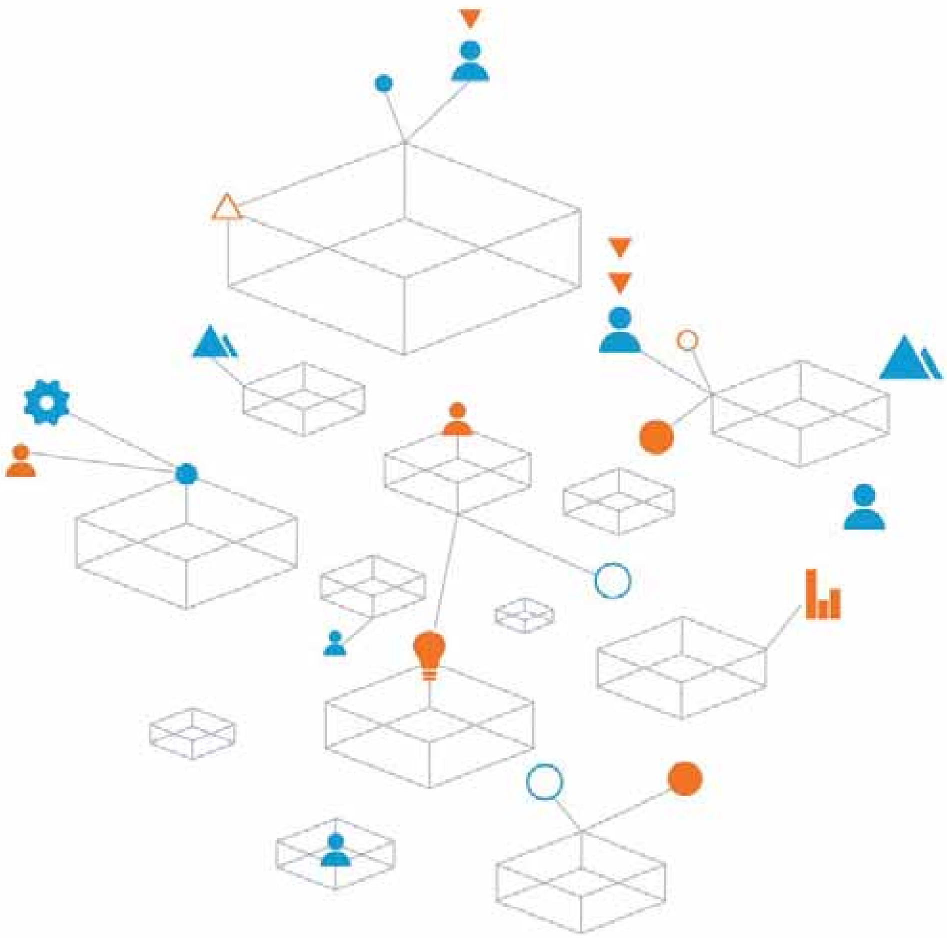
Tharisa plc
Tharisa Chrome and PGM Mine, South Africa
Competent Persons Report
(Effective Date 30 September 2016)
Lead Competent Person: Ken Lomberg (Pr.Sci.Nat.)



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EXECUTIVE SUMMARY

Coffey Mining (South Africa) (Proprietary) Limited (Coffey) was requested by Tharisa plc, formerly Tharisa Limited (Tharisa or the Company), to complete a Competent Persons Report (CPR) in support of the Mineral Resource and Mineral Reserve declaration 2016, in respect of the Tharisa Mine located in the North West Province of South Africa.

The Mineral Resources and Reserves are reported in accordance with the guidelines of “The South African Code for Reporting of Exploration Results, Mineral Resources and Reserves (2016)” (SAMREC Code).

This report is dated 30 September 2016 and Tharisa has advised Coffey that no material change has occurred to the Tharisa Mine since this date.

Participants

The participants consist of a number of technical experts brought together by Coffey to complete the CPR and are all Competent Person’s as defined in (SAMREC Code). The compilation of the CPR in accordance with the reporting requirements of the LSE was supervised by Mr Lomberg. The participants in the compilation of the CPR and their individual areas of responsibility are listed as follows:-

Ken Lomberg, Senior Principal Consultant, Coffey
Project management, mineral resources, geological interpretations, site visits, report preparation.

Jaco Lotheringen, Associate Consultant – Ukwazi Mining Solutions
Mining engineering, mineral reserve estimation, infrastructure, economic valuation, site visits, report preparation.

John James, Associate Consultant – Celtis Geotechnical
Geotechnical Engineering, site visits, report preparation.

Independence

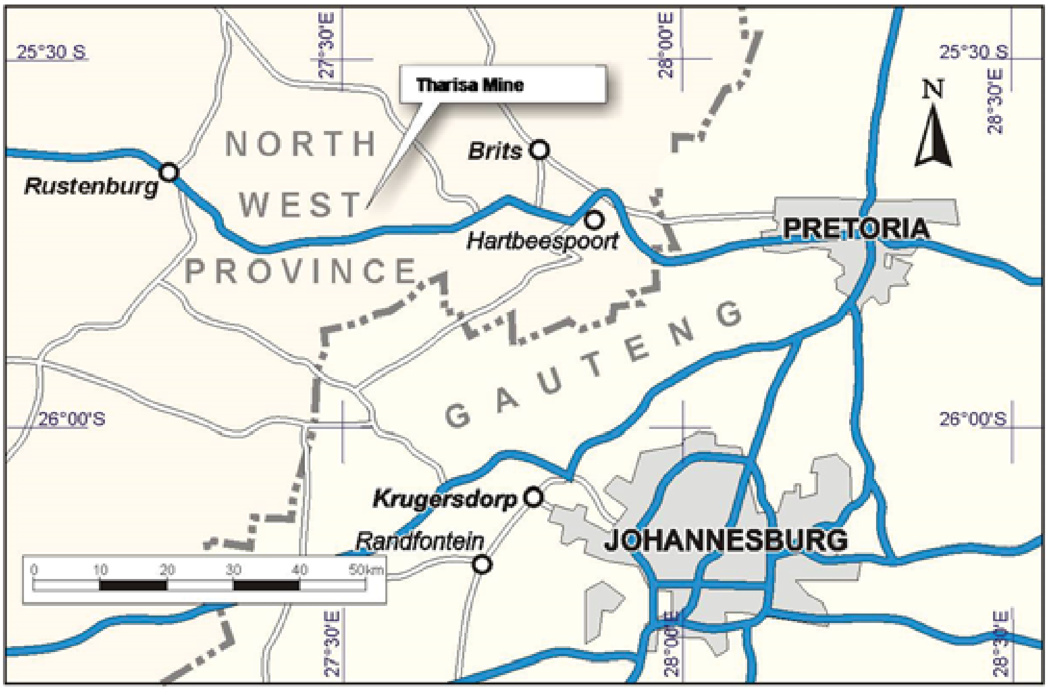
Coffey is an independent technical consulting group, with no direct or indirect interests in Tharisa. Neither Coffey, nor the key personnel responsible for the work, has any material interest in Tharisa, the companies associated with this project, their subsidiaries or their mineral properties. All work done by Coffey for Tharisa, is strictly in return for professional fees. Payment for the work is not in any way dependent on the outcome of the work or on the success or otherwise of Tharisa’s own business dealings. There is no conflict of interest in Coffey undertaking this work as contained in this document.

Ownership and Property Description

The Tharisa Mine a platinum group metal (PGM) and Chrome Mining Operation exploiting the Middle Group (MG) Chromitite Layers on two properties, being portions of the property Farm 342JQ and the whole of the property Rooikoppies 297JQ, located in the North West Province some 35km east of the city of Rustenburg and 95km from Johannesburg (Figure 1). The Tharisa Mine was developed by

Tharisa Minerals (Pty) Ltd (Tharisa Minerals) which holds a mining right, granted by the Department of Mineral Resources (DMR) on 19 September 2008 and registered on 13 August 2009, to various portions of Farm 342JQ (in respect of PGMs, gold, silver, nickel, copper and chrome ore) and Rooikoppies 297JQ (PGMs, gold, silver, nickel, copper and chrome ore contained within the MG Chromitite Layers only).

Figure 1
Location of the Tharisa Mine



A main road bisects the property in a north-south direction. The road provides access to the town of Marikana. The nearest major road, the N4 National Road links Pretoria with Rustenburg and crosses the south-eastern corner of the Farm 342JQ property immediately south of the outcrop of the Middle Group (MG) Chromitite Layers. The east west Rustenburg-Brits railway line bisects the Rooikoppies property with a station located in the town of Marikana on the Rooikoppies property.

History of the Tharisa Mine Ownership

Thari Resources (Pty) Ltd (Thari) which was incorporated in January 2005, acquired prospecting rights for chrome and PGMs over various portions of the property Farm 342JQ and to the property Rooikoppies 297JQ in March 2006. Thari is a Historically Disadvantaged South African (HDSA) and woman controlled company focused on the minerals and energy sectors.

In March 2006 Thari established Tharisa Minerals as a wholly owned subsidiary. During September 2008, February 2009 and March 2009 the prospecting rights held by Thari were transferred to Tharisa

Minerals after obtaining the necessary Ministerial approval in terms of Section 11 of the Mineral and Petroleum Resources Development Act, 2002 (MPRDA).

In March 2008, the mining rights for chrome ore, over portions 96 and 183 of Farm 342JQ were purchased from South African Producers and Beneficiators of Chrome Ore (Pty) Ltd. On 19 September 2008, the prospecting rights, including those for PGM and chrome ore, over various portions of Farm 342JQ and the whole of Rooikoppies held by Tharisa Minerals, were converted into mining rights in terms of Section 16 of the MPRDA.

Tharisa plc was incorporated in February 2008 and after obtaining the necessary Ministerial approval acquired 74% of Tharisa Minerals on 9 February 2009. The remaining 26% is currently held by Thari (20%) and the Tharisa Community Trust (6%). In July 2011 the Tharisa Minerals mining right 49/2009 (MR) was amended in terms of Section 102 of the MPRDA to include portions 96 and 183 of Farm 342JQ in respect of PGM, and to include PGM and chrome ore in respect of portion 286 of Farm 342JQ.

The Tharisa Mine started trial mining in October 2008 and commenced production of ore on a small scale from March 2009, achieving an average throughput rate of 38,000 tpm Run of Mine (RoM) ore with a small chrome concentrator. From 2010 to 2012 the mine undertook a number of process facility expansions to increase processing capacity to 400,000 tpm RoM ore).

Tharisa plc was listed on the Johannesburg Stock Exchange and commenced trading on 10 April 2014.

Current Mining Operations

The mining operation is divided into the east pit and west pit, located on either side of the Sterkstroom River that runs north-south through the Tharisa Mine (Tharisa) property. The pits are designed to protect the water course and the local infrastructure running parallel to the river (Figure 5). The east pit extends to the eastern boundary of the mining right while the west pit extends to where the Mineral Resource is defined on the far western portion of the mine. MCC Contracts (Pty) Ltd is the appointed mining contractor and has extensive open pit contract mining experience in Africa.

Tharisa produces largely fresh material from four groups of the MG Chromitite Layers, namely, MG4 (MG4A and MG4), MG3, MG2 and MG1. Some mining occurred on the UG1 Chromitite Layer in the past. The shallow MG1 was mined underground, by the previous mining right holder, to a limited extent on the eastern boundary of the property. Currently, no mining is conducted on MG0 Chromitite Layer.

The mining schedule is co-ordinated to match the capacity of the processing facility. At steady state Tharisa will mine and process 5.0Mtpa RoM ore.

The open pit operations maintain planned production levels until 2029, then transitions to underground bord and pillar mining. The last open pit tonnage is mined in 2036.

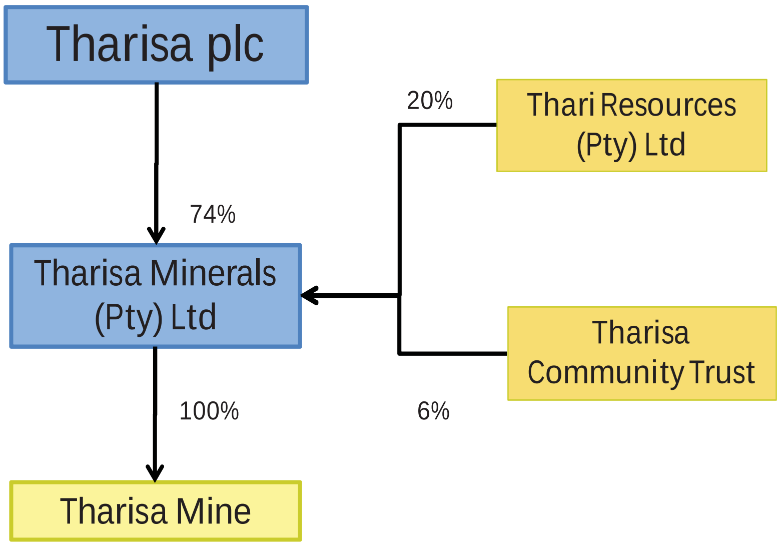
The open pit design and schedule including the mine design and scheduling of the future underground operation, was undertaken by Ukwazi Mining Solutions (Pty) Ltd (Ukwazi). The two schedules were combined into a joint production schedule.

Legal Aspects and Legal Tenure

The Tharisa Mine was developed by Tharisa Minerals which holds a mining right, granted by the DMR on 19 September 2008, to various portions of the property Farm 342JQ and to the property Rooikoppies 297JQ.

The corporate holding structure of the Tharisa Mine is represented in Figure 2.

Figure 2
Corporate Holdings Structure for the Tharisa Mine



Geology and Mineralisation

The Tharisa Mine is situated on the south-western limb of the Bushveld Complex and is underlain by the Middle Group (MG) Chromitite Layers.

The MG Chromitite Layers outcrop on Farm 342JQ striking roughly east - west and dipping at 12 -15° to the north. Towards the western extent of the outcrop, the dip is steeper, with a gentle change in strike to NW-SE. The stratigraphy typically narrows to the west and the dip steepens. The dip typically shallows out at depth across the extent of the mine area.

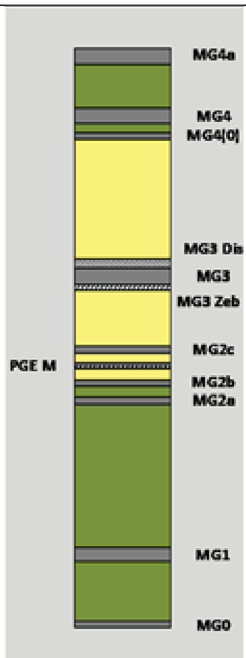
The MG Chromitite Layer package consists of five groups of chromitite layers being the MG0 Chromitite Layer, MG1 Chromitite Layer, the MG2 Chromitite Layer (subdivided into C, B and A chromitite layers), the MG3 Chromitite Layer and the MG4 Chromitite Layer (subdivided into MG4(0), MG4 and MG4A Chromitite Layers) (Figure 3). The layers between the chromitite layers frequently include stringers or disseminations of chromite. The structural interpretation of the Tharisa Mine is based on the aeromagnetic data and the drilling data. The MG Chromitite Layers at the Tharisa Mine are a typical stack of tabular deposits (Figure 3 and Figure 4).

The Upper Group (UG) 1 Chromitite Layer ranges between 165m to 18m stratigraphically above the MG4A Chromitite Layer on the Farm 342JQ property and 163m (down dip) to 18m (near surface) on the Rooikoppies property. The UG1 Chromitite Layer outcrops on the Farm 342JQ property. Both the UG2 Chromitite Layer (which ranges between 300m to 150m above the MG4A Chromitite Layer) and Merensky Reef (which ranges between 400m (east) to 290m (west) above the MG4A Chromitite Layer) outcrop on the Rooikoppies property. Poorly developed chromitite layers below the MG Chromitite Layer were intersected in boreholes and are interpreted as the Lower Group (LG) Chromitite Layers.

Figure 3

Summary of Stratigraphic Units modelled

Stratigraphic Column

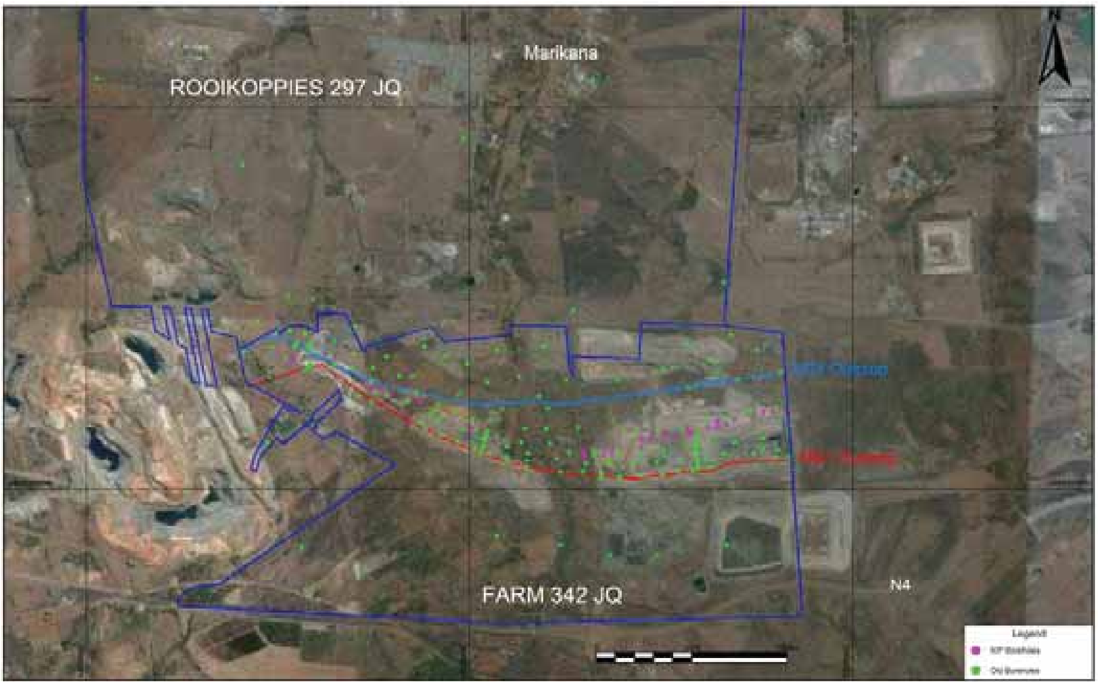


Unit	
MG4A Chromitite Layer	
MG4 Chromitite Layer	**MG4 Chromitite** **MG4 – MG4 (0) Parting** **MG4(0) Chromitite**
MG3 Chromitite Layer	**MG3 Disseminated** **MG3 Chromitite** **MG3 Zebra**
MG2 Chromitite Layer	**MG2C Chromitite** **PGEM Hangingwall** **PGE Marker** **PGEM Footwall** **MG2B Chromitite** **MG2A – MG2B Parting** **MG2A Chromitite**
MG1 Chromitite Layer	
MG0 Chromitite Layer	

The structural interpretation of the Tharisa Mine was previously based on the aeromagnetic data and the drilling data. The only significant fault is a steeply dipping NW-SE trending normal fault with a downthrow of less than 30m to the east. This fault occurs only on the far north-eastern corner of the property and will have little effect on mining of the MG Chromitite Layers on Farm 342JQ. This fault was confirmed in both Lonmin plc (Lonmin) underground operations and Samancor stopes located immediately east of the mine. A NE-SW sub-vertical dyke of some 10m thickness was interpreted on the aeromagnetic survey. This dyke was not fully intersected in any of the boreholes but has been intersected in the East Mine box-cut and is 11m wide. The dyke is not expected to have a major impact on mining. The only other major feature of interest is the Spruitfontein upfold or pothole which is located on the properties immediately west of the mine. It affects the UG2 Chromitite Layer as well as the rest of the Critical Zone below. The area around the pothole is on the adjacent property and was not accessible for further investigation.

Figure 4

Google Image of the Tharisa Mine plan showing borehole locations and outcrop positions of UG1 and MG1 Chromitite Layers



The UG1 Chromitite Layer is stratigraphically situated in the Upper Critical Zone and is well developed in the Bushveld Complex. It comprises of massive chromitite, chromitiferous pyroxenite, bands of anorthosite, chromitite, norites and stringers of chromitites. The UG1 Chromitite Layer has an east-west strike and dips to the north. The dip angle varies from 10º in the east to 25º in the west. The thickness of the UG1 Chromitite Layer ranges from a few centimetres up to 3m in places. The lenses of anorthosite and pyroxenite are seen impregnated with numerous chromite grains in places. The hanging wall changes from pyroxenite to anorthositic norites. The footwall is formed by bifurcated bands of anorthosite and chromite lenses.

Exploration and Geology

The Tharisa Mine has been explored for its mineral potential since the early 1900s. Initially this was in the form of erratic exploration activities which included trenching and small open pits.

Various trenches were excavated on both the UG1 and the MG Chromitite Layers. The MG Chromitite Layers were previously exploited from three known pits, excavated by previous tenement holders and which remain unrehabilitated.

Six diamond boreholes were drilled during January 1997 by an entrepreneur, Mr Hennie Botha in the northwest part of Farm 342JQ property and on the adjacent property, Spruitfontein 341JQ. Five NQ size, vertical diamond boreholes were drilled along strike on Farm 342JQ during 2006 by Thari Resources under the supervision of Coffey. A total of 121 vertical boreholes and 23 deflections, representing some 22,500m were drilled from March 2007 to October 2007. The drilling programme was designed so that boreholes would intersect the base of the MG1 Chromitite Layer at approximately 30m, 60m, 120m, 180m, 300m, 500m and 1000m below surface. A line of boreholes that intersected at 220m below surface was later added for greater coverage of the deposit. The drilling programme was designed to drill the deposit closest to the outcrop at higher density than further downdip so that the subsequent mineral resource estimate close to the outcrop could confidently be declared as an indicated and/or measured mineral resource in preparation for a feasibility study and the consideration of open pit mining. The programme for the deeper boreholes on the Rooikoppies property, where Lonmin is mining the Merensky Reef and UG2 Chromitite Layer, was revised due to various difficulties relating to the siting of boreholes to prevent holing into existing underground infrastructure. Fewer, more widely spaced boreholes were therefore drilled.

Two fence lines (down dip) were drilled with TNW core size for metallurgical test purposes, intersecting the chromitite layers at 10m depth increments down to 60m below surface on the MG4 Chromitite Layer. Two NQ boreholes were drilled for geotechnical logging, sampling and to conduct rock strength tests. Six boreholes were drilled around the proposed civil engineering sites which coincide with the LG6 Chromitite Layer outcrop to ensure that a possible economical deposit was not being sterilised. A total of 10 boreholes were drilled on the Rooikoppies property to test the extension of the MG Chromitite Layer package down dip.

The collars of all the boreholes were surveyed. Downhole surveys were completed for all the boreholes drilled to a depth greater than 120m. All geological and sampling protocols used are to international standards. The precious metal analyses (Pt, Pd, Rh, Au, Ru, Ir, Os) were undertaken using NiS/MS analytical method and base metals analysis using the ICP Fusion D/OES analytical method, at Genalysis (Johannesburg).

A comprehensive quality assurance and quality control (QA/QC) programme was carried out concurrent with drilling. This included three certified reference standards, blanks and field duplicates. Each quality control aspect used was introduced in a ratio of 1:20. All assay issues were resolved and the assay data confirmed to be reliable and acceptable for a mineral resource estimate.

Some 35 boreholes (21 boreholes from East Pit and 14 from West Pit) were selected in 2016 (Figure 4). The boreholes were completely reclogged and sampled with the samples being sent to Intertek Genalysis Laboratory Services in Johannesburg for assay. The precious metal analyses (Pt, Pd, Rh, Au, Ru, Ir, Os) were undertaken using NiS/MS analytical method and base metals analysis using the XRF Fusion method. An appropriate QA/QC programme was instituted.

The geological modelling confirmed the tabular nature of the deposit and identified the major structural features (dykes and faults). The models were validated to ensure that the stratigraphic integrity was maintained. The result is five planar surfaces stacked on top of each other demonstrating the tabular nature of the deposit. The geological modelling utilised the other structural information gained from the aeromagnetic survey, trenching etc. It was noted that the dip flattens with depth.

Mineral Resource

The mineral resource estimate was completed over the mining right of Tharisa Minerals to a depth of 750m for the MG Chromitite Layers and UG1Chromitite Layer:-

- MG4A Chromitite Layer
- MG4 Chromitite Layer consisting of the MG4(0) and MG4 Chromitite Layer with the parting between them
- MG3 Chromitite Layer with the disseminated material above and the disseminated chromitite below ("zebra")
- MG2 Chromitite Layer including the MG2A, MG2B, MG2C Chromitite Layers, the parting between the MG2A and MG2B Chromitite Layers as well as the PGM layer between the MG2B and MG2C Chromitite Layers and the associated partings
- MG1 Chromitite Layer
- MG0 Chromitite Layer
- UG1 Chromitite Layer

<u>MG Chromitite Layer</u>

The data was coded for the different units within the MG and UG1 Chromitite Layer packages. Statistical analysis was then completed on both the raw and composite data grouped by unit type after examination of the data indicated that the units defined different geological populations and are statistically distinct.

Each intersection was composited after coding for all stratigraphic layers. The Pt, Pd, Rh, Au, Ru, Ir, Os, Cu, Ni, Al, Ca, Cr, Cr_2O_3, Fe, Mg and Si concentrations were composited utilising the weighting by thicknesses and densities. An analysis of the unit thickness showed that there is little correlation between the concentration and thickness confirming that the use of concentration was appropriate in the mineral resource estimate.

An assessment of the high-grade composites was completed to determine whether high-grade cutting was required. Based on the above assessment, no high grade cutting or capping was undertaken.

A series of two-dimensional grade estimates were generated based on geologically and geochemically defined units within the MG Chromitite Layer cycle. The mineral resource estimation was completed using either an inverse distance (power 2) method for each variable within each unit could be modelled. The method of estimation was determined after consideration of the data and after unsuccessfully being able to generate variograms for the various components.

The concentration of Pt (g/t), Pd (g/t), Rh (g/t), Au (g/t), Ru (g/t), Ir (g/t), Os (g/t), Cu (ppm), Ni (ppm), Al/Al_2O_3 (%), Ca/CaO (%), Cr/Cr_2O_3 (%), Fe/Fe_2O_3 (%), Mg/MgO (%), Si/SiO_2 (%), K_2O (%), MnO (%), NaO_2 (%), P_2O_5 (%), TiO_2 (%) and V_2O_5 (%) for each of the units identified within the MG Chromitite Layers utilising the composite grade over the thickness of that unit (seam model approach). In addition the bulk density was estimated for each unit.

A block size of 100m x 100m was selected. The search criteria included an isotropic search volume of 1000m that expanded to 1500m then 3000m if the criteria of a minimum of four and a maximum of 12 composite data for each block estimate were not met.

After examination of the various geological features where the MG Chromitite Layers are not developed viz. dykes, faults, potholes, mafic pegmatites, the geological loss for the East Pit was set at 5%, 7.5% for the West Pit and 15% for the Far West Pit. A geological loss of 15% was applied for the remaining areas.

The classification of the mineral resources was undertaken in accordance with the guidelines of the SAMREC Code (2016). The Competent Persons responsible for the mineral resource estimation and classification is Mr Ken Lomberg Pr.Sci.Nat..

<u>UG1Chromitite Layer</u>

The UG1 Chromitite Layer comprises a top chromitite layer, a middling (pyroxenite/anorthosite) and a bottom chromitite layer. It was necessary to model these individual layers separately due to their different geochemical characteristics.

The East and West Mine areas were modelled independently as it was noted that they are of different populations. The East Mine was further divided into two domains due to geology and grade considerations in the far eastern side. In total seven datasets were distinguished and modeled independently i.e. West (top, middling, and bottom), East (top, middling and bottom) and Far East (one model).

As a result of the confidence in the geological model, each of the stratigraphic units was estimated independently as a layer and a hard boundary was used. Each of the (Al_2O_3 (%), CaO (%), MgO (%), Fe_2O_3(%), K_2O(%), MnO (%), Na_2O(%), P_2O_5(%), Cr_2O_3(%), Pt (g/t), Pd (g/t), Rh (g/t), Ru (g/t), Ir (g/t), Au (g/t), width(m) and density values were estimated independently using inverse power of distance (power of 2).

The classification of the mineral resources was undertaken in accordance with the guidelines of the SAMREC Code. The Competent Person responsible for the mineral resource estimation and classification is Mr Ken Lomberg Pr.Sci.Nat.

In classification of the mineral resource estimate for the UG1 Chromitite Layer, consideration was given to the reasonable prospects for eventual economic extraction. As a result the declaration was made only for the areas where MG Chromitite Layer mining is anticipated to occur in the planned open pit. The expansion of the declaration will require a financial assessment incorporating the potential movement of dumps and other surface infrastructure.

The mineral resource estimates for the MG and UG1 Chromitite Layers were estimated with an effective date 30 September 2016 (Table 1).

Mining Engineering

A feasibility study was concluded in October 2008. Various revisions to the mine plan were undertaken to match the requirements of the processing facilities and included both open pit and underground mine design and scheduling.

The open pit operation targeted MG1, MG2, MG3, MG4 and MG4A Chromitite Layers in an operation that was split into a West pit and an East pit. The mine was planned for two phases, an initial open pit mine followed by an underground mining operation. The open pit plan was based on fixed contract rates and volumes as determined through a detailed planning process. Based on a maximum of a 200m high wall, at an average of 430ktpm production profile, the open pit operation maintains planned production levels until 2029 before mining underground starts. The last open pit tonnage is mined in 2035. The underground mining target the MG2 and MG4 Chromitite layers and starts towards the end of the open pit operations from 2029 onwards to ensure a steady production profile throughout the LOM.

Table 1

Mineral Resource Statement for the Tharisa Mine (30 September 2016)

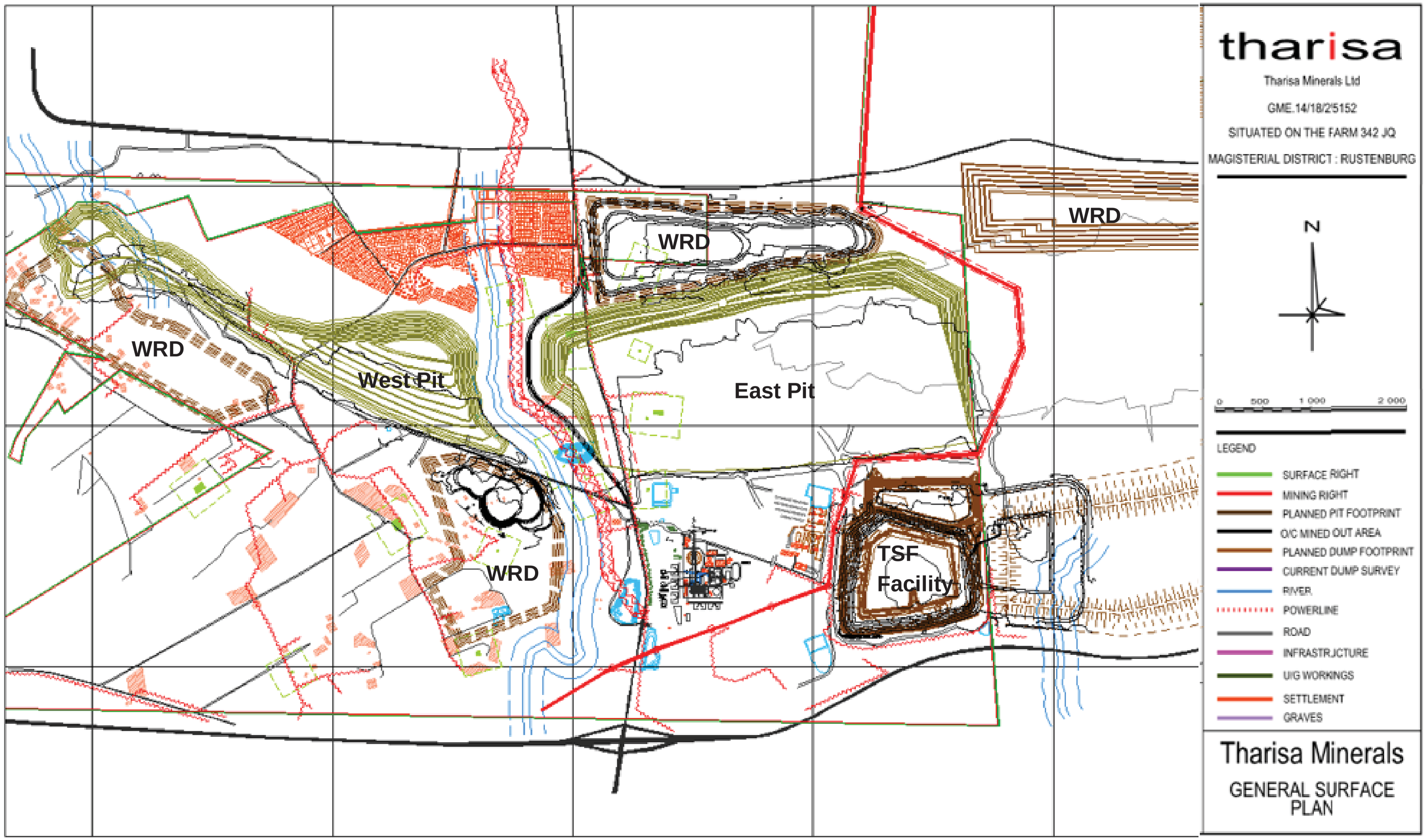
MG4A CHROMITITE LAYER																	
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE +Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	10.163	1.52	3.67	23.35	0.40	0.15	0.12	0.00	0.25	0.04	0.05	0.67	59:23:18:0	1.02	39:15:12:0:25:4:5	1.17	333	767
Indicated	13.515	1.54	3.69	26.08	0.43	0.17	0.14	0.00	0.28	0.05	0.06	0.74	58:23:18:0	1.13	38:15:12:0:25:4:5	1.31	490	770
Inferred	70.400	1.48	3.70	22.64	0.36	0.14	0.12	0.00	0.24	0.04	0.05	0.62	58:23:19:1	0.95	38:15:12:0:25:4:5	1.30	2,143	682
MG4 and MG4(0) CHROMITITE LAYER Package																	
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE +Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	16.450	2.41	3.74	27.00	1.11	0.25	0.22	0.00	0.39	0.08	0.12	1.58	70:16:14:0	2.17	51:11:10:0:18:4:5	1.29	1,145	801
Indicated	25.787	2.96	3.67	25.02	1.07	0.23	0.21	0.00	0.37	0.08	0.11	1.51	71:15:14:0	2.07	52:11:10:0:18:4:5	1.20	1,712	737
Inferred	178.033	3.86	3.56	22.69	0.97	0.18	0.19	0.00	0.34	0.07	0.10	1.34	72:14:14:0	1.86	52:10:10:0:18:4:6	1.14	10,643	700
MG3 CHROMITITE LAYER																	
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE +Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm))
Measured	12.680	2.07	3.47	20.18	0.82	0.50	0.21	0.01	0.31	0.06	0.08	1.54	53:33:14:0	1.99	41:25:10:0:15:3:4	1.18	810	675
Indicated	19.769	2.49	3.45	19.38	0.80	0.49	0.21	0.01	0.29	0.06	0.08	1.50	53:33:14:0	1.93	41:25:11:0:15:3:4	1.11	1,228	649
Inferred	89.191	2.05	3.53	21.15	0.86	0.50	0.23	0.00	0.32	0.06	0.09	1.60	54:31:14:0	2.07	42:24:11:0:16:3:4	1.17	5,938	689
MG2 CHROMITITE LAYER																	
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE +Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	22.787	3.82	3.61	20.35	1.13	0.30	0.16	0.00	0.28	0.05	0.08	1.59	71:19:10:0	2.01	56:15:8:0:14:3:4	1.07	1,470	753
Indicated	33.587	4.32	3.58	18.17	0.99	0.29	0.15	0.00	0.25	0.05	0.07	1.43	69:20:10:0	1.80	55:16:8:0:14:3:4	0.99	1,946	739
Inferred	283.454	6.56	3.50	14.33	0.76	0.23	0.12	0.00	0.20	0.04	0.06	1.11	69:20:11:0	1.41	54:16:8:0:14:3:4	0.82	12,876	689

MG1 CHROMITITE LAYER																	
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	6.839	1.30	3.84	32.74	0.32	0.21	0.11	0.00	0.45	0.07	0.07	0.63	50:33:17:1	1.23	26:17:9:0:37:6:6	1.43	271	803
Indicated	10.096	1.10	3.89	32.45	0.33	0.21	0.11	0.00	0.46	0.08	0.07	0.66	51:32:17:1	1.27	26:17:9:0:36:6:6	1.45	413	806
Inferred	69.487	1.54	3.85	30.59	0.33	0.19	0.10	0.00	0.43	0.07	0.07	0.62	53:30:17:1	1.19	28:16:9:0:36:6:6	1.44	2,656	778
MG0 CHROMITITE LAYER																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	3.69	0.52	3.71	25.54	0.46	0.16	0.15	0.00	0.28	0.05	0.06	0.77	60:20:19:1	1.16	40:13:13:0:24:4:5	1.25	137	740
Indicated	5.78	0.59	3.70	25.98	0.59	0.18	0.16	0.00	0.32	0.05	0.07	0.95	63:19:17:0	1.39	43:13:12:0:23:4:5	1.27	258	735
Inferred																		
UG1 CHROMITITE LAYER																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured																		
Indicated	3.54	2.24	3.69	23.68	0.86	0.29	0.17	0.00	0.32	-	0.07	0.72	46:31:19:5	1.16	28:19:12:3:32:0:6	1.14	133	-
Inferred	2.45	3.35	3.68	22.75	0.81	0.28	0.17	0.01	0.32	-	0.07	0.66	41:35:18:6	1.05	26:22:12:4:31:0:6	1.11	83	-
TOTAL MINERAL RESOURCE																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	72.612	11.6	3.67	23.68	0.86	0.29	0.17	0.00	0.32	0.06	0.08	1.32	65:22:13:0	1.78	48:16:10:0:18:3:5	1.20	4,166	756
Indicated	112.073	15.3	3.72	22.75	0.81	0.28	0.17	0.01	0.32	0.06	0.08	1.26	64:22:13:0	1.72	47:16:10:0:18:3:5	1.17	6,180	732
Inferred	693.009	18.8	3.62	19.85	0.74	0.24	0.15	0.00	0.28	0.05	0.07	1.13	66:21:13:0	1.54	48:15:10:0:18:4:5	1.07	34,339	700
Total	**877.694**	**17.9**	**3.63**	**20.54**	**0.76**	**0.25**	**0.15**	**0.00**	**0.29**	**0.06**	**0.07**	**1.16**	**65:21:13:0**	**1.58**	**48:16:10:0:18:4:5**	**1.09**	**44,685**	**709**

Note: The mineral resource is declared to a depth of 750m below surface.
Grades and tonnages are reported at shaft head
The consideration of realistic eventual extraction necessitates that the mineral resource considers the MG Chromitite Layer to be a geological unit and that all platiniferous and chromiferous horizons will be mined and all PGM, Cu, Ni and Cr_2O_3 recovered.
The UG1 Chromitite Layer is declared for the part that falls within the current proposed open pit
The mineral resource is reported inclusive of the mineral reserve

Figure 5

Map showing the Mining Layout



<u>Open Pit</u>

The pit optimisation was undertaken in 2016 using Geovia Whittle (Whittle) pit optimisation software. A pit optimisation process was completed taking into consideration the updated mining, cost, revenue and financial parameters. The results confirmed that the current East pit high wall with minimal pit design changes required for the West pit. The Far West pit increased in size due to additional resources that were included towards the south west.

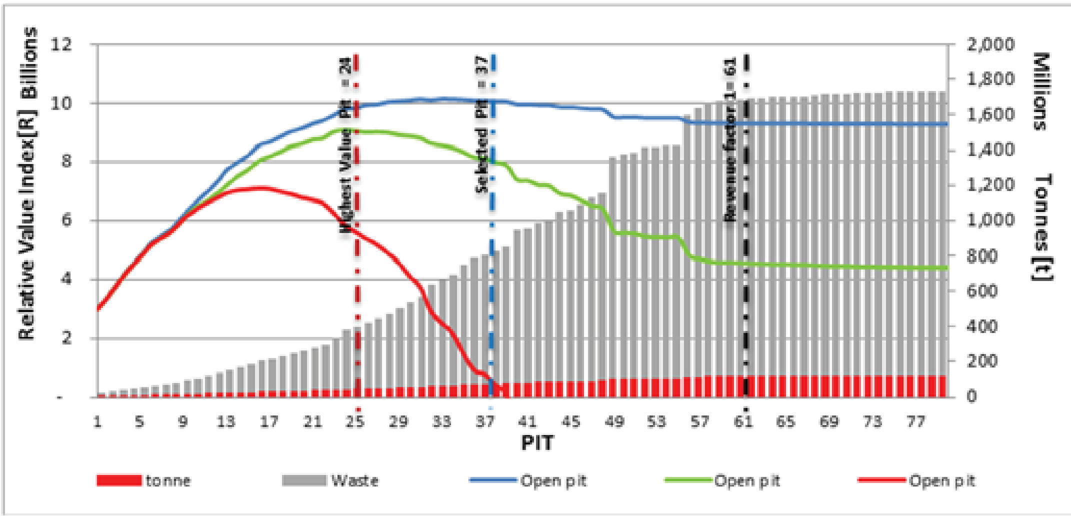
The Whittle optimisation process identifies which blocks should be mined and which ones should be left in the ground. The pit optimisation parameters considered were:

- Pit slope angles
- Mining related modifying factors
- Physical characteristics
- Mining parameters
- Processing cost and revenue parameters.

The pit shell selection strategy applied considers the relative value of the selected case, but maximises the life of the operation. Pit 37 was selected and contains a total of 73.4Mt ROM ore. The selected pit perimeter shell is in line with the current infrastructure placement and previous optimisations conducted on the incremental pit analysis at a maximum high wall depth of 200m. Figure 6 shows the Whittle pit-by-pit graph.

Figure 6

Graphic showing the Relative values of the Whittle Pit optimisation



The ultimate pit was selected using the high monetary value and extended life pit selection strategy as per Tharisa's corporate strategy. The strategy considered the optimum relative value and extended life based on the specified case of the Whittle pit-by-pit graph. Pit shell 37 was selected and contained 73.4Mt ROM ore and 729Mt of waste with a stripping ratio of 9.9t/t basis.

Mining is undertaken by an established mining contractor with a track record on similar operations. Mine planning is conducted in conjunction with the mining contractor to ensure that operational plans are achieved.

The mining related modifying factors applied were based on study work, testwork, observation and measurement. A geotechnical slope angle of 45º, with a 10m safety berm at an overall slope angle of 35º was used for the top 20m of topsoil and soft overburden while an average overall 53º slope angle was applied at depth. Geological losses were applied at 5% in the less steeply dipping eastern section where more information existed, whilst a 7.5% geological loss was applied around the current mining areas in the West pit. A 15% geological loss was applied on the much steeper dipping further western area in the West pit. The geological loss accounted for unknown geological features that resulted in a loss of available Mineral Resources. The total of 6% mining losses was based on the available Mineral Resource mined, with losses allowed for drilling, blasting and loading activities. External dilution was applied based on the mining methodology employed per ore layer. The wider MG Chromitite Layers that are blasted with the surrounding waste and loaded selectively attracted a higher dilution percentage while the MG1 layer which is mined selectively attracted a lower dilution percentage. The average dilution applied amounted to 20.3% measured on a tonnage basis. A reconciliation concluded that an improvement of 10% on dilution is required to achieve the targeted dilution percentages. Excessive losses and dilution pose a material risk and have a material negative effect on the profitability of the operation.

Excavators (65t to 90t class) are used to load 40t to 80t class articulated dump trucks in the chromitite layer and waste parting zones. The ROM ore is hauled directly from the pit to the ROM pad or placed on a designated stockpile or fed directly through the mobile primary crusher and sized to -200mm. Mining operations in West pit is restricted to day-light hours compared to the 24 hour operation in East pit. East pit is equipped with appropriate lighting plants on each production face with quality control enforced by grade control technicians.

Bulk waste above the MG4A Chromitite Layer is excavated using 360t excavators and hauled with 150t dump trucks. Haul roads were designed at a maximum inclination of 10% and with a width of 30m, taking into consideration the 150t truck dimensions for safe two-way traffic.

Operational costs used in the optimisation process were based on the plant and infrastructure operational budget, overheads and contractual mining rates. PGM metal prices were adjusted to incorporate the off take terms and the government royalty.

Plant recoveries were based on actual performance while capacities were based on design capacity. The PGM recoveries on oxidised and fresh ore are shown in Figure 7. The mass yield applied was based on the supplied yield curves as indicated in Figure 8.

Bulk waste is blasted in 20m benches. Depending on the dump location, waste is hauled to either the dump located on the outcrop side or hauled through temporary ramps on the interim highwall to a dump located on the highwall side of the pit. Backfilling is maximised but remains 100m behind the active working faces at all time. An estimated 47% of the waste is backfilled over the life of the operation. It must be noted that, due to the lowwall ramps and a minimum 100m down dip lag between the backfill and the working faces, the 46% backfill is reasonable and in line with similar operations.

Figure 7

Graph of the PGM Grade Recovery Curves

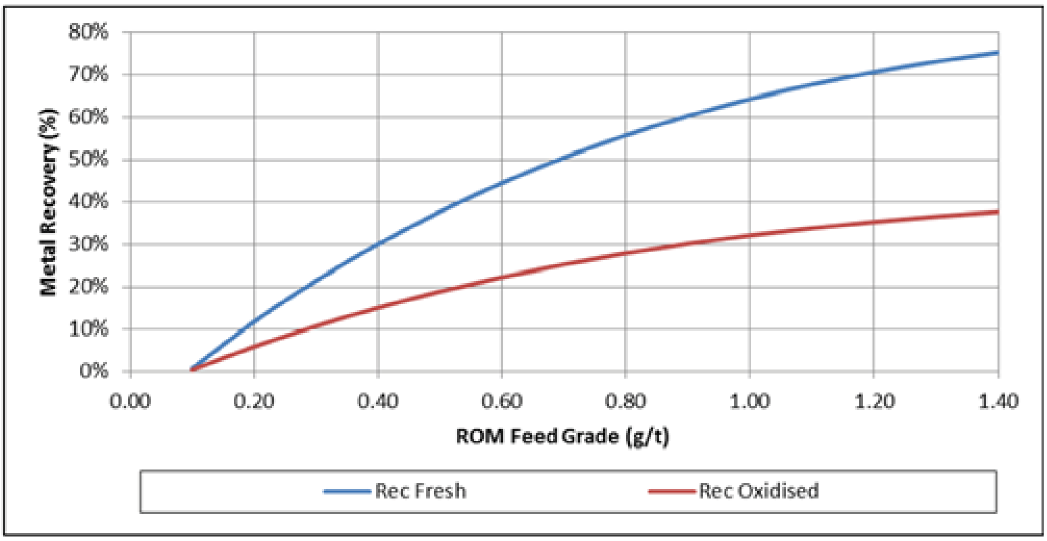
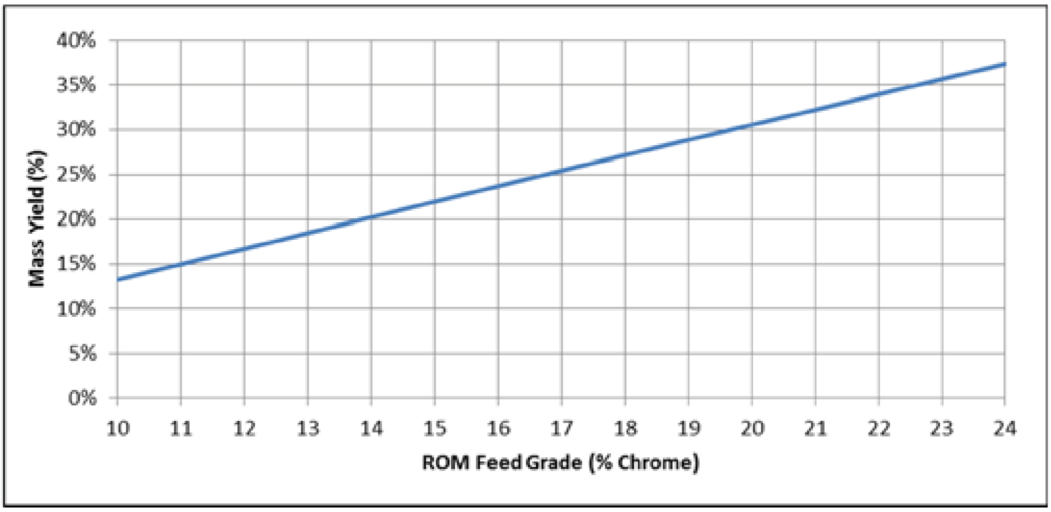


Figure 8

Graph of the Chrome Grade Recovery Curves



Steady state waste stripping requirements were set at approximately 1.4 million bank cubic metres per month (BCM/m) from the two pits. A total of 430ktpm ROM ore is produced from the two pits. Steady state production levels are maintained from the open pits up to 2029 with a gradual ramp up of production from underground sources. The last open pit tonnage is mined in 2035.

A total of 54.3Mt of Proved Mineral Reserve and 26.0Mt of Probable Mineral Reserve is declared for the open pits (Table 2).

Table 2
Open Pit: Mineral Reserve Estimation Summary
(30 September 2016)

Reserve category	Tonnes (Mt)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	3PGE+Au (g/t)	5PGE+Au (g/t)	Cr_2O_3 (%)
Proved	54.2	0.77	0.24	0.15	0.004	1.17	1.53	21.1
Probable	26.0	0.72	0.23	0.13	0.004	1.09	1.42	18.6
Total	**80.2**	**0.75**	**0.24**	**0.14**	**0.004**	**1.14**	**1.49**	**20.3**

Note: The Mineral Reserve is declared in terms of the guidelines of the SAMREC Code
The reserve does not report Os as it typically not included in the revenue generated from the sale of PGEs.

Underground Mine Design

An underground mining study was conducted as part of the 2013 CPR. No subsequent study work was completed. The sections contained in this document describing the underground mining and design methodologies are an extract of the 2013 report.

Small portions exist within the mine design for which Tharisa does not currently hold the mining right and/ or where the surface rights have yet to be acquired. These areas were not excluded from the mine design based on the reasonable expectation that exists that the necessary permitting and ownership could be in place by the time mining is undertaken in these areas.

The minimum strategic design requirements for the underground section was a ROM production of 400ktpm as a continuation of the open pit production profile with sustained production levels during the transition period. The health and safety aspects considered must provide for a low safety risk and profitable underground mining.

To successfully define a single go-forward case for the mining exploitation strategy, the mining method, access selection, mine design, scheduling, mining equipment selection, and the preparation of an operational and capital cost schedule up to steady state production was considered. The footprint area for underground mining was constrained by the open pit perimeter and crown pillar to the south, the 750m depth cut-off to the north and the mining right boundaries to the east and west. The overall exploitation strategy applied was to maximise the economic open pit limits followed by underground mining from the pit high walls.

A mining method selection study was undertaken to evaluate the productivity, equipment suitability, capital costs, operating costs, environmental aspects, and health and safety risks associated with various methods. A trackless bord and pillar was selected as the preferred mining method. Bord and pillar mining is widely employed for the extraction of similar flat dipping deposits with the advantages that:

- Development rates are faster compared to conventional systems
- The mining method offers good flexibility in terms of dealing with geological and quality anomalies
- Safety is enhanced as fewer people are involved and most of the work is conducted from the protection of machinery
- Personnel, equipment and consumables are moved efficiently and almost directly to the working faces
- Shift change-over times are reduced
- Supervision is improved and working places can be visited with less effort compared to conventional methods.

An analysis was undertaken to select the appropriate mining horizons. From this analysis, MG2 and MG4 Chromitite Layers were selected. After further scrutiny, it was concluded that MG2C Chromitite Layer must be excluded from the mining cut to reduce internal dilution and only MG2A and MG2B Chromitite Layers and the waste parting are planned to be mined. The combined thickness of MG2A Chromitite Layer, waste parting and MG2B Chromitite Layer in the greater part of the underground design area is in excess of 1.8m and meets the minimum requirements of the equipment selected. A further constraint was applied that the maximum mining width must not exceed 4m. The mining cut was re-stated for MG2A only, taking the minimum width into consideration.

The MG4 Chromitite Layer, at an average in situ thickness of 3.0m, was selected as the second mining horizon as it was wide enough for trackless bord and pillar mining. The parting between MG2 and MG4 Chromitite Layers varys between 15m to 20m thick. The selected mining cut included MG4 Chromitite Layer, the pyroxenite parting and MG4(0) Chromitite Layer below. The same maximum and minimum width criteria were used. Where the MG4 Chromitite Layer package thickness exceeded 4m, only the MG4 Chromitite Layer was selected for the mining cut.

Mining extraction in the bord and pillar mining method was achieved by developing a series of roadways (rooms or bords) on the chromitite layer and connecting them by holings or cut-throughs to form pillars that provide support for the overlying strata. Mining extraction in this method is a function of the pillar sizes which is a function of the depth below surface.

To accommodate the equipment sizes, production requirements and geotechnical considerations, minimum and maximum mining cuts were set at 1.8m and 4.5m respectively. Layers thinner than 1.8m were diluted up to a minimum height of 1.8m in the production sections and 4.5m in the declines.

Access to the underground mine is gained through three sets of on reef declines. The advantages of this system are that all development is undertaken on the reef horizon, more information on the geology is obtained during development and waste development is required to access the chromitite layers.

The main disadvantage of this option is the lack of surge capacity. Two decline systems with a capacity of 150ktpm each were planned from the highwall of East pit for MG2 and MG4 Chromitite Layers respectively. Additional declines must be developed on MG2 from West pit highwall that services both MG2 and MG4 Chromitite Layers at a capacity of 50ktpm from each chromitite layer.

