THE MSA GROUP

Specialist Consultants to the Mining Industry

MOKOPANE VANADIUM PROJECT

PRE-FEASIBILITY STUDY

PREPARED BY THE MSA GROUP (PTY) LTD FOR:
Bushveld Minerals Limited

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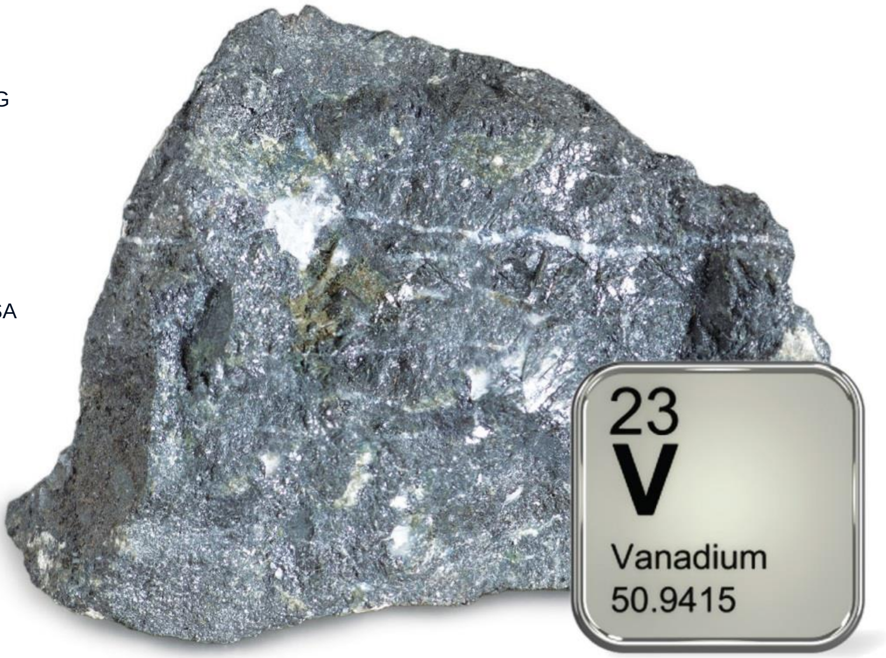

IMPORTANT NOTICE

This notice is an integral component of the Mokopane 2015 Pre-Feasibility Study (the PFS) and should be read in its entirety and must accompany every copy made of the PFS.

As detailed in Sections 1 and 2, the PFS has been prepared for Bushveld Minerals Limited (BML) by The MSA Group (Pty) Ltd (MSA), WorleyParsonsRSA (WP), Trueground Consulting (Trueground), Hatch Goba (Pty) Ltd (Hatch), Epoch Resources (Pty) Ltd (Epoch), Digby Wells Environmental (Digby Wells), BML and Hindsight Financial and Commercial Solutions (Pty) Ltd (Hindsight)) as the PFS Contributors.

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The conclusions and estimates stated in the PFS are to the accuracy stated in the PFS only and rely on assumptions stated in the PFS. The results of further work may indicate that the conclusions, estimates and assumptions in the 2015 PFS would need to be revised or reviewed.

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

The PFS should be construed in light of the methodology, procedures and techniques used to prepare the PFS. This report is intended to be read as a whole and sections or parts of the PFS should not be read or relied upon out of context.

The PFS is intended to be used by BML, subject to the terms and conditions of its contract with the PFS Contributors. Recognizing that BML has legal and regulatory obligations, the PFS Contributors have consented to the filing of the PFS with AIM.

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1 SUMMARY

This Technical Report has been prepared by the MSA Group (Pty) Ltd, WorleyParsonsRSA, TrueGround Consulting, Hatch Goba (Pty) Ltd, Epoch Resources (Pty) Ltd, Digby Wells Environmental, and Hindsight Financial and Commercial Solutions (Pty) Ltd (collectively the Consultants) on behalf of Bushveld Minerals Limited (BML). The Consultants were commissioned to prepare a Technical Report for a Pre-Feasibility Study (PFS) on the Mokopane Vanadium Project (the Project) located in the Limpopo Province, Republic of South Africa. BML is an exploration and project development company listed on AIM, the London Stock Exchange's international market for smaller growing companies, under the symbol "BMN".

The Project is 64 % owned by BML through its wholly owned subsidiary Bushveld Resources Limited (BRL), which in turn holds 64 % of Pamish Investments No 39 (Pty) Ltd (Pamish).

Pamish hold a Prospecting Right (LP 95 PR) for an area including the following farm portions: RE of Vogelstruisfontein 765LR, RE of Vliegekraal 783LR, RE of Vriesland 781LR, RE of Schoonoord 786LR, RE and Portions 1, 2, 3, 4, 5 and 6 of Bellevue 808LR. The Prospecting Right allowed for the prospecting of iron ore, vanadium, titanium and other minerals. The combined area of the PR area is 10,072.795 hectares (ha).

