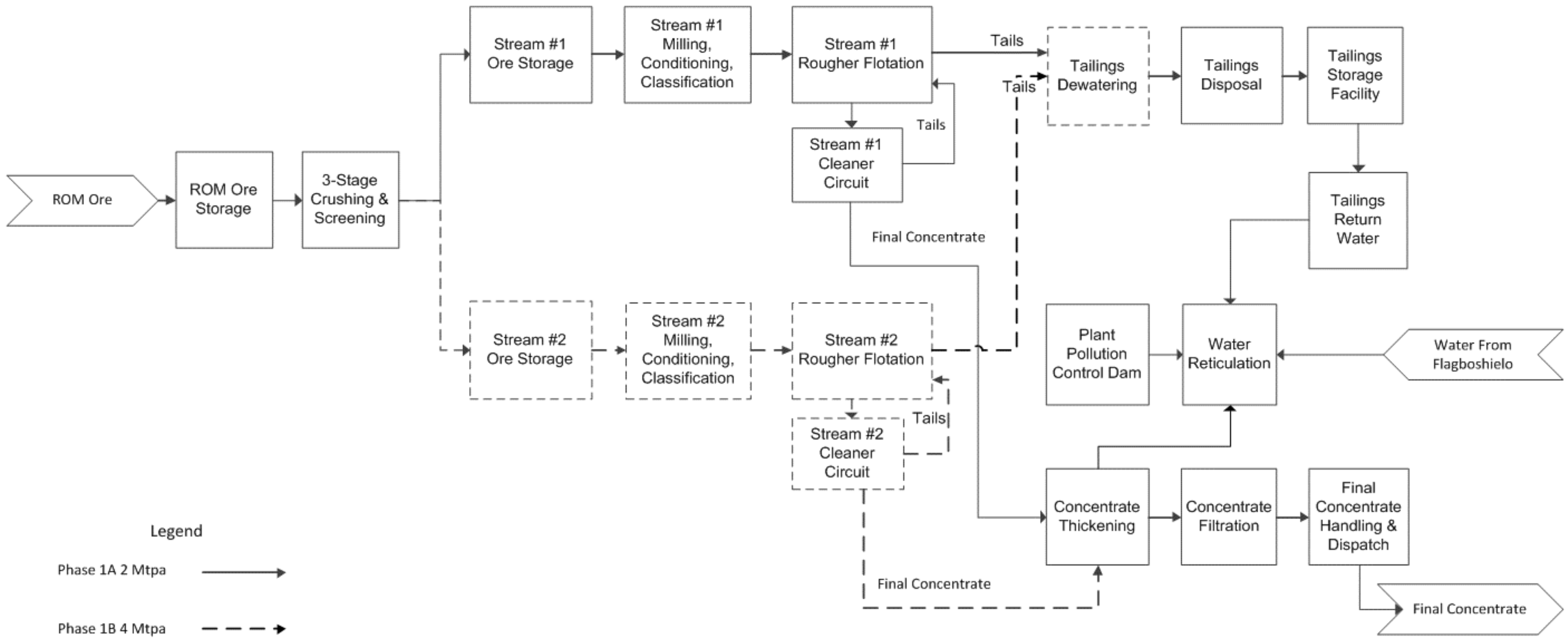
The plant will consist of a concentrator for all three production phases of 4 Mtpa, 8 Mtpa, and 12 Mtpa.

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

Phase 1 includes the construction of a 4 Mtpa concentrator and other associated infrastructure in 2020, in two modules of 2 Mtpa. Phase 2 includes a ramp-up to 8 Mtpa in 2024, and Phase 3 a further ramp-up to a plant capacity of 12 Mtpa in 2028.

A two phased production approach was used for the Phase 1 flow sheet development and design. The selected flowsheet comprises a common 3-stage crushing circuit, feeding crushed material to two milling-flotation modules 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 tailings handling and concentrate thickening filtration and storage. The process description is presented in Figure 1.8.

Figure 1.8 Concentrator Flowsheet



Note: 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 (typical rainfall approximately 600 mm per annum) will require significant additional capacity to meet the growing demand 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 that need to be addressed in order to ensure that sufficient supply is achieved.

The Olifants River Water Resource Development Project (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 Project. From this point, the Pruissen Pipeline Project will be developed to deliver water to the communities and mining projects on the Northern Limb.

Under the ORWRDP, a pipeline is to be constructed between Flag Boshielo Dam on the Olifants River and Pruissen, located to the south of Mokopane. The Pruissen Pipeline Project will be developed to deliver water from Pruissen to the communities and mining projects on the Northern Limb, including the Platreef Project. The Flag Boshielo dam was raised 5 m to its final height in 2006.

The Pruissen pipeline scheme forms part of the larger ORWRDP regional expansion of water distribution, which also includes the construction of the De Hoop Dam on the Steelpoort River and related pipeline infrastructure. Construction of this dam began in March 2007, but has been intermittent since then due to funding issues and engineering redesigns. Based on current forecasts, the projected partial impoundment of the De Hoop Dam was achieved in Q3 2011, and raw water was available for delivery in Q2 2012. An outline of the ORWRDP is shown in Figure 1.9.

Figure 1.9 Olifants River Water Resources Development Project

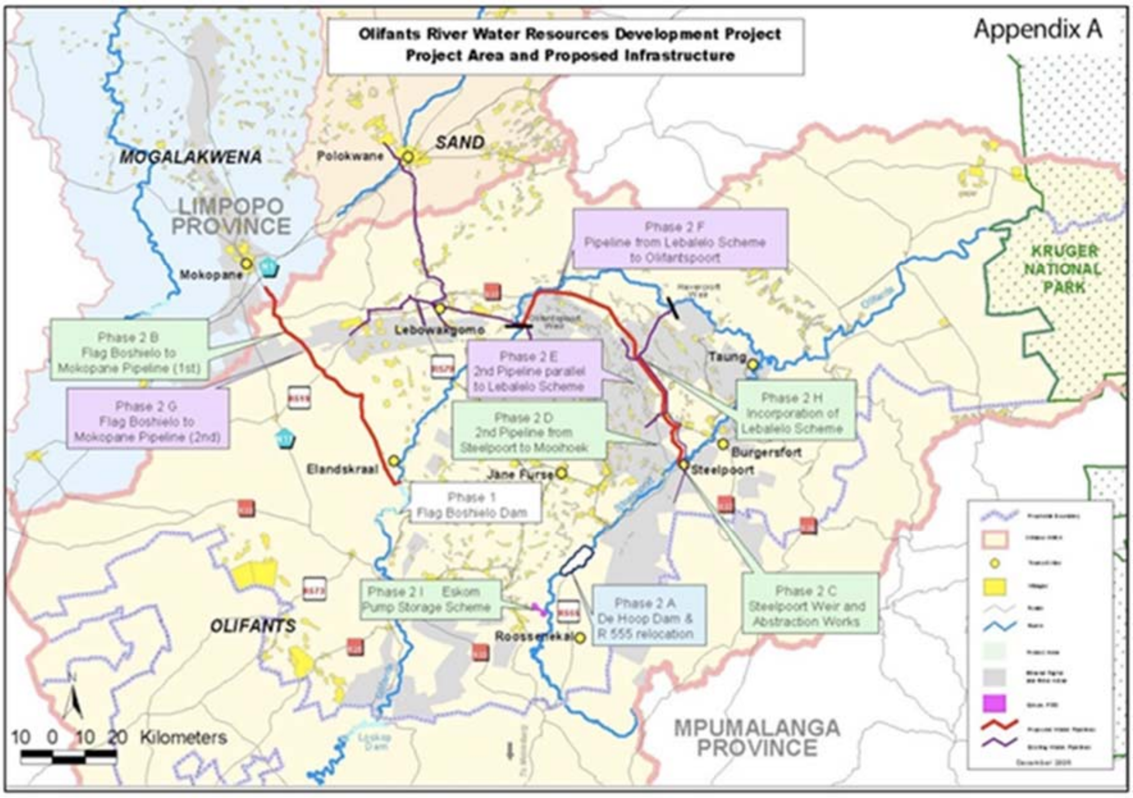


Figure supplied by Ivanhoe 2013.

Platreef Resources 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 co-ordinate discussions with the various participants in the ORWRDP 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 ORWRDP 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. The Platreef Project's water requirement for the 8 Mtpa base case scenario would be approximately 22 million litres per day.

The available water in the region is fully allocated despite the construction of the De Hoop Dam and the raising of Flag Boshielo Dam wall. The yields of both these dams have been reduced due to revision of the hydrology. In addition to water demand from the mines, there is also the need to meet environmental water requirements in the catchment, which are currently not being met in full.

As a consequence, the coal mines in the Witbank area have commissioned a study to determine the potential of excess acid mine water in the Highveld Coal Fields in the upper Olifants catchment as a resource to supplement the demands of mining in the middle Olifants catchment. The preliminary results of this study have been published.

The Platreef Resource is committed to working with the JWF to develop the ORWRDP as the primary source of bulk water to service the needs of the Project.

1.18.2 Highway Re-Alignment

The N11 National Highway connects Mokopane to the South Africa/Botswana border. This road currently runs directly through the Turfspruit and Macalacaskop farms, and serves the operating Anglo Platinum 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 Mokopane.

The major mining operations in the area, mainly platinum, are located some 25 km north-west of Mokopane. The current transportation to and from the mines is by road which has resulted in concerns regarding the deteriorating level of service of the existing road (N11).

A study was completed in 2009 in respect of the proposed re-routing of approximately 18.4 km and the upgrading of approximately 3.6 km of Section 13 of the N11 (N11-13), as well as the widening of approximately 2.7 km of the Provincial Road (R101), between Armoede to the north of Mokopane and Planknek to the east of same in the Limpopo Province.

The realignment route will bypass the Turfspruit and Macalacaskop farms but will bisect the Rietfontein farm and has therefore been considered in the Tailings Storage Facility footprint.

1.18.3 Project Power Supply

Eskom has advised that sufficient power is not presently available in the Mokopane area due to transmission line limitations and generating shortfalls. The generating shortfall should be alleviated with the first unit of the new Medupi Power Station due to come on line in Q4 2014 and then at nine month intervals for the remaining five units.

The Medupi Power Station is a new dry-cooled coal fired power station being built by Eskom near Lephalale (approximately 180 km east-north-east of Mokopane) in Limpopo province, South Africa. When completed, the Medupi Power Station will have six boilers (Six-Pack design) each powering a 800 MW turbine, producing 4,800 MW of power into the national grid. This will be the largest dry-cooled coal fired power station in the world. Contracts have been placed with Hitachi to supply the boilers and Alstom to provide the steam turbines for this plant. Medupi Power Station will be supplied by coal from Exxaro's Grootegeluk coal mine, located adjacent to the Medupi Power Station. Eskom has placed a contract with Exxaro to supply coal for 40 years.

A new Main Transmission Substation (MTS), called the Borutho MTS (400 kV/132 kV/22 kV) is sized at 2x500 MVA (extendable) and will come on line in 2014. The Borutho substation is approximately 26 km's from the Platreef Project site (Figure 1.10).

Should future demand exceed the capacity of the new Borutho MTS, an additional transformer could be installed at the MTS to meet the new demand and to increase capacity to 1000 MVA.

There are two transmission alternatives to supply power to the Platreef Project that have been defined. They are: the standard and the premium supply schemes.

The standard supply scheme essentially consists of a radial line to the project site. The disadvantage of this is that should the line be damaged, there will be no alternative power transmission system to the plant site.

The premium supply scheme would include a loop in and loop out from the project site totalling 8 km. This will allow redundancy in power transmission lines should one of the lines go down. The Platreef Project has requested that Eskom complete a budget quote study that considers the premium supply option.

Figure 1.10 Proposed Transmission

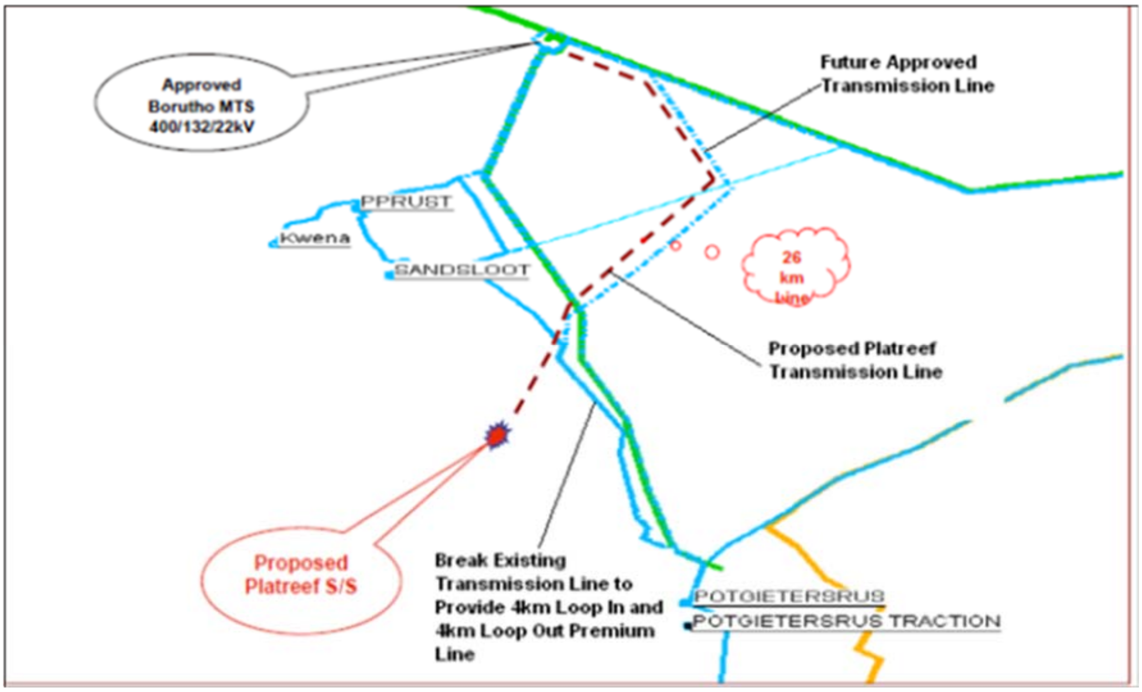


Figure supplied by Ivanhoe 2013.

1.19 Market Studies and Contracts

Ivanhoe plans a phased expansion of the operation with Phase 1 run of mine (ROM) production of 4 Mtpa (approximately 150 ktpa concentrate and 9 ktpa nickel) followed by Phase 2 at 8 Mtpa and Phase 3 at 12 Mtpa. The capacities of the expansions after Phase 1 would be determined at the time through additional studies. 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. Ivanhoe commissioned a marketing study on the sale of concentrates, furnace mattes, converter mattes and PGM concentrates in 2013. The marketing assumptions in the Platreef 2014 PEA are based on scoping studies, not contracts.

South Africa has a number of smaller PGM mining firms, toll smelting and refining contracts and purchase agreements have become more prevalent in South Africa than in the past. The major PGM 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, PGM 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 PEA.

Marketing of the products from the Platreef Project is of significant importance to the project value. The terms of payment are highly variable and subject to change as conditions for any one potential purchaser alter. Each are subject to competitive pressures, capacity pressures, tolerance for impurities (Cr_2O_3, etc) changing feed mixes, cost pressures, exchange rates, the impact of product quality discounts/premiums and process efficiency improvements. Thus it is difficult to generalise about the value available from various products. However, the assumptions in the Platreef 2014 PEA represent a reasonable estimate based upon the current market knowledge.

The estimates in the Platreef 2014 PEA are based on knowledge of concentrates sales contracts that have been agreed in Southern Africa with local purchasers and for the offshore purchase of mattes and PGM concentrates. As the local purchase of furnace mattes or converter mattes has only occurred infrequently and for small volumes, an estimate has been based on the operating costs and an approximation for a required margin. Actual terms may vary and will be dependent on the negotiations at the time the contracts are agreed.

Whilst there is sufficient furnace capacity in South Africa, the converting and sulphur removal capacity is constrained by environmental sulphur emissions permits. There is some available converting and acid plant capacity, but the high iron and sulphur levels in the Ivanhoe concentrates will likely fill this very quickly, and additional capital expenditure would then be required.

The marketing study concluded that nickel refining capacity could accommodate Phase 2 of Ivanhoe's development plan, for an expansion from 9 ktpa to 18 ktpa nickel. Whilst the current refining gross footprint would not need expansion, further modules of production will need to be added to accommodate the Ivanhoe converter mattes.

The local opportunity for Ivanhoe in the initial phases of their operations is thus limited to utilising the spare nickel capacity that is available but this will require solutions to the environmental issues and successful negotiations between Ivanhoe and the three smelters and refiners. The availability of excess capacity required for Phase 2 of the project is largely dependent on the state of the PGM industry in 2024 (the current estimate for the start of Phase 2).

Opportunities for using flash furnace capacity and selling/tolling the resultant matte either locally or offshore remain. However, the technical problem related to the high MgO needs resolution and there may be investment needed in sulphur removal.

Precious metal refining capacity in South Africa could be expanded so as to be sufficient to treat all of the currently planned output from Ivanhoe. The capital required will be small when compared with the BMR investment.

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 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 that may be 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 environmental and social impact studies 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; and
- 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.

Ivanhoe 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 Mineral and Petroleum Resources Development Act (MPRDA), the National Environmental Management Act (NEMA) as well as the EP and IFC Performance Standards;
- Stakeholder Engagement Process (SEP) in accordance with the NEMA 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 Conclusions

1.21.1 Platreef 2014 PEA

The Platreef 2014 PEA compares the results of three phases at scoping level. Further work and studies are required to bring the Project to a Pre-Feasibility Study level. The 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.

1.21.2 Geology and Mineral Resource Estimates

Dr Parker and Mr Kuhl are of the opinion that the Mineral Resources for the 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 2010 CIM Definition Standards.

Mineral Resources have been estimated on an externally undiluted basis and without consideration for mining recovery. Dilution and recoveries will vary with the geometry (dip, thickness, faulting and or irregularities in contacts) of the mineralization 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% to 30%, and mining recoveries range from 70% to 100% using the mining methods considered for evaluation of reasonable prospects of economic extraction.

Permitting, environmental, legal and socio-economic issues taxation and infrastructure considerations that may also impact the Mineral Resource estimates at the Platreef Project are typical of advanced-stage exploration and development projects in Southern Africa. It is the opinion of Dr Parker and Mr Kuhl that there is a reasonable expectation that Ivanhoe and various stakeholders can reach agreement to develop the Project; however, this process may require additional consultation and negotiation and may require some form of compensation payments.

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

- Renewal of the Rietfontein prospecting licence has not been granted. The Mineral Resources amenable to open pit methods on Macalacaskop are not expected to be affected; however, Mineral Resources amenable to open pit methods as declared for Turfspruit and Rietfontein would have to be re-evaluated without a valid prospecting licence on Rietfontein.
- Confirmation that a mining right will be granted for the Macalacaskop and Turfspruit area; the application has been lodged and grant was pending at the Report effective date.
- Assumptions as to the amenability of local communities to allow surface access for Ivanhoe exploration and sampling programmes with appropriate negotiation and compensation.
- Assumptions used to generate the conceptual data for consideration of reasonable prospects of 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 mineralization layers may result in changes to the metallurgical recovery and smelter payable assumptions used to evaluate reasonable prospects of economic extraction.

1.21.3 Mining

The deposit geometry and geotechnical conditions will allow the use of highly productive mechanized mining methods.

The NSR cut-off grades of $100 and $80 used in estimating the PEA inventory are significantly higher than the estimated break-even cut-off grades.

The total PEA inventory estimated for this study is more than adequate to support mining at the production rates considered.

More detailed design and scheduling needs to be performed during the pre-feasibility study (PFS) in order to obtain better estimates of the length of the pre-production period and the time required to ramp the mine up to full production.

The staged approach to mine development and project expansion allows for logical project growth based on actual site experience and understanding of mining conditions.

1.21.4 Metallurgy

Mr Michael Valenta is of the opinion that for the purposes of a scoping study the test work is very comprehensive and considers all the options surrounding a metallurgical complex for the processing of PGMs and nickel and copper. 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 test work and chemical analysis has been conducted with the necessary diligence and accuracy.

The selection of samples submitted for the metallurgical test work has considered the open pit and underground mineralized material and for the purposes of the scoping study are deemed to be sufficient.

During the course of the test work programme Mr Michael Valenta was included in regular feedback meetings and update meetings.

It is the opinion of Mr Michael Valenta that 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 the option of least risk to the project and recommended for this stage of the scoping study.

The proposed flotation circuit is an accurate interpretation of the findings of the flotation test work. The sizing of the various flotation stages is adequate for this level of scoping study and can be refined during the future variability test work.

The proposed modular approach used for Phases 1, 2, and 3 is considered a wise approach for this level of study. The option of introducing a smelter and BMR needs to be evaluated particularly from a timing perspective.

1.21.5 Infrastructure

The availability of power and water to a suitable level for this level of study. The project team has addressed the issues in suitable detail for this level of study. It is however the opinion of Mr Michael Valenta that the availability of water and the timely delivery of the various infrastructure projects must not be taken for granted.

A number of mining projects are in the development phase on the Bushveld Igneous Complex that all require water and power. The risk of over-allocation and under-delivery by the various entities must not be underestimated. Anglo Platinum has announced that the expansion of the Mogolokwena North operation by an additional 600 ktpm is being considered. This will place significant strain on the existing infrastructure and proposed expansions.

The environmental impact of the proposed water projects on the 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.

1.22 Recommendations

1.22.1 Platreef 2014 PEA

The results of the Platreef 2014 PEA suggest that progress to a Pre-Feasibility Study (PFS) can be reasonably justified. It is recommended that Ivanhoe continue to optimise the PFS scope of work and execution plan. The PFS should be based on Phase 1 and evaluation of Phase 2 and Phase 3 scenarios should continue at scoping 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 2014 PEA.

1.22.2 Geology, Exploration, and Mineral Resources

AMEC has recommended a two-phase work programme in support of updates to the geology and Mineral Resource estimates. Each phase can be conducted concurrently, and the results the proposed drilling programme can be incorporated as available in the second work phase. The first phase is estimated at approximately $15 M, the second at $235,000 to $320,000.

Ivanhoe has plans to continue exploration and step-out (resource expansion) drilling in 2014. Drill plans include 95 drillholes (92,800 m). The drilling is targeted in four main areas and there is also a provision for some contingency drilling. The drill programme assumes the all-inclusive drill costs of for 2014 will be US$160/m. The 2014 programme has an overall estimated total budget of $15 M. AMEC considers that this is the first work phase.

Ivanhoe is currently undertaking a comprehensive re-logging of available drill core and reinterpretation of the Project geology and deposit setting. Once this programme is complete, the following second-phase work should be undertaken in support of an updated Mineral Resource estimate:

- AMEC recommends that the geological model is rebuilt to combine the ATS, AMK, and UMT (bulk mineable) resource models into a single model or if not feasible, build separate models that are defined on a common basis. This will put all models on the same litho-stratigraphic and assay (total) basis. In AMEC's experience, rebuilding the model is estimated at approximately $100,000–$150,000.

- As part of this model rebuild programme, AMEC also recommends that Ivanhoe also undertakes a validation of the entire Project database to verify that no errors have arisen during the transfer from the former Fusion database software to the current acQuire database and to ensure that the information supporting the remodeling is acceptable. AMEC estimates this work to be in the range of $45,000–$60,000.

- Mineral Resources should be re-estimated based on the validated database and the updated geological model. The updated estimate should be budgeted at approximately $45,000–$60,000.

1.22.3 Mining Recommendations

Following are Stantec's recommendations for additional work and modifications to the current mine plans during the PFS:

- Mine layouts and designs should be refined and optimized to the extent possible to enable more accurate scheduling and cost estimates.
- Stoping layouts should be prepared in greater detail in order to better estimate and account for both internal and external dilution and overall mining recovery factors.
- 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.
- More detailed development and production schedules should be prepared to obtain better estimates of the length of the pre-production period and of the time required to ramp the mine up to full production.
- Capital and operating cost estimates specific to the Platreef Project should be developed using first principles in order to refine project cost estimates. Labor rates and materials and equipment costs should be updated to better reflect local South African costs.
- Consideration should be given to sinking Shaft 3 concurrently with Shafts 1 and 2 and equipping it for temporary hoisting of development muck. Use of Shaft 3 as a temporary development platform could help shorten the pre-production period significantly.
- Additional geotechnical data should be collected in the rock units in the footwall of the deposit to assess ground conditions there. Current designs assume that a large portion of the underground infrastructure will be located in the footwall units.
- Shaft 4 should be moved to the west to allow for earlier mining of longhole stoping resources located within the area of the current shaft pillar.
- Alternative types and sources for backfill should be evaluated for the time period between production start-up and commissioning of the paste backfill plant.
- Additional ventilation and refrigeration studies need to be conducted as part of the PFS.

1.22.4 Metallurgical

The metallurgical test work thus far has been comprehensive and has met the requirements of the scoping study. It has however confirmed a number of potential risks that need to be addressed in the subsequent stages of the project. These include:

- The fact that the mineralized material is considered to be very hard;
- The modest concentrate grades and the effect of chasing higher grades on the PGE recovery;
- The seemingly complex reagent suite and the requirement of lengthy condition time in order to be able to meet the recovery and grade targets.
- The variability of the mineralized material grade and process recoveries;
- The need to include more of the footwall into the mill feed.

Further test work is recommended on compiste samples to address the first four risks in the earlier feasibility studies. Variability test work in relation to the mining plan should be considered at a later stage of the project once a better understanding of the mineralized material characteristics has been developed.

The proposed metallurgical plant design encompasses the major findings of the test work; however, the operability and interaction of the various stages needs to be considered in the later stages. A better understanding of the mineralized 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 buildup of floatable contaminants in the flotation circuit.

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

The metallurgical test work and the proposed concentrator design are at a suitable stage to justify that the project should progress to a pre-feasibility stage.

Alternative comminution circuits need to be considered once further comminution results become available. The application of alternative comminution technology can also be considered in light of the successes achieved on other operations in the vicinity of the project. The need to install a concentrate filter in Phase 1 needs to be critically assessed.

1.22.5 Infrastructure

The progress with the various infrastructure projects needs to be continuously monitored. A number of mining projects are in the feasibility stage within the footprint of the Bushveld Igneous Complex and the risk of over-allocation and under-delivery is a reality.

2 INTRODUCTION

Ivanhoe is a mineral exploration and development company, whose principal properties are 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 90% interest in South African Prospecting Right LP30/5/1/1/2/872PR, while a 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) (the Japanese Consortium) holds the remaining 10% interest. 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 (the Platreef Project). Holdings in the Platreef Project are through South African subsidiary, Platreef Resources (Pty) Limited (Platreef Resources).

For the purposes of the Platreef 2014 PEA, the name Ivanhoe refers interchangeably to, Ivanhoe Mines Ltd., the predecessor company named Ivanhoe Nickel and Platinum Limited and to Ivanplats Limited. The subsidiary company Platreef Resources was formerly named African Minerals.

2.1 Terms of Reference

The Platreef 2014 PEA is an Independent NI 43-101 Technical Report for the wholly-owned Platreef nickel–copper–gold–platinum group element (PGE) project (the Project) located near Mokopane, in the Limpopo Province of the Republic of South Africa.

The Platreef 2014 PEA is a Preliminary Economic Assessment with an effective date of 25 March 2014 that has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The following companies have undertaken work in preparation of the Platreef 2014 PEA:

- OreWin: Overall report preparation, general and administration costs, financial model.
- AMEC: Mineral Resource estimation.
- SRK Consulting Inc.: Mine geotechnical recommendations.
- Stantec Inc.: Underground mine plan.
- Geo tail: Tailing Storage Facility.
- Metallicon Process Consulting (Pty) Ltd: Process Engineering and Infrastructure.

The Platreef 2014 PEA uses metric measurements. The currency used is 2014 United States dollars ($).

2.2 Qualified Persons

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

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining was responsible for: Sections 1.1 to 1.5, 1.19, 1.20, 1.21.1, 1.22.1 ; Sections 2; Section 3; Section 15; Section 16.2.2.1; Section 19; Section 20; Section 21.1, 21.2, 21.3, 21.10; Section 22; Section 23; Section 24; Section 25.1; Section 26.1 and 26.6 and Section 27.
- Dr Harry Parker, SME Registered Member (2460450), Technical Director, AMEC E&C Services Inc was responsible for: Sections 1.6 to 1.14, 1.21.2, and 1.22.2; Section 2; Section 3.1, 3.2, 3.3; Section 4; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.8, 10.9, 10.10.1; Section 11; Section 12; Section 14; Sections 25.2 and 25.3; Section 26.2 and Section 27.
- Timothy Kuhl, SME Registered Member (1802300), Principal Geologist, AMEC E&C Services Inc was responsible for: Sections 1.6 to 1.14, 1.21.2, and 1.22.2; Section 2; Section 3.1, 3.2, 3.3; Section 4; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.8, 10.9, 10.10.1; Section 11; Section 12; Section 14; Sections 25.2 and 25.3; Section 26.2 and Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting Inc., as Corporate Consultant, was responsible for: Sections 2, Section 3; Section 16.1.
- Mel Lawson, B. Eng. (Mining), SME Registered Member (1859650), Mining Principal, Stantec Consulting International LLC was responsible for: Section 1.16, Section 1.21.3, Section 1.22.3, Section 2, Section 3, Section 16.2.1, Section 16.2.2, Section 16.2.2.2, Section 16.2.2.3, Section 16.2.3 to 16.2.7, Section 21.4, Section 21.7, Section 25.4, Section 26.3, Section 27.
- Guillaume de Swardt, P. Eng., Eng Council Sth Africa (950429) Director, Geo Tail (Pty) Limited was responsible for: Sections 2, Section 3; Sections 18.16; Section 21.6 and 21.9.
- Michael Valenta, ECSA P.Eng (970402), Pr. Eng (Int) (200360005), FSAIMM (55644), Managing Director Metallicon Process Consulting (Pty) Ltd. was responsible for: Section 1.15, 1.17, 1.18, 1.21.4,1.21.5, 1.22.4, 1.22.5, Sections 2, Section 3, Section 5, Section 10.7, 10.10.2; Section 13, Section 17, Section 18.1 to 18.15, Section 21.5, 21.8, Sections 25.5 and 25.6; Sections 26.4 and 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; and again on 8 November 2012. 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 Project site from September 2001 to September 2003, in 2009, 2010, 2011, and most recently from 16 to 21 November 2012. During the site visits, Dr Parker personally inspected core and surface outcrops, drill platforms, and sample cutting and logging areas; held discussions on geology and mineralization 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. During these trips, he audited drill data obtained since AMEC's 2007 database audit (DaSilva, 2007), obtained QA/QC data, field checked drill collars, and collected witness samples for check assays. He also inspected core, surface outcrops, and sample cutting and logging areas. Discussions were held with Ivanhoe's staff about project geology and mineralization; geological interpretations were reviewed, and potential locations of major infrastructure were viewed.

Mr William Joughin visited the site for one day during 2011 to inspect borehole core prior to reviewing the geotechnical investigation and carrying out the geotechnical design. The SRK team carried out geotechnical logging during June and July 2013 under the supervision of Mr. Greg Dyke of SRK. Mr. Greg Dyke visited the site on 8 of June 20111 and again for three days in July 2011 for quality control purposes. Mr William Joughin has subsequently visited the site for two days on 23 and 24 May 2013 to inspect borehole core.

Mr. Mel Lawson participated in the Platreef Project kick-off 07–09 November 2012 in South Africa, which included meetings in Ivanhoe's Sandton, Johannesburg offices with Ivanhoe staff and project consultants. A site visit on 08 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. Mr Guillaume de Swardt visited the Platreef property during a site visit conducted on 27 October 2011.

Mr Valenta has not visited the Project site, but did visit the Mintek laboratory where the current metallurgical test work is underway.

2.4 Effective Dates

There are a number of effective dates, for 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: 3 April 2013.
- Date of the Bikkuri Mineral Resource estimates that are amenable to selective underground mining methods: 8 May 2013.
- Date of the supply of the last drillhole information used in the UMT models: 26 October 2012.

- Date of the supply of the last information on the ongoing drill programme on the Project: 18 February 2014.
- Date of supply of the latest information on surface rights and mineral tenure: 25 March 2014.

2.5 Information Sources and References

Reports and documents listed in Section 3 and Section 27 of the Platreef 2014 PEA 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 information was sourced from Ivanhoe in their areas of expertise as required, providing supporting information for the QPs.

All measurement units used in the Platreef 2014 PEA are metric, and currency is expressed in US dollars unless stated otherwise.

The following abbreviations are commonly used in the metallurgical sections of this report: PGE = platinum group elements; PGM = platinum group minerals; BMS = base metal sulphides. 3E or 3PE denotes Pt + Pd + Au; 3PE+Au denotes Pt + Pd + Au + Rh.

3 RELIANCE ON OTHER EXPERTS

3.1 Project Ownership, Mineral Tenure, Permits and Agreements

Dr Parker and Mr Kuhl, the AMEC QPs, have not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Project area, underlying property agreements or permits. The QPs have fully relied upon, and disclaim responsibility for, information derived from legal experts for this information through the following documents:

- Webber Wentzel 2014: Legal Opinion: The South African Mineral Title held by Platreef Resources (Proprietary) Limited: letter opinion prepared by Webber Wenzel, Attorneys, on behalf of Ivanhoe Mines Ltd., 25 March 2014.

This information is used in Sections 1.6, 4.4, 4.5, 4.8, 4.10 and 25.2 of the Report.

- Webber Wentzel, 2012a: Legal Opinion: The South African Mineral Title held by Platreef Resources (Proprietary) Limited: letter opinion prepared by Webber Wenzel, Attorneys, on behalf of Ivanplats Ltd, BMO Nesbitt Burns Inc., Morgan Stanley Canada Limited, AMEC E&C Services Inc, 7 September 2012.
- Webber Wentzel, 2012b: Plateau Resources (Proprietary) Limited: Prospecting Right 740PR in respect of the Farm Rietfontein 2 KS: letter opinion prepared by Webber Wenzel, Attorneys, on behalf of Ivanplats Ltd, BMO Nesbitt Burns Inc., Morgan Stanley Canada Limited, AMEC E&C Services Inc, 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 Pty 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 Limited, dated 12 September 2010.

This information is used in Sections 4.5, 4.6.2, 4.6.3 and 4.7 of the Report.

The AMEC QPs have viewed information from Plateau Resources in relation to the renewal of the prospecting licence over the Rietfontein Farm through the following document:

- Application In Terms Of Section 18 Of The MPRDA To Renew A Prospecting Right Over The Farm Rietfontein 2 KS-Plateau Resources (Pty) Ltd: letter from Plateau Resources to the Department of Mineral Resources advising of hard-copy renewal application, dated 22 August 2011.

This information is consistent with the information provided by the legal experts on mineral title which is used in Section 4.5.3 of the Report.

3.2 Surface Rights

The AMEC QPs have fully relied upon, and disclaim responsibility for, information supplied by Ivanhoe's staff and experts retained by Ivanhoe for information relating to the status of the current Surface Rights as follows:

- Webber Wentzel 2014: Legal Opinion: The South African Mineral Title held by Platreef Resources (Proprietary) Limited: letter opinion prepared by Webber Wenzel, Attorneys, on behalf of Ivanhoe Mines Ltd., 25 March 2014.
- Ivanhoe Nickel and Platinum Limited and Atlatsa Resources Corporation, 2009: Settlement and New Project Agreement: agreement signed between Ivanhoe Nickel and Platinum Limited and Atlatsa Resources Corporation, effective date 11 December 2009.
- Broughton, D., 2012: Platreef Project: letter prepared by David Broughton, Ivanplats Vice President Exploration for AMEC on the status of mineral tenure and surface rights for the Project, dated 14 June 2012.

This information is used in Section 4.6 and 25.2 of the Report.

3.3 Royalties and Taxes

The assumptions for royalties and taxes have been provided by Ivanhoe and are based on the letter from KPMG to Ivanhoe dated 6 March 2014 subject: Platreef Resources: Updated commentary on specific tax consequences applicable to an operating mine in the Republic of South Africa. Bernard Peters the QP for these assumptions has relied on Ivanhoe and disclaims responsibility for the assumptions and work relating to royalties and taxes presented in Sections 1, 4, and 22.

3.4 Environmental

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

- 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 of the Executive Summary of the August 2003 Environmental Baseline Report for the Platreef Project S0242, September 2007: unpublished report prepared by WSP Walmsley, Sandton, Republic of South Africa for Ivanplats.
- Wessels, B., 2013: Platreef Updated Technical Report: email from Barbara Wessels, Digby Wells Consultant to AMEC providing updates on ongoing environmental studies.
- Field D, 2014, Platreef Hydrogeology Report, 26 March 2014 provided by Platreef.
- Van Wyk & Veermak 2014, Platreef Project: Summary of Progress on Golder Water And Waste Studies, February 2014 By Golder Associates.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

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

4.2 Property and Title in South Africa

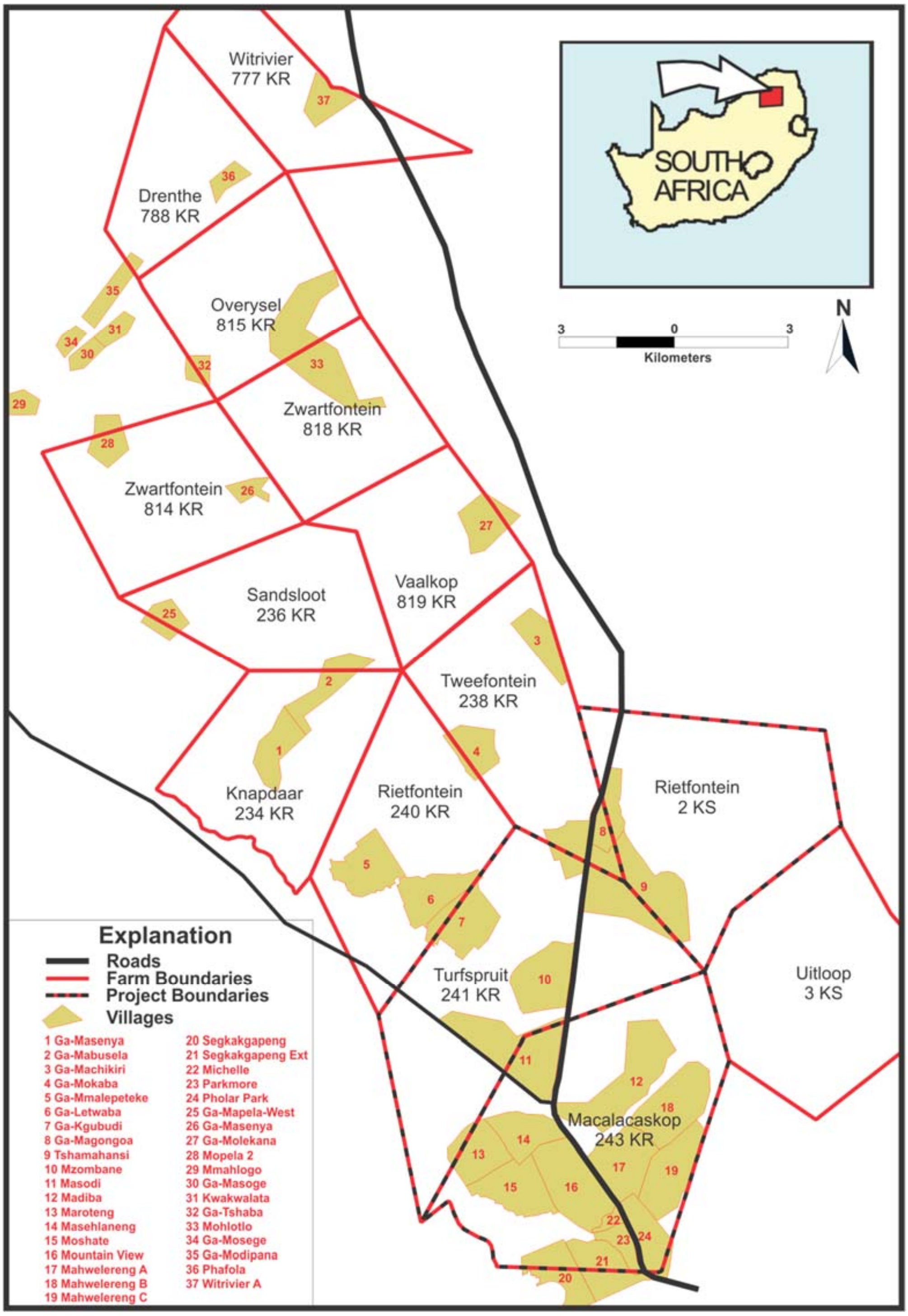
I Information in this sub-section was based on public domain sources, and the country review prepared by Ivanhoe in support of their 2012 Initial Public Offering (Ivanhoe, 2012). The AMEC QPs have not independently verified the information.

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

The Mineral and Petroleum Resources Development Act No 28 of 2002 (MPRDA), 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 MPRDA 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 MPRDA 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 MPRDA. The DMR has discretionary powers for awarding conversions of mining rights from the 1991 Act to the MPRDA. These powers are primarily used in relation to Black Economic Empowerment (BEE) and social-upliftment objectives.

Under the South African Mining Charter of 2004 (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.

Figure 4.1 Project Location and Farm Boundaries



Note: Prospecting licence LP30/5/1/1/2/872PR boundaries correspond to the perimeter boundaries of the Macalacaskop (243 KR) and Turfspruit (241 KR) farms. The boundaries for prospecting licence LP30/5/111/2/740PR correspond to the perimeter of the Rietfontein (2 KS) farm. Collectively, the two prospecting licences form the Project area. Khaki areas on the plan are the main settlements and townships. Figure by AMEC, 2013; data courtesy Ivanhoe.

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 (the "old right") to a 2004 exploration and mining right (a "new right"), 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 MPRDA, to ensure that a mineral resource is optimally exploited. In addition, a rights holder is only entitled to a mining or prospecting right to the extent that the ground holding is actively worked. A planned exploration or mining work programme is required, and must be followed, or corrective measures may be taken by the DMR.

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

4.2.1 Mineral Property Title

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

A mining right is a new-order right issued in terms of the MPRDA 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 mining right can be obtained either by the conversion of an existing old- order mining right, or as a new-order right subject to the exercise of the exclusive right of the holder of a new-order prospecting right, or subject to an application for a new mining right.

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 MPRDA replaced this common-law position, and the 1991 Act was repealed by the MPRDA.

Although the MPRDA 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. The regulations were designed to align the 2006 environmental regulations with the National Environmental Management Act (NEMA), and to streamline the EIA process. Within the regulations, specified timeframes for receipt of Governmental assessment 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 authorization prior to commencement were revised to three 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: contains activities that will only require an environmental authorization 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.

4.3 Republic of South Africa Fiscal Environment

The taxes and royalties that apply to the Republic of South Africa are described in Section 22.3.4.

4.4 Project Ownership

Ivanhoe Mines Ltd. effectively holds 90% of Platreef Resources (Pty) Ltd, directly and indirectly, through an interest in Beales Sàrl. The minority interests held in the Platreef Project and in Beales 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). Figure 4.2 and Figure 4.3 present the Platreef Projects' ownership structure before and after the proposed Broad-Based Black Economic Empowerment (B-BBEE) transaction, respectively.

Ivanhoe is the operator of the Platreef Project.

Figure 4.2 Ownership Structure of the Platreef Project Before The Proposed B-BBEE Transaction

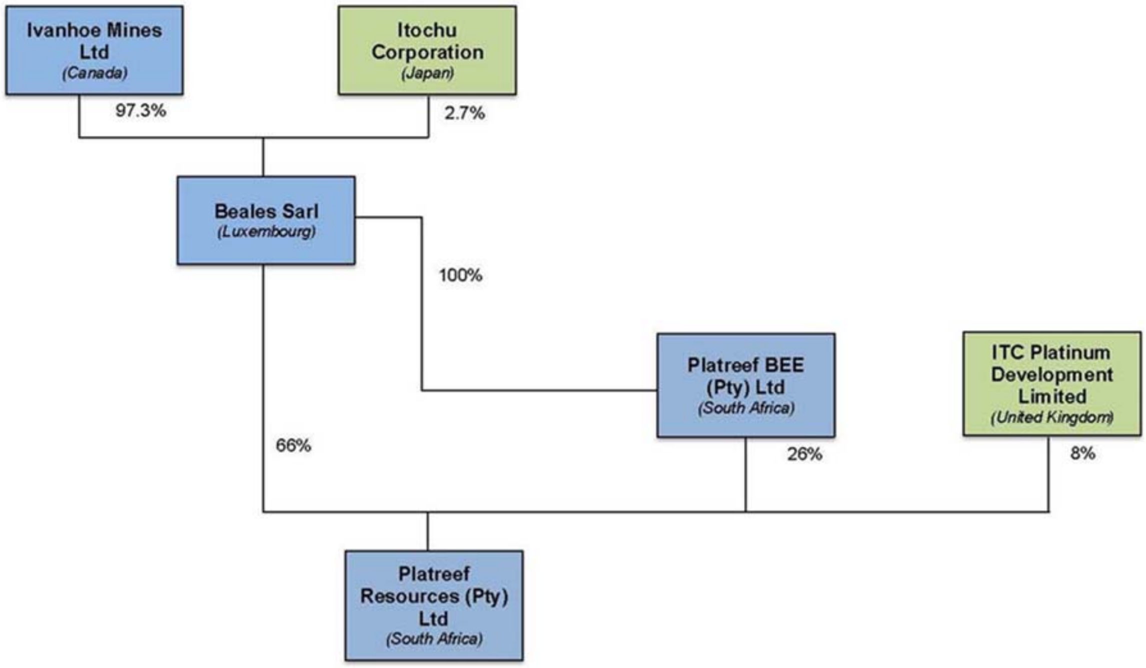
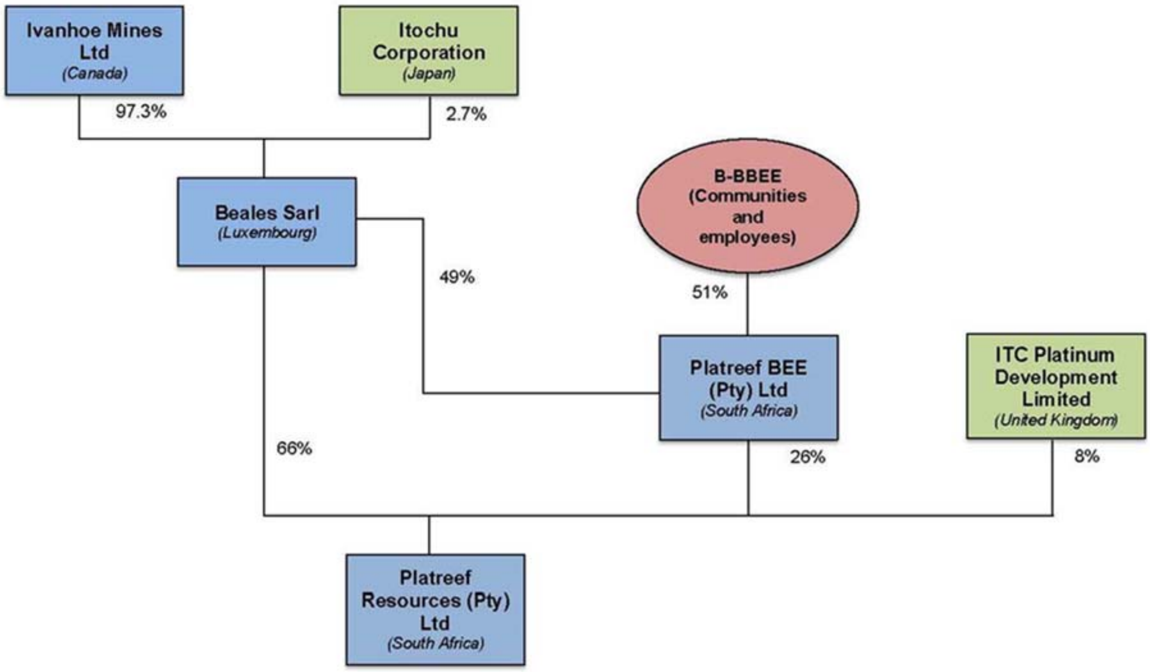


Figure 4.3 Ownership Structure For The Platreef Project After The Proposed B-BBEE Transaction



In October 2010, Itochu acquired a 2% interest in Prospecting Right LP30/5/1/1/2/872PR from Ivanhoe for US$10 million (840 M 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 Platinum Development Ltd for an additional US$280 M (22.4 B yen), and has concluded a Joint Operation and Investment Agreement with Ivanhoe (Itochu, 2011). Consequently, Itochu and ITC Platinum (collectively the Itochu Consortium) holds an aggregated interest of 10% in Prospecting Right LP30/5/1/1/2/872PR; Ivanplats Limited (now Ivanhoe Mines Ltd.) owns the remaining 90%. The Itochu Consortium's cash contribution will be applied to exploration and development activities on the prospecting right.

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 company. Additional information on the Itochu Agreement is included in Section 4.8.1.

Ivanhoe has entered into a series of agreements in order to comply with Section 2(d) of the Mineral and Petroleum Resources Development Act 2002 (MPRDA)

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.

A successful application for a mining right requires Platreef Resources to comply with the Black Economic Empowerment (BEE) requirements of the MPRDA and therefore Ivanhoe entered into a BEE transaction which introduces a 26% HDSA shareholding. The BEE Transaction will empower local communities and employees (the BEE Partners).

4.5 Mineral Tenure

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

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

4.5.1 Prospecting Right No. MPT No. 55/2006 PR (LP30/5/1/1/2/872PR

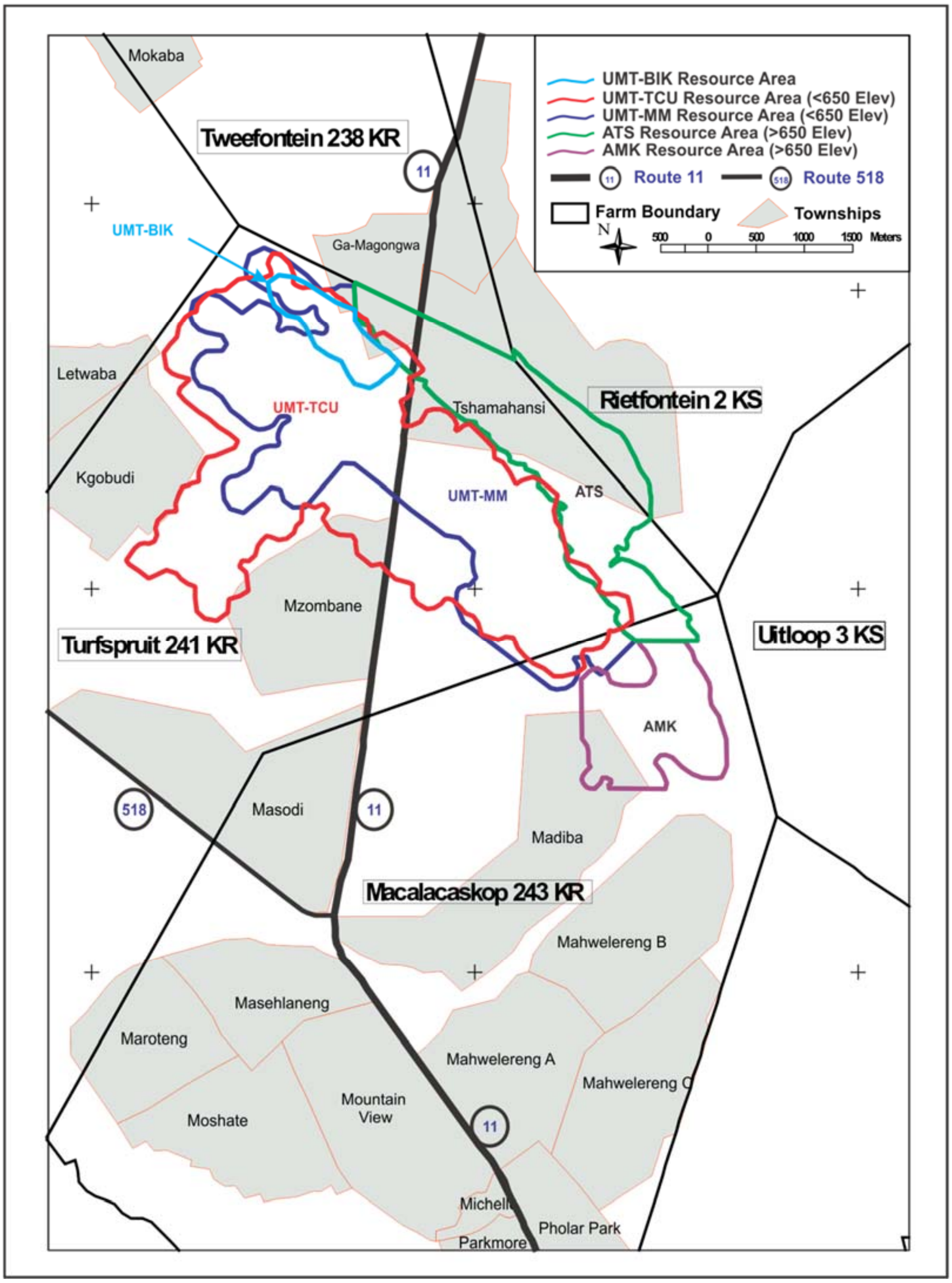
AMEC was supplied with legal opinions and annexes dated 20 November 2006, 21 November 2006, 12 November 2009, 7 September 2012, and 25 March 2014, which reviewed the legal status of the mineral lease K2921/2001 on the Turfspruit Number 241 KR and Macalacaskop Number 243 KR Farms.

These documents support that Platreef Resources (Pty) Limited, registration number 1988/000334/07, a subsidiary of Ivanhoe Mines Ltd holds exclusive prospecting rights to prospect for base and precious metals on farms Turfspruit 241 KR and Macalacaskop 243 KR.

At the outset, these rights were granted in accordance with the 1991 Act. The mineral right became legally effective in October 2002.

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

Figure 4.4 Major Township and Farm Locations



Note: The main road indicated on this plan is Highway N11. The UMT and Bikkuri areas would be mined from underground; the AMK and ATS areas would be mined from open-pits. The boundary of the ATS deposit is constrained by the Turfspruit farm (mineral tenure) boundary, and the northeastern boundary of UMT. Information on figure courtesy Ivanhoe, generated by AMEC 2013.

In terms of Section 18(4) of the MPRDA, the Platreef Prospecting Right may be renewed once for a period not exceeding three years. Platreef Resources made application to renew the Prospecting Right for a three-year extension of term prior to the expiry date. Platreef Resources was notified by the Department of Mineral Resources on 4 May 2011 that the prospecting right 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 MPRDA. The renewal was registered in the Mineral and Petroleum Titles Registration Office on 4 November 2011.

For the title to continue to be maintained, Platreef Resources must pay the required annual title fees and comply with the relevant obligations and work programs relating to its prospecting activities on the prospecting right. Platreef Resources advised AMEC that the required payments have been made as at the Report effective date.

4.5.2 Application for Mining Right over Mineral Prospecting Right MPT No. 55/2006 PR

A Prospecting Right can only be renewed for one three-year period under MPRDA. This renewal has occurred in respect of the Platreef Prospecting Right. To maintain tenure continuity over Mineral Prospecting Right MPT No. 55/2006, Platreef Resources would need to apply for a Mining Right prior to expiry of the Prospecting Right. Subject to complying with the provisions of the Prospecting Right, the holder of a Prospecting Right has the exclusive right, in Section 19(1)(b) of the MPRDA, to apply for and be granted a Mining Right 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 Mining Right in terms of Section 22 of the MPRDA in respect of platinum group metals and all associated metals and minerals mined out of necessity and convenience together with the platinum group metals, over the farms Macalacaskop 243-KR and Turfspruit 241-KR, 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 mining right application consisted of a pack of documents with the three main components being the following:

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

On 17 July 2013, the Department of Mineral Resources (DMR) notified Ivanhoe that it had accepted its application for a mining right on the farms Macalacaskop 243-KR and Turfspruit 241-KR and requested Platreef to comply with inter alia the following:

- To conduct an environmental impact assessment 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 apararent typing error), the DMR issued a revised acceptance letter on 26 August 2013, in terms of the DMR inter alia confirmed that the Mining Right Application had been accepted and directed Platreef to:

- Conduct an environmental impact assessment 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 environmental impact assessment 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 AMEC that the Mining Right Application submitted by Platreef Resources meets all the above requirements.

4.5.3 Prospecting Right No. MPT 76/2007 PR (LP30/5/1/1/2/740PR)

Atlatsa Resources Corporation (Atlatsa; formerly Anooraq Resources Corporation) is a company incorporated in British Columbia, Canada. Its South African interests are held in the name of Plateau Resources Limited, a company incorporated under the laws of the Republic of South Africa.

Plateau Resources legally holds exclusive prospecting rights to prospect for base and precious metals on the farm Rietfontein 2 KS. The mineral lease is identified as Prospecting Right No. MPT 76/2007 PR, which has a DMR reference of LP30/5/1/1/2/740 PR. The Prospecting Right was valid for a five-year period, and was to expire on 27 November, 2011. Prior to the expiry date, on 22 August 2011, Plateau Resources lodged an application to renew the prospecting right for a three-year extension of term.

At the effective date of the Report, the renewal was still pending. Legal opinion provided to AMEC notes that under the terms of section 18(3) of the MPRDA a Prospecting Right in respect of which an application for renewal has been lodged shall, despite its expiry date, remain in force until such time as the application has been granted or refused.

Based on legal opinion, and discussions with Ivanhoe, AMEC considers that, until there is legal opinion to the contrary or formal documentation of refusal of the renewal from the appropriate South African regulatory authorities, it is a reasonable expectation that the renewal will be forthcoming, and that therefore Mineral Resources can be declared on Prospecting Right No. MPT 76/2007 PR.

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, Mountian View, and Michelle Communities community are the lawful occupiers of the Macalacaskop Farm, and the Tshamahansi (Hlongwane, Baloyi and Matjeke), Kgobudi, Masodi, and Magongoa communities Communities are the lawful occupiers of the Turfspruit Farm (see Figure 4.4) Rights to prospect and mine the land are granted by the State.

In terms of Section 5 of the MPRDA, 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 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.

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 AMEC 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.

AMEC notes that there may be some additional community consultations required, as 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.

AMEC considers that, 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 lesae 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 to AMEC that Ivanhoe 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 farm has been claimed by the Mamashela Community, and the claim has been gazetted.

Legal opinion provided to AMEC indicated that in a letter from the DRDLR, dated 16 April 2012, the department confirmed that a claim for restitution has been lodged over the Turfspruit 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 and Turfspruit Farms

The Ivanhoe-controlled farms, Macalacaskop No 243-KR and Turfspruit No 241-KR, are contiguous, sharing a common boundary along the north-west border of Macalacaskop and the south-eastern border of Turfspruit. Macalacaskop contains 4,281 ha of land. Turfspruit contains 3,561 ha of land. The combined total is 7,842 ha.

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

4.6.3 Rietfontein Farm

Rietfontein Farm, No 2 KS, has a contiguous border with Turfspruit Number 241-KR, sharing a common boundary along the south-western border of Rietfontein and the north-eastern border of Turfspruit. Rietfontein Farm has an area of 2,878 ha.

Plot surveys and land area calculations were performed by a Professional Land Surveyor.

The farm was legally surveyed in the past, and the original surveys are on file at the Office of the Surveyor-General of the Limpopo Province (formerly Northern Province) of South Africa.

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 MPRDA, 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; the government is not a community. 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" (the 2009 Agreement), dated 11 December 2009, was concluded between Ivanhoe Nickel and Platinum Ltd (the former name of Ivanplats) and Anooraq Resources Corporation (Anooraq), now known as Atlatsa Resources Corporation (Atlatsa; the holding company of Plateau Resources). The 2009 Agreement superseded and replaced respective rights and obligations of Ivanhoe and Atlatsa 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 Agreement also terminated arbitration and other proceedings, and created a new legal and business relationship between the two parties.

The 2009 Agreement contained the following key elements:

- Anooraq contributed the Rietfontein prospecting right and the Rietfontein 2 KS farm. Ivanhoe contributed a defined portion of the Turfspruit prospecting right and the Turfspruit 241 KR farm. This area became collectively the joint venture "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 companies 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 s 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 governmental 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 the dilution. Alternatively, Anooraq can relinquish its interest in the Property for a 5% net smelter return royalty payable on any mineral products extracted from the Rietfontein prospecting lease.
- A BEE provision will be required to obtain a Mining Right. In this instance, the 2009 Agreement states that Anooraq will not be obliged to reduce its interest in the Property, but that the Ivanhoe’s interest will be reduced. BEE from an ownership perspective 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 Anooraq meets the local ownership requirements for a BEE transaction.

A provision was made within the 2009 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 project 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 the Anooraq Group, 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 the Ivanhoe Mines Group, 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 (b), any proposed underground mine that extracts 100% of its mineral products from Turfspruit would result in a 100% interest for Ivanhoe.

Legal opinion provided to AMEC indicated that while the settlement agreement that was entered into between Ivanhoe Mines and Anooraq/Atlatsa (the holding companies of the South African subsidiaries that hold the prospecting rights through Platreef and Plateau) 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 Prospecting Rights will be required 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 MPRDA.

AMEC considers it a reasonable expectation that at the current stage of Project knowledge, such agreements could be enacted, and that therefore declaration of Mineral Resources on Prospecting Right No. LP30/5/1/1/2/872PR can be supported.

Ivanhoe Mines advised AMEC that an offer has been made to Anooraq/Atlatsa to outright purchase the Anooraq/Atlatsa rights to the Rietfontein farm; Anooraq/Atlatsa were considering the offer as of the Report effective date.

4.8.2 Itochu Agreement

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

On 26 May 2011, Itochu acquired, through its affiliate ITC Platinum Development Ltd, an additional 8% interest in the 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 Joint Operation and Investment Agreement on 6 June 2011, the effective participating interests in the underlying Platreef Project became as follows: Ivanhoe – 90%, Itochu – 2% and ITC Platinum Development Ltd. – 8%.

Under the JOIA, Ivanhoe granted Itochu and ITC Platinum Development Ltd (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.

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 Joint Operation and Investment Agreement 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 AMEC that exploration activities have typically been conducted in compliance with applicable laws in South Africa. The exception is 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 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 MPRDA. 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.

4.10.2 Proposed Bulk Sampling Programme

On 21 September, 2012, Ivanhoe applied for permission to conduct a bulk sampling programme on the Project. The application envisaged an exploration shaft that would allow for collection of an approximate 1,500 t bulk sample from approximately 780 m depth below surface.

On 29 August 2013, the Deputy Director-General, through powers delegated to him by the Minister, granted permission to Ivanhoe to remove and dispose of base minerals and precious metals recovered during the course of prospecting operations in or on the farms Macalacaskop 243-KR and Turfspruit 241-KR (excluding residential areas). The permission to remove bulk samples will lapse upon the lapsing of the Prospecting Right.

On 30 August 2013, the DMR approved Ivanhoe’s amended Environmental Management Plan (EMPlan) in respect of the removal of bulk samples, drilling of additional boreholes and the erection of associated infrastructure, subject to certain terms and conditions.

On 3 January 2014, Ivanhoe was granted an exemption by the Limpopo Provincial Department of Economic Development, Environment and Tourism from compliance with Regulation No. 53(2) of Government Notice No. R543 relating to comprehensive public participation. A number of objections have been advanced as follows:

- Lawyers for Human Rights, acting for and on behalf of the Mokopane Interested and Affected Community Committee lodged a notice of appeal.
- Mr Aubrey Langa, allegedly an individual from the local community, lodged an internal appeal in terms of Section 96 of the MPRDA against the DMR's decision to grant the bulk sampling permission to Ivanhoe; Ivanhoe has responded to the DMR, and the internal appeal is currently pending.

Mr Langa (purportedly acting on behalf of the Masehlaneng Community, the Madiba Community and the Kgobudi Community) applied to High Court of South Africa, Gauteng Division, Pretoria, for an urgent interdict prohibiting Ivanhoe from commencing bulk sampling activities. At the hearing on 19 November 2013, the Court struck the application from the roll for lack of averments to justify urgency. As at the effective date of the 2014 legal opinion, Mr Langa had not re-enrolled the application for hearing.

4.11 Significant Risk Factors

In AMEC’s opinion, there are two significant permitting risks to Project development.

The first lies with any requirement for resettlement of occupants of townships on the three farms. This risk is much less if underground mining operations are conducted versus open-pit operations. The second is the requirement for renewal of the Reitfontein licence.

4.12 Comments on Section 4

In the opinion of the QPs, the information discussed in this section supports the declaration of Mineral Resources. The QPs note the following:

- Information provided by legal experts and Ivanhoe support Ivanhoe's ownership claims to Prospecting Right MPT 55/2006 PR. Ivanhoe applied for a mining right on 6 June 2013, prior to expiry of Prospecting Right MPT 55/2006 PR in May 2014.
- A joint venture with Atlatsa, over the Rietfontein farm is covered under the terms of the 11 December 2009 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 agreement.
- Ivanhoe advised AMEC that Ivanhoe has submitted an offer to outright purchase the rights to Rietfontein from Atlatsa. Atlatsa is still considering the offer.
- The Rietfontein licence renewal application was lodged on 22 August 2011; however, at the Report effective date, the renewal is still pending. Based on discussions with Ivanhoe and legal opinion provided, AMEC considers that, until there is legal opinion to the contrary, it a reasonable expectation that the renewal will be forthcoming, and that therefore declaration of Mineral Resources on Prospecting Right MPT 55/2006 PR and Prospecting Right MPT 76/2007 PR can be supported.
- Rietfontein is critical to open-pit mining on Turfspruit. Underground mining is not contemplated for Reitfontein; however some of the necessary infrastructure to support such activities, such as a tailings dam, may be located within the Reitfontein area.
- If the Prospecting Right on Rietfontein is not renewed, Mineral Resources amenable to extraction by open-pit mining methods on both Rietfontein and Turfspruit would have to be re-evaluated.
- The AMEC QPs note that should an open-pit operation be envisaged, then there will likely be mine disturbances associated with the development and mining of any open-pit projecting beyond the current Project boundary, particularly onto the adjacent Tweefontein Farm (refer to figures in Section 6). 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. An application for a mining right has been lodged for the ground that is held by Ivanhoe under MPT 55/2006 PR. No mining right application has been made for Reitfontein, as renewal of the Prospecting Right has not yet been granted.
- Surface rights within the areas of the Rietfontein, Macalacaskop and Turfspruit Farm areas belong to the national government. AMEC considers that 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 which other communities may lawfully occupy the Rietfontein 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 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 farm; information provided to Ivanhoe by the Department of Rural Development and Land Reform indicates a non-gazetted claim by the Mokopane Tribe over the area covered by Prospecting Right MPT 76/2007 PR.
- 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.
- AMEC notes that there have been instances where drill programmes have been affected by short-term access issues, most recently in 2012. Over the 12+ years 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.
- AMEC considers that, 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.
- Through their actions to date, Ivanhoe has shown their understanding of, and accepts the importance of, proactive community relations, and is continuing to liaise with representatives of the local communities.
- To the extent known, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Project is located approximately 280 km north-east of Johannesburg. Year-round access is by four-lane, paved, all-weather road to Mokopane (formerly Potgietersrus). From Mokopane the access continues as a two-lane, paved, all-weather national highway. The highway passes through the Project. Access to drill sites and other areas within the Project is by gravel all-weather roads or by unpaved tracks.

The closest major international airport is Johannesburg, about a four-hour drive from Mokopane, and the regional hub is at Polokwane (formerly Pietersburg) 56 km to the north of Mokopane.

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 Province. The closest railhead to the Project is in Mokopane.

5.2 Climate

The climate is semi-arid, with precipitation occurring as rain. Average annual rainfall is around 300 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; however, Golder Associates (2011) noted the highest monthly rainfall as 112 mm in January 1923.

High daily temperatures occur throughout the year; the mean maximum monthly temperatures range from 21°C to 33°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 mean minimum monthly temperature ranges from 6°C to 20°C.

Golder Associates (2011) 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.

Electrical energy, telephone service, and other infrastructure components are available in Mokopane. Mokopane's town centre is located approximately 11 km from the centre of the Project. Large-scale infrastructure, such as high-voltage electrical lines and large volumes of water, are situated within moderate distances from the Project. The main line of the national railroad system passes approximately 6 km east of the Project.

5.2.1 Local Labour Resources

There is a moderate level of mining activity within a 100 km radius around the farms. A large, unskilled labour force lives in urban areas near and on the farms. These people can be trained for many job assignments if a project is developed, as is demonstrated by Ivanhoe's employment of approximately 80 staff from the Mokopane area. Some skilled trade positions and professional staff will have to be recruited from outside the area.

Adequate town-site facilities and infrastructure exist to support an influx of personnel. Housing may have to be constructed or subsidised for some positions.

It is the opinion of the qualified person responsible for this section of the report, Mr Michael Valenta, that the labour situation in this region will not pose a challenge if formal training programmes are established ahead of time and the staff are sensitised to the operation prior to production commencing. The maintaining of good relations with the community will ensure minimal labour unrest as the community understand and benefits from the mining operations.

5.2.2 Power Supply

The section of the report considers the current status of the power supply to the area. A more detailed discussion of the power supply and associated infrastructure is given in Section 18.6.

The results of a 2007 study (Pienaar and Erwee) indicated that extension of the existing national power grid transmission and distribution systems from the Eskom substations to the project area was feasible and could be undertaken following completion of an agreement between Eskom and Ivanhoe. An agreement was entered into with Eskom to supply a permanent 70 MVA of power from an expansion of the national grid which will bring an additional high voltage line near the project.

As power is required for the initial mine development (shaft sinking), prior to the completion date of the permanent supply from 4,800 MW Medupi Power Station currently under construction, an agreement for 5 MVA of temporary construction power was concluded with Eskom. This power will be supplied from a local sub-station close to Mokopane.

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

Based on these studies and discussions, Mr Michael Valenta is of the opinion that there is a reasonable expectation that the electrical need for any proposed project development can be met.

5.2.3 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.

The 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 inititives to supply water to the area please refer to Section 18.4.

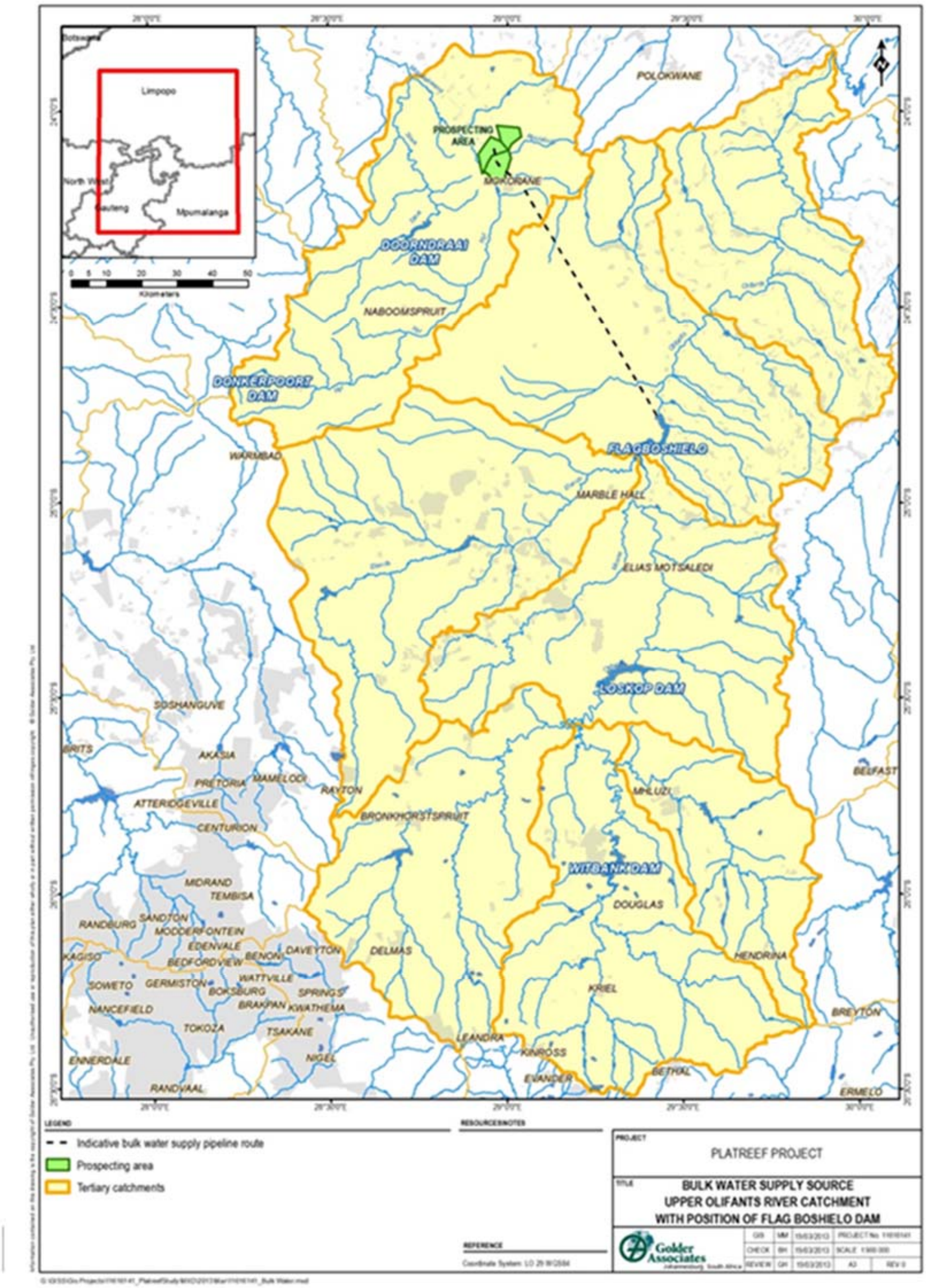
Ivanhoe is a participant in the Olifants River Water Resource Development Project (ORWRDP) which is designed to deliver water for domestic and industrial (mining) purposes to the Eastern and Northern limbs of the Bushveld Complex. Ivanhoe is also a member of the Joint Water Forum (JWF) which facilitates and co-ordinates 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 capacity required 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 and other projects (Figure 5.1). Ivanhoe'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 Project area. Ground water sources have been identified, and Ivanhoe has applied for water-use licenses from the DWA (Figure 5.2).

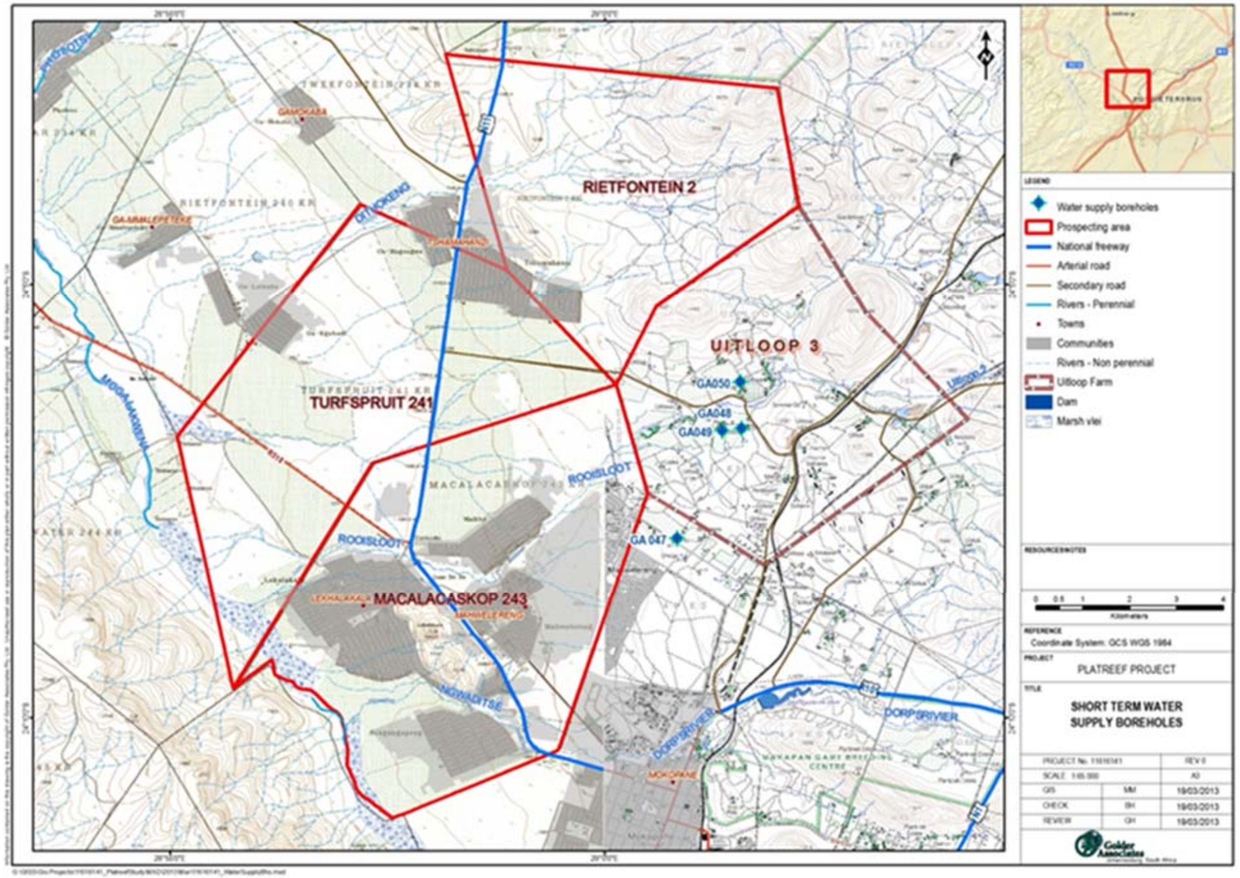
It is the opinion of Mr Michael Valenta that adequate planning and discussions have been ongoing in ensuring a supply of water for the project and future expansion. It is reasonable to expect that adequate water will be available and the necessary infrastructure will be in place to supply such.

Figure 5.1 Location Plan Flag Boshielo Dam and Proposed Water Pipeline



Note: Figure courtesy Ivanhoe, 2013.

Figure 5.2 Water Bore Location Plan



Note: Figure courtesy Ivanhoe, 2013.

5.2.4 Highway Re-Alignment

The N11 national highway connects Mokopane with the South Africa/Botswana border. The current road runs directly through the Turfspruit and Macalacaskop farms, and serves the operating Mogalakawena (formerly Anglo Platinum PPL) mine and the Lonmin's Akanani Project. The road may need to be re-aligned away from the footprint of any future open-pit. Re-alignment is not expected to be required for underground operations.

5.3 Physiography

The Rietfontein, Macalacaskop, and Turfspruit 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 farm, adjacent to Turfspruit. Figure 5.3 is a photograph taken in the Project area illustrating the general topography.

The elevation on the farms ranges from a maximum of about 1,140 metres above seal level (masl) in northern Turfspruit to about 1,060 masl on Macalacaskop.

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.

5.4 Sufficiency of Surface Rights

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

Figure 5.3 Project Physiography

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

6 HISTORY

During the 1970s, regional exploration was undertaken over 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 and Macalacaskop 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 2014 PEA.

Ivanhoe acquired a prospecting permit for both Turfspruit and Macalacaskop farms in February 1998, and subsequently Ivanhoe entered into a joint venture (JV) with Atlatsa over the Rietfontein 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 2002 to 2013.

Ivanhoe contracted Hatch Engineering in 2003 (Hatch: 2003a, 2003b, 2003c, 2003d; Matyas, 2003) to provide a conceptual-level study for a ‘greenfield’ Ni–PGE concentrator/smelter. The study considered the smelter would treat an average of 1,200 t/d of concentrate. An update to the report assessed smelting of 1,850 t/d of concentrate.

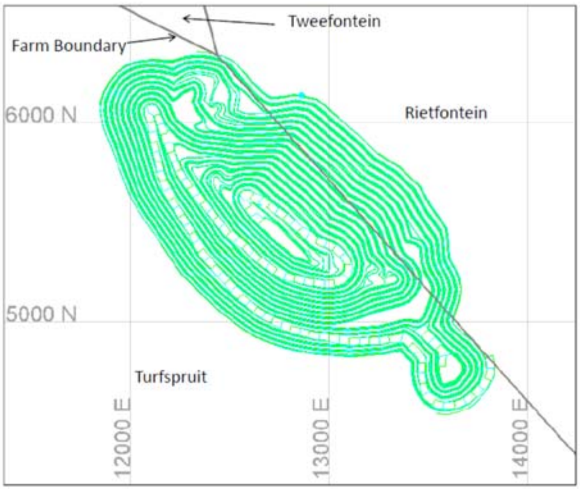
Between June 2003 and December 2003, Mineral Development Services Ltd. (MDS) of South Africa (Lawrence, 2003) reviewed metallurgical testwork undertaken to June 2003, developed conceptual flow sheets, and prepared mechanical equipment lists for a concentrator. MDS (2003, 2004) also developed preliminary (±35%) capital and operating cost estimates.

In 2003 to 2004, Ivanhoe completed a second overall study on the Platreef Project. This internal study (African Minerals, 2004) was used to assess the amenability of the deposit to open-pit mining (AMEC, 2003a, 2004a, 2004b). Results of the work indicated that the mineralization on the Turfspruit/Rietfontein farms was more likely to support a mining operation than the mineralization on the Macalacaskop farm.

In 2003 to 2004, Ivanhoe completed a second overall study on the Platreef Project. This internal study (African Minerals, 2004) was used to assess the amenability of the deposit to open-pit mining (AMEC, 2003a, 2004a, 2004b). Figure 6.1 and Figure 6.2 show three-dimensional views of the conceptual ultimate pits. Results of the work indicated that the mineralization on the Turfspruit/Rietfontein farms was more likely to support a mining operation than the mineralization on the Macalacaskop farm.

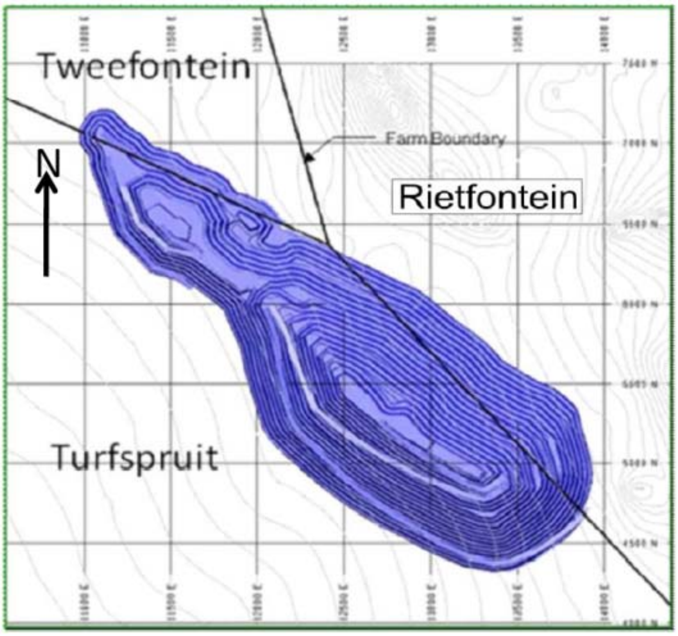
The AMEC QPs, Dr Parker and Mr Kuhl, consider the studies to be useful as background support when considering reasonable prospects for economic extraction for open-pit Mineral Resources in Section 14 of this Report.

Figure 6.1 Conceptual Pit Designed to Depth of Approximately 500 m



Note: Figure prepared by AMEC, 2003.

Figure 6.2 Conceptual Pit Designed to Depth of Approximately 560 m



Note: Grid squares on plan are 500 m x 500 m; Figure prepared by AMEC, 2003.

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 mineralization down-dip within the Turfspruit 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 mineralization 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 mineralization considered amenable to open-pit and underground mining methods was publicly disclosed in Parker et al. (2012). A Mineral Resource estimate update assuming selective and mass-mineable underground mining methods was prepared in March–April 2013. Estimates for the Bikkuri Reef were prepared in May 2013.

7 GEOLOGICAL SETTING AND MINERALIZATION

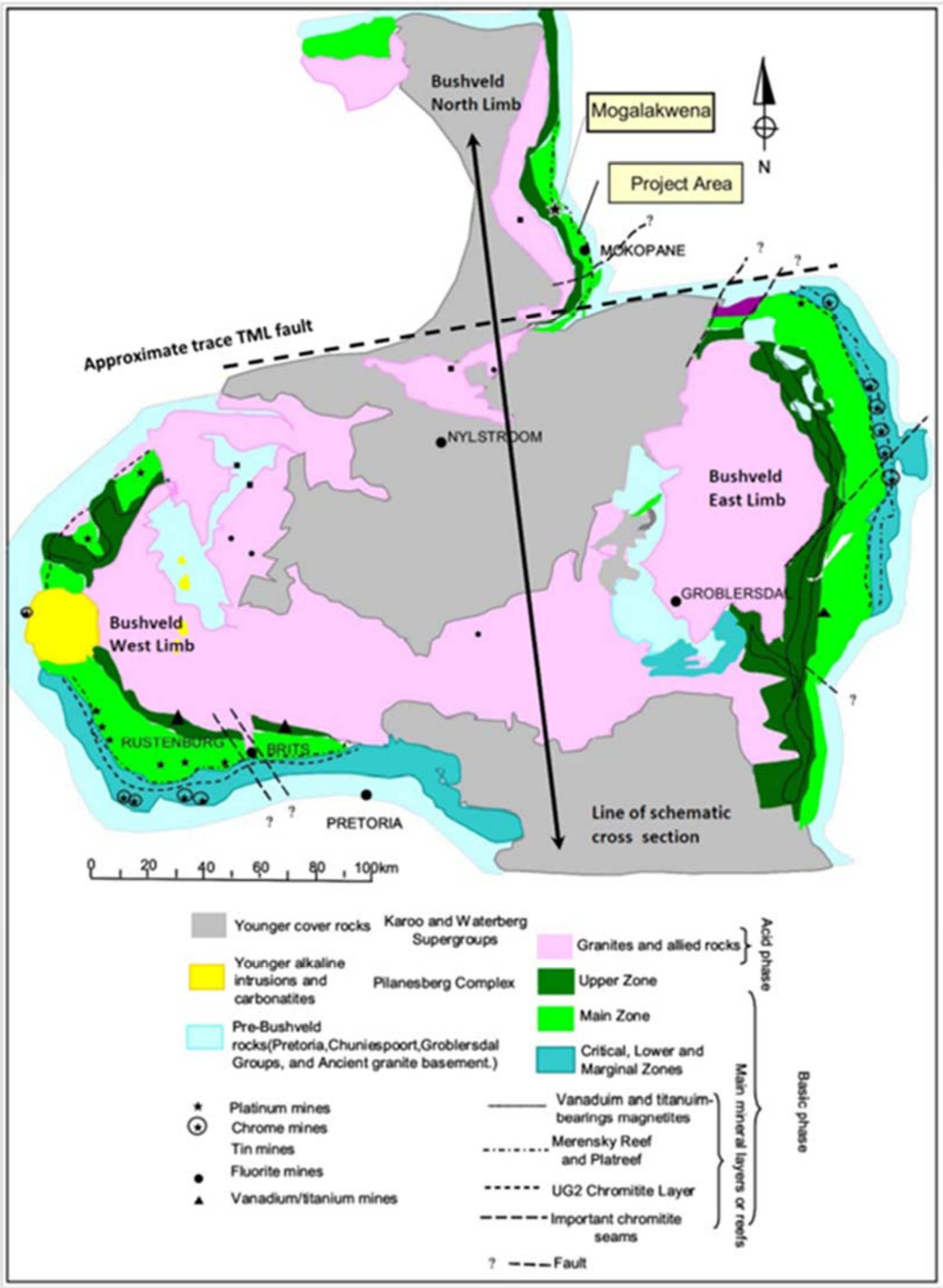
Information on the Project regional setting and context is summarized from Parker et.al. (2012, 2013). The project-level geology section has largely been prepared by or draws on the work of Ivanhoe personnel, specifically Dr. Danie Grobler and Shane Neilsen.

7.1 Regional Geology

Within the Kaapvaal Craton, the ~2.06 Ga Bushveld Igneous Complex (BIC) intrudes the Transvaal Supergroup, forming the world's largest layered intrusion and the world's largest source of platinum (Cawthorn, 1999). The Eastern and Western Limbs of the BIC are well exposed and researched, whereas understanding of the Northern Limb, host to the Platreef Project, is still evolving.

Typically the BIC consists of a mafic–ultramafic layered suite, a granite suite, and a package of predominantly felsic volcanic rocks. The Rustenburg Layered Suite (RLS) hosts major deposits of platinum group elements (PGEs), chromite, vanadium and nickel (Figure 7.1).

Figure 7.1 Regional Geological Plan of the Bushveld Complex



Note: Modified after Viljoen and Schürmann, 1998, section line represents location of section in Figure 7.2.

The suite has been subdivided into a number of zones described from the basal units to the top; descriptions are abstracted from Kinnaird et al., (2005), and Cawthorn (1999 and 2005):

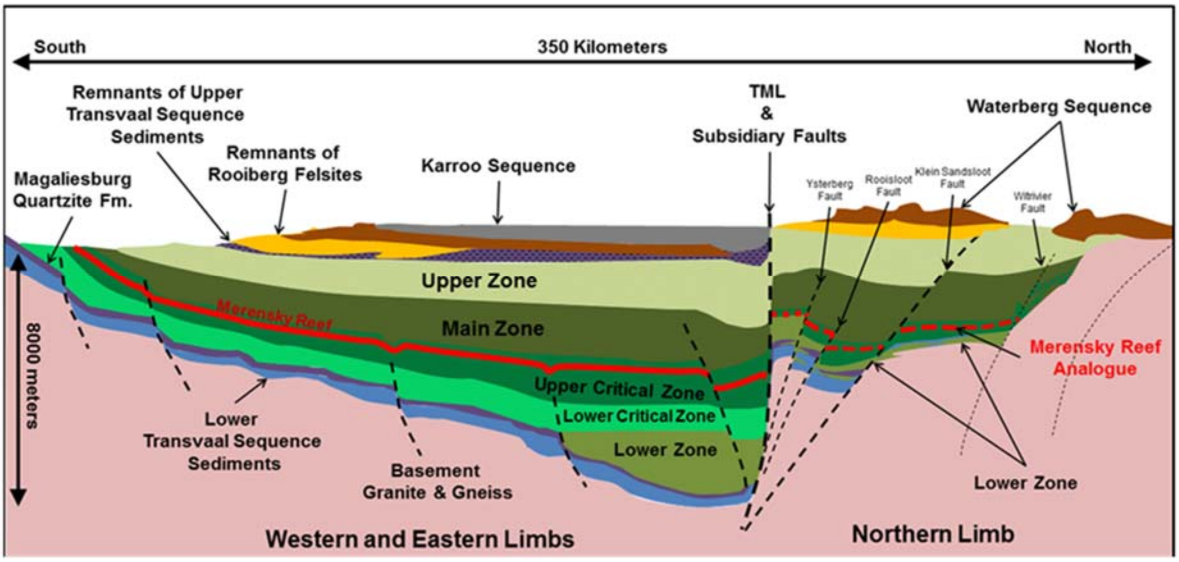
- The Marginal Zone (MZN) — Norites with variable proportions of accessory clinopyroxene, quartz, biotite and hornblende, indicating magma contamination from the underlying sediments. This unit is not always present.
- Lower Zone (LZ) — upper and lower peridotites separated by a central harzburgite.
- Critical Zone (CZ) — Lower Critical Zone comprises orthopyroxenitic cumulates; Upper Critical Zone consists of 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 being the Merensky and the Upper Group 2 (UG2) reef of the Eastern and Western Limbs. These range on average from 0.4 to 1.5 m thickness, and PGEs typically range from 4 to 10 g/t (Cawthorn, 2005).
- Main Zone (MZ) — a succession of gabbronorites with occasional anorthosite and pyroxenite bands.
- Upper Zone (UZ) — gabbroic succession.

7.2 Northern Limb

The Northern Limb (formerly Potgietersrus Limb) of the BIC hosts the Platreef mineralization on the Project. Figure 7.2 shows a schematic section through the BIC that has been modified by Ivanhoe to illustrate the interpreted Merensky Reef analogue within the Project area.

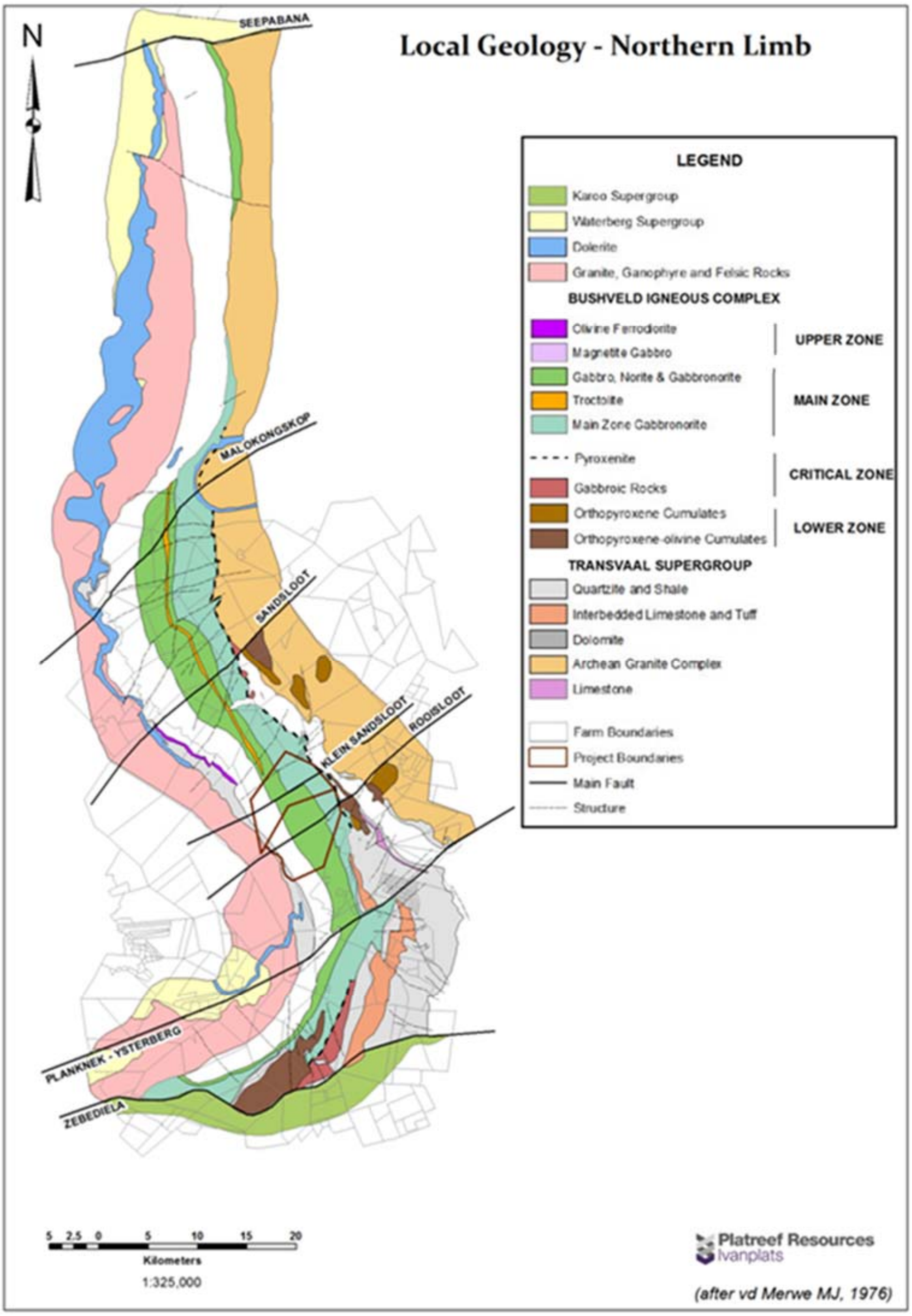
Structurally, the Northern Limb is separated from the rest of the BIC by a craton-wide linear structure called the Thabazimbi–Murchison Lineament (TML). In the area around Mokopane the near-surface expressions of the lineament are the Zebediela and Planknek–Ysterberg faults (Figure 7.3). In broad terms, Upper Zone and Main Zone rocks in the Northern Limb are similar to those in the Eastern and Western Limbs across the TML, but the underlying PGE-bearing mafic-ultramafic rocks of the RLS in the Northern Limb are much thinner, and form a poorly understood composite intrusion with "contact-style" mineralization adjacent to the floor rocks, termed the Platreef. Recent work by Ivanhoe's geologists provides evidence for a detailed stratigraphic correlation between the Platreef and its mineralized zones, and the well-known mineralized stratigraphy of the Upper Critical Zone and its Merensky and UG2 reefs, south of the TML.

Figure 7.2 Schematic Cross-Section Through Bushveld Igneous Complex



Note: Figure 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 Geological Plan of the Northern Limb of the BIC



Note: Figure courtesy Ivanhoe, 2013, modified after van der Merwe (1976).

7.2.1 Lithologies

South of the Planknek-Ysterberg fault, the lower part of the Northern Limb stratigraphy and its mineralized zones have been correlated with Upper Critical Zone (UCZ) rocks (Maier et al., 2008).

North of the Planknek-Ysterberg fault, PGE–Ni–Cu mineralization occurs within the Platreef, a moderately to steeply west-dipping, <100 m to 200–400 m thick, composite mafic–ultramafic intrusion emplaced into Transvaal Supergroup metasedimentary rocks and (to the north) into Archean granite–gneiss (Kinnaird, 2005).

Crude internal stratigraphy within the Platreef has been documented but has been difficult to correlate between properties. Where present it takes the form of a lower, usually heterogeneous and less well mineralized zone of variable lithologies including pyroxenite, harzburgite, norite, gabbronorite and xenoliths or rafts of variably digested and metamorphosed Transvaal Supergroup rocks; an overlying well-mineralized zone comprised mainly of pyroxenite and locally harzburgite; and an upper, generally poorly mineralized pyroxenite (Kinnaird, 2005 and references therein; Comline et al., 2007). Interpretation of Platreef lithologies, stratigraphy, and mineralization is hampered by widespread contamination of the mafic-ultramafic magmas through the digestion/assimilation process, particularly in the lower heterogeneous (contaminated) zone. Previous subdivision of the mineralized upper Platreef in the Project area (Parker et al., 2012) distinguished an Upper Top Loaded Zone (UTLZ) and a Lower Top Loaded Zone (LTLZ), mainly within the B-Pyroxenite (BP).

In 2012, Ivanhoe staff demonstrated that high-grade PGE–Ni–Cu mineralization is consistently hosted within an unconformable, non-cumulate, pegmatoidal, mafic to ultramafic sequence, commonly bound by chromitite stringers and containing coarse-grained to pegmatoidal sulphides. Overlying this pegmatoidal package is a barren feldspathic pyroxenite unit of variable thickness. A separate mineralized zone, of disseminated, medium- to coarse-grained sulphides, is perched near the top of this non-pegmatoidal feldspathic pyroxenite. This mineralization was previously identified as the Upper and Lower Top-Loaded Zones.

Although generally thicker, these two zones share many similarities with the M1 and M2 mineralized zones of the Merensky Reef (Davey, 1992; Lea, 1996), and the host sequence of lithologies corresponds directly to the Merensky Cyclic Unit (MCU) (Grobler et al., 2012). Hangingwall and footwall lithologies correspond to those described for the Merensky Reef in the northern part of the Western Limb (Viljoen 1994 and 1999; Viljoen et al., 1986a; Viljoen et al., 1986b; Viring and Cowell 1999).

7.2.2 Structure

Emplacement of the BIC is generally considered to be associated with anorogenic magmatism caused by intracratonic rifting (Friese, 2012). The Eastern Limb is compartmentalized by several north-east-trending faults, whose map patterns indicate they influenced the development of the Marginal, Lower, and Critical Zones.

In the Northern Limb, similar compartmentalization is suggested by the presence of north-east to north-south faults that appear to separate "sub-basins" with different internal stratigraphy (Figure 7.2 and Figure 7.3). Details of this require a better understanding and correlation of stratigraphy within the Northern Limb. Pre-Bushveld fold structures may also have influenced the development of the Platreef and, in general, of the Northern Limb stratigraphy (Friese, 2003; Friese and Chunnett, 2004; Nex, 2005).

7.2.3 Mineralization

The discussion in this section reflects the generally-held views in the geological community. Work by Ivanhoes' geologists during 2012–2013 led to the revised provisional mineralization interpretation for the Platreef Project, as described in Section 7.3.2. Geological studies and interpretations remain ongoing and the interpretations as presented in Section 7.3.2 may change.

The Platreef, which extends northward from Mokopane for at least 30 km, is defined as a Ni–PGE–Cu–Au-bearing mafic–ultramafic package with a hangingwall of MZ gabbronorite and a footwall of Transvaal Supergroup meta-sedimentary rocks in the south and Archaean granite gneiss in the north (Kinnaird, 2005). This definition includes the mineralization on the Project, but excludes mineralization in Main Zone gabbronorites north of the Drenthe farm, or south of Mokopane where mineralization occurs in recognizably layered rocks of the Critical Zone (Kinnaird, 2005, Maier et al., 2008).

Sulphide mineralization concentrated in the lower parts of the Platreef typically has Pt/Pd ratios of less than 1, whereas the upper portion of the Platreef may have ratios >1, and locally >2. PGE studies on the Platreef by Kinnaird et al., (2005) noted a poor correlation between PGE and sulfur. The study concluded that PGE mineralization was not simply controlled by segregating sulphide melt but rather the addition of semi-metals to the magma as a result of contamination. This produced a complex PGE assemblage including tellurides, bismuthides, and arsenides, in addition to PGE–sulphides.

Chromitite layers or stringers have been observed throughout the strike length of the Platreef, but tend to be discontinuous (White, 1994; Holwell and McDonald, 2006; Yudovskaya and Kinnaird, 2010). Platreef chromitites have been interpreted as 'chromite-rich zones' in pyroxenite, rather than continuous, persistent stratigraphic layers as in the Critical Zone south of the TML (McDonald and Holwell, 2011).

Correlation of the Platreef mineralized zones with the reefs of the Upper Critical Zone south of the TML has been suggested (Vermaak and van der Merwe, 2002; Maier et al., 2008) but remains contentious. Maier et al., (2008) argued that Platreef-type mineralization relates to a marginal setting whereby mineralization is focused along the margin of the Bushveld intrusion, at the base of the Rustenburg Layered Suite in a 'contact-style' manner.

7.3 Project Geology

7.3.1 Overview

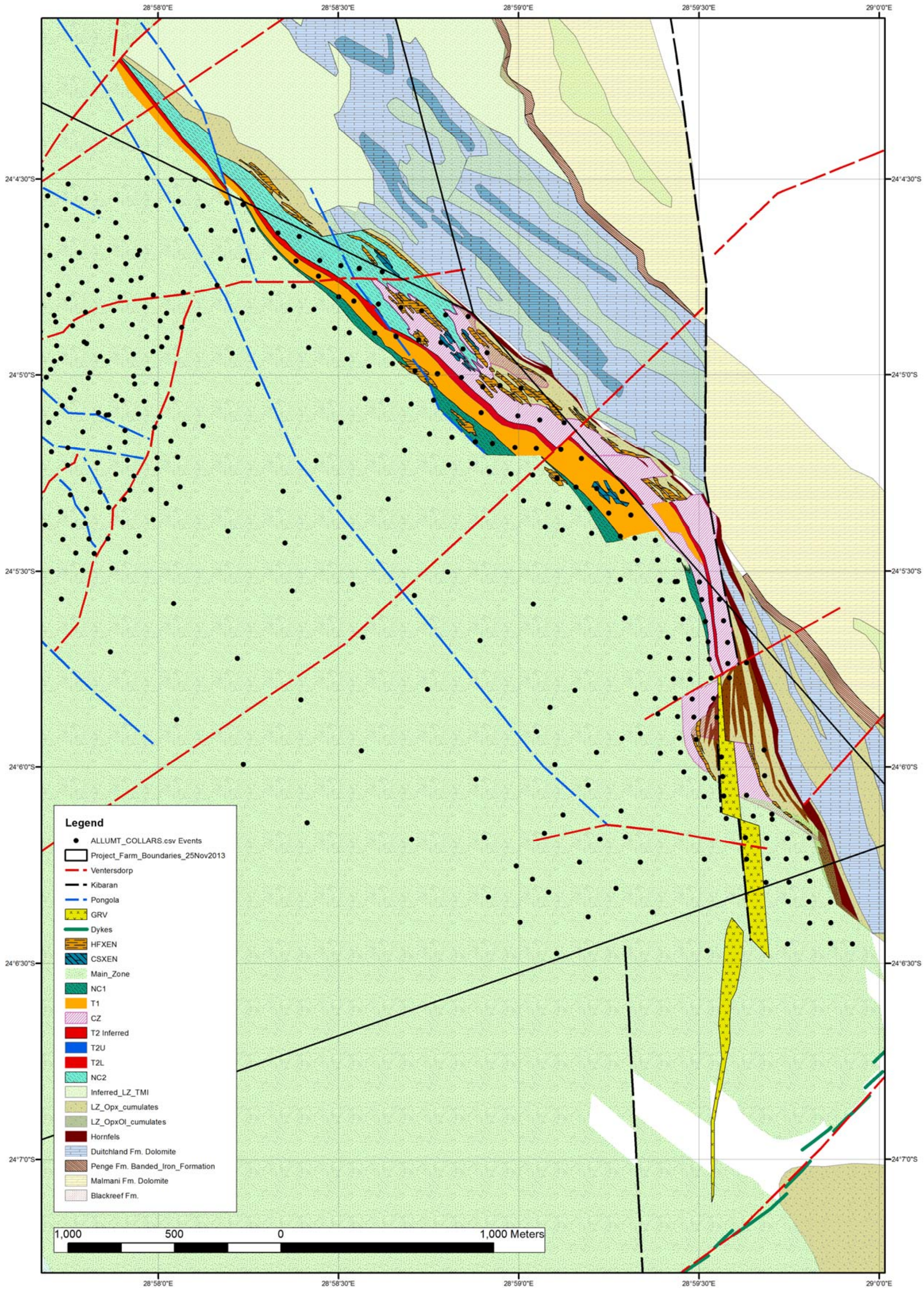
The generalized geology of the Project area is shown in Figure 7.4. Historically, the bulk of scientific and exploration work on the Bushveld's Northern Limb has been performed on the near-surface, generally steeply dipping, part of the Platreef that is potentially amenable to open-pit mining methods.

The floor of the Platreef (contact with Lower Zone or Transvaal) shows steep dips to depths of 900 m (elevation 200 masl). There is a strong right-angle bend at the south end of Figure 7.5, reflecting a possible fault off-set between the Turfspruit and Macalacaskop basins of Kinnard (2005). Below the 200 m elevation is a zone of more gentle dips with local discontinuities that probably also reflect faulting. This area of near constant elevation has been called the Flatreef.

As shown in Figure 7.6, the top of the Platreef (contact with Main Zone) broadly reflects the topography of the floor contact, but is smoother, particularly the Flatreef (G1). There are two depressions that may have been influenced by the floor contact (G3, G4); these are the locus of potholes (see discussion in Section 7.3.5) with thickened TCU. The G2 area does not appear as a depression related to the floor contact, but there is a slight sag in the top contact (see Figure 7.6) and thickened TCU below (see Figure 7.16, Figure 7.20, and Figure 7.21).

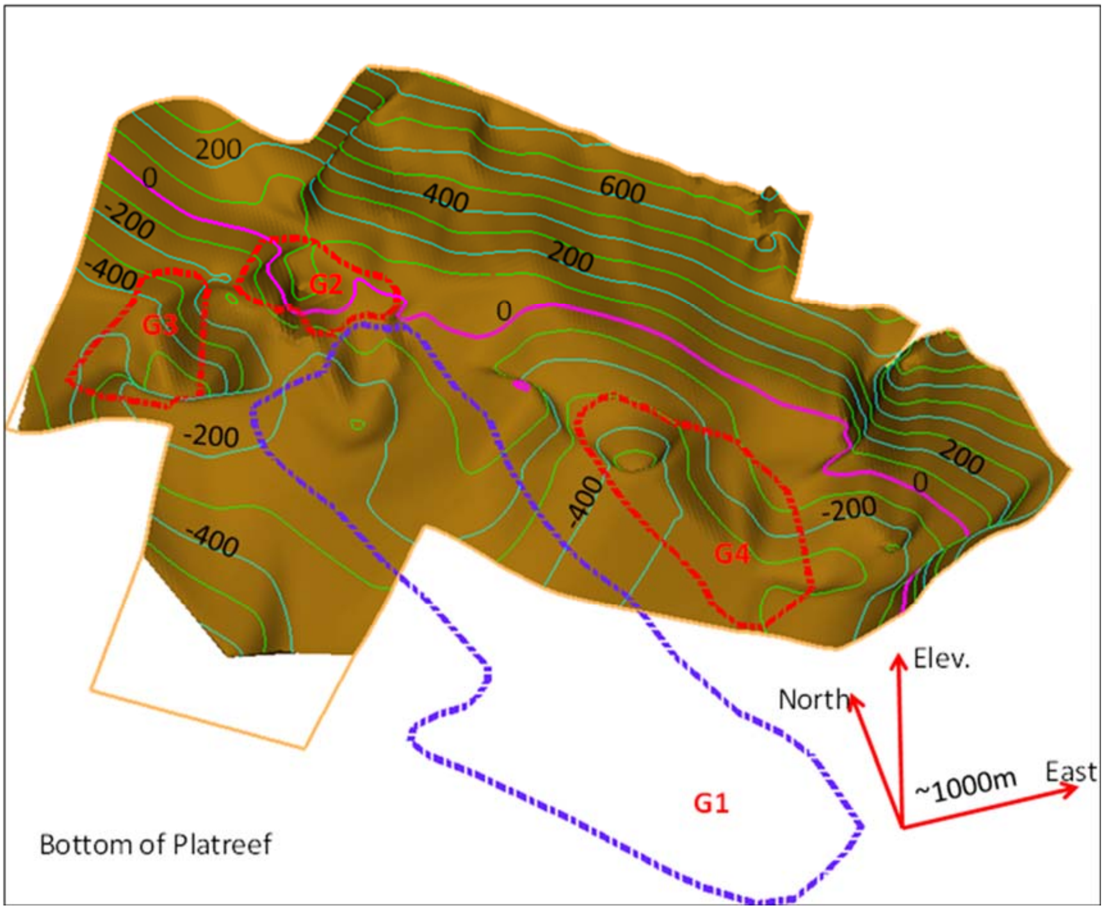
The change in dip of the Platreef at approximately 200 masl in the floor, 400 masl in the top, is accompanied by fundamental changes in geology. Unlike the steeply-dipping up-dip area, the Platreef footwall here comprises apparently uncontaminated pyroxenite, harzburgite, and dunite of Critical Zone affinity. In this area it was first recognized that a cyclic magmatic stratigraphy exists and can be correlated from drillhole to drillhole within the upper portion of the Platreef. This upper portion contains well-developed, stratigraphically-constrained zones of chromitite and pegmatoid associated with elevated PGE grades (Grobler et al., 2012). Ivanhoe geologists correlate these strata named by them the Turfspruit Cyclic Unit (TCU) with the Merensky Cyclic Unit (MCU) of the Upper Critical Zone south of the TML.

Figure 7.4 Project Geology Plan



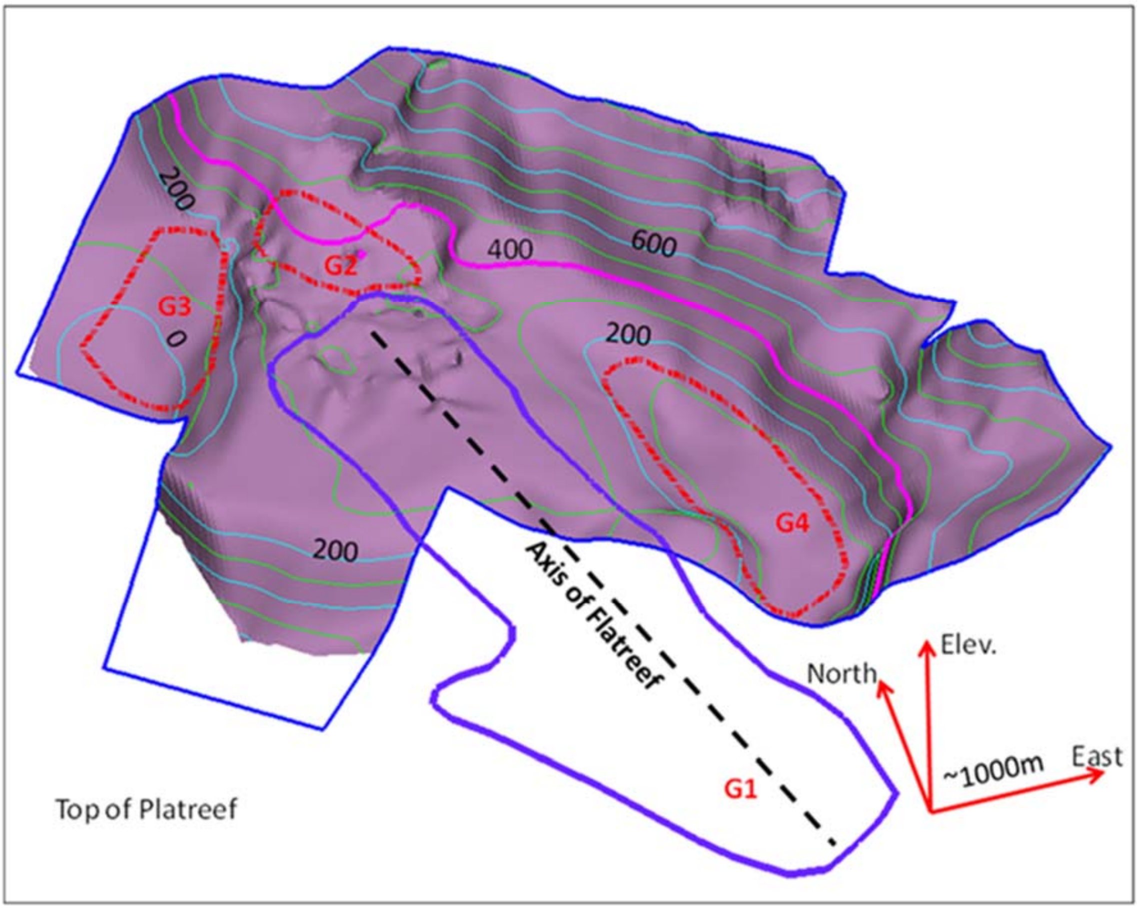
Note: Figure Courtesy Ivanhoe, 2014. Figure shows the sub-outcrop position of the TCU units on the eastern side of the Turfspruit farm; Pongola faults are normal dip slip, Kibaran faults are normal dip slip, and Ventersdorp faults are dextral strike slip. The figure also shows the inferred sub-outcrop position of the T2 mineralized zone (Merensky-analogue) from surface mapping and drillhole information. Also note sub-outcrop of Lower Zone ultramafic units (Brits and Nielsen 2013).

Figure 7.5 Perspective View, Platreef Floor Looking North-North-East



Note: Figure prepared by AMEC and Ivanhoe, 2013. Contours in masl. View is from data available to December 2011. G1 = Flatreef, with projection from gravity survey in 2012; G2, G3, G4 label areas where TCU is thickened.

Figure 7.6 Perspective View, Platreef Top, Looking North-North-East

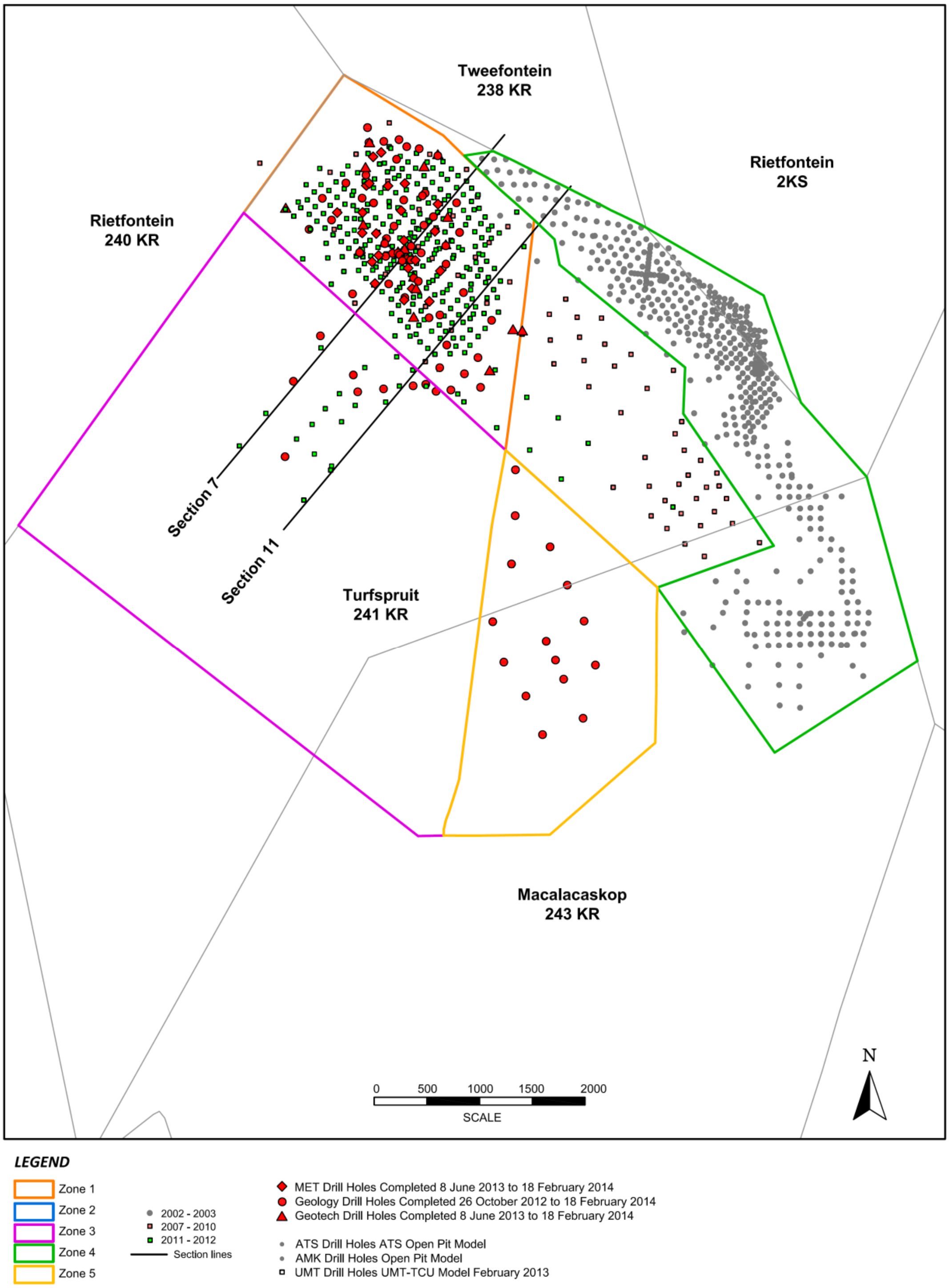


Note: Figure prepared by AMEC and Ivanhoe, 2013. Contours in m. View is from data available to December 2011. G1 = Flatreef, with projection from gravity survey in 2012; G2, G3, G4 label areas where TCU is thickened.