The geotechnical parameters considered for this study were based on the work conducted as part of the feasibility study concluded in 2008 and additional work completed in 2012. Initial pillar designs were modified in line with best practices employed at similar mines in the area. Consequently, pillar sizes of 6m x 6m on 8m bord spans and 6m holings were used in the stoping designs. The pillars were designed to increase with depth from 6m x 6m in the upper levels to 8m x 8m in the bottom stopes. Additional geotechnical modelling is required to refine these parameters in due course. This modelling must include a study of the waste partings between the layers to form the basis of possible future inclusion of portions of MG1 and MG3 Chromitite Layers.

The mining dilution factors were estimated from first principles to assume an overbreak of 10cm waste from both the hanging and footwall horizons of the mined Layer. Depending on the dip of the chromitite layer, some waste is mined to maintain safe and horizontal underfoot conditions. The dilution factors decrease with depth from 16% to 13% for MG2 Chromitite Layer and from 15% to 12% for MG4 Chromitite Layer. This is in direct proportion to the pillar sizes which increase with depth. Mining recovery for both horizons was set at the historical mining average for similar operations at 98%. The extraction is a function of the pillar sizes and was estimated from first principles. A decreasing trend with depth is shown from 79% in the upper levels to 71% in the lower levels for both chromitite layers.

Ten production sections are required to meet the planned 200ktpm ROM production for MG2 Chromitite Layer based on the LHD requirement estimate. A total of 12 production sections are required to meet the planned 200ktpm ROM production from MG4 Chromitite Layer. Based on a production profile of 400ktpm, the scheduled underground production commences with the production ramp up during FY 2029 and continues up to FY 2073, with an underground mine life of 24 years at steady state production.

The scheduling strategy, which is a key driver to the overall project costs and economic value, was set to establish the eastern decline system initially before moving to the western decline system. This strategy was chosen to minimise project risk by starting off with areas of higher geological confidence and chromitite layer thicknesses. The sinking of the MG2 Chromitite Layer east triple declines is set to start five years before the depletion of open pit operations. At the planned advance rates, the mining of the triple MG2 Chromitite Layer declines to Level 4, including the ledging and ventilation provisions, will be completed within 24 months with the ramp up to steady state within 48 months. Sinking and production ramp up for the MG4 Chromitite Layer declines will be executed over the next three years and steady state production of 400ktpm is expected in year five from project inception. This ramp up is timed to maintain production rates with the depletion of the open pit Mineral Reserves.

The underground operations will make use of some of the existing infrastructure established for open pit operations such as electricity, water, plant, houses, offices, transport and communications networks as this is operational when the underground operations are conducted. Additional infrastructure provided in the capital cost estimate includes:

- The ventilation network
- Underground workshops and fuelling facilities
- Pumping arrangements
- Washrooms and lamp room facilities
- Emergency facilities.

The mining operating costs were sourced mainly from an internal cost database and from relevant service providers. A total of 18.6Mt of underground RoM was declared as a Probable Mineral Reserve (Table 3).

Table 3
Underground Mine: Mineral Reserve Statement
(30 September 2016)

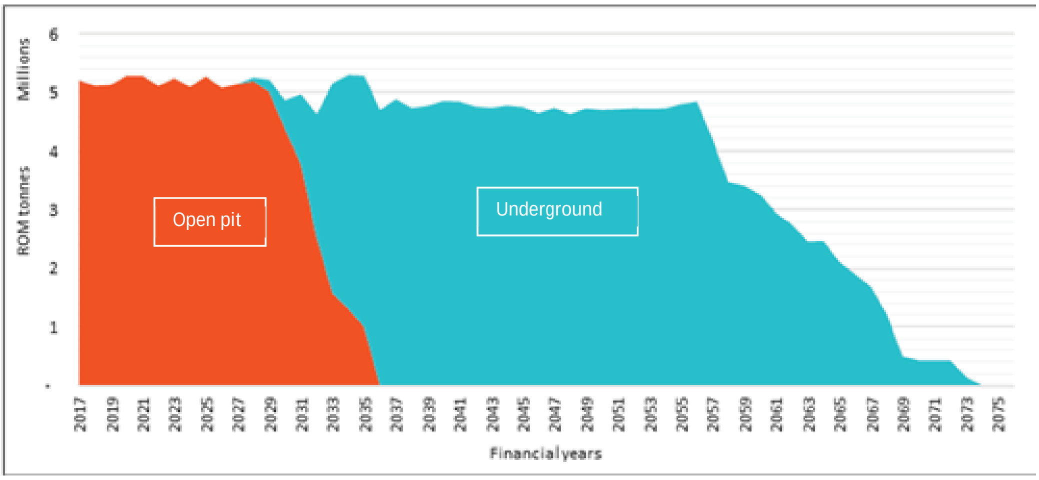
Reserve category	Tonnes (Mt)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	3PGE+Au (g/t)	5PGE+Au (g/t)	Cr_2O_3 (%)
Proved	-	-	-	-	-	-	-	-
Probable	18.6	0.82	0.19	0.15	0.002	1.17	1.52	19.3
Total Reserve	**18.6**	**0.82**	**0.19**	**0.15**	**0.002**	**1.17**	**1.52**	**19.3**

Note: *The Mineral Reserve is declared in terms of the guidelines of the SAMREC Code.*
The reserve does not report Os as it typically not included in the revenue generated from the sale of PGEs.

<u>Production Schedule</u>

The combined LOM schedule for the current open pit and planned underground operations is presented in Figure 9.

Figure 9
Integrated Mining Production Schedule



Mineral Reserves

Modifying factors were applied to the Mineral Resource to convert it to a Mineral Reserve. The modifying factors applied were geological losses at 5% in the less steeply dipping eastern section where more information existed whilst a 7.5% geological loss was applied around the current mining areas in the west pit. A 15% geological loss was applied on the much steeper dipping far western area in West pit, mining recovery (mining loss of 6%) and mining dilution (20.3% tonnage basis on average). Metallurgical recoveries were applied according to metal recovery curves for oxidised and fresh ore respectively and Cr_2O_3 recovery was based on the process recovery curve. The combined open pit and underground Mineral Reserve estimate is presented in Table 4.

Table 4
Total Mine: Mineral Reserve Statement
(30 September 2016)

Reserve Category	Tonnes (Mt)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	3PGE+Au (g/t)	5PGE+Au (g/t)	Cr_2O_3 (%)
Proved	54.2	0.77	0.24	0.15	0.004	1.17	1.53	21.1
Probable	44.6	0.76	0.22	0.14	0.003	1.12	1.46	18.9
Total Reserve	**98.8**	**0.77**	**0.23**	**0.15**	**0.003**	**1.15**	**1.50**	**20.1**

The reserve does not report Os as it typically not included in the revenue generated from the sale of PGMs.
5PGE = Pt+Ir+Ru+Rh+Pd

Geotechnical Engineering

On the most recent visit to the mine it was observed that the working pit slopes are stable and the benches and slopes conform to the design. No major risks were observed.

In the 2013 design study, data was collected from geotechnical logging in the then current east and central pits of Tharisa Mine to determine stable slope angles. Acceptable design methodologies were used to quantify the appropriate slope angles that will allow for safe and effective extraction of the resource. Slope angles of 45° in saprolitic material and 53° overall slope angles in fresh rock up to an overall slope height of 210m were shown to be stable. Kinematic analysis suggests a possibility for toppling failure. Catch berms must be maintained as instability is expected to be on a bench scale. Beside this potential minor mode of failure the safety factors are high.

An earlier geotechnical investigation was carried out by logging eight boreholes and sampling the lithological units prior to strength testing the samples. The pillar strengths and N' values for underground mining were calculated and from this pillar sizes and stope spans designed. Mining aspects require that the bord spans be limited to 6m. The planned support for the stoping and development has also been designed incorporating these design parameters.

On-going joint mapping using laser scanning is being undertaken to define hazardous areas.

Risk Summary

A summary of the perceived risks associated with the mine is presented in Table 5.

Table 5

Tharisa Minerals Technical Risk Summary

Item	Relative Risk
Geology and Mineral Resources	Low
Mining Engineering and Mineral Reserves	Low to Medium
Geotechnical Engineering	Low
Infrastructure	Low to Medium

Based on the above risk summary, Coffey considers the Tharisa Mine to have an overall **Low to Medium Risk**

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1 INTRODUCTION AND TERMS OF REFERENCE

1.1 Scope of the Report

Coffey Mining (South Africa) (Proprietary) Limited (Coffey) was requested by Tharisa plc, formerly Tharisa Limited (Tharisa or the Company), estimate the Mineral Resource and Mineral Reserve for the Annual Report 2016 in respect of the Tharisa Mine located in the North West Province, South Africa.

1.2 Site Visits

Dr James and Messrs Lomberg, Mans and Lotheringen have visited the property on a regular basis over a period of approximately nine years since 2007.

1.3 Competent Persons Report

This report has been prepared in accordance with the guidelines of "The South African Code for Reporting of Exploration Results, Mineral Resources and Reserves (The SAMREC Code) 2016 Edition. The SAMREC Code was updated and released in May 2016. Although Tharisa is not yet required to report against the SAMREC Code (2016), as it only becomes mandatory in January 2017, Tharisa has chosen to be proactive and this report has applied the requirements of the latest version. An important aspect is that where a material change to a significant project occurs, the checklist or Table 1 (Appendix A) of the SAMREC Code needs to be reviewed on an 'if not, why not" basis.

1.4 Qualifications and Experience

The participants consist of a number of technical experts brought together by Coffey to complete the CPR and are all "Competent Persons" as defined in the SAMREC code. The compilation of the CPR was prepared by Mr Lomberg. The participants in the CPR and their individual areas of responsibility are listed as follows:-

Ken Lomberg, Senior Principal Consultant, Coffey
B.Sc. (Hons) Geology, B.Com., M.Eng., FGSSA, Pr.Sci.Nat.
Project management, mineral resources, geological interpretations, site visits, report preparation.

Mr Lomberg has some 30 years experience in the minerals industry (especially platinum and gold). He has been involved in exploration and mine geology and has had the privilege of assisting in bringing a mine to full production. His expertise is especially in project management, mineral reserve and resource estimation.

Mr Lomberg has undertaken mineral resource and reserve estimations and reviews for platinum, chromite, gold, copper, uranium and fluorite projects. He has assisted with the reviews or estimation of diamond and coal projects. He has assisted with or compiled

Competent Persons Reports/NI 43-101 for various companies that have been listed on the TSX, JSE and AIM.

Jaco Lotheringen, Associate Consultant – Ukwazi Mining Solutions
B.Eng., MSAIMM, Pr.Eng.
Mining engineering, Mineral Reserve estimation, infrastructure, site visits, report preparation.

Mr. Lotheringen is a member in good standing of the Southern African Institute of Mining and Metallurgy (SAIMM) and is a registered Professional Mining Engineer with the Engineering Council of South Africa (ECSA). He has 20 years' experience in the Mining and Minerals industries with the last 15 years focussed primarily on the estimation and audit of Mineral Reserve estimates. Mr. Lotheringen has more than five years relevant experience in the planning and reserve estimation of similar platinum and chrome open cast operations. Mr Lotheringen has undertaken Mineral Reserve estimations and reviews for platinum, gold, copper, chrome, manganese, coal and iron ore projects. He has assisted on Mineral Expert Reports/NI 43-101 for various projects that have been listed on the TSX, JSE and AIM.

Dr John James, Associate Consultant – Celtis Geotechnical
B.Sc. (Hons) (Geology), PhD, FSAIMM, FSANIR, MGSSA
Geotechnical Engineering, site visits, report preparation.

Dr James is the principal consultant for Celtis Geotechnical CC, consulting to various mining companies on projects in South Africa, Zambia, Botswana and Australia. While with Rodio SA, he managed exploration drilling, grouting, surface and underground geotechnical contracts in Turkey and South Africa.

He has experience in open pit mining, involved with supervising slope stability consultants at the then JCI's Platinum Mines and while with Rand Mines on outcrop mining. He has a total of 20 years experience in practical rock mechanics and design on gold mines, with Anglo-American, Rand Mines and JCI; this includes considerable experience in wide orebody mining, geology and all aspects of support design and backfill behaviour and placement; the Technology, Rock Mechanics and Design of hard rock, coal and base metal mines as well as tunnelling, and has also directed projects and research into mine design, technology transfer and auditing and assessment systems.

He was jointly awarded the M D G Salamon prize for the most important contribution to Rock Mechanics in 1997. He has published numerous technical papers and articles on rock support and other relevant rock engineering topics.

1.5 Independence

Neither Coffey, nor the key personnel nominated for the completed and reviewed work, has any interest (present or contingent) in Tharisa plc and its subsidiaries, its directors, senior management, advisers or the mineral properties reported on in this report. The proposed work, and any other work done by Coffey for Tharisa plc, is strictly in return for professional fees. Payment for the work is not in any way dependent on the outcome of the work, nor on the success or otherwise of Tharisa plc's own business dealings. There is no conflict of interest in Coffey undertaking the CPR as contained in this document.

1.6 Legal Proceedings

The CP has relied on information provided by Tharisa that all necessary statutory mining authorisations, permits and licences have been obtained or where they have not, the process for application is underway.

It is understood that there are no legal impediment for the continued mining operation or that would affect the likely viability of the Mine and/or on the estimation and classification of the Mineral Reserve and Mineral Resource as reported in this document.

Coffey is not aware of any legal proceedings against the Company that could adversely affect its ability or right to exploit the Tharisa Mine mineral resource and reserve.

2 DISCLAIMER

This report was prepared as a Competent Persons Report, in accordance with both the SAMREC Code for Tharisa plc, by Coffey. The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in Coffey's services and based on:

i) information made available at the time of preparation by Tharisa plc and its subsidiaries,

ii) third party technical reports prepared by Government agencies and previous tenement holders, along with other relevant published and unpublished third party information, and

iii) the assumptions, conditions and qualifications set forth in this report.

This report is intended to be used by Tharisa plc, subject to the terms and conditions of its contract with Coffey.

Neither the whole nor any part of this report nor any reference thereto may be included in or with or attached to any document or used for any other purpose, without Coffey's written consent to the form and context in which it appears.

A final draft of this report was provided to Tharisa plc, along with a written request to identify any material errors or omissions, prior to lodgement.

Neither Coffey, nor the authors of this report, are qualified to provide extensive comment on legal facets associated with ownership and other rights pertaining to Tharisa Minerals', mineral properties. Coffey did not see or carry out any legal due diligence confirming the legal title of Tharisa Minerals, to the mineral properties.

3 PROPERTY DESCRIPTION AND LOCATION

3.1 Mine Description and Location

Tharisa Minerals, a 74% held subsidiary of Tharisa plc, operates the Tharisa Mine. Tharisa Minerals holds a mining right, granted by the Department of Mineral Resources (DMR) on 19 September 2008 and registered on 13 August 2009, to various portions of the property of Farm 342JQ (in respect of PGMs (Platinum Group Metals), nickel, copper, silver and chrome) and to the whole property of Rooikoppies 297JQ (in respect of the PGMs, nickel, copper, silver and chrome contained within the MG Chromitite Layers only). The Tharisa Mine is located in the North West Province some 35km east of the city of Rustenburg (Figure 3.1_1) in the Marikana section of the south-western limb of the Bushveld Complex (Figure 3.1_2). The Marikana section is separated from the Brits section to the east by Wolhulterskop and from the Rustenburg section to the west by the Spruitfontein upfold.

The Tharisa Mine is located approximately 5km north of the Magaliesberg Mountains. These mountains are formed by quartzites (Transvaal Sequence), which are common as floor or basement rocks to the Bushveld Complex.

The nearest major road is the N4 National Road which links Pretoria with Rustenburg and crosses the south-eastern corner of the Farm 342JQ property immediately south of the outcrop of the Middle Group (MG) Chromitite Layers. A secondary road bisects the property in a north-south direction providing access to the town of Marikana. The east west Rustenburg-Brits railway line bisects the Rooikoppies property with a station located in the town of Marikana on the Rooikoppies property.

3.2 Climate

A typical summer rainfall climate prevails in the area. Summer rain occurs mainly in the form of thunderstorms with a mean annual precipitation of approximately 680mm, and evaporation is about 1,800mm per year. Winds are generally light and blow predominantly from the north-west. Winters are cool and dry. Extreme weather conditions occur in the form of frost (2 to 20 occurrences per annum) and the occasional hail storm.

The average annual temperature for the year is approximately 19°C, with average maximum temperatures ranging between 22°C and 32°C and average minimum temperatures ranging between 2°C and 18°C. The hottest months are December to February. During April and May there is a noticeable drop in temperature, which signals the commencement of winter. The coldest months are June and July.

The area generally has a high S-Pan evaporation rate in the summer months from November to January. This gives rise to a high relative humidity. Evaporation is greater in summer than in winter, due to higher ambient temperatures.

Figure 3.1_1
Map Showing the Geographic Location of the Tharisa Mine

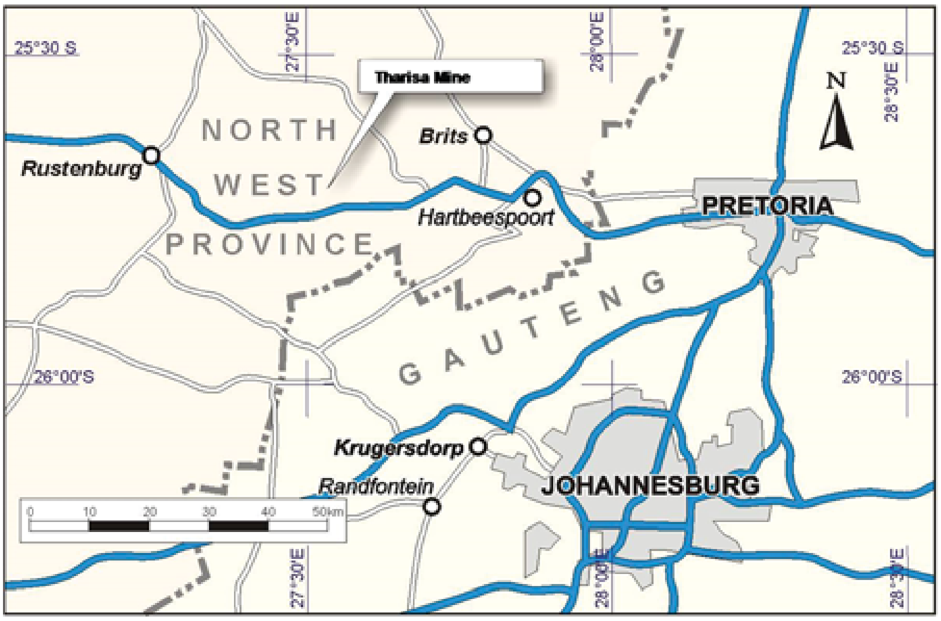
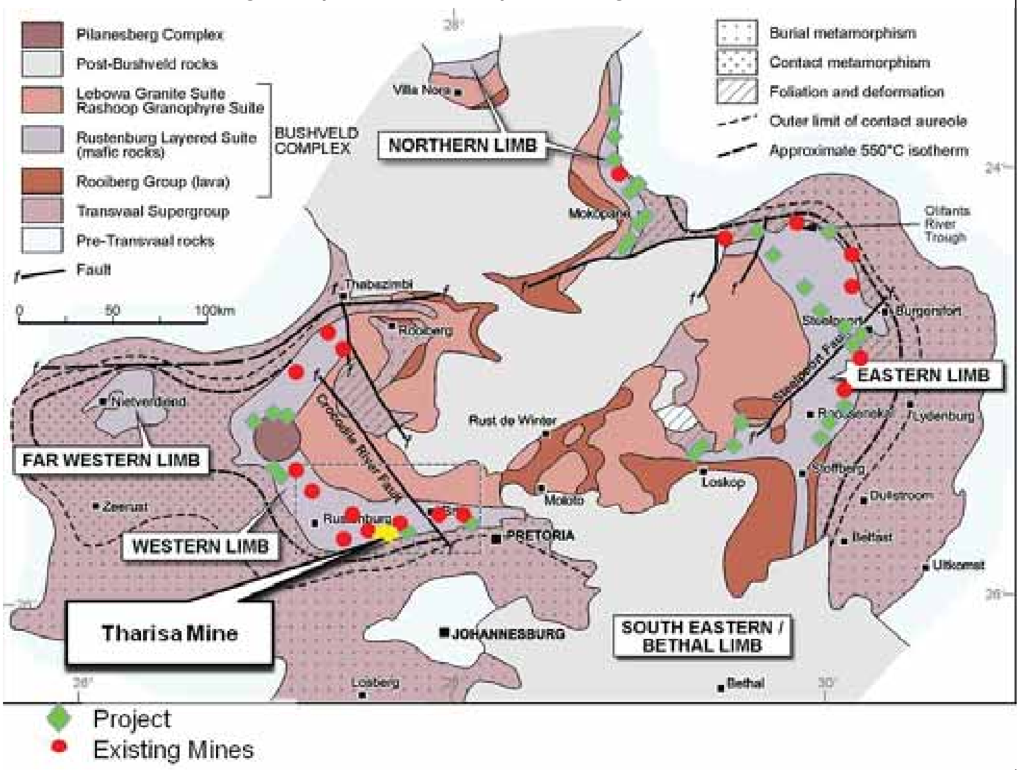


Figure 3.1_2
Generalised Geological Map of Bushveld Complex Showing the Location of the Tharisa Mine



3.3 Physiography

The topography on the Tharisa Mine property is gently undulating. The elevation ranges from 1,140m in the south-west to approximately 1,320m in the north. Immediately north of the project are a number of gabbro-norite hills. Approximately 5km to the south of the mine is the Magaliesberg Mountain range where the peaks rise to approximately 1400m above mean sea level (amsl). The perennial Sterkstroom and various non-perennial tributaries run through the mine area.

This area is located within the savannah biome, and consists typically of scattered trees and shrubs with continuous grass ground cover. Shrub and tree density increases along rivers and in the gabbro-norite hills. Land use is predominantly agricultural in the south with the Marikana operations of Lonmin plc (Lonmin) being situated on the northern part of the Rooikoppies property and the chrome operations of Samancor situated to the east of the mine.

A topographic map of the mine and the local area is presented in Figure 3.3_1

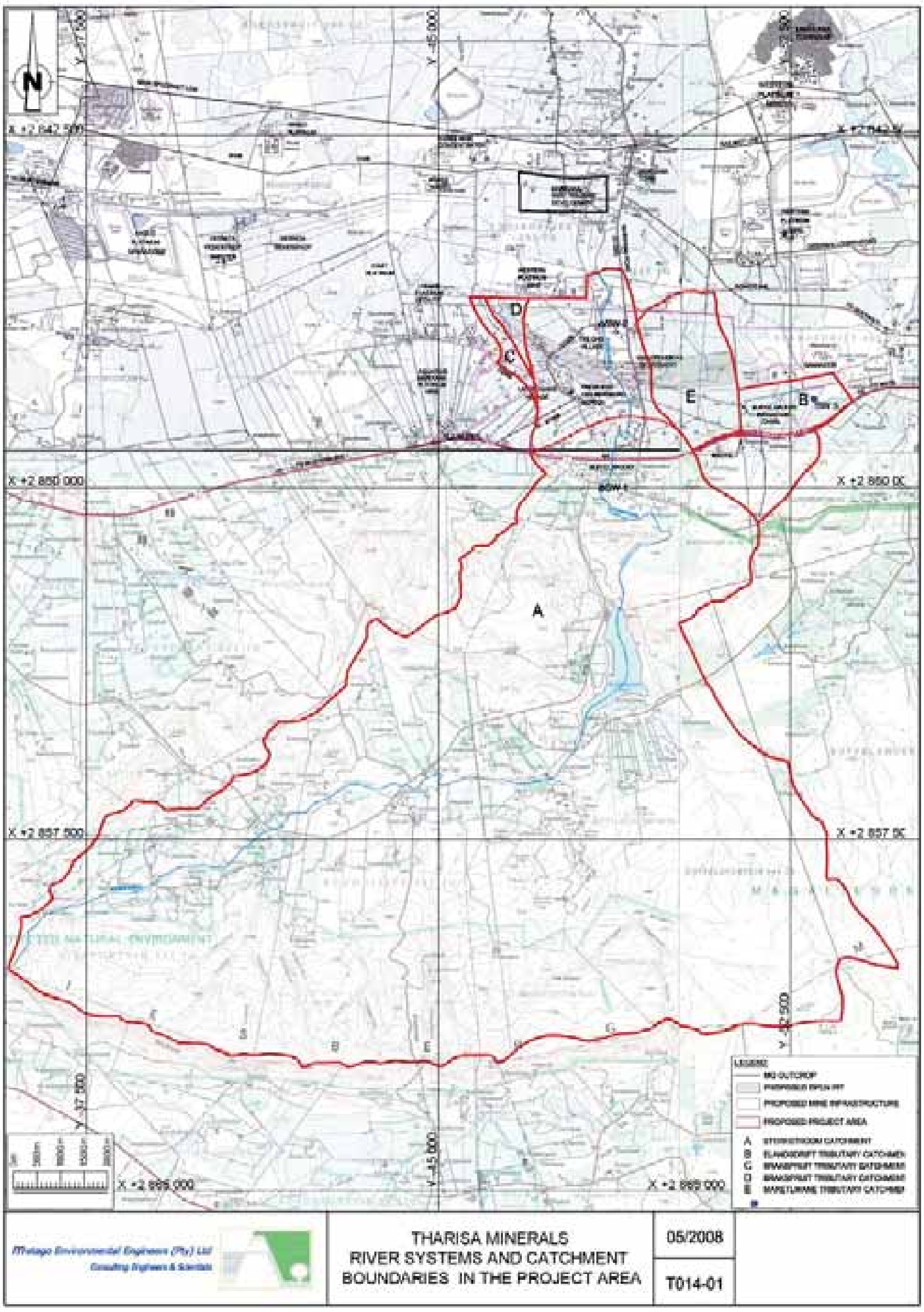
3.4 Land Use

Land use around the Tharisa Mine consists of a mixture of farming, mining, residential, small business and general community activities. It is expected that agricultural production took place in the area for both subsistence farming by informal settlers and commercial farming, including crop production (maize, sunflowers, wheat, livestock feed) and livestock grazing. Due to overgrazing and subsistence farming practices by informal dwellers as well as the collection of vegetation mainly for firewood, parts of the general area were transformed. River systems within the area also show evidence of disturbance by agricultural activities.

A 275KV power line associated Eskom servitude, crosses through the eastern boundary of the mine area in a north-south direction. Smaller rural power lines and telephone lines currently service the residential areas within the western and eastern sections of the mine area. Infrastructure (pipes and canals) associated with the Buffelspoort Irrigation Board traverse various sections of the mine area in a south-north direction. There is also a network of tarred and gravel roads which exists in the area.

Figure 3.3_1

River Systems and Catchment Boundaries in the Mine Area



3.5 License Status - Mining Right

Tharisa Minerals holds a mining right, granted by the DMR (then the DME) in terms of the MPRDA on 19 September 2008, for a period of 30 years, to various portions of the property Farm 342JQ (in respect of PGMs, gold, nickel, copper, silver and chrome) and the whole of the property Rooikoppies 297JQ (in respect of PGMs, gold, nickel, copper, silver and chrome contained within the MG Chromitite Layers only) (Figure 3.5_1). On 13 August 2009, the mining right was registered in the Mining and Petroleum Titles Registration Office, under Reference No 49/2009(MR).

On 7 March 2008 a mining right in respect of chrome was granted over Portions 96 and 183 of the property Farm 342JQ to South African Producers and Beneficiators of Chrome Ore (Pty) Ltd and registered on 27 July 2009. These rights were purchased by Tharisa Minerals on 18 March 2008.

In July 2011, an application was granted in terms of Section 102 of the MPRDA, to amend the existing mining right by the addition of Portions 96 (46.38ha), 183 (15.18ha) and 286 (13.29ha) of the property Farm 342JQ to the mining right 49/2009(MR).

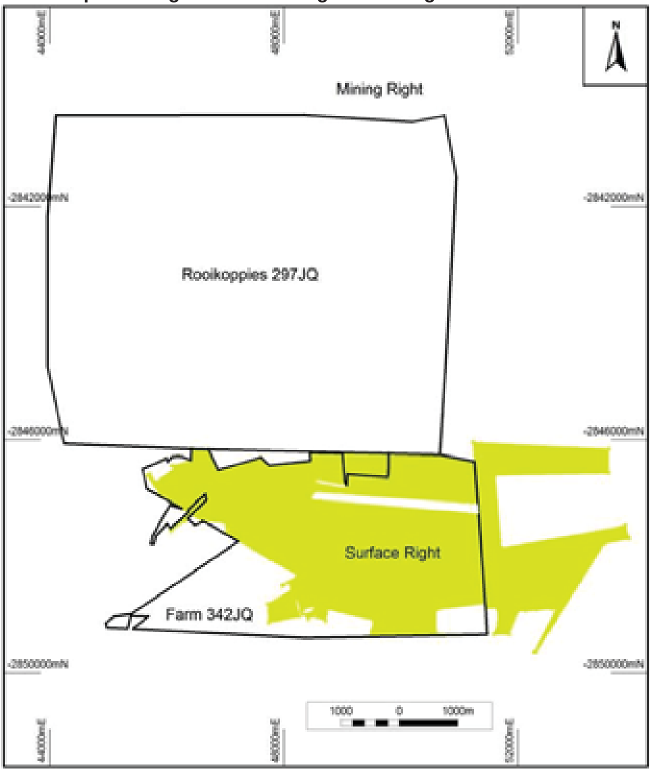
Figure 3.5_1

Map showing the Mineral Rights of the Tharisa Mine

3.6 Surface Rights

The surface rights of several of the portions of Farm 342JQ have been purchased by Tharisa Minerals with the stated intent of obtaining other surface rights (Figure 3.6_1). It should be noted that should Tharisa Minerals not acquire all the surface rights of the area defined in the mining right, it will not be precluded from mining there.

Figure 3.6_1
Map showing the Surface Rights Holdings of Tharisa Mine



4 HISTORY

4.1 Ownership History

Thari Resources (Pty) Ltd (Thari) which was incorporated in January 2005, acquired prospecting rights for chrome and PGMs over various portions of the property Farm 342JQ and to the property Rooikoppies 297JQ in March 2006. Thari is a HDSA and woman controlled company focused on the minerals and energy sectors.

In March 2006 Thari established Tharisa Minerals as a wholly owned subsidiary. In September 2008, the prospecting rights were transferred from Thari to Tharisa Minerals after obtaining the necessary Ministerial approval in terms of Section 11 of the MPRDA.

Tharisa plc was incorporated in February 2008 and after obtaining the necessary Ministerial approval acquired 74% of Tharisa Minerals on 9 February 2009. The remaining 26% is held by Thari (20%) and The Tharisa Community Trust (6%).

On 19 September 2008, the prospecting rights, for PGM and chrome, over various portions of Farm 342JQ and the whole of Rooikoppies, held by Tharisa Minerals, were converted into a mining right with the approval of the DMR. This mining right was registered to Tharisa Minerals on 13 August 2009. Subsequently, the mining right for chrome over portions 96 and 183 of the Farm 342 JQ was purchased from South African Producers and Beneficiators of Chrome Ore (Pty) Limited.

In July 2011, an application was granted in terms of Section 102 of the MPRDA, to amend the existing mining right by the addition of Portions 96, 183 and 286 of the property Farm 342JQ to the mining right 49/2009(MR).

4.2 Work undertaken by the Previous License Holders

Prior to Thari obtaining the prospecting rights, the only known exploration activities undertaken on the properties had been the regional mapping undertaken by the Geological Survey (now Council of Geoscience) and the drilling of six cored boreholes by an entrepreneur Mr Hennie Botha on Farm 342JQ and the adjacent property Spruitfontein 341JQ.

4.3 Historical Mineral Resources and Mineral Reserves

The mineral resource was initially estimated in 2008 and depleted based on the tonnage mined. The mineral resource reported as at December 2013 is presented in Table 4.3_1. The mineral reserve has been re-estimated a number of times utilising revised mining approaches and revised revenue and cost projections. The Mineral Resources and Reserves of December 2015 are reported in Tables 4.3_2 and 4.3_3.

Table 4.3_1

Mineral Resource Statement for the Tharisa Mine (31 December 2015)

	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+ Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
MG4A CHROMITITE LAYER																		
Measured	6.234	1.43	3.69	24.82	0.40	0.15	0.12	0.003	0.26	0.04	0.05	0.67	59:22:18:0	1.02	39:15:12:0:25:4:5	1.12	204	760
Indicated	15.885	1.59	3.70	24.29	0.40	0.15	0.13	0.003	0.25	0.04	0.05	0.68	59:23:18:1	1.03	39:15:12:0:25:4:5	1.10	525	762
Inferred	68.476	1.43	3.70	25.18	0.39	0.14	0.13	0.004	0.26	0.05	0.05	0.67	59:21:19:1	1.03	38:14:12:0:26:4:5	1.11	2,263	763
MG4 and MG4(0) CHROMITITE LAYER Package																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+ Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	17.920	4.09	3.74	26.39	0.69	0.19	0.17	0.003	0.32	0.06	0.08	1.06	66:18:16:0	1.51	46:13:11:0:21:4:5	1.17	872	781
Indicated	29.790	2.99	3.65	24.75	1.08	0.22	0.21	0.003	0.36	0.08	0.11	1.51	71:15:14:0	2.06	52:11:10:0:18:4:6	1.20	1,972	730
Inferred	170.678	3.70	3.62	22.60	0.99	0.19	0.19	0.003	0.34	0.07	0.10	1.36	72:14:14:0	1.88	53:10:10:0:18:4:6	1.15	10,313	697
MG3 CHROMITITE LAYER																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+ Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm))
Measured	10.417	3.73	3.26	13.22	0.60	0.35	0.15	0.005	0.22	0.04	0.06	1.11	54:32:14:0	1.43	42:25:11:0:15:3:4	0.99	479	482
Indicated	23.412	4.28	3.22	17.99	0.75	0.44	0.19	0.005	0.27	0.05	0.08	1.39	54:32:14:0	1.79	42:25:11:0:15:3:4	1.08	1,347	603
Inferred	67.415	3.21	3.20	25.65	1.01	0.58	0.26	0.005	0.38	0.08	0.10	1.86	54:31:14:0	2.42	42:24:11:0:16:3:4	1.13	5,245	785
MG2 CHROMITITE LAYER																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+ Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	13.092	3.96	3.62	19.33	1.07	0.28	0.15	0.004	0.26	0.05	0.08	1.50	71:18:10:0	1.89	56:15:8:0:14:3:4	0.97	796	730
Indicated	42.716	4.37	3.67	17.80	0.98	0.28	0.15	0.004	0.24	0.05	0.07	1.42	69:20:10:0	1.78	55:16:8:0:14:3:4	0.92	2,388	733
Inferred	286.164	6.68	3.62	13.26	0.70	0.21	0.11	0.004	0.19	0.04	0.05	1.02	69:20:11:0	1.30	54:16:8:0:15:3:4	0.75	11,975	674

	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
MG1 CHROMITITE LAYER																		
Measured																	-	
Indicated	14.041	1.24	3.91	33.44	0.34	0.22	0.11	0.004	0.48	0.08	0.08	0.67	50:32:17:1	1.30	26:17:9:0:37:6:6	1.34	589	811
Inferred	57.245	1.23	3.89	32.26	0.33	0.20	0.11	0.003	0.45	0.08	0.07	0.64	51:31:17:1	1.24	26:16:9:0:36:6:6	1.29	2,276	803
MG0 CHROMITITE LAYER																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	1.801	0.50	3.74	26.07	0.57	0.18	0.16	0.004	0.30	0.05	0.07	0.92	62:19:18:0	1.33	43:13:12:0:22:4:5	1.19	77	747
Indicated	3.188	0.71	3.75	27.08	0.61	0.19	0.17	0.004	0.32	0.06	0.07	0.98	62:20:17:0	1.44	43:14:12:0:22:4:5	1.22	147	752
Inferred	0.011	0.17	3.73	23.76	0.45	0.17	0.15	0.006	0.24	0.04	0.05	0.77	58:22:19:1	1.11	41:15:13:1:22:4:5	1.11	0.40	711
UG1 CHROMITITE LAYER																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured																		
Indicated	1.500	2.17	3.75	23.68	0.36	0.28	0.14	0.030	0.21			0.82	44:35:17:4			1.12	39	
Inferred																		
TOTAL MINERAL RESOURCE																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	49.464	2.68	3.73	21.51	0.73	0.24	0.16	0.004	0.28	0.05	0.07	1.13	64:21:14:0	1.53	48:16:10:0:18:3:5	1.09	2,428	699
Indicated	128.033	2.45	3.67	22.22	0.80	0.27	0.16	0.004	0.31	0.06	0.08	1.24	65:22:13:0	1.69	48:16:10:0:18:3:5	1.10	7,007	713
Inferred	651.488	3.11	3.74	19.88	0.74	0.23	0.15	0.004	0.28	0.05	0.07	1.13	66:21:13:0	1.53	49:15:10:0:18:4:5	1.00	32,072	712
Total	**828.984**	**2.95**	**3.73**	**20.38**	**0.75**	**0.24**	**0.15**	**0.004**	**0.28**	**0.05**	**0.07**	**1.15**	**66:21:13:0**	**1.56**	**48:15:10:0:18:4:5**	**1.02**	**41,507**	**712**

Note: The mineral resource is declared to a depth of 750m below surface.
Grades and tonnages are reported at shaft head
The consideration of realistic eventual extraction necessitates that the mineral resource considers the MG Chromitite Layer to be a geological unit and that all platiniferous and chromiferous horizons will be mined and all PGM, Cu, Ni and Cr_2O_3 recovered.
The UG1 Chromitite Layer is declared for the part that falls within the current proposed open pit
The mineral resource is reported inclusive of the mineral reserve

Table 4.3_2

Tharisa Mine: Open Pit Mineral Reserve (December 2013) (SAMREC Code)

Proved Mineral Reserve													
Chromitite Layer	**Tonnes ('000)**	**Pt (g/t)**	**Pd(g/t)**	**Rh(g/t)**	**Au (g/t)**	**3PGE+Au (g/t)**	**Ru(g/t)**	**Ir(g/t)**	**5PGE+Au (g/t)**	**Cr_2O_3 (%)**	**Cu (%)**	**Ni (%)**	**Cr (%)**
MG0													
MG1													
MG2	13.2	0.85	0.27	0.13	0.004	1.27	0.23	0.07	1.57	15.9	0.003	0.060	10.8
MG3	11.1	0.55	0.32	0.14	0.005	1.01	0.20	0.05	1.26	11.9	0.003	0.045	8.1
MG4	11.0	1.00	0.22	0.20	0.003	1.43	0.34	0.10	1.87	24.2	0.002	0.071	16.6
MG4A	6.1	0.35	0.13	0.11	0.003	0.59	0.22	0.04	0.85	21.3	0.003	0.066	14.6
Total	**41.4**	**0.74**	**0.25**	**0.15**	**0.004**	**1.14**	**0.25**	**0.07**	**1.46**	**17.8**	**0.003**	**0.060**	**12.2**
Probable Mineral Reserve													
Chromitite Layer	**Tonnes ('000)**	**Pt(g/t)**	**Pd(g/t)**	**Rh(g/t)**	**Au (g/t)**	**3PGE+Au (g/t)**	**Ru(g/t)**	**Ir(g/t)**	**5PGE+Au (g/t)**	**Cr_2O_3 (%)**	**Cu (%)**	**Ni (%)**	**Cr (%)**
MG0													
MG1	6.8	0.32	0.20	0.11	0.004	0.63	0.45	0.07	1.15	32.1	0.002	0.077	22.0
MG2	14.6	0.85	0.30	0.14	0.004	1.29	0.23	0.06	1.58	15.9	0.002	0.061	10.9
MG3	13.2	0.58	0.33	0.15	0.004	1.05	0.21	0.06	1.32	12.8	0.003	0.047	8.7
MG4	6.8	1.04	0.24	0.20	0.003	1.48	0.35	0.11	1.94	24.0	0.002	0.070	16.4
MG4A	5.0	0.34	0.14	0.11	0.004	0.59	0.22	0.04	0.85	20.7	0.003	0.066	14.2
Total	**46.4**	**0.67**	**0.27**	**0.14**	**0.004**	**1.08**	**0.27**	**0.07**	**1.42**	**19.1**	**0.002**	**0.061**	**13.1**
Total Mineral Reserve													
Chromitite Layer	**Tonnes ('000)**	**Pt(g/t)**	**Pd(g/t)**	**Rh(g/t)**	**Au (g/t)**	**3PGE+Au (g/t)**	**Ru(g/t)**	**Ir(g/t)**	**5PGE+Au (g/t)**	**Cr_2O_3 (%)**	**Cu (%)**	**Ni (%)**	**Cr (%)**
MG0													
MG1	6.8	0.32	0.20	0.11	0.004	0.63	0.45	0.07	1.15	32.1	0.002	0.077	22.0
MG2	27.8	0.85	0.28	0.14	0.004	1.28	0.23	0.07	1.58	15.9	0.003	0.061	10.9
MG3	24.4	0.56	0.32	0.14	0.005	1.03	0.20	0.06	1.29	12.4	0.003	0.046	8.5
MG4	17.7	1.02	0.23	0.20	0.003	1.45	0.34	0.11	1.90	24.2	0.002	0.071	16.5
MG4A	11.1	0.34	0.13	0.11	0.003	0.59	0.22	0.04	0.85	21.0	0.003	0.066	14.4
Total	**87.8**	**0.70**	**0.26**	**0.14**	**0.004**	**1.11**	**0.26**	**0.07**	**1.44**	**18.5**	**0.002**	**0.061**	**12.7**

Table 4.3_3

Tharisa Mine: Underground Mine Mineral Reserve (September 2016)
Reported in terms of the guidelines of the SAMREC Code

Chromitite Layer	Tonnes ('000)	Pt (g/t)	Pd(g/t)	Rh(g/t)	Au (g/t)	3PGE+Au (g/t)	Ru(g/t)	Ir(g/t)	5PGE+Au (g/t)	Cr_2O_3 (%)	Ni (%)	Cu (%)	Cr (%)
Proved Mineral Reserve													
MG2AB	-	-	-	-	-	-	-	-	-	-	-	-	-
MG4	-	-	-	-	-	-	-	-	-	-	-	-	-
Total	-	-	-	-	-	-	-	-	-	-	-	-	-
Probable Mineral Reserve													
Chromitite Layer	**Tonnes ('000)**	**Pt(g/t)**	**Pd(g/t)**	**Rh(g/t)**	**Au (g/t)**	**3PGE+Au (g/t)**	**Ru(g/t)**	**Ir(g/t)**	**5PGE+Au (g/t)**	**Cr_2O_3 (%)**	**Ni (%)**	**Cu (%)**	**Cr (%)**
MG2AB	6.6	0.70	0.21	0.10	0.002	1.02	0.20	0.05	1.27	17.4	0.060	0.002	11.9
MG4	12.0	0.89	0.18	0.17	0.002	1.25	0.31	0.10	1.66	20.4	0.061	0.002	14.1
Total	**18.6**	**0.82**	**0.19**	**0.15**	**0.002**	**1.17**	**0.27**	**0.08**	**1.52**	**19.3**	**0.060**	**0.002**	**13.3**
Total Mineral Reserve													
Chromitite Layer	**Tonnes ('000)**	**Pt(g/t)**	**Pd(g/t)**	**Rh(g/t)**	**Au (g/t)**	**3PGE+Au (g/t)**	**Ru(g/t)**	**Ir(g/t)**	**5PGE+Au (g/t)**	**Cr_2O_3 (%)**	**Ni (%)**	**Cu (%)**	**Cr (%)**
MG2AB	6.6	0.70	0.21	0.10	0.002	1.02	0.20	0.05	1.27	17.4	0.060	0.002	11.9
MG4	12.0	0.89	0.18	0.17	0.002	1.25	0.31	0.10	1.66	20.4	0.061	0.002	14.1
Total	**18.6**	**0.82**	**0.19**	**0.15**	**0.002**	**1.17**	**0.27**	**0.08**	**1.52**	**19.3**	**0.060**	**0.002**	**13.3**

4.4 Current Operations

In Phase 2 of the mine's development, the mining rate was increased to 100ktpm, to feed the Phase 2 processing facility expansion. This plant was commissioned in 2012 and increased the pilot plant throughput capacity to 100ktpm and the incorporation of a PGM recovery and additional chrome scavenging circuits. A 300ktpm concentrator was commissioned to treat the increased ROM production in parallel to the existing 100ktpm Phase 2 plant. The current mine capacity is 4.8Mtpa. With minor improvements to the processing plant the capacity increased to 5.1Mtpa.

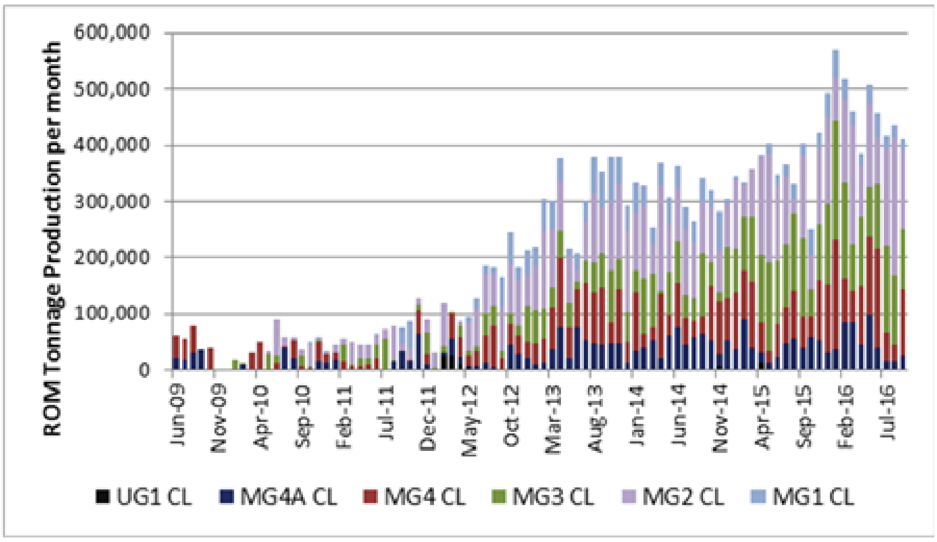
The historical mine production is presented in Figure 4.4_1:

- As at 30 September 2016. the Tharisa Mine has produced 2,830,521t of 42% Cr_2O_3 chromite concentrate
- The mining cost is currently R248 per Run of Mine (RoM) tonne

Tharisa Mine has secured sufficient supply of water and electricity to meet its requirements for steady state production for the LOM.

Figure 4.4_1

Historical Mining Production from the UG1 and MG Chromitite Layers



The open pit production is being undertaken by MCC, who is the appointed mining contractor. MCC is a well established mining contractor in the area and have extensive knowledge of the project.

While the Phase 2 and 3 process facility expansions were underway, mine production was limited to 38ktpm ROM ore throughput. The 100kt plant was commissioned in February 2012 followed bythe commissioning of the 300ktpm plant in December 2012. Following improvements to the current plant infrastructure, the ROM production has increased to 430ktpm.

The maximum depth of mining is currently at 75m in the East pit and produces a predominantly fresh material from the MG chromitite layers, comprising the MG4A, MG4, MG3, MG2 and MG1 Chromitite Layers. The shallow MG1 Chromitite Layer was mined underground to a limited extent on the eastern boundary of the property by the previous mining right holder.

The current mine plan is based on two open pit operations east and west of the Sterkstroom river which runs north south through the Tharisa Mine area. The pits were designed to protect the water course and the local infrastructure running parallel to the river. Current ROM production is 430ktpm.

The open pit production fulfils the production requirements until 2029, after which production will transition to an underground bord and pillar mining operation. The last open pit tonnage will be mined in 2035.

The mine design and schedule was completed by Ukwazi. The production profile was designed to ensure a practical constant delivery of ore to the processing facility.

5 GEOLOGICAL SETTING

5.1 Regional Setting

The stable Kaapvaal and Zimbabwe Cratons in southern Africa are characterised by the presence of large mafic to ultramafic layered complexes, the best known of which are the Great Dyke in the Zimbabwe Craton and the Bushveld and Molopo Complexes in the Kaapvaal Craton. By far the largest, best-known and economically most important of these is the Bushveld Complex, which was intruded about 2,060 million years ago into rocks of the Transvaal Supergroup, largely along an unconformity between the Magaliesberg quartzite of the Pretoria Group and the overlying Rooiberg felsites. The total estimated extent of the Bushveld Complex is some 66,000 km^2, of which about 55% is covered by younger formations. The mafic rocks of the Bushveld Complex host layers rich in PGM, chromium and vanadium, and constitute the world's largest known resource of these metals.

5.1.1 Bushveld Complex Stratigraphy

The mafic rocks (collectively termed the Rustenburg Layered Suite) can be divided into five zones known as the Marginal, Lower, Critical, Main and Upper Zones from the base upwards (Figure 5.1.1_1).

The **Marginal Zone** is comprised of generally finer grained rocks than those of the interior of the Bushveld Complex and contains abundant xenoliths of country rock. It is highly variable in thickness and may be completely absent in some areas and contains no known economic mineralisation.

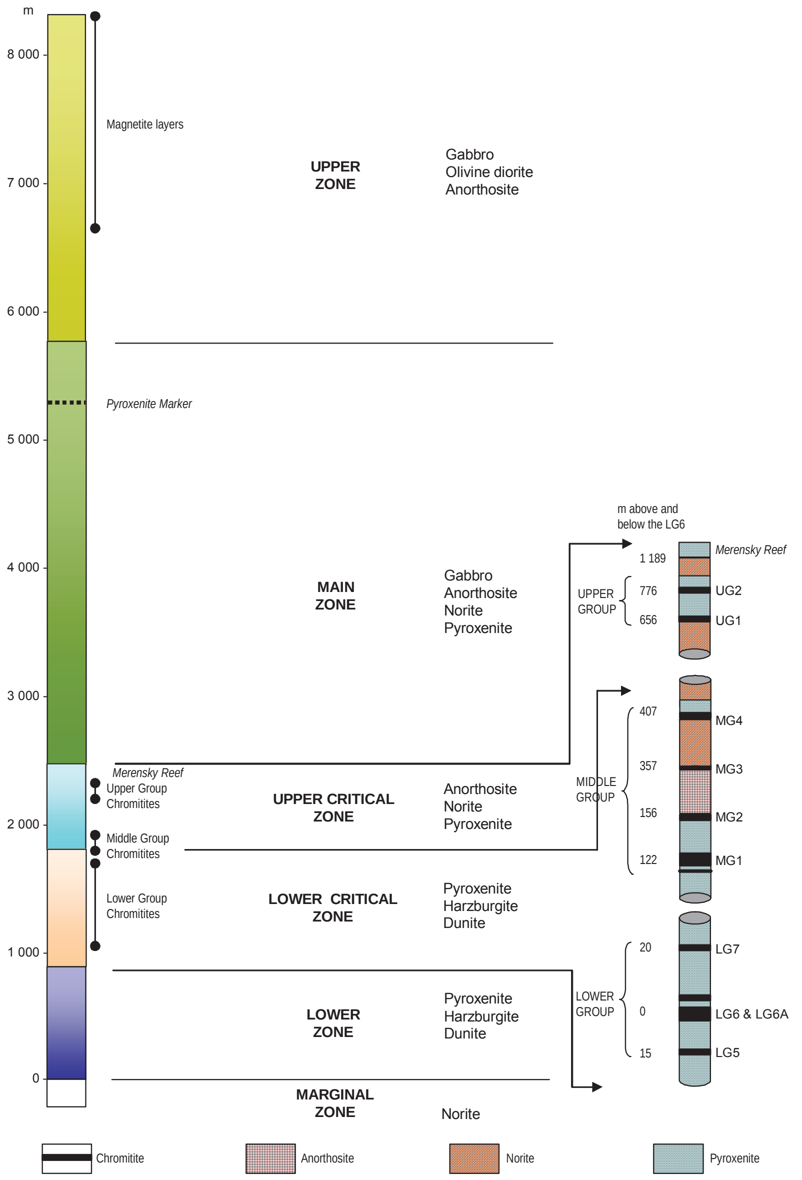
The **Lower Zone** is dominated by orthopyroxenite with associated olivine-rich cumulates in the form of harzburgites and dunites. The Lower Zone may be completely absent in some areas.

The **Critical Zone** is characterised by regular and often fine-scale rhythmic, or cyclic, layering of well-defined layers of cumulus chromite within pyroxenites, olivine-rich rocks and plagioclase-rich rocks (norites, anorthosites etc). The economically important PGM deposits are part of the Critical Zone.

The Critical Zone hosts all the chromitite layers of the Bushveld Complex, of which up to 14 have been identified. The first important cycle is the lower of the two Upper Group (UG) Chromitite Layers (the UG1 Chromitite Layer). This unit consists of a chromitite layer and underlying footwall chromitite layers that are interlayered with anorthosite. The most important of the chromite cycles for PGM mineralisation is the upper of the two UG Chromitite Layers (the UG2 Chromitite Layer) which averages some 1m in thickness and is mined throughout the Bushveld Complex.

Underlying the UG Chromitite Layers are the MG Chromitite Layers which consists of five groups of chromitite layers over an overall thickness of 50 – 80m. These chromitite layers are important as they contain significant concentrations of chromite and PGMs.

Figure 5.1.1_1
Generalised Stratigraphic Column of Rustenburg Layered Suite, Bushveld Complex
(after Viljoen and Schürmann, 1998)



The two uppermost units of the Critical Zone are the Merensky and Bastard units. The former is also of great economic importance as it contains at its base the PGM-bearing Merensky Reef, a feldspathic pyroxenitic assemblage with associated thin chromitite layers that rarely exceeds 1m in thickness. The top of the Critical Zone is generally defined as the top of the robust anorthosite (the Giant Mottled Anorthosite) that forms the top of the Bastard cyclic unit.

The Critical Zone may be subdivided into the Upper and Lower Critical Zones based on the last appearance of cumulus feldspar. This boundary is considered to be between the UG and MG Chromitite Layers.