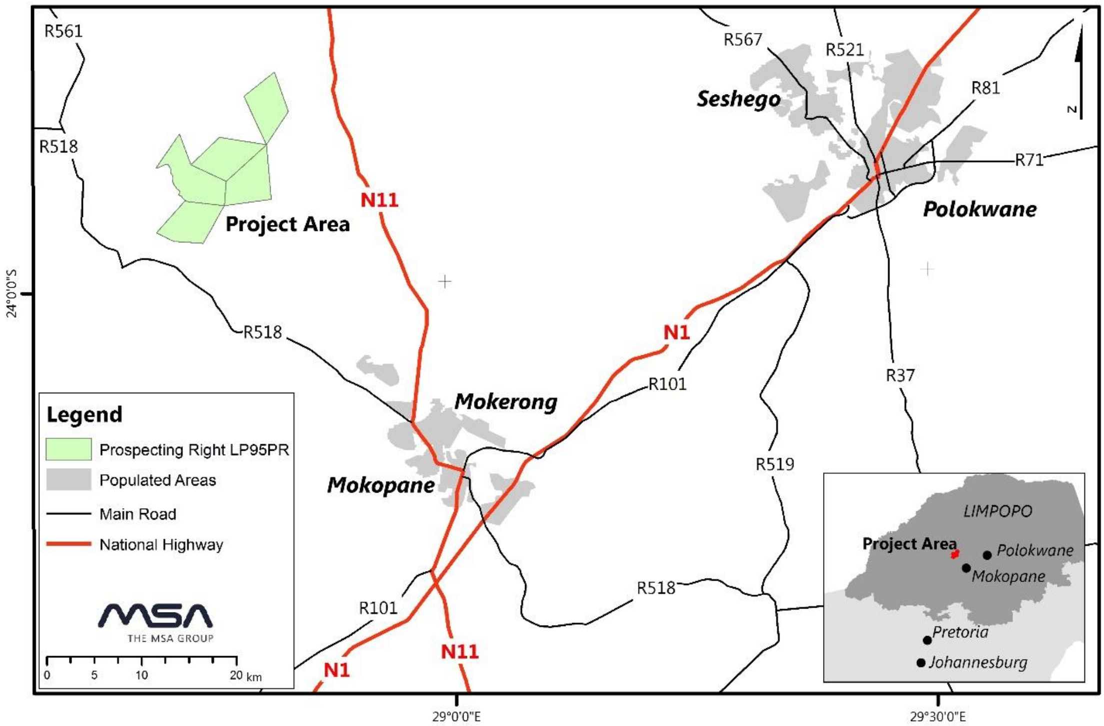
BML proposes to develop the Project using an open pit mining method to supply the concentrator and salt roast plant. The ore will be crushed and milled on site, concentrated using magnetic separation and beneficiated in a salt roast plant to produce vanadium pentoxide (V_2O_5) in flakes. The salt roast plant will be a separate entity held in the name of SaltCo (Pty) Ltd; a new company to be registered.

Unless specified otherwise, all costings have been undertaken at an accuracy level of ± 25 %.

1.1 Property Description and Location

The Project is located on the central portion of the Northern Limb of the Bushveld Complex in the Limpopo Province of South Africa. It is located within the Prospecting Right LP95PR, issued in accordance with the , in the Mokopane District, Mogalakwena Local Municipality, Waterberg District Municipality, approximately 65 km west of Polokwane and 45 km northwest of Mokopane in the Mokopane District, Limpopo Province, Republic of South Africa (Figure 1-1).

Figure 1-1
Regional locality map of the Project Area



Primary access to the Project is via a tarred road linking Mokopane and the village of Bakenberg and secondarily through a tarred road (main access to Vogelstruisfontein 765LR) connecting with the N11 to Mokopane. This access is enhanced by a good network of secondary gravel roads and tracks that exist within the area (Figure 1-1 and Figure 1-2).

The Project Area is at an elevation of about 1,000 m above sea level and has a semi-arid climate with a summer rainy season and a pronounced dry spell during winter. Average annual rainfall is 495 mm, with December and January being the wettest months and July the driest.