A geographical demarcation of the Project area into five zones was developed based on the date of the database closure in October 2012 that supports the resource estimation in Section 14. The locations of the zones are shown in Figure 7.7. The zones are based on different exploration target concept areas rather than on geological criteria.

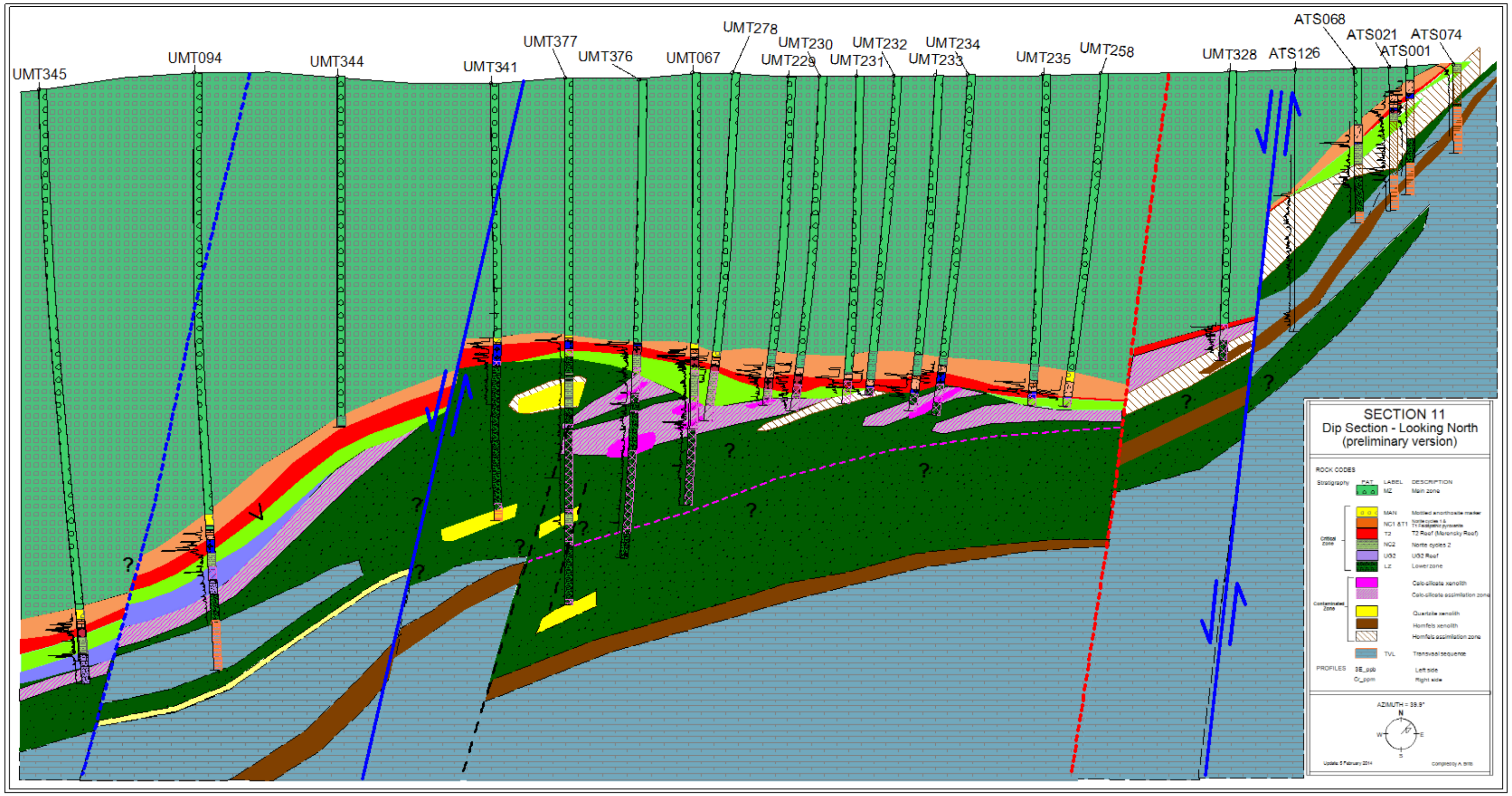
Mineralized intercepts for the Flatreef are shown in Table 7.1 and are based on a selection of the drillholes shown in Figure 7.8. Grade shells were used to constrain grade estimation (see Section 14.2). They are also useful to demonstrate the distribution of mineralization. Table 7.2 shows the average true thicknesses of grade shells for the TCU.

Figure 7.7 Project Zones Plan



Note: Figure courtesy Ivanhoe, 2012; position of cross-sections shown in Figure 7.8 and Figure 7.16 are shown.

Figure 7.8 Cross-Section Along Dip Section 11 Showing TCU



Note: Figure courtesy Ivanhoe 2014; detailed geological and structural interpretations as shown on the figure are provisional and may be subject to change. Drill intercepts calculated by AMEC for selected drillholes from this section are provided in Table 7.1. Location of section line is indicated in Figure 7.7. Section is from west to east across the Flatreef.

Table 7.1 Intercepts Grading > 2 g/t and > 3 g/t 2PE + Au Located on Section Shown in Figure 7.8

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

3 g/t 2PE + Au Composites									
DHID	**From (m)**	**To (m)**	**Drilled Length (m)**	**% Ni**	**% Cu**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**2PE + Au (g/t)**
ATS068	199.30	202.34	3.04	0.53	0.30	3.95	3.36	0.51	7.82
ATS126	No 3 g/t 2PE + Au intercept	–	–	–	–	–	–	–	–
UMT328	699.00	703.00	4.00	0.39	0.23	1.16	1.56	0.35	3.07
UMT258	865.00	882.00	17.00	0.16	0.29	2.32	1.95	0.58	4.85
UMT256	836.93	841.00	4.07	0.42	0.26	2.91	2.55	0.78	6.24
UMT235	850.00	861.09	11.09	0.33	0.21	2.39	2.22	0.34	7.95
UMT234	810.00	836.06	26.06	0.38	0.20	2.12	2.32	0.32	4.75
UMT233	843.00	856.00	13.00	0.38	0.21	2.05	1.98	0.29	4.32

3 g/t 2PE + Au Composites									
DHID	From (m)	To (m)	Drilled Length (m)	% Ni	% Cu	Pt (g/t)	Pd (g/t)	Au (g/t)	2PE + Au (g/t)
UMT232	835.00	851.26	16.26	0.39	0.20	2.17	1.72	0.26	4.15
UMT231	815.00	821.44	6.44	0.47	0.23	2.19	2.42	0.28	4.89
UMT230	803.00	812.00	9.00	0.39	0.18	1.73	2.19	0.23	4.16
UMT229	807.43	812.18	4.75	0.17	0.08	2.44	1.51	0.22	4.17
UMT278	780.00	793.00	13.00	0.39	0.19	2.27	2.52	0.29	5.08
UMT067	758.28	770.26	11.98	0.46	0.21	2.12	3.12	0.31	5.54
UMT376	708.33	716.00	7.67	0.49	0.25	2.99	3.52	0.43	6.94
UMT377	700.00	706.00	6.00	0.25	0.12	2.92	2.33	0.39	5.65
UMT341D1	695.00	706.00	11.00	0.33	0.16	1.72	1.57	0.37	3.66
UMT094	1257.82	1275.79	17.97	0.26	0.13	2.44	2.45	0.30	5.20
UMT345	1430.00	1440.00	10.00	0.30	0.14	1.97	1.40	0.51	3.88

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

Table 7.2 Average Grade Shell True Thicknesses

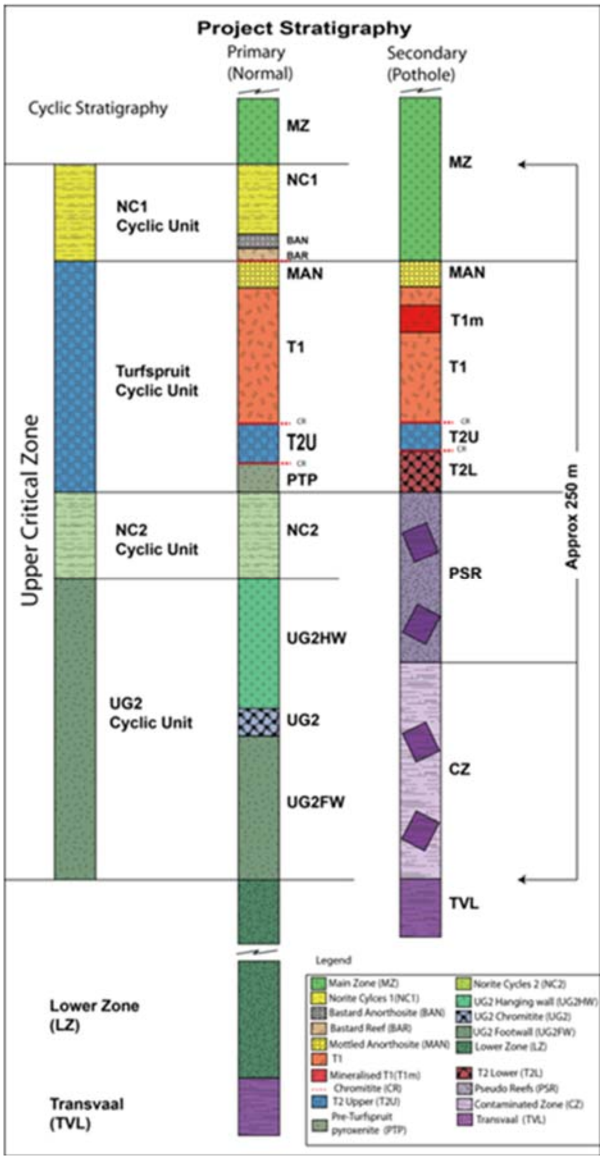
Nested Grade Shell	Indicated (Zone 1, 100 x 100 m spacing)	Inferred (Zones 2, 3)
3 g/t 2PE + Au	17.1 m	12.9 m
2+3 g/t 2PE + Au	24.3 m	18.0 m
1+2+3 g/t 2PE + Au	29.1 m	23.6 m

Note: Computed where grade shell is present and at least 3 m thick.

7.3.2 2012–2013 Geological Re-interpretation and Correlation with Upper Critical Zone

A new subdivision and nomenclature termed the Turfspruit Cyclic Unit (TCU) was proposed for the upper Platreef on Turfspruit (Grobler and Nielsen, 2012; Grobler et al., 2012). Re-interpretation of historical drillholes has allowed the extension of this interpretation across the entire Project area, including the shallow mineralization considered potentially amenable to open-pit mining methods that is located towards the eastern boundary of Turfspruit. Although large areas show major contamination in this area, the cyclic stratigraphy remains recognizable, and the sub-outcrop position of the TCU has been established (refer to Figure 7.4). The general stratigraphic sequence, as recently developed in the Project area is depicted in Figure 7.9.

Figure 7.9 Proposed Cyclic Stratigraphic Framework



Note: Figure courtesy Ivanhoe, 2012. PTP is an unmineralised orthopyroxenite locally present at the base of the TCU.

The cyclic magmatic sequence now recognized within the Project area has been subdivided into five major cyclic units, and correlated with the Upper Critical Zone (UCZ) south of the TML (Grobler et al., 2012). The units include from top to bottom:

- Norite Cycles 1 (NC1) as hangingwall to the TCU; this unit is an analogue of the Bastard Cyclic Unit (BCU), containing the "Giant Mottled Anorthosite" (GMA) at its top.
- Turfspruit Cyclic Unit (TCU), the main mineralized cyclic unit; this unit is analogous to the Merensky Cyclic Unit (MCU) that contains the Merensky anorthosite and pyroxenite.
- Norite Cycles 2 (NC2) as footwall to the TCU; this unit is an analogue of the Merensky footwall in the Western and Eastern BIC.

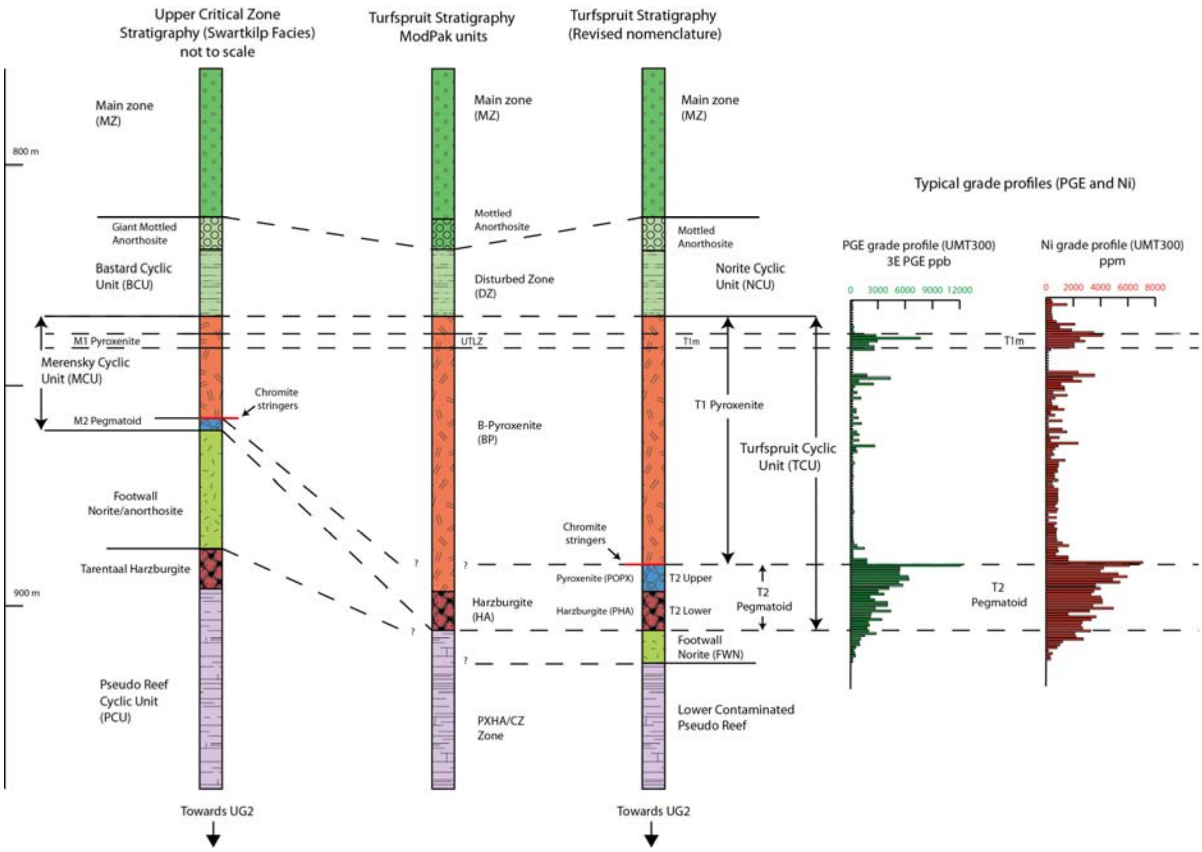
- Pseudo Reef Cyclic Units (PSR) that are restricted to large regional pothole depressions and are analogous to the Pseudo Reefs of the Swartklip Facies (Northwestern BIC) found in the interval between the Merensky Reef and UG2 reef.
- UG2 Cyclic Unit (UG2CU) is an analogue to the UG2 Cyclic Unit of the Western and Eastern Limbs, exhibiting many similarities including "leader triplets" chromitite stringers.

The main difference between the UCZ cyclic units on Turfspruit and those in the Eastern and Western Limbs is their greater thickness, particularly with regards the TCU.

Figure 7.10 shows the correlation suggested by Ivanhoe staff between the TCU and cyclic units located on the Eastern and Western Limbs of the BIC. "Turfspruit Stratigraphy Modpak units" were used to estimate the mass mineable underground resources (see Section 14.3); "Turfspruit Stratigraphy Revised nomenclature" was used to estimate the selectively mineable underground resources (see Section 14.2).

Contamination by Transvaal sediments can occur within any of the stratigraphic horizons; however, it is predominantly confined to the units below the TCU. Contamination hampers footwall stratigraphic identification and is grouped into "Contaminated Zone" (CZ) where dominant. To date, no evidence of the existence of Lower Critical Zone lithologies have been found within the Turfspruit area. However, Lower Zone mafic to ultramafic rocks have been intersected in many deep holes within the Project area. Work is ongoing to understand and potentially resolve the stratigraphy of the CZ.

Figure 7.10 Revised Stratigraphic Interpretation, Turfspruit Cyclic Unit



Note: Figure courtesy Ivanhoe, modified by AMEC, 2012. ModPak units were used in resource modelling in 2011 and revised nomenclature from 2012.

7.3.2.1 Turfspruit Cyclic Unit (TCU)

The TCU is the best-developed cyclical unit recognized in the Project area and hosts the principal mineralized reefs. The TCU is in general subdivided from the base upwards into the following zones:

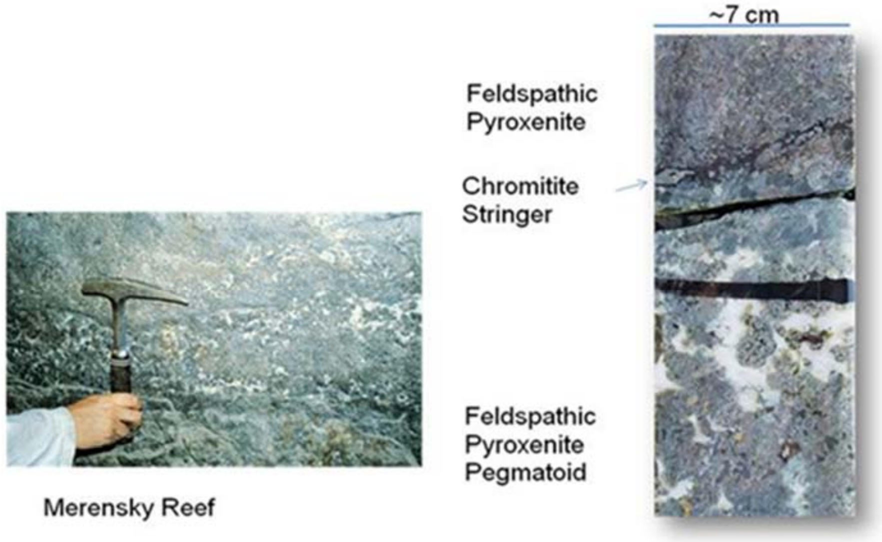
1. Mineralized pegmatoidal harzburgite and/or pegmatoidal olivine-bearing pyroxenite (T2 Lower), locally with a chromitite stringer on its bottom contact.
2. Mineralized pegmatoidal orthopyroxenite, commonly with a thin (~0.5 cm) chromitite stringer marking its upper contact (T2 Upper).
3. Non-mineralized non-pegmatoidal medium-grained feldspathic pyroxenite (T1), with a generally non-pegmatoidal mineralized zone near its top (T1m).
4. Mottled anorthosite-norite on the hangingwall contact (T1).

The pegmatoidal T2 Lower harzburgite exhibits a poikilitic texture whereby large orthopyroxene oikocrysts enclose smaller olivine ± chromite grains. Higher PGE and Ni-Cu grades (>4 g/t PGE, >0.4% Ni, >0.2% Cu) are commonly associated with the pegmatoid and chromitite, although mineralization also persists into both the over- and underlying rocks. Pt/Pd ratios also tend to be higher (>1.0) in association with chromitite and pegmatoid. The T2 Upper + Lower pegmatoidal zone averages ~4 to >20 m in thickness.

The T1 pyroxenite is medium to coarse-grained, variably feldspathic, and usually comprises the thickest unit within the TCU, averaging ~15 m, but thickening to as much as 85 m in places (Nielsen and Grobler, 2012). The T1 pyroxenite contains the mineralized T1 (T1m) zone, which consists of disseminated, medium to coarse-grained sulphides hosted within the typically non-pegmatoidal feldspathic pyroxenite, locally containing chromitite stringers. The T1m contact is gradational with adjacent weakly to un-mineralized T1 pyroxenite. The basal 1–2 m of the T1 (directly above the T2 contact) is commonly weakly to moderately mineralized, and may contain millimetre-thick chromitite "leaders".

Davey (1992) and Lea (1996) also describe two zones of PGE enrichment associated with the Merensky Cyclic Unit in the eastern Bushveld, the M1 pyroxenite and M2 pegmatoid. Figure 7.11 shows a comparison of the Merensky Reef and the TCU.

Figure 7.11 Comparison of Merensky Reef and the TCU



Note: Left photograph 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 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.

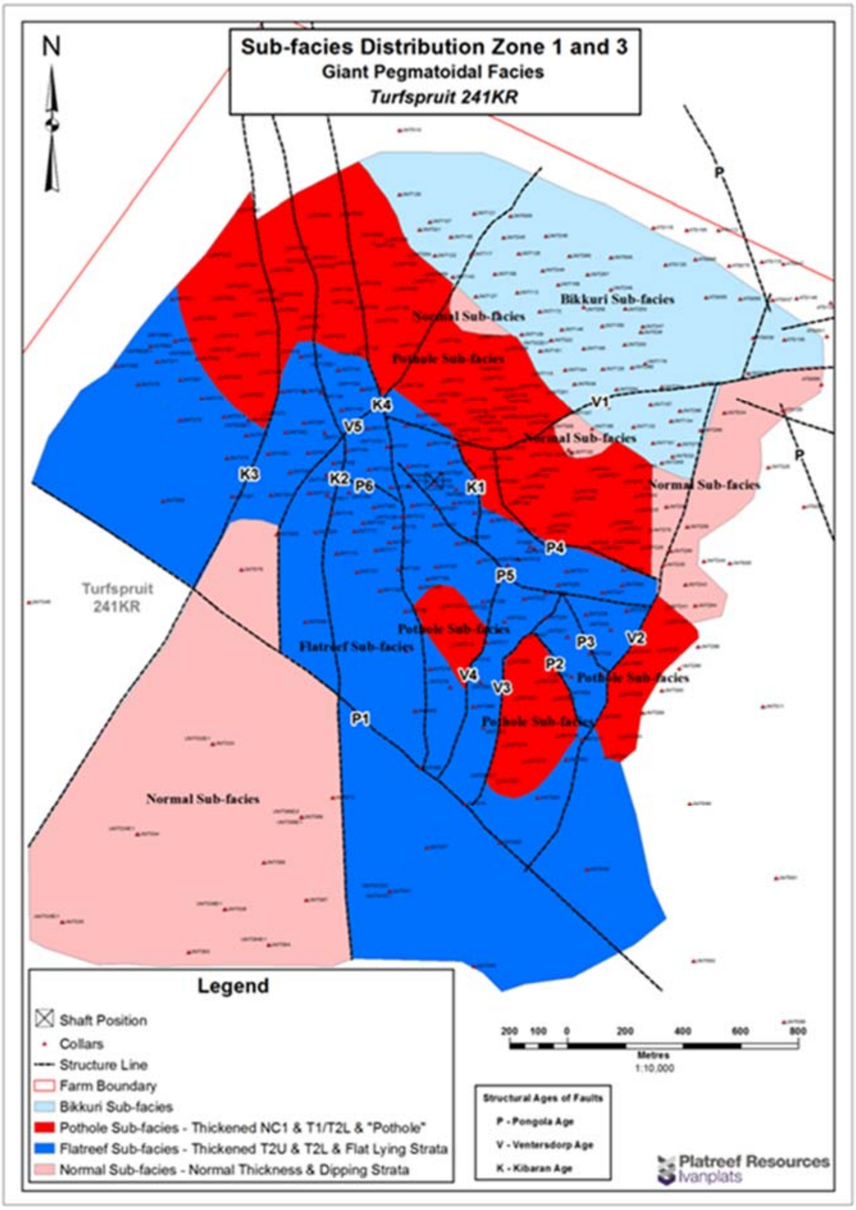
7.3.3 Delineation of New Platreef Regional Facies and Sub-Facies

Rocks of the Upper Critical Zone in the Western Limb of the BIC have been divided into the "Swartklip facies" to the north, and the "Rustenburg facies" to the south (Wagner, 1929); the demarcating line being the Pilanesberg Fault. Four sub-facies have been recognised in the Rustenburg facies, based on the morphology of pothole structures and thickness variations of the Merensky Reef, and two sub-facies within the Swartklip facies (Viljoen et al., 1986a; Viljoen, 1999).

Ivanhoe has subsequently recognized what may be a new regional facies, the "Giant Pegmatoidal facies" (Grobler et al., 2013) in the Upper Critical Zone (UCZ). Establishment of the "Giant Pegmatoidal facies" is based on the significant thickness difference of the mineralized pegmatoidal T2 reef (~4 to >20 m), compared to the much thinner pegmatoidal Merensky Reef (~ 1 m) (Grobler et al., 2013).

Consistent with the revised interpretation and correlation with the Upper Critical Zone (UCZ), the UCZ recognized in the Project area is proposed to represent a new Four magmatic sub-facies were postulated within the Giant Pegmatoidal facies, and were defined in an attempt to describe the T1m and T2 mineralized zones (Figure 7.12). These facies are based on the distribution and development of the different lithological units that represent the TCU. The distribution and morphology of these sub-facies are comparable to that of "potholes" described in the Merensky Reef south of the TML, (Section 7.3.5).

Figure 7.12 Giant Pegmatoidal Facies Zone 1 and Zone 3 Sub-facies Distribution



Note: Figure courtesy Ivanhoe, 2013, after Grobler et al., 2013. Shaft position is planned. Refer to Section 7.3.6 for a discussion of the faulting indicated on the plan.

7.3.4 Stratigraphic Correlations and Nomenclature

The stratigraphy of the UCZ is depicted in Table 7.3. In this table, a comparison is made between the revised UCZ stratigraphy, the previous ModPak distinctions used by Ivanhoe from 2010 to 2012 (Parker et al., 2012), the historical Platreef interpretation and the recognised eastern and western Bushveld stratigraphy. ModPak units were used in resource modelling in 2011 and revised nomenclature from 2012. Table 7.4 provides a breakdown of the units recognized in the Upper Critical Zone as a result of the Ivanhoe work in 2012. Table 7.5 summarises the stratigraphic column and provides generalised rock compositions for each stratigraphic unit. Table 7.5 also summarizes the regional and sub-facies descriptions. Table 7.6 provides a list of the acronyms for the key lithologies that are used in the geotechnical discussions in Section 16.1.

Table 7.3 Stratigraphic Correlations Summary

Platreef (White, 1994; Kinnaird et al., 2005)	Modpak (Parker et al., 2012)	Turfspruit (Grobler and Nielsen, 2012)	
Upper Zone	Upper Zone	Upper Zone	Upper Zone
Unconformity			
Main Zone	Main Zone (MZ)	Main Zone	Main Zone
Unconformity			
Disturbed Zone	Disturbed Zone (DZ)	Norite Cycle 1	NC1
		Bastard Anorthosite (BAN)	
		Bastard Reef (BAR)	
		Mottled Anorthosite (MAN)	TCU
C Pyroxenite	B pyroxenite (BP)	T1	
B Pyroxenite	Pegmatoidal B pyroxenite (PBP)	T2U	
	Harzburgite (HA)	T2L	
Unconformity			
A Pyroxenite	Pyroxenites and harzburgites (PXHA)	Norite Cycle 2	NC2
		Pseudo Reef	PSR
		UG2 Hangingwall	UG2CU
Malmali subgroup	Contaminated Zone (CZ)	UG2 Chromitite	
		UG2 Footwall	
Unconformity			
Lower Zone	Lower Zone (LZ)	Lower Zone	Lower Zone
Marginal Zone	Marginal Zone (MZN)	Marginal Zone	Marginal Zone
Transvaal	Transvaal (FL)	Transvaal	Transvaal

Note: Contaminated Zone can occur at any stratigraphic level below the TCU.

Table 7.4 Upper Critical Zone (UCZ) on Turfspruit and Macalacaskop (Grobler et al., 2013)

Turfspruit UCZ		Lithology	Petrographic Description	Thickness (m)	BIC UCZ
NC1CU	NC1	Interlayered sequence of anorthosite, norite and feldspathic pyroxenite	Comprises cyclical units of GN, AN, N, FPX and locally chromite.	0–100	Bastard Cyclic Unit
	BAR	Feldspathic pyroxenite	Mineralised FPX located within the NC1, with a thin chrome stringer often developed on the basal contact.	0–5	–
TCU	MAN	Mottled anorthosite	An anorthosite containing globules of pyroxene ranging in from mm sized spotted to cm sized mottled anorthosite.	0–20	Merensky Cyclic Unit
	T1	Feldspathic pyroxenite	Composed of ±20% plagioclase feldspar; ±80% orthopyroxene, with minor clinopyroxene. Medium to coarse grained, with pyroxene cumulates. Weak talc-tremolite-chlorite, as well as biotite alteration.	0–50	–
	T1m	Mineralized feldspathic pyroxenite	Portion of the T1 feldspathic pyroxenite that contains fine- to medium-grained sulphides.	0–10	–
	T2U	Orthopyroxenite	Poikilitic pegmatoidal orthopyroxenite, typically with a chrome stringer marking the top contact. Composed of >70% orthopyroxene, <10% clinopyroxene, <10% plagioclase and <10% olivine.	0–25	–
	T2L	Harzburgite	Composed of >40% to <90% olivine; >10% to <40% orthopyroxene and <10% clinopyroxene and plagioclase <10%. Often has a characteristic poikilitic texture.	0–30	–
NC2CU	NC2	Interlayered sequence of anorthosite, norite and feldspathic pyroxenite	Comprises cyclical units of AN, N, FPX and occasional Cr stringer in AN. Sporadically mineralised.	0–100	Footwall Norite 1-12

Turfspruit UCZ		Lithology	Petrographic Description	Thickness (m)	BIC UCZ
PSRCU	PSR	Harzburgite (“Tarentaal”)	Composed of feldspathic harzburgite with subhedral plagioclase grains. Well mineralised with base metal sulphides (Ni and Cu) and less PGEs.	0–200	Pseudo Reef
UG2CU	UG2HW	Feldspathic pyroxenite	Composed of medium-grained cumulus orthopyroxene and intercumulus plagioclase with minor cumulus clinopyroxene, massive igneous texture.	0-150	UG2 Cyclic Unit
	UG2	Chromitite	Fine to medium-grained chromitite band. Subhedral to euhedral chromitite band consists of orthopyroxenes, clinopyroxene and plagioclase grains.	0–1.5	–
	UG2FW	Olivine orthopyroxenite	Composed of medium-grained abundant intercumulus plagioclase and rounded cumulus olivine commonly enclosed by oikocrysts of clinopyroxene, crudely layered poikilitic igneous texture.	0–50	–
LZ	LZ	Orthopyroxenite, harzburgite and dunite	Composed of orthopyroxenite, harzburgite and dunite. Fine to medium grained orthopyroxenite with interstitial plagioclase is characterized by very fine-grained disseminated sulphides.	0–200	Lower Zone
CZ	CZ	Parapyroxenite, calc-silicate, hornfels, quartzite xenoliths	A mixture of parapyroxenite (a mixture of metamorphosed metasediments, pyroxenite and serpentinite) and calc-silicate (a partially melted dolomite with inter-bedded chert). Associated with massive sulphide bodies.	0–350	–
TVL	TVL	Quartzite, hornfels, dolomite	Very fine to fine-grained metasedimentary rocks.	0–250	Transvaal

Table 7.5 Facies and Sub-Facies Description

Facies	Sub-Facies	Abbreviation	Description
Giant Pegmatoidal Facies (GPF)	Bikkuri Reef	BRS	"Surprise reef" intersected at shallower than expected depths and forming a double reef package on the eastern side of Zone 1.
	Normal Reef	NRS	Reef exhibiting more normal thicknesses. Noritic footwall stratigraphy with a well-developed UG2 Cyclic Unit.
	Flatreef	FRS	Flat-lying anomalously thick T2 reef.
	Pothole Reef	PRS	Thickening of lithologies into large depression structures. Development of olivine-bearing harzburgitic units at or near the base of the TCU. Footwall of thick Pseudo Reef pyroxenite and feldspathic harzburgite.

Note: Table after Grobler et al., 2013.

Table 7.6 Platreef Lithologies

Lithology	Code	Composition
Anorthosite	AN	Plagioclase, minor clinopyroxene, minor orthopyroxene.
Assimilation	AZ	Orthopyroxene, plagioclase, sulphides, clinopyroxene, chromite, assimilated xenoliths.
Feldspathic pyroxenite	FPX	Orthopyroxene, plagioclase, sulphides, clinopyroxene, chromite.
Gabbronorite	GN	Clinopyroxene, plagioclase, orthopyroxene.
Granitic Vein	GRV	Potassium feldspar, plagioclase feldspar, quartz, biotite and/or amphibole.
Melagabbronorite	MGN	Clinopyroxene, plagioclase (<30%), orthopyroxene.
Magnetite	MT	Magnetite, clinopyroxene, plagioclase.
Norite	N	Clinopyroxene, orthopyroxene, plagioclase, minor sulphides.
Norite cycles	NC	Orthopyroxene, plagioclase, clinopyroxene.
Olivine Orthopyroxenite	OLOPX	Orthopyroxene, plagioclase, sulphides, minor clinopyroxene, olivine (<40%).
Olivine Pyroxenite	OLPX	Orthopyroxene, plagioclase, sulphides, clinopyroxene, olivine (<40%).
Ortho Pyroxenite	OPX	Orthopyroxene, plagioclase, sulphides, minor clinopyroxene, chromite.
Para Pyroxenite	PAPX	Clinopyroxene, anhydrite, sulphides, calcite.
Pyroxenite	PX	Orthopyroxene, sulphides, clinopyroxene, plagioclase, olivine, chromite, sulphides.

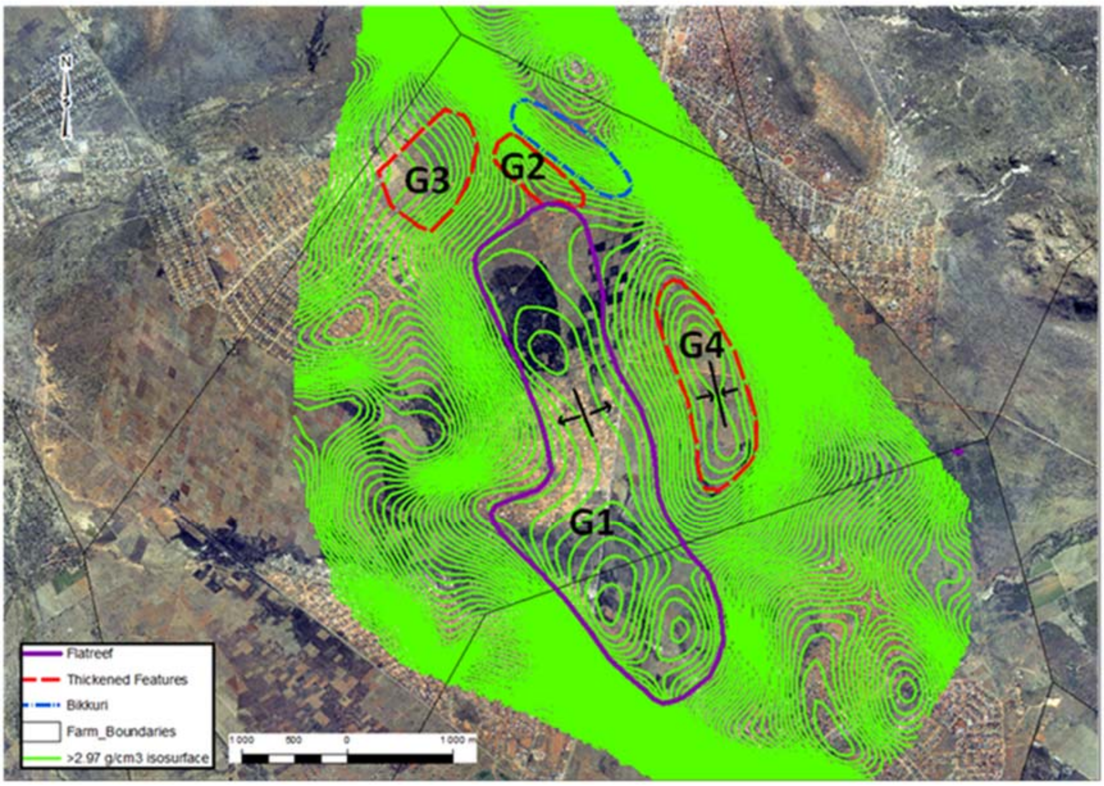
7.3.5 Geological Features of the UCZ in Project Area

Three distinct geological features (Figure 7.13) are recognized within the Project area and include the following:

- A double reef package within the north-eastern part of Zone 1 (informally termed the Bikkuri Reef).
- Three different areas (G2, G3, G4) where UCZ lithologies show significant thickening into what appear to be large depressions or "pothole" depressions, at least two of which (G3, G4) are possibly controlled by existing pre-Bushveld fold structures and/or faults. On the other hand in detail potholes can reflect magmatic (thermal) erosion on a local scale, as shown in Figure 7.14 Such erosion may have occurred at G2.
- Presence of a flat-lying portion of the TCU again related to structural control.

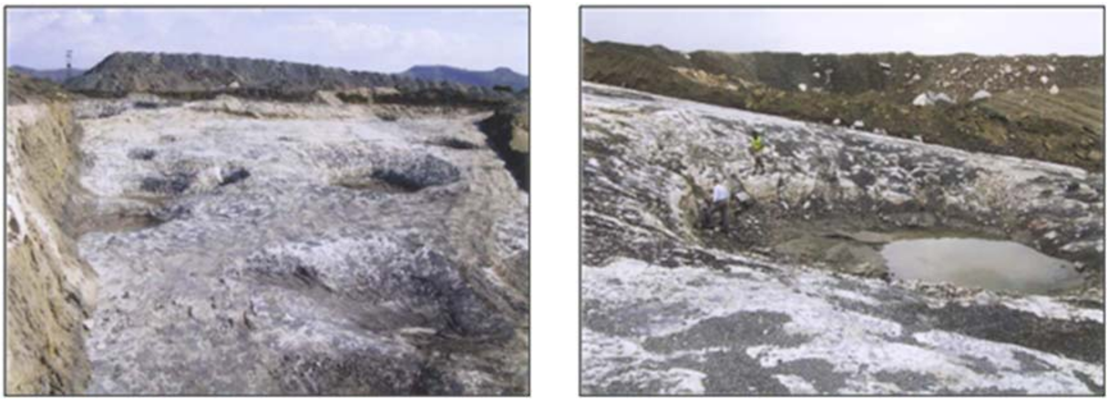
Figure 7.15 shows an isopach map of the thickness of the 2+3 g/t grade shell. The thickest zones correlate with the depressions described below. The axis of the Flatreef tends to have the thinnest zones. The contour patterns, particularly in the well drilled inset portion of Figure 7.15 are consistent with the pothole topography shown in Figure 7.14. In addition to and in advance of underground exploration, a detailed seismic survey will help to determine accurately the dimensions and amplitudes of potholes. This survey has been conducted, and the data are currently being processed.

Figure 7.13 Major Geological Features on Geophysical Plan, Falcon Gravity Data



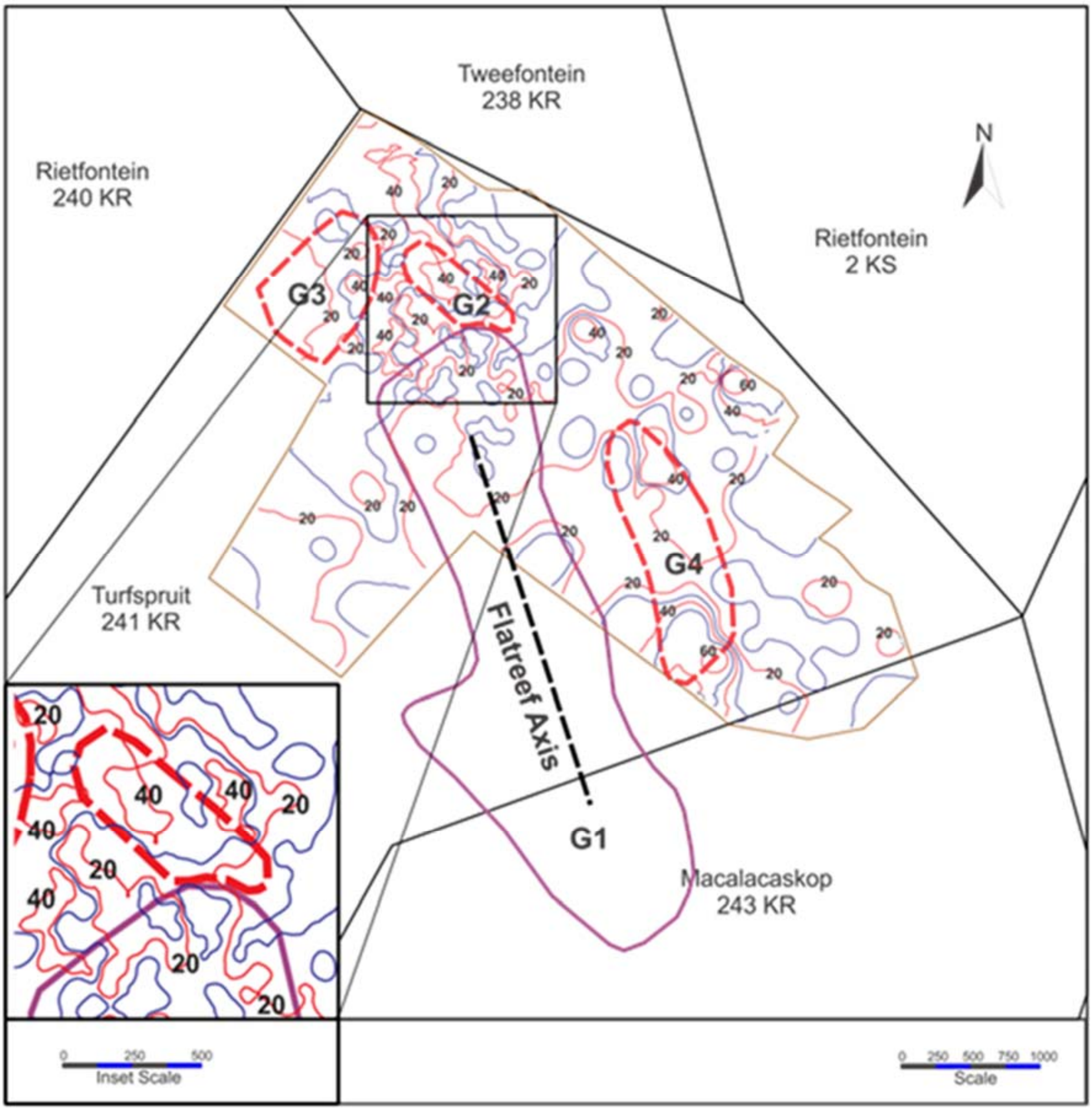
Note: Figure courtesy Ivanhoe, 2013. G1 represents the Flatreef; G2, G3, and G4 represent areas with thickened TCU; blue dashed perimeter indicates location of Bikkuri Reef.

Figure 7.14 Pothole Structures, Karee Platinum Mine (Rustenburg Area)



Note: Photographs from van der Merwe and Cawthorn, 2005. Photographs show the occurrence of “pothole” structures at the base of the UG2 chromitite reef. Note human figures standing inside pothole structure in photograph on the right, which provide scale.

Figure 7.15 Isopach Map of the 2+3 g/t Grade Shell (units m)



Note: Figure courtesy Ivanhoe, 2013. Based on UMT-TCU model and gravity survey. G1 represents the Flatreef; G2, G3, G4 are areas with thickened TCU.

The largest feature recognized within the TCU is a depression (G2) that occurs within the eastern part of Zone 1, where significant thickening of the NC1 and the upper (T1) stratigraphic layer of the TCU occur. This depression contains a distinct thickening of the T1 feldspathic pyroxenite. A second similar depression (only partly drilled) is present towards the north-western edge of Zone 1 (G3). However, in this case, thickening of the both the TCU as well as its footwall units (still in the Platreef) appear to have occurred. A third depression (G4) occurs mainly in the Zone 2 area in the southern part of the Turfspruit. Features related to these structures appear to be reminiscent of "regional pothole" structures as described from other parts of the BIC (Viljoen et al., 1986a; Viljoen et al., 1986b; Viring and Cowell, 1999).

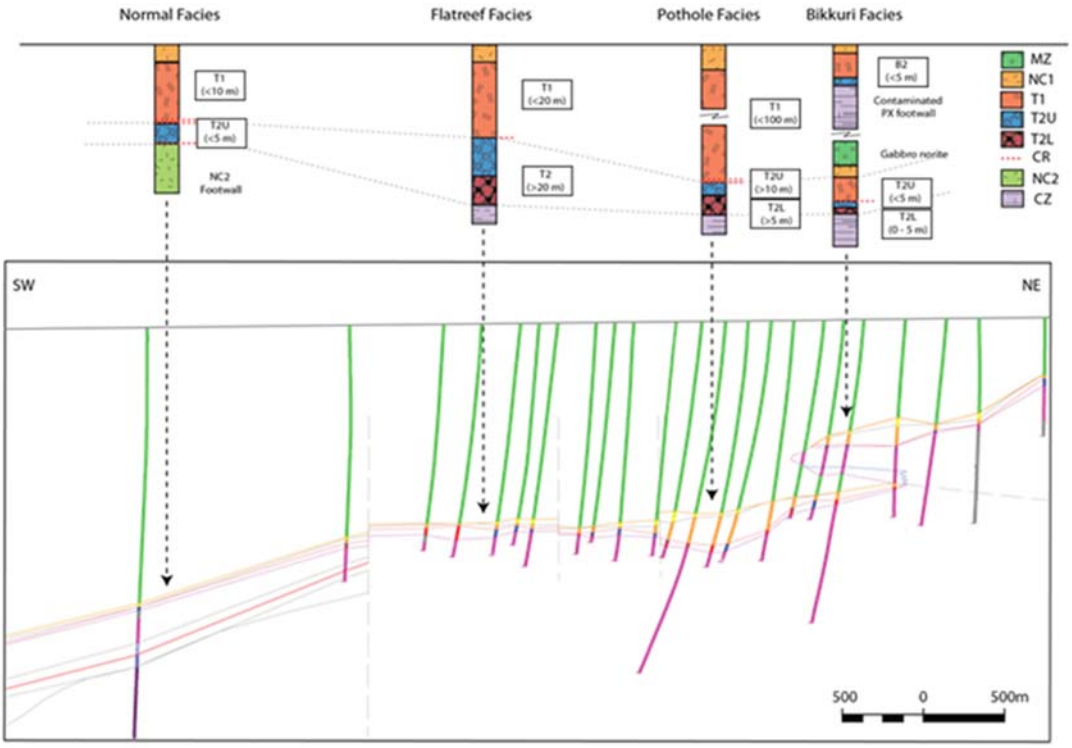
Smaller potholes appear to be present within the Flatreef based on the distribution of T2 Lower olivine-bearing lithologies.

The regional pothole and normal sub-facies are very similar to those recognized and described from the Western and Eastern Limbs of the BIC.

A further, unique feature recognized within Zone 1 is the Flatreef portion of the Platreef, initially recognized as being flat-lying compared to the steeper-dipping reefs within the Zone 4 area (the portion of the deposit where mineral resources amenable to open-pit mining are located). Ongoing studies suggest that intrusion of the TCU appears to have been partially controlled by pre-Bushveld folds and faults evident in the Transvaal Supergroup sedimentary rocks. The presence of identical centimetre to metre scale primary cyclic magmatic layering in both steep and flat-dipping parts of the Project area suggests that post-Bushveld faulting tilted originally flatlying zones to current steep orientations in the up-dip area, while the Flatreef maintained more or less its original geometry. The Flatreef in essence appears to contain better-mineralized T2 mafic to ultramafic units compared to the surrounding areas, where the T1m and T2 reefs occur in closer proximity to each other. Juxtaposition of different T2 lithologies (possibly “normal” and “pothole” sub-facies) is difficult to model at the current drill spacing, but may be due to strike-slip displacement across fault structures.

Figure 7.16 is a cross-section through the mineralized zone and illustrates the different sub-facies recognized across the Turfspruit drilled extent. Structural displacement is evident on cross-sections and plan views on distribution plots of the different facies (refer to Section 7.3.6 for a discussion of Ivanhoe’s structural interpretation of the Flatreef area).

Figure 7.16 Cross-Section along Dip Section 7 through Zone 1 Turfspruit Cyclic Unit



Note: Figure courtesy Ivanhoe, 2013. Figure displays the occurrence of the different sub-facies within the interpreted Giant Pegmatoidal Facies. Each strip log at the top of the figure corresponds with the geology observed in the drill section below. Cross-section location shown in Figure 7.7.

7.3.6 Structure

Ivanhoe has used the regional structural interpretations of Friese (2012) as a framework to guide structural interpretations in the Project area.

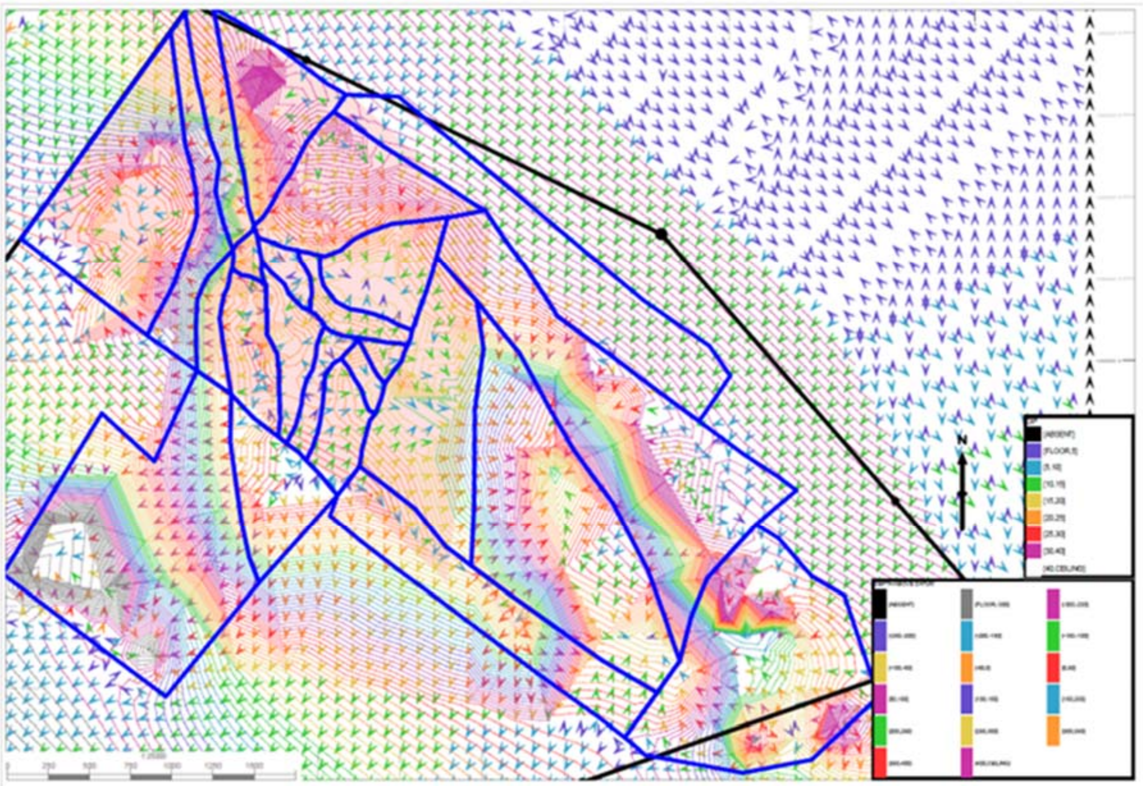
Five major groups of faults have been recognized regionally in the Bushveld according to their relative timing, orientation, and kinematics (Table 7.7, after Friese, 2012). On the Project, Ivanhoe geologists have interpreted the Pongola (P), Kibaran (K) and Ventersdorp (V) fault sets mainly in Zone 1 where there is detailed drilling. Fault interpretations were generated by Ivanhoe staff, using a combination of contour mapping and dip direction data (Figure 7.17, Figure 7.18) with Figure 7.19 providing a summary plan. This information provides the Project-wide framework for a conceptual model of the structural anomaly locally termed the Flatreef in Zone 1. Current interpretations show that the mineralization in the Flatreef area has been both upfaulted and downthrown. This is illustrated in the sections included as Figure 7.20 to Figure 7.22. Figure 7.19 shows the locations of the section lines.

Table 7.7 Summary of Structural Features in the Bushveld and Project Area

Name	Azimuth	Dip	Type	Sense	Age	Features	Comment	Association
Pongola Extensional Faults	NW	Steep	Normal	Extensional	Murchison Orogeny (2.98–2.96 Ga)	<40 cm brittle ductile tectonite with thin calcite veins, occasional granitic healing	Displaced by all three other fault groups, 10 - 35 m of net vertical displacement, dextral strike-slip reactivation	Granite dykes
Group 2 Thrusts	SE	Shallow	Thrust	Reverse	Ubendian Orogeny and syn-Bushveld Complex (2.1–1.86 Ga)	Semi-ductile, thin (3 - 60 cm), quartz veins and serpentinization on thrust planes	Subsidiary compressive structures	–
Group 1 Thrusts	NW	Shallow	Layer-parallel	Reverse followed by normal reactivation	Ubendian Orogeny and syn-Bushveld Complex (2.1–1.73 Ga)	Semi-ductile, thin (3 - 60 cm), quartz veins and serpentinization on thrust planes	Flexural slip thrusts common on stratigraphic contacts	Granite sills
Kibaran Extensional faults	N	Steep	Normal	Extensional	Kibaran Orogeny (1.35–1.2 Ga)	Intensely sheared fracture surfaces with brittle-ductile tectonite infill and thin calcite veins	Major basement detachments with undulating dips Cross-cut by re-activated Pongola and Ventersdorp orientated faults.	–
Ventersdorp Shear Zones	NE	Sub vertical	Strike-Slip	Dextral followed by sinistral reactivation	Reactivation syn-Bushveld Complex of Limpopo Orogeny rifts. (1.15–1.1 Ga)	Intensely deformed country rock now present as <50 cm thick ductile tectonite with slickenside features on fault surfaces, thin calcite veins and serpentinization	Prominent regional structures, 75 m of net vertical displacement. First and second order shears form large-scale dextral strike-slip duplexes	Granite and dolerite dykes

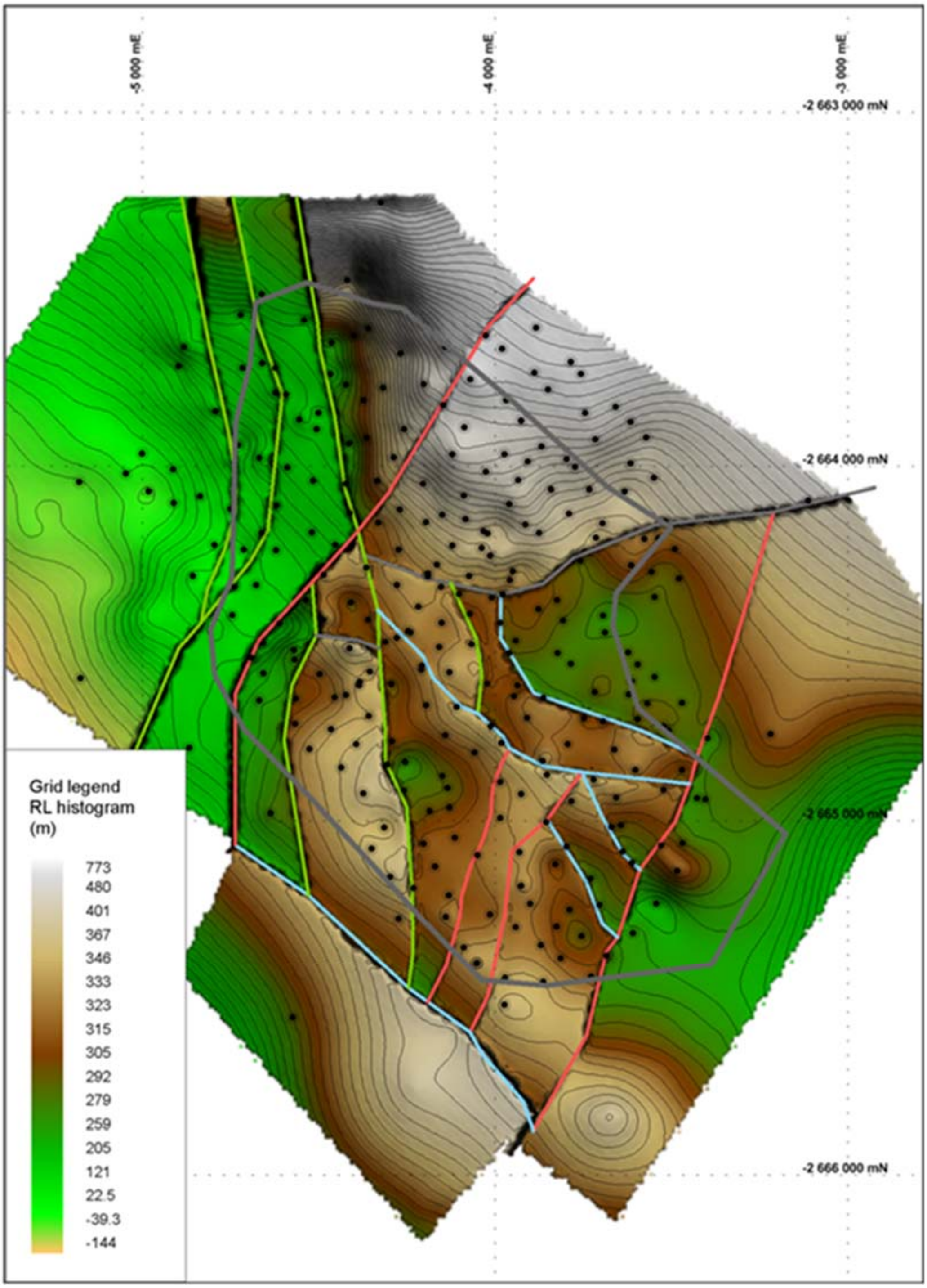
Note: from Freise (2012).

Figure 7.17 Contour–Dip Direction Plan, Flatreef (Zone 1) Area



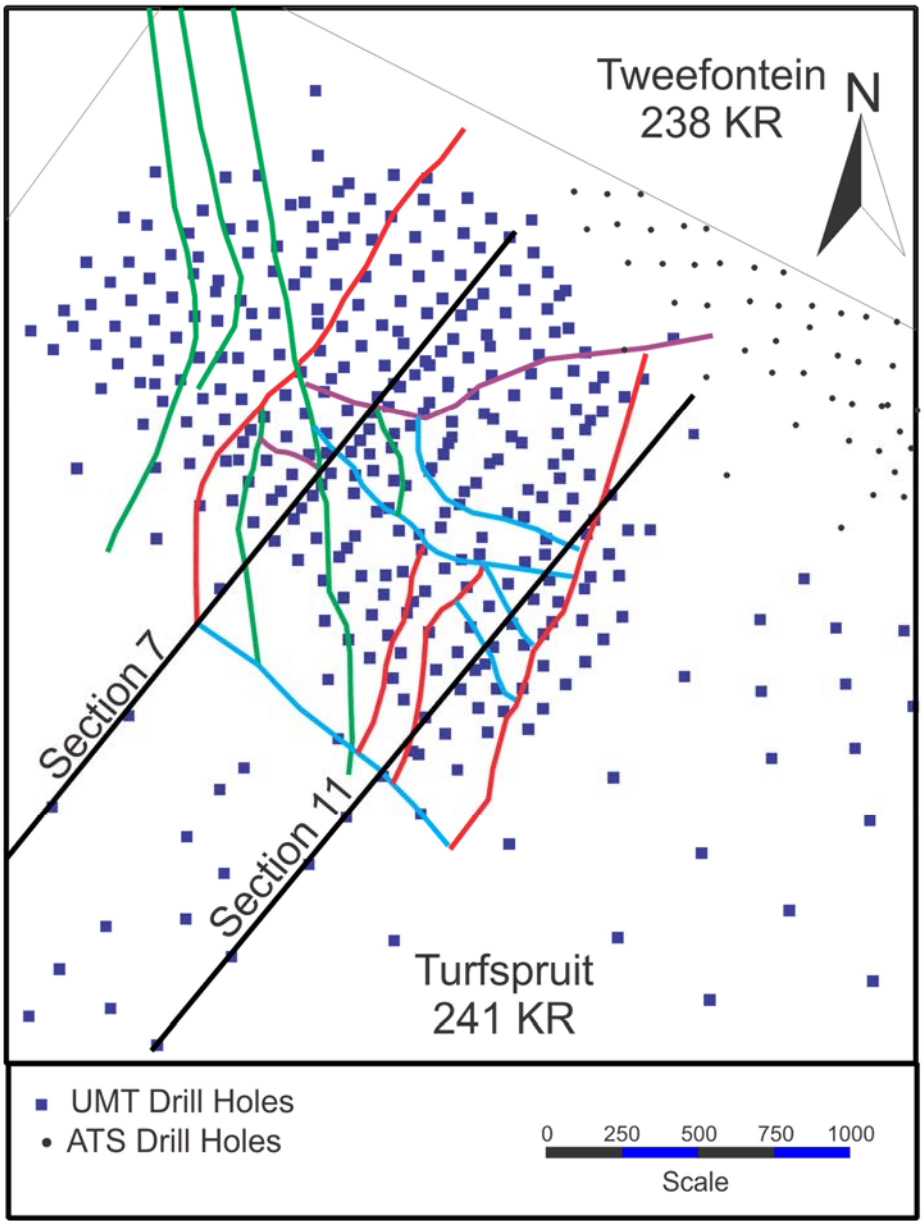
Note: Figure courtesy Ivanhoe, 2013. Zone 1 boundaries are shown in black; contours and dips related to the T1 – Main Zone contact as per drilling data.

Figure 7.18 Inset Plan, "Flatreef" (Zone 1) Area



Note: Figure courtesy Ivanhoe, 2013. Backdrop is the Falcon gravity survey – based elevation contour plan for the Main Zone-Platreef isosurface shown as RL in metres above sea level. Ventersdorp dextral strike-slip faults are indicated as red lines, Pongola normal dip-slip faults as pale blue or grey lines, and Kibaran normal dip-slip faults as green lines. Drill collars indicated as filled black circles.

Figure 7.19 Simplified Structural Plan Showing Locations of Wire-Frame Drill Sections



Note: Figure courtesy Ivanhoe, 2013. Ventersdorp dextral strike-slip faults are indicated as red lines, Pongola normal dip-slip faults as blue lines, and Kibaran normal dip-slip faults as green lines. Drill collars indicated as filled blue boxes.

Figure 7.20 TCU Dip Section 7, Looking North-West

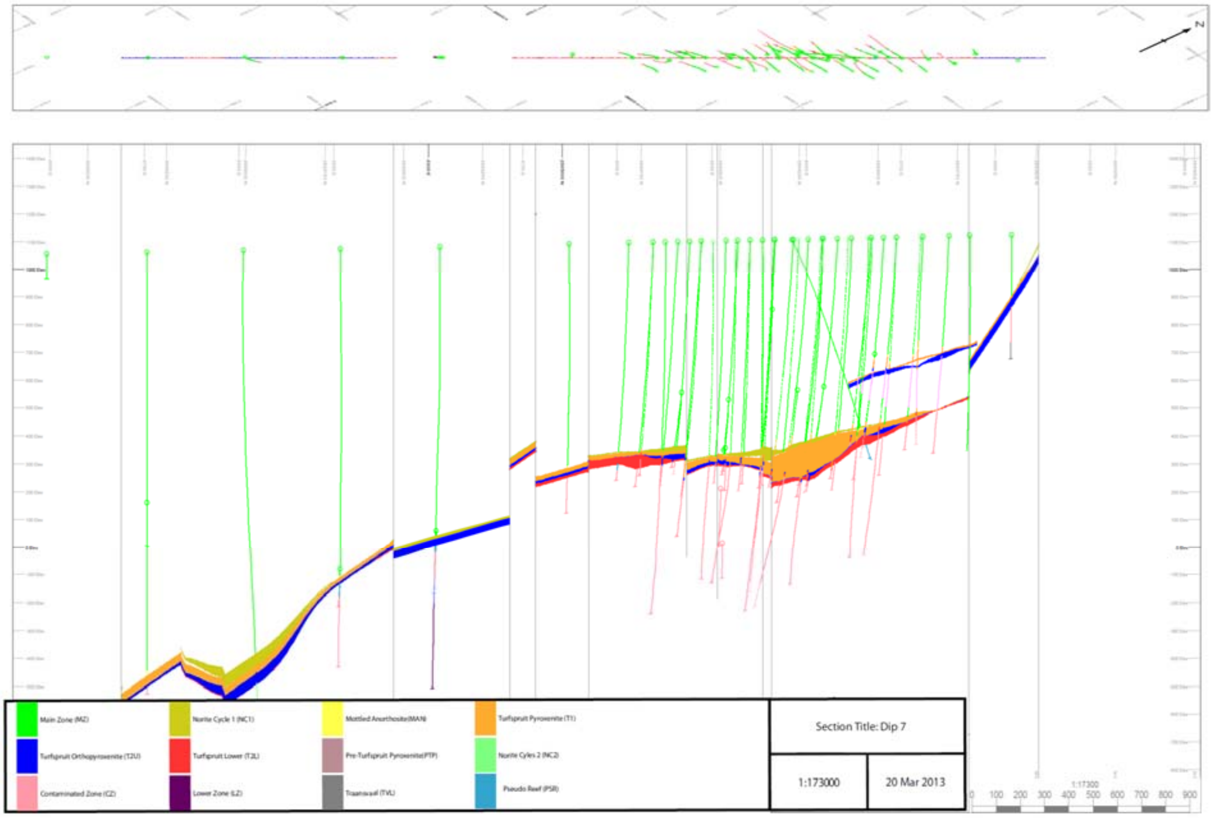
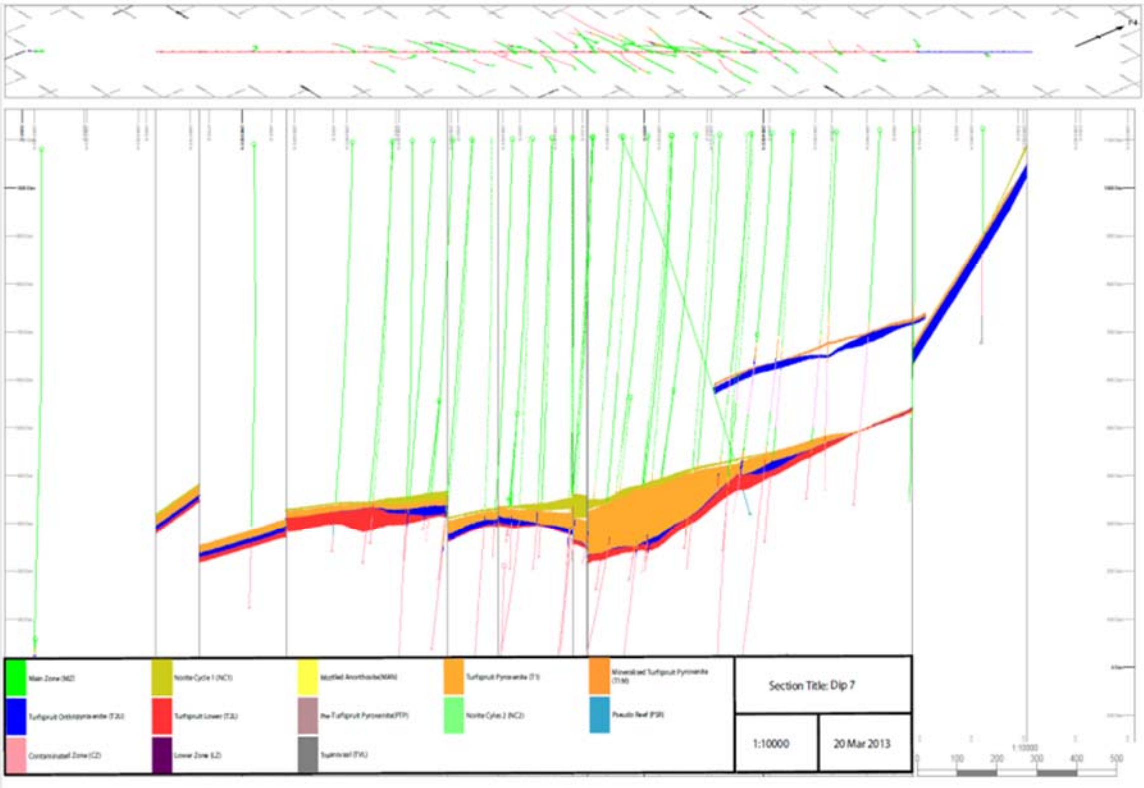
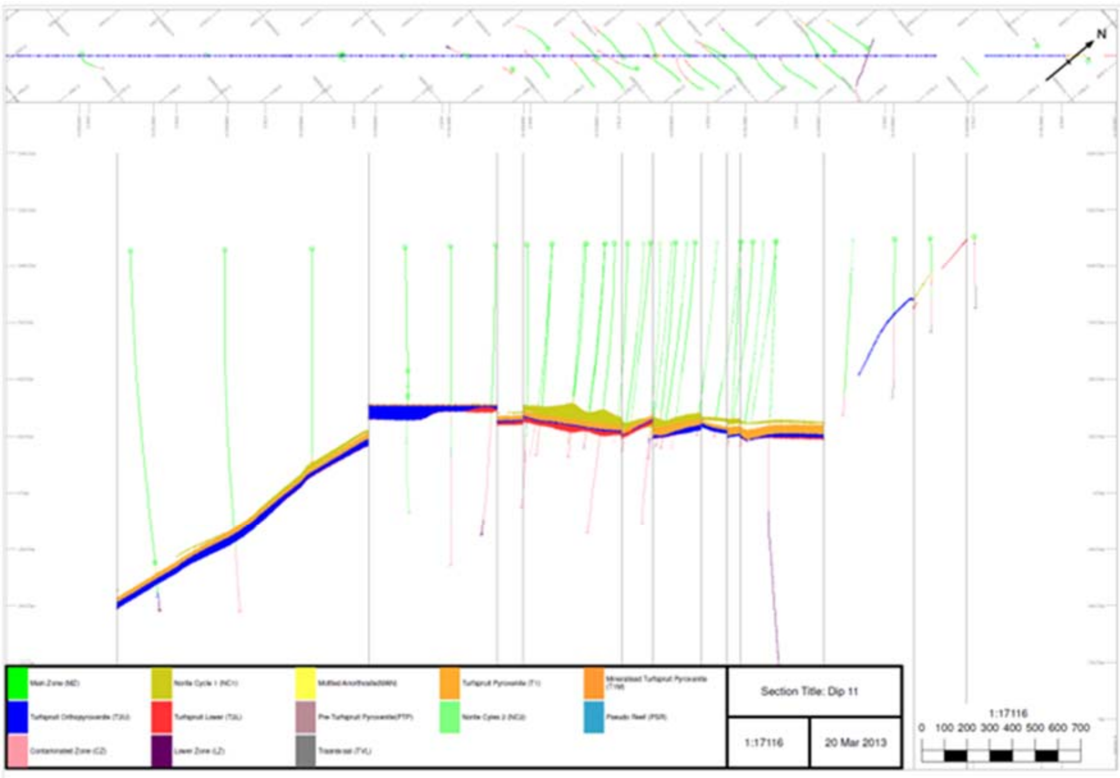


Figure 7.21 TCU Dip Section 7 (inset), Looking North-West



Note: Figure courtesy Ivanhoe, 2013.

Figure 7.22 T1 Wireframe, Dip Section 11, Looking North-West



Note: Figure courtesy Ivanhoe, 2013.

7.3.7 Mineralogy of PGE-Base Metal Mineralization in the Project Area

There are six separate mineralized zones found throughout the Upper Critical Zone on the Project (refer to Table 7.4). The T1 and T2 are by far the best developed and shows good continuity across the property.

The other mineralized zones mostly contain erratic mineralization and disrupted continuity along strike and dip.

The primary magmatic mineralization on Turfspruit 241-KR, in general, exhibits the same geological characteristics as described for the Merensky Reef within the Upper Critical Zone of the BIC.

Although much thicker than the normal Merensky Reef, the Turfspruit T1 and T2 reefs are much less affected by contamination from sedimentary xenoliths than most of the other units below. The Pseudo- and UG2 Reefs found stratigraphically below the TCU usually are less continuous, being disrupted by sedimentary xenoliths and associated contamination/alteration.

PGM and Base Metal Sulphide Occurrence

Unless otherwise referenced, the remainder of this section is based on work conducted on the ATS and AMK deposits by Hutchison (2003), and Hutchison and Kinnaird (2005); and is summarized from the UMT model report (AMEC, 2010b). Much of the work described here was confirmed by mineralogy conducted in association with metallurgical testwork (see Section 13.4).

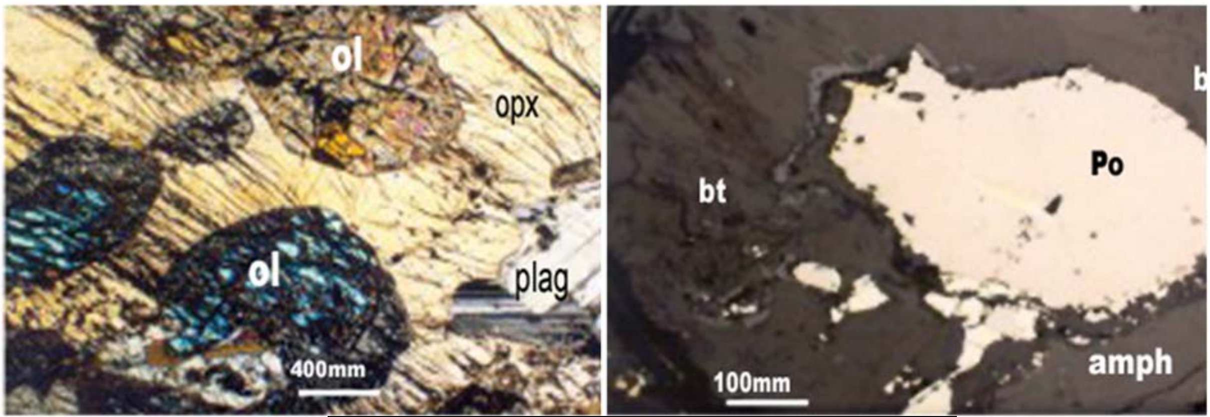
Within the Platreef, both base metal sulphides and PGMs occur as disseminations. The sulphides range from 5 µm to less than 2 cm in size and may form within primary silicates, often as oriented intergrowths, or interstitial to the primary silicates, or within the alteration assemblage of talc, tremolite and serpentine. Much of the sulphide component appears to be associated with intergranular quartzo–feldspathic veinlets, and chalcopyrite is also common in irregular veinlets and infilling small fractures.

There are a wide variety of PGE phases, mainly as tellurides, arsenides, and antimonides of Pd, Pt, Rh, Ag, and variable amounts of Bi occurring as bismutho-antimonides and complex bismuthotellurides. PGMs occur as small micrometre-sized (typically less than 5 µm) satellite grains around composite sulphide grains, or they are dispersed within the primary silicate phases unrelated to any sulphides. PGMs also occur in alteration assemblages dominated by talc, tremolite and serpentine. Less commonly, PGMs occur as inclusions within the disseminated and net-textured sulphides. There appear to be several phases of sulphide formation. An early phase is dominated by irregular blebs of disseminated pyrrhotite and pentlandite, and a later phase exists where chalcopyrite is more abundant, and the sulphides are associated with quartz–feldspathic material, both interstitial to primary silicates and thin veinlets. Figure 7.23 shows core with sulphides in drillhole UMT083 at approximately 1,323 m drill depth. Figure 7.24 shows photomicrographs of four Platreef samples.

Figure 7.23 Core Photograph from UMT083 at 1,323 m Depth, Illustrating Sulphide Mineralization

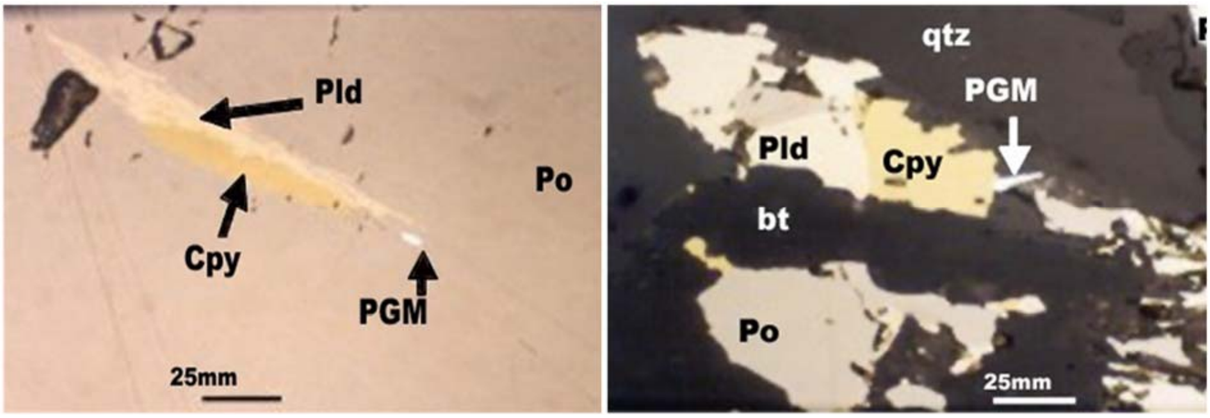
Note: The yellowish mineral is chalcopyrite; the dull purplish mineral is pyrrhotite; the light cream mineral with higher reflectance and some cleavage is pentlandite. Core photographs courtesy Ivanhoe, 2011.

Figure 7.24 Transmitted and Reflected Light Photomicrographs of Four Platreef Samples



Transmitted Reflected

Cpy – chalcopyrite	Plag – plagioclase
Pld – pentlandite	Ol – olivine
PGM – platinum group mineral	Bt – biotite
Po – pyrrhotite	Qtz – quartz
Opx – orthopyroxene	Amph – amphibole



Reflected Reflected

Note: Figure courtesy Ivanhoe, 2003.

Base-Metal Sulphides

Within the Platreef at Macalacaskop and Turfspruit farms, sulphide contents range modally from less than 1% to more than 25%; rare sections of core may have massive sulphides over a scale of tens of centimetres. Textures vary from centimetre-sized blebs to sub-millimetre grains, composed of pyrrhotite, pentlandite and chalcopyrite, to irregular intergranular monomineralic grains composed of pyrrhotite or pentlandite. Veins and net-textured mineralization predominantly consisting of pyrrhotite are common in the peridotites (harzburgites), pyroxenites and hornfelses.

Samples from drillholes show a number of textures, the most frequent being large fractionated blebs together with smaller disseminated monomineralic grains.

The sulphides occur typically as disseminated grains, varying in size from a few micrometres (mainly in serpentinized peridotite and calc silicate rocks) to 2 cm blebs. The base metal sulphides are, in places, intergrown with secondary silicate replacing primary silicates, and in secondary hydrothermal veins traversing the primary or secondary silicates.

In 2010, a mineralogical study of three Platreef UMT metallurgical samples indicated most base-metal sulphides were liberated at a 75 µm grind (Duarte and Theron, 2010).

Platinum Group Minerals (PGMs)

The distribution of the discrete PGMs within the Platreef tends to be broadly controlled by stratigraphic position, with the uppermost part of the Platreef commonly carrying the highest PGE grades. On a hand-specimen scale the distribution can be erratic. Some samples are poorly mineralized, and others contain numerous grains, often with similar PGM minerals clustered together. The majority of the PGM grain sizes in the Platreef and footwall are very small (less than 10 µm to 5 µm), with few exceptions where grain sizes range between 20 µm and 60 µm.

Based on work done by Armitage et al. (2002) and internal research by Ivanhoe, the PGMs identified can be classed according to temperature of formation and sulfosalt affinities as:

- High-temperature alloys; high-temperature semi-metalloids (arsenides and antimonoarsenides).
- Lower-temperature semi-metalloids (antimonides, tellurides, bismuthotellurides).
- Lower-temperature alloys (Pt–Pd–Ge–Pb, Pd–Au, and Au–Ag alloys).

The recognized textural and mineralogical associations of PGM alloys are in base metal sulphides, on the rims of base metal sulphides, oxides, primary silicates, and alteration silicates.

7.3.8 Mineralized Units

The main mineralized units in the Project area are summarized in Table 7.8.

Table 7.8 Cyclic Unit Mineralization

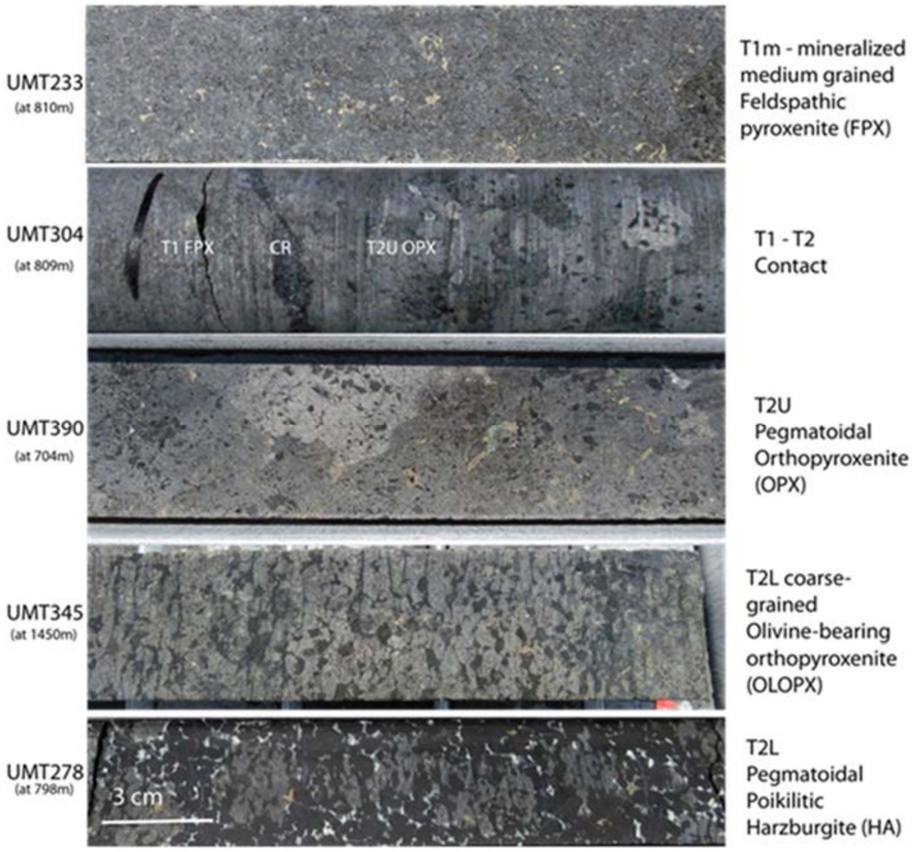
Cyclic Unit	Mineralized Zone	Description
NC1CU	BAR	Fine to medium grained magmatic sulphides hosted in feldspathic pyroxenite. BMS are predominantly chalcopyrite, pentlandite and pyrrhotite.
TCU	T1	Medium to coarse grained magmatic sulphides grains hosted in feldspathic pyroxenite.
	T2	Very coarse grained magmatic sulphides hosted in pegmatoidal orthopyroxenite and pegmatoidal poikilitic harzburgite. The top of the mineralized zone is commonly marked by a chromite stringer.
PSRCU	PSR	Medium to coarse grained magmatic sulphide hosted in pyroxenite and feldspathic harzburgite. 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
Other	Platreef contact style mineralization	Massive sulphide bodies hosted on the Bushveld - Transvaal contact. Predominantly Ni and Cu rich with minor PGEs

Mineralization within the TCU

The main mineralized zone of primary magmatic nature occurs within pegmatoidal T2 rocks at the base of the TCU (Figure 7.25). The vertical distribution of mineralization generally takes the following form:

- In Normal Reef facies (refer to Table 7.7), the pegmatitic harzburgite (T2 Lower) is absent, and mineralization is contained mainly within the pegmatitic pyroxenite (T2 Upper). However, mineralization typically persists a variable distance below the T2 Upper into the underlying pyroxenite-norite-anorthosite NC2 cyclic unit.
- In Pothole Reef facies, mineralization is continuous between T2 Upper pegmatitic pyroxenite and T2 Lower pegmatitic poikilitic feldspathic harzburgite and/or pegmatitic olivine-bearing pyroxenite (T2 Lower). The T2 Upper + T2 Lower mineralization generally shows good lateral continuity.
- A third scenario occurs when the T2 Upper is in contact with contaminated footwall lithologies (Contaminated Zone); however, with careful observation these may be interpreted as either one of the T2 Lower or NC2 units.

Figure 7.25 TCU Mineralization Shown in Typical TCU Lithologies



Note: Figure courtesy Ivanhoe, 2013. Black minerals are serpentinized olivine.

The T2 reef is marked at its top contact by an abrupt change in grain size and texture, usually marked by a chromitite stringer of variable thickness. The sulphide grains also show an increase in grain size across the contact, becoming medium to coarse-grained, within both the T2 Upper and T2 Lower.

An interval of feldspathic pyroxenite (T1) barren of Ni–Cu–PGE mineralization is generally found above the T2 pegmatoid. The thickness of this interval can vary between 0 m and 85 m, depending on the structural setting (Nielsen and Grobler, 2012).

The T1m is a mineralized zone that is perched near the top of the T1 feldspathic pyroxenite. It is an approximately 4.5 m thick zone and contains between 1% and 5% sulphides. Chromitite stringers are only occasionally associated with the zone and tend to occur within the zone rather than marking the top and bottom as seen in the T2. Where present, the PGE mineralization is extremely elevated and shows high (> 1) Pt:Pd ratios.

7.4 Current Geological Mapping and Interpretation Programme

Geological investigations at the Platreef Project are ongoing and include:

- High resolution three-dimensional (3D) seismic surveying was completed in Q1 of 2014. The seismic data will be interpreted during the remainder of 2014 to enhance the understanding of the the Platreef structural setting.
- A detailed review is ongoing of drill core and core photographs noting the bedding angles, breccia zones and fault zones to better identify locations of faults versus folds or rolls in the stratigraphy.
- Detailed re-logging drill core and review of core photographs is also underway in support of generation of better drill correlations of the mineralization of the T1 and T2 units and for correlation of hydrothermal breccias or granitic dykes.

Re-logging of core in the footwall of the TCU is ongoing to decipher the footwall stratigraphy and better understand the geological controls of the footwall mineralization, and to provide information on the controls and stratigraphy of the Contaminated Zone.

During early February 2014, Ivanhoe staff met with AMEC staff in Reno, US, and Ivanhoe provided an update on the status of the preliminary geological mapping and the provisional stratigraphic, structural and Project setting re-interpretions. AMEC notes that when complete, this work will provide a much improved geological model for the Platreef and better define the controls of Platreef mineralization.

As the work is early-stage, a number of hypotheses remain to be tested, and indications to date are of local changes, rather than major interpretational changes in the mineralized horizons, AMEC considers that the geological information used in constructing the resource models in Section 14 remain current. The next resource model update should incorporate all of the findings.

7.5 Comments on Section 7

In the opinion of Dr Parker and Mr Kuhl, knowledge of the deposit settings, lithologies, mineralization style and setting, and structural and alteration controls on mineralization within the AMK, ATS and UMT 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 (with Floor units) located in the north-west portion of Zone 1 presented in this Report and in Parker et al., (2013) is based on close-spaced and more detailed drilling undertaken between 2011 and 2012 and ongoing geological review and interpretation.

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. Recent investigations have shown that the bedding angles in core indicate layers steepen in these areas suggesting flatter-lying zones are connected to each other by slopes. In future model updates, this interpretation will obviate the need for the inclusion of the many small faults contained in the 2013 model.

Locally the faulting may reduce the thickness and grade of the selectively mineable zones; however, this local variability is not expected to materially impact the Mineral Resource estimate.

Re-logging of core in the footwall of the TCU is under way to decipher the stratigraphy and better understand the geological controls of the mineralization. Where space permits, Critical Zone cyclical units (NC2) are observed to form below the T2. This can extend to the development of the UG2. This enhanced geological interpretation will support both Mineral Resource estimation and conceptual mine planning using selective methods.

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 some 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"); and
- Contact-style mineralisation at the base of the intrusion (Platreef-type) occurs mainly in the North Limb.

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

- The Platreef remains a complex and enigmatic deposit.
- Stratigraphic relationships with other stratiform deposits such as the Merensky and UG2 Reefs have been suggested.
- The extent to which the Northern Limb was connected to the rest of the complex across the Thabazimbi–Murchison Lineament (refer to Figure 7.1 where this is shown as the TML fault) remains to be established.
- The Platreef represents a complex of sills intruded into basement granite-gneiss, Transvaal Supergroup sediments or pre-Platreef Lower Zone intrusions.
- Intrusive relationships of the Main Zone gabbronorites, into solidified and deformed Platreef, removes the Main Zone as a source of metals for the Platreef.
- Mineral chemistry, bulk geochemistry, and Sr, Nd, and Os isotope geochemistry of the Platreef are most consistent with an ultramafic (Critical or Lower zone) component.
- Platreef εNd values and $^{187}Os/^{188}Os$ initial isotope ratios overlap clearly with the Merensky Reef but not the Upper Critical Zone.
- 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 mineralization.

Two emplacement models are considered to be the most likely to explain the mineralization (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 text in the previous sub-section reflects study of norites, pyroxenites and harzburgites contaminated with floor rocks that occur on the steep (north-east) limb to depths of 700 m. In places (flatter embayments) thin massive sulphide zones occur at the contact between mafic and ultramafic intrusives and the metasediments of the floor.

Drilling in 2010 on Ivanhoe’s Turfspruit 241 KR property north of Mokopane discovered an area where the Platreef changes from steeply west-dipping to gently-dipping or flat-lying. This Flatreef is under-lain by ultramafic rocks of Lower Zone affinity, where contamination from floor rocks is lessened, and stratigraphic correlation of lithologies in the uppermost Platreef is possible. Following a 2012 re-logging programme completed by Ivanhoe, the Flatreef can be considered as a Merensky-reef analogue. Current Ivanhoe interpretations have the host sequence of lithologies (TCU) corresponding directly to the Merensky Cyclic Unit (MCU). Hangingwall and footwall lithologies correspond to those described for the Merensky Reef in the northern part of the Western Limb. The footwall to the TCU is contaminated by assimilated Transvaal Supergroup siltstones, quartzites and dolomites, which added calcium and silica to the magma. Discontinuous but concordant sulphide bearing lenses occur in the footwall.

Dr Parker and Mr Kuhl consider that the mineralization delineated at the Turfspruit, Macalacaskop and Rietfontein farms is typical of Platreef-style mineralization 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

Drillhole coordinates were given in the Hartebeesthoek 1994 LO29 national coordinate system.

- Origin: the co-ordinates of the origin are defined on the South African National Co-ordinate System at +4,324.66Y, +2,669,596.48X, LO 29°E, WGS84 reference ellipsoid, Hartebeesthoek Datum. The origin is defined as x = +10 000.00 and y = +1000.0.
- Co-ordinate axis: The system is defined as a normal Cartesian co-ordinate system. The Y-axis is a line extending from the origin increasing to the north. The line is oriented north-south. The X-axis is a line extending from the origin increasing to the east. The line is oriented east-west. The Y-axis is perpendicular to the X-axis.
- Measurement of angles: Angles are measured in a counter-clockwise direction from the X-axis.
  - 0°: East (+ X-axis).
  - 90°: North (+ Y-axis).
  - 180°: West (- X-axis).
  - 270°: South (- Y-axis).
- Height datum: The height datum will be mean sea level as indicated by benchmark BMA2 at the intersection of the 00 and N7 geological gridlines. BMA2:= 1,155.36 m.

Conversion from the Hartebeesthoek 1994 LO29 coordinate system to the local project coordinate system is accomplished by:

- X_Local = X_LO29 +14324.26.
- Y_Local = Y_LO29 + 2670596.48.
- Z_Local = Z_LO29.

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. A geological map combining the field mapping with drillhole information is included as Figure 7.4 in Section 7.

9.3 Geochemical Sampling

Geochemical sampling of surface trenches proved to be ineffective in delineating stratigraphic control on mineralisation.

A detailed geochemical study based on the stratigraphic interpretation outlined in Section 7.2 and Section 7.3, and concentrating on selected borehole intersections is currently underway.

9.4 Geophysics

Since Project inception, geophysical survey methods have included aeromagnetics, gravity gradiometer and a number of downhole geophysical methods. The downhole methods include: 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 polarization (IP); density; neutron; induction and vertical seismic profile (VSP).

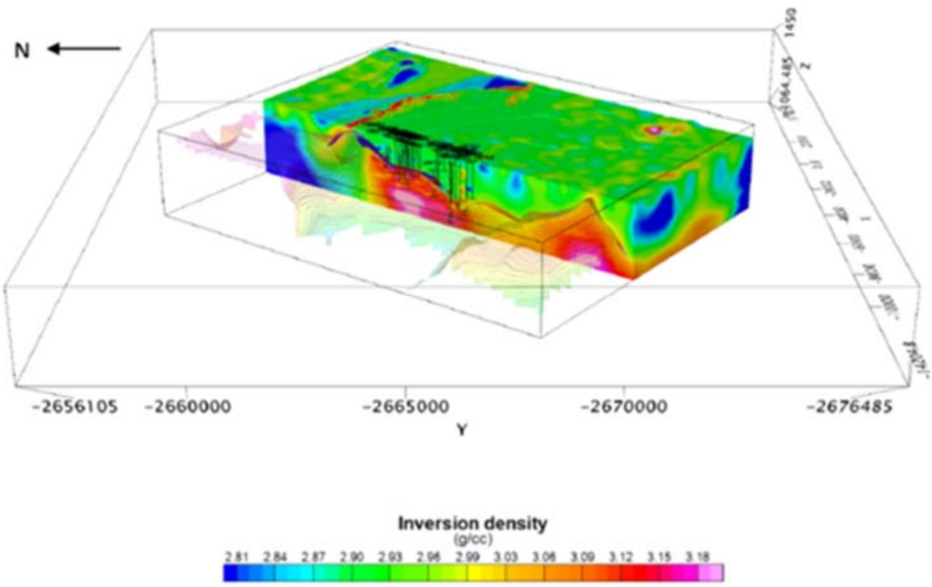
Various downhole geophysical surveys were conducted from 2008 to 2012 within a total of 27 boreholes. The most recent downhole geophysical survey work was conducted for GT008 that was drilled within the area selected as the site of the exploration bulk sample shaft.

In early 2012, Ivanhoe acquired 130 km^2 of Falcon gravity data that were geologically-constrained and inverted by N. Williams of Ivanhoe Australia Ltd., through the use of proprietary algorithms. The Falcon airborne gravity gradiometer system was developed by BHP Minerals (now BHP Billiton) to aid in the discovery of mineral deposits. All rights were purchased by Fugro Airborne Surveys in 2009.

A >2.97 g/cm^3 three-dimensional (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 9.1).

The Falcon data supplements previous geophysical work conducted in the project area and indicates that the Flatreef could potentially extend to the south of Zone 1 for >3 km.

Figure 9.1 Geologically-constrained Falcon Gravity Inversion Interpretation



Note: Figure courtesy Ivanhoe, and sourced from Williams (2012). Inversion sliced along NE oriented section. Image shows computed depth to >2.97 g/cm^3 isosurface which maps the gabbronorite/pyroxenite contact and thereby depicts the approximate structure of the mineralised reef horizon.

9.5 Petrology, Mineralogy, and Research Studies

Mineralogical studies based on samples from the TCU including the mineralised T1 and T2 reefs are currently being planned. In addition, researchers from the University of the Witwatersrand have taken selected samples of the various reefs and host lithologies of the entire UCZ stratigraphy including the Lower Zone intersected below the TCU. Detailed geochemical and mineralogical studies are currently being performed on those samples.

An MSc research student from Cardiff University is currently completing a mineralogical and geochemical study on samples selected across the TCU stratigraphy from uncontaminated rock in selected drillholes from Zone 3.

9.6 Exploration Potential

The Platreef mineralization remains open along strike and down-dip. There is excellent opportunity to expand the extent of known mineralization with further drilling, based on the Falcon information in Figure 9.1. In particular Zone 5 has significant Flatreef exploration potential, as identified by the Falcon geophysical data in Figure 7.13.

9.7 Comments on Section 9

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

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

Continued field work will be required to support any future pre-feasibility study and exploration shaft design and sinking.

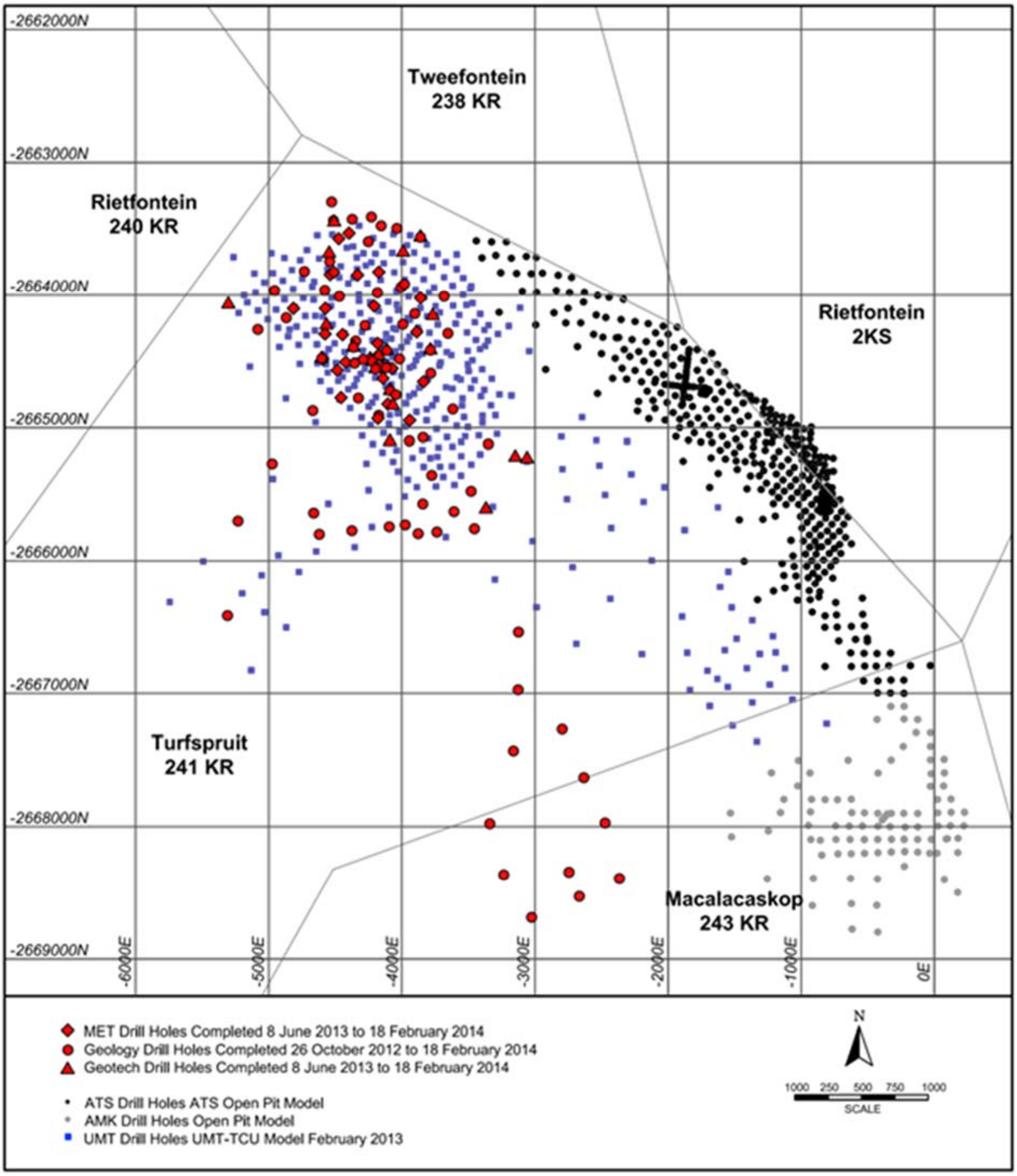
10 DRILLING

10.1 Drill Programmes

Drilling on the Project has been undertaken in two major phases; the first from 2001 to 2003 is termed the open pit programme (designated AMK at Macalacaskop and ATS at Turfspruit/Reitfontein). The open-pit programme drillholes are located in Zone 4 (see Figure 7.7). The second phase commenced in 2007 and is ongoing. This second drill phase is termed the underground programme, is designated UMT (including Bikkuri), and nearly all drilling is on Turfspruit. These drillholes are situated in Zones 1 to 3 and Zone 5.

From the 954 core drillholes (excluding re-drilled mother holes and all deflections) a total of 624,248 m were drilled and completed by 26 October 2012; this included 555 holes (194,591 m) from the open-pit programme and 399 holes (429,657 m) from the underground programme. A drillhole location plan is included as Figure 10.1.

Figure 10.1 Drill Collar Location Plan



Note: Figure courtesy Ivanhoe, modified by AMEC, 2014.

Following a notice from the Department of Mineral Resources (DMR) regarding community concerns over compensation issues, 29 holes and exploration drilling activities were put on hold in October 2012. On 28 May 2013, following ratification of community compensation agreements, the DMR rescinded the Section 93 Directive. Drilling recommenced on 8 June 2013.

From 26 October 2012 to 18 February 2014, a total of 64 drillholes (44,712 m) and 59 deflections (55,886 m) have been completed at Platreef. The additional drillholes have been completed for geotechnical data, metallurgical samples, and geology/resource drilling (Figure 10.1). Nine drillholes were in progress on 18 February 2014.

The additional drilling has included:

- Zone 1 Geotechical data - 23 drillholes (7,708 m) and 7 deflections (7,054 m).
- Zone 1 Metallurgial Samples - 16 drillholes (14,302 m) and 27 deflections (24,639 m).
- Geology Zone 1 - 6 drillholes (5,279 m) and 20 deflections (18,839 m).
- Geology Zone 2 - 4 drillholes (3,528 m).
- Geology Zone 3 - 5 drillholes (4,655 m).
- Geology Zone 5 - 10 drillholes (9,239 m) and 5 deflections (4,353 m).
- Assay data are available for 19 drillholes and 30 deflections as at 18 February 2014.

10.1.1 Zone 4

Drillhole prefixes for the open-pit programme are prefixed AMK; ARF; ATM; ATS; DTS; GT(001 to 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, whilst the ATS initial drill spacing is approximately 120 m to 140 m and generally follows an east-north-east-oriented drilling grid which 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. A “mining simulation” drill grid was completed at a 10 m x 10 m drill spacing (DTS drillholes), and an infill programme (ITS drillholes) was completed to increase the drill density to approximately 100 x 75 m or 75 x 75 m.

10.1.2 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 to 2012 a total of 425,918 m were drilled from 394 drillholes. Drillholes were collared as vertical until UMT105; after that, holes were drilled at an 85° inclination with the exception of UMT330 which had a 60° 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 m to 500 m between holes).

10.1.3 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 ceased due to community concerns over compensation issues. Three drillholes were completed together with their respective three deflections, whilst two drillholes were suspended. A total of 3,739 m were drilled with all holes collared at 85° and completed on a nominal drill spacing of 400 x 800 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 diameter) and HQ (63.3 mm diameter) core, and a limited amount of geotechnical drilling was completed with oriented NQ3 (44.9 mm diameter) core from stabilized 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 2 ending in 2011, whilst drilling in Zone 1 and 3 is on-going, and Zone 5 is the latest explored area. All drilling extracts 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 the average hole length is 1,043 m; the minimum hole length is 212 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 is 300 m to 600 m thick at Turfspruit. 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 galvanized-plate core boxes. Storage facilities consist of lockable brick and corrugated steel sheds where the core boxes are placed on two metre-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 finds them 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 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, whilst there is an option for the geologist to record a comment(s).

Logs were then captured by a geologist into a Fusion database (a product of Datamine®) and independently checked by the onsite database manager.

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 was commonly 100%. The recoveries only show a substantial decrease within faulted/sheared zones.

10.5 Collar Surveys

A contracted certified land surveyor (Mr. Louis Nel) 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.

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 to 583 m. The ATS and AMK drillholes were downhole surveyed using multi-shot Reflex and Maxibor instruments. Multiple survey shots were taken at 3 to 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 to 5 m intervals. In Zones 1 to 3 and Zone 5, there are 30 drillholes without surveys. UMT001 to UMT106 were drilled vertically, while UMT107 to UMT394 were drilled either at -85° or -90°. 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 76 instances where the EMS has been selected, due to erroneous or uncompleted gyro surveys.

10.7 Metallurgical Drilling

The area sampled was Zone 1 and all UMT borehole data was incorporated in order to define a representative characteristic grade distribution per Geomet unit. The lithological base used in sample selection is the Geomet units as modelled, namely the T1, T2 Upper (T2U), and T2 Lower (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 grade shell cut-off data. The selection criteria included 2PE+Au grade, Ni grade, Pt/Pd ratio, and rock type.

The drill map below indicates the holes selected for the Mintek and SGS laboratories on which the metrallurgical 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 is included as Table 10.1.

Table 10.1 Drill Intercept Summary Table

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

10.9 Comparison of 2013 Drilling

Ivanhoe recommenced drilling on the Platreef project on 8 June 2013.

AMEC compared the the available geological data for the additional drilling as at 18 February 2014 to the current resource models and concluded the new drilling in Zone 1, Zone 2, and Zone 3 supports the assumptions made in the geological models. Zone 5 is not part of the current resource model.

10.10 Comments on Section 10

10.10.1 Geology and Resource Drilling

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 mineralization, drill intercept widths are approximately equivalent to true widths for most UMT drillholes. Drill orientations are generally appropriate for the mineralization style. In the open-pit areas, 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 angle 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 mineralization. The sections display typical drillhole orientations for the deposits.
- Drillhole intercepts as summarized in Table 10.1 appropriately reflect the nature of the PGE–Au–Ni–Cu mineralization.

10.10.2 Metallurgical

It is the opinion of the qualified person responsible for the metallurgical aspects of the project, Mr Michael Valenta, that adequate sample was provided for metallurgical test work and mineralogical analysis in order to provide a representative sample for the purposes of a scoping study.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

From the time of Ivanhoe's initiation of the 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 in the Platreef 2014 PEA.

Drill core is sawn in half using a wet saw. A study completed during 2011 by AMEC (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

For the open-pit resource drilling between 2001 and 2003, assay sampling began where the geologist's observations indicated the top of bedrock. Before Q4 2001, some Platreef drill intervals lacking visual mineralization were not submitted for assay. This practice was reviewed in 2002, and additional core intervals were subsequently sampled.

Prior to May 2003, unsampled drilled intervals were limited primarily to soil, oxidized bedrock, and non-prospective hangingwall and footwall rocks. In addition, a small number of xenoliths within the Platreef Complex were deemed barren from geological observations and were assigned assay values of zero for resource estimation purposes. Beginning in May 2003, oxidized bedrock has been assay sampled, and a remedial assay sampling programme including the oxidized portion of drillholes was completed by May 2003.

The sampling of drill core was completed by Ivanhoe's employees at the Platreef project offices in Mokopane.

A sample length of one metre was initially selected for efficient sample handling and preparation. In May 2003, the nominal sample length was increased to 2.5 m, based on recommendations by Francois-Bongarcon (2002).

Dr Francois-Bongarcon determined that a sample length of 2.5 m would not significantly degrade the assay quality, and Set Point Laboratories determined that the larger samples could be effectively prepared for assay.

The depth from the drillhole collar for each sample was recorded in an electronic spreadsheet. Each sample was given a unique identification code. Sample boundaries were marked on the drill core, and the core was sawn longitudinally in half. Ivanhoe employees bagged the half-core intervals and assigned a drillhole identifier and sample number to each sample.

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 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. 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 to 220 samples and include ±10 standard reference materials (SRMs) 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. An Excel spreadsheet is constructed that includes the drillhole ID, laboratory ID and sample number.

Sampling is completed by at least two people. Sample weights are captured in the Excel file for the sample batch. Photographs are taken of each sample displaying the bag sample number and the sample tags and labels inside the sample bag. Sampling is conducted in sets of 10 samples, and after every tenth sample, the samples are inspected to ensure sample numbers are correct, the Excel spreadsheet corresponds, and the sample bags are not damaged.

11.2 Density Determinations

11.2.1 AMK and ATS Bulk Density

At AMK bulk densities were determined for wet and dry rock fragments representing the major lithologies (Table 11.1). The wet determinations include moisture content within the rock. In order to quantify the moisture content, the samples were placed into a drying oven and then sealed in wax. The difference between the two determinations is negligible; this result is consistent with the observed lack of porosity. To be consistent with previously reported "wet" tonnages, the "wet" (i.e. unsealed) determinations have been used for tonnage calculations.

Table 11.1 Average Bulk Densities for AMK Area

Rock Group	Wet (Unsealed) Determinations			Dry (Wax Coat) Determinations		
	No.	Average of Wet Density g/cm³	Weighted Average g/cm³	No.	Average of Dry Density g/cm³	Weighted Average g/cm³
Main Zone	11	2.85	–	9	2.84	–
Platreef	69	3.04	3.07	60	3.03	3.07
Xenolith	29	2.80	2.84	26	2.75	2.80
Serpentinized Peridotite	11	3.06	3.06	9	3.04	3.04
Footwall and Lower Zone	9	3.13	–	7	3.10	–

At ATS, bulk density samples were selected to ensure high-quality coverage, both spatially and lithologically. Samples from various geographic areas and important lithological units were selected from the sawn half-core. Pieces selected were approximately 10 cm long. About 1,088 samples selected from 230 different drillholes were analyzed. Table 11.2 shows mean densities by rock type used for tonnage estimations in the ATS area.

Table 11.2 Average Bulk Densities for ATS Area

Lithology	Average Density g/cm³	Comments
Soil	1.76	From Glover, T.J., Pocket Ref, 2nd ed., Sequoia Publishing, Littleton, CO
Main Zone	2.89	Based on measurements
Serpentinite	3.01	Based on measurements
Hornfels	2.85	Based on measurements
Norite/pyroxenite	2.99	Based on measurements
Floor	2.85	Based on measurements

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:

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

where Ma = Mass in Air; and

Mw = Mass in Water

The database contains over 40,000 density determinations that were recorded from 2007 to 2012 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 11.3, where the proportions of the samples for each broad stratigraphic unit are displayed. More than half the samples taken across the lease area are from the Main Zone (MZ) gabbronorite (GN). A total of 3,005 samples have been taken within the Turfspruit Cyclic Unit (TCU) that is the main focus for Mineral Resource calculations, and over 12,500 density samples from the footwall of the TCU.

Table 11.3 Stratigraphic Unit Density

	Total	Hangingwall			Bikkuri			TCU			Footwall		
		MZ	NC1	MAN	Total	B1	B2	Total	T1	T2	NC2	CZ	LZ
Ave	2.99	2.90	2.98	2.84	3.12	3.12	3.13	3.15	3.18	3.11	3.06	3.1	3.17
Min	2.05	2.37	2.58	2.52	2.6	2.6	2.62	2.57	2.58	2.57	2.61	2.05	2.60
Max	4.45	3.5	3.49	3.02	3.37	3.37	3.37	3.60	3.60	3.42	3.46	4.45	3.42
Std dev	0.17	0.1	0.15	0.07	0.15	0.16	0.13	0.13	0.11	0.16	0.17	0.17	0.19
# of Samples	40,168	22,756	918	176	263	189	74	3,005	1,878	1,127	317	9,816	457

Abbreviations used in this table are explained in Table 7.4.

The separate lithologies from the entire stratigraphy are shown in Table 11.4. The more felsic rocks understandably have a lower SG with the GN averaging 2.91; the orthopyroxenite (OPX) and the feldspathic pyroxenite (FPX) average 3.19. The SG decreases as olivine is added to the system. This is counterintuitive, but is due to the serpentinization of the olivine. Within the TCU the SG decreases with the increase of the olivine content. The olivine bearing orthopyroxenite (OLOPX) averages 3.10 and the HA averages 2.99. The granite veins (GRV) have the lowest SG of any of the rock (2.66) and occur throughout the stratigraphy.

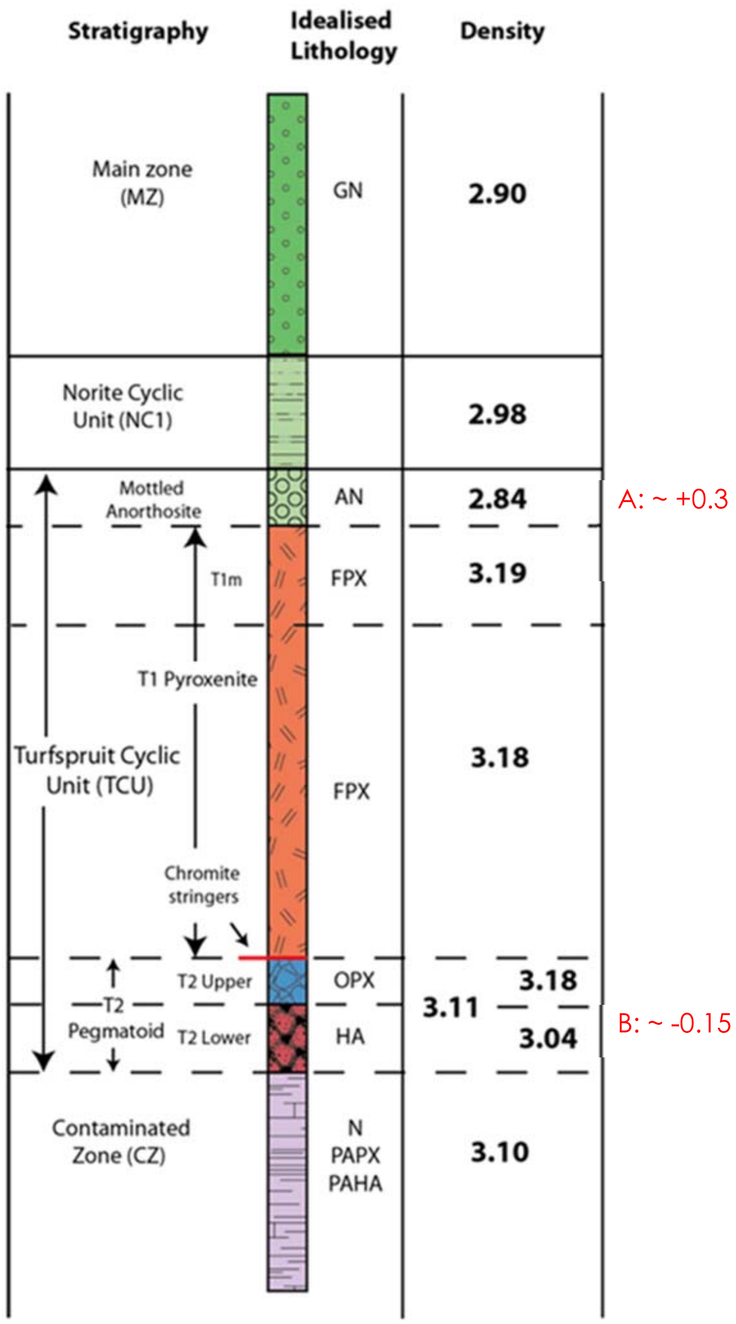
Table 11.4 Lithological Density

	Total	GN	GRV	AN	FPX	OPX	OLOPX	HA	PAPX	PAHA	CR	PX
Ave	2.99	2.91	2.66	2.84	3.19	3.19	3.10	2.99	.12	3.08	3.30	3.16
Min	2.05	2.37	2.44	2.43	2.68	2.78	2.60	2.52	2.05	2.27	3.09	2.13
Max	4.45	3.55	3.48	3.24	3.60	3.41	3.98	3.41	4.45	3.6	3.44	4.3
Std dev	–	0.07	0.11	0.07	0.08	0.10	0.19	0.14	0.17	0.16	0.14	0.16
# of Samples	40,168	20,153	1,195	591	2,447	593	423	628	3,713	831	9	1,413

Figure 11.1 shows an idealised strip log with the associated densities and two horizons of large density contrast are marked A and B;

- There is a ~0.3 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 range of 3.19–3.18) and the HA or T2L (2.96–3.04). When the T2 is combined, the overall average SG is 3.11.

Figure 11.1 Idealized Density Strip Log



Note: Figure courtesy Ivanhoe, 2013.

11.3 Analytical and Test Laboratories

To date, laboratories utilized for the Platreef Project include the primary laboratories Set Point Laboratories (Set Point; Johannesburg, RSA) and Ultra Trace Laboratory (Ultra Trace; Perth, Australia); the check laboratories were Lakefield (Lakefield Johannesburg; Johannesburg, RSA) and Genalysis Laboratory Services (Genalysis; Perth, Australia and Johannesburg, RSA).

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 specialized in the inspection, analysis, audit, and certification, and management systems in relation to regulatory or voluntary standards.

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

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

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

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

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

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

In late 2010, Acme Laboratories (Acme) of Vancouver, Canada, became the check laboratory. The laboratory holds ISO/IEC 17025:2005 accreditation for specified analytical techniques. In the third quarter of 2011, Ultra Trace could no longer accommodate all of the 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.

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 analyzed samples until capacity was reached in 2002. From November 2002 all prepared samples were analyzed by Ultra Trace.

11.4.1 AMK and ATS Sample Preparation

Prior to May 2003, sample bags were transported by a private freight contractor to Set Point in Johannesburg. After May 2003, sample preparation was completed at Set Point's new facility in Mokopane, and samples were delivered the same day they were loaded for transport.

Initial sample preparation by Set Point included crushing and pulverizing the entire sample to a nominal grind of 60% passing 200 mesh (75 µm) using a jaw crusher, a rolls crusher, and Labtechnics LM-2 pulverizers. In March 2002 a more stringent grind of 90% passing 150 mesh (106 µm) was established.

In August 2002, long grinding times began to outstrip the capacity of the Set Point preparation laboratory. Consequently, a splitting step was introduced between the sample crushing and the final pulverization. Francois-Bongarcon (2002) specified a criterion of 95% passing (p95) 10 mesh (1.7 mm). This was the desired average, and to implement it, a lower control limit of 90% passing 1.7 mm was specified. This has resulted in a test average, excluding failures that trigger re-grinding, of about 94% passing 1.7 mm.

11.4.2 AMK and ATS Sample Analysis

Samples were assayed by Set Point until 2002 when the capacity was exceeded. After November 2002, all samples were submitted to Ultra Trace for analysis after sample preparation was completed at Set Point.

Set Point analysis initially included Au, Pt, Pd, Rh, Cu, Ni, S, Cr, Co, V, Rb, Sr, and Sc. Au, Pt, and Pd were assayed by fire assay with a lead collector. The dissolved bead was analyzed by inductively-coupled plasma (ICP). Rhodium (Rh) was determined in a separate fire assay utilizing a gold inquart. The other elements were determined by X-ray fluorescence (XRF) analysis of a pressed pellet of sample pulp mixed with a binding agent. The sulphur and Rh assays were discontinued in October 2002 due to their expense and believed limited usefulness at the time.