The economically viable chromite reserves of the Bushveld Complex, most of which are hosted in the Critical Zone, are estimated at 68% of the world's total, whilst the Bushveld Complex also contains 56% of all known platinum group metals. The Merensky Reef, which developed near the top of the Critical Zone, can be traced along strike for 280km and is estimated to contain 60,000t of PGM to a depth of 1 200m below surface. The pyroxenitic Platreef mineralisation, north of Mokopane (formerly Potgietersrus), contains a wide zone of more disseminated style platinum mineralisation, along with higher grades of nickel and copper than occur in the rest of the Bushveld Complex.

The well-developed **Main Zone** consists of norites grading upwards into gabbronorites. It includes several mottled anorthosite layers in its lower sector and a distinctive pyroxenite layer two thirds of the way up, termed the Pyroxenite Marker.

The base of the overlying **Upper Zone** is defined by the first appearance of cumulus magnetite above the Pyroxenite Marker. In all, 25 layers of cumulus magnetite punctuate the Upper Zone, the fourth (Main Magnetite layer) being the most prominent. This is a significant marker, some 2m thick, resting upon anorthosite, and is exploited for its vanadium content in the eastern and western limbs of the Bushveld Complex.

5.1.2 Platinum Mineralisation

The Merensky Reef has traditionally been the most important platinum producing layer in the Bushveld Complex. Seismic surveys undertaken by the Council for Geoscience indicate that reflectors associated with the Merensky Reef can be traced as far as 50km down dip, to depths of 6,000m below surface. The Merensky Reef varies considerably in its nature, but can be broadly defined as a mineralised zone within, or closely associated with the ultramafic cumulate at the base of the Merensky cyclic unit.

In addition to the PGM mineralisation associated with the Merensky Reef, all chromitites in the Critical Zone at times contain elevated concentrations of PGMs. The UG2 Chromitite Layer is the only chromitite layer that is significantly exploited for PGMs at present.

The major geological features that affect the UG2 Chromitite Layer are faults, dykes, potholes and mafic/ultramafic pegmatites. Potholes are features of subsidence or erosion where the igneous layer is absent or occurs at a lower elevation in a modified form. Typically the PGM concentration and the thickness of the layer are modified. Potholes typically approach a circular shape. Potholes occur within all stratigraphic units of the Bushveld Complex including the MG Chromitite Layer. Poor ground conditions may be associated with potholes and pothole edges. On some mines, such as Bokoni (formerly known as Atok) and Northam, potholes may cause a geological loss of ground of up to 25%.

Another unique feature of the geology of the Bushveld Complex are the mafic/ultramafic pegmatites sometimes referred to as iron rich ultramafic pegmatites (IRUP's) or replacement pegmatites. While these often destroy the structure of the chromitite layer, the PGMs may be unaffected. However, it can result in a mining problem, especially underground, as it becomes difficult to identify the mineralised horizons.

5.1.3 Chromite Mineralisation

The first record of chrome in the Bushveld Complex is noted as an outcrop in the Hex River near Rustenburg in 1865. By the 1920s the various chromitite layers had been identified and traced over the known extent of the Bushveld Complex. Chromite mining started in earnest at about that time but it was not until the 1960s that South Africa became a major producer.

The Bushveld Complex hosts stratiform chromite deposits that are present as layers of massive chromitite. These layers are present in the Critical Zone and have been designated as the Lower Group (LG), MG and UG Chromitite Layers. The lower Critical Zone is host to the LG Chromitite Layers that consists of seven chromitite layers. The thickest and most significant being the LG6 Chromitite Layer. The MG Chromitite Layers consist of five individual chromite packages of which three are in the lower Critical Zone and two are in the upper Critical Zone. There are two UG Chromitite Layers with the UG2 Chromitite Layer being the most significant as a major source of PGM mineralisation.

Although remarkably consistent and continuous across the Bushveld Complex, the variations along strike have allowed the definition of 14 sections each with a unique character. The Tharisa Mine is located in the Marikana Section.

The LG6, MG1 and UG2 Chromitite Layers are the most exploited because of their mineralogical composition and because they can be mined by mechanised equipment both in open pit and underground. The LG6 Chromitite Layer is typically up to 1.05m thick and has a Cr_2O_3 grade of 46% to 48% and a Cr:Fe ratio of 1.56 – 1.60. Locally the LG Chromitite Layers may have much higher Cr:Fe ratios such as at Grasvaley (2.13 – 2.83) and Nietverdeind (1.88 – 2.06). The grade at Nietverdiend ranges from 48% to 51% Cr_2O_3.

The UG2 Chromitite Layer is typically up to 1m thick and has a Cr_2O_3 grade of 43.6% and a Cr:Fe ratio of 1.26 to 1.40. It has a significant PGM grade and so has been mined extensively to recover the PGMs.

The MG1 Chromitite Layer has been sporadically mined with the largest underground mining section being immediately east of the Tharisa Mine and mined by Samancor.

5.2 Local Geology

5.2.1 Tharisa Mine Area

The Tharisa Mine is located on the south-western limb of the Bushveld Complex in the Marikana section, on the properties Farm 342JQ and Rooikoppies 297JQ. The Marikana section is separated from the Brits section to the east by Wolhulterskop and the Rustenburg section to the west by the Spruitfontein upfold (Figure 5.2.1_1).

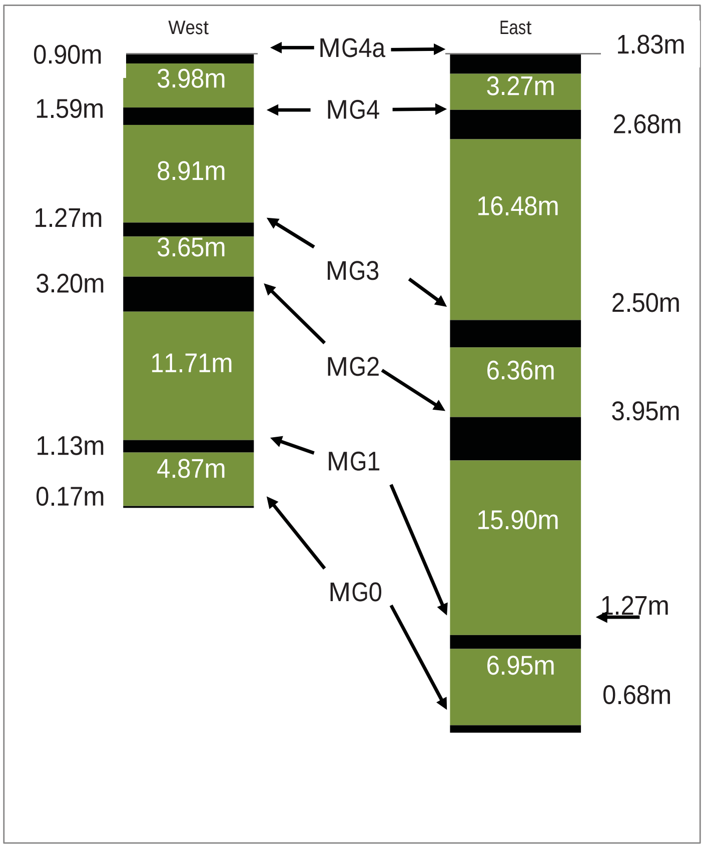
The MG Chromitite Layers outcrop on Farm 342JQ striking roughly east - west and dipping at 12-15° to the north to a depth estimated at over 1,000m with the total strike length being some 5,600m. Towards the west the dip is steeper with a gentle change in strike to NW-SE (Figure 5.2.1_2) and the stratigraphy narrows (Figure 5.2.1_3). The dip typically shallows out at depth across the extent of the mine area. The UG1 Chromitite Layer which occurs between 165m to

18m stratigraphically above the MG4A Chromitite Layer on the Farm 342JQ property and 163m (downdip) to 18m (near surface) on the Rooikoppies property also outcrops on the Farm 342JQ property. Both the UG2 Chromitite Layer (between 300m to 150m above MG4A Chromitite Layer) and the Merensky Reef (between 400m (east) to 290m (west) above MG4A Chromitite Layer) outcrop on the Rooikoppies property. Poorly developed chromitite layers below the MG Chromitite Layer were intersected in boreholes and are interpreted as the LG Chromitite Layers.

Figure 5.2.1_1

Tharisa Mine

Summary of Stratigraphic Unit across Tharisa Mine



Chromite Layer thickness from the block model – Measured and indicated areas
Parting – based on wireframes for indicated and measured area

Unit
MG4A Chromitite Layer
MG4 Chromitite Layer
MG3 Chromitite Layer
MG2 Chromitite Layer
MG1 Chromitite Layer
MG0 Chromitite Layer

Figure 5.2.1_2

Map of the Western Bushveld Complex showing the location of the Tharisa Mine

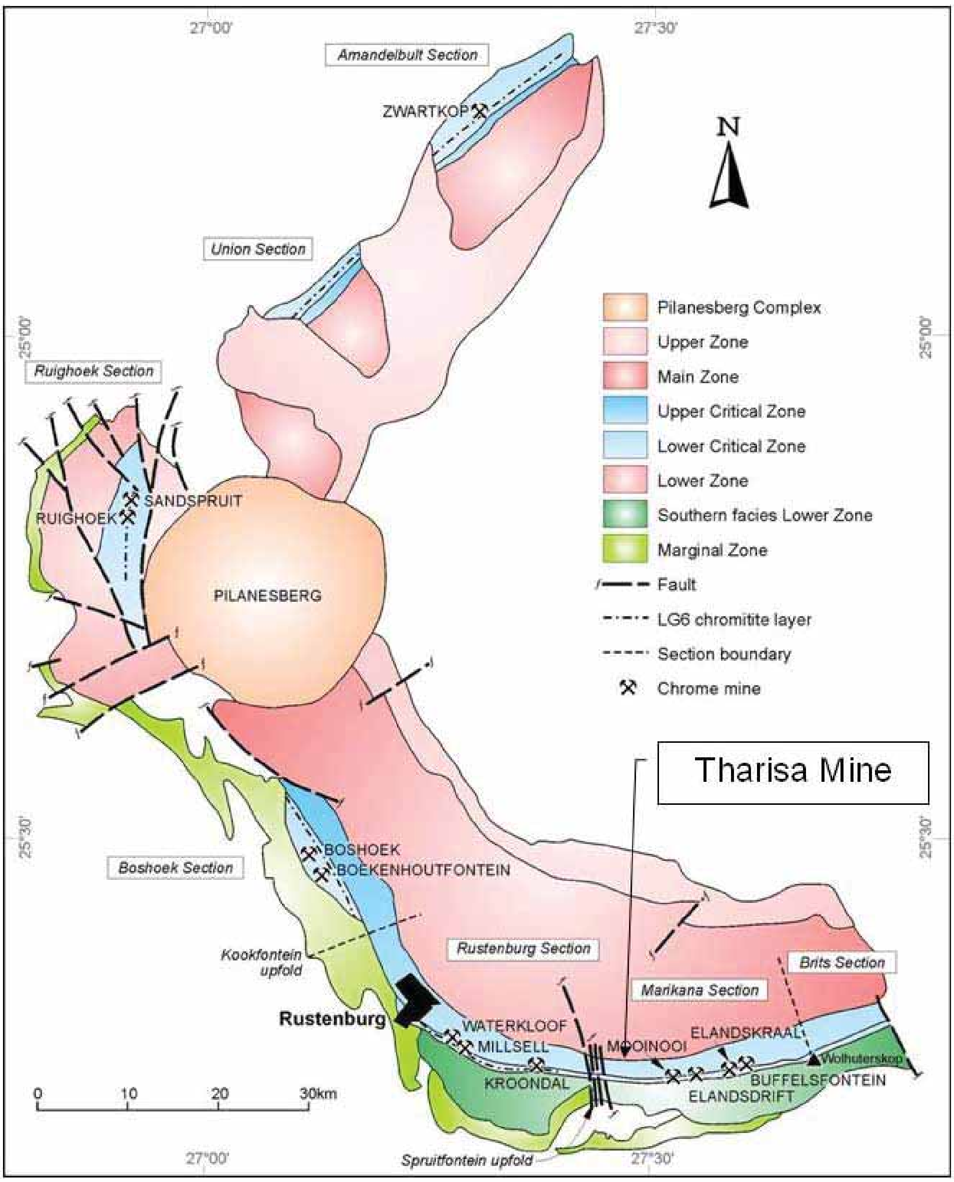
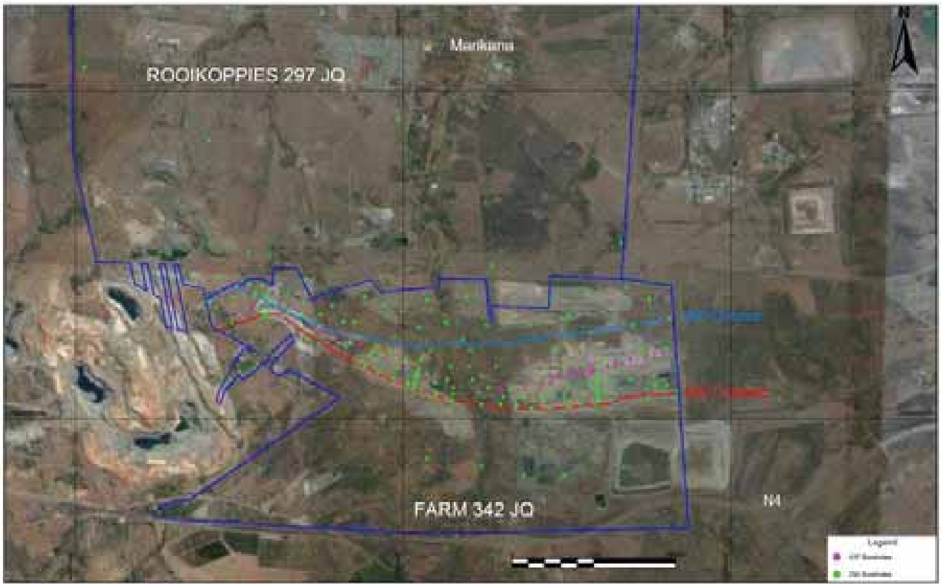


Figure 5.2.1_3

Borehole locations and outcrop positions of UG1 and MG1 Chromitite Layers represented on a Google Image of the Tharisa Mine



Note: additional boreholes (KIP) indicated

5.2.2 Middle Group Chromitite Layers

The MG Chromitite Layer package consists of five groups of chromitite layers (the MG0 Chromitite Layer, MG1 Chromitite Layer, the MG2 Chromitite Layer (subdivided into C, B and A Chromitite Layers), the MG3 Chromitite Layer and the MG4 Chromitite Layer (subdivided into the MG4(0), MG4 and MG4A Chromitite Layers) (Figure 5.2.2_1). The MG0 Chromitite Layer may be defined but formation of these chromitites is very erratic, thin and generally considered uneconomical in the mine area. However, where the MG1 Chromitite Layer immediately above is mined, there is merit in mining the MG0 Chromitite Layer as well. The MG0 Chromitite Layer Mineral Resource is declared for the area of the planned open pit.

The MG Chromitite Layer package (MG1 Chromitite Layer to MG4A Chromitite Layer) is developed over an average thickness of 57m in the East but thins to 46m in the West. The average thickness of the various units and subunits and a summary of the composite statistics are presented in Table 5.2.2_1. Down dip all partings thickness increase except for the MG4A – MG4 Chromitite Layer parting that decreases downdip. Figure 5.2.2_2 and Figure 5.2.2_3 are schematic representations of the variation within the MG Chromitite Layer packages and the parting thicknesses along strike and down dip respectively.

The entire MG and LG Chromitite Layers are truncated by the UG2 Chromitite Layer in the west at the neighbouring Spruitfontein upfold. The UG2 Chromitite Layer is reported to have a pothole morphology where it overlies the Transvaal Sequence rocks and truncates the MG and LG Chromitite Layers.

Figure 5.2.2_1

Generalised Stratigraphic Column of the MG Chromitite Layer at the Tharisa Mine

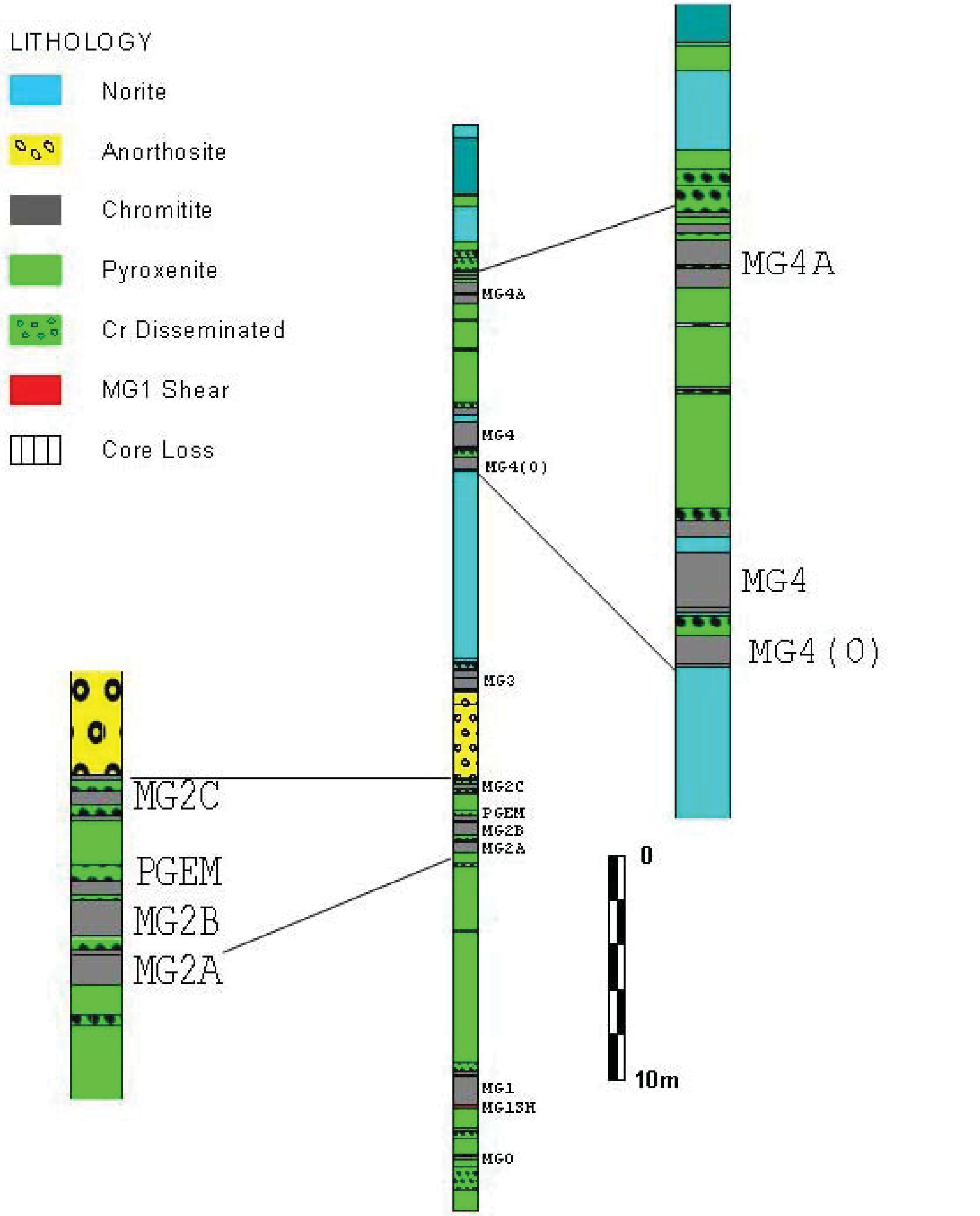


Table 5.2.2_1

Average Intersection Thicknesses of the MG Chromitite Layers and Partings

Unit or sub unit	Mine Average (m)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	Cr_2O_3 (%)	Cr:Fe
MG4 Chromitite Layer					
MG4A Chromitite Layer	1.37	0.74	60:22:18:1	24.56	1.07
Parting MG4A-MG4	3.78	0.22	57:24:18:1	5.91	0.36
MG4 Chromitite Layer	1.47	1.76	69:15:15:0	28.60	1.20
Parting MG4-MG4(0)	0.56	1.09	75:14:11:0	17.91	1.00
MG4(0) Chromitite Layer	0.56	1.29	69:17:14:0	28.89	1.20
MG3 Chromitite Layer					
Parting MG4(0)-MG3	11.10				
MG3 Disseminated	0.60	0.90	48:38:14:1	8.03	0.65
MG3 Chromitite Layer	1.38	1.77	55:31:14:0	25.67	1.12
MG3 - Zebra	0.18	0.77	64:21:14:0	8.77	0.72
MG2 Chromitite Layer					
Parting MG3-MG2C	4.95				
MG2C Chromitite Layer	0.64	1.98	69:20:11:0	28.35	1.17
PGEM HW	0.72	1.24	75:16:9:0	7.80	0.44
PGEM	0.51	2.62	72:18:10:0	17.01	0.85
PGEM FW	0.68	1.26	72:18:10:0	10.55	0.55
Parting MG2C-MG2B	0.56	1.32	68:18:14:0	31.06	1.22
MG2B Chromitite Layer	0.53	0.94	69:18:12:0	17.44	0.77
Parting MG2B-MG2A	0.58	1.94	70:22:8:0	28.59	1.16
MG2A Chromitite Layer	0.60	2.01	71:21:8:0	29.09	1.20
MG1 Chromitite Layer					
Parting MG2A-MG1	13.38				
MG1 Chromitite Layer	1.30	0.61	51:31:17:1	30.77	1.24
MG0 Chromitite Layer					
Parting MG1 - MG0	5.52				
MG0 Chromitite Layer	0.59	0.83	61:19:19:1	26.13	1.15

Figure 5.2.2_2

Along strike section showing the variations in MG Chromitite Layer partings

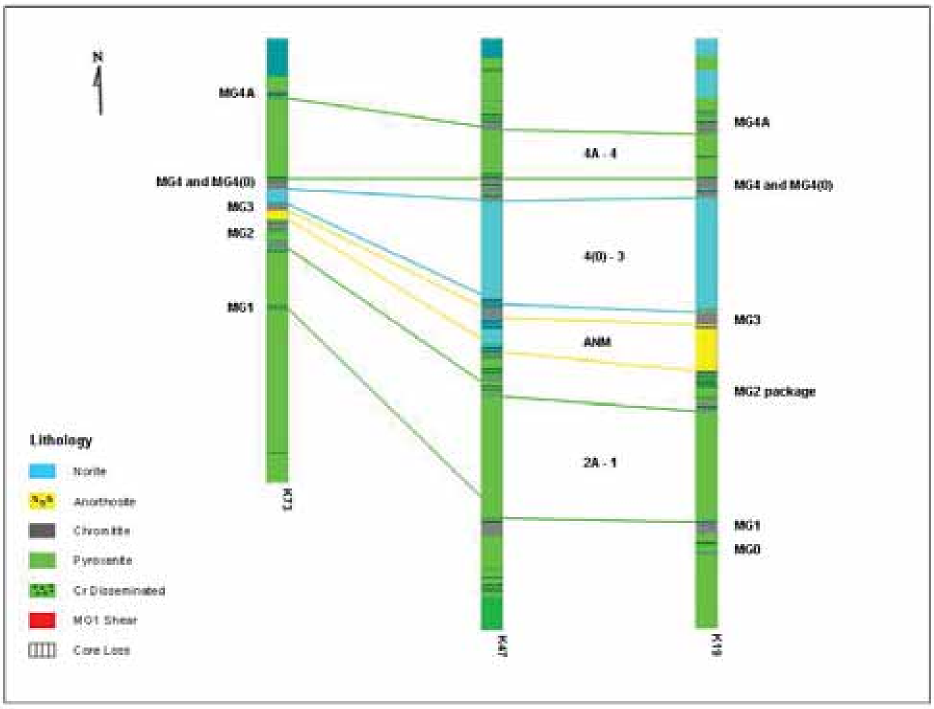
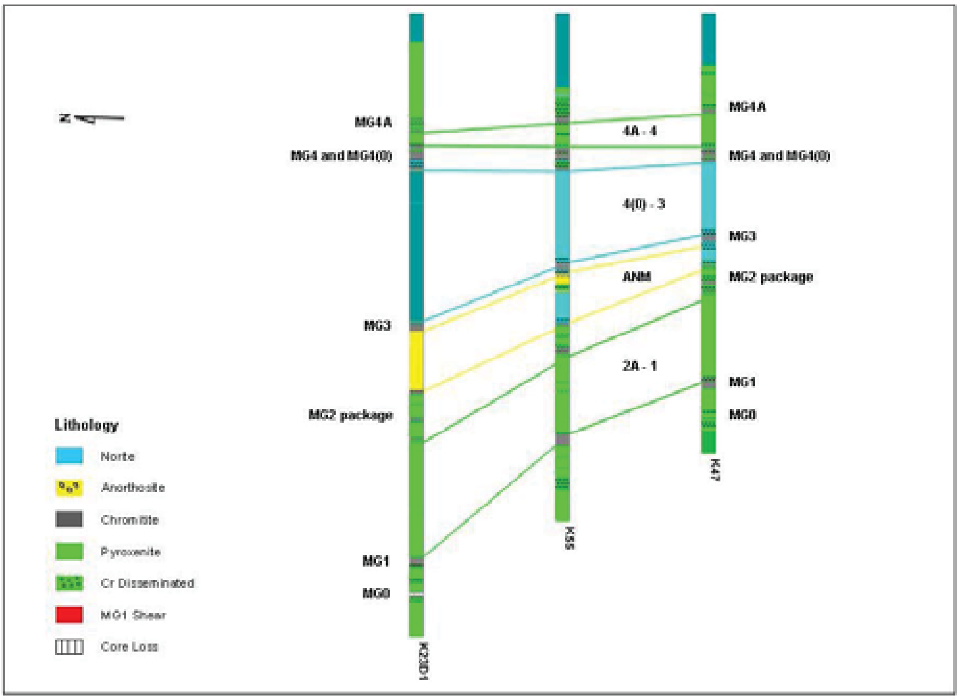


Figure 5.2.2_3

Downdip section showing the variations in MG Chromitite Layer partings



Description of the MG0 Chromitite Layer

Some dissemination and more chromitite layers and stringers are developed in the footwall pyroxenite of the MG1 Chromitite Layer. These are termed the MG0 Chromitite Layer. (Figure 5.2.2_4 shows the thickness based on the borehole intersections. Note the very thin/absent layer in the west. The number of stringers and layers vary and little consistency was noticed within the MG0 Chromitite Layer.

Description of the MG1 Chromitite Layer

At the base of the MG Chromitite Layer Package is the MG1 Chromitite Layer (1.3m thick) with a feldspathic pyroxenite developed above for some 13.4m and which underlies the MG2 Chromitite Layer. The MG1 Chromitite Layer is typically a massive chromitite with minor feldspathic pyroxenite partings or layering. In some areas the MG1 Chromitite Layer has developed into two chromitite layers separated by a feldspathic pyroxenite. A textural feature called mottling is common in both the MG1 Chromitite Layer and MG2B Chromitite Layer. The mottles reflect large rounded individual silicate crystals (5mm in diameter), called oikocrysts (Schurmann, 1998). The MG1 Chromitite Layer becomes thinner to the west with a transition from 1.3m thick in the east to an average of 0.75m thick in the west. The MG1 Chromitite Layer has a relatively simple structure.

Borehole intersections and trench exposures clearly demonstrate that the MG1 Chromitite Layer thins towards the NW near surface and eventually disappears. Although outcrop of the MG1 Chromitite Layer disappears, it was intersected again downdip below 50m depth. It is not uncommon for the MG1 Chromitite Layer to split into more than one layer. The facies outlines defined are single, multiple (where the MG1 Chromitite Layer splits into various bands), thinning and missing (Figure 5.2.2_5). Shearing in and around the MG1 Chromitite Layer is common and can occasionally be present in the hanging wall but is more common within the MG1 Chromitite Layer or its immediate footwall.

Figure 5.2.2_4

MG0 Chromitite Layer Thickness Contours

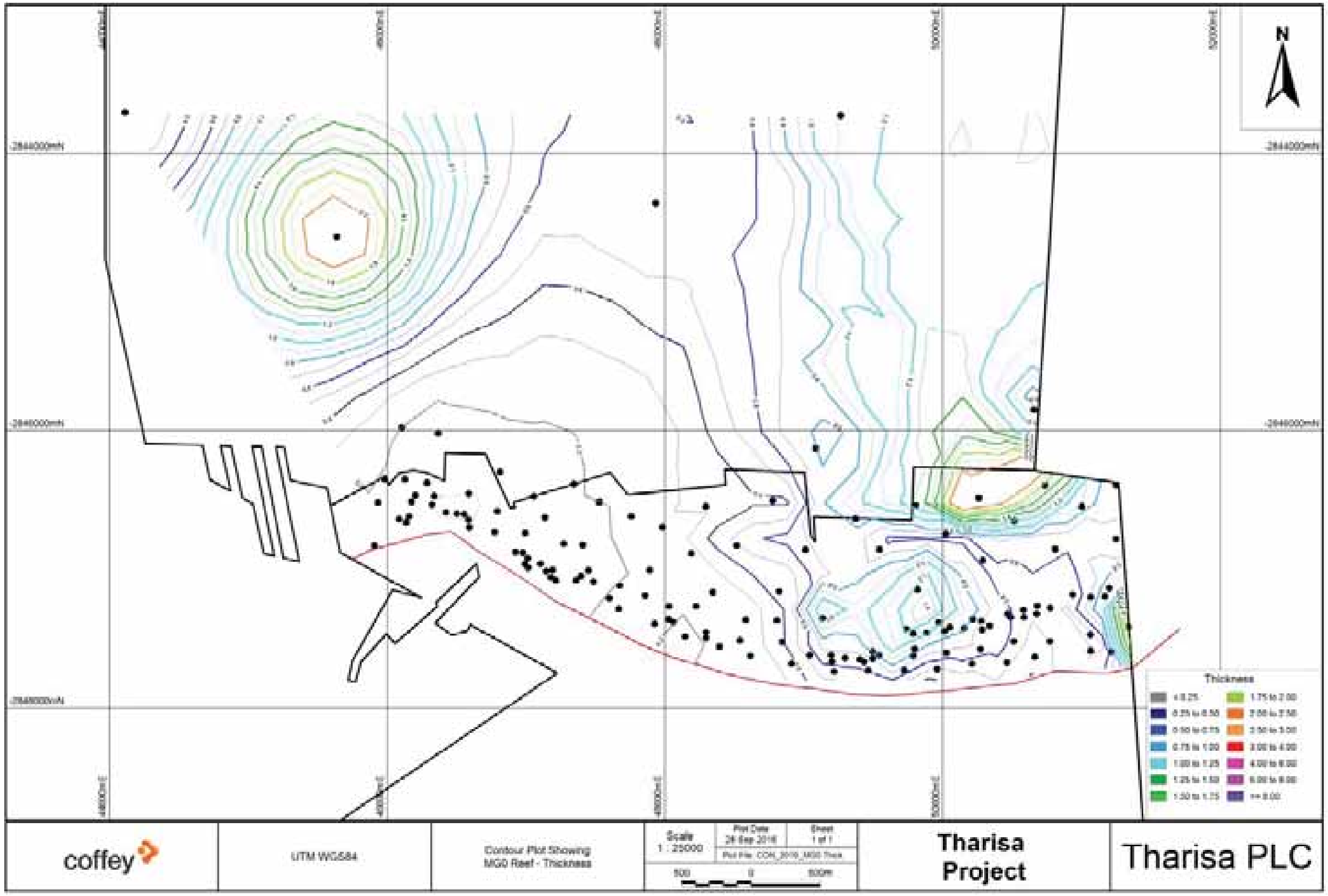
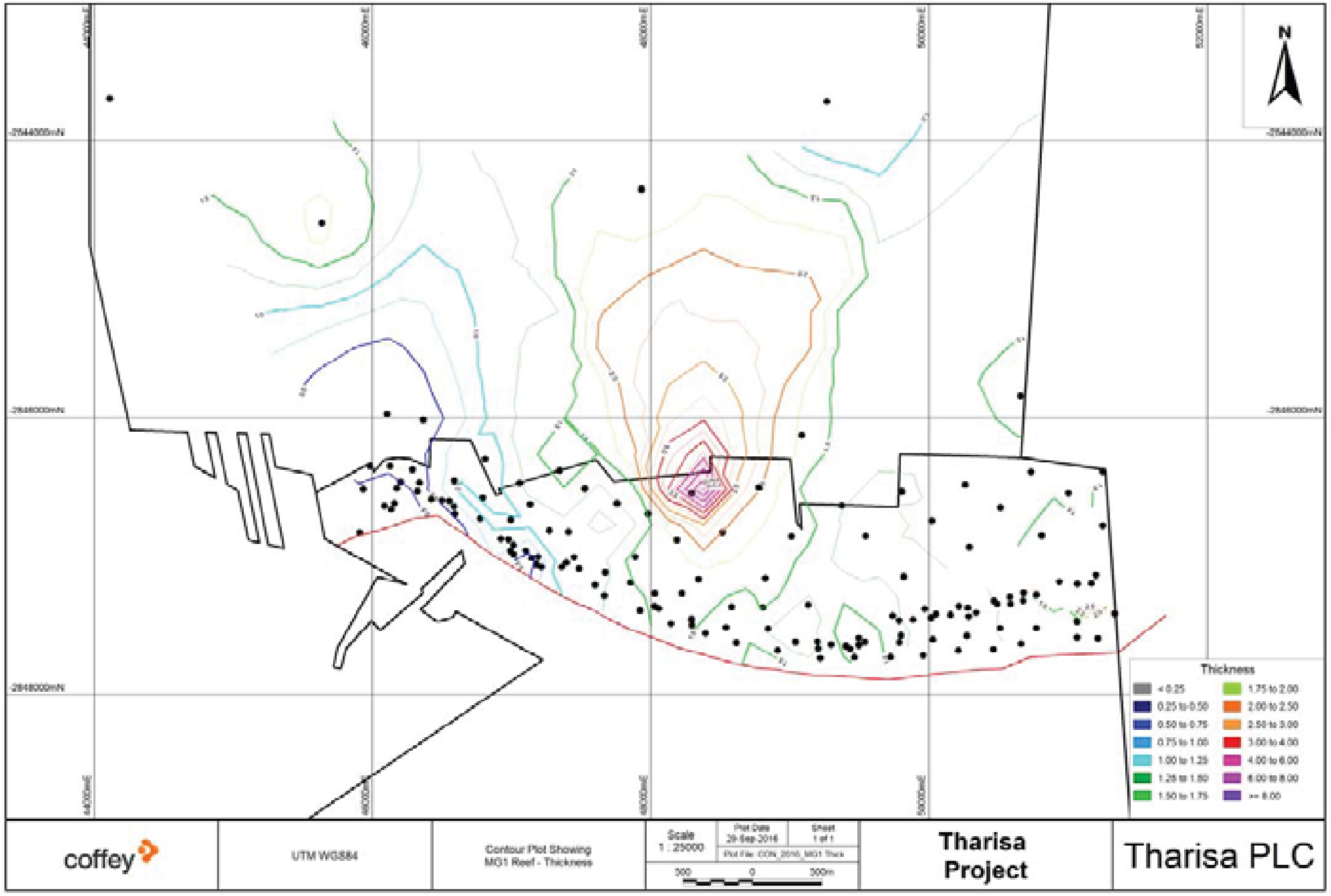


Figure 5.2.2_5

MG1 Chromitite Layer Thickness Contours

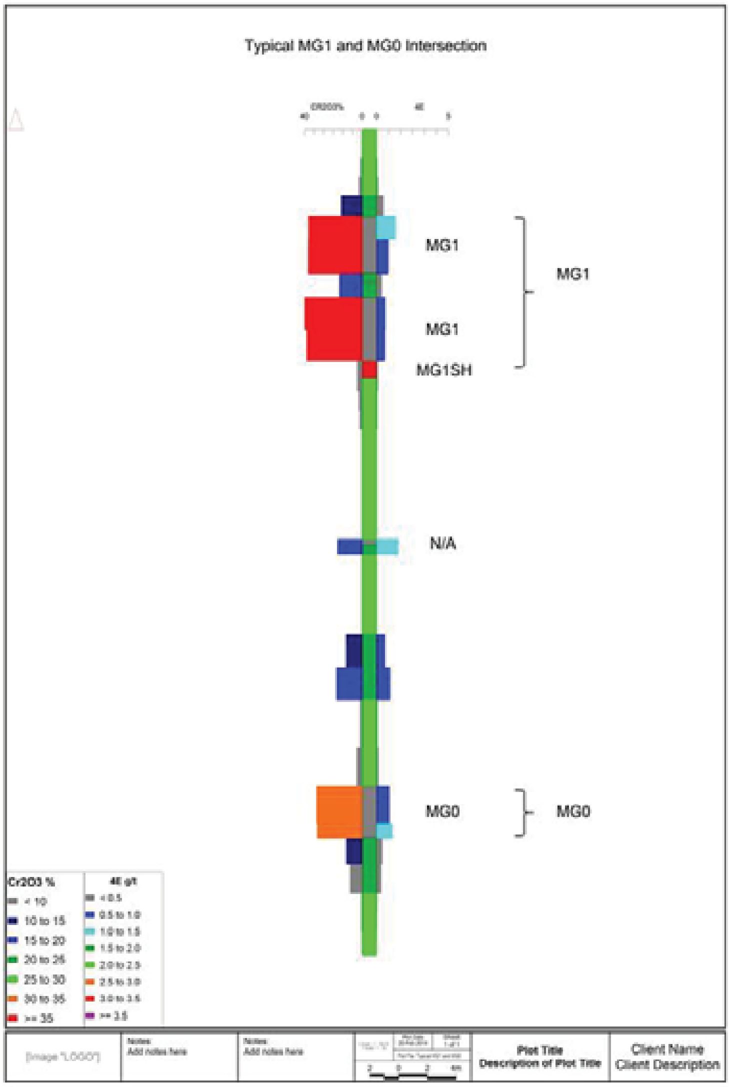


The MG1 Chromitite Layer carries the highest Cr content of all the MG Chromitite Layers with an average Cr2O3 grade of 30.8% and a Cr:Fe ratio of 1.24. The PGM concentration is low (0.6g/t 3PGM+Au). A definite geochemical signature is recognised where the top contact of the MG1 Chromitite Layer has the highest PGM concentrations grading down linearly to its bottom contact (Figure 5.2.2_6).

Midway between the MG1 Chromitite Layer and the overlaying MG2A Chromitite Layer, a thin chromitite stringer or some chromite dissemination is typically present within the felspathic pyroxenite.

Figure 5.2.2_6

Typical MG1 Chromitite Layer Cr_2O_3 and 3PGE+Au geochemical signature



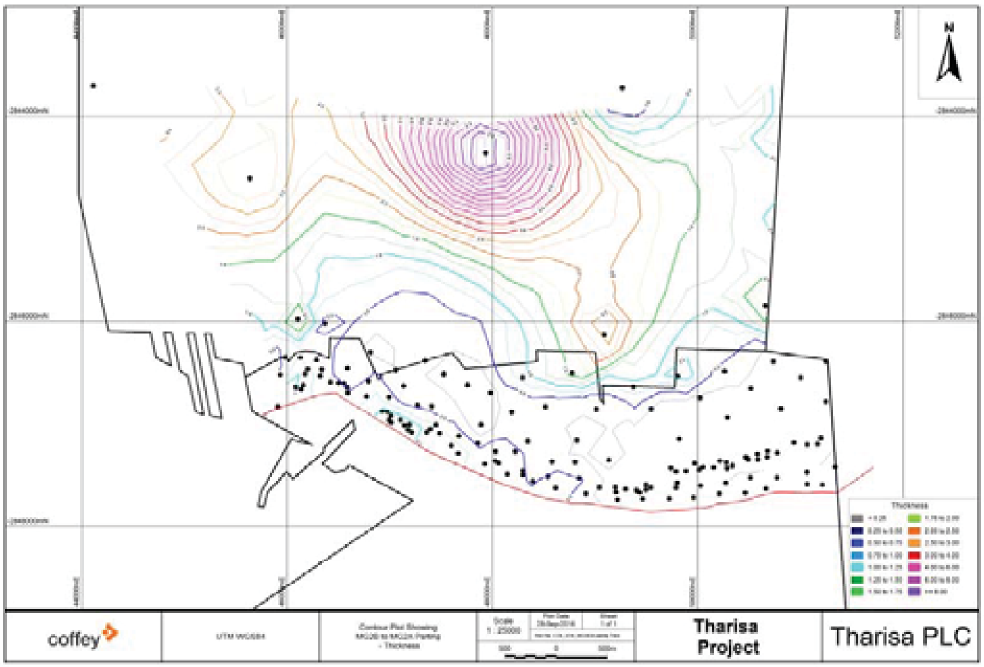
Description of the MG2 Chromitite Layer

The MG2 Chromitite Layer (some 5.4m thick) consists of three groupings of chromitite layers which from the base are the MG2A Chromitite Layer (0.6m thick), MG2B Chromitite Layer (0.6m thick) and the MG2C Chromitite Layer (0.6m thick). The partings are typically feldspathic pyroxenite with the parting between the MG2A Chromitite Layer and MG2B Chromitite Layer being on average 0.9m thick. The parting between the MG2B Chromitite Layer and MG2C Chromitite Layer is typically 2.5m thick and includes a platiniferous chromitite stringer (PGEM). Some 5m above the MG2C Chromitite Layer is the MG3 Chromitite Layer. The parting is generally an anorthosite or norite which forms the overlaying Anorthosite Marker.

The MG2A Chromitite Layer separates from the MG2B Chromitite Layer towards the NW along strike and downdip, with more than a metre separation closer to surface and up to 9m further downdip. Figure 5.2.2_7 presents the parting thickness between the MG2B and MG2A Chromitite Layers.

Figure 5.2.2_7

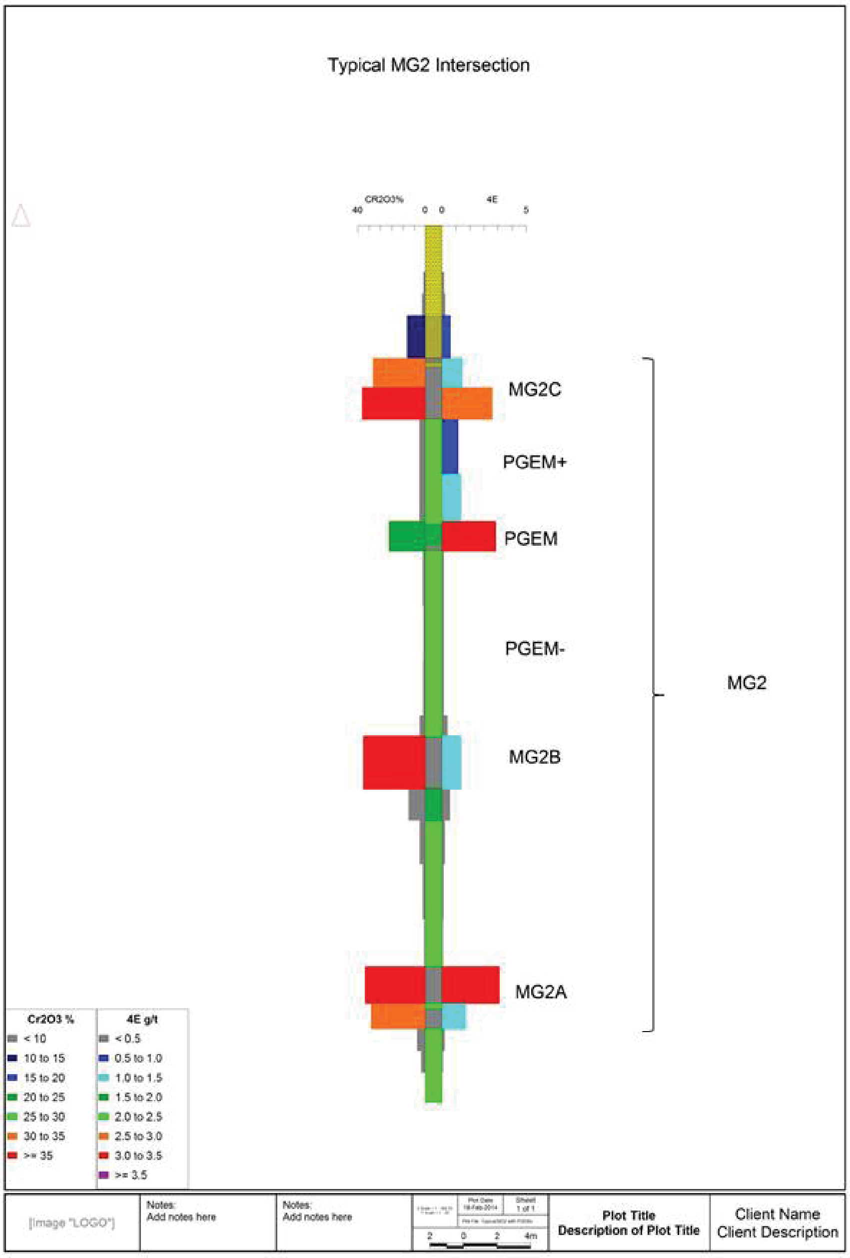
Thickness (Down Hole Intersection Length) isopachs for the parting between MG2B and MG2A Chromitite Layers



The MG2A and MG2B Chromitite Layers occasionally form a single chromitite layer but can be distinguished by a definite analytical signature. PGM concentrations are much higher in the MG2C and MG2A Chromitite Layers (±2g/t (3PGE+Au)) with a much lower concentration in the MG2B Chromitite Layer (±1.3g/t (3PGE+Au)). A few chromitite stringers, disseminated chromite within the middling pyroxenite and sometimes a chromitite layer at the base of these stringers, appear between the MG2C and MG2B Chromitite Layers. These have been coded PGEM and carry the highest concentration of PGMs within the MG2 Chromitite Layer at approximately 2.7g/t (3PGE+Au). A typical geochemical signature is presented in Figure 5.2.2_8. Typically an increase in PGM concentration from the MG2C Chromitite Layer top contact to the MG2C Chromitite Layer bottom contact can be noted. The MG2A Chromitite Layer displays the opposite signature.

Figure 5.2.2_8

Typical MG2 Chromitite Layer Cr_2O_3 and 3PGE+Au geochemical signature



The Anorthosite Marker (ANM), a prominent anorthosite, norite or a combination of the two, separates the MG2 Chromitite Layer from the overlying MG3 Chromitite Layer. Chromitite stringers are often present within the marker close to the top and bottom contacts and they may have high PGM concentration.

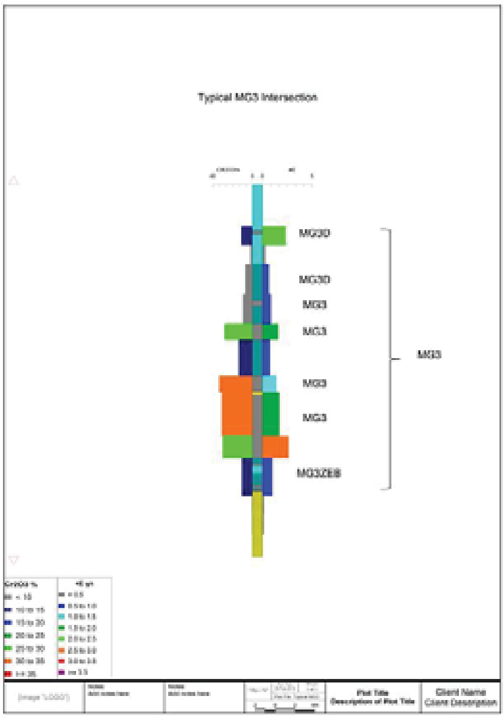
<u>Description of the MG3 Chromitite Layer</u>

The MG3 Chromitite Layer is occasionally a massive chromitite layer but more often a very irregular, assemblage of chromitite layers and stringers within a norite and/or anorthosite, which is difficult to correlate. The top of the package typically consists of thin chromitite stringers and dissemination of chromite in norite which develops into a more massive layer at the base. Due to numerous chromitite layers and stringers comprising the MG3 Chromitite Layer, it is not easy to define the core of the MG3 Chromitite Layer package or the most appropriate mining unit. The mining unit is defined largely by the presence of massive chromitite. The upper or lower limits of the mining cut was defined where the immediate hanging or footwall becomes largely noritic or anorthositic with disseminations of chromite. This typically correlates with the reduction in PGM concentration. The chromitite is mineralised with PGM bearing minerals with the disseminated chromite bearing lithologies being much less mineralised or barren. The top contact of the MG3 Chromitite Layer is not always very clearly defined and hence the use of the bottom contact as the reference contact.

The mining cut of the MG3 Chromitite Layer (2.1m thick) consists of a chromitite with disseminated chromite in a norite or anorthosite immediately above and below the chromitite (Figure 5.2.2_9). The PGM concentrations are very erratic and no definite geochemical signature is defined (Figure 5.2.2_9). The cut determination is based on the ability of the metal content to add value to the overall potential cut. To determine the cumulative contribution, the commodity price, recovery and cost are considered.

Figure 5.2.2_9

Typical MG3 Chromitite Layer Cr_2O_3 and 3PGE+Au geochemical signature



MG3 Chromitite Later: Determinant for the MG3 Disseminated and MG3 Zebra cuts

In 2015 consideration was given to including the material immediately below and above the MG3 Chromitite Layer as being metalliferous and being able to contribute to the revenue of the mine. The cuts were selected on a geological basis with the presence of grade for Cr and/or PGM being considered. A review of this methodology in 2016 suggests that the material does not have "reasonable prospects for eventual economic extraction" as required by the SAMREC Code (2016). The cuts need to be optimised to include technical and economic aspects. The methodology adopted considers where the cumulative cut starting from the MG3 Chromitite Layer, will contribute financially to the operation. In order to assess the impact of these assumptions on revenue, cost and recovery were considered (Table 5.2.2_2).

Table 5.2.2_2
Summary of Economic Assumptions applied to the determination of the MG3 Disseminated and MG3 Zebra cuts.

RoM Feed Grade		Processed value		Price: (Rand)	
6 PGE+Au grade (g/t)	**1.624**	R 467.90	R/tonne	6 PGE+Au basket (R/oz)	R 9,100.00
Chrome grade (%)	**18.8**	R 454.66	R/tonne	Chrome (R/tonne)	R 1,600.00
Total revenue		**R 922.56**	R/tonne		
Plant cost		R 135.00	R/tonne		
Mining cost		R 291.30	R/tonne		
Overhead cost		R 28.00	R/tonne		
Contribution		**R 468.26**	R/tonne		

Based on break even grades for PGMs and chrome

The contribution for each sampling interval was calculated and the cumulative contribution determination made. Where the cut no longer contributed on a cumulative basis, the top MG3 Disseminated) or bottom (MG3 Zebra) of the cut was established (Figure 5.2.2_10 and Figure 5.2.2_11). These cuts were then used in the estimation.

Figure 5.2.2_10

Tabulation showing the methodology to determine the cut for the MG3 Disseminated

BHID	Cr2O3 (%)	6 PGE +Au (g/t)	Cumulative thickness from the top of MG3 CR	Sample width (m)	Reef		Cumulative contribution per tonne	R/ unit
KIP171	5.89	1.101	1.86	0.04	3D	0	- 310.64	**-6.20**
KIP171	5.89	1.101	1.82	0.01	3D	0	- 304.44	**-1.55**
KIP171	5.89	1.101	1.81	0.01	3D	0	- 302.89	**-1.55**
KIP171	2.32	0.422	1.80	0.22	3D	0	- 301.34	**- 230.18**
KIP171	7.05	0.908	1.58	0.16	3D	0	- 71.16	**-45.65**
KIP171	7.05	0.908	1.42	0.02	3D	0	- 25.51	**-5.71**
KIP171	7.05	0.908	1.40	0.01	3D	0	- 19.81	**-2.85**
KIP171	4.60	0.597	1.39	0.25	3D	0	- 16.95	**- 197.93**
KIP171	4.32	0.471	1.14	0.22		1	180.97	**- 198.97**
KIP171	13.70	1.938	0.92	0.01		1	379.94	**15.74**
KIP171	13.70	1.938	0.91	0.02		1	364.20	**31.48**
KIP171	13.70	1.938	0.89	0.12		1	332.73	**188.85**
KIP171	13.70	1.938	0.77	0.03		1	143.87	**47.21**
KIP171	13.70	1.938	0.74	0.04		1	96.66	**62.95**
KIP171	8.50	0.806	0.70	0.2		1	33.71	**-58.29**
KIP171	15.97	1.823	0.50	0.16		1	92.00	**269.05**
KIP171	6.36	0.64	0.34	0.15		0	- 177.05	**-93.99**
KIP171	7.52	0.739	0.19	0.005		0	- 83.05	**-2.19**
KIP171	7.52	0.739	0.19	0.015		0	- 80.87	**-6.56**
KIP171	7.52	0.739	0.17	0.04		0	- 74.31	**-17.49**
KIP171	7.52	0.739	0.13	0.12		0	- 56.83	**-52.46**
KIP171	7.52	0.739	0.01	0.01		0	- 4.37	**-4.37**
MG3 Chromitite								

Figure 5.2.2_11

Tabulation showing the methodology to determine the cut for the MG3 Zebra

BHID	Cr2O3 (%)	6 PGE +Au (g/t)	Cumulative thickness from the base of MG3 CR	Sample width (m)	Reef		Cumulative contribution per tonne	R/ unit
MG3 Chromitite								
K80	0.46	0.23	0.20	0.20		0	-202.44	- 202.44
K80	0.54	0.23	0.40	0.20		0	-405.07	- 202.63
K80	26.10	3.14	0.52	0.13		1	171.35	576.42
K80	26.10	3.14	0.55	0.03		1	286.63	115.28
K80	8.60	1.61	0.56	0.01		1	290.16	3.53
K80	8.60	1.61	0.59	0.02		1	298.98	8.82
K80	8.60	1.61	0.70	0.12		1	339.57	40.59
K80	1.21	0.22	1.18	0.48	3ZEB	0	-154.17	- 493.74
K8	1.22	0.23	1.38	0.20	3ZEB	0	-363.44	- 209.27
K80	3.50	0.45	1.40	0.02	3ZEB	0	-381.43	-17.99
K80	3.50	0.45	1.60	0.20	3ZEB	0	-556.86	- 175.43
K80	17.84	0.93	1.61	0.02	3ZEB	0	-553.50	3.36
K80	17.84	0.93	1.67	0.05	3ZEB	0	-541.20	12.30
K80	0.46	0.23	0.20	0.20	3ZEB	0	-202.44	- 202.44

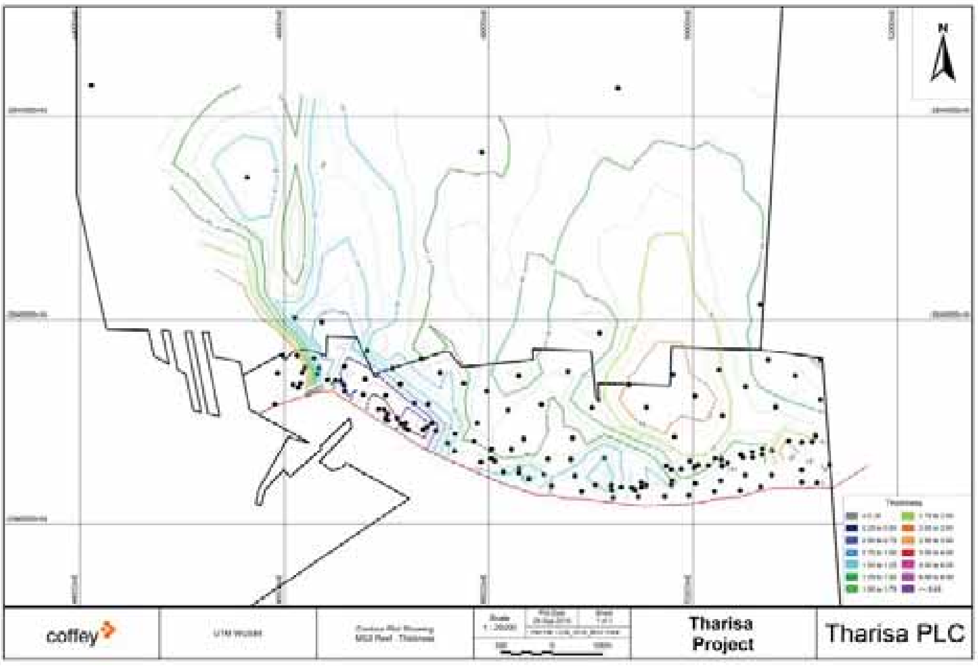
Above the massive MG3 Chromitite Layer, a layer containing disseminated chromitite with an average thickness of 0.9m has been identified. This unit has sufficient lateral continuity that it has been possible to identify it in within the open pit and within exploration boreholes. The unit is referred to as the MG3 Disseminated or Hangingwall and coded as MG3D.