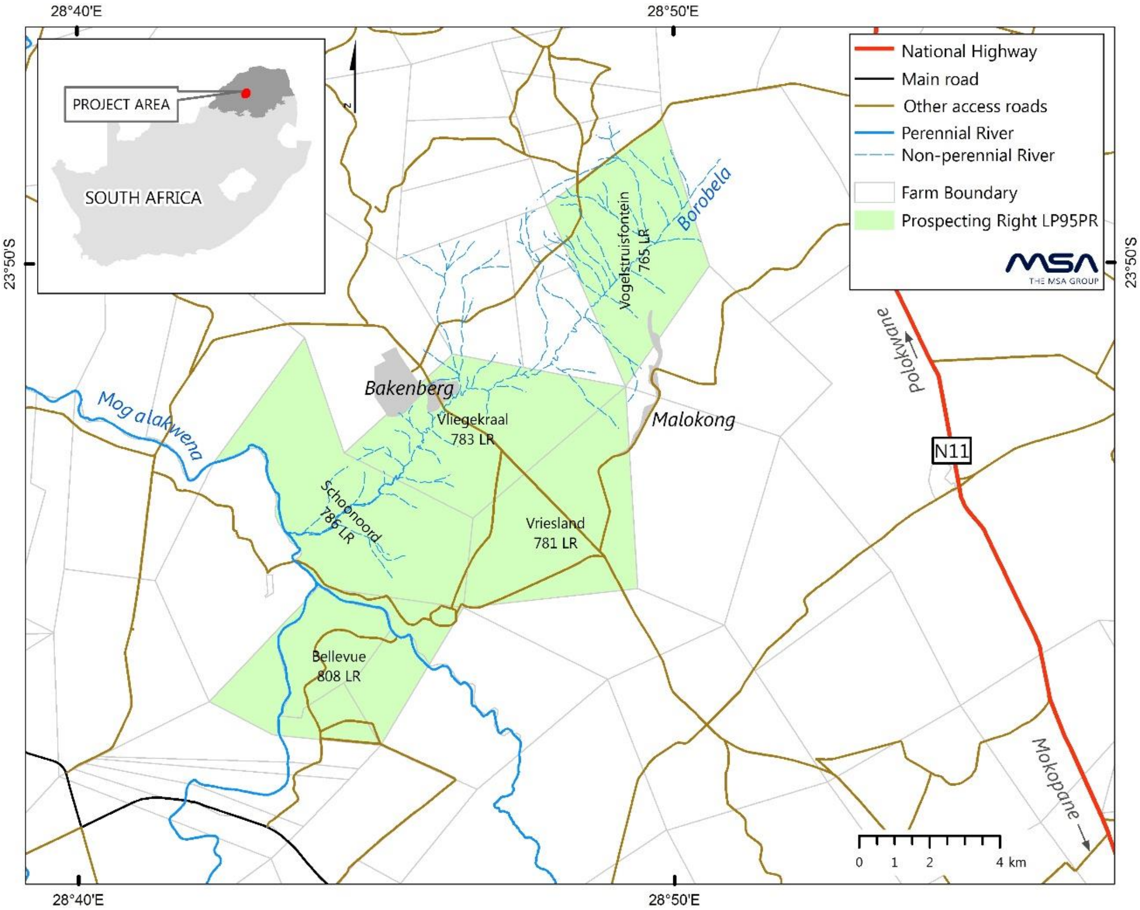
The general area is characterised by flat lying to gently sloping ground punctuated by a series of northerly trending hills in the east and the higher plateau of Bushveld granite and diabase-capped hills to the west. Drainage in the Project Area is from the north-northeast to south-southeast via the seasonal Borobela River and its weak tributary network.

The area is classified as Makhado Sweet Bushveld. The hilly areas are bush covered whilst the flat lying areas support a mixture of bush and cultivated fields. Soil cover varies from thin brown residual soils with bedrock outcrop in the east, thick (>5 m) residual and transported "black turf" soils along the broad valley of the Borobela River in the central portions, and red residual soils in the west.

Land use is dominated by traditional grazing with summer dryland subsistence agriculture, and the land is generally in a degraded condition.

Figure 1-2

Location of the five farms comprising the Pamish Prospecting Right



1.2 Mineral Tenure

Pamish holds a Prospecting Right, LP95PR, which covers the farms Vriesland 781LR, Vliegekraal 783LR, Vogelstruisfontein 765LR, Schoonoord 786LR and Bellevue 808LR, and which was granted for iron ore, vanadium, titanium and all minerals that may be found in intimate association with the latter, as well as nickel, copper, cobalt, chrome, platinum group metals and gold. Phosphate ore was added in February 2014.

The status of the Prospecting Right is based on information and copies of documents provided by BML. These include a legal opinion confirming that Pamish Investments No 39 (Pty) Ltd (Pamish) remains the Prospecting Right holder for LP95PR beyond the expiry date (15 March 2015) and during the processing period of the Mining Right application (which was submitted on 13 March 2015), until such time as the Right may be approved or not by the Department of Mineral Resources (DMR) (Table 1-1). The Consultants have not independently verified, nor are they qualified to verify, the legal status of the Prospecting Right and assume that the Mokopane Vanadium Project will prove lawfully accessible for further exploitation.

Table 1-1
Details of the Prospecting Rights pertaining to the Mokopane Vanadium Project

Company	BRL Interest (%)	Farm Names	Minerals	Area (ha)	PR No.	Status
Pamish Investments No 39 (Pty) Ltd (Pamish)	64 %	Vogelstruisfontein 765LR, Vriesland 781LR, Vliegekraal 783LR, Schoonoord 786LR and Bellevue 808LR (the latter two farms were added in February 2014)	Iron Ore, Vanadium, Titanium and all minerals that may be found in intimate association with the latter, Platinum Group Metals, Gold, Cobalt, Copper, Nickel and Chrome. Phosphate Ore was added in February 2014	10072.7949	LP95PR	The Prospecting Right was renewed on 30 May 2011 for 3 years. An application to include the two additional farms Schoonoord 786 LR and Bellevue 808 LR, was approved in January 2013, executed on 19 February 2014 and registered with the Title Deeds office on 27 October 2014. The Prospecting Right expired on 15 March 2015 and Pamish submitted an application for a Mining Right on 13 March 2015.

Whilst the Consultants made sufficient inquiry about the legal status of the Rights, this does not constitute a legal opinion. However, the Consultants are satisfied that the Rights and the corporate structure presented is a fair reflection of the current holdings.