Ultra Trace performed a similar fire assay to determine Au, Pt, and Pd by using an ICP mass spectrometer (MS) finish that provided a lower detection limit of 1 ppb. Ultra Trace did not assay for Rh. Ultra Trace determined Cu and Ni by multi-acid digestion sufficiently robust to provide dissolution of all minerals (“total” metal assay). For a short time, other metals were assayed by XRF using the same protocol as Set Point. In December 2002, the assay suite was reduced to Au, Pt, Pd, Cu, and Ni.

After May 2003, a separate protocol for oxidized samples was introduced. This involved relatively few samples showing signs of oxidation: Ni and Cu were analyzed via aqua regia (partial) digestion and standard “total” acid digestion. Fire assays were used to analyze Au, Pt, and Pd.

The p95 -1.7 mm coarse reject was saved. For every 20th sample, a split of the coarse reject was inserted into the sample stream and a “CRD” suffix (designating coarse-reject duplicate) was added to the sample name. These served as a quality check of the splitting protocol.

Set Point completed four screen tests of coarse reject samples each shift. The average of the four tests is recorded and was considered to constitute a failure (warranting corrective steps) if the mean fell below 90% passing 1.7 mm. In addition, approximately 2% of the coarse rejects were sent to Lakefield Johannesburg to check compliance of crush size specifications.

Upon receipt at Ultra Trace, approximately 2% of the sample pulps are tested by wet-screening a portion through a 106 µm screen. If more than one sample in the submission fails the test, the entire submission is repulverized, and the same samples are re-tested to confirm the grind criterion has been achieved.

All pulp rejects and coarse rejects are returned to the Platreef facilities in Mokopane.

11.4.3 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 Platreef. 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 percent 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 Platreef.

After return to the 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 Ultra Trace. From the fourth quarter of 2011, samples have also been shipped to Genalysis (Johannesburg) and ALS Chemex (Vancouver) and Acme (Vancouver).

During 2012, Genalysis and Ultratrace were used as both primary and check laboratories on work performed, with Ultratrace re-assaying Genalysis samples and vice-versa. The same laboratories are used for the 2013–2014 drill programme that is currently in progress.

11.4.4 UMT Sample Analysis

Ultra Trace used a multi-acid digestion followed by ICP-OES reading to determine total Ni, Cu, Cr, and sulphur. Samples were also assayed for sulphur using a LECO furnace (controlled combustion of sample pulp with infrared reading of SO2 gas); the LECO and ICP sulphur results show close agreement. Lead flux (collector) fire assays with an ICP-MS finish were used to determine Pt, Pd, and Au.

Set Point used the following assay methods (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; Ni (Code 255).
- S by Leco (Code 255).
- Fire assay Pd collector followed by ICP-MS for Au, Pt, Rh (Code 415).

- NiS Collection for Au, Pt, Pd, Rh, Ir, Ru, Os (Code 419).

Ultra Trace used the following analytical methods:

- Fire assay lead collection followed by ICP MS for Au, Pt and Pd (Doc 600).
- Total acid digestion followed by ICP OES for Cr; Cu; Ni and S (Doc 200).
- Small-scale aqua regia digestion followed by ICP OES for Cr, Cu, Ni, S (AR201).

Genalysis used the following analytical methods:

- Fire assay lead collection followed by ICP MS for Au, Pt and Pd (method code FA25/MS].
- Aqua regia digestion followed by ICP/OES for Cu, Cr, Ni, S (method code AR01/OM).
- Multi acid digestion followed by ICP OES for Cu, Cr, Ni, S (method code 4A/OM).

11.4.5 Check Sample Analysis

Check sampling at ALS Chemex used a fire assay technique followed by ICP MS for Au, Pt and Pd (ALS method PGM-MS24). A four-acid-mixed acid digest followed by ICP-MS was completed to determine a 48-element suite (ME-MS61). Sulphur assays were performed using a LECO furnace (S-IR08).

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, S (4A/OM).
- Sieve test as indicated by individual sample breakdown (SV02).

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

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

ACME used the following protocols:

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

11.5 Quality Assurance and Quality Control

11.5.1 AMK and ATS QA/QC

Control Samples

All laboratories used in the Platreef Project exercise quality control in the form of duplicates, SRMs and blanks. These controls are included in each assay report.

After November 2001, Platreef inserted "blind" quality-control materials with each submission. The insertion frequency ranged from 3% to 6%. The controls included in-house prepared standard reference materials (SRMs) with "best values" determined by round-robin submission to five reputable independent commercial laboratories located on three continents: Lakefield and Set Point (RSA), Genalysis and Ultra Trace (Australia), and G&T (Canada).

All control results were analyzed for Au, Pt, Pd, Cu, and Ni. Failures were identified, and pulps from batches associated with failures were re-assayed. Platreef blind duplicate results (Set Point only) were analyzed to identify outliers and sample mis-orderings.

Blanks

Blank material used during drilling of AMK and ATS drillholes consisted of unsampled drill core of barren basement rock drilled at the end of each drillhole. Two to 10 m of basement rock was routinely drilled to obtain a reliable contact depth. The basement rock reliably returned Au, Pt, and Pd results of less than 20 ppb, but had low concentrations of Cu and Ni (low hundreds of ppm).

One or two blanks were submitted per drillhole. These were placed between samples visually identified as mineralized, in order to detect contamination during preparation. Results of blanks were compared to the results of the preceding sample number, under the presumption that in most cases this sample would precede the blank through the same crusher and pulverizer. No trend of the blank grade compared to that of the preceding sample was seen, except for a brief period when the blank did show a slight upward trend in grade (low tens of ppb for PGMs) with respect to grade of the preceding sample. Although the effect was too small to impact Mineral Resource estimation, it was investigated. It was found that technicians had stopped cleaning the spoon used to transfer pulp when splitting the pulp samples for shipping. When cleaning recommenced, the correlation ceased.

After a new project geologist elected to increase sample interval lengths of some rock types up to 6 m, blanks started to show episodic high results. AMEC investigated and found that this was caused by the need to use two or more trays for drying the sample in the preparation laboratory when the sample interval exceeded about 1.5 m. There was only one sample tag, and trays were getting mixed up. In conjunction with the Set Point laboratory manager, AMEC introduced additional safeguards to prevent this, and blanks returned to normal performance. Blank results indicate that contamination and sample mix-ups were sufficiently rare as to present no risk to Mineral Resource estimations.

Standard Reference Materials (SRMs)

Very few SRMs were commercially available for PGEs, and the decision was made to formulate in-house SRMs.

Initial Platreef SRMs were constructed from a selection of pulp rejects. After the change in preparation protocol in August 2002, coarse rejects were used to make 20–50 kg of SRM material. These SRMs from the coarse rejects were prepared and tested for homogeneity by SGS Lakefield Laboratories in Johannesburg. Materials showing inadequate homogeneity were re-milled or discarded. Each reference material was rotary split multiple times to produce single-use packets of approximately 80 g. The packets were thoroughly randomized to prevent any slight variations in grade (introduced during the splitting process) producing misleading shifts in the obtained average results over different time periods.

The round-robin consisted of submitting multiple packets of each reference material to a minimum of five independent commercial assay laboratories well-versed in assaying for the elements of interest. The best value is taken as the median of the population of each laboratory's mean. Using the median counters the effect of any extreme results produced by one laboratory, and eliminates the need to reject any laboratory's results.

Packets of SRMs were routinely inserted into every sample submission. Because the preparation and assay laboratory are run by separate companies, the insertions are blind to the assay laboratory. The SRMs provide a reliable check on laboratory performance.

Check Assays

Check assays were performed at Lakefield until June 2002. After June 2002, check assays were performed by Genalysis Laboratory (Genalysis) in Perth, Australia. The check assay programme included the same assay suite as the original assays. In addition, the check assays used a less robust aqua regia digestion for Ni and Cu.

In addition to the blind controls, every 20th sample pulp was submitted to an independent laboratory for check assay. Blind reference materials were included in each of these submissions. These results were analysed and combined with the control analysis to identify samples for re-assay.

A separate programme to validate the routine method of analysis for PGE (fire assay by lead collection) included assaying 2% of samples by nickel sulphide collector.

11.5.2 UMT QA/QC

Control Samples

As is prevalent throughout the industry, all laboratories employed by the 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 Project inserted coarse reject duplicates, field blanks, and packets of CRMs in order to independently monitor laboratory performance.

Blanks

Blanks utilized natural rock materials that have less than 5 ppb concentrations of Au, Pt, and Pd, but have copper concentrations of 10 to 20 ppm and Ni concentrations of 20 to 30 ppm. Blanks underwent preparation steps and therefore provide an upper limit on levels of contamination caused by preparation.

More than 1,700 blind blanks were included within the Ultra Trace submissions in 2010–2011. Very high levels in a blank for several elements would indicate sample mix-ups. None of the inserted blanks showed very high levels for multiple elements, although approximately 20 samples (<0.5%) showed moderately elevated levels for several elements, which might be ascribed to either a sample mix-up or to contamination during preparation. Excluding one anomalous Au result of 74 ppb, Au results for blanks did not exceed 15 ppb. No Pt or Pd results exceeded 70 ppb, and excluding two samples with anomalously high levels of Au and Pt and Pd, no blank result exceeded 40 ppb for Pt or Pd. Average blank results for Au, Pt, and Pd are all less than 3 ppb. The maximum Cu and Ni blank results are 122 and 548 ppm respectively. Excluding the highest blank Ni result, which had elevated results for Cu, Cr and S, the highest Ni result in a blank is 250 ppm. Performance on blanks is adequate for Mineral Resource estimation purposes.

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 certified reference materials (CRMs) purchased from commercial African Mineral Standards (AMIS, Johannesburg), and/or in-house SRMs made from composites of drill sample coarse rejects that were prepared by SGS (Johannesburg), with best values assigned by AMEC based upon round-robin results. Details are provided in Reid (2011) and Long (2010). In-house SRMs were phased out as appropriate materials became available from AMIS.

Fifteen CRMs and SRMs were used extensively enough to compare Ultra Trace's mean results of each for comparison to best values. Excluding outliers that triggered follow-up investigation (for control insertion mix-ups) and in very rare cases remedial re-assaying of some laboratory batches, the average of the Ultra Trace results is within 10% of the certified value for the major elements of interest (Ni, Pt, Pd, Au, Cu) and in most cases for the added element, 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. 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 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 mineralized 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.

11.6 Databases

The drillhole data are 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 in the second quarter of 2013.

11.6.1 AMT and ATS Data Entry

All geological information from drilling logs was double-entered into the computer by two different individuals. A relational database was used to compare the two files and to identify entry errors. Each non-matching dual entry was checked, and the correct entry identified for the final database.

All assay reports were transmitted from the laboratory to Platreef via email or electronic bulletin boards. The transmitted files are imported into an Excel template and checked for proper formatting, extraction of duplicate data and header information, and for reasonableness checks using various metal ratios. The checks identified records with missing or erroneous entries sourced at the laboratory.

Data were managed in Access and in SQL Server. Back-ups of data onto hard media (CD, DVD) were routinely performed. Some back-ups are stored in Ivanhoe's offices in Johannesburg.

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 (co-ordinate surveys, total depth, downhole 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 authorized 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.

11.7 Sample Security

AMT and ATS pulp rejects and coarse rejects are returned to the Platreef offices in Mokopane, where they are 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 PGE, Au, Ni, Rh and Cu 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, vary from 1 m to 2.5 m intervals in the AMK and ATS areas, and are 1 m intervals in the UMT area; the sample intervals are considered to be adequately representative of the mineralization.
- 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 PGE–Au–Ni–Rh–Cu deposits.
- Core drill programmes were analyzed 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 co-ordinates, 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 reviews and those performed by independent consultants. A database audit was performed by AMEC during 2012 to ensure its suitability for resource estimation. A further audit has been made in 2014.

12.1 McDonald Speijers Audit (2002, 2004)

Ivanhoe contracted McDonald Speijers (MS), an Australian mining consultancy, to review the technical aspects of the Platreef project in 2002 and 2004 (McDonald and Speijers, 2003 and 2004). After the first review, MS stated that the sampling and assay programmes were in general appropriate to the type of mineralization, the QA/QC programme was consistent with good industry standards, and mineral resource modelling construction was very thorough and competent. MS also recommended incorporation of the geological interpretation into the resource models, as the data became available. This was done in the next model update (model Q). They also recommended further improvements in modeling via mineralized zone interpretations. These modeling recommendations have been implemented where data are available to support them.

12.2 External Review of ATS Model (2003)

Dr J.M. Rendu (2003) completed a peer review of the inverse-distance weighted grade model for ATS that was available in March 2003. He found the model formed a reasonable basis for the evaluation and reporting of mineral resources and concluded that the model could be used as a basis for preliminary studies.

12.3 AMEC AMK and ATS Database Reviews (2007, 2010)

In addition to automated data verification procedures, spot inspections of the drill logs were completed. There was a period of time where barren gaps in the drill core were unsampled and unlogged. These gaps were re-examined, and the uncut core portions were identified and logged. Unsampled intervals were either sampled or assigned grades equal to zero.

AMEC completed a database audit in June 2007 for all available drilling at that time (DaSilva, 2007). Under the supervision of AMEC, Ivanhoe personnel completed a 5% random check of the ATS and AMK exploration database and a 100% check of collar coordinate surveys of holes used in resource estimation. The records selected for inspection were checked against primary sources of information (assay certificates, drill logs, survey certificates). No issues that could affect Mineral Resource estimation were noted.

12.4 AMEC Site Visits

12.4.1 Site Visits by QPs During UMT Drilling

During the April 2010 site visit (Kuhl, 2010), AMEC undertook the following:

- Compared database entries against supporting documents for collar, survey, density and geology coding for five UMT drillholes.
- Provided suggestions for improvement of database procedures.
- Performed field checks for nine drillhole collars using a Garmin GPS unit.

No significant errors were noted that could affect Mineral Resource estimation.

Dr Harry Parker visited the project site in March 2011 and reviewed logging of Modpak in 12 drillholes (Parker, 2011). The Modpak interpretations have since been superseded by the geological interpretations discussed in Section 7.3.2. Mr Kuhl also visited site in July–August 2011, and observed drilling operations and reviewed Modpak logging.

Mr. Kuhl later visited the project between 25 January and 2 February 2012. During this site visit, the new geological interpretation for the TCU was reviewed in both cross-section and drill core. Mr. Kuhl also visited drilling locations.

Dr. Harry Parker visited the site from 16 to 22 November 2012. He inspected core and checked new logging units in nine holes; he verified collar coordinates for 10 holes using a hand-held GPS unit; he also 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 Project between 25 November and 12 December 2012. During this site visit, the geological interpretation for the TCU was reviewed in both cross-section and drill core, preliminary exploratory data analysis was completed, and work began on constructing the geological model. Mr. Kuhl retrieved 20 witness samples from Set Point (Mokopane) and supervised the packageing and shipment to Ultra Trace Laboratory.

12.4.2 Other 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.

Dr Parker visited the site on numerous occasions during the 2001–2003 AMK-ATS drilling programme. He reviewed all aspects of the geology and resource modelling for the open-pit Mineral Resources.

12.5 AMEC 2012 Database Reviews

12.5.1 August 2012 Review

In August 2012, AMEC compared the collar, downhole, geology logs and assay data (Au, Pt, Pd, Ni, and Cu) from the previously-audited database (August 2010) to the August 2012 database as a check on data integrity (Yennamani, 2012). The additional drillholes reviewed were completed after the database close-out date for the Mineral Resource estimate. The review noted:

- Some discrepancies between the original collar survey documents and the database; further investigation was recommended to resolve the differences.
- Minor errors with downhole survey and lithology data; these were corrected in the database by Ivanhoe staff.
- Minor issues related to the import of sample results from samples submitted for re-assays were found during the database audit check; these were corrected in the database by Ivanhoe staff.

AMEC considered that at the close of the review, that 99% of the Ivanhoe assay database conformed to assay certificates from the laboratories and found that the assay database is acceptable to support future Mineral Resource estimation.

12.5.2 December 2012 Review

In December 2012, AMEC compared the downhole, geology logs and assay data (Au, Pt, Pd, Ni, and Cu) from the previously-audited database (August 2012) as a check on data integrity (Yennamani, 2013). Eight new drillhole collars were added to the December 2012 database; information was checked on five of these holes. Collar data were unavailable for review. No issues were noted with the downhole surveys or assay certificates. Minor discrepancies were noted with lithology coding in the database when compared to the logs; these were sent to Ivanhoe staff for correction. The assay database was considered acceptable to support future Mineral Resource estimation.

12.6 Quality Assurance and Quality Control Results

12.6.1 AMK and ATS QA/QC

The open-pit resource drilling programme was one of the largest sampling and assaying programmes executed in South Africa at the time.

The original sample preparation protocol greatly exceeded usual industry practice; the entire sample of half core was pulverized. A change of protocol to produce a 10 mesh product and then pulverise a split of that product was considered. This is the most typical protocol used for sample preparation. A sampling expert evaluated the protocol, which is a 'best practice'. The controls on that protocol (coarse reject duplicates, daily screen tests and second-party screen tests) exceed industry standards.

Initial assaying was typical of that obtained in South Africa. Improvements were made, commencing in 2001, with the introduction of blind standards, blanks, extensive check assays and remedial assays. Check assays on samples assayed prior to 2001 have not identified any problems prior to these improvements. Samples identified as having inaccurate results were re-assayed. The re-assays are subjected to the same quality control. This approach exceeds industry standards.

Ivanhoe's rigorous data entry and validation programmes were above industry-standard.

In 2010, AMEC attempted to again review QA/QC data for the open-pit resource drilling, but was unable to do so because the data were being migrated from an old server to a new database. However, as the original QA/QC programme was set up by AMEC, and AMEC personnel made periodic visits to check on implementation during 2001 to 2003, AMEC is of the opinion that the resource drilling results for the area amenable to be mined using open-pit methods are valid to support Mineral Resource estimation.

All final assay results are suitable for use in Mineral Resource estimation.

12.6.2 UMT QA/QC

AMEC obtained and reviewed the available QA/QC data for the UMT drilling. Blanks, duplicates, and CRMs are checked when results return and 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, including reports by ALS Chemex, and results from some of these jobs are pending. A few failed batches by Ultra Trace have been remediated.

AMEC noted:

- All Ultra Trace means on SRMs are within 5% of recommended values for the five major elements of economic interest (Ni, Cu, Pt, Pd, Au). 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. AMEC is unsure of the cause for this issue, but suggests additional samples of the blank material be submitted for assay in order to better determine the Cu and Ni content of the blank. SRMs with low Cu and Ni content should accompany these samples.
- Genalysis results for Cu, Pt, Pd, and Au were in line with the SRMs, but Genalysis showed a low bias for Ni. The bias varies from batch to batch but is about 8% to 12% low overall. Results in the >6,000 ppm Ni grade range show good agreement. There are 13 pairs of results where Ni is greater than 6,000 ppm Ni, and these do not agree well between laboratories. There are not enough data in this range to be conclusive. AMEC 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 are approximately 10% higher for PGM fire assays compared to Ultra Trace results. Inserted CRMs in both Ultra Trace and Acme submissions indicate this can be accounted for by a slight low bias in the Ultra Trace results and a slight high bias in the Acme PGM results. The Ultra Trace results likely slightly underestimate PGMs 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.

Ivanhoe supplied results from coarse reject duplicate samples for Au, Pt, Pd, Ni, and Cu assays. Successful or 'passing' duplicates were identified by calculating the Absolute Value of the Relative Difference (AVRD).

AMEC evaluates the duplicate samples by calculating the 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%: AVRD values are 20%, 11%, 7%, and 5%, respectively. Pt exceeded the threshold, with AVRD values of 28% at the 90th percentile. AMEC noted that Ivanhoe were not submitting pulp duplicates as part of their QA/QC programme, and recommended that Ivanhoe use Ultra Trace's reported pulp-duplicate results to assess the precision of pulp duplicates.

12.6.3 QA/QC Drilling Completed Between March 2011 and June 2012

AMEC performed a review on the QA/QC data available for drilling completed between March 2011 and June 2012. Results were:

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 Project's increased drilling rate necessitated using Genalysis and Set Point laboratories, in addition to UltraTrace. AMEC separated the results by laboratory and calculated each laboratory's median result for each element of interest for each AMIS Certified Reference Material. 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 UltraTrace 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 UltraTrace 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.

12.7 AMEC Witness Samples

Three groups of witness samples have been collected at Platreef by AMEC, in April 2010, February 2011, and November 2012. The purpose of collecting these samples is to confirm the presence of mineralization.

12.7.1 April 2010

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

12.7.2 February 2011

AMEC (Long, 2011a) first identified diamond drillholes (UMT prefix) having mineralized intercepts of PGEs (Au, Pt, and Pd). These were divided into ten groups based upon their drillhole number, to provide grouping by similar time periods. There were approximately seven drillholes in each group. A drillhole was randomly selected from each group. The best mineralized intercepts were identified in each drillhole. The intercepts were compared to available information on sample intervals that had been re-sampled, including AMEC's previous witness samples, material taken for metallurgical study, and material collected for mineralogical studies. In cases where a mineralized interval had already been sampled, a different drillhole was selected. The follow up selection was made so that the selected drillholes had collar locations spread across the entire mineralized extent of the project drilling.

In order to reduce the impact of core shifting within a core box, mineralized intervals were assayed in their entirety. A minimum of two core boxes of samples was maintained, with as many as five core boxes used in some cases. Enough core boxes were selected so that lower-grade intervals (near background) would be included. Once a core box was selected, all of the core in the box was quartered, and all the quartered core sampled, except for the partial samples in the first and last core box selected for each drillhole.

Quarter core samples were prepared in the same way as routine samples: dried, weighed, crushed to greater than a control limit of 90% passing 10 mesh (using a jaw crusher for primary crushing and about 10 seconds in an LM2 as secondary crushing, and a criterion that an average sample achieves 95% passing 10 mesh), then split in a riffle splitter once or twice to obtain a sample pulp with a nominal weight of 500 g. The pulp split was then pulverized in an LM2 to a criterion of >90% passing 106 µm. Pulverization was checked by screen tests.

All samples were submitted to Ultra Trace for the current standard suite of analysis: Au, Pt, and Pd by Pb 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).

For the 260 samples collected, 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, and these differences are statistically significant by a two-tailed student's t-test at the 5% level. This means there is less than a 5% chance that a re-sampling of the half core population would result in the mean observed in the quarter core witness samples. However, the results of the inserted CRMs indicated that the Pt and Pd results had a low (but within the acceptable range) bias for Pt and Pd, or around 5%, and the CRMs associated with the original results for these samples did not.

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

12.7.3 2013

A third set of witness samples were taken in November 2012, and assay results received in January 2013.

Original and witness assay values were compared for Pt, Pd, Au, 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.

Comparison of means of witness samples to means of original results show agreement within 5% for base metals, sulphur, and Pd, but not for Au and Pt. 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.

Paired student's t-tests on each element found no significant differences between means at the $p<0.05$ level. A non-parametric statistical test, the Sign Test, was performed on the Au and Pt results; this 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 by chance (at the $p <.05$ level). In the case of Pt, nine out of 20 pairs had a lower Pt result for the original assay.

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

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

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

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.

12.8 Verification of Grind-Assay Function

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

12.9 Comparison of UltraTrace and Mintek Assays

In the first half of 2013, AMEC conducted a number of comparisons of UltraTrace (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, conform to the UltraTrace assays which are the basis for the Mineral Resource estimates. The evaluation commenced with a January 2013 submission of stored pulp splits of exploration drill samples corresponding to drill sample intervals that were used to make up a 2012 bulk sample for metallurgical test work at Mintek.

Mintek's assays included fire assay results for Au, Pt, Pd, and some Rh (only on samples with elevated PGEs); Leco total sulphur; and two sets of ICP (optical emission) determinations for base metals using an aqua regia digestion and a more robust fusion (followed by acid dissolution of the fused pellet) method. These Mintek assay methods were the same as those used for Mintek's metallurgical test work.

The principal finding from this initial submission was a high bias of 10 to 15% in Mintek's Ni results; this was shown both by the blind insertions of AMIS standards and by comparison with the original UltraTrace results on a split of the same pulp. In March 2013, AMEC informed Mintek of their poor Ni accuracy and requested re-assay using an atomic absorption instrument. AMEC 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 also made a new submission of blind Platreef standard reference materials (former in-house standards) together with AMIS certified reference materials much more of greater variety and number than what was included in AMEC'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 investigation of Mintek internal quality controls revealed that they were relying upon two standard reference materials that had not been assayed by any other laboratory.

Later in March 2013, Platreef QC Manager Annelien Parsons obtained all available pulp rejects from Mintek test work, together with Mintek's assay results for those materials. These samples included various kinds of tails and concentrate samples. A few samples of Mintek's standard reference materials were obtained as well. These samples were submitted to UltraTrace for base metal analysis in April 2013. The UltraTrace 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 (2013):

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

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 standard reference materials were also found to overestimate Ni by a similar amount, indicating a long-standing high bias in Mintek Ni results that covers all the Mintek metallurgical test work performed on Platreef samples in 4Q 2012 (Long, 2013b).

12.9.1 February 2014 Data Review

AMEC obtained drill sample assay, location (downhole suvery and collar location),and geotechnical (drilling recovery) data exported from the Platreef project's databases (as ASCII csv format computer files) along with electronic copies (Adobe PDF, or ASCII CSV files) of source survey information. This information covered the period 8 June 2013 to 18 February 2014.

As at 1 March 2014, AMEC had verified over 95% of the assay results against copies of original laboratory reports. All sample assays have been accounted for, and no disagreements between the assay reports and the database report were found.

Approximately 10% of the assay reports underwent remedial follow-up assays as a result of portions (batches of approximately 24 samples) not passing Platreef's stringent external blind quality controls, consisting of blanks, AMIS certified reference materials, and coarse reject duplicates. Of the remedial assays, two samples showed small (<0.1 ppm) differences for fire assay results, and Cu results from one report where Cu was reported as <2 ppm, was entered in the database as 2 ppm. AMEC recommended these assays be assigned as 1 ppm Cu.

The quality control coverage of sample submissions is consistent and thorough. In cases where the result falls outside a control limit, it is usually near the limit, and the remedial results of the re-assayed samples are typically similar to the original results with a few rare exceptions. The re-assayed CRMs usually returned results inside control limits upon re-assay.

Duplicate performance shown by coarse rejects and by pulp sample results demonstrates very good precision. There are very rare samples having Au grade greater than about 0.75 ppm, which show substantial variability, indicating a gold particle size (nugget effect) is likely present.

All the collar surveys were checked against the surveyor's source document, which is reported in Platreef's local grid coordinate system. This audit only verified the local grid data entry and did not investigate any grid coordinate conversion calculations.

Downhole surveys were checked for anomalous deviations using AMEC's proprietary software method. Some drillholes have EMS surveys which depend upon magnetic readings and can therefore be influenced by magnetite. The majority of 2013 drillhole surveys are gyroscopic and do not depend upon magnetic field readings. The typical spacing of readings for either of these surveys is either 3 m or 6 m. A small number of anomalous readings (less than 100 out of more than 10,000) have been identified for further review. Based on review of these thus far, it is estimated that about 20 survey data points may be recommended for rejection. This will not noticeably impact drill sample locations.

As in previous campaigns, drill core recovery in the data reviewed for this campaign was found to very good with very rare exceptions.

AMEC finds the drill data provided in the database extract that comprises drilling from 8 June 2013 to 18 February 2014 to be adequate for Mineral Resource estimation purposes.

12.10 Comments on Section 12

AMEC 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 and Mr Kuhl, sufficient verification has been conducted to provide support that the data collected are suitable for use as a basis for Mineral Resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Sampling and Sample Analysis

SGS performed metallurgical test work in 2010. SGS used a conventional (lead collector) fire assay for PGE assays, and a peroxide fusion for its assay of base metals. Most Platreef samples do not have sulphur contents high enough to warrant a peroxide fusion rather than a lithium borate one, which is the more commonly-used fusion method on samples with less than 10% sulphur. Three sample pulps of head samples were obtained from SGS and submitted to Ultra Trace along with blind-inserted SRMs.

Ivanhoe supplied samples of drill-core to Xstrata Process Support (XPS) in 2011 for use in two phases of scoped metallurgical and mineralogical testwork. The analytical work was performed by ALS Chemex at their Vancouver laboratory. Analysis of Pt, Pd, Au, and Rh was performed using fire assay. The Pt, Pd, and Au were determined by lead collection with cupellation to a low-temperature prill, followed by acid dissolution and ICP finish (ALS Chemex method PGM-ICP23). The Rh was separately assayed using a specialist gold collection fire assay method and ICP-MS finish (ALS Chemex method MS25). The base metals were determined by a sodium peroxide fusion followed by acid dissolution and ICP finish (ALS Chemex method ME–ICP81).

In August 2012, material for a 'Phase 2' sample was selected and sent to Mintek Laboratories. Samples were selected using the representative distribution method recommended by Jorge Oliviera of Xstrata Process support (XPS). Three separate zones were selected for sampling, under the guidance of Dr. Mike Bryson (Mintek), based on the following parameters:

- Proximity to proposed shaft.
- Structurally favourable reef (flat-lying).
- Connectivity and quality of mineralization.

Once the spatial zones were selected, all samples within 2 g/t 2PE+Au composites within the zone were tabulated in Excel. Due to sample length variability, samples were normalized using the length weighted average function. The normalized 2PE+Au and Ni grades were rounded to 1,000 ppm and 0.1% respectively. These rounded values were then categorized into histogram bins.

Various sample distribution statistics were calculated to ensure that a representative sub-sample could be collected. The sample numbers were cross-checked with previous metallurgical sample work to check if ½ NQ core was still available. A new table was created, and samples were added individually from all available drillholes until the appropriate distribution was achieved. Sample mass was estimated under the assumption that 1.0 metre of ¼ NQ core weighs 1.1 kg.

The primary distribution parameters were 2PE+Au grade, Ni grade and rock type. Given the recent results of metallurgical work at XPS, it was decided to keep the grade distribution even, but favour samples with higher Pt/Pd ratios. A consequence of this is that the sample is slightly biased towards material from the top of the composite.

Two sources of error in the selection method are known. Firstly, the rounding and categorizing of data into sample bins and secondly, the assumption of uniform sample weight. These factors are not expected to have a significant influence on the overall result of this work.

Mintek assays for 3PE+Au used a standard fire assay procedure with analysis of dissolved prills by ICP-OES. Nickel and copper assays required that the sample was subjected to an aqua-regia digest followed by ICP/OES determination. Total sulphur is analysed using LECO combustion.

13.2 Previous Metallurgical Testwork

Various metallurgical test work 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 2012 a series of metallurgical test work campaigns were carried out on the Platreef mineralized material as detailed below.

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) and TLZ-SP (Top Loaded Zone – Serpentinite). The test work programme included grind optimization testing, reagent scouting tests and locked cycle tests.

The results of locked cycle tests using the optimized test conditions are presented in Table 13.1 and Table 13.2 below.

Table 13.1 Phase 1 Locked Cycle Test Results for the TLZ-PX Composite Sample

Product	Mass		Grade				% Distribution			
	g	%	2PGE + Au (g/t)	Cu (%)	Ni (%)	S (%)	2PGE + Au (g/t)	Cu (%)	Ni (%)	S (%)
ReCleaner Conc.	49.2	4.9	73.7	3.8	6.6	15.6	77.4	86.7	73.0	57.4
Rougher Tail	957.7	95.1	1.1	0.0	0.1	0.6	22.6	13.3	27.0	42.6
Head (Calculated)	1,006.8	100	4.7	0.2	0.4	1.3	100	100	100	100

Table 13.2 Phase 1 Locked Cycle Test Results for the TLZ-SP Composite Sample

Product	Mass		Grade				% Distribution			
	g	%	2PGE + Au (g/t)	Cu (%)	Ni (%)	S (%)	2PGE + Au (g/t)	Cu (%)	Ni (%)	S (%)
ReCleaner Conc.	31.9	3.2	57.9	5.8	10.4	27.0	68.7	88.0	70.0	78.6
Rougher Tail	974.1	96.8	0.9	0.0	0.1	0.2	31.3	12.0	30.0	21.4
Head (Calculated)	1,006.0	100	2.7	0.2	0.5	1.1	100	100	100	100

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

A mineralogical analysis was performed on the locked cycle test cleaner tailings which indicated that the optimum grind size was to be no more than 70% passing 75µm. PGE's were found to be predominantly occurring as PGE-bismuthides, PGE-bismutho-tellurides, PGE-arsenides and PGE-antimonides with very little association with sulphides. A circuit that allowed for 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 the South African PGE industry.

Phase 2: Xstrata Process Support

In 2011 a metallurgical test work 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 Upper Critical Zone (UCZ). This phase of testing included grind optimization testing followed by baseline rougher flotation tests.

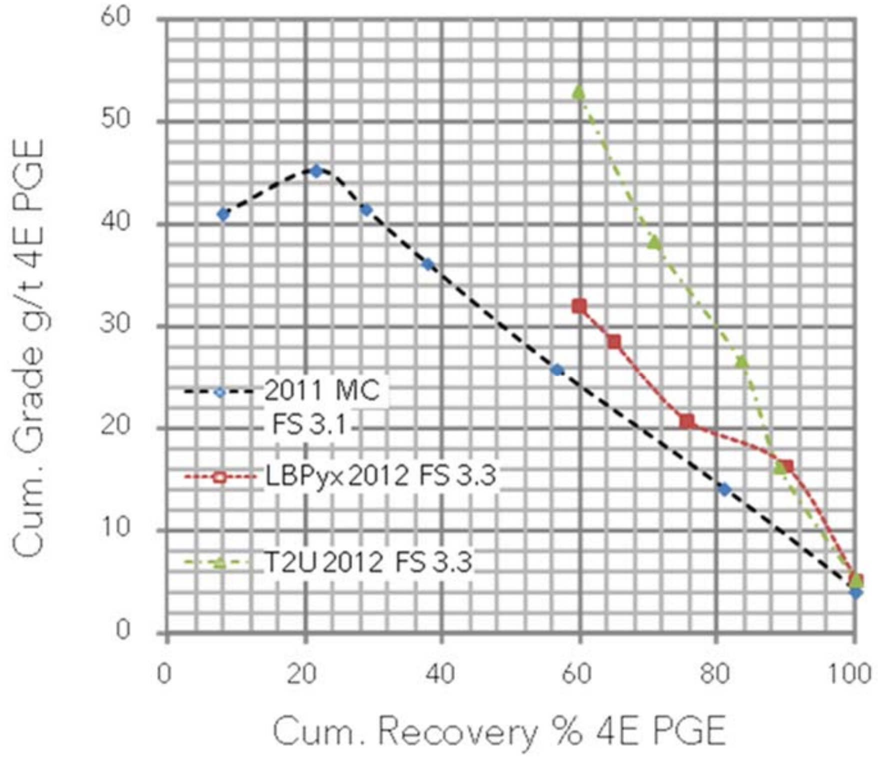
In early 2012, the Platreef stratigraphic interpretation was re-assessed and three geometallurgical units were identified for the newly revised nomenclature. A second phase of testing was initiated at XPS, to determine the metallurgical response based on the updated stratigraphic interpretation for the deposit. A summary of the updated classification as compared to the classification used prior to 2012 is presented in Figure 7.10.

The flowsheet development testing at XPS focused on mineralogy, obtaining baseline flotation tests and producing grade recovery relationships. An optimized two stage milling and two stage flotation flowsheet, commonly referred to as a MF2 circuit in South African processing terms was developed based on results from grind optimization testing and reagent dosage testing.

The optimized grade recovery curves derived at XPS during the Phase 2 testing are presented in Figure 13.1 below. It was evident that the performance of the newly classified geometallurgical unit T2U gave a better grade and recovery profile than the previously classified LBP (Lower B-Pyroxenite) unit and the 2011 master composite made up of the five geometallurgical units from the UCZ.

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%. For both the LBP 2012 and T2U 2012 test the PGE recovery in the rougher flotation was approximately 90% at a rougher concentrate grade of 16 g/t.

Figure 13.1 XPS Phase 2 Optimized PGE Grade-Recovery Curves



13.3 Current Metallurgical Test Work

This section summarizes the metallurgical test work carried out up to October 2013 used to develop the scoping study flowsheet. Development testwork is ongoing and variability and mini pilot plant work is planned for mid 2014.

Following on from the identification of the T1, T2U, and T2L geometallurgical units during the 2012 geological re-assement and subsequent to the Phase 2 XPS testing, further flowsheet development testing and reagent scouting tests were conducted at SGS Lakefield (testwork Phase 3). During this phase of testing a reagent suite that included oxalic acid and thiourea with conditioning in the mill prior to flotation indicated PGE recovery and concentrate grade improvements.

Based on these findings, further flowsheet development test work was initiated at Mintek (Phase 4 and Phase 6) and SGS Lakefield (Phase 5) between March 2012 and October 2013.

Test work was performed on the Platreef resource from March 2012 to October 2013 at Mintek and SGS Lakefield as follows:

- Phase 3: Test work on Master Composite II sample conducted at SGS Lakefield from July 2012 – December 2012 under the management of Ivanhoe and AMEC.
- Phase 4: Test work on geometallurgical units T1, T2U, and T2L conducted at Mintek from March 2012 – January 2013 under the management of Ivanhoe and AMEC.
- Phase 5: Test work on geometallurgical units T1, T2U, and T2L conducted at SGS Lakefield from March 2013 – October 2013 under the management of Ivanhoe and DRA.
- Phase 6: Test work on geometallurgical units T1, T2U, and T2L conducted at Mintek from March 2013 – October 2013 under the management of Ivanhoe and DRA.

The Phase 3 samples were selected by Platreef based on the geometallurgical classification used prior to the re-assement in early 2012 when the Platreef stratigraphic interpretation was updated. The geometallurgical units used to make up the Phase II Master Composite sample contained material that included the geometallurgical units T1m and T2L as per the updated 2012 classification but was noted as having some form of disproportion based on the updated classification based as derived in early 2012 and presented in Figure 13.1.

Phase 4–Phase 6 testing was conducted on drill core samples representing the newly classified geometallurgical namely, T1, T2 upper (T2U), and T2 Lower (T2L).

13.4 Mineralogy

A mineralogical analysis was performed at Mintek in Phase 4 on samples of each of the three geometallurgical units namely, T1, T2U, and T2L.

A PGE and BMS search was conducted on samples of each geometallurgical unit which had been milled to 80% passing 75 µm. The results of this mineralogical investigation are presented below.

13.4.1 PGE Occurrence

- PGE-bismutho-tellurides are the major PGM species present in all of the samples, with lesser amounts of PGE-arsenides, PGE-sulphides and PGE-alloys.
- The T1 sample was comprised predominantly of PGE-bismutho-tellurides with an approximate 50% by volume, with PGE-sulphides accounting for 26% by volume and PGE-alloys comprising 11% by volume with the PGE-arsenides making up less than 10% by volume.
- The T2U sample was comprised predominantly of PGE-bismutho-tellurides with an approximate 34% by volume, with PGE-alloys accounting for 30% by volume and PGE-sulphides comprising 19% by volume with the PGE-arsenides making up 15% by volume.
- The T2L sample was comprised predominantly of PGE-arsenides with an approximate 35% by volume, with PGE-bismutho-tellurides accounting for 32% by volume and PGE-alloys accounting for 22% by volume with the PGE-sulphides making up 10% by volume.

The PGE mode of occurrence is summarized in Table 13.3.

Table 13.3 Summary of the Platreef PGE Mode of Occurrence

PGM Mode of Occurrence Class	Description	T1 PGM (%v/v)	T2U PGM (%v/v)	T2L PGM (%v/v)
L	Liberated PGM	61.3	58.6	62.3
SL	PGM associated with BMS only (i.e. a binary PGM-BMS particle)	0.5	2.1	19.5
AG	PGM attached to silicate or oxide gangue (i.e. PGM exposed at particle perimeter)	29.6	28.1	13.8
SAG	PGM associated with BMS attached to silicate or oxide gangue (i.e. BMS exposed at particle perimeter)	4.2	4.8	0.5
SG	PGM associated with BMS locked within silicate or oxide gangue (i.e. no exposure of BMS or PGM at particle perimeter)	0.1	0.3	0.0
G	PGM locked within silicate or oxide gangue (i.e. no exposure of BMS or PGM at particle perimeter)	4.3	6.1	3.9
		100	**100**	**100**

The PGE occurrence data indicated a high fraction of liberated and partially liberated PGE for all geometallurgical units with 3.9% - 6.4% indicated as locked.

13.4.2 Base Metal Sulphide (BMS) Analysis

Relative portions of Base Metal Sulphide (BMS) minerals present in each of the geometallurgical units were measured by QEMSCAN SMS analysis. The results of this BMS analysis are presented in Table 13.4.

Table 13.4 Summary of the BMS Occurrence in Each Composite

Mineral	T1 Composite (%m/m)	T2U Composite (%m/m)	T2L Composite (%m/m)
Pentlandite	30.1	35.5	38.6
Chalcopyrite	19.2	19.9	20.6
Pyrrhotite	49.6	44.0	39.6
Pyrite	0.8	0.4	1.0
Other BMS	0.3	0.2	0.3
Total	**100**	**100**	**100**

13.4.3 Conclusions

- PGE-bismutho-tellurides are the major PGE species present in all of the samples, with lesser amounts of PGE-arsenides, PGE-sulphides and PGE-alloys.
- Pyrrhotite, pentlandite and chalcopyrite were found to be the major BMS minerals present in all of the samples.
- At a target grind of 80% passing 75µm, PGE liberation was found to be in excess of 90% for all samples.
- At a target grind of 80% passing 75µm, BMS liberation was found to be in excess of 80% for all samples.

13.5 Comminution Test Work

Comminution test work 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.

A summary of the comminution test results are presented in Table 13.5.

Table 13.5 Summary of the Comminution Test Results

Sample Name	Relative Density	JK Parameters			Work Indices (kWh/t)			Abrasion Index (g)
		A x b	t_{a1}	DWI	RWI	BWI 100 mesh	BWI 150 mesh	
T1M	3.19	43.6	0.35	7.3	14.5	19.5	20.0	–
T2U	3.18	39.5	0.33	8.0	14.7	18.4	20.1	–
T2L	3.08	44.5	0.37	7.0	16.1	21.3	22.3	–
Comp FPX	3.19	35.0	0.29	9.1	17.3	19.4	20.3	0.362
Comp N	3.13	44.2	0.36	7.1	–	–	–	0.344
Comp OLOPX-OPX	3.17	32.8	0.27	9.6	15.9	20.4	–	0.303
Comp PAPX	3.00	29.9	0.26	10.1	20.7	22.3	22.3	0.344

The comminution testing indicated that the footwall composite samples (FPX, N, OLOPX-OPX and PAPX) are abrasive and that due to the high competency of both the mineralized material and the footwall composites with regard to SAG milling, a crusher and ball milling circuit would be better suited to the Platreef process. 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 ball work index test (< 3.35 mm).

13.6 Flotation Test Work Summary

13.6.1 Testing on Master Composite II (MC II)

In Phase 3, test work was conducted at SGS Lakefield in 2012 to select a suitable flowsheet and flotation reagent suite for the Platreef process. This testing was conducted on the Master Composite II (MC II) sample.

Table 13.6 Head Assay for Phase 3 Master Composite II

	Head Grade							
Sample ID	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)	S (%)
Master Composite II	1.35	1.95	–	0.23	3.53	0.18	0.38	0.93

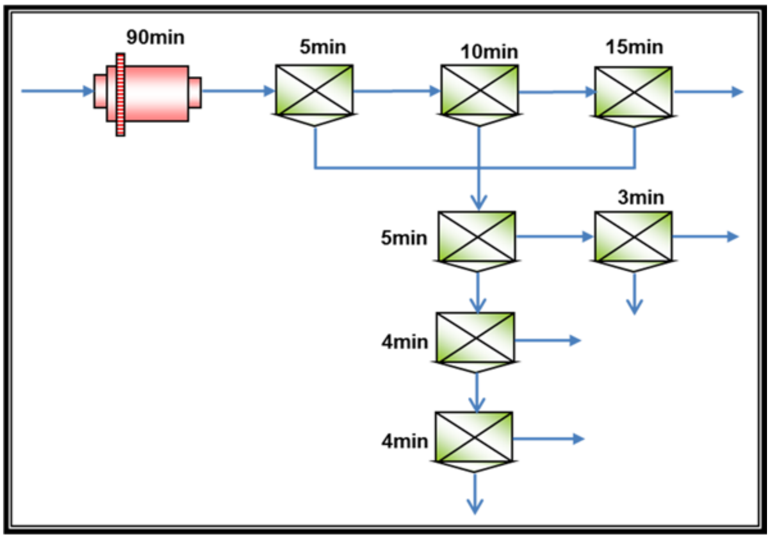
A series of tests were conducted on the MC II sample to evaluate the effect of various reagent combinations as presented in Table 13.7.

Table 13.7 Flotation Conditions Used for Reagent Suite Testing on Master Composite II

Test #	Purpose:	Primary Grind								Flotation Reagents (g/t)						
		Time (min)	K80 (µm)	CuSO4 (g/t)	Oxalic Acid (g/t)	Thiourea (g/t)	DETA (g/t)	pH	Ep (mV)	SIPX	3477	HP700	Oxalic Acid	DETA	Thiourea	CMC
F1	Baseline	90	76	250	–	–	–	8.6	+80	92.5	92.5	75.0	0	0	0	15
F2	Oxalic acid + DETA instead of CuSO4	90	83	–	200	–	50	8.7	+120	92.5	92.5	77.5	80	20	0	15
F3	Oxalic acid + Thiourea instead of CuSO4	90	79	–	200	50	–	8.8	+140	92.5	92.5	77.5	80	0	20	85
F4	CMC in 1st cleaner	90	83	250	–	–	–	8.6	+150	92.5	92.5	75.0	0	0	0	103
F5	Mag sep before flotation	85	73	250	–	–	–	8.6	+160	92.5	92.5	75.0	0	0	0	45
F6	Oxalic acid alone	90	83	–	200	–	–	8.8	+140	92.5	92.5	77.5	130	0	0	175

The flowsheet used for tests F1–F6 is summarized in Figure 13.2.

Figure 13.2 Flowsheet Used for Reagent Suite Testing on Master Composite II



The results of open circuit batch cleaner tests F1–F6 on MC II are presented in Table 13.8.

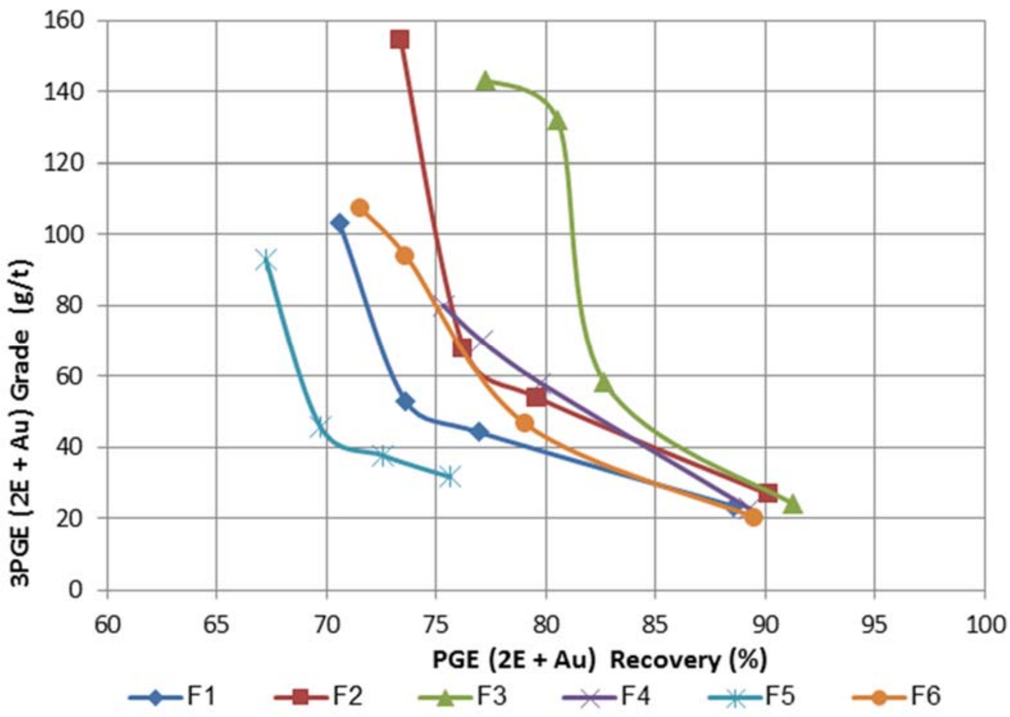
Table 13.8 Summary of Batch Cleaner Test Results for Reagent Suite Selection Testing on Master Composite II

			Grade				Recovery			
Test No.	Products	Mass (%)	**2E+Au (g/t)**	Cu (%)	Ni (%)	S (%)	**2E+Au (%)**	Cu (%)	Ni (%)	S (%)
F1	ReReCl.C	2.79	**103.26**	4.88	7.99	22.40	**70.58**	73.33	61.29	67.71
	Rou. Conc	15.43	**23.41**	1.08	1.90	4.76	**88.57**	89.87	80.87	79.61
F2	ReReCl.C	2.17	**154.29**	6.62	10.00	20.40	**73.38**	79.59	59.03	49.16
	Rou. Conc	15.40	**26.72**	1.08	1.89	4.97	**90.19**	92.28	79.08	84.98
F3	ReReCl.C	2.30	**143.17**	5.86	9.92	16.90	**77.26**	77.81	62.06	61.59
	Rou. Conc	16.14	**24.11**	1.00	1.81	3.60	**91.25**	92.74	79.49	92.03
F4	ReReCl.C	3.97	**79.99**	3.52	5.83	16.80	**75.32**	73.61	62.26	72.92
	Rou. Conc	16.61	**22.65**	1.01	1.77	5.00	**89.31**	88.13	79.34	90.87
F5	ReReCl.C	3.03	**92.92**	4.18	6.73	18.80	**67.22**	68.03	56.29	64.97
	Rou. Conc	10.01	**31.67**	1.42	2.40	6.59	**75.63**	76.21	66.30	75.21
F6	ReReCl.C	2.72	**107.19**	5.16	7.98	20.60	**71.55**	77.48	58.89	64.41
	Rou. Conc	17.89	**20.37**	0.92	1.65	4.31	**89.51**	90.93	80.16	88.66

The grade recovery curves for tests F1–F6 are presented in Figure 13.3 to Figure 13.5. Test F3 was found to give the best grade and recovery profile for both PGEs and nickel with a PGE (2PE+Au) recovery of 77.3% at a grade of 143 g/t and a nickel recovery of 62.2% at a grade of 9.9%. Test F3 and F2 were found to give similar performance in terms of copper grade and recovery with an average recovery of 78.7% at a grade of 6.2%.

Based on the results of the open circuit batch cleaner reagent testing (F1–F6), the reagent suite and conditions for test F3 was selected as the optimum test result in terms of PGE (2PE+Au) grade and recovery.

Figure 13.3 PGE Grade-Recovery Curves for Reagent Suite Testing on Master Composite II



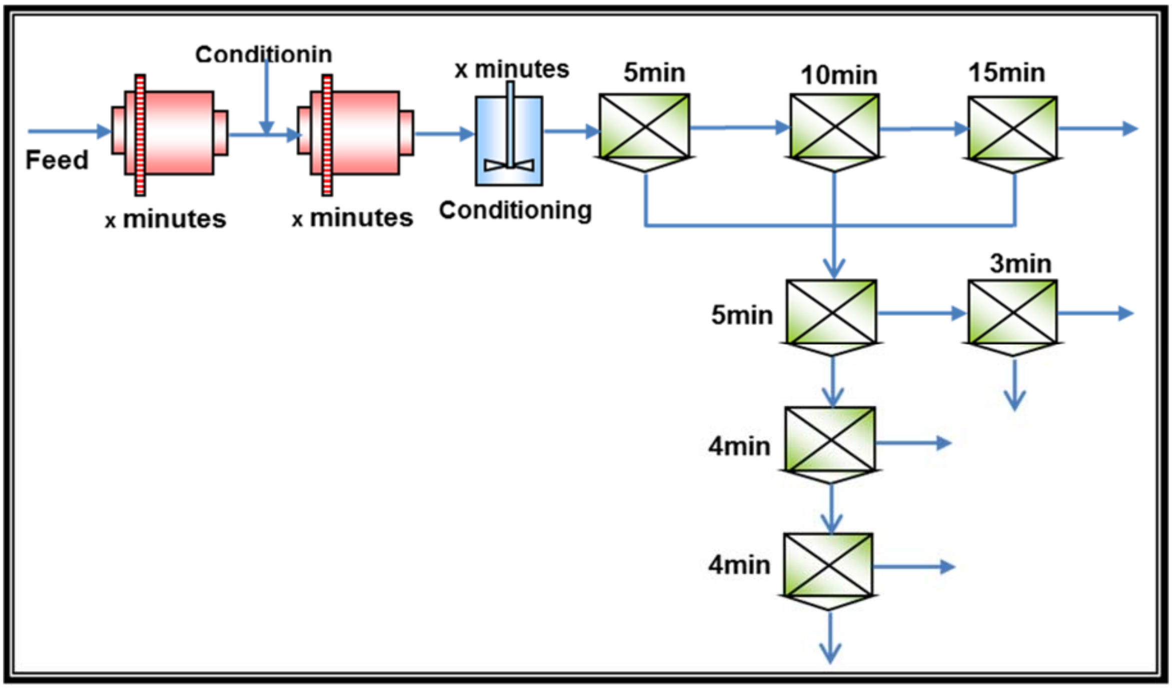
Based on the optimal reagent suite as selected for test F3, a further series of tests were conducted on the MC II sample to evaluate the effect of oxalic acid, thiourea, and conditioning time in the mill as presented in Table 13.9.

Table 13.9 Flotation Conditions Used for Conditioning Tests on Master Composite II

Test #	Purpose:	Primary Grind							Flotation Reagents (g/t)					
		Time (min)	K80 (µm)	CuSO4 (g/t)	Oxalic Acid (g/t)	Thiourea (g/t)	pH	Ep (mV)	SIPX	3477	HP700	Oxalic Acid	Thiourea	CMC
F13	As F3:Oxalic & thiourea from primary mill to conditioner 30 min	90	90	–	–	–	8.4	+40	92.5	92.5	80.0	280	70	85
F14	As F3:Oxalic & thiourea from primary mill to conditioner 10 min	90	84	–	–	–	9.4	+40	92.5	92.5	80.0	280	70	85
F15	As F3:Oxalic & Thiourea to last 15 minutes of grinding	90	84	–	200	50	9.2	+20	92.5	92.5	77.5	80	20	85

The flowsheet used for tests F13–F15 as presented in Table 13.9 is summarized in Figure 13.4.

Figure 13.4 Flowsheet Used for Conditioning Tests on Master Composite II



The results of open circuit batch cleaner tests F13–F15 relative to test F3 are presented in Table 13.10.

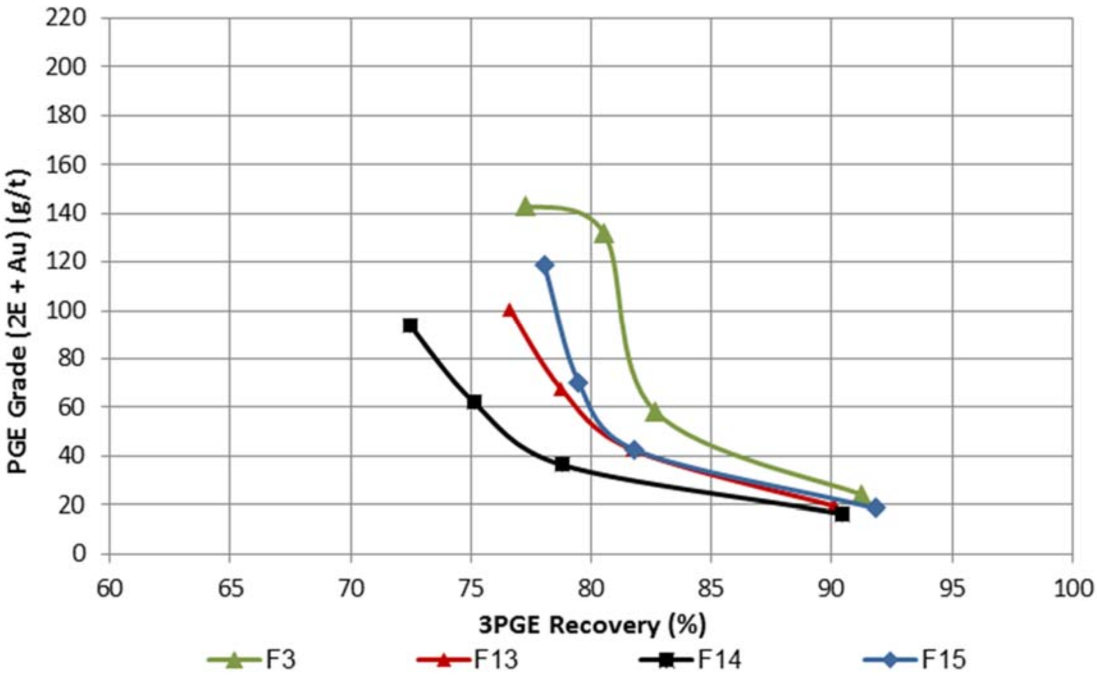
Table 13.10 Summary of Batch Cleaner Test Results for Flowsheet Optimization Testing on Master Composite II

			Grade				Recovery			
Test No.	Products	Mass (%)	2E+Au (g/t)	Cu (%)	Ni (%)	S (%)	2E+Au (%)	Cu (%)	Ni (%)	S (%)
F3	ReReCl.C	2.30	**143.17**	5.86	9.92	16.90	**77.26**	77.81	62.06	61.59
	Rou. Conc	16.14	**24.11**	1.00	1.81	3.60	**91.25**	92.74	79.49	92.03
F13	ReReCl.C	3.10	**100.32**	4.75	8.15	21.40	**76.63**	78.05	66.09	72.25
	Rou. Conc	18.80	**19.48**	0.91	1.67	4.42	**90.11**	90.33	82.18	90.29
F14	ReReCl.C	2.84	**93.77**	4.98	8.28	21.60	**72.47**	77.75	61.47	67.33
	Rou. Conc	20.21	**16.41**	0.82	1.53	4.03	**90.43**	91.65	81.09	89.47
F15	ReReCl.C	3.05	**118.65**	4.78	7.66	19.50	**78.05**	80.64	61.89	67.99
	Rou. Conc	22.88	**18.58**	0.73	1.35	3.38	**91.84**	92.95	82.00	88.53

For tests F13 and F14 oxalic acid and thiourea were added post milling with a conditioning time pre-flotation of 30 and 10 minutes respectively. In test F15 the above reagents were added into the mill but only during the last 15 minutes of grinding.

The PGE grade recovery curves for tests F13–F15 as compared to test F3 are presented in Figure 13.5.

Figure 13.5 PGE Grade-Recovery Curves for Reagent Conditioning Tests on Master Composite II



Based on all the open circuit batch cleaner tests on the MC II sample, the following was concluded:

- A reagent suite that includes flotation pre-conditioning with oxalic acid and thiourea gave the best results in terms of 2PE+Au grade and recovery.
- The overall PGE grade and recovery was found to be sensitive to oxalic acid and thiourea conditioning time in the mill. Tests in which oxalic acid and thiourea were conditioned in the mill for 90 minutes gave the best results with respect to PGE grade and recovery to final concentrate.

The SGS testing on the MC II sample indicated that a reagent suite containing oxalic acid and thiourea was suited to the Platreef mineralized material and formed the basis for further flowsheet development work.

Further flowsheet development work was initiated at SGS Lakefield and Mintek in Phase 4 and Phase 6 in order to address the following:

- Phase 3 testing was conducted on Master Composite II which was not made up of material fully representative of the newly defined geometallurgical units T1, T2U, and T2L as per the stratigraphic reassessment in mid-2012.
- The results obtained were based on oxalic acid and thiourea conditioning within the mill for 90 minutes of milling time. Typical in-mill residence times for full scale milling operations are in the range 4–11 minutes and as such it would not be possible to replicate the 90 minute laboratory in-mill conditioning time on commercial scale. Tests in which the reagent was not added to the mill and conditioned prior to flotation indicated lower grades for PGE, copper, and nickel in final concentrate thus further testing was required.

13.7 Flotation Test Work on Geometallurgical Units T1, T2U and T2L

As part of the Phase 4 and Phase 6 metallurgical testwork campaigns, testing was completed at Mintek in Johannesburg and SGS Lakefield on various composite samples from the Platreef deposit representing the geometallurgical units T1, T2U, and T2L as per the updated mid-2012 assement. Initial indications from mining were that an average mine ratio of 15% T1, 42.5% T2U, and 42.5% T2L would be processed. In addition to this mineralized footwall would also form part of the mine blend, the percentage of this was unknown.

13.7.1 SGS Lakefield–Phase 5

In Phase 5, 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 head assays for the samples used during the Phase 5 test work campaign at SGS are summarized in Table 13.11.

Table 13.11 Phase 5 SGS Sample Head Assays

	Head Grade							
Sample ID	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)	S (%)
T1	1.46	1.11	–	0.31	2.88	0.14	0.26	0.75
T2U	2.00	1.71	–	0.37	4.08	0.15	0.28	0.73
T2L	1.64	1.63	–	0.22	3.49	0.16	0.33	0.79
15% T1 : 42.5% T2U : 42.5% T2L	1.64	1.63	–	0.22	3.49	0.16	0.30	0.74
80% Min Zone:20% Footwall	1.53	1.52	–	0.20	3.25	0.15	0.30	0.73

13.7.2 Mintek – Phase 4 and Phase 6

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 composites of indicated mine blends.

The head assays off the samples at Mintek used during the Phase 4 (Mintek Phase II) and Phase 6 (Mintek Phase III) test work campaigns are summarized in Table 13.12 and Table 13.13.

Table 13.12 Phase 4 Mintek Sample Head Assays

	Head Grade							
Sample ID	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)	S (%)
T1	2.52	2.26	0.12	0.35	5.25	0.20	0.40	0.98
T2U	2.02	2.03	0.13	0.32	4.50	0.25	0.46	1.02
T2L	2.08	2.47	0.16	0.29	5.00	0.25	0.51	1.05
50%T2U : 50%T2L	2.05	2.13	0.13	0.28	4.59	0.24	0.48	1.08
15%T1 : 42.5%T2U : 42.5%T2L	2.10	2.10	0.12	0.29	4.61	0.23	0.46	0.97

Table 13.13 Phase 6 Mintek Sample Head Assays

	Head Grade							
Sample ID	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	3PE+Au (g/t)	Cu (%)	Ni (%)	S (%)
T2U	1.99	1.98	0.09	0.33	4.39	0.21	0.40	1.12
T2L	1.99	2.00	0.13	0.31	4.43	0.23	0.47	1.30
15%T1 : 42.5%T2U : 42.5%T2L	2.10	1.99	0.10	0.31	4.50	0.22	0.43	1.20

13.7.3 Rougher Kinetic Testing to Determine Optimum Grind

A rougher rate test was performed on each of the geometallurgical units at various grinds at Mintek during Phase 4 and at SGS during Phase 5 in order to determine the effect of grind on flotation kinetics, overall rougher recovery and concentrate grade as follows:

- Mintek Phase 4: Rougher rate tests at 40%, 60% and 80% passing 75µm.
- SGS Phase 5: Rougher rate tests at 80% passing 150, 106, 75 and 53 µm.

Summary Results for T1

Table 13.14 Summary of Rougher Kinetic Testing to Determine Optimum Grind For T1

T1 Composite				Grade				Recovery			
Laboratory	Products	Time (min)	Mass (%)	3PE+Au (g/t)	Cu (%)	Ni (%)	S (%)	3PE+Au (%)	Cu (%)	Ni (%)	S (%)
Mintek Phase 4	40% -75µm	30	14.82	32.97	1.19	2.20	5.83	92.13	83.78	76.16	92.69
	60% -75µm	30	20.55	24.73	0.85	1.70	4.51	96.76	84.58	82.88	97.49
	80% -75µm	30	22.29	22.65	0.79	1.47	4.27	96.29	84.98	83.20	97.61
SGS Phase 5	80% -150µm	30	29.56	7.72	0.45	0.77	2.43	90.75	91.36	84.57	92.74
	80% -106µm	30	30.59	8.23	0.45	0.75	2.25	92.83	92.03	83.93	94.30
	80% -75µm	30	36.43	7.85	0.38	0.65	2.00	94.74	91.98	85.54	95.03
	80% -53µm	30	33.01	9.41	0.42	0.72	2.10	93.92	92.42	84.93	93.66

The rougher kinetic testing at Mintek indicated that for T1 a grind of 60% passing 75 µm gave similar results in terms of PGE recovery and rougher kinetics when compared to a grind of 80% passing 75 µm. Nickel recovery improved by 2% when the grind was increased from 60% to 80% passing 75 µm. The coarser grind of 40% passing 75 µm gave the worst overall result with the lowest PGE, copper and nickel recoveries. The testing at SGS compared to the Mintek findings with an indicated optimum grind of 80% passing 75 µm for T1 based on improved PGE recovery as grind was increased from 80% passing 150 µm to 80% passing 75 µm, with no evident recovery benefit at a finer grind of 80% passing 53 µm.

Summary Results for T2U

Table 13.15 Summary of Rougher Kinetic Testing to Determine Optimum Grind For T2U

T2U Composite				Grade				Recovery			
Laboratory	Products	Time (min)	Mass (%)	3PE+Au (g/t)	Cu (%)	Ni (%)	S (%)	3PE+Au (%)	Cu (%)	Ni (%)	S (%)
Mintek Phase 4	40% -75µm	18.01	22.49	1.20	2.08	4.67	90.47	84.03	82.01	85.07	18.01
	60% -75µm	22.57	18.14	0.95	1.72	3.83	94.84	84.70	84.35	92.55	22.57
	80% -75µm	24.44	17.90	0.86	1.56	3.82	96.92	84.70	84.83	95.37	24.44
SGS Phase 5	80% -150µm	28.02	13.05	0.49	0.83	2.24	92.03	90.43	84.16	87.05	28.02
	80% -106µm	30.70	12.78	0.45	0.77	2.10	94.81	92.63	87.05	90.30	30.70
	80% -75µm	35.03	11.06	0.43	0.77	2.17	95.36	94.34	88.66	93.61	35.03
	80% -53µm	33.81	10.26	0.43	0.73	1.98	94.24	96.46	85.40	91.01	33.81

The rougher kinetic testing at both Mintek and SGS indicated that for T2U a grind of 80% passing 75 µm gave the best PGE and Nickel recoveries. At Mintek the copper recovery was found to be insensitive to grind with similar copper recoveries achieved for grinds of 40% – 80% passing 75 µm, whereas the SGS testing indicated that copper recovery improved as the fineness of grind was increased from 80% passing 150 µm to 80% passing 53 µm.

Summary Results for T2L

Table 13.16 Summary of Rougher Kinetic Testing to Determine Optimum Grind For T2L

T2L Composite				Grade				Recovery			
Laboratory	Products	Time (min)	Mass (%)	3PE+Au (g/t)	Cu (%)	Ni (%)	S (%)	3PE+Au (%)	Cu (%)	Ni (%)	S (%)
Mintek Phase 4	40% -75µm	18.01	22.49	1.20	2.08	4.67	90.47	84.03	82.01	85.07	18.01
	60% -75µm	22.57	18.14	0.95	1.72	3.83	94.84	84.70	84.35	92.55	22.57
	80% -75µm	24.44	17.90	0.86	1.56	3.82	96.92	84.70	84.83	95.37	24.44
SGS Phase 5	80% -150µm	28.02	13.05	0.49	0.83	2.24	92.03	90.43	84.16	87.05	28.02
	80% -106µm	30.70	12.78	0.45	0.77	2.10	94.81	92.63	87.05	90.30	30.70
	80% -75µm	35.03	11.06	0.43	0.77	2.17	95.36	94.34	88.66	93.61	35.03
	80% -53µm	33.81	10.26	0.43	0.73	1.98	94.24	96.46	85.40	91.01	33.81

The rougher kinetic testing at both Mintek and SGS indicated that for T2L a grind of 80% passing 75 µm gave the best PGE, and nickel recovery. The copper recovery was found to improve as the fineness of grind was increased with the best copper recovery achieved at a target grind of 80% passing 53 µm at SGS.

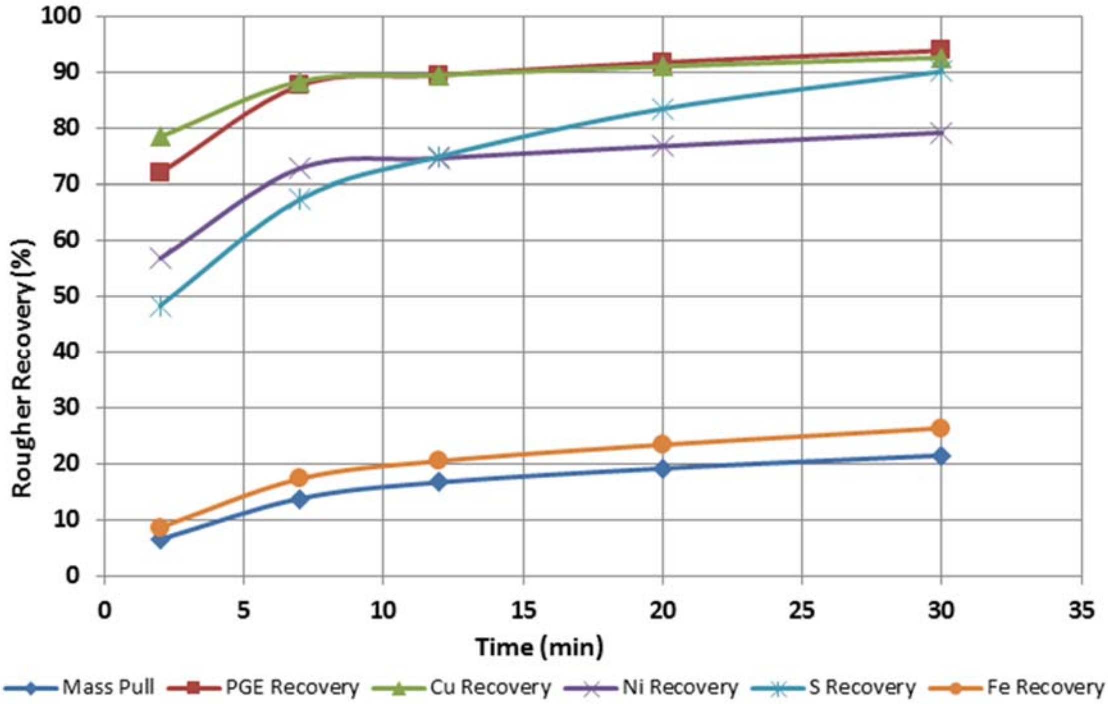
Based on both the SGS and Mintek rougher kinetic testing at various grinds, a grind of 80% passing 75 µm was selected as optimal grind for all further flotation test work.

Based on the selected grind of 80% passing 75 µm two rougher kinetic tests were conducted on a mineralized zone composite sample comprising 15% T1, 42.5% T2U, and 42.5% T2L. The results of the two rougher kinetic tests as presented in Table 13.17 were used to model the flotation response and the rate curves are presented in Figure 13.6.

Table 13.17 SGS Lakefield Summary Results of Rougher Kinetic Tests on the Mineralised Zone Blend Composite at 80% Passing 75 µm

Composite (15% T1, 42.5% T2U, 42.5% T2L)				Grade				Recovery			
Laboratory	Products	Time (min)	Mass (%)	2E+Au (g/t)	Cu (%)	Ni (%)	S (%)	2E+Au (%)	Cu (%)	Ni (%)	S (%)
SGS Phase 5	80% -75µm	30	19.70	18.72	0.74	1.23	3.50	92.64	92.14	75.84	88.20
	80% -75µm	30	21.38	17.19	0.69	1.21	–	92.76	91.24	79.85	–

Figure 13.6 Rougher Kinetic Curves for the Blend Composite At 80% Passing 75 µm



The results of rougher kinetic testing on the mineralised zone blend composite at 80% passing 75 µm indicated the following:

- A rougher mass pull of 20% was achieved at a rougher flotation time of 30 minutes.
- PGE, copper and nickel demonstrated fast flotation kinetics with recoveries of 87.6%, 88.4%, and 72.8% respectively within the first seven minutes of rougher flotation.
- Sulphur flotation kinetics was slower and sulphur recovery increased from 67.3% after seven minutes to 90.2% at thirty minutes.
- Iron recovery increased from 17.3% after seven minutes to 26.4% at thirty minutes.
- PGE, copper and nickel demonstrated the presence of a slow floating fraction with an additional 2.11%, 1.6%, and 2.4% of recovery in last ten minutes of flotation respectively.

13.7.4 Evaluation of Reagent Conditioning Parameters for a 3 Stage Cleaning Circuit

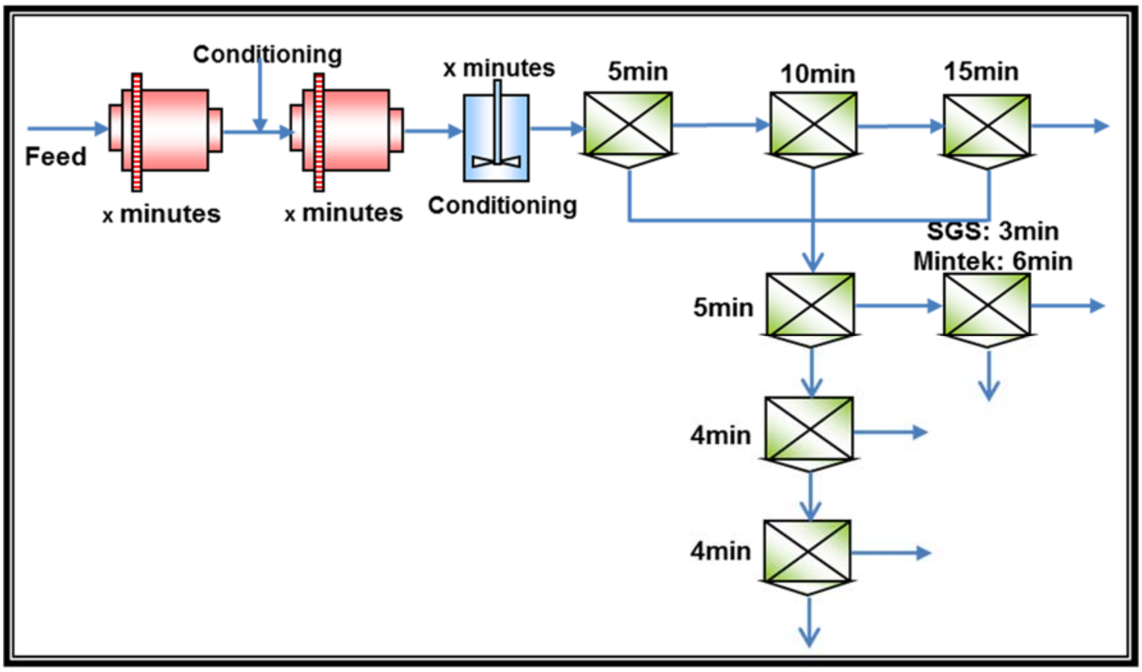
A series of open circuit cleaner batch flotation tests were performed at SGS and Mintek in Phase 5 and 6 on a blend composite containing 15% T1, 42.4% T2U, and 42.5% T2L at a target grind of 80% passing 75 µm, in order to establish the optimal oxalic acid and thiourea addition requirements and conditioning parameters.

In order to further understand the effect of in-mill conditioning time and to compare the difference in performance when oxalic acid and thiourea were added to the mill for the entire laboratory milling duration (> 90 minutes) as compared to tests where oxalic acid and thiourea were added to the mill for a shorter duration of the total milling time, further tests were conducted. These included the addition of post mill conditioning in an agitated flotation cell. An initial series of scouting tests were performed with an in-mill conditioning time of 15 minutes. These tests indicated that the overall PGE recovery and grade were sensitive to conditioning time in the mill. Further testing was initiated in which the in-mill conditioning time was further reduced to 5 minutes in an attempt to simulate commercial milling conditions, in which typical residence times are in the range 4–11 minutes.

Tests with 5 Minutes of in-Mill Conditioning Time

The 5 minute in-mill conditioning tests were conducted using a flowsheet and flotation residence times as presented in Figure 13.7.

Figure 13.7 Summary of the Flowsheet Used for the 5 Minute In-Mill Conditioning Tests



A summary of the test conditions used is presented in Table 13.18.

Table 13.18 Summary of the Test Conditions Used for the 5 Minute In-Mill Conditioning Tests

Test ID	Description	Grinding Media	In Mill Conditioning			Post Mill Conditioning			Rougher Concentrate Conditioning			Flotation Reagents [g/t] (1% solution)					
		Type	Time (min)	Oxalic (g/t)	Thiourea (g/t)	Type	Oxalic (g/t)	Thiourea (g/t)	Time (min)	Oxalic (g/t)	Thiourea (g/t)	SIPX	Aero 3477	Sen Froth 522	Oxalic (g/t)	Thiourea (g/t)	Sendep 30E
Baseline	Mintek Phase 6	Stainless Steel	124	200	50	–	–	–	2	80	20	93	93	78	–	–	85
Test 1		Stainless Steel	5	200	50	30 min 30% Solids	–	–	2	80	20	93	93	78	–	–	85
Test 2		Stainless Steel	5	200	50	30 min 30% Solids	–	–	10	80	20	93	93	78	40	10	85
Test 3		Carbon Steel	5	200	50	30 min 30% Solids	–	–	10	80	20	93	93	78	40	10	85
Test 4		Stainless Steel	5	200	50	60 min 30% Solids	–	–	2	80	20	93	93	78	40	10	85
Test 5		Stainless Steel	5	200	50	30 min 30% Solids	200	50	2	80	20	93	93	78	40	10	85
Test 6		Stainless Steel	5	200	50	30 min 60% Solids	–	–	2	80	20	93	93	78	40	10	85
Test 7		Stainless Steel	5	100	25	30 min 60% Solids	100	25	2	80	20	93	93	78	40	10	85
Baseline BC-F1	SGS Phase 5	High Chrome 440C	80	200	50	–	–	–	2	80	20	93	93	78	–	–	85
BC-F2		High Chrome 440C	5	200	50	30min 60% Solids	200	50	2	120	30	93	93	78	–	–	85
BC-F7		High Chrome 440C	5	200	50	60min 30% Solids	–	–	2	80	20	93	93	85	40	10	135
BC-F9		High Chrome 440C	5	200	50	60min 30% Solids	–	–	2	80	20	93	93	85	80	20	135
BC-F10		High Chrome 440C	5	200	50	60min 30% Solids	–	–	10	80	20	93	93	85	40	10	75

The results of this testing are presented in Table 13.19.

Table 13.19 Summary of the Results for the 5 Minute In-Mill Conditioning Tests

				Grade				Recovery			
Laboratory	**Test No.**	**Products**	**Mass (%)**	**3PE+Au* (g/t)**	**Cu (%)**	**Ni (%)**	**S (%)**	**3PE+Au* (%)**	**Cu (%)**	**Ni (%)**	**S (%)**
Mintek Phase 6	Baseline	ReReCl.C	2.89	**121.37**	6.28	10.80	24.10	**79.97**	83.13	67.48	69.16
		Rou. Conc	21.58	**19.18**	0.96	1.74	4.45	**94.39**	94.72	81.35	95.33
	1	ReReCl.C	3.02	**107.58**	6.05	10.00	26.90	**73.60**	81.79	66.64	68.58
		Rou. Conc	19.74	**20.13**	1.06	1.89	5.12	**89.91**	93.18	82.32	85.12
	2	ReReCl.C	2.97	**114.68**	6.17	9.08	24.80	**75.55**	81.24	61.34	67.44
		Rou. Conc	18.79	**21.71**	1.14	1.91	5.21	**90.52**	94.67	81.53	89.59
	3	ReReCl.C	2.75	**109.71**	5.98	9.78	25.80	**67.69**	76.04	65.72	64.78
		Rou. Conc	20.08	**20.32**	1.01	1.79	4.97	**91.65**	93.75	88.08	91.24
	4	ReReCl.C	3.18	**105.92**	5.55	9.60	25.82	**77.44**	78.67	65.56	74.24
		Rou. Conc	22.69	**17.78**	0.93	1.71	4.47	**92.67**	94.15	83.41	91.62
	5	ReReCl.C	3.06	**104.28**	5.71	9.74	26.04	**74.81**	78.26	64.30	71.58
		Rou. Conc	24.22	**16.43**	0.86	1.60	4.13	**93.18**	93.69	83.67	89.80
	6	ReReCl.C	2.93	**109.87**	6.22	9.78	25.10	**76.65**	82.10	63.81	67.80
		Rou. Conc	20.02	**19.38**	1.04	1.84	4.70	**92.37**	93.88	82.01	86.73
	7	ReReCl.C	2.71	**115.71**	6.50	10.40	26.86	**72.75**	79.38	60.16	65.07
		Rou. Conc	18.54	**21.21**	1.11	2.09	5.43	**91.14**	92.89	82.63	89.82
SGS Phase 5	Baseline BC-F1	ReReCl.C	1.96	**122.27**	6.27	9.98	23.80	**72.73**	80.67	62.46	61.78
		Rou. Conc	15.71	**18.45**	0.87	1.52	3.80	**88.20**	90.02	76.66	79.29
	BC-F2	ReReCl.C	1.94	**135.58**	6.06	9.40	26.80	**72.41**	75.37	58.67	68.16
		Rou. Conc	15.06	**21.64**	0.91	1.56	4.43	**89.50**	88.05	75.33	87.22
	BC-F10	ReReCl.C	1.91	**145.76**	5.95	9.19	–	**74.72**	78.34	56.57	–
		Rou. Conc	11.26	**29.72**	1.18	2.03	–	**89.55**	91.15	73.47	–
	BC-F3	Comb. Conc	1.77	**164.90**	6.58	10.59	25.66	**75.34**	77.05	59.08	61.87
		Rou. Conc	14.08	**24.76**	0.94	1.65	4.10	**89.82**	87.24	73.25	78.39
	BC-F5	Comb. Conc	2.66	**96.66**	4.46	6.73	22.52	**65.85**	68.77	52.50	74.30
		Rou. Conc	13.92	**24.77**	1.08	1.88	5.79	**88.41**	87.51	76.88	100.00
	BC-F6	Comb. Conc	1.89	**128.32**	6.51	9.78	25.44	**70.10**	79.71	58.10	63.66
		Rou. Conc	11.98	**25.04**	1.17	1.98	5.42	**86.77**	90.88	74.54	86.01
	BC-F7	ReReCl.C	2.14	**130.40**	5.42	8.31	23.50	**73.87**	77.19	57.29	69.97
		Rou. Conc	12.63	**27.00**	1.08	1.87	5.28	**90.33**	90.99	76.07	92.71
	BC-F9	ReReCl.C	2.15	**124.40**	5.55	8.23	22.80	**69.40**	75.20	56.05	63.51
		Rou. Conc	13.96	**24.50**	1.02	1.71	4.79	**88.70**	89.69	75.72	86.61
	BC-F10	ReReCl.C	1.91	**150.40**	5.95	9.19	24.10	**74.30**	78.34	56.57	62.09
		Rou. Conc	11.26	**26.50**	1.18	2.03	5.22	**89.20**	91.15	73.47	92.71

The results of open circuit batch cleaner testing with 5 minutes of in-mill conditioning time at Mintek indicated that when using stainless steel grinding media, PGE recovery was in the range of 72.8% – 77.4% as compared to 79.9% for the baseline test which used 90 minutes of in-mill conditioning time. The best test result was achieved for test 4 which had 60 minutes of post mill conditioning at 30% solids and test 6 which had 30 minutes of post mill conditioning at 60% solids. The results of test 5 and 7 indicated that additional oxalic acid and thiourea addition to the conditioning stage did not provide any grade or recovery benefit.

The results of open circuit batch cleaner testing with 5 minutes of in-mill conditioning time at SGS Lakefield indicated that for tests with 5 minutes of in-mill conditioning time the PGE recovery to final concentrate was in the range 69.5% – 74.7% as compared to the SGS baseline test (BC-F1, using 90 minutes of in-mill conditioning) with a PGE recovery of 72.7%. The best PGE recovery was achieved for test BC-F10 which had 60 minutes of post mill conditioning time at 30% solids. Similar to the findings at Mintek, additional oxalic acid and thiourea addition to the post mill conditioning stage in test BC-F2 did not improve PGE recovery.

In contrast to the Phase 6 Mintek results test BC-F10 with 5 minutes of in-mill conditioning and 60 minutes post mill conditioning in a flotation cell gave a better recovery and grade profile than the baseline test BC-F1 with conditioning within the mill for the full milling duration. This was attributed to the fact that the measured grind for BC-F1 was 80% passing 89 µm for 80 minutes of milling time. The milling time was adjusted for tests BC-F7, BC-F9, and BC-F10 and the actual grinds achieved are shown in Table 13.20.

Table 13.20 Test Grinds Achieved

Test ID	P80
BC-FI	89
BC-F2	95
BC-F7	78
BC-F9	77
BC-F10	77

Mintek test 3 was conducted using carbon steel grinding media. This was found to negatively impact 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. The grade recovery profile for nickel was similar to that found when testing with stainless steel media.

The grade recovery curves for the 5 minute in-mill conditioning tests are presented in Figure 13.8 to Figure 13.10.

Figure 13.8 PGE Grade – Recovery Curves for the 5 minute In-Mill Conditioning Tests

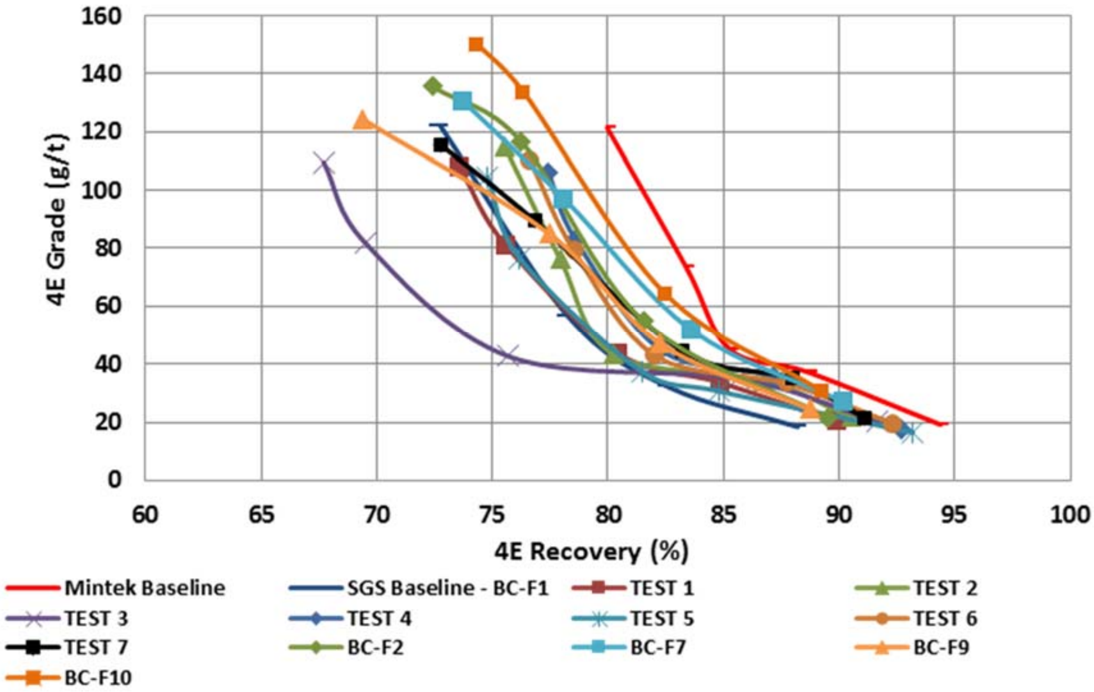


Figure 13.9 Copper Grade – Recovery Curves for the In-Mill Conditioning Tests

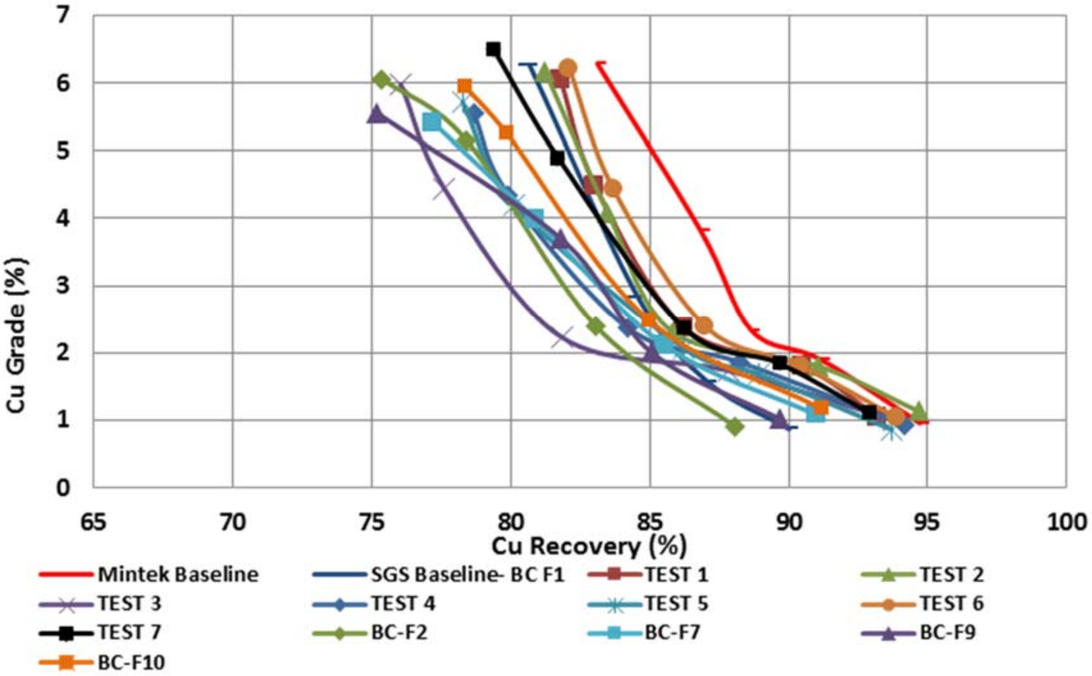
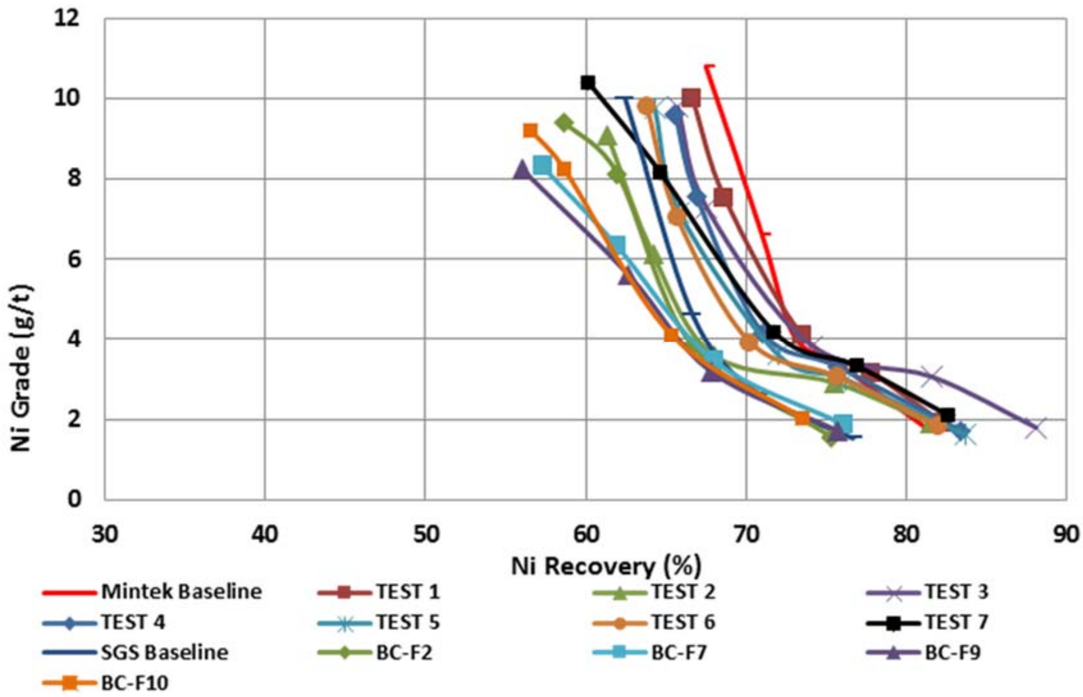


Figure 13.10 Nickel Grade – Recovery Curves for the Mintek In-Mill Conditioning Tests



Based on the 5 minute in-mill conditioning tests, the conditions used in Mintek test 4 and SGS test BC-F10 were selected as the optimum conditions for further testing. It was however noted that Mintek test 6 gave similar results to Mintek test 4.

Based on the finding that the carbon steel grinding media used in Mintek test 3 had a negative impact on PGE grade and recovery, it was decided to continue testing with either stainless steel or high chrome grinding rods, depending on the availability at each laboratory.

13.7.5 Cleaner Circuit Configuration Testing

A review of the rougher kinetic test data as presented in Section 13.7.1 indicated that due to the high fraction of fast floating PGE, copper, and nickel material there may be benefit in a split cleaner circuit configuration in which the fast floating fraction is treated in a separate cleaner to the medium and slow floating fractions. A series of tests was conducted at Mintek in Phase 6 and SGS in Phase 5 to evaluate this flowsheet option.

The conditions used for the split cleaner circuit configuration testing in Phase 5 and 6 are presented in Table 13.21.

Table 13.21 Summary of the Conditions Used for the Split Cleaner Configuration Tests

Float Conditions																		
Test ID	Description	Grinding Media	In Mill Conditioning			Post Mill Conditioning			Rougher Concentrate Conditioning			Flotation Reagents [g/t] (1% solution)					
		Type	Time (min)	Oxalic (g/t)	Thiourea (g/t)	Type	Oxalic (g/t)	Thiourea (g/t)	Description	Oxalic (g/t)	Thiourea (g/t)	SIPX	Aero 3477	Sen Froth 522	Oxalic	Thiourea	Sendep 30E
Test 8	Mintek Phase 6	Stainless Steel	5	200	50	30 min 60% Solids 1200 RPM	100	25	2 min	80	20	93	93	80	40	10	110
Test 9		Stainless Steel	5	200	50	30 min 60% Solids 1200 RPM	100	25	LG 10 min Polish Grind	80	20	93	93	80	80	20	110
Test 12		Stainless Steel	5	200	50	60 min 30% Solids 1200 RPM	–	–	10 min HG, MG, & LG	80	20	93	93	83	–	–	83
BC-F5	SGS Phase 5	High Chrome 440C	5	200	50	30min 60% Solids	100	25	2 min HG Con 10 min LG Con	160	40	90	90	83	–	–	165
BC-F6		High Chrome 440C	5	200	50	30min 60% Solids	100	25	2 min HG Con 10 min LG Con Polish Grind	160	40	90	90	80	–	–	155

Two distinct cleaner circuit configurations were tested using an in-mill conditioning time of five minutes as follows:

- Mintek test 8 and 9 and SGS test BC-F5 and BC-F6: A two-stage high-grade cleaner circuit operated in open circuit treating a high grade rougher concentrate and a three stage low grade cleaner circuit treating the combined medium and low grade rougher concentrates with no scavenger cleaner circuit was tested (only two stages of low grade cleaning were used for BC-F5 and BC-F6). In test 9 and BC-F6 a polishing regrind was included on the combined medium grade and low grade rougher concentrates prior to cleaner flotation.
- Test 12: A two-stage high-grade cleaner circuit treating a high grade rougher concentrate and a three stage medium-grade cleaner circuit treating the combined medium-grade rougher concentrates and scavenger cleaner concentrate with a scavenger cleaner circuit treating low grade rougher concentrate was tested. The high-grade cleaner circuit tails were recycled to the medium cleaner circuit and the medium cleaner circuit tails were recycled to the scavenger cleaner circuit.

The flowsheets for the split cleaner tests are presented in Figure 13.11 and Figure 13.12 below.

Figure 13.11 Flowsheet Used for Cleaner Configuration Tests 8 and 9 And BC-F5 and BC-F6

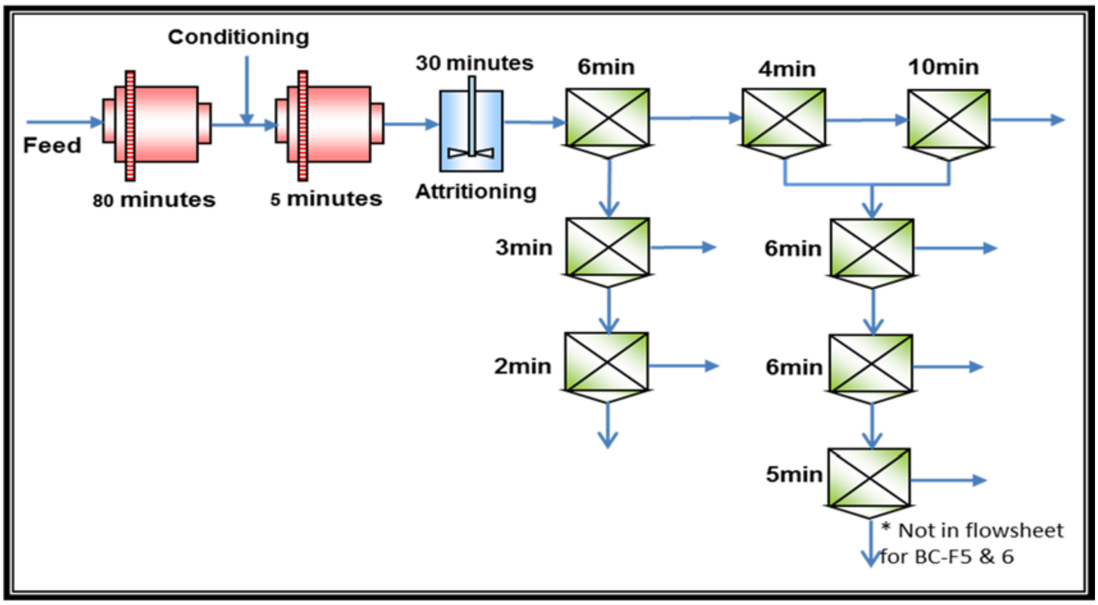
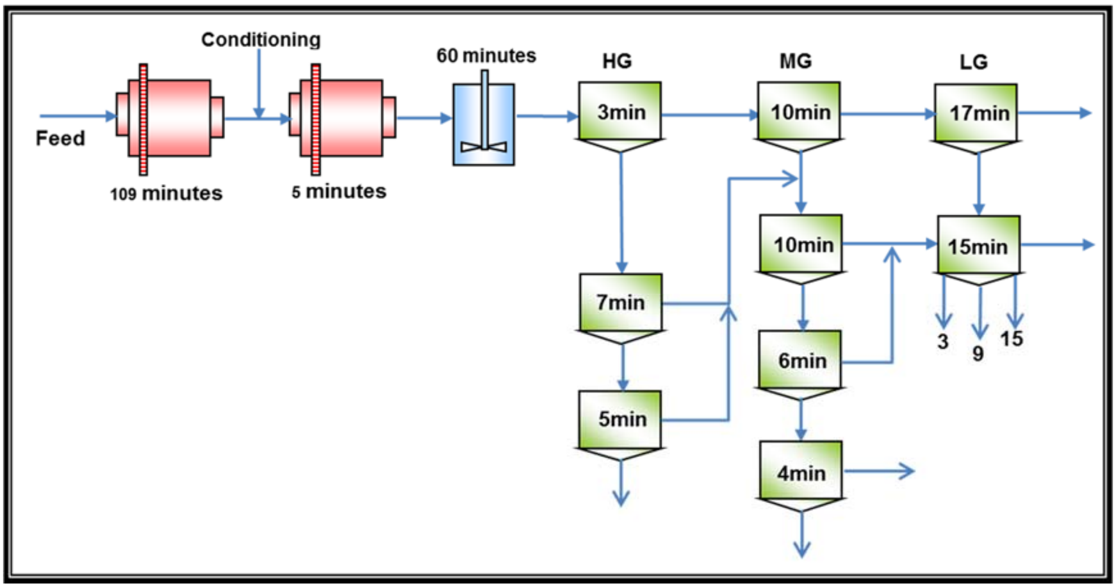


Figure 13.12 Flowsheet Used for Cleaner Configuration Test 12



The results for split cleaner configuration tests are presented in Table 13.22.

Table 13.22 Summary of the Results for the Split Cleaner Circuit Tests

				Grade				Recovery			
Laboratory	**Test No.**	**Products**	**Mass (%)**	**3PE+Au (g/t)**	**Cu (%)**	**Ni (%)**	**S (%)**	**3PE+Au (%)**	**Cu (%)**	**Ni (%)**	**S (%)**
Mintek Phase 6	8	Comb. Conc	2.84	**96.16**	5.76	9.04	24.52	**64.46**	75.86	55.09	59.71
		Rou. Conc	18.31	**20.45**	1.09	2.10	5.48	**88.37**	92.29	82.48	85.99
	9	Comb. Conc	3.14	**90.72**	5.14	8.38	22.68	**67.39**	76.62	56.38	63.12
		Rou. Conc	21.35	**17.82**	0.91	1.82	4.63	**89.91**	92.43	83.17	87.46
	12	Comb. Conc	3.15	**109.08**	5.48	10.07	23.85	**81.99**	84.55	72.32	69.25
		Rou. Conc	18.19	**21.24**	1.04	2.01	5.47	**92.27**	92.79	83.21	91.70
SGS Phase 5	BC-F5	Comb. Conc	2.66	**96.66**	4.46	6.73	22.52	**65.85**	68.77	52.50	74.30
		Rou. Conc	13.92	**24.77**	1.08	1.88	5.79	**88.41**	87.51	76.88	100.00
	BC-F6	Comb. Conc	1.89	**128.32**	6.51	9.78	25.44	**70.10**	79.71	58.10	63.66
		Rou. Conc	11.98	**25.04**	1.17	1.98	5.42	**86.77**	90.88	74.54	86.01

The results of the split cleaner circuit testing with 5 minutes of in-mill conditioning time indicated that a split cleaner circuit as per the flowsheet derived for test 12 gave a PGE recovery of 82% at a grade of 109.1 g/t as compared to 77.4% at a grade of 105.9 g/t for the optimized simple cleaner circuit used for test 4.

The grade recovery curves for the circuit configuration tests relative to the performance of the optimal 3 stage cleaner test 4 are presented in Figure 13.13 to Figure 13.15.

Figure 13.13 PGE Grade – Recovery Curves for the Split Cleaner Circuit Configuration Tests

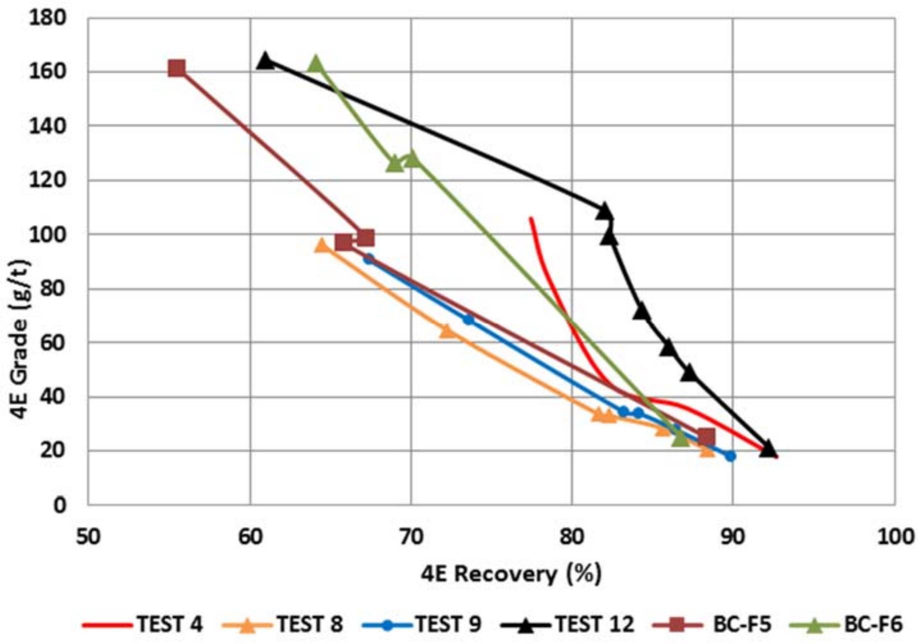


Figure 13.14 Copper Grade – Recovery Curves for the Split Cleaner Circuit Configuration Tests

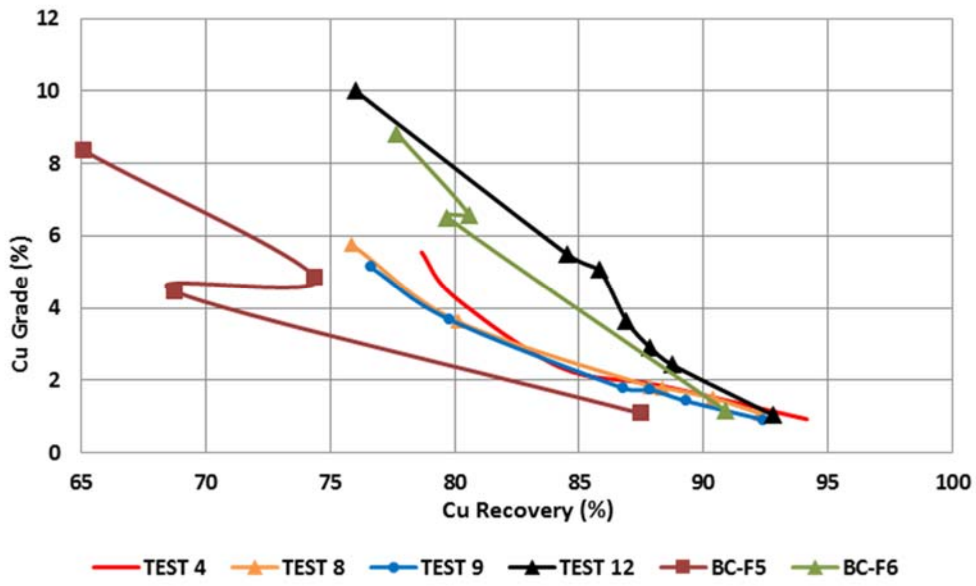
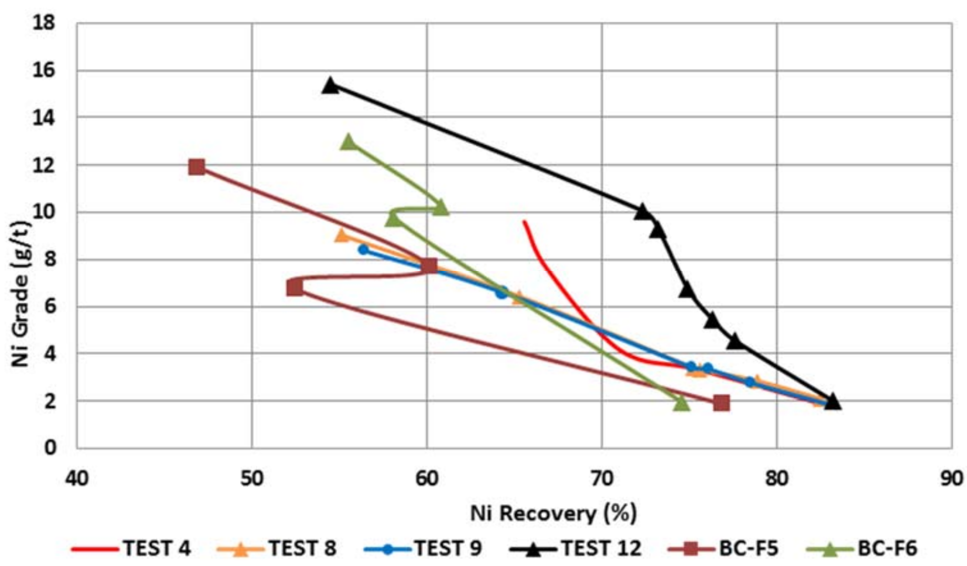


Figure 13.15 Nickel Grade – Recovery Curves for the Split Cleaner Circuit Configuration Tests



13.7.6 Evaluation of Conditioning Parameters for a Split Cleaner Circuit

Based on the results of the split cleaner circuit configuration testing as presented in Section 13.7.3, it was decided to re-evaluate the oxalic acid and thiourea conditioning parameters for a split cleaner circuit configuration using the optimized test conditions in Mintek test 12. This testing was conducted at Mintek using the Phase 6 composite sample containing 15% T1, 42.5% T2U, and 42.5% T2L.

The test conditions and flowsheet used are presented in Table 13.23 and Figure 13.16.

Figure 13.16 Flowsheet Used for Conditioning Tests Using a Split Cleaner Configuration

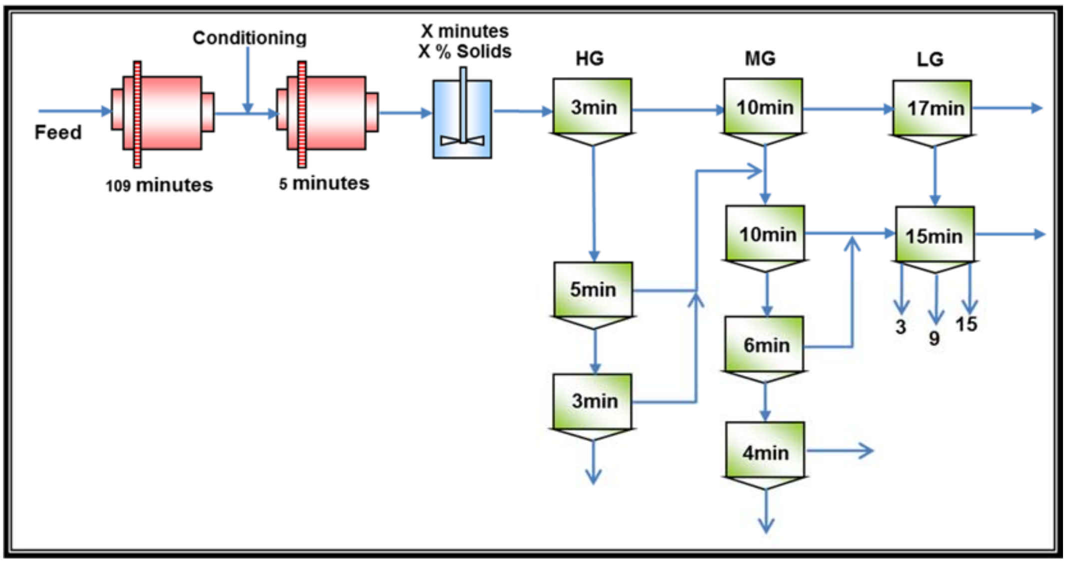


Table 13.23 Summary of the Conditions Used for Conditioning Tests Using a Split Cleaner Configuration

Test ID	Description	Grinding Media	In Mill Conditioning			Post Mill Conditioning			Rougher Concentrate Conditioning			Flotation Reagents [g/t] (1% solution)					
	Flowsheet Configuration	Type	Time (min)	Oxalic (g/t)	Thiourea (g/t)	Description	Oxalic (g/t)	Thiourea (g/t)	Description	Oxalic (g/t)	Thiourea (g/t)	SIPX	Aero 3477	Sen Froth 522	Oxalic	Thiourea	Sendep 30E
Test 16	MF1 with Split Cleaner Circuit (HG, MG & LG)	Stainless Steel	5	200	50	–	–	–	LG 10 min	30	7	93	93	80	50	12	83
Test 17	MF1 with Split Cleaner Circuit (HG, MG & LG)	Stainless Steel	5	200	50	30 min 60% Solids 1200 RPM	–	–	LG 10 min	30	7	93	93	80	50	12	83
Test 18	MF1 with Split Cleaner Circuit (HG, MG & LG)	Stainless Steel	5	200	50	15 min 60% Solids 1200 RPM	–	–	LG 10 min	30	7	93	93	80	50	12	83
Test 19	MF1 with Split Cleaner Circuit (HG, MG & LG)	Stainless Steel	5	200	50	–	–	–	LG 10 min Polish Grind with Reagent	30	7	93	93	80	70	17	85
Test 20	MF1 with Split Cleaner Circuit (HG, MG & LG)	Stainless Steel	5	200	50	60 min 60% Solids 400 RPM	–	–	5 min HG & MG	50	12	93	93	80	30	7	85

Table 13.24 Summary Results for the Split Cleaner Circuit Conditioning Tests

Effects of Oxalic acid & Thiourea ConditioningTime										
Test No.	Products	Mass (%)	Grade				Recovery			
			3PE+Au (g/t)	Cu (%)	Ni (%)	S (%)	3PE+Au (%)	Cu (%)	Ni (%)	S (%)
12	Comb. Conc	3.15	**109.08**	5.48	10.07	23.85	**81.99**	84.55	72.32	69.25
	Rou. Conc	18.19	**21.24**	1.04	2.01	5.47	**92.27**	92.79	83.21	91.70
16	Comb. Conc	3.68	**86.80**	4.83	7.10	18.73	**72.67**	83.76	58.15	60.50
	Rou. Conc	20.38	**20.14**	0.98	1.78	5.09	**93.30**	93.85	80.52	90.92
17	Comb. Conc	4.56	**81.23**	4.18	7.01	20.26	**83.65**	85.07	68.94	81.94
	Rou. Conc	21.94	**19.20**	0.95	1.76	4.89	**95.14**	93.44	83.18	95.15
18	Comb. Conc	5.26	**68.01**	3.57	5.91	16.76	**78.65**	85.33	66.04	77.50
	Rou. Conc	23.59	**18.15**	0.88	1.67	4.60	**94.07**	94.45	83.79	95.30
19	Comb. Conc	2.98	**106.67**	5.65	9.69	22.50	**73.73**	80.86	62.32	62.60
	Rou. Conc	22.80	**17.41**	0.85	1.66	4.29	**92.13**	92.99	81.66	91.34
20	Comb. Conc	2.78	**108.30**	6.32	9.39	22.93	**67.43**	76.30	56.60	57.59
	Rou. Conc	21.24	**19.78**	0.99	1.80	4.76	**94.24**	91.67	82.91	91.45

The results of Mintek split cleaner circuit conditioning tests indicated that overall PGE recovery and grade are sensitive to conditioning parameters.

The conditions used for test 17 with 30 minutes of post mill conditioning at 60% solids were selected for further testing due to the higher PGE recovery. It was noted that this test resulted in a lower overall concentrate grade of 80.2 g/t as compared to 109 g/t for test 12 in which the post mill condition stage was operated for 60 minutes at 30% solids. The mass pull for test 17 was 4.6% as compared to 3.2% for test 12.

The grade-recovery curves for the Phase 6 Mintek split cleaner circuit conditioning testing with of in-mill conditioning are presented in Figure 13.17 to Figure 13.19.

Figure 13.17 PGE Grade – Recovery Curves for the Split Cleaner Conditioning Tests

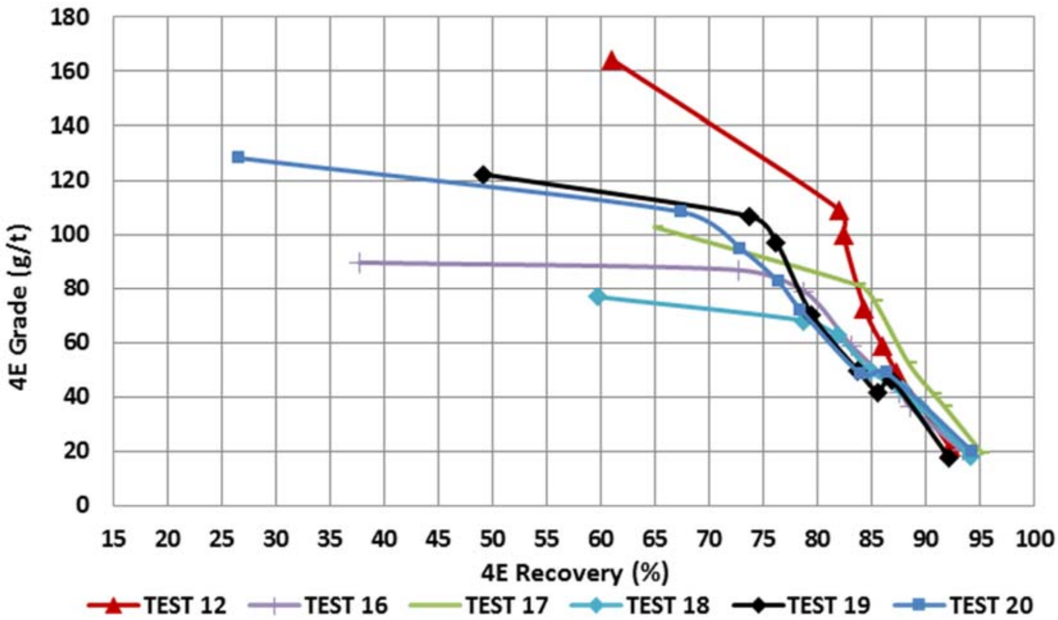


Figure 13.18 Copper Grade – Recovery Curves for the Split Cleaner Conditioning Tests

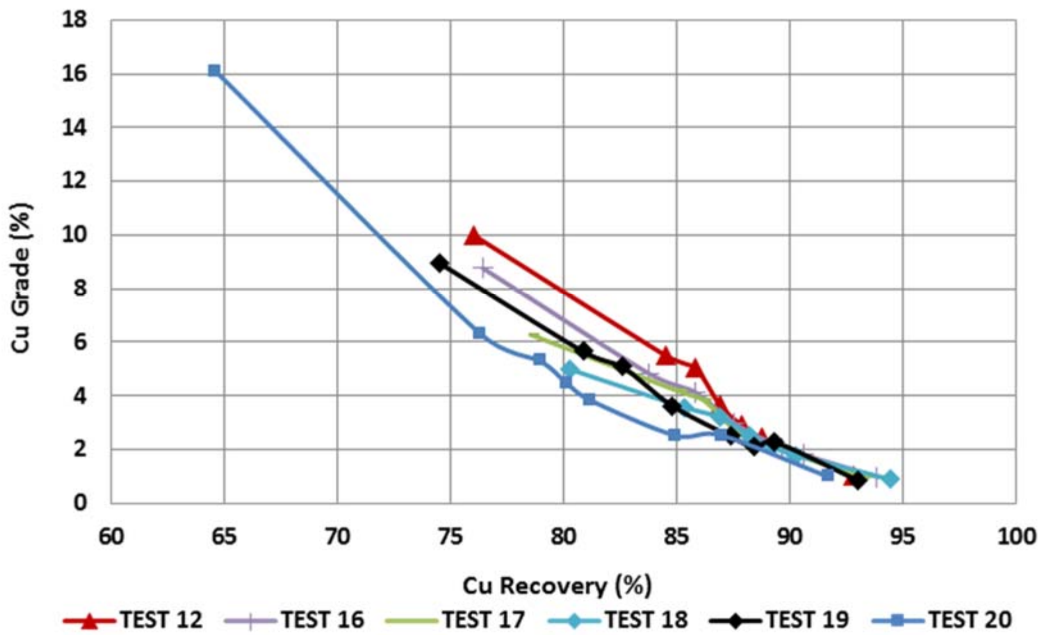
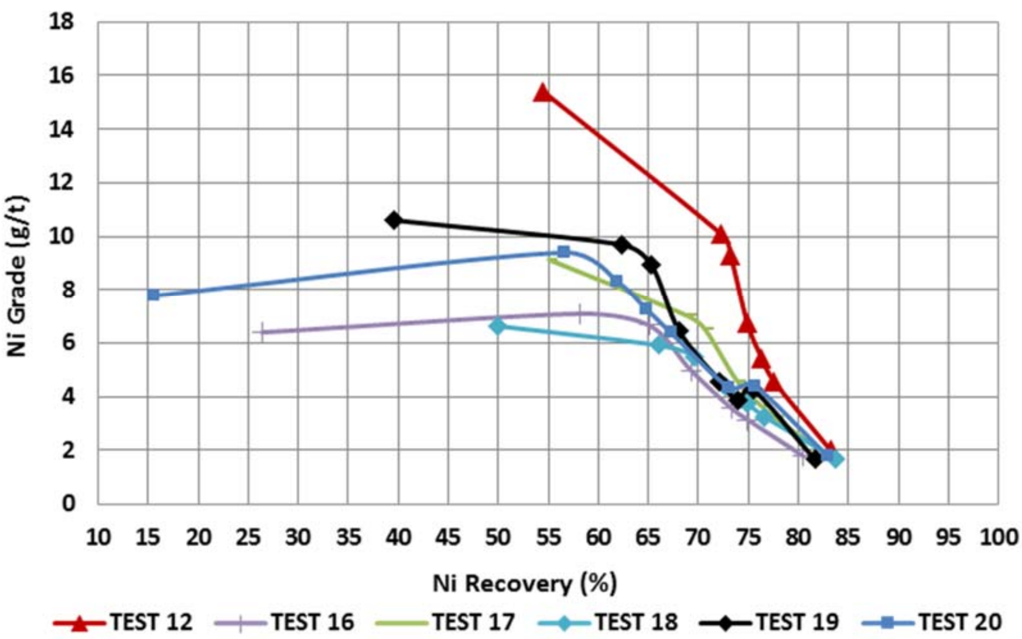


Figure 13.19 Nickel Grade – Recovery Curves for the Split Cleaner Conditioning Tests



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

During Phase 4 and 5 three (3) locked cycle tests were performed using a simple three stage cleaning circuit treating a combined rougher concentrate. In addition to this, three locked cycle tests were performed using the split cleaner circuit configuration and 5 minutes of in-mill conditioning time. The results of this testing are presented in Sections 13.8.1 and 13.8.2.