Immediately below the massive MG3 Chromitite Layer a zone in which chromitite layers are developed between layer of anorthosite and norite or disseminated within these lithologies, is developed with an average thickness of 0.5m. This zone is also of sufficient lateral continuity such that it has been possible to identify and was considered of economic significance. The zone is referred to as the MG3 Zebra because of the stripey appearance.

Based on geological and geochemical features, various facies of the MG3 Chromitite Layer can be defined (Figure 5.2.2_12). The MG4(0) Chromitite Layer is some 12m above the MG3 Chromitite Layer.

Figure 5.2.2_12

MG3 Chromitite Layer Thickness (m) Contour Plot



Description of the MG4 Chromitite Layer and MG4(0) Chromitite Layer

The MG4 Chromitite Layer consists of a lower chromitite (MG4(0) Chromitite Layer) (approximately 0.6m thick) immediately overlain by a norite (approximately 0.85m thick) followed by the chromitite layer of the MG4 Chromitite Layer (approximately 1.5m thick), overlain by another parting, of feldspathic pyroxenite composition, some 4m thick and finally overlain by the chromitite of the MG4A Chromitite Layer (approximately 1.5m thick).

The MG4 Chromitite Layer is consistent throughout the property in that it has a pyroxenite hangingwall and a norite footwall. At its base a chromitite layer (or layers) namely the MG4(0)

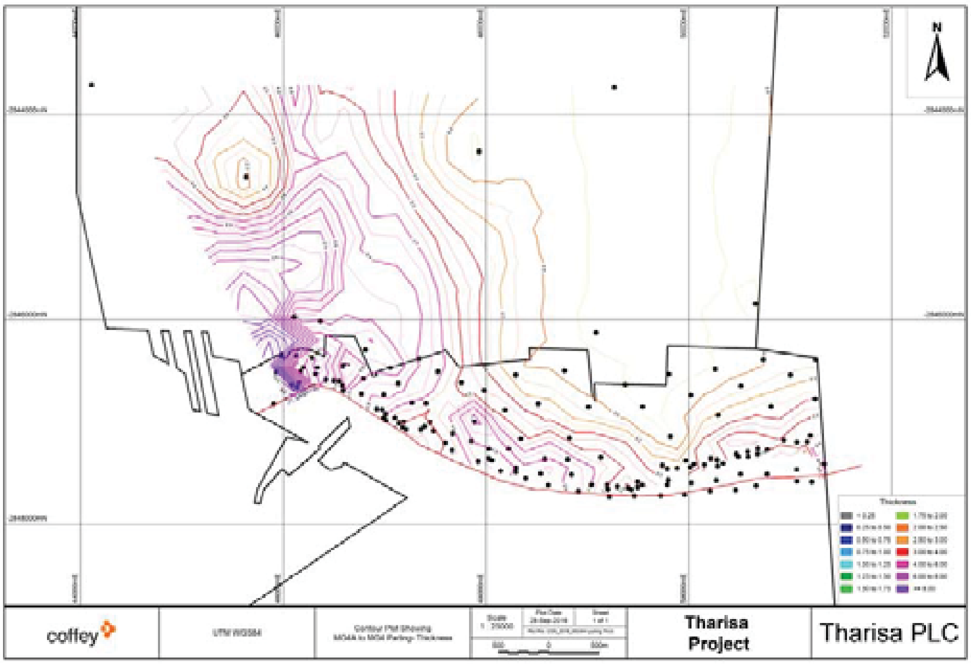
Chromitite Layer. This subdivision is based on a geochemical signature which does not necessarily correspond to an obvious parting above the last chromitite layer.

The MG4 Chromitite Layer has a relatively simple structure similar to the MG1, MG2 and MG3 Chromitite Layers.

Both the MG4 and MG4(0) Chromitite Layers may comprise more than one chromitite layer. The parting between MG4 and MG4(0) Chromitite Layers is mostly a norite with disseminated chromite or disseminated chromite in pyroxenite. The parting is up to 2.5m thick at its thickest but can also be entirely absent. Based on the geology of the MG4 and MG4(0) Chromitite Layers, various facies are defined (Figure 5.2.2_13).

Figure 5.2.2_13

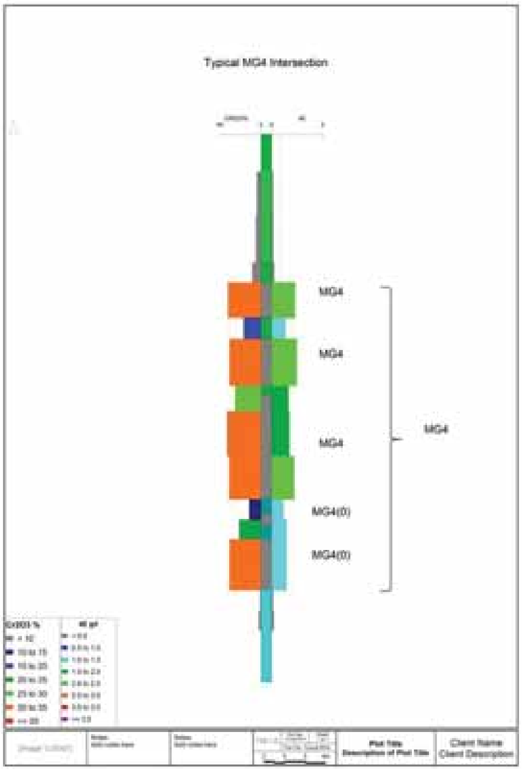
Contour Plot of the Separation between MG4(0) and MG4 Chromitite Layers



The typical geochemical signatures of MG4 and MG4(0) Chromitite Layers are presented in Figure 5.2.2_14. The PGM concentration of the MG4(0) Chromitite Layer is approximately 1.3g/t (3PGE+Au) lower than the grade of the MG4 Chromitite Layer which has a PGM concentration of approximately 1.7g/t (3PGE+Au).

Figure 5.2.2_14

Typical MG4(0) and MG4 Chromitite Layer Cr_2O_3 and 3PGE+Au geochemical signature



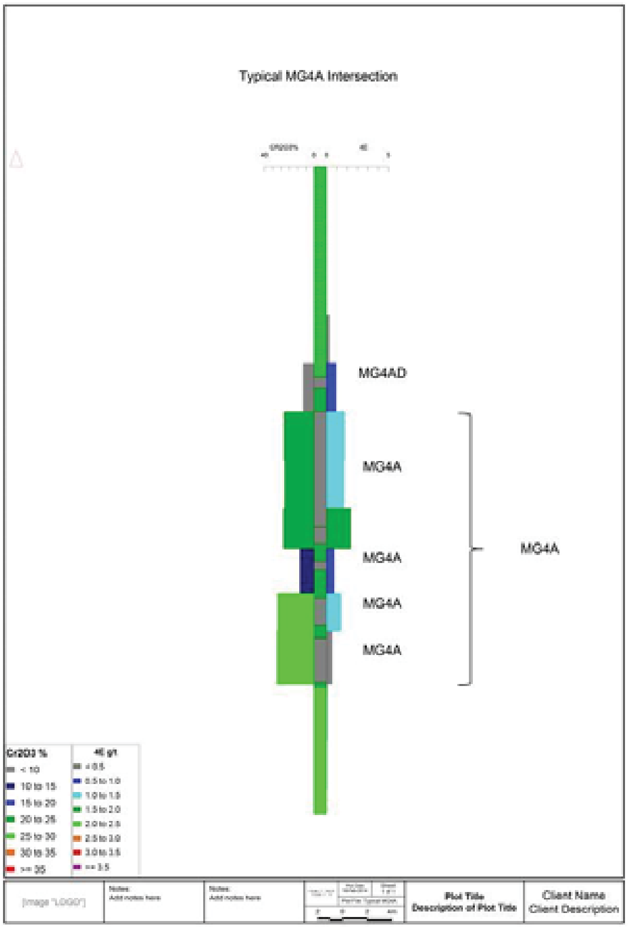
Description of the MG4A Chromitite Layer

Above the MG4 Chromitite Layer is a 4m thick feldspathic pyroxenite parting overlain by the chromitite of the MG4A Chromitite Layer (1.5m thick). The MG4A Chromitite Layer consists of a number of chromitite layers within a pyroxenite host rock. Midway between the MG4A and MG4 Chromitite Layers, chromitite stringers and disseminated chromite may be present. The MG4A Chromitite Layer, as with the MG3 Chromitite Layer, has a less well defined top contact and hence the bottom contact was contoured. A norite/melanorite is consistent prelude to the pyroxenite in the hanging wall of the MG4A Chromitite Layer.

The concentrations of Cr_2O_3 and PGM in the MG4A Chromitite Layer are low at 25% and 0.7g/t (3PGE+Au) respectively. The typical geochemical profile is presented as Figure 5.2.2_15.

Figure 5.2.2_15

Typical MG4A Chromitite Layer Cr_2O_3 and 3PGE+Au geochemical signature



5.3 Geology of the UG1 Chromitite Layer

The UG1 Chromitite Layer is stratigraphically situated in the Upper Critical Zone and is well developed in the Bushveld Complex. It comprises the massive chromitite, chromitiferous pyroxenite, bands of anorthosite, chromitite and norites and stringers of chromitites. The UG1 Chromitite Layer has a strike direction of east-west and dips to the north with the dip varying from 10º in the east to 25º in the west.

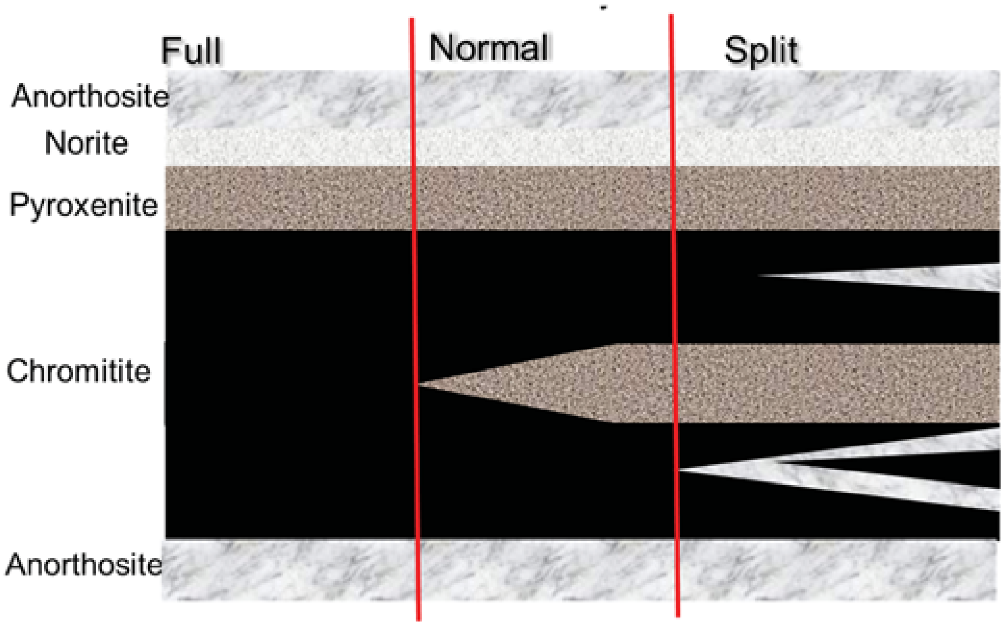
The thickness of the UG1 Chromitite Layer ranges from few centimetres up to 3m in places. The lenses of anorthosite and pyroxenite are seen impregnated with numerous chromite grains in places. The hanging wall changes from pyroxenite to anorthositic norites. The footwall is formed by bifurcated bands of anorthosite and chromite lenses.

At Tharisa Mine, the UG1 Chromitite Layer has three distinguishable facies (Figure 5.3_1):

- Full UG1 Chromitite Layer
- Normal Reef
- Split Reef Facies

Figure 5.3_1

Schematic Diagram of the UG1 Chromitite Layer Model



5.3.1 Full UG1 Chromitite Layer

This facies contributes 1% of the UG1 Chromitite Layer at Tharisa Mine. It is more prevalent to the west. It comprises a single massive chromitite layer with an average thickness of 2.5m.

5.3.2 Normal Reef

The Normal Reef facies of the UG1 Chromitite Layer comprises the massive chromitite with 10cm to 100cm internal waste. The top and bottom chromitite layers have different geochemistry signatures suggesting that they were formed under different conditions and from different sources. The thicknesses of top and bottom layers differ considerably throughout the property. The thickness varies from 0.5m to 1.50m per layer.

This facies contribute 95% of the UG1 Chromitite Layer in the property.

5.3.3 Split Reef Facies

The Split Reef facies contributes 4% of the UG1 Chromitite Layer at Tharisa. It comprises of numerous layers of chromitite, anorthosite and pyroxenite as shown in Figure 5.3.3_1.

The UG1 Chromitite Layer is affected by geological structures such as reef rolls, faults, potholing, intrusives such as iron-rich ultramafic pegmatites and dykes.

Figure 5.3.3_1

Structures in the UG1 Chromitite Layer

5.4 Structure

The structural interpretation of the Tharisa Mine area is based on the aeromagnetic data and the drilling data. The MG Chromitite Layers at the Tharisa Mine are a stack of tabular deposits.

An Air Tractor 402A aircraft was used to conduct a high resolution aeromagnetic survey over the Tharisa Mine area during August 2007 (Figure 5.4_1). Total field magnetics were calculated with the use of 2 Cesium Vapour magnetometers. A DTM was constructed using real time differential GPS and a laser altimeter. A total of 900 line-km were covered. The survey lines were 0 degrees (true north) with 100m spacing. Tie lines perpendicular to the survey lines were spaced at 500m. Sample spacing was at 6.5m along the flight lines and ground clearance was 40m.

The only significant fault in the mine area is a steeply dipping NW-SE trending normal fault (Figure 5.4_1) with a downthrow of less than 30m to the east. This fault occurs only on the far north-eastern corner of the property and will have little effect on mining of the MG Chromitite Layers on Farm 342JQ. This fault was confirmed in both Lonmin underground operations and Samancor stopes.

A low angled WNW-ESE trending thrust fault (Lonmin interpretation) is a prominent lineation on the aeromagnetic image. The fault is expected to have little impact on the mining of the MG Chromitite Layers.

A NE-SW striking sub-vertical dyke of approximately 10m thickness was interpreted from the aeromagnetic survey. This dyke was not fully intersected in any of the boreholes but was intersected in the East Mine box-cut and is 11m wide.

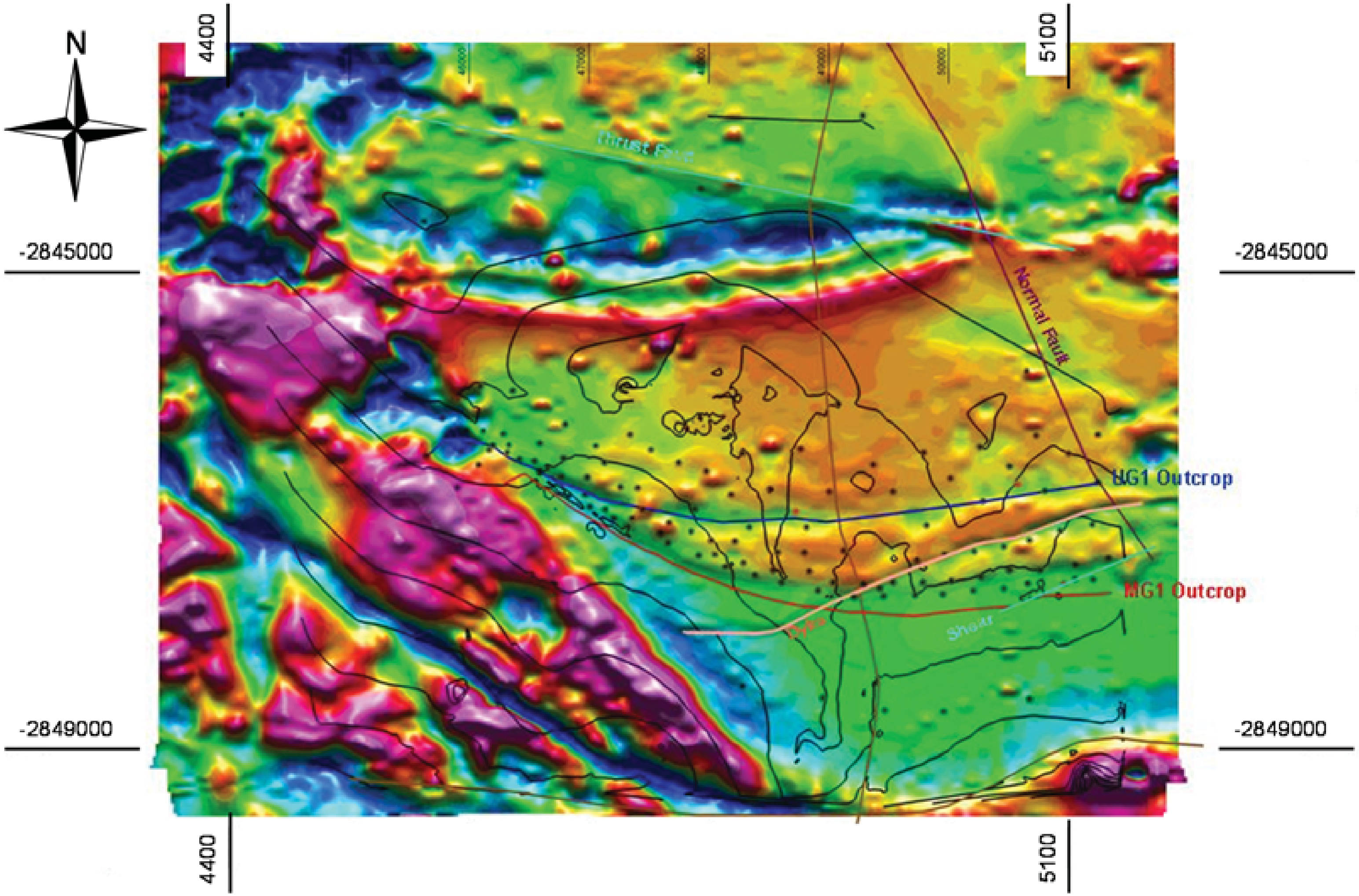
A NE-SW trending sub-vertical shear is exposed in the far eastern pit on Farm 342JQ. Evidence of this shear was seen in boreholes K94, K6A and K20. It is evident as a lineation on the aeromagnetic survey. The MG1 Chromitite Layer thickness is reduced around the shear. Future open pit activities are not affected as the thinned MG1 Chromitite Layer has already been exploited in the area around the shear.

An aeromagnetic anomaly north of the MG Chromitite outcrops, following the north-westerly curve along strike is interpreted as the anorthosite and norite in the UG1 Chromitite Layer footwall.

The only other major structural feature of interest is the Spruitfontein upfold or pothole to the west of the Tharisa Mine. It affects the UG2 Chromitite Layer as well as the rest of the Critical Zone below. The area around the pothole which is on the adjacent property was not accessible to further investigation.

Figure 5.4_1

Map Showing High Resolution Aeromagnetic Survey Interpretation



6 EXPLORATION AND DRILLING

6.1 Previous Exploration

The Tharisa Mine area has been explored for its mineral potential since the early 1900s. Initially this was in the form of erratic exploration activities which included trenching and small open pits.

6.2 Exploration by Thari

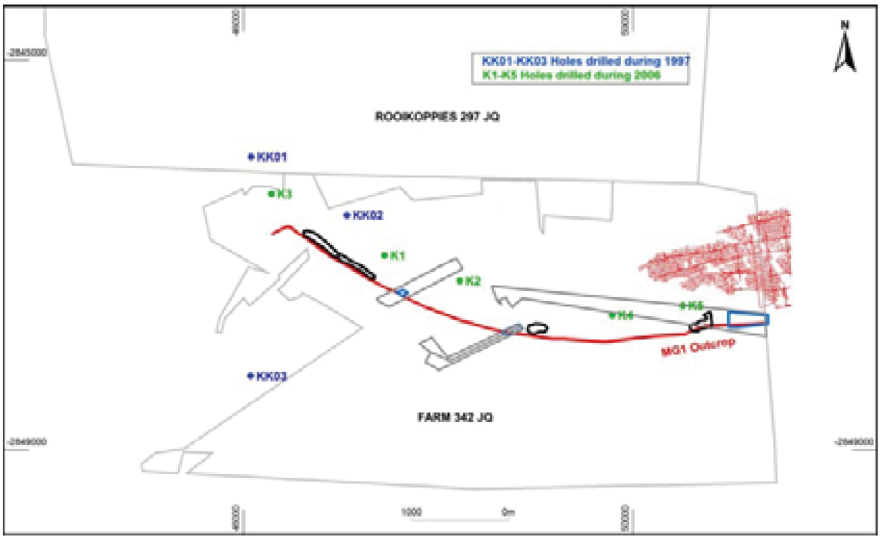
The mineral resource estimate is based predominately on a diamond drilling exploration programme managed by Coffey in 2007. Trenching was undertaken and utilised for geological understanding and geological modelling. Drilling for metallurgical sampling purposes was also undertaken but the associated assay data was not included in this modelling.

6.3 Trenching and Pit Excavations

Various trenches were historically excavated on both the UG1 and the MG Chromitite Layers. During the 2007 exploration programme additional trenching was undertaken on the MG Chromitite Layers. The MG Chromitite Layers were previously exploited from three known pits, excavated by previous tenement holders and which remain unrehabilitated. An additional two pits, one on portion 96 (Farm 342JQ) and another on portions 361/362 (Farm 342JQ), were excavated and exposed the lower half of the MG Chromitite Layer package and were subsequently rehabilitated (backfilled). A sixth pit was opened and backfilled during 2007 on portion 286 of Farm 342JQ. The details of these excavations are presented in Figure 6.3_1. A photograph taken in 2006 of the pit on portion 286 (Farm 342JQ) is presented in Figure 6.3_2. The MG1 Chromitite Layer was mined out underground by Samancor on the eastern side of the Farm 342JQ property.

Figure 6.3_1

Map Showing Distribution of the Historic Boreholes, Various Pits and the Stoped Out MG1 Chromitite Layer on the Eastern Side of Farm 342JQ.



Pits shown in blue have been rehabilitated

Figure 6.3_2

Photograph of Pit Sidewall showing the relationship of the various MG Chromitite Layers (Pit on Portion 286)

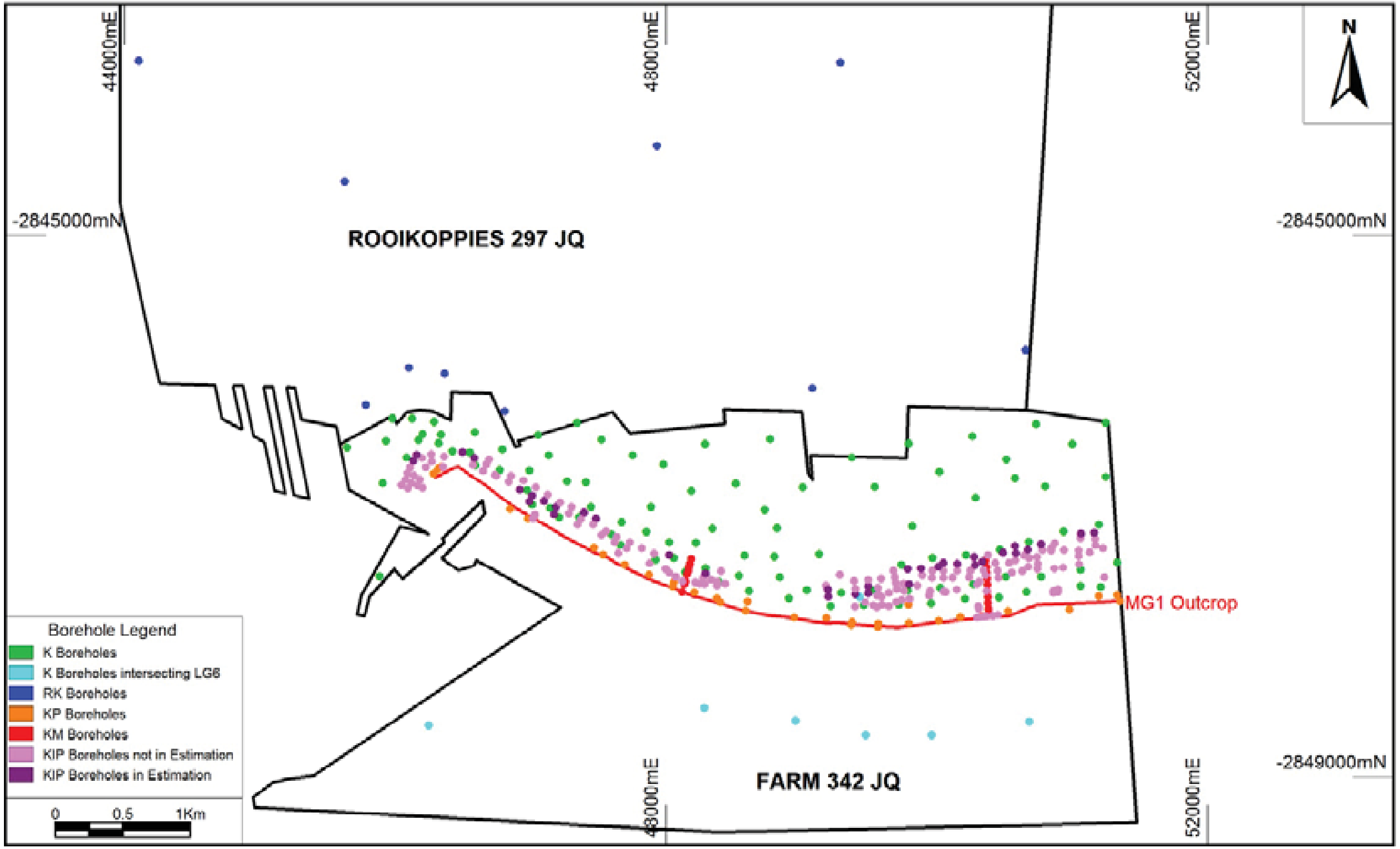
6.4 Drilling

Six diamond boreholes were drilled during January 1997 by a local entrepreneur, Mr Hennie Botha, in the northwest part of Farm 342JQ property (K01, K02 and K03) and on the adjacent property, Spruitfontein 341JQ (BSB01, BSB02 and BSB03). A report was subsequently compiled by LW Schurmann. The only data available from this exploration programme are five of the logs included in the report. The core was not made available to Coffey. The original logs provide insufficient and inaccurate detail compared to geology of diamond boreholes drilled nearby during the 2007 Thari drilling programme. The collar positions could also not be verified. The data is therefore considered unreliable and was not included in the mineral resource estimate.

Five NQ diameter, vertical diamond boreholes totalling 654m were drilled along strike on Farm 342JQ during 2006 by Thari under the supervision of Coffey. One TNW diameter diamond borehole (K4M1) was drilled 5m away from K4 for metallurgical testwork. The collar positions of these boreholes were surveyed by Clive Macintosh Surveys.

Figure 6.4_1

Location Plan for Drilling Programme on the Tharisa Mine



A total of 121 vertical boreholes and 23 deflections, representing 22,500m of drilling were completed in the period from March 2007 to October 2007 (Figure 6.4_1). Drilling was mainly of NQ (47.50mm) diameter except for 18 boreholes of TNW (60.4mm) diameter completed for metallurgical testwork. Four deep boreholes drilled on Rooikoppies were drilled BQ (36.27mm) diameter. A total of 13 NQ diameter deflections were drilled off selected mother boreholes for lithological comparison. Ten TNW diameter deflections were drilled to contribute bulk material for the metallurgical testwork. Shallow percussion boreholes were drilled along the full strike extent on the MG1 Chromitite Layer, on the Farm 342JQ property, to accurately demarcate it. A total of 31 boreholes were drilled (see orange coloured collars in Figure 6.4_1); the boreholes averaged 15m in depth. All borehole locations were clearly marked with cement beacons and a PVC rod. However, where the land has since been cultivated or illegally occupied, the beacons have been either displaced or destroyed.

The drilling programme was designed so that boreholes would intersect the base of the MG1 Chromitite Layer at approximately 30m, 60m, 120m, 180m, 300m, 500m and 1000m below surface. A line of boreholes that intersected at 220m below surface later added for greater coverage of the deposit. The drilling programme was designed to drill the deposit closest to the outcrop at higher density than further downdip so that the subsequent mineral resource estimate close to the outcrop could confidently be declared as an indicated and/or measured mineral resource in preparation for a feasibility study and the consideration of open pit mining. The programme for the deeper boreholes on the Rooikoppies property, where Lonmin was then mining the Merensky Reef and UG2 Chromitite Layer, was revised due to various difficulties relating to siting the boreholes to avoid holing into existing underground infrastructure. Fewer, more widely spaced boreholes were therefore drilled.

Two fence lines (oriented in the down dip direction) were drilled with TNW diameter core for metallurgical test purposes, intersecting the chromitite layers at 10m depth increments down to 60m below surface on the MG4 Chromitite Layer. These boreholes are shown in red on Figure 6.4_1 as KM101 to KM120.

Two NQ boreholes, K96 and K24, were drilled at the request of Coffey for geotechnical logging, sampling and to conduct rock strength tests.

Six sterilisation boreholes (K100 and K124 to K128 indicated in cyan, Figure 6.4_1) were drilled around the proposed civil engineering sites which coincide with the LG6 Chromitite Layer outcrop. One borehole, K95, was drilled to intersect both the MG Chromitite Layer package and the LG Chromitite Layer package.

A total of 10 boreholes (in dark blue Figure 6.4_1) were drilled on the Rooikoppies property to test the extension of the MG Chromitite Layer package down dip.

The X, Y and Z coordinates of all drill collars have been accurately determined by a qualified surveyor of Trevor Cufflin Surveys cc. Downhole surveys were undertaken on all the boreholes drilled deeper than 120m by Reflex Africa.

The surface topography data was generated from an airborne survey.

All diamond drilling was undertaken by reputable drilling contractors to industry standard. Core recoveries were estimated to average >95%. Intersections of mineralisation with lower than 95% core recovery were redrilled. Core recovery over the MG1 Chromitite Layer averaged 80% due to the presence of a fault gouge commonly present or adjacent to the MG1 Chromitite Layer. The fault gouge within the more competent rock rendered core loss inevitable.

Since the commencement of the open pit mine, 255 diamond drill holes (KIP Boreholes) have been drilled inside the operating open pit mine on a 50m (NS) x 100m (EW) grid ahead of the mining for exploration and grade control purposes. These holes were drilled vertically to produce NQ size ore. Some 35 boreholes (21 boreholes from East Pit and 14 from West Pit) were selected in 2016 (Figure 6.4_1) and were completely re-logged and sampled, with the samples being sent to Intertek Genalysis Laboratory Services in Johannesburg for assay. An appropriate QA/QC programme was instituted. The position of the boreholes was surveyored by the mine surveyors. Due to the short length of hole, it was not deemed necessary to undertake downhole surveys.

The core was originally logged by the pit geologists. The selected holes were re-logged and sampled by GeoActiv. The protocols used were similar to those used in the original drilling campaign making the data comparable.

These boreholes are considered to provide a significant amount of information close to the current workings and as a result it has been possible to review the mineral resource classification.

6.5 Logging of Boreholes

A detailed geological log of each borehole was undertaken. A geotechnician marked 1m intervals on the core with a black paint marker prior to logging by a geologist. Core was logged in detail, coding the various lithologies, dip angles, grain size, rock texture, alteration, weathering, mineralisation and structures. Chromitite layers were assigned friability (friable, semi-friable or hard) and were coded in a separate stratigraphic column on the logsheets.

Data from these hardcopy logsheets were captured into a SABLE database and validated.

For all chromitite layer intersections below 60m depth a rock quality designation (RQD) was calculated starting 20m above the reef top contact. A RQD for each drill run length was calculated. Intersections within the run length with joints/fractures less than 10cm apart were measured with a clinorule and all these lengths were added together and the total then subtracted from the total drill run length. A percentage of intact core (>10cm pieces) was then recorded as the RQD for that run length.

6.6 Sampling and Data Verification

After logging, representative samples over various chromitite layer intersections were marked out on the core with a paint marker. Unique sample numbers were assigned and information for each sample recorded in a sample ticket book. Core with samples marked out was photographed with a digital camera both dry and wet. Subsequently the core was cut in half vertically along its length and across to obtain the marked out samples. Only half core was submitted for analyses. The other half was retained in the core tray for future reference.

The focus during sampling was to choose sample intervals according to lithologies in order to separate the chromitite from the host rock. Each designated unit (MG1, MG2, MG3, MG4(0) and MG4 Chromitite Layer) was sampled such that the geochemistry of the unit could be investigated.

The units were sampled as indicated below:

- The MG4 Chromitite Layer was sampled continuously from the top of MG4A Chromitite Layer to the base of MG4(0) Chromitite Layer separating the chromitite within into different samples.
- The MG3 Chromitite Layer was sampled continuously from the bottom to the top contact. Above the chromitite layer, sampling the core was sampled to a point where chromite was no longer viisble and coded as MG3 Disseminated. Below the chromitie layer, the core contining chromitite stringers and disseminations was sampled and coded as MG3 Zebra.
- The MG2 Chromitite Layer was sampled continuously from the base of the MG2A Chromitite Layer to the top of the MG2C Chromitite Layer. The sampling was also undertaken so as to obtain the geochemical signatures of the chromitite layers separately from the partings.
- The MG1 Chromitite Layer and MG0 Chromitite Layer were sampled continuously from the bottom contact to the top contact.
- Two non-mineralised footwall and hangingwall samples were taken.

Sample intervals varied from an absolute minimum of 15cm for NQ core (20cm for BQ) to a maximum of 50cm. Chromitite samples included a 0.5 to 2cm host rock margin to avoid PGM and chrome loss during the core cutting process. This is the recognised standard for sampling of PGM deposits in the industry

Quality control monitoring protocols involved submission of sample blanks, duplicates and certified standards with the core sample batches. AMIS0010 and SARM8 were originally alternated as standards but AMIS0010 was later replaced with AMIS0006 due to lack of availability of AMIS0010. In the 2016 campaign AMIS0027, AMIS0053, AMIS0064 and AMIS0075 were used as standards.

Each sample was bagged separately with a numbered ticket inside the bag and the sample number also written on the outside of the sample bag. A dispatch form was submitted along with samples to ensure the total number of samples and correct sample numbers were recorded.

The sampling methodology is appropriate and supports the mineral resource estimate and classification made.

6.6.1 Analytical Procedures

Pre 2016 Drilling Campaigns

Analyses were undertaken by Genalysis, a certified laboratory. Genalysis is an accredited Laboratory with the South African National Analytical Standards (SANAS) with reference number T0464-11-2013.

Sample preparation was undertaken in the Genalysis facility in Johannesburg prior to a pulp being air freighted to Genalysis Perth for analysis. The sample preparation was undertaken using a jaw crusher to crush samples to minus 10mm in size. Pulverising of the samples was undertaken to achieve 85% minus 75µm in size. All samples were assayed for PGM by 7E NiS/MS and for base metals by ICP Fusion D/OES.

Detection limits are presented in Table 6.6.1_1.

Table 6.6.1_1

Detection Limits Applicable to Pre 2016 Sampling

Pt	2	Cu	20
Pd	2	Ni	20
Rh	1	Cr	50
Ru	2		
Os	2		
Ir	2		
Au	5		

2016 Sampling Campaign

Analyses were undertaken by Intertek Genalysis, a certified laboratory. Genalysis is an accredited Laboratory with the South African National Analytical Standards (SANAS) with reference number T0464-04-2016. The precious metal analyses are undertaken in Perth, Western Australia. This facility is certified by the National Association of Testing Authorities, Australia (NATA).

The samples were analysed for the PGMs using the Nickel Sulphide Lead Collection (Ni/S) fire assay method. The precious metals are collected in a nickel sulphide matte which is dissolved leaving the Au and PGEs as a residue. This residue is filtered off, dissolved in aqua regia and read on an ICP-MS for low ppb detection limits (Table 6.6.1_2).

The base metals were analysed by Li borate fusion XRF with a single point LOI (1000°C). The method recognises the highly refractory nature of chromite ores which requires a specialist approach in the fusion process to ensure that the spinel structure is decomposed and the entire sample is dissolved in the fusion disk.

Table 6.6.1_2

Detection Limits and Analytical Techniques Applicable to 2016 Sampling Campaign

Element	Description	Detection Limits	Element	Description	Detection Limits
Cr_2O_3	XRF	0.005%-100%	MgO	XRF	0.01%-100%
SiO_2		0.01%-100%	Al_2O_3		0.01%-100%
MnO		0.01%-100%	SO_3		0.002%-100%
CaO		0.01%-100%	Na_2O		0.01%-100%
TiO_2		0.01%-100%	Cu		0.005%- 5%
Ni		0.005% - 5%	V_2O_5		0.005%-10%
Fe_2O_3		0.01%-100%	P_2O_5		0.002%-100%
LOI	1000°C	0.01%- 100%	K_2O		0.01%-100%
Element	**Description**	**Lower Detection Limit (ppb)**	**Element**	**Description**	**Lower Detection Limit (ppb)**
Pt	25g NiS fire assay / ICP-MS	1	Os	25g NiS fire assay / ICP-MS	1
Pd		1	Ir		1
Rh		1	Au		2
Ru		1			

The assay techniques used are considered appropriate for the PGM and base metal analyses and the mineral resource estimate.

6.6.2 Analytical Quality Control Data

A comprehensive QA/QC programme was undertaken, both for the original drilling and for the subsequent sampling campiagn in 2016. The QA/QC programme identifies various aspects of the results that could have negatively influenced the subsequent resource estimate. It was possible to identify samples that had been swapped, missing samples, incorrect labelling amongst other aspects. Further, the QA/QC aims to confirm both the precision and accuracy of the laboratory and thereby confirm that the data used in the mineral resource estimate is of sufficient quality.

The control samples used comprised of two different certified standards, a blank and a duplicate for every 20 samples submitted. The intended aim was 5% coverage for each of the control sample types. Further control on data integrity was achieved through re-submittal of not less than 5% of the total samples to a referee laboratory (SGS Lakefield, Johannesburg) from the original drilling campaign. The quality control data was analysed on an on-going basis and generated numerous queries with the laboratory. All queries were satisfactorily resolved.

SGS Lakefield is an accredited Laboratory with the South African National Analytical Standards (SANAS) with reference number T0107-10-2013.

Definition of terms related to the QA/QC protocols applied and subsequent evaluations are provided below:

A **standard** is a reference sample with a known (statistically) element abundance and standard deviation. Reference standards are used to gauge the accuracy of analytical reporting by comparing the pre-determined values to those reported by the laboratory used during an exploration project.

A **blank** is a standard with abundance of the element of interest below the level of detection of the analytical technique.

A **duplicate** is the split of a sample taken at a particular stage of the sampling process; e.g. Field Duplicate.

The precision and accuracy will be discussed in terms of the following statistical measures routinely applied by Coffey:-

Thompson and Howarth Plot showing the mean relative percentage error of grouped assay pairs across the entire grade range, used to visualise precision levels by comparing against given control lines.

Rank HARD Plot, which ranks all assay pairs in terms of precision levels measured as half of the absolute relative difference from the mean of the assay pairs (HARD), used to visualise relative precision levels and to determine the percentage of the assay pairs population occurring at a certain precision level.

Mean vs HARD Plot, used as another way of illustrating relative precision levels by showing the range of HARD over the grade range.

Mean vs HRD Plot is similar to the above, but the sign is retained, thus allowing negative or positive differences to be computed. This plot gives an overall impression of precision and also shows whether or not there is significant bias between the assay pairs by illustrating the mean percent half relative difference between the assay pairs (mean HRD).

Correlation Plot is a simple plot of the value of assay 1 against assay 2. This plot allows an overall visualisation of precision and bias over selected grade ranges. Correlation coefficients are also used.

Quantile-Quantile (Q-Q) Plot is a means where the marginal distributions of two datasets can be compared. Similar distributions should be noted if the data is unbiased.

6.6.3 Assay Quality Control Data Assessment

Pre 2016 Drilling Campaigns

The quality control protocol required the use of two different certified standards, a blank and a coarse reject duplicate for every 20 samples. The intended aim was 5% coverage of each control. In addition some 5% of the samples were analysed by a referee laboratory (SGS Lakefield) (Table 6.6.3_1)

Table 6.6.3_1
Summary of the Number of Control Samples: Pre 2016 Drilling Campaigns

	Submitted	Samples	Proportion
Standard SARM8	567	11,344	4.9%
Standard AMIS0006	240	11,344	2.1%
Standard AMIS0010	324	11,344	2.9%
Coarse Reject Duplicates	563	11,344	4.9%
Blanks	571	11,344	5.0%
Referee samples (pulps)	483	9,079 (actual samples)	5.3%
Referee control samples (pulps)	119	2,265 (control samples)	5.3%

The following summary conclusions were made based on the review of the QA/QC data analyses.

Blanks:

Pt, Pd, Rh, Au, Ru, Os, Ir:

The analyses for the precious metals include occasional blank samples with anomalously high values. All anomalous values were considered and where appropriate, the laboratory was requested to rerun selected samples in the batch. The number of anomalous grades is considered sufficiently low to consider analytical data from associated batches of samples to be acceptable for use in a mineral resource estimate.

Cr, Cu, Ni:

The analyses for the base metals include occasional blank samples with anomalously high values. The values of chrome for these blanks is very low. Contamination is therefore not considered an issue in the laboratory. The number of anomalous grades is considered sufficiently low to consider analytical data from associated batches of samples to be acceptable for use in a mineral resource estimate.

Al, Ca, Fe, Mg, Si:

Examination of the blank analyses for the major elements confirms that contamination and sample management are not laboratory issues and that the analytical data from associated batches of samples to be acceptable for use in a resource estimate.

<u>Standards</u>

A summary of the certified values of the selected certified reference materials is presented in Table 6.6.3_2. The selection of standards was done based firstly on comparable matrix (similar geochemistry) and secondly on the basis of PGE concentration. The first was selected to be close to the expected mean PGE grade and the other for a grade close to the anticipated marginal grade. A third standard was selected to confirm the accuracy of the chromium assays.

Table 6.6.3_2

Summary of Certified Reference Materials: Pre 2016 Drilling Campaigns

Certified Control Values							
	AMIS0006		**AMIS0010**		**SARM 8**		
Element	Grade	Range	Grade	Range		Grade	Range
Pt (g/t)	1.43	±0.15	2.05	±0.29	Al_2O_3 (%)	10.5	10.4 - 10.6
Pd (g/t)	0.94	±0.10	1.33	±0.10	CaO (%)	0.26	0.24 – 0.27
Rh (g/t)	0.29	±0.03	0.41*	±0.08	Cr_2O_3 (%)	48.9	48.9 – 49.0
Au (g/t)	0.02*		0.026*		Fe (%)	14.1	14.0 – 14.1
Cu (ppm)	838	±79	716*	±0146	MgO (%)	14.6	14.6 – 14.7
Ni (ppm)	820*	±65	1116	±118	MnO (%)	0.25	0.24 – 0.26
Cr (%)	7.89*	±0.41	15.84	±1.08	SiO_2 (%)	4.30	4.26 – 4.34

AMIS0006: Certified Feed Grade UG2 PGE Reference Material supplied by African Mineral Standards
AMIS00010: Certified High Feed Grade UG2 PGE Reference Material supplied by African Mineral Standards
SARM 8: Certified Chromium ore (Potgietersrus) Reference Material supplied by Mintek
* - Provisional results or indicated grade

The performance of the certified reference materials (standards) is summarised in Table 6.6.3_3.

Table 6.6.3_3

Summary of Certified Reference Materials (Standards) Performance: Pre 2016 Drilling Campaigns

	Pt	Pd	Rh	Au	Ru	Ir	Cr	Cu	Ni	Al	Fe	Mg	Si
AMIS0006													
Tolerance	97.9	97.5	95.8			94.5	0	87.5	99.2				
Bias	0.11	-0.46	3.05			3.09	46.76	2.90	2.09				
AMIS0010													
Tolerance	95.4	99.1	40.1	52.2	97.2	91.7	97.5	97.2	97.5				
Bias	3.08	1.03	10.18	-2.97	-0.23	4.75	-1.26	-2.96	1.31				
SARM8													
							98.4			95.1	97.9	98.6	84.7
							-2.25			-4.57	-0.15	-0.30	8.30

Tolerance: Proportion of assays within specification
Bias Percentage difference in grade of certified value against assays of standards

AMIS0006: Pt, Pd, Ru, Ir:

The analyses for the precious metals for which the standard is certified were all within the specified limits (greater than 95% of the analyses within the 10% limits of analyses). All anomalous values were reviewed and appropriate action taken with the laboratory. The accuracy of analytical data from associated batches of samples are considered acceptable for use in a mineral resource estimate.

AMIS0010: Pt, Pd, Ru:

The analyses for the precious metals for which the standard is certified were all within the specified limits (greater than 95% of the analyses within the 10% limits of analyses). All anomalous values were reviewed and appropriate action taken with the laboratory. The accuracy of analytical data from associated batches of samples are considered acceptable for use in a mineral resource estimate.

AMIS0006: Cr, Cu, Ni:

The analyses for the base metals for which the standard is certified were all within the specified limits (greater than 95% of the analyses within the 10% limits of analyses). All anomalous values were reviewed and appropriate action taken with the laboratory. The accuracy of analytical data from associated batches of samples are considered acceptable for use in a mineral resource estimate.

AMIS0010: Cr, Cu, Ni:

The analyses for the base metals for which the standard is certified were all within the specified limits (greater than 95% of the analyses within the 10% limits of analyses). All anomalous values were reviewed and appropriate action taken with the laboratory. The accuracy of analytical data from associated batches of samples are considered acceptable for use in a mineral resource estimate.

SARM8: Cr

The analyses for Cr were all within the specified limits (greater than 95% of the analyses within the 10% limits of analyses). All anomalous values were reviewed and appropriate action taken with the laboratory. The accuracy of analytical data from associated batches of samples are considered acceptable for use in a mineral resource estimate.

Duplicates

Overall the consideration of the duplicate analyses confirms that the laboratory demonstrates acceptable precision. The patterns of comparison show typically poorer correlation at lower grades and close to the level of detection. However there is good agreement between the mean, standard deviation and coefficient of variation between the original and repeat sample grades. The correlations are typically in the high nineties which is statically significant and acceptable considering for the large size of the dataset.

Referee Analyses

The correlation of the PGEs is considered at a level that confirms the accuracy of the primary laboratory. An examination of the standards used for the referee analysis confirms that the referee analyses demonstrate accuracy and precision for the PGEs. An examination of the standards for the referee analyses suggests that the referee analyses are unreliable.

Conclusion

The conclusion drawn is that the precision and accuracy of the assay data is acceptable for use in a mineral resource estimate.

2016 Sampling Campaign

Quality control monitoring protocols involved submission of blanks, duplicates and certified reference standards with the core sample batches. After every 8th sample an alternating blank or duplicate was allocated to the sampling sequence followed by a standard as the 10th sample. The actual numbers of control samples submitted are shown in Table 6.6.3_4. For duplicates, an empty sample bag was submitted for the laboratory and instructions given to take a second split of the preceding sample after crushing and pulverising. Four different standards were used in the sampling program of varying grades. A summary of the expected values for all four standards can be seen in Table Table 6.6.3_5. All standards were supplied by African Mineral Standards (Pty) Ltd. Pool filter sand was used as the blank material.

Table 6.6.3_4

Summary of the Number of Control Samples: 2016 Sampling Campaign

	Submitted	Samples	Coverage Proportion
AMIS0027	14		0.4%
AMIS0053	62		1.9%
AMIS0064	102		3.1%
AMIS0075	156	3342	4.7%
Blanks	167		5.0%
Duplicates	167		5.0%
Referee samples (pulps)	0		0.0%

Table 6.6.3_5

Summary of Standards Samples Performance: 2016 Sampling Campaign

Element	Unit	AMIS0027		AMIS0053	
		Expected Value	Two std Deviations	Expected Value	Two std Deviations
Pt	ppm	2.45	0.3	2.52	0.2
Pd	ppm	1.62	0.18	1.23	0.08
Rh	ppm	0.49*	0.08	0.18	0.016
Au	ppm			0.22*	0.036
Ir	ppm	0.18*	0.02	0.06*	0.01
Ru	ppm	0.77*	0.14	0.33	0.03
Cu	ppm	142*	20	810	46
Ni	ppm	1197	94	1719	118
Cr	%	13.74	0.87	0.4895	0.019
Element	**Unit**	**AMIS0064**		**AMIS0075**	
		Expected Value	**Expected Value**	**Expected Value**	**Two std Deviations**
Pt	ppm	1.28	1.28	1.2	0.08
Pd	ppm	0.59	0.59	1.5	0.08
Rh	ppm			0.25*	0.04
Au	ppm	0.1*	0.1*	0.07*	0.016
Ir	ppm	0.023*	0.023*	0.085	0.01
Ru	ppm	0.123*	0.123*	0.35*	0.04
Cu	ppm	654	654	227*	46
Ni	ppm	1509	1509	1114*	78
Cr	%	0.5144	0.5144	6.49	0.27

AMIS0027

All elements except Cu have more than 93% of results within two standard deviations of the expected value (EV). The bias for Pd, Rh and Ru is >6% and therefore high but as the results are within limits this is not considered an issue. Cu returns poor results with only 64% within tolerance and a bias of 12%. This is possibly due to the EV being 142ppm which is less than three times the detection limit for the method used and therefore the method might not be accurate at these low levels.

AMIS0053

Au, Ir, Ni and Cr have greater than 95% of results falling within two standard deviations with a bias of 4.6% or less. Pt, Pd, Rh, Ru and Cu have between 82% and 89% of results falling within two standard deviations of the EV, with bias of 2.4% or less. However, most results fall within three standard deviations and therefore are considered warnings. When the results falling within three standard deviations are included greater than 95% of all results are within tolerance.

AMIS0064

All elements except Pt have greater than 95% of results falling within two standard deviations of the EV with a bias of 3.2% or less. Ir has a slightly abnormal bias of -5.4% possibly due to the low grade. For Pt, when the results falling within three standard deviations are included greater than 96% of results are within tolerance, but the bias is excessive at -6.3% and the reason for this cannot be deduced.

AMIS0075

Rh, Au, Cu, Ni and Cr have greater than 95% of results falling within two standard deviations of the EV, with a bias < 2.5% except for Cu which has a high bias of 9.3%. Pd, Ir and Ru have between 79% and 93% of results falling within two standard deviations, with bias of -3.5% or less. When the results falling within three standard deviations are included in these elements, greater than 95% of all results are within tolerance. Pt only has 76% within two standard deviations and only 79% within three standard deviations, with a bias of -4%. The poor reporting of this element cannot be deduced.

Blanks

Minor sporadic contamination is evident especially in the first few batches. However, the contamination levels are low and not considered an issue.

Duplicates

For Pt, Pd, Rh, Ir, Ni and Cr more than 90% of the data pairs are within 10% HARD limits and are considered acceptable. For Au and Cu most of the data pairs fall close to or below ten times the detection limit for the methods used and therefore accuracy will be poor. For Ru, 88% of the data pairs are within 10% HARD limits. If the data pairs which return results below 10 time detection limits are removed then 90% of pairs are within 10% HARD limits.