1.3 Geology

The Project Area is situated within the Northern Limb of the Bushveld Complex (BC) and covers the upper portion of the Main Zone (MZ) and the entire Upper Zone (UZ) of the Rustenburg Layered Suite (RLS). The UZ is approximately 1,250 m thick and dips gently (15° to 25°) to the west. The UZ is characterised by the presence of vanadiferous titano-magnetite (VTM) layers hosted predominantly by VTM-enriched gabbro, gabbronorite, leuconorite, anorthosite and olivine diorite. The VTM layers include disseminated, semi-massive and massive VTM intervals of variable thicknesses and variable proportions of oxide (Ti-magnetite) and silicate minerals (feldspar, pyroxene and olivine).

The RLS is the World's largest and economically most important layered complex and is known for the remarkable geological and geochemical continuity of the magmatic stratigraphy. In common with other layered intrusions, such as the Great Dyke in Zimbabwe (Wilson, 1997), Molopo Farms Complex in Botswana (Reichhardt, 1994) and the Stillwater Complex in the USA (Irvine *et al.*, 1983), the intrusive ultramafic to mafic magma has undergone a differentiation process which has resulted in the formation of magnesium-, chromium-, nickel- and precious metal-rich units in the lower portion of the RLS with iron-, titanium-, vanadium- and phosphorus-rich layers in the upper portion.

The UZ consists of numerous cyclic units of alternating and well-layered rocks and is subdivided into three subzones:

- Subzone A is dominated by gabbroic rocks,
- Subzone B is dominated by the presence of modal olivine in the rocks; and
- Subzone C is dominated by the presence of modal apatite in the rocks.

The rocks of the RLS show remarkable continuity and individual layers can generally be traced along strike for tens of kilometres.

Since 2010, exploration by BML focussed on the Main Magnetite Layer (MML) and the stratigraphically higher semi-massive to massive Ti-magnetite layers N, O, P and Q. The P and Q VTM layers together with their enclosing gabbroic host rocks, which can contain considerable quantities of disseminated VTM, have been collectively termed the P-Q Zone.

The Project is based on the three mineralised layers associated with the MML which is part of the UZ of the BC. These are the MML, the MML Hanging Wall (MML HW) and the AB Zone located in the footwall of the MML.

The MML mineralised zone occurs near the base of the UZ and consists of an upper VTM-rich interval (MAG3) which is separated from a lower VTM-rich interval (MAG4) by a VTM-poorer leucogabbronorite "parting". The MML was intersected during the 2010 to 2013 exploration programme in 13 vertical drillholes, and has an average true thickness of 9.8 m, including the VTM-poor parting, and dips between 18° and 24° to the west. The MAG3 ranges between 2.59 m and 7.65 m and averages 4.09 m in true thickness. The MAG4 ranges between 2.48 m and 6.30 m and averages 3.59 m in true thickness. The parting ranges from 0.93 m to 4.06 m and averages 2.16 m in true thickness.

The MML HW comprises fourteen continuous layers defined by geological logging and VTM content, consisting of alternating layers of relatively high-grade semi-massive to massive VTM, lower-grade gabbronorite and barren anorthosite. These fourteen layers of the MML HW package are conformable with the MML and have a combined average true thickness of approximately 72 m.

The AB Zone represents the stratigraphically lowest accumulation of abundant VTM and occurs approximately 100 m below the MML near the base of the UZ of the RLS. The AB Zone consists of a relatively higher-grade upper and lower layer of strongly disseminated VTM, separated by a lower grade parting. The layers of the AB Zone have an average dip of 21° to the west with a combined average true thickness of approximately 9.3 m.

The Scoping Study undertaken in July 2014 and this Pre-Feasibility Study have been based on the MML only. It is anticipated that the lower grade MML HW layers in the immediate hanging wall of the MML which forms part of the stripping during the mining of the MML, will be stock piled for potential future processing.

1.4 Previous Work

Prior to BML's systematic drilling programmes, the Project Area had not been explored for its Ti-magnetite potential but was covered by a regional geochemical soil sampling and geological mapping campaign by the South African Council for Geoscience (CGS). The latter work was published in 1985 at 1:250,000 scale as the 2328 Pietersburg Geological Series map. The soil sampling was conducted at 1 km intervals and the samples were analysed by XRF and ICP-MS for over 40 elements including Fe_2O_3, V, TiO_2, Cu and Ni. Significant vanadium and titanium anomalies occur and generally coincide with areas mapped as the UZ.

A regional aeromagnetic and radiometric survey was conducted in the 1990's and processed by the CGS. The data show northerly-trending magnetic zones which have been correlated with the two most prominent VTM-rich stratigraphic units, namely the Main Magnetite Group and the N-Q Zone comprising the N, O, P and Q Ti-magnetite layers.

A stratigraphic drillhole BV-1 was drilled by the CGS in 1991 on the farm Bellevue 808LR, some 2 km south-west of the Project Area. The 2,950 m deep hole covered the entire Upper Zone stratigraphy and intersected 32 discrete layers of VTM-rich rocks (>20 % opaque minerals) ranging in thickness between 7 cm and 13 m (Ashwal *et al.*, 2005). Most prominent are the uppermost semi-massive Ti-magnetite layer (Q layer) which has a thickness of 13 m and an approximately 8 m thick vanadium-rich layer with variable Ti-magnetite content. The latter is some 175 m above the base of the UZ and can be correlated with the MML. The occurrence of the two most prominent Ti-magnetite layers in drillhole BV-1 at depths of approximately 600 m and 1,400 m illustrates the remarkable spatial continuity of these layers.