13.8.1 Locked Cycle Testing of a Three Stage Cleaning Circuit Configuration

Three locked cycle tests were conducted using a simple three stage cleaning circuit with an in-mill conditioning time in excess of 80 minutes during Phase 4 and Phase 5 at Mintek and SGS respectively. The flowsheet used for these tests is presented in Figure 13.20 and the test conditions are presented in Table 13.25.

Figure 13.20 Flowsheet Used for the Three Stage Cleaner Locked Cycle Tests in Phase 4 and 5

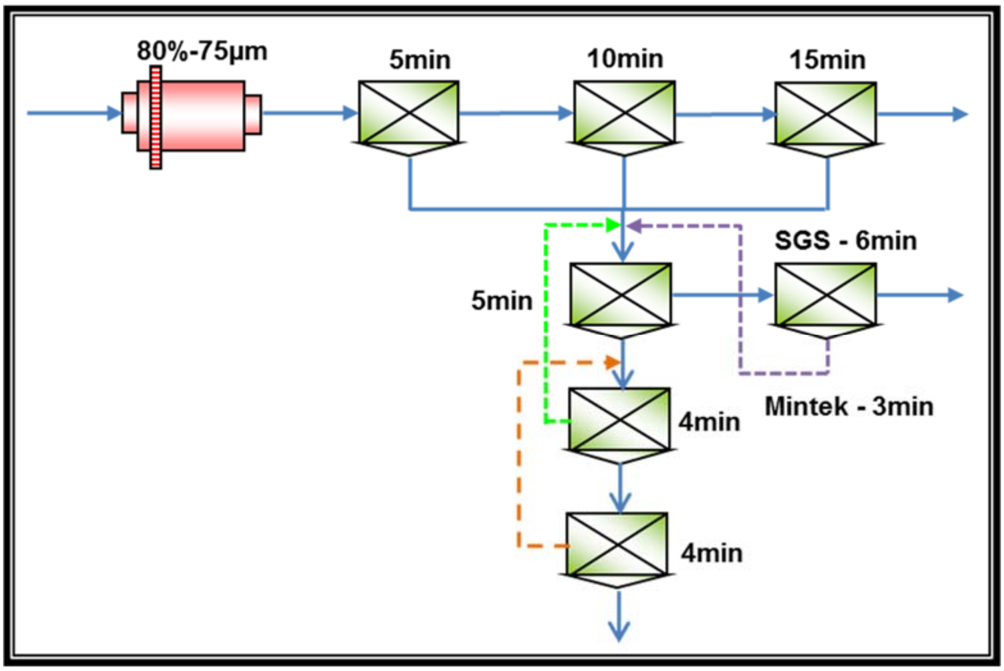


Table 13.25 Three Stage Cleaner Circuit Locked Cycle Test Conditions

Float Conditions									
Stage	**Reagents [g/t] (1% solution)**						**Time [min]**		
	SIPX	**3477**	**Senfroth**	**Oxalic**	**Thiourea**	**Sendep**	**Mill**	**Condition**	**Float**
Mill	–	–	–	200	50	–	Xmin	–	–
Rougher Circuit									
Rougher 1	25	25	35	–	–	–	–	1	5
Rougher 2	25	25	10	–	–		–	1	10
Rougher 3	25	25	10	–	–		–	1	15
Cleaner Circuit	–	–	–	80	20		–	2	–
Cleaner 1	10	10	5	–	–	50	–	1	5
Cleaner Scav.	2.5	2.5	5	–	–		–	1	3 (Mintek) 6 (SGS)
ReCleaner Stage	2.5	2.5	5	–	–	25	–	1	4
ReReCleaner Stage	2.5	2.5	7.5	–	–	10	–	1	4
Total	**93**	**93**	**78**	**280**	**70**	**85**			

The results of the simple three stage cleaner circuit locked cycle tests are presented in Table 13.26.

Table 13.26 Three Stage Cleaner Circuit 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 4	15% T1, 42.5% T2U & 42.5% T2L	18.49	2.44	0.11	0.22	1.10	25.33	9.62	9.98	8.11	8.68	20.29	19.10	18.42
Mintek Phase 4	50% T2U & 50% T2L	19.59	2.76	0.12	0.37	1.74	27.05	10.30	11.65	8.76	13.96	31.52	19.05	19.54
SGS Phase 5	80% Min Zone & 20% Footwall	16.67	19.34	0.76	1.32	3.42	21.73	12.00	89.76	92.89	76.55	84.67	2.15	21.73

For tests conducted on the mineralized zone only (Mintek Phase 4):

- The final concentrate mass pull was 3.0% – 3.2%.
- The projected PGE (3E + Au) recovery was in the range 83.8% – 85.2% at a final concentrate grade of 120g/t – 126 g/t.
- The projected copper recovery was 76.5% at a final concentrate copper grade of 5.9% – 6.7%.
- The projected nickel recovery was in the range 67.9% – 72.5% at a final concentrate nickel grade of 10.7% – 11.3%.

The test conducted at SGS in Phase 5 was conducted on a sample blend containing 80% mineralized zone and 20% footwall, the results were as follows:

- The final concentrate mass pull was 3.55%
- The projected PGE recovery was 81.9% at a final concentrate grade of 82.9 g/t.
- The projected copper recovery was 87.3% at a final concentrate grade of 3.4%.
- The projected nickel recovery was 66.6% at a final concentrate grade of 5.4%.

Locked cycle test II in Phases 4 and 5 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.

13.8.2 Locked Cycle Testing of the Split Cleaner Circuit Configuration

Three locked cycle tests were conducted using a split cleaner circuit with an in-mill conditioning time of 5 minutes during Phase 5 and Phase 6 at SGS and Mintek respectively. The flowsheet used for these tests is presented in Figure 13.21 and the test conditions are presented in Table 13.27.

The results of the split cleaner circuit locked cycle tests with 5 minutes of in-mill conditioning time are presented in Table 13.28.

Figure 13.21 Flowsheet Used for the Phase 5 and 6 Locked Cycle Tests Using a Split Cleaner Circuit Configuration

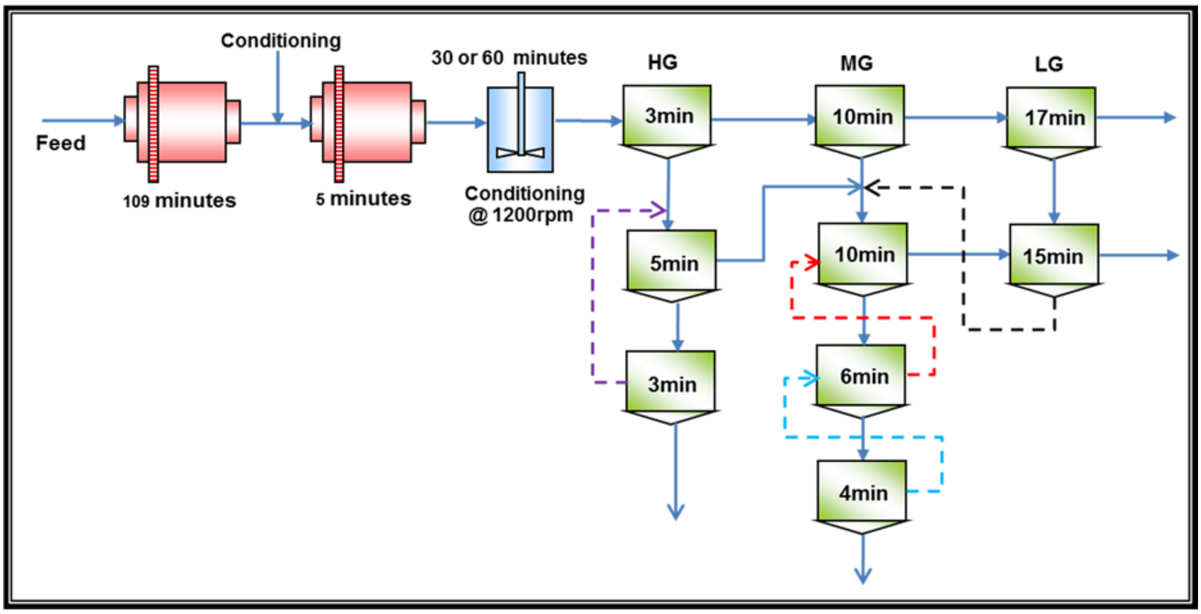


Table 13.27 Split Cleaner Circuit Locked Cycle Test Conditions

Stage	Reagents [g/t] (1% solution)								
	SIPX	Aero 3477	Senfroth 522	Oxalic	Thiourea	Sendep 30E	Mill	Condition	Float
Mill	–	–	–	–	–	–	Xmin	–	–
Mill	–	–	–	200	50	–	5	–	–
Pre-Conditioning	–	–	–	–	–	30 min @60% Solids or 60 min @30% Solids	–	30	–
Rougher Circuit						–			
Rougher 1	25	25	35	–	–	–	–	1	3
Rougher 2	25	25	10	–	–	–	–	1	10
Rougher 3	25	25	10	–	–	–	–	1	17
High Grade Cleaner Circuit	–	–	–	30	7	–	–	5	–
HG 1st Cleaner	5	5	7.5	–	–	15 + 7.5 After ~1 min of Chalcopyrite float	–	1	5
HG 2nd Cleaner	2.5	2.5	5	–	–	2.5	–	1	3
Mid Grade Cleaner Circuit	–	–	–	30	7	–	–	5	–
MG 1st Cleaner	5	5	5	–	–	50 + 5 After ~1 min of Chalcopyrite float	–	1	10
MG 2nd Cleaner	2.5	2.5	5	20	5	15	–	2	6
MG 3nd Cleaner	–	–	–	–	–		–	1	4
Low Grade Cleaner Circuit	–	–	–	30	7	–	–	10	–
LG 1st Cleaner	2.5	2.5	5	–	–	–		1	15
Total	**93**	**93**	**83**	**310**	**76**	**80**			

Table 13.28 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 (%)
SGS Phase 5	15% T1, 42.5% T2U & 42.5% T2L	4.62	68.17	2.69	4.54	12.77	14.86	21.41	86.00	88.29	69.91	80.37	3.02	10.14
SGS Phase 5	15% T1, 42.5% T2U & 42.5% T2L	3.50	87.67	3.61	5.90	15.77	12.41	24.66	84.88	86.71	67.96	76.73	1.87	8.88
Mintek Phase 6	15% T1, 42.5% T2U & 42.5% T2L	4.07	93.00	4.91	8.33	24.35	6.02	29.85	87.80	88.38	73.15	87.15	0.99	13.59

For tests conducted on the mineralized zone only in the ratio 15% T1, 42.5% T2U, and 42.5% T2L:

- The final concentrate mass pull was in the range 3.5% – 4.6%.
- The projected PGE (3E + Au) recovery was in the range 86.0% – 87.8% at a final concentrate grade of 68 g/t – 93 g/t.
- The projected copper recovery was in the range 86.7% – 88.4% at a final concentrate copper grade of 2.7% – 4.9%.
- The projected nickel recovery was in the range 67.9% – 73.1% at a final concentrate nickel grade of 4.5% – 8.3%.

The locked cycle testing in Phases 5 and 6 indicated that the split cleaner circuit configuration with 5 minutes of in-mill conditioning improved PGE (3E + Au) and copper recovery with an approximate 1.7% PGE recovery improvement and 11% copper recovery improvement as compared to the Mintek Phase 4 locked cycle testing on the mineralized zone sample. The mass pull achieved for the split cleaner circuit tests was 0.5% – 1% higher than that achieved for the simple cleaner circuit tests and as a result the concentrate grade was lower. Saleable concentrate grades of >80 g/t were still achieved for these tests.

Locked cycle test II in Phases 4 and 5 had an oxalic acid and thiourea in-mill conditioning time in excess of 80 minutes which would not be possible to replicate for a full scale milling operation and as such the locked cycle testing results in Phases 4 and 5 are not considered to be representative of the grade and recovery that can be achieved for a commercial scale operation.

13.8.3 Concentrate Analysis

An XRD analysis was performed on the combined final concentrate for the Phase 6 Mintek split cleaner locked cycle test and the results are presented in Table 13.29.

Table 13.29 Mintek Phase 6 Locked Cycle Test Concentrate Results by XRD Analysis

Mineral	Ideal Chemical Formulae	Mass %
Actinolite	$Ca_2(Mg,Fe)_5(Si_8O_{22})(OH)_2$	2
Chalcopyrite	$CuFeS_2$	16
Clinochlore	$(Mg,Fe)_5Al(AlSi_3O_{10})(OH)_8$	3
Lizardite	$Mg_3(Si_2O_5)(OH)_4$	0
Pentlandite	$(Fe,Ni)_9S_8$	31
Pyrrhotite	Fe_7S_8	25
Talc	$Mg_3Si_4O_{10}(OH)_2$	8
Anorthite	$CaAl_2Si_2O_8$	6
Chromite	$FeO.Cr_2O_3$	1
Magnetite	Fe_3O_4	2
Dickite	$Al_2(Si_2O_5)(OH)_4$	6
Quartz	SiO_2	0

The detailed concentrate analysis for the Mintek Phase 6 locked cycle test is presented in Table 13.30 to Table 13.33. The analysis is for a combined final concentrate produced from a split cleaner circuit.

Table 13.30 Mintek Phase 6 Locked Cycle Test Concentrate PGM Analysis

Sample Name Descreption	Rep	Pt ppm	Pd ppm	Rh ppm	Au ppm	3PE+Au ppm	Pt/Pd Ratio ppm
Locked Cycle Final Conc. HG+MG Composite	1	40.20	46.40	2.71	4.91	94.22	0.87
Locked Cycle Final Conc. HG+MG Composite	2	41.80	47.00	2.88	5.04	96.72	0.89

Table 13.31 Mintek Phase 6 Locked Cycle Test Concentrate Results by ICP

Sample Name Descreption	Rep	ICP1 Mg %	Al %	Si %	Ca %	Ti %	V %	Cr %	Mn %	MgO %	SiO2 %	Fe %	Co %	Ni %	Cu %	Zn %	Pb %
Locked Cycle Final Conc. HG+MG Composite	1	4.09	**0.77**	7	0.69	<0.05	<0.05	<0.05	<0.05	6.78	14.98	32.3	0.2	8.88	4.73	<0.05	<0.05
Locked Cycle Final Conc. HG+MG Composite	2	4.1	**0.77**	6.96	0.69	<0.05	<0.05	<0.05	<0.05	6.80	14.89	32	0.2	8.81	4.64	<0.05	<0.05

Table 13.32 Mintek Phase 6 Locked Cycle Test Concentrate Results by ICP-MS

Sample Name Descreption	Rep	Ti ppm	V ppm	As ppm	Se ppm	Sr ppm	Y ppm	Li ppm
Locked Cycle Final Conc. HG+MG Composite	1	384	**24**	15.1	102	10.1	2.01	3.48
Locked Cycle Final Conc. HG+MG Composite	2	380	**25.2**	15.3	100	10.4	1.9	3.31

Table 13.33 Mintek Phase 6 Locked Cycle Test Concentrate Results by XRF

Sample Name Description	**Al %**	**Si %**	**P %**	**S %**	**Cl %**	**K %**	**Ca %**	**Ti %**	**V %**	**Cr %**	**Mn %**	**Fe %**	**Co ppm**	**Ni ppm**	**Cu ppm**	**Zn ppm**
Locked Cycle Final Conc. HG+MG Composite	<0.050	8.000	<0.005	15.140	<0.003	<0.002	0.649	0.039	0.006	0.063	0.049	28.530	2015	67900	36800	479
	Ga ppm	**Ge ppm**	**As ppm**	**Se ppm**	**Br ppm**	**Rb ppm**	**Sr ppm**	**Y ppm**	**Zr ppm**	**Nb ppm**	**Mo ppm**	**Ag ppm**	**Cd ppm**	**In ppm**	**Sn ppm**	**Sb ppm**
	22.4	5.90	15.4	94.9	14.8	<1.0	10.9	5.40	7.60	<1.00	14.6	11.7	2.80	1.30	1.60	<1.50
	Te ppm	**I ppm**	**Cs ppm**	**Ba ppm**	**La ppm**	**Ce ppm**	**Hf ppm**	**Ta ppm**	**W ppm**	**Hg ppm**	**Tl ppm**	**Pb ppm**	**Bi ppm**	**Th ppm**	**U ppm**	
	16.0	1.70	3.40	7.20	5.00	14.9	<2.00	<2.00	<2.00	4.50	5.20	50.0	15.3	<2.00	5.60	

13.9 Process Plant Recovery Estimate

The plant recovery estimates are based on locked cycle testing conducted at SGS and Mintek as presented in Section 13.8. The locked cycle test results that were used were for tests conducted on a composite sample containing material from the mineralized zone in the ratio 15% T1, 42.5% T2U, and 42.5% T2L using a split cleaner circuit configuration. Locked cycle test results were used as these results are considered to provide a more accurate estimate of final recovery and concentrate grade than open circuit testing. All results are based on bench scale laboratory test work conducted on a small scale in a controlled environment. It should be noted that the samples used did not contain any footwall dilution which would bepresent in full-scale mining operations.

13.9.1 Locked Cycle Tests Used to Derive the Recovery Estimate

The locked cycle test conducted at Mintek in Phase 6 was conducted using the optimized conditions as presented in Section 13.8 above. The PGE, copper and nickel recovery and final concentrate grades achieved for this test are presented in Table 13.34.

Table 13.34 Mintek Phase 6 Locked Cycle Test Results

		Grade				Recovery			
	Mass (%)	**3PE+Au (g/t)**	**Cu (%)**	**Ni (%)**	**Total S (%)**	**3PE+Au (%)**	**Cu (%)**	**Ni (%)**	**Total S (%)**
HG ReCleaner Conc	2.06	115	8	10	30	55	73	46	54
MG ReReCleaner Conc	2.01	70	2	6	19	33	15	27	33
Combined Conc	**4.07**	**93**	**5**	**8**	**24**	**88**	**88**	**73**	**87**
LG Cleaner Scavenger Tails	29.13	1.04	0.06	0.19	0.30	7	7	12	8
Rougher Tails	66.81	0.33	0.01	0.11	0.09	5	4	15	5
Head Calculated	100.00	4.31	0.23	0.46	1.14	100.0	100.0	100.0	100.0
Head Measured	–	4.50	0.22	0.46	1.18	–	–	–	–

The locked cycle test III conducted at SGS in Phase 5 was conducted using the optimized conditions as presented in Section 13.8 above but with 60 minutes of post mill conditioning at 30% solids. Based on the preliminary open circuit testing, this conditioning regime appeared to give similar results to tests with 30 minutes post conditioning at 60% solids. The PGE, copper and nickel recovery and final concentrate grades achieved for this test are presented in Table 13.35. The test performed at SGS gave lower concentrate grades and recoveries than the Mintek test. This is probably due to the fact that the head grade of the SGS sample is lower than that of the Mintek sample.

Table 13.35 SGS Phase 5 Locked Cycle Test III Results

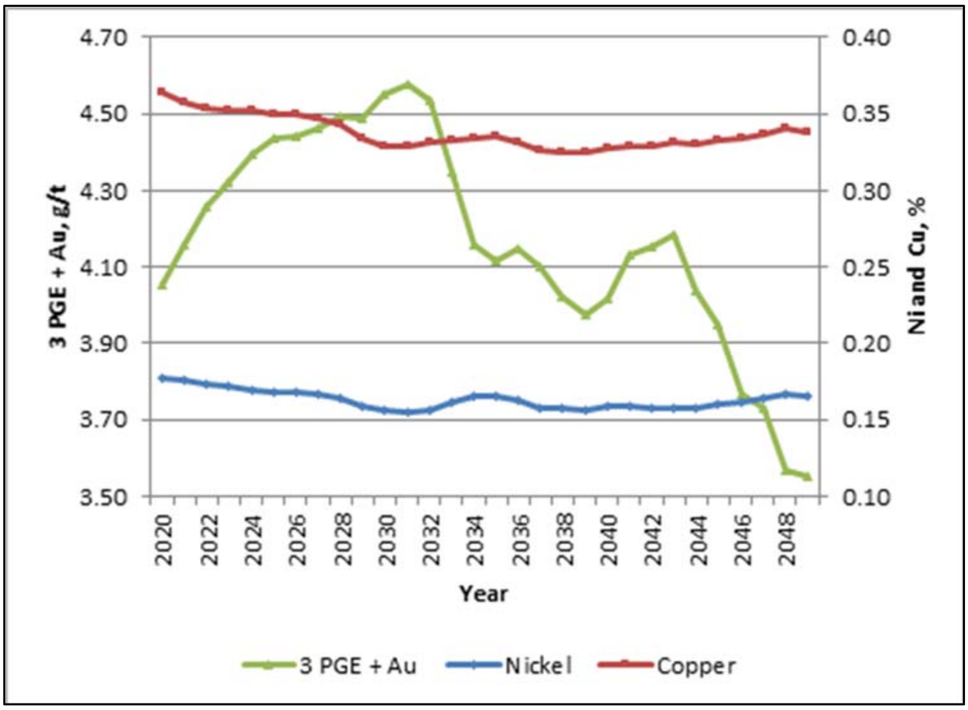
Product	Mass	Assay (g/t), (%)					Distribution (%)				
	%	2PE+Au	Rh	Cu	Ni	S	2PE+Au	Rh	Cu	Ni	S
HG 2nd Cl Conc	1.5	154	5	8	11	22	67	62	80	55	46
MG 3rd Cl Conc	2.0	32	1	0.49	2	11	18	17	7	13	30
Combined Cl Conc	**3.5**	**85**	**3**	**4**	**6**	**16**	**85**	**80**	**87**	**68**	**77**
Rougher Conc	13	25	0.82	1	2	5	91	86	92	75	84
LG Cl Tail	9.4	2.0	0.08	0.080	0.21	0.58	5	6	5	7	8
Rougher Tail	87.1	0.38	<0.02	0.014	0.09	0.13	9	14	8	25	16
Head (calc.)	100.0	3.50	0.12	0.15	0.30	0.72	100.0	100.0	100.0	100.0	100.0
(direct)	–	3.49	0.06	0.16	0.30	0.74	–	–	–	–	–

SGS reported a calculated 2PE + Au head grade of 3.50 g/t with Rhodium at 0.12 g/t the comparative 4PGE head grade (3PE + Au) was thus 3.61 g/t.

13.9.2 Plant Recovery Estimate

Taking the results as presented in Table 13.35 and Table 13.36, a recovery estimate for full scale plant operations was derived based on 3PE + Au, copper and nickel head grades from the mine plan. A summary of these head grades is presented in Figure 13.22.

Figure 13.22 Base Case 4 Mtpa Mine Plan, PGE, Copper and Nickel Head Grades

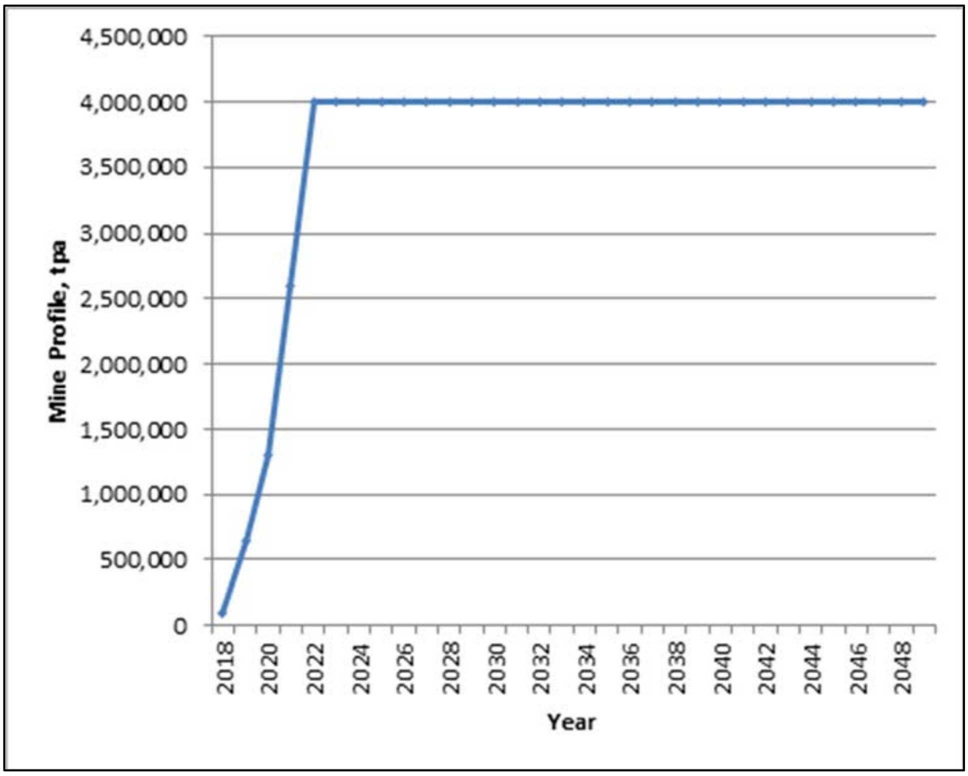


In the mine plan as presented in Figure 13.23:

- PGE head grade varies between 3.55 g/t and 4.58 g/t with an average LOM head grade of 4.17 g/t for the base case mining plan.
- Copper head grade varies between 0.16% and 0.18% with an average LOM head grade of 0.16% for the base case mining plan.
- Nickel head grade varies between 0.37% and 0.32% with an average LOM head grade of 0.34% for the base case mining plan.

The expected tonnage profile for the 4 Mtpa base case mining plan is presented in Figure 13.23.

Figure 13.23 Base Case Mine Plan Production Profile



The mine plan as presented in Figure 13.22 and Figure 13.23 was used to derive a process plant production schedule for the 4 Mtpa base case mining scenario. It must be noted that footwall dilution and hangingwall dilution are minor.

Using the locked cycle test results as presented in Table 13.35 and Table 13.36, a final residue grade for the combined tailings stream comprising of rougher tailings and scavenger cleaner tailings was calculated for each of the PGE elements (3PE + Au) as well as copper and nickel. These calculated combined tailings final residue grades are presented in Table 13.36.

Table 13.36 Combined Tails Estimates Based on Locked Cycle Test Results

	Mass (%)	PGE (g/t)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Cu (%)	Ni (%)
SGS Phase 5 Locked Cycle Test III								
Head Grade	100	3.61	1.61	1.65	0.12	0.23	0.15	0.30
Rougher Tail	87.1	0.39	0.17	0.16	0.02	0.04	0.01	0.09
Scav Tail	9.4	2.04	0.80	0.97	0.08	0.20	0.08	0.21
Combined Tail	96.5	0.55	0.23	0.24	0.03	0.06	0.02	0.10
Mintek Phase 6 Locked Cycle Test								
Head Grade	100	4.31	1.89	2.06	0.11	0.25	0.23	0.46
Rougher Tail	66.8	0.33	0.14	0.16	0.00	0.04	0.01	0.11
Scav Tail	29.1	1.04	0.43	0.48	0.03	0.11	0.06	0.19
Combined Tail	95.9	0.55	0.22	0.25	0.01	0.06	0.03	0.13

- The final PGE residue grade for the SGS test was 0.55 g/t for a calculated PGE (3E + Au) test head grade of 3.61 g/t, the locked cycle test at Mintek gave a similar PGE residue grade of 0.55 g/t for a calculated head grade of 4.31 g/t.
- The final copper residue grade for the SGS test was 0.02% for a calculated test head grade of 0.14% as compared to a copper residue grade of 0.03% for a calculated head grade of 0.23% at Mintek.
- The final nickel residue grade for the SGS test was 0.10% for a calculated test head grade of 0.30% as compared to a copper residue grade of 0.13% for a calculated head grade of 0.46% at Mintek.

The SGS locked cycle test used samples that had lower measured PGE (3E + Au) , copper and nickel head grades of 3.55 g/t, 0.14% and 0.30% respectively as compared to 4.50 g/t, 0.23%, and 0.46% for PGEs (3PE + Au), copper and nickel respectively in the Mintek test. The lower head grades in the SGS resulted in noticeably lower recoveries for the SGS test as compared to Mintek.

A review of the process plant feed grades for each of these elements as presented in Figure 13.22 indicates that for PGEs the plant feed grade is will be in the range 3.55 g/t – 4.58 g/t, which means that both the SGS and Mintek test results provide an indication of the process plant recovery that can be expected for the range of PGE head grades in the mine plan.

Based on the indicated variation in head grade, and similar final residue grades in test work resulting in overall test recovery variations, it was decided to use both the Mintek and SGS data in order to estimate the expected PGE, copper and nickel recoveries.

These steady state recovery estimates are presented in Table 13.37.

Table 13.37 Platreef Recovery Estimate

Description	Estimated Recovery	
	Test Calculated Head Grade PGE (g/t), Cu,Ni (%)	Test Recovery
Copper (Cu)	0.15–0.23	87%–88%
Nickel (Ni)	0.30–0.46	68%–73%
Platinum (Pt)	1.61–1.89	86%–89%
Palladium (Pd)	1.65–2.06	86%–88%
Gold (Au)	0.23–0.25	77%
Rhodium (Rh)	0.11–0.12	80%–87%
PGE (3PE+Au)	3.61–4.31	86%–88%

The steady state recovery estimate is based on the locked cycle test results.

- The steady state recovery copper recovery was estimated to be 87% – 88% with an estimated concentrate grade of > 4%.
- The steady state nickel recovery was estimated to be 68% – 73% with an estimated concentrate grade of > 6%.
- The steady state PGE (3E + Au) recovery was estimated to be 86% – 88% with a concentrate grade of >85 g/t.

13.10 Platreef 2014 PEA Recovery Assumptions

The formulae and constants used to calculate the recoveries in the Platreef 2014 PEA are shown in Table 13.38 to Table 13.40.

Table 13.38 Recovery Constants

Recovery Algorithm Basis	Head Grade Range That Algorithm is Valid	Linear Correlation		
		m	HG	Tail
Cu	0.15g/t – 0.23g/t	9.824	0.230	0.027
Ni	0.30g/t – 0.46g/t	5.754	0.460	0.130
Pt	1.61g/t – 1.89g/t	-50.199	1.890	0.223
Pd	1.65g/t – 2.06g/t	32.921	2.060	0.252
Au	0.23g/t – 0.25g/t	3.790	0.250	0.060
Rh	0.11g/t – 0.12g/t	3.472	0.106	0.012
Mass Pull	4.0%			

Table 13.39 Tailings Grade Calculation

$$\big(HG - Feed\ Grade - (m * Tail)\big)/(-m)$$

Table 13.40 Concentrator Recovery Calculation

$$\frac{\big(100 * Feed\ Grade - 100 * Tailings\ Grade * (1 - Mass\ Pull)\big)}{(100 * Feed\ Grade\ )}$$

13.11 Metallurgical Variability

A full suite of variabilty test work has not been conducted; however, the geometallurgical units chosen and the sample blends tested are expected to be representative of the mineralized zone material in the Platreef deposit. Further testing on footwall samples is required to quantify the impact of footwall dilution on PGE, copper and nickel recovery and grade. In addition to this variability drilling and test work would be required to quantify the variability across the deposit and between each of the geometallurgical units as identified in the mine plan.

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. Test work 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.

13.12 Comments on Section 13

It is the opinion of the qualified person responsible for this section of the report, Mr Michael Valenta, that, given the fact that this is a scoping study, the amount of test work that has been conducted is more than adequate. The test work has confirmed the findings of the mineralogical analysis and the recovery prediction made by the Mintek mineralogy department.

In addition, the results indicate that grinding finer than 80% passing 75 micron would have a benefit in terms of PGE recovery. In some cases finer grinding of MCII and T2L mineralized material may also see a benefit in BMS recovery due to the finer BMS grain size distribution compared to the other two mineralized material types.

A PGE recovery of circa circa 87% can be expected at a saleable grade. Higher recoveries can be achieved however the test work indicates that the production of a saleable grade may force the operation to reduce mass pull thus inhibiting the recovery.

The benefit of increased recovery with finer grind will come with an increase in capital and operating cost and a focus of the pre-feasibility study should be to quantify the benefit and cost of such a decision.

Further work needs to be done to understand the chemistry in the pulp in order to understand the need for high chrome steel balls, a complicated reagent suite and excessive conditioning time. A greater understanding of the behaviour of the major gangue phases needs to be developed in order to understand the mechanism whereby the gangue phases appear to be activated. This has an adverse effect on the concentrate grade and recovery.

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

- Further test work on grind–recovery relationships.
- Flotation kinetics and the identification of fast and slow floating minerals.
- Opportunities to optimise any future flotation circuit configuration.
- Opportunities to optimise an appropriate reagent suite.
- Identifying the mode of occurrence of the recovery losses with the objective of identifying alternative process routes: e.g. separation on size and targeting areas of recovery loss.
- Variability test work.

Based on the test work performed as part of the scoping study, two processing risks have been identified.

13.12.1 Reagent Suite

The Platreef flotation circuit development testing indicated that a reagent suite with oxalic acid and thiourea addition, prior to flotation in a conditioning stage, allowed for improved PGE recovery and grade in final concentrate. The use of oxalic acid and thiourea is not uncommon for Russian PGE operations and this reagent suite has been published in a flotation reagent text book "Handbook of Flotation Reagents". However, the use of this reagent suite has not been employed at any South African Platinum flotation operation.

During the circuit development testing the flowsheet evolved from a simple 3 stage cleaning circuit to a split cleaner circuit configuration with the inclusion of a post mill conditioning stage. The effect of a standard flotation reagent suite similar to that employed by Platinum flotation operations in the vicinity of the Platreef deposit has not been tested on the optimized flowsheet configuration as presented.

Confirmatory reagent suite testing using the optimized flowsheet configuration is required during the Pre-Feasibility Study to finalize the Platreef reagent suite.

13.12.2 Recovery Estimate

The recovery estimate derived for the scoping study as presented in Section 13.9, was based on the results achieved from two locked cycle tests. The mine plan as presented in Figure 13.22 indicates that the mine plan includes geometallurgical units T1, T2U, and T2L as well as fractions of hangingwall and footwall. The samples for the locked cycle tests used to derive the recovery estimate were made up of mineralized material only in the ratio 15% T1, 42.5% T2U, and 42.5% T2L and did not include any footwall or hangingwall. Further testing 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 grade. Further to this detailed variability testing would be required to more accurately quantify the expected recovery and grade thus highlighting what degree of variability could be expected.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

There are four Mineral Resource estimates for the Platreef Project:

- Mineral Resources that are amenable to underground selective mining methods. This consists of material within and adjacent to grade shells in the TCU, and is all below the 650 m elevation. This Mineral Resource has been updated using the revised geological interpretation of Grobler and Nielsen (2012) and incorporation of extensive additional drilling in Zone 1 and some new drilling in Zone 2 and Zone 3. The Mineral Resource amenable to selective underground mining methods is supported by the UMT-TCU model.
- Mineral Resources that are amenable to underground mass mining methods. In the 31 March 2011 Mineral Resource estimate, this included the mineral resource amenable to underground selective mining. The resource model has not been updated, but has been trimmed so as to now be mutually exclusive from the Mineral resource that is amenable to underground selective mining. The Mineral Resources amenable to underground mass mining are below the 650 m elevation. Within the "trimmed" Mineral Resources there has been limited additional drilling. The Mineral Resources amenable to mass underground mining are supported by the UMT-MM model, formerly referred to as the UMT bulk model.
- Mineral Resources that are amenable to open-pit mining. The model has not been updated, as there has been no new drilling. The stated Mineral Resources are unchanged and have an effective date of 31 March 2011. Mineral Resources amenable to open-pit mining are situated above the 650 m elevation.
- Bikkuri area Mineral Resources that are amenable to underground selective mining methods. This consists of material within and adjacent to grade shells in the Bikkuri Reef. This Mineral Resource has been estimated using revised geological interpretation and incorporation of additional drilling in Zone 1 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.

Indicated and Inferred Mineral Resources were estimated for the UMT-TCU area. Recognition of lithological controls (TCU stratigraphy) on grade has enabled declaration of Inferred Mineral Resources at wider drill spacings than would normally be possible. Additional infill drilling in Zone 1 permitted the declaration of Indicated Mineral Resources in that portion of the Project area.

Additional drilling down-dip 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.

The UMT-TCU deposit is the main focus of the Project moving forward. The UMT-TCU Mineral Resource estimate is now considered the base-case resource estimate. The UMT-MM underground deposit is an additional, mutually-exclusive, case.

The computer resource models for the Mineral Resource amenable to open-pit mining methods (AMK and ATS models) were built by AMEC in 2002 and 2003. The Mineral Resource estimate for the UMT-MM model was completed in March 2011. The updated Mineral Resource estimate for the UMT-TCU represents what was previously considered the Mineral Resource amenable to selective mining methods, and was completed in March 2013.

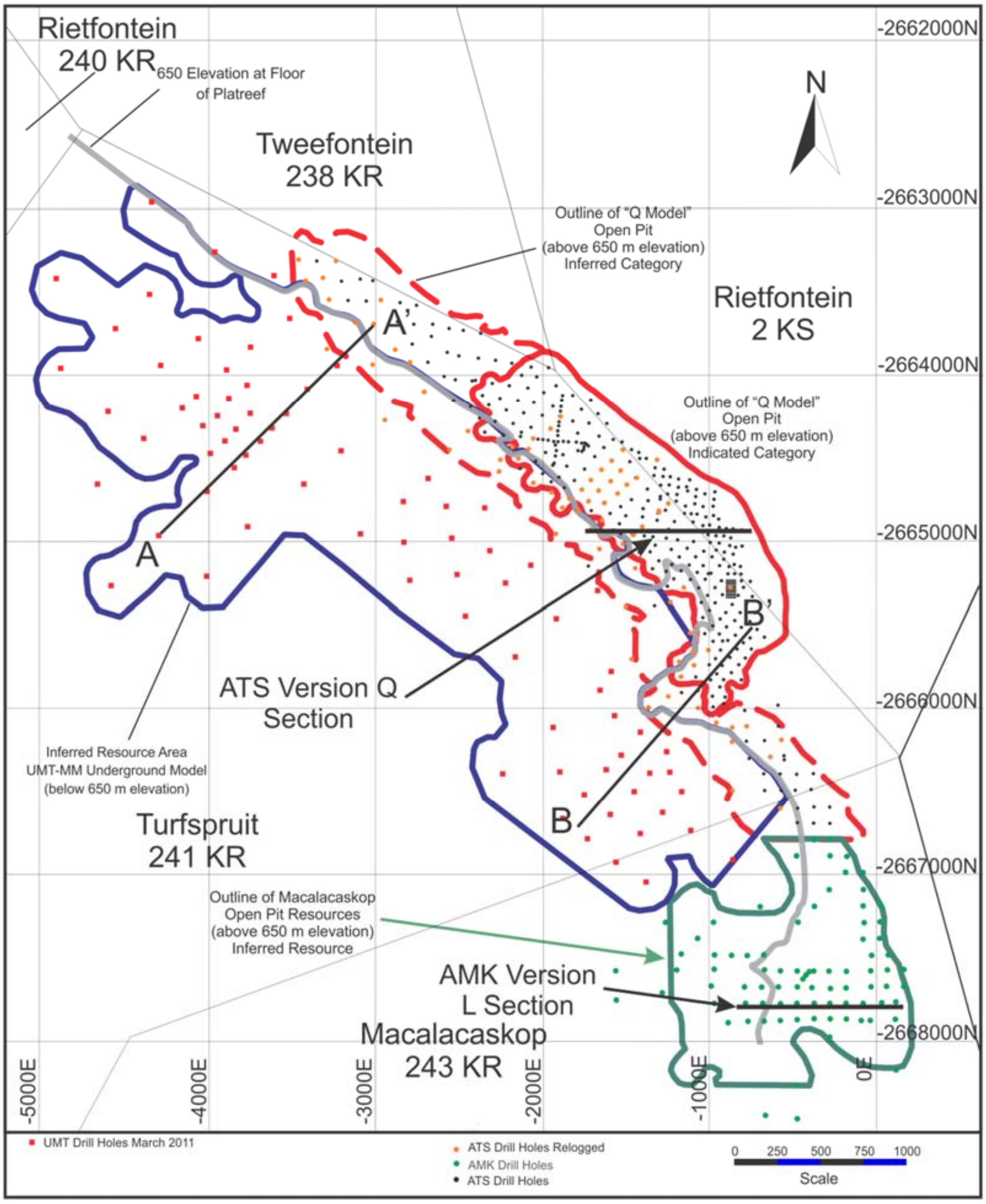
The limits of Platreef Indicated and Inferred Mineral Resource for the UMT-MM area and the Mineral Resources amenable to open-pit mining are shown on Figure 14.1. The Mineral Resource areas that are amenable to open-pit mining are shown in red (ATS) and green (AMK). The limits of the Inferred Mineral Resources for the UMT-MM model are shown in blue. The limits of the UMT-TCU area Mineral Resource estimate are shown in Figure 14.2. The UMT-TCU resource area is formed of the densely-drilled Area 1 (predominantly Zone 1), and the less densely drilled Area 2 (predominantly Zones 2 and 3). The limits of the UMT-BIK area Mineral Resource estimate are also shown in Figure 14.1. The UMT-BIK Mineral Resource is in Zone 1 and is stratigraphically above the UMT-TCU resource model.

14.2 UMT-TCU Resource Model

The UMT-TCU model is located on Turfspruit farm. Mineralization is considered amenable to selective underground mining methods. In the discussion which follows, some tables include information for the Bikkuri model so as to avoid the need for duplicate presentation of model parameters. The Bikkuri reef is interpreted as a slump block of the TCU.

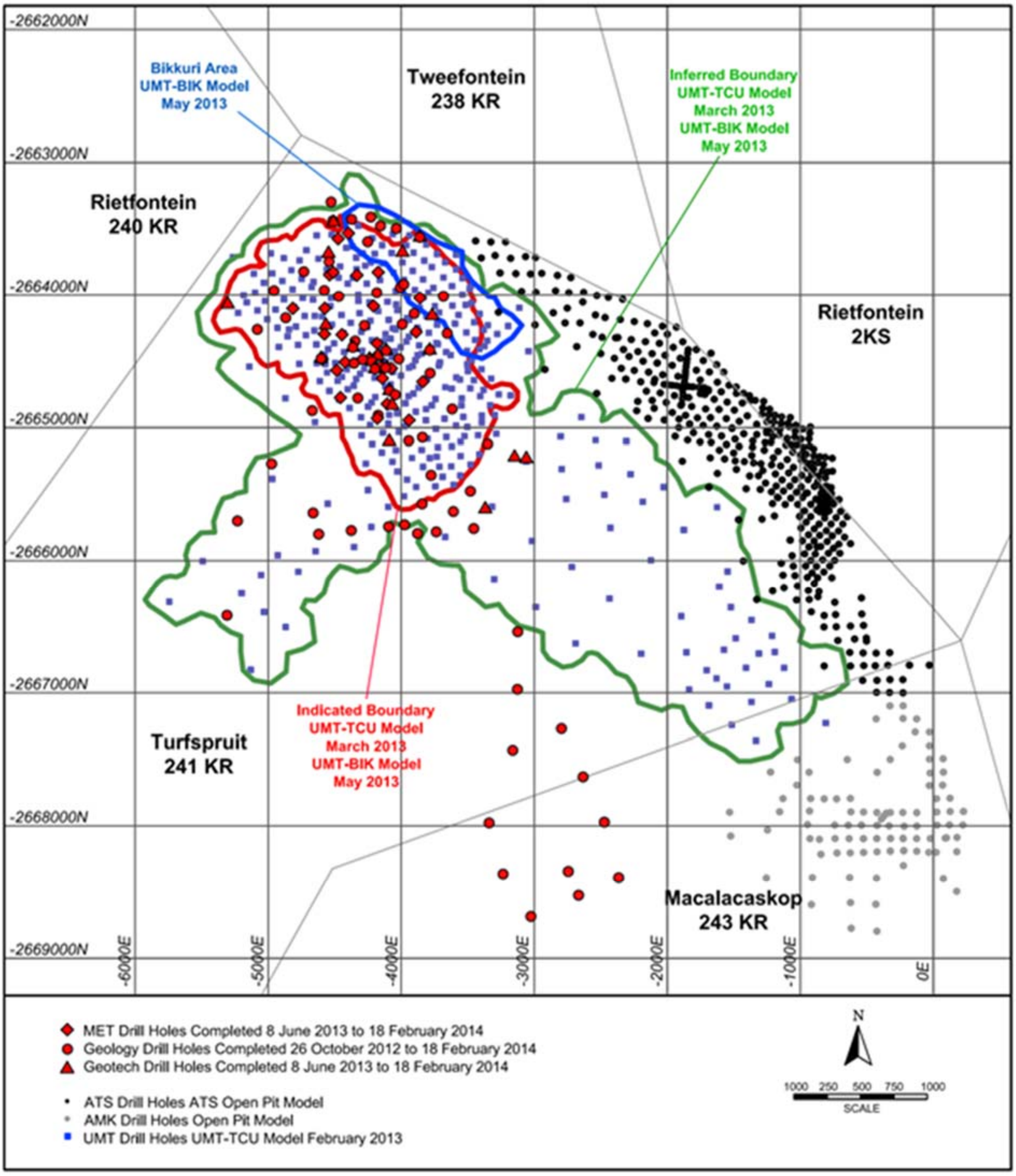
The UMT-TCU resource model update was limited to that portion of the UMT area defined by the stratigraphic sequence referred to as the TCU. The UMT-TCU model in analogous to what was previously (Parker et al., 2012) referred to as the higher-grade selectively-minable model and is now considered to be the base case Mineral Resource estimate for the mineralization amenable to underground mining methods, moving forward. The UMT-TCU model is contained within an envelope defined to include the TCU stratigraphic sequences. The upper surface of the envelope was set at 25 m above the top of the NC1. The lower surface of the envelope was set at 75 m below the bottom of the T2L unit.

Figure 14.1 Mineral Resource Areas for the UMT-MM and Open-Pit



Note: Figure prepared by AMEC, 2013.

Figure 14.2 Mineral Resource Areas for the UMT-TCU and UMT-BIK



Note: Figure prepared by AMEC, 2014.

14.2.1 Drillhole Data

The drilling cut-off date for the UMT model was 26 October 2012. The Platreef database contains 954 core drillholes (excluding re-drilled pilot holes and all deflections). A total of 624,248 m were drilled and completed by 26 October 2012 and this includes 555 holes (194,591 m) from the open-pit programme and 399 holes (429,657 m) from the underground programme (refer to Figure 7.7).

A total of 399 UMT drillholes (429,657 m) were used in the UMT-TCU Mineral Resource model update. All UMT drillholes have been re-logged for consideration of the TCU. The UMT-TCU model includes 34 drillholes (38,537 m) from the open-pit area that have been re-logged for the TCU geology.

14.2.2 Geological Model

Geological interpretations for the UMT-TCU area were developed by Ivanhoe personnel. The re-logging work summarized in the lithological variable STRAT (Table 14.1) is the basis for the geological model. A numeric model code (MCODE) was assigned to each lithology.

Table 14.1 Model Package Description

Model	ModPak (UMT-MM)	UNIT (UMT-MM)	ZCODE (UMT-MM)	STRAT (UMT-BIK, (UMT-TCU)	MCODE (UMT-BIK, (UMT-TCU)
Bikkuri	MZ	MZ	10	NCBK	50
				MANBK	51
				B1	52
				B2	55
				FWBK	57
TCU (Top Loaded Zone)	DZ	UDZ	22	NC1	20
				MAN	21
	BP	UBP	23	T1	22
				T2U	25
	HA	UHABP	26	T2L	26
	PXHA	UPXHA	24	FW	27
	CZ	UCZ	27	FW	27
	HF	HF	28	FW	27
	HA	UHA	29	FW	27
	DZ	LDZ	32	FW	27
Platreef: Lower Unit (Bottom Loaded Zone)	BP	LBP	33	FW	27
	PXHA	LPXHA	34	FW	27
	LPX	LLPX	35	FW	27
	HABP	LHABP	36	FW	27
	CZ	LCZ	37	FW	27
	HA	LHA	39	FW	27
Floor (not estimated)	FL	FL	40	FW	27

Note: The current interpretation is that the Bikkuri Reef is likely a slump block of the TCu.

Two-dimensional gridded-seam models were completed for five stratigraphic horizons used in the construction of the geological model (NC1, MAN, T1, T2U, and T2L). Wireframe surfaces were constructed from the gridded seam models of the NC1, MAN, T1, T2U, and T2L.

14.2.3 High-Grade Shells – UMT-TCU

Ivanhoe personnel identified nested 2PE + Au (Pt + Pd + Au) grade shells using a minimum of 3 m of 1 g/t 2PE + Au, 2 g/t 2PE + Au, and 3 g/t 2PE + Au. 2PE + Au grade shells were used rather than 3PE+Au because rhodium assaying was incomplete at the time the shells were constructed. The grade shells were constructed as a tool for constraining grade estimates.

The nested grade shells were identified in two mineralized zones (T1MZ and T2MZ). The T1MZ grade shells are associated with the T1 stratigraphic unit. The T2MZ grade shells are associated with the T2 stratigraphic units (T2U and T2L). Two-dimensional gridded-seam models were completed for the T1MZ and T2MZ grade shells. Wireframe surfaces were constructed from the gridded seam models of the T1MZ and T2MZ seam models. The T1MZ and T2MZ are defined by their 2PE + Au assays and are not restricted to specific stratigraphic horizons within the T1 and T2 stratigraphic units respectively.

Grade shell codes (GCODES) were used to identify blocks within and outside the grade shells. The GCODES are summarized in Table 14.2.

Table 14.2 Summary of GCODE for TCU and Bikurri (All Elements)

Model	Grade Shell	GCODE
TCU-BIK	NCBK	0
	MANBK	0
	B1MZ 1g	301
	B1MZ 2g	302
	B1MZ 3g	303
	B1	3
	B2MZ 1g	401
	B2MZ 2g	402
	B2MZ 3g	403
	B2	4
	FWBK	0
UMT-TCU	T1MZ 1g 2PE + Au	101
	T1MZ 2g 2PE + Au	102
	T1MZ 3g 2PE + Au	103
	NC1	0
	MAN	0
	T1	1
	T2MZ 1g 2PE + Au	201
	T2MZ 2g 2PE + Au	202
	T2MZ 3g 2PE + Au	203
	T2	2
	FW	0

Note: The current interpretation is that the Bikkuri Reef is likely a slump block of the TCU. The T1MZ and T2MZ are not restricted to specific stratigraphic horizons within the T1 and T2 stratigraphic units.

14.2.4 Mineralization Adjacent to the TCU Mineralized Zones

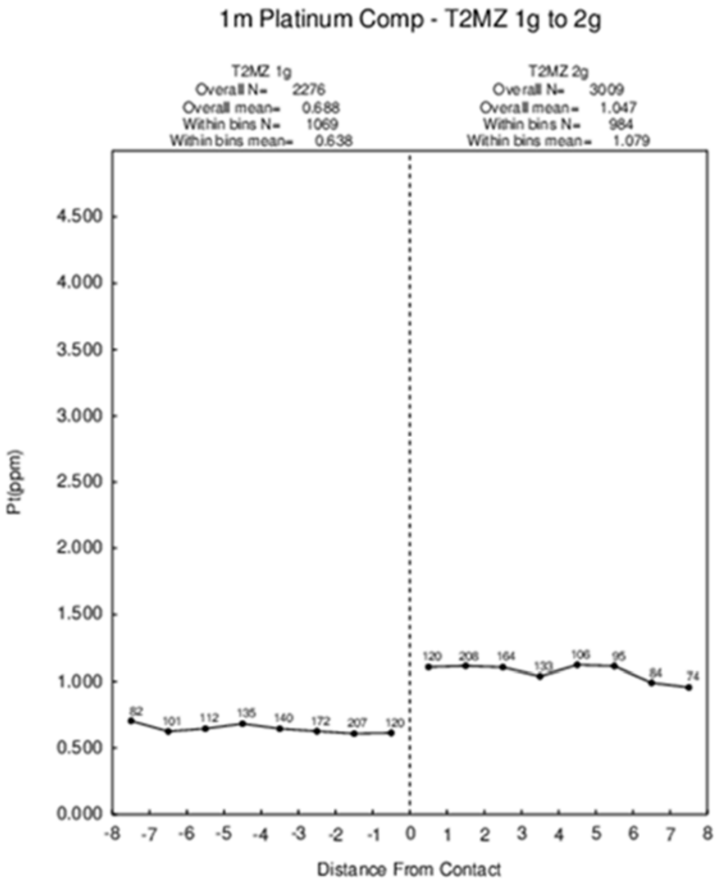
There is scattered mineralization adjacent to the TCU mineralized zones that is locally continuous. Ivanhoe plans to use floating stope software in scoping studies. Mineralization adjacent to the TCU mineralized zones may be included in the resultant stopes; hence there is a need to estimate grades in blocks in an envelope around the TCU mineralized zones.

14.2.5 Compositing and Exploratory Data Analysis (EDA) for UMT-TCU Model

The resource model update occurs within the stratigraphic sequence referred to as the Turfspruit Cyclic Unit discussed in Section 7.3.2. The drillhole database was composited to 1 m length composites within the UMT-TCU model envelope. The compositing was controlled by the nested 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 14.3 displays the contact profile between the T2MZ 1 g/t 2PE + Au, and T2MZ 2 g/t 2PE + Au shells.

Figure 14.3 Contact Profile for Platinum Between 1 g/t and 2 g/t 2PE + Au Shells

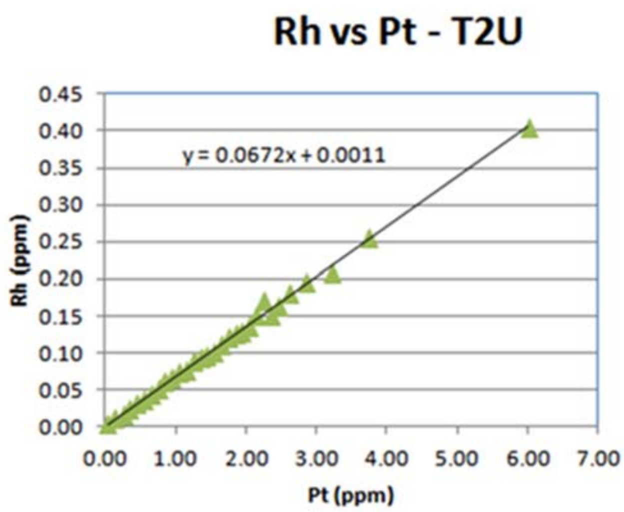


Note: Figure prepared by AMEC, 2013.

Rhodium analyses are only partially complete on the assay database, and rhodium to platinum regressions were constructed to estimate the rhodium content for samples missing rhodium analysis. The Rh and Pt values were binned using a series of platinum thresholds, and the regression equation was calculated using the bin averages.

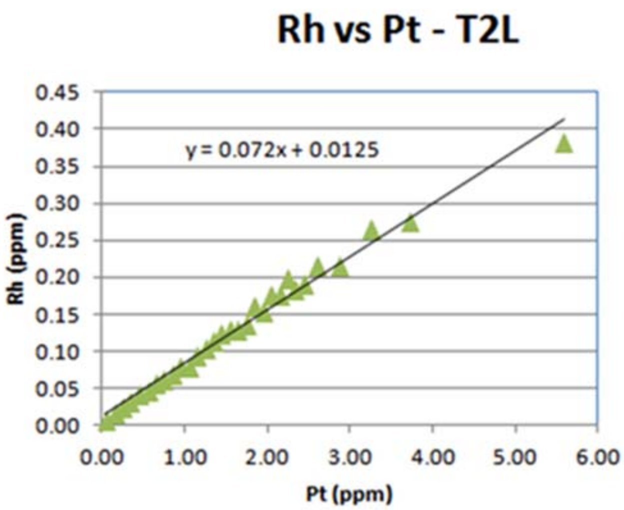
Figure 14.4 and Figure 14.5 show rhodium as a function of platinum regression for the T2U and T2L respectively. Table 14.3 summarizes the proportions of assays with rhodium analysis within the grade shells and by stratigraphic unit. The proportion of rhodium assays exceeds 50% within the 2 g/t 2PE + Au shell and exceeds 60% in the in T2U and T2L.

Figure 14.4 Rhodium Regression for the T2U



Note: Figure prepared by AMEC, 2013: points represent estimated average Rh grades within Pt bins.

Figure 14.5 Rhodium Regression for the T2L



Note: Figure prepared by AMEC, 2013: points represent estimated average Rh grades within Pt bins.

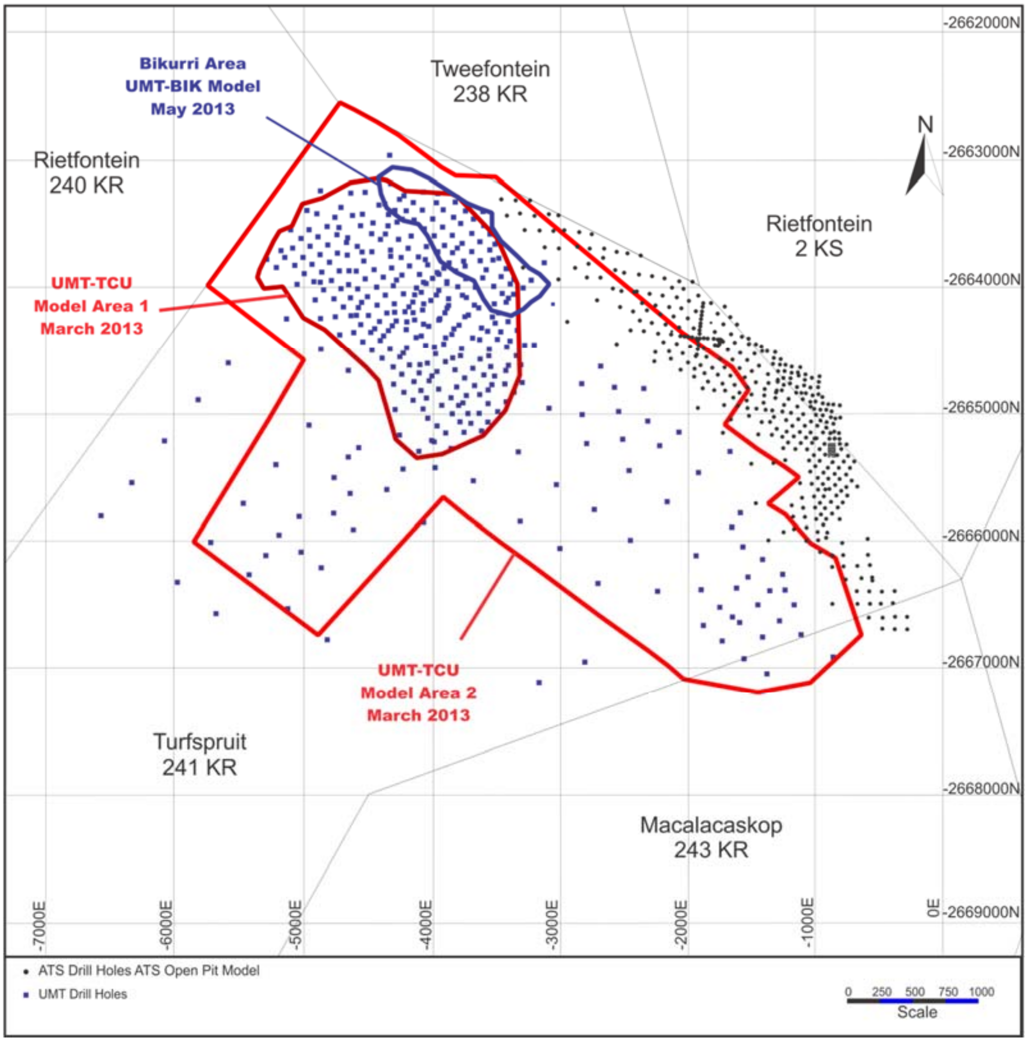
Table 14.3 Proportions of Rhodium Assays by Strat Code and Grade Shell

STRAT	Assays	Rh Analysis	%
HW	9,052	1,091	12.1
NC1	6,470	236	3.6
MAN	825	108	13.1
T1	13,526	5,124	37.9
T2U	4,098	2,473	60.3
T2L	4,167	2,506	60.1
FW	72,080	4,871	6.8
FW	110,218	16,409	14.9
Grade Shell	**Assays**	**Rh Analysis**	**%**
T1MZ 1 g/t	1,024	401	39.2
T1MZ 2 g/t	613	351	57.3
T1MZ 3 g/t	923	564	61.1
T2MZ 1 g/t	2,577	967	37.5
T2MZ 2 g/t	3,349	1,791	53.5
T2MZ 3g/t	5,088	3,348	65.8

14.2.6 Block Model and Grade Estimation

The UMT-TCU block model was constructed over the area of UMT drilling (Figure 14.6). Blocks were oriented parallel to the national coordinate system. The block model used a parent block size of 20 m x 20 m x 1 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 regularized to 10 m x 10 m x 2 m.

Figure 14.6 Extents of the UMT-TCU and TCU-BIK Resource Model Areas



Note: Figure prepared by AMEC, 2013.

Block Grade Estimation

Inverse distance to the third power (ID3) interpolation was used to estimate Platreef grades into blocks in the UMT-TCU model. Kriging was used in Estimation Area 1 within the T1MZ and T2MZ. Variograms (using the correlogram method) were completed by grade shell and combined shells. Figure 14.7 and Figure 14.8 are examples of platinum downhole and directional variograms.

To eliminate the effects of the 24 structural domains discussed in Section 7.3, the estimation was completed by hanging the T1MZ and T2MZ at the 1,000 m elevation. Nearest-neighbour (NN) models representing declustered composite distributions were generated for validation checks. This allows the effect of discontinuities at fault boundaries to be removed.

Figure 14.7 Downhole Correlogram Model for Platinum

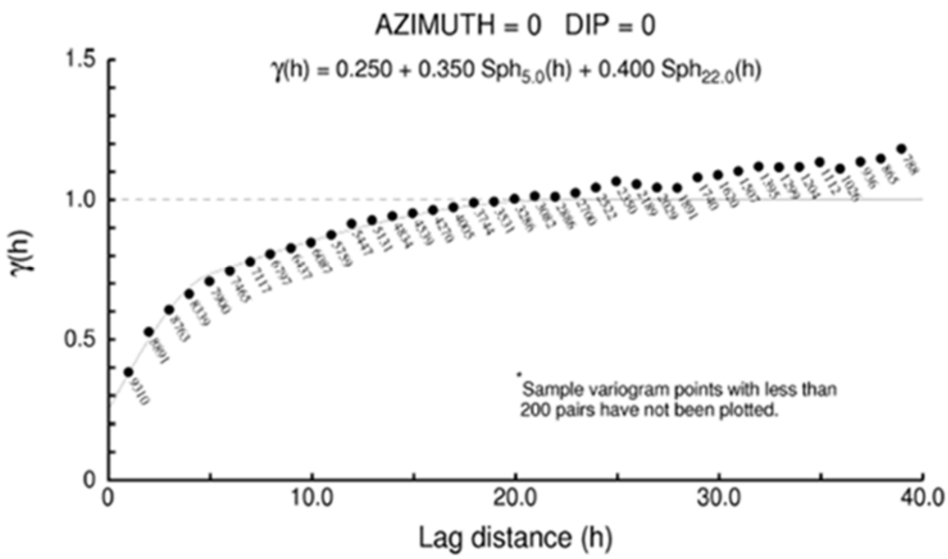
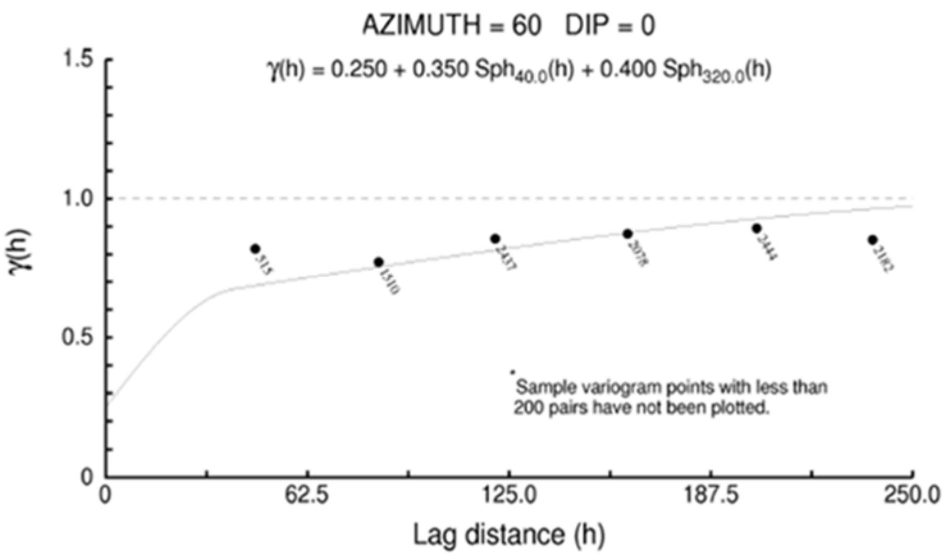


Figure 14.8 Directional Correlogram Model for Platinumat Azimuth 60



Grade Estimation

T1MZ

Grade estimation in the T1MZ included block and composite matching by GCODE (see Table 14.2). Estimation was completed by kriging and inverse distance to the third power (ID3) in Estimation Area 1 and by ID3 in Estimation Area 2 (Zones 2 and 3). A NN estimate was completed for model validation. After grade estimation, all blocks and composites were back-transformed back to the original elevation.

T2MZ

Grade estimation in the T2MZ included block and composite matching by GCODE and MCODE (see Table 14.1 and Table 14.2) matching in Pass 1. In Pass 2, matching was only by GCODE. Estimation was completed by kriging and ID3 in Estimation Area 1 and by ID3 in Estimation Area 2. A NN estimate was completed for model validation. After grade estimation, all blocks and composites were transformed back to the original elevation.

T1 and NC1

Grade estimation in the T1, NC1, and MAN stratigraphic units not located within the nested grade shells were estimated by matching blocks and composites by MCODE. These blocks were estimated after transformation with respect to the the T1MZ virtual elevation of 1,000 m. Estimation was completed by ID3. A NN estimate was completed for model validation.

T2 and FW

Grade estimation in the T2 and FW stratigraphic units and not within the nested grade shells were estimated by matching blocks and composites by MCODE. These blocks were estimated after transformation with respect to the T2MZ virtual elevation of 1,000 m. Estimation was completed by ID3. A NN estimate was completed for model validation.

Search Volumes

Estimations were completed in Datamine for Pass 1, Pass 2 and Pass 3 using expanding search volumes. Search volumes are summarized in Table 14.4.

Table 14.4 Search Strategy for Grade Estimation (All Elements)

Search Volume	Search Distances			Min	Max	Max/
	X	Y	Z	Samples	Samples	DH
1	250	250	50	4	15	3
2	500	500	100	4	15	3
3	1500	1500	300	1	15	3

Outlier Restriction

No grade capping was implemented within the nested grade shells. An outlier restriction was applied in the host rocks outside the grade shells. The outlier thresholds were selected from probability plots of 1 m composites and are summarized in Table 14.5. All boundaries were considered hard. Composites with grades above the grade threshold and with distances from composite to block center beyond the distance threshsolds were not used in estimation.

Table 14.5 Outlier Restriction Thresholds

Zone	Ni(%)	Cu (%)	Pt (%)	Pd (g/t)	Au (g/t)	Rh (g/t)
Grade	0.25	0.25	1.00	1.50	0.25	0.10
Distance	150 m	150 m	150 m	150 m	150 m	150 m

Blocks not estimated were given the mean grade of the Stratigraphic unit. The mean grades used are summarized in Table 14.6. 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.

Table 14.6 Mean Grades to Fill Blocks Not Estimated

Zone	GCODE	Cu (%)	Ni (%)	Pt (ppm)	Pd (ppm)	Au (ppm)	Rh (ppm)
	T1MZ 1g	0.121	0.224	0.616	0.511	0.211	0.029
Area 1	T1MZ 2g	0.135	0.255	1.413	0.949	0.284	0.560
	T1MZ 3g	0.186	0.358	2.432	2.026	0.523	0.119
	T1MZ 1g	0.124	0.228	0.609	0.561	0.177	0.032
Area 2	T1MZ 2g	0.133	0.264	1.193	0.988	0.279	0.063
	T1MZ 3g	0.183	0.358	2.284	1.909	0.454	0.108
Area 1	T1	0.040	0.103	0.199	0.159	0.073	0.011
Area 2	T1	0.037	0.095	0.163	0.120	0.061	0.009
	T2MZ 1g	0.113	0.238	0.650	0.750	0.119	0.047
Area 1	T2MZ 2g	0.137	0.283	1.046	1.184	0.187	0.074
	T2MZ 3g	0.188	0.386	2.460	2.532	0.333	0.172
	T2MZ 1g	0.114	0.232	0.741	0.759	0.125	0.053
Area 2	T2MZ 2g	0.142	0.288	1.022	1.133	0.185	0.073
	T2MZ 3g	0.191	0.396	2.203	2.210	0.338	0.156
Area 1	T2	0.079	0.175	0.393	0.456	0.076	0.033
Area 2	T2	0.074	0.149	0.291	0.338	0.064	0.027
All	NC1	0.025	0.055	0.065	0.069	0.023	0.008
All	MAN	0.007	0.016	0.032	0.026	0.007	0.002
All	FW	0.076	0.132	0.291	0.364	0.059	0.024

Regularization

Upon completion of the estimation, the UMT-TCU block model was regularized from the 20 m x 20 m x 1 m (sub-celled to 10 m x 10 m x 0.5 m) to a 10 m x 10 m x 2 m (no sub-cells) model. 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.

14.2.7 Bulk Density

Densities were also coded to the blocks by stratigraphic unit using the mean density values for each stratigraphic unit (Table 14.7).

Table 14.7 Bulk Density Values

Zone	Mean Density	CV	Maximum SG	Minimum SG
HW	2.91	0.04	4.47	2.04
NC1	2.98	0.05	4.28	2.09
MAN	2.84	0.03	3.18	2.42
T1	3.19	0.04	4.33	2.58
T2U	3.19	0.04	3.80	2.30
T2L	3.04	0.05	4.34	2.33
FW	3.10	0.06	4.45	2.05

14.2.8 Mineral Resource Classification

Mineral Resources have been classified using the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2010):

> “A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth’s crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.”

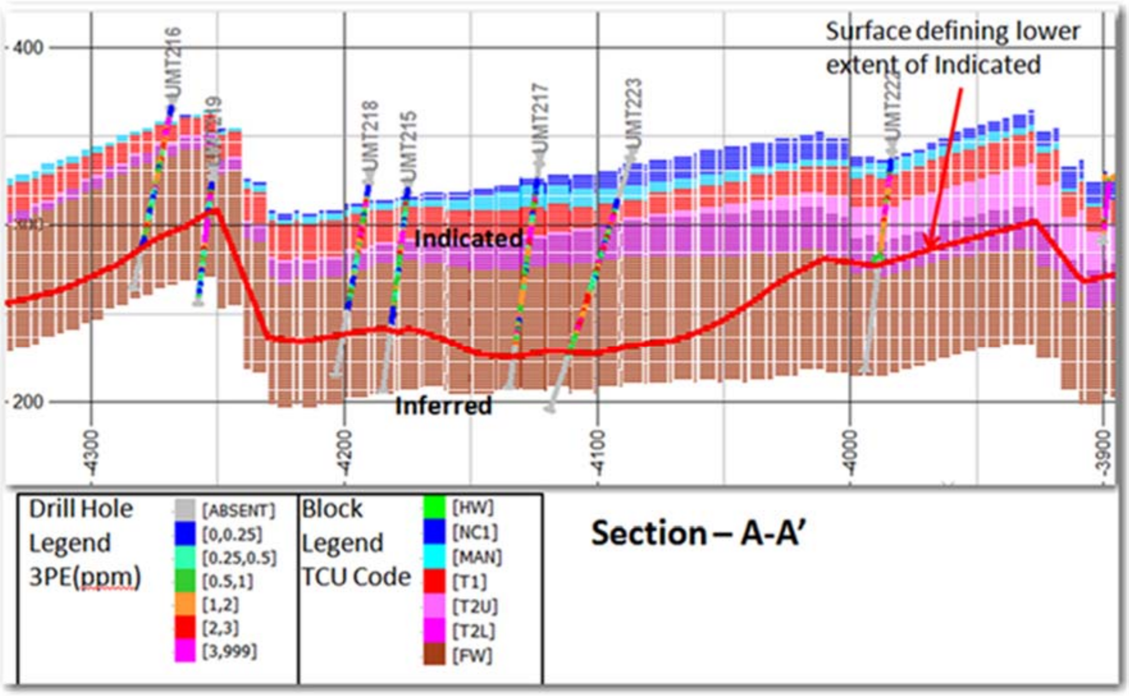
> “An ‘Inferred Mineral Resource’ is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings, and drillholes.”

> “An ‘Indicated Mineral Resource’ is that part of the Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.”

UMT-TCU Model Classification

Inferred Mineral Resources are declared where the drillhole spacing is 400 m to 800 m (predominately Area 2). The Inferred Mineral Resources are permitted at a wider drillhole spacing than would normally apply because of the well defined geology of the TCU. In Area 1, much of the FW stratigraphic unit is classified Inferred Mineral Resources because many of the tails of the drillholes were not sampled due to the focus on the TCU. It is expected that once these drillholes are sampled, a higher conficence category may be able to be assigned to the estimated Mineral Resources. Figure 14.9 displays the regions of Indicated and Inferred Mineral Resources on a typical cross-section.

Figure 14.9 Surface Defining Lower Extent of Indicated Mineral Resources (looking north-west)



Note: Figure prepared by AMEC, 2013. Coordinates shown are WGS system.

14.2.9 UMT-TCU Model Validation

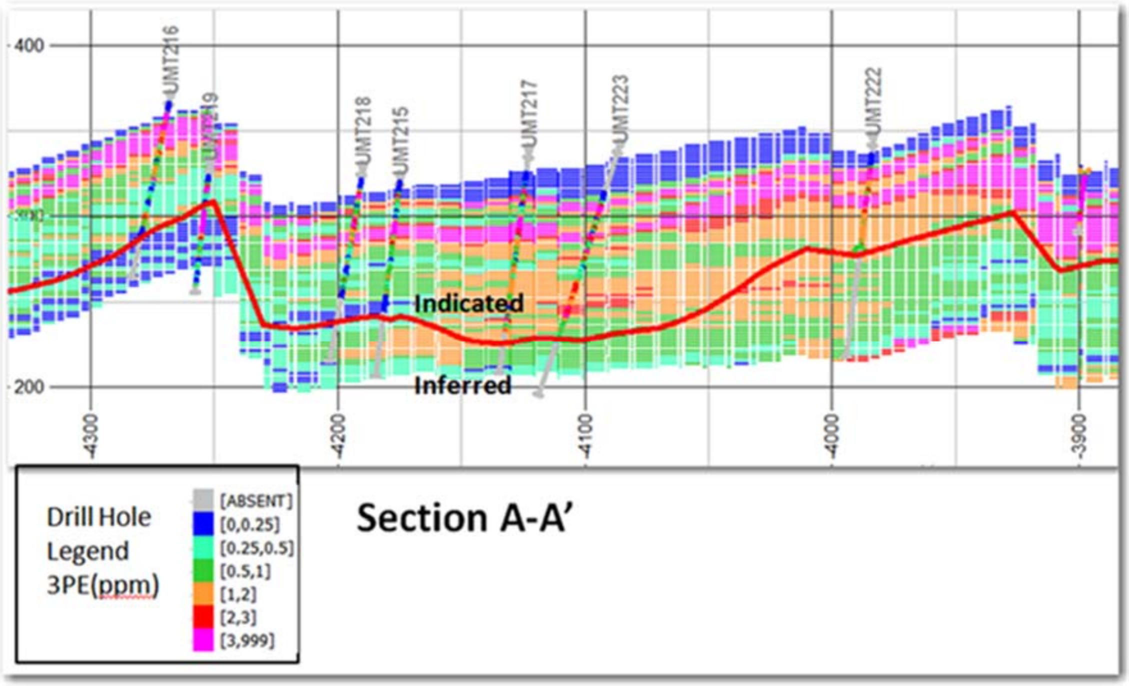
Model validation included visual inspection of block grades relative to composite grades on cross-sections and level plans. Statistical comparisons consisting of box plots and grade profiles tabulated in different directions (swaths) for each metal by stratigraphic unit and grade shell were constructed to compare the Kriged (where present), ID3 grade estimates, NN estimates, and 1 m composites.

Visual Validation and Box Plots

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 2PE + Au and Ni are presented in Figure 14.10 and Figure 14.11 respectively (section lines A-A').

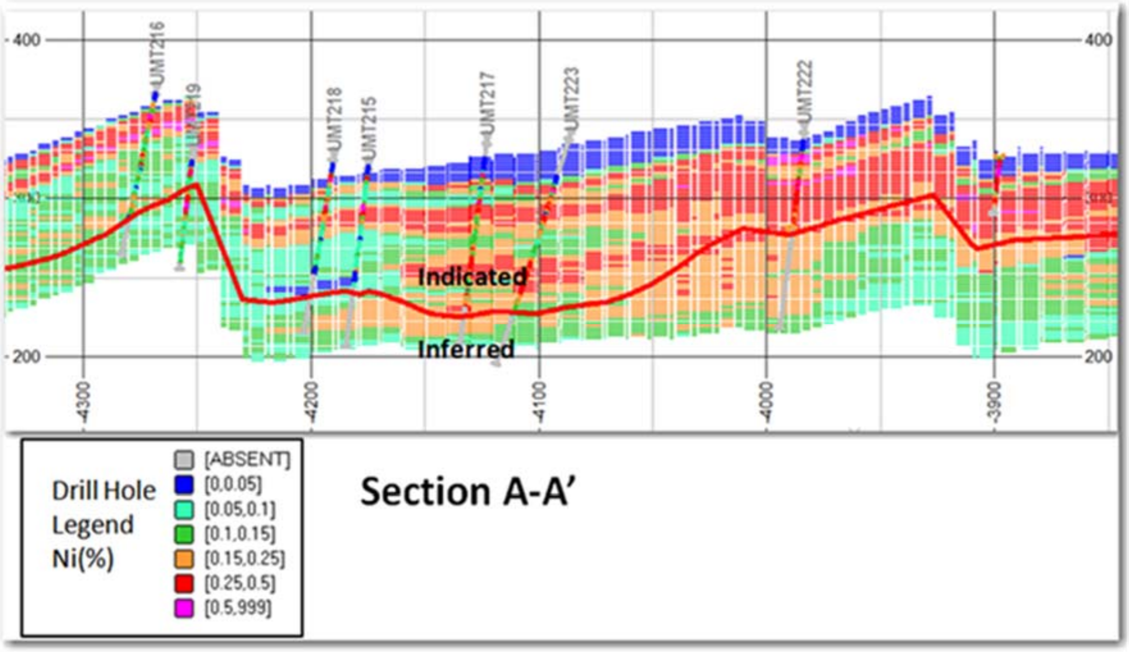
The global means and grade distributions for each metal from the ID3 model, NN model and 1 m composites checked within reasonable levels, suggesting the ID3 model is globally un-biased.

Figure 14.10 Section AA' Displaying 2PE + Au Block and Composite Grades (looking north-west)



Note: In the legend to this figure, 3PE = Pt + Pd + Au.

Figure 14.11 Section AA' Displaying Ni Block and Composite Grades (looking north-west)



Note: Figures prepared by AMEC, 2013.

Swath Plots

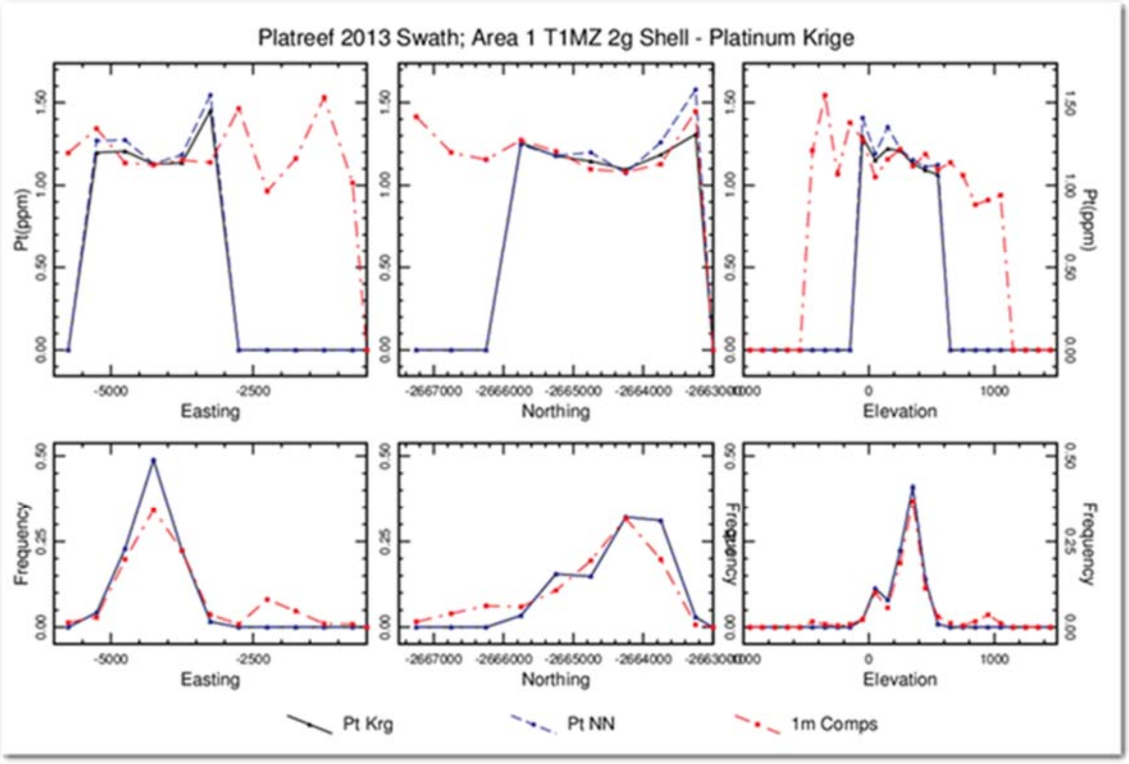
Swath plots (width of 200 m) of the ID3 model, NN model and 1 m composites were completed for Cu, Ni, Pt, Pd, Au, and Rh.

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

AMEC commonly focuses swath plot analysis on blocks classified as Measured and Indicated. Model validation was completed for Estimation Areas 1 (Indicated) and 2 (Inferred) and no local biases were observed.

The platinum swath plot for the T1MZ 2 g/t 2PE + Au shell is presented in Figure 14.12.

Figure 14.12 Platinum Swath Plot for T1MZ – 2 g/t 2PE + Au Shell



Note: Figure prepared by AMEC, 2013.

Comments on the UMT-TCU Model

As currently configured the UMT-TCU model covers the TCU Stratigraphic units and includes what was formerly referred to as the selectively-mineable model. The UMT-TCU model also includes estimation of grades in blocks adjacent to the TCU, up to 25 m on the hangingwall – effectively to the barren Main Zone gabbro norite, and up to 75 m into the FW – and usually stops short of the Floor of the Platreef.

Additional drilling is required in the area classified as Inferred (Area 2) to better define the stratigraphic units and the fault domains. Additional geological work is required to decipher the stratigraphy in the FW unit in both Areas 1 and 2. Upon the completion of the geological interpretation in the FW of the UMT-TCU model, an update of the UMT-Mass Model (UMT-MM Model) can be completed. There will then be one model for the UMT area.

Local bias is expected in Area 2 because of the wide-spaced drilling and large search distances required for grade estimation. Additional drilling should permit better grade estimations in Area 2.

Future models should consider a 10 m x 10 m x 2 m parent cell block size.

14.3 UMT-MM Model

Inferred Mineral Resources were estimated for UMT-MM Model in March 2011 (Parker et al., 2012). Recognition of lithological controls (referred to as the ModPak) on grade has enabled declaration of Inferred Mineral Resources at wider drill spacings than would normally be possible. The UMT-MM Model is partially included in the update of the UMT-TCU model. That portion of the UMT-Bulk Model included in UMT-TCU Model has been identified, and this portion of the UMT-Bulk resources has been removed from the resource tabulation. The limits of the UMT-Bulk Inferred Mineral Resource are shown on Figure 14.1 and generally are located beneath the UMT-TCU Mineral Resource estimate (see Figure 14.17).

14.3.1 Geological Model

Geological interpretations for the UMT area were developed by Ivanhoe and AMEC personnel. The re-logging work summarized in the lithological variable Modpak (refer to Table 14.1) is the basis for the UMT-MM geological model. A zone code (Zcode) was assigned to each lithology. Table 14.1 summarizes the correlation between the stratigraphic designations between the UMT-MM and UMT-TCU models.

Wireframe surfaces were constructed for the bottom of MZ (Main Zone-undisturbed), UDZ (Upper Disturbed Zone), UBP (Upper B-Pyroxenite), UPXHA (Upper Pyroxenite-Harzburgite)/Top of Lower Unit, LDZ (Lower Disturbed Zone), LBP (Lower B-Pyroxenite), and LPXHA (Lower Pyroxenite-Harzburgite). A wireframe model of the top of Floor was also built.

Wireframe solids were constructed for discontinuous and/or discordant Modpak units including BPHA (Harzburgite associated with bottom of B Pyroxenite), HA (Harzburgite not associated with B-Pyroxenite), CZ (Contaminated Zone, calc-silicates) and HF (Hornfels).

14.3.2 High-Grade Shells

A review of the drillhole assays relative to the geological interpretation and preliminary statistics indicated the need to use additional controls besides lithology for block-grade estimation. For this stage of the Project, and because drilling is widely spaced, grade shells were chosen as the appropriate tool for constraining grade estimates.

High-Grade Shell – UMT-MM Model

A grade shell (built as a wireframe model) based on a 1 g/t 2PE + Au cut-off applied to 5 m composites (termed 5m1g shell) was built. This grade shell was used to tag blocks falling within the shell relative to the block centroid. Blocks within the 5m1g shell were coded as high-grade (HG), and blocks outside the shell were coded as low-grade (LG). The grade shell is typically in close association with the B Pyroxenite and Harzburgite contact as defined by the ModPak geological interpretation.

14.3.3 Exploratory Data Analysis and Grade Estimation Domains

Compositing and EDA for UMT-MM Model

Five-metre length composites, controlled by the geological surfaces and wireframes were constructed. Composites that were less than 2.5 m in length were stitched onto the previous up-hole composite. Composites were tagged as either "HG" for inside the 5m1g shell or "LG" for outside of the shell.

Exploratory data analysis (EDA) was completed on 5 m composites. Inspection of preliminary sections and plans showed a propensity for unreasonable overextension of high Ni, Cu and PGE grades in an uncapped model. This is typical for models supported by wide-spaced data, and AMEC views its capping and outlier restriction as appropriate to control unreasonable overextension of high PGE grades.

Histograms and probability plots for each metal were used to determine capping level thresholds (Table 14.8).

Table 14.8 Composite Capping Levels for UMT-MM Model

Zone	Zone Code	Ni (%)	Cu (%)	Pt (g/t)	Pd (g/t)	Au (g/t)
MZ	10	NE	NE	NE	NE	NE
UDZ	22	0.35	0.22	No Cap	1.20	0.35
UBP	23	0.70	0.32	4.00	3.00	0.65
UPXHA	24	1.00	0.60	2.50	2.50	0.50
UHABP	26	No Cap	0.35	2.50	3.00	0.50
UCZ	27	0.40	0.25	1.50	1.80	0.40
HF	28	0.28	0.40	0.90	1.10	No Cap
LDZ	32	0.40	0.30	0.60	No Cap	No Cap
LBP	33	0.45	0.30	No Cap	1.50	No Cap
LPXHA	34	0.90	0.70	3.50	2.00	0.50
LLPX	35	0.70	0.40	No Cap	1.50	0.40
LHABP	36	No Cap	0.35	No Cap	2.20	0.40
LCZ	37	0.38	0.28	No Cap	No Cap	0.28
LHA	39	No Cap	No Cap	No Cap	No Cap	No Cap
FL	40	NE	NE	NE	NE	NE

Note: NE = Not Estimated.

Outlier restriction grade thresholds and distances (Table 14.9) were determined from a review of cross-sections (after a preliminary grade estimation run). The capping levels and outlier restrictions shown apply to both the LG and HG composites. The estimation of the LG zone matched LG blocks and LG composites using zcode.

The estimation of the HG zone matched HG blocks and HG composites, but was not constrained on zcode domains and sample sharing was between all rock types.

Table 14.9 Outlier Restriction Thresholds for UMT-MM Model

Metal	Distance Threshold	Grade Threshold
Ni	150 m	0.40 %
Cu	150 m	0.30 %
Pt	100 m	1.0 g/t
Pd	100 m	1.0 g/t
Au	100 m	0.30 g/t

Note: Composite grade is capped beyond distance threshold.

The capping and outlier restriction resulted in reduction in the estimated metal in the capped/outlier-restricted model of 2% of the Ni and Cu, 12% of the Pt and Pd and 3% of the Au compared to an uncapped inverse distance model.

Box plots for each metal were used to assess mineralization ranges and domain associations. These plots show higher mean grades for Harzburgite than for any other lithology. Contact profiles were used to determine soft/hard contacts and composite sharing between the Zcode domains for grade estimation. Statistics were reviewed to determine sample sharing for Pt, Pd, and Au in the LG zone (Table 14.10) and HG zone (Table 14.11).

Table 14.10 Grade Estimation Composite Sharing for LG Zone – UMT – MM Model

Zone (Block)	Zcode (Block)	Ni Sharing	Cu Sharing	Pt (LG) Sharing	Pd (LG) Sharing	Au (LG) Sharing
UDZ	22		23	–	–	–
UBP	23	24	22, 24	22, 24, 26	24	26
UPXHA	24	23	23, 32	23, 32	23, 32	32
UHABP	26	–	–	23	–	23
UCZ	27	28	23, 24	–	–	28
HF	28	27	–	–	–	27
LDZ	32	–	24, 33	–	24, 33	24
LBP	33	–	32	32	32	–
LPXHA	34	35, 37	35	24, 35, 37	35, 37	36, 37
LLPX	35	34	34	34	34	34
LHABP	36	–	–	–	–	–
LCZ	37	34	–	34	34	34

Note: (1) Zcode = Zone Code. See Table 14.1.

Table 14.11 Grade Estimation Composite Sharing for HG Zone – UMT – MM Model

Zone (Block)	Zcode (Block)	Ni Sharing	Cu Sharing	Pt (HG) Sharing	Pd (HG) Sharing	Au (HG) Sharing
UDZ	22	–	–	23, 26	23, 26	–
UBP	23	–	–	22, 26	22, 26	26
UPXHA	24	–	–	27	27	32
UHABP	26	–	–	22, 23	22, 23	23
UCZ	27	–	–	24	24	28
HF	28	–	–	–	–	–
LDZ	32	–	–	33, 34, 35, 36, 37	33, 34, 35	33
LBP	33	–	–	33, 34, 35, 36, 37	32, 34, 35	32
LPXHA	34	–	–	33, 34, 35, 36, 37	33, 35, 37	36, 37
LLPX	35	–	–	33, 34, 35, 36, 37	32, 33, 34	34
LHABP	36	–	–	33, 34, 35, 36, 37	37	34, 37
LCZ	37	–	–	33, 34, 35, 36, 37	36	34, 36

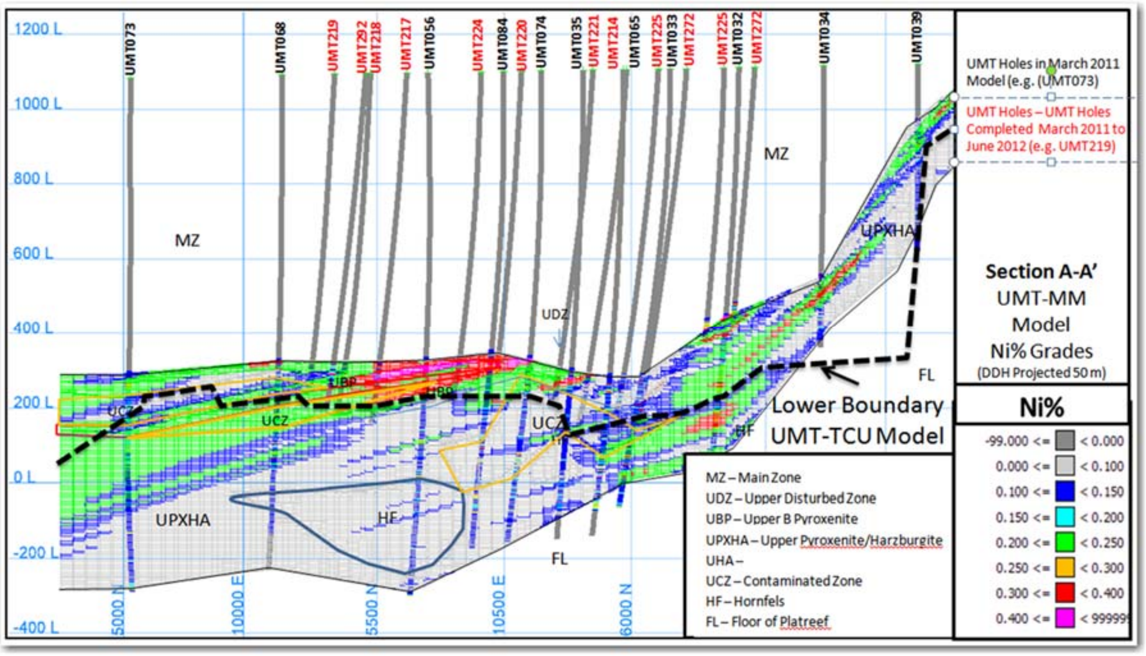
Note: (1) Zcode = Zone Code. See Table 14.1.

14.3.4 UMT-MM Block Model and Grade Estimation

The UMT-MM block model was constructed over the area of UMT drilling (refer to Figure 14.1). The blocks were oriented parallel to the mine grid coordinates. The overall model parent block size was 50 m x 50 m x 50 m, with the parent block size within the Platreef for grade estimation being 25 m x 25 m x 5 m. The geological surfaces and wireframes, and grade zones (HG for bulk model and nested shells for selective model) were coded to the blocks, allowing sub-celling along the contacts to a minimum size of 5 m x 5 m x 1 m. Densities were also coded to the blocks using the average values by zone code (refer to Table 11.3)

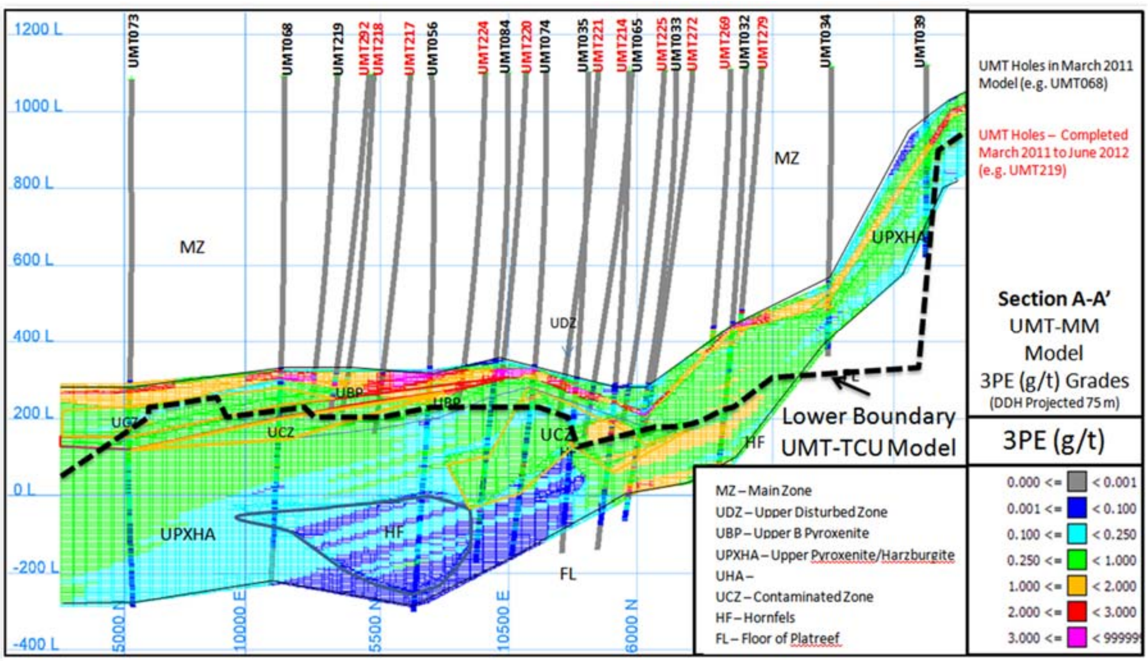
Figure 14.13 to Figure 14.16 show representative sections through the UMT-MM Model. The sections show nickel and 2PE + Au grades in drillholes and blocks. The dashed line shows the lower boundary of the UMT-TCU resource model discussed above.

Figure 14.13 UMT-MM Model – Cross-Section A-A' (looking north-west) Showing Ni%



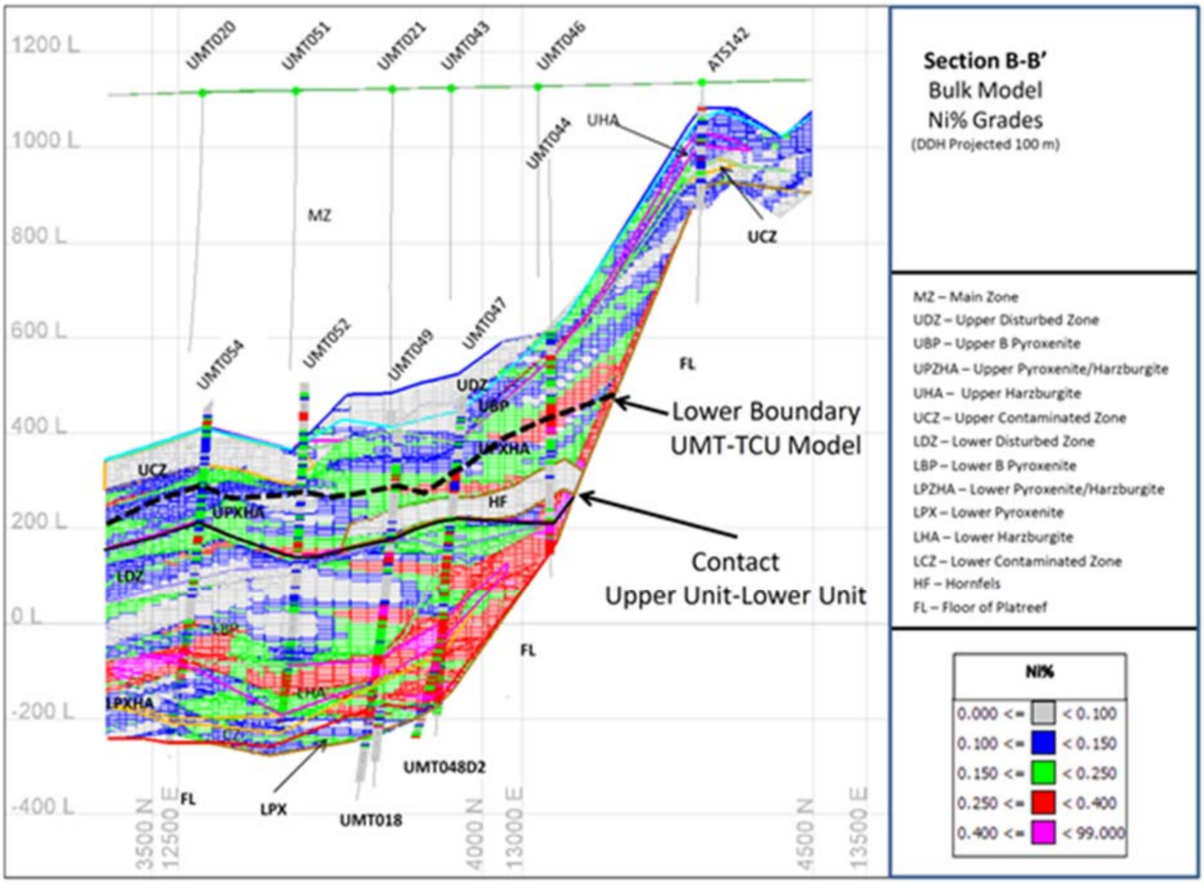
Note: Figure generated by AMEC, 2012. Coordinates shown are in the Local system. In the legend of this figure 3PE = Pt + Pd + Au.

Figure 14.14 UMT-MM Model – Cross-Section A-A' (looking north-west) showing 2PE + Au (g/t)



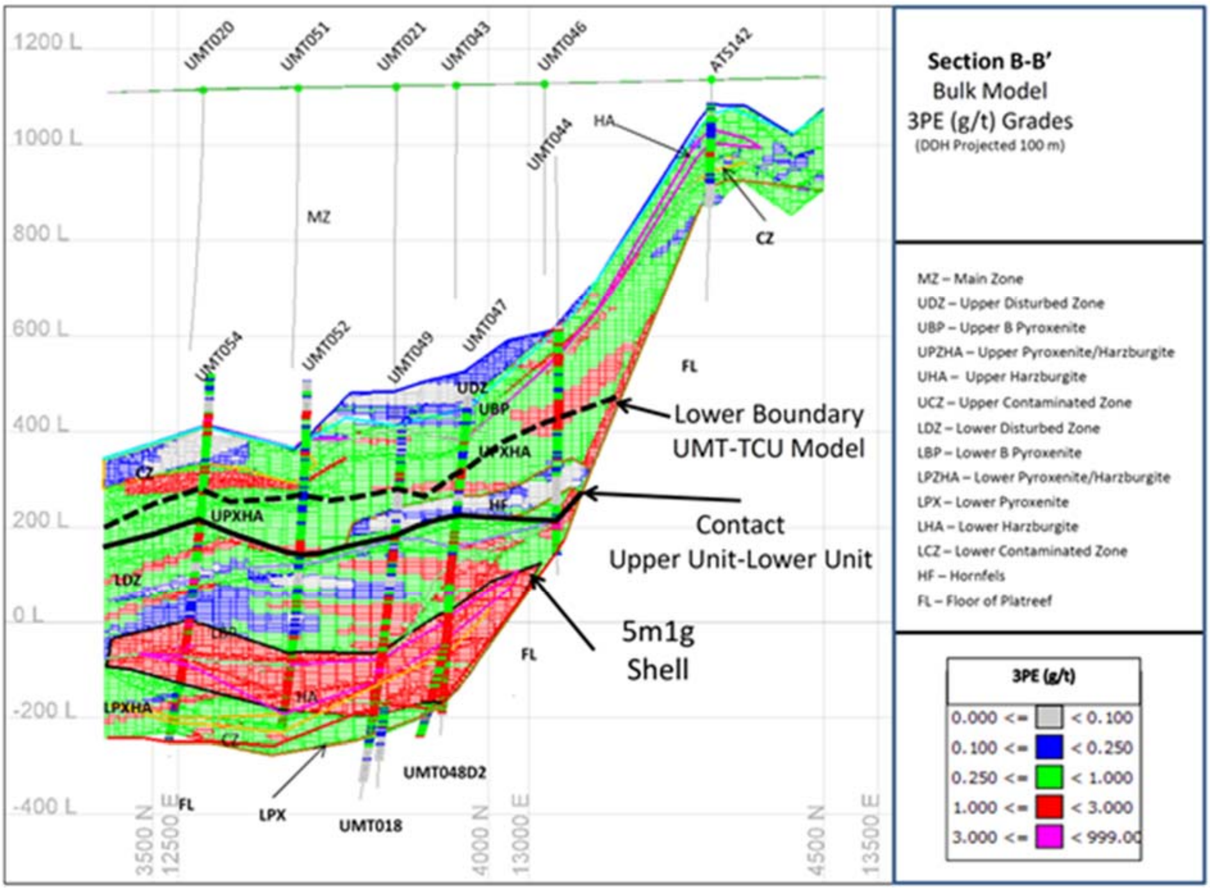
Note: Figure generated by AMEC, 2012. Coordinates shown are in the Local system.

Figure 14.15 UMT-MM Model – Cross-Section B-B' (looking north-west) Showing Ni%



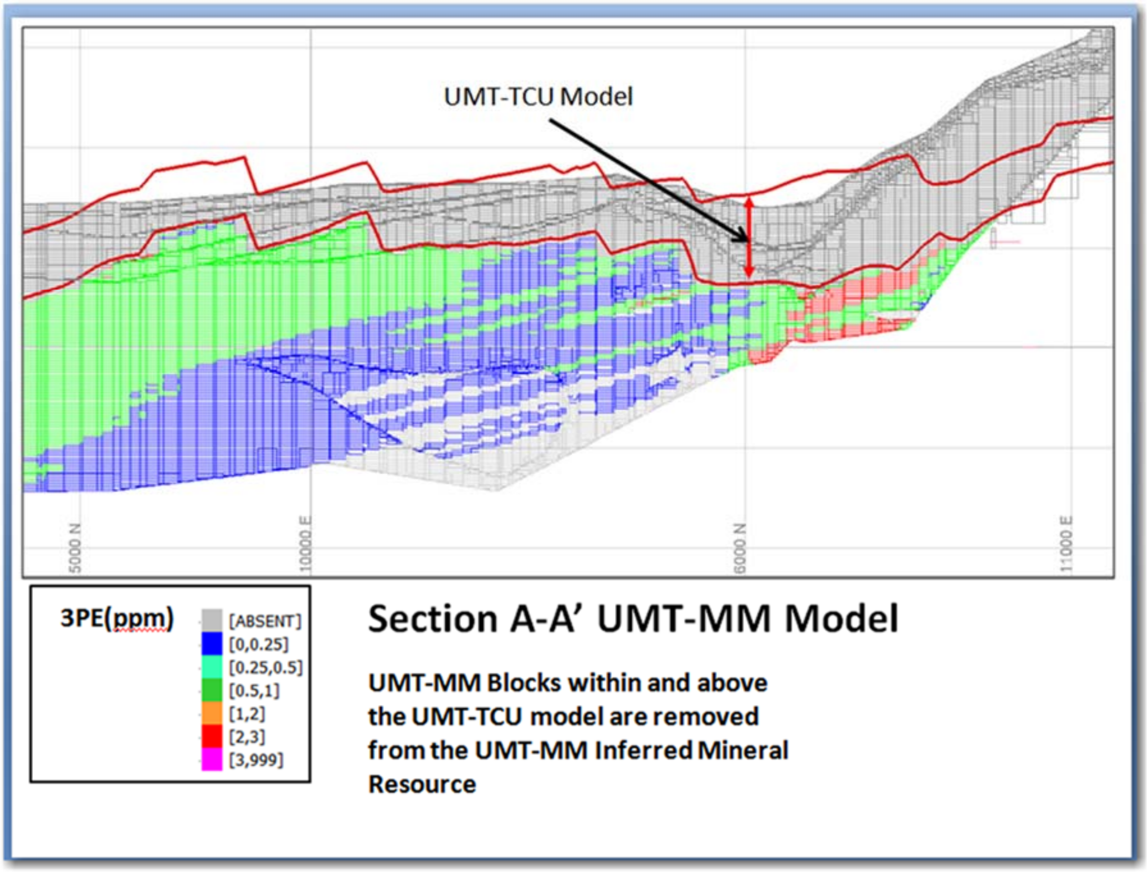
Note: Figure generated by AMEC, 2012. Coordinates shown are in the Local system.

Figure 14.16 UMT-MM Model – Cross-Section B-B' (looking north-west) Showing 2PE + Au (g/t)



Note: Figure generated by AMEC, 2012. Coordinates shown are in the Local system. . In the legend of this figure 3PE = Pt + Pd + Au.

Figure 14.17 UMT-MM Model Inferred Mineral Resources Below UMT-TCU Model (looking north-west)



Note: Figure generated by AMEC, 2012. Coordinates shown are in the Local system. In the legend of this figure 3PE = Pt + Pd + Au.

14.4 Open Pit Resource Models

A summary is provided below; a more detailed description is provided in the September 2012 Technical Report.

The methodologies used for creating the block models and estimating Mineral Resources amenable to open-pit methods at AMK and ATS are similar.

Mineralization within the AMK deposit was modelled in 2003 and is referenced as the 'Version L' model (methodology used is described in Parker and Francis, 2002). The ATS resource model ('Version Q' model) was developed during 2002 and 2003 (AMEC 2003b). This model was used in conceptual studies (AMEC 2004c), which were used to support Mineral Resources amenable to open-pit mining methods at ATS.

The drillhole locations and open-pit resource model limits are also shown on Figure 14.1.

14.4.1 Geological Models (Open Pit)

At both AMK and ATS, the hangingwall of the Platreef intrusive sequence is the Main Zone gabbronorite, and the footwall is the Transvaal Formation. Within the Platreef norites and pyroxenites (NPX), hornfels xenoliths (XE) of the Transvaal Formation and higher grade serpentinized peridotite–pyroxenite (SP) layers have been identified.

Hangingwall and footwall contacts were modelled as wireframe surfaces at ATS and AMK. The XE and SP units were modeled as wireframes for ATS. For AMK, these units were modelled probabilistically using indicator variables.

14.4.2 EDA and Grade Estimation Domains (Open Pit)

Grade estimation plans were developed by reviewing drillhole data on cross-sections and level plans, and producing a series of statistical plots (on 5 m composites and NN estimates) including box plots, histograms, probability plots, contact plots and variograms. Summaries of the observations made from this work are included in this sub-section.

AMK Resource Area

The Platreef in the AMK model area is divided into two subzones. The main mineralized zone is Subzone 1 that occurs at the base of the Platreef and is predominantly ultramafic rocks (pyroxenites and serpentinized peridotites). Subzone 2 is spatially limited and occurs above Subzone 1; it is dominated by norites and gabbros and is not part of the Mineral Resource estimate.

Within Subzone 1, SP is the highest-grade lithology with the lowest-grade variability of the Platreef lithologies. NPX lithologies are lower-grade with similar variability to SP, particularly for platinum. XE is the lowest-grade unit with generally high grade variability. There is local mineralization within XE, particularly in dolomite. The low variability and levelling out at the top end of the grade distributions indicates an upper limit on grade, making it unnecessary to cap (or top-cut) the distributions before grade estimation. Such limits are commonly found where the geochemical environment constrains enrichment; i.e., there is equilibrium and no remobilization/concentration by secondary processes.

Contact grade profiles between the three groups in Subzone 1 indicated use of hard contacts for grade estimation was appropriate.

In general, correlations between the metals are very strong and linear, indicating origination from a high-temperature mono-sulphide solid-solution that exsolved pyrrhotite, chalcopyrite, pentlandite, and PGMs as it cooled with little evidence of significant metal zoning.

ATS Resource Area

ATS was divided into four spatial domains (from south-east to north-west: South, Middle, Embayment, and North) based on metal ratios and mineralization widths. Each domain was partitioned by a continuous to locally disrupted hornfels and/or marginal zone norite (MZN). The hornfels may have partitioned the magmas and/or prevented mixing or interaction of the magma with the floor. Each domain was subdivided into serpentinized peridotite (SP) and norites and pyroxenites (NPX).

Grade capping for the five metals was considered unnecessary because of the low-grade variability suggested by the CVs.

Minor grade discontinuities were identified at geological zone contacts. Geological zones with similar average grades were grouped for grade estimation. Hard boundaries were used for each of the regions and domains, except for XE.

Variograms (using the correlogram method) were estimated using Sage2001™ software for Cu, Ni, Au, Pt, Pd, and dollar values inside each zone and hornfels unit using 5 m composites. Most variograms proved noisy and difficult to model; this was particularly true for zones with few composites.

14.4.3 Block Model and Grade Estimation (Open Pit)

The estimation methodology for the AMK and ATS models was developed to improve local grade estimates without compromising global grade distributions. Grade estimates within the Platreef were completed for Cu, Ni, Au, Pt, and Pd using inverse distance to an appropriate power (IDP). The power value was adjusted such that the co-efficients of variation (CVs) of block grade distributions reasonably matched selective mining unit (SMU) targets. The variance of the block distribution is inversely proportion to the power used in inverse distance interpolation. The CVs of the SMU distributions were estimated using the volume–variance relationship which states that the variance is inversely proportional to the volume of support. Conceptual mining engineering studies completed on the ATS deposit determined a 15 m x 15 m x 10 m SMU was likely to be appropriate for the deposit.

For both models, the blocks were oriented parallel to the local mine grid, with domain coding and grade estimates completed on a whole block basis (i.e., no sub-celling). The proportion of each block below the topographic surface was stored.

No allowances were made in the open-pit models for external dilution; or contact boundary loss/waste dilution. Outside the Platreef, the lack of sulphides and generally low-grade assays permit a reasonable assumption that these blocks can be assigned as waste.

Data available at the time ATS and AMK were studied indicated that only sulphide copper and nickel were recoverable by metallurgical treatment processes; therefore, nickel and copper values are reported on a 'sulphide head grade' basis with no allowance made for metallurgical recovery.

AMK Resource Model

A block size of 25 m x 25 m x 5 m was used for the AMK model. A single-pass estimation run was made for each metal. Search ellipsoids and inverse distance power varied with domain (refer to Table 14.12).

Table 14.12 AMK Inverse Distance Estimation Parameters

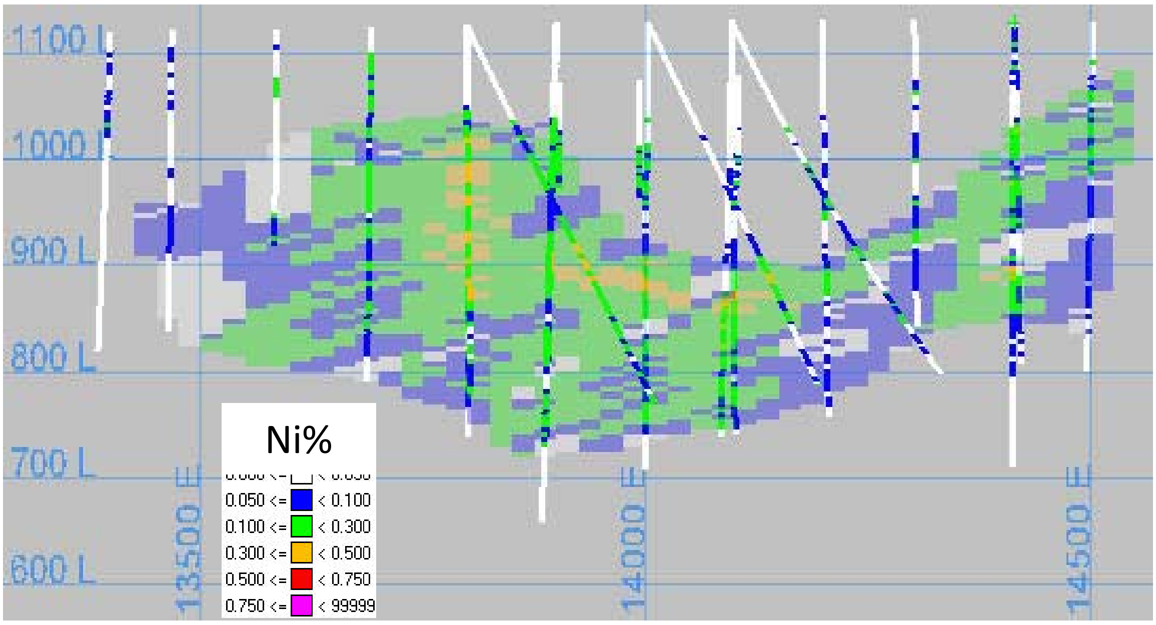
Domain	ID Power	Ellipsoid Orientation(deg)		Search Distances (m)		
		Strike	Dip	X	Y	Z
SP	2.5	37	36 NW	200	200	30
XE	1.4	311	21 SW	200	200	30
NPX	6.0	311	21 SW	200	200	30

Note: The lengths of the search ellipsoid axes are shown in terms of rotated X, Y, Z, with Z being the pole to the "equatorial" plane of the ellipsoid.

An octant search requiring a minimum of three octants with a maximum of two composites per octant was imposed for data selection. A block grade must be estimated with a minimum of six and a maximum of 24 composites.