Conclusion

Whilst a number of the standard results fall outside the expected two standard deviation limits, most fall within three standard deviations and are therefore considered acceptable. However, this should have been monitored throughout the assaying process and reported back to the laboratory so they could try to deduce why some elements were erratic. The graphs also show variations in the results over time, possible drift or re-calibrations, and should also have been queried with the laboratory.

Both the Blanks and the Duplicates do not show any problems such as transpositions or contamination of samples during preparation and as most standard results return within three standard deviations of the EV they are considered acceptable for use in the Resource Estimation.

6.6.4 Bulk Density Measurements

Bulk density data determinations were derived via the Archimedean ‘weight in air/weight in water’ technique, using an appropriate procedure and an accurate balance. The core is essentially impermeable and contains no vugs or voids. These density determinations are therefore considered appropriate for bulk density. In total, 8,814 bulk density measurements were taken representing samples submitted for chemical analysis and representing the various lithologies of the MG Chromitite Layers. The density measurements were not taken from sheared and fractured cores as they are permeable.

6.7 UG1 Chromitite Layer

The UG1 Chromitite Layer was not logged in detail in the previous drilling campaigns as it was not deemed economic. In 2012, the core was relogged and sampled to determine the nature of the UG1 Chromitite Layer and allow the estimation of a mineral resource. An outcrop position of the UG1 Chromitite Layer was projected based on the present mining of the UG1 Chromitite Layer and the borehole intersections.

The layers have a north-south dip direction. All drilled boreholes on the northern side of the outcrop intersected the Layer at the anticipated depths; an indication of continuity of mineralization and consistency in dip angle. All boreholes that intersected the UG1 Chromitite Layer were logged and sampled. The logging was done 1m above and below the UG1 Chromitite Layer.

6.7.1 Sampling Methodology

Representative samples over various UG1 Chromitite Layer intersections were marked out on the core with a paint marker. Unique sample numbers were assigned and information for each sample recorded in a sample ticket book. Core with samples marked out was photographed with a digital camera both dry and wet. Subsequently the core was cut in half vertically along its length and across to obtain the marked out samples. Only half core was submitted for analyses. The other half remained in the core tray for future reference.

The focus during sampling was to choose sample intervals according to lithologies in order to separate the mineralized layer from the host rock.

Sample intervals varied from an absolute minimum of 15cm for NQ core (20cm for BQ) to a maximum of 35cm. Chromitite samples included a 0.5 to 2cm host rock margin to avoid PGM and chrome loss during the core cutting process. This is the recognised standard for sampling of chromitite and PGM deposits in the industry

Quality control monitoring protocols involved submission of sample blanks, duplicates and certified standards with the core sample batches.

Each sample was bagged separately with the ticket number inside and the sample number also written on the outside of the sample bag. A dispatch form was submitted along with samples to ensure the total number of samples and correct sample numbers were recorded.

The sampling methodology is appropriate and supports the mineral resource estimate and classification made.

6.7.2 Analytical Procedures

Sample preparation was undertaken in the SGS Lakefield laboratory in Johannesburg. The sample preparation was undertaken using a jaw crusher to crush samples to minus 10mm in size. Pulverising of the samples is undertaken to achieve 85% minus 75µm in size.

Analyses were undertaken by SGS Lakefield, a certified laboratory. All samples were assayed for major oxides by XRF fusion and PGM by 6E NiS/MS. Selected samples were analysed for base metals by ICP Fusion D/OES.

The assay techniques used are considered appropriate for the major elements, PGM and base metal analyses and suitable for use in a mineral resource estimate.

6.7.3 Chain of Custody – Responsibility and accountability

The full chain of custody was implemented for the sample submission by the geologists to the analytical laboratory. The details of the samples to be submitted were recorded on standard documentation on site. The samples were checked by sampling personnel and the geologists prior to shipment. All details were provided on the despatch notes. The assay certificates were e-mailed to the Geologist as csv and pdf files. Cross checking of the assay certificates with the results was possible as these included details of each batch.

6.7.4 Bulk Density Measurements

Bulk density data determinations were derived via the Archimedean 'weight in air/weight in water' technique, using an appropriate procedure and an accurate balance. The core is essentially impermeable and contains no vugs or voids. These density determinations are therefore considered appropriate for bulk density. In total, 534 bulk density measurements were taken representing samples submitted for chemical analysis and representing the various lithologies of the UG1 Chromitite Layers. The density measurements were not taken from sheared and fractured cores as they are permeable.

6.8 Summary

The geological, collar and downhole survey data is considered to conform to international standards and to be suitable for use in a mineral resource estimation. The assay data are considered acceptable in terms of both assay precision and accuracy.

7 MINERAL RESOURCE ESTIMATION

7.1 Database

7.1.1 Borehole Database Development

Coffey was commissioned to manage the drill programme in 2007. The following key digital data relevant to the resource estimation study was compiled by Coffey:

- A borehole database that included collar location, downhole survey, assay, and geology data was compiled.
- Bulk density data and documentation.
- Assay quality control data.

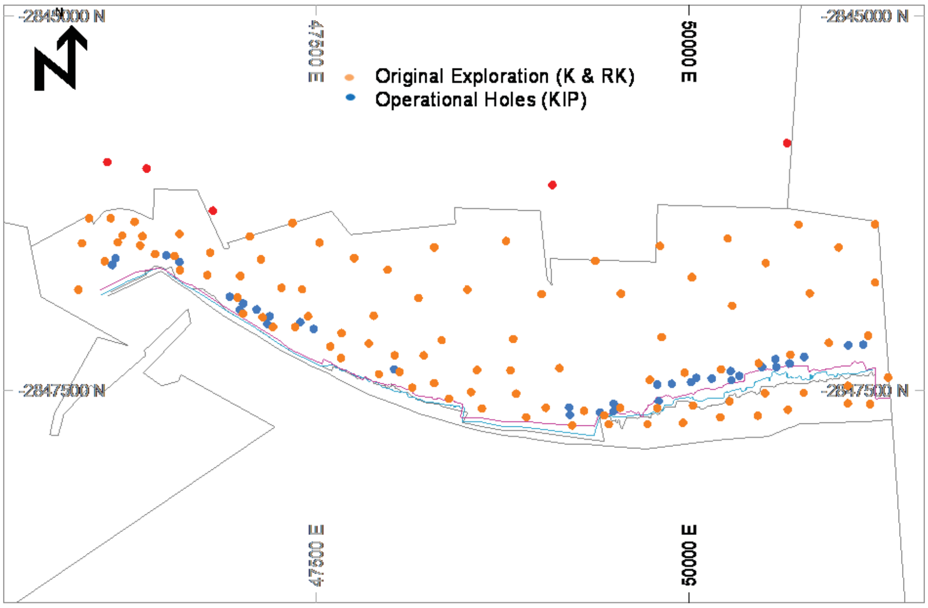
In November 2013, Coffey updated the borehole database utilising the knowledge gained during the exploration phase in 2008 and the subsequent knowledge gained during the open pit mining operation. The update consisted of the coding and re-coding of the various stratigraphic layers that constitute the MG Chromitite Layer packages and adding additional codes for units for which a better understanding had been gained. The following are new units that were not present in the initial database.

- The MG4AD Layer which consists of disseminated mineralisation identified above the MG4A Chromitite Layer.
- The MG3D Layer which consists of disseminated mineralisation has been defined. It is located directly above the primary MG3 Chromitite Layer.
- The MG3 Zebra Layer has been defined. It consists of an accumulation of thinly laminated chromitite layers located directly below the MG3 Chromitite Layer.
- Sub units within the parting between the MG2C and MG2B Chromitite Layers have been identified. These are as follows:
  - A layer named the PGEM Layer has been identified within the parting and known to be platiniferous.
  - Between this layer and the MG2C Chromitite Layer above is the PGEM HW Layer.
  - Between the PGEM Layer and the MG2B Chromitite Layer below is the PGEM FW Layer.

During 2016 35 boreholes were sampled and assayed and added to the database for inclusion in the mineral resource estimate (Figure 7.1.1_1).

Figure 7.1.1_1

Map showing the Boreholes used in the Mineral Resource Estimate



7.1.2 Borehole Database Validation

The drilling data was reviewed and validated prior to the resource evaluation studies.

The following general activities were undertaken during database validation:-

- Ensuring compatibility of total borehole depth data in each of the collar, survey, assay and geology database files.
- During the drilling programme the geological model was continuously updated and the boreholes validated on an individual basis.
- Inspections of the borehole core and consideration of the assay data to ensure understanding of the mineralisation and eliminate problems with the correlation of assay results and geology.
- Checking of borehole survey data for unusual or suspect downhole deviations.
- Ensuring sequential downhole depth and interval data in the survey, assay and geology files.
- Replacements of “less than detection limit” character entries with nominal low-grade values (half detection limit).
- Coding and re-coding of the various stratigraphic layers of the borehole database utilising the knowledge gained during the exploration phase in 2008 and the subsequent knowledge gained during the open pit mining operation.

7.1.3 Assay Quality Control Data Assessment

The quality control protocol implemented during the exploration drilling required the use of two different certified standards, a blank and a coarse reject duplicate for every 20 samples. The intended aim was 5% coverage of each control. In addition some 5% of the samples were analysed by a referee laboratory (SGS Lakefield) (Table 6.6.3_1)

7.1.4 Conclusion

The conclusion drawn is that the precision and accuracy of the assay data is acceptable for use in a mineral resource estimate.

7.2 Bulk Density Database

Bulk density data was collected routinely. In total, 8,814 bulk density measurements were taken, representing samples submitted for chemical analysis and representing the various lithologies of the MG Chromitite Layer. The data was collected from all diamond drill boreholes in the latest drilling campaign. Examination of the data confirmed internal constancy with the ranges and averages typical of the lithologies represented.

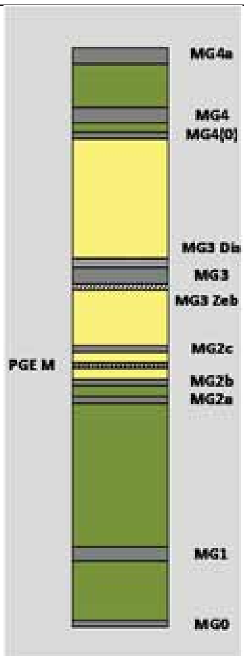
The data was analysed to determine the average density of the major lithologies. Where the density data was not measured, the average density was assigned to the lithology based on the analysis of the measured data.

7.3 Geological Modelling

The Tharisa Mine deposit was modelled using the 3D software packages Datamine™Studio Version 3.24.73.0 and Micromine™ Version 11. The geological modelling consisted of defining and then modelling the most appropriate contact in each Chromitite Layer across the property (Table 7.3_1).

Table 7.3_1

Summary of Stratigraphic Units modeled

Stratigraphic Column	Unit	
		
	MG4A Chromitite Layer	
	MG4 Chromitite Layer	MG4 Chromitite MG4 – MG4 (0) Parting MG4(0) Chromitite
	MG3 Chromitite Layer	MG3 Disseminated MG3 Chromitite MG3 Zebra
	MG2 Chromitite Layer	MG2C Chromitite PGEM Hangingwall PGE Marker PGEM Footwall MG2B Chromitite MG2A – MG2B Parting MG2A Chromitite
	MG1 Chromitite Layer	
	MG0 Chromitite Layer	

Wireframe surfaces for each of the five Chromitite Layer were modelled based on the borehole intersections. The models were validated to ensure that they did not cross and that the stratigraphic sequence was maintained. It was noted that the dip flattens with depth and the deepest borehole provided unusual data.

For the open pit area, more detail was required. Wireframe surfaces for each of the five mining units were modelled based on the borehole intersections. The thickness of some of the units i.e. the vertical distance between some of the surfaces is relatively small compared to the lateral distance between points of intersection. The models were validated to ensure that they represented the geometry of the units and that the stratigraphic sequence was maintained. The resulting surfaces are stacked on top of each other demonstrating the tabular nature of the deposit. The modelling utilised the other structural information gained from the aeromagnetic survey, in pit observations, surface mapping, trenching etc.

An examination of the geology revealed that it changes from east to west. In the east the stratigraphy was typically well defined with all the layers being recognisable. Towards the west, the geology becomes more complex. The identification and delineation of all stratigraphic units becomes more difficult as the separation of the units became is narrower with some units overlying other units directly. Based on these observations a cut off was defined separating the eastern side of the property which is more constant geologically from the western part where the geology is significantly more complex. The current understanding has allowed the mineral resource declaration to include this far wetern area although more work is required to fully undertstand the geology here.

7.4 Statistics

The data was coded for the different units within the MG Chromitite Layer package. Statistical analysis was then completed on both the raw and composite data grouped by unit type after examination of the data indicated that the units defined different geologically distinct populations and are well defined statistically (Table 7.4_1). Summary descriptive statistical analysis was completed based on the various geological units of the MG Chromitite Layer package

Table 7.4_1

Coding for the various units of the MG Chromitite Layer Package

DESCRIPTION	LAYER	STATIGRAPHY
4A Disseminated Hangingwall	4AD	MG4
MG4A	4A	
Parting MG4A-MG4	4A4	
MG4	4CR	
Parting MG4-MG0	44Z	
MG4(0)	4Z	
Parting MG4-MG3	4Z3	
3CR Disseminated Hangingwall	3D	MG3
MG3	3CR	
Zebra 3CR Footwall	3ZEB	
Parting MG3-MG2	2CHW	
MG2C	2C	MG2
PGEM Hangingwall	PGEM HW	
PGEM Layer	PGEM	
PGEM Footwall	PGEM FW	
MG2B	2B	
Parting MG2B-MG2A	BA	
MG2A	2A	
Parting MG2-MG1	2A1	
MG1	1CR	MG1
MG0	MG0	MG0

7.5 Compositing

Each intersection was composited across the full thickness of each unit as defined in the coding in Table 7.4_1. The Pt, Pd, Rh, Au, Ru, Ir, Os, Cu, Ni, Al, Ca, Cr, Cr_2O_3, Fe, Mg and Si concentrations were composited utilising the weighting by density and thickness. This is considered necessary as the lithologies have significantly different densities. An analysis of the unit thickness showed that there is little correlation between the concentration and thickness confirming that the use of concentration was appropriate for use in grade estimation.

7.6 Data Cutting

An assessment of the high-grade composites was completed to determine whether high-grade cutting was required. The approach taken to the assessment of the high-grade composites and outliers is summarised as:-

- Detailed review of histograms and probability plots with significant breaks in populations interpreted as possible outliers.
- Investigation of clustering of the higher grade data. High-grade data which clustered were considered to be real while high grade composites not clustered with other high grade data were considered to be a possible outlier and requiring further consideration either through cutting and/or search restriction.
- The ranking of the composite data and the investigation of the influence of individual composites on the mean and standard deviation plots.

Where possible outliers were identified, an examination of the data was undertaken to confirm whether this was indeed an outlier. The potential influence on the mineral resource estimate was also considered. After this examination and assessment, no high grade cutting or capping was undertaken.

7.7 Variography

Variography is used to describe the spatial variability or correlation of an attribute (Pt, Pd, Rh, Au, Cu, Ni etc). The spatial variability is traditionally measured by means of a variogram, which is generated by determining the averaged squared difference of data points at a nominated distance (h), or lag (Srivastava and Isaacs, 1989). The averaged squared difference (variogram or γ(h)) for each lag distance is plotted on a bivariate plot, where the X-axis is the lag distance and the Y-axis represents the average squared differences (γ(h)) for the nominated lag distance.

The variography was calculated and modelled in the geostatistical software, Datamine. The experimental variography models developed were considered very poor and unsuitable for estimation.

7.8 Block Model Development

A series of two-dimensional seam model-type estimates based on geologically and geochemically defined units within the MG Chromitite Layer cycle, was undertaken (Table 7.4_1). Based on the average spacing of surface boreholes and the requirements of the mine design, a parent block size of 100m x 100 was used. No rotation of the model was undertaken.

For each unit, grade variables and layer thicknesses were interpolated.

- The MG0 Chromitite Layer was estimated as a single unit.
- The MG1 Chromitite Layer was estimated as a single unit.
- The MG2 Chromitite Layer was estimated as seven units – three chromitite layers (MG2A Chromitite Layer, MG2B Chromitite Layer and MG2C Chromitite Layer) with the two partings being estimated independently due to the different geological and geochemical characteristics. The upper parting is further subdivided by a platiniferous layer (PGEM) into a lower parting (PGEM FW) and an upper parting (PGEM HW). Seven units – MG2C, PGEM HW, PGEM, PGEM FW, MG2B, MG2B-MG2A parting and MG2A Chromitite Layer.
- The MG3 Chromitite Layer was estimated three separate units - MG3D, MG3 Chromitite Layer and MG3 Zebra.
- The MG4 and MG4A Chromitite Layers were estimated as five units – three chromitite layers (MG4(0) Chromitite Layer, MG4 Chromitite Layer and MG4A Chromitite Layer) with the two partings being estimated independently due to the different geological and geochemical characteristics.

The data supplied included the ‘collar’ coordinates and survey data for both the mother holes and deflections. The data from the deflections thus formed part of the database as if it were an independent borehole. Each deflection within the borehole database was retained as separate data. These deflections have been offset from the surveyed chromitite layer intersection location of the mother hole by a nominal 1° at the top of wedge position. Where multiple deflections are developed, the deflections have been distributed around the borehole. The choice of displacement is arbitrary, given the scale of the borehole spacing. Maintaining the individual deflections as separate data rather than compositing the deflections to a single intersection composite is preferred.

In addition to the mineral resource estimate, the block model was utilised for subsequent mining studies. The precision of a block estimate is a function of the block size, related to the distribution of local data and the variogram structure. Although the MG Chromitite Layers have lateral variations, based on the distribution of data it is not considered possible to identify and hence it is considered impractical to selectively mine the higher grade blocks. Most of the selectivity is based on geological and geochemical characteristics of the different chromitite layers within the MG Chromitite Layer package i.e. selectivity dependent on the vertical stratigraphy.

7.9 Grade Estimation

The mineral resource estimation for the Tharisa Mine was completed using inverse distance weighting of borehole data. The method of grade estimation was considered after examination of the data and after attempting to generate variograms for the various components.

The intersected width, the density and the concentration of Pt (g/t), Pd (g/t), Rh (g/t), Au (g/t), Ru (g/t), Ir (g/t), Os (g/t), Cu (ppm), Ni (ppm), Al/Al_2O_3 (%), Ca/CaO (%), Cr/Cr_2O_3 (%), Fe/Fe_2O_3 (%), Mg/MgO (%), Si/SiO_2 (%), K_2O (%), MnO (%), NaO_2 (%), P_2O_5 (%), TiO_2 (%) and V_2O_5 (%) for each of the units identified within the MG Chromitite Layers where the concentration or grade is for the composite over the thickness of that unit. The mineral resource estimate was completed for the area of the mining right of Tharisa Minerals.

The relationship between grade and thickness was examined for the most economically important elements namely 3PGE+Au (g/t) and Cr_2O_3 (%). Based on this analysis, the concentration of each element was estimated independently from the thickness (LENGTH) of the units.

7.9.1 Search Criteria

Based on the understanding of spatial variation of the data and of the geology, a spherical search was adopted. A number of search radii were tested for the different elements. The final selection of the search criteria was made after the various options were tested on the various units. The selection was based on an examination of the global grades as well as consideration for the geological variability and the observed east – west grade trends. The grade estimation utilised the search parameters presented in Table 7.9.1_1.

7.9.2 Model Validation

A visual and statistical review was completed on the estimates prior to accepting the model. Acceptable levels of mean reproduction are noted between the block model and input composite data. Comparisons with previous estimates are also considered.

Table 7.9.1_1
Sample Search Parameters

	First Search Volume			Second Search Volume			Third Search Volume		
	Search radius (m)	Min. No of Samples	Max. No of Samples	Search radius (m)	Min. No of Samples	Max. No of Samples	Search radius (m)	Min. No of Samples	Max. No of Samples
MG4AD	1000	4	12	1500	4	12	3000	3	20
MG4A	1000	4	12	1500	4	12	3000	3	20
MG4A-MG4 Parting	1000	4	12	1500	4	12	3000	3	20
MG4	1000	4	12	1500	4	12	3000	3	20
Parting MG4 – MG4(0)	1000	4	12	1500	4	12	3000	3	20
MG4(0)	1000	4	12	1500	4	12	3000	3	20
MG3D	1000	4	12	1500	4	12	3000	3	20
MG3CR	1000	4	12	1500	4	12	3000	3	20
MG3-Zebra	1000	4	12	1500	4	12	3000	3	20
MG2C	1000	4	12	1500	4	12	3000	3	20
PGEM HW	1000	4	12	1500	4	12	3000	3	20
PGEM	1000	4	12	1500	4	12	3000	3	20
PGEM FW	1000	4	12	1500	4	12	3000	3	20
MG2B	1000	4	12	1500	4	12	3000	3	20
Parting MG2B – MG2A	1000	4	12	1500	4	12	3000	3	20
MG2A	1000	4	12	1500	4	12	3000	3	20
MG1	1000	4	12	1500	4	12	3000	3	20
MG0	1000	4	12	1500	4	12	3000	3	20

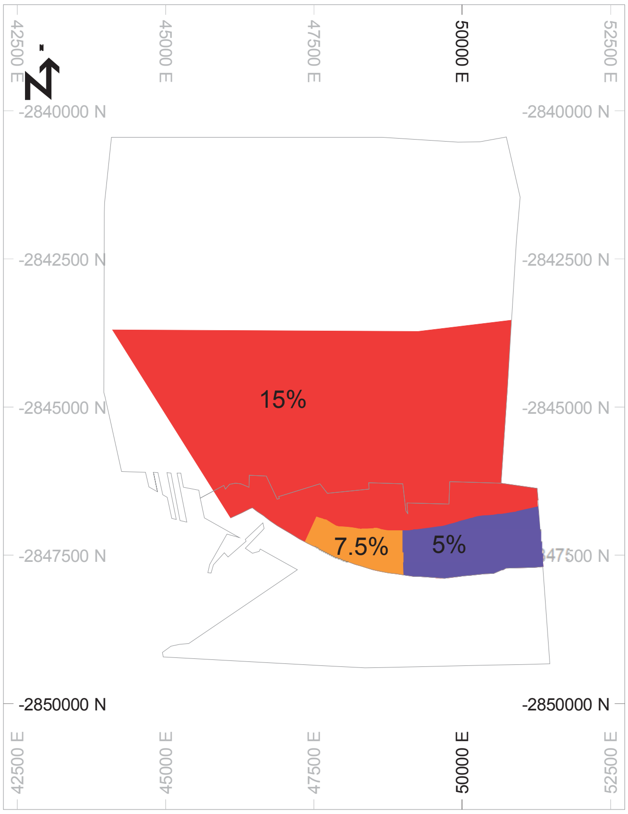
7.10 Geological Loss

The geological features where the Middle Group Chromitite Layer is not developed include faults, dykes, potholes and mafic/ultramafic pegmatites constitute the geological loss. The various features interested in the pit have been recorded and measured and use to assess the geological loss. This information serves to inform the declaration of the geological loss in the areas that are anticipated to be mined by open pit. These details have been collated and an assessment of the effect on the chromitite layers estimated. After examination of the various geological features where the MG Chromitite Layers are not developed viz. dykes, faults, potholes, mafic pegmatites, the geological loss for the East Pit was set at 5%, 7.5% for the West Pit and 15% for the Far West Pit. The details are depicted din Figure 7.10_1.

A geological loss of 15% was applied for the remaining areas. For these areas the geological model presents a series of tabular deposits intersected by some dykes and a few large displacement faults crossing the property. In addition larger potholes have been delineated. However, the smaller scale faulting (<10m throw) and the presence of smaller potholes cannot be delineated but must be considered. The estimated geological loss is based on the existing knowledge of the deposit.

Figure 7.10_1

Map showing the Applied Geological Loss



7.11 MG Chromitite Layers Mineral Resource Reporting

The classification of the mineral resources was undertaken in accordance with the guidelines of the SAMREC Code (2016). The Competent Persons responsible for the mineral resource estimation and classification is Mr. Ken Lomberg Pr.Sci.Nat..

The classification of the mineral resource was based on the robustness of the various data sources available, confidence in the geological interpretation, variography and various estimation service variables (e.g. distance to data, number of data, maximum search radii etc).

7.11.1 Criteria for Mineral Resource Categorisation

The resource estimate was classified as a combination of Measured, Indicated and Inferred Resource based on the criteria set out in Table 7.11.1_1.

Table 7.11.1_1

Confidence Levels of Key Criteria for Classification of MG Chromitite Layers of the Tharisa Mine

Items	Discussion	Confidence				
		MG0	MG1	MG2	MG3	MG4/MG4A
Drilling Techniques	Diamond drilling to International Standard.	High	High	High	High	High
Logging	Standard nomenclature and procedures to international standards.	High	High	High	High	High
Drill Sample Recovery	The core recovery is estimated as >95% and is considered acceptable for mineral resource estimation.	High	High/Moderate (Core very friable with generally <90% recovery)	High	High	High
Sub-sampling Techniques and Sample Preparation	International standard for Diamond Drilling.	High	High	High	High	High
Quality of Assay Data	Available data is of international quality.	High	High	High	High	High
Verification of Sampling and Assaying	Complete QA/QC programme employed.	High	High	High	High	High
Location of Sampling Points	Survey of all collars with downhole survey.	High	High	High	High	High
Data Density and Distribution	Drilled with a spacing of 250m to 2000m.	Classification based on borehole density and understanding of the underlying geology and geochemistry				
Audits or Reviews		None	None	None	None	None
Database Integrity	Errors identified and rectified.	High	High	High	High	High
Geological Interpretation	Geological interpretation of each chromitite layer. Continuity of geology adequately demonstrated. Major structures identified.	High	High	High	High	High
Mineralisation Type	Able to correlate Chromitite Layers across the project.	High	High	High	High	High
Estimation and Modelling Techniques	Ordinary Kriging.	High	High	High	High	High
Cut-off Grades	Geological interpretation of the mineralised horizon for grade composting	High	High	High	High	High
Mining Factors or Assumptions	None.	High	High	High	High	High

It should be noted that the core recovery on the MG1 Chromitite Layer was considerably more difficult due to the very friable nature of the chromitite layer. Previously this was considered to indicate a lower confidence in the assays and hence the lower classification of the mineral resource. With the additional sampling and operational understanding, sufficient confidence has been found to upgrade the classification of this part of the mineral resource.

7.11.2 Mineral Resource Classification

The resource classification considers the above assessment and confidence in exploration data, geological understanding and grade estimation. The classification is presented in Figure 7.11.2_1 for MG1 Chromitite Layer and Figure 7.11.2_2 for the other Chromitite Layers.

Figure 7.11.2_1

Plan showing the location of the various classified mineral resources for the MG0 and MG1 Chromitite Layers of the Tharisa Mine

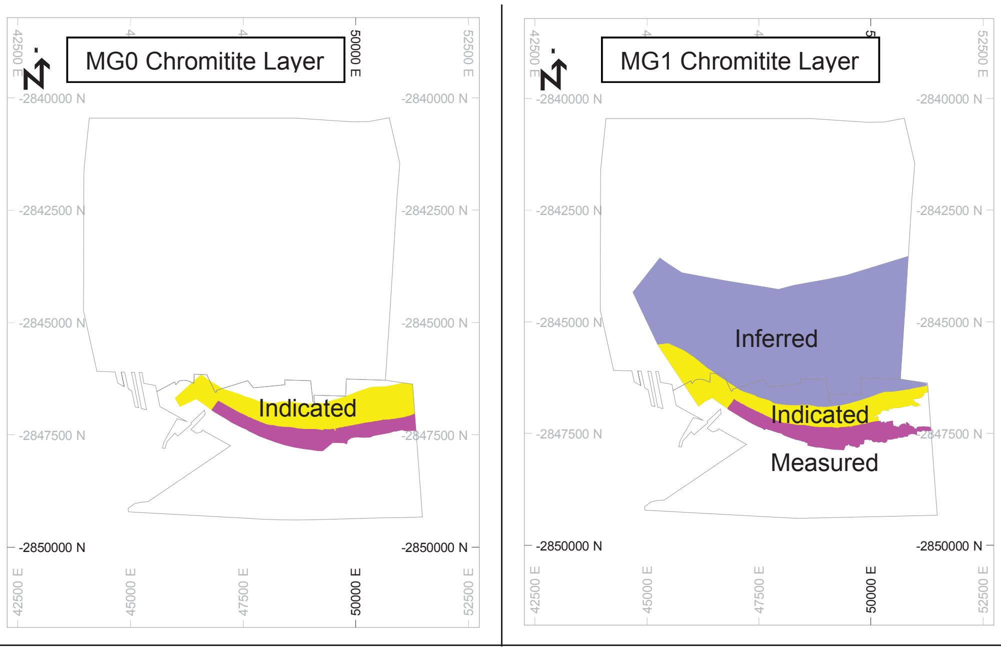
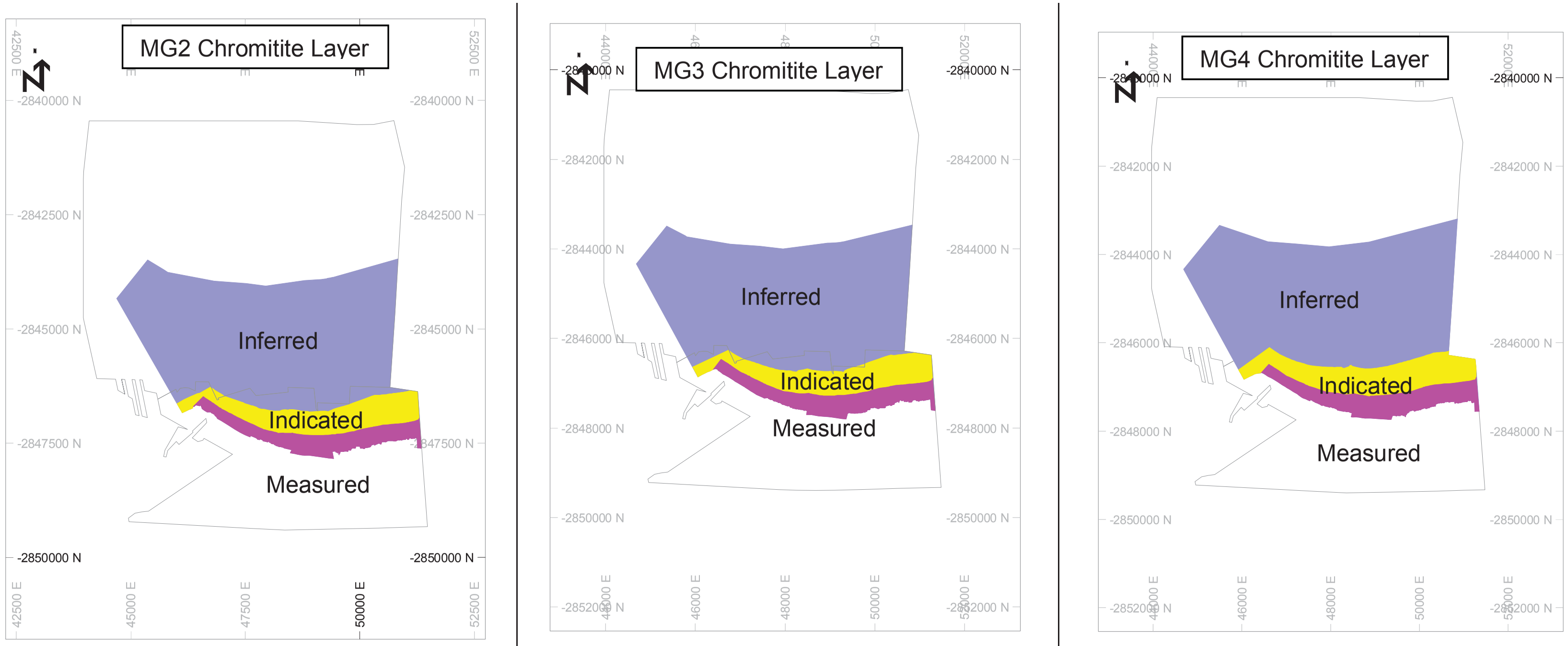


Figure 7.11.2_2

Plan showing the Location of the Various Classified Mineral Resources for the MG2, MG3 and MG4 Chromitite Layers of the Tharisa Mine



7.12 Estimate of the Mineral Resources – 30 September 2016

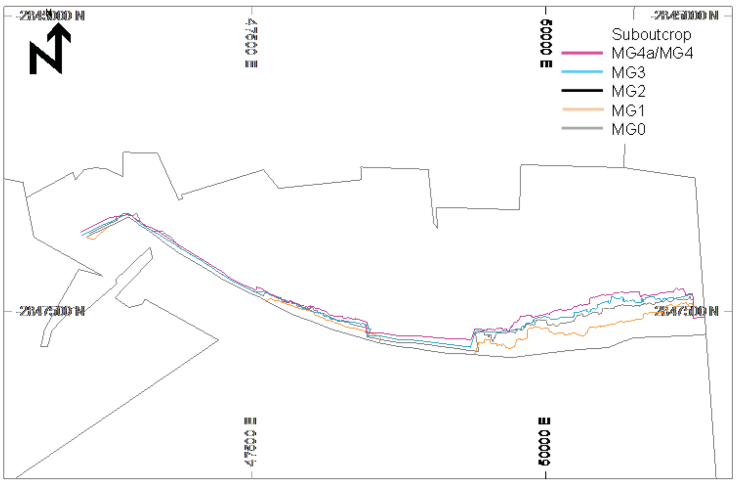
The September 2013 mineral resource statement was based on the interpretation of the structure and assay values available at that time. The mineral resource statement dated September 2014 was derived by depleting this estimate based on production figures of both tonnage and grade. The mineral resource statement dated September 2015 was derived by a further depletion based on production figures of both tonnage and grade.

In December 2015, the Datamine block model was updated with this estimate forming the basis of the latest estimate of the mineral resources. No further exploration drilling had been included in the estimate as the sampling and assaying of the exploration boreholes had not been undertaken. The open pit mining had allowed various aspects of the geology to be reviewed based on geological observations.

The outcrop of each unit was surveyed (Figure 7.12_1).

Figure 7.12_1

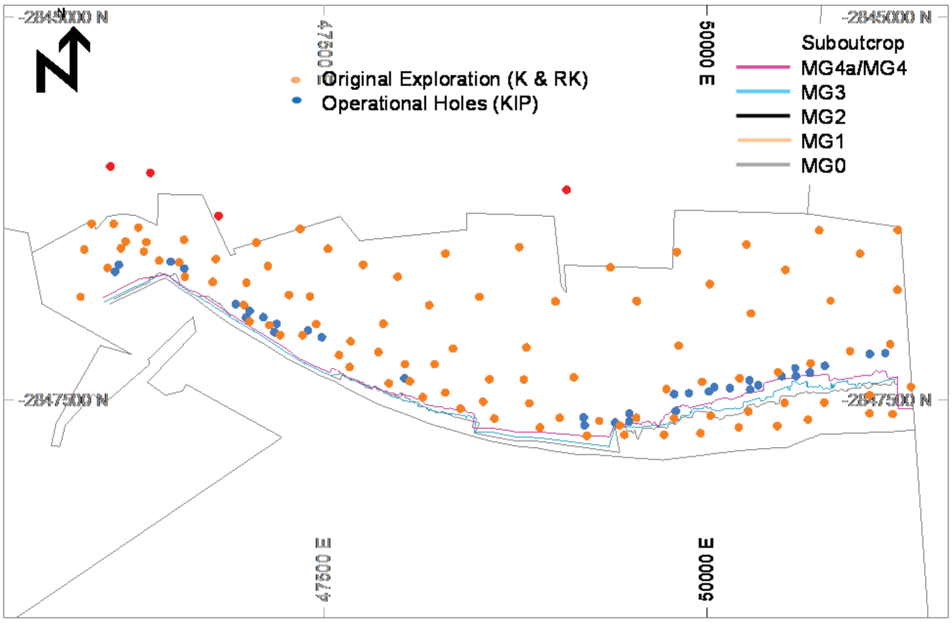
Plan showing the location of the MG Chromitite Layers of the Open Pit



The estimate includes additional exploration boreholes (35 KIP boreholes) and geological observation sin the pit. In particular a better understanding of the geology in the far west exists (Figure 7.12_2).

Figure 7.12_2

Plan showing the location of the Boreholes used in the Mineral Resource Estimate



7.12.1 Geological Loss

The geological losses were revised based on pit observations of geological disturbances (Section 7.10).

7.12.2 Classification of Mineral Resources

Previously the MG1 Chromitite Layer Resource was classified as Indicated and Inferred because of the friable nature of the chromite made full recovery difficult. Based on observations and experience from the open pit mine, sufficient confidence now exists to classify part of the resource as Measured.

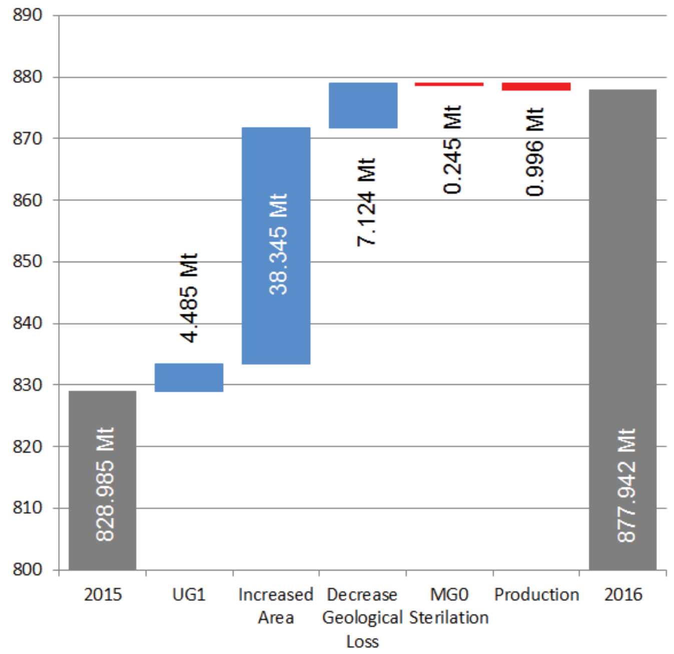
The additional boreholes included in this estimate have allowed the northern boundary of the measured mineral resource to be moved downdip (or to the north) due to the additional information in this area. However, the indicated boundary remains as no information has been obtained to move it. Further to this, the work undertaken on the mine suggests that the transition from the Marikana facies to the Rustenburg facies can be broadly defined on the western side of the mine. This has also been considered in defining the areas of the various mineral resource classifications

7.12.3 Comparison with Previous Estimation

The comparison with the previous estimates highlights the following changes (Figure 7.12.3_1):

Figure 7.12.3_1

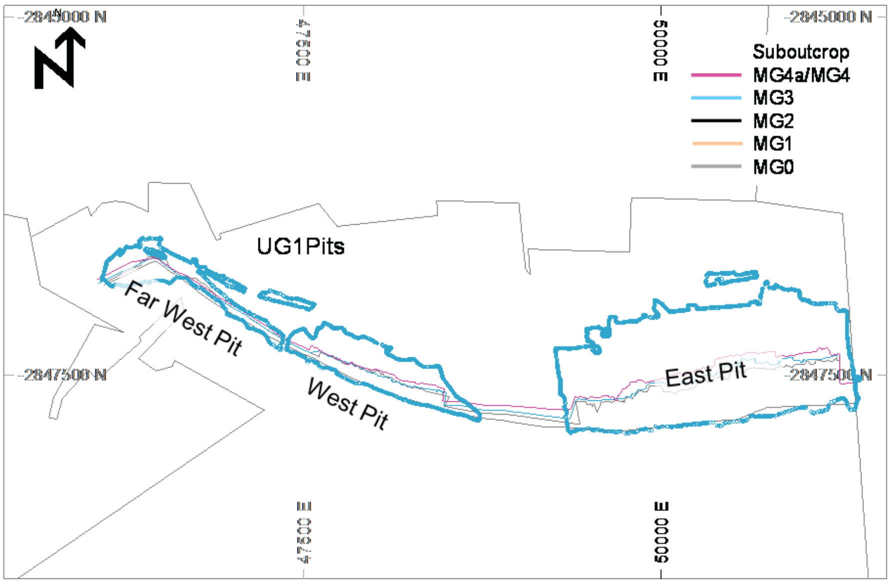
Waterfall Graph showing the major changes from the 2015 Mineral Resource Declaration



Production: - The area that has been mined has been determined based on the surveyed position of the various chromite layers (Figure 7.12.3_2).

Figure 7.12.3_2

Plan showing the location of the MG Chromitite Layers in the planned Open Pit



UG1 Chromitite Layer: - an increase of 4Mt as a result of the extension other mineral resource towards the west and the retabulation based on the 2016 planned final pit outlines.

Additional Area: - the mineral resource has been extended towards the far west as new information has provided a better understanding of the geology and the additional boreholes provided geological and grade information in this area

Reduced Geological loss: - the overall geological loss has been reduced based on the revision presented by the Senior Exploration Geologist.

MG0 Chromitite Layer sterilisation: - a small area of the MG0 Chromitite Layer has been sterilised due to dumping of material on the outcrop. It is consider unlikely that the chromitite under these dumps would be mined due to the now additional cost of removing the dump.

MG3 Disseminated and MG3 Zebra revision: - The material immediately above and below the MG3 Chromitite Layer has been reconsidered to provide an optimal mining cut for the Mineral Resources and Mineral Reserves. There has been a loss of tonnage as a result.

7.12.4 Mineral Resource Statement

The Mineral Resource Statement for the Tharisa Mine with an effective date of 30 September 2016 is presented in Table 7.12.4_1.

Table 7.12.4_1

Mineral Resource Statement for the Tharisa Mine (30 September 2016)

	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+ Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
MG4A CHROMITITE LAYER																		
Measured	10.163	1.52	3.67	23.35	0.40	0.15	0.12	0.00	0.25	0.04	0.05	0.67	59:23:18:0	1.02	39:15:12:0:25:4:5	1.17	333	767
Indicated	13.515	1.54	3.69	26.08	0.43	0.17	0.14	0.00	0.28	0.05	0.06	0.74	58:23:18:0	1.13	38:15:12:0:25:4:5	1.31	490	770
Inferred	70.400	1.48	3.70	22.64	0.36	0.14	0.12	0.00	0.24	0.04	0.05	0.62	58:23:19:1	0.95	38:15:12:0:25:4:5	1.30	2,143	682
MG4 and MG4(0) CHROMITITE LAYER Package																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+ Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	16.450	2.41	3.74	27.00	1.11	0.25	0.22	0.00	0.39	0.08	0.12	1.58	70:16:14:0	2.17	51:11:10:0:18:4:5	1.29	1,145	801
Indicated	25.787	2.96	3.67	25.02	1.07	0.23	0.21	0.00	0.37	0.08	0.11	1.51	71:15:14:0	2.07	52:11:10:0:18:4:5	1.20	1,712	737
Inferred	178.033	3.86	3.56	22.69	0.97	0.18	0.19	0.00	0.34	0.07	0.10	1.34	72:14:14:0	1.86	52:10:10:0:18:4:6	1.14	10,643	700
MG3 CHROMITITE LAYER																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+ Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm))
Measured	12.680	2.07	3.47	20.18	0.82	0.50	0.21	0.01	0.31	0.06	0.08	1.54	53:33:14:0	1.99	41:25:10:0:15:3:4	1.18	810	675
Indicated	19.769	2.49	3.45	19.38	0.80	0.49	0.21	0.01	0.29	0.06	0.08	1.50	53:33:14:0	1.93	41:25:11:0:15:3:4	1.11	1,228	649
Inferred	89.191	2.05	3.53	21.15	0.86	0.50	0.23	0.00	0.32	0.06	0.09	1.60	54:31:14:0	2.07	42:24:11:0:16:3:4	1.17	5,938	689
MG2 CHROMITITE LAYER																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+ Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	22.787	3.82	3.61	20.35	1.13	0.30	0.16	0.00	0.28	0.05	0.08	1.59	71:19:10:0	2.01	56:15:8:0:14:3:4	1.07	1,470	753
Indicated	33.587	4.32	3.58	18.17	0.99	0.29	0.15	0.00	0.25	0.05	0.07	1.43	69:20:10:0	1.80	55:16:8:0:14:3:4	0.99	1,946	739
Inferred	283.454	6.56	3.50	14.33	0.76	0.23	0.12	0.00	0.20	0.04	0.06	1.11	69:20:11:0	1.41	54:16:8:0:14:3:4	0.82	12,876	689

MG1 CHROMITITE LAYER																	
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	6.839	1.30	3.84	32.74	0.32	0.21	0.11	0.00	0.45	0.07	0.07	0.63	50:33:17:1	1.23	26:17:9:0:37:6:6	1.43	271	803
Indicated	10.096	1.10	3.89	32.45	0.33	0.21	0.11	0.00	0.46	0.08	0.07	0.66	51:32:17:1	1.27	26:17:9:0:36:6:6	1.45	413	806
Inferred	69.487	1.54	3.85	30.59	0.33	0.19	0.10	0.00	0.43	0.07	0.07	0.62	53:30:17:1	1.19	28:16:9:0:36:6:6	1.44	2,656	778
MG0 CHROMITITE LAYER																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	3.69	0.52	3.71	25.54	0.46	0.16	0.15	0.00	0.28	0.05	0.06	0.77	60:20:19:1	1.16	40:13:13:0:24:4:5	1.25	137	740
Indicated	5.78	0.59	3.70	25.98	0.59	0.18	0.16	0.00	0.32	0.05	0.07	0.95	63:19:17:0	1.39	43:13:12:0:23:4:5	1.27	258	735
Inferred																		
UG1 CHROMITITE LAYER																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured																		
Indicated	3.54	2.24	3.69	23.68	0.86	0.29	0.17	0.00	0.32	-	0.07	0.72	46:31:19:5	1.16	28:19:12:3:32:0:6	1.14	133	-
Inferred	2.45	3.35	3.68	22.75	0.81	0.28	0.17	0.01	0.32	-	0.07	0.66	41:35:18:6	1.05	26:22:12:4:31:0:6	1.11	83	-
TOTAL MINERAL RESOURCE																		
	Tonnage (Mt)	True Thick (m)	Bulk Density (t/m^3)	Cr_2O_3 (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ru (g/t)	Os (g/t)	Ir (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	6PGE+Au (g/t)	Pt:Pd:Rh:Au:Ru:Os:Ir	Cr:Fe	6PGE+Au (koz)	Ni (ppm)
Measured	72.612	11.6	3.67	23.68	0.86	0.29	0.17	0.00	0.32	0.06	0.08	1.32	65:22:13:0	1.78	48:16:10:0:18:3:5	1.20	4,166	756
Indicated	112.073	15.3	3.72	22.75	0.81	0.28	0.17	0.01	0.32	0.06	0.08	1.26	64:22:13:0	1.72	47:16:10:0:18:3:5	1.17	6,180	732
Inferred	693.009	18.8	3.62	19.85	0.74	0.24	0.15	0.00	0.28	0.05	0.07	1.13	66:21:13:0	1.54	48:15:10:0:18:4:5	1.07	34,339	700
Total	**877.694**	**17.9**	**3.63**	**20.54**	**0.76**	**0.25**	**0.15**	**0.00**	**0.29**	**0.06**	**0.07**	**1.16**	**65:21:13:0**	**1.58**	**48:16:10:0:18:4:5**	**1.09**	**44,685**	**709**

Note: The mineral resource is declared to a depth of 750m below surface.
Grades and tonnages are reported at shaft head
The consideration of realistic eventual extraction necessitates that the mineral resource considers the MG Chromitite Layer to be a geological unit and that all platiniferous and chromiferous horizons will be mined and all PGM, Cu, Ni and Cr_2O_3 recovered.
The UG1 Chromitite Layer is declared for the part that falls within the current proposed open pit
The mineral resource is reported inclusive of the mineral reserve

7.13 UG1 Chromitite Layer

7.13.1 Methodology

The UG1 Chromitite Layer was modelled using the 3D software package Datamine™. The UG1 Chromitite Layer comprises the top chromitite layer, middling (pyroxenite/anorthosite) and bottom chromitite layers. It was necessary to further model individual layers because of the independent geochemical characteristics. The three layers were therefore modelled independently.

A plan showing the UG1 Chromitite Layer is presented in Figure 7.13.1_1. East and West Mines were modelled independently as it was noted that they are of different populations. The boundary between east and west mines was put at the Sterkstroom river. East Mine was further divided into two domains due to geology and grade considerations in the far eastern side.

In total seven databases were distinguished and modeled independently i.e. West (top, middling, and bottom), East (top, middling and bottom) and Far East (one model).

As a result of the confidence in the geological model, each of the stratigraphic units was estimated independently as a layer and hard boundary was used. Each of the (Al_2O_3(%), CaO(%), MgO(%), Fe_2O_3(%), K_2O(%), MnO(%), Na_2O(%), P_2O_5(%), Cr_2O_3(%), (Pt (g/t), Pd(g/t), Rh(g/t), Ru(g/t), Ir(g/t), Au(g/t), width(m) and density) values were estimated independently using inverse power of distance (power of 2).

Mean densities for each domain were used in tonnage calculations as the variability was low.

7.13.2 Compositing

The data was composited by stratigraphic unit (UG1 Chromitite Layer) to produce a “reef only” grade as well as composited to sub-stratigraphic zones (i.e. Top, Middling and Bottom Chromitite Layers) and domains within UG1 Chromitite Layer (i.e. West and East’s Top, Middling and Bottom Chromitite Layers and Far East).

7.13.3 Statistical Analysis

A detailed statistical analysis was undertaken according to the geological model developed for each mineralised domain and for each metal element per composite. The composited data shows more or less normal distributions.

7.13.4 Geological Losses

The deposit is known to be intersected by few faults, barren mafic and ultramafic dykes as well as potholes and replacement pegmatites which both have an effect on stratigraphic and grade continuity. A geological loss of 15% was applied.

7.13.5 UG1 Chromitite Layer Mineral Resource Reporting

The mineral resource in respect of the UG1 Chromitite Layer is reported in Table 7.13.5_1. The classification of the mineral resources was undertaken in accordance with the guidelines of the SAMREC Code. The Competent Person responsible for the mineral resource estimation and classification is Mr. Ken Lomberg Pr.Sci.Nat.

The classification of the mineral resource was based on the robustness of the various data sources available, confidence in the geological interpretation, variography and various estimation service variables (e.g.: distance to data, number of data, maximum search radii etc).

Additional consideration has been given to the stand alone potential based on reasonable expectation of eventual economic extraction. It is therefore assumed that the UG1 Chromitite Layer is mined together with the Middle Group (MG) Chromitite Layers in the same open pit.

Figure 7.13.1_1
Plan view of the UG1 Chromitite Layer's Mineral Resource.

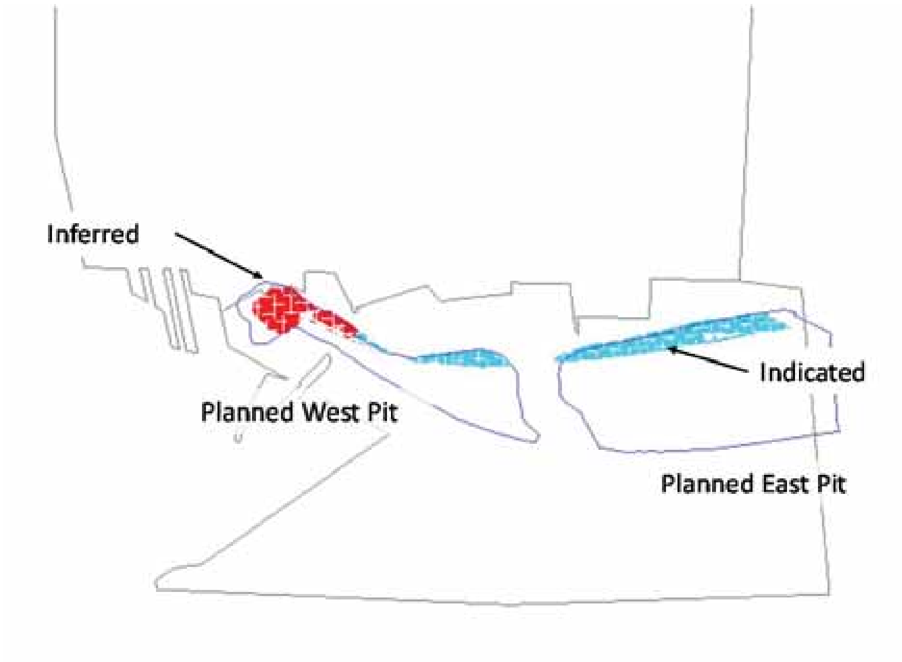


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Table 7.13.5_1

Tharisa Minerals

UG1 Chromitite Mineral Resource Estimation

30 September 2016

Layer	Tonnage (Mt)	Thickness (m)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	3PGE+Au (g/t)	Pt:Pd:Rh:Au	3PGE+Au (koz)	Cr_2O_3 (%)
INDICATED MINERAL RESOURCE										
Top	1.57	0.97	0.27	0.24	0.13	0.04	0.68	40:36:19:6	55.83	25.12
Bottom	1.97	1.27	0.38	0.21	0.14	0.03	0.76	50:28:18:4	76.73	21.08
TOTAL	**3.54**	**2.24**	**0.33**	**0.22**	**0.13**	**0.03**	**0.72**	**46:31:19:5**	**132.56**	**22.88**
INFERRED MINERAL RESOURCE										
Top	0.99	1.31	0.32	0.35	0.13	0.03	0.83	38:42:16:4	40.71	24.99
Bottom	1.46	2.04	0.23	0.14	0.11	0.04	0.54	44:27:21:8	41.92	17.31
TOTAL	**2.45**	**3.35**	**0.27**	**0.23**	**0.12**	**0.04**	**0.66**	**41:35:18:6**	**82.63**	**20.42**
TOTAL RESOURCE	**5.98**	**5.59**	**0.31**	**0.23**	**0.13**	**0.04**	**0.70**	**44:32:19:5**	**215.19**	**21.87**

*Assuming UG1 Chromitite Layer is mined together with the Middle Group (MG) Chromitite Layers

Grades and tonnages are reported at shaft head

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8 TECHNICAL STUDIES

8.1 Introduction

A feasibility study was concluded in October 2008. Various revisions to the mine plan were undertaken to match the requirements of the processing facilities, including both open pit and underground mine design and scheduling. The last revision was undertaken using the 2016 Mineral Resource update.