The N-Q Layers in the Project Area had not been identified prior to BML's exploration activities. The MML was only partially portrayed on the maps existing at the time and had been interpreted from exposures in isolated outcrops. No historic Mineral Resource Estimates (MRE) had been carried out in the Project Area.

1.5 Previous Mineral Resource Estimates

1.5.1 2010 and 2011

A total of 4,234.06 m were drilled in 17 diamond drillholes during the 2010-2011 drilling campaigns on the farms Vliegekraal 783LR and Vriesland 781LR. This included four drillholes totalling 902.02 m on the MML and 10 drillholes totalling 2,583.77 m on the P-Q Zone. The stratigraphically lower N and O layers were excluded from the MRE.

The results from these 17 drillholes, together with information about the Project, were presented in a report entitled "JORC Competent Person's Report and MRE for the Mokopane Fe-V-Ti Project covering the farms Vriesland 781LR, Vliegekraal 783LR, Malokong 784LR and Vogelstruisfontein 765LR near Mokopane, Limpopo Province, South Africa", dated 25 November 2011. The following Mineral Resources were reported for the MML (Table 1-2) in November 2011 in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, The JORC Code 2004 Edition (JORC 2004). A 40 % Fe_2O_3 cut-off was used for the MML.

Table 1-2
MML Inferred Mineral Resources, <100 m depth at 40 % Fe_2O_3 cut-off, as at 25 Nov 2011

Cut Off Fe_2O_3 %	Million Tonnes	Density t/m^3	Fe %	Fe_2O_3 %	P_2O_5 %	TiO_2 %	V_2O_5 %	SiO_2 %	Al_2O_3 %
40	66.21	3.83	37.1	53.1	0.01	9.2	1.24	17.9	11.1

1.5.2 2012

1.5.2.1 General

In terms of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, The JORC Code 2012 Edition (JORC 2012), MRE's may include not only mineralisation that has the potential to be economically viable using currently practised mining and extraction technology, but also mineralisation that in the opinion of the Competent Person has reasonable potential to become economically viable with advances in mining and extraction technology within the foreseeable future. Mineralisation within the MML in the Project Area appears to be fairly continuous to depths well below those currently considered to be of economic viability. Cognisance has been taken of the substantial mineralisation that is likely to be present at depth; and depth cut-offs have been imposed based on simplistic bulk stripping ratios that, while considerably beyond the limits of current commercial mining practice, might conceivably become viable in the future. The estimates do not, however, take any account of the additional costs that might prove to be associated with the extraction of saleable metals from the mineralogically complex Ti-magnetite material (relative to more conventional iron ore deposits), irrespective of the metallurgical test work that has been undertaken to date.

The results of the 2012 drilling campaign were summarised in a report entitled "JORC Competent Person's Report and Mineral Resource Estimate for the Mokopane Fe-V-Ti Project, Limpopo Province, South Africa", dated 12 April 2013.

1.5.2.2 MML

During 2012, 13 drillholes totalling 927.49 m were drilled, of which nine intersected the MML, one hole was stopped approximately 100 m above the MML and three holes were drilled into the footwall to the MML.

For the March 2013 MRE, the MML was subdivided into the two semi-massive to massive VTM layers, namely MAG3 and MAG4 and the VTM-poor, feldspar-rich parting (the "Parting") between MAG3 and MAG4. The Parting has a low abundance of VTM, ranging between 5 % and 30 %, whereas MAG3 and MAG4 contain between 35 % and 90 % VTM. As the Parting has average Fe_2O_3 grades below 40 %, it was not regarded as a Mineral Resource.

The drilling increased the confidence of the shallow mineralisation such that an Indicated Mineral Resource was declared for the MML from surface to a vertical depth of 120 m. The Mineral Resource for the MML on the farms Vriesland 781LR and Vliegekraal 783LR (Table 1-3) was reported at a cut-off-grade of 40 % Fe_2O_3 and was prepared in accordance with JORC 2012. The grades and tonnages for the Parting are shown in Table 1-4.

Table 1-3
MML Indicated Mineral Resource, <120 m vertical depth, as at 20 March 2013

Layer Name	Thickness (m)	Tonnes (million)	Density (t/m³)	Fe (%)	Fe_2O_3 (%)	Fe Metal (million tonnes)	TiO_2 %	V_2O_5 %	SiO_2 %	Al_2O_3 %	P_2O_5 %	S %
MAG3	4.09	27.50	4.08	45.5	65.1	12.51	10.0	1.50	10.6	7.8	0.01	0.12
MAG4	3.59	24.31	4.00	43.9	62.7	10.66	9.3	1.46	11.8	8.9	0.01	0.24
Total	**7.68**	**51.81**	**4.04**	**44.7**	**64.0**	**23.17**	**9.7**	**1.48**	**11.2**	**8.3**	**0.01**	**0.18**