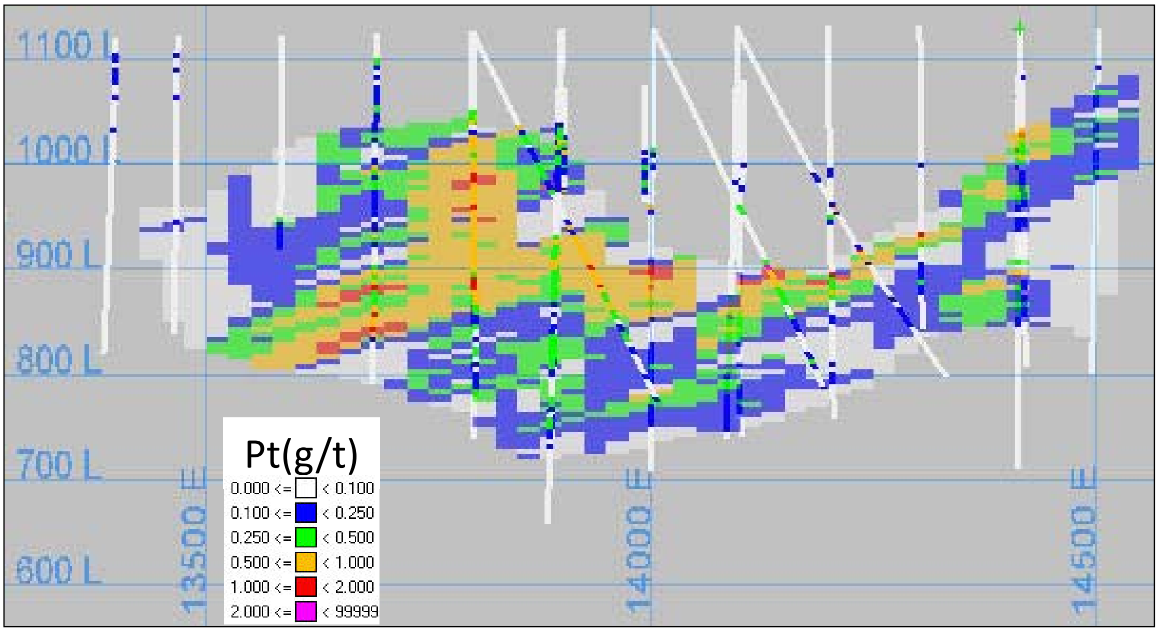
Figure 14.18 and Figure 14.19 show nickel and platinum grades for a section through the AMK deposit.

Figure 14.18 AMK Sulphide Ni (%) Block Estimates and Composites (cross-section 2500N, (Version L Model, 2003), looking north)



Note: Section Looking North. Figure generated by AMEC, 2003. Coordinates shown are in the Local system.

Figure 14.19 AMK Pt (g/t) Block Estimates and Composites



Note: Section Looking North. Figure generated by AMEC, 2003. Coordinates shown are in the Local system.

ATS Resource Model

A block size of 15 m x 15 m x 10 m was selected. Grades were estimated within the four regions, and three lithological domains (SP, XE, and NPX) of the Platreef. The same estimation plan was used for each element (Table 14.13). A three-pass setup was used for the estimation of grades with each pass having a progressively larger search ellipsoid; most blocks were estimated in Pass 1.

A sampling and assay programme of the drill core from oxidized Platreef rocks was completed in late 2003, and a wireframe model of the oxidized and soil horizons was constructed. The average depth of oxidation is 30 m. Preliminary metallurgical results of oxidized material are not encouraging, and oxidized material is treated as waste.

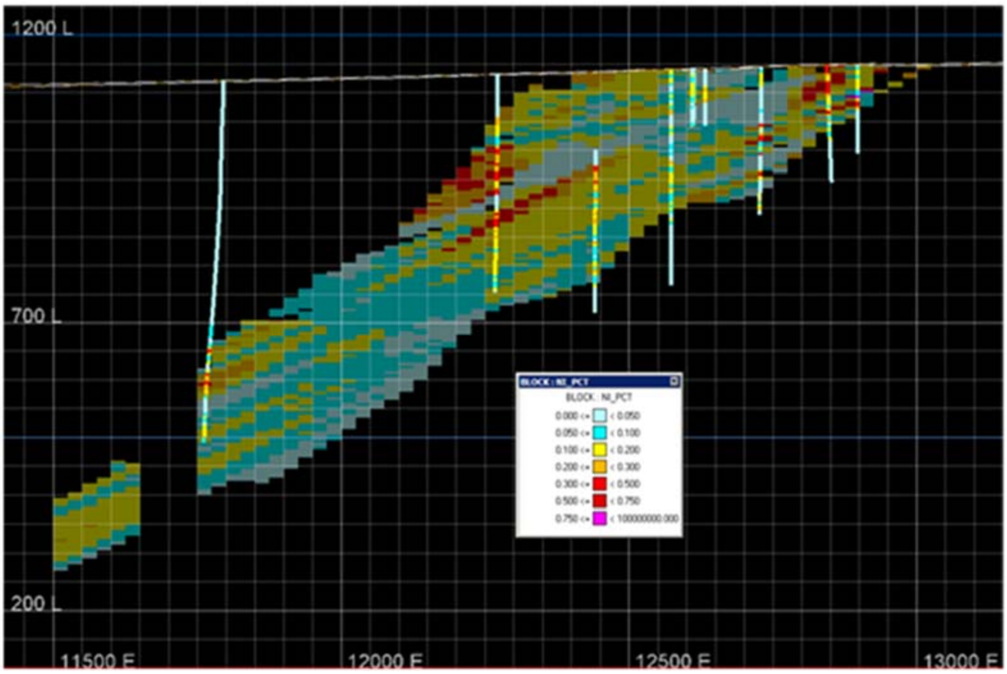
Table 14.13 ATS Inverse Distance Parameters

Domain	ID Power	Orientation(deg)		Pass 1 (m)			Pass 2 (m)			Pass 3 (m)		
		Strike	Dip	X	Y	Z	X	Y	Z	X	Y	Z
North												
SP	0.9	308	35 SW	150	150	25	250	250	25	300	300	50
XE	1.4	320	26 SW	150	150	25	250	250	25	300	300	50
NPX	5.0	308	35 SW	150	150	25	250	250	25	300	300	50
Middle												
SP	2.0	311	41 SW	150	150	25	250	250	25	300	300	50
XE	1.4	320	26 SW	150	150	25	250	250	25	300	300	50
NPX	4.4	311	41 SW	150	150	25	250	250	25	300	300	50
Embayment												
SP	3.5	308	38 SW	150	150	25	250	250	25	300	300	50
XE	1.4	320	26 SW	150	150	25	250	250	25	300	300	50
NPX	4.4	308	38 SW	150	150	25	250	250	25	300	300	50
South												
SP	2.5	48	25 SW	150	150	25	250	250	25	300	300	50
XE	1.4	320	26 SW	150	150	25	250	250	25	300	300	50
NPX	2.5	48	25 SW	150	150	25	250	250	25	300	300	50

Note: The lengths of the search ellipsoid axes are shown in terms of rotated X, Y, Z with Z being the pole to the "equatorial" plane of the ellipsoid.

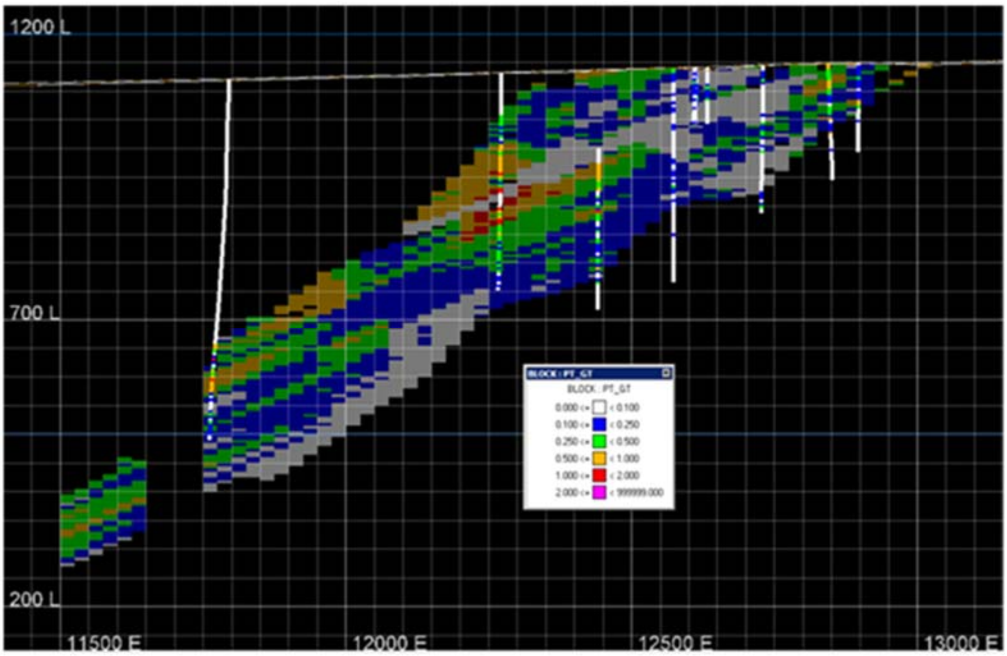
Figure 14.20 and Figure 14.21 show nickel and platinum grades for a section through the ATS deposit.

Figure 14.20 ATS Sulphide Ni (%) Block Estimates and Composites (cross-section 5850N (Version Q Model, 2003) looking north)



Note: Figure generated by AMEC, 2003. Coordinates shown are in the Local system.

Figure 14.21 ATS Pt (g/t) Block Estimates and Composites (cross-section 5850N (Version Q Model, 2003) looking north)



Note: Figure generated by AMEC, 2003. Coordinates shown are in the Local system.

14.4.4 Density (Open-Pit Models)

Bulk densities measured on drill core at site were used in tonnage estimates (Table 14.14). Section 11.2.1 contains the description of the density sampling and measurement methodology.

Table 14.14 Density Values for Tonnage Estimations

Rock	Density (g/cm^3)
AMK	
Platreef	3.07
Xenolith	2.80
Serpentinized Peridotite	3.04
ATS	
Main Zone	2.89
Serpentinite	3.01
Hornfels	2.85
Norite/Pyroxenite	2.99
Floor	2.85

14.4.5 Comments on Open Pit Models

Although the AMK model dates from 2003, it is suitable for conceptual studies. It does not contain allowances for block boundary loss or waste dilution at contacts. Eventually the AMK model should be connected with the UMT model to the north.

The ATS model is acceptable for preliminary mine planning, but will be locally inaccurate. The model will be high-biased in areas estimated to be high-grade and low-biased in areas estimated to be low-grade. The bias has been mitigated to some extent by infill drilling in the area where the drill spacing was reduced to 75 m x 75 m or 75 m x 100 m spacing. Additional infill drilling will be required to improve accuracy of grade estimation. The ATS model does not contain allowance for block boundary dilution or waste dilution at contacts, including the hangingwall and footwall of the Platreef and the contacts between XE (hornfels) and NPX or XE and SP.

Currently the TCU stratigraphic units are being correlated up-dip from the UMT-TCU model into the open-pit area (ATS Model) and along strike southwesterly into the AMK area. Re-logging of ATS drillholes have identified the TCU sequence, though highly contaminated, in the area of the resources considered amenable to open-pit methods. Though the Mineral Resource amenable to open-pit mining is not presently the focus of the Platreef project, the reinterpretation of the geology in the open-pittable area could provide better information that support some part of the mineralization being included in a revised UMT-TCU resource estimate above the current 650 m elevation.

14.4.6 Mineral Resource Classification (Open Pit Models)

All Mineral Resources declared for AMK are in the Inferred category. Drill spacing is nominally on a 100 m grid, with Mineral Resources projected up to 200 m beyond drilling. At this spacing there is broad-scale continuity in grade and lithological domains, but on a local basis there is sufficient uncertainty as to domain position and grade to prevent classification of the Mineral Resources as Indicated.

Indicated and Inferred Mineral Resources reported for ATS are located above the 650 m elevation. Indicated Mineral Resources are drilled at 100 m x 75 m (locally 75 m x 75 m) spacings; at both of these spacings continuity of lithological domains and grade can be assumed. Inferred Mineral Resources are drilled at 100 to 200 m spacings.

14.5 UMT-BIK Bikkuri Reef Resource Estimate

The Mineral Resources for the Bikkuri Reef are located on Turfspruit farm and includes mineralization that is amenable to underground selective mining methods and consists of material within and adjacent to grade shells for the Bikkuri Reef (refer to Figure 14.2).

The Mineral Resource for the Bikkuri Reef has been constructed using a revised geological interpretation that is similar to the TCU interpretation of the main Platreef. Current interpretations suggest the Bikkuri Reef is a slump block of the main Platreef (see Section 7.3.5 and Figure 7.16). The Mineral Resource estimate is based on the UMT-BIK model.

Controls for mineralization on the Bikkuri Reef are similar to those recognized on the Platreef (TCU stratigraphy) but the mineralization is typically lower in grade. The UMT-BIK model was completed in May 2013. The limits of the UMT-BIK Indicated and Inferred Mineral Resource estimates are shown on Figure 14.2 and are the same as the UMT-TCU resource model.

The UMT-BIK model is contained within an envelope defined to include the Bikkuri stratigraphic sequences. The upper surface of the envelope was set at 75 m above the top of the 'Norite Cycles in Bik Sequence' (NCBK). The lower surface of the envelope was set at 75 m below the bottom of the B2 stratigraphic horizon or the top of the TCU of the main Platreef where the distance is less than 75 m.

14.5.1 Drillhole Data – UMT-BIK

The drillhole data for the UMT-BIK resource model are a subset of the Platreef database and include 58 drillholes (50,363 m). All UMT drillholes have been re-logged for consideration of the TCU and the Bikkuri Reef.

14.5.2 Geological Model (UMT-BIK)

Geological interpretations for the UMT-BIK resource model were developed by Ivanhoe personnel. The re-logging work summarized in the lithological variable STRAT (refer to Table 14.1) is the basis for the geological model. A numeric model code (MCODE) was assigned to each lithology. Table 14.1 compares the coding scheme for the UMT-TCU, UMT-MM, and UMT-BIK models.

Two-dimensional gridded-seam models were completed for three stratigraphic horizons used in the construction of the Bikkuri geological model (NCMANBK, B1, and B2). Four wire frame surfaces were constructed from the gridded seam models for the NCMANBK, B1, and B2 (top and bottom).

14.5.3 High-Grade Shells – UMT-BIK

Ivanhoe personnel identified nested 2PE + Au (Pt + Pd + Au) grade shells using a minimum of 3 m grading 1 g/t 2PE + Au, 2 g/t 2PE + Au, and 3 g/t 2PE + Au. Grade shells using 2PE + Au were used rather than 3PE + Au because rhodium assaying was incomplete at the time the shells were constructed. The grade shells were constructed as a tool for constraining grade estimates.

The nested grade shells were identified in two mineralized zones (B1MZ and B2MZ). The B1MZ grade shells are associated with the B1 stratigraphic unit. The B2MZ grade shells are associated with the B2 stratigraphic unit. Two-dimensional gridded-seam models were completed for the B1MZ and B2MZ grade shells, and wireframe surfaces were constructed from the gridded seam models.

Grade shell codes (GCODES) were used to identify blocks within and outside the grade shells. Table 14.2 summarizes the GCODES that were used for the UMT-BIK resource model.

14.5.4 Mineralization Adjacent to the Bikkuri Mineralized Zones

There is scattered mineralization locally adjacent to the Bikkuri mineralized zones (B1MZ and B2MZ). Mineralization adjacent to the Bikkuri mineralized zones may be included in mine development, and therefore there is a need to estimate grades in blocks in an envelope around the Bikkuri mineralized zones.

14.5.5 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.

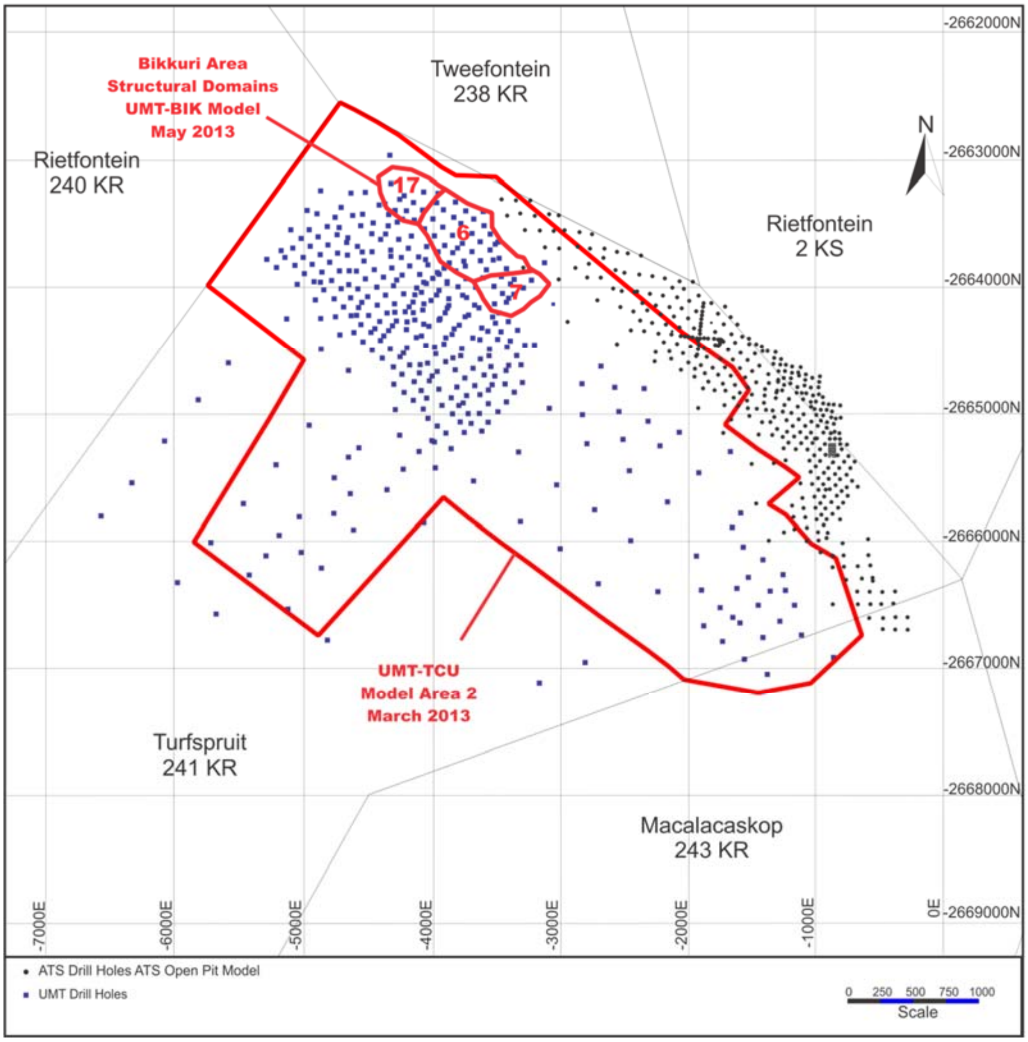
EDA was completed using box plots, histograms, probability plots and contact profiles. Discontinuities in grade profiles near contacts suggested the grade shells and stratigraphic boundaries should be considered hard boundaries.

The rhodium analyses are only partially complete on the Bikkuri drillholes. Because of the limited rhodium data in the Bikkuri drilling subset, the rhodium regressions for the UMT-TCU model were used for the Bikkuri Reef (see Section 14.2.5).

14.5.6 Block Model and Grade Estimation

The UMT-BIK block model was constructed over the area where the Bikkuri Reef has been identified (Figure 14.22). Blocks were oriented parallel to the national co-ordinate system. The block model used a parent block size of 10 m x 10 m x 1 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 regularized to 10 m x 10 m x 2 m block sizes.

Figure 14.22 Extent of UMT-BIK Resource Model, Showing Estimation Domains



Note: The UMT-TCU and UMT-BIK resource areas are separated by elevation. Figure prepared by AMEC, 2013. Co-ordinates use the WGS system.

14.5.6.1 Grade Estimation – UMT-BIK

B1MZ and B2MZ

Grade estimation in the B1MZ and B2MZ included block and composite matching by GCODE. Grade estimation was completed by structural domain using the dynamic anisotropic option in Datamine. Estimation was completed by ID3. A NN estimate was completed for model validation.

Blocks Adjacent Grade Shells

Grade estimation in the NCBK, MANBK, B1, B2 and FWBK stratigraphic units for blocks not located within the nested grade shells were estimated by matching blocks and composites by MCODE. Estimation was completed by ID3 using dynamic anisotropy. A NN estimate was completed for model validation.

Estimations were completed in Datamine using expanding search volumes. Search volumes are summarized in Table 14.15.

Table 14.15 Search Strategy for 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

Grade Capping and Outlier Restriction

No grade capping was implemented within the nested grade shells. No outlier restriction was applied.

Regularization

Upon completion of the estimation, the UMT-BIK block model was regularized from the 20 m x 20 m x 1 m (sub-celled to 10 m x 10 m x 0.5 m) to a 10 m x 10 m x 2 m (no sub-cells) model. 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.

14.5.7 Bulk Density

Densities were also coded to the blocks by stratigraphic unit using the mean density values for each stratigraphic unit (Table 14.16).

Table 14.16 Bulk Density Values

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

14.5.8 Mineral Resource Classification

Mineral Resources at Bikkuri have been classified using the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2010), as discussed in Section 14.2.8.

The boundaries of Indicated and Inferred Mineral Resources for the TCU-BIK resource model are shown in Figure 14.2. The drill spacing in the Indicated Mineral Resource is nominally 100 m. Drill spacing in the Inferred Mineral Resource ranges up to 400 m.

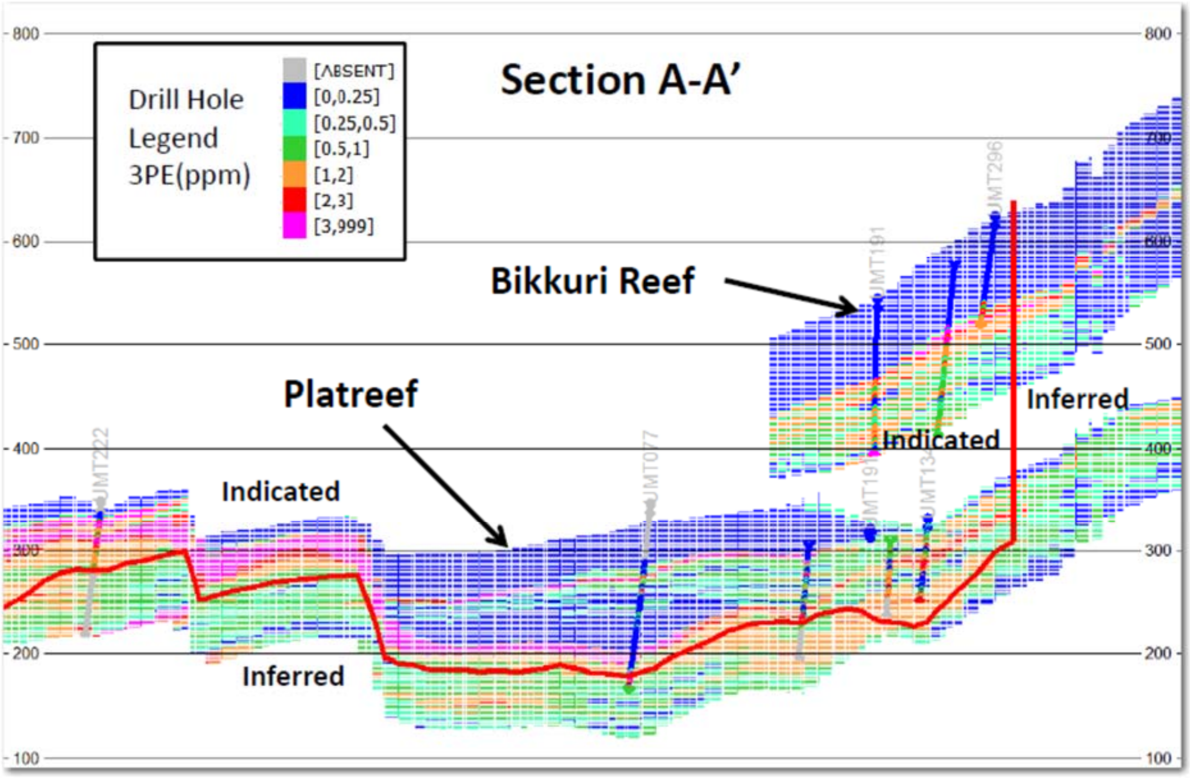
14.5.9 UMT-BIK Model Validation

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 grade shell were constructed to compare the kriged (where present), ID3 grade estimates, NN estimates and 1 m composites.

14.5.9.1 Visual Validation and Box Plots

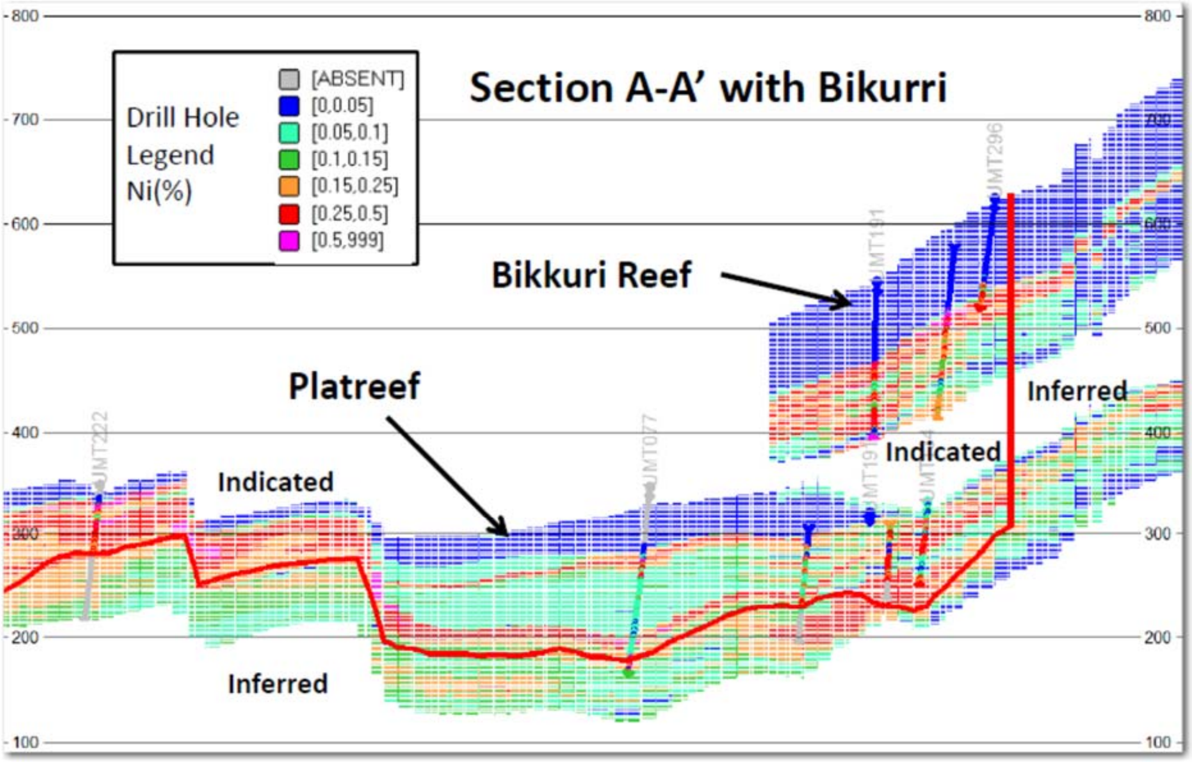
Block grades (ID3) were compared to composite grades (for each metal) by visual inspection on cross-sections, long sections and level plans. In general, the composite grades were honoured in the block distributions. Representative cross-sections for 2PE + Au and Ni are presented in Figure 14.23 and Figure 14.24 respectively (section lines A-A').

Figure 14.23 Section AA' Displaying 2PE + Au Block and Composite Grades (looking north)



Note: Figure prepared by AMEC, 2013. Co-ordinates use the WGS system. In the legend of this figure 3PE = Pt + Pd + Au.

Figure 14.24 Section AA' Displaying Ni Block and Composite Grades (looking north)



Note: Figure prepared by AMEC, 2013. Co-ordinates use the WGS system.

The global means and grade distributions for each metal from the ID3 model, NN model, and 5 m composites were checked and found to compare within reasonable levels, suggesting the ID3 model is globally un-biased.

14.5.9.2 Swath Plots

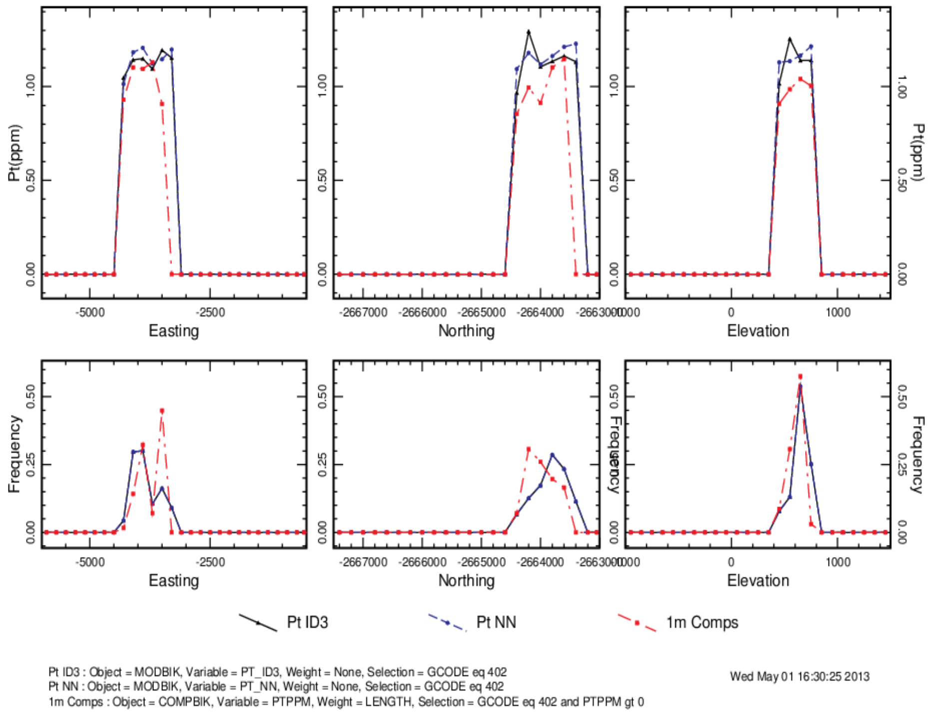
Swath plots (width of 200 m) of the ID3 model, NN model and 1 m composites were completed for Cu, Ni, Pt, Pd, Au, and Rh.

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

AMEC commonly focuses swath plot analysis on blocks classified as Measured and Indicated. Because of the limited extent of the UMT-BIK resource model, swath plots were completed for the entire UMT-BIK resource model.

The platinum swath plot for the B2MZ 2 g/t shell is presented in Figure 14.25.

Figure 14.25 Platinum Swath Plot for B2MZ – 2 g/t Shell



Note: Figure prepared by AMEC, 2013.

14.5.10 Comments on the UMT-BIK Model

As currently configured the UMT-BIK model covers the stratigraphic units comprising the Bikkuri Reef.

The UMT-BIK model locally includes estimation of grades in blocks adjacent to the Bikkuri stratigraphic units (B1 and B2), up to 75 m on the hangingwall and up to 75 m into the Bikkuri footwall, but does not extend to the model envelope for the UMT-TCU resource model. Additional drilling is required to better define the lateral extents of the Bikkuri mineralization and the boundary between the Bikkuri stratigraphic units and the TCU stratigraphic units.

14.6 Assumptions Made to Assess Reasonable Prospects for Economic Extraction

AMEC undertook a conceptual analysis to assess reasonable prospects for economic extraction for declaration of Mineral Resources. Underground mining methods considered are conventional, mechanized mining methods that have a reasonable safety factor.

14.6.1 Commodity Prices

AMEC considers that consensus long-term commodity prices should be used in declaration of Mineral Reserves and for purposes of financial analyses. For the Mineral Resource estimates, the following prices were used: $8.81/lb for Ni, $2.73/lb for Cu, $1,699/troy ounce for Pt, $667/troy ounce for Pd, and $1,315/troy ounce for Au. For rhodium, AMEC has used $2,065/troy ounce, which is based on the average of two values obtained from technical reports that were filed on SEDAR ($1,875, $2,255) in 2012.

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 $38/t. Process, concentrate transport and general and administrative (G+A) costs for this case were estimated at an average of $12.50/t of mill feed.

For the MM underground scenario, mining costs could vary from $9 to $35/t depending on whether block caving or some form of sub-level mining is used. For the MM underground case and open-pit case, a production rate of 10 Mtpa was assumed. Process, concentrate transport and G+A costs were estimated at an average of $12/t of mill feed.

14.6.3 Process Recoveries

For the selective high-grade option, process recoveries are based on equations shown in Table 14.17. These recoveries were available from Base Data Template 13, provided by Ivanhoe and AMEC process engineers on 21 March 2013.

Table 14.17 Metallurgical Recovery Equations (21 March 2013)

	Metallurgical Domain		
	T1	**T2U**	**T2L**
Mass Pull (%)	3.690	3.082	3.399
Tailings Grade			
Pt (g/t)	0.300	0.324	0.324
Pd (g/t)	0.250	0.282	0.282
Au (g/t)	0.100	0.106	0.106
Rh (g/t)	0.007	0.007	0.007
4PGE (g/t)	0.630	0.712	0.754
Ni (%)	0.090	0.113	0.113
Cu (%)	0.030	0.030	0.030
Typical Head			
Grades			
Pt (g/t)	2.998	2.998	2.998
Pd (g/t)	2.931	2.931	2.931
Au (g/t)	0.244	0.244	0.244
Rh (g/t)	0.152	0.152	0.152
4PGE (g/t)	6.325	6.325	6.325
Ni (%)	0.297	0.297	0.297
Cu (%)	0.108	0.108	0.108
Recoveries (%)			
Pt	90.4	89.5	89.6
Pd	91.8	90.7	90.7
Au	60.5	57.9	58.0
Rh	95.6	95.5	95.6
4PGE	90.4	89.1	88.5
Ni	70.8	63.1	63.2
Cu	73.2	73.1	73.2

Note: Mass Pull = percentage weight recovery to concentrates
Recovery = [(head grade – (1-mass pull)/100)/(head grade)]100

For the open-pit cases, the process recoveries are based on equations calculated using Ni(s) and Cu(s) for nickel and copper:

*Nickel recovery = ((9.3 * Ln(Ni head grade) + 99.1) * % non-serpentine mineralization) + ((9.3 * Ln(Ni head grade) + 84.9) * % serpentine mineralization)*

*Copper recovery= ((10.3 * Ln(Ni head grade) + 93.3)* % non-serpentine mineralization) + ((10.3 * Ln(Ni head grade) + 92.6) * % serpentine mineralization)*

*Platinum recovery= ((4.8 * Ln(Ni head grade) + 80.9) * % non-serpentine mineralization) + ((4.8 * Ln(Ni head grade)+ 80.1) * % serpentine mineralization)*

*Palladium recovery= ((8.9 * Ln(Ni head grade) + 83.5) * % non-serpentine mineralization) + ((8.9 * Ln(Ni head grade) + 84.2) * % serpentine mineralization)*

*Gold recovery= ((11.9 * Ln(Ni head grade) + 78.5) * % non-serpentine mineralization) + ((11.9 * Ln(Ni head grade) + 79.4) * % serpentine mineralization).*

For the MM underground case, the process recoveries are taken from the same equations for serpentinite regardless of the block lithology. This is a conservative assumption, given limited testwork. The equations are applied to total nickel (Ni).

14.6.4 Smelter Payables

AMEC 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 $22/t of concentrates (approximately $1/t of mineralized 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'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.

14.6.6 NSR (Net Smelter Return)

The NSR calculation assumes the smelter payables.

14.7 Mineral Resource Statement

Mineral Resource statements for Mineral Resources amenable to underground mining methods (UMT) and Mineral Resources amenable to open-pit mining methods (ATS and AMK) are tabulated in this section. The term "base case" has been used to indicate the tonnage and grade estimate that are considered by AMEC to provide a starting point for scoping studies and preliminary economic assessments. 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.

The following considerations were taken into account when making the decision to use a 650 m elevation to demarcate the base-case limit between Mineral Resources amenable to open-pit mining methods and Mineral Resources amenable to underground mining methods:

- The ATS and UMT models overlap; therefore, a method for differentiating Mineral Resources amenable to open-pit or underground mining methods was required. Selecting the elevation is difficult at the current state of project knowledge. Various open-pit cases have been evaluated over the years resulting in pits bottoming at between the 660 m and 450 m elevation (540 m to 700 m vertical depths). The two cases run (and detailed) in 2003 (AMEC, 2003) bottomed at about the 660 m and 590 m elevations. The Indicated and some Inferred Mineral Resources lie within these pits.
- In the ATS and AMK deposits, metallurgical recoveries were stated on a sulphide nickel basis, and the open-pit Mineral Resource models have sulphide nickel and copper. For the UMT deposit, metallurgical recoveries were stated on a total nickel basis, and the underground Mineral Resource models use total nickel and copper. The distinction between sulphide and total nickel is necessary to perform the assessments of reasonable prospects of economic extraction for mineralization within the AMK, ATS, and UMT deposits.
- At elevations lower than 650 m, the open-pit models do not extend the entire way across the Platreef, and at elevations higher than 650 m, the underground models do not extend the entire way across the Platreef.

Future mining studies are likely to provide a reassessment of reasonable prospects of economic extraction for open-pit and underground scenarios. This may involve redefinition of the demarcation between Mineral Resources amenable to open-pit or underground mining methods.

14.7.1 Mineral Resources Amenable to Underground Mining Methods

There are two mining scenarios that could exploit mineralization at depth within the Platreef:

- Selective mining within and adjacent to TCU mineralized zones.
- Mass mining.

The selectively-mineable option is considered the base-case Mineral Resource estimate for the purposes of this Report. AMEC reviewed the potential to mass-mine lower-grade material, and presents the results as an additional and mutually-exclusive case.

Other considerations are:

- Process, concentrate transport and site G+A costs must be covered for reporting Mineral Resources.

Mining costs have been considered in setting the cut-off ($38/t for the selective case, and for the bulk case from $9/t to $35/t depending on whether block caving or some method of sub-level mining was used.

14.7.2 Mineralization 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 mineralization that could be mined using underground selective methods such as long-hole open-stoping, drift/cut and bench, bench-and-fill or drift-and-fill.

Table 14.18 shows Mineral Resources lying within and adjacent to the TCU mineralized zones.

Table 14.18 Mineral Resources Amenable to Selective Mining Methods 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)	Ni (%)	Cu (%)
3 g/t	137	2.27	2.31	0.35	0.15	5.09	0.38	0.18
2 g/t	214	1.83	1.89	0.29	0.12	4.13	0.34	0.17
1 g/t	387	1.28	1.34	0.21	0.09	2.92	0.28	0.14
Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Ni (Mlbs)	Cu (Mlbs)
3 g/t	–	10.0	10.2	1.53	0.67	22.4	1,133	558
2 g/t	–	12.6	13.0	2.00	0.85	28.5	1,610	794
1 g/t	–	15.9	16.7	2.67	1.09	36.3	2,408	1,189
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)	Ni (%)	Cu (%)
3 g/t	211	2.09	2.06	0.34	0.14	4.63	0.38	0.18
2 g/t	415	1.57	1.59	0.27	0.11	3.54	0.33	0.16
1 g/t	1,054	0.96	1.02	0.18	0.07	2.23	0.26	0.13
Contained Metal								
Cut-off 3PE+Au	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Ni (Mlbs)	Cu (Mlbs)
3 g/t	–	14.2	14.0	2.29	0.97	31.5	1,764	855
2 g/t	–	20.9	21.3	3.58	1.44	47.2	3,032	1,490
1 g/t	–	32.7	34.7	6.95	2.32	75.7	5,935	3,035

Notes:

1. Mineral Resources have an effective date of 3 April 2013. The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME.
2. Mineral Resources estimated assuming underground selective mining methods within and adjacent to the TCU are exclusive of the Mineral Resources estimated assuming mass-mining methods. The 2 g/t 3PE+Au cut-off is considered the base case estimate; the 3 g/t 3PE+Au cut-off is also being considered.
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 m x 400 m (locally to 400 m x 200 m and 200 m x 200 m) spacing.
4. Reasonable prospects for economic extraction were determined using the following assumptions. Assumed commodity prices are Ni: $8.81/lb, Cu: $2.73/lb, Pt: $1,699/oz, Pd: $667/oz, Au: $1,315/oz, and Rh: $2,065/oz. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $40/t) and process, G&A, and concentrate transport costs (average $12.50/t of mill feed for 4 Mt/a operation) would be covered. The process recoveries vary with block grade but typically would be 85–90% for Pt, Pd, and Rh; 75% for Au and 70% for Ni and 85% for Cu.
5. Totals may not sum due to rounding.

AMEC tested the Mineral Resources for reasonable prospects for economic extraction. At a 2 g/t 4PE 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 long-hole 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 4PE cut-off generate an NSR of $10/t, meaning they will cover nearly all process, concentrate transport, and G&A costs.

Table 14.19 shows the Indicated and Inferred Mineral Resources lying within the nested 1 g/t, 2 g/t, and 3 g/t 2PE + Au grade shells. This case illustrates what high-grade mineralization is present within the 1 g/t, 2 g/t, and 3 g/t 2PE + Au grade shells.

Table 14.19 Mineral Resources Within Grade Shells Assuming Underground Selective Mining Methods (base case is highlighted)

Tonnage and Grades, Indicated Mineral Resources								
Grade Shells	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Ni (%)	Cu (%)
3 g/t	121	2.34	2.38	0.35	0.16	5.22	0.38	0.19
2 + 3 g/t	204	1.83	1.90	0.29	0.12	4.14	0.34	0.17
1 + 2 + 3 g/t	265	1.57	1.64	0.26	0.11	3.57	0.32	0.16
Contained Metal, Indicated Mineral Resources								
Grade Shells	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Ni (Mlbs)	Cu (Mlbs)
3 g/t	–	9.09	9.25	1.38	0.61	20.3	1,011	498
2 + 3 g/t	–	12.0	12.4	1.91	0.81	27.1	1,539	758
1 + 2 + 3 g/t	–	13.3	13.9	2.18	0.91	30.4	1,866	914
Tonnage and Grades, Inferred Mineral Resources								
Grade Shells	Mt	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)	Ni (%)	Cu (%)
3 g/t	185	2.13	2.12	0.35	0.15	4.74	0.39	0.19
2 + 3 g/t	355	1.63	1.63	0.28	0.11	3.66	0.34	0.17
1 + 2 + 3 g/t	548	1.31	1.32	0.23	0.09	2.95	0.30	0.15
Contained Metal, Inferred Mineral Resources								
Grade Shells	–	Pt (Moz)	Pd (Moz)	Au (Moz)	Rh (Moz)	3PE+Au (Moz)	Ni (Mlbs)	Cu (Mlbs)
3 g/t	–	12.6	12.6	2.07	0.87	28.2	1,580	762
2 + 3 g/t	–	18.6	18.7	3.19	1.27	41.7	2,653	1,288
1 + 2 + 3 g/t	–	23.0	23.3	4.05	1.58	52.0	3,639	1,776

Notes:

1. Mineral Resources have an effective date of 3 April 2013. The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME.
2. Mineral Resources estimated assuming underground selective mining methods within the TCU grade shells are exclusive of the Mineral Resources estimated outside the TCU grade shells, the Mineral Resources estimated assuming mass-mining methods, and the Bikkuri Reef estimates. The grade shell rows are also not additive. The 3 g/t 2PE + Au shell is included in the 2 + 3 g/t 2PE + Au shell, which is in turn included in the 1 + 2 + 3 g/t 2PE + Au shell.
3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately -200 m to 650 m elevation. Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 m x 400 m (locally to 400 m x 200 m and 200 m x 200 m) spacing.
4. Reasonable prospects for economic extraction were determined using the following assumptions. Assumed commodity prices are Ni: $8.81/lb, Cu: $2.73/lb, Pt: $1,699/oz, Pd: $667/oz, Au: $1,315/oz, and Rh: $2,065/oz. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $40/t) and process, G&A, and concentrate transport costs (average $12.50/t of mill feed for a 4 Mtpa operation) would be covered. The process recoveries vary with block grade but typically would be 85–90% for Pt, Pd and Rh; 75% for Au and 70% for Ni and 85% for Cu. The Mineral Resources within the 1+2+3, 2+3 or 3 g/t 2PE + Au grade shells have been estimated (at nominal cut-off grades of 1, 2, and 3 g/t 2PE + Au respectively) to show sensitivity to cut-off grade and to provide multiple options for consideration in future mining studies. No allowances for mining recovery and external dilution have been applied.
5. Totals may not sum due to rounding.
6. In this table 3PE + Au = Pt + Pd + Rh + Au. 2PE + Au = Pt + Pd + Au.

For the selectively-mineable underground base case resource estimate, cut-offs for constraining grade shells have been presented in terms of 2PE + Au, because the majority of the value is attributable to PGEs and gold. The Mineral Resources within the 1+2+3 g/t 2PE + Au (nominal cut-off grade of 1 g/t 2PE + Au), 2+3 g/t 3PE (nominal cut-off grade of 2 g/t 2PE + Au), or 3 g/t 2PE + Au (nominal cut-off grade of 3 g/t 3PE) grade shells have been estimated so that scoping mining studies could be performed using multiple options. The grade shells were constructed using 2PE + Au = Pt + Pd + Au because rhodium assaying was incomplete at the time. Rhodium was subsequently assayed or estimated using regression, and its estimates are presented in Table 14.19 together with a 4PE estimate (Pt + Pd + Au + Rh).

Table 14.20 tabulates the Indicated and Inferred Mineral Resources that are adjacent to the grade shells.

Table 14.20 Mineral Resources Adjacent to Grade Shells Assuming Underground Selective Mining Methods

Indicated Minerals Resource								
Mineralized Zone	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**Rh (g/t)**	**3PE+Au (g/t)**	**Ni (%)**	**Cu (%)**
HW	1.70	1.37	1.12	0.34	0.07	2.90	0.25	0.12
Middle	6.91	1.30	1.13	0.22	0.09	2.74	0.22	0.10
FW	6.09	1.18	1.15	0.16	0.08	2.57	0.25	0.14
Total	**14.7**	**1.26**	**1.14**	**0.21**	**0.08**	**2.69**	**0.23**	**0.12**
Inferred Mineral Resource								
Mineralized Zone	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**Rh (g/t)**	**3PE+Au (g/t)**	**Ni (%)**	**Cu (%)**
HW	2.22	0.95	1.08	0.24	0.05	2.32	0.26	0.17
Middle	8.68	1.32	1.12	0.24	0.08	2.76	0.23	0.11
FW	44.0	1.03	1.21	0.17	0.08	2.49	0.27	0.15
Total	**54.9**	**1.07**	**1.19**	**0.19**	**0.08**	**2.52**	**0.26**	**0.14**

Notes:

1. Mineral Resources have an effective date of 3 April 2013. The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME.
2. HW = above T1MZ, Middle = between T1MZ and T2MZ, FW = below T1MZ. Mineral Resources estimated assuming underground selective mining methods that are adjacent to grade shells are exclusive of the Mineral Resources estimated assuming selective mining within grade shells and mass-mining methods and the Bikkuri Reef estimates. The estimate is reported using a 2 g/t 3PE+Au cut-off.
3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately -200 m to 650 m elevation. Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 m x 400 m (locally to 400 m x 200 m and 200 m x 200 m) spacing.
4. Reasonable prospects for economic extraction were determined using assumed commodity prices of Ni: \$8.81/lb, Cu: \$2.73/lb, Pt: \$1,699/oz, Pd: \$667/oz, Au: \$1,315/oz, Rh: \$2,065/oz. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average \$40/t) and process, G&A, and concentrate transport costs (average \$12.50/t of mill feed for a 4 Mtpa operation) would be covered. The process recoveries vary with block grade but typically would be 85-90% for Pt, Pd and Rh; 75% for Au and 70% for Ni and 85% for Cu.
5. Totals may not sum due to rounding.
6. In this table 3PE+Au = Pt + Pd + Rh + Au and 2PE + Au = Pt + Pd + Au.

14.7.3 Mineral Resource Statement for Mineralization Amenable to Underground Mining Methods (Estimate Assuming Mass-Mining Methods)

The Mineral Resources amenable to underground mass mining methods have been revised for transfer of much of the previous Upper Unit Top Loaded Zone to the Mineral Resources amenable to underground selective mining methods.

Table 14.21 shows a tabulation of the Mineral Resources amenable to underground mass-mining methods for the Platreef Project. The metallurgical laboratories (SGS Johannesburg, XPS Falconbridge Ontario and Mintek Johannesburg) report on a total nickel basis; therefore, this Mineral Resource is reported on a total nickel basis. Inferred Mineral Resources are reported at 0.15% Ni cut-off grade. This cut-off grade is justified using the commodity price assumptions discussed in Section 14.5. The Mineral Resources amenable to mass mining methods are all located in areas of wide-spaced drilling (typically 400 x 400 m spacing). Only Inferred Mineral Resources are declared.

Table 14.21 Inferred Mineral Resources (at 0.15% Ni (total) Cut-Off)

Tonnage and Grades							
Property	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**2PE + Au (g/t)**	**% Ni**	**% Cu**
Turfspruit	1,870	0.40	0.49	0.09	0.98	0.21	0.13
Macalacaskop	40	0.28	0.39	0.09	0.76	0.21	0.14
Total 2PE + Au	**1,910**	**0.40**	**0.49**	**0.09**	**0.98**	**0.21**	**0.13**
Contained Metal							
Property	–	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**2PE + Au (Moz)**	**Ni (Mlbs)**	**Cu (Mlbs)**
Turfspruit	–	24.0	29.4	5.5	58.9	8,740	5,520
Macalacaskop	–	0.4	0.5	0.1	1.0	190	120
Total	**–**	**24.4**	**29.9**	**5.6**	**59.9**	**8,930**	**5,650**

Notes:

1. Mineral Resources have an effective date of 3 April 2013. The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME.
2. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from the 650 m elevation (-500 m depth) downward to approximately -400 m elevation (-1,550 m depth). The 2011 block model has been trimmed to exclude the 2013 block model for Mineral Resources amenable to selective mining methods. Inferred Mineral Resources are based on an area drilled on approximately 400 m x 400 m (locally 400 m x 200 m and 200 m x 200 m) spacing.
3. The estimate is reported at a cut-off grade of 0.15% Ni. Mineral Resources at the 0.15% Ni cut-off grade occur in continuous zones; there are a relatively minor number of blocks inside these zones that are below cut-off and have been excluded.
4. The cut-off grade (0.15% Ni) used to report the base case for reasonable prospects of economic extraction assumes commodity prices of Ni: $8.81/lb, Cu: $2.73/lb, Pt: $1,699/oz, Pd: $667/oz, Au: $1,315/oz. It has been assumed that payable metals would be 82% from smelter/refinery and that a mix of block cave and sub-level mining costs (averaging $20/t, and ranging from $9/t to $35/t), and process, G&A, and concentrate transport costs (average of $12/t of mill feed) would be covered for a conceptual 10 Mtpa operation. Process recoveries are taken from metal-specific equations for serpentinite. Nickel is presented as an example where nickel recovery = ((9.3 *Ln (Ni head grade) + 84.9).
5. Totals may not sum due to rounding.
6. 2PE + Au = Pt + Pd + Au.

14.7.4 Comments on Mineral Resources Amenable to Mass Mining Methods

The geological logging and interpretation is the foundation for the underground resource model. Recognition of lithological controls on grade has enabled declaration of Inferred Mineral Resources at a wider spacing than would otherwise be possible. The Inferred Mineral Resource boundary is shown on Figure 14.2. The drillhole spacing is 400 m x 400 m or 400 m x 200 m, with local 200 m x 200 m coverage. Around the margins, the boundary is nominally 200 m from the closest drillhole. The Mineral Resource amenable to mass mining methods also includes material in Zone 1, where not all holes reached the footwall, and the spacing was effectively 400 x 400 m.

The geometry of the Platreef in the Mineral Resource area that is amenable to underground mining methods is relatively complex, both in terms of thickness (150 m to 650 m) and dip (0° to 75°). Selection of mining methods and preparation of stope layouts is likely to require trade-off studies.

The declared Mineral Resources at the 0.15% Ni cut-off will provide an NSR/t of approximately $30. This should cover average mining, processing, concentrate transport and G&A costs.

The resource model has been constructed using relatively wide-spaced data, and is therefore suitable for planning using bulk underground methods such as block caving, sub-level caving, or sub-level open stoping. The degree of smoothing in the block model results from block size and estimation plans used, which reflects the currently-available data. The model is not suitable for planning a high-grade option using more selective mining methods typical of vein mining such as narrow open stoping or drift-and-fill mining.

14.7.5 Mineral Resource Statement for Mineralisation Amenable to Open-Pit Mining Methods

The Mineral Resources amenable to open-pit mining methods have not been updated since March 2011(refer to Parker et al., 2012).

Mineral Resources that could be exploited by open-pit mining methods include Platreef mineralization at ATS and AMK.

About half of the Mineral Resources that could be extracted using open-pit mining methods are classified as Inferred Mineral Resources. Infill drilling has been completed for a portion of ATS at a nominal drillhole spacing of 75 m x 100 m (refer to Figure 14.2), and this area has been classified as Indicated Mineral Resources. Inferred Mineral Resources are estimated outside the area of the infill drilling and above the 650 m elevation (or to an approximate depth below topography of 500 m).

Grade and lithological continuity at the infill drilling density appear to be reasonably predictable. No Indicated Mineral Resources exist outside of the area of infill drilling because the drillhole spacing is often too wide to assume continuity of lithology (and thereby grade, which depends on lithology) between points of observation.

A 0.10% sulphide Ni cut-off was selected to declare Mineral Resources that are amenable to open-pit mining methods, as the total precious and base metals grade (metal content) of the blocks above this cut-off grade are considered to cover projected conceptual operating costs.

Table 14.22 provides a tabulation of Mineral Resources that could be mined using open-pit methods.

Table 14.22 Indicated and Inferred Mineral Resources that are Amenable to Open-Pit Mining Methods

Property/Deposit	Mt	% Ni Sulphide	% Cu Sulphide	Pt (g/t)	Pd (g/t)	Au (g/t)	2PE + Au (g/t)
ATS – Indicated							
Turfspruit 241 KR	470	0.20	0.14	0.34	0.45	0.09	0.87
Rietfontein 2 KS	40	0.21	0.17	0.28	0.41	0.09	0.78
Total ATS Indicated	**520**	**0.20**	**0.14**	**0.33**	**0.44**	**0.09**	**0.86**
		Contained Ni Sulphide (Mlb)	**Contained Cu Sulphide (Mlb)**	**Contained Pt (Moz)**	**Contained Pd (Moz)**	**Contained Au (Moz)**	**Contained 2PE + Au (Moz)**
Turfspruit 241 KR		2,038	1,484	5.14	6.80	1.31	13.2
Rietfontein 2 KS		202	161	0.39	0.57	0.12	1.08
Total ATS Indicated		**2,240**	**1,645**	**5.53**	**7.37**	**1.43**	**14.3**
ATS – Inferred							
	Mt	**% Ni Sulphide**	**% Cu Sulphide**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**2PE + Au (g/t)**
Turfspruit 241 KR	260	0.16	0.10	0.41	0.47	0.10	0.97
Rietfontein 2 KS	0	0.00	0.00	0.00	0.00	0.00	0.00
Total ATS Inferred	**260**	**0.16**	**0.10**	**0.41**	**0.47**	**0.10**	**0.97**
		Contained Ni Sulphide (Mlb)	**Contained Cu Sulphide (Mlb)**	**Contained Pt (Moz)**	**Contained Pd (Moz)**	**Contained Au (Moz)**	**Contained 2PE + Au (Moz)**
Turfspruit 241 KR		899	589	3.41	3.88	0.80	8.10
Rietfontein 2 KS		0	0	0.0	0.0	0.0	0.00
Total ATS Inferred		**899**	**589**	**3.41**	**3.88**	**0.80**	**8.1**
AMK – Inferred							
	Mt	**% Ni Sulphide**	**% Cu Sulphide**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**2PE + Au (g/t)**
Macalacaskop 243 KR	250	0.17	0.11	0.52	0.55	0.10	1.18
		Contained Ni Sulphide (Mlb)	**Contained Cu Sulphide (Mlb)**	**Contained Pt (Moz)**	**Contained Pd (Moz**	**Contained Au (Moz**	**Contained 2PE + Au (Moz**
Macalacaskop 243 KR		913	581	4.20	4.45	0.84	9.49

Total – Open-Pit (AMK + ATS)							
	Mt	**% Ni Sulphide**	**% Cu Sulphide**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**2PE + Au (g/t)**
Indicated	520	0.20	0.14	0.33	0.44	0.09	0.86
Inferred	510	0.16	0.10	0.46	0.51	0.10	1.07
		Contained Ni Sulphide (Mlb)	**Contained Cu Sulphide (Mlb)**	**Contained Pt (Moz)**	**Contained Pd (Moz)**	**Contained Au (Moz)**	**Contained 2PE + Au (Moz)**
Indicated		2240	1645	5.53	7.37	1.43	14.3
Inferred		1812	1171	7.61	8.34	1.64	17.6

Notes:

1. Mineral Resources have an effective date of 31 March 2011. The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME.
2. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from 650 m elevation to surface (approximately 500 m depth extent). A selective mining unit (SMU) of 15 m x 15 m x 10 m has been assumed. External dilution has not been applied. Indicated Mineral Resources are based on an area drilled on approximately 75 m x 100 m spacings. Inferred Mineral Resources are based on an area drilled on approximately 120 m x 140 m spacings.
3. Mineral Resources are reported assuming a 0.10% sulphide nickel cut-off grade. At a 0.1% sulphide nickel cut-off grade, the mineralization is continuous.
4. The 0.10% sulphide Ni cut-off grade is based on assumed costs and metal prices for the purposes of assessing reasonable prospects of economic extraction. Commodity prices were assumed to be Ni: $9.20/lb, Cu: $3.00/lb, Pt: $1,785/oz, Pd: $650/oz, Au: $1,265/oz. Concentrator, G&A and concentrate transport costs are estimated to average $11/t of mill feed for a conceptual 10 Mt/a operation. Mining costs are estimated at an average of $5/t.
5. Totals may not sum due to rounding.
6. 2PE + Au = Pt + Pd + Au.

14.7.6 Comments on Mineral Resources Amenable to Open-Pit Mining Methods

Mineral Resources should at least pay for concentrator, site G&A and concentrator transport costs, which are assumed to be approximately $11/t. An $11/t NSR cut-off is approximately equivalent to a 0.05% sulphide nickel cut-off.

There has been little metallurgical testwork on samples below 0.15% sulphide nickel (AMEC, 2003); hence the Mineral Resources base-case for the mineralization that may be amenable to open-pit extraction methods are declared at a 0.10% sulphide nickel cut-off grade. This cut-off grade is approximately equivalent to an NSR cut-off grade of $25/t (ATS) to $30/t (AMK).

The current drillhole spacing for Indicated Mineral Resources (75 m x 100 m) is sufficient to assume continuity of geology and mineralization between points of observation. The drillhole spacing will need to be reduced in order to support a feasibility study.

The aim would be to increase the overall resource estimation confidence for the deposit, with the intention of improving the classification of some or all of the Mineral Resources from Indicated to Measured, and potentially from Inferred to Indicated.

Additional detailed drilling is required to evaluate the effect of block boundary loss and external dilution at contacts between the Platreef and floor rocks, norite/pyroxenite and xenoliths, and serpentinized pyroxenite and xenoliths.

This could result in a downside impact on grade and/or tonnage of Mineral Resources, which is not able to be predicted with the current data. Alternatively the ability to mine more selectively on a local basis should be studied, and this could have a positive impact on future Project economics.

Under the 2010 CIM Definition Standards (CIM, 2010), a Mineral Resource must have reasonable prospects for economic extraction. The Inferred Mineral Resources at both Turfspruit and Macalacaskop lie within 500 m of the surface and are potentially accessible by open-pit mining methods. The Indicated Mineral Resources have been shown to fall within a conceptual pit shell. Part of the Inferred Mineral Resources lies below the conceptual pit shell. Scoping studies are required to define the interface between the open-pit and underground methods, and there is a risk that a portion of the Inferred Mineral Resources may be shown to be sub-economic.

14.7.7 Bikkuri Reef Resource Estimate

Table 14.23 provides the total Mineral Resource estimate for minerallization lying within and adjacent to grade shells for the Bikkuri Reef. Table 14.24 summarizes the Mineral Resources considered amenable to selective underground mining methods for the Bikkuri Reef that fall within the grade shells (within grade shells). Table 14.25 presents the Mineral Resources considered amenable to selective underground mining methods that are outside the grade shells (adjacent grade shells) in the Bikkuri Reef.

Table 14.23 Mineral Resource Estimates for the Bikkuri Reef Within and Adjacent to Grade Shells Assuming Selective Underground Mining Methods (basecase is highlighted)

Tonnage and Grades, Indicated Mineral Resources								
Grade Shells	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**Rh (g/t)**	**3PE+Au (g/t)**	**Ni (%)**	**Cu (%)**
3 g/t	2.3	1.67	1.45	0.36	0.09	3.57	0.40	0.22
2 + 3 g/t	5.6	1.34	1.20	0.30	0.08	2.92	0.36	0.20
1 + 2 + 3 g/t	17.2	0.83	0.79	0.19	0.05	1.86	0.27	0.15
Contained Metal, Indicated Mineral Resources								
Grade Shells	–	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Ni (Mlbs)**	**Cu (Mlbs)**
3 g/t	–	0.12	0.11	0.03	0.01	0.26	20.2	11.1
2 + 3 g/t	–	0.24	0.22	0.05	0.01	0.52	44.0	24.4
1 + 2 + 3 g/t	–	0.46	0.44	0.10	0.03	1.03	101.7	58.2
Tonnage and Grades, Inferred Mineral Resources								
Grade Shells	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**Rh (g/t)**	**3PE+Au (g/t)**	**Ni (%)**	**Cu (%)**
3 g/t	0.9	1.57	1.41	0.37	0.08	3.42	0.40	0.21
2 + 3 g/t	2.3	1.30	1.16	0.31	0.07	2.84	0.34	0.18
1 + 2 + 3 g/t	10.0	0.75	0.73	0.18	0.04	1.70	0.25	0.14
Contained Metal, Inferred Mineral Resources								
Grade Shells	–	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Ni (Mlbs)**	**Cu (Mlbs)**
3 g/t	–	0.05	0.04	0.01	0.00	0.10	7.8	4.0
2 + 3 g/t	–	0.10	0.09	0.02	0.01	0.21	17.4	9.1
1 + 2 + 3 g/t	–	0.24	0.23	0.06	0.01	0.55	55.0	30.6

Note:

1. Mineral Resources have an effective date of 8 May, 2013. The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME.
2. Mineral Resources estimated assuming underground selective mining methods for the Bikkuri Reef are exclusive of the Mineral Resources estimated assuming mass-mining methods and the Mineral Resources estimated within and adjacent to the TCU. The grade shell rows are also not additive. The 3 g/t 2PE + Au shell is included in the 2 + 3 g/t 2PE + Au shell, which is in turn included in the 1 + 2 + 3 g/t 2PE + Au shell. The 2 + 3 g/t 2PE + Au shell is considered the base case estimate.
3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately 400 m to 800 m elevation. Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 m x 400 m (locally to 400 m x 200 m and 200 m x 200 m) spacing.
4. Reasonable prospects of economic extraction were determined using assumed commodity prices of Ni: $8.81/lb, Cu: $2.73/lb, Pt: $1,699/oz, Pd: $667/oz, Au: $1,315/oz, Rh: $2.065/oz. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $40/t) and process, G&A, and concentrate transport costs (average $12.50/t of mill feed for a 4 Mtpa operation) would be covered. The process recoveries vary with block grade but typically would be 85–90% for Pt, Pd and Rh; 75% for Au and 70% for Ni and 85% for Cu. The Mineral Resources within the 1+2+3, 2+3 or 3 g/t 2PE + Au grade shells have been estimated (at nominal cut-off grades of 1, 2, and 3 g/t 2PE + Au respectively) to show sensitivity to cut-off grade and to provide multiple options for consideration in future mining studies. No allowances for mining recovery and external dilution have been applied.
5. Totals may not sum due to rounding.
6. In this table 3PE + Au = Pt + Pd + Rh + Au. 2PE + Au = Pt + Pd + Au.

Table 14.24 Mineral Resource Estimates for the Bikkuri Reef Within Grade Shells Assuming Selective Underground Mining Methods (base case is highlighted)

Tonnage and Grades, Indicated Mineral Resources								
Grade Shells	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**Rh (g/t)**	**3PE+Au (g/t)**	**Ni (%)**	**Cu (%)**
3 g/t	2.48	1.57	1.38	0.35	0.09	3.38	0.39	0.21
2 + 3 g/t	5.09	1.34	1.19	0.30	0.07	2.89	0.35	0.20
1 + 2 + 3 g/t	12.2	0.91	0.84	0.21	0.05	2.02	0.28	0.16
Contained Metal, Indicated Mineral Resources								
Grade Shells	**–**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Ni (Mlbs)**	**Cu (Mlbs)**
3 g/t	–	0.13	0.11	0.03	0.01	0.27	21.1	11.6
2 + 3 g/t	–	0.22	0.19	0.05	0.01	0.47	39.5	21.8
1 + 2 + 3 g/t	–	0.36	0.33	0.08	0.02	0.79	75.4	42.6
Tonnage and Grades, Inferred Mineral Resources								
Grade Shells	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**Rh (g/t)**	**3PE+Au (g/t)**	**Ni (%)**	**Cu (%)**
3 g/t	0.82	1.51	1.41	0.33	0.08	3.34	0.38	0.19
2 + 3 g/t	2.31	1.28	1.14	0.30	0.07	2.78	0.33	0.17
1 + 2 + 3 g/t	5.88	0.85	0.79	0.20	0.05	1.89	0.26	0.14
Contained Metal, Inferred Mineral Resources								
Grade Shells	**–**	**Pt (Moz)**	**Pd (Moz)**	**Au (Moz)**	**Rh (Moz)**	**3PE+Au (Moz)**	**Ni (Mlbs)**	**Cu (Mlbs)**
3 g/t	–	0.04	0.04	0.01	0.00	0.09	6.78	3.45
2 + 3 g/t	–	0.09	0.08	0.02	0.00	0.21	17.0	8.76
1 + 2 + 3 g/t	–	0.16	0.15	0.04	0.01	0.36	33.8	18.0

Notes:

1. Mineral Resources have an effective date of 8 May, 2013. The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME.
2. Mineral Resources estimated assuming underground selective mining methods for the Bikkuri Reef inside the grade shells are exclusive of the Mineral Resources estimated outside the grade shells, Mineral Resources estimated assuming mass-mining methods, and the Mineral Resources estimated within and adjacent to the TCU. The grade shell rows are also not additive. The 3 g/t 2PE + Au shell is included in the 2 + 3 g/t 2PE + Au shell, which is in turn included in the 1 + 2 + 3 g/t 2PE + Au shell. The 2 + 3 g/t 2PE + Au shell is considered the base case estimate.
3. Mineral Resources are reported on a 100% basis. Mineral Resources are stated from approximately 400 m to 800 m elevation. Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 m x 400 m (locally to 400 m x 200 m and 200 m x 200 m) spacing.
4. Reasonable prospects of economic extraction were determined using assumed commodity prices of Ni: \$8.81/lb, Cu: \$2.73/lb, Pt: \$1,699/oz, Pd: \$667/oz, Au: \$1,315/oz, Rh: \$2.065/oz. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average \$40/t) and process, G&A, and concentrate transport costs (average \$12.50/t of mill feed for a 4 Mt/a operation) would be covered. The process recoveries vary with block grade but typically would be 85–90% for Pt, Pd and Rh; 75% for Au and 70% for Ni and 85% for Cu. The Mineral Resources within the 1+2+3, 2+3 or 3 g/t 2PE + Au grade shells have been estimated (at nominal cut-off grades of 1, 2, and 3 g/t 2PE + Au respectively) to show sensitivity to cut-off grade and to provide multiple options for consideration in future mining studies. No allowances for mining recovery and external dilution have been applied.
5. Totals may not sum due to rounding.

6. In this table 3PE + Au = Pt + Pd + Rh + Au. 2PE + Au = Pt + Pd + Au.

Table 14.25 Mineral Resources for the Bikkuri Reef Adjacent Grade Shells Assuming Selective Underground Mining Methods

Tonnes and Grade Indicated Mineral Resource								
Mineralized Zone	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**Rh (g/t)**	**3PE+Au (g/t)**	**Ni (%)**	**Cu (%)**
HW	0.03	1.12	0.93	0.32	0.06	2.42	0.25	0.13
Middle	0.04	1.01	0.91	0.24	0.06	2.23	0.31	0.17
FW	0.31	0.94	1.14	0.18	0.06	2.33	0.34	0.19
Total	**0.38**	**0.96**	**1.10**	**0.20**	**0.06**	**2.32**	**0.33**	**0.18**
Tonnes and Grade Inferred Mineral Resource								
Mineralized Zone	**Mt**	**Pt (g/t)**	**Pd (g/t)**	**Au (g/t)**	**Rh (g/t)**	**3PE+Au (g/t)**	**Ni (%)**	**Cu (%)**
HW	0.02	0.92	0.91	0.24	0.05	2.11	0.31	0.15
Middle	0.01	1.02	0.91	0.24	0.06	2.22	0.32	0.16
FW	0.12	0.99	0.92	0.22	0.07	2.20	0.32	0.20
Total	**0.15**	**0.98**	**0.91**	**0.23**	**0.07**	**2.19**	**0.32**	**0.19**

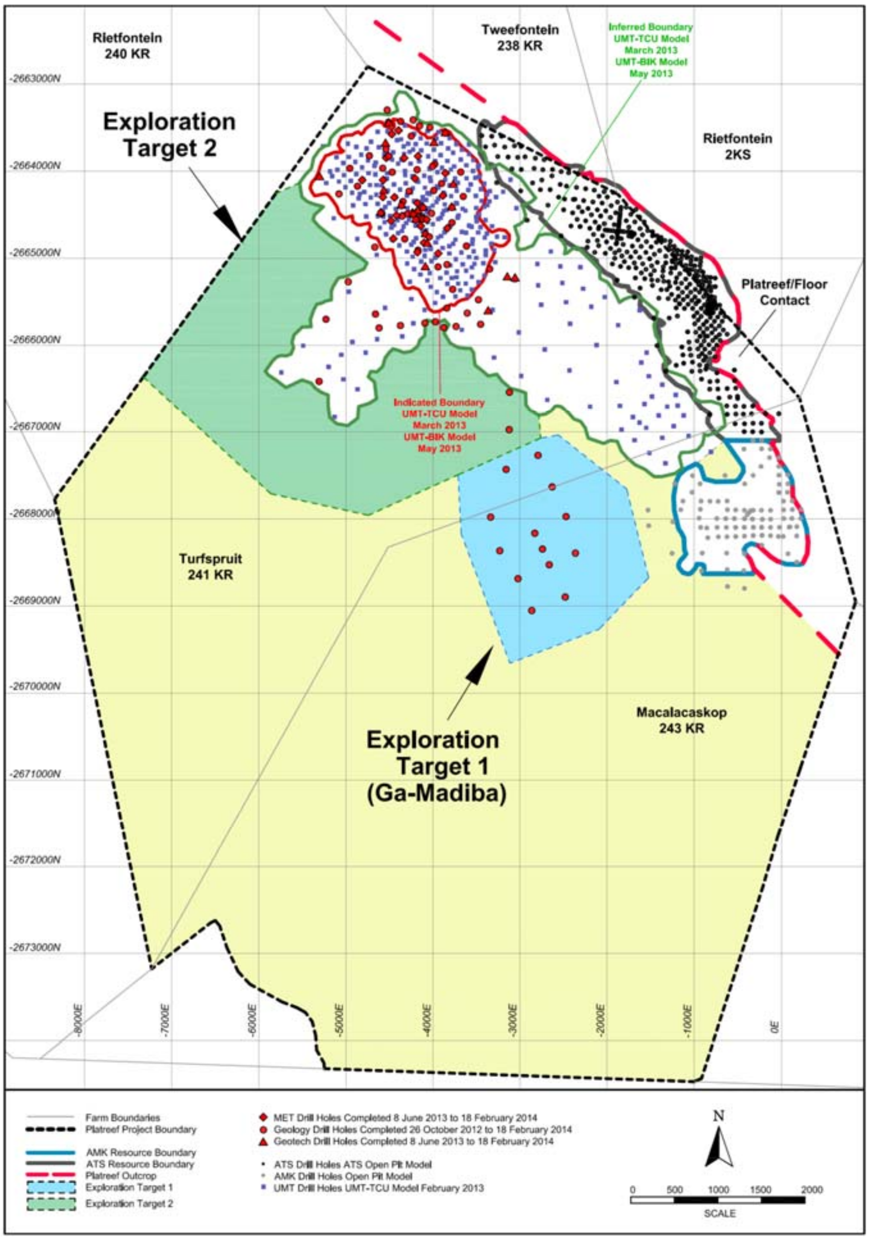
Notes:

1. Mineral Resources have an effective date of 8 May, 2013. The Qualified Persons for the estimate are Dr Harry Parker, RM SME, and Mr Timothy Kuhl, RM SME.
2. HW = above B1MZ, Middle = between B1MZ and B2MZ, FW = below B2MZ and above the TCU. Mineral Resources estimated assuming underground selective mining methods for the Bikkuri Reef adjacent to the grade shells are exclusive of the Mineral Resources estimated inside the grade shells at Bikkuri, Mineral Resources estimated assuming mass-mining methods, and the Mineral Resources estimated within and adjacent to the TCU.
3. Mineral Resources are reported on a 100% basis and at a cut-off grade of 2 g/t 3PE+Au. Mineral Resources are stated from approximately -200 m to 650 m elevation. Indicated Mineral Resources are drilled on approximately 100 x 100 m spacing; Inferred Mineral Resources are drilled on 400 m x 400 m (locally to 400 m x 200 m and 200 m x 200 m) spacing.
4. Reasonable prospects of economic extraction were determined using assumed commodity prices of Ni: $8.81/lb, Cu: $2.73/lb, Pt: $1,699/oz, Pd: $667/oz, Au: $1,315/oz, Rh: $2,065/oz. It has been assumed that payable metals would be 82% from smelter/refinery and that mining costs (average $40/t) and process, G&A, and concentrate transport costs (average $12.50/t of mill feed for a 4 Mtpa operation) would be covered. The process recoveries vary with block grade but typically would be 85–90% for Pt, Pd and Rh; 75% for Au and 70% for Ni and 85% for Cu. No allowances for mining recovery and external dilution have been applied.
5. Totals may not sum due to rounding.
6. 3PE + Au = Pt + Pd + Rh + Au.

14.8 Exploration Targets

Beyond the current Mineral Resources, mineralization is open to expansion to the south and west. Two exploration targets have been identified (Figure 14.26).

Figure 14.26 Exploration Targets



Note: Figure prepared by AMEC, 2014. Coordinates use the WGS system.

Target 1 is based on results from 14 wide-spaced step-out drillholes completed between 26 October 2012 and 18 February 2014. Target 1 could contain up to an additional 115 to 235 Mt grading 3.1 to 4.5 g/t 3PE+Au (comprising 1.2 to 1.7 g/t Pt, 1.7 to 2.3 g/t Pd, 0.06 to 0.14 g/t Rh, 0.17 to 0.26 g/t Au), 0.23% to 0.28% Ni and 0.11% to 0.14% Cu over an area of 3.7 km^2. The tonnage and grades are based on intersections of 2 g/t 3PE + Au mineralization in drillholes completed in Target 1.

Target 2, which surrounds the currently estimated mineral resources in Zones 1 and 2, could contain an estimated additional 260 to 450 Mt grading 3.4 to 4.5 g/t 3PE+Au (comprising 1.7 to 2.4 g/t Pt, 1.2 to 1.6 g/t Pd, 0.14 to 0.20 g/t Rh, 0.26 to 0.33 g/t Au), 0.30% to 0.35% Ni and 0.15% to 0.18% Cu over an area of 7.6 km^2. The tonnage and grades are based on 2 g/t 3PE+Au intersections of mineralization in 19 widely-spaced drillholes completed in Target 2 and adjacent drillholes within the Inferred Mineral Resources. These drillholes were also completed between 26 October 2012, and 18 February 2014.

AMEC 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.

Beyond these exploration target areas is approximately 37 km^2 of unexplored ground on the property under which the Platreef is projected to lie. It is not possible to estimate a range of tonnages and grades for this ground. There is excellent potential for mineralization to significantly increase with further step-out drilling to the south-west.

14.9 Comments on Section 14

Dr Parker and Mr Kuhl are of the opinion that the Mineral Resources for the Project, which have been estimated using core-drill data, have been performed to industry best practices (CIM, 2003), and conform to the requirements of CIM Definition Standards, 2010.

Since the commencement of exploration in the UMT area, iterative Mineral Resource estimates between 2010 and 2011 have led to a progressive increase in the tonnage of Inferred Mineral Resources. With the inclusion of results from the ongoing drill programme in an update of the block model, higher confidence categories upgrades are supported, and should permit completion of more detailed mining studies.

As noted in Section 7.5, 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.Recent investigations have shown that the bedding angles in core indicate layers steepen in these areas suggesting flatter-lying zones are connected to each other by slopes. In future model updates, this interpretation will obviate the need for the inclusion of the many small faults contained in the 2013 model. Locally the faulting may reduce the thickness and grade of the SMZs; however, this local variability is not expected to materially impact the Mineral Resource estimate.

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

- Confirmation of the renewal of the Rietfontein prospecting licence has not been received. The Mineral Resources amenable to open-pit methods on Macalacaskop are not expected to be affected; however, Mineral Resources amenable to open-pit methods as declared for Turfspruit and Rietfontein would have to be re-evaluated without a valid prospecting licence on Rietfontein.
- Confirmation that a mining right will be granted for the Macalacaskop and Turfspruit area; the application has been lodged and grant was pending at the Report effective date.
- Assumptions as to the amenability of local communities to allow surface access for Ivanhoe exploration and sampling programmes with appropriate negotiation and compensation.
- Permitting, environmental, legal and socio-economic assumptions.
- Assumptions used to generate the conceptual data for consideration of reasonable prospects of 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 mineralization layers may result in changes to the metallurgical recovery and smelter payables assumptions used to evaluate reasonable prospects of economic extraction.

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 mineralization 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% to 30%, and mining recoveries range from 70% to 100% using the mining methods considered for evaluation of reasonable prospects of economic extraction.

AMEC 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.
- Combining the ATS, AMK and UMT (bulk mineable) resource models into a single model or if not feasible, separate models defined on a common basis.

- Revising metallurgical recovery equations for ATS and AMK so they are on a total nickel basis.

This will put all models on the same litho-stratigraphic and assay (total) basis.

15 MINERAL RESERVE ESTIMATES

There are no Mineral Reserves in the Platreef 2014 PEA. The Platreef 2014 PEA 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 categorized as Mineral Reserves, and there is no certainty that the results will be realized.

16 MINING METHODS

16.1 Mine Geotechnical

A geotechnical study has been completed for the PLatreef 2014 PEA. The Platreef Project is located in Mokopane, Limpopo Province, South Africa. The primary objectives of the study were to:

- Make use of selected exploration boreholes to facilitate the assimilation of geotechnical data.
- Select representative core specimens from the respective lithological units for geomechanical testing, specifically Uniaxial Compressive Strength (UCS) testing.
- Carry out both empirical and numerical analyses in order to propose an appropriate mining method, identify potential geotechnical risks and derive preliminary mining spans and support requirements.

Eight exploration boreholes, equating to some 1,515 m of core, were geotechnically logged on site, and the quality of the rock mass was classified according to two rock mass classification systems, namely Laubscher's (1990) Mining Rock Mass Rating (MRMR) and the Norwegian Geotechnical Institute's Q-System (Barton et al, 1974).

Typically, three joint sets are developed within the mining area, with micro-joint surfaces generally being rough undulating. Infill, when present, is typically chloritic in nature. Serpentinised joints, while not common, were identified in borehole UMT075.

Analyses also included the identification of potential zones of weakness, with all broken and highly sheared and serpentinised zones, together with zones of persistent sub-vertical jointing, being noted.

Currently, not a lot is known about the groundwater regime across the mining area. However, this aspect of the project is currently being investigated by Golder Associates.

Laboratory test results on twenty-three representative core specimens indicated UCS values of between 55 MPa (Olivine Pyroxenite) and 257 MPa (Melagabbronorite).

RMR and MRMR values derived for the respective lithological units varied according to rock type, with pyroxenite having the lowest value (25) and gabbro norite having the highest (94). Similarly, the MRMR values were between 23 and 84 respectively. Mean RMR and MRMR values varied between 49 (Para Pyroxenite) and 64 (Para B Pyroxenite) with respect to RMR values and 44 and 57 with respect to MRMR values.

In terms of Q' values, values varied between 0 (pyroxenite) and 400 (pyroxenite), with a mean value of 20. The mean Q' for the mining zone is 23. The Matthews/Potvin Stability Graph for supported stopes was used to gain an appreciation of stope stability and Barton's Q chart was used for development support.

Mining of the Flatreef will be between a depth of 800 m to 1,000 m. The horizontal to vertical stress ratio is not known and could be as high as 2, but is expected to tend towards 1.0 at a depth of 1,000 m. The height of the target mining zone is determined from the grade distribution in the deposit.

Mining methods will depend on the height of the target zone. Longhole stope dimensions considered at this stage are 15 m wide by the height of the target stoping zone. The stope length will range between 40 m and 60 m, ensuring that a hydraulic radius of 8, for the stope walls, is not exceeded. Stope back support will comprise 4 m long cables to ensure stability of the stope back and to minimise dilution.

A primary, secondary stoping sequence with post filling of all stopes is recommended. Where the target stoping zone is narrower, benching, and drift-and-fill methods will be used. Mesh or shotcrete should be installed in the backs, where personnel will be exposed.

Backfill will need to be free-standing during secondary stope extraction. Initial calculations indicate that a backfill strength of 300 kPa will be required over a 20 m height, with a 790 kPa strength being required over a 40 m height.

A shaft pillar will be required to protect the shaft. A preliminary estimate of the shaft pillar radius is 250 m for the dual shaft system and 200 m for the ventilation shafts. This will need to be verified during the pre-feasibility study. Early shaft reef extraction can be considered as an alternative to leaving a shaft pillar.

Mine development will require rockbolts installed in a 1.5 m x 1.5 m pattern with mesh or shotcrete. The length of support should be at least half of the excavation width.

16.1.1 Introduction

SRK Consulting (Vancouver) (Pty) Ltd submitted a proposal to Ivanhoe Mines in Canada for SRK to undertake a Geotechnical Investigation for the Platreef Project, Limpopo Province, South Africa. SRK Consulting (South Africa) (Pty) Ltd (SRK) formed an integral part of the project team by providing experienced technical analytical capability and QA/QC for the project, with SRK (Vancouver) providing a peer review function.

16.1.2 Geotechnical Objectives

The primary objectives of the study were:

- To assimilate geotechnical data from existing, selected exploration boreholes to facilitate the quantification of the quality of the rock mass.
- To identify representative core specimens from the various lithological units intersected in the boreholes in order to carry out strength tests on the specimens, thereby gaining a quantitative appreciation of the strength characteristics of the various lithological units.
- To carry out empirical and numerical analyses in order to propose an appropriate mining method, identify potential geotechnical risks and to derive preliminary mining spans and support requirements.

16.1.3 Scope of Work

The scope for the Geotechnical Investigation included, inter alia:

- An initial site visit to liaise with Platreef Resources (Pty) Ltd personnel on the ground, to gain an appreciation of the quantity and quality of data available for the Platreef 2014 PEA and to visit the site of the proposed underground mining operation.
- A review of all available geological and exploration data pertaining to the Platreef Platreef 2014 PEA.
- The selection of representative exploration boreholes for geotechnical borehole logging purposes, with a view to achieving an adequate coverage of the mining area.
- The geotechnical logging of eight (8) exploration boreholes of which four were selected from the northern portion, and four from the southern portion of the Mine 1 Area (now known as Zone 1) respectively.
- The identification, selection and submission of representative core specimens from the respective lithological units intersected in the boreholes from the northern portion of the Mine 1 Area, the southern exploration boreholes having already been assayed.
- The identification of discrete geotechnical domains and the characterization of the rock mass quality of each geotechnical domain using both the Laubscher (1990) Mining Rock Mass Rating Classification System and the Norwegian Geotechnical Institutes Q-System (Barton et al, 1974).

16.1.4 Geotechnical Project Team

The project team comprised the SRK team members outlined in Table 16.1.

Table 16.1 SRK Project Team

Mr. Alan Naismith	Principal Mining Geotechnical Engineer and Partner (Internal Review)
Mr. William Joughin	Principal Mining Geotechnical Engineer and Partner
Mr. Greg Dyke	Principal Engineering Geologist (Project Manager)
Mr. Robert Armstrong	Senior Structural Geologist
Ms. Sharla Coetsee	Engineering Geologist
Mr Ovy Pillay	Engineering Geologist
Ms. Candice Maduray	Engineering Geologist
Mr. Rory Bush	Geologist
Mr Peter Terbrugge	Principal Consultant

16.1.5 Geotechnical Core Logging

A total of eight (8) exploration boreholes, deemed to be representative of the Mine 1 Area, were geotechnically logged on site by a representative of SRK. Of the eight boreholes, four were selected from the northern portion of the mining area (UMT 109, UMT 123, UMT 130 and UMT 146) and four from the southern portion of the mining area (UMT 070, UMT 075, UMT 096 and UMT 100). In approaching the geotechnical logging of the exploration boreholes for the Platreef 2014 PEA, the approach was to only:

- Geotechnically log the upper 50 m of the immediate hangingwall to the mining zone.
- Geotechnically log the mining zone.
- Geotechnically log the immediate 50 m of footwall below the mining zone.

In total, some 1,515 m of core was geotechnically logged for the Platreef 2014 PEA. A summary of the geotechnical logging is presented as Table 16.2.

Table 16.2 Summary of Geotechnical Core Logging for Rock Mass Classification

Borehole No.	Portion of Mine 1 Area	Depth (m)		Total Metres Logged (m)
		From	To	
UMT 070	Southern	757.43	981.43	224.00
UMT 075	Southern	733.00	891.00	158.00
UMT 096	Southern	779.00	924.20	145.00
UMT 100	Southern	776.00	923.00	147.00
UMT 109	Northern	856.38	1056.38	200.00
UMT 123	Northern	718.42	864.48	146.06
UMT 130	Northern	695.43	876.27	180.84
UMT 146	Northern	386.00	546.00	160.00
UMT 146	Northern	612.00	766.00	154.00
Total Metres Logged				**1,514.90**

The geotechnical logging entailed the assimilation of data with respect to:

- The identification and vertical extent of geotechnical zones, specifically within the hangingwall units and the mining zone.
- The Total and Solid Core Recovery (TCR and SCR).
- The Rock Quality Designation (RQD).
- The quality of solid rock versus weak rock.
- Rock competence, i.e. the ratio of solid rock to matrix and the quality of the matrix material defects (faults, shear zones, intense fracturing and zones of deformable material).
- The degree and nature of rock weathering.
- The Intact Rock Strength (IRS)/hardness.

- The relative orientation of rock structures.
- The spacing between the sets of rock structures.
- The total number/density/frequency of structures.
- The condition of structures, i.e. roughness profile, wall alteration and infilling.
- The groundwater conditions.

16.1.6 Joint Analysis

- Joint Sets

  A total of 336 joint readings were taken from the eight exploration boreholes that were geotechnically logged. An analysis of the joint orientation data indicates that, generally, three joints sets are developed, with 47% of the joint orientation data comprising three sets. In those instances were only two sets are developed (41%), the orientation of the joint sets are oblique and sub-horizontal to the core axis. Where only one joint set is developed, the orientation of the joint sets is typically oblique to the core axis.

- Micro Joint Condition

  A total of 786 joint surfaces were inspected during the geotechnical logging of the exploration cores. Analysis of the joint data shows that, typically, micro-joint surfaces are rough undulating (41%) followed by rough planar (25%), slickensided undulating (18%) and smooth planar (10%), which make up the remainder of the predominant micro-joint conditions.

- Infill

  When present, infill material is typically chloritic in nature and classifies as a soft-sheared fine material. Serpentinised joints were identified in borehole UMT075, between a depth of 828 m and 830 m, in a zone of highly sheared pyroxenite. Although chlorite infill dominates, non-softening fill material, in the form of calcite, is also developed on joint surfaces, mainly in boreholes UMT123 and UMT130.

16.1.7 Potential Structural Weaknesses

Included in the geotechnical analysis was the identification of zones of structural weakness that may negatively impact the hangingwall and pillars during mining operations. To this end, zones of veining, fracturing and shearing were identified in each of the boreholes geotechnically logged. A summary of these poorer zones is presented in Table 16.3.

Table 16.3 Summary of Laboratory Testing Programme

Borehole No.	Depth (m)		Description
	From	To	
UMT 070	843.21	843.43	Broken zone.
	940.57	940.69	Broken zone.
UMT 075	828.64	828.66	Highly sheared; serpentinised infill.
UMT 100	807.81	808.00	Calcite vein.
	897.28	897.33	Calcite vein.
	916.81	921.40	Series of sub-vertical joints.
	921.40	921.90	Sub-vertical joint.
UMT 109	856.38	858.38	Sub-vertical joint with calcite infill.
UMT 109	882.38	888.38	Sub-vertical joint with calcite infill.
	957.20	859.08	Slickensided shear zone.
	997.21	1,002.76	Highly fractured with chlorite/sericite infill.
	1,014.38	1,020.38	Highly fractured with chlorite/sericite infill.
UMT 123	768.42	774.27	Sub-vertical joints.
	774.27	780.00	Shear zone.
UMT 130	856.38	858.38	Sub-vertical joints with calcite.
	858.38	900.38	Bad zone.
UMT 146	386.00	391.22	Sub-vertical fracture.
	537.46	537.93	Sub-vertical fracture.
	645.29	646.00	Sub-vertical fracture.
	740.01	744.36	Bad zone.

16.1.8 Goniometry Analysis

The angle of the joint to the core axis (α) and the circumference angle (β) were measured using a goniometer in order to calculate the true dip and dip direction of joints from oriented exploration borehole cores, i.e. boreholes UMT 109, UMT123, UMT130, and UMT146.
A composite equal angle, lower hemisphere contour plot of the 556 measurements is presented in Figure 16.1, from which parallel, north-west–south-east trends are evident. The stereonet plots for the individual boreholes are shown as Figure 16.2, Figure 16.3, Figure 16.4 and Figure 16.5 respectively.

Figure 16.1 Lower Hemisphere Plot

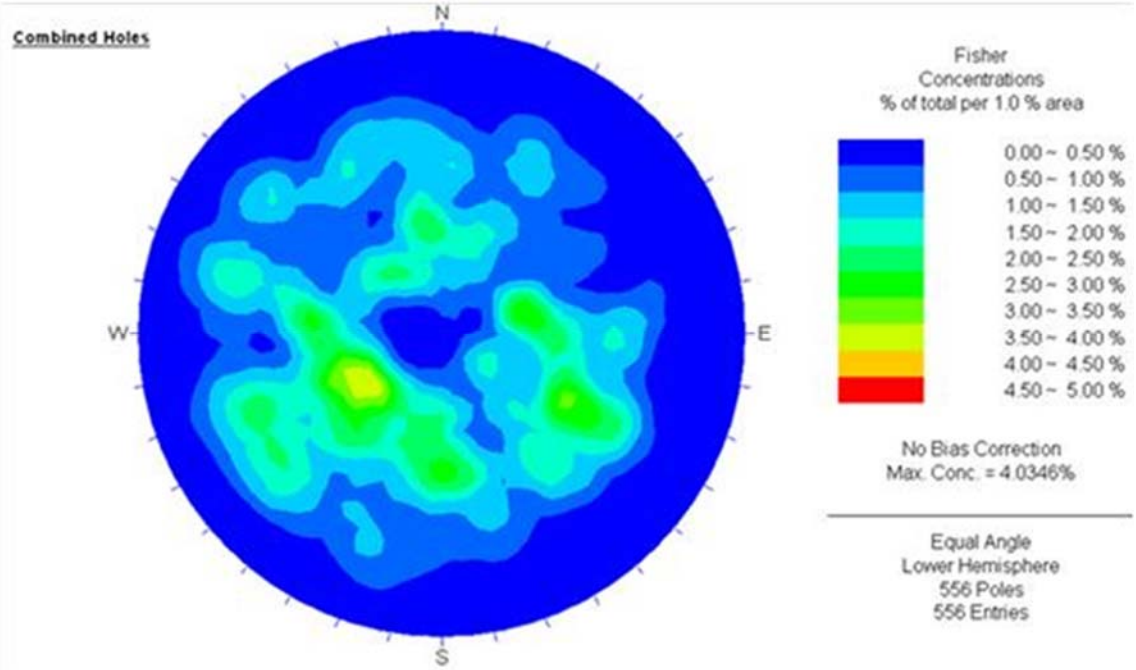


Figure 16.2 Pole Plot for Borehole UMT123

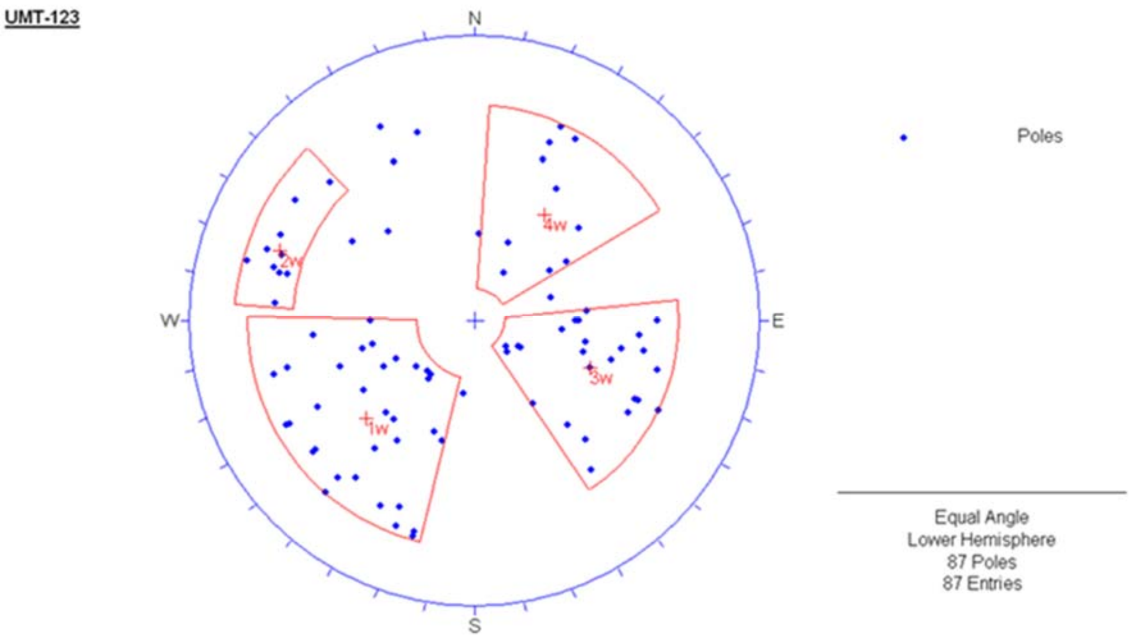


Figure 16.3 Pole Plot for Borehole UMT109

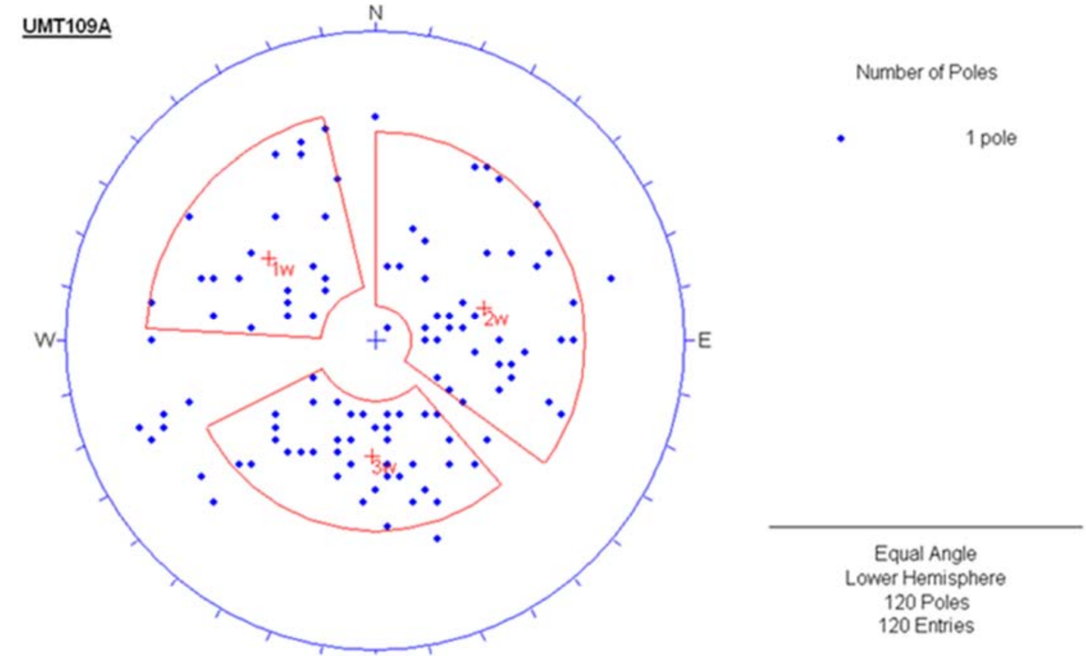


Figure 16.4 Pole Plot for Borehole UMT130

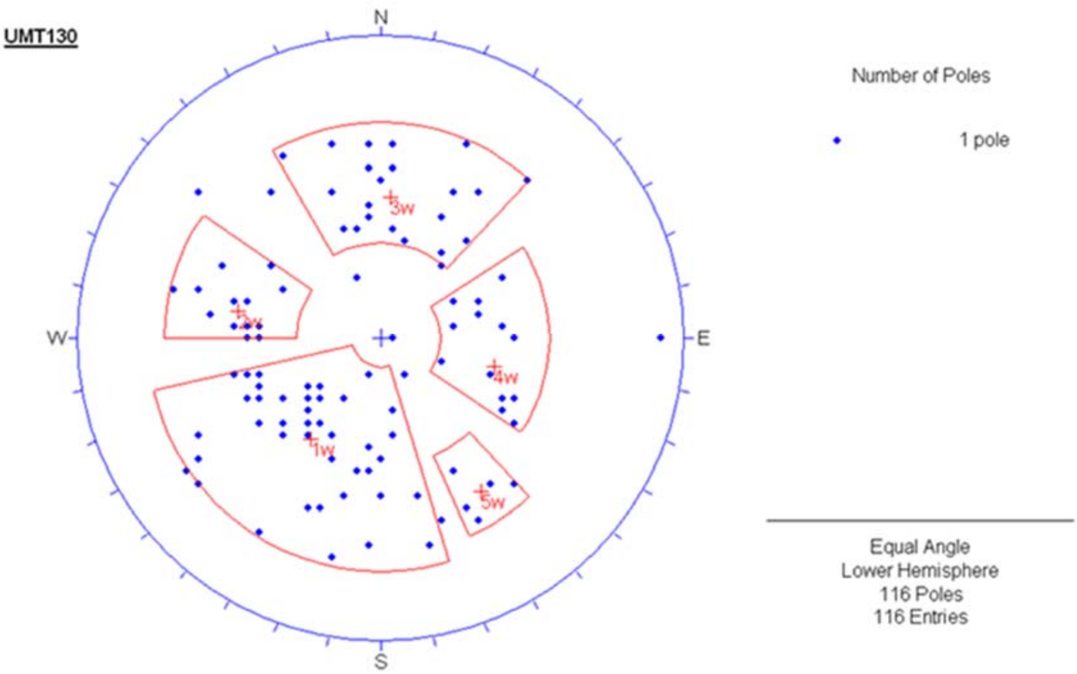
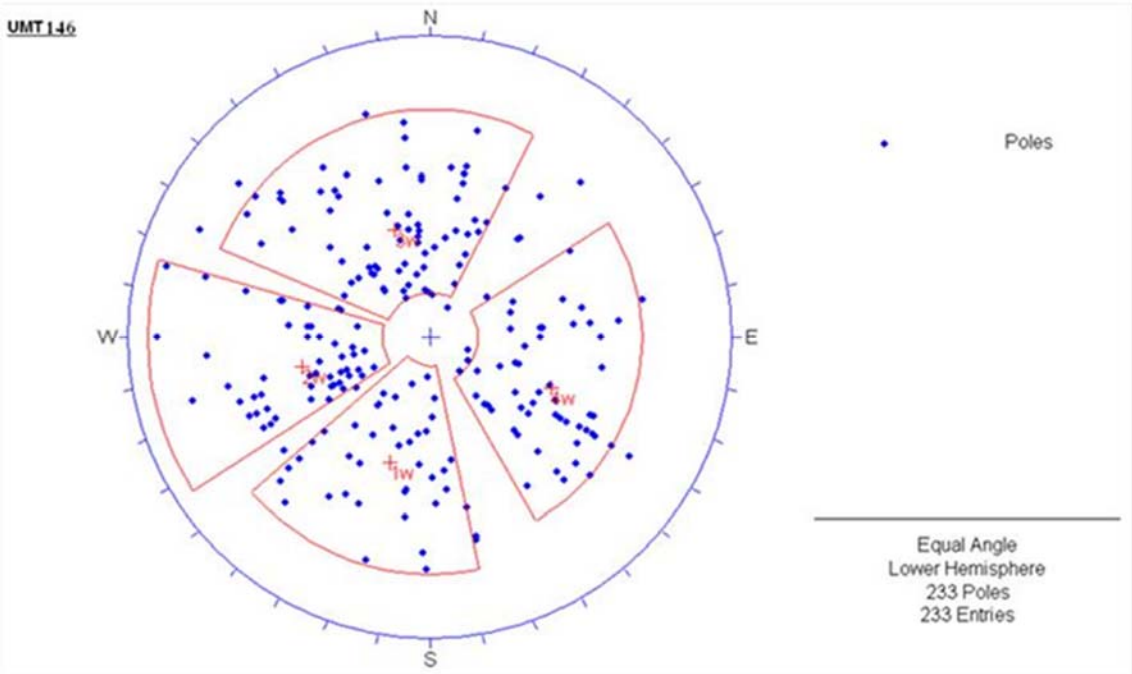


Figure 16.5 Pole Plot for Borehole UMT146



Based on an analysis of the stereonets, a summary of the primary joint sets identified is presented as Table 16.4.

Table 16.4 Summary of Joint Orientations

Borehole No.	Joint Set Orientation (Dip/Dip Direction)				
	Set 1	Set 2	Set 3	Set 4	Set 5
UMT 123	54/048	46/291	46/215	72/109	–
UMT 109	38/256	41/359	44/133	–	–
UMT 130	44/032	48/182	41/283	50/101	61/326
UMT 146	40/162	46/291	46/077	46/017	–

16.1.9 Groundwater Regime

Currently, there is limited groundwater data available for the Platreef Project, with most of what is available concentrating primarily on available water supplies for mining operations. However, Golder Associates is currently busy carrying out a hydrological baseline study for the Platreef Project. Consequently, the potential impact of groundwater on the stability of the underground workings has yet to be determined.

16.1.10 Laboratory Testing

A total of 23 representative core specimens were submitted to CSIR Miningtek for geomechanical testing, comprising Uniaxial Compression (UCS) tests and Uniaxial Compression tests with strain gauges (UCM) respectively. A summary of the laboratory test programme is presented as Table 16.5.

Table 16.5 Summary of Laboratory Testing Programme

Rock Type	Relative Mining Position	Laboratory Test Required
Gabbronorite	Hangingwall	UCS and UCM
Granite Vein	Hangingwall	UCS
Melagabbronorite	Hangingwall	UCS
Anorthosite	Hangingwall	UCS
Pyroxenite	Mining zone	UCS and UCM
Olivine Pyroxenite	Mining zone	UCS
Feldspathic pyroxenite	Mining zone	UCS
Norite	Mining zone	UCS
Altered Zone	Footwall	UCS
Pyroxenite	Footwall	UCS

A summary of the laboratory test results is presented in Table 16.6 and Table 16.7 respectively.

Table 16.6 Summary of UCS Test Results

Rock Type	Mean Density (kg/m³)	Strength (UCS) (MPa)		
		Min	Max	Mean
Gabbronorite	2650	141.8	141.8	141.8
Melagabbronorite	2875	161.2	256.5	208.9
Gabbronorite	2880	111.3	120.4	115.9
Pyroxenite	3195	64.3	207.5	127.9
Anorthosite	2860	153.9	153.9	153.9
Olivine Pyroxenite	3170	54.7	216.4	135.6
Feldspathic pyroxenite	3720	199.2	199.2	199.2
Norite	2960	86.5	86.5	86.5
Altered Zone	3180	92.2	92.2	92.2

Table 16.7 Summary of UCM Test Results

Rock Type	Mean Density (kg/m³)	Strength (UCS) (MPa)			Tangent @50% UCS		Mean Modulus Ratio
		Min	Max	Mean	Mean Poisson's Ratio	Mean Deformation Modulus (GPa)	
Gabbronorite	2980	117.8	117.8	117.8	0.161	87.8	745
Pyroxenite	3226	113.8	200.0	149.6	0.257	164.5	1030

16.1.11 Rock Mass Classification (Laubscher, 1990)

Use was made of Laubscher's (1990) Mining Rock Mass Classification System (MRMR) to characterise the quality of the rock mass. Each lithological unit was individually evaluated and discrete geotechnical domains identified, specifically with respect to fracture frequency, lithological boundaries, hardness (Intact Rock Strength, IRS) and degree of weathering. This classification system, as used by SRK, has been successfully used for the preliminary evaluation of pit slope walls and the determination of rock mass strength characteristics. The MRMR classification system is an extremely robust method of utilising all of the relevant rock mass parameters to assist with mine design.

The following classification parameters were assessed for each geotechnical domain identified:

- Core recovery.
- Rock Quality Designation (RQD).
- Rock (core) competency.
- Field estimation of intact rock strength (IRS).
- Degree of weathering.
- Number of joints grouped in 30° (0–30°, 30°–60° and 60°–90°) intervals measured from core axis.
- Total number of joints in an interval i.e. sum of all the joints.
- Joint surface conditions.

These parameters are then assessed in accordance with the MRMR classification system and are allocated ratings within the following ranges:

- IRS – 0 to 20
- RQD – 0 to 15
- Js – 0 to 25
- FF – 0 to 40
- Jc – 0 to 40

The calculation of Laubscher's (1990) rock mass ratings made use of the Intact Rock Strength (IRS), Fracture Frequency (FF) and Joint Wall Condition factor (Jc). The influence of groundwater on the shear strength of the discontinuities is initially taken into account in the selection of the parameters for joint condition. In this regard, there are the options to select either dry, moist, moderate pressure or high pressure conditions depending on whether drainage or dewatering are being considered. The insitu rock mass rating value, out of a total of 100, is then adjusted to account for the expected mining environment, i.e. weathering, structural orientation, induced stress and blasting. The adjustments to the in-situ rock mass rating is introduced in recognition of the type of excavation proposed and the time dependant behaviour of the rock mass. The possible percentage adjustments are as follows:

- Weathering – 30% to 100%
- Orientation – 63% to 100%
- Induced Stress – 60% to 120%
- Blasting – 80% to 100%

Although the percentages are empirical, the principle has proven sound, and as such, it forces the designer to allow for these important factors during the logging process. In effect, the anticipated deterioration of the rock mass, once exposed in the mine environment, is provided for in these adjustments (see Table 16.8).

Table 16.8 Summary of Rock Mass Rating Adjustments

Weathering	Joint Orientation	Induced Stress	Blasting	Total Adjustment
1.00	0.95	1.00	0.94	0.89
The oxide material is weathered.	Three joint sets plus random joints have been assumed.	Moderate stress levels anticipated (Depth 600 m to 1000 m).	Good conventional; blasting is assumed.	–

A summary of the unadjusted RMR and MRMR values for the hangingwall, mining zone and footwall, are presented in Table 16.9.

Table 16.9 Summary of RMR and MRMR Values Per Mining Unit

MRMR per Mining Unit	RMR Value			MRMR Value		
	Min	Max	Mean	Min	Max	Mean
Hangingwall	40	72	57	36	64	51
Mining zone	28	94	54	25	84	48
Footwall	25	86	53	23	77	48

However, due the variable mineralisation distribution within the mining zone, i.e. the base of the Main Zone (MZ) does not necessarily constitute the immediate hangingwall to the mining zone, rock mass rating values were also derived per lithological unit, as units such as the B-Pyroxenite may also constitute the immediate hangingwall to the mining zone. A summary of the resultant RMR and MRMR values are presented in Table 16.10.

Table 16.10 Summary of RMR and MRMR Values Per Lithological Unit

MRMR per Lithological Unit	RMR Value			MRMR Value		
	Min	Max	Mean	Min	Max	Mean
Altered Zone	49	74	59	43	66	53
Anorthosite	46	63	56	42	56	51
Calcsilicate	52	68	57	47	60	51
F Pyroxene	49	77	61	44	68	54
Gabbro Norite	40	94	58	36	84	52
GRV	42	68	56	38	60	50
Harzburgite	49	71	56	44	63	51
Hornfels	46	72	58	41	64	52
Melagabbronorite	55	72	60	49	64	54
Mottled Anorthosite	47	68	58	42	61	52
Norite	35	82	51	32	74	46
OL Pyroxene	40	64	51	36	57	46
OLFPX	44	64	53	41	57	49
Para Pyroxenite	45	60	49	40	53	44
Para B Pyroxenite	45	86	64	40	77	57
Pyroxene	25	86	52	23	77	47
Quartz Feldspathic Vein	44	66	55	39	59	50

16.1.12 Rock Mass Classification (Barton et al, 1974)

Use was also made of the Norwegian Geotechnical Institute's Q-System (1974) to facilitate the derivation of Q' (excluding Jw and SRF) and ultimately the Stability Number (N') to facilitate the evaluation of the stability of open stopes. A summary of the Q'-values, for the mining and lithological units respectively, is presented in Table 16.11 and Table 16.12 respectively.

Table 16.11 Summary of Q' Values Per Mining Unit

Q' per Mining Unit	Min	Max	Mean
Hangingwall	8	400	26
Mining zone	2	320	23
Footwall	0	71	21

Table 16.12 Summary of Q' Values Per Lithological Unit

Q' per Lithological Unit	Min	Max	Mean
Altered Zone	8	73	30
Anorthosite	10	73	30
Calcsilicate	17	37	24
F Pyroxene	8	200	31
Gabbro Norite	10	191	27
GRV	9	40	26
Harzburgite	11	66	26
Hornfels	12	65	31
M Gabbro Norite	8	36	24
Mottled Anorthosite	13	35	24
Norite	3	71	21
OL Pyroxene	8	46	16
OLFPX	18	36	27
Para Pyroxenite	7	37	20
Para B Pyroxenite	12	68	28
Pyroxene	0	400	20
Quartz Feldspathic Vein	12	75	26

16.1.13 Stress Regime

Based on a collation and evaluation of in-situ stress measurements from South Africa, by Stacey & Wesseloo (1998), the following may be noted:

- Horizontal principal stresses typically tend to be aligned approximately NW-SE and NE-SW.
- Generally, horizontal stress is equal to, or greater than, vertical stress.
- A significant difference in the maximum and minimum stress causes shearing along joint surfaces in a jointed rock mass.

Stress conditions in many Bushveld operations in South Africa are characterized by a high ratio of horizontal to vertical geological stresses (K ratio), particularly at shallow depth. Whereas in most deep level gold mining operations, the K ratio is in the order of 0.5, in Bushveld operations, a ratio of 2 is not uncommon. Figure 16.6 shows the Major (K1) and Minor (K3) horizontal to vertical stress ratio versus depth. At 1,000 m depth, the average K ratio is about 1.0 and the very high values are less common.

Figure 16.6 Major (K1) and Minor (K3) Horizontal to Vertical Stress Ratios

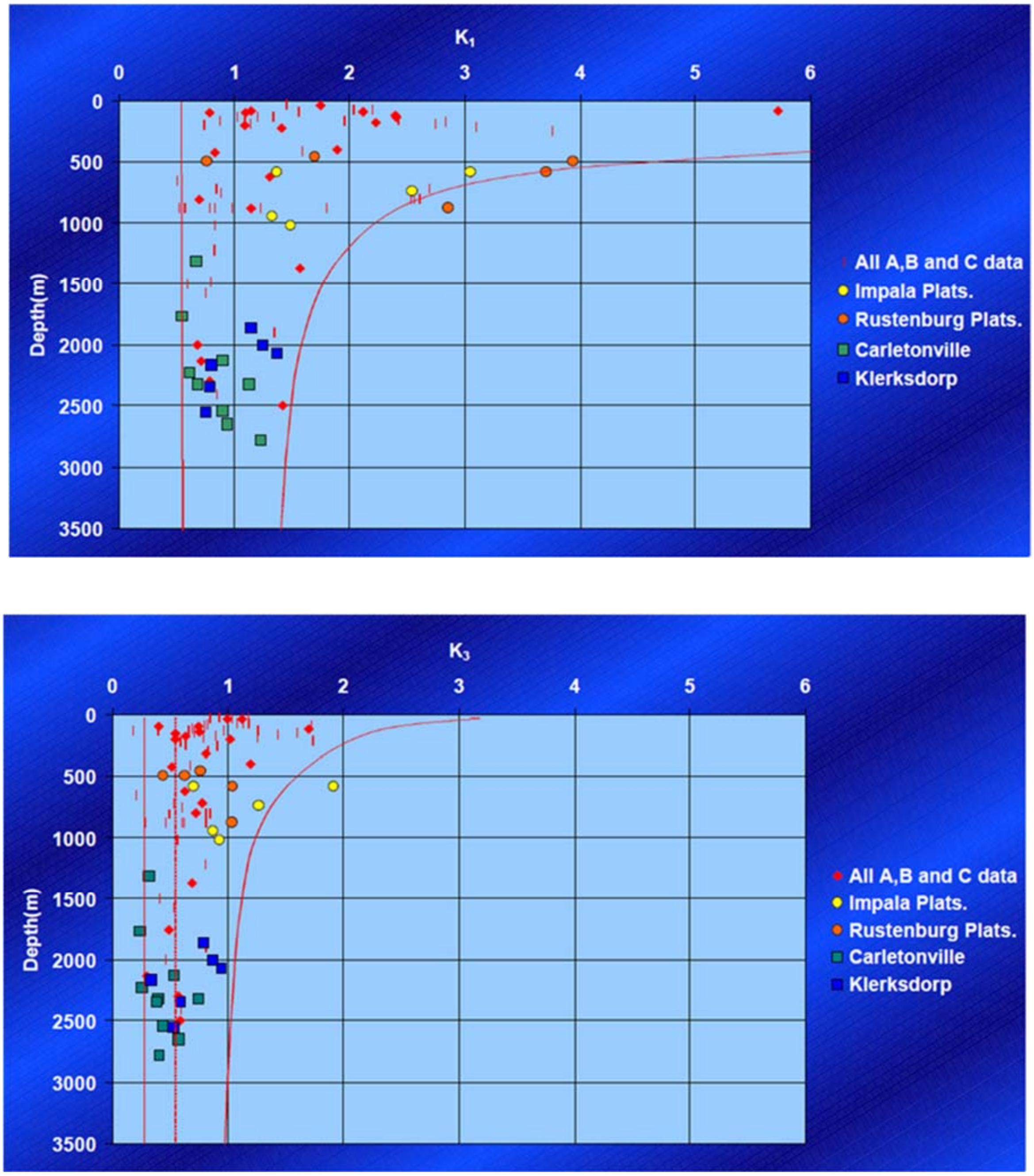


Figure After Stacey and Wesseloo, 1998.

16.1.14 Stope Stability

The target stoping zone is defined by a grade cut-off. The stope back and walls are expected to lie within what was originally defined as the mining zone in Section 16.

The assessment of stope stability and potential dilution is based on the Matthews/Potvin Stability Graph (Figure 16.7). It should be noted that this empirical method is based on stopes with rock walls rather than backfill. It is not strictly correct to apply this method to stopes with backfill on either side, but it does give an indication of stability, providing there is tight filling and the backfill provides some measure of support.

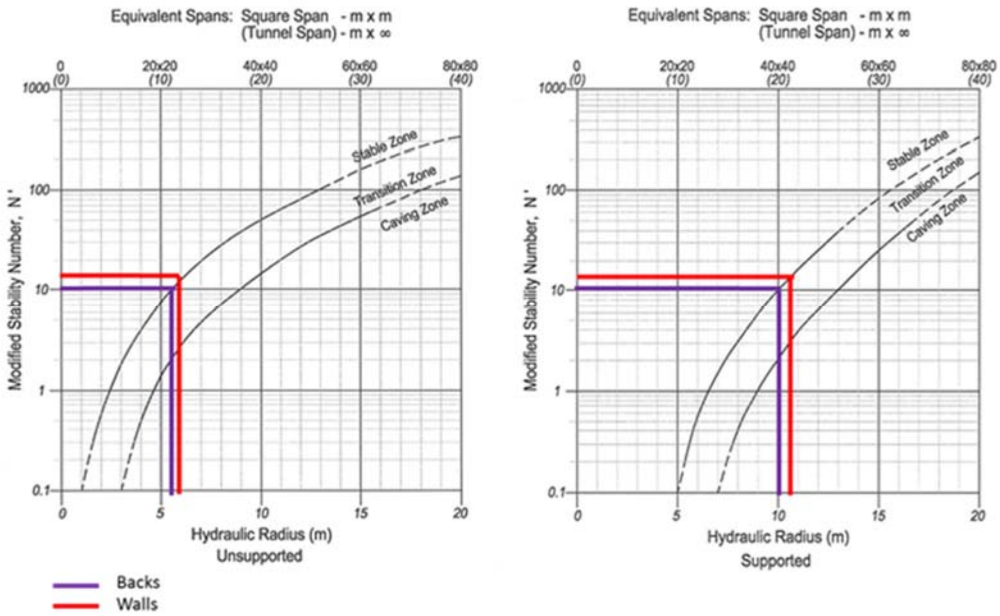
The stability number (N') is determined as follows:

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

- The mean Q' for the mining zone is 23 (Table 16.10).
- A is the Rock Stress Factor, which is a function of the UCS and the maximum induced stress acting at the centre of the stope wall. Simple elastic modelling was used to determine the stress in the back and walls of the stope (Figure 16.7).
- B is the Joint Orientation Factor. This is based on the angle between the critical joint and the wall. Table 16.3 shows that, although the strike is variable, the critical joint dip is likely to be in the order of 45°. These can be coated with serpentenite and chlorite, which have low shear strength.
- C is the Gravity Adjustment Factor. Gravity falls will be the most likely mode of failure for both the flat and inclined reef. Since the flat reef is sub-horizontal, the C value would be 2.

The parameters used for determination of 'N' and the wall hydraulic radius are listed in Table 16.13.

Figure 16.7 Matthews/Potvin Stability Graph for Stope Backs and Walls



It should be noted that chrome stringers, and other reef parallel weaknesses, do occur above the UG2 and Merensky Reefs resulting in potential instability.