The selected exploitation strategy is the combined mining of MG1, MG2, MG3, MG4, MG4(0) and MG4A Chromitite Layers which extend from the surface to a depth of 750mbs at dips varying from 13° in the east to 16° in the west.

8.2 Geotechnical Assessment

The mine is being excavated following the slope designs undertaken by Celtis Geotechnical and Open House Management Services. The current pit slopes are generally much shallower than the designed slope angles of 53º for sound rock and 45º for weathered rock and soil. This is due to the low stripping ratio, pursued for economic reasons. The current ultimate or final pit design overall slope is lower than the proposed slope angle due to a highwall ramp included in the East pit highwall.

The slope assessment was based on the on fracture logging and rockmass classification of 10 boreholes (eight geological boreholes and two additional boreholes to collect samples for rock strength testing) (James, 2008) and geotechnical data collected by Open House Management Solutions (Pty) Ltd (OHMS) in the current east and central pits of Tharisa Mine to determine stable slope angles (Cilliers and Bosman, 2013).

Further data collection and reassessment of the slope design will be undertaken as mining continues. Laser scans to map jointing and asses hazards has commenced.

8.2.1 Geotechnical Environment

During the visit to the mine for this review the following observations were made:

- There are no slopes which exceed the maximum design slope of 53º so all slopes are stable
- The benches and berms are being mined to design standards
- In the Far West Pit the initial vertical benches in highly weathered pyroxenite and soil are being cut over 10 m high. The rock engineering consultant has recommended that these benches be pushed back with 3 m benches
- The stripping ratio is low and will have to be increased to achieve the planned final depth
- The deepest level of the pits is about 60 m below surface
- No critical risks were observed.

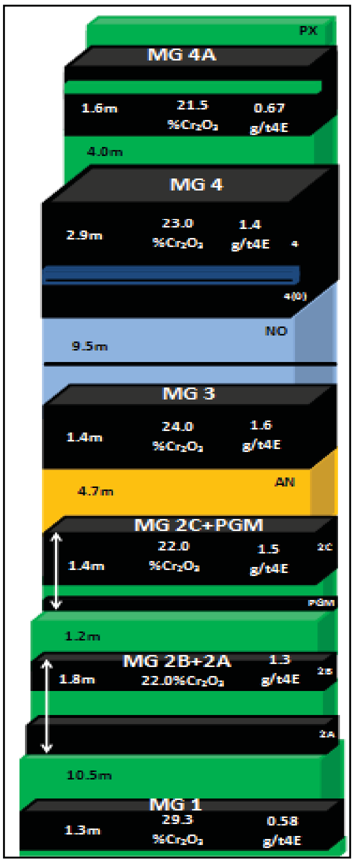
In 2013 a detailed geotechnical study was undertaken by OHMS at the mine consisting of face mapping in the existing east and central pits. Samples were collected from existing exploration boreholes for rock strength testing. The major lithological units in the ore body were tested for Uniaxial Compressive Strength (UCS), Density, Elastic Modulus and Poisson's Ratio.

These boreholes were selected to be at the location of the final pit walls.

There was also a previous geotechnical investigation in 2008 which included fracture logging and rock mass ratings of eight geological boreholes before splitting. The boreholes were selected to sample the area of the ore body and two additional geotechnical boreholes were drilled for sampling and strength testing.

It is planned to mine all the MG Chromitite Layers from the MG0 to the MG4A Chromitite Layers in the open pit (Figure 8.2.1_1). The MG Chromitite Layers sub outcrop beneath black turf soil and are separated by middlings of pyroxenite, anorthosite and norite. The footwall of the MG0 Chromitite Layer consists of pyroxenite.

Figure 8.2.1_1
Generalised Stratigraphy of the MG Chromitite Layers



<u>Structure and rock fabric</u>

In order to quantify the predominant orientation of geological structures in the various rock types, OHMS took measurements of exposed discontinuity surfaces in the east and central pits.

The measurements were analysed using lower hemispherical stereonet projections (Figure 8.2.1_2). Distinct joint sets were defined from Fischer concentration contours of poles. A total of 137 observations were mapped at various locations in the current pits. Four distinct clusters were identified and grouped in sets (Table 8.2.1_1). A number of randomly orientated joints, not conforming to the identified sets, were identified. Only two of these joint sets were identified as prominent, the flat dipping joints were identified as related to the igneous layering.

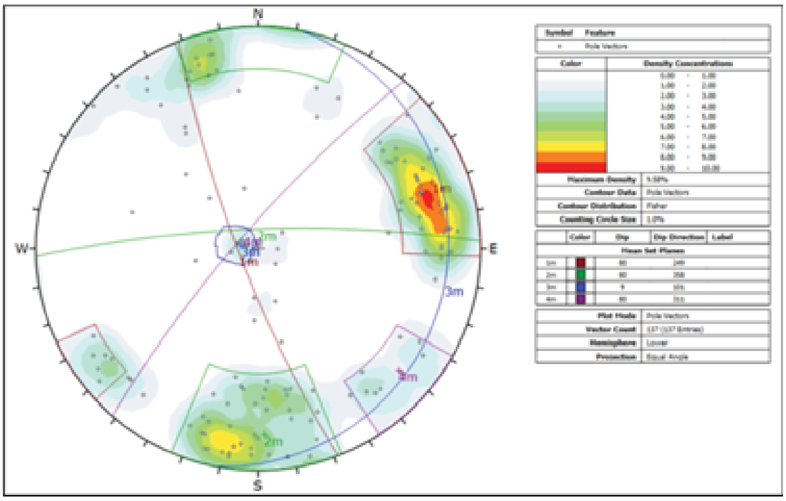
Table 8.2.1_1
Tharisa Minerals
Summary of Joint Sets Identified in the Open Pit

Joint set	Dip (degree)	Dip Direction
J1	80	249
J2	80	358
J3	9	101
J4	80	311

.

The exposed rock surfaces in the open pits were also limited as most of the areas were affected by blasting damage. Unfortunately the mapping could therefore not be performed in each lithology. No regional structures were mapped or logged.

Figure 8.2.1_2
Stereonet of Joints Mapped from In Pit Observations



<u>Geological Structure</u>

The only geological structures of note are a major fault which strikes approximately east west and is near vertical. It should have no major effect on the open pit mining. Although faulting is limited in the area, the majority of minor faults are anticipated to be of the high angle-normal or reverse faults.

A thin shear zone which is often altered is located below or in the MG1 Chromitite Layer. Due to its position it should have no effect on the design of the open pit. However in localised areas it may mean additional support or larger pillars needed in the underground mine.

From previous site visits the following observations were made:

- The drill core from the geological drilling campaign is in a good state and is stored in the core shed on the property.

<u>Rock mass quality</u>

The rock mass quality was quantified by OHMS using the RMR methodology proposed by Bieniawski and for the purpose of comparison the Bartons Q rating was also determined. The rock mass classification was done from exposures in the current east and central pit. Figure 8.2.1_3 illustrates the methodology for rating. The results are presented in Table 8.2.1_2.

Figure 8.2.1_3
Photograph Demonstrating the Method of Rock Mass Rating Determination

Table 8.2.1_2 Tharisa Minerals Summary of Rock Mass Ratings						
Area	**MG1 Chromitite Layer**	**MG2 Chromitite Layer**	**MG3 Chromitite Layer**	**MG4 Chromitite Layer**	**MG1- MG2 Chromitite Layers Parting**	**MG2- MG3 Chromitite Layers Parting**
RMR	68	69	65	71	74	73
Q Rating	6.01	13.4	10.05	13.4	13.99	13.4

.

An adjusted MRMR value is used to take into account weathering. The rock mass ratings used for design purposes also allowed for existing blast damage. An MRMR average value of 53 was derived for the rock mass.

Rock strength testing

Samples were selected for a series of uniaxial and triaxial strength tests. All tests were conducted strictly according to the prescribed ISRM procedures.

The uniaxial compressive strength tests, of core samples collected from fresh rock, were performed to also quantify the Young's modulus and Poisson's ratio of the rock types. The UCS values obtained from the laboratory tests were evaluated using the Modulus ratio method: In addition Brazilian indirect tensile strength (UTB) testing was carried out which also confirmed the accuracy of the UCS values obtained as it is generally assumed that the UTB value approximates 10% of the UCS (Table 8.2.1_3).

Table 8.2.1_3 Tharisa Minerals Summary of Rock Strengths		
Lithology	**UTB method**	**Modulus Ratio method**
Anorthosite	270.5MPa	229.08MPa
Pyroxenite	197.0MPa	186MPa

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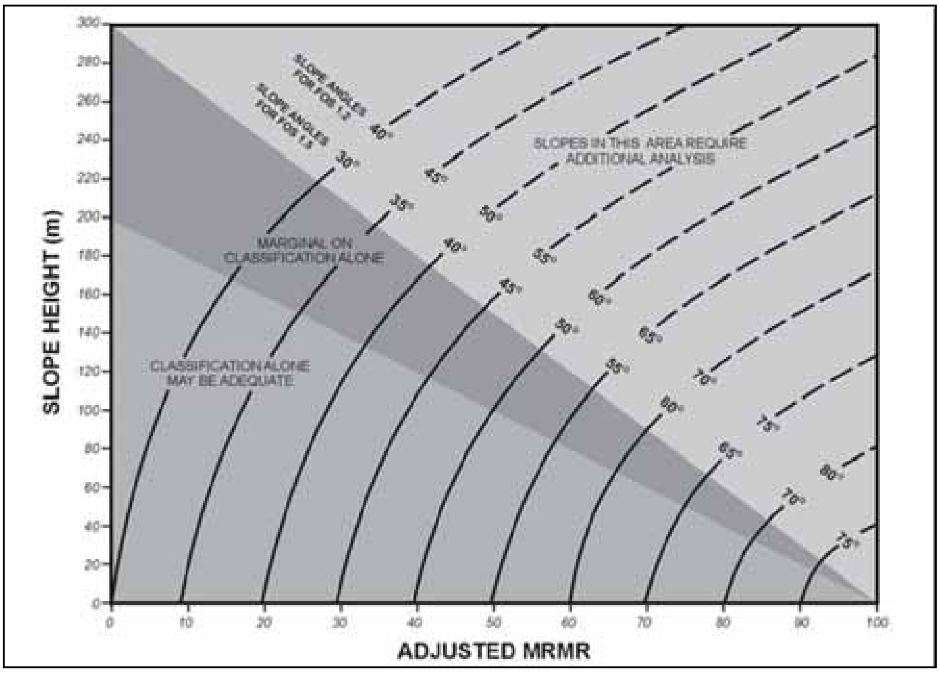
Hydrogeology

During the visits there was evidence of groundwater seepage from the exposed highwalls. Pit dewatering is conducted from toe drains at the advancing highwall. The hydrogeology is being monitored for environmental reasons as the mine deepens, this data should be incorporated in the geotechnical data base. The OHMS slope design is based on a dry slope as the pit will be dewatered.

8.2.2 Open Pit Slope Design

For indicative purposes the Haines and Terbrugge empirical design chart was used to assess the probable safe slope angles (Figure 8.2.2_1). The adjusted MRMR value of 51 for fresh rock was used in the assessment.

Figure 8.2.2_1
Haines and Terbrugge Empirical Design Chart used to Assess Safe Pit Slope Angles



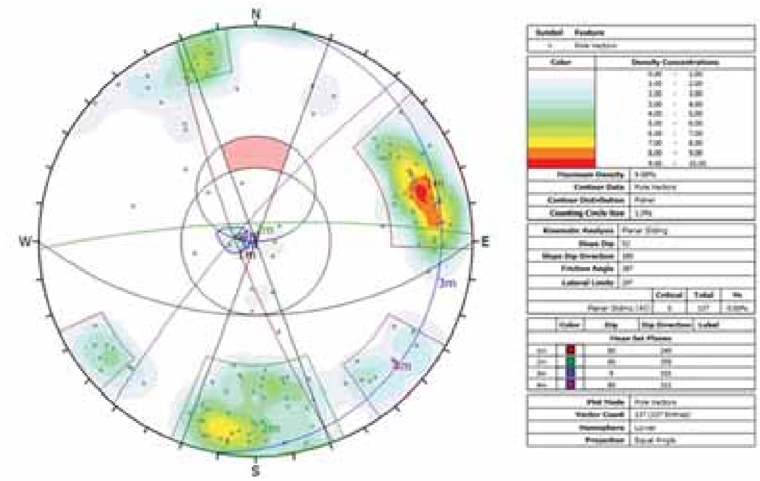
The Haines and Terbrugge design chart suggests that an overall slope angle of 52° in fresh rock will have a factor of safety of 1.2. This was taken as a guideline for further investigation using numerical modelling and kinematic analysis.

Kinematic analysis

The potential for structurally controlled failure modes of the northern highwall was investigated. The discontinuities measured on the outcrops were used for a kinematic analysis. A slope angle of 52° was assessed. For planar sliding to occur, a discontinuity must daylight in the slope and the dip of the discontinuity must be lower than the friction angle. The analysis is presented in Figure 8.2.2_2.

Figure 8.2.2_2

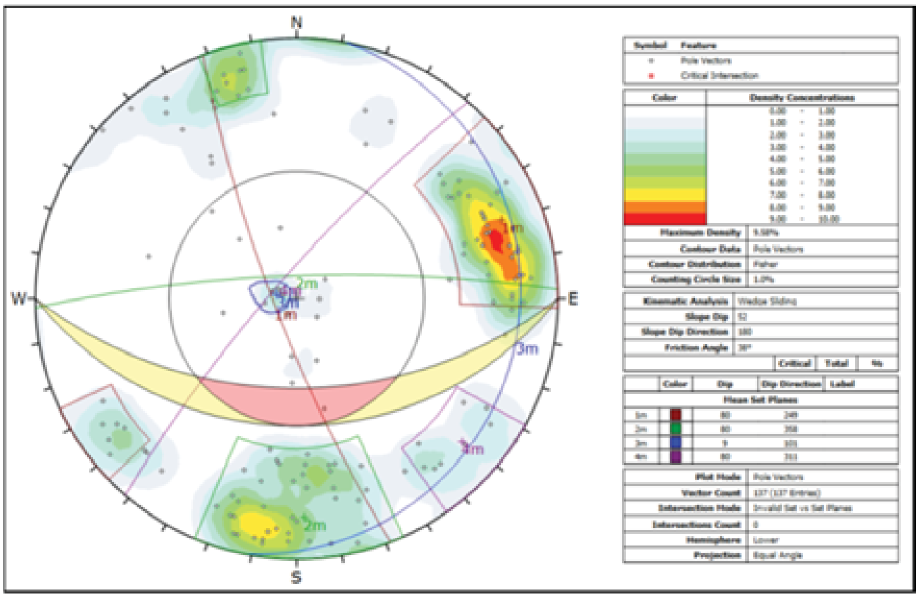
Stereonet and Analysis of Planar sliding in the Northern Slope



The wedge sliding kinematic analysis is based on the analysis of intersections of joint sets (Figure 8.2.2_3).

Figure 8.2.2_3

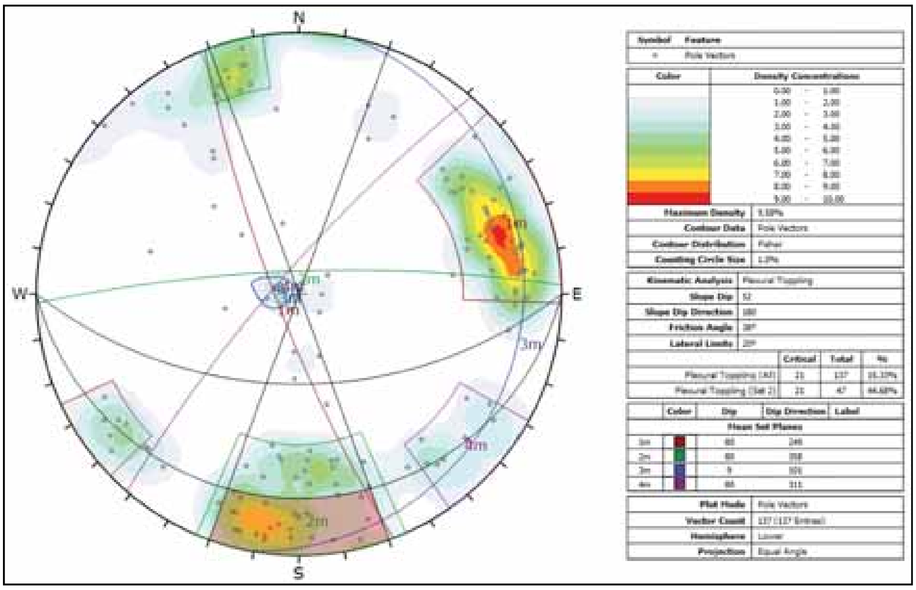
Stereonet and Analysis of Wedge Failure in the Northern Slope



The critical zone for flexural toppling is the highlighted region between the slip limit plane, stereonet perimeter and the 20° lateral limits. Any poles plotting in this region represent a potential risk of flexural toppling (Figure 8.2.2_4).

Figure 8.2.2_4

Stereonet and Analysis of Flexural Toppling Failure in the Northern Slope



From the stereonets it was concluded that no planar or wedge type failures are anticipated in the final highwall slope. The orientation of Joint Set 2 indicates that toppling failure is possible. The scale of this was not assessed and the potential would depend on joint continuity and cohesion.

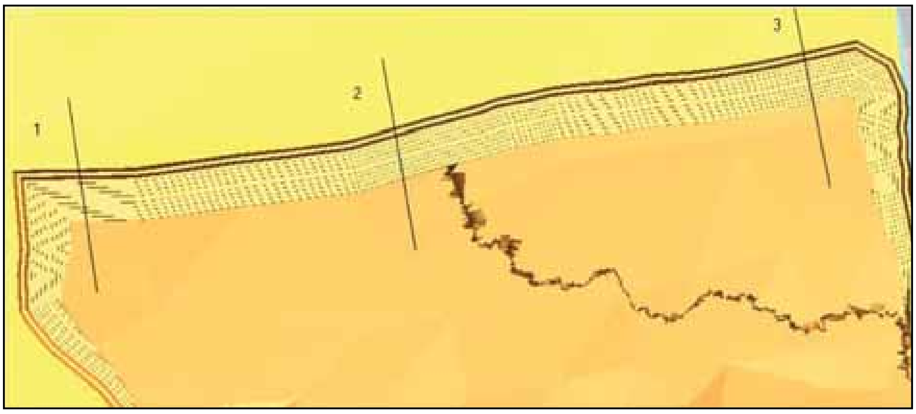
It was concluded that in the fresh rock, overall slope angles of 52° should be stable with catch berms of 9.4m wide.

Numerical Modelling

The slope stability was assessed using the Phase 2D, two dimensional, finite element software. The sections modelled for East Mine are shown in Figure 8.2.2_5. Models of 3 sections through the pit were constructed using the material properties as defined from laboratory tests and rock mass properties quantified using the RocData software program.

Figure 8.2.2_5

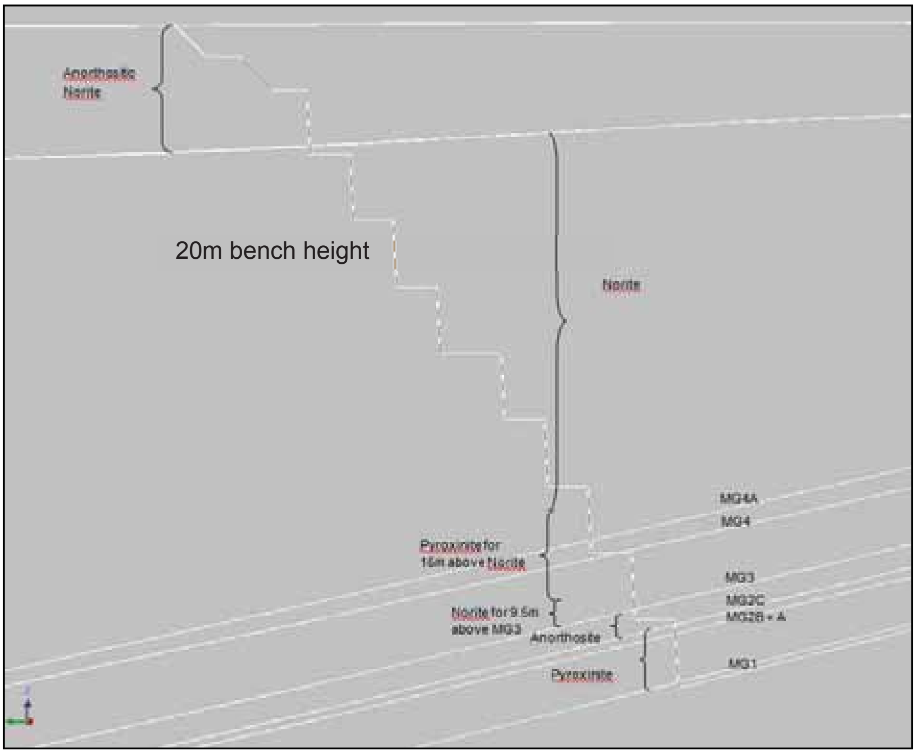
Schematic Plan of East Mine showing the positions of the Sections Modelled



Only saprolitic and fresh rock material properties were used for the Highwall slope (Figure 8.2.2_6). An overall angle of 53° was used to investigate the stability of the slope. The angle modelled for saprolitic rock was 45°.

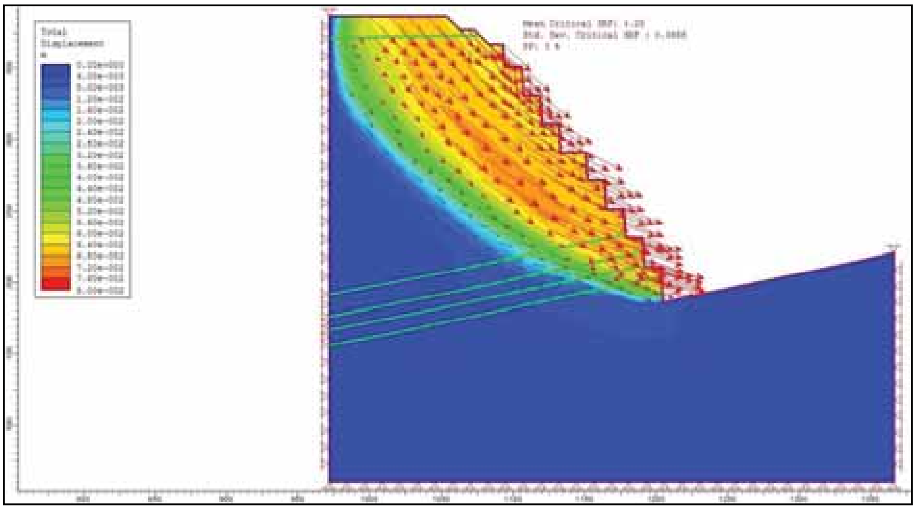
Figure 8.2.2_6

Diagram of the Proposed Highwall configuration showing Geology



The models simulated completely dry slopes, as it was assumed that an effective dewatering program will be implemented. An example of the numerical modelling is presented in Figure 8.2.2_7.

Figure 8.2.2_7
Graphic Example of the Hydrological Numerical Modelling Results



The Finite Element models calculated contours of displacement for the highwall. The Factor of Safety (FoS) and the Probability of Failure (PoF) were determined from these models and presented in Table 8.2.2_1. The likelihood of failure occurring was shown to be remote given the high Factor of Safety and low Probability of Failure.

Table 8.2.2_1
Tharisa Minerals
Summary of Rock Fall Hazard Analysis

Northern slope	Slope angle (fresh rock)	Slope angle (saprolitic rock)	FoS	PoF
Section 1	53°	45°	4.27	0
Section 2	53°	45°	4.25	0
Section 3	53°	45°	4.60	0

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Rock fall hazard analysis

OHMS used The Trajec3D rigid body dynamics software to simulate the trajectory of probable fall bodies. This software simulated the fall paths for three dimensional bodies, over a three dimensional surface, representing a pit geometry. The aim is to determine fall body velocity and kinetic energy at impact with road ways or catch berms. Three fall body geometries were selected for comparison, with two masses. The fall body geometries were selected to effectively simulate the most likely rock fall shape.

None of the falling bodies roll down the pit slopes and therefore it was concluded that the width of the catch berms will be sufficient to catch possible falls.

Seismic Hazard

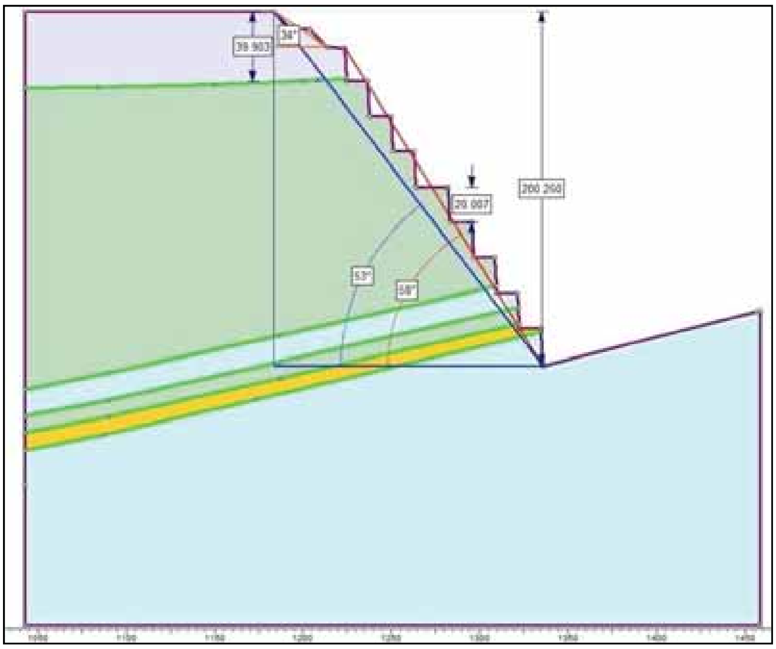
Using the seismic hazard map for South Africa produced by the South African Council for Geoscience it was concluded that Tharisa Mine does not fall within any of the zones of known seismic activity, whether natural or mining induced. The historic peak ground acceleration values are of the lowest in the subcontinent and therefore it was concluded that the potential influence of seismic activity on the stability of the mine is negligible and was not a consideration in the design of the slopes.

Conclusions

During the OHMS investigation, analyses and design, the following was carried out:

- The geotechnical conditions have been comprehensively assessed and the results found to be similar those of the previous investigation.
- Slope angles were determined from the Haines and Terbrugge design chart suggest overall slope angles of approximately 52° with a Factor of Safety of 1.2 in fresh rock. The proposed final design is presented in Figure 8.2.2_8.

Figure 8.2.2_8
The Proposed Final Pit Slope Design

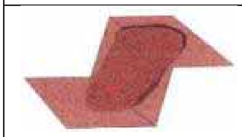
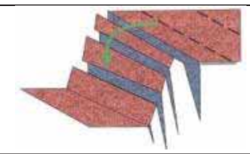
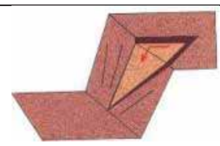


- Transitional surfaces between residual soil and saprolite, and between saprolite and fresh rock, were constructed from borehole information.

- Slope stability was assessed using Phase2D Finite Element Model. Factor of Safety and Probability of Failure suggested that overall slope angles of 45° in saprolitic rock and 53° in fresh rock, will yield very stable slopes.
- Kinematic failure was investigated and it was found that the Highwall may have some probability of toppling type failure related to Joint Set 2. Adequate catch berms are required.
- Rock fall hazard analysis was performed and it was concluded that catch berms with 9.4m widths were determined to be sufficient.
- No seismic activity is anticipated during the mining process.

The quantification of critical input parameters and level of detail considered in the design is sufficient for Life of Mine design. Various modes of failure were considered. These are illustrated in Figure 8.2.2_9.

Figure 8.2.2_9
Illustration of the Types of Slope Failure Considered

Modes of failure	Parameter	Modes of failure	Parameter
Circular		**Circular**	
			
Very unlikely, as shown by the Numerical Modelling	Most likely shown by Kinetic Analysis. However depends on the continuity of the jointing and will be halted by catch berms	Very unlikely as shown by Kinetic Analysis	Very unlikely as shown by Kinetic Analysis

The overall slope angle derived in the OHMS study may be conservative as the kinetic analysis indicated that toppling failure was a potential problem but all the other assessments indicated high factors of safety.

The toppling may be limited to small failures depending on joint continuity, and can be controlled with catch berms. Toppling failure is sensitive to bench slope and not to the overall slope. Further studies could steepen the overall slope of the final highwalls with attendant economic advantages.

No major geotechnical risks are anticipated.

8.2.3 Underground Mining

With regard to the future underground mining operation, the middlings between the various chromitite layers are a factor to consider in geotechnical design as with middlings of less than 12m it is usually necessary to superimpose the pillars. However the middlings between the MG1 and MG2A Chromitite Layers in most of the proposed underground mining areas are typically 12m to 15m or greater. The MG2C to MG4(0) Chromitite Layer middling is mainly

12m to 20m or greater. Thus interaction between the chromitite layers is not considered to be a concern. However this must be reassessed in localised areas once underground mining commences.

The mechanised trackless bord and pillar was deemed to be the best mining method for the mining resource under consideration.

The MG2 and MG4 Chromitite Layers were selected for underground mining. The combined thickness of the MG2A Chromitite Layer, parting and MG2B Chromitite Layer, in the greater part of the underground area, will be in excess of 1.8m. The MG4 Chromitite Layer is on average 3.0m thick and is wide enough for trackless Bord and Pillar mining and selected as the second mining horizon. Minimum and maximum mining cuts were set at 1.8m and 4.5m respectively.

The Potvin stability graph method was used to design stable panel spans for each chromitite layer. This method is widely used in South African platinum mines and incorporates the relevant geotechnical information based on a modification of Q, the Modified Stability Number N'. The maximum spans were calculated for used in a hybrid mining system. However recent findings indicate that in the MG1 Chromitite Layer, spans in conventional mining with mine poles and a middling to the MG2 Chromitite Layer of less than 15m, should be restricted to 15m.

Celtis Geotechnical investigated the maximum stable spans and pillar sizes for the underground mining as shown in Table 8.2.3_1.

Table 8.2.3_1
Tharisa Minerals
Summary of the Relevant Geotechnical Data for Underground Mine Design

Lithological Unit	Average N'	Average N"	Minimum N'	Minimum N"	Hydraulic Radius Minimum N' Unsupported
MG4 hangingwall	38.86	15.55	7.30	2.92	4.75
MG4A hangingwall	55.61	22.25	5.72	2.29	4.00
MG4- 4A middling	53.59	21.43	6.57	2.63	4.50
MG2 hangingwall	56.09	22.43	4.65	1.86	4.25
MG2 footwall MG1 hangingwall	39.45	15.78	5.92	2.37	4.50

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However, for the planned trackless bord and pillar mining, a bord width of 6m will be used throughout.

The DRMS or rock mass strength for each chromitite layer to be mined was calculated taking into account the effects of weathering, joint orientation and method of excavation. This was used to calculate the size of the in-panel pillars. A range of pillar sizes for the various depths and mining widths were calculated. Rigid pillars will be left to prevent plug failure and back-break problems. Down to a depth of 600m, the pillars were designed as non-yielding pillars which can support the whole over burden load from surface. The stress was calculated using tributary areas theory, and the pillar strengths were calculated by the Hedley and Grant (H&G) formula. As the mining will all be below 200m below surface where tributary areas theory over-estimates the pillar loading, Factors of Safety in excess of 1.3 were considered stable. Below 350 m, crush pillars can be considered, sized to suit the mining width of each chromitite layer.

The primary support in Bord and Pillar mining is the in situ pillars. A pattern of 2.4m grouted roofbolts, or equivalent splitsets, spaced at 2m apart in the hanging wall should be sufficient under normal conditions. Long anchor tendon support will be installed if faulted areas are encountered.

Access to the underground workings will be through a triple decline shaft system on reef from portals in the highwall of the opencast mining to the MG2 Chromitite Layer. This decline set will also be used as the main intake airways for the mine. Initial access will be on apparent dip. The decline support will depend on local geotechnical conditions and excavation dimensions. Below 350m it is anticipated that the geological losses in the area may provide sufficient regional support. In some areas, specific regional pillars may need to be designed on the stoping horizon.

In order to proceed with the study for the future underground expansion of the mine, additional work will be required to verify the geotechnical conditions at the selected portal positions.

8.2.4 Rock Engineering

The mine has appointed a competent rock engineering consultant to undertake regular visits and inspections to the mine including the collection of geotechnical data required to ratify the slope designs. The mine is visited monthly and reports are made on the visits and any salient issues.

Mines in South Africa are required to have a Code of Practice (CoP) to combat rockfalls drawn up according to the guidelines of the Department of Mineral Resources. There is a CoP in place in the mine, which complies with the guidelines. The CoP was revised in September 2015. It is due to be reviewed in 2017.

The rock engineering service has put in place regular laser scanning of all pits to find joint orientations (Figure 8.2.4_1) and produce hazard plans (Figure 8.2.4_2).

Figure 8.2.4_9

Map showing the Joint mapping of the East Pit using laser scanning

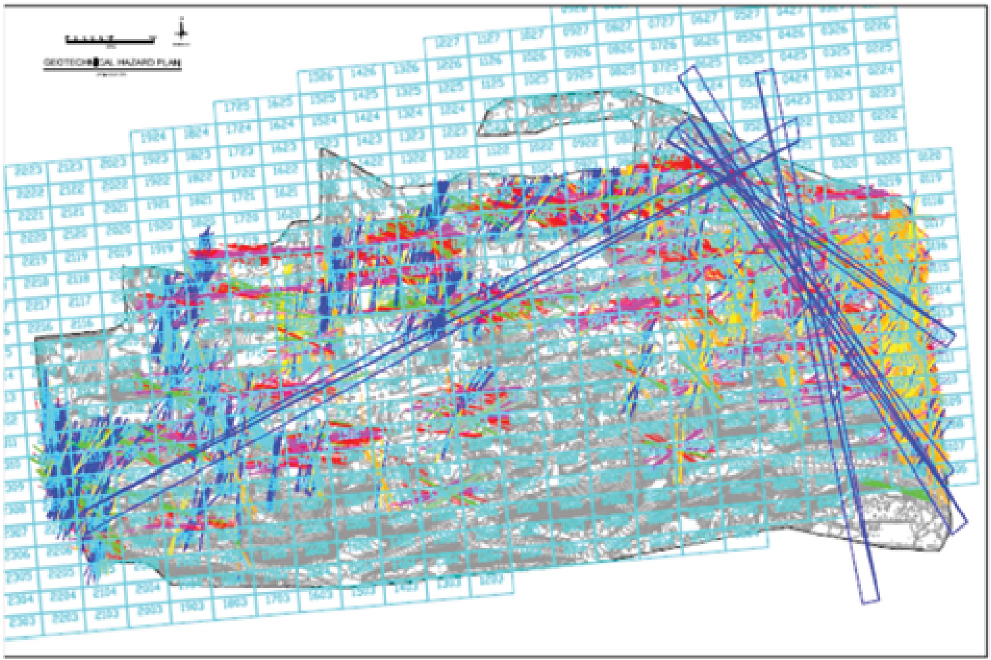
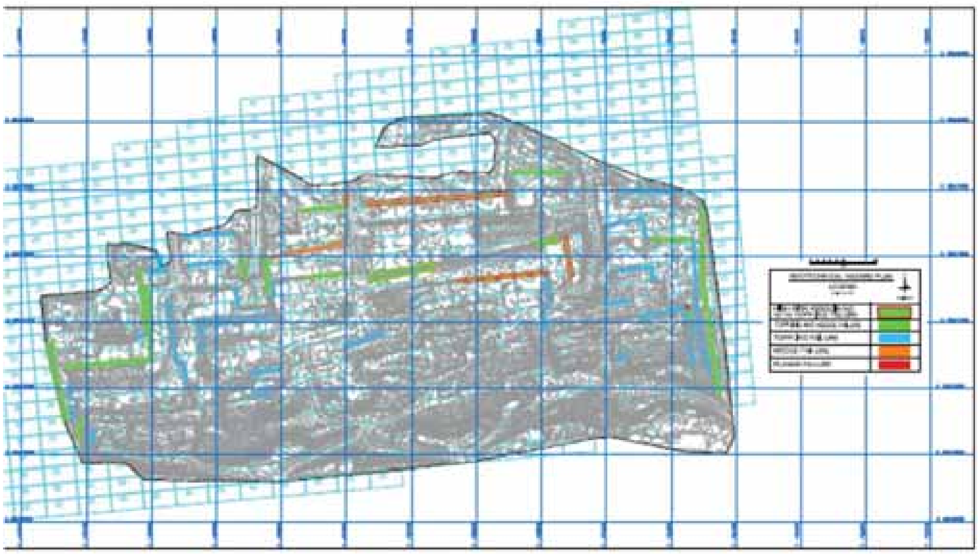


Figure 8.2.4_9

A Hazard map derived from Joint Mapping



8.2.5 Conclusions

The current mining has reached a depth of about 60 m. Current slopes are shallower than the design slopes. Designed bench geometry is being followed. The stripping ratio is low and will have to be increased to achieve the planned final pit depths.

The planned surface mining method has been devised with consideration of the geotechnical conditions anticipated in the ore body. The slope design is based on the study undertaken by OHMS. This is based on structural and geotechnical information obtained from in-pit joint mapping and the establishment of a geotechnical database.

The study ratified the design of the highwalls by dynamic analysis and numerical modelling.

Regular monitoring of the pit wall conditions and rock conditions is being carried out and reports on conditions and stability are being produced.

Measurement of jointing orientations and prospective hazards using laser scanning has been done on all the pits by Maptech. It is planned to repeat this regularly.

Groundwater level measurement and monitoring is being conducted for environmental management purposes. This data should be included in the geotechnical database.

The underground mining design has been conducted using modified stability number studies for stope spans and the Hedley and Grant methodology to calculate stable pillar sizes.

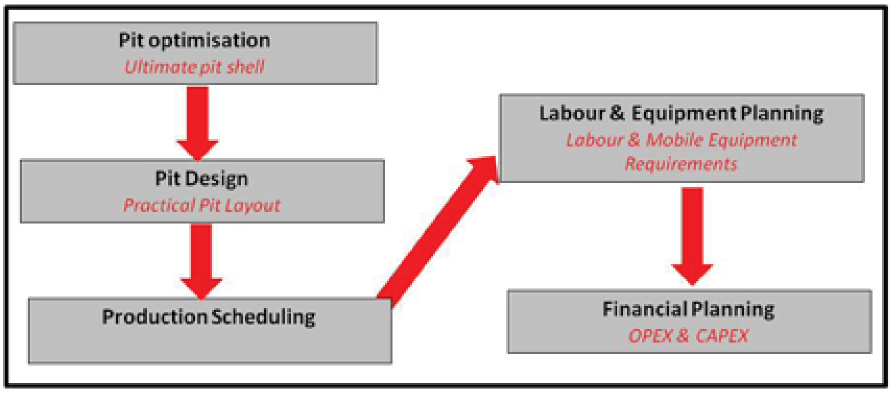
No major geotechnical risks are anticipated.

8.3 Open Pit Mine Design Study

A LOM planning process was followed to declare a Mineral Reserve for the open pits and the transition into underground mining. Practical limitations were considered to balance pit life and economic value. The final pit dimensions were selected to maximise value and LOM, considering factors such as modifying factors, scheduling constraints, unit costs and potential revenue. Mining contractor costs, transport costs, overhead costs, product selling price, and infrastructure costs were the major drivers in the cost model. The mining engineering process followed during the 2016 open pit mine design study is depicted in Figure 8.3_1. A pit optimisation with a West pit re-design was conducted for the 2016 Mineral Reserve estimation process.

Figure 8.3_1

Open Pit Mining Engineering Process Flow



8.3.1 Design Criteria

The design criteria were applied throughout the planning process to ensure that the work was undertaken in line with the guidelines of the SAMREC Code with a transparent reporting process and an executable plan.

<u>Safety berm</u>

The dimensions of the safety berm were calculated using global standards of good mining practice.

- Berm height = 1.7m
- Width of berm = 4.9m.

<u>Haul roads</u>

All mining equipment operate within the mining industry standard gradient of 1:10 (10% or 6°). The width of the haul road was based on the design criteria of a 3.5 multiple of the equipment width, plus the width of the safety berm with provision for a drainage channel to a minimum haul road width of 30m.

<u>Haul road width: Two way traffic</u>

- Width of equipment = 7m
- Width of haul road surface for two way traffic =23m
- Safety berm = 5m
- Drainage channel = 0.8m
- Design width = 30m.

Minimum operating width

The minimum operating width for the pit is limited by the equipment selection. For a 360t class hydraulic shovel, a minimum width of 40m is required for double sided loading. The 150t class haul trucks have a minimum turning diameter of 27.5m. A minimum mining operating width of 50m is sufficient for the bulk waste mining operations for a double side loading configuration.

Bench height

A bench height of 20m for bulk waste was selected to accommodate the large sized equipment. The first bench in the weathered zone must be battered at an overall slope angle of 35°. The ore is loaded in flitches depending on the MG Chromitite Layer thickness, using 65t excavators.

Waste Backfill

Waste backfill into the final void was considered during the haul road placement to optimise the available floor area available for dumping. Approximately 47% of all waste mined is dumped in-pit on the exposed pit floor. This has a material cost advantage relative to dozing or loading and hauling of the waste material from out-of-pit waste rock dumps (WRDs) during making safe process of the final void.

Initial waste material from the bulk waste above MG4A Chromitite Layer and the internal waste partings between the chromitite layers is used for the construction of tailings storage facility (TSF) walls. Further waste material is dumped on the permanent WRDs that are constructed to a maximum height of 60m, in 15m lifts, with an overall slope angle of 16°. A WRD is constructed at a safe distance north of the east pit high wall (WRD 1). Waste from the west pit is hauled to the south of the outcrop (WRD 2). An additional waste dump was constructed towards the north east of East pit highwall. Existing dwellings to the south of the west pit were relocated to the north of the west pit. An additional WRD with a capacity of 60 million LCM is required for East pit to accommodate the balance of the waste material.

Other Considerations

Various infrastructure constraints were considered during the detailed and operational planning processes. A road and a water canal is in the process to be diverted with expected completion date end of 2017.

8.3.2 Equipment Selection

MCC is required to supply the required mining equipment. MCC has similar contracts at adjacent mines with similar equipment and has extensive experience in hard-rock open pit mining.

Excavators (65t to 90t class) are used to load 40t to 80t class articulated dump trucks in the chromitite layer and waste parting zones. RoM ore is hauled directly from the pit to the RoM pad or placed on a designated stockpile or fed directly through the mobile primary crusher and sized to 200mm. Mining operations in the west pit is restricted to day-light hours compared to 24 hour operation in the east pit. The east pit is equipped with appropriate lighting plants on each production face with quality control enforced by grade control technicians.

Bulk waste above MG4A Chromitite Layer is loaded with 360t excavators and hauled with 150t dump trucks. Haul roads were designed at a maximum inclination of 10% and with a width of 30m, taking into consideration the 150t truck dimensions for safe two-way traffic.

8.3.3 Pit Optimisation

The pit optimisation was undertaken in 2016 using Geovia Whittle ("Whittle") pit optimisation software. The Whittle optimisation process identified which blocks should be mined and which ones should be left in the ground. In an effort to identify the blocks to be mined, an economic block model was created from a mining block model. This was done by assuming production, mining and process costs and long term view on chromite and PGM prices as recommended by Tharisa.

Using the economic block values, each positive block was checked to determine whether its value pays for the removal of overlying waste blocks. The analysis of pit limits which maximizes the relative net present value (NPV) required that the time value of money was taken into account to define which blocks should be mined and which blocks should be left in the ground or mined as waste during the life of the project. Pit optimisation parameters considered were:

- Pit slope angles
- Mining related modifying factors
- Physical characteristics
- Mining parameters
- Processing cost and revenue parameters.

The optimisation process is divided into two processes:

1. Creation of a range of nested pit shells of increasing sizes, that is constructed by varying the product price and generate a pit shell at each price point.
2. The selection of the optimal pit shell is achieved by generating various production schedules for each pit shell to calculate the relative value for each schedule. The output of this process is a series of "pit-versus-value" curves.

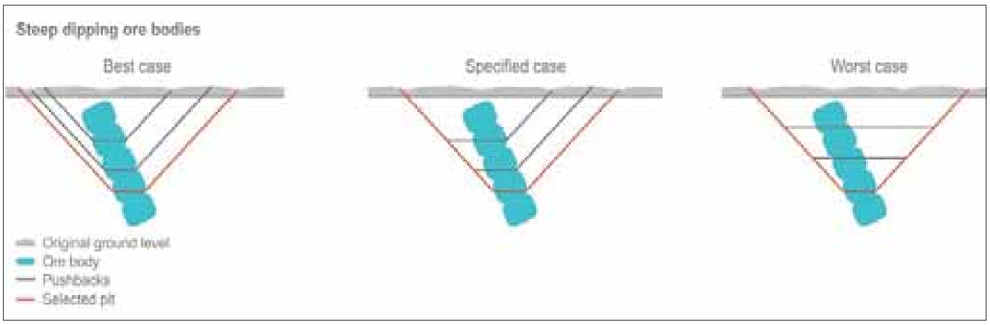
Three pit-versus-value curves are generated:

1. **Best case:** corresponds to minimum stripping in which mining follows the sequence of nested pit shells. Although this method gives you the highest relative value, it is not practical. It serves to provide the upper limit with regards to pit size
2. **Worst case:** waste material is removed level for level to correspond to the maximum stripping scenario and therefore lowest relative value. It serves to provide the lower limit with regards to pit size
3. **Specified case:** a case between the best and worst cases and models the influence of pre-stripping on the value curve.

The optimum specified Whittle shell is identified where the specified case is maximised. A simplified illustration of the definitions is shown in Figure 8.3.3_1.

Figure 8.3.3_1

Schematic showing the Hypothetical Best case, Specified Case and Worst Case Pit shells



8.3.4 Pit Optimisation parameters

A pit slope angle of 35° was assigned to the top 20m of overburden while 52° was assigned to the fresh hard rock portion. This resulted in an overall average slope angle of 48°. Mining costs were based on the agreed contractual MCC mining unit rates, current actual processing costs and existing bulk and local infrastructure costs. Long term forecast PGE prices were applied and converted to a Free on Mine optimisation basis. A discount rate of 9.24% was applied.

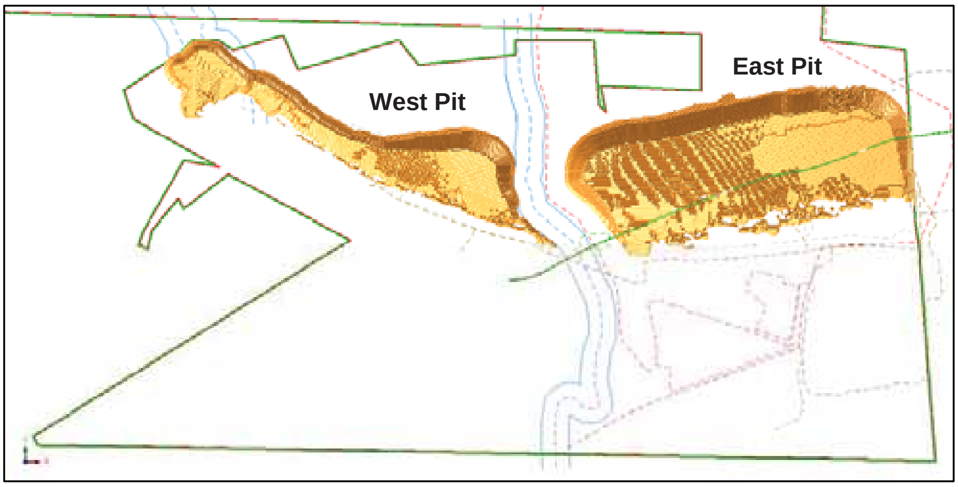
8.3.5 Pit Optimisation results

The Whittle optimisation process selects a combination of resource blocks that can be exposed to deliver the highest value in an open pit operation for a given set of design, operating and economic assumptions. It must be noted that the value stated in the optimisation process is a relative value based on the Whittle schedule including fixed and variable operational cost. The resultant pit shell represents the optimum pit based on the criteria used. A selection process to select the ultimate pit included four pit selection strategies viz. optimum relative value, maximised relative value with extended life, revenue factors and incremental cut-off cost. A single case is selected as a basis for the preparation of the detailed mine plan.

The pit selection strategy used considers the relative value of the selected case, but also maximises the life of the operation. Pit 37 was selected based on the selection criteria and contains a total of 73.4Mt ROM ore. The selected pit perimeter shell is in line with the current infrastructure placement and previous optimisations conducted on the incremental pit analysis at a maximum high wall depth of 200m (Figure 8.3.5_2).

Figure 8.3.5_1

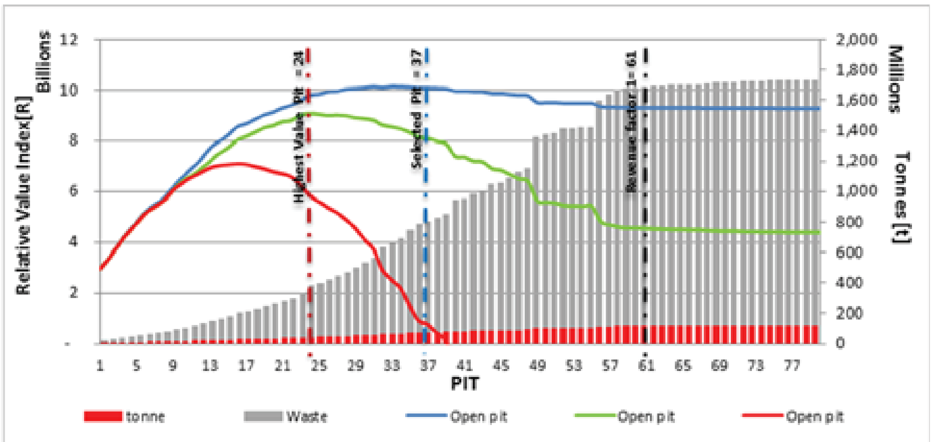
Schematic Diagram Showing the Pit Configuration with Existing Infrastructure Outlines



The ultimate pit was selected using the high value and extended life pit selection strategy as per Tharisa's corporate strategy. The strategy considered the optimum relative value and extended life based on the specified case of the Whittle pit-by-pit graph (Figure 8.3.5_2). Pit shell 37 was selected and contained 73.4Mt ROM ore and 729Mt of waste with a stripping ratio of 9.9t/t basis.

Figure 8.3.5_2

Graphic showing the Relative values of the Whittle Pit optimisation



8.3.6 Pit Design

Permanent ramps were designed on the high-wall of the east pit, thus reducing the overall high-wall slope angle from the previously accepted 53° to 48° on the latest east pit design. For the purposes of the strategic plan and Mineral Reserve estimate, the pit shell was modified in areas along faults where impractical 'waste islands' were placed and in areas where slumps in the pit floor were planned. The position of low wall access ramps were considered and are critical to the sustainability of ROM production. The surface layout is presented in Figure 8.3.6_1.

8.3.7 Mining Methodology

Waste is blasted in 20m benches. Depending on the dump location, waste is hauled to the dump located on the outcrop side or hauled through temporary ramps on the interim high wall to a dump located on the highwall side of the pit. Backfill are maximised and always maintained 100m behind the production faces. An estimated 47% of the waste is backfilled over the life of the operation. The backfill percentage is reasonable due to the low wall ramps, envisaged underground infrastructure and a minimum 100m down dip lag between the backfill and the working faces. The underground portals are established from the highwall.

The current reef mining methodology requires that MG1 is blasted selectively with MG2, MG3, MG4 and MG4a Chromitite Layers blasted with their respective surrounding waste. All the materials are loaded in 5m flitches with 65t to 90t class hydraulic excavators. In-pit grade control 'spotters' do all ore and waste classifications to control losses and dilutions based on the selected mining cuts.

Figure 8.3.6_1

Open Pit Surface Layout

8.3.8 Destination Scheduling

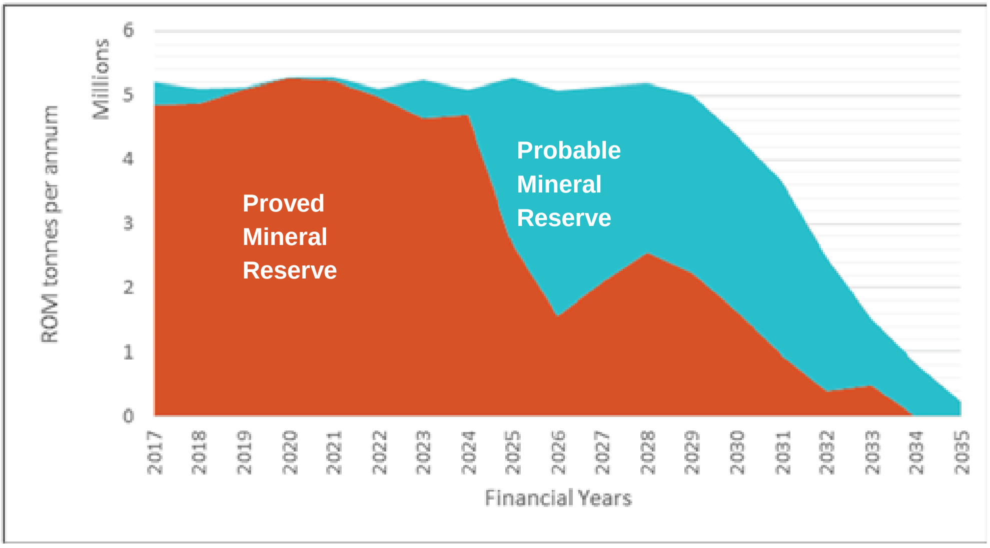
Hauling distances per period are estimated from the schedule based on the specific blast block mined, the dump destination and the haul route. Distances from the mined block to the closest ramp on each level are determined and added to the ramp and surface hauling distances. An appropriate cost model based on the contractual hauling rates and scheduled hauling distances were compiled.