Note: *Mineral Resource is reported at a 40 % Fe_2O_3 cut-off.*

Table 1-4
Grade and Tonnage* for MML Parting, <120 m vertical depth, as at 20 March 2013

Layer Name	Thickness (m)	Tonnes (million)	Density (t/m³)	Fe (%)	Fe_2O_3 (%)	TiO_2 %	V_2O_5 %	SiO_2 %	Al_2O_3 %	P_2O_5 %	S %
PARTING	2.16	11.43	3.16	20.9	29.9	3.5	0.58	34.5	19.0	0.01	0.17

Note: ** The MML Parting does not constitute a Mineral Resource as the mineralization is below the cut-off grade of 40 % Fe_2O_3.*

1.6 Mineral Resource Estimate (2013 and 2014)

Since November 2013, the primary focus of the Project shifted to vanadium, as opposed to iron. As a result, the MRE presented for the MML and MML Hanging Wall was reported according to a vanadium cut-off grade of ≥0.3 % V_2O_5 rather than Fe_2O_3 which was used in previous MREs for the MML.

The following section presents the MRE conducted by The MSA Group during 2014. This is the current Mineral Resource estimate for the MML and MML Hanging Wall for the Project.

1.6.1 MML and MML Hanging Wall

The MRE was based on additional sampling and assay data from the MML HW succession collected from the 17 drillholes drilled on the MML between 2010 and 2012.

The MML HW was subdivided into fourteen continuous layers defined by geological logging and VTM content, consisting of alternating layers of relatively high-grade semi-massive to massive VTM, lower-grade gabbronorite and barren anorthosite. The main target horizons were the VTM layers, particularly those averaging ≥0.30 % V_2O_5. These fourteen layers of the MML HW package are conformable with the MML and have a combined average true thickness of approximately 72 m.

Fewer drillhole intersections were available on the MML HW layers relative to the MML. Consequently, the MML HW Mineral Resource is classified as an Inferred Mineral Resource while the MML is classified as an Indicated Mineral Resource. The MML and MML HW Mineral Resources have been prepared in accordance with JORC 2012, and are both reported at a cut-off grade of 0.30 % V_2O_5 and to a vertical depth of 120 m below surface.

The MML HW Mineral Resource forms part of the Project and it is expected that the MML HW will be co-extracted with the MML. Although grades are generally lower in the MML HW layers relative to the MML, the cost of mining the MML HW is expected to be minimal as much of the mining cost will be attributed to the stripping of MML HW required to access the MML.

Reasonable Prospects for Eventual Economic Extraction for the MML HW VTM layers are dependent on its co-extraction with the MML, and it is unlikely that the MML HW VTM layers could be extracted economically as a standalone project.

The Mineral Resource for the MML and MML HW on the farms Vriesland 781LR and Vliegekraal 783LR is presented in Table 1-5.

A further 31 diamond drillholes (totalling 1,831.61 m) were drilled in 2014, of which 22 holes (1,295.82 m) targeted the MML and MML HW, and nine holes (535.79 m) targeted a disseminated VTM mineralisation (A-B Zone) approximately 180 m stratigraphically below the MML. Sampling and assaying of the 31 holes had not been completed at the time of reporting.

Table 1-5

MML and MML HW Mineral Resources at a 0.30% V_2O_5 cut-off, ≤120 m depth, as at 6 Nov 2014

Layer Name	Mineral Resource Category	Width (m)	Tonnes (Mt^1)	Density (t/m^3)	V_2O_5 (%)	Fe (%)	Fe_2O_3 (%)	TiO_2 (%)	SiO_2* (%)	Al_2O_3* (%)	P_2O_5* (%)	S* (%)	V_2O_5 (kt^2)	Fe (Mt^1)
UG-C	Inferred	4.04	31.8	3.48	0.64	25.7	36.7	5.9	30.2	15.4	0.01	0.12	202.8	8.2
UG-A	Inferred	1.64	12.7	3.31	0.59	23.2	33.1	5.3	32.5	17.5	0.01	0.01	75.6	3.0
UMG1	Inferred	3.24	25.5	3.30	0.59	22.9	32.7	5.4	32.6	17.6	0.01	0.01	150.4	5.8
UMG2	Inferred	2.03	15.7	3.40	0.69	25.9	37.0	6.2	29.4	16.7	0.01	0.01	107.7	4.1
MAG1 HW GAB**	Inferred	17.53	72.3	3.02	0.31	13.1	18.8	2.9	42.0	21.9	0.01	0.12	223.3	9.5
MAG1	Inferred	1.31	12.0	3.96	1.07	40.0	57.1	9.7	15.6	10.8	0.01	0.06	128.7	4.8
MAG2	Inferred	1.10	9.2	3.57	0.83	30.2	43.1	7.2	25.1	15.1	0.01	0.06	76.3	2.8
MML HW	Inferred	5.89	42.3	3.01	0.32	13.4	19.2	2.5	42.2	21.6	0.02	0.11	136.0	5.7
Total	**Inferred**	36.77	221.5	3.21	0.50	19.8	28.3	4.4	35.7	18.9	0.01	0.08	1,100.8	43.8
MAG3	Indicated	4.09	27.5	4.08	1.50	45.5	65.1	10.0	10.6	7.8	0.01	0.12	412.5	12.5
PART	Indicated	2.16	11.4	3.16	0.58	20.9	29.9	3.5	34.5	19.0	0.01	0.17	66.3	2.4
MAG4	Indicated	3.59	24.3	4.00	1.46	43.9	62.7	9.3	11.8	8.9	0.01	0.24	354.9	10.7
Total	**Indicated**	9.84	63.2	3.85	1.32	40.4	57.8	8.6	15.4	10.2	0.01	0.18	833.7	25.6
Total Mineral **Resources** [3]		**46.61**	**284.8**	**3.33**	**0.68**	**24.4**	**34.8**	**5.4**	**31.2**	**17.0**	**0.01**	**0.10**	**1,934.5**	**69.4**