The determination of N' for backs does not take chrome stringers or reef parallel weaknesses as the critical joint, but it is planned to support the backs with cables to manage this risk.

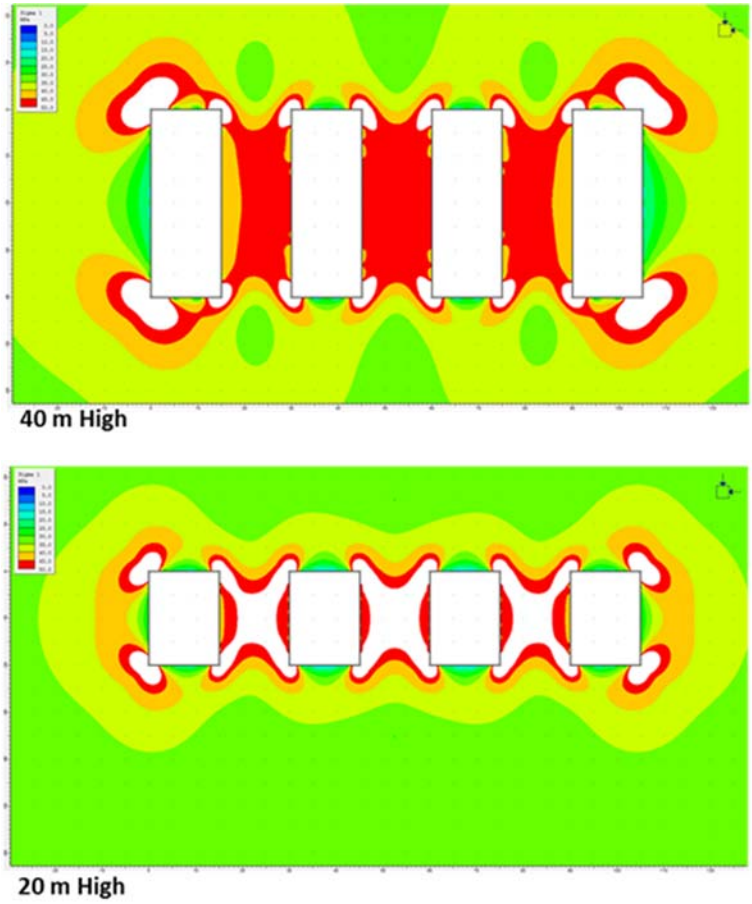
Table 16.13 Determination of Stability Number

Parameter	Backs	Walls
Q'	23	23
Stress	25	45
Strength	127	127
Stress to strength ratio	5.1	2.8
A	0.45	0.19
Wall dip	0°	90°
Dip of critical joint	45°	45°
Angle between joint and wall	45°	45°
B	0.5	0.5
Mechanism	Gravity	Sliding
Wall dip/dip of critical joint	0	45°
C	2	6
N'	10.3	13.3
Unsupported hydraulic radius	5.5	6
Supported hydraulic radius	10	11

The stope walls have been designed with a hydraulic radius of 8, based on an earlier iteration of this analysis. It should be noted that 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. If the conditions of the walls deteriorate, then the stope can stopped and backfill can be placed. If there is some sloughing of the walls, secondary blasting may be required to remove this material. In extreme cases, there will be minor losses. The failures will effectively reduce the volume of the secondary stopes, which will be replaced by backfill. It will be possible to implement a management system to minimise losses and disruptions to production.

More data will be used during the PFS to assess the potential stope conditions.

Figure 16.8 Section View Showing 2 D Elastic Modelling of Stopes



16.1.15 Mining Method, Layout, Sequence and Support Requirements

The height of the target stoping zone varies considerably. At this stage, it is planned to use longhole stoping where the deposit exceeds 18 m height and benching or drift-and-fill is being considered for the lower stoping zone heights.

Longhole Stoping With Post Backfilling

The stope dimensions considered at this stage are 15 m wide x height of the target stoping zone. The length of the stopes will need to be limited to between 40 m and 60 m, to ensure that the hydraulic radius of the walls does not exceed 8 m. The hydraulic radius of the stope back will vary between 5.5 m and 6 m, which is slightly larger than the required unsupported hydraulic radius (5.5 m) for the backs. It is also planned to widen the overcut to 10 m, to improve production drilling accuracy. The overcut crown should be supported with 4 m cable anchors and, to provide protection for the workers during development and production drilling and to limit dilution after blasting. Providing the undercut and other access drives are less than 5.5 m wide, 2.4 m long rockbolts in a 1.5 m x 1.5 m pattern, will provide adequate support for both the crown and sidewalls.

The crowns of all drives, where personnel are exposed, should be supported with mesh or shotcrete to ensure the safety of personal.

Once the stope has been blasted, personnel access must be prevented. The broken material should be removed with remote LHDs.

At 1,000 m depth, minor rockbursts 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 rockbursts. 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 (see Section 16.1.3). Adjacent secondary stopes should not be mined simultaneously and backfill should be placed prior to mining the next secondary stope.

Benching, Drift-and-Fill

It is suggested that 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 sidewalls. 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 material.

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.

16.1.16 Backfill Requirements

At this stage of the project, it is recommended that all stopes, drifts and benches are backfilled. The backfill could take the form of cemented rock fill (CRF), hydraulic fill or paste fill. The availability of waste material underground is limited and would need to be imported and transported at considerable expense. Hydraulic fill systems are simple, but consolidation takes time, drainage water needs to be handled and there is an increased risk of backfill failures. The fines need to be removed from the tailings stream, through cyclone classification, to ensure proper drainage, which could create problems if the mill grind is very fine. Paste fill systems use the full tailings stream and are optimal for quick filling and productivity, but the horizontal transport may be a challenge and will need to be properly designed. A trade-off analysis for different backfill systems will be necessary during the pre-feasibility study.

A paste backfill and consolidated hydraulic fill will have a water content by mass of 20%. Given the density of 3,200 kg/m^3 (see Table 16.5 and Table 16.6); the following backfill characteristics are anticipated:

Solids density (kg/m^3)	3,200
Water content (% mass)	20
Pulp density (% mass)	80
Slurry density (kg/m^3)	2,230
Porosity (%)	45
Solids (% volume)	55

Theoretically, 55% of the mined material will need to be returned to underground stopes as tailings backfill. This provides an indication of the backfill requirements.

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. 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. The strength of the backfill to ensure the necessary free-standing ability can be calculated as follows:

$$UCS = \frac{F.B\gamma_b(2H - D)}{2H + 2B - D}$$

Where:

F =factor of safety

γ_b = Unit weight of the backfill

B = Stope length

D= Stope width

H= Stope height

The binder requirement can be estimated with the following equation:

c=n×(UCS/27×F)^((1⁄(1.54)))

Where η = porosity

The backfill strength and binder requirements are provided in Figure 16.9 and Figure 16.10 respectively.

Figure 16.9 Backfill Strength Requirements

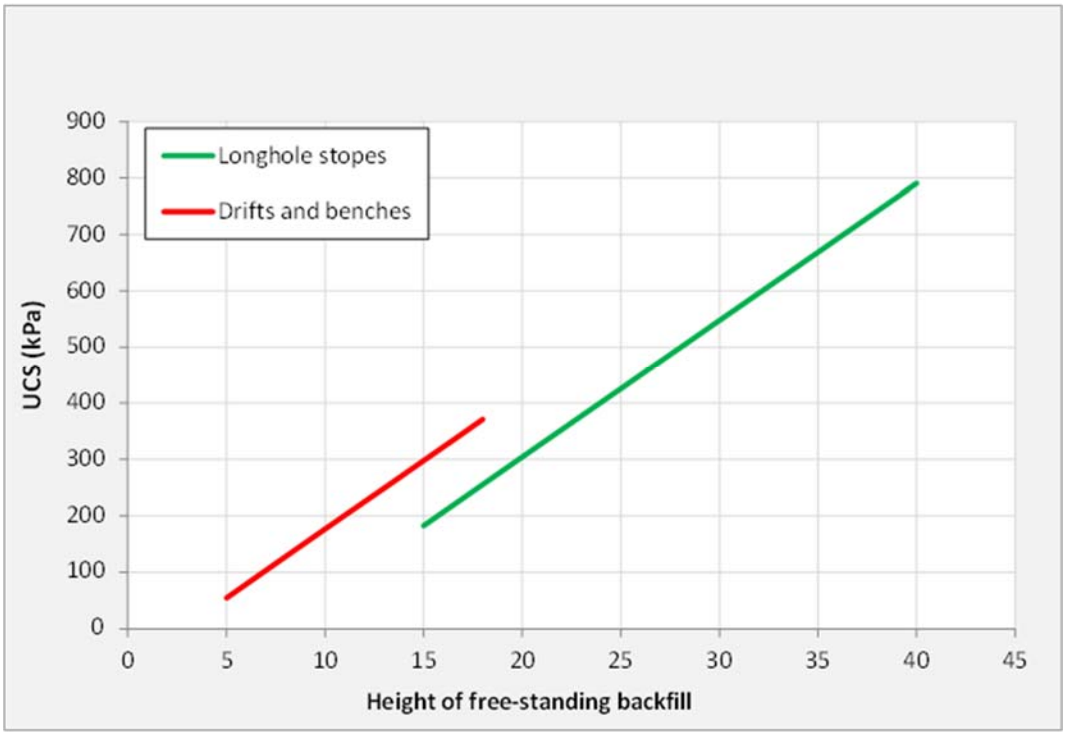
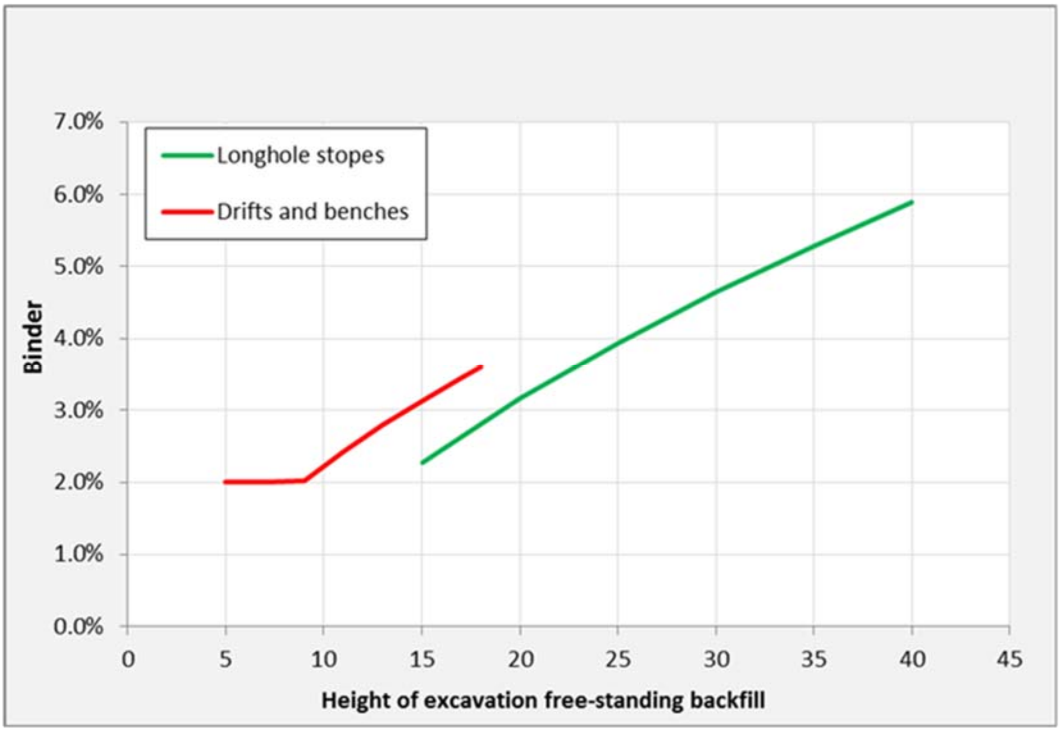


Figure 16.10 Backfill Binder Requirements



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. If hydraulic backfill is used, it will be necessary to provided drainage inside the stopes and behind the bulkhead. This is typically in the form of perforated pipes wrapped in a geotextile drainage fabric.

16.1.17 Protection of Shafts

The shaft needs to be protected from excessive mining induced stress and deformation. Once extensive mining has taken place, high mining induced vertical compressive stresses can cause steelwork and lining damage in the shaft barrel. The basic criterion for shaft pillar design is that the vertical field stress in the centre of the pillar should not exceed about 50% of the strength of the rock. Also the compressive strain should not exceed 0.4 mm/m and 0.7 mm/m for normal shaft steelwork and linings.

At 1,000 m depth the shaft pillar radius is typically about 200 m to 250 m in strong rock. For the purposes of the Platreef 2014 PEA, the shaft pillar radius should be 250 m for the dual shaft system and 200 m for the single ventilation shafts.

Once the minable shape of the deposit and extent of mining has been defined, the shaft pillar will need to be modelled to verify the shaft pillar dimensions.

Early extraction of the shaft reef has been done successfully for three deep shafts in South Africa and these shafts are still operating effectively with no shaft pillars. This approach requires considerable modification to the shaft steelwork and infrastructure and requires further rinvestigation. It is an option, which can be considered in the pre-feasibility stage.

Any two shafts should be separated by a distance of at least three times the combined diameter of the shafts to avoid stress interaction.

In SRK report 450790/2, the geotechnical characteristics of the rock in which the main shaft is planned were evaluated and support recommendations were provided. A single, triple tube, rotary-cored, vertical borehole (GT008) was drilled at the centre of the proposed production shaft position. The hole was geotechnically logged to a final depth of 1,225.97 m, which was 74.03 m short of the planned 1,300 m hole completion depth. This was due to extremely poor ground conditions being encountered at the end of hole.

The primary aim of the borehole logging was to:

- Classify the rock mass, over the full depth of the borehole, using Laubscher's (1990) Mining Rock Mass Rating Classification System and Barton et al's (1974) Norwegian Geotechnical Institutes Q-System.
- Identify any potentially adverse ground conditions that may impact the stability of the production shaft.
- Provide data with respect to the sub-surface geology, ground conditions and support recommendations.

Laboratory testing was carried out on selected, representative samples collected from borehole GT008; testing comprised:

- Uniaxial Compressive Tests.
- Uniaxial Compressive Tests with Elastic Modulus and Poisson's Ratio.
- Brazilian Tensile Strength Tests.
- Base Friction tests based on direct shear tests on saw-cuts.

Analysis and interpretation consisted of:

- A qualitative description of the sub-surface geology, based on the geotechnical borehole log.
- The interpretation of the weathering regime, based on the geotechnical borehole log.
- The interpretation of the jointing with depth, including micro-condition, macro-condition, joint infill and joint alteration.
- The quantification of the quality of the rock mass with depth.
- The provision of support recommendations with depth.

It was found that the rock mass was of suitable quality for sinking to a depth of approximately 1,106 m. 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 1,106 m, an extremely poor quality fractured hornfels, extends to a depth of at least 1,226 m. It is believed that the poor rock mass conditions are due to a local, but very large xenolith. It is not recommended that the shaft should be sunk through this material. If it 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.

16.1.18 Stability of Mine Development

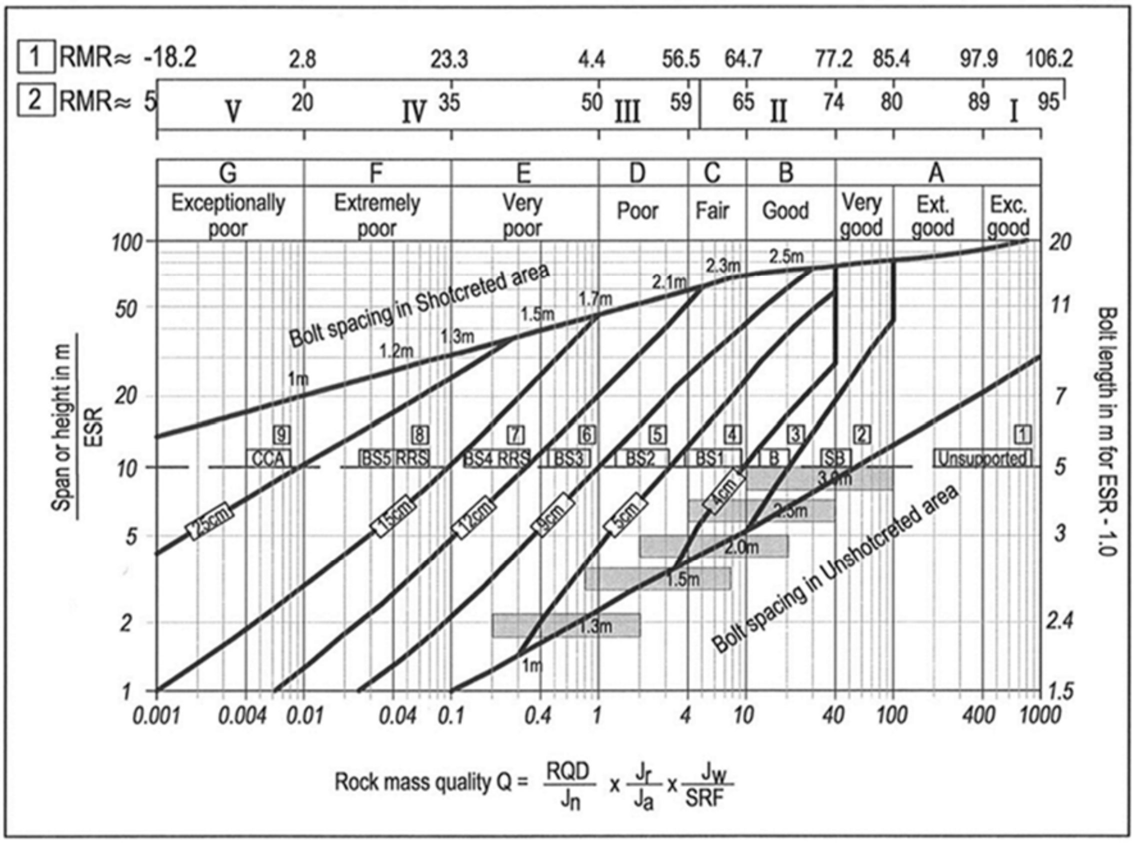
Large, important excavations should be developed in good quality rock, in areas that are not influenced by abutment stresses. Access development should avoid high stress abutments as far as possible and additional support will need to be installed if tunnels are likely to experience high stress during their life. Excavations should be separated by three times the combined diameter to avoid stress interaction. Where this cannot easily be avoided, it will be necessary to model the stress interaction and determine the additional support requirements.

The mean Q’ values for the mining zone and footwall are 23 and 20 respectively (see Table 16.10). Water is generally not expected to influence the stability of excavations, except near surface in the shafts. Jw can, therefore, generally be taken as 1. The SRF (Stress Reduction Factor) takes weakness zones and stress effects into consideration. Weakness zones are expected to be infrequent and additional support will be required in these areas. On other mines in the Bushveld Complex, minor stress damage occurs at depths below 800 m, under virgin stress conditions.

Moderate to severe stress damage can occur in excavations in brittle anorthosite. An SRF value of 2 is considered appropriate for typical conditions. A typical Q value of 10 can, therefore, be considered.

Regarding the Q support chart (Figure 16.11), an Excavation Support Ratio (ESR) value of 1.6 should be considered for permanent mine openings. Rockbolt support will be adequate for 5 m x 5 m tunnels under typical conditions (spacing 1.5 m x 1.5 m). The crowns of excavations should be supported with mesh or shotcrete to ensure the safety of personnel. Additional support will be required where the conditions are poor due to geological structures or local stress conditions. Large permanent openings will require reinforced shotcrete and long cable anchors as permanent support. The length of rockbolts should be half the width of the excavation.

Figure 16.11 Q Support Chart



16.1.19 Recommendations for Feasibility Study

The geotechnical investigation and design in this report is adequate for the Platreef 2014 PEA. The following additional work will be required to bring the project to feasibility level.

- Development of a geotechnical domain model.
- Derivation of Hydraulic Radii, mining spans, pillar dimensions and backfill requirements using empirical methods.
- Review of the proposed mining methods and sequences.
- Elastic modelling of the proposed mining, to investigate stress concentrations and the effect on access development and shaft stability.
- Assessment of underground support requirements.
- Conceptual, two-dimensional, non-linear modelling of stopes and critical infrastructure to verify stability and support requirements.
- The evaluation of the geotechnical risk associated with the various mining methods.

16.2 Mining

16.2.1 Introduction

Mining zones included in the current Platreef mine plans occur at depths ranging from approximately 500 m to 1,600 m. Access to the mine will be via multiple vertical shafts. Mining will be performed using highly productive mechanized methods and paste backfill will be utilized to fill open stopes. When available, excess waste rock will be used as backfill where cemented backfill is not required.

In November 2011, Stantec completed a scoping study that evaluated underground mining of the Platreef resource. Ivanhoe has subsequently performed additional drilling and completed a major reinterpretation of the geologic model. In November 2012, Ivanhoe requested that Stantec prepare an update to the scoping study, utilizing an updated resource block model that was being prepared by AMEC E&C Services Inc. (AMEC). The resource model was completed in March 2013 and is the basis for the Platreef 2014 PEA.

The Platreef 2014 PEA evaluates options for mining the indicated and inferred resources at Platreef based on a combination of longhole and drift-and-fill stoping methods, with paste backfill and supplemental waste rock fill where required. The three project phases are:

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

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 a further ramp-up to a plant capacity of 12 Mtpa.

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

16.2.2 Mineral Resources

The mineral resources available for mining at the Platreef were based on the resource block model developed by AMEC and provided to Stantec in March 2013. In addition to the grades of the individual metal, the block model includes a variable named 3PE+Au. The 3PE+Au is calculated by adding the grades for platinum, palladium, gold, and rhodium. The 2 g/t 3PE+Au grade shell was used by AMEC as a base case to assess the available mineral resource. The Mineral Resources are reported in Section 14.

16.2.2.1 Mining Block Model

The resource block model for selective mining within and adjacent to the TCU was used to prepare a mining block model for mine planning. Preliminary estimates of price and metallurgical parameters were estimated to calculate the Net Smelter Return (NSR) for each block in the resuource block model. 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.

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 16.14 and Table 16.15. The BDT13 metal recovery to concentrate was based on a fixed tail grade (Table 16.16) and a constant mass pull of 3.23%.

Table 16.14 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 16.15 BDT13 Realisation Assumptions

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

Table 16.16 BDT13 Fixed Tail Grade

Domain	Pt g/t	Pd g/t	Au g/t	Rh g/t	4PGE g/t	Ni %	Cu %
T1	0.30000	0.25000	0.10000	0.00674	0.63000	0.09000	0.02500
T2U	0.32417	0.28242	0.10598	0.00674	0.71236	0.11273	0.03000
T2L	0.32417	0.28242	0.10598	0.00674	0.75403	0.11273	0.03000

The BDT13 prices and parameters are similar to the final prices and parameters used in the economic analysis for the Platreef 2014 PEA which were defined at the end of 2013 and in early 2014. The final parametsr resulted in an average increase of approximately $10/t NSR or approximately 10%. There was also an increase in unit costs over the same period.

16.2.2.2 PEA Inventory Definition

High grade PGE mineralization at Platreef occurs in two main zones within the Turfspruit cyclic unit, the T1 pyroxenite unit (Zone T1) and the T2 pegmatoid unit (Zone T2). Zone T1 is located above T2, relatively thin, and with thicknesses seldom exceeding 10 m. The T2 unit ranges in thickness from less than 5 m to over 50 m and is thus well suited for large scale mechanized mining. The two units are separated vertically by a variable thickness of more weakly mineralized rock.

The current evaluation concentrates on mining of the thicker T2 resource. Material from T1 is only mined in areas where it is separated by 4 m or less from the T2 zone and the grade of the T1 zone is sufficient to justify mining of the intervening low grade material. Identification of the potential mining blocks within the Platreef resource was accomplished using the Vulcan Envisage three-dimensional underground design software package and Stantec's proprietary Vertical Miner software tool.

The longhole stoping mining method will be used for mining the thicker portions of the resource zones. A minimum mineable thickness of 18 m is used to identify potentially mineable longhole stoping resources.

The drift-and-fill stoping method has been assumed for mining resource zones thinner than 18 m. A minimum mining thickness of 4 m has also been applied to the resource in these areas, which allows the use of suitably sized mechanized mining equipment. For the Platreef 2014 PEA it has been assumed that all thinner material will be extracted using the drift-and-fill method. Drift-and-bench stoping and other possible stoping methods need to be assessed in future studies for the thicker portions of the zones under 18 m.

NSR cut-off grades were evaluated utilizing mining costs from previous scoping studies as well as processing and G&A costs from Ivanhoe's previous economic evaluations. Table 16.17 summarizes those costs for longhole and drift-and-fill stoping.

Table 16.17 Estimated Mining Costs

Description	Rate	Cost ($/t)	
		Longhole Stoping	Drift-and-Fill Stoping
Mining	–	10.93	26.01
Backfill	–	11.91	11.91
Indirects	30%	6.85	11.38
Contingency	10%	2.97	4.93
Power	–	2.61	2.61
Total Mining Cost	**–**	**35.27**	**56.84**
Processing	–	10.00	10.00
G&A (Variable)	–	3.00	3.00
G&A (Fixed)	–	0.50	0.50
Total Cost	**–**	**48.77**	**70.34**

Minimum economic NSR cut-off values of $50 for longhole stoping and $70 for drift-and-fill stoping were assumed in Stantec's Vertical Miner programme. This programme identifies potentially mineable resource zones by evaluating NSR values for vertical columns of blocks within the resource model. Assumptions are made for NSR cut-off grades, minimum thickness requirements, and maximum allowed internal dilution.

The programme begins by finding the first economic resource block near the base of the mining zone and then evaluates the economics of mining the blocks immediately above it. Economic mining zones are identified where the cut-off grade, thickness, and internal dilution requirements are satisfied.

In-place Vertical Miner resources are estimated for various minimum NSR values for each potential mining method and the location of the selected resources are plotted in plan view. Polygons are drawn around the potentially mineable mining zones and the in-place resources occurring in each polygon are estimated. Small, isolated mining zones are excluded from the estimates.

Dilution and recovery factors are then applied to the in-place resources for each potential mining area to produce an estimate of recovered resources and define the PEA Inventory to be included in the Platreef 2014 PEA.

Dilution grades are estimated using the average grade of the mining blocks immediately above the mining zones. Dilution is applied as a percentage of the total vertical miner tonnes. Total diluted tonnes and diluted grades are then calculated.

Due to irregularities in the geometry of the mineralized zones, not all cut-off grade material can be mined without incurring excessive dilution. 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 mineralized 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.

Table 16.18 shows the dilution and mining recovery factors used in the helping define the PEA Inventory.

Table 16.18 Dilution and Mining Recovery Factors

Mining Method	% Dilution	% Recovery
Longhole Stoping	15%	80%
Drift-and-Fill Stoping	10%	85%

Table 16.19, Table 16.20, and Table 16.21 summarize the estimated PEA inventories from which the mining production schedules were subsequently developed for Phase 1 (4 Mtpa), Phase 2 (8 Mtpa), and Phase 3 (12 Mtpa). A plan view showing the boundaries of the areas included in the PEA Inventories is show in Figure 16.12.

Table 16.19 Summary of PEA Inventory – Phase 1 (4 Mtpa)

Resource Category	Mining Area	PEA Inventory (Mt)	NSR ($/t)	Cu (%)	Ni (%)	Pt g/t	Pd g/t	Au g/t	Rh g/t	3PE+Au (g/t)
Indicated	Area 1	123.5	$139.26	0.16	0.34	1.80	1.90	0.27	0.13	4.10
	Southeast Area	0.0	–	–	–	–	–	–	–	–
	Southwest Area	0.0	–	–	–	–	–	–	–	–
	Total	**123.5**	**$139.26**	**0.16**	**0.34**	**1.80**	**1.90**	**0.27**	**0.13**	**4.10**
Inferred	Area 1	0.0	–	–	–	–	–	–	–	–
	Southeast Area	0.0	–	–	–	–	–	–	–	–
	Southwest Area	0.0	–	–	–	–	–	–	–	–
	Total	**0.0**	–	–	–	–	–	–	–	–
All Categories	Area 1	123.5	$139.26	0.16	0.34	1.80	1.90	0.27	0.13	4.10
	Southeast Area	0.0	–	–	–	–	–	–	–	–
	Southwest Area	0.0	–	–	–	–	–	–	–	–
	Total	**123.5**	**$139.26**	**0.16**	**0.34**	**1.80**	**1.90**	**0.27**	**0.13**	**4.10**

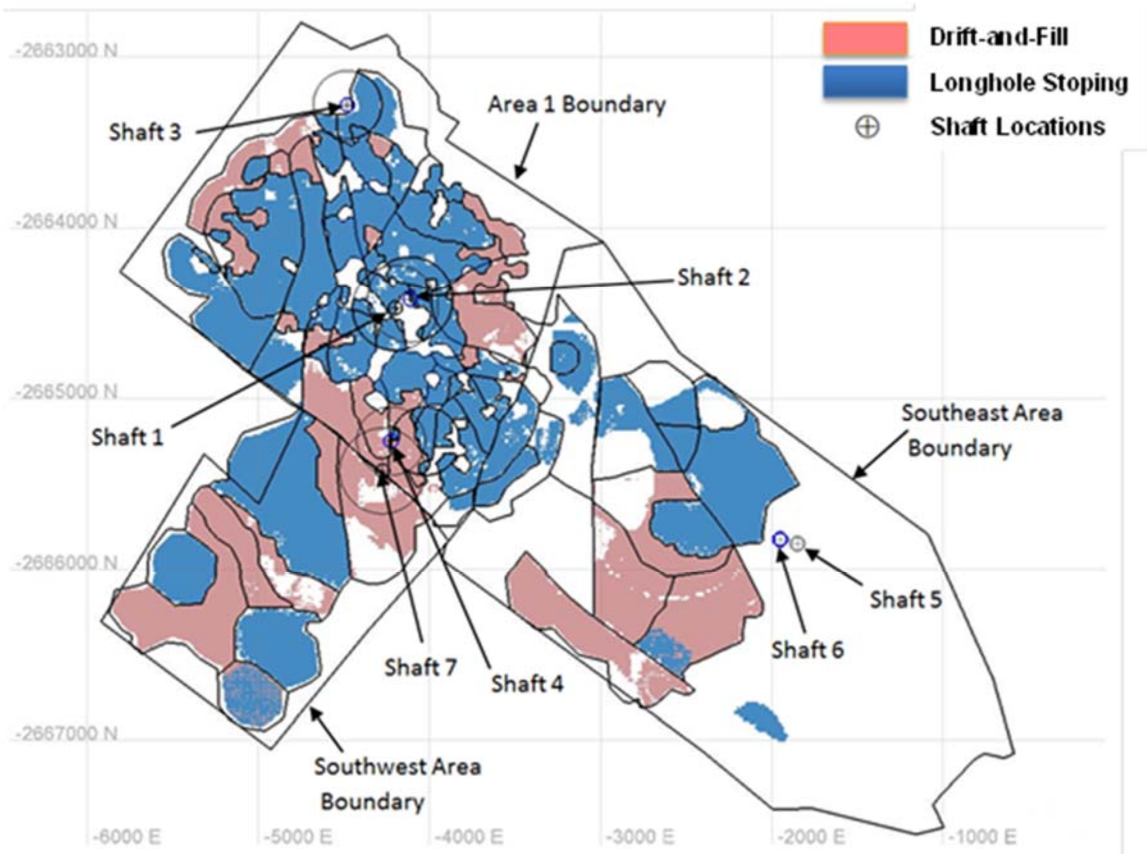
Table 16.20 Summary of PEA Inventory – Phase 2 (8 Mtpa)

Resource Category	Mining Section	PEA Inventory (Mt)	NSR ($/t)	Cu (%)	Ni (%)	Pt g/t	Pd g/t	Au g/t	Rh g/t	3PE+Au (g/t)
Indicated	Area 1	123.5	$139.26	0.16	0.34	1.80	1.90	0.27	0.13	4.10
	Southeast Area	0.0	–	–	–	–	–	–	–	–
	Southwest Area	0.0	–	–	–	–	–	–	–	–
	Total	**123.5**	**$139.26**	**0.16**	**0.34**	**1.80**	**1.90**	**0.27**	**0.13**	**4.10**
Inferred	Area 1	26.9	$138.01	0.15	0.31	1.85	1.94	0.26	0.14	4.19
	Southeast Area	41.4	$115.73	0.15	0.31	1.50	1.60	0.24	0.11	3.46
	Southwest Area	39.7	$115.26	0.15	0.31	1.51	1.61	0.24	0.11	3.46
	Total	**108.1**	**$121.11**	**0.15**	**0.31**	**1.59**	**1.69**	**0.25**	**0.13**	**3.64**
All Categories	Area 1	150.4	$139.03	0.16	0.33	1.81	1.91	0.27	0.13	4.12
	Southeast Area	41.4	$115.73	0.15	0.31	1.50	1.60	0.24	0.11	3.46
	Southwest Area	39.7	$115.26	0.15	0.31	1.51	1.61	0.24	0.11	3.46
	Total	**231.5**	**$130.79**	**0.16**	**0.33**	**1.70**	**1.80**	**0.26**	**0.12**	**3.89**

Table 16.21 Summary of PEA Inventory – Phase 3 (12 Mtpa)

Resource Category	Mining Section	PEA Inventory (Mt)	NSR ($/t)	Cu (%)	Ni (%)	Pt g/t	Pd g/t	Au g/t	Rh g/t	3PE+Au (g/t)
Indicated	Area 1	123.5	$139.26	0.16	0.34	1.80	1.90	0.27	0.13	4.10
	Southeast Area	0.0	–	–	–	–	–	–	–	–
	Southwest Area	0.0	–	–	–	–	–	–	–	–
	Total	**123.5**	**$139.26**	**0.16**	**0.34**	**1.80**	**1.90**	**0.27**	**0.13**	**4.10**
Inferred	Area 1	30.6	$132.07	0.15	0.30	1.78	1.85	0.26	0.13	4.03
	Southeast Area	128.7	$118.30	0.18	0.36	1.41	1.45	0.26	0.10	3.21
	Southwest Area	84.3	$126.81	0.14	0.30	1.73	1.74	0.26	0.13	3.87
	Total	**243.6**	**$122.97**	**0.16**	**0.33**	**1.57**	**1.60**	**0.26**	**0.11**	**3.54**
All Categories	Area 1	154.1	$137.83	0.16	0.33	1.80	1.89	0.27	0.13	4.09
	Southeast Area	128.7	$118.30	0.18	0.36	1.41	1.45	0.26	0.10	3.21
	Southwest Area	84.3	$126.81	0.14	0.30	1.73	1.74	0.26	0.13	3.87
	Total	**367.1**	**$128.45**	**0.16**	**0.34**	**1.65**	**1.70**	**0.27**	**0.12**	**3.73**

Figure 16.12 Plan View of Mining Areas



16.2.2.3 Production Schedules

The Platreef 2014 PEA includes three phases of underground mining and concentrator expansion. The three phases are:

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

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

A production schedule for each phase was developed for a total production life of 30 years after mill start up. The total project life is 36 years including pre-production development, construction, and production ramp up prior to mill start-up.

The following list presents other operating criteria included in the mine plan.

- The underground mine operates 360 days per year.
- Production includes all above cut-off material from development and stope production.
- Development of the bulk sampling shaft (Shaft 1) begins in the second quarter of 2014 (Year -6).
- Development of the production shaft (Shaft 2) begins in the third quarter of 2014 (Year -6).
- Production from stope development activities begins in the third quarter of 2018 (Year -2).
- Stope production begins in the first quarter of 2019 (Year -1).
- Mill start up occurs in the first quarter of 2020 (Year 1).
- A declining cut-off grade scenario is utilized in the early years of production in order to increase the initial mill head grade and to shorten the payback period.

Production rates of 4 Mtpa, 8 Mtpa, and 12 Mtpa were selected for evaluation based on the results of the maximum sustainable production rate study. The total PEA inventories for these three production scenarios and a 30-year mine life are shown in Table 16.22.

Table 16.22 Phase PEA Inventories Mined

Phase	Mt	NSR ($/t)	Cu (%)	Ni (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)
Phase 1 – 4 Mtpa	117	$141	0.16	0.34	1.84	1.93	0.27	0.13	4.17
Phase 2 – 8 Mtpa	219	$134	0.16	0.35	1.70	1.78	0.27	0.12	3.87
Phase 3 – 12 Mtpa	310	$133	0.16	0.34	1.71	1.77	0.27	0.12	3.87

Phase 1 – 4 Mtpa

A 4 Mtpa production rate was selected by Ivanhoe as the basis for the initial evaluation based on the results of a survey estimating the amount of available nickel refining capacity in South Africa. At Ivanhoe's direction, only Indicated Mineral Resources are included in the production plan.

The schedule was prepared using the strategy of staying closer to the production shaft and above a depth of 1,000 m during the early years of production. An NSR cut-off grade of $100 was used in defining these initial mining areas in order to maximize overall mined grade. The cut-off grade was lowered to $80 NSR as mining proceeded into outlying and deeper areas.

Longhole stoping is planned to be used exclusively for the first 10 years of production. Limited drift-and-fill mining begins in some of the thinner, high grade zones after that point, supplementing the longhole stoping tonnes at a rate of 0.72 Mtpa. Mining of resources located in shaft pillar zones is delayed until the last few years of the mine life.

Figure 16.13 shows the areas of the Platreef resource selected for mining at 4 Mtpa by proposed mining methods. Table 16.23 presents the estimated tonnes and average grades by mining method for this production scenario.

Figure 16.13 Plan View of Mining Areas – Phase 1 – 4 Mtpa

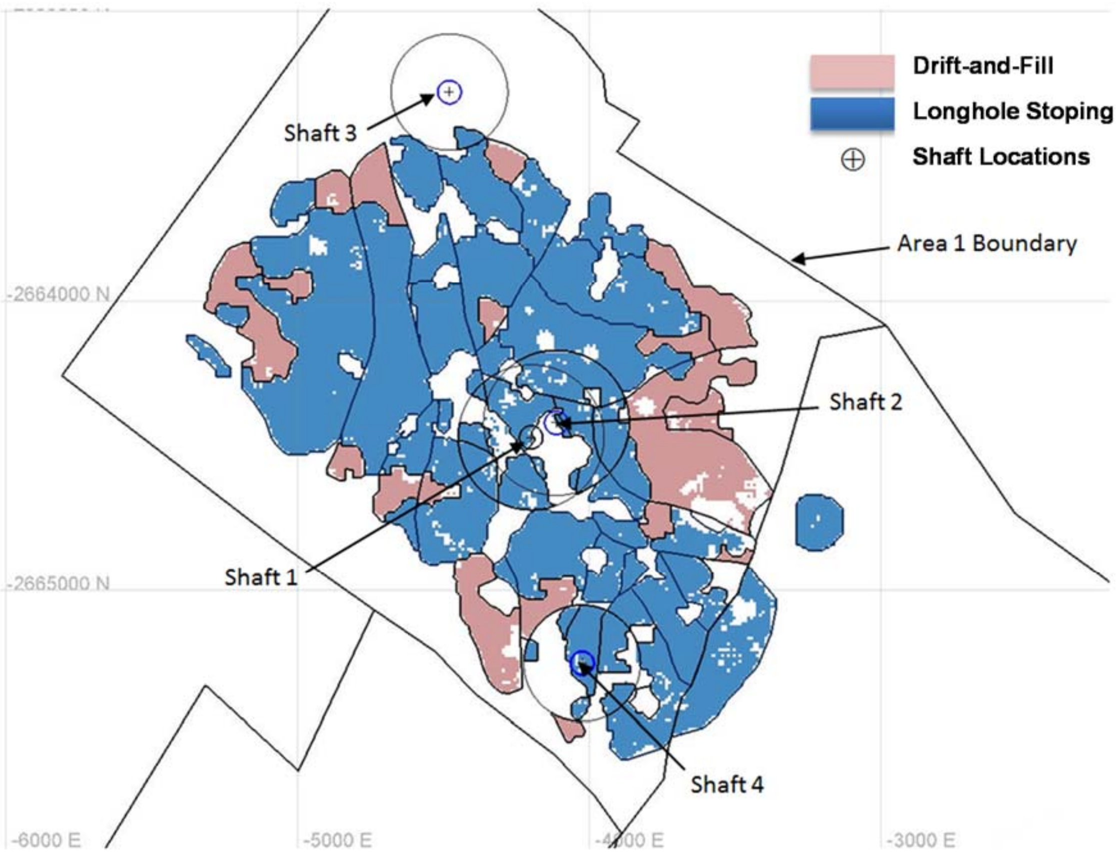


Table 16.23 PEA Inventories by Mining Method – Phase 1 – 4 Mtpa

Section	Mt	NSR ($ / t)	Cu (%)	Ni (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)
Longhole Stoping	105.6	$139.41	0.163	0.337	1.804	1.914	0.271	0.126	4.12
Cut-and-Fill	11.1	$158.07	0.161	0.335	2.131	2.131	0.308	0.151	4.72
Total	**116.7**	**$141.18**	**0.16**	**0.34**	**1.84**	**1.93**	**0.27**	**0.13**	**4.17**

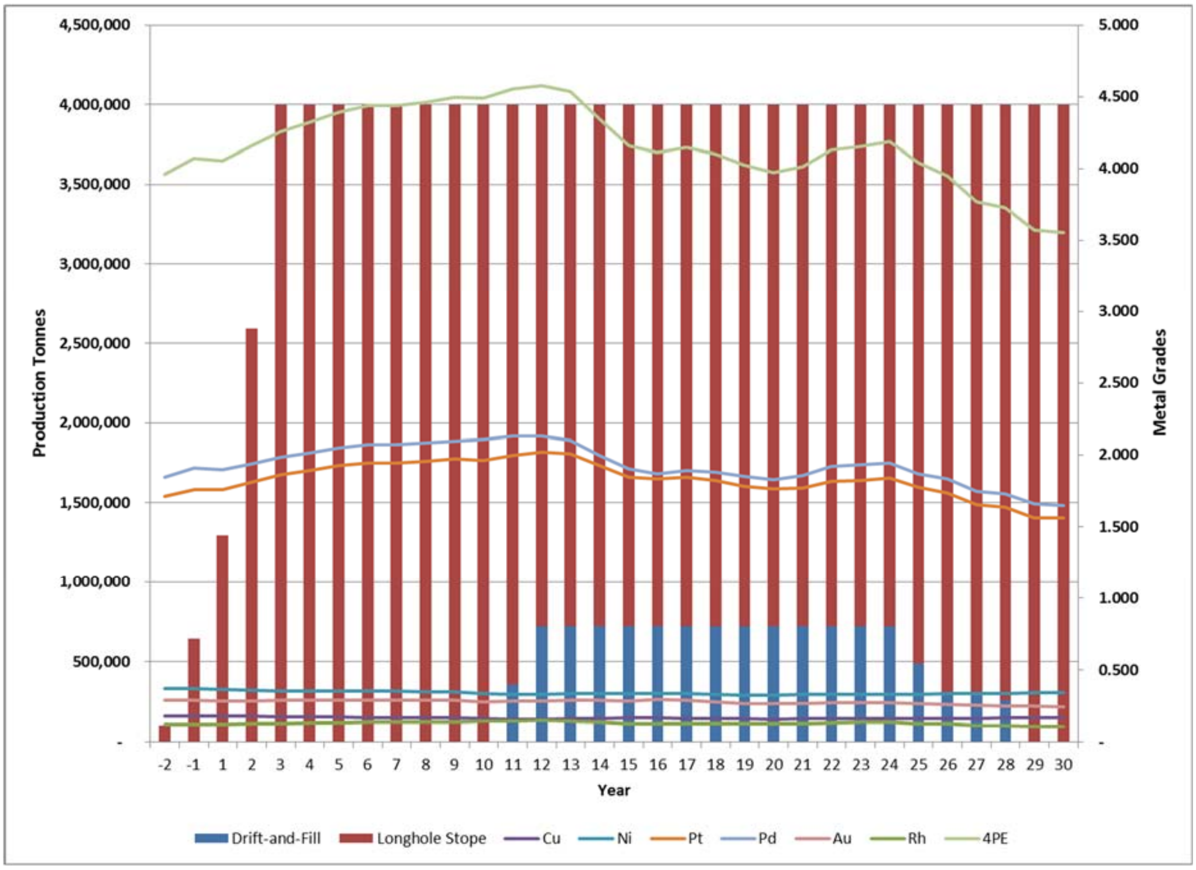
Table 16.24 and Figure 16.14 present summarized annual production schedules for Phase 1. Production build up occurs over a period of approximately 3.5 years.

Table 16.24 Production Schedule for Mining – Phase 1 – 4 Mtpa

Year	t	NSR ($ / t)	Cu (%)	Ni (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)
-2	100,000	$139.83	0.18	0.37	1.71	1.85	0.29	0.12	3.96
-1	648,000	$142.80	0.18	0.37	1.76	1.91	0.29	0.12	4.07
1	1,296,000	$141.55	0.18	0.36	1.76	1.90	0.28	0.12	4.06
2	2,592,000	$143.78	0.18	0.36	1.81	1.94	0.28	0.12	4.16
3	4,000,000	$146.08	0.17	0.35	1.86	1.99	0.29	0.13	4.26
4	4,000,000	$147.67	0.17	0.35	1.89	2.02	0.29	0.13	4.32
5	4,000,000	$149.72	0.17	0.35	1.92	2.05	0.29	0.13	4.39
6	4,000,000	$150.58	0.17	0.35	1.94	2.07	0.29	0.13	4.44
7	4,000,000	$150.68	0.17	0.35	1.94	2.07	0.29	0.13	4.44
8	4,000,000	$150.89	0.17	0.35	1.96	2.08	0.29	0.13	4.46
9	4,000,000	$151.14	0.16	0.34	1.97	2.10	0.29	0.14	4.49
10	4,000,000	$149.10	0.16	0.33	1.96	2.11	0.28	0.14	4.49
11	4,000,000	$150.29	0.16	0.33	2.00	2.13	0.28	0.14	4.55
12	4,000,000	$151.19	0.16	0.33	2.02	2.13	0.28	0.15	4.58
13	4,000,000	$150.79	0.16	0.33	2.01	2.10	0.29	0.14	4.54
14	4,000,000	$146.07	0.16	0.33	1.93	2.00	0.29	0.14	4.35
15	4,000,000	$141.11	0.16	0.33	1.85	1.90	0.28	0.13	4.16
16	4,000,000	$140.29	0.16	0.34	1.83	1.87	0.29	0.12	4.11
17	4,000,000	$140.47	0.16	0.33	1.84	1.89	0.29	0.13	4.15
18	4,000,000	$137.72	0.16	0.33	1.82	1.88	0.28	0.13	4.10
19	4,000,000	$134.97	0.16	0.32	1.78	1.85	0.27	0.12	4.02
20	4,000,000	$133.64	0.16	0.32	1.76	1.83	0.26	0.12	3.97
21	4,000,000	$135.19	0.16	0.33	1.77	1.86	0.26	0.13	4.02
22	4,000,000	$138.55	0.16	0.33	1.81	1.92	0.27	0.13	4.13
23	4,000,000	$138.91	0.16	0.33	1.82	1.93	0.27	0.13	4.15

Year	t	NSR ($ / t)	Cu (%)	Ni (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)
24	4,000,000	$140.29	0.16	0.33	1.84	1.94	0.27	0.14	4.18
25	4,000,000	$136.27	0.16	0.33	1.78	1.87	0.26	0.13	4.04
26	4,000,000	$134.07	0.16	0.33	1.73	1.83	0.26	0.12	3.95
27	4,000,000	$129.17	0.16	0.33	1.65	1.75	0.25	0.12	3.77
28	4,000,000	$128.38	0.16	0.34	1.64	1.73	0.25	0.11	3.73
29	4,000,000	$124.20	0.17	0.34	1.56	1.66	0.24	0.11	3.57
30	4,000,000	$123.53	0.16	0.34	1.56	1.65	0.24	0.11	3.55
Total	**116,636,000**	**$141.18**	**0.16**	**0.34**	**1.84**	**1.93**	**0.27**	**0.13**	**4.17**

Figure 16.14 Production Schedule – Phase 1 – 4 Mtpa



Phase 2 – 8 Mtpa

Phase 2 – 8 Mtpa includes the Indicated Mineral Resource areas included in Phase 1 – 4 Mtpa as well as certain high-grade Inferred Mineral Resource zones in Area 1 and the Southeast and Southwest Areas, which were excluded from Phase 1. Higher-grade bulk mineable zones in the Southeast Area were targeted for the initial production build up outside of Area 1. Mining zones at shallower depths were mined first and deeper zones were sequenced later in the mine life.

An NSR cut-off grade of $80 was used in defining mining zones in most of the inferred resource areas. As in Phase 1 – 4 Mtpa, longhole stoping is used exclusively for the first 10 years of production. Limited drift-and-fill mining begins after that point and supplements the longhole stoping tonnes at rates of 1.0 to 1.5 Mtpa for the remaining mine life. Mining of resources located in shaft pillar zones is delayed until the last few years of the mine life.

Figure 16.15 presents the additional areas of the Platreef resource selected for mining at 8 Mtpa. Table 16.25 presents the estimated tonnes and average grades by mining method for this production scenario.

Figure 16.15 Plan View of Mining Areas – Phase 2 – 8 Mtpa

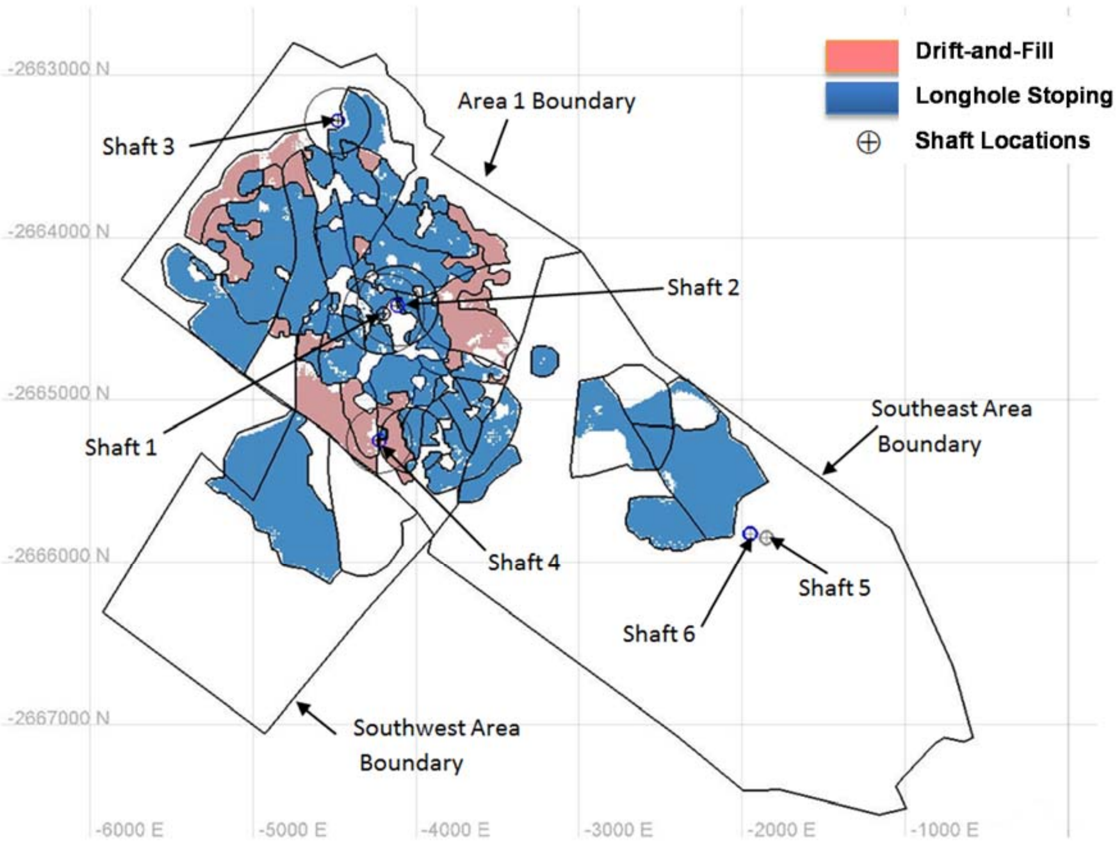


Table 16.25 Resources Recovered by Mining Method – Phase 2 – 8 Mtpa

Section	Mt	NSR ($ / t)	Cu (%)	Ni (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au
Longhole Stoping	197.2	$131.87	0.164	0.347	1.655	1.746	0.269	0.118	3.79
Drift-and-Fill	22.1	$154.63	0.158	0.331	2.097	2.108	0.287	0.150	4.64
Total	**219.3**	**134.16**	**0.16**	**0.35**	**1.70**	**1.78**	**0.27**	**0.12**	**3.87**

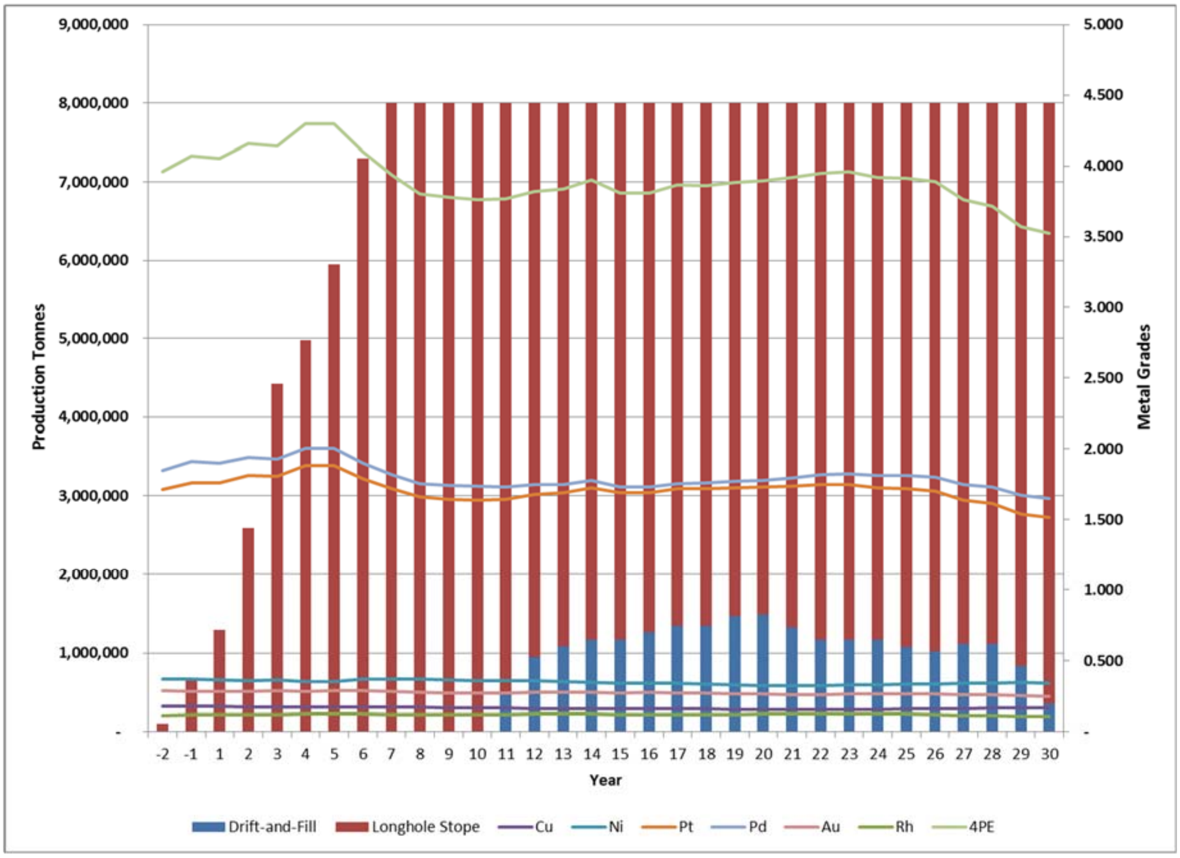
Table 16.26 and Figure 16.16 present summarized annual production schedules for Phase 2. Initial production build up in Area 1 is similar to Phase 1. Production from inferred resource zones begins in Year 3. Production builds up to 8 Mtpa over a period of approximately 7.5 years.

Table 16.26 Production Schedule for Mining – Phase 2 – 8 Mtpa

Year	t	NSR ($ / t)	Cu (%)	Ni (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)
-2	100,000	$139.83	0.18	0.37	1.71	1.85	0.29	0.12	3.96
-1	648,000	$142.80	0.18	0.37	1.76	1.91	0.29	0.12	4.07
1	1,296,000	$141.55	0.18	0.36	1.76	1.90	0.28	0.12	4.06
2	2,592,000	$143.78	0.18	0.36	1.81	1.94	0.28	0.12	4.16
3	4,424,000	$144.41	0.18	0.36	1.81	1.93	0.29	0.12	4.14
4	4,972,000	$146.92	0.17	0.35	1.88	2.00	0.29	0.13	4.30
5	5,944,000	$147.58	0.17	0.36	1.88	2.00	0.29	0.13	4.30
6	7,296,000	$144.33	0.17	0.37	1.79	1.90	0.29	0.12	4.10
7	8,000,000	$140.06	0.17	0.37	1.72	1.82	0.28	0.12	3.94
8	8,000,000	$136.18	0.17	0.37	1.66	1.75	0.28	0.12	3.81
9	8,000,000	$134.47	0.17	0.36	1.65	1.74	0.27	0.12	3.78
10	8,000,000	$133.35	0.17	0.36	1.64	1.73	0.27	0.12	3.76
11	8,000,000	$133.45	0.17	0.36	1.65	1.73	0.27	0.12	3.77
12	8,000,000	$134.73	0.16	0.36	1.68	1.74	0.28	0.12	3.82
13	8,000,000	$134.79	0.16	0.35	1.69	1.75	0.28	0.12	3.84
14	8,000,000	$135.65	0.16	0.35	1.72	1.78	0.28	0.12	3.90
15	8,000,000	$132.51	0.16	0.34	1.69	1.73	0.27	0.12	3.81
16	8,000,000	$132.84	0.16	0.34	1.69	1.73	0.28	0.12	3.81
17	8,000,000	$133.82	0.16	0.34	1.72	1.75	0.27	0.12	3.87
18	8,000,000	$132.85	0.16	0.33	1.72	1.76	0.27	0.12	3.86
19	8,000,000	$132.46	0.16	0.33	1.73	1.77	0.27	0.12	3.88
20	8,000,000	$132.15	0.15	0.33	1.73	1.78	0.27	0.12	3.90
21	8,000,000	$131.93	0.15	0.32	1.74	1.80	0.26	0.13	3.92
22	8,000,000	$133.19	0.15	0.33	1.75	1.82	0.26	0.13	3.95

Year	t	NSR ($ / t)	Cu (%)	Ni (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)
23	8,000,000	$133.64	0.15	0.33	1.75	1.82	0.26	0.13	3.96
24	8,000,000	$132.95	0.16	0.33	1.72	1.81	0.26	0.13	392
25	8,000,000	$133.38	0.16	0.33	1.72	1.81	0.27	0.12	3.92
26	8,000,000	$132.95	0.16	0.34	1.70	1.80	0.27	0.12	3.89
27	8,000,000	$129.67	0.16	0.34	1.64	1.75	0.26	0.12	3.76
28	8,000,000	$128.28	0.16	0.34	1.61	1.73	0.26	0.12	3.71
29	8,000,000	$124.57	0.17	0.35	1.54	1.67	0.25	0.11	3.57
30	8,000,000	$122.09	0.17	0.34	1.52	1.65	0.25	0.11	3.52
Total	**219,272,000**	**$134.16**	**0.16**	**0.35**	**1.70**	**1.78**	**0.27**	**0.12**	**3.87**

Figure 16.16 Production Schedule – Phase 2 – 8 Mtpa



Phase 3 – 12 Mtpa

Phase 3 – 12 Mtpa includes all Indicated and Inferred Mineral Resource areas included in Phase 2 – 8 Mtpa as well as additional Inferred Mineral Resources from deeper, high-grade zones in the Southwest Area and intermediate-grade zones from the Southeast Area.

An NSR cut-off grade of $80 was used in defining mining zones in most of the inferred resource areas. As in Phase 1, longhole stoping is used exclusively for the first 10 years of production. Drift-and-fill mining begins after that point and supplements the longhole stoping tonnes at a rate of 3.0 to 3.5 Mtpa for the remaining mine life. Mining of resources located in shaft pillar zones is delayed until the last few years of the mine life.

Figure 16.17 presents the additional areas of the Platreef resource selected for the expansion to 12 Mtpa. Table 16.27 presents the estimated tonnes and average grades by mining method for this production scenario.

Figure 16.17 Plan View of Mining Areas – Phase 3 – 12 Mtpa

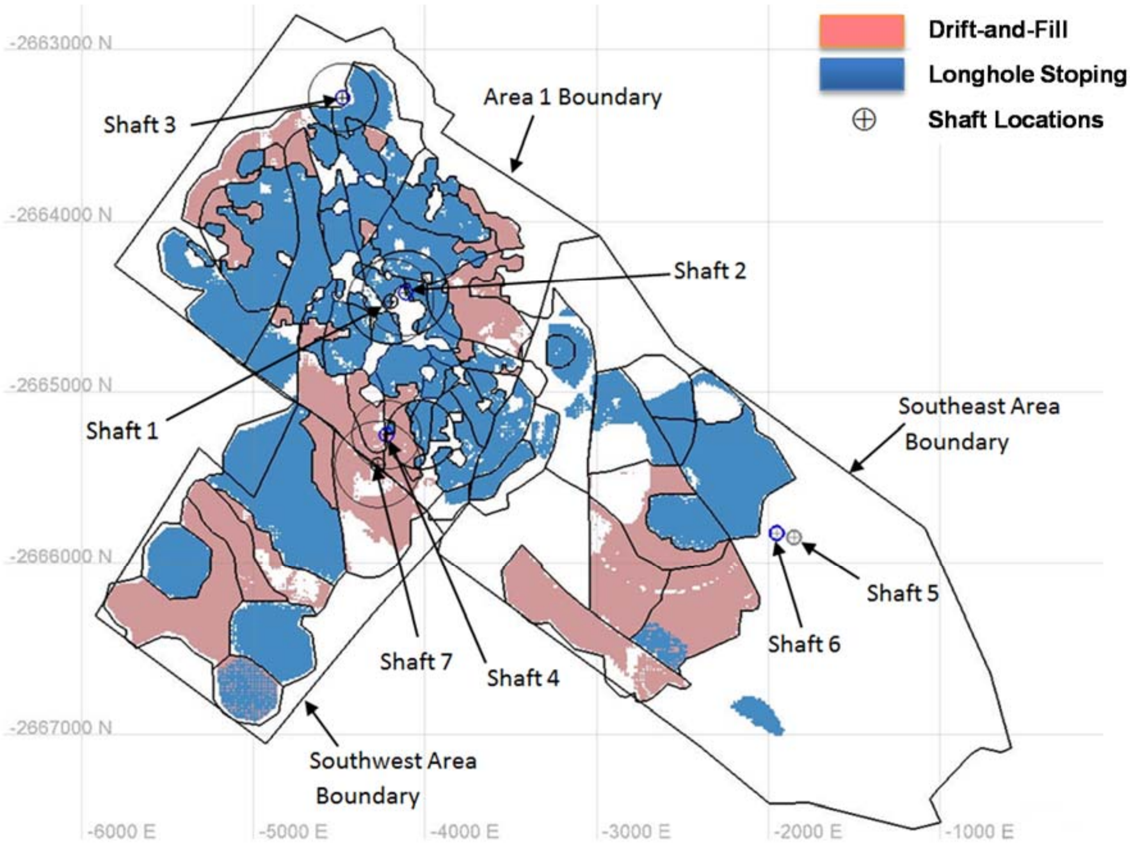


Table 16.27 Resources Recovered by Mining Method – Phase 3 – 12 Mtpa

Section	Mt	NSR ($ / t)	Cu (%)	Ni (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)
Longhole Stoping	244.6	$128.94	0.159	0.336	1.645	1.717	0.265	0.117	3.74
Drift-and-Fill	65.3	$145.99	0.160	0.332	1.952	1.961	0.289	0.139	4.34
Total	**309.9**	**$132.53**	**0.16**	**0.34**	**1.71**	**1.77**	**0.27**	**0.12**	**3.87**

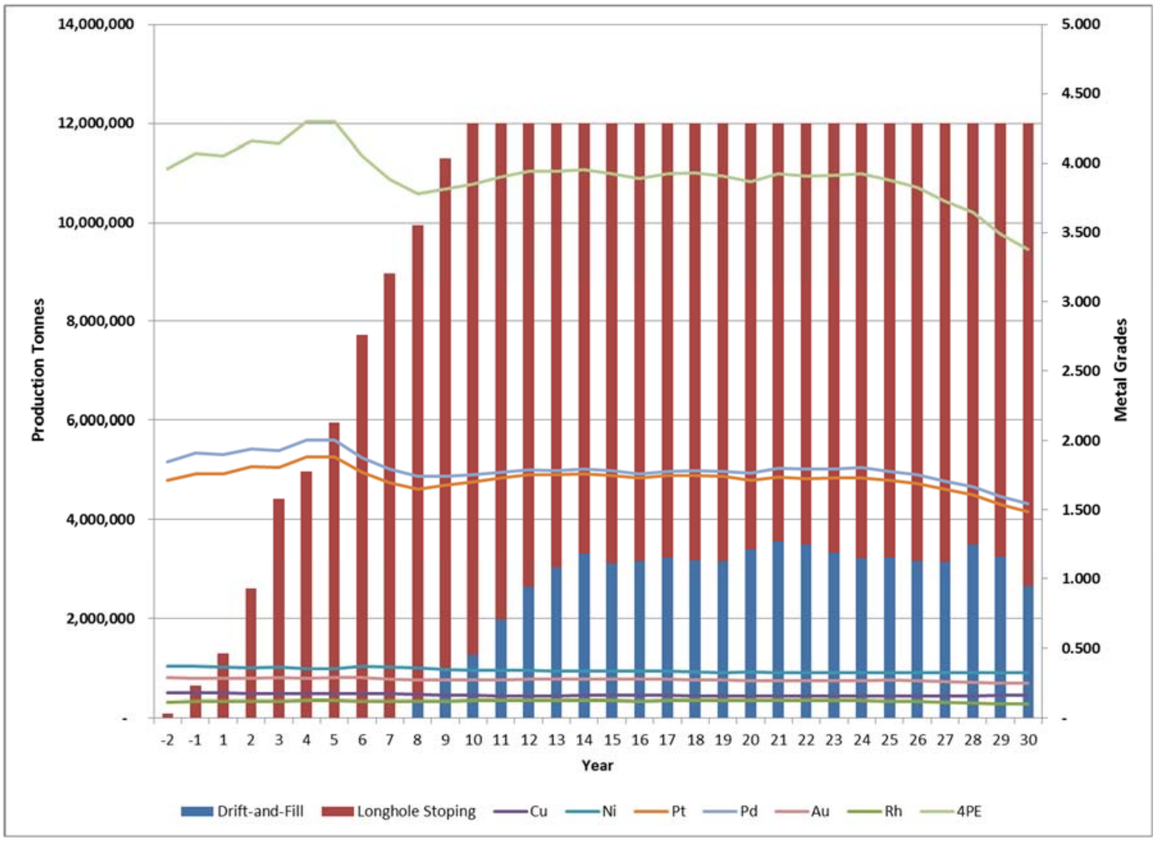
Table 16.28 and Figure 16.18 present summarized annual production schedules for Phase 3. Initial production build up is similar to Phase 2. Production builds up to 12 Mtpa over a period of approximately 10.5 years.

Table 16.28 Production Schedule for Mining – Phase 3 – 12 Mtpa

Year	t	NSR ($ / t)	Cu (%)	Ni (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)
-2	100,000	$139.83	0.18	0.37	1.71	1.85	0.29	0.12	3.96
-1	648,000	$142.80	0.18	0.37	1.76	1.91	0.29	0.12	4.07
1	1,296,000	$141.55	0.18	0.36	1.76	1.90	0.28	0.12	4.06
2	2,592,000	$143.78	0.18	0.36	1.81	1.94	0.28	0.12	4.16
3	4,424,000	$144.41	0.18	0.36	1.81	1.93	0.29	0.12	4.14
4	4,972,000	$146.92	0.17	0.35	1.88	2.00	0.29	0.13	4.30
5	5,944,000	$147.58	0.17	0.36	1.88	2.00	0.29	0.13	4.30
6	7,720,000	$142.99	0.17	0.37	1.77	1.88	0.29	0.12	4.05
7	8,972,000	$137.37	0.17	0.36	1.70	1.79	0.28	0.12	3.89
8	9,944,000	$133.47	0.17	0.36	1.65	1.74	0.27	0.12	3.78
9	11,296,000	$132.80	0.16	0.35	1.68	1.74	0.27	0.12	3.81
10	12,000,000	$133.12	0.16	0.34	1.70	1.76	0.27	0.12	3.85
11	12,000,000	$134.54	0.16	0.34	1.73	1.77	0.27	0.13	3.90
12	12,000,000	$135.39	0.16	0.34	1.75	1.79	0.28	0.13	3.94
13	12,000,000	$135.23	0.16	0.34	1.75	1.78	0.28	0.13	3.94
14	12,000,000	$135.23	0.16	0.34	1.76	1.79	0.28	0.13	3.96
15	12,000,000	$134.52	0.16	0.34	1.75	1.78	0.27	0.12	3.93
16	12,000,000	$133.66	0.16	0.34	1.73	1.76	0.28	0.12	3.89
17	12,000,000	$134.19	0.16	0.33	1.75	1.78	0.27	0.13	3.92
18	12,000,000	$133.38	0.16	0.33	1.75	1.78	0.27	0.13	3.93
19	12,000,000	$132.65	0.16	0.33	1.74	1.78	0.27	0.13	3.91
20	12,000,000	$131.55	0.16	0.33	1.71	1.76	0.27	0.12	3.87
21	12,000,000	$132.26	0.16	0.32	1.74	1.80	0.27	0.12	3.93
22	12,000,000	$131.60	0.15	0.32	1.73	1.79	0.26	0.12	3.91
23	12,000,000	$131.70	0.15	0.32	1.73	1.80	0.26	0.12	3.91

Year	t	NSR ($ / t)	Cu (%)	Ni (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	3PE+Au (g/t)
24	12,000,000	$132.22	0.15	0.32	1.73	1.80	0.27	0.13	3.93
25	12,000,000	$131.09	0.16	0.33	1.71	1.78	0.27	0.12	3.88
26	12,000,000	$129.89	0.16	0.33	1.69	1.75	0.27	0.12	3.83
27	12,000,000	$126.78	0.16	0.32	1.65	1.70	0.26	0.11	3.73
28	12,000,000	$124.47	0.16	0.33	1.61	1.67	0.26	0.11	3.64
29	12,000,000	$120.11	0.16	0.32	1.54	1.60	0.25	0.11	3.49
30	12,000,000	$116.39	0.16	0.32	1.49	1.54	0.25	0.10	3.38
Total	**309,908,000**	**$132.53**	**0.16**	**0.34**	**1.71**	**1.77**	**0.27**	**0.12**	**3.87**

Figure 16.18 Production Schedule – Phase 3 – 12 Mtpa



16.2.3 Mine Design

16.2.3.1 Phase 1 – 4 Mtpa

For Phase 1, primary access to the mine is via a 1,100 m deep, 7.3 m diameter production shaft (Shaft 2) Secondary access is via a 950 m deep, 7.3 m diameter ventilation/bulk sampling shaft (Shaft 1). During initial mine development, Shaft 1 will serve as a ventilation intake and Shaft 2 will serve as an exhaust. At mine production, both of these shafts serve as ventilation intakes. Two additional ventilation exhaust shafts (Shafts 3 and 4) are planned. Shaft 3 is 950 m deep, 7.3 m in diameter, and is located on the northern side of the mining area. Shaft 4 is 750 m deep, 6.0 m in diameter, and is located on the southern side. Shaft locations are shown in plan view in Figure 16.13.

Shaft functions and design parameters for Phase 1 are summarized in Table 16.29.

Table 16.29 Shaft Functions and Design Parameters – Phase 1 – 4 Mtpa

Shaft	Function	Diameter (m)	Depth (m)
1	Bulk Sample / Escape/ Vent Intake	7.3	950
2	Production - 4 Mtpa / Service / Vent Intake	7.3	1,100
3	Vent Exhaust	7.3	950
4	Vent Exhaust	6.0	750

Stope production will be loaded and trammed using LHD units to either local passes or directly loaded on to trucks for transport to the main pass system located at the production shaft. The local passes will transfer stope production to either of the two main haulage levels, i.e., 750 and 950 m levels, where trucks will then haul the material to the main pass. The main pass will feed an underground crusher system located below the 950 m level. The crusher discharges into a fines bin and then the material is fed onto a conveyor belt on the 1,050 m level that transfers to a loading pocket at the shaft.

Access ramps and mining sublevels provide access to mining areas. Production from below the 950 m level is loaded into trucks and hauled via a ramp to the 950 m level and to the main crusher bin at the production shaft.

Other major underground infrastructure for the Platreef project includes underground maintenance shops, fuel stations, pump stations, explosive magazines, storage areas, and offices. Primary ventilation fans are located underground.

16.2.3.2 Phase 2 – 8 Mtpa

Infrastructure for Phase 2 is similar to that for Phase 14 with the following modifications and additions.

- The diameter of Shaft 2 was increased to 10.6 m and the hoisting capacity was increased to accommodate the increased production rate.
- The diameter of Shaft 4 was increased to 8.7 m and the depth was increased to 950 m to provide additional exhaust capacity to ventilate resource areas mined in the Southwest Area. The shaft location was also moved approximately 250 m to the north-west.
- Two additional 7.3 m diameter ventilation shafts (Shafts 5 and 6) were added in the Southeast Area to support mining in that area. Shaft 5 is 950 m deep and Shaft 6 is 1,050 m deep.
- The 750 m and 950 m levels were extended to provide haulage and to support mining from the Southeast and Southwest Areas.
- Additional access ramps and sublevels are provided for the Southeast and Southwest Areas.

Shaft locations are shown in plan view in Figure 16.15. Shaft functions and design parameters for Phase 2 are summarized in Table 16.30.

Table 16.30 Shaft Functions and Design Parameters – Phase 2 – 8 Mtpa

Shaft	Function	Diameter (m)	Depth (m)
1	Bulk Sample / Escape/ Vent Intake	7.3	950
2	Production - 8 Mtpa / Service / Vent Intake	10.6	1100
3	Vent Exhaust	7.3	950
4	Vent Exhaust	8.7	950
5	Vent Exhaust	7.3	950
6	Escape / Vent Intake	7.3	1050

16.2.3.3 Phase 3 – 12 Mtpa

Infrastructure for Phase 3 is similar to that for Phase 2 with the following modifications and additions.

- The diameters of Shafts 5 and 6 are increased to 8.7 m to provide additional ventilation capacity in the Southeast Area.
- An additional 8.7 m diameter, 1,500 m deep production shaft (Shaft 7) was added in the Southwest Area to support mining of deeper high grade resources in that area.
- Crushing and materials handling systems were added for Shaft 7.
- Haulage Levels are developed on the 1,150 m and 1,350 m levels in the Southwest Area.
- Additional access ramps and sublevels are provided for the Southeast and Southwest Areas.

Shaft locations are shown in plan view in Figure 16.17. Shaft functions and design parameters for Phase 3 are summarized in Table 16.31.

Table 16.31 Shaft Functions and Design Parameters – Phase 3 – 12 Mtpa

Shaft	Function	Diameter (m)	Depth (m)
1	Bulk Sample / Escape/ Vent Intake	7.3	950
2	Production - 8 Mtpa / Service / Vent Intake	10.6	1100
3	Vent Exhaust	7.3	950
4	Vent Exhaust	10.6	950
5	Vent Exhaust	8.7	950
6	Escape / Vent Intake	8.7	1,050
7	Production - 4 Mtpa / Service / Vent Intake	8.7	1500

Figure 16.19 is an elevated view showing the mining areas and preliminary layouts for Phase 3.

Figure 16.19 Phase 3 –12 Mtpa Development (Looking South-east)

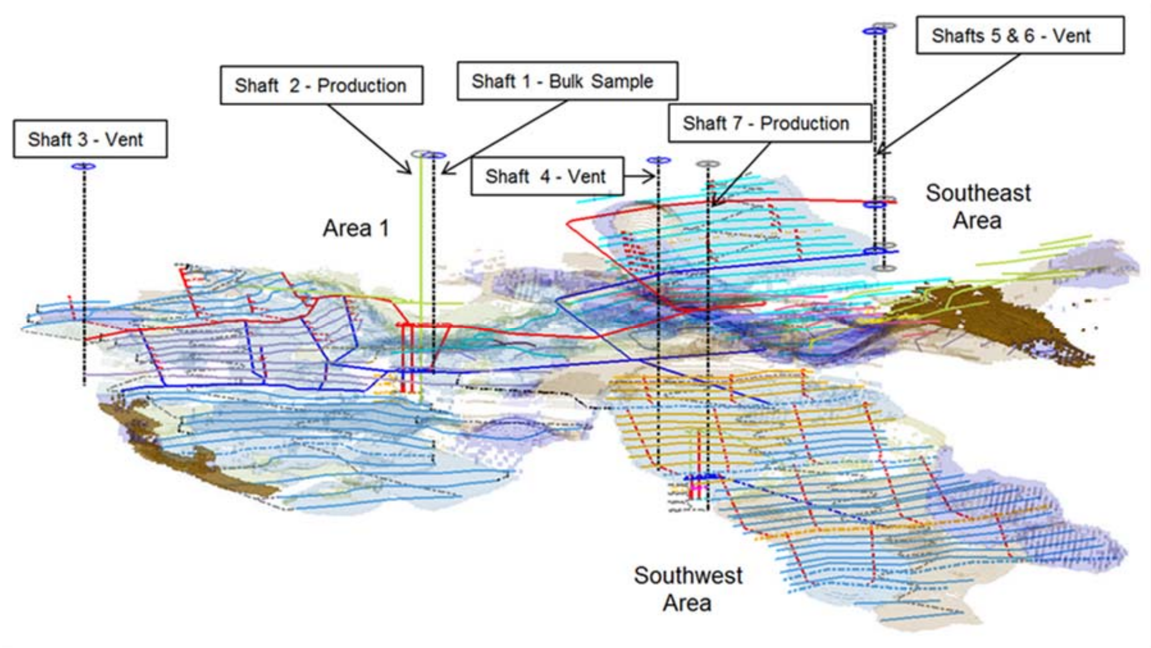


Figure by Stantec. Note: Colors are for presentation only; they have no technical significance.

Figure 16.20 is a similar view that roughly illustrates the relative mine expansions for Phases 1, 2, and 3.

Figure 16.20 Phase1 to 3 Mine Expansions (Looking South-east)

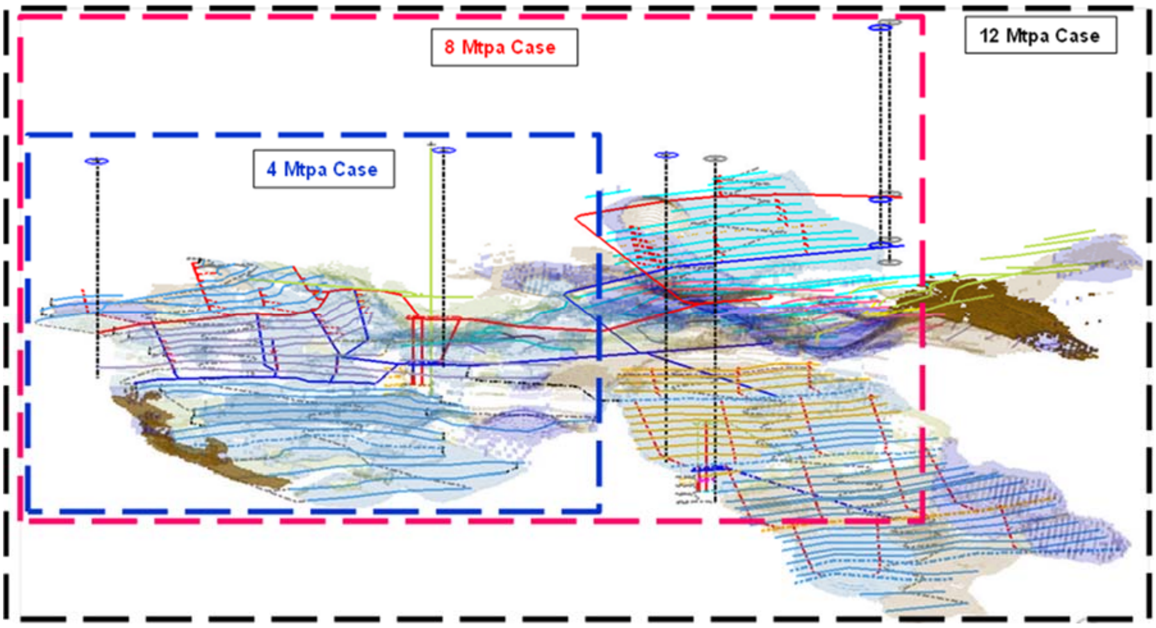


Figure by Stantec. Note: Colors are for presentation only; they have no technical significance.

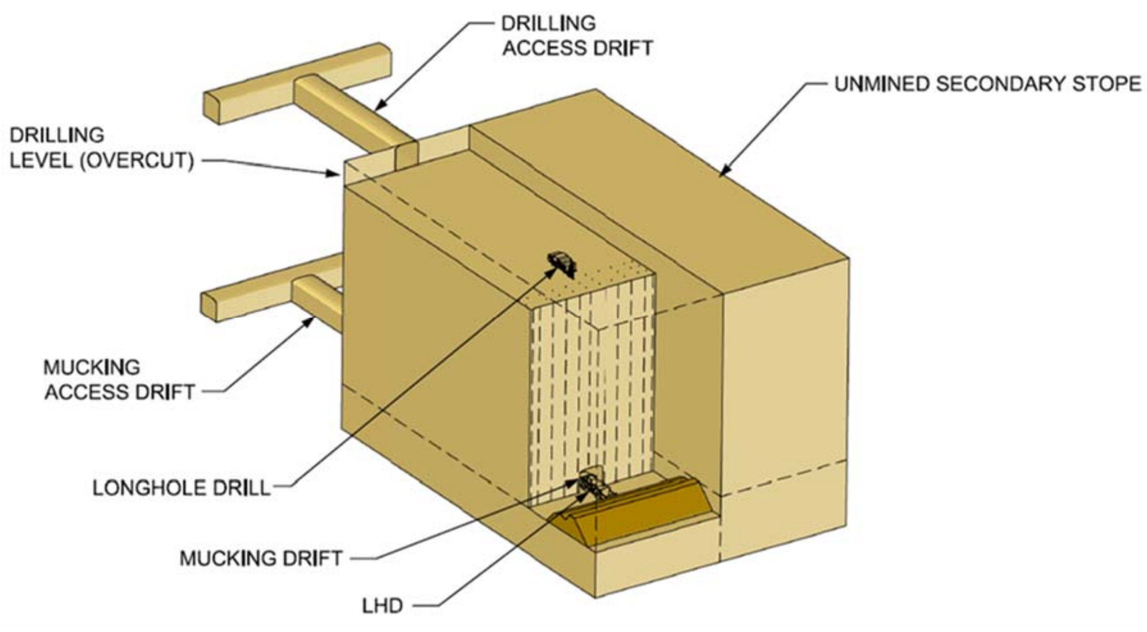
The design parameters for the infrastructure, stope development, and stopes can be found in Section 16.2.5

16.2.4 Mining Methods

A combination of mining methods is proposed for extraction of the Platreef PEA inventory. Mining zones with vertical thicknesses greater than or equal to 18 m will be mined using the longhole stoping method while thinner zones will be mined using mechanized drift-and-fill or drift. Stope method selection will be based in part on the information gained from a delineation drilling programme that will be done from diamond drill stations that have been excavated with footwall drift advance. This in-fill drill programme will provide detailed local geological information that will be used for stope planning and production forecasting.

The primary mining method selected for the Platreef project is longhole stoping utilizing paste backfill for post-mining support. The longhole stoping method is considered a bulk mining method with good recovery and minimal dilution. Each stope includes a drill drift at the top of the stope and an extraction drift at the bottom, leaving a bench in between the two drifts. Once the drifts are mined, the bench is then drilled, blasted, and excavated. The open stope is then filled with paste backfill and allowed to cure before extraction of the adjacent stope blocks. A typical stope layout is presented in Figure 16.21.

Figure 16.21 Typical Longhole Stope Layout

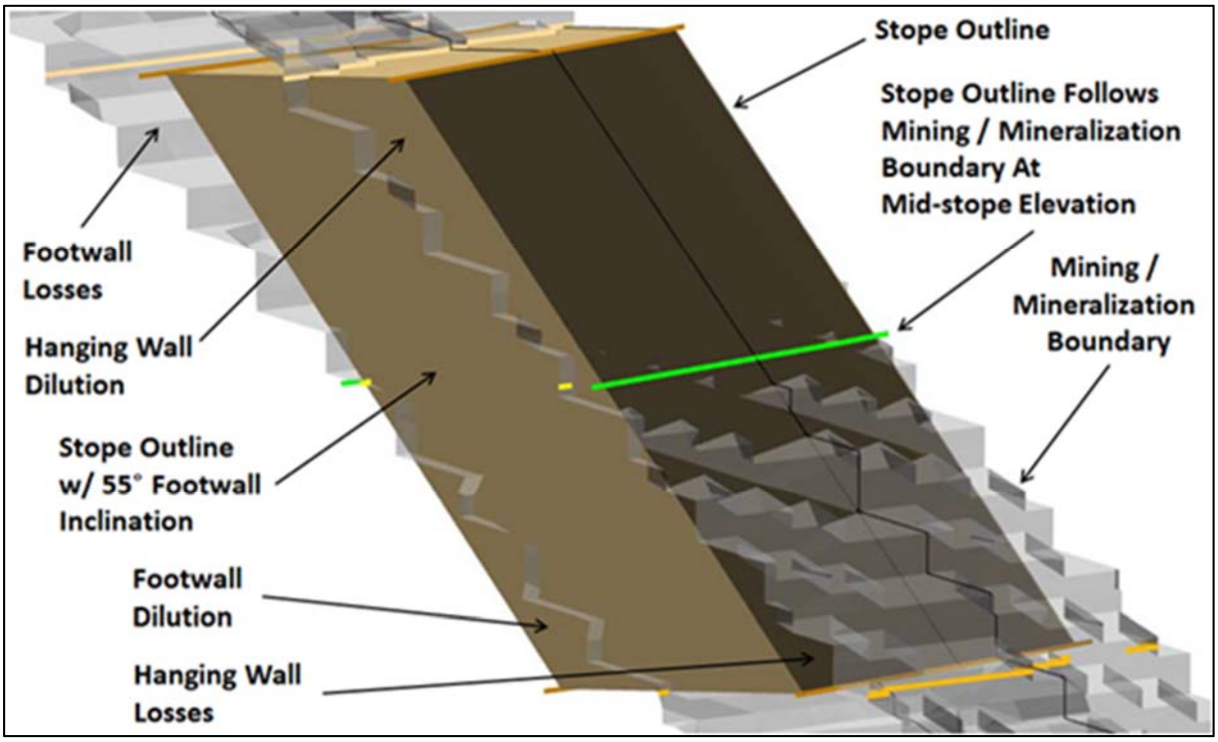


SRK Consulting (SRK) has provided preliminary design parameters for stope design. In general, unsupported stope walls will be designed with a maximum hydraulic radius of 8 m. For cable bolted stope backs, a maximum hydraulic radius of 6.5 m is recommended.

The dip of the mining zones at Platreef varies from nearly flat to as much as 45° to 50°. Different approaches to longhole stoping are taken for accessing and mining depending on the dip of the individual mining zones.

In the moderately dipping zones, the stopes are oriented approximately perpendicular to the strike of the mining zone (transverse longhole stoping). Stopes will be designed to be 30 m high by 15 m wide. SRK's recommendations limit the open stope length to approximately 35 m. Stopes longer than 35 m will be mined in two or more segments. The footwall of the stope will be designed with a minimum inclination of 55° in order to assure proper flow along the footwall to the mucking level at the stope bottom. Significant dilution and mining losses will occur on the footwall since the dip of the mining zone is less than 55°. Stope hangingwalls will be drilled as flat as possible in order to minimize dilution and mining losses. Figure 16.22 illustrates the manner by which stopes will be designed in moderately dipping mining zones.

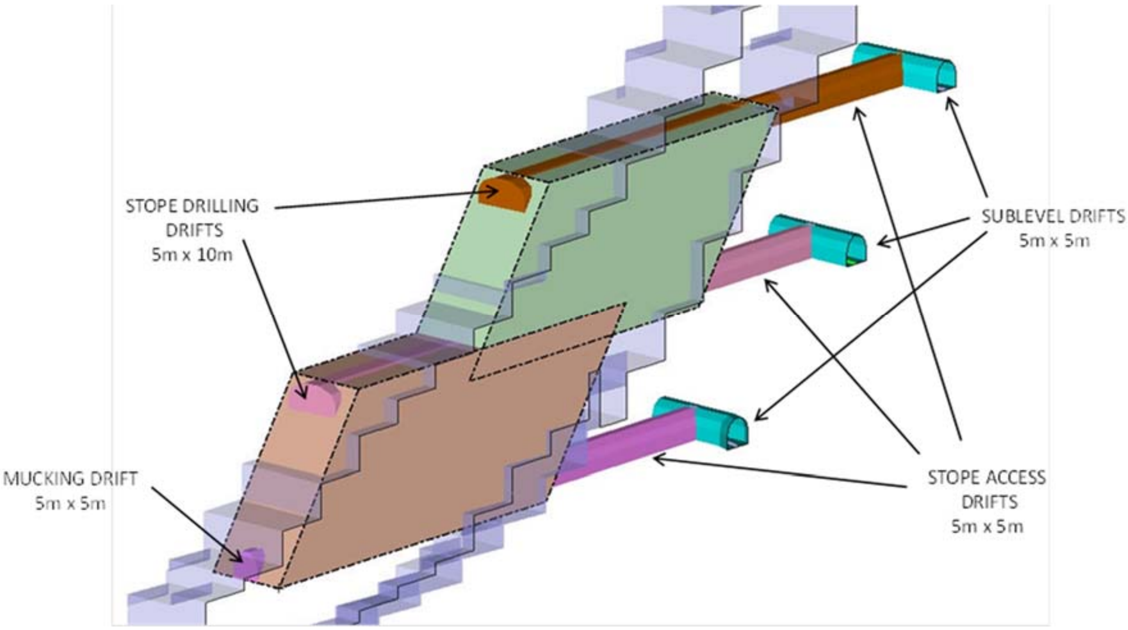
Figure 16.22 Transverse Longhole Stope Design Parameters – Moderately Dipping Zones



Access to transverse longhole stopes is from sublevels driven in the underlying footwall waste rock, parallel to the mining zone and on 25 m vertical intervals. Stope access drifts measuring 5 m x 5 m are driven from the sublevels to the mining zones. The bottom stope access drift is driven through the mining zone from footwall to hangingwall at 5 m x 5 m. The upper access drift is widened to10 m upon reaching the mining zone. A slot raise is blasted at the end of the stope and the remainder of the stope is ring drilled and blasted on retreat toward the stope entrance.

Figure 16.23 shows an example of a typical transverse longhole stope. In similar 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 16.23 Example Transverse Longhole Stope Configuration – Moderately Dipping Mining Zones



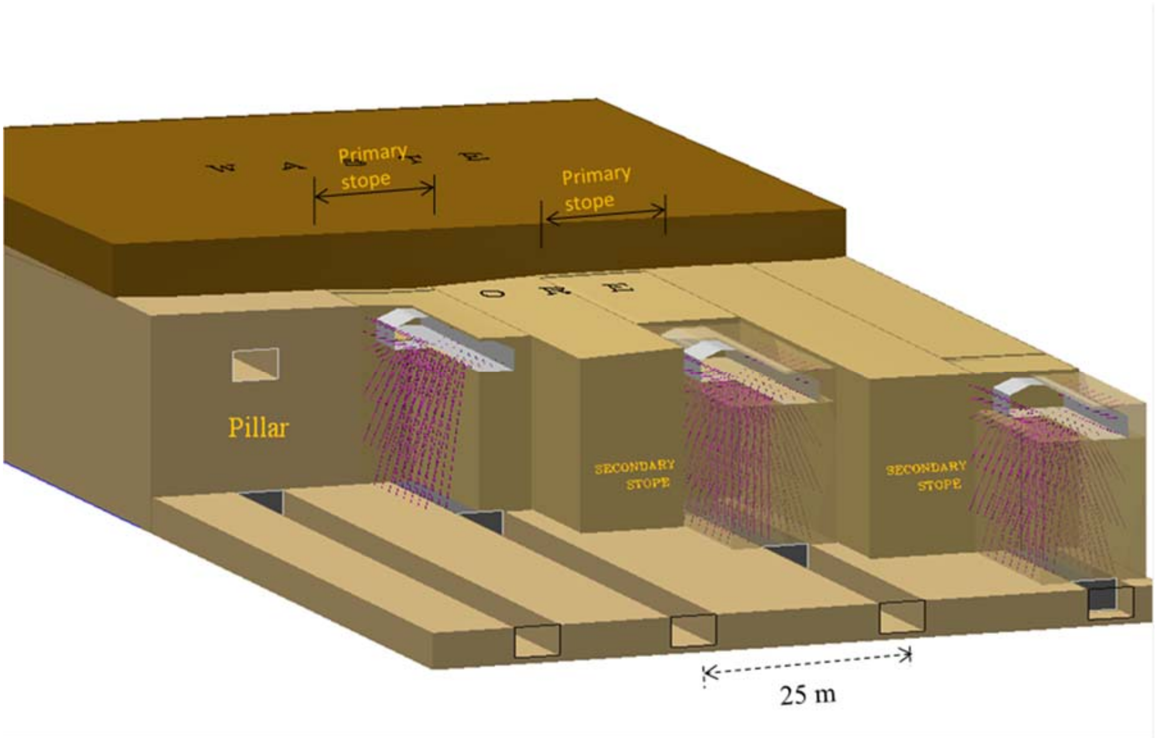
The design parameters for the stopes are discussed in Section 16.2.4.1.

In the gently dipping mining zones, the thickness of the mineralized material is variable and both the hangingwall and footwall contacts are irregular. Stopes will be designed with their long axes parallel to the overall strike of the mining zone (longitudinal longhole stoping). Stope widths will be 15 m. Stope heights will be variable with a minimum height of 18 m and a maximum height of approximately 31 m. Where the vertical thickness of the mining zone exceeds 31 m, stopes will be mined in two lifts. The lower lift is mined and backfilled prior to mining the upper lift. The overall length of individual stopes will vary with the width of the mining 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. 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 continues in a retreating fashion until the remainder of the stope is extracted.

The stopes are mined using a primary and secondary extraction sequence. The primary stopes are mined and backfilled first, followed by the mining and backfilling of the secondary stopes. Figure 16.24 shows a typical primary / secondary longitudinal longhole stope mining sequence. The design parameters for the stopes are discussed in Section 16.2.5.

Access to longitudinal longhole stopes is from sublevels driven in the underlying footwall waste rock, parallel to the mining zone and on approximately 25 m vertical intervals. Stope access drifts measuring 5 m x 5 m are driven from the sublevels on approximately 150 m centers at both the top and bottom of the mining zone. Due to the potentially large horizontal expanse of the mining zone defined by the development accesses, multiple 75 m x 150 m stopes can be accessed from each pair of access drifts. The upper access drift is widened to10 m upon reaching the limits of the mining zone and slot raises are opened at the stope ends to allow the remainder of the stope to be ring drilled and blasted on retreat toward the stope access.

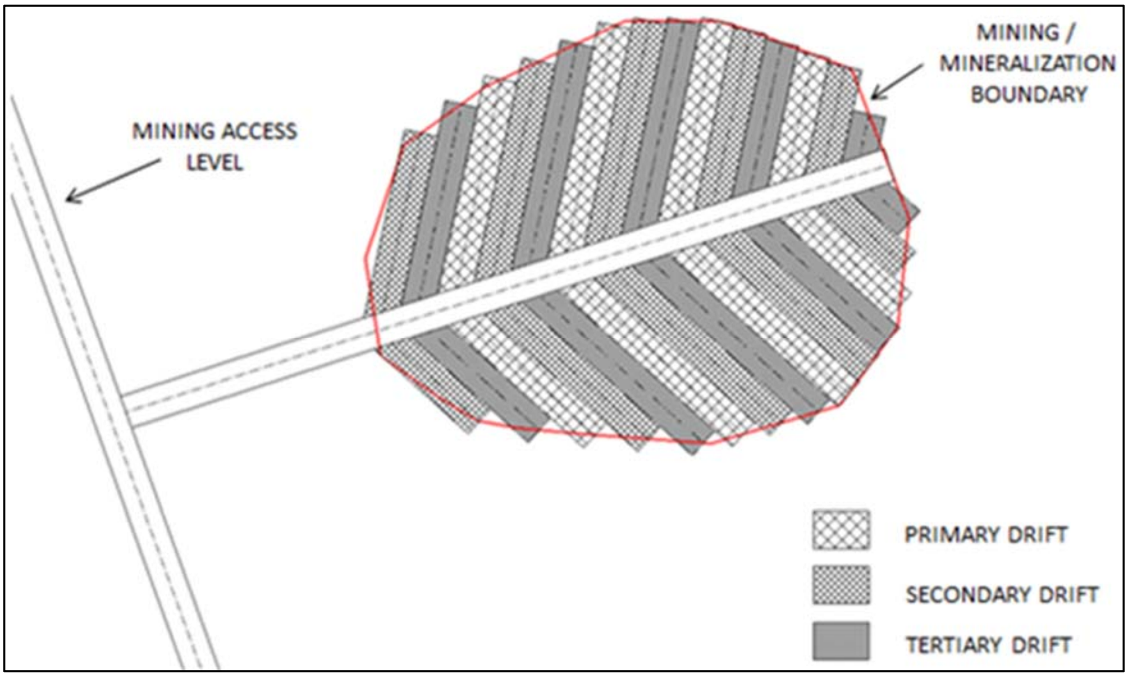
Figure 16.24 Longitudinal Longhole Stope Mining Sequence – Gently Dipping Zones



16.2.4.1 Drift-and-Fill

In drift-and-fill mining, the mining zone is divided into horizontal slices, and drifts are mined adjacent to one another. Upon completion of each drift, the void is backfilled with paste backfill. After the backfill has consolidated, another drift is driven next to the fill. Mining progresses in this manner until the entire horizontal slice is removed. 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 production from an individual slice. Figure 16.25 shows a typical primary-secondary-tertiary drift-and-fill layout.

Figure 16.25 Typical Drift-and-Fill Level Layout



Where ground conditions are poor, it may be necessary to mine and backfill the drifts individually on retreat from the far end of the level toward the entrance. Productivity is reduced in such cases.

Drift-and-fill mining can be performed in either an overhand or underhand fashion. In overhand mining, after each complete horizontal slice is mined and backfilled, mining progresses to the slice above. In underhand mining, after each complete horizontal slice is mined and backfilled, mining progresses to the slice below and mining takes place beneath the cemented backfill placed in the lift above. Stantec recommends the use of overhand methods at Platreef because the relative weakness of paste backfill would make underhand mining dangerous without the addition of large amounts of cement to the backfill.

Drift-and-fill mining is a very flexible mining method that allows nearly complete recovery of the mining zones. Mining is completed with the same equipment used for mine development. Dilution from waste external to the mining zone is minimal. Productivity is relatively low due to the small blast sizes. 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.

16.2.5 Mine Design Parameters

The design parameters for the mining areas are based on geotechnical recommendations provided by SRK. The geotechnical information is utilized for determining basic drive dimensions, stope width and height, 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.

The ground support criteria utilized for all development includes threaded rebar bolts and wire mesh with a bolt spacing of 1.5 m. Ground support designs are at a conceptual level and are generalized to meet the minimum support requirements.

Ore development for the longhole stopes consists of a drill drift at the top and an extraction drift at the bottom of each stope. The drill drift provides access for production drilling of the stope. The drill drift is developed 5 m high by 10 m wide. Cable bolts are installed to ensure stability of the back. The bottom extraction drift is designed at 5 m high by 5 m wide. Production drilling and blasting will widen the stope to 15 m over its full height. The drive dimensions for the longhole stopes are presented in Table 16.32.

Table 16.32 Longhole Stoping Development Heading Sizes

Ore Development	Height (m)	Width (m)
Sublevel Drift	5.0	5.0
Top Stope Access Drift	5.0	5.0
Top Drill Drift	5.0	10.0
Bottom Stope Access Drift	5.0	5.0
Bottom Extraction Drift	5.0	5.0

Longhole stopes in the moderately dipping mining zones are assumed to be 30 m high by 15 m wide. Overall stope length is variable depending on the thickness of the mining zones. SRK's preliminary design criteria limit the maximum open stope segment length to approximately 35 m. The stopes are designed with a minimum inclination of 55° in order to ensure flow to the extraction drift and to minimize losses and dilution from the hangingwall and footwall. Stope design parameters for the moderately dipping zones are presented in Table 16.33.

Table 16.33 Stope Design Parameters – Moderately Dipping Mining Zones

Dimensions	30 m high by 15 m wide by variable length
Maximum Open Stope Length	35 m
Dilution Assumption	15%
Minimum Stope Inclination	55°

The stopes in the gently dipping mining zones are also assumed to be 15 m wide. Stope heights will vary depending on the thickness of the mining zone. Overall stope lengths and maximum open stope segment also vary with stope height. All stope walls are vertical except at the hangingwall and footwall contacts.

Table 16.34 provides the stope design details for the gently dipping mining zones.

Table 16.34 Stope Design Parameters – Gently Dipping Mining Zones

Dimensions	Variable height by 15 m wide by variable length
Maximum Open Stope Length	Varies by stope height
Dilution Assumption	15%
Stope Wall Angles	All vertical (except at hangingwall and footwall contacts)

16.2.6 Mine Development Schedule

The current project development schedule assumes that development of Shaft 1 (bulk sampling shaft) begins in April 2014 and that development of Shaft 2 begins in July 2014. Shaft 1 is sunk initially to a depth of approximately 800 m, where it intersects the mining zone. A bulk sample for metallurgical testing is taken at that elevation. Sinking then resumes and proceeds to a final depth of 975 m. During sinking, shaft stations are established at the 450, 750, 850, and 950 m levels.

Shaft 2 is sunk to a final depth of 1,100 m. Shaft stations are established at the 450, 750, 850, 950, and 1,050 m levels. After completion of sinking, the shaft is equipped and the production hoisting systems is commissioned. Shaft 2 is completed in December 2016.

After sinking of Shaft 1 is completed, a temporary loading station is installed on the 950 m level. The loading station capacity is approximately 1,500 t/d. The loading station allows LHDs to directly load the shaft sinking buckets. A waste pass connecting the 750, 850, and 950 m levels is also developed. The loading station and waste pass will allow development to proceed from those three levels prior to commissioning of the production shaft. Shaft 1 is completed in April 2016.

After completion of Shaft 1, a programme of drifting and diamond drilling is performed for the purpose of confirming continuity of the mineralized zone. When Shaft 2 reaches the 950 m level, a ventilation loop to surface is established, allowing the rate of underground development to increase. Development will initially focus on excavations required for construction of the main truck dumps, crusher, materials handling system, and other infrastructure located near the shafts. Driving of ventilation drifts toward Shafts 3 and 4 also begins shortly after completion of Shaft 1.

Sinking of Shaft 3 begins after Shaft 1 is completed and sinking of Shaft 4 begins after Shaft 2 is completed. Shaft 3 is completed in June 2017 and Shaft 4 is completed in September 2017.

Commissioning of the crusher and materials handling systems for the production shaft occurs in December 2018. Initial production from stope development begins in July 2018. Production from longhole stoping begins in January 2019.

Major milestones in the pre-production development and construction schedule are shown in Table 16.35.

Table 16.35 Pre-production Development Milestones

Milestone	Date
Begin Shaft 1 Construction	Apr 2014
Begin Shaft 2 Construction	Jul 2014
Complete Bulk Sampling Programme	Nov 2015
Complete Shaft 1	Apr 2016
Complete Shaft 2	Dec 2016
Complete Shaft 3	Jun 2017
Complete Shaft 4	Sep 2017
Commission Materials Handling System	Dec 2017
Begin Stope Development	Jul 2018
Begin Stope Production	Jan 2019

16.2.7 Mine Equipment Requirements

The mine equipment requirements are split into two categories, fixed equipment and mobile equipment. The equipment requirements for each category are estimated at a conceptual level of accuracy. The following are the design criteria for sizing, selection, and quantifying fixed and mobile equipment.

- Mining Method.
- Mine Production Rate.
- Ventilation Requirements.
- Mine Design Criteria.

A list of major fixed equipment by category is presented in Table 16.36.

Table 16.36 Fixed Mine Equipment (12 Mtpa)

Shatfs and Hoists	Material Handling
Production Hoist Service Hoist Auxiliary Hoists (2) Head Frames (2) Hoist Wire Ropes, Sheaves, and Rope Attachments Loading Pocket Procduction 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 Rescure Equipment for 15 Persons Cap Lamps & Chargers Self Rescuers Miscellaneous Sanitary Supplies and Units Refuge Chambers
Electrical	**Undergound Shop**
Main Substations U/G Substation Mine Load Centers 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

Table 16.37 provides a list of selected equipment, maximum quantity required LOM and does not include replacement equipment.

The mobile equipment selected is based on the criteria listed above and information from similar projects. The mobile equipment quantities are estimated at a conceptual level and based on historic performance rates and number of crews scheduled throughout the production schedule. The equipment quantity calculation includes projected equipment availability and spares.

The overall quantities fluctuate over the LOM to match the production and development schedule requirements at any given time.

The equipment rebuild and replacement methodology utilized for the Platreef 2014 PEA assumes the equipment is rebuilt three years after purchase, and then replaced three years later. Based on historical rebuild and replacement schedules, a factor of 15% of the total initial equipment purchase cost, applied to each year, covers mobile equipment rebuild and replacement costs.

Table 16.37 Mobile Equipment (Maximum Operating Quantities)

Mobile Equipment	Maximum Quantity Phase 1	Maximum Quantity Phase 2	Maximum Quantity Phase 2
Drill Jumbo, 2-Boom, Sandvik, DD420-40 6-9500-I Jumbo	11	19	30
Drill Jumbo, Single Boom, Atlas Copco Boltec, Rock Bolter (1 Boom With Screen Handler)	6	10	16
Drill Jumbo, Single Boom, Cable Bolting, Atlas Copco Cabletec LC, Cable Bolter	2	3	4
Production Elec/Hydraulic Drill, Sandvik, DL310-7 7-6325-F	8	15	15
LHD, 5.4m^3, Sandvik, LH514 with Remote package	16	27	39
Haul Truck – Sandvik, TH540	12	22	26
Explosives Truck/Jumbo, ANFO, Maclean Engineering, AC-3 ANFO Charger	7	12	16
Shotcrete Sprayer	1	1	2
Shotcrete Carrier	2	2	4
Scissor Lift Truck, Maclean Engineering, SI-3 Mine-Mate	4	4	4
Tractor, General Purpose, Kubota, RTV900 4x4	10	14	18
U.G. Road Grader, Caterpillar,12 m	2	2	3
Cassette Carrier, Maclean Engineering, CS-3 (approx 5-Tonne Payload)	4	6	8
Fuel Cassette, Maclean Engineering, CS-3 Fuel Cassette, 5,000 Liter Tank	2	3	4
Fuel/Lube Cassette, Maclean Engineering, CS-3 Fuel /Lube Cassette, 2,000 Liter Tank	2	3	4
Flatbed Cassette, Maclean Engineering, CS-3 2.4 m Wide Flatbed Cassette	2	3	4
Water Spraying	2	2	3
Subtotal	**92**	**148**	**200**

17 RECOVERY METHODS

17.1 Introduction

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

The development scenarios for the Platreef Project are shown in Figure 17.1 below.

Figure 17.1 Platreef Project Development Scenarios

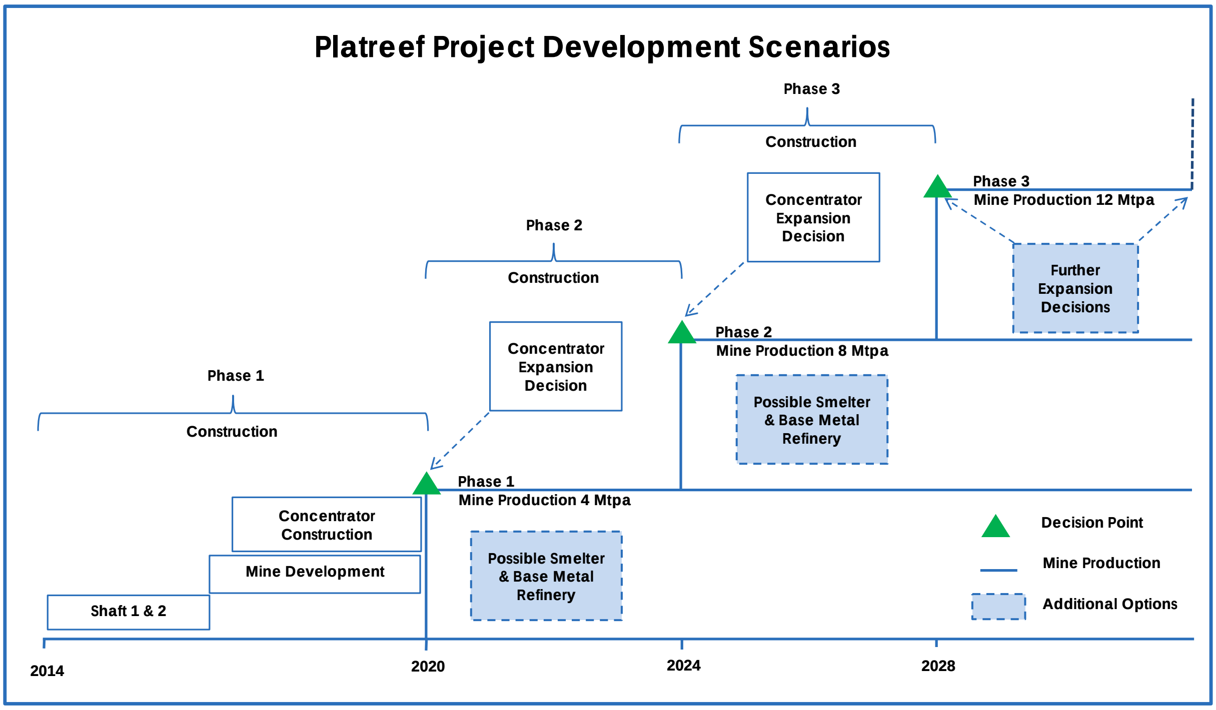


Figure by OreWin

Phase 1 includes the construction of a 4 Mtpa concentrator and other associated infrastructure in 2020. Phase 2 includes a ramp-up to 8 Mtpa in 2024, and in Phase 3 a further ramp-up to a plant capacity of 12 Mtpa in 2028.

A two staged production approach was used for the Phase 1 flow sheet development and design. The plant will be constructed in increments of 2 Mtpa, the selected flowsheet comprises of a common 3-stage crushing circuit (2-stage crushing during stage 1A), two by 2 Mtpa milling-flotation modules, followed by a common thickening and filtration circuit.

17.2 Process Design Criteria

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

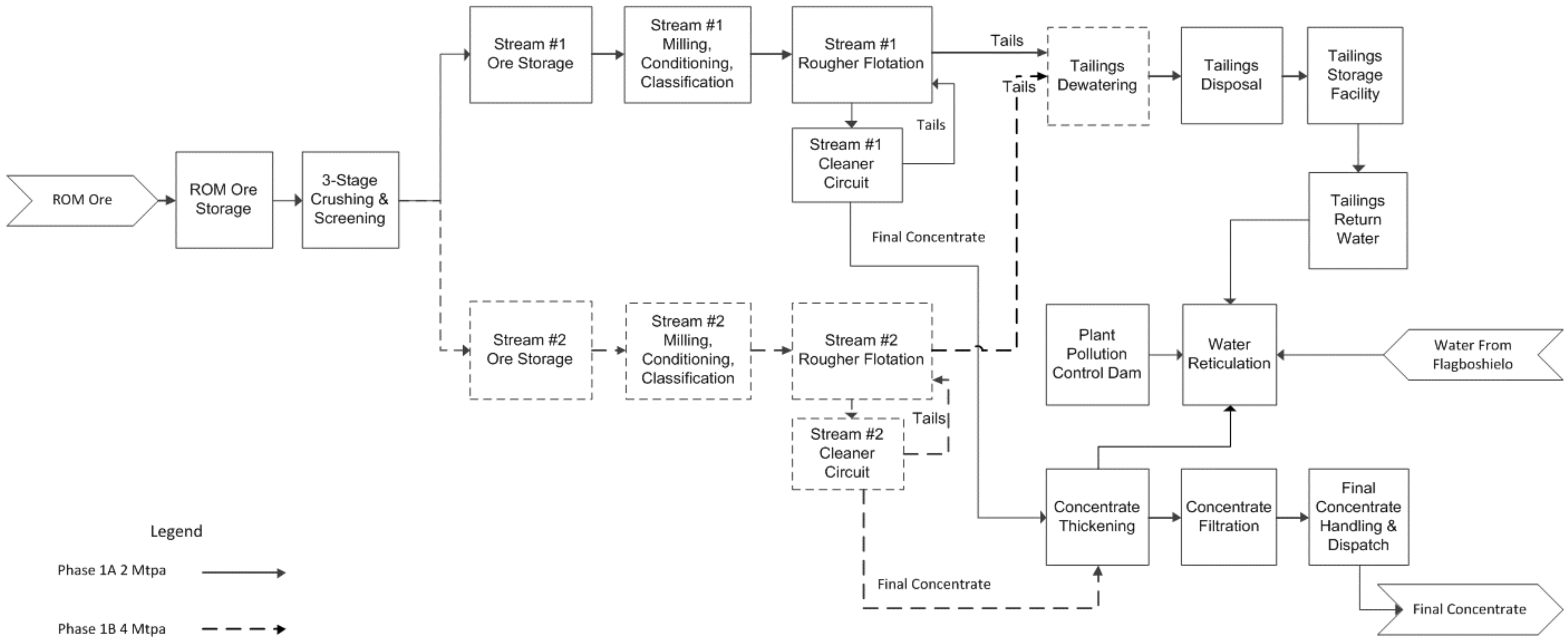
Table 17.1 Basic Process Design Criteria

Criteria	Units	Nominal	Design
Production Summary			
Mining			
Annual Mining Rate year –1	tpa	–	24 828
Annual Mining Rate year 1	tpa	–	802 368
Annual Mining Rate year 2	Mtpa	–	2.56
Annual Mining Rate year 3	Mtpa	–	3.77
Annual Mining Rate year 4+	Mtpa	–	4.04
Life of Mine	Years	–	33
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	1.60	2.05
Palladium	g/t	1.68	2.20
Rhodium	g/t	0.10	0.16
Gold	g/t	0.25	0.33
3PE+Au	g/t	3.64	4.70
Copper	%	0.15	0.19
Nickel	%	0.30	0.37
Overall Recovery			
3PE+Au recovery	%	–	85
Nickel recovery	%	–	67
Copper recovery	%	–	88
Concentrate Grades			
3PE+Au grade target	g/t	–	90
Nickel grade target	%	–	8
Copper grade target	%	–	5
Crushing Operating Schedule			
Operating days per annum	days	–	365
Operating shifts per day	#	–	2
Hours per shift	h	–	8
Availability	%	–	70
Utilisation	%	–	90
Crushing circuit running time	%	–	63
Overall running time	h/annum	–	5519
Circuit maximum feed rate Phase 1A	t/h	362	500
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	%	–	92
Utilisation	%	–	98
Milling circuit running time	%	–	90
Overall running time	h/annum	–	7898
Circuit feed rate	t/h/module	253	279
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 17.2 for a high level block flow diagram of the Platreef Project concentrator plant.

Figure 17.2 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 250 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 vibrating feeder, via a 50 t reclaim bin. The reclaimed material is conveyed to the RoM silo.

During the production phase the RoM material will be conveyed to the 6,000 t RoM silo. 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 final crusher circuit will consist 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 classification 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 (P_{100} 93 mm) is conveyed, together with the secondary crusher product, to the coarse screening section.

The primary and secondary crusher product is collected in two 110 t coarse screening feed bins, from where it is extracted and fed to two double deck vibrating coarse screens by means of vibrating feeders. The top decks of the coarse 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 is choke fed by means of a vibrating secondary crusher feeder discharging onto the secondary crusher feed conveyor, which in turn feeds the secondary crusher. The secondary crusher product has a P_{80} of 50 mm (P_{100} 89 mm), and is transferred back to the coarse screening section. The bottom decks of the coarse 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 P_{80} of 11 mm (P_{100} 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 screening section. The tertiary screening product (- 11 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.

17.3.3 Mill Feed Storage

The crusher circuit product (-11 mm) is stored in two 6,000 t 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 are 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.

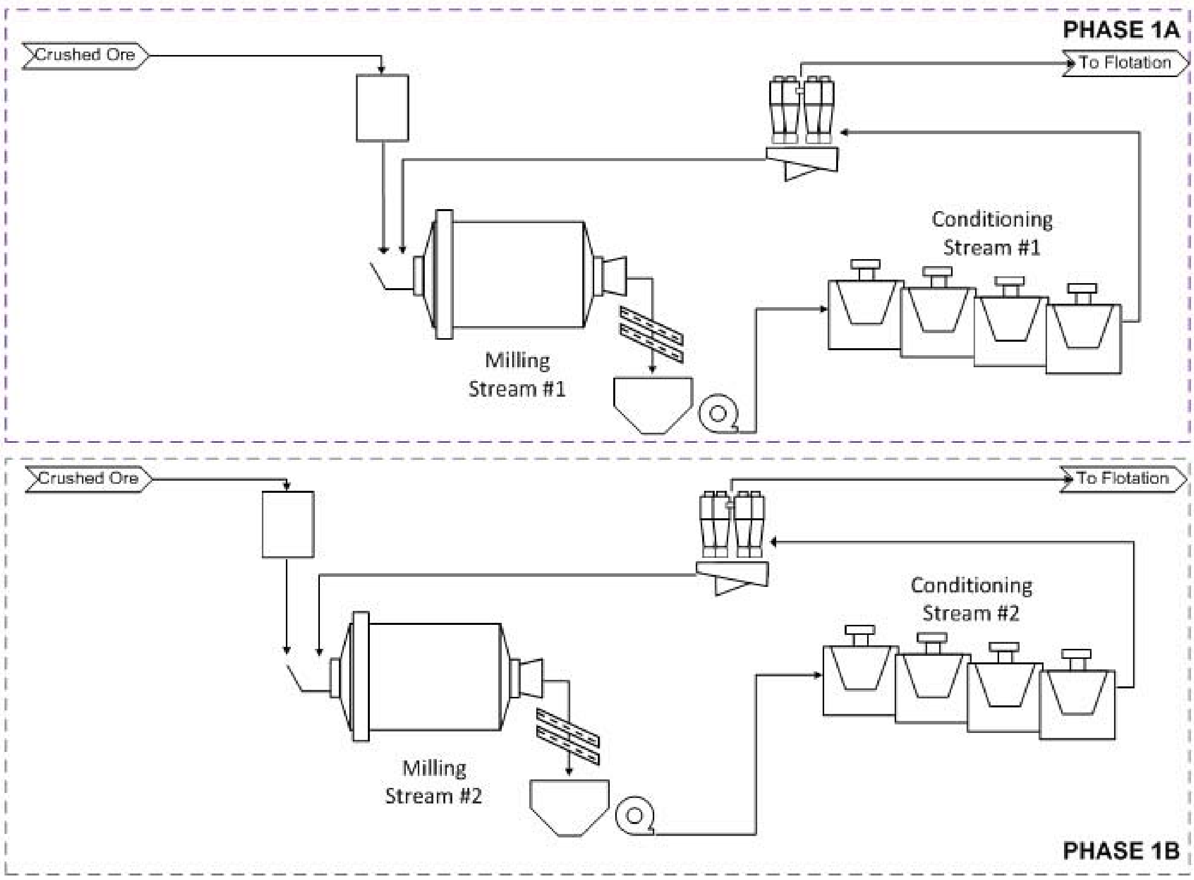
The milling circuit consists of an 8 MW, 21 ft diameter x 30 ft effective grinding length (EGL), grate discharge ball mill, operating in closed circuit with a classification cyclone cluster. The mill feed material (F_{100} 11 mm, F_{80} 7 mm) is fed to the mill feed hopper where process water is added for in-mill density control. Thiourea and oxalic acid are also added to the mill feed hopper.