8.3.9 Life of Mine Plan

Most of the material mined from the first ten years of the schedule is from the Measured Mineral Resource category which was converted into Proved Reserves (Figure 8.3.9_1). Indicated Mineral Resources were converted to Probable Reserves.

Figure 8.3.9_1

Open Pit Mine Schedule with Mineral Reserve Classification



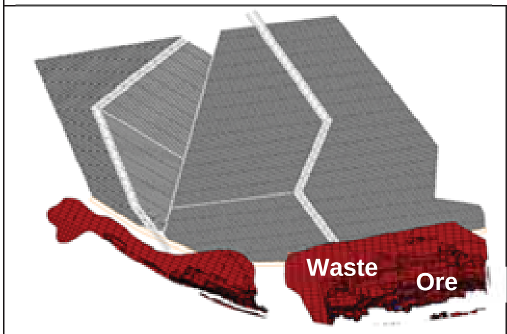
The schedule delivers an average 3PGE+Au grade of 1.14g/t over the life of the operation and 1.49g/t on a 5PGE+Au basis and RoM Chromite grade delivered at an average of 20.2% Cr2O3. During the previous financial year (October 2015 to September 2016) the combined plant feed grades averaged 18.0% for chrome with a lower than expected feed quantity of high chrome grade MG1 material, the PGM feed grade for the same period averaged 1.03g/t (4E). The production schedule indicates medium term plant feed grades of 20.9% and 20.6% (chrome) for FY2017 and FY2018 respectively. The medium term 4E PGM plant feed grades for FY2017 and FY2018 is 1.12g/t and 1.17g/t (4E). The schedule aim to produce a steady state RoM production of 430ktpm from the two. No physical mining or processing constraints was identified that would materially negatively impact planned production rates. The average production per month for the previous 12 months were 440ktpm. Steady state waste stripping requirements are set at 1.4 million BCM/month in total from the two pits. Steady state production from the open pit is maintained to 2029 when the underground production ramp up is planned.

A depiction of the mining schedule is provided in Figure 8.3.9_2.

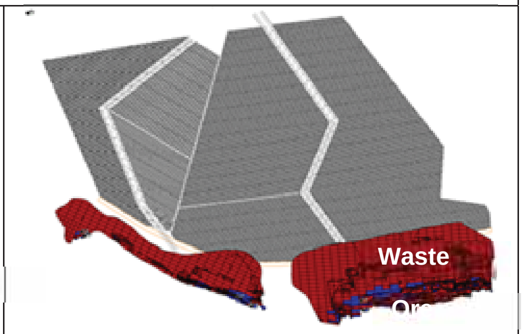
Figure 8.3.9_2

Open Pit Mine Schedule

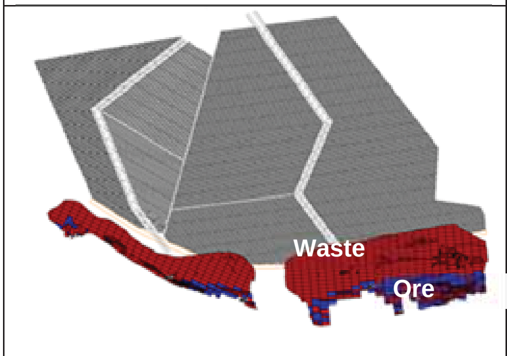
The figures below is a graphic representation of the mining sequence with the red indicated as the bulk waste and blue showing mining blocks that contain chromitite layers or selective waste.



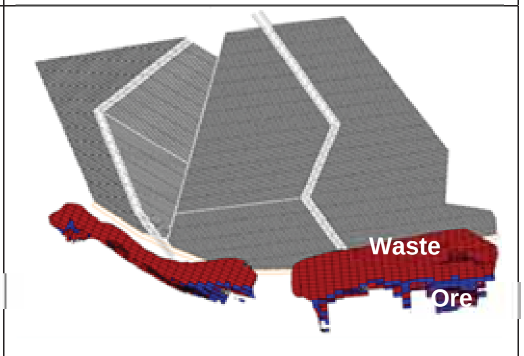
End September 2016



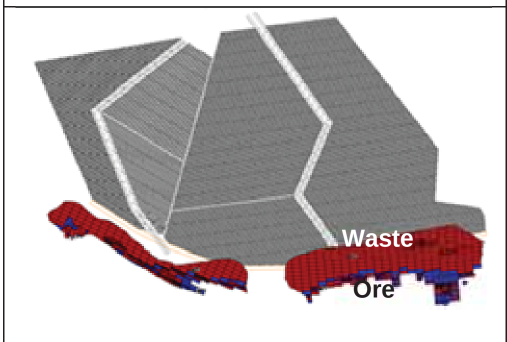
January 2017



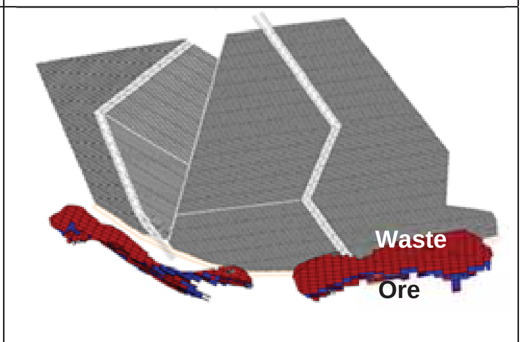
January 2019



January 2021



January 2023

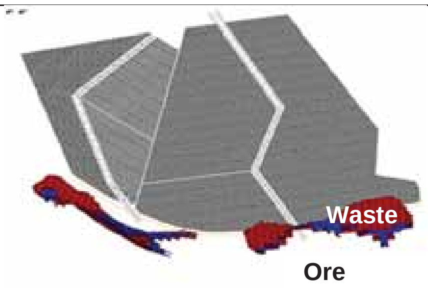


January 2025

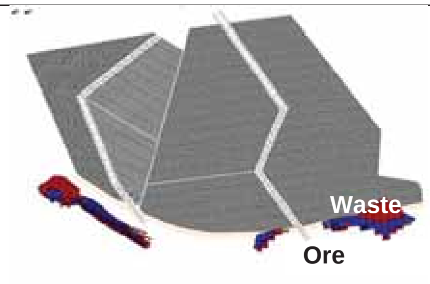
Figure 8.3.9_2

Open Pit Mine Schedule

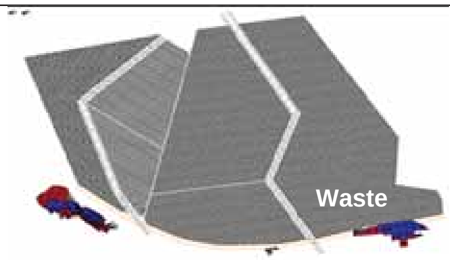
The figures below is a graphic representation of the mining sequence with the red indicated as the bulk waste and blue showing mining blocks that contain chromitite layers or selective waste.



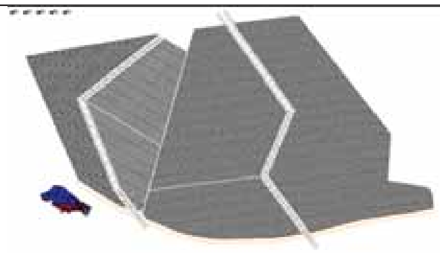
January 2027



January 2029



January 2031



January 2033

8.4 Underground Mining

8.4.1 Introduction

The design requirements identified for the underground section included:

- An underground RoM production of 400ktpm as a continuation of the open pit production profile. Underground mining is planned to commence in 2029
- Health and safety aspects were considered to deliver a relatively low risk operation
- Maintain profitability.

8.4.2 Mining Method Selection

The critical aspects considered during the mining method selection included safety, the Chromitite Layer widths, the dip parameters, the required volume of RoM ore, minimised waste development and mining cost. Four mining methods were considered:

- Conventional breast stoping;
- Hybrid mining;
- Mechanised dip mining;
- Trackless Bord and Pillar.

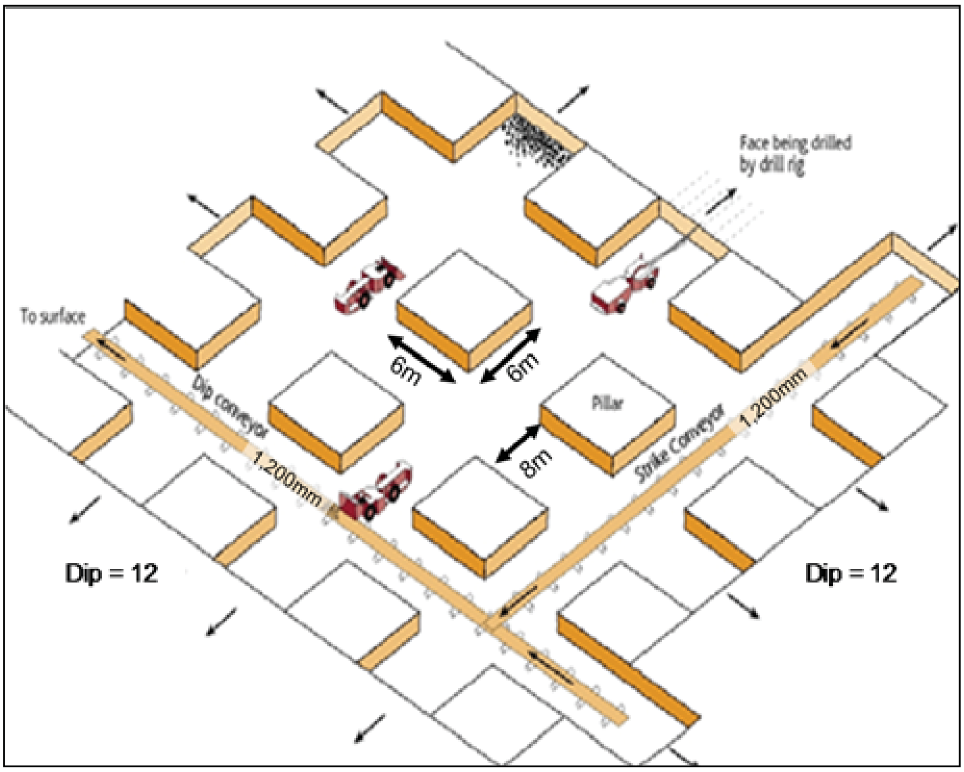
Trackless Bord and Pillar mining was selected as the preferred mining method. Compared to the other systems, it offers the following advantages:

- Development rates are faster.
- Flexibility in dealing with geological structures.
- Safety is enhanced as people are removed from high energy contact sources. Supervision is improved through mobile access to the workings.
- Mining extraction is achieved by developing a series of bords on reef and connecting them via holings to form pillars that provide support for the overlying strata (Figure 8.4.2_1).

Three active faces are allowed in each section for drilling, three for support, three for cleaning operations and a further three as production contingency.

Figure 8.4.2_1

Typical Bord and Pillar Layout



Each section with a dip width of 168m is equipped with a 1,200mm advancing strike conveyor which is maintained not more than 80m from the active stoping faces to minimise LHD hauling distance. Each conveyor is equipped with a grizzly feeder to screen out boulders that is either crushed or scalped as waste. The main conveyor capacity was set at 400t/h and tips directly onto the 1,200mm main surface stockpile feed conveyor.

8.4.3 Chromitite Layer Selection

The MG2 and MG4 Chromitite Layers were selected for underground mining. The combined thickness of MG2A, parting and MG2B in the greater part of the underground area, will be in excess of 1.8m. This matches the minimum stoping width requirements for the selected trackless equipment.

The MG1 Chromitite Layer has an average in situ thickness of 1.3m which is not ideally suitable for mechanised bord and pillar mining. Excessive dilution would result in the application of mining related modifying factors. This chromitite layer was mined using conventional mining methods on the adjacent property. The mined-out workings exist within the current open pit perimeter and within the planned underground footprint area. Due to the low, 10m average, inter burden parting between MG1 and MG2 Chromitite Layers, only one chromitite layer was selected.

The MG3 Chromitite Layer is relatively thin at an average in situ thickness of 1.4m and is midway between the MG2 and MG4 Chromitite Layers. This layer was excluded from the underground investigation on the same principle as the MG1 Chromitite Layer due to the low in situ thickness.

The MG4 Chromitite Layer is on average 3.0m thick and is of sufficient thickness for trackless bord and pillar mining and was selected as the second mining horizon.

8.4.4 Mining Cut

<u>MG4 Chromitite Layer</u>

The MG4 Chromitite Layer was selected as the second mining horizon with an average in situ thickness of 3.0m which is wide enough for trackless bord and pillar mining. The selected mining cut includes MG4 Chromitite Layer, the pyroxenite parting and MG4(0) Chromitite Layer below. A maximum mining cut of 4m, with a minimum of 1.8m, was used as criteria for the mining cut selection. However where the thickness exceeded 4m, only the MG4 Chromitite Layer was selected for the mining cut.

<u>MG2 Chromitite Layer</u>

The mining cut is taken as the MG2A Chromitite Layer to the MG2B Chromitite Layer. The MG2C Chromitite Layer was not considered as part of the mining cut due to the width of the parting. The mining cut was optimised to allow for a minimum of 1.8m and a maximum 4.0m mining height. Where the chromitite layer exceeds 4.0m, the MG2A Chromitite Layer was targeted.

8.4.5 Underground Access Options

Various options to access the targeted chromitite layers were considered and after a systematic analysis the top three options were:

- Option I: A vertical shaft at the centre of gravity of the resource.
- Option II: A footwall decline 20m below the targeted chromitite layers.
- Option III: Declines on reef.

The on-reef declines, Option III, was considered to be the most suitable access system for the underground project. Plans showing the underground mining layout are presented in Figure 8.4.5_1 (MG2 Chromitite Layer) and Figure 8.4.5_2 (MG4 Chromitite Layer).

The advantages of this system are:

- All development is on reef.
- More information on the geology is obtained during development.
- No cross cut development in waste to reef horizons.

The main disadvantage of this option is the lack of surge capacity. A breakdown on the strike conveyor has a direct impact on production as operations can only proceed once the ore handling system is functional.

The triple on-reef decline system is used as the main ventilation intake airways for the mine and consists of:

- **Services Decline** for access by trackless mobile equipment.
- **Main Conveyor Decline** for ore handling. This decline accommodates other services such as pumping columns, potable water pipes, fuel lines, compressed air lines, power lines and a walkway. From investigations carried out, a 1,200mm size trough conveyor at a speed of 4m/s in this decline has the capacity to handle the planned tonnage including allowances for maintenance and unplanned disruptions.
- **Chairlift Decline** primarily for the transportation of men to and from the working faces.

The dimensions of the three declines have been set at 6.0m wide by 4.5m high. All the declines will be developed at an apparent dip of 9^0 to facilitate access with mobile machinery. A crown pillar of 50m on dip separating the surface and underground operations was allowed for in the design. The RoM production capacity for each set of declines is presented in Table 8.4.5_1.

Table 8.4.5_1
Decline System Design Capacities

Decline system	Capacity per month [RoM tpm]	Decline system	Capacity per month [RoM tpm]
MG2 East Decline	150,000	MG2 West Decline	50,000
MG4 East Decline	150,000	MG4 West Decline	50,000

Figure 8.4.5_1

Underground Layout for theMG2 Chromitite Layer

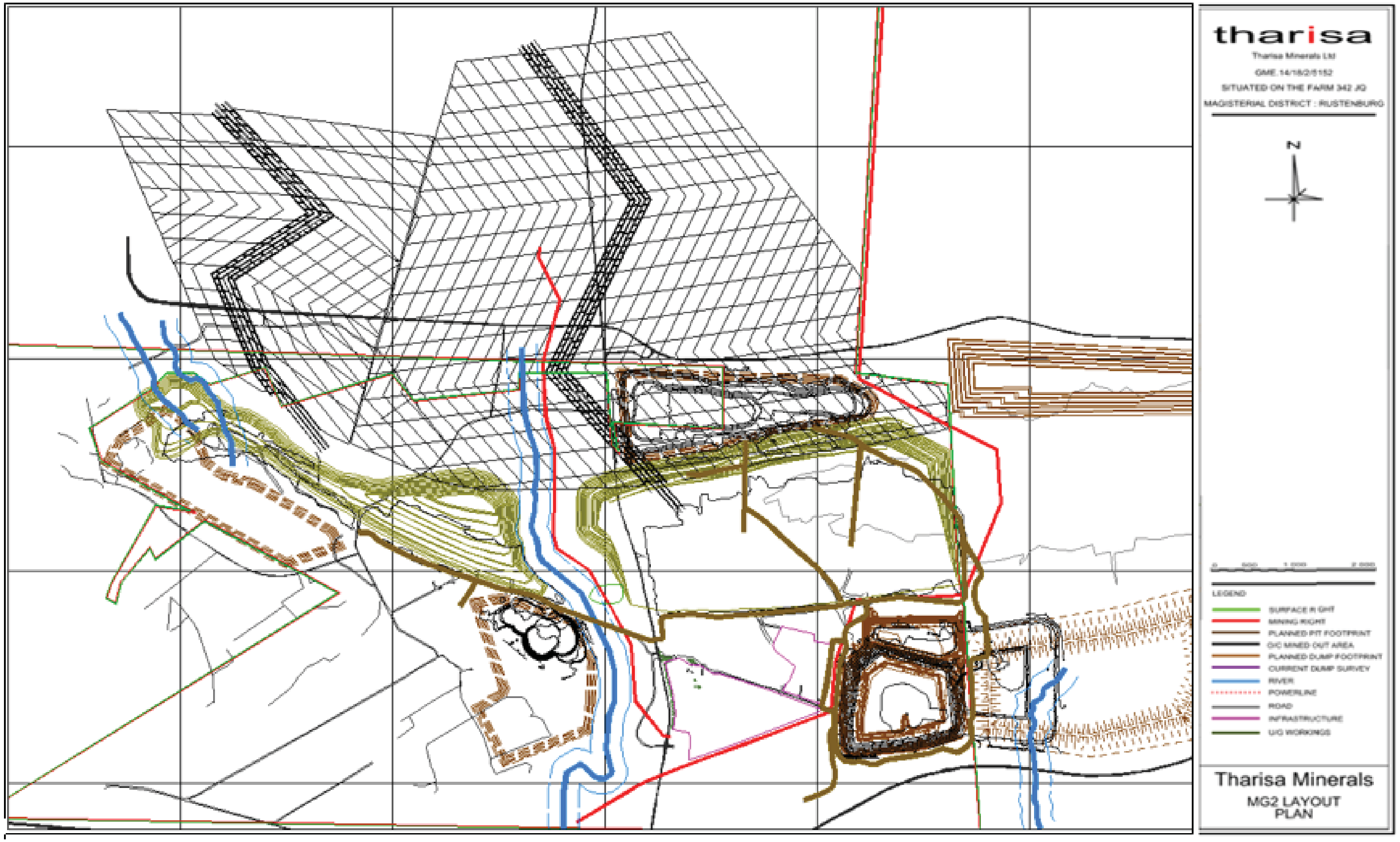
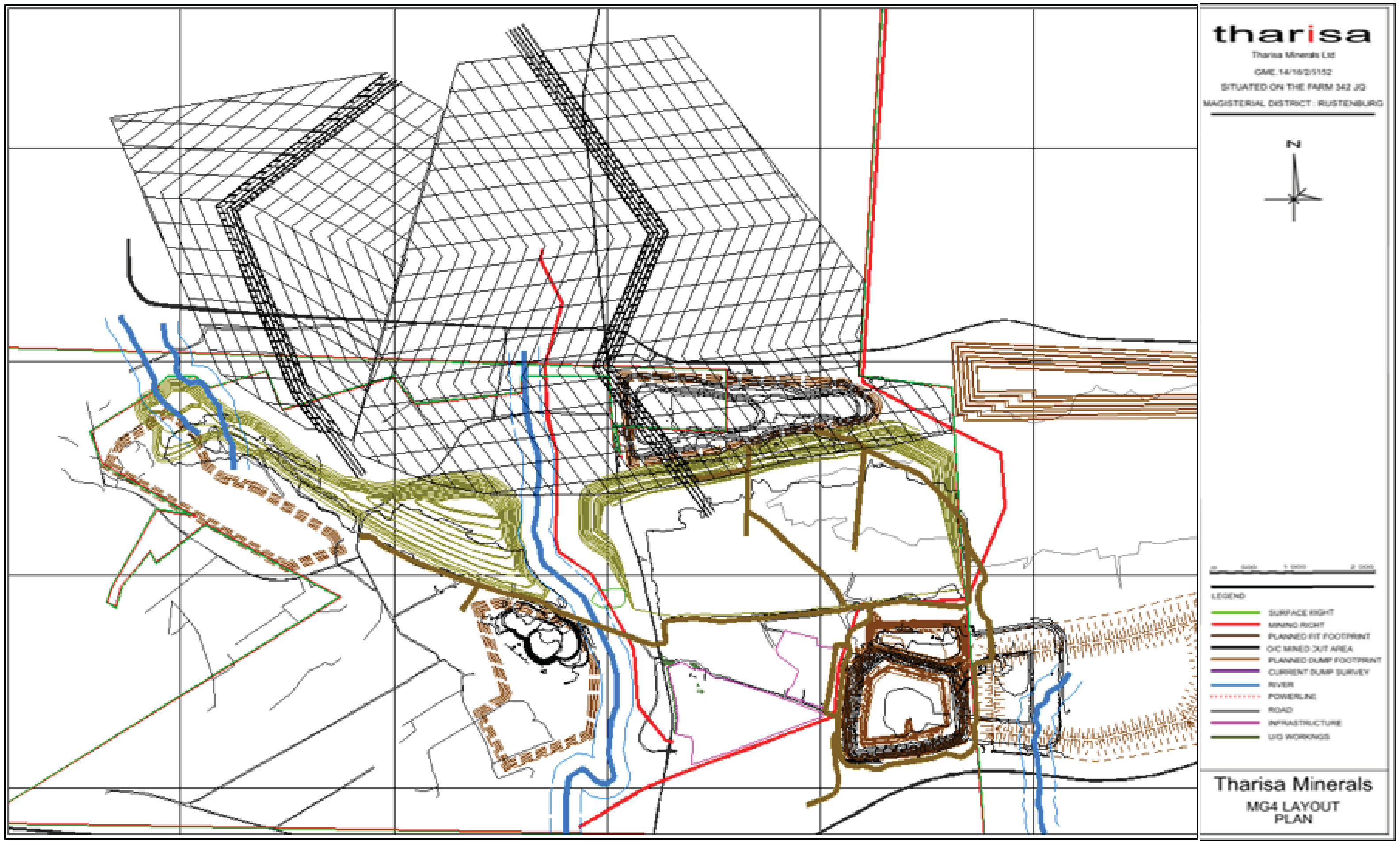


Figure 8.4.5_2

Underground Layout for theMG4 Chromitite Layer



8.4.6 Geotechnical/ Hydrological Considerations

The geotechnical parameters and pillar designs were recommended by Dr J James, Geotechnical Engineer. The recommendation is for 6m x 6m pillars on 8m bord spans and 6m holings for the stoping designs. The pillars are designed to increase with depth from 6m x 6m in the upper levels to 8m x 8m in the lower areas.

MG2A, MG2B and MG4 Chromitite Layer hanging walls are competent. A support pattern of 2.4m grouted roof bolts, or equivalent split sets, spaced on a 2m x 2m grid in the hanging wall was considered sufficient under normal conditions. Additional spot bolts are required if faulted areas are encountered as mining progresses.

The general hydrological conditions for the area were described as wet and the shallow open pit being mined at the time of compiling this report is being pumped almost continuously to maintain workable underfoot conditions.

Excessive water is not expected to cause any material risk to the planned underground operations. An appropriate water reticulation system was provided for in the capital cost. To minimize the inflow of water into underground workings, diversion trenches or embankments are installed around all the decline portals. Surface ventilation holings are protected from surface run-off water.

8.4.7 Equipment Selection

Equipment units were selected based on the planned production rates, chromitite layer geometry, excavation sizes and available technology. The minimum height that is traversed safely and efficiently by low profile machines is currently 1.8m.

A LH209L or equivalent LHD is suitable based on the above criteria. This LHD is 1.69m high and has a bucket reach of almost 5m making it an appropriate match for the planned mining cuts.

The Sandvik DL230L or equivalent drill rig, with a tramming height of 1.4m is the best fit.

8.4.8 Shift Cycle

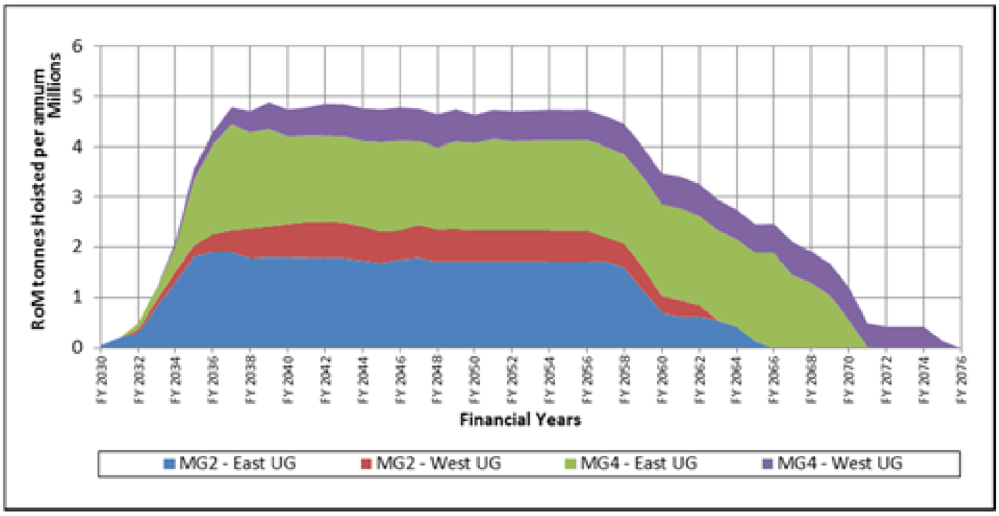
Mining production for the underground operations were planned for two ten hour shifts, five days per week. Drilling, blasting, lashing and supporting are the main activities on the morning shift while the back shift is mainly for lashing. Blasting is carried out once per day at the end of the morning shift while blasting during the sinking of the declines was set at twice per day during the first 18 months. A period of at least three hours was allowed for before re-entry after blasting.

8.4.9 Production Scheduling

Based on a production profile of 400ktpm, the scheduled underground production commences during financial year 2029 with initial development and continues to 2073 resulting in a mine life of 24 years at steady state production (Figure 8.4.9_1). The mine plans for MG2 and MG4 Chromitite Layer underground mining are presented in Figure 8.4.9_2 and Figure 8.4.9_3.

Figure 8.4.9_1

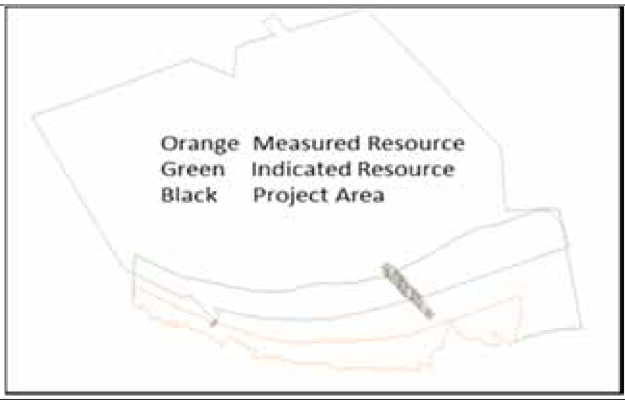
Underground Mine Production Schedule



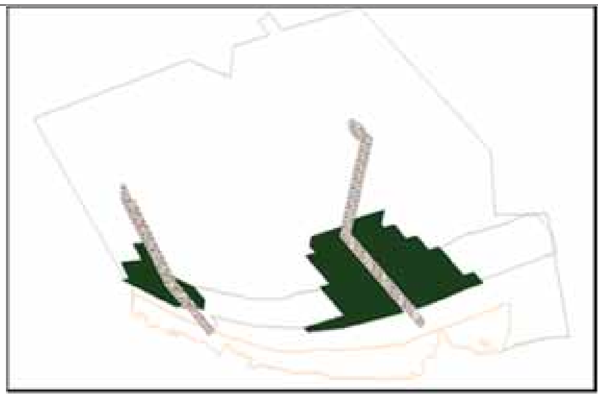
The scheduling strategy, which is a key driver to the overall project costs and economic value, was to establish the eastern decline system initially before moving to the western decline system. This strategy was chosen to minimise the project risk by starting off with areas of higher geological confidence and layer thicknesses. The sinking of the MG2 Chromitite Layer east triple declines system, starts five years before the projected winding down of open pit operations. At the planned advance rate, the mining of the triple MG2 declines and ledging to the Level 3, will be completed within 24 months with the ramp up to steady state within 48 months.

Figure 8.4.9_2

Underground Period Progress Plots: MG2 Chromitite Layer

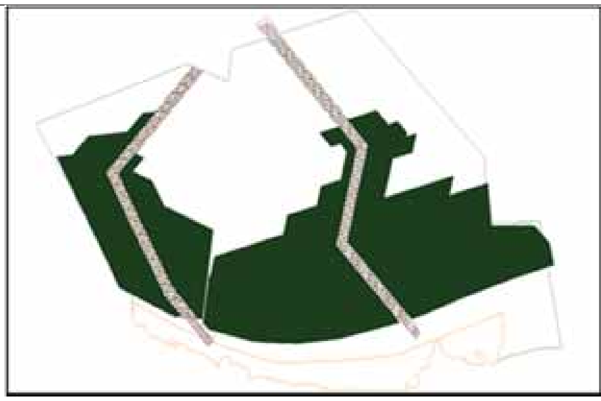


January 2030

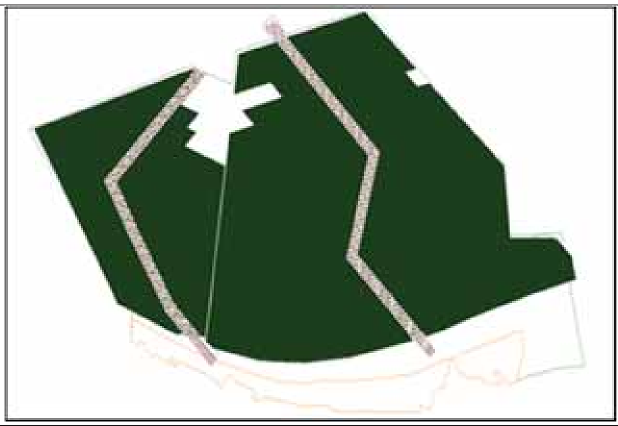


January 2032

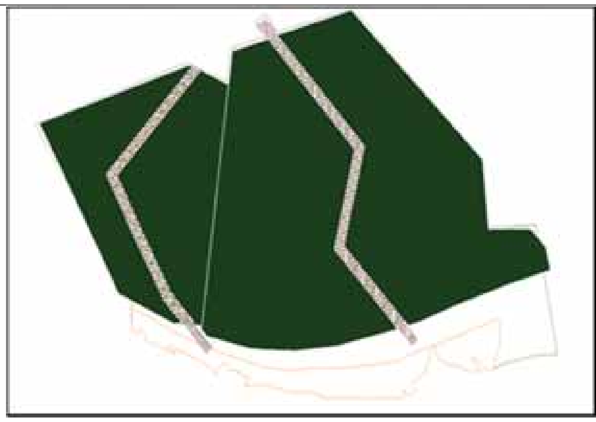
January 2042



January 2052



January 2062

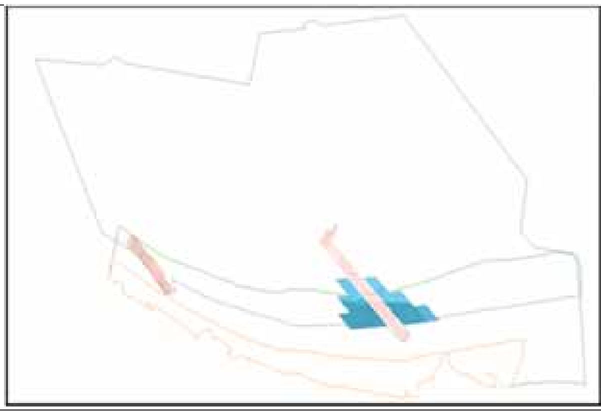


January 2066

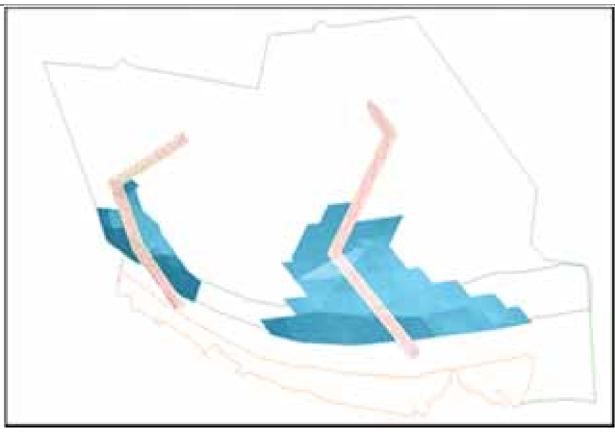
Figure 8.4.9_3

Underground Period Progress Plots: MG4 Chromitite Layer

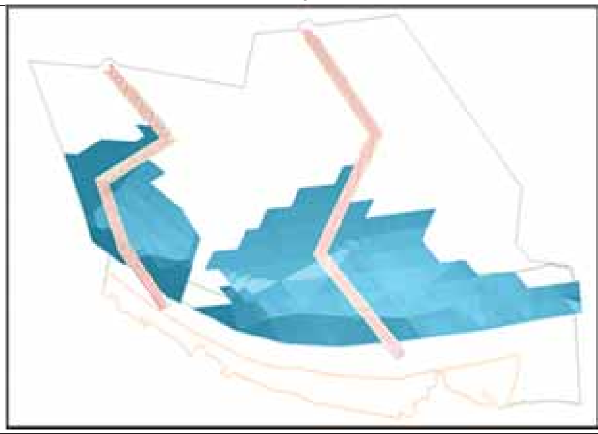
January 2032



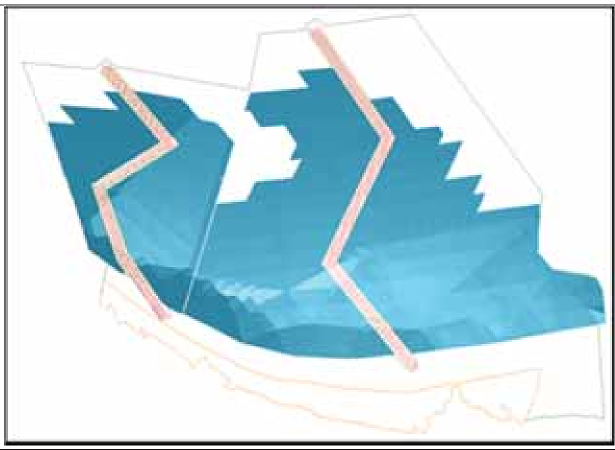
January 2034



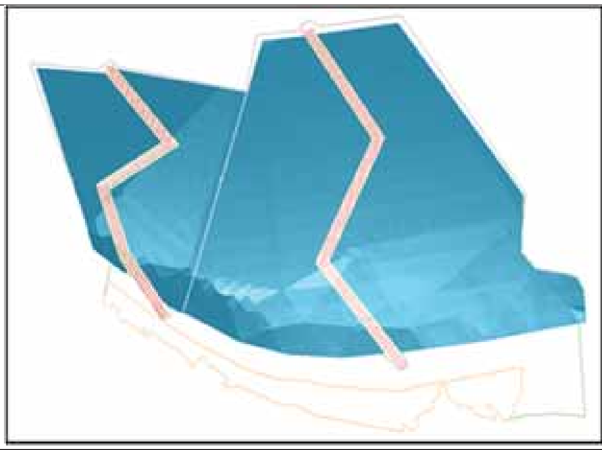
January 2044



January 2054



January 2064



January 2068

8.4.10 Infrastructure Requirements

The underground operations will leverage off existing infrastructure for open pit operations such as electricity, water, the plant, houses, offices, transport and communications networks that are in place when the underground operations commence. Additional infrastructure provided for in the capital cost estimate includes:

- The ventilation network
- Underground workshops and fuelling facilities
- Pumping arrangements
- Washrooms and lamp room facilities
- Emergency Facilities.

8.4.11 Labour

Except for a core owner's team, the majority of the labour force is contracted labour. Tharisa is located in a prime mining area with an experienced pool of labour to choose from. The owner's team, including the supervisory and management staff are retained from the open pit operations. Appropriate induction and training is required to ensure a smooth transition to underground operations.

8.4.12 The Underground Cost Model

An underground cost model was compiled from first principles and based on a 2013 schedule of rates. A contract mining site establishment fee of R20m per decline was assumed.

Capital Costs

A capital cost outlay of R2.23 billion including a 10% contingency is required to move the project to steady state production at a rate of 400ktpm over a period of five years. A summary of the initial major capital costs include:-

- R1,516m for decline development, equipping and conveyor installations
- R140m for site establishment, Preliminary and general and electricity costs
- R175m for portal establishment and support
- A 10% contingency.

Mining Operating Costs

The mining operating costs were sourced from the Ukwazi database and from relevant service providers. The operating expenditure estimate of R508/t (including a 10% contingency) compares favourably with other similar operations in the country employing the same mining method.

9 MINERAL RESERVES

9.1 Open Pit Mineral Reserve Estimation

A LOM planning process was followed to declare a Mineral Reserve for the open pits and the transition into underground mining. Various technical aspects were considered in the mine design and schedule including the determination of the economic pit limits, geotechnical parameters, mining methodology and sequence, pit access, ramp placement, equipment capability, production rates and practical mining considerations. The mining related modifying factors applied included geological losses, mining loss and mining dilution.

9.1.1 Geological Losses

Geological losses were applied at 5% for the East pit, 7.5% for the areas in the west pit where current production is taking place and 15% for the far west area in the west pit. This is in accordance with the recommendation of the competent person.

9.1.2 Mining Recovery (Mining Loss)

Mining losses was based on 6%, estimated on previous performance and determined by observation and measurement in the existing operation. The sources of mining losses included mining activities close to geological features, misalignment of reef excavator bucket size with the chromitite layer thickness, incorrect loading on the roof and floor of the chromitite layers and losses due to blasting activities.

9.1.3 Mining Dilution

Appropriate dilution was applied in the LOM plan and Mineral Reserve estimate. A mining dilution of 20.3% (estimated on a tonnage basis) was applied based on a reconciliation conducted between actual grades achieved and modelled grades with calculated dilution for the corresponding periods. The reconciliation consisted of production data for the preceding twelve months, based on actual plant feed grades achieved in both the Genesis and Voyager plants for the period. Dilution was planned for every chromitite layer based on the mining methodology employed for that specific chromitite layer. The chromitite layers that were mined with the surrounding waste rock were classified as non-selective mining and thus attracted a higher percentage dilution. Non selective mining units included MG2, MG3, MG4 and MG4A Chromitite Layers. The chromitite layers mined as selective was allocated a lower relative percentage dilution. The only chromitite layer that was deemed to mine selectively was MG1 Chromitite Layer.

The current actual dilution is 30% with a 10% improvement required to achieve the planned. Certain controls are put in place and entails the following:

- Grade controllers are employed on every operating mining face
- Experinced pit geologist are employed in the pit
- Adequate destination will be shown to the operators to ensure mining losses is minimised

- Proper lighting plants are positioned during night time ore loading
- Loading operations will be audited consistently to ensure correct loading practices are applied
- On-going in-fill drilling with sampling are completed to ensure the correct mining cuts are mined
- Monthly reconciliations are done between mining and plant feed grades.

9.1.4 Metallurgical Recoveries

Plant recoveries were based on actual performance while capacities were based on design capacity. The PGM recoveries are shown in Figure 9.1.4_1. The mass yield applied for a metallurgical grade chromite product based on the supplied yield curves as indicated in Figure 9.1.4_2.

Figure 9.1.4_1

PGM Grade Recovery Curves

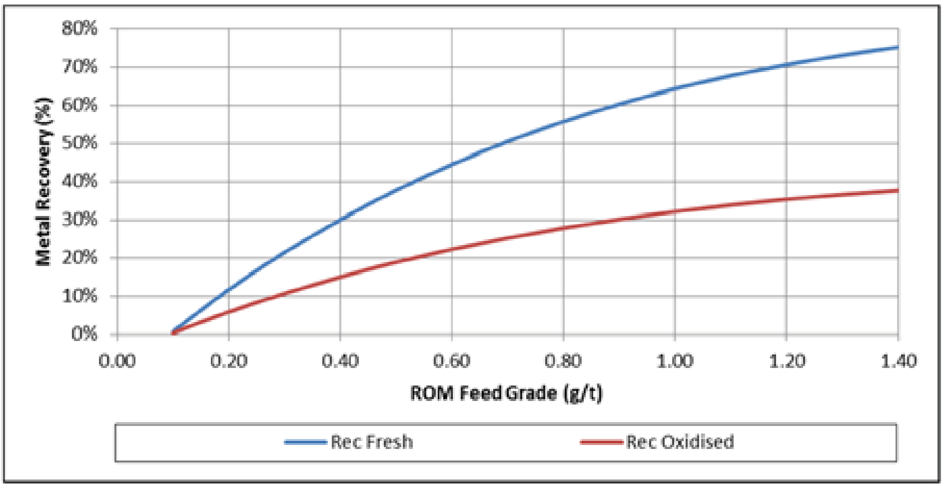
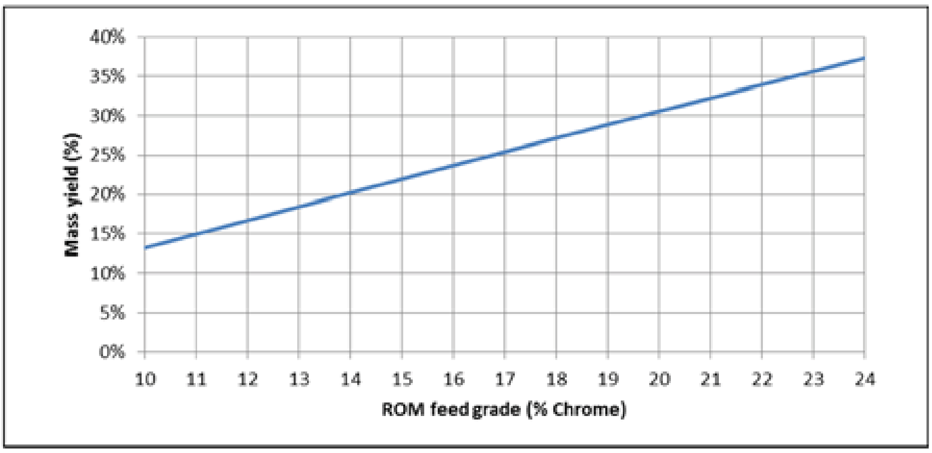


Figure 10.1.4_2

Mass Yield Curve – Chrome Concentrate



9.1.5 Financial and Revenue Parameters

The revenue parameters used in the financial assessment to allow declaration of a Mineral Reserve are presented in Table 9.1.5_1. The PGM prices were reduced as the metals are sold as a concentrate, and only attract a percentage of the metal value. No selling cost was assigned to the PGM's and a royalty of 4.7% was included. A Cost, Insurance and Freight (CIF) cost was allowed for transport and associated costs of the chrome concentrate to the ultimate destination in China.

Table 9.1.5_1

Open Pit: Financial and Revenue Parameters

Parameter	Unit	Value
Revenue		
Pt	US$/oz.	1 350
Pd	US$/oz.	847
Rh	US$/oz.	1 035
Au	US$/oz.	1 250
42% Cr_2O_3	R/t	2 327
Financial		
Discount rate	%	9.24
Royalty fee (% of revenue)	%	4.7
Chrome transport cost	R/t	750

Note: the economic parameters used to optimise the mining operation and determine the viability of the mining operation in order to declare a mineral reserve, may be different from those used in the valuation of the mine as a whole.

The commodity prices and foreign exchange rates used in the model were based on Tharisa's recommendations.

9.1.6 Capital and Operating Costs

The mining cost was based on the approved contract rates of the current mining contractor. The rate included drilling, blasting, loading and hauling on a semi selective mining basis.

Minimal capital is required for the mining operation as MCC supplies all the mining equipment. The capital is in effect incorporated into the mining rate which is captured in the mine operating cost estimate.

9.1.7 Mineral Reserve Tabulation

With the applicable modifying factors identified and evaluated as being reasonable, and the financial model yielding positive economic returns, the Mineral Resource within the mining footprint was converted to a Mineral Reserve. The Mineral Reserve was declared exclusive of UG1 and MG(0) Chromitite Layers.

The Mineral Reserve Estimate for the open pit section for Tharisa Mine is presented in Table 9.1.7_1 in accordance with the SAMREC guidelines.

Table 9.1.7_1

Tharisa Open Pit Mineral Reserve Estimation (30 September 2016)

Chromitite Layer	Tonnes (Mt)	Pt (g/t)	Pd(g/t)	Rh(g/t)	Au (g/t)	3PGE+Au (g/t)	Ru(g/t)	Ir(g/t)	5PGE+Au (g/t)	Cr_2O_3 (%)	Cu (%)	Ni (%)	Cr (%)
Proved Mineral Reserve													
MG0													
MG1	5 349	0.293	0.197	0.098	0.004	0.592	0.424	0.065	1.085	30.52	0.003	0.074	20.88
MG2	18 340	0.990	0.267	0.141	0.004	1.402	0.245	0.070	1.721	17.68	0.003	0.066	12.10
MG3	7 384	0.681	0.404	0.173	0.005	1.263	0.257	0.070	1.595	17.04	0.003	0.057	11.66
MG4	14 208	0.966	0.215	0.191	0.003	1.376	0.337	0.102	1.818	23.47	0.002	0.069	16.05
MG4A	8 972	0.369	0.135	0.111	0.003	0.619	0.234	0.046	0.902	21.76	0.004	0.069	14.89
Total	**54 253**	**0.770**	**0.244**	**0.149**	**0.004**	**1.167**	**0.286**	**0.073**	**1.530**	**21.05**	**0.003**	**0.067**	**14.40**
Probable Mineral Reserve													
Chromitite Layer	**Tonnes (Mt)**	**Pt(g/t)**	**Pd(g/t)**	**Rh(g/t)**	**Au (g/t)**	**3PGE+Au (g/t)**	**Ru(g/t)**	**Ir(g/t)**	**5PGE+Au (g/t)**	**Cr_2O_3 (%)**	**Cu (%)**	**Ni (%)**	**Cr (%)**
MG0													
MG1	3 332	0.287	0.181	0.095	0.003	0.567	0.406	0.063	1.039	27.99	0.002	0.068	19.14
MG2	11 902	0.864	0.241	0.127	0.004	1.236	0.215	0.062	1.517	15.43	0.003	0.062	10.55
MG3	3 780	0.655	0.362	0.162	0.004	1.183	0.244	0.067	1.498	16.49	0.003	0.054	11.28
MG4	4 458	0.916	0.207	0.177	0.003	1.303	0.309	0.096	1.711	21.56	0.002	0.063	14.75
MG4A	2 490	0.333	0.133	0.102	0.003	0.571	0.210	0.040	0.824	19.55	0.003	0.063	13.37
Total	**25 962**	**0.717**	**0.234**	**0.134**	**0.004**	**1.089**	**0.259**	**0.066**	**1.418**	**18.64**	**0.003**	**0.062**	**12.75**
Total Mineral Reserve													
Chromitite Layer	**Tonnes (Mt)**	**Pt(g/t)**	**Pd(g/t)**	**Rh(g/t)**	**Au (g/t)**	**3PGE+Au (g/t)**	**Ru(g/t)**	**Ir(g/t)**	**5PGE+Au (g/t)**	**Cr_2O_3 (%)**	**Cu (%)**	**Ni (%)**	**Cr (%)**
MG0													
MG1	8 681	0.291	0.191	0.097	0.004	0.582	0.417	0.064	1.067	29.55	0.002	0.072	20.21
MG2	30 242	0.941	0.256	0.136	0.004	1.337	0.233	0.067	1.641	16.80	0.003	0.064	11.49
MG3	11 164	0.672	0.390	0.170	0.005	1.236	0.252	0.069	1.562	16.86	0.003	0.056	11.53
MG4	18 666	0.954	0.213	0.188	0.003	1.358	0.331	0.100	1.793	23.01	0.002	0.068	15.74
MG4A	11 462	0.361	0.135	0.109	0.003	0.609	0.229	0.044	0.885	21.28	0.004	0.068	14.56
Total	**80 215**	**0.753**	**0.241**	**0.144**	**0.004**	**1.142**	**0.277**	**0.071**	**1.494**	**20.27**	**0.003**	**0.065**	**13.87**

Grades and tonnages are reported at shaft head

9.2 Underground Mineral Reserve Estimation

Mining related modifying factors applicable to the underground design were applied to convert the Mineral Resources to Mineral Reserves.

9.2.1 Geological Losses

A geological loss of 15% was applied based on the recommendations of the Competent Person.

9.2.2 Mining External Dilution

The mining dilution factors were calculated from first principles with the following assumptions:

- A 10cm layer of waste from the hanging and footwall horizons of the mined chromitite layer will be mined and conveyed as RoM ore.
- Depending on dip of the chromitite layer, some waste is mined to maintain safe and horizontal underfoot conditions as per design.

The dilution factors decrease with depth from 16.1% to 13.2% for MG2 Chromitite Layer and from 15.0% to 11.7% for MG4 Chromitite Layer. This is in direct proportion to the pillars sizes that increase with depth.

9.2.3 Mining Recovery

Mining recovery for both chromitite layers was set at the historical mining average for similar operations at 98%.

9.2.4 Mining Extraction before Geological Losses

This is mainly a function of the pillar size and was estimated from first principles. A decreasing trend with depth is indciated from 78.6% in the upper levels to 71.4% in the lower levels for both chromitite layers.

9.2.5 Mineral Reserve Tabulation

Indicated Resources included in the mine plan were converted to Probable Mineral Reserves.

This project includes Probable Mineral Reserves and material from Inferred Mineral Resources. The Mineral Reserve estimate for Tharisa is presented in Table 9.2.5_1 in accordance with the SAMREC guidelines.

The Mineral Reserve declaration is in respect of tonnage and grade delivered to the processing facility.

Table 9.2.5_1

Tharisa Underground Mineral Reserve Estimate (30 September 2016)

Chromitite Layer	Tonnes (Mt)	Pt (g/t)	Pd(g/t)	Rh(g/t)	Au (g/t)	3PGE+Au (g/t)	Ru(g/t)	Ir(g/t)	5PGE+Au (g/t)	Cr_2O_3 (%)	Ni (%)	Cu (%)	Cr (%)
Proved Mineral Reserve													
MG2AB	-	-	-	-	-	-	-	-	-	-	-	-	-
MG4	-	-	-	-	-	-	-	-	-	-	-	-	-
Total	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Probable Mineral Reserve													
Chromitite Layer	**Tonnes (Mt)**	**Pt(g/t)**	**Pd(g/t)**	**Rh(g/t)**	**Au (g/t)**	**3PGE+Au (g/t)**	**Ru(g/t)**	**Ir(g/t)**	**5PGE+Au (g/t)**	**Cr_2O_3 (%)**	**Ni (%)**	**Cu (%)**	**Cr (%)**
MG2AB	6.6	0.70	0.21	0.10	0.002	1.02	0.20	0.05	1.27	17.4	0.060	0.002	11.9
MG4	12.0	0.89	0.18	0.17	0.002	1.25	0.31	0.10	1.66	20.4	0.061	0.002	14.1
Total	**18.6**	**0.82**	**0.19**	**0.15**	**0.002**	**1.17**	**0.27**	**0.08**	**1.52**	**19.3**	**0.060**	**0.002**	**13.3**
Total Mineral Reserve													
Chromitite Layer	**Tonnes (‘000)**	**Pt(g/t)**	**Pd(g/t)**	**Rh(g/t)**	**Au (g/t)**	**3PGE+Au (g/t)**	**Ru(g/t)**	**Ir(g/t)**	**5PGE+Au (g/t)**	**Cr_2O_3 (%)**	**Ni (%)**	**Cu (%)**	**Cr (%)**
MG2AB	6.6	0.70	0.21	0.10	0.002	1.02	0.20	0.05	1.27	17.4	0.060	0.002	11.9
MG4	12.0	0.89	0.18	0.17	0.002	1.25	0.31	0.10	1.66	20.4	0.061	0.002	14.1
Total	**18.6**	**0.82**	**0.19**	**0.15**	**0.002**	**1.17**	**0.27**	**0.08**	**1.52**	**19.3**	**0.060**	**0.002**	**13.3**

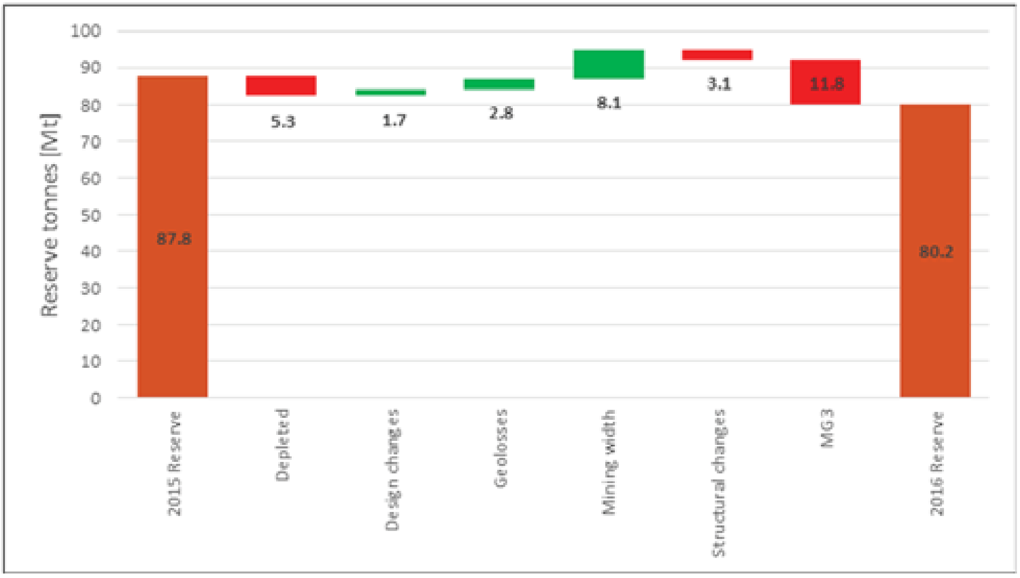
Grades and tonnages are reported at shaft head

9.3 Mineral Reserve Reconciliation

A Mineral Reserve reconciliation was conducted between the 30 September 2015 and 30 September 2016 reported Mineral Reserve. Figure 9.3_1 shows the variance in the 2015 and 2016 Mineral Reserve estimate.