Note: *[1]Mt = million tonnes; [2]kt = thousand tonnes; [3]Rounding may cause computational errors*

**Included for informative purposes only, no value will be derived from these materials*

***A 0.30 % V_2O_5 cut-off has been applied laterally across this layer such than only material > 0.30 % V_2O_5 is included in the tonnage listed in this table.*

1.6.2 The AB Zone

A Mineral Resource Estimate, undertaken on the AB Zone during 2015 in accordance with JORC 2012, estimated a 12.5 Mt Inferred Mineral Resource. Davis Tube Tests (DTT) were undertaken on the samples from this Inferred Mineral Resource. Results indicated concentrate grades of between 2.01 % and 2.65 % V_2O_5 with limited variability, an average concentrate grade of 2.21 % V_2O_5, and vanadium recoveries of up 97.82 %. However, this AB Zone Mineral Resource, summarised in Table 1-6, does not form part of the scope of the PFS.

Table 1-6
AB Zone Mineral Resources estimate at a 0.3 % V_2O_5 cut-off, ≤ 120 m vertical depth, as at 16 July 2015

Mineral Resource Category	Width (m)	Tonnes (Mt)	Density (t/m3)	V_2O_5 (%)	Fe_2O_3 (%)	TiO_2 (%)
Inferred	9.30	12.5	3.18	0.70	27.9	4.2

1.7 Ore Reserve Estimate

The following modifying factors and mine design criteria and assumptions were applied to the Mineral Resource in order to determine an Ore Reserve for the Project:

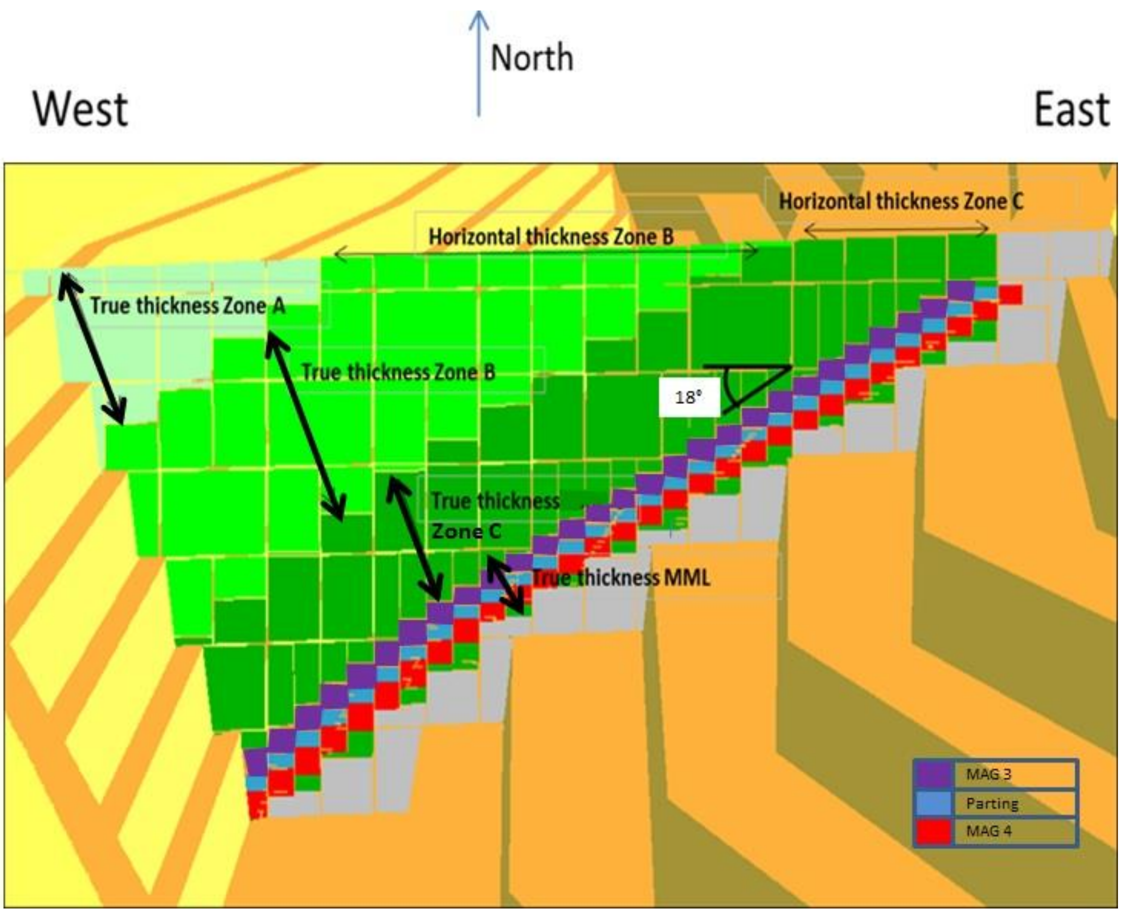
- a thirty year Life of Mine (LoM), mining 952,000 tonnes of undiluted MML per annum;
- two pits will be mined, Pit 1 and Pit 2, using the same open-pit mining method and the same fleet of mining equipment;
- a maximum depth of 80 m below original ground level is to be mined;
- an overall high wall slope angle of 55° will be applied;
- three zones of mineralisation, (Zone A, Zone B and Zone C (Table 1-7 and Figure 1-3)), which occur in the hanging wall above the MML, have been identified as having potential economic mineralisation and will be stored separately on the Low Grade and Lower Grade Stockpile for possible future treatment;
- an allowance for dilution of 5 % with low grade mineralisation (Zones A, B and C, and MML parting) and a 5 % loss of MML material to the Low and Lower Grade stockpile;
- the production rate required is readily achievable and should allow for an accurate separation of MAG 3 and MAG4 from the low grade Zone A and MML parting;
- the mining of the foot-wall was not considered;
- a ramp exacavated from the high wall will provide access to the working faces in the pits;
- Contractors are to be employed as the mining practitioners; and
- mining operations will take place from Monday to Friday on a 24 hour three shift per day cycle.