The milled material (P_{80} 200 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 reports to the mill classification cyclone cluster, which produces an overflow product of P_{80} 75 µm. The cyclone underflow is recycled to the mill feed hopper for 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 17.3.

Figure 17.3 Milling Circuit Flow Diagram



17.3.5 Pre-Conditioning

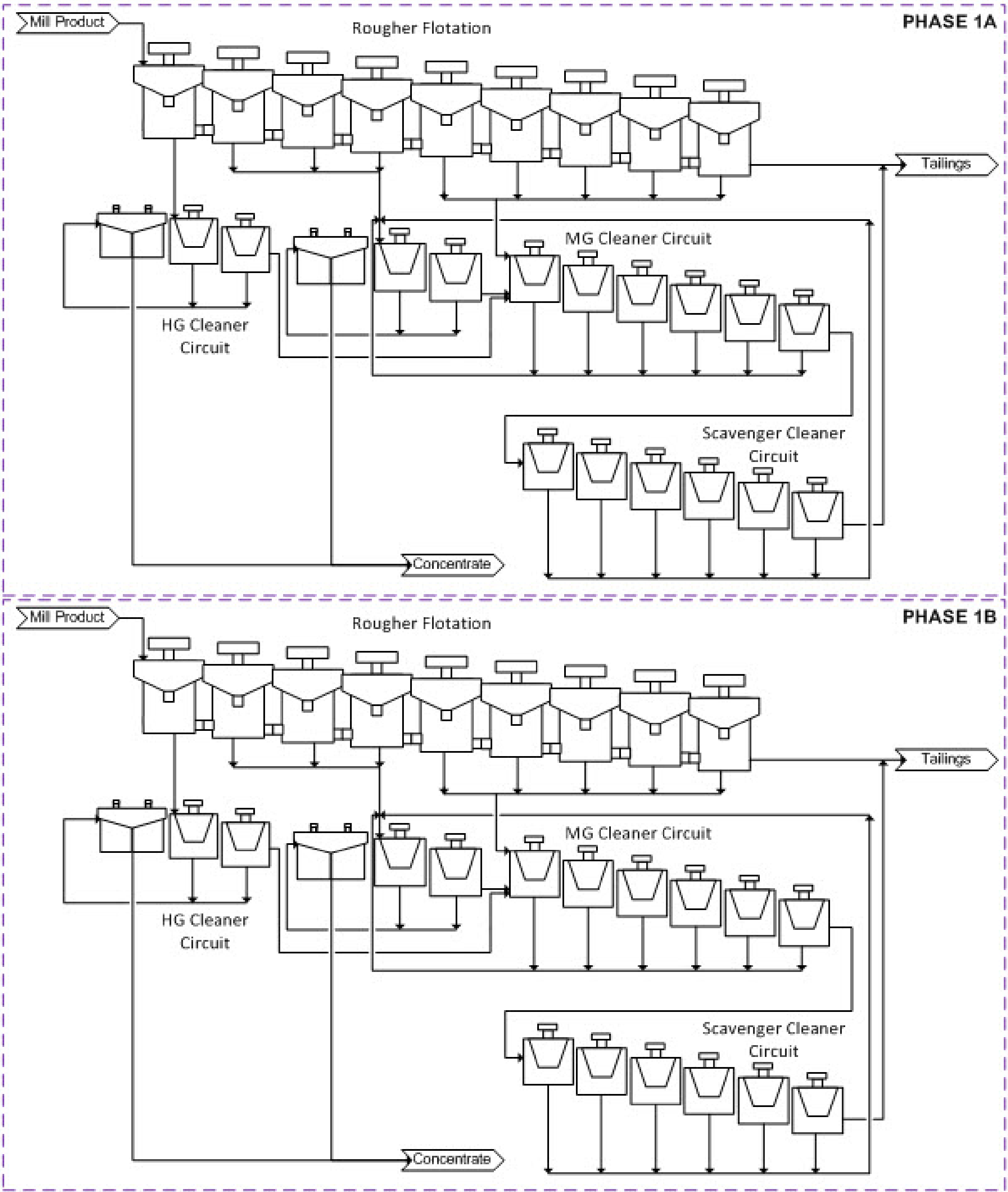
The pre-conditioning circuit for Phase 1B (4 Mtpa processing rate) consists of 2 identical installations, each capable of treating 2 Mtpa. Only one of the two streams will be installed during Phase 1A. A single pre-conditioning stream is described below.

The milled product is pumped to the first of four 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 Flotation Circuit

The flotation circuit is schematically presented in Figure 17.4.

Figure 17.4 Flotation Flow Diagram



17.3.6.1 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 streams will be installed during Phase 1A. A single stream is described below.

The milling classification cyclone overflow, at approximately 35% solids (w/w), reports to the rougher flotation bank via a sampler. The rougher flotation bank consists of 9 × 100 m^3, 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 and the scavenger cleaner tails gravitate to a tails sump via a two stage sampling system. The rougher tailings, together with the 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 roughers; while Oxalic acid and Thiourea is added to each of the concentrate sumps.

17.3.6.2 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 streams will be installed during Phase 1A. A single stream is described below.

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

The high grade rougher concentrate reports to the high grade cleaner circuit (2 × 20 m^3 forced air tank cells, 12 minutes total residence time). The high grade cleaner concentrate reports to the high grade recleaner flotation bank (2 × 5 m^3 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 is pumped to the medium grade cleaner cells.

The medium grade rougher concentrate, together with the scavenger cleaner concentrate and the high grade cleaner tailings, report to the medium grade cleaner circuit (6 × 30 m^3 forced air tank cells, 25 minutes total residence time). The medium grade cleaner concentrate reports to the medium grade recleaner flotation bank (2 × 20 m^3 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 m^3 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 feeds to the medium grade cleaner bank, while the medium grade cleaner tails reports to the scavenger cleaner flotation bank, together with the low grade rougher concentrate.

SIPX, frother, depressant and Aero 3477 is 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.

17.3.6.3 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 streams will be installed during Phase 1A. A single stream 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 m^3, 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 is added to the scavenger cleaner cells.

17.3.7 Tailings Dewatering & Disposal

During Phase 1A the flotation tailings product is pumped to the tailings storage facility (TSF) as a dilute slurry (33% 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.

With the commissioning of Phase 1B the flotation tailings product, at approximately 34% solids (w/w) is pumped to the tails dewatering cyclone cluster. The thickened 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 40 g/t to produce a thickener underflow product of 50% solids (w/w) which is pumped to the tailings disposal tank, where it combines with the dewatering cyclone overflow 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% (w/w) by means of a duty/standby pump train installation.

17.3.8 Concentrate Handling & Filtration

The final concentrate from the cleaner flotation circuit (at a density 1.21 t/m^3) 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 40 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 m^2 expandable to 96 m^2). The filter cake, with a moisture content of 12%, 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.

17.3.9 Sampling and Ancilliaries

17.3.9.1 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 external analysis. 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.

17.3.9.2 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.

17.3.9.3 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 delivieries.
- Delivery of various operating spares and consumables.

17.3.9.4 Process Plant Water Services

The design includes allowance for various water recovery and treatment services which include the following:

- Potable water supply.
- Tailings return water.
- Domestic sewerage and sewerage water return facility.
- Process water filtration.
- Pollution control dam water return.

17.3.9.5 Process Plant Workshops and Stores

The design includes an allowance for the workshops and stores which include the following:

- Reagents storage.
- Lubricant and chemical storage.
- Gas bottle storage.
- Mechanical and electrical workshops.
- Tool store room.
- Rigging workshop and store.
- Salvage yard.
- Cable yard.

- Closed store and open storage yard.
- Grinding media storage.

17.3.9.6 Process Plant Buildings

The design includes an allowance for the plant ancillary buildings which include the following:

- Sample preparation room.
- Change house to service the process plant.
- Control room.
- Plant offices and ablution facilities.
- Canteen.
- Laundry room.
- A gate house.
- A taxi rank and bus shelter.
- Lubrication rooms for the crushing and milling areas.

17.3.10 Reagents

17.3.10.1 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.

17.3.10.2 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.

17.3.10.3 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.

17.3.10.4 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.

17.3.10.5 Reagents – Thiourea

Thiourea granules are delivered in 25 kg bags. The granules are dissolved and diluted to 15% 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.

17.3.10.6 Reagents – Oxalic acid

Oxalic acid granules are delivered in 25 kg bags. The granules are dissolved and diluted to 15% 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.

17.3.10.7 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 80 g/t.

17.3.11 Water Services

Raw water from the Flag Boshielo dam is stored in a bulk raw water dam from where it is distributed to the mining complex and the processing plant. The processing 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, the tailings thickener overflow product, excess concentrate thickener overflow product, as well as any plant run-off from the plant pollution control dam. 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.

A pollution control dam, equipped with a submersible pump, is provided for plant run-off collection. Water from this dam will be pumped to the process water circuit.

17.3.12 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 each.

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 Process Production Schedules

The concentrate production and the average concentrate metal grades along the years are shown on Figure 17.5 and Figure 17.6 for Phase 1.

Figure 17.7 and Figure 17.8 for Phase 2, and Figure 17.9 and Figure 17.10 for Phase 3.

The process production schedule for Phase 1 is summarized on Table 17.2 to Table 17.4, Phase 2 on Table 17.5 to Table 17.7 and Phase 3 on Table 17.8 to Table 17.10.

Figure 17.5 Phase 1 Concentrate Production with Ni and Cu Grades

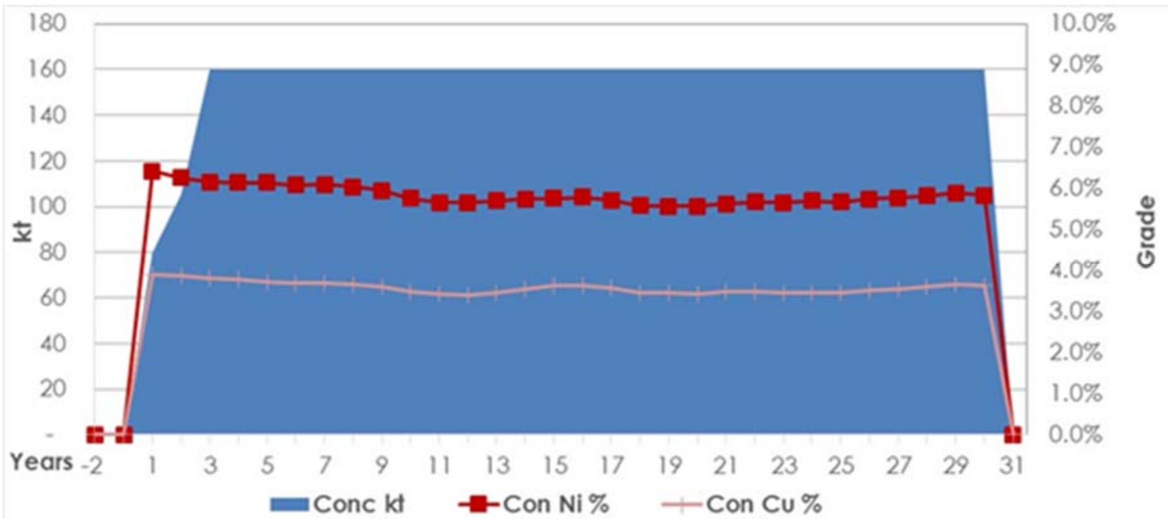


Figure 17.6 Phase 1 Concentrate Production with 3PE+Au Grades

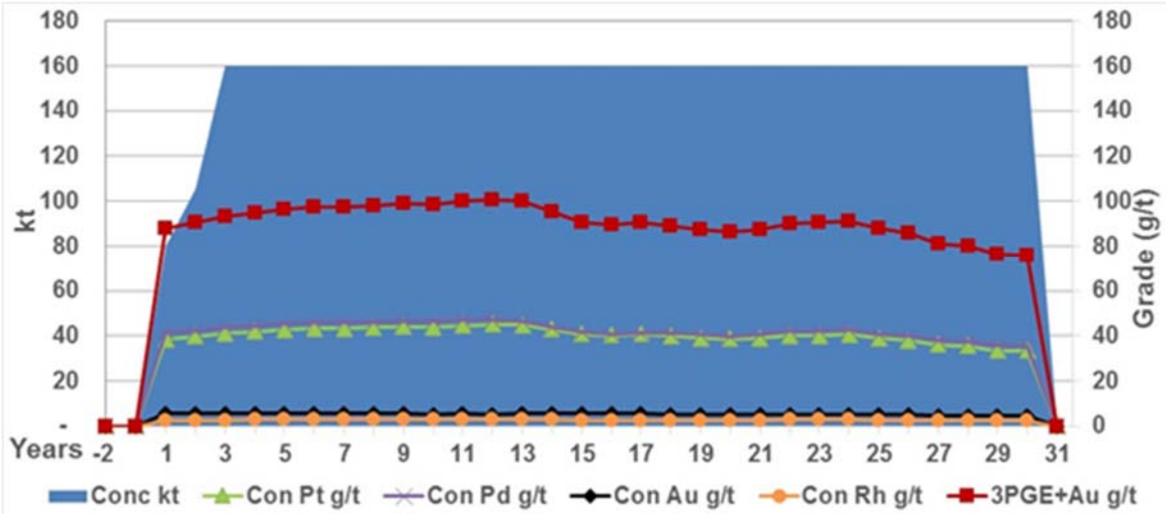


Table 17.2 Phase 1 Process Production Schedule (Years -2 to 11)

4 Mtpa Case Description	Unit	Total	Year												
			-2	-1	1	2	3	4	5	6	7	8	9	10	11
	Conc kt	4,665	–	–	80	105	160	160	160	160	160	160	160	160	160
	Con Ni %	0.06	–	–	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06
	Con Pt g/t	40.50	–	–	38.44	39.83	41.13	41.90	42.75	43.28	43.29	43.62	44.03	43.78	44.62
	Con Pd g/t	42.41	–	–	41.52	42.49	43.65	44.39	45.17	45.72	45.78	45.96	46.36	46.59	47.11
	Con Cu %	0.04	–	–	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.03	0.03
	Con Au g/t	5.26	–	–	5.44	5.45	5.47	5.46	5.51	5.51	5.50	5.52	5.46	5.32	5.43
	Con Rh g/t	2.76	–	–	2.58	2.64	2.71	2.76	2.82	2.85	2.85	2.87	2.93	2.96	3.02
Contained Metal	Ni (klb)	598,862	–	–	11,326	14,539	21,734	21,649	21,653	21,478	21,496	21,277	20,980	20,293	19,946
	Pt (koz)	6,075.14	–	–	98.87	135.02	211.58	215.55	219.92	222.61	222.68	224.37	226.50	225.20	229.51
	Pd (koz)	6,361.51	–	–	106.79	144.04	224.52	228.33	232.37	235.19	235.48	236.40	238.49	239.64	242.31
	Cu (klb)	367,653	–	–	6,882	8,971	13,409	13,285	13,164	13,031	13,050	12,920	12,697	12,314	12,038
	Au (koz)	788.53	–	–	13.99	18.46	28.12	28.09	28.37	28.34	28.29	28.38	28.10	27.38	27.93
	Rh (koz)	413.28	–	–	6.64	8.95	13.94	14.20	14.49	14.67	14.67	14.76	15.06	15.25	15.54

Table 17.3 Phase 1 Process Production Schedule (Years 12 to 24)

4 Mtpa Case Description	Unit	Year												
		12	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**
	Conc kt	160	160	160	160	160	160	160	160	160	160	160	160	160
	Con Ni %	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06
	Con Pt g/t	45.15	44.87	42.81	40.79	40.38	40.71	40.09	39.07	38.63	38.85	39.94	40.11	40.55
	Con Pd g/t	47.19	46.43	43.96	41.62	40.79	41.37	41.06	40.43	39.76	40.48	42.02	42.33	42.61
	Con Cu %	0.03	0.03	0.04	0.04	0.04	0.04	0.03	0.03	0.03	0.03	0.03	0.03	0.03
	Con Au g/t	5.38	5.46	5.49	5.45	5.58	5.46	5.28	5.12	5.02	5.06	5.13	5.13	5.16
	Con Rh g/t	3.07	3.04	2.88	2.72	2.68	2.73	2.70	2.68	2.68	2.71	2.82	2.86	2.88
Contained Metal	Ni (klb)	19,944	20,127	20,252	20,307	20,415	20,166	19,707	19,617	19,614	19,852	19,976	19,931	20,088
	Pt (koz)	232.27	230.82	220.22	209.84	207.73	209.42	206.25	200.96	198.70	199.84	205.46	206.32	208.59
	Pd (koz)	242.78	238.86	226.14	214.08	209.81	212.81	211.20	207.97	204.54	208.22	216.16	217.74	219.17
	Cu (klb)	12,008	12,135	12,501	12,768	12,784	12,582	12,220	12,160	12,087	12,317	12,327	12,220	12,225
	Au (koz)	27.69	28.10	28.23	28.02	28.68	28.09	27.16	26.33	25.85	26.02	26.38	26.38	26.53
	Rh (koz)	15.79	15.63	14.80	14.00	13.80	14.03	13.91	13.81	13.80	13.92	14.52	14.70	14.79

Table 17.4 Phase 1 Process Production Schedule (Years 25 to 30)

4 Mtpa Case Description	Unit	Year					
		25	26	27	28	29	30
	Conc kt	160	160	160	160	160	160
	Con Ni %	0.06	0.06	0.06	0.06	0.06	0.06
	Con Pt g/t	39.04	37.91	35.91	35.40	33.45	33.47
	Con Pd g/t	40.85	39.90	37.82	37.44	35.75	35.39
	Con Cu %	0.03	0.04	0.04	0.04	0.04	0.04
	Con Au g/t	5.07	5.01	4.84	4.79	4.70	4.67
	Con Rh g/t	2.73	2.66	2.51	2.48	2.36	2.34
Contained Metal	Ni (klb)	20,017	20,237	20,349	20,534	20,774	20,587
	Pt (koz)	200.85	195.01	184.71	182.12	172.05	172.15
	Pd (koz)	210.14	205.25	194.53	192.58	183.91	182.03
	Cu (klb)	12,209	12,416	12,537	12,697	12,922	12,776
	Au (koz)	26.10	25.78	24.90	24.66	24.15	24.03
	Rh (koz)	14.05	13.69	12.93	12.75	12.13	12.05

Figure 17.7 Phase 2 Concentrate Production with Ni and Cu Grades

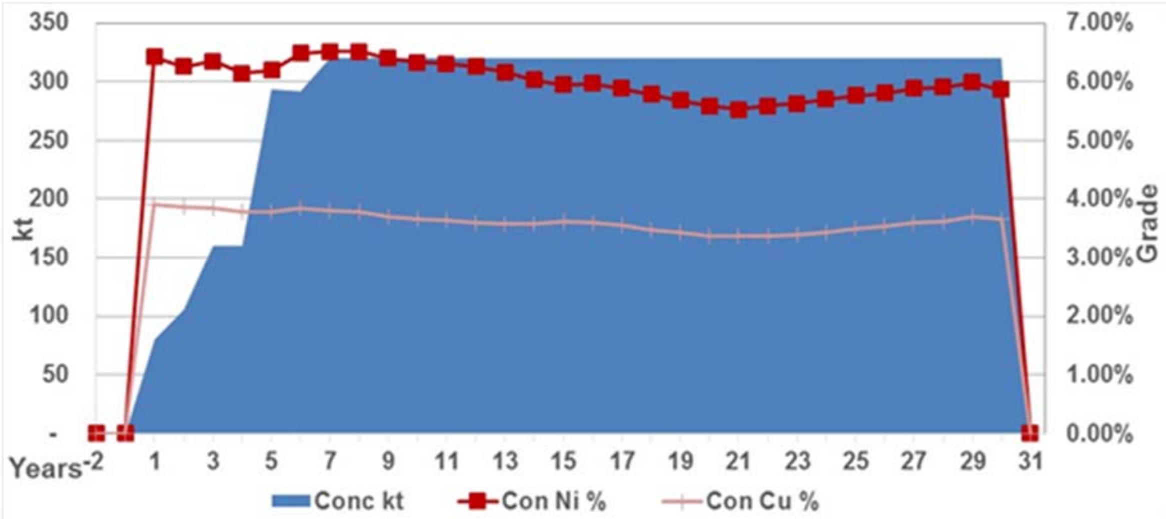


Figure 17.8 Phase 2 Concentrate Production with 3PE+Au Grades

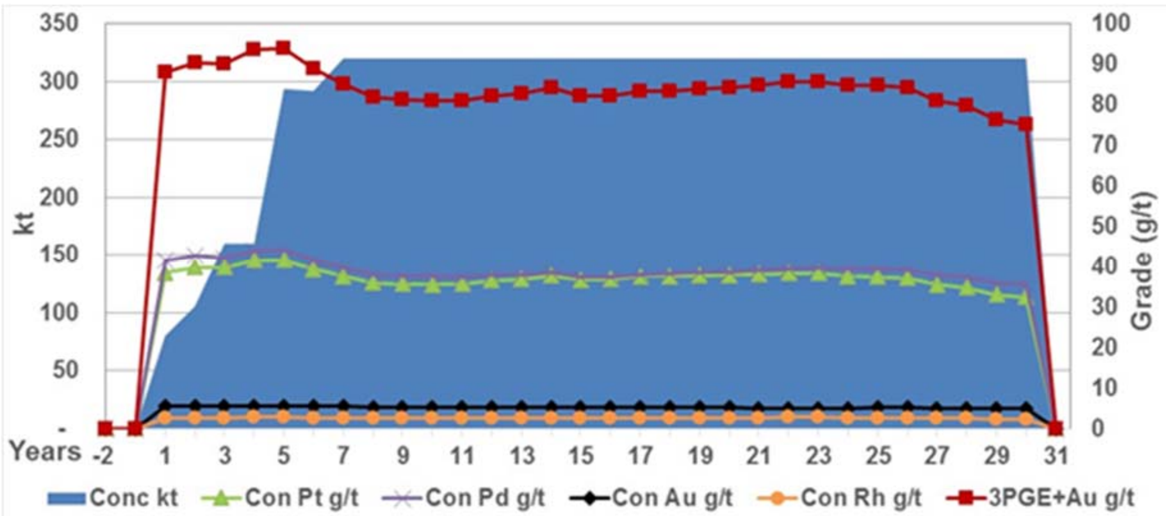


Table 17.5 Phase 2 Process Production Schedule (Years -2 to 11)

8 Mtpa Case Description	Unit	Total	Year											
			- 2	- 1	1	2	3	4	5	6	7	8	9	10	11
	Conc kt	8,771	–	–	80	105	160	160	294	292	320	320	320	320	320
	Con Ni %	0.06	–	–	0.06	0.06	0.06	0.06	0.06	0.06	0.07	0.07	0.06	0.06	0.06
	Con Pt g/t	37.04	–	–	38.44	39.83	39.80	41.47	41.59	39.34	37.55	36.00	35.66	35.45	35.66
	Con Pd g/t	38.71	–	–	41.52	42.49	42.18	43.94	44.03	41.49	39.55	37.97	37.75	37.53	37.40
	Con Cu %	0.04	–	–	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04
	Con Au g/t	5.19	–	–	5.44	5.45	5.50	5.47	5.50	5.52	5.42	5.33	5.26	5.24	5.23
	Con Rh g/t	2.63	–	–	2.58	2.64	2.66	2.74	2.75	2.67	2.61	2.57	2.58	2.60	2.62
Contained Metal	Ni (klb)	1,160,490	–	–	11,326	14,539	22,400	21,666	40,086	41,732	45,942	45,917	45,157	44,666	44,529
	Pt (koz)	10,444.18	–	–	98.87	135.02	204.73	213.35	392.55	369.16	386.31	370.38	366.92	364.73	366.89
	Pd (koz)	10,915.40	–	–	106.79	144.04	217.00	226.03	415.64	389.28	406.93	390.65	388.40	386.13	384.73
	Cu (klb)	695,411	–	–	6,882	8,971	13,590	13,332	24,426	24,696	26,891	26,663	26,155	25,766	25,637
	Au (koz)	1,462.36	–	–	13.99	18.46	28.27	28.14	51.90	51.76	55.80	54.81	54.12	53.89	53.82
	Rh (koz)	741.77	–	–	6.64	8.95	13.68	14.08	25.92	25.05	26.88	26.42	26.58	26.74	26.99

Table 17.6 Phase 2 Process Production Schedule (Years 12 to 24)

8 Mtpa Case Description	Unit	Year												
		12	13	14	15	16	17	18	19	20	21	22	23	24
	Conc kt	320	320	320	320	320	320	320	320	320	320	320	320	320
	Con Ni %	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06
	Con Pt g/t	36.47	36.84	37.69	36.72	36.81	37.48	37.49	37.70	37.87	37.98	38.23	38.25	37.60
	Con Pd g/t	37.79	37.92	38.63	37.47	37.40	38.02	38.09	38.37	38.56	39.02	39.52	39.67	39.37
	Con Cu %	0.04	0.04	0.04	0.04	0.04	0.04	0.03	0.03	0.03	0.03	0.03	0.03	0.03
	Con Au g/t	5.30	5.30	5.28	5.22	5.28	5.26	5.17	5.11	5.09	5.02	5.03	5.05	5.07
	Con Rh g/t	2.67	2.68	2.69	2.60	2.58	2.62	2.63	2.66	2.68	2.70	2.74	2.75	2.71
Contained Metal	Ni (klb)	44,102	43,502	42,532	41,945	42,078	41,513	40,765	40,037	39,415	38,981	39,404	39,675	40,180
	Pt (koz)	375.23	378.98	387.78	377.82	378.71	385.64	385.71	387.88	389.60	390.75	393.32	393.50	386.84
	Pd (koz)	388.78	390.09	397.43	385.53	384.82	391.21	391.84	394.72	396.75	401.42	406.64	408.13	405.07
	Cu (klb)	25,342	25,294	25,277	25,449	25,434	25,051	24,553	24,155	23,813	23,729	23,785	23,862	24,246
	Au (koz)	54.49	54.55	54.30	53.71	54.31	54.10	53.23	52.60	52.36	51.66	51.77	51.96	52.11
	Rh (koz)	27.50	27.54	27.63	26.73	26.53	26.96	27.07	27.33	27.58	27.75	28.21	28.32	27.90

Table 17.7 Phase 2 Process Production Schedule (Years 25 to 30)

8 Mtpa Case Description	Unit	Year					
		25	26	27	28	29	30
	Conc kt	320	320	320	320	320	320
	Con Ni %	0.06	0.06	0.06	0.06	0.06	0.06
	Con Pt g/t	37.47	37.10	35.51	34.82	33.00	32.38
	Con Pd g/t	39.38	39.17	37.80	37.42	36.00	35.48
	Con Cu %	0.03	0.04	0.04	0.04	0.04	0.04
	Con Au g/t	5.10	5.09	5.00	4.95	4.84	4.79
	Con Rh g/t	2.69	2.66	2.55	2.52	2.42	2.39
Contained Metal	Ni (klb)	40,645	40,959	41,488	41,674	42,281	41,352
	Pt (koz)	385.52	381.74	365.35	358.24	339.47	333.18
	Pd (koz)	405.12	402.95	388.94	384.99	370.37	365.00
	Cu (klb)	24,656	24,888	25,320	25,583	26,110	25,852
	Au (koz)	52.45	52.39	51.45	50.88	49.81	49.27
	Rh (koz)	27.71	27.39	26.27	25.93	24.87	24.63

Figure 17.9 Phase 3 Concentrate Production with Ni and Cu Grades

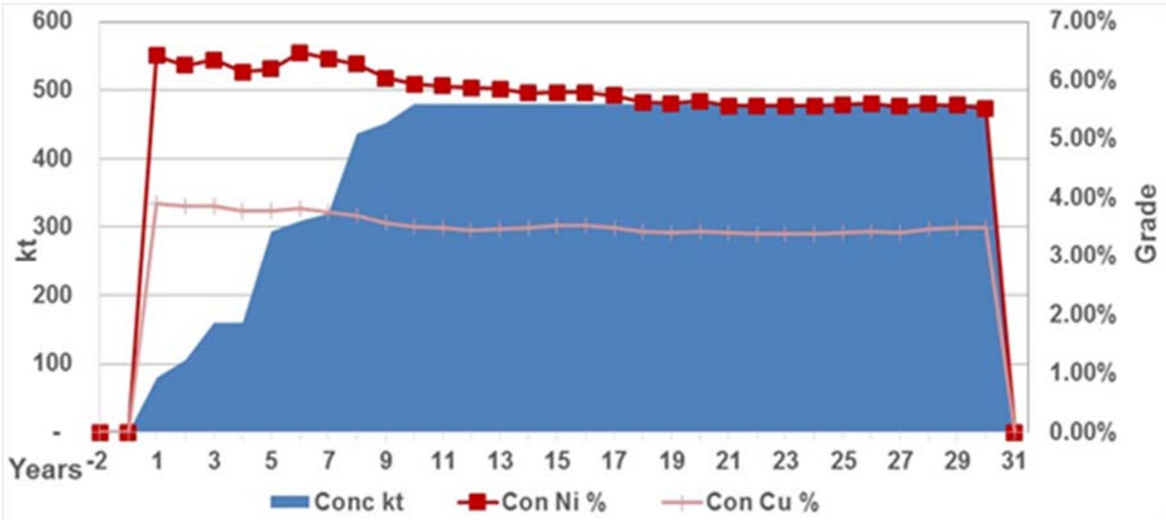


Figure 17.10 Phase 3 Concentrate Production with 3PE+Au Grades

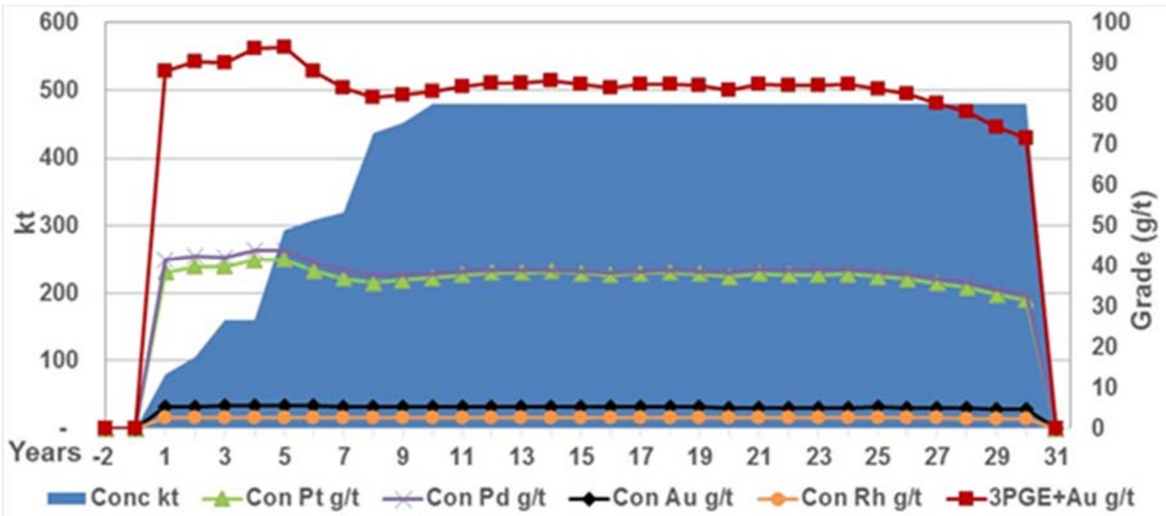


Table 17.8 Phase 3 Process Production Schedule (Years -2 to 11)

12 Mtpa Case Description	Unit	Total	Year												
			-2	-1	1	2	3	4	5	6	7	8	9	10	11
	Conc kt	12,396	–	–	80	105	160	160	294	309	320	437	452	480	480
	Con Ni %	0.06	–	–	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06
	Con Pt g/t	37.30	–	–	38.44	39.83	39.80	41.47	41.59	38.86	36.96	35.91	36.48	37.04	37.83
	Con Pd g/t	38.37	–	–	41.52	42.49	42.18	43.94	44.03	40.96	39.00	37.77	37.78	38.07	38.48
	Con Cu %	0.04	–	–	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.03
	Con Au g/t	5.18	–	–	5.44	5.45	5.50	5.47	5.50	5.48	5.33	5.24	5.23	5.23	5.26
	Con Rh g/t	2.64	–	–	2.58	2.64	2.66	2.74	2.75	2.65	2.60	2.58	2.63	2.67	2.73
Contained Metal	Ni (klb)	1,581,547	–	–	11,326	14,539	22,400	21,666	40,086	44,058	44,948	60,424	60,120	62,800	62,490
	Pt (koz)	14,864.03	–	–	98.87	135.02	204.73	213.35	392.55	385.84	380.27	504.11	529.90	571.67	583.75
	Pd (koz)	15,294.12	–	–	106.79	144.04	217.00	226.03	415.64	406.65	401.21	530.23	548.80	587.49	593.91
	Cu (klb)	957,299	–	–	6,882	8,971	13,590	13,332	24,426	26,046	26,455	35,497	35,474	37,085	36,794
	Au (koz)	2,064.71	–	–	13.99	18.46	28.27	28.14	51.90	54.44	54.88	73.60	75.97	80.67	81.17
	Rh (koz)	1,050.37	–	–	6.64	8.95	13.68	14.08	25.92	26.35	26.76	36.23	38.18	41.25	42.14

Table 17.9 Phase 3 Process Production Schedule (Years 12 to 24)

12 Mtpa Case Description	Unit	Year												
		12	13	14	15	16	17	18	19	20	21	22	23	24
	Conc kt	480	480	480	480	480	480	480	480	480	480	480	480	480
	Con Ni %	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06
	Con Pt g/t	38.37	38.41	38.60	38.22	37.85	38.21	38.31	38.08	37.40	38.03	37.73	37.78	37.89
	Con Pd g/t	38.79	38.74	38.96	38.68	38.21	38.61	38.74	38.54	38.28	39.09	38.97	39.05	39.24
	Con Cu %	0.03	0.03	0.03	0.04	0.04	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03
	Con Au g/t	5.30	5.30	5.29	5.27	5.29	5.27	5.21	5.18	5.14	5.11	5.07	5.07	5.11
	Con Rh g/t	2.76	2.75	2.74	2.69	2.66	2.70	2.71	2.71	2.67	2.69	2.68	2.69	2.70
Contained Metal	Ni (klb)	62,152	61,891	61,224	61,334	61,289	60,777	59,513	59,330	59,797	58,867	58,912	58,866	58,876
	Pt (koz)	592.09	592.73	595.70	589.83	584.08	589.64	591.18	587.66	577.10	586.87	582.29	582.98	584.67
	Pd (koz)	598.56	597.79	601.27	596.95	589.66	595.89	597.88	594.78	590.74	603.21	601.36	602.68	605.64
	Cu (klb)	36,546	36,763	36,940	37,325	37,296	36,901	36,207	36,014	36,317	36,044	35,865	35,788	35,856
	Au (koz)	81.77	81.84	81.57	81.28	81.69	81.28	80.33	79.89	79.39	78.88	78.31	78.17	78.85
	Rh (koz)	42.56	42.40	42.22	41.50	41.06	41.63	41.77	41.77	41.18	41.54	41.35	41.55	41.61

Table 17.10 Phase 3 Process Production Schedule (Years 25 to 30)

12 Mtpa Case Description	Unit	Year					
		25	26	27	28	29	30
	Conc kt	480	480	480	480	480	480
	Con Ni %	0.06	0.06	0.06	0.06	0.06	0.06
	Con Pt g/t	37.30	36.74	35.75	34.74	32.96	31.62
	Con Pd g/t	38.54	38.00	36.83	35.92	34.21	32.92
	Con Cu %	0.03	0.03	0.03	0.03	0.03	0.03
	Con Au g/t	5.17	5.14	5.03	4.92	4.82	4.72
	Con Rh g/t	2.64	2.59	2.51	2.45	2.33	2.25
Contained Metal	Ni (klb)	59,138	59,355	58,800	59,167	58,983	58,418
	Pt (koz)	575.68	567.02	551.74	536.09	508.62	488.01
	Pd (koz)	594.80	586.47	568.30	554.37	527.98	508.00
	Cu (klb)	36,087	36,284	36,089	36,580	36,936	36,908
	Au (koz)	79.74	79.38	77.58	75.97	74.37	72.92
	Rh (koz)	40.72	39.98	38.77	37.76	36.02	34.80

17.5 Comments on Section 17

The qualified person responsible for this section, Mr Michael Valenta, is of the opinion that the proposed plant design has captured the findings of the test work 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 project. It is most likely that the size of the flotation circuit may reduce in size once further variability test work 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 test work. The impact of the footwall and added dilution may have a material impact on the final circuit configuration.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

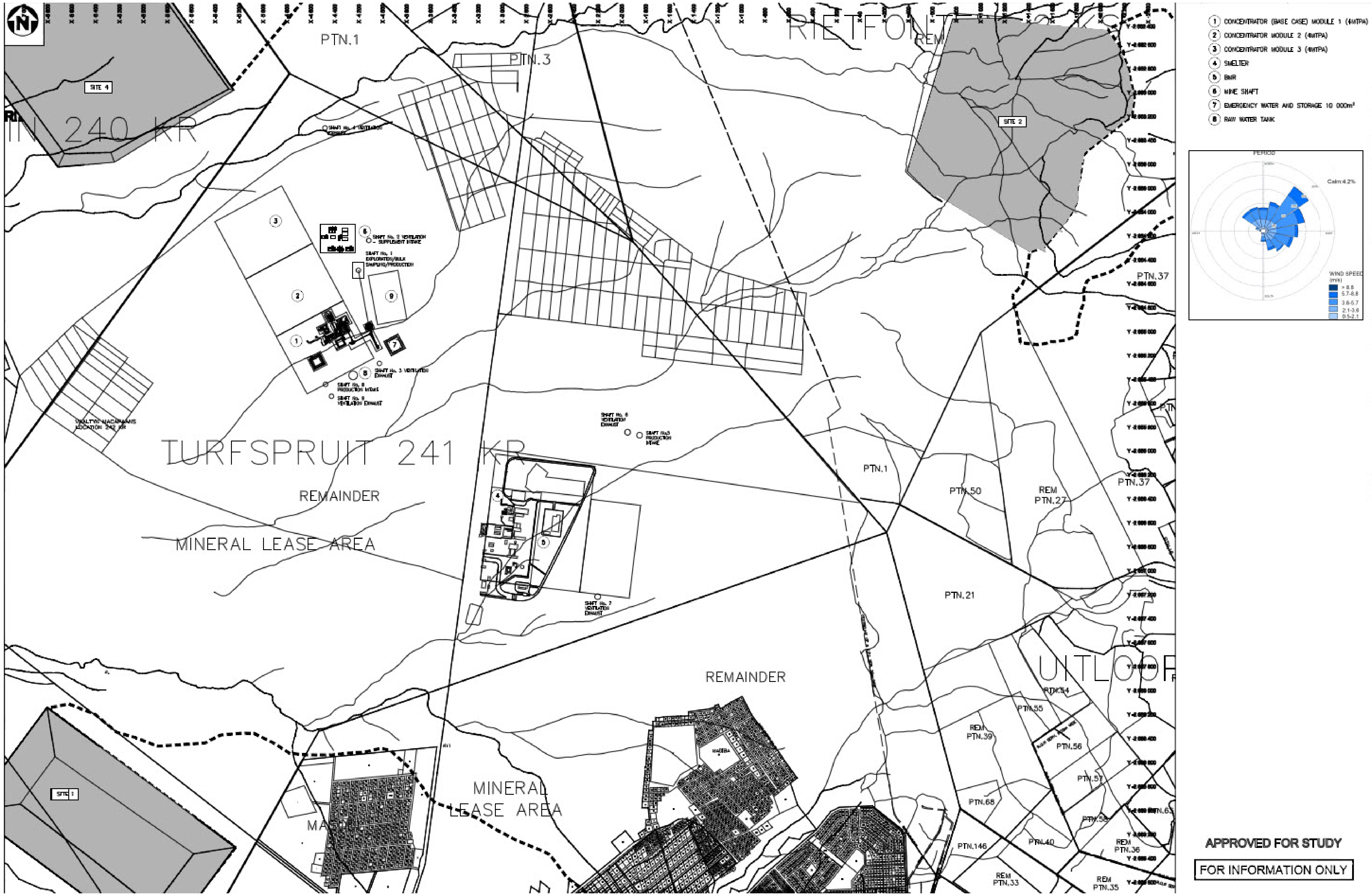
The Project site is located approximately 280 km north-east of Johannesburg. Year-round access to the site is by paved, all-weather national highway (N1) to Mokopane (formerly Potgietersrus). From Mokopane the access continues as a paved, all-weather national highway (N11). The N11 cuts through the deposit in the northern portion of the Turfspruit farm area where Turfspruit abuts Tweefointein. This road is a two-lane tarmac road suitable for heavy loads year round.

Access to drill sites and other areas within the Project is by gravel all-weather roads or by unpaved tracks. The site is currently served by an improved gravel road that is oriented east/west and intersects the site at approximately the southern end of the Turfspruit area. This road tees off the N11 and is suitable for heavy loads when conditions are dry. After sustained rains the road must be graded and allowed to thoroughly dry before the surface is suitable for sustained heavy traffic.

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 Project is in Mokopane.

Figure 18.1 Site Map



18.2 Local Resources and Infrastructure

Electrical energy, telephone service, accommodation, potable water and other infrastructure components are available in Mokopane. The Mokopane town centre is approximately 11 km from the centre of the Project. The main line of the national railroad system passes approximately 6 km east of the Project.

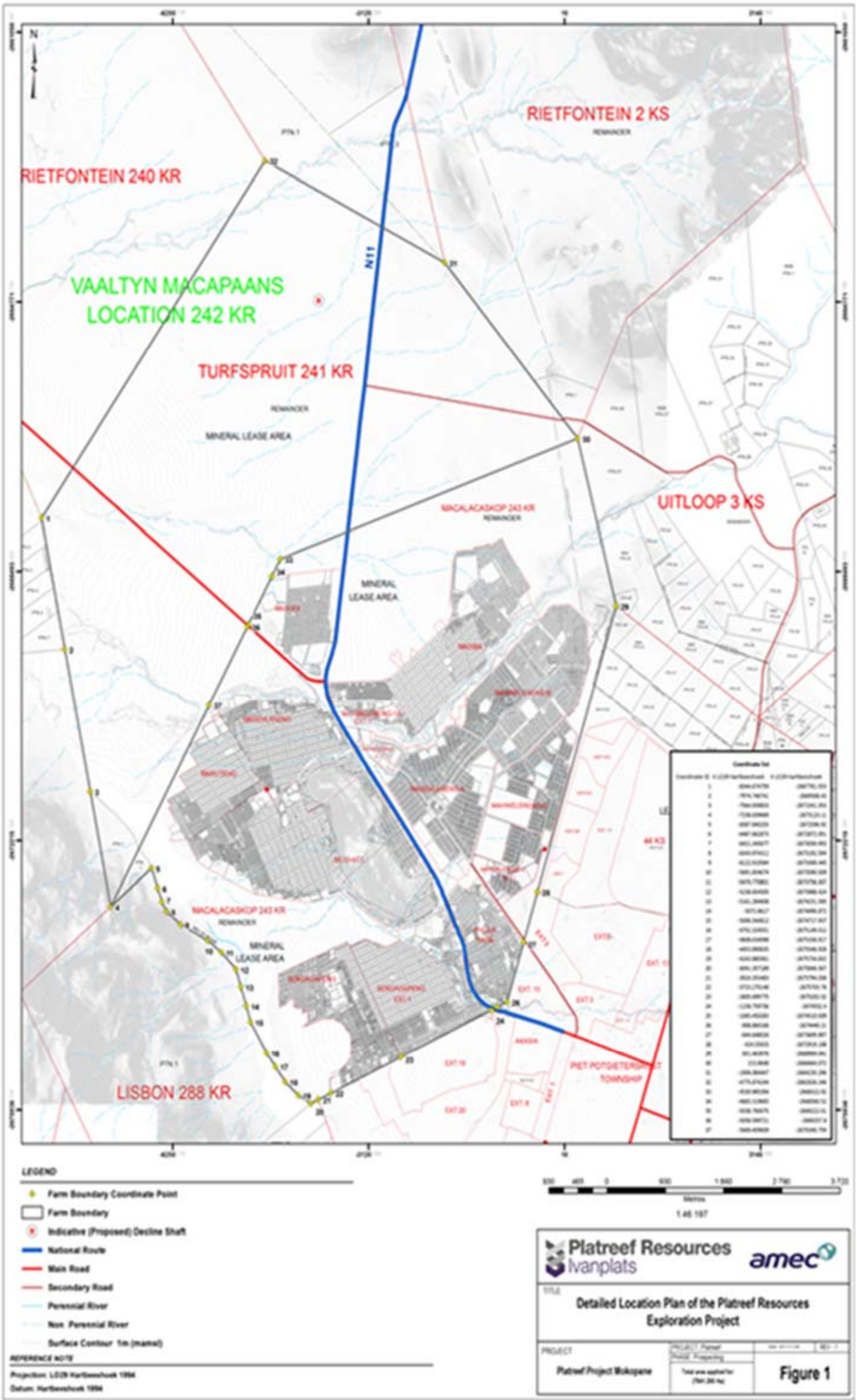
18.3 Local Labour Resources

Mining activity is moderate within a 100 km radius. A large, potential labour force lives within close proximity of the Project. The local communities can be trained in order to develop the necessary skills to support the Project. Skilled trade positions and professional staff will have to be recruited from outside the area.

Adequate town-site facilities and infrastructure exist to support an influx of personnel. Housing may have to be constructed or subsidised for some senior positions. An accommodation 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 of just more than 1,400 employees. It should be noted that this will only be feasible if all accommodation options within the current database can be secured. This will likely have immense cost implications.

The majority of available accommodation is within the hospitality sector, followed by properties which can be purchased. It should be noted that to secure the quantity of accommodation within the hospitality sector on the scale that mine intends to, will require intensive logistical arrangements (in terms of pre-booking, etc.). This process will be complicated by the uncertainty regarding the periods for which accommodation will be require. The viability of using existing accommodation infrastructure will depend on a) whether or not employees will relocate to Mokopane with their families and b) whether these properties will be used only for the construction period or for the duration of the project.

Figure 18.2 Locality Map Showing Major Townships and Roads



18.4 Water and Wastewater Systems

18.4.1 Bulk Water

South Africa is a country of relatively low rainfall and, in particular, the Limpopo province (typical rainfall ~600 mm per annum) will require significant additional capacity to meet the growing demand 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 supply is achieved. According to the Olifants Water Resource Development Reconcuiliation Strategy prospects are good to obtain sufficient water for the 3 phases.

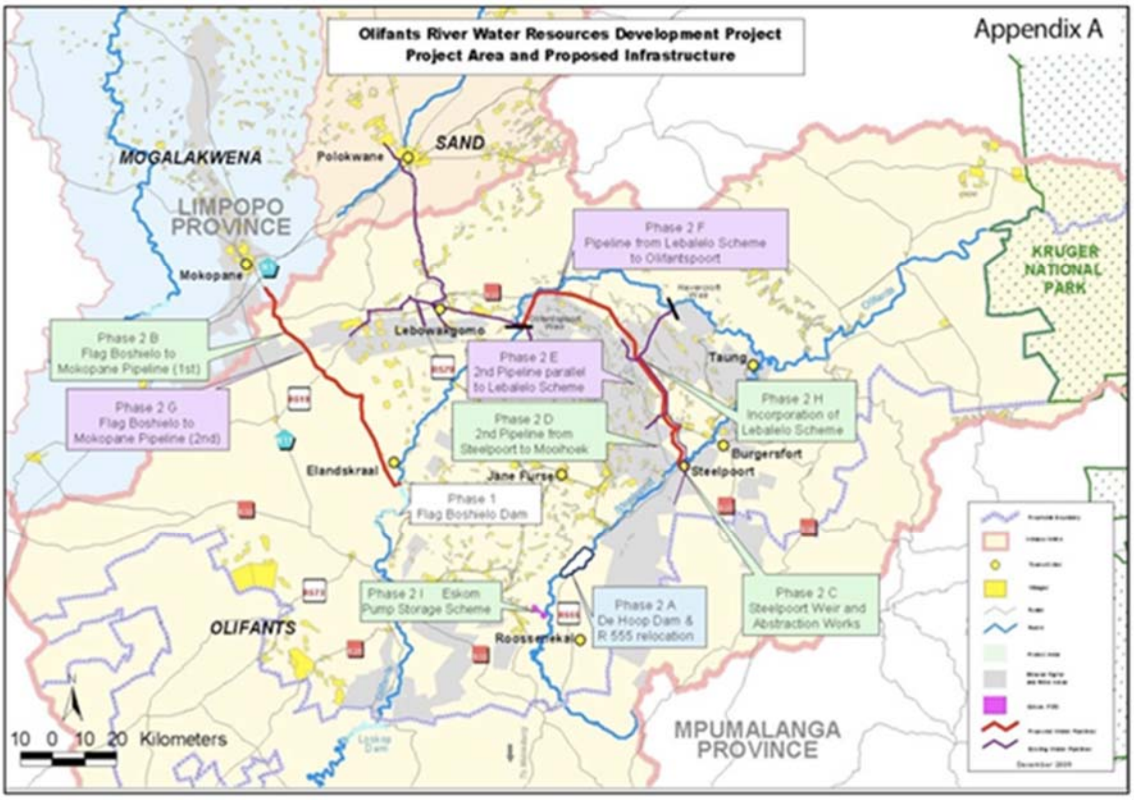
The Olifants River Water Resource Development Project (ORWRDP) is designed to deliver water to the Eastern and Northern Limbs of the Bushveld Igneous Complex within the Limpopo province of South Africa. The project consists of the construction of the new De Hoop dam which is nearing completion, the wall of the Flag Boshielo dam which has been raised by 5 m and related pipeline infrastructure which will ultimately deliver water to Pruissen, located to the south-east of Mokopane and 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.

Under the ORWRDP, a pipeline is to be constructed between Flag Boshielo Dam on the Oliphants River and Pruissen, located to the south of Mokopane. The Pruissen pipeline project will be developed to deliver water from Pruissen to the communities and mining projects on the Northern Limb, including the Platreef Project.

The Pruissen pipeline scheme forms part of the larger ORWRDP regional expansion of water distribution, which also includes the construction of the De Hoop dam on the Steelpoort River and related pipeline infrastructure. Construction of the De Hoop dam began in March 2007. Construction has been intermittent due to funding issues and engineering redesigns.

An outline of the ORWRDP is shown in Figure 18.3.

Figure 18.3 Olifants River Water Resources Development Project



Platreef Resources is a member of the Joint Water Forum (JWF) (part of the ORWRDP) and the Pruissen Water Forum. 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. The Platreef Project's water requirement for the 8 Mtpa base case scenario would be approximately 22 million litres per day.

The available water in the region is fully allocated despite the construction of the De Hoop Dam and the raising of Flag Boshielo Dam wall. The yields of both these dams have been reduced due to revision of the hydrology. In addition to water demand from the mines, there is also the need to meet environmental water requirements in the catchment which are currently not being met in full.

As a consequence, the mines in the area have commissioned a study to determine the potential of excess mine water in the Highveld Coal Fields in the upper Olifants catchment as a resource to supplement the demands of mining in the middle Olifants catchment. The preliminary results of this study have been published.

The Platreef Project is committed to working with the JWF to develop the ORWRDP as the primary source of bulk water to service the needs of the Project.

18.4.2 Potential Alternative Sources of Water

A study has been undertaken to provide an alternative source for bulk water supply to the Phase 2B scheme (Figure 18.3) proposed by the Joint Water Forum. This technical feasibility study considers the routing and construction of bulk water supply pipelines from three alternative sources at a conceptual level. An order of magnitude estimate of costs has been prepared to support this study.

The alternative bulk water supply sources identified for this study are:

- A pipeline from the existing Flag Boshielo Dam to site.
- A pipeline from existing Rooiwal Waste Water Treatment Works to site.
- A pipeline from existing Seekoeigat Waste Water Treatment Works.

In addition to the above, further investigations are currently underway with the relevant municipalities to determine the actual quantity of water available and a more detailed investigation into these sources of water will be conducted during the pre-feasibility study.

This study has provided a conceptual level assessment of the engineering feasibility, routing and cost of construction and operation and maintenance for a bulk water supply of 10Mm^3/a to Platreef mine from three potential sources noted above:

- Flag Boshielo Dam - length of pipeline 120,770 m.
- Rooiwal Waste Water Treatment Works – length of pipeline 209,509 m.
- Zeekoiegat Waste Water Treatment Works – length of pipeline 217,673 m.

The preliminary cost for construction, operation and maintenance of the pipelines has been prepared resulting in the following cost per cubic metre of water, assuming a 20 year endowment period and including a 10% contingency:

- Flag Boshielo Dam - R7.94/m^3.
- Rooiwal Waste Water Treatment Works – R13.19//m^3.
- Zeekoiegat Waste Water Treatment Works – R13.52/m^3.

Although the Flag Boshielo dam appears to be the most viable option, other options, such as waste water from Mokopane waste water treatment works is being further investigated.

Platreef is also investigating, together with the Mokopane Municipality, the potential supply of waste water from the Mokopane Waste Water Treatment Works.

18.4.3 Water Balance

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 will be developed during the pre-feasibility study. The objective of the water balance is to calculate the volumes of make-up water that will be required for the mine and process plant facilities under equilibrium conditions and to ensure effective water use by the proposed project in all sections of the process.

The main sources of water are anticipated as follows:

- Bulk water supply – 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.

The main water losses from the mine water circuits include:

- Evaporation from dams and tailings storage facilities;
- Entrainment and retention in tailings dam;
- Seepage to groundwater; and
- Discharge to surface streams and water courses.

During the pre-feasibility study, tailings disposal methods such as paste and filter cake disposal will be investigated to determine the most environmentally responsible and economical method of tailings disposal.

18.4.4 Potable Water

It is expected that potable water for ablution facilities and kitchens will be obtained from the bulk water supply system or from treated groundwater resources currently being investigated.

18.4.5 Potable Water Reticulation

Potable water will be distributed via pipe racks and sleeper ways along with other services where possible and routed underground as necessary.

18.4.6 Wastewater

Sewage will be drained via underground sewers to a sewage treatment plant and treated to produce an effluent of a suitably safe standard for process use. Floor washings that are organically contaminated, from kitchens and ablution blocks, will also drain via the sewers to the treatment plant.

Sludge from the sewage treatment plant will be pumped out for disposal with tailings or dried on engineered drying beds and buried.

Floor washings that are potentially contaminated with mineral oils (workshops, refuelling and lube and diesel storage areas) will be passed through oil skimmers and drained via oily-water sewers to the run-off dam.

18.5 Highway Re-Alignment

The N11 national highway connects Mokopane with the South Africa/Botswana border. The current road runs directly through the Turfspruit and Macalacaskop 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 Mokopane.

The major mining operations in the area, mainly platinum, are located some 25 km north-west of Mokopane. The current transportation means to and from the mines is by road which has resulted in concerns regarding the deteriorating level of service of the existing N11.

A study was completed in 2009 in respect of the proposed re-routing of approximately 18.4 km and the upgrading of approximately 3.6 km of National Route 11 Section 13 (N11-13), as well as the widening of approximately 2.7 km of the R101, between Armoede to the north of Mokopane and Planknek to the east of same in the Limpopo Province.

The realignment route will bypass the Turfspruit and Macalacaskop farms but will bisect the Rietfontein farm and has therefore been considered in the Tailings Storage Facility footprint.

18.6 Bulk Power Supply

Eskom have 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 with the first unit of the new Medupi power station due to come on line end 2014 and then at nine month intervals for the remaining five units.

The Medupi Power Station is a new dry-cooled coal fired power station being built by Eskom near Lephalale (approximately 180 km east-north-east of Mokopane) in Limpopo province, South Africa. When completed, the power station will have six boilers (“Six-Pack” design) each powering a 800 MW turbine, producing 4,800 MW of power into the national grid. This will be the largest dry-cooled coal fired power station in the world. Contracts have been placed with Hitachi to supply the boilers and Alstom to provide the steam turbines for this plant.

Medupi will be supplied by coal from Exxaro’s Grootegeluk coal mine, located to the north of the site. Eskom has placed a contract with Exxaro to supply 14.6 Mt coal per year for 40 years.

A new Main Transmission Substation (MTS), called the Borutho MTS (400 kV/132 kV/22 kV) is sized at 2x500 MVA (extendable) and will come on line in 2014. The Borutho substation is approximately 26 km from the Platreef Project site (shown in Figure 18.4).

Should future demand exceed the capacity of the new Borutho MTS, an additional transformer could be installed at the MTS to meet the new demand and to increase capacity to 1000 MVA.

There are essentially two transmission schemes to supply power to the Platreef Project, namely standard or premium supply schemes.

The standard supply scheme essentially consists of a radial line to the project site. The disadvantage of this is that should the line be damaged, there will be no alternative power transmission system to the plant site.

The premium supply option will include a loop in and loop out from the project site totalling 8 km. This will allow redundancy in power transmission lines should one of the lines go down. The Platreef Project has requested that Eskom complete a desk top study that considers the premium supply option.

Figure 18.4 Proposed Transmission Solution

18.6.1 Permanent Power

A power application form has been submitted to Eskom for the supply of bulk power for the start of Platreef Project operations by Q3 2016. The submission of a power application form is a precursor to an Eskom power desktop feasibility study. The power application form notes the Platreef Base Case Scenario being a 3 Mtpa underground mine and a Platinum Group Element (PGE) concentrator and will include all supporting infrastructure. The Project maximum demand under this scenario is estimated at 70 MVA.

The Base Case power that will be required in three phases:

- Construction Power ~ 5 MVA (Q1 2014).
- Mining and Plant Ramp Up – 56 to 292 MVA (Q2 2014 to Q4 2015).
- Full Scale Operation 292 MVA (Q4 2015 onwards).

There are three stages of project development that need to be followed in order to secure power for the Platreef Project:

- Desktop Study Phase which will prepare a study and cost to 65% accuracy and has been completed.
- Budget Quotation Phase. To initiate the budget quotation phase, Platreef paid a commitment fee of R19 million for the following (i) EIA, (ii) engineering and design and (iii) procurement of long lead items if required. The budget quotation will be accurate to within 85% and will be completed six months after receipt of the commitment fee. Eskom have advised the Project that the EIA could take up to 18 months to complete.
- The final stage is the Construction Phase and is estimated to take 12 months to complete.

There are two types of agreements/standards in place between Eskom and various consumers. These agreements are required in an effort to reduce power in a controlled manner during periods of emergency load reduction and system restoration. The following standards are currently being implemented:

- NRS 048-9 – This standard defines the principles during load shedding and load curtailment. Stage 1 and 2 requires a 10% curtailment, and stage 3 requires 20%.
- Once the power crisis has normalised, all new contracts will require that all consumers will be required to reduce load by 10% in times of critical system load.

18.6.2 Construction Power

Eskom have been requested to investigate the availability of nearby power lines and substations to provide construction power for the Project. A number of options are available, Eskom are currently investigating these options through a desktop feasibility study and have given initial indication that the following substations may be available to provide construction power to the Project:

- 132/22 kV Sandsloot substation, or
- 132/33 kV Potgietersrus substation, or

- 33/11 kV Mahwelereng substation located near the Project site and is feeding from 132/33 kV Potgietersrus substation.
- A commitment fee of R 641,000 has been paid to Eskom to provide a budget quotation.
- Any power requirements prior to the supply of temporary construction power will be supplied by diesel generated sets.

18.7 Access Roads

The plant site is located approximately 11 km North-North-East of Mokopane, on the farm Turfspruit 241KR. The site is accessed from the N11, a single-carriageway public highway with a bitumen surface. The N11 is at present in good condition.

It will be necessary to secure permission from the road authorities to construct an intersection on the N11 for access to the plant site.

18.8 Fuel Unloading and Storage Facility

Light vehicles petrol stations will obtain fuel from local petrol stations.

For the mine, a fuel depot to dispense petrol and diesel has been planned. It is normal practice in South Africa for bulk fuel suppliers such as Engen or BP to provide the necessary bowsers and pumps, etc. free of charge in return for a fuel supply contract.

18.9 Fire Protection and Detection

The water supply for fire-fighting shall 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 shall 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. Connections to the firewater system for any other service will not be permitted.

Fire hydrants will be located when the plot plan has been finalized. Hydrants shall be “industrial standard” (not “commercial 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.

Hose reels should 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 m^2 of plant area.

Deluge systems shall be considered for:

- Pumps handling flammable liquids.
- Lube oil consoles for critical equipment.
- Transformer Bays.
- Flammable Liquid Storage areas.

- All other hazardous materials areas.

Placement of hand extinguishers shall meet prescribed specifications. Hand extinguishers to be placed in fire hazardous process areas, non-fire hazardous process areas, electrical rooms, non-process buildings, control rooms, and all company vehicles.

Fire detection equipment shall include a fire indicator panel (FIP) located in the main control room area and local intelligent units or sub fire indicator panels (SFIP) as required located around the site. The total control and indicating equipment shall be arranged as a distributed system preferably interconnected by a single communications and power loop on a loop in loop out basis at each device to minimise site wiring requirements.

The fire detection system will be independent of the PCS and shall be specified as part of the overall fire protection system for the plant, which shall include the fire water system and inert gas suppression systems.

18.10 Run-off and Diversions

The Stormwater management measures will include the following features:

- Clean water cut-off drains to intercept storm runoff uphill of the plant site and divert it around the site in order to minimise the volume of potentially contaminated water.
- Plant run-off drains to collect rainwater run-off and drain it into a run-off dam,
- Run-off dam to contain run-off on a short-term basis for use elsewhere and;
- Silt traps to collect water from the plant run-off drains and remove grit before it is discharged into the run-off dam.

Water will be pumped out from the dam for use in the process plant or in dust suppression, bearing in mind that this source is a minor component in the overall water balance.

Unless stated otherwise in the environmental management plan (EMP), the dam will be sized for a 1:50 year 24-hour rainfall event, as required by law. The drains and the hydraulic structures at the dam and silt trap will be designed, similarly, for the 1:50 year peak flow rate.

18.11 Onsite Hospital and Medical Facilities

A clinic and first-aid facility will be housed together at a suitable position near the main access gate.

18.12 Administration and Changehouse Facilities

Administration facilities will be allowed for the Plant and Mine. The building will house office space for Executive Management, Human Resources, Accounting and Finance, Health Safety and Environmental, and General Administration. An office complement of approximately 50 people is estimated, with a floor space of 1,217 m^2. This allows approximately 18 m^2 per person and accounts for additional services such as a boardroom, rest room facilities, etc.

Two change houses of approximately 800 m^2 in total are also planned. The one facility will serve concentrator personnel and visitors; this will be approximately 500 m^2. The second will service the mining personnel and will be approximately 300 m^2.

18.13 Stores and Workshops

A centralized stores facility is planned to service the processing facilities and the mine. The size will be 1,600 m^2 and will contain offices and an overhead crane. The stores facility will be located near the Administration building at the processing plant.

One workshop has been planned for the processing facility. The workshop will be 450 m^2 and will include an overhead crane.

The mining workshop has been allowed separately under the mining budget.

A light vehicle maintenance shop has not been planned, as there is sufficient capacity in Mokopane to outsource the maintenance of light vehicles. Vehicles included in the Light Vehicle category include automobiles and LDV.

18.14 Logistics

The assumption for the scoping study is that all Platreef concentrate will be sold to local smelters:

- Anglo American Platinum Limited operates three smelting complexes, namely Mortimer (Union Section), Waterval (Rustenburg) and Polokwane.
- Impala Operates one smelter in Rustenburg area.
- Lonmin Operates one smelter in Rustenburg area.

The Platreef concentrate will be transported to one of the above smelters via road transport using covered side tipper trucks. This is standard mode of transport for moving concentrate between concentrator and smelter in the area.

The concentrate is envisaged as being transported via covered side tipper trucks with a net payload of 32 t to 34 t and this service will be provided on a contract basis. Loading at the mine is at the processing plant via overhead bins/ hoppers and off-loading at the smelter is by means of side-tipping into a grid from where conveyers transport the concentrate to the smelter.

Considering the 3 Mtpa base case scenario and assuming a ~6% mass pull through the concentrator, it is expected that 180,000 t/a of concentrate will be produced. Based on this, standard 32 t to 34 t side tipper trucks would be ideal for this size of operation. This equates to 440 to 468 truck loads per month or ~15 truck loads per day.

There are various side tipper configurations and the type used will be dependant of concentrate physical properties.

18.15 Comments

The findings of the infrastructure study are a true reflection of the current situation existing in the country. The status of the various projects, particularly those concerned with power and water, must be regularly updated. Many projects have been caught unawares by construction delays on some of the major infrastructure projects.

Ongoing liaison with government entities must be maintained and the necessary discussions must be progressed regarding the changing of roads and the use of infrastructure.

18.16 Tailings Storage Facility

18.16.1 Project Requirements

The tailings storage facility (TSF) requirements of the scoping phase of the study have been as follow:

- Design a tailings storage facility that can accommodate platinum tailings.
- The tailings deposition rate will be approximately 3.8 dry Mtpa (Phase 1) increasing to 11.4 dry Mtpa in Phase 3.
- The design life for the tailings storage facility is approximately 30 years.
- Investigate the potential for increasing the deposition rate to approximately 11.4 Mtpa over the life of the project.

18.16.2 Design Objectives

- Create a safe and stable tailings storage facility 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, mineralization 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 facility.
- Cost effective construction, operation and closure.
- The tailings storage facility be situated so that it will not sterilise or be in conflict with any mining activity.

18.16.3 Facility Description

The Rietfontein 2 KS site is the preferred site for the 3,8 Mtpa Phase 1 scenario. Together, the Rietfontein 2 KS and Bultongfontein sites can accommodate the 7.6 Mtpa Phase 2 and 11,4 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 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 an upstream construction method.

The Bultongfontein site is proposed to be developed in time for the Phase 2 production scenario. A preliminary footprint for a tailings storage facility of sufficient capacity for Phase 2 and Phase 3 has been identified. However, Eksom has advised that there is an easement for high voltage power lines within the footprint. It is understood that it may be possible to move the easement. Eksom has not been yet been approached to negotiate a realignment of the easement.

Tailings samples were not made available for laboratory testing during the scoping study phase. It is recommended that the geotechnical properties of a representative composite tailings sample should be concluded during the feasibility study phases.

The following site investigations should be concluded as part of the feasibility study phases:

- Geotechnical investigation.
- Geohydrological investigation and modeling, including geochemistry.

19 MARKET STUDIES AND CONTRACTS

19.1 Summary

Ivanhoe plans a phased expansion of the operation with Phase 1 run of mine (ROM) production of 4 Mtpa (approximately 150 ktpa concentrate and 9 ktpa Ni) followed by Phase 2 at 8 Mtpa and Phase 3 at 12 Mtpa. The capacities of the expansions after Phase 1 would be determined at the time trough additional studies. 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. Ivanhoe commissioned a marketing study on the sale of concentrates, furnace mattes, converter mattes and PGM concentrates in 2013.

With the advent of a number of smaller PGM mining firms, toll smelting and refining contracts and purchase agreements have become more prevalent in South Africa than in the past. The major PGM 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, PGM 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 PEA.

The key companies in the South African PGM industry are: Impala, Anglo Platinum Northam Platinum, Lonmin and Aquarius Platinum:

- The major producers, Impala and Anglo Platinum have a full suite of process facilities and hence tend to be purchasers rather than sellers of any PGM containing materials. The only exception being that Anglo Platinum has sold some PGM residues from the Base Metal refinery to Umicore, a European refiner, on a long-term contract.
- Northam Platinum has been one of the smaller producers that opted for investment into their own smelting and partial base metal refining facilities; they sell their nickel as a sulphate, copper is electro-won and their high grade PGM concentrates are sold to a Hereaus, a European refiner with some local operations.
- Lonmin has similar smelting and base metal refining operations, thus sells its nickel sulphate and electro-won copper but processes its own PGM concentrates in a dedicated PGM refinery.
- Aquarius Platinum, a once significant producer, does no smelting or refining so sells all its flotation concentrates to either Impala or Anglo Platinum.

The sale of PGM 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 delivery and payment. The pipeline for some metals (e.g. rhodium) is as long as 9 months such that these terms can have a serious 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 have not found a market offshore and are all sold to local smelters.

Marketing of the products from the Platreef Project is of significant importance to the project value. The terms of payment are highly variable and subject to change as conditions for any one potential purchaser alter. Each are subject to competitive pressures, capacity pressures, tolerance for impurities (Cr2O3, etc) changing feed mixes, cost pressures, exchange rates, the impact of product quality discounts/premiums and process efficiency improvements. Thus it is difficult to generalise about the value available from various products. However, the assumptions in the Platreef 2014 PEA represent a reasonable estimate based upon the current market knowledge.

The estimates in the Platreef 2014 PEA are based on knowledge of concentrates sales contracts that have been agreed in Southern Africa with local purchasers and for the offshore purchase of mattes and PGM concentrates. As the local purchase of furnace mattes or converter mattes has only occurred infrequently and for small volumes, an estimate has been based on the operating costs and an approximation for a required margin. Actual terms may vary and will be dependent on the negotiations at the time the contracts are agreed.

Whilst there is sufficient furnace capacity in South Africa, the converting and sulphur removal capacity is constrained by environmental sulphur emissions permits. There is some available converting and acid plant capacity, but the high iron and sulphur levels in the Ivanhoe concentrates will likely fill this very quickly, and additional capital expenditure would then be required.

The marketing study concluded that nickel refining capacity could accommodate Phase 2 of Ivanhoe's development plan, for an expansion from 9 ktpa to 18 ktpa nickel. Whilst the current refining gross footprint would not need expansion, further modules of production will need to be added to accommodate the Ivanhoe converter mattes.

The local opportunity for Ivanhoe in the initial phases of their operations is thus limited to utilising the spare nickel capacity that is available but this will require solutions to the environmental issues and successful negotiations between Ivanhoe and the three smelters and refiners. The availability of excess capacity required for Phase 2 of the project is largely dependent on the state of the PGM industry in 2024 (the current estimate for the start of Phase 2).

19.2 Flotation Concentrates

The sale of PGM 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 delivery and payment. The pipeline for some metals (e.g. rhodium) is as long as 9 months such that these terms can have a serious 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 have not found a market offshore and are all sold to local smelters.

19.3 Intermediate Products

The sale of PGM rich intermediate products from further downstream of the concentrators such as furnace mattes or converter mattes is less developed as a market and no regular sales are being made.

However, some nickel-copper matte after magnetic removal of the PGM-containing alloys has been sold both locally and offshore from South Africa and Botwana. The most significant purchasers are within South Africa. The terms for PGM in mattes terms are normally less that the terms for concentrates but the base metal terms can be an improvement relative to the terms for concentrates.

19.4 PGM Concentrates

The PGM concentrates that are left 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 PGM refiners and fabricators globally. The amphoteric elements (Sb, Bi, Te, Se, As, etc) are seen as deleterious and penalties are a possibility for unusual amounts of these contaminants. Local refiners could be quite competitive as their operating costs are low compared to offshore operators but no regular contracts are in place for comparison.

19.5 Available Capacity

19.5.1 Available Smelter Capacity

Currently furnace capacity for concentrates is available in South Africa.

Sulphur capture facilities are strained such that added sulphur in their feed is to be avoided until they invest further in acid plants.

Northam has little spare capacity as the operation is a single, small 15 MW furnace which treats platinum material as well as their own concentrates. Impala has historically taken the view that they are in the toll smelting business and endeavour to keep some capacity available for the junior small-scale PGM producers.

The conventional operations also have limited capacity for UG2 concentrates as the level of Cr_2O_3 is a concern in the traditional six-in-line furnaces. Hence high penalties are applied to Cr_2O_3 levels of more than 1% whilst generally concentrates with Cr_2O_3 in excess of 3% are rejected as feeds to conventional smelters. With the concentrates expected from Platreef this will not be a concern. The iron (Fe) and sulphur (S) in the Ivanhoe concentrates would require increased converting and acid plant capacity. It will be necessary to study in further detail the SO_2 abatement. Obtaining environmental permission for this increase may prove difficult.

The marketing study concluded that there is adequate furnace capacity in Southern Africa operations for at least the first three phases of Ivanhoe production however the high Fe/S content of Ivanhoe concentrates exceeds the available converting and acid plant capacity.

19.5.2 Base Metal Refining Capacity

The converter mattes produced from any smelting/converting operations can be sold to offshore nickel producers or refined in the local base metal refineries. There has been some recent expansions that are yet to achieve design capacity. In the medium to longer term further expansions are expected that will add somewhat to the nickel loading at the refinery.

19.5.3 PGM Final Concentrate Refining Capacity

The marketing study concluded that there is at present spare capacity available for PGM concentrate refining in Southern Africa.

19.6 Smelting and Refining Contracts and Cost Structures

Based on the marketing analysis studies by Ivanhoe the realisation assumptions used for the Platreef 2014 PEA are shown in Table 19.1. The tendency in the market for concentrates has been to levy a treatment charge in ZAR/ton of dry concentrate to cover drying, smelting, converting and acid production costs as within the smelter complex the tonnage of concentrate is the largest cost driver. These treatment charges are typically in the range of R1,300 to R2,000 per ton of dry concentrate treated and are payable within 5–7 days after the close of the month of delivery.

Metal losses in the smelting and refining complex as well as base metal and PGM 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.

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 costly as well. There are no really typical refining charges but locally such charges are expected to be based on operating costs, so around ZAR25-40,000/t of refined metal would not be surprising. Offshore such costs are often quoted as a charge for each recoverable pound of nickel and copper.

At present exchange rates this would be around ZAR25-35,000/t. 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 recovery of PGMs from high grade refinery concentrates (40% PGM) is very high. Most purchasers of these materials will give payable amounts of around 99% for Pt and Pd and a few percentage points less for Rh, Au, Ir and Ru. PGM refining charges of ZAR80–90/kg material plus some ZAR3, 200–4,200/kg (Pt, Pd, and Au) and ZAR17,000–22,000/kg (Rh, Ru, Ir) are typical.

Table 19.1 PEA Realisation Assumptions

Product	Net return as a % of 3PE+Au plus Cu/Ni**	Minimum Specs	Treatment Charge	BM Terms – Ni, Cu, Co	PGM Terms	Pipelines	Penalties
Flotation Concentrates	81–83	>80 g/t 3PE+Au; < 1.5% Cr_2O_3	R 0 to R200 /t	82-86% for Ni, Cu no payment for Co	82-86% Pt, Pd, Rh, Au; 0% for Ir, Ru	BM 2 months, Pt, Pd & Au 3 months, Rh 6 months*	>1% Cr_2O_3 with >3% rejected; low PGM grades

Table shows Approximate Values of PGM Products in the Value Chain based on Platreef Material.

* Often the purchaser of flotation concentrates absorbs the pipeline cost in the general terms offered, however; the forward price risk often remains with the seller. Thus a large percentage of the payment (70-90%) for all metals can be received in the month following the month of delivery and final payment is adjusted for prices at the stipulated pipeline.

** Total payment less the charges divided by the 3PE+Au PGM plus Cu/Ni contained metal value.

***Total payment less the charges divided by the 3PE+Au PGM contained metal value.

19.6.1 Metal Recoveries

Actual metal recoveries by the major PGM company process divisions (smelting, base metal refining and PGM 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 low-Fe mattes produced.

Table 19.2 Typical Metal Recoveries (from Concentrates)

Metal Concerned	Recovery Range %
Base Metals	
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 recovered and sold. These metals have limited markets that have been historically in oversupply. Large stockpiles are often held by some of the producers. 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.

19.6.2 Payment Pipelines

The process required to produce pure metals from both refining operations takes a significant time and a large inventory of valuable 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 but most refiners simply apply a single fixed period to each of the metals 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.

19.6.3 Penalties

Penalties can be levied against the seller of concentrates for high moistures, low PGM grades and high chromite levels. Within the refining circuit elements such as Fe, As, Bi, Sb, Se, Te, Pb, Zn, and SiO_2 can be problematic such that buyers of mattes and PGM concentrates may levy penalties for some of these elements.

19.6.4 Terminal Sale Agreements

The final refiners of the metals are often in the position of selling their 'pure metals' at a discount to the LME ruling prices either because of the quality of the metal (base metals often attract discounts or premiums to the LME based on quality) or because of agreements with bulk purchasers (PGM metals are often discounted from the free market price for large volume buyers). Precious metal discounts have been as much as 3% and base metal discounts can be as high as 10% for poor quality products or can carry premiums of as much as 20% in the case of high grade nickel for the electroplating market.

In effect these discounts/premiums are absorbed by the refiners because the purchase contracts are often written based upon LME ruling prices at the time of purchase; this is another reason for the wide variety of terms offered to producers. South African base metal refiners produce lower grades of copper metal and only stainless steel grades of nickel metal hence copper is discounted from the LME by as much as 20% whilst nickel sells at LME or just below.

19.7 Conclusion

Marketing of the products from the Platreef Project is of significant importance to the project value. The terms of payment are highly variable and subject to change as conditions for any one potential purchaser alter. Each are subject to competitive pressures, capacity pressures, tolerance for impurities (Cr_2O_3, etc) changing feed mixes, cost pressures, exchange rates, the impact of product quality discounts/premiums and process efficiency improvements. Thus it is difficult to generalise about the value available from various products. However, the assumptions in the Platreef 2014 PEA represent a reasonable estimate based upon the current market knowledge.

The estimates in the Platreef 2014 PEA are based on knowledge of concentrates sales contracts that have been agreed in Southern Africa with local purchasers and for the offshore purchase of mattes and PGM concentrates. As the local purchase of furnace mattes or converter mattes has only occurred infrequently and for small volumes, an estimate has been based on the operating costs and an approximation for a required margin. Actual terms may vary and will be dependent on the negotiations at the time the contracts are agreed.

Whilst there is sufficient furnace capacity in South Africa the converting and sulphur removal capacity is constrained by environmental sulphur emissions permits. There is some available converting and acid plant capacity but the high iron and sulphur levels in the Ivanhoe concentrates will likely fill this very quickly and additional capital expenditure would then be required.

The marketing study concluded that nickel refining capacity could accommodate Phase 2 of Ivanhoe development plan, for an expansion from 10 ktpa to 20 ktpa nickel. Whilst the current refining gross footprint would not need expansion, further modules of production will need to be added to accommodate the Ivanhoe converter mattes.

The local opportunity for Ivanhoe in the initial phases of their operations is thus limited to utilising the spare nickel capacity that is available but this will require solutions to the environmental issues. This would require extensive successful negotiations between Ivanhoe and the three smelters and refiners. The availability of excess capacity required for Phase 2 of the project is largely dependent on the state of the PGM industry in 2024 (the current estimate for start of Phase 2).

Opportunities for using flash furnace capacity and selling/tolling the resultant matte either locally or offshore remain, however the high MgO technical problem needs resolution and there may be investment needed in sulphur removal.

Precious metal refining capacity in South Africa could be expanded to be sufficient to treat all of the currently planned output from Ivanhoe. The capital required will be small when compared to the base metal refining investment.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

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 farm Rietfontein 2 KS. Ivanhoe intends to apply for a mining right over all three farms. The farm Bultongfontein 239 KR also formed part of the investigations.

The Project site lies in a north westerly direction, approximately 8 km from the town of Mokopane (previously known as Potgietersrus). The 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 Terms of Reference

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 mining right application (MRA) that was submitted on 6 June 2013. These studies 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; and
- 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.

20.3 Summary of Relevant Baseline Results

20.3.1 Wetland Systems

In light of the fact that the Rooisloot, Ngwaditse and Dithokeng rivers are ephemeral systems and as a result, no ecological state assessment could be conducted during the field survey (November 2011) due to these systems being predominantly dry, it is recommended that an additional aquatic assessment high flow survey be conducted. This would then provide a baseline description for these systems which is currently limited. It is recommended that a high flow assessment be conducted during February/March 2012 at selected sites on the three respective systems where possible. In addition to this, samples sites should also be selected on the Mogalakwena River.

20.3.2 Fauna and Flora

20.3.2.1 Vegetation Survey

Plant Communities Identified

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. 135 species were identified throughout the Project area. Six vegetation communities were identified, including:

- Ridge Bushveld;
- Impacted Ridge Bushveld;
- Degraded Mixed Bushveld;
- Secondary Grassland and Agricultural fields;
- Wetland vegetation; and
- Residential areas.

Red Data Plant Species

Red Data species identified by the PRECIS data considered during the field survey were not identified; 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 (Sheperds tree); and
- Sclerocarya birrea (Marula).

Exotic and Invasive Plant Species

Declared weeds and invaders have the tendency to dominate or replace the canopy or herbaceous layer of natural ecosystems, thereby transforming the structure, composition and function of natural ecosystems. The amended Regulations (Regulation 15) of the Conservation of Agricultural Resources Act, 1983 (Act No. 43 of 1983) identify three categories of problem plants:

- Category 1 plants may not occur on any land other than a biological control reserve and must be controlled or eradicated. Therefore, no person shall establish plant, maintain, propagate or sell/import any category 1 plant species. Three Category 1 species were identified.
- Category 2 plants are plants with commercial application and may only be cultivated in demarcated areas (such as biological control reserves) otherwise they must be controlled. One Category 1 species was identified.
- Category 3 plants are ornamentally used plants and may no longer be planted, except those species already in existence at the time of the commencement of the regulations (30 March 2001), unless they occur within 30 m of a 1:50 year floodline and must be prevented from spreading. Three Category 3 species were identified.

Cultural Important Plant Species

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. Animal survey.

20.3.2.2 Animal Survey

Desktop studies on animals included identifying animals potentially occurring in the area of interest and identifying their Red Data status. Eleven mammal species were found during the field survey and two of these species have a protected status.

Mammals Found During the Field Survey

A full species list of mammals observed has been recorded. Two of the species found are protected under Schedule 3 of the Limpopo Environmental Management Act. Both of these species, the Grey rhebok and Yellow-spot dassie, were found to the north-east of the project area, which is a ridge range that forms part of the Witvinger Nature Reserve.

Red Data and Protected Mammals

The probability of occurrence of the Red Data species considered 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 project area.

20.3.2.3 Birds

Bird Species Found During the Field Survey

During the field survey 49 species were observed. This list cannot be considered as a complete list as many other birds can be present within any given season or day of the year. During the dry season survey, bird activity was greatly reduced. Therefore it is recommended that any development should take place during the dry season so that the fledgling numbers are reduced.

Red Data and Protected Bird Species

The Yellow-Billed Stork and African Spoonbill are protected by the Agreement on the Conservation of African-Eurasian Migratory Waterbirds (AEWA). 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 were 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. 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.

20.3.2.4 Herpetofauna

Red Data Herpetofauna

No Red Data status reptiles were found during the surveys.

20.3.3 Air Quality

20.3.3.1 Suspended Particulates

The ambient concentrations of suspended particulates highlight the impact of seasonal changes on dust load in the atmosphere. In the wet season (late Spring and Summer) the recorded concentrations were below the recommended SA 24-hour guideline value (125 µg/m^3). Concentrations measured from April to June (with the exception of May) fell above the SA 24-hour guideline, presumably as a result of the dry season.

The temporal changes in suspended dust load indicate that the pre-mining environment is characterised by seasonal changes in dust load, which results in varying dust loads in the ambient air, depending on the time of the year.

The contribution of very fine fraction particles (PM2.5) as a fraction of total PM10 is significant, especially during the months (May to June). During the month of June these particles accounted for at least 40% of the total concentration.

20.3.3.2 Fallout Dust

During the period of Autumn to Spring, the results were characterised by heavy dustfall (i.e. above the South African Department of Environmental Affairs Guidelines value of 1,200 mg/m^2/day. From September 2003 to February 2004 the measured dustfall decreased significantly from heavy to moderate fallout (i.e. below 750 mg/m^2/day). The very heavy dustfalls at Mokopane indicate that the pre-mining environment is characterised by heavy dustfalls during the dry season.

The community lands and Madiba (which is open land) show consistently heavy dustfall for most of the monitoring. The measured dustfall in these sites is mainly attributed to ploughing carried out by local farmers. The ploughing activities tend to increase the surface susceptibility to wind erosion because particle cohesion is reduced. The fallout samples collected from sites Ms Pohotona and Madiba also show heavy dustfall. These sites are located in Mahwelereng Township and are intended to measure and quantify population exposure to nuisance dust in the pre-mining environment.

The results obtained from these two sites show that the population is exposed to varying concentrations of dust. From these preliminary results it can be deducted that during the dry winter months dust load in the area is heavy.

20.3.4 Noise

The results of the ambient noise measurements taken at the relevant communities near the proposed mining activities indicated that the baseline noise levels are mostly higher than that of the SANS 10103:2008 night time guideline levels for suburban districts. 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 20.1.

Table 20.1 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
	Indicates $L_{Aeq,T}$ levels above either the daytime rating limit or the night time rating limit					

Note: LAeq,T is the equivalent continuous A-weighted sound pressure level, in decibels, determined over a time period of 16 hours during the daytime and 8 hours during the night time. The Maximum/Minimum is the highest/lowest reading during the specified time period over which the measurement was taken. 'A-weighted' is a standard weighting of the audible frequencies designed to reflect the response of the human ear to noise.

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 typically traffic and crickets respectively.

20.3.5 Water Balance Results

The water balance is a deficit water balance, and significant quantities of make-up water will be required. The make-up requirements are seasonal. During the wet season, storm water will be harvested from the tailings storage facility basin and return water dam. Returns consequently increase and make-up requirements decrease. The opposite is true during the dry season.

20.3.5.1 Management Systems

Proposed process water management systems include:

- Decant barge pumping system.
- Water Storage System.
- Water Return System.
- Diversion Channels.
- Environmental Control Measures.

20.3.6 Closure Considerations

The closure plan for the TSF will be developed during the life of the facility. The purpose of preparing a closure plan is to ensure that the TSF design, construction and operation procedures are compatible with the achievement of final closure and rehabilitation to acceptable environmental standards and at a reasonable cost. It is anticipated that the closure plan will be updated periodically before the preparation of the final closure plan. The closure plan will be prepared in accordance with "best practice" and the requirements of the environment.

20.4 Baseline Groundwater Monitoring Plan

Ivanhoe has enagaged Golder Associates to carryout groundwater studies on the Platreef Prroject the studies are examining the results from hydrocensus surveys, geophysical surveys, exploratory drilling, test pumping, numerical modelling and hydrogeological assessment of the local groundwater conditionsThe groundwater quality from the 2013 water quality assessment of the Platreef Monitoring Network (PMN) and the initial hydrogeological baseline study in August 2012 is not significantly different. It is; however, it is clear that excluding monitor sites GPR-13 and GPR-14, the Platreef study area's water quality is less impacted than the surrounding areas covered during the August 2012 hydrocensus survey.

Long term monitoring of groundwater levels and groundwater quality is essential to establish a history of the status of the natural (pre-mining) groundwater situation and to satisfy compliance requirements of the DWA and EIA. The monitoring database is then available to confirm the baseline situation against which any impacts from mining operations on the groundwater resources of the aquifers present can be measured. The monitoring data form the basis for long term groundwater management.

The water use in the area is already high and will require substantial management and collaboration between water users to prevent unnecessary impeachments.

The general groundwater quality status of the PSA is good (Class 1/2) compared to the area cover by the larger hydrocensus survey. There are places outside the PSA where the groundwater quality is poor and not suitable for human/domestic consumption. It is evident that due to the groundwater flow regime (based on the piezometric surface showing a distinct westerly flow direction) deterioration of the groundwater quality will developed in future.

20.4.1 Recommendations

- The groundwater level and quality monitoring programme on the PMN should remain operational in the light of the mine infrastructure development phase ahead. This activity is to secure a sound groundwater quality and level baseline and trend information data base which can be used for reference and planning in the future.
- The monitoring programme should be updated and maintained throughout Life of Mine and into closure to observe changes in relation to the baseline reference status obtained during the Baseline and later study phases

- The current groundwater monitor network and operations should remain as a minimum monitoring requirement for time being. The monitor protocol and configuration will have to be adopted to cover the final production phase of the mine. It might also be required to consider a long-term monitoring strategy for the mine decommissioning phase as required by the regulators/authorities. Any monitoring site (borehole) that becomes unusable due to vandalism or construction deterioration should be replaced immediately to prevent the development of data/information gaps. An updated monitoring network should be implemented in 2014
- A monitoring borehole must be drilled close to each vertical shaft to monitor the shallow aquifer water level responses. These monitoring boreholes are to be equipped with automatic water level data loggers.
- Groundwater quality samples must continue to be analysed by an accredited laboratory.
- Independent quarterly and annual monitoring reports should be prepared
- A numerical model of the Platreef Project should be maintained and updated as data becomes available and changes occur. This can then be used to simulate differing scenarios and assist with implementation of strategies designed to manage impacts.
- The reason for the poor groundwater quality observed NW of the proposed mine infrastructure area should be investigated. Although there was no definite trend observed since the sequential quarterly sampling of the groundwater at monitoring site GPR-13 started in 2011, the water quality at GPR-13 is a concern and probably from an external pollution source.
- An assessment of the groundwater quality and level monitoring results should be conducted by the end of the current hydrological cycle to plan/economise the monitoring programme for the 2014 hydrological cycles.
- To conclude, consideration of expanding the sequential groundwater level monitor programme will have to be done; especially to include a few external (>1,000 m) monitoring sites before shaft sinking and other water consuming activities start.
- Ivanhoe should consider establishing a local water resources monitoring committee where issues around water uses, potential impacts and water quality trends can be discussed and shared.

20.5 Licences and Permits

20.5.1 Mineral Rights

Ivanhoe is the holder of a converted old order prospecting right over the farms Macalacaskop No. 243, Registration Division KR and Turfspruit No. 241, Registration Division KR ("Prospecting Area") granted in terms of Schedule II of the Mineral and Petroleum Resources Development Act No. 28 of 2002 ("the MPRDA") which entitles it to exclusively prospect for base minerals and precious metals in, on and over the Prospecting Area. The Prospecting Right is 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 MPRDA 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 grants 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 MPRDA over the Prospecting Area for the same minerals. This application has been submitted to the DMR electronically through the SAMRAD portal on 6 June 2013, and was accepted by the Regional Manager on 17 July 2013, but is still pending approval. The application number in respect of the mining right is LP30/5/1/2/2/10067MR.

Platreef also has an agreement in place with Plateau Resources (Pty) Ltd, the holder of a prospecting right over the adjacent farm Rietfontein No. 2, Registration Division KS, which entitles Platreef to the base minerals and precious metals.

20.5.2 Mining Right Applications

The Project applied for a Mining Right in Q2 2013 andthe formal environmental Scoping Phaseand EIA phase has been completed. Environmental Authorisations will need to comply with the requirements of the National Environmental Management Act, No 107 of 1998, and the Mineral and Petroleum Resources Development Act, No 28 of 2002.

In addition, the authorisations and applications detailed in Table 20.2 needed to be initiated prior to the Mining Right application.

Table 20.2 MRA Authorisations and Applications

Authorisation/Application	Legal Requirement	Timeframe
Mining Right Application	MPRDA 28 of 2002, s 22 & GN R.527 Reg 10 & 11	12 months
Approved EIA & EMP	Mineral and Petroleum Resources Development Act, Act 28 of 2002, s39(1) & GN R.527 Reg 48 -51	12 months
Environmental Authorisation and EIAR/EMP approval	National Environmental Management Act, Act 107 of 1998, s24(1), (2) & (5) & GN R543, GN R544, GN R545 and GN R546	12–18 months
Waste License Application	National Environmental Management; Waste Act, Act 59 of 2008, s20 & GN R 718	12–18 months
Rezoning Application	Town-Planning and Townships Ordinance No.15 of 1986	6–12 months
Integrated Water Use Licence Application	National Water Act, Act 36 of 1998, s21 & s40	18–24 months
Industrial water supply agreement	Water Services Act, 108 of 1997, s 7	No timeframes

20.5.3 IWULA, WULA, and NEMWA

20.5.3.1 General Authorisation (GA), Integrated Water Use Licence Application and Integrated Waste Water Management Plan

An assessment of the GA from four properties (1,077 ha) on the farm Uitloop 3KS was prepared and submitted to DWA Limpopo Region (21 October 2013). An acknowledgement of receipt was obtained. A request from DWA for Ivanhoe to submit copies of agreements for the taking of water from two privately owned properties on the farm Uitloop 3KS was subsequently received. Golder drafted agreements for each of the two landowners. These were submitted to Ivanhoe for signature by the landowners and return. Subsequently the signed agreements have been submitted to DWA to finalise registration of the water use on the WRMS data base.

The BSS IWULA and IWWMP were submitted to DWA, Limpopo Office on 6 November 2013. Receipts were obtained and copies provided to Ivanhoe.

A meeting with 4 members of the DWA 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: As a result of the visit the following additional information was requested:

- Borehole abstraction forms (although they have been submitted for the General Authorisation, DWA decided they should also be included in the IWULA);
- The Water Balance should be presented in a clearer format;
- An alternative means of sewage disposal needs to be considered such as a package plant since the Mokopane and Polokwane STWs are overloaded and poorly operated and DWA would not approve of disposal in these facilities;
- The design drawings for the PCD must be signed and the Engineers registration included.
- These additional items were to be submitted to DWA by end of February 2014.

20.5.3.2 IWULA Main Mine

Ivanhoe has signed all the forms and these have been presented to DWA. The TSF Report has been received and At the meeting held on 17 February at Ivanhoe, DWA requested that 1 month's notice is given of our intention to present the Main Mine IWULA. A meeting request has accordingly be arranged for the end of March 2014. It will not impact on the approval timeframes as the same officials work on all the Ivanhoe Applications.

20.5.4 NEMA

An application was made in terms of the National Environmental Management Act No. 107 of 1998 (NEMA), reference number 12/1/9/2-W32; LIM/EIA/0000538/2013.

20.5.5 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.

20.6 Socio-Economic Baseline

The proposed project is located within the Waterberg District, in the Mogalakwena LM. More specifically the projects area of interest coincides with 13 (Ward 13, 18-31) of the 32 local municipal wards. There are 15 villages located within these wards.

20.6.1 Land Claims and Ownership

A land claims enquiry on the farms within the Platreef prospecting area noted that there were claims on Turfspruit 241-KR and Rietfontein 2-KS; another land claim exists on Bultongfontein 239-KR, which is one of the properties considered for the placement of operational infrastructure. Claims for these properties are still pending on behalf of the Mokopane Trust and Mamahsela community. These are still in the process of being validated by the land commissioner. At the time of the enquiry the land claims had not been gazetted and the status of the claim was 'research' therefore the claim is still under investigation.

The majority of the prospecting area is owned by the Government of the Republic of South Africa, but is identified as Indigenous/traditional land. This means that the Traditional Authority has jurisdiction over the land and holds the land in trust for its people. It needs to be noted that several factions within the community do not recognise the TA and/or the local chiefs.

20.6.2 Mine-Community Relations

Community perceptions about-and attitudes towards the proposed Platreef Project can been shaped by social political events and/or existing attitudes towards mining activities within the project area. Anglo American's Mogalakwena operation and Lonmin's prospecting activities are located relatively close to the Platreef operation, and are also focussed on Platinum extraction.

Communities affected by the Anglo, Lonmin and Platreef projects respectively, have in the past launched protest actions against these cooperation's. This might indicate widespread discontent against Platinum mining houses within the study area.

20.6.3 Potential Social Risks

Social risk is linked to a project's stakeholders and can either be a risk to a project as a result of the impact on stakeholders or stakeholders' impact on the project. In most cases a risk can be financial, delay or reputational. The potential social risks which the project might be exposed to are:

- Community expectations.
- Social unrest and community opposition.
- Political tensions.

- Failure to acquire a social license to operate.
- Physical and economic displacement.
- Land claims.

20.6.4 Social and Community Impact

Several large urban communities inhabit portions of the Platreef Project area (refer to Figure 4.4).

Some portion of the communities may be required to be relocated or economically relocated, with more relocation required to enable open-pit mining of the ATS deposit versus open-pit mining of the AMK deposit or underground mining of the UMT deposit. Such relocations have previously been performed in South Africa; however, any full relocation of Tshamahansi would be amongst the larger moves contemplated. Future detailed studies would be necessary should such relocation be required.

In 2007, a detailed study was performed of the requirements to support relocation of selected communities if an open-pit mine was developed (Synergy, 2007). The study identified three key areas requiring careful management and lead-in times, including:

- Timing: The time required for effective community relocation was considered to be dependent on successful community engagement, and the capability of municipal and provincial authorities to provide necessary infrastructure. Estimates of the time required for planning, construction and relocation varied between seven and nine years, based on the community sizes and areas in 2007.
- Costs: The costs of relocation are likely to be higher than those estimated in 2007. In general terms, building costs in South Africa have been increasing at a rate in excess of official inflation largely as a result of substantial growth and demand in this sector.
- Land: Identification of suitable land for relocations considering land availability and costs and project infrastructural requirement.

Synergy (2007) noted that, at a minimum, the following points are likely to require consideration in future evaluations:

- Identification of issues, liabilities and costs relating to any proposed resettlement. This should be supported by studies of "lessons-learned" outcomes from resettlements undertaken by other mining companies in South Africa, in particular that of Anglo Platinum at Ga Pila.
- Assessment of alternatives, including assessment of different potential sites, land status evaluations, whether the resettlement should be completed as a single phase or multi-phase operation and over what timeframe.
- Evaluation of government and community support for any proposed move.
- Inclusion of appropriate organizations, whether governmental, civil, or traditional, in decision-making processes related to any proposed resettlement.

- Ability of existing governmental, civil, and traditional authorities to manage any proposed resettlement, and to provide support for the communities post-relocation, ability of these various parties to manage any conflicts or disputes that may result.
- Provision by Ivanhoe for stakeholders to provide feedback on the process, and provision for establishment of accepted grievance mechanisms so that stakeholders' dissatisfactions are addressed in a timely manner.
- Flexibility of Ivanhoe to react to changing perceptions in the community or to changes to the stakeholders involved both during the duration of any proposed move and the life of the Project.
- Consideration of likely social (e.g. inheritance) or economic (e.g. awarding of construction contracts) impacts on the communities.
- Consideration of Equator Principles and International Finance Corporation guidelines.

There is currently no overarching integrated legislation or policy that governs involuntary land settlement in South Africa, and consideration will have to be given to a number of different legislative instruments as well as the experience of similar settlement relocations undertaken by other companies. Additional legislation may be passed into law in the future, and there can be no guarantee that relocation of the settlements can be agreed between the parties involved, or that the timeframe, or the terms under which an agreement may be completed will remain in agreement.

Ivanhoe considers that a full census to identify affected villages and communities, and identification of any affected infrastructure, such as schools, churches, and recreation facilities, will be required, but the census should be conducted when the most appropriate mining and development routes have been determined for the Project. Such a census will be conducted by an appropriately qualified Independent Consultant once any mining right application has been submitted.

Ivanhoe has already instituted policies and consultation procedures and a dedicated Community and Government Relations (CGR) Team that will promote community relations. Regular meetings have been held with community representatives.

20.6.5 Platreef Skills and Business Survey

In March 2012, Digby Wells Environmental was appointed by Platreef to undertake a sample survey of skilled individuals and small business enterprises in the Mokopane area. The primary purpose of this survey was to establish an electronic database and knowledge repository of formal and informal businesses as well as skilled individuals within the proposed project's primary labour sending area.

During the survey, a total of 8,634 respondents registered their skills on the database. In general individuals who registered are relatively young, with 89% being younger than 40 years. With regards to gender females (52%) slightly outnumber males (48%). About 90% of respondents have at least some secondary education, while a quarter have attained some kind of tertiary education.

The majority (87%) of individuals who registered on the database are unemployed. However, most of them were previously employed and have some workplace experience. The majority of individuals were previously employed in the retail (12%), administration (10%) and service sectors (10%). Another 7% of individuals were previously employed in the mining sector.

A total of 537 respondents registered their businesses on the business database. Unlike the residency of respondents registered on the skills database, the results derived from the business survey, showed that a larger number of businesses are located near the immediate study area.

Most businesses specialise in building and construction (20%), providing services (12%), and catering (10%). About 25% of these businesses are located in Mahwelereng Village, and have been trading for more than three years. Nearly 80% of businesses are registered as Close Corporations, while only 5% are informal or unregistered businesses.

With regards to the total workforce of registered businesses, most companies indicated that they employ less than five employees, while 35% indicated that they employ five to 19 employees. Only 12% of companies indicated that their total workforce ranges between 20 and 99.

Business owners were requested to indicate whether they are involved in contract work, and just less than one third indicated that they were. These businesses mostly specialise in the construction, service provision, and supply sectors. Of the businesses regularly involved in contract work, the experience with mining contracts includes catering, maintenance, construction, service provision, and supply.

It should be noted that the results of the survey only reflect the current trends in the skills and business domains and, as a result, the usefulness of the databases will be limited in future by the dated nature of the information they contain. To maintain and improve the utility of these databases, Digby Wells recommended that they should be updated on a regular basis, either by updating existing skills profiles or by allowing new registrations.

20.7 Recommendations

To comply with local and international requirements, a project of this magnitude would require the following:

- ESIA in accordance with the Mineral and Petroleum Resources Development Act (MPRDA), the National Environmental Management Act (NEMA) as well as the EP and IFC Performance Standards;
- Stakeholder Engagement Process (SEP) in accordance with the NEMA 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).

20.7.1 Legal Requirements

20.7.1.1 Local Legislation

Prior to construction and operation of an underground mine, the following local legislative authorisation would be required:

- In support of a MRA, authorisation in terms of Section 22 of the Mineral and Petroleum Resources Development Act, 2002 (Act No. 28 of 2002) (MPRDA) by the Department of Mineral Resources (DMR) is required;
- Authorisation of listed activities in terms of the National Environmental Management Act, 1998 (Act No. 107 of 1998) (NEMA) and Associated EIA Regulations (GNR. 543, 544 and 545 of 18 June 2010) from the Limpopo Department of Economic Development, Environment and Tourism (LEDET);
- Authorisation of water uses by obtaining an IWULA in terms of Section 21 of the National Water Act, 1998 (Act No. 36 of 1998) from the Department of Water Affairs (DWA); and
- A Waste Management License for categorised waste activities in terms of the National Environmental Management Waste Act, 2008 (Act No. 59 of 2008) (NEMWA) will be submitted to the National Department of Environmental Affairs (NDEA) or equivalent regional authority depending on the classification of certain waste streams.

20.7.1.2 International Standards

All environmental and social studies completed for the proposed project will be in conformance with the framework provided in the World Bank Group (WBG) and IFC policies and guidelines for Environmental Assessment (EA).

With this in mind, the main guidelines that will be followed in developing the ESIA scoping 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).

20.7.1.3 Equator Principles

The Equator Principles (EP) is a set of environmental and social benchmarks. Once adopted by banks and other financial institutions, the Equator Principles 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.

20.7.2 Environmental and Social Impact Assessment

The following activities will be conducted during the compilation of an ESIA to comply with local and international requirements.

20.7.2.1 Scoping Phase

- Compilation and distribution of 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.

20.7.2.2 EIA Phase

- Conduct specialist investigations and compile specialist reports;
- Compile Environmental Legal Register for construction, operations and closure;
- Compile the following draft reports:
- MPRDA compliant ESIA report;
- NEMA and IFC compliance ESIA report;
- Draft Environmental Management Plan (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.

20.7.2.3 Permitting Requirements

The following application forms will be submitted during the course of the proposed project:

- IWULA in terms of the NWA; and
- Integrated waste management license application in terms of the NEMWA.

20.7.2.4 ESIA Specialist Studies

The following specialist studies are to be conducted for the compilation of the Environmental and Social Impact Assessment Report to ensure compliance with local and international requirements:

- Legal Assessment;
- Soils and Land Capability Assessment;
- Fauna and Flora (Biodiversity) Assessment;
- Aquatic and Wetlands Assessment;
- Noise Assessment;
- Air Quality Assessment and dust monitoring;
- Topography Assessment;
- Visual Assessment
- Heritage and Archaeology Assessment;
- GIS and Mapping;
- Greenhouse Gas Inventory (in compliance with international requirements);
- Health Baseline Assessment (in compliance with international requirements);
- Socio-economic Assessment;
- Resettlement Action Plan (RAP);
- Rehabilitation and Closure Cost Assessment;
- Detailed terms of reference for each of these studies are available on request.

21 CAPITAL AND OPERATING COSTS

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: R10=$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 pre-production capital cost including contingency for each case is shown in Table 21.1. The pre-production, sustaining, and total capital costs are shown in Table 21.2 and Table 21.3.

Table 21.1 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
Capitalized 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 & Studies	–	19	19
Mining: Indirects	55	58	58
Mining: EPCM	80	93	97
Processing & Infrastructure: EPCM	37	37	37
Subtotal	**172**	**207**	**211**
Owners Cost			
Capitalized G&A	26	26	26
Mining	60	79	79
Processing & Infrastructure	17	18	17
Subtotal	**103**	**123**	**122**
Capital Expenditure Before Contingency	**1,185**	**1,338**	**1,376**
Mining Contingency	221	259	272
Processing & Infrastructure Contingency	120	122	120
Capital Expenditure After Contingency	**1,525**	**1,719**	**1,769**

Table 21.2 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
Capitalized 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 & Studies	–	–	19
Mining: Indirects	–	–	–
Mining: EPCM	–	–	–
Processing & Infrastructure: EPCM	4	33	63
Subtotal	**4**	**33**	**81**
Owners Cost			
Capitalized G&A			
Mining	–	–	–
Processing & Infrastructure	2	14	29
Subtotal	**2**	**14**	**29**
Capital Expenditure Before Contingency	**814**	**2,029**	**3,282**
Mining Contingency	124	354	572
Processing & Infrastructure Contingency	36	146	266
Capital Expenditure After Contingency	**974**	**2,528**	**4,120**

Table 21.3 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
Capitalized 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 & Studies	–	19	38
Mining: Indirects	55	58	58
Mining: EPCM	80	93	97
Processing & Infrastructure: EPCM	41	70	99
Subtotal	**177**	**240**	**292**
Owners Cost			
Capitalized 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 & Infrastructure Contingency	156	268	386
Capital Expenditure After Contingency	**2,499**	**4,247**	**5,888**

21.3 Operating Cost Summary

The operating costs are summarised in Table 21.4 and Table 21.6. The cash costs denominated in US$/oz Payable 3PE+Au show that Phase 2 has a slightly lower cash cost for the Phases.

Table 21.4 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**
Nickel Credits	-367	-411	-397
Copper Credits	-81	-89	-86
Total Cash Costs After Credits	**367**	**341**	**371**

Table 21.5 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: Realization Costs			
Transport	334	628	887
Refining Charges	4,207	7,540	10,577
Government Royalty	938	1,628	2,261
Total Realization 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 & 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 21.6 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: Realization 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 Realization 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**

21.4 Mining Capital Costs

This section describes the parameters, basis, and exclusions for capital estimates for mining the Platreef resource. The overall cost estimate for the 4 Mtpa production option was compiled to include pre-production capital, sustaining capital, and mine operating costs. The capital costs are estimated at a conceptual level of accuracy. The unit costs are based on the most recent cost information from similar projects and adjusted where required to fit the mine plan. A summary is provided of pre-production, sustaining capital, and total capital costs for the 4 Mtpa option and is shown in Table 21.7.

Table 21.7 Overall Cost Summary for 4 Mtpa Option

Description	4 Mtpa
Total Mined (Mt)	116.64
Pre-production Capital Costs (US$M)	735.22
Sustaining Capital Costs (US$M)	678.96
Contingency on Capital (US$M)	344.40
Total Capital Cost (US$M)	1,758.58
Mine Operating Costs (US$M)	3,753.70
Total Life-of-Mine Expenditures (US$M)	**5,512.28**
Average Cost Per tonne (US$/t)	**47.26**

Overall cost estimates for the 8 Mtpa and 12 Mtpa options were developed by factoring the costs for the 4 Mtpa option then adding the costs of additional infrastructure such as shafts and hoisting facilities. Overall costs are summarized in Table 21.8.

Table 21.8 Overall Cost Summary for 8 Mtpa and 12 Mtpa Options

Description	8 Mtpa	12 Mtpa
Total PEA Inventory Extracted (Mt)	219	310
Pre-production Capital Costs (US$M)	862	908
Sustaining Capital Costs (US$M)	1,524	2,347
Contingency on Capital (US$M)	613	844
Total Capital Cost (US$M)	2,999	4,098
Mine Operating Costs (US$M)	7,109	10,874
Total Life-of-Mine Expenditures (US$M)	**10,108**	**14,972**
Average Cost Per tonne (US$/t)	**46.10**	**48.31**

Note: Any inaccuracy in summation is due to rounding.

All costs are based on 2014 $US. Summaries of the annual cost expenditures for each production rate option are provided in Figure 21.1, Figure 21.2, and Figure 21.3.

Figure 21.1 Life-of-Mine Expenditure Schedule – Phase 1 – 4 Mtpa

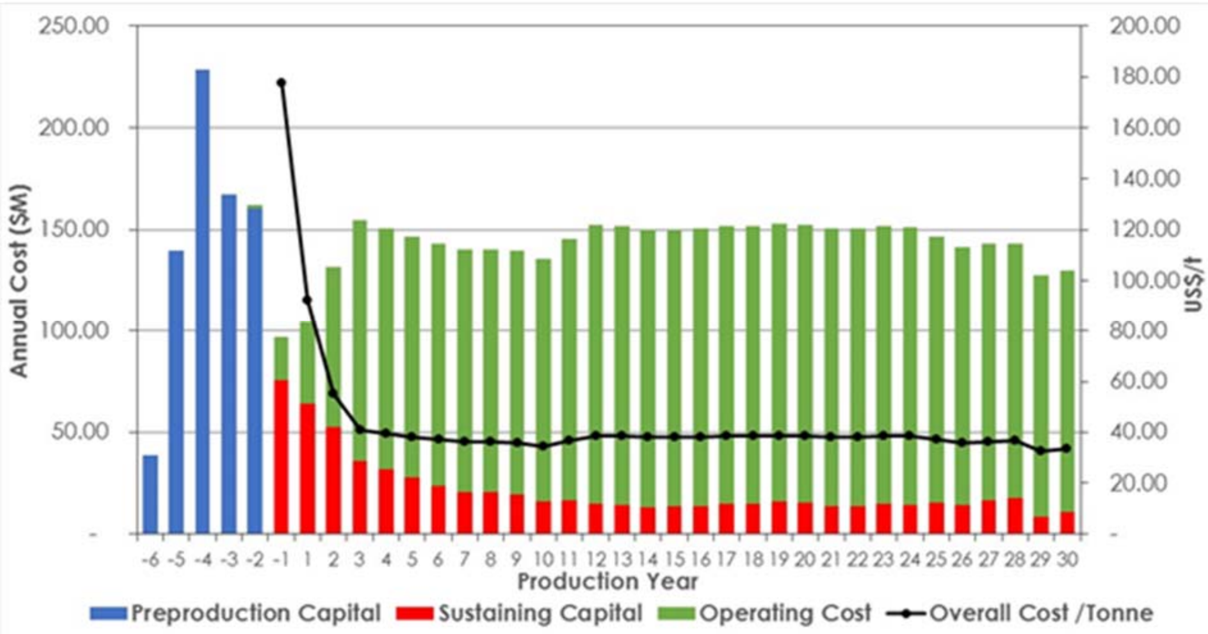


Figure 21.2 Life-of-Mine Expenditure Schedule – Phase 2 – 8 Mtpa

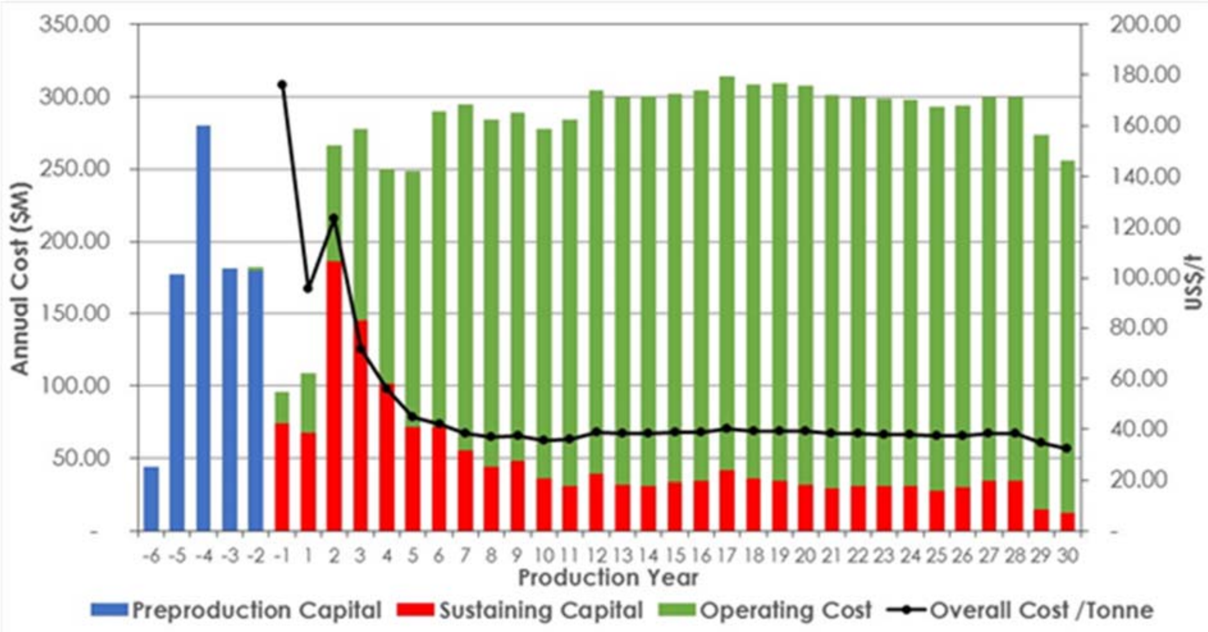
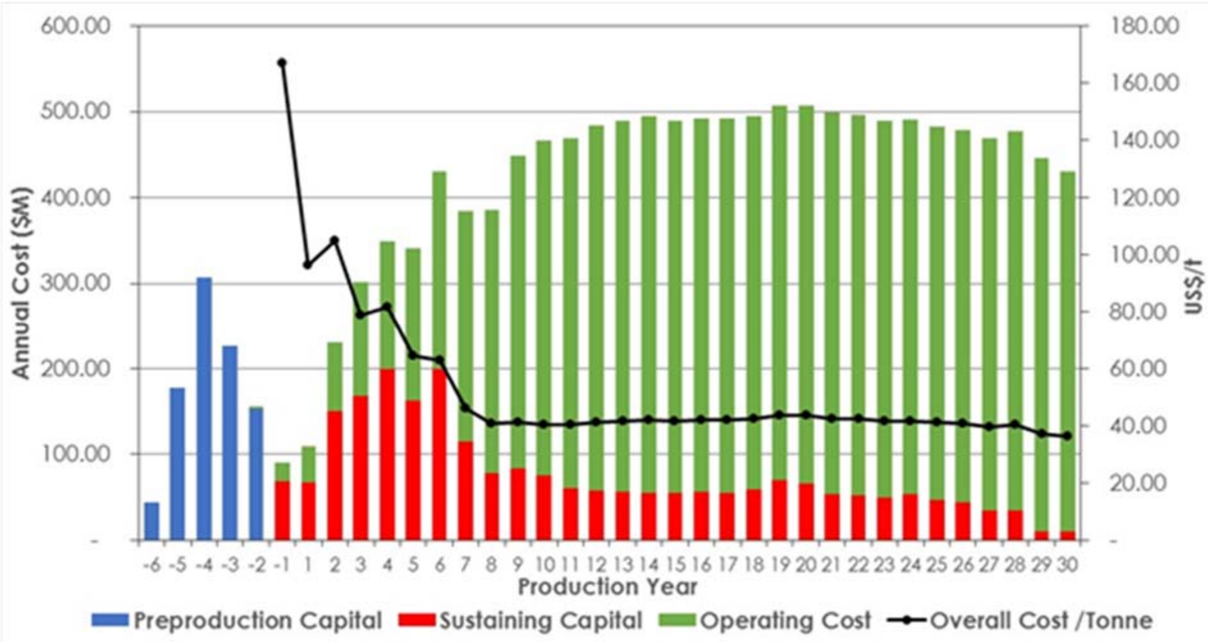


Figure 21.3 Life-of-Mine Expenditure Schedule – Phase 3 – 12 Mtpa



21.4.1 Capital Costs

The total capital cost for each option 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.

Sustaining capital is comprised of ongoing capital development and construction as well as additional mobile equipment purchases during production ramp up and mobile equipment rebuild and replacement costs throughout the mine life.

A capital contingency of 10% is included in the estimate for mobile equipment related items. A contingency of 30% is applied to all other capital expenditures.

The pre-production and sustaining capital costs for the 4 Mtpa, 8 Mtpa, and 12 Mtpa phases are summarized in Table 21.9, Table 21.10, and Table 21.11.

Table 21.9 Capital Cost Summary – Phase 1 – 4 Mtpa

Description	Pre-production (Millions $US)	Sustaining (Millions $US)	Total (Millions $US)
Contractor Costs			
Shaft Sinking	296.4	–	296.4
Direct Costs*	243.5	–	243.5
Indirect Costs*	10.8	–	10.8
Margins / Insurances / Bonding*	44.5	–	44.5
Subtotal Contractor Costs	**595.2**	**–**	**595.2**
Owner Costs			
Surface Capital Infrastructure	–	–	–
Hoists and Conveyances	19.9	–	19.9
Fixed Equipment	23.9	–	23.9
Mobile Equipment (Initial Purchase)	15.9	–	15.9
Mobile Equipment Purchase, Rebuild, and Replacement	–	312.3	312.3
EPCM	49.1	–	49.1
Owner's Team	19.6	–	19.6
Builder's Risk Insurance	2.0	–	2.0
Water Supply	0.9	–	0.9
Waste Management	0.1	–	0.1
Power During Pre-production	8.7	–	8.7
Lateral / Vertical Development and Construction	–	308.7	308.7
Miscellaneous Construction – Undefined Allowance	–	58.0	58.0
Subtotal Owner's Costs	**140.1**	**679.0**	**819.0**
Total Contractor and Owner's Costs	**735.2**	**679.0**	**1,414.2**
Contingency	220.6	123.8	344.4
Total Capital Cost	**955.8**	**802.8**	**1,758.6**

*Shaft sinking cost includes sinking contractor's direct and indirect costs, margins, insurances, and bonding.