Figure 10.3_1

Mineral Reserve Reconciliation



A volume variance occurred due to structural updates, mainly due to wireframe thickness variations between the estimated Mineral Resource wireframes and updated survey information as measured during the preceding 12 months. Planned dilution increased due to the change in mining methodology that resulted in an increase in Mineral Reserves of approximately 8.1Mt. The geological losses were estimated through on-mine measurements and resulted in the East pit geological losses to decrease from 7.5% to 5% and a decrease from 15% to 7.5% in the West pit near the current workings. Reserve depletions for the period were estimated at 5.3Mt. Local design changes accounted for 1.7Mt based increase in final open pit depth in the West pit and the additional Mineral Resources that were added in the far west area within the West pit. The mining cut for the MG3 layer reduced materially in the attempt to increase the chromite feed grade to the plant.

10 RISK ANALYSIS

10.1 Introduction

The risk analysis presented here is not a formal risk assessment. Coffey approach is to highlight areas of risk and the potential impacts of that risk that would normally be expected in similar operations. The focus is on highlighting areas of risk that are of relevance to project financiers or to potential project purchasers or investors.

In this report the risk analysis determines the level of risk which is classified from minor to major, as presented in Table 10.1_1.

Table 10.1_1
Definitions of the Levels of Risk

Level of Risk	Explanation
Major Risk	The factor poses an immediate danger of a failure, which if uncorrected, will have a material effect (>15% to 20%) on the project cash flow and performance and could potentially lead to project failure.
Moderate Risk	The factor, if uncorrected, could have a significant effect (10% to 15% or 20%) on the project cash flow and performance unless mitigated by some corrective action.
Minor Risk	the factor, if uncorrected, will have little or no effect (<10%) on project cash flow and performance.

The likelihood of a risk must also be considered for this analysis this is defined as the likelihood that within a seven year period an event may occur and is classified as likely (will probably occur), possible (may occur) or unlikely (unlikely to occur).

The impact of a risk and its likelihood are combined into an overall risk assessment as presented in Table 15.1_2.

Table 10.1_2
Overall Risk Assessment Matrix

Likelihood of Risk (within a 7 year period)	Level of Risk		
	Minor	**Moderate**	**Major**
Likely	Medium	High	High
Possible	Low	Medium	High
Unlikely	Low	Low	Medium

10.2 Risk Summary

Based on the sections above, a summary of the perceived risks to the Tharisa Mine are presented in Table 10.2_1.

Table 10.2_1
Overall Risk Assessment Analysis

Hazard/Risk Issue	Likelihood	Consequence Rating	Overall Risk Assessment
Geology and Mineral Resources			
Significant Variance in Resource Tonnage	Unlikely	Moderate	Low
Resource Grade Variation	Unlikely	Moderate	Low
Significant Variance in Geological losses	Unlikely	Minor	Low
Western Extend of Mineral Resource	Possible	Minor	Low
Geotechnical Engineering			
Open Pit Slope failure	Unlikely	Moderate	Low
Underground Fall of Ground	Probable	Minor	Moderate
Underground Pillar failure	Possible	Minor	Low
Mining Engineering			
Tonnage variation	Possible	Moderate	Medium
Grade Variation	Possible	Moderate	Medium
Open Pit Mining Method	Unlikely	Minor	Low
Production Schedule	Unlikely	Moderate	Low
Highwall Collapse	Possible	Moderate	Medium
Underground Mining Method	Unlikely	Minor	Low
Negative change in Opex	Possible	Moderate	Medium
Negative change in Capex	Possible	Moderate	Medium
Infrastructural			
Water Supply	Possible	Moderate to Minor	Medium to Low
Power Supply	Unlikely	Moderate	Low

Based on the above risk summary, Coffey considers the Tharisa Mine to have an overall ***Low to Medium Risk*****.**

10.3 Geology and Mineral Resources

The level of technical risk is defined as the likelihood of variation of resource tonnage and/or grade from the stated values.

The geological model developed by Coffey and the application to the mineral resource estimate.

- The geological model developed presents a tabular deposit with some dykes and faults crossing the property. Smaller scale faulting (<10m throw) must be considered. No potholes have been delineated although it is considered likely that some potholing of the MG Chromitite Layers has occurred. As these Chromitite Layers are not mined extensively elsewhere, it is difficult to assess the degree of potholing or the presence of small scale faulting. The application of a 7.5% - 15% geological loss is made based on knowledge of the Bushveld Complex and is intended to represent those areas where the MG Chromitite Layer is replaced by mafic pegmatites, intersected by faults or dykes, or disrupted by potholes.

The interpretation of the position of the most westerly point where a mineral resource can be declared is subjective.

- The interpreted position is considered to represent the likely extent of the deposit that can realistically be exploited based on the current data available, the current understanding of the geology and the macroeconomic understanding. It is possible that this boundary could move. It is considered more likely to move westward, effectively increasing the mineral resource base.

The overall geological risk is therefore considered ***Low***.

10.4 Mining

Coffey Mining associates a ***medium risk*** rating for the mining operation due to a concern relating to the amount of dilution which may report to the RoM ore and into the processing facility. Tharisa must place special emphasis on grade control and mining the width of the ore zone with limited dilution.

Any delay in the relocation of the roads, overhead power lines and water canals in the east pit area poses a scheduling risk. Reasonable time allocations were made in the LOM schedule for these relocations. Sufficient flexibility exists in the mining plan to reschedule activities to maintain the planned build-up profile.

The planned construction of a dam from the pit void at the end of the economic life of the operation poses a risk since the required regulatory approval must still be obtained. This application is in process and it is reasonable to assume that it will be approved.

10.5 Geotechnical Engineering

Geotechnical open pit slope and underground bord and pillar designs have been carried out using a probability based design, numerical modelling and dynamic wedge analysis, developed from detailed rock mass and rock material data coupled with structural data collected, which provide for greater certainty in the geotechnical design that is at an acceptable level of confidence for a mine of this size.

Coffey associates a **Low Risk** with the Geotechnical Engineering.

10.6 Infrastructure

Tharisa Minerals has obtained commitments to water and power that are suitable for the operations of the mine. According to the mine water consultant, there is adequate water to take the mine up to 400,000 tpm and maintain it at steady state production. This is in agreement with the water licence and water balance, however there may be a risk of water shortages during extreme dry times. If the amendment of the water licence is approved, allowing use of agricultural water, in the risk during extreme dry seasons will be reduced and allow the mine to function as required. Tharisa Minerals has finalised the arrangements with Eskom for provision of power as required, ensuring sufficient power for steady production.

Coffey associates a **Medium to Low Risk** rating for infrastructure.

10.7 Risk Summary

Based on the sections above, a summary of the perceived risks to the Tharisa Mine are presented in Table 10.7_1.

Table 10.7_1
Tharisa Mine Technical Risk Summary

Item	Relative Risk
Geology and Mineral Resources	Low
Mining Engineering and Mineral Reserves	Low to Medium
Geotechnical Engineering	Low
Infrastructure	Low to Medium

Based on the above risk summary, Coffey considers the Tharisa Mine to have an overall ***Low to Medium Risk*.**

11 GLOSSARY OF DEFINITIONS AND TECHNICAL TERMS

Term	Description
Au	Chemical symbol for Gold
Ir	Chemical symbol for Iridium
Os	Chemical symbol for Osmium
Pd	Chemical symbol for Palladium
Pt	Chemical symbol for Platinum
Rh	Chemical symbol for Rhodium
Ru	Chemical symbol for Ruthenium
3PGE+Au	Pt, Pd, Rh and Au
4E	Pt, Pd, Rh and Au
5PGE+Au	Pt, Pd, Rh, Ru, Ir and Au
6PGE+Au	Pt, Pd, Rh, Ru, Ir, Os and Au
7E	Pt, Pd, Rh, Ru, Ir, Os and Au
aeromagnetic survey	A geophysical survey method to measure the strength of the earth magnetic field using a magnetometer aboard or towed behind an aircraft.
AIDS	Acquired immune deficiency syndrome or acquired immunodeficiency syndrome (AIDS) is a disease of the human immune system caused by the human immunodeficiency virus (HIV)
anorthosite	A rock comprised of largely feldspar minerals and minor mafic iron-magnesium minerals
Arxo	Arxo Logistics (Pty) Ltd, a company registered and incorporated in South Africa. Arxo is the appointed logistics contractor for the Tharisa Mine.
Bushveld Complex	A major intrusive igneous body in the northern part of South Africa, that has undergone remarkable magmatic differentiation. It is by far the largest layered intrusion known. The Bushveld Complex is a leading source of chromium and PGMs.
Chromitite	A rock composed essentially of chromite, that typically occurs as layers or irregular masses exclusively associated with magmatic complexes. The bulk of the world's exploitable chromitite occurs almost exclusively in layered complexes.
Chromitite layers	Thick accumulations of chromite grains to form almost monomineralic bands or layers. Chromitite Layers are typically greater than 30cm thick..
chromium	The element chromium (Cr) is classified as a metal and is situated between other metals such as vanadium (V), manganese (Mn) and molybdenum (Mo) in the Periodic Chart of Elements.
Chromite	a hard, black, refractory chromium-spinel mineral consisting of varying proportions of the oxides of iron chromium, aluminium and magnesium.
Chrome mass yield	Chrome mass yield is calculated by dividing the chrome concentrate tonnes by the total feed tonnes and expressed as a percentage
Coffey	Coffey Mining SA (Pty) Ltd, a company registered and incorporated in South Africa.
Composite	A weighted accumulation of the intersection value to a specific length or over a specific stratigraphic unit
CPI	Consumer Price Index
CPR	Competent Persons Report
Critical Zone	A stratigraphic zone within the Bushveld Complex where a wide variety of different igneous rock types occur which host the bulk of the significant PGM and chrome mineralization i.e. Merensky Reef and UG2 Chromitite Layer.
DME	Department of Minerals and Energy – in 2009 the DME was split into the Department of Mineral Resources (DMR) and the Department of Energy (DoE)
DMR	Department of Mineral Resources
DTM	Digital Terrain Model
dyke	A wall-like body of igneous rock that is intruded (usually vertically) into the surrounding rock in such a way that it cuts across the stratification (layering) of this rock.

Term	Description
DWA	Department of Water Affairs
Eskom	South African electrical utility company
fault	A fractured surface in the earth's crust along which rocks have moved relative to each other.
Feasibility Study	The original feasibility study conducted by Coffey on the Tharisa Mine, which was concluded in October 2008
EIA	Environmental Impact Assessment
EMP	Environmental Management Programme
EPCM	Engineering, Procurement and Construction Management
FOB	Free on board
GATT	General Agreement on Tariffs and Trade
GDP	Gross Domestic Product
geostatistics	A branch of statistics focusing on the understanding of spatial data
GPS	Global Positioning system
HDSA	Historically Disadvantaged South Africans
highwall	The unexcavated face of exposed overburden of an opencast mine
HIV	Human immunodeficiency virus
IAPs	Interested and Affected Parties
ICP Fusion D/OES	Analytical technique to measure the concentration of trace elements
Indicated Mineral Resource (SAMREC)	An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.
Inferred Mineral Resource (SAMREC)	An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
IRUP	Iron-Rich Ultramafic Pegmatite – a type of rock which typically intruded into the Rustenburg Layered Suite of the Bushveld Complex, generally after the main mineralized layers were formed. IRUPs can replace the normal stratigraphic sequence over extensive areas, and can have a greater or lesser effect on the mineralized layers. They occur as pipes, dykes and sheets.
JSE	Johannesburg Stock Exchange South Africa. JSE Limited, a licensed exchange under the Securities Services Act, 2004
Farm 342JQ	The Farm 342JQ, registration division JQ, located in the Bojanala Municipal District in the North West Province, South Africa.
LG Chromitite Layer	Lower Group Chromitite Layer
LSE	London Stock Exchange
Lower Zone	Stratigraphic unit of the Bushveld Complex
mafic pegmatites	a suite of coarse-grained rocks that form discordant bodies within the layered sequence of the Bushveld Complex.
mamsl	metres above mean sea level
MCC	MCC (Pty) Ltd, a company registered and incorporated in South Africa. MCC is the appointed open pit mining contractor at the Tharisa Mine.
MDM Engineering	MDM Engineering (Pty) Ltd, a company registered and incorporated in South Africa. MDM is the appointed engineering contractor responsible for the construction of the new 300,000 tonne

Term	Description
	per month concentrator at the Tharisa Mine.
Measured Mineral Resource (SAMREC)	A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proved Mineral Reserve or to a Probable Mineral Reserve.
Metago	Metago Environmental Engineers (Pty) Ltd (now trading as SLR Consulting (Africa) (Proprietary) Limited), a company registered and incorporated in South Africa.
Merensky Reef	A pyroxenitic tabular layer or band within the Bushveld Complex containing economic concentrations of PGMs. The Merensky Reef is one of the principle PGM ore bodies within the Bushveld Complex and is mined extensively.
MG	Middle Group with reference to MG Chromitite Layers
MG Chromitite Layers	Group of five chromitite layers that are known in the lower and upper Critical Zone of the Bushveld Complex
MG0 Chromitite Layer	Specific chromitite layer contained within the MG Chromitite Layer package
MG1 Chromitite Layer	Specific chromitite layer contained within the MG Chromitite Layer package
MG2 Chromitite Layer	Specific chromitite layer contained within the MG Chromitite Layer package
MG3 Chromitite Layer	Specific chromitite layer contained within the MG Chromitite Layer package
MG4 Chromitite Layer	Specific chromitite layer contained within the MG Chromitite Layer package
MG4A Chromitite Layer	Specific chromitite layer contained within the MG Chromitite Layer package
MHSA	Mine Health and Safety Act, Act 29 of 1996
Competent Persons Report (SAMREC)	A report on the technical aspects of a project or mine prepared by a Competent Person (CP). The contents are determined by the nature/status of the project/mine being reported and may include appropriate for the level of study.
Mineral Reserve (SAMREC)	A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified. The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.
Mineral Resources (SAMREC)	A ‘Mineral Resource’ is a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.
Mining Right	A mining right is the permission granted by the State through the Department of Mineral Resources which gives you the authority to mine minerals within a certain area. A mining right may not exceed a period of 30 years.
MPRDA	The Mineral and Petroleum Resources Development Act 28 of 2002 of South Africa
MRMR	mining rock mass rating system
Mt	million tonnes
MVA	megavolt – ampere – a measure of required electrical power
NiS/MS	Specialist analytical technique used to determine the concentration of PGMs
norite	A coarse-grained, basic igneous rock consisting of essential plagioclase feldspar, orthopyroxene (hypersthene or bronzite), and clinopyroxene (augite), often with accessory ilmenite.

Term	Description
oz	fine ounce or troy ounce (31.1035g), used as a measure for the mass of precious metals
PGM	Platinum Group Metals, being platinum, palladium, rhodium, ruthenium, iridium, osmium, and, for the purposes of this report and in accordance with industry practice, gold.
pillar	Natural underground support system using unmined parts of the ore body
potholes	A geological feature frequently occurring in the Bushveld Complex in which one layer of the Bushveld Complex transgresses its footwall and forms a basin-shaped depression.
Pr.Sci.Nat.	Professional Natural Scientist in accordance with the rules of the South African Council for Natural Scientific Professionals which identifies him/her as a highly skilled professional with technical knowledge and competence.
Probable Mineral Reserve (SAMREC)	A Probable Mineral Reserve is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proved Mineral Reserve.
Prospecting Right	A prospecting right is a permit which allows a company or an individual to survey or investigate an area of land for the purpose of identifying an actual or probable mineral deposit.
Proved Mineral Reserve (SAMREC)	A Proved Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proved Mineral Reserve implies a high degree of confidence in the Modifying Factors.
Pyroxenite	refers to a relatively uncommon dark-coloured rock consisting chiefly of pyroxene; pyroxene is a type of rock containing sodium, calcium, magnesium, iron, titanium and aluminium combined with oxygen.
QA/QC programme	A programme of testing, used particularly for assays, to assist to confirm that the data used in a mineral resource estimation is reliable and comparable
RMR	The rock mass rating (RMR) system is a geomechanical classification system for rocks, developed by Z. T. Bieniawski between 1972 and 1973.[1]
RoM	Run of Mine
Rooikoppies 297JQ	The farm Rooikoppies 297, registration division JQ, located in the Bojanala Municipal District in the North West Province, South Africa.
Royalty Act	Mineral and Petroleum Resources Royalty Act, Act 28 of 2008.
RQD	Rock quality designation which is a description using geotechnical engineering principles which that determines the quality of rock that was recovered when taking a core sample.
SAG mill	Semi autogenous grinding mill
SAMREC Code (2016)	The South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves (The SAMREC Code) 2016 Edition
Sponsor	Macquarie Capital Securities Limited
tailings	that portion of the ore from which most of the valuable material has been removed by concentration and which is therefore low in value and rejected.
Tharisa	Tharisa plc formerly Tharisa Limited, a company registered and incorporated in the Republic of Cyprus.
Tharisa Mine	The existing chrome and PGM mine and processing operations, owned by Tharisa Minerals, located in the Bushveld Complex, which is situated in the Magisterial District of Rustenburg, North West Province, South Africa
Tharisa Minerals	Tharisa Minerals (Pty) Ltd, a company registered and incorporated in the Republic of South Africa, the developer and operator of the Tharisa Mine, held 74% by Tharisa.
The Company	Tharisa plc, formerly Tharisa Limited, a company registered and incorporated in the Republic of Cyprus.
tpa	tonnes per annum
tph	tonnes per hour
tpm	tonnes per month
TSF	Tailings Storage Facility
UCS	Uniaxial Compressive strength
UG2 Chromitite Layer	Upper Group 2 Chromitite Layer of the Bushveld Complex that is well known and typically

Term	Description
	contains PGMs in a concentration that is sufficient for economic extraction
Uniaxial Compressive Strength	Measure of the capacity of a material to withstand pushing forces
Ukwazi	Ukwazi Mining Solutions (Pty) Ltd, a company registered and incorporated in South Africa. Ukwazi is the appointed mine design and scheduling contractor at the Tharisa Mine.
US$	United States Dollar (currency)
variogram	The variogram is the key mathematical and graphical function in geostatistics as it is used to describe or fit a model of the spatial correlation of the observed phenomenon.
VAT	Value added tax
WTO	World Trade Organisation
ZAR	South African Rand (currency)

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The South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves (The SAMREC Code) 2016 Edition)

Site information and reports – existing operation

Appendix A

Authors Certificates

Authors Certificate Ken Lomberg (Senior Principal)

As the Lead Competent Person and Complier of the report entitled "**Tharisa Chrome and PGM Mine, South Africa Competent Persons Report"** with an effective date of 30 September 2016", I hereby state:-

1. My name is Kenneth Graham Lomberg and I am the Senior Principal for Coffey Mining South Africa Pty Ltd.
2. I am a practicing geologist and a registered with the South African Council for Natural Professionals.
3. I have a BSc (Hons) (Geology), BCom (Economics and Statistics) and an MEng (Mining Engineering)
4. I have been 31 years mining industry experience (especially platinum and gold). I have practiced my profession continuously since 1985. I have over 5 years of relevant experience having completed mineral resource estimations on various properties located on the Bushveld Complex hosting Magmatic Layered Intrusive style mineralization.

   I have the relevant experience of the type of deposit and of resource estimation that is the subject of this report. I have performed consultant work on various projects on the Bushveld Complex including Aurora, Kransplaats, Atok Mine, Mecklenburg, Smokey Hills, Kalplats, Garatau, Kennedy's Vale, Kalkfontein, Blue Ridge Mine, Eland Mine, WBJV, Palmietfontein, Stellite, Townlands and Tharisa. I have assisted with approximately 15 of the estimated 20 Junior Platinum Exploration and Mining Projects in South Africa. These assignments have ranged from listings documents, CPRs, ITRs, feasibility studies, NI43-101 compliant resource estimations and valuations.
5. I am a 'Competent Person' as defined in the SAMREC Code.
6. I am responsible for the Mineral Resources declaration and the compilation of the CPR.
7. I am not aware of any material fact or material change with respect to the subject matter of the Report that is not reflected in the Report, the omission of which would make the Report misleading.
8. I declare that this Report appropriately reflects the Competent Person's/author's view.
9. I am independent of Tharisa plc.
10. I have read the SAMREC Code (2016) and the Report has been prepared in accordance with the guidelines of the SAMREC Code.
11. At the effective date of the Report, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated at Johannesburg and 30 September 2016.

Mr Kenneth Lomberg
Senior Principal

Authors Certificate Jaco Lotheringen (Principal Mining Engineer)

As the Lead Competent Person and Complier of the report entitled "**Tharisa Chrome and PGM Mine, South Africa Competent Persons Report"** with an effective date of 30 September 2016", I hereby state:-

1. My name is Jaco Lotheringen and I am the Principal Mining Engineer at Ukwazi Mining Solutions (Pty) Ltd.
2. I am a practicing mining engineer registered with the South African Institute of Mining and Metallurgy (Registration Number 701237), and the Engineering Council of South Africa (Registration number 20030022).
3. I have a BEng (Mining) from the University of Pretoria and a Mine Managers certificate of competency.
4. I am a practicing mining engineer with 19 years' experience and the principal mining engineer and director of Ukwazi Mining Solutions. I have practiced my profession continuously since 1997. I have more than five years relevant experience, and I have completed Mineral Reserve estimations on various projects and commodities.
5. I am a 'Competent Person' as defined in the SAMREC Code.
6. I am responsible for the Mineral Reerves declaration and the compilation of related sections of the CPR .
7. I am not aware of any material fact or material change with respect to the subject matter of the Report that is not reflected in the Report, the omission of which would make the Report misleading.
8. I declare that this Report appropriately reflects the Competent Person's/author's view.
9. I am independent of Tharisa plc.
10. I have read the SAMREC Code (2016) and the Report has been prepared in accordance with the guidelines of the SAMREC Code.
11. At the effective date of the Report, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated at Pretoria and 30 September 2016.

Mr Jaco Lotheringen
Mining Engineer (Pr Eng)

Appendix B

SAMREC Code – Table 1

SAMREC Section	SAMREC Contents	Report Section
Section 1: Project Outline		
Property Description		
T1.1 (i)	Brief description of the scope of project (i.e. whether in preliminary sampling, advanced exploration, scoping, pre-feasibility, or feasibility phase, Life of Mine plan for an ongoing mining operation or closure).	Sect 3.1
T1.1 (ii)	Describe (noting any conditions that may affect possible prospecting/mining activities) topography, elevation, drainage and vegetation, the means and ease of access to the property, the proximity of the property to a population centre, and the nature of transport, the climate, known associated climatic risks and the length of the operating season and to the extent relevant to the mineral project, the sufficiency of surface rights for mining operations including the availability and sources of power, water, mining personnel, potential tailings storage areas, potential waste disposal areas, heap leach pad areas, and potential processing plant sites.	Sect 3.2, 3.3, 3.4, 3.5, 3.6
T1.1 (iii)	Specify the details of the personal inspector on the property by each CP or, if applicable, the reason why a personal inspection has not been completed.	Sect 1.2
Location		
T1.2 (i)	Description of location and map (country, province, and closest town/city, coordinate systems and ranges, etc.).	Sect 3.1, 5.2
T1.2 (ii)	Country Profile: describe information pertaining to the project host country that is pertinent to the project, including relevant applicable legislation, etc. Also provide a brief overview of the environmental and social context within which the project is located. Assess, at a high level, relevant technical, environmental, social, economic, political and other key risks.	Previous CPRS
T1.2 (iii)	Topo-cadastral map detail based on level of reporting.	Sect 3.4,
Adjacent Property		
T1.3 (i)	Discuss details of relevant adjacent properties If adjacent or nearby properties have an important bearing on the report, then their location and common mineralized structures should be included on the maps. Reference all information used from other sources.	Previous CPRS **Error! Reference source not found.**
History		
T1.4 (i)	State historical background to the project and adjacent areas concerned, including known results of previous exploration and mining activities (type, amount, quantity and development work), previous ownership and changes thereto.	Sect. 4**Error! Reference source not found.**
T1.4 (ii)	Present details of previous successes or failures should be referred to transparently with reasons why the project should now be considered potentially economic.	Sect. 4.4
T1.4 (iii)	Discuss known or existing historical Mineral Resource estimates and performance statistics to actual production	Sect. 4.3

SAMREC Section	SAMREC Contents	Report Section
	for past and current operations, including the reliability of these and how they relate to the SAMREC Code.	
Legal Aspects and Permitting		
T1.5 (i)	Discuss the nature of the issuer’s rights (e.g. prospecting and/or mining) and the right to use the surface of the properties to which these rights relate. The date of expiry and other relevant details must be disclosed.	Sect. 3.5, 3.6**Error! Reference source not found.**
T1.5 (ii)	Present the principal terms and conditions of all existing agreements, and details of those still to be obtained, (such as, but not limited to, concessions, partnerships, joint ventures, access rights, leases, historical and cultural sites, wilderness or national park and environmental settings, royalties, consents, permission, permits or authorizations).	Sect. 3.5, 3.6, Previous CPRs**Error! Reference source not found.**
T1.5 (iii)	Present the security of the tenure held at the time of reporting or that is reasonably expected to be granted in the future along with any known impediments to obtaining the right to operate in the area. State details of applications that have been made.	Sect. 3.5
T1.5 (iv)	Provide a statement of any legal proceedings for example land claims, that may have an influence on the rights to prospect or mine for minerals, or an appropriate negative statement.	Sect. 1.6**Error! Reference source not found.**
T1.5 (v)	Provide a statement should be provided to the effect that such governmental requirements as may be required have been approved.	Sect. 1.6
Royalties		
T1.6 (i)	Describe the royalties that are payable in respect of each property.	Sect. 1.6**Error! Reference source not found.**
Liabilities		
T1.7 (i)	Describe the liabilities, including rehabilitation guarantees, that are pertinent to the project. Provide a description of the rehabilitation liability, including, but not limited to, legislative requirements, assumptions and limitations.	Previous CPRs
Section 2: Geological Setting, Deposit and Mineralisation		
Geological Setting, Deposit and Mineralisation		
T2.1 (i)	Briefly describe the regional geology.	Sect. 5.1
T2.1 (ii)	Describe the project geology including deposit type, geological setting and style of mineralisation.	Sect. 5.1, 5.2, 5.3, 5.4
T2.1 (iii)	Discuss the geological model or concepts being applied in the investigation and on the basis of which the exploration program is planned. Describe the inferences made from this model.	Sect. 5.2, 5.3, 5.4
T2.1 (iv)	Discuss data density, distribution and reliability and whether the quality and quantity of information are sufficient to support statements, made or inferred, concerning the exploration target or deposit.	Sect. 6

SAMREC Section	SAMREC Contents	Report Section
T2.1 (v)	Discuss the significant minerals present in the deposit, their frequency, size and other characteristics. Includes minor and gangue minerals where these will have an effect on the processing steps. Indicate the variability of each important mineral within the deposit.	Sect. 5.2, 5.3**Error! Reference source not found.**
T2.1 (vi)	Describe the significant mineralised zones encountered on the property, including a summary of the surrounding rock types, relevant geological controls, and the length, width, depth, and continuity of the mineralisation, together with a description of the type, character, and distribution of the mineralisation	Sect. 5.2, 5.3
T2.1 (vii)	Confirm that reliable geological models and / or maps and cross sections that support interpretations exist.	Sect. 6, 7.1,7.3
Section 3: Exploration and Drilling, Sampling Techniques and Data		
Exploration		
T3.1 (i)	Describe the data acquisition or exploration techniques and the nature, level of detail, and confidence in the geological data used (i.e. geological observations, remote sensing results, stratigraphy, lithology, structure, alteration, mineralisation, hydrology, geophysical, geochemical, petrography, mineralogy, geochronology, bulk density, potential deleterious or contaminating substances, stratigraphy, geotechnical and rock characteristics, moisture content, bulk samples etc.). Data sets should include all relevant metadata, such as unique sample number, sample mass, collection date, spatial location etc.	Sect. 6, 7.1
T3.1 (ii)	Identify and comment on the primary data elements (observation and measurements) used for the project and describe the management and verification of these data or the database. This should describe the following relevant processes: acquisition (capture or transfer), validation, integration, control, storage, retrieval and backup processes. It is assumed that data are stored digitally but hand-printed tables with well organized data and information may also constitute a database.	Sect. 6, 7.1
T3.1 (iii)	Acknowledge and appraise data from other parties and reference all data and information used from other sources.	Sect. 7.1
T3.1 (iv)	Clearly distinguish between data/information from the property under discussion and that derived from surrounding properties	Sect. 6.3, 6.4
T3.1 (v)	Describe the survey methods, techniques and expected accuracies of data. Specify the grid system used.	Sect. 6.4
T3.1 (vi)	Discuss whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the estimation procedure(s) and classifications applied.	Sect. 6.4**Error! Reference source not found.**
T3.1 (vii)	Present representative models and/or maps and cross sections or other two or three dimensional illustrations of results should exist, showing location of samples, accurate drill-hole collar positions, down-hole surveys, exploration pits, underground workings, relevant geological data, etc	Sect. 7.3

SAMREC Section	SAMREC Contents	Report Section
T3.1 (viii)	Report the relationship between mineralisation widths and and intercept lengths and (particularly important) the geometry of the mineralisation with respect to the drill-hole angle. If it is not known and only the downhole lengths are reported, confirm it with a clear statement to this effect (e.g. ‘downhole length, true with not known’).	Sect. 7.4, 7.5,7.6
Drilling Technique		
T3.2 (i)	Present the type of drilling undertaken (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Banka, sonic, etc) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	Sect. 6.4
T3.2 (ii)	Describe whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.	Sect. 6.4, 6.5, 6.6
T3.2 (iii)	Describe whether logging is qualtative or quantative in nature, indicate if core photography (or costean, channel, etc) was undertaken.	Sect. 6.4
T3.2 (iv)	Present the total length and percentage of the relevant intersections logged.	Sect. 6.4
T3.2 (v)	Discuss the results of any downhole surveys of the drill-holes.	Sect. 6.4
Sampling Governance		
T3.3 (i)	Describe the nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.	Sect. 6.6, 6.7
T3.3 (ii)	Describe the sampling processes, including sub-sampling stages to maximize representivity of samples. This should include whether sample sizes are appropriate to the grain size of the material being sampled. Indicate whether sample compositing has been applied.	Sect. 6.6, 6.7
T3.3 (iii)	Appropriately describe each data set (e.g. geology, grade, density, quality, diamond breakage, geo-metallurgical characteristics etc.), sample type, sample-size selection and collection methods	Sect. 6.6, 6.7
T3.3 (iv)	If the geometry of the mineralisation with respect to the drill-hole angle is known, its nature should be reported. State whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. If the intersection angle is not known and only the downhole lengths are reported, that should be stated.	Sect. 5.2, 5.3,5.4
T3.3 (v)	Describe retention policy and storage of physical samples (e.g. core, sample reject, etc.)	Sect. 6.4
T3.3 (vi)	Describe the method of recording and assessing core and chip sample recoveries and results assessed, measures taken to maximise sample recovery and ensure representative nature of the samples and whether a	Sect. 6.4, 6.5

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	relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	
T3.3 (vii)	If a drill-core sample is taken, state whether it was split or sawn and whether quarter, half or full core was submitted for analysis. If a non-core sample, state whether the sample was riffled, tube sampled, rotary split etc. and whether it was sampled wet or dry.	Sect. 6.6, 6.7
Sampling Preparation and Analysis		
T3.4 (i)	Identify the laboratory(s) and state the accreditation status and Registration Number of the laboratory. Laboratories should be appropriately accredited. If not, this fact should be disclosed.	Sect. 6.6
T3.4 (ii)	Identify the analytical method. Discuss the nature, quality and appropriateness of the assaying and laboratory processes and procedures used and whether the technique is considered partial or total.	Sect. 6.6**Error! Reference source not found.**
T3.4 (iii)	Describe the process and method used for sample preparation, sub-sampling and size reduction, and likelihood of inadequate or non-representative samples (i.e. improper size reduction, contamination, screen sizes, granulometry, mass balance, etc.).	Sect. **Error! Reference source not found.**, 6.6**Error! Reference source not found.**
Sample Governance		
T3.5 (i)	Discuss the governance of the sampling campaign and process, to ensure quality and representivity of samples and data, such as sample recovery, high grading, selective losses or contamination, core/hole diameter, internal and external QA/QC, and any other factors that may have resulted in or identified sample bias.	Sect. 6.6, 6.7, 7.1**Error! Reference source not found.**
T3.5 (ii)	Describe the measures taken to ensure sample security and the Chain of Custody.	Sect. 6.6, 6.7**Error! Reference source not found.**
T3.5 (iii)	Describe the validation procedures used to ensure the integrity of the data, e.g. transcription, input or other errors, between its initial collection and its future use for modelling (e.g. geology, grade, density, etc.).	Sect. 7.1**Error! Reference source not found.**
T3.5 (iv)	Describe the audit process and frequency (including dates of these audits) and disclose any material risks identified.	No information on audits provided by client**Error! Reference source not found.**
Quality Control/Quality Assurance		
T3.6 (i)	Demonstrate that adequate field sampling process verification techniques (QA/QC) have been applied, e.g. the level of duplicates, blanks, reference material standards, process audits, analysis, etc. If indirect methods of	Sect. 6.6, 6.7**Error! Reference source not**

SAMREC Section	SAMREC Contents	Report Section
	measurement were used (e.g. geophysical methods), these should be described, with attention given to the confidence of interpretation.	**found.**
Bulk Density		
T3.7 (i)	Describe the method of bulk density determination with reference to the frequency of measurements, the size, nature and representativeness of the samples.	Sect. 7.2**Error! Reference source not found.**
T3.7 (ii)	If target tonnage ranges are reported then the preliminary estimates or basis of assumptions made for bulk density must be stated.	Not applicable**Error! Reference source not found.**
T3.7 (iii)	Discuss the representivity of the bulk density samples of the material for which a grade range is reported.	Sect. 7.2**Error! Reference source not found.**
T3.7 (iv)	Describe the adequacy of the method of bulk density determination for bulk material with special reference to accounting for void spaces (vugs, porosity etc.), moisture and differences between rock and alteration zones within the deposit.	Sect. 7.2**Error! Reference source not found.**
Bulk Sampling and/or Trial Mining		
T3.8 (i)	Indicate the location of individual samples (including map).	Not applicable
T3.8 (ii)	The Size of samples, spacing/density of samples recovered should be described. Whether sample sizes and distribution are appropriate to the grain size of the material being sampled should be described.	Not applicable **Error! Reference source not found.**
T3.8 (iii)	Describe the method of mining and treatment.	Sect. 8.3, 8.4
T3.8 (iv)	Indicate the degree to which the samples are representative of the various types and styles of mineralisation and the mineral deposit as a whole.	Sect. 7.3, 7.4
Section 4: Estimation and Reporting of Exploration Results and Mineral Resources		
Geological Model and Interpretation		
T4.1 (i)	Describe the geological model, construction technique and assumptions that forms the basis for the Exploration Results or Mineral Resource estimate. Discuss the sufficiency of data density to assure continuity of mineralisation and geology and provide an adequate basis for the estimation and classification procedures applied.	Sect. 7.3
T4.1 (ii)	Describe the nature, detail and reliability of geological information with which lithological, structural, mineralogical, alteration or other geological, geotechnical and geo-metallurgical characteristics were recorded.	Sect. 7.1, 7.3
T4.1 (iii)	Describe any obvious geological, mining, metallurgical, environmental, social, infrastructural, legal and economic	Sect. 5.1, 5.2,7.1, 7.3

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	factors that could have a significant effect on the prospects of any possible exploration target or deposit.	
T4.1 (iv)	Discuss all known geological data that could materially influence the estimated quantity and quality of the Mineral Resource.	Sect. 5.1, 5.2
T4.1 (v)	Discuss whether consideration was given to alternative interpretations or models and their possible effect (or potential risk) if any, on the Mineral Resource estimate.	Sect. 7.3
T4.1 (vi)	Discuss geological discounts (e.g. magnitude, per reef, domain, etc.), applied in the model, whether applied to mineralized and / or un-mineralized material (e.g. potholes, faults, dykes, etc).	Sect. 7.10, 7.13
Estimation and Modelling Techniques		
T4.2 (i)	Describe in detail the estimation techniques and assumptions used to determine the grade and tonnage ranges..	Sect. 7.9
T4.2 (ii)	Discuss the nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values (cutting or capping), compositing (including by length and/or density), domaining, sample spacing, estimation unit size (block size), selective mining units, interpolation parameters and maximum distance of extrapolation from data points.	Sect. 7.9
T4.2 (iii)	Describe assumptions and justification of correlations made between variables.	Not applicable
T4.2 (iv)	Provide details any relevant specialized computer program (software) used should be named (with the version number) together with the parameters used.	Sect. 7.9
T4.2 (v)	State the processes of checking and validation, the comparison of model information to sample data and use of reconciliation data, and whether the Mineral Resource estimate takes account of such information.	Sect. 7.9
T4.2 (vi)	Describe the assumptions made regarding the estimation of any co-products, by-products or deleterious elements.	Sect. 7.9
Reasonable and Realistic Prospects for Eventual Economic Extraction		
T4.3 (i)	Disclose and discuss the geological parameters. These would include (but not be limited to) volume / tonnage, grade and value / quality estimates, cut-off grades, strip ratios, upper- and lower- screen sizes.	Sect. 9.1, 9.2
T4.3 (ii)	Discuss and justify the engineering parameters. These would include mining method, processing, geotechnical, geohydraulic and metallurgical) parameters.	Sect. 9.1, 9.2
T4.3 (iii)	Disclose and discuss the infrastructural including, but not limited to, power, water, site-access.	Sect. 9.1, 9.2
T4.3 (iv)	Disclose and discuss the legal, governmental, permitting, statuary parameters.	Sect. 9.1, 9.2
T4.3 (v)	Disclose and discuss the environmental and social (or community) parameters.	Sect. 9.1, 9.2
T4.3 (vi)	Disclose and discuss the marketing parameters.	Sect. 9.1, 9.2
T4.3 (vii)	Disclose and discuss the economic assumptions and parameters. These factors will include, but not limited to,	Sect. 9.1, 9.2

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	commodity prices and potential capital and operating costs.	
T4.3 (viii)	Disclose and discuss any material risks.	Sect. 9.1, 9.2
T4.3 (ix)	Discuss the parameters used to support the concept of "eventual".	Sect. 9.1, 9.2
Classification Criteria		
T4.4 (i)	Describe the criteria and methods used as the basis for the classification of the Mineral Resources into varying confidence categories.	Sect. 7.12
Reporting		
T4.5 (i)	Discuss the reported low and high grades and widths together with their spatial location to avoid misleading the reporting of Exploration Results, Mineral Resources and Mineral Reserves.	Sect. 7.4, 7.5**Error! Reference source not found.**
T4.5 (ii)	Discuss whether the reported grades are regional averages or if they are selected individual samples taken from the property under discussion.	Sect. 7.4, 7.5**Error! Reference source not found.**
T4.5 (iii)	State assumptions regarding mining methods, infrastructure, metallurgy, environmental and social parameters. State and discuss where no mining-related assumptions have been made.	Sect. 9.1, 9.2**Error! Reference source not found.**
T4.5 (iv)	State the specific quantities and grades that are reported in ranges and/or widths and explain the basis for reporting.	Not applicable **Error! Reference source not found.**
T4.5 (v)	Present the detail, e.g. open pit, underground, residue stockpile, remnants, tailings, and existing pillars or other sources in the Mineral Resource Statement	Not applicable **Error! Reference source not found.**
T4.5 (vi)	Present a reconcilliation with any previous Mineral Resource estimate. Where appropriate report and comment on any historical trends (e.g. global bias).	Sect. 7.12**Error! Reference source not found.**
T4.5 (vii)	Present the defined reference point for the tonnages and grades reported as Mineral Resources. State the reference point if the point is where the run of mine material is delivered to the processing plant. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.	Sect. 7.12**Error! Reference source not found.**
T4.5 (viii)	If the CP is relying on a report, opinion, or statement of another expert who is not a CP, disclose the date, title, and author of the report, opinion, or statement, the qualifications of the other expert and why it is reasonable for	Sect. 7.12, 7.13, 9.1, 9.2**Error! Reference**

SAMREC Section	SAMREC Contents	Report Section
	the CP to rely on other expert, any significant risks, and any steps the CP took to verify the information provided.	**source not found.**
T4.5 (ix)	State the basis of equivalent formulae, if required	Not applicable
Section 5: Technical Studies		
Introduction (Level of Study)		
T5.1 (i)	For Mineral Resource state the level of study – whether scoping, prefeasibility, feasibility or ongoing Life of Mine. For Mineral State the level of study – whether prefeasibility, feasibility or ongoing Life of Mine. The Code requires that a study to at least a Pre-Feasibility level has been undertaken to convert Mineral Resource to Mineral Reserve. Such studies will have been carried out and will include a mine plan or production schedule that is technically achievable and economically viable, and that all Modifying Factors have been considered.	Sect. 8.1**Error! Reference source not found.**
T5.1 (ii)	Provide a summary table of the Modifying Factors used to convert the Mineral Resource to Mineral Reserve for Pre-Feasibility, Feasibility or ongoining Life-of-Mine studies.	Sect. 8.3, 8.4**Error! Reference source not found.**
Mine Design		
T5.2 (i)	State assumptions regarding mining methods and parameters when estimating Mineral Resources or explain where no mining assumptions have been made.	Sect. 8.3, 8.4**Error! Reference source not found.**
T5.2 (ii)	State and justify all modifying factors and assumptions made regarding mining methods, minimum mining dimensions (or pit shell) and internal and, if applicable, external) mining dilution and mining losses used for the techno-economic study and signed-off, such as mining method, mine design criteria, infrastructure, capacities, production schedule, mining efficiencies, grade control, geotechnical and hydrological considerations, closure plans, and personnel requirements.	Sect. 8.3, 8.4**Error! Reference source not found.**
T5.2 (iii)	State what resource models have been used in the study.	Sect. 8.3, 8.4**Error! Reference source not found.**
T5.2 (iv)	Explain the basis of (the adopted) cut-off grade(s) or quality parameters applied should be explained. Include metal equivalents if relevant	Not applicable **Error! Reference source not found.**
T5.2 (v)	Describe and justify of mining method(s) to be used.	Sect. 8.3, 8.4**Error! Reference source not found.**

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T5.2 (vi)	For open pit mines, include a discussion of pit slopes, slope stability, and strip ratio.	Sect. 8.2**Error! Reference source not found.**
T5.2 (vii)	For underground mines, include a discussion of mining method, geotechnical considerations, mine design characteristics, and ventilation/cooling requirements.	Sect. 8.2**Error! Reference source not found.**
T5.2 (viii)	Discuss the mining rate, equipment selected, grade control methods, geotechnical and hydrogeological considerations, health and safety of the workforce, staffing requirements, dilution, and recovery.	Sect. 8.3, 8.4**Error! Reference source not found.**
T5.2 (ix)	State the optimization methods used in planning, list of constraints (practicality, plant, access, exposed reserves, stripped reserves, bottlenecks, draw control).	Sect. 8.3, 8.4**Error! Reference source not found.**
Metallurgical and Testwork		
T5.3 (i)	Discuss the source of the sample and the techniques to obtain the sample, laboratory and metallurgical testing technique.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.3 (ii)	Explain the basis for assumptions or predictions regarding metallurgical amenability and any preliminary mineralogical testwork already carried out.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.3 (iii)	For Mineral Resources the CP should discuss the possible processing methods and any processing factors that could have a material effect on the likelihood of eventual economic extraction. Discuss the appropriateness of the processing methods to the style of mineralisation. For a mineral reserve describe and justify the processing method(s) to be used, equipment, plant capacity, efficiencies, and personnel requirements.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.3 (iv)	Discuss the nature, amount and representativeness of metallurgical testwork undertaken and the recovery factors used. A detailed flow sheet / diagram and a mass balance should exist ,especially for multi-product operations from which the saleable materials are priced for different chemical and physical characteristics.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.3 (v)	State what assumptions or allowances have been made for deleterious elements and the existence of any bulk-sample or pilot-scale testwork and the degree to which such samples are representative of the ore body as a whole.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.3 (vi)	State whether the metallurgical process is well-tested technology or novel in nature.	Sect. 9.1, 9.2**Error! Reference source not found.**

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Infrastructure		
T5.4 (i)	Comment on the current state of infrastructure or the ease with which the infrastructure can be provided or accessed.	Previous CPRs**Error! Reference source not found.**
T5.4 (ii)	report in sufficient detail to demonstrate that the necessary facilities have been allowed for (which may include, but not be limited to, processing plant, tailings dam, leaching facilities, waste dumps, road, rail or port facilities, water and power supply, offices, housing, security, resource sterilization testing etc.). Detailed maps showing locations of facilities should exist.	Previous CPRs **Error! Reference source not found.**
T5.4 (iii)	Provide a statement showing that all necessary logistics have been considered.	Previous CPRs **Error! Reference source not found.**
Environmental and Social		
T5.5 (i)	Confirm that the company holding the tenement has addressed the host country's environmental legal compliance requirements and any mandatory and/or voluntary standards or guidelines to which it subscribes.	Previous CPRs **Error! Reference source not found.**
T5.5 (ii)	Identify the necessary permits that will be required and their status. Where not yet obtained confirm that there is a reasonable bsis to believe that all permits required for the project will be obtained.	Previous CPRs **Error! Reference source not found.**
T5.5 (iii)	Identify and discuss any sensitive areas that may affect the project as well as any other environmental factors, including interested and affected parties and/or studies that could have a material effect on the likelihood of eventual economic extraction. Discuss possible means of mitigation.	Previous CPRs **Error! Reference source not found.**
T5.5 (iv)	Identify any legislated social management programmes that may be required and content and status of these.	Previous CPRs
T5.5 (v)	Outline and quantify the material socio-economics and cultural impacts that need to be mitigated, and their mitigation measures and, where appropriate the associated costs.	Previous CPRs **Error! Reference source not found.**
Market Studies and Economic Criteria		
T5.6 (i)	Describe the valuable and potentially valuable product(s) including suitability of products, co-products and by products to market.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.6 (ii)	Describe product to be sold, customer specifications, testing, and acceptance requirements. Discuss whether	Sect. 9.1, 9.2**Error!**

SAMREC Section	SAMREC Contents	Report Section
	there exists a ready market for the product and whether contracts for the sale of the product are in place or expected to be readily obtained. Price and volume forecasts and the basis for the forecast.	**Reference source not found.**
T5.6 (iii)	State, describe and justify all economic criteria that have been used for the study such as capital and operating costs, exchange rates, revenue / price curves, royalties, cut-off grades, reserve pay limits.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.6 (iv)	Provide a summary description, source and confidence of method used to estimate the commodity price/value profiles used for cut-off grade calculation, economic analysis and project valuation, including applicable taxes, inflation indices, discount rate and exchange rates.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.6 (v)	Present the details of the point of reference for the tonnages and grades reported as Mineral Reserves must be in respect of a point of reference (e.g. material delivered to the processing facility or saleable product(s)). It is important that, in any situation where the reference point is different, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.6 (vi)	Justify assumptions made concerning production cost including transportation, treatment, penalties, exchange rates, marketing and other costs. Allowances should be made for the content of deleterious elements and the cost of penalties.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.6 (vii)	State type, extent and condition of plant and equipment that is significant to the existing operation(s).	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.6 (viii)	Provide details of all environmental, social and labour costs should be considered.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.6 (ix)	Describe the valuable and potentially valuable product(s) including suitability of products, co-products and by products to market.	Sect. 9.1, 9.2**Error! Reference source not found.**
Risk Analysis		
T5.7 (i)	Report an assessment the technical, environmental, social, economic, political and other key risks to the project. Describe actions that will be taken to mitigate and/or manage the identified risks.	Sect. 10**Error! Reference source not found.**
Economic Analysis		
T5.8 (i)	Applicable to Mineral Resources and Mineral Reserves and should at the appropriate level (scoping study, PFS, FS, or ongoing LoM) provide an economic analysis.	Sect. 9.1, 9.2**Error! Reference source not**

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		found.
T5.8 (ii)	Cash Flow forecast on an annual basis using Mineral Reserves or Mineral Resources OR an annual production schedule for the life of the project.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.8 (iii)	A discussion of net present value (NPV), internal rate of return (IRR) and payback period of capital.	Sect. 9.1, 9.2**Error! Reference source not found.**
T5.8 (iv)	Sensitivity or other analysis using variants in commodity price, grade, capital and operating costs, or other significant parameters, as appropriate and discuss the impact of the results.	Sect. 9.1, 9.2**Error! Reference source not found.**
Section 6: Estimation and Reporting of Mineral Reserves		
Estimation and Modelling Techniques		
T6.1 (i)	Describe the Mineral Resource estimate used as a basis for the conversion to a Mineral Reserve.	Sect. 8.3, 8.4**Error! Reference source not found.**
T6.1 (ii)	Report on the Mineral Reserve Statement with sufficient detail indicating if the mining is open pit or underground plus the source and type of mineralisation, facies or ore body, surface dumps, stockpiles and all other sources.	Sect. 9.1, 9.2**Error! Reference source not found.**
T6.1 (iii)	Provide a reconciliation reporting historic reliability and reconciliation of the performance parameters, assumptions and modifying factors. This should include a comparison with the previous Reserve quantity and qualities, if available. Where appropriate, report and comment on any historic trends (e.g. global bias).	Sect. 9.1, 9.2**Error! Reference source not found.**
Classification Criteria		
T6.2 (i)	Describe and justify criteria and methods used as the basis for the classification of the Mineral Reserves into varying confidence categories, which should be based on the Mineral Resource category, and include consideration of the confidence in all the modifying factors.	Sect. 9.1, 9.2**Error! Reference source not found.**
Reporting		
T6.3 (i)	Discuss the proportion of Probable Mineral Reserves, which have been derived from Measured Mineral Resources (if any), including the reason(s) therefore.	Sect. 9.1, 9.2**Error! Reference source not found.**
T6.3 (iI)	Present details of, for example open pit, underground, residue stockpile, remnants, tailings, and existing pillars or	Sect. 9.1, 9.2**Error!**

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	other sources in respect of the Mineral Reserve statement.	**Reference source not found.**
T6.3 (iii)	Present the details of the defined reference point for the Mineral Reserve. State where the reference point is the point where the run of mine material is delivered to the processing plant. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported. The tonnages and grades reported for Mineral Reserves should state clearly whether these are in respect of material delivered to the plant or after recovery.	Sect 9.1, 9.2**Error! Reference source not found.**
T6.3 (iv)	Presenta a reconciliation with the previous Mineral Reserve estimates. Where appropriate, report and comment on any historic trends (e.g. global bias).	Sect. 9.1, 9.2**Error! Reference source not found.**
T6.3 (v)	Only Measured and Indicated Mineral Resources can be considered for inclusion in the Mineral Reserve.	Sect. 9.1, 9.2**Error! Reference source not found.**
T6.3 (vi)	State whether the Mineral Resources are inclusive or exclusive of Mineral Reserves.	Sect. 9.1, 9.2**Error! Reference source not found.**
Section 7: Audits and Reviews		
Audits and Reviews		
T7.1 (i)	State type of review/audit (e.g. independent, external), area (e.g. laboratory, drilling, data, environmental compliance, etc), date and name of the reviewer(s) together with their recognized professional qualifications.	No Audit details Disclosed**Error! Reference source not found.**
T7.1 (ii)	Disclose the conclusions of relevant audits or reviews. Note where significant deficiencies and remedial actions are required.	No Audit details Disclosed **Error! Reference source not found.**
Section 8: Other Relevant Information		
Environmental		
T8.1 (1)	Discuss all other relevant and material information not discussed elsewhere.	Previous CPRs **Error! Reference source not found.**
Section 9: Qualifications of Competent Person(s) and Other Key Staff. Date and Signature Page		

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Qualifications		
T9.1 (i)	State the full name, address, registration number and name of the professional body or RPO for all CPs. State the relevant experience of the CP(s) and other key technical staff who prepared and are responsible for the Public Report, of which he/she is a member, and relevant experience, together with other key technical staff who prepared and are responsible for the Public Report.	Appendix A, 1.4, 1.5
T9.1 (ii)	State the CPs relationship to the issuer of the report.	Appendix A, 1.5
T9.1 (iii)	Provide the Certificate of the CP (Appendix 2) including the date of sign-off and the effective date, in the Public Report.	Appendix A