Note: Any inaccuracy in summation is due to rounding.

Table 21.10 Capital Cost Summary – Phase 2 – 8 Mtpa – Base Case

Description	Pre-production (Millions $US)	Sustaining (Millions $US)	Total (Millions $US)
Contractor Costs			
Shaft Sinking	372.6	145.7	518.3
Direct Costs*	260.1	56.1	316.2
Indirect Costs*	10.8	16.8	27.6
Margins / Insurances / Bonding*	47.4	12.8	60.2
Subtotal Contractor Costs	**690.8**	**231.4**	**922.2**
Owner Costs			
Surface Capital Infrastructure	–	1.5	1.5
Hoists and Conveyances	30.2	–	30.2
Fixed Equipment	23.9	12.3	36.2
Mobile Equipment (Initial Purchase)	24.7	-	24.7
Mobile Equipment Purchase, Rebuild, and Replacement	–	516.5	516.5
EPCM	57.7	–	57.7
Owner's Team	23.1	–	23.1
Builder's Risk Insurance	2.3	–	2.3
Water Supply	0.9	–	0.9
Waste Management	0.1	–	0.1
Power During Pre-production	8.7	–	8.7
Lateral / Vertical Development and Construction	–	678.3	678.3
Miscellaneous Construction – Undefined Allowance	–	84.0	84.0
Subtotal Owner's Costs	**171.6**	**1,292.6**	**1,464.2**
Total Contractor and Owner's Costs	**862.5**	**1,524.0**	**2,386.5**
Contingency	258.7	353.9	612.6
Total Capital Cost	**1,121.2**	**1,877.9**	**2,999.1**

*Shaft sinking cost includes sinking contractor's direct and indirect costs, margins, insurances, and bonding.
Note: Any inaccuracy in summation is due to rounding.

Table 21.11 Capital Cost Summary – Phase 3 – 12 Mtpa

Description	Pre-production (Millions $US)	Sustaining (Millions $US)	Total (Millions $US)
Contractor Costs			
Shaft Sinking	413.4	317.9	731.3
Direct Costs*	260.1	139.6	399.7
Indirect Costs*	10.8	41.9	52.7
Margins / Insurances / Bonding*	47.4	31.8	79.2
Subtotal Contractor Costs	**731.7**	**531.2**	**1,262.8**
Owner Costs			
Surface Capital Infrastructure	–	2.8	2.8
Hoists and Conveyances	30.2	19.9	50.1
Fixed Equipment	23.9	22.3	46.1
Mobile Equipment (Initial Purchase)	24.7	–	24.7
Mobile Equipment Purchase, Rebuild, and Replacement	–	661.2	661.2
EPCM	60.8	–	60.8
Owner's Team	24.3	–	24.3
Builder's Risk Insurance	2.4	–	2.4
Water Supply	0.9	–	0.9
Waste Management	0.1	–	0.1
Power During Pre-production	8.7	–	8.7
Lateral / Vertical Development and Construction	–	1,025.3	1,025.3
Miscellaneous Construction – Undefined Allowance	–	84.0	84.0
Subtotal Owner's Costs	**176.0**	**1,815.5**	**1,991.5**
Total Contractor and Owner's Costs	**907.7**	**2,346.7**	**3,254.3**
Contingency	272.3	571.8	844.1
Total Capital Cost	**1,180.0**	**2,918.4**	**4,098.4**

*Shaft sinking cost includes sinking contractor's direct and indirect costs, margins, insurances, and bonding.
Note: Any inaccuracy in summation is due to rounding.

The capital costs included in this evaluation are discussed below.

21.4.1.1 Contractor Direct Costs

Contractor costs include cost elements listed below. Key aspects of each cost component are also noted.

- Labor
  - Labor includes a combination of direct and indirect labor required to complete the specified task. Labor can include hourly and staff personnel depending on the type of activity.
- Permanent Materials
  - Permanent materials includes 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 specialized 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 Labor.
- Mechanical and Electrical Maintenance Labor.
- 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 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 labor, 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."

Builder's Risk Insurance

Builder's risk insurance is assessed at 0.3% against the "Total Contractors Cost + Surface Infrastructure Cost + Permanent Fixed Capital Equipment Costs + Permanent Mobile Capital Equipment Costs."

Electric Power Consumption

Power costs associated with pre-production mine development and construction are assessed at a calculated rate of $0.075 per kWh.

Water Supply

Water supply costs associated with pre-production mine development and construction are assessed at a rate of $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.

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%.

Quantity Development

The design engineering team developed all quantities included in, associated with, or related to the scope of work based on conceptual mine designs.

A mine equipment list was developed based on the infrastructure design and production productivity. A major portion of the equipment prices is based on industry averages and historical data from similar projects.

21.5 Plant Capital Cost Estimate

21.5.1 Summary

The capital cost estimate in this section covers the process plant and plant infrastructure as per the selected flowsheet illustrated in Section 17 of this report, as well as additional general infrastructure requirements.

The capital cost represented in this section is based on a modular approach using two milling-flotation modules with a capacity of 2 Mtpa to achieve 4 Mtpa normal throughput. Future expansion cases to 8 Mtpa and 12 Mtpa will be modular to the 4 Mtpa processing plant and provision has been made for this in the overall block plan.

The plant capital cost estimate is summarised in Table 21.12.

The capital expansion scenarios for the 8 and 12 Mtpa process plants were factored from the 4 Mtpa base case. The following assumptions were made:

- Process plant cost for 4 Mtpa was used for the 8 Mtpa and 12 Mtpa expansion estimates.
- 50% of process plant infrastructure cost was used for 8 Mtpa and 12 Mtpa.
- To allow for general infrastructure upgrades, 50% of bulk permanent power and bulk water costs as per the November 2011 estimates by AMEC were used.
- Other general infrastructure such as earthworks, services, external roads, fencing, waste management were not increased for the expansion cases.
- Sustaining capital was calculated at 2.5% of the total capital cost. There are no sustaining capital costs estimated in Year 1 and the final four years of production in each Phase.
- Sustaining capital on the tailings storage facility was included in the TSF estimate.

Refer to Table 21.13 below for a summary of the expansion scenarios.

Table 21.12 Plant Capital Cost Summary for 4 Mtpa

Sustaining Capital Factor:	2.5%	
Base Date:	Dec-13	
Rate of exchange:	10.00	ZAR = 1 USD

	Pre-Production Capital		Sustaining Capital	Total Capital
	R ('000)	$ ('000)	$ ('000)	$ ('000)
PROCESS PLANT & PLANT INFRASTRUCTURE				
Process Plant Supply & Install	1,291,316	129,132	80,707	209,839
Process Plant Infrastructure Supply & Install	222,307	22,231	–	22,231
Tailings lines & Corridor & Return Water	75,000	7,500	–	7,500
Silos	24,749	2,475	–	2,475
Preliminary and General Cost- Plant (P&G)	362,204	36,220	–	36,220
Preliminary and General Cost- Infrastructure (P&G)	96,741	9,674	–	9,674
Subtotal Process Plant & Inplant Infrastructure	***2,072,317***	***207,232***	***80,707***	***287,939***
MISCELLANEOUS				
Spares	26,925	2,692	–	2,692
Transport	31,903	3,190	–	3,190
Consumables & First Fills	59,339	5,934	–	5,934
Mobile Equipment	48,000	4,800	–	4,800
Subtotal Miscellaneous	***166,167***	***16,617***	**–**	***16,617***
TOTAL PROCESS PLANT	**2,238,484**	**223,848**	**80,707**	**304,556**
GENERAL INFRASTRUCTURE				
Bulk Permanent Power	250,000	25,000	–	25,000
Construction Power	23,800	2,380	–	2,380
Bulk Water	569,000	56,900	–	56,900
Earthworks	79,500	7,950	–	7,950
Services	25,000	2,500	–	2,500
Electrical	18,000	1,800	–	1,800
Control	5,000	500	–	500
Substations	24,500	2,450	–	2,450
Buildings	42,600	4,260	–	4,260
Bulk Water Supply	7,500	750	–	750
External Roads	50,580	5,058	–	5,058
Fuel & Lubrication	13,000	1,300	–	1,300
Fire Protection	3,000	300	–	300
Pipe & Cable Racking	5,000	500	–	500
Fencing & Access Control	30,000	3,000	–	3,000
Waste Management	15,500	1,550	–	1,550
TOTAL GENERAL INFRASTRUCTURE	***1,161,980***	***116,198***	–	***116,198***
TOTAL DIRECT COSTS	**3,400,464**	**340,046**	**80,707**	**420,754**
INDIRECT COSTS				
Temporary Project Facilities @ 2% of DC	**68,009**	6,801	–	6,801
Project Services & Owners Team @ 3% of DC	**102,014**	10,201	–	10,201
EPCM @ 12% of DC	**408,056**	40,806	–	40,806
TOTAL INDIRECT COSTS	**578,079**	**57,808**	–	**57,808**
TOTAL PROJECT (EXCL. CONTINGENCY)	**3,978,543**	**397,854**	**80,707**	**478,562**
Growth, Contingency, Risk (30%)	**1,193,563**	**119,356**	**24,212**	**143,568**
PROJECT GRAND TOTAL	**5,172,106**	**517,211**	**104,919**	**622,130**

Table 21.13 Expansion Scenarios

Capital Expenditure	4 Mtpa $ ('000)	8Mtpa $ ('000)	12 Mtpa $ ('000)
Pre-production Capital	517,211	371,194	371,194
Sustaining Capital	104,919	99,254	86,663
Project Grand Total	**622,130**	**470,448**	**457,857**

21.5.2 Basis Of Estimate

The base date for this capital cost estimate is January 2014. South African Rands (R) was used as a base currency for the estimate, with an exchange rate of R10 = $1.00. A project contingency, risk and estimating inaccuracy of 30% on the total project direct and indirect cost has been included. Escalation cost has been excluded and the estimate which is stated in real money terms.

The estimating approach followed was to price all major process/mechanical equipment on the mechanical equipment list by means of budget quotations. The equipment supply costs were used as the basis for estimating costs for the balance of other engineering disciplines i.e. earthworks, civil, piping, steelwork, plate work, electrical, control and instrumentation. Ratios and factors were applied to the respective engineering disciplines based on benchmarking against similar platinum projects executed by DRA. Platreef Project specific adjustments have been taken into consideration for battery limits, exclusions, founding and ground conditions and the proposed location of the project.

21.5.3 Input Documents

The following were used as the main inputs to the capital estimate for the plant:

- DRA mechanical equipment list based on the updated process design and selected flowsheet.
- Area Infrastructure for bulk water supply, external power supply (Eskom 70 MVA) and construction power (Eskom 5 MVA) were copied form the 2011 AMEC Scoping Study report.
- Additional general area infrastructure outside the process plant area has been identified using the product work breakdown structure and included in the capital estimate.

21.5.4 Assumptions

Preliminary geotechnical information has indicated poor ground conditions may exist, therefore it is assumed that piling will be required at the ROM silos, crushing and screening buildings, milling building, flotation plant, filter plant, and workshops.

Capital for waste rock disposal earthmoving equipment has been allowed for under the miscellaneous section of the Process Plant Direct Cost.

Ore stockpile management has been allowed for in the operating cost and is assumed to be done by trackless loading and hauling operation on surface. No allowance is made for automated stacking and reclaiming stockpile management equipment.

It is further assumed that the loading and hauling operations will be subcontracted to others and therefore no allowance was made in the capital estimate for purchasing of loaders and haul trucks.

21.5.5 Allowance for Additional General Infrastructure

Additional General Infrastructure items have been added to the capital estimate compared to the bulk water and power allowances only included in the AMEC Scoping Study Report. The additional infrastructure items have been included below the bulk water and power line items under the General Infrastructure section part of the Process Plant estimate.

21.5.6 Summary of Concentrator Plant Capital Cost Estimate

21.5.6.1 Earthworks & Infrastructure

The following were covered:

- Bulk earthworks and plant terracing, inclusive of the front end crushing area.
- Clear and Grub for the tailings line.
- Raw water, pollution control and run off dams.
- Plant roads & Maintenance access roads.
- Security Fencing.
- Offices, workshops, stores and laboratory.

Further to this the removal of topsoil to the depth of 3 m and the replacement of more suitable material from a borrow pit within the Platreef lease area has been allowed for.

21.5.6.2 Civil Works

Civil works (excluding piling) have been factorised from the mechanical equipment costs. Architectural buildings (such as offices) and other buildings (such as workshops and stores) have been included under civil works.

Preliminary geotechnical information has indicated poor ground conditions may exist, therefore it is assumed that piling will be required at the ROM silos, crushing and screening buildings, milling building, flotation plant, filter plant and workshops.

Piling was estimated based on database information from a previous concentrator project. An average depth of pile was determined from a geological test pits that was drilled in the area of the process plant. A ratio was applied to allow for the difference in average pile depths between the Platreef and the benchmark concentrator and thereafter escalation was applied.

No consideration has been given to potential ground water.

21.5.6.3 Structural Steel

All structural steel supply and installation has been factorised from mechanical equipment costs.

21.5.6.4 Mechanical Equipment & Conveyors

A mechanical equipment list has been developed based on current flow sheets, and used as input to the estimate. Each piece of equipment was priced by means of budget quotations obtained from reputable equipment suppliers. Mechanical installation costs have been factorised from the mechanical equipment costs.

Conveyors have been estimated in length, height, and belt width by means of a high level conceptual layout; and thereafter factorised in terms of an estimated rate per meter obtained from database information from previous projects. The rate for the conveyors includes the mechanical equipment associated directly with the conveyors.

21.5.6.5 Piping and Valves

Piping supply and installation have been factorised from the mechanical equipment cost. This cost covers the process plant piping, utilities, dust and fire suppression, conveyors and crushing area. The cost for the tailings and return water line has not been included here and is covered in process plant infrastructure.

21.5.6.6 Electrical and Instrumentation

Electrical and instrumentation supply and installation costs have been factorised from the mechanical equipment cost. This allowance includes cabling, racking, termination, and testing.

21.5.6.7 Installation and Erection

Installation and erection cost has been factorised from the mechanical equipment costs for each discipline, i.e. civil, structural, mechanical, piping, electrical and instrumentation.

21.5.6.8 Sustaining Capital

Sustaining capital was calculated at 2.5% of the total capital costs . There are no sustaining capital costs estimated in Year 1 and the final four years of production in each Phase.

21.5.6.9 Transport / Freight / Insurance

Transport, freight and insurance have been factorised form the mechanical equipment costs. The applied factor is based on database information from previous platinum projects and only includes a limited allowance for ocean freight.

21.5.6.10 Bulk Water

The esimates for bulk water capital were based on production sizing requirements.

21.5.6.11 Bulk Power

A capital allowance to be paid to Eskom has been allowed for in the estmate.

21.5.6.12 Consumables and First Fills

First fill of oils and lubricants, reagents and grinding media has been allowed.

21.5.6.13 Spares

An allowance for commissioning, strategic and a limited amount of operating spares have been factorised from the mechanical equipment cost.

21.5.6.14 Commissioning

Commissioning of mechanical and/or process equipment has been allowed for in the mechanical supply cost and EPCM cost. This covers vendor representation on site during assembly and commissioning, including hot commissioning.

21.5.6.15 EPCM Cost

EPCM cost has been estimated at 12% of the total direct project cost for the process plant and general infrastructure sections combined. These costs cover the project management, engineering, design, procurement, construction management and commissioning management directly associated with the implementation of the project.

21.5.6.16 Temporary Project Facilities Costs

An allowance for temporary project facilities have been made by a factorisation based on 2% of the total direct project cost. This allowance includes:

- Construction Power, water, laydowns, offices and stores.
- Construction vehicles.
- PPE & Safety equipment.
- Temporary fuel and waste systems.
- Medical facilities.
- Owners construction facilities.

21.5.6.17 Project Services & Owners Team Costs

Project Services have been factored as 3% of the total direct project cost and include:

- Project Consulting services such as material flow studies, social and environmental, QS services, water studies, ventilation consultants, surface geotechnical investigations etc.
- Owners team.
- Waste control.
- Training and development services.
- Site access and security.
- Construction power and water usage.
- Risk management services.
- QC and expediting.

21.6 TSF Capital and Closure Cost Estimates

21.6.1 Introduction

The estimate base date has been defined as February 2014. All costs exclude VAT. The base currency for the Platreef Project is in United States Dollars ($). All costs were estimated in South African Rand (R) and converted to $ at the following rate: R10=$1.00.

21.6.2 TSF Capital Cost

The capital cost for the pre-deposition civil works associated with the tailings storage facilities can be summarised as follows:

Table 21.14 Capital Cost TSF's

	Phase 1 US$M	Phase 2 US$M	Phase 3 US$M	Basis for Estimate
Rietfontein TSF	39.1	0	39.1	Benchmarking against similar projects and where necessary on first level quantification and market related rates +/- 30% accuracy level).
Bultongfontein TSF	0	46.0	46.0	Factored approach
Total Cost	**39.1**	**46.0**	**85.1**	

21.6.3 Closure Costs

A provisional allowance has been made for the final year of operation for closure. The applicable costs are summarised in the table below:

Table 21.15 Closure Cost TSF's

	Phase 1 US$M	Phase 2 US$M	Phase 3 US$M	Basis for Estimate
Rietfontein TSF	9.3	0	9.3	Benchmarking against similar projects and where necessary on first level quantification and market related rates +/- 30% accuracy level).
Bultongfontein TSF	0	10.8	10.8	Factored approach
Total Cost	**9.3**	**10.8**	**20.1**	

21.7 Mining Operating Costs

Unit operating costs were prepared for both longhole stoping and drift-and-fill mining methods, with paste backfill, and annual operating costs were generated based on the tonnes produced in each year. Total operating costs are summarized in Table 21.16, Table 21.17, and Table 21.18, for the 4 Mtpa, 8 Mtpa, and 12 Mtpa options, respectively.

Table 21.16 Operating Cost Summary – Phase 1 – 4 Mtpa

Description	Total Cost (US$M)	Unit Cost (US$/t)
Production Direct Costs	1,482.5	12.71
Backfill Costs	802.8	6.88
Incremental Haulage Costs	19.0	0.16
Indirect Operating Costs	691.3	5.93
Mine Air Cooling	116.6	1.00
Power Costs	269.6	2.31
Water Cost	72.3	0.62
Undefined Allowance	299.6	2.57
Total Operating Cost	**3,753.7**	**32.18**
(116.6 Mt Mined)		

Table 21.17 Operating Cost Summary – Phase 2 – 8 Mtpa

Description	Total Cost (US$M)	Unit Cost (US$/t)
Production Direct Costs	2,806.5	12.8
Backfill Costs	1,512.1	6.90
Incremental Haulage Costs	74.5	0.34
Indirect Operating Costs	1,317.9	6.01
Mine Air Cooling	219.3	1.00
Power Costs	501.8	2.29
Water Cost	105.9	0.48
Undefined Allowance	571.1	2.6
Total Operating Cost	**7,109.1**	**32.42**
(219.3 Mt Mined)		

Table 21.18 Operating Cost Summary – Phase 3 – 12 Mtpa

Description	Total Cost US$M	Unit Cost US$/t
Production Direct Costs	4,491.1	14.49
Backfill Costs	2,180.3	7.04
Incremental haulage Costs	124.0	0.40
Indirect Operating Costs	2,038.6	6.58
Mine Air Cooling	309.9	1.00
Power Costs	706.8	2.28
Water Cost	139.7	0.45
Undefined Allowance	883.4	2.85
Total Operating Cost	**10,873.8**	**35.09**
(309.9 Mt Mined)		

21.7.1.1 Operating Cost Estimate Scope Definition

These cost estimates are intended to cover all expenses required to operate the mine. The operating costs are subdivided into the following cost centers.

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. Also included are costs for development (drill/draw drift development and drill drift slashing) and stope access drift development. Allowances have also been made for mine air cooling costs.

Backfill Costs

Backfill costs consists of all costs associated with preparing and filling stopes, including cement costs and freight, backfill plant operating costs and operating labor, and delivery costs.

Incremental Haulage Costs

Direct production costs includes an allowance for truck haulage for an average distance of 1.0 km from passes located in the mining areas to the main passes near the hoisting shaft(s). An incremental allowance is applied to haulage from deeper and more remotely located areas where truck haulage distances exceed 1.0 km. The incremental haulage allowance assumes an average haulage distance of 2.0 km.

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.

Power Costs

Representative power costs were based on power loads developed from similar projects of similar size. A detailed breakdown for power was not prepared at this level of evaluation.

Water Costs

A preliminary estimate of water usage was prepared by Stantec in December 2012 for a separate mine plan with an annual production rate of 3.0 Mtpa. This estimate was factored for the current three phases. Unit costs for water were supplied by Ivanhoe to estimate annual costs.

Miscellaneous Construction – Undefined Allowance

An allowance has been included as 10% of the operating costs for miscellaneous underground excavations that may be required for efficient operation of the Platreef project.

Operating Cost – Undefined Allowance

An allowance has been included as 10% of the operating costs for miscellaneous costs that may not be accounted for elsewhere in the estimate.

21.8 Plant Operating Cost Estimate

21.8.1 Summary

The operating cost estimate is based on a metallurgical processing facility with a design throughput of 4 Mtpa. Expansion to 8 Mtpa and 12 Mtpa will be done with the addition of 4 Mtpa processing plant modules – and thus, the operational costs calculated for the 4 Mtpa processing plant can be used for the modular expansions, taking into consideration the required escalation factors to compensate for the planned execution time frames and economies of scale.

Table 21.19 below summarises the estimated annual operational costs for the Platreef Project concentrator plant.

Table 21.19 Plant Operational Cost Estimate

Description	Phase 1A 2 Mtpa	Phase 1B 4 Mtpa
Fixed Costs	**R**	**R**
Labour Cost	40 913 362	58 799 326
Stores & Maintenance	41 338 372	64 227 884
Concentrate Transport	6 917 647	13 835 294
Stockpile Reclamation	8 280 000	2 760 000
Assays	3 061 800	5 136 480
Fixed Cost Total	**100 511 182**	**144 758 984**
Variable Costs	**R**	**R**
Utilities	89 490 259	157 391 715
Consumables	62 973 024	125 946 047
Variable Cost Total	**152 463 282**	**283 337 762**
Total Annual Cost	**252 974 464**	**428 096 747**
Total Unit Cost (R/t)	**126.5**	**107.0**
Total Unit Cost ($/t) Excluding contingency	**12.65**	**10.70**

Refer to Figure 21.4 for an illustration of the plant operational cost estimate splits for each phase.

Figure 21.4 Operational Cost Splits

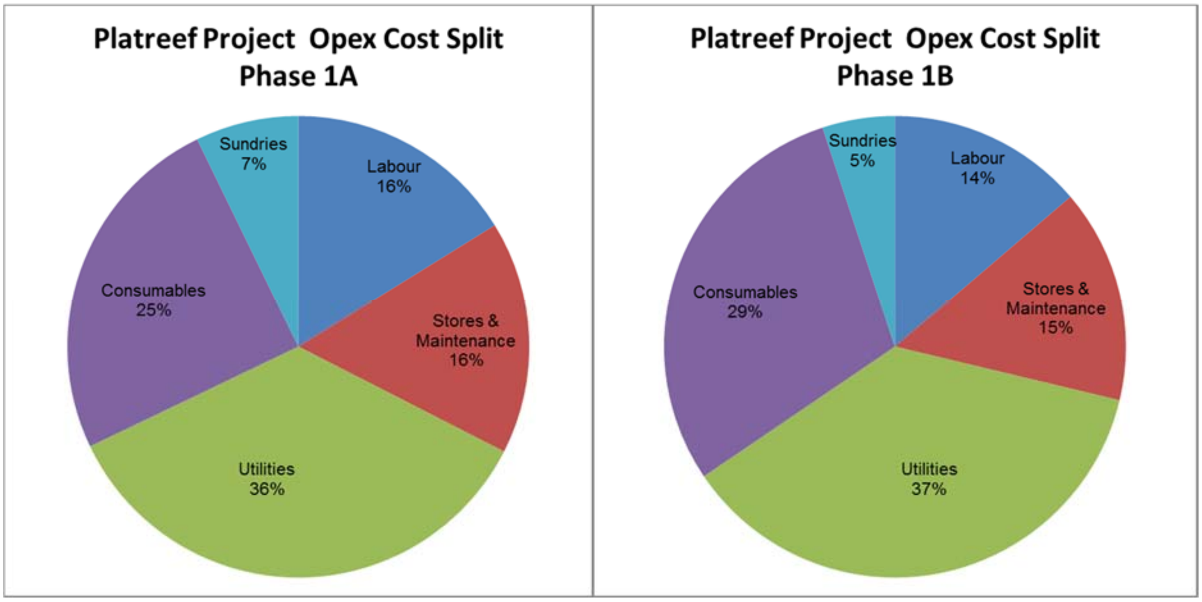


Table 21.20 Operating Cost Estimate for the Expansion Scenarios

Description	4 Mtpa	8 Mtpa	12 Mtpa
Fixed Costs	**R**	**R**	**R**
Labour Cost	58 799 326	74 712 260	91 398 906
Stores & Maintenance	64 227 884	128 455 769	192 683 653
Concentrate Transport	13 835 294	27 670 588	41 505 882
Stockpile Reclamation	2 760 000	5 520 000	8 280 000
Assays	5 136 480	6 123 600	9 185 400
Fixed Cost Total	**144 758 984**	**242 482 217**	**343 053 841**
Variable Costs	**R**	**R**	**R**
Utilities	157 391 715	314 713 709	472 035 704
Consumables	125 946 047	251 892 094	377 838 141
Variable Cost Total	**283 337 762**	**566 605 804**	**849 873 845**
Total Annual Cost	**428 096 747**	**809 088 021**	**1 192 927 686**
Total Unit Cost (R/t)	**107.0**	**101.1**	**99.4**
Total Unit Cost ($/t)	**10.4**	**9.8**	**9.6**

21.8.2 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 preliminary test work and engineering input and is expected to have an accuracy of ±25 to ±35%. The base currency of the Platreef Project is in United States Dollars ($). All costs have been estimated in Soith African Rand and converted to USD at the following rate: R 10 = $1.00. A base date of January 2014 was used.

21.8.3 Plant Operation Costs Inputs

21.8.3.1 Labour

The process plant labour costs are based on a staffing model and rates of similar platinum processing plants of a similar size and complexity.

21.8.3.2 Power

The power costs 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 R 0.46/kWh. Additional administration costs were added based on the Eskom tariffs booklet.

21.8.3.3 Water

The monthly water cost estimate is based on a rate of R 10.00/m^3, typical costs for the supply from Lebalelo water user association. Raw water consumption figures are based on the scoping level plant mass balance and expected losses (evaporation and interstitial losses) at the tailings storage facility.

21.8.3.4 Consumables

The consumption figures used in determining the operating cost estimate are based on test work data and supported by the scoping level plant mass balances. The reagent and grinding media costs used in the operating cost estimate is based on supply rates obtained from reputable reagent suppliers to the platinum industry in South Africa. These prices are consistent with the rates currently being paid by other platinum producers.

A single supply price was obtained for all reagents, except for oxalic acid and thiourea where three prices were obtained for each. The highest of the three supply prices was used in the estimate. The price difference 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 sources are reliable. The vendor also commented that they keep approximately two months' stock spare in South Africa.

21.8.3.5 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.

The crusher and mill liner cost were based on supply rates obtained from reputable vendors. Abrasion indexes were used to calculate the average life span of the crusher and mill liners.

21.8.3.6 Concentrate Transport

Concentrate transport was based on a transported distance of 300 km from site. A rate of $0.21 /t/km has been used.

21.8.3.7 Stockpile Management

It was assumed that a total of 30 000 tpm of stockpile material would be reclaimed in Phase 1A, and an average of 10,000 tpm would be reclaimed in Phase 1B. A rate of 23 R/t was used for the stockpile movement, based on rates obtained from earth moving contractors.

21.8.3.8 Assays

The assay cost was estimated based on the assumption that an an external company would be used for assaying.

21.9 TSF Operating Costs

The operation cost, benchmarking against similar projects and where necessary on first level quantification and market related rates, is approximately R1.75/t ($0.19/t).

21.10 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 base on the variable and fixed costs as summarised in Table 21.21. The general and administration costs estimate splits is presented in

Figure 21.5. The total general and administration costs of the 4 Mtpa, 8 Mtpa, and 12 Mtpa phases are presented in Table 21.22.

Table 21.21 General and Administration Cost Assumptions

Description	Unit	G&A Costs
G&A Variable	$/t	0.51
G&A Per Annum Fixed	$ 000pa	13,658

Figure 21.5 General and Administration Operating Costs

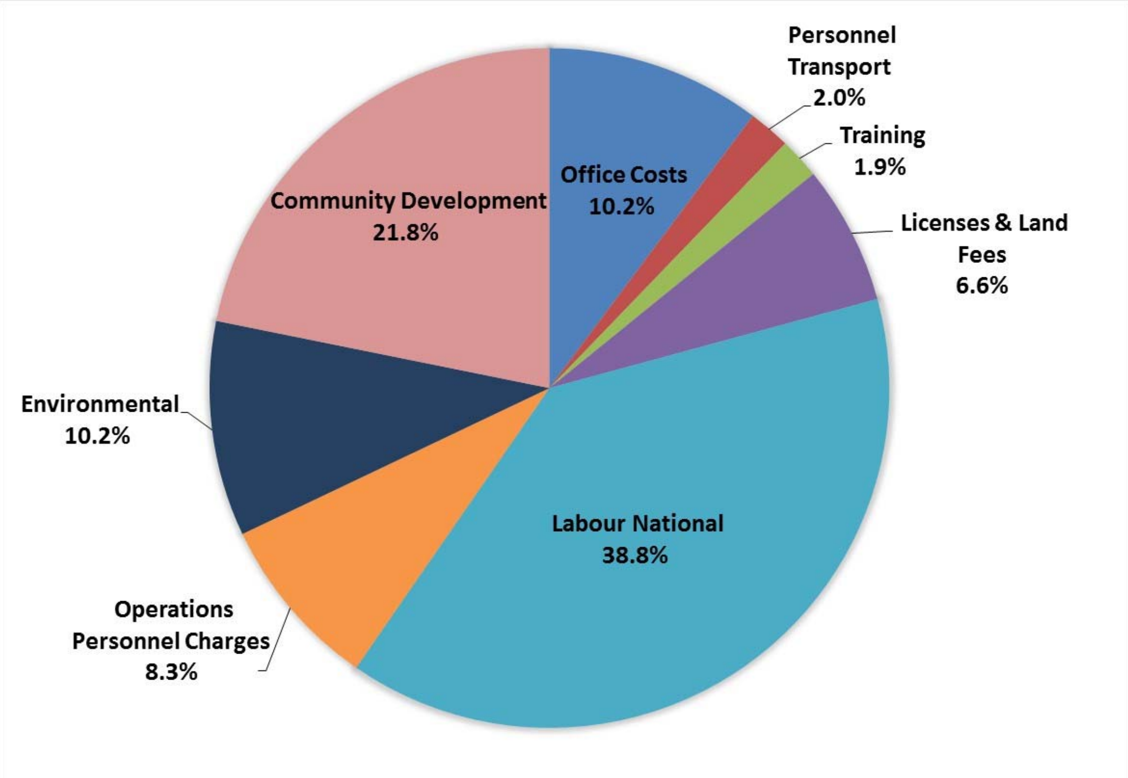


Table 21.22 General and Administration Costs

Description	Unit	4 Mtpa	8 Mtpa	12 Mtpa
G&A Per Annum Fixed	$ 000pa	13,658	13,658	13,658
G&A Variable	$ 000pa	2,035	4,069	6,104
Total	$ 000pa	15,693	17,727	19,762
Unit Cost (Life of Mine Average)	$/t	3.92	2.22	1.65

22 ECONOMIC ANALYSIS

22.1 Platreef Development Scenarios

Ivanhoe's development plan for Platreef considers three phases of underground mining and concentrator expansion. The three phases 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 is Phase 2, the 8 Mtpa concentrator case. The development scenarios and additional options for the Platreef Project are shown shown in Figure 22.1. 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.

Figure 22.1 Platreef Project Development Scenarios

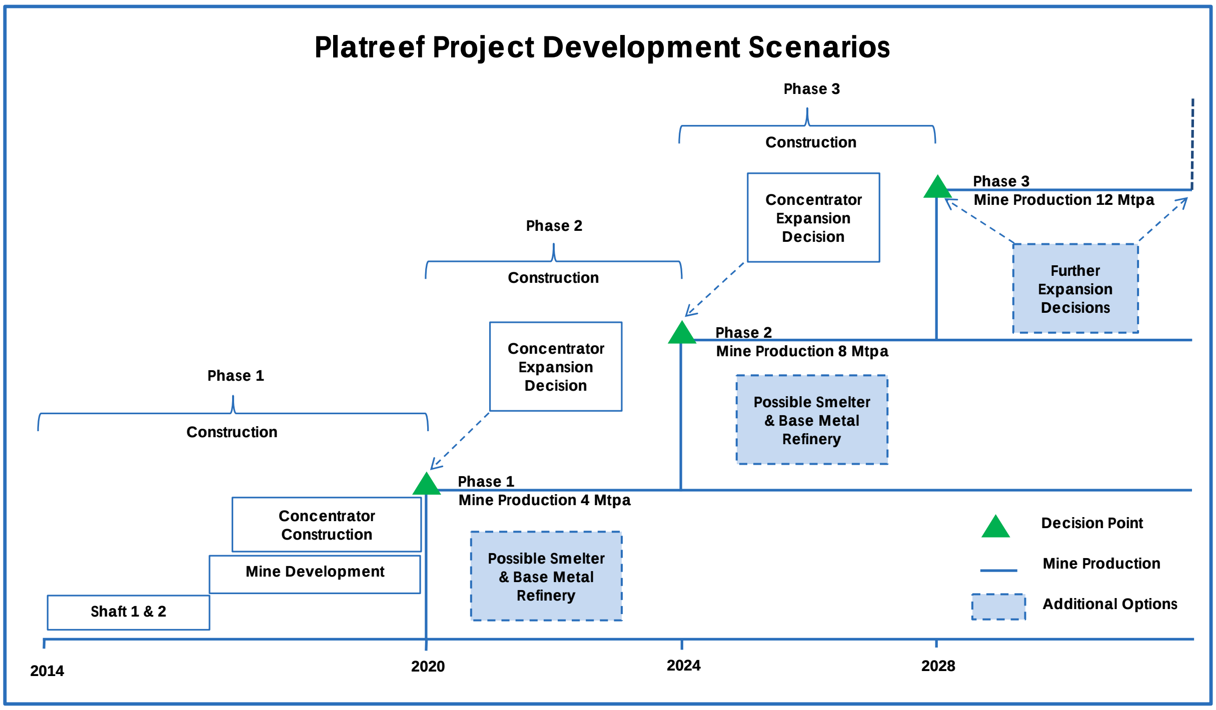


Figure by OreWin 2014.

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 in 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.

22.2 Summary of Financial Results

The economic analysis uses price assumptions of US$8.35/lb nickel, US$1,700/oz platinum, US$820/oz palladium, US$1,300/oz gold, US$3.00/lb for copper and US$1,700/oz rhodium. The prices are based on a review of consensus price forecasts from a 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.

Comparison of the results of the financial analysis for each case shows that there is a progressive increase in NPV for the three phases. The After Tax NPV at an 8% discount rate (NPV8), IRR and payback for each case are shown in Table 22.1 and Figure 22.2. 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 (Figure 22.3). As the phased expansions progress the payback period increases as capital is committed over a longer time horizon (Figure 22.4).

The undiscounted and discounted cash flows are plotted together in Figure 22.5. Cash flow increases with each concentrator expansion, which can also be seen on Figure 22.6 and Figure 22.7 (After Tax), and Figure 22.8 and Figure 22.9 (before tax).

Table 22.1 After Tax Financial Results

		Phase 1 4 Mtpa	Phase 2 8 Mtpa	Phase 3 12 Mtpa
Net Present Value (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 22.2 After Tax NPV @ 8% Discount Rate

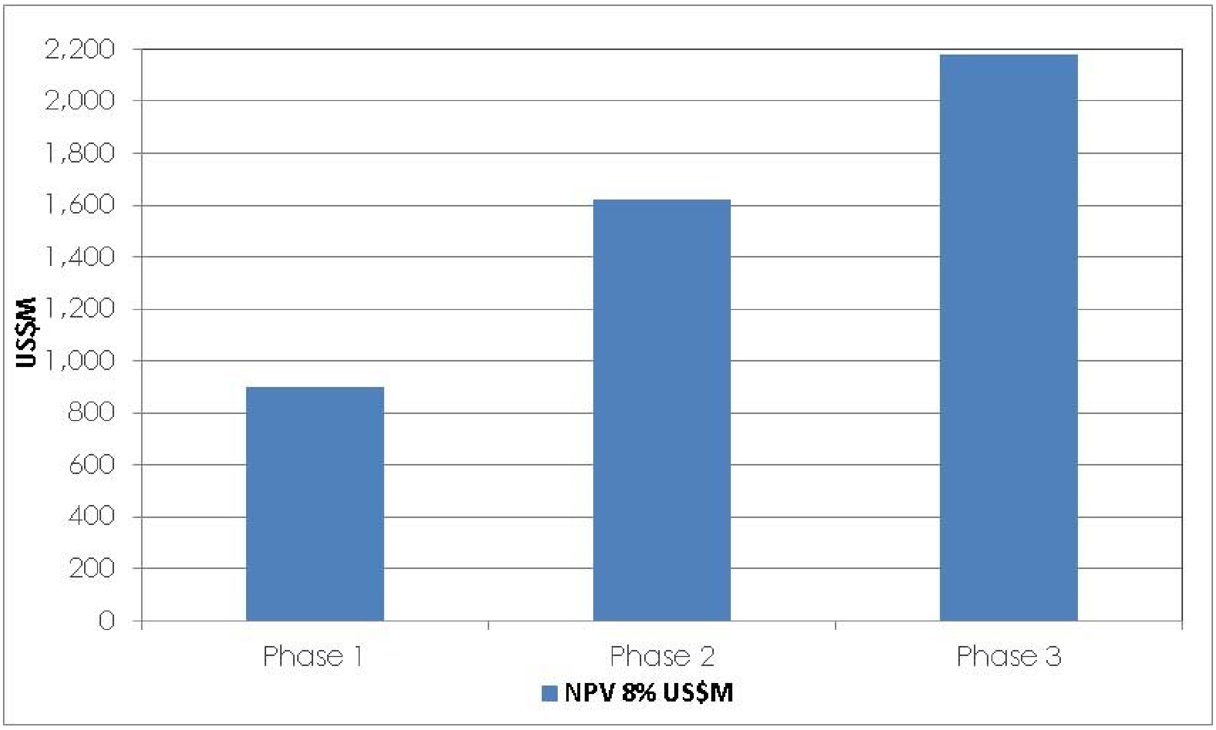


Figure by OreWin 2014.

Figure 22.3 After Tax IRR

Figure by OreWin 2014.

Figure 22.4 After Tax Project Payback Period

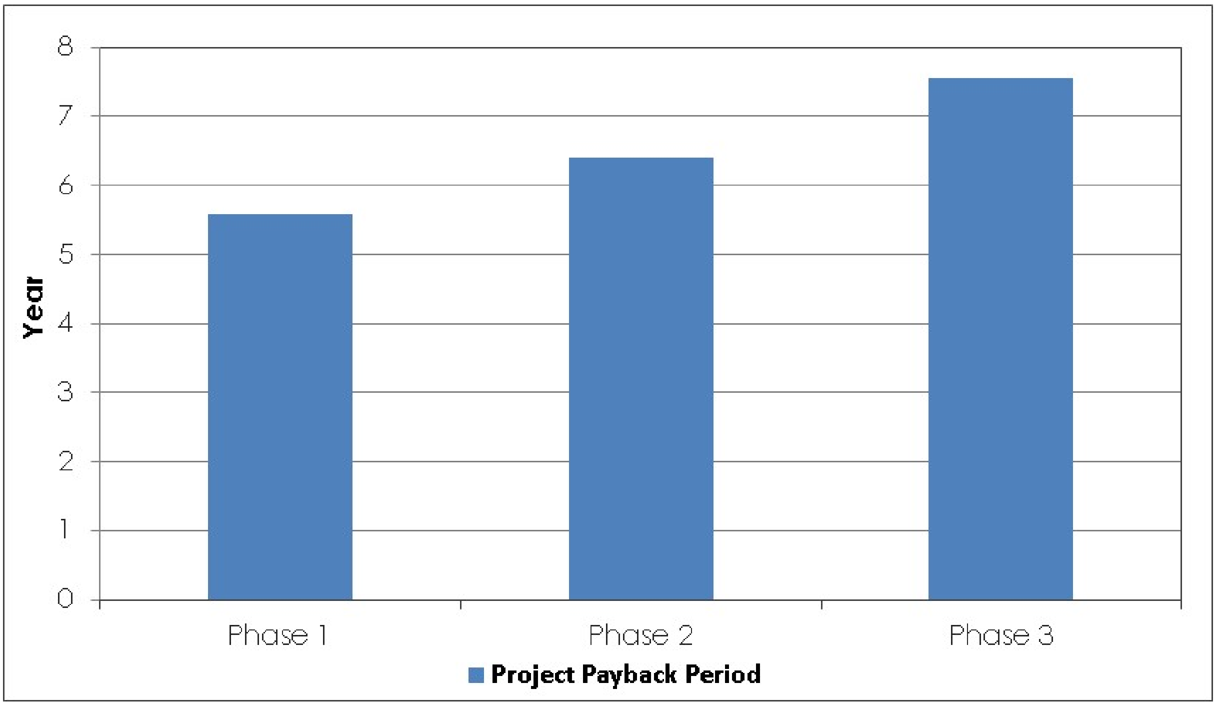


Figure by OreWin 2014.

Figure 22.5 After Tax Undiscounted and Discounted Cash Flows

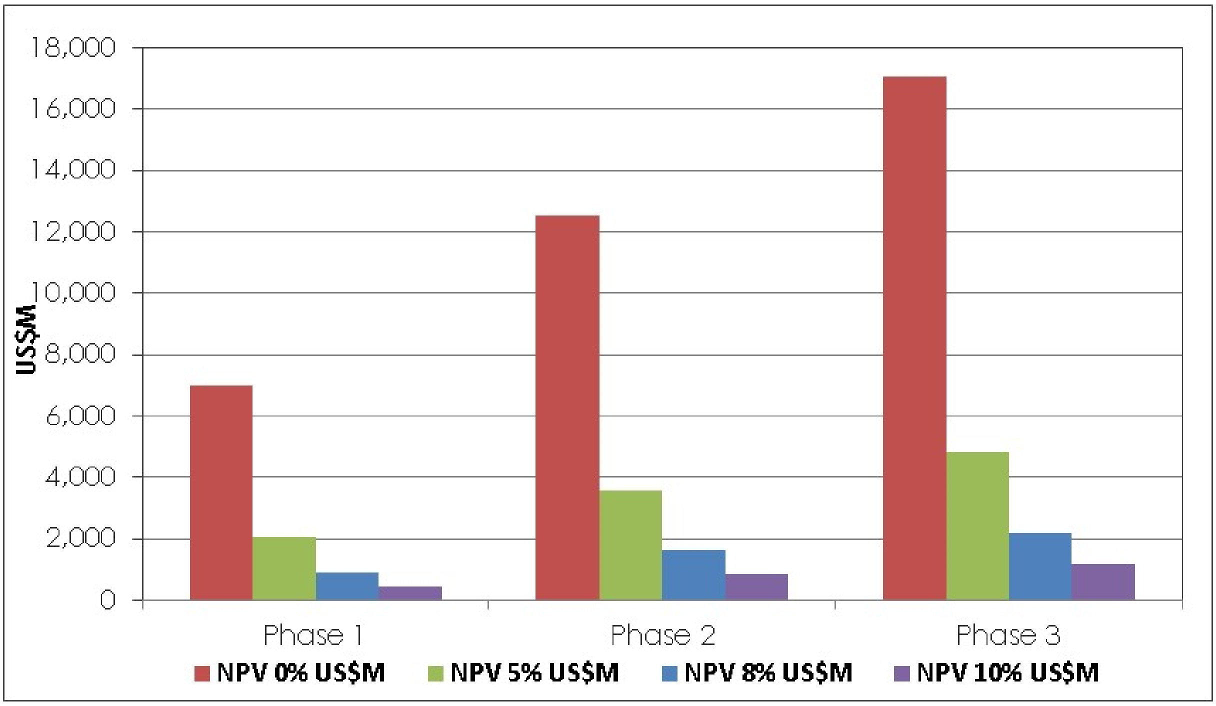


Figure by OreWin 2014.

Figure 22.6 Cumulative Cashflow After Tax

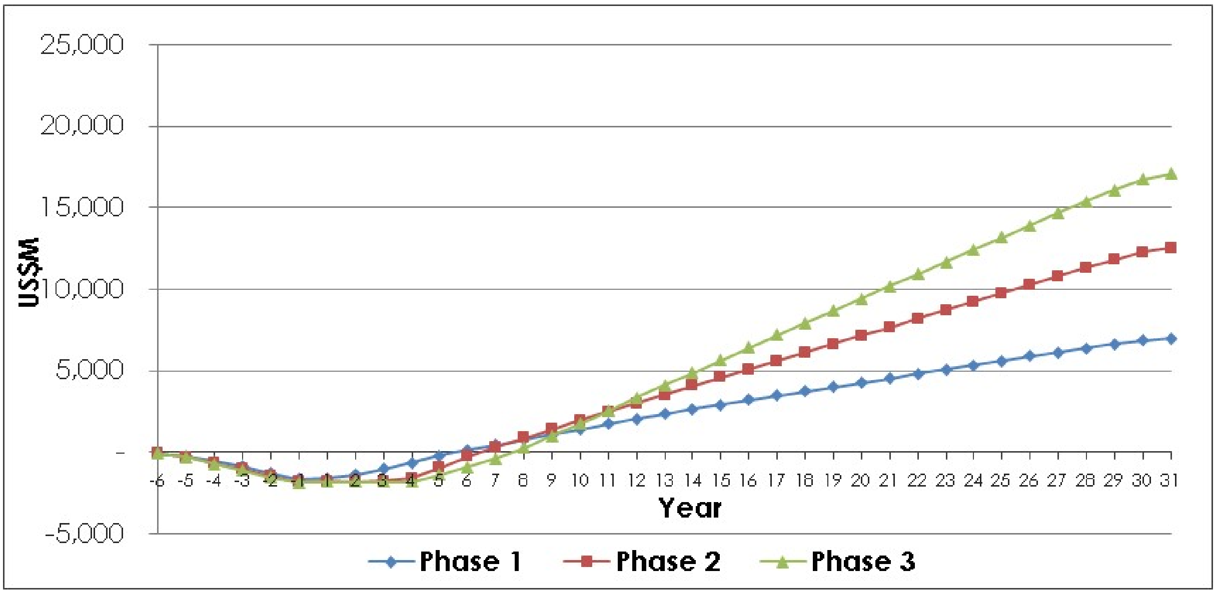


Figure by OreWin 2014.

Figure 22.7 Cumulative Cashflow After Tax (Initial Years)

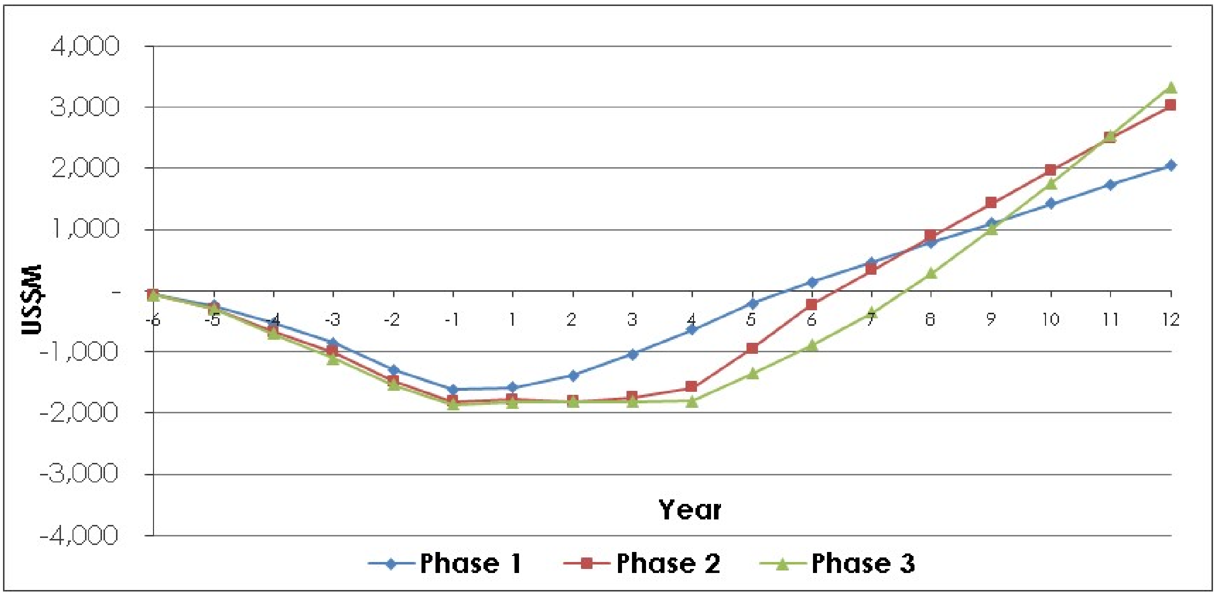


Figure by OreWin 2014.

Table 22.2 Before Tax Financial Results

		Phase 1 4 Mtpa	Phase 2 8 Mtpa	Phase 3 12 Mtpa
Net Present Value (US$M)	Undiscounted	9,656	17,346	23,667
	5%	2,955	5,179	6,918
	8%	**1,416**	**2,496**	**3,312**
	10%	814	1,474	1,963
	12%	410	801	1,087
	15%	31	183	300
IRR		**15.3%**	**16.4%**	**16.9%**
Project Payback Period	**(Years)**	**5.48**	**6.29**	**7.41**

Figure 22.8 Cumulative Cashflow Before Tax

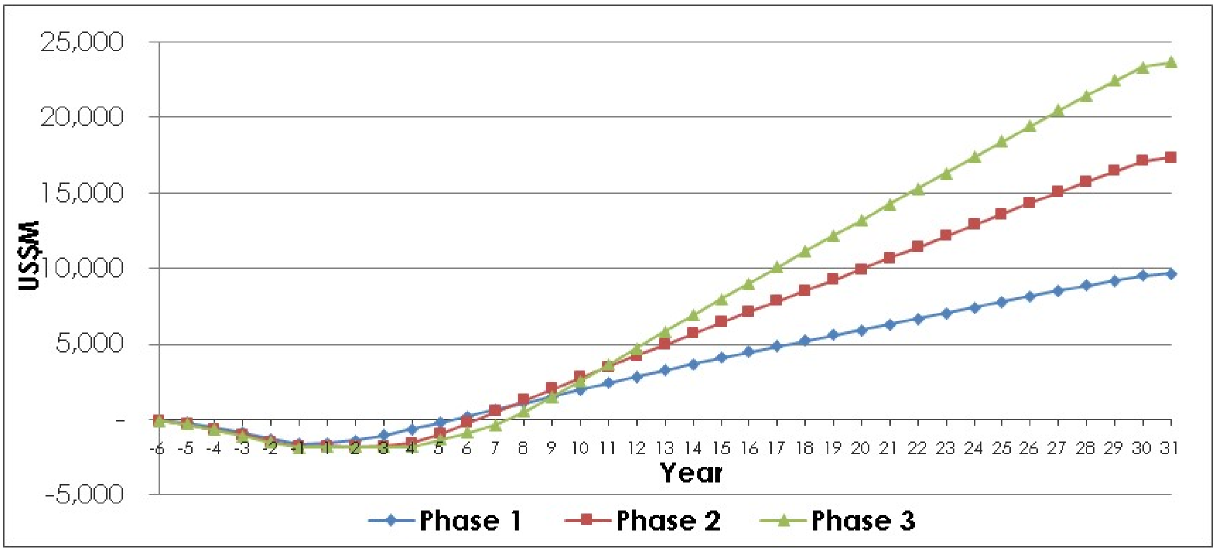


Figure by OreWin 2014.

Figure 22.9 Cumulative Cashflow Before Tax (Initial Years)

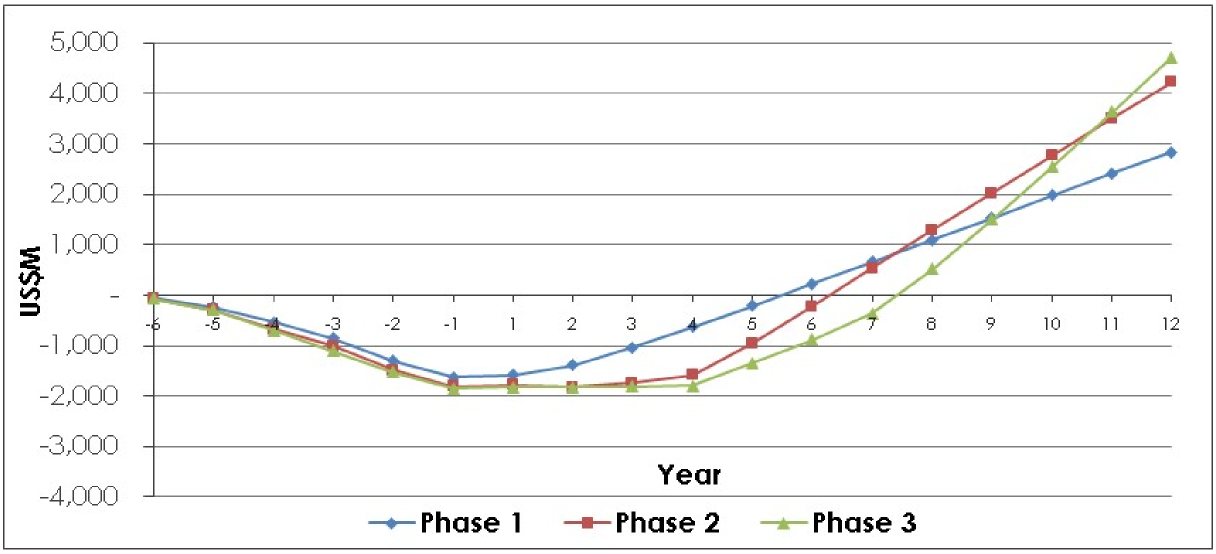


Figure by OreWin 2014.

22.3 Model Assumptions

22.3.1 Pricing and Discount Rate Assumptions

The Project level financial models begin on 1 January 2014. It is presented in 2014 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%.

The economic analysis of all seven scenarios uses price assumptions of US$8.35/lb nickel, US$1,700/oz platinum, US$820/oz palladium, US$1,300/oz gold, US$3.00/lb for copper and US$1,700/oz rhodium. The prices are based on a review of consensus price forecasts from a financial institutions and similar studies that have recently been published.

22.3.2 Realisation Costs

Realisation costs are described in Section 19.

22.3.3 Inflation

No allowance for inflation has been made in the analyses.

22.3.4 Republic of South Africa Fiscal Environment

The following taxes comprise the majority of taxes and fees payable to the government under Republic of South Africa legislation:

Corporate income tax 28%

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.

22.4 Management Fees

Ivanhoe shall earn a management fee, to be paid by the Project, equal to 3% of all items, directly or indirectly, incurred or accrued in relation to operating the Project other than capital costs. The NPV8 of the management fee is $37M (Phase 1), $62M (Phase 2) and $84M, respectively

22.5 Production Summary

The key production totals are compared for each of the cases in Table 22.3. The key average annual production results over the 30 year mine life are shown in Table 22.4.

Table 22.3 Production Summary

Production Summary				
		Phase 1	Phase 2	Phase 3
		4 Mtpa	8 Mtpa	12 Mtpa
Total Mined & Processed	**Mt**	**117**	**219**	**310**
Nickel	%	0.34	0.35	0.34
Platinum	g/t	1.84	1.70	1.71
Palladium	g/t	1.93	1.78	1.77
Copper	%	0.16	0.16	0.16
Gold	g/t	0.27	0.27	0.27
Rhodium	g/t	0.13	0.12	0.12
Recoveries				
Nickel Recovery	%	69.13	69.47	69.05
Platinum Recovery	%	88.21	87.15	87.24
Palladium Recovery	%	87.63	86.85	86.77
Copper Recovery	%	87.89	87.90	87.84
Gold Recovery	%	76.69	76.72	76.72
Rhodium Recovery	%	85.92	86.62	86.62
Concentrate Produced				
Concentrate	kt	4,665	8,771	12,396
Nickel	%	5.8	6.0	5.8
Platinum	g/t	40.5	37.0	37.3
Palladium	g/t	42.4	38.7	38.4
Copper	%	3.6	3.6	3.5
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
Metal Sold				
Nickel	Mlb	599	1,160	1,582
Platinum	koz	6,075	10,444	14,864
Palladium	koz	6,362	10,915	15,294
Copper	Mlb	368	695	957
Gold	koz	789	1,462	2,065
Rhodium	koz	413	742	1,050
3PE + Au	koz	13,638	23,564	33,273

Table 22.4 Life of Mine Average Production Summary

Life of Mine Average Production Summary				
		Phase 1	Phase 2	Phase 3
		4 Mtpa	8 Mtpa	12 Mtpa
Life of Mine	**Years**	**30**	**30**	**30**
Mined and Processed (Includes ramp up)	**Mtpa**	**3.9**	**7.3**	**10.3**
Nickel	%	0.34	0.35	0.34
Platinum	g/t	1.84	1.70	1.71
Palladium	g/t	1.93	1.78	1.77
Copper	%	0.16	0.16	0.16
Gold	g/t	0.27	0.27	0.27
Rhodium	g/t	0.13	0.12	0.12
Recoveries (Life of Mine Average)				
Nickel Recovery	%	69.13	69.47	69.05
Platinum Recovery	%	88.21	87.15	87.24
Palladium Recovery	%	87.63	86.85	86.77
Copper Recovery	%	87.89	87.90	87.84
Gold Recovery	%	76.69	76.72	76.72
Rhodium Recovery	%	85.92	86.62	86.62
Concentrate Produced (Life of Mine Average)				
Concentrate	ktpa	156	292	413
Nickel	%	5.8	6.0	5.8
Platinum	g/t	40.5	37.0	37.3
Palladium	g/t	42.4	38.7	38.4
Copper	%	3.6	3.6	3.5
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
Metal Sold (Life of Mine Average Annual Production)				
Nickel	Mlb	20	39	53
Platinum	koz	203	348	495
Palladium	koz	212	363.8	510
Copper	Mlb	12	23.2	32
Gold	koz	26	48.7	69
Rhodium	koz	14	24.7	35
3PE + Au	koz	455	785	1,109

22.6 Capital and Operating Cost Summary

The pre-production capital cost including contingency for each case is shown in Figure 22.10 and Table 22.5. The pre-production, sustaining, and total capital costs are shown in Table 22.5, Table 22.6, and Table 22.7. The operating costs are summarised in Table 22.8 to Table 22.10. The cash costs denominated in US$/oz Payable 3PE+Au show that Phase 2 is slightly lower cash cost for the Phases and the Options.

Figure 22.10 Pre-Production Capital

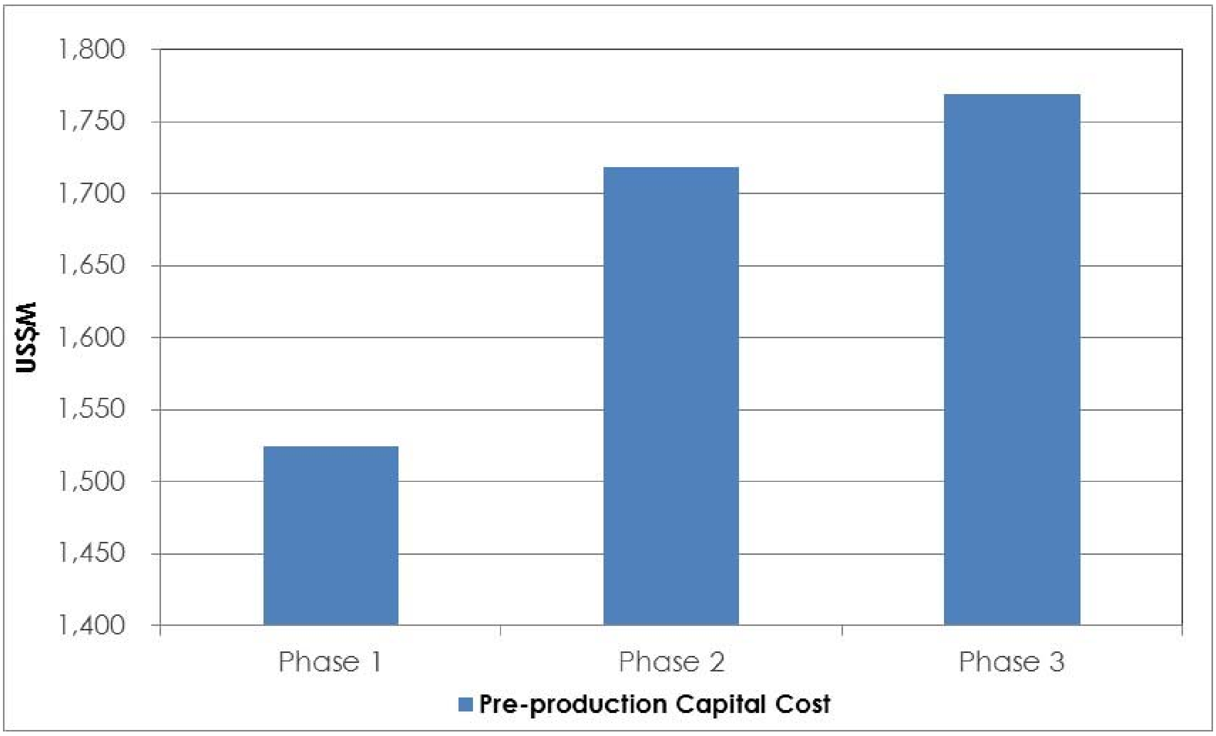


Figure by OreWin 2014.

Table 22.5 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
Capitalized 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
Subtotal	**144**	**151**	**144**
Indirects			
Drilling & Studies	–	19	19
Mining: Indirects	55	58	58
Mining: EPCM	80	93	97
Processing & Infrastructure: EPCM	37	37	37
Subtotal	**172**	**207**	**211**
Owners Cost			
Capitalized G&A	26	26	26
Mining	60	79	79
Processing & Infrastructure	17	18	17
Subtotal	**103**	**123**	**122**
Capital Expenditure Before Contingency	**1,185**	**1,338**	**1,376**
Mining Contingency	221	259	272
Processing & Infrastructure Contingency	120	122	120
Capital Expenditure After Contingency	**1,525**	**1,719**	**1,769**

Table 22.6 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
Capitalized 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 & Studies	–	–	19
Mining: Indirects	–	–	–
Mining: EPCM	–	–	–
Processing & Infrastructure: EPCM	4	33	63
Subtotal	**4**	**33**	**81**
Owners Cost			
Capitalized G&A			
Mining	–	–	–
Processing & Infrastructure	2	14	29
Subtotal	**2**	**14**	**29**
Capital Expenditure Before Contingency	**814**	**2,029**	**3,282**
Mining Contingency	124	354	572
Processing & Infrastructure Contingency	36	146	266
Capital Expenditure After Contingency	**974**	**2,528**	**4,120**

Table 22.7 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
Capitalized 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 & Studies	–	19	38
Mining: Indirects	55	58	58
Mining: EPCM	80	93	97
Processing & Infrastructure: EPCM	41	70	99
Subtotal	**177**	**240**	**292**
Owners Cost			
Capitalized 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 & Infrastructure Contingency	156	268	386
Capital Expenditure After Contingency	**2,499**	**4,247**	**5,888**

Table 22.8 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**
Nickel Credits	-367	-411	-397
Copper Credits	-81	-89	-86
Total Cash Costs After Credits	**367**	**341**	**371**

Table 22.9 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: Realization Costs			
Transport	334	628	887
Refining Charges	4,207	7,540	10,577
Government Royalty	938	1,628	2,261
Total Realization 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 & 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 22.10 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: Realization 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 Realization 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**

22.7 Project Cash Flows

The cash flows for the three phases are shown in Figure 22.11 to Figure 22.13 and in Table 22.11 to Table 22.13.

Figure 22.11 Phase 1 Concentrator 4 Mtpa Cumulative Cash Flow

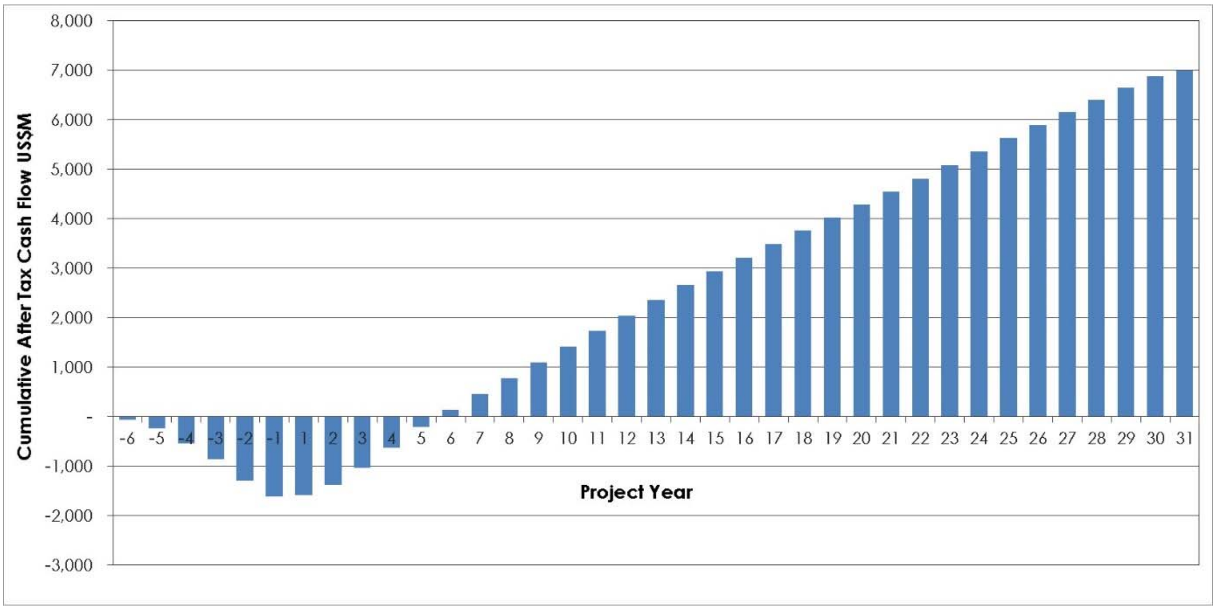


Figure by OreWin 2014.

Figure 22.12 Phase 2 Concentrator 8 Mtpa Cumulative Cash Flow

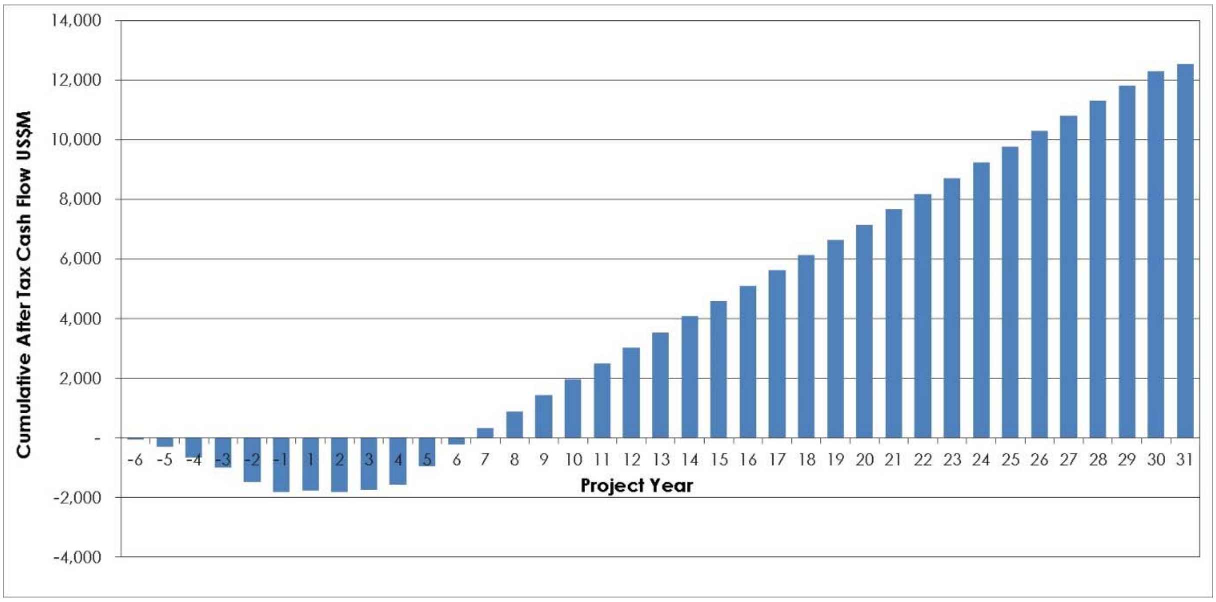


Figure by OreWin 2014.

Figure 22.13 Phase 3 Concentrator 12 Mtpa Cumulative Cash Flow

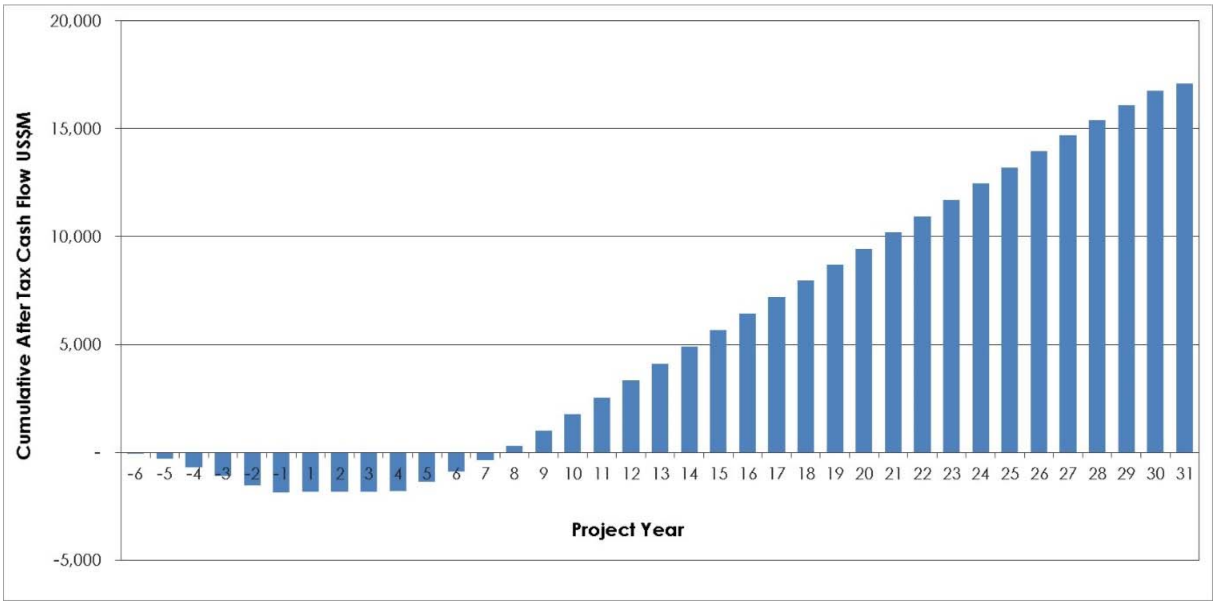


Figure by OreWin 2014.

Table 22.11 Phase 1 – 4 Mtpa Cumulative Cash Flow

Project Year	**-6**	**-5**	**-4**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6 to 10**	**11 to 20**	**21 to LOM**	**Total**
Item	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M
Total Gross Revenue	–	–	–	–	–	–	400	535	826	835	846	4,260	8,107	7,566	23,375
Total Realisation Costs	–	–	–	–	–	–	78	122	193	195	198	994	1,904	1,795	5,479
Net Sales Revenue	–	–	–	–	–	–	**323**	**413**	**632**	**640**	**649**	**3,267**	**6,203**	**5,771**	**17,896**
Site Operating Costs						–									
Mining	–	–	–	–	3	21	41	79	119	119	119	598	1,361	1,295	3,755
Processing & Tailings & Smelter & BMR	–	–	–	–	–	–	21	28	43	43	43	214	429	429	1,251
General & Administration	4	4	4	4	4	4	19	19	20	20	20	99	198	198	618
Total Site Operating Costs	**4**	**4**	**4**	**4**	**7**	**26**	**81**	**126**	**182**	**182**	**182**	**912**	**1,988**	**1,922**	**5,624**
Operating Surplus / (Deficit)	**-4**	**-4**	**-4**	**-4**	**-7**	**-26**	**242**	**287**	**451**	**458**	**467**	**2,355**	**4,214**	**3,849**	**12,273**
Indirect Costs	0	0	0	0	0	1	1,424	113	103	99	99	248	269	261	2,617
Net Profit Before Income Tax	**-4**	**-4**	**-4**	**-4**	**-7**	**-26**	**-1,182**	**174**	**348**	**358**	**368**	**2,107**	**3,945**	**3,588**	**9,656**
Income Tax Expense	–	–	–	–	–	–	–	–	–	–	–	554	1,105	1,005	2,663
Net Profit After Income Tax	**-4**	**-4**	**-4**	**-4**	**-7**	**-26**	**-1,182**	**174**	**348**	**358**	**368**	**1,553**	**2,841**	**2,583**	**6,992**
Capital Expenditure	-49	-179	-291	-319	-435	-296	-130	-68	-48	-43	-38	-140	-210	-203	-2,448
Depreciation Less Working Capital	0	–	–	–	0	1	1,351	88	47	92	91	220	227	329	2,448
Net Cash Flow After Tax	**-53**	**-183**	**-295**	**-323**	**-441**	**-321**	**39**	**193**	**347**	**408**	**421**	**1,633**	**2,858**	**2,709**	**6,992**

Table 22.12 Phase 2 – 8 Mtpa Cumulative Cash Flow

Project Year	-6	-5	-4	-3	-2	-1	1	2	3	4	5	6 to 10	11 to 20	21 to LOM	Total
Item	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M
Total Gross Revenue	–	–	–	–	–	–	400	535	814	829	1,528	7,597	15,108	14,832	41,644
Total Realisation Costs	–	–	–	–	–	–	78	104	170	176	358	1,797	3,585	3,529	9,796
Net Sales Revenue	–	–	–	–	–	–	**323**	**431**	**644**	**653**	**1,170**	**5,801**	**11,523**	**11,303**	**31,849**
Site Operating Costs															
Mining	–	–	–	–	3	22	41	80	133	149	177	1,180	2,689	2,636	7,109
Processing & Tailings & Smelter & BMR	–	–	–	–	–	–	21	28	43	43	74	398	809	809	2,226
General & Administration	4	4	4	4	4	4	19	19	20	20	22	109	219	219	670
Total Site Operating Costs	**4**	**4**	**4**	**4**	**7**	**26**	**81**	**127**	**195**	**211**	**273**	**1,687**	**3,717**	**3,664**	**10,006**
Operating Surplus / (Deficit)	**-4**	**-4**	**-4**	**-4**	**-7**	**-26**	**241**	**304**	**449**	**442**	**897**	**4,114**	**7,806**	**7,639**	**21,843**
Indirect Costs	0	0	0	0	0	1	1,623	355	364	310	204	506	618	514	4,497
Net Profit Before Income Tax	**-4**	**-4**	**-4**	**-4**	**-7**	**-27**	**-1,382**	**-51**	**85**	**132**	**693**	**3,608**	**7,188**	**7,125**	**17,346**
Income Tax Expense	–	–	–	–	–	–	–	–	–	–	–	809	2,013	1,998	4,819
Net Profit After Income Tax	**-4**	**-4**	**-4**	**-4**	**-7**	**-27**	**-1,382**	**-51**	**85**	**132**	**693**	**2,799**	**5,175**	**5,127**	**12,527**
Capital Expenditure	-57	-231	-364	-339	-461	-308	-135	-319	-321	-271	-134	-354	-497	-404	-4,197
Depreciation Less Working Capital	0	–	–	–	0	2	1,551	330	312	302	71	467	509	653	4,197
Net Cash Flow After Tax	**-61**	**-235**	**-369**	**-343**	**-468**	**-333**	**34**	**-40**	**75**	**163**	**630**	**2,911**	**5,187**	**5,376**	**12,527**

Table 22.13 Phase 3 – 12 Mtpa Cumulative Cash Flow

Project Year	**-6**	**-5**	**-4**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6 to 10**	**11 to 20**	**21 to LOM**	**Total**
Item	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M
Total Gross Revenue	–	–	–	–	–	–	400	535	814	829	1,528	9,545	22,845	21,861	58,358
Total Realisation Costs	–	–	–	–	–	–	78	106	164	164	349	2,235	5,414	5,216	13,726
Net Sales Revenue	–	–	–	–	–	–	**323**	**429**	**650**	**665**	**1,179**	**7,310**	**17,431**	**16,645**	**44,632**
Site Operating Costs															
Mining	–	–	–	–	3	22	42	81	134	150	178	1,565	4,331	4,369	10,874
Processing & Tailings & Smelter & BMR	–	–	–	–	–	–	21	28	43	43	74	499	1,194	1,194	3,096
General & Administration	4	4	4	4	4	4	19	19	20	20	22	114	239	239	717
Total Site Operating Costs	**4**	**4**	**4**	**4**	**7**	**26**	**82**	**128**	**196**	**212**	**274**	**2,178**	**5,764**	**5,802**	**14,687**
Operating Surplus / (Deficit)	**-4**	**-4**	**-4**	**-4**	**-7**	**-26**	**241**	**301**	**453**	**453**	**905**	**5,132**	**11,666**	**10,843**	**29,945**
Indirect Costs	0	0	0	0	0	1	1,663	307	434	463	381	1,228	1,051	750	6,278
Net Profit Before Income Tax	**-4**	**-4**	**-4**	**-4**	**-7**	**-27**	**-1,422**	**-6**	**20**	**-10**	**524**	**3,904**	**10,616**	**10,093**	**23,667**
Income Tax Expense	–	–	–	–	–	–	–	–	–	–	–	788	2,972	2,829	6,589
Net Profit After Income Tax	**-4**	**-4**	**-4**	**-4**	**-7**	**-27**	**-1,422**	**-6**	**20**	**-10**	**524**	**3,116**	**7,643**	**7,264**	**17,078**
Capital Expenditure	-57	-231	-398	-398	-422	-298	-135	-270	-398	-428	-326	-1,054	-846	-576	-5,837
Depreciation Less Working Capital	0	–	–	–	0	2	1,590	282	381	454	248	1,048	885	946	5,837
Net Cash Flow After Tax	**-61**	**- 235**	**-402**	**-402**	**-428**	**-323**	**33**	**5**	**3**	**17**	**445**	**3,110**	**7,682**	**7,635**	**17,078**

22.8 Price Sensitivity Analysis

A price sensitivity analysis of the NPV8 was performed for each one of the seven scenarios, taking into account the variation on prices of Platinum (from US$1,400/oz to US$2,000/oz) and Nickel (from US$6.85/lb to US$9.85/lb). The results are shown in Table 22.14 to Table 22.16 and Figure 22.14 to Figure 22.19.

Table 22.14 NPV8 v Price – Phase 1

NPV8 US$M	Platinum Price - US$/lb				
Nickel Price - US / lb	**1,400**	**1,600**	**1,700**	**1,800**	**2,000**
6.85	521	689	773	857	1,026
7.85	603	771	856	940	1,108
8.35	644	813	897	981	1,149
8.85	685	854	938	1,023	1,190
9.85	768	937	1,021	1,105	1,272

Figure 22.14 Platinum Price v NPV8 – Phase 1

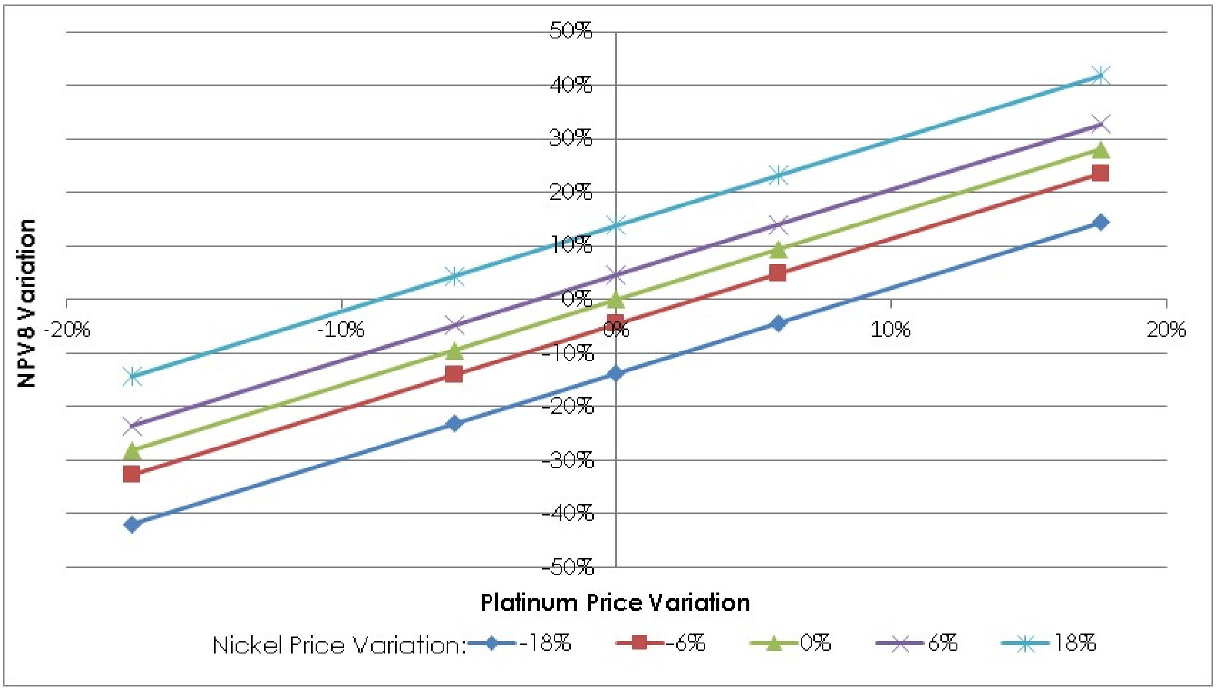


Figure by OreWin 2014.

Figure 22.15 Nickel Price v NPV8 – Phase 1

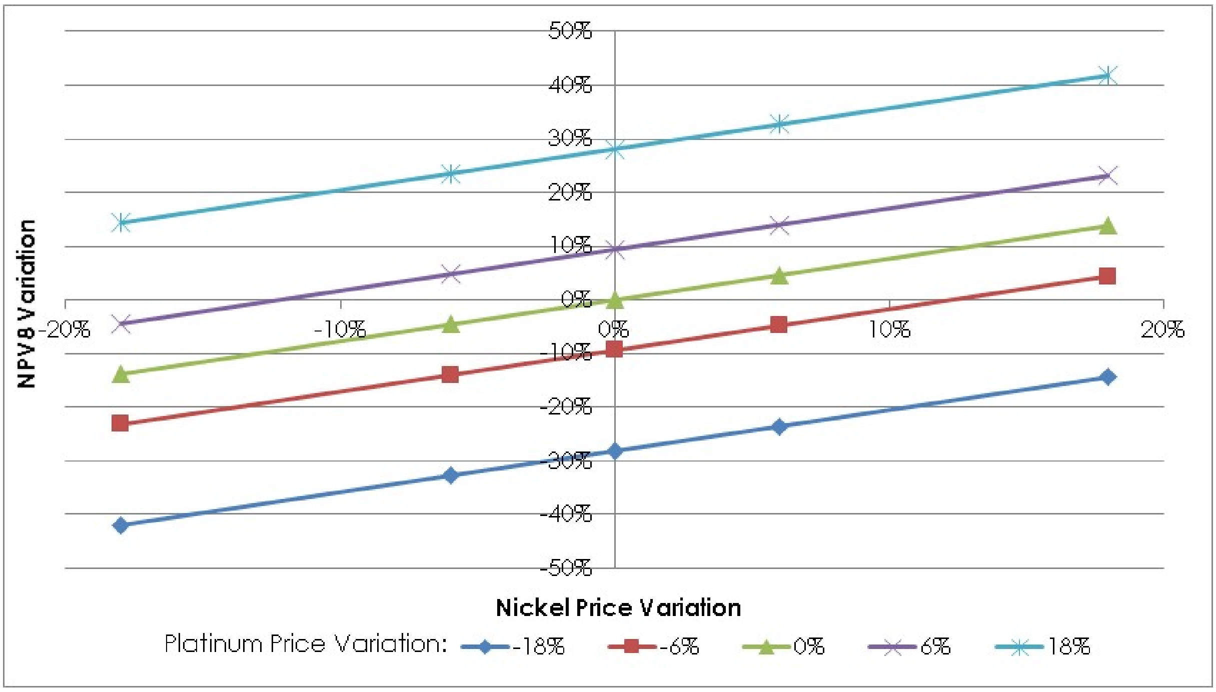


Figure by OreWin 2014.

Table 22.15 NPV8 v PRICE – Phase 2

NPV8 US$M	Platinum Price - US$/lb				
Nickel Price - US / lb	**1,400**	**1,600**	**1,700**	**1,800**	**2,000**
6.85	1,000	1,265	1,397	1,530	1,794
7.85	1,148	1,413	1,546	1,678	1,940
8.35	1,222	1,487	1,620	1,751	2,014
8.85	1,297	1,561	1,694	1,825	2,087
9.85	1,445	1,709	1,841	1,972	2,234

Figure 22.16 Platinum Price v NPV8 – Phase 2

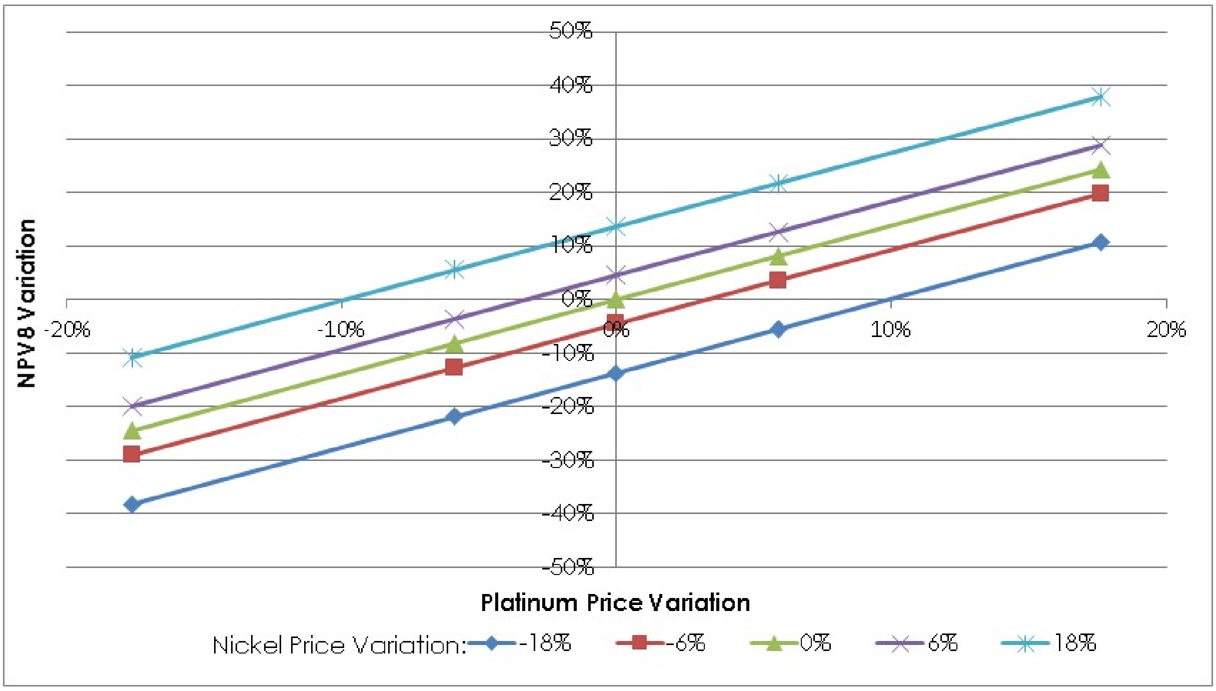


Figure by OreWin 2014.

Figure 22.17 Nickel Price v NPV8 – Phase 2

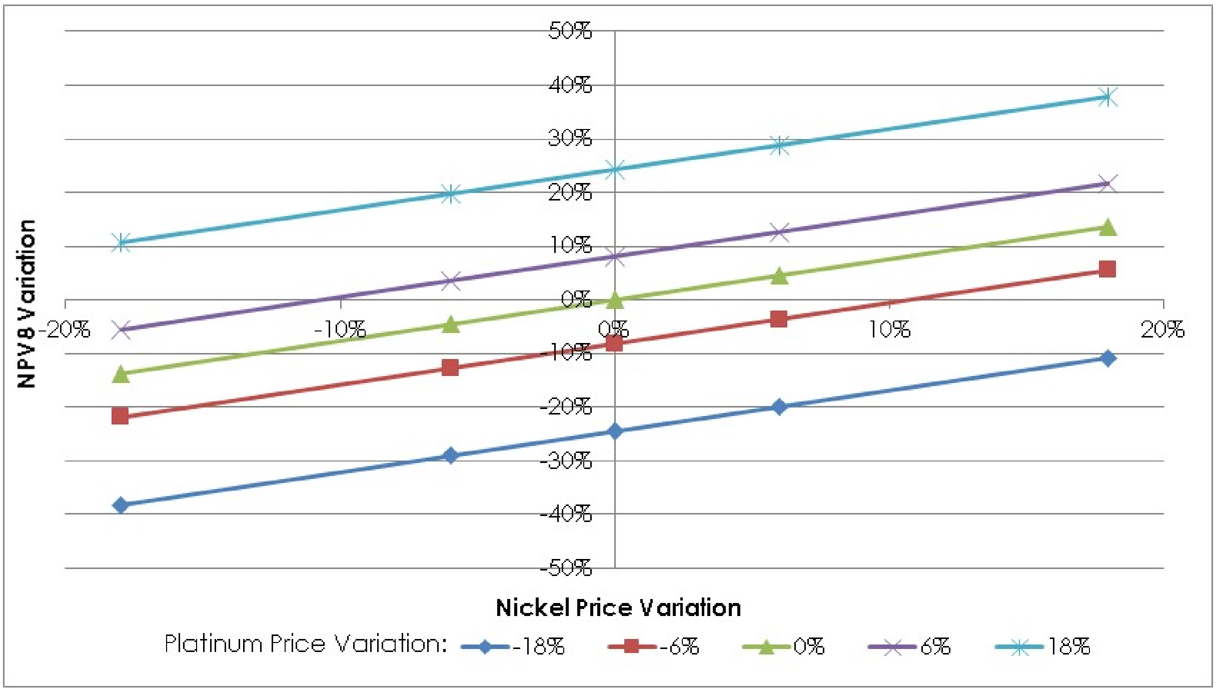


Figure by OreWin 2014.

Table 22.16 NPV8 v Price – Phase 3

NPV8 US$M	Platinum Price - US$/lb				
Nickel Price - US / lb	**1,400**	**1,600**	**1,700**	**1,800**	**2,000**
6.85	1,370	1,722	1,897	2,072	2,422
7.85	1,559	1,910	2,085	2,260	2,609
8.35	1,654	2,004	2,179	2,354	2,702
8.85	1,748	2,098	2,273	2,448	2,795
9.85	1,936	2,286	2,460	2,634	2,982

Figure 22.18 Platinum Price v NPV8 – Phase 3

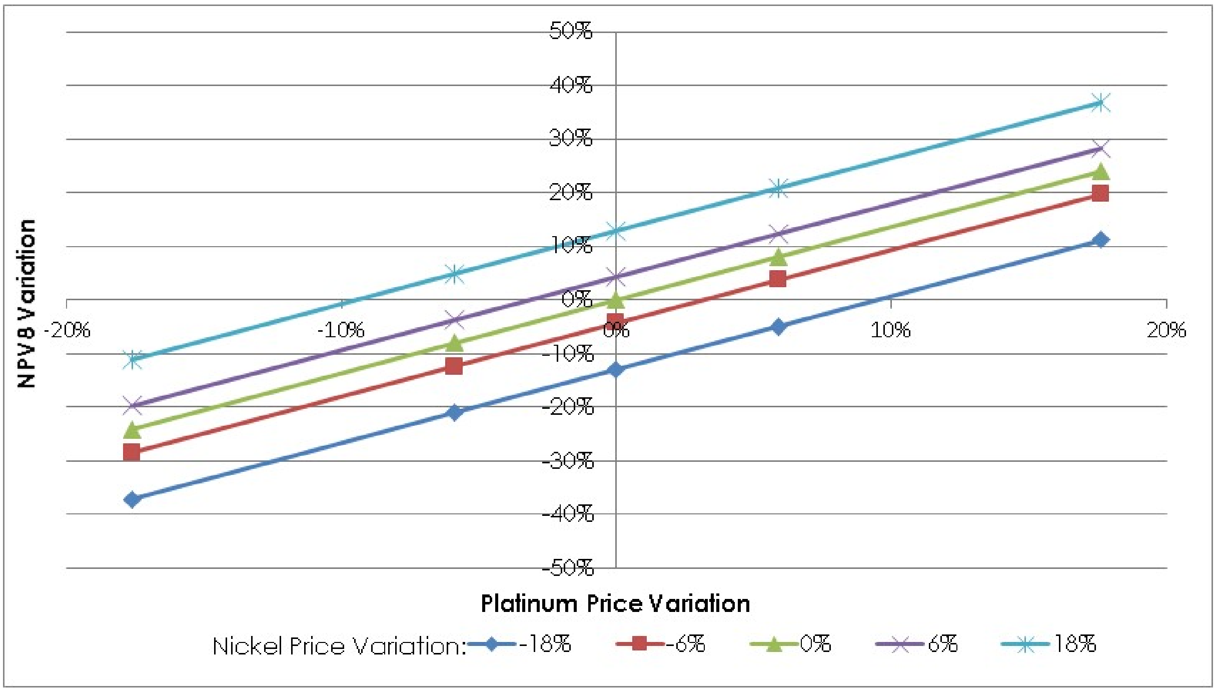


Figure by OreWin 2014.

Figure 22.19 Nickel Price v NPV8 – Phase 3

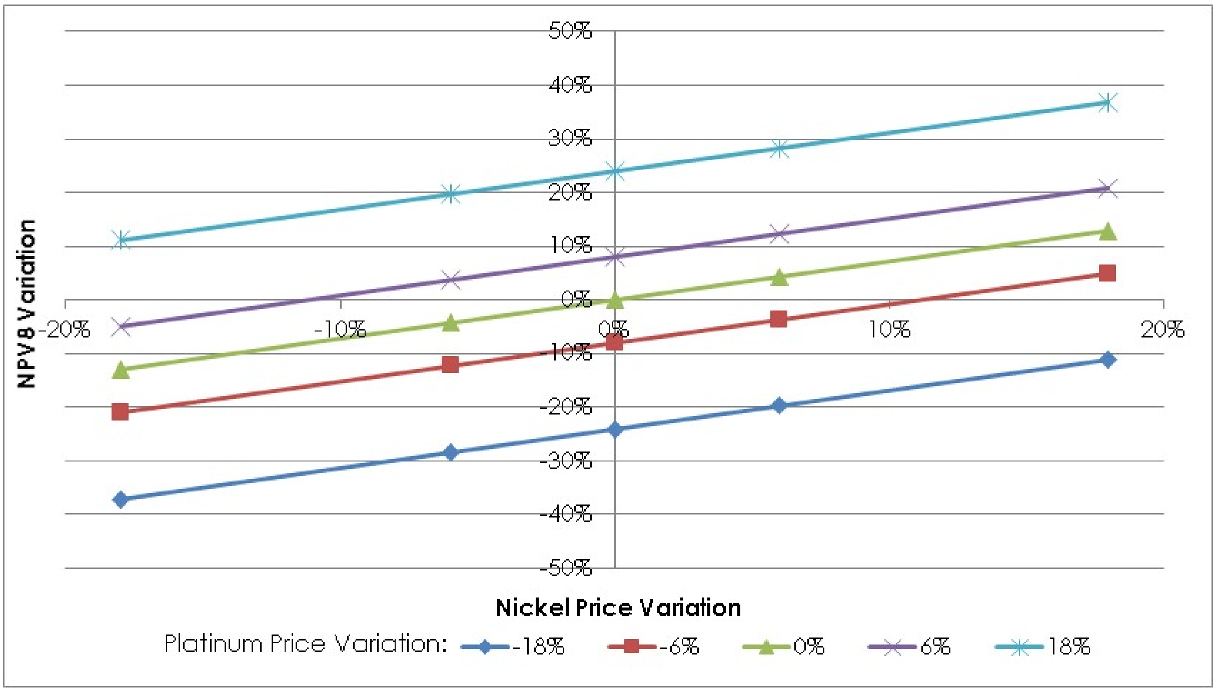


Figure by OreWin 2014.

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 PEA

The Platreef 2014 PEA compares the results of three phases at scoping level. Further work and studies are required to bring the Project to a Pre-Feasibility Study level. The 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 Tenure and Surface Rights

Legal opinion was provided to AMEC that supports that Ivanhoe holds indirect title to MPT 55/2006 PR. This prospecting right is valid to 31 May 2014. Ivanhoe has applied for a mining right over the prospecting licence area, and at the effective date of this Report, the licence grant was pending.

Atlatsa Resources Corporation indirectly holds title to MPT 76/2007 PR. The prospecting right was valid for a five-year period, and was to expire on 27 November 2011. Prior to the expiry date, on 22 August 2011, an application was lodged to renew the prospecting right for a three-year extension of term. At the effective date of the Report, the renewal was still pending.

Surface rights over the Macalacaskop and Turfspruit farms 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, Mountian View, and Michelle Communities community are the lawful occupiers of the Macalacaskop Farm, and the Tshamahansi (Hlongwane, Baloyi and Matjeke), Kgobudi, Masodi, and Magongoa communities Communities are the lawful occupiers of the Turfspruit Farm (see Figure 4.4) Rights to prospect and mine the land are granted by the State.

Objections have been lodged with respect to both the planned bulk sampling programme and the mining rights application. These objections are currently under review by the relevant administrator authority.

25.3 Geology and Resources

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

Four major cyclic units have been recognized which correlate well with the Upper Critical Zone (UCZ) rock sequence described for the main Bushveld Complex.

The TCU is laterally continuous across large parts of the Project area. Mineralization in the TCU shows generally good continuity and is mostly confined to pegmatoidal orthopyroxenite and harzburgite. Variations across the Project area in the stratigraphic footwall of the TCU, and in thickness and lithology (facies) within the TCU, are interpreted as related to potholes.

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, mineralization style and setting, and structural and alteration controls on mineralization within the AMK, ATS, and UMT deposits are sufficient to support Mineral Resource estimation. The mineralization delineated at the Turfspruit, Macalacaskop and Rietfontein farms is typical of Platreef-style mineralization within the Northern Limb of the Bushveld Complex. Exploration programmes developed using the Merensky-reef analogue are appropriate to the deposit style.

25.3.1 Database

Drilling on the 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 18 February 2014, a total of 64 drillholes (44,712 m) and 59 deflections (55,886 m) have been completed at Platreef. The additional drillholes have been completed for geotechnical data, metallurgical samples, and geology/resource drilling. Nine drillholes were in progress on 18 February 2014.

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 are downhole 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 standard reference material (SRM) samples.

The quality assurance and quality control (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 PGE, Au, Ni, Rh, and Cu analytical data are sufficiently reliable to support Mineral Resource estimation.

25.3.2 Mineral Resource Estimates

Dr Parker and Mr Kuhl are of the opinion that the Mineral Resources for the 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 2010 CIM Definition Standards.

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

- Confirmation of the renewal of the Rietfontein prospecting licence has not been granted. The Mineral Resources amenable to open-pit methods on Macalacaskop are not expected to be affected; however, Mineral Resources amenable to open-cut-off pit methods as declared for Turfspruit and Rietfontein would have to be re-evaluated without a valid prospecting licence on Rietfontein.
- Confirmation that a mining right will be granted for the Macalacaskop and Turfspruit area; the application has been lodged and grant was pending at the Report effective date.
- Assumptions as to the amenability of local communities to allow surface access for Ivanhoe exploration and sampling programmes with appropriate negotiation and compensation.
- Assumptions used to generate the conceptual data for consideration of reasonable prospects of 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 mineralization layers may result in changes to the metallurgical recoveries and smelter payables assumptions used to evaluate reasonable prospects of economic extraction.

Mineral Resources have been estimated on an externally undiluted basis and without consideration for mining recovery. Dilution and recoveries will vary with the geometry (dip, thickness, faulting and or irregularities in contacts) of the mineralization and the eventual mining method used. These factors can only be estimated after life-of-mine plans are prepared. In AMEC's experience, dilution (low-grade or waste materials) ranges from 10% to 30%, and mining recoveries range from 70% to 100% using the mining methods considered for evaluation of reasonable prospects of economic extraction.

25.4 Mining Risks and Opportunities

This section presents identified risks and opportunities associated with the mining plan at 4 Mtpa and the expansion phases evaluated for the underground mining at Platreef. These identified risks and opportunities are subdivided into those common between the three mine plans presented (4 Mtpa, 8 Mtpa, and 12 Mtpa options), and those specific to the expansion phases.

25.4.1 Risks

25.4.1.1 Common

Presented below are risks that are common to all the production rate scenarios.

- Mine designs and associated work was prepared at a conceptual level, which requires additional engineering and design, to ensure the level of design is commensurate with any financial decisions that Ivanhoe is considering related to the mining of the Platreef.
- Shaft development rates utilized in this study might be considered aggressive. Slower development rates would increase the pre-production schedule and defer revenue, affecting the overall economics.
- Development and production scheduling has not been performed in detail. Experience factors were applied to assist in estimating reasonable completion dates. Detailed scheduling could result in delays to the initial production dates and to the production ramp up schedules.
- Much of the data and information used for developing the schedules and costs for the Platreef project were estimated using experience on other similar projects. Additional data required to confirm or refute the assumptions made in this report is listed below.
  - Geotechnical data at depth to ensure the mining method is suitable and ground support requirements can be designed to suit.
  - Evaluation of hydrological data to confirm that water will not significantly impact costs and schedules and identification of dewatering systems required to address the water issues if required.
  - Updated labor and equipment and material costs will need to be developed for the next phase.
- Equipment availability is currently a risk being realized by all mine operators. Significant lead times are currently required for procurement and delivery of new mobile and fixed equipment from vendors and suppliers.
- Variations in the floor and roof of the gently dipping areas will necessitate definition drilling to better define the resources prior to mining these areas. In some of these areas, these variations could result in excessive dilution and/or mining losses.
- Small scale geological features may result in vertical displacement of the mining zones in the gently dipping zones, which may result in higher ground support costs and/or increased dilution within the stopes.

- Sourcing and training of a large workforce presents many challenges. Mining productivities and costs would be affected adversely if sufficient numbers of trained workers cannot be provided.
- Stope production is scheduled to start in early 2019, approximately one year prior to mill start-up and therefore start-up of the paste backfill plant. An alternative source of backfill will be needed during this time period.

25.4.1.2 Expansion Phases

Presented below are risks that are specific to the expansion phases.

- There are very few operations globally that mine by longhole stoping with backfill at the suggested production rates of 22,000 t/d to 33,000 t/d (8 Mtpa and 12 Mtpa). Though the deposit appears to contain adequate resources, there may be a risk of having to schedule additional mining areas to ensure that the number of required active stopes is met and maintained. This would increase the ramp up periods for build up to full production.
- Costs applied for haulage from the southeast area may be overly optimistic. A more detailed analysis should be performed during the pre-feasibility study (PFS).
- Total manpower requirements for the expansion phases are estimated in the order of 1,000 to 1,300 persons. Sourcing and training this level of manpower presents challenges. Mining productivities and costs would be affected adversely if sufficient numbers of trained workers cannot be provided.

25.4.2 Opportunities

25.4.2.1 Common

Presented below are opportunities common to all the production rate scenarios.

- Additional opportunities may exist related to early mining of higher-grade material but have not been fully identified. This will be evaluated during the PFS.
- An opportunity may exist to shorten the pre-production period by sinking Shaft 3 concurrently with Shafts 1 and 2.
- Relocation of Shaft 4 to the west could alleviate some shaft pillar issues and allow for earlier mining of longhole stoping resources in that area.
- General mine design opportunities that will optimize overall costs and associated schedules exist for both options.

25.4.2.2 Expansion Phases

Presented below are opportunities specific to the expansion phases.

- Alternative haulage methods such as rail or conveyor should be investigated for production coming from the south-east and south-west areas.

25.4.3 Mining Conclusions

The deposit geometry and geotechnical conditions will allow the use of highly productive mechanized mining methods.

The NSR cut-off grades of US$100 and US$80 used in estimating the PEA inventory are significantly higher than the estimated break-even cut-off grades.

The total PEA inventory estimated for this study is more than adequate to support mining at the production rates considered.

More detailed design and scheduling needs to be performed during the pre-feasibility study (PFS) in order to obtain better estimates of the length of the pre-production period and the time required to ramp the mine up to full production.

The staged approach to mine development and project expansion allows for logical project growth based on actual site experience and understanding of mining conditions.

25.5 Metallurgy

It is the opinion of the qualified person responsible for the metallurgical aspects of the PEA, Mr Michael Valenta, that a comprehensive metallurgical test work programme was conducted on the samples provided by the geologists. The range of samples tested appears to span the limits of the mineralized material from a grade perspective; however, further variability test work will be required in the future.

The comprehensive mineralogical analysis on the mineralized material samples provided invaluable information on the mineralisation within the mineralized 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 PGEs.

Given the level of study of this scoping study the amount of metallurgical test work far exceeds what is commonly expected from a study of this accuracy. Furthermore, the quality and high standard of the test work is illustrated by the checks and balances that have been applied throughout the programme.

The test work confirmed that the mineralized material can be classified as a very hard mineralized material that will require careful consideration from a crushing and milling perspective. The selected comminution circuit mitigates this riskroute to follow. Further test work in the next stages of the project can consider other comminution technologies as are already being applied in nearby metallurgical operations.

The flotation test work confirmed that the mineralized material contains various gangue mineral species that will challenge the operation in achieving saleable grade concentrate. The test work was however successful in overcoming the PGE grade challenge without a significant loss in PGE recovery. The test work results were confirmed at two independent metallurgical laboratories and therefore the risk of poor inaccurate results has been mitigated.

Observations and conclusions from the test work are accurately interpreted and included in the proposed concentrator design. Further test work is required to test the robustness of the design and to optimise the various stages in the process.

25.6 Infrastructure

The availability of power and water to a suitable level for this level of study. The project team has addressed the issues in suitable detail for this level of study. It is however the opinion of Mr Michael Valenta that the availability of water and the timely delivery of the various infrastructure projects must not be taken for granted.

A number of mining projects are in the development phase on the Bushveld Igneous Complex that all require water and power. The risk of over-allocation and under-delivery by the various entities must not be underestimated. Anglo Platinum has announced that the expansion of the Mogolokwena North operation by an additional 600 ktpm is being considered. This will place significant strain on the existing infrastructure and proposed expansions.

The environmental impact of the proposed water projects on the 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.

26 RECOMMENDATIONS

26.1 Platreef 2014 PEA

The results of the Platreef 2014 PEA suggest that progress to a Pre-Feasibility Study (PFS) can be reasonably justified. It is recommended that Ivanhoe continue to optimise the PFS scope of work and execution plan. The PFS should be based on Phase 1 and evaluation of Phase 2 and 3 scenarios should continue at scoping 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 2014 PEA.

26.2 Geology, Exploration, and Mineral Resources

AMEC has recommended a two-phase work programme in support of updates to the geology and Mineral Resources. Each phase can be conducted concurrently, and the results the proposed drilling programme can be incorporated as available in the second work phase. The first phase is estimated at approximately US$15 M, the second at US$235,000 to US$320,000.

Ivanhoe has provided AMEC with a planned drill programme for 2014. Dr Parker and Mr Kuhl have reviewed the programme and consider it to be appropriate for the project development stage.

26.2.1 2014

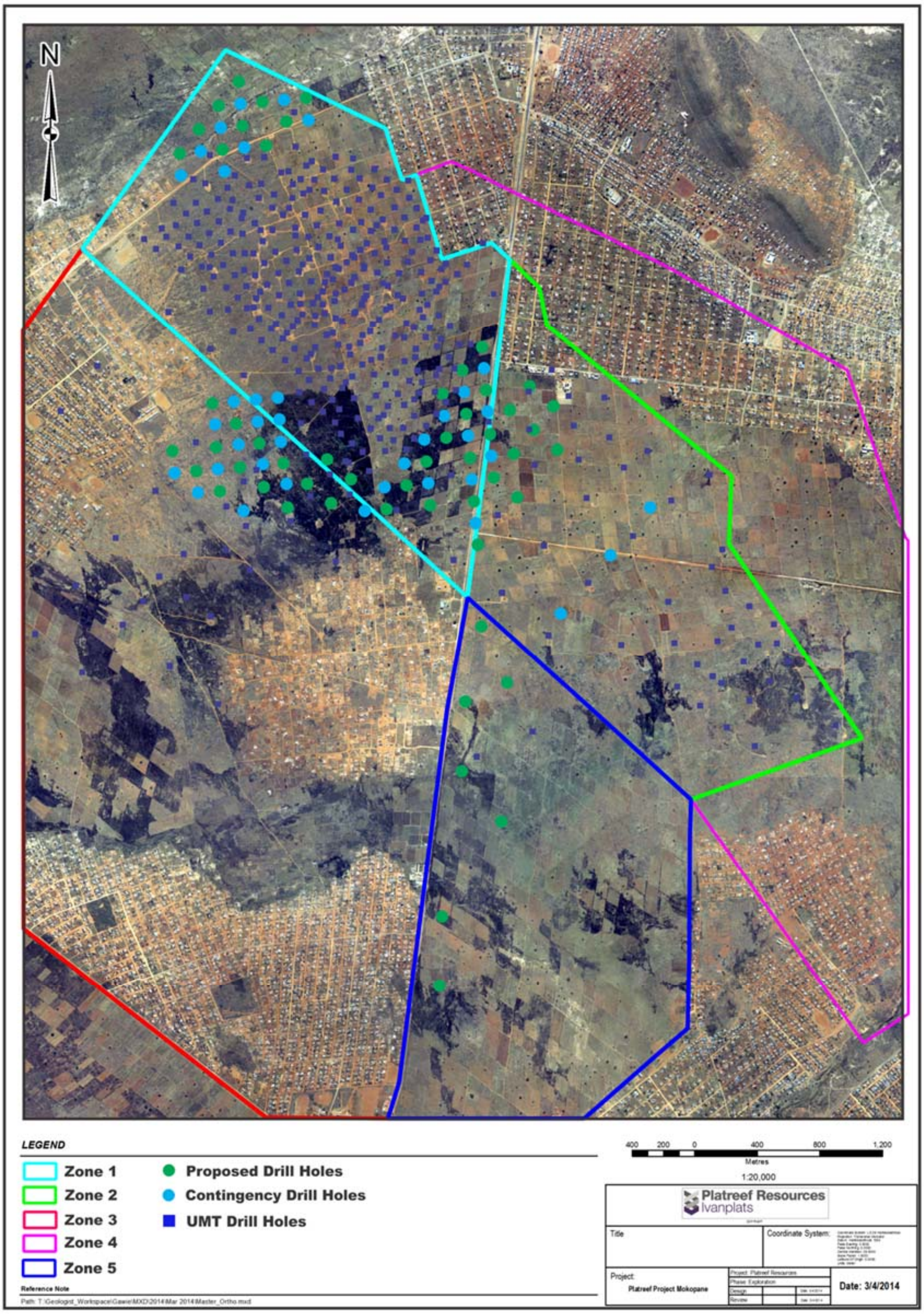
Ivanhoe has plans to continue exploration and step-out (resource expansion) drilling in 2014. Drill plans include 95 drillholes (92,800 m). Figure 26.1 also displays the proposed drill plan. The drilling is targeted in four main areas:

- Madiba Area (Zone 5): An additional 10 drillholes (9,350 m) are proposed on a 400 x 400 m grid. The proposed drilling will continue to test the projection of the Flatreef into the Madiba area.
- Zone 1: A total of 18 drillholes (16,988 m) are planned. An intitial 10 drillholes are planned to be completed on a 200 x 200 m grid. A planned follow up programme includes eight drillholes to infill the 200 x 200 m drilling with a dice 5 drill pattern (also known as a five-spot pattern whereby the drilling forms a square pattern with a single hole in the middle of the square). The locations of the infill drillholes are tentative and will be finalized based on the results of the 200 x 200 m drilling.
- Zone 2: A total of 32 drillholes (31,500 m) are proposed for Zone 2. Initial drilling is proposed at 200 m spacing to test the expansion of the Flatreef. Infill drilling is proposed using a dice 5 drill pattern. The locations of the infill drillholes are tentative and will be finalized based on the results of the 200 x 200 m drilling.
- Zone 3: A total of 19 drillholes (28,995 m) are proposed for Zone 3. Initial drilling is proposed at 200 m spacing to test the expansion of the Flatreef. Infill drilling is proposed using a dice 5 (five- spot) drill pattern. The locations of the infill drillholes are tentative and will be based on the results of the 200 x 200 m drilling.

- Contingency Drilling: The budget includes 6,000 m (6 drillholes) for contingency purposes. Drillhole locations are tentative and final collar locations, if the drilling is undertaken, will be determined based on the results of the 2014 drill campaign.

A summary of the proposed drilling and estimated budget for the programme is included as Table 26.1. Drill programmes assume the all-inclusive drill costs of for 2014 of US$160/m. The 2014 programme has an overall estimated total budget of US$15 M.

Figure 26.1 Proposed Drilling



Note: Figure prepared by AMEC, 2014.

Table 26.1 Drill Plan for 2014

Drilling Programme	Drillholes	Metres
Madiba 400 m	10	9,350
Zone 1 Expansion - 200	10	9,480
Zone 1 Expansion – Dice 5	8	7,500
Zone 2 N Expansion – 200	22	21,500
Zone 2 N Expansion – Dice 5	10	10,000
Zone 3 Expansion – 200	15	14,930
Zone 3 Expansion – Dice 5	14	14,065
Contingency	6	6,000
Total	**95**	**92,836**

Note: Drill programmes assume the all-inclusive drill costs of for 2014 of US$160/m. The 2014 programme has an overall estimated total budget of US$15 M.

26.2.2 Resource Estimate Update

Ivanhoe is currently undertaking a comprehensive relogging of available drill core and reinterpretation of the Project geology and deposit setting. Once this programme is complete, the following second-phase work should be undertaken in support of an updated Mineral Resource estimate. This second phase is estimated at between US$235,000 and US$320,000.

AMEC recommends that the geological model is rebuilt to combine the ATS, AMK, and UMT (bulk mineable) resource models into a single model or if not feasible, build separate models that are defined on a common basis. This will put all models on the same litho-stratigraphic and assay (total) basis. In AMEC’s experience, rebuilding the model is estimated at approximately US$100,000–US$150,000.

As part of this model rebuild programme, AMEC also recommends that Ivanhoe also undertakes a validation of the entire Project database to verify that no errors have arisen during the transfer from the former Fusion database software to the current acQuire database and to ensure that the information supporting the remodeling is acceptable. AMEC estimates this work to be in the range of US$45,000–US$60,000.

Mineral Resources should be re-estimated based on the validated database and the updated geological model. The updated estimate should be budgeted at approximately US$90,000–US$110,000.

The validation step and resource estimate update can be performed concurrently with the exploration drill programme recommended in Section 26.2 and are independent of that drill programme.

26.3 Mining Recommendations

Following are Stantec's recommendations for additional work and modifications to the current mine plans during the PFS.

- Mine layouts and designs should be refined and optimized to the extent possible to enable more accurate scheduling and cost estimates.
- Stoping layouts should be prepared in greater detail in order to better estimate and account for both internal and external dilution and overall mining recovery factors.
- 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.
- More detailed development and production schedules should be prepared to obtain better estimates of the length of the pre-production period and of the time required to ramp the mine up to full production.
- Capital and operating cost estimates specific to the Platreef Project should be developed using first principles in order to refine project cost estimates. Labor rates and materials and equipment costs should be updated to better reflect local South African costs.
- Consideration should be given to sinking Shaft 3 concurrently with Shafts 1 and 2 and equipping it for temporary hoisting of development muck. Use of Shaft 3 as a temporary development platform could help shorten the pre-production period significantly.
- Additional geotechnical data should be collected in the rock units in the footwall of the deposit to assess ground conditions there. Current designs assume that a large portion of the underground infrastructure will be located in the footwall units.
- Shaft 4 should be moved to the west to allow for earlier mining of longhole stoping resources located within the area of the current shaft pillar.
- Alternative types and sources for backfill should be evaluated for the time period between production start-up and commissioning of the paste backfill plant.
- Additional ventilation and refrigeration studies need to be conducted as part of the pre-feasibility study.

26.4 Metallurgical Recommendations

The metallurgical test work thus far has been comprehensive and has met the requirements of the scoping study. It has however confirmed a number of potential risks that need to be addressed in the subsequent stages of the project. These include:

- The fact that the mineralized material is considered to be very hard;
- The poor concentrate grades and the effect of chasing higher grades on the PGE recovery;
- The seemingly complex reagent suite and the requirement of lengthy condition time in order to be able to meet the recovery and grade targets.
- The variability of the mineralized material grade and process recoveries;
- The need to include more of the footwall into the mill feed.

Further test work is recommended on compiste samples to address the first four risks in the earlier feasibility studies. Variability test work in relation tio the mining plan should be considered at a later stage of the project once a better understanding of the mineralized material characteristics has been developed.

The proposed metallurgical plant design encompasses the major findings of the test work; however, the operability and interaction of the various stages needs to be considered in the later stages. A better understanding of the mineralized 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 buildup of floatable contaminants in the flotation circuit.

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

26.5 Infrastructure

The progress with the various infrastructure projects needs to be continuously monitored. A number of mining projects are in the feasibility stage within the footprint of the Bushveld Igneous Complex and the risk of over–allocation and under–delivery is a reality.

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 include:

- ESIA in accordance with the Mineral and Petroleum Resources Development Act (MPRDA), the National Environmental Management Act (NEMA) as well as the EP and IFC Performance Standards;

- Stakeholder Engagement Process (SEP) in accordance with the NEMA 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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