Applying the above design criteria and factors, a total of 28.56 Mt of MML mineralisation is estimated as mineable, this will be depleted at a rate of 952,000 tonnes per annum over a period of 30 years as indicated below in Table 1-8.

Table 1-7
Correlation between the geological layers and hanging wall zones

Geological Layer	Hanging wall mineralisation
UG-C	Zone A
UG-A	
UMG1	
UMG2	
MAG1 HW GAB**	Zone B
MAG1	Zone C
MAG2	
MML HW	

Figure 1-3
Representative cross-section of the mine indicating the hanging wall zones of mineralisation, and the MML Target zone



Note: *This diagram is for visualisation purposes only and is not to scale*

Table 1-8
Probable Ore Reserves

Orebody	True Thickness (m)	SG (t/m^3)	Tonnes (million)	V_2O_5 (%)
MML Upper, MAG 3	4.09	4.08	15,342	1.425
MML Lower, MAG 4	3.59	4.00	13,154	1.387
***Total/Average**	**7.68**	**4.04**	**28,496**	**1.41**

1.8 Mining

Prior to the commencement of mining of the MML, bush clearing, topsoil and/or black turf removal and storage, and non-MML and waste removal and storage will be required.

The MML ore will be mined in two pits, each following the dip of the MML (i.e. approximately 18°), one located to the north of the provincial road and one to the south. Each pit will utilise an access ramp excavated from the highwall (on the west) mined down at an angle of 8° to intersect the upper contact of the MML. Once the MML is intersected mining will follow it along strike on a level gradient to create an open pit width of approximately 30 m. This open pit will then advance via working faces to the north and south utilising the central, common ramp.

It was assumed for the purposes of this PFS that a specialised mining contractor will be appointed to undertake the excavation of the pit and the mining and transportation of the ore. Budget prices for the mining were procured from contractors who have knowledge of the area and have experience in the expected mining conditions. The proposed mining method is within acceptable practices in South Africa.

The Ore Reserves will be depleted at a rate of 952,000 tpa with an allowance of an additional 5 % of non-MML included in the material delivered to the plant. At this rate the life of mine is expected to be just under 30 years.

The mining study was completed to an accuracy of ±20 %.

1.9 Mineral Processing and Metallurgical Testwork

Testwork activities were carried out prior to the commencement of the PFS. The results of these previous testwork activities were used to determine various process parameters in the current plant design and are summarised in this report for reference. No further metallurgical testwork has been carried out during the current Project phase.

1.9.1 Concentrator

Negligible Mineral Resource feed grade and V_2O_5 recovery variability was expected, based on the understanding of the Project geology. This was further confirmed by testing of samples from various drill holes, both down dip and along strike.

Davis tube testing was conducted on composites of samples received from the various drill holes to investigate the optimum grind size for beneficiation based on vanadium grade and recovery achieved. The size specifications provided for the testing are summarised in Table 1-9.

With respect to product grades, results indicate that the feed material is fully liberated at the grind of 80 % < 53 µm. Tests of specific samples showed that the product grade is not particularly sensitive to the fineness of grind between 38 µm and 212 µm. The expected product grade composition as well as the variations around the average grade composition at a grind size of 80 % < 53 µm is summarised in Table 1-10.

Table 1-9
Grind specifications for liberation tests

Grind Specifications
80 % < 212 µm
80 % < 106 µm
80 % < 75 µm
80 % < 53 µm
80 % < 38 µm

Table 1-10
The average product grade composition and its range at a grind size of 80 % < 53 µm

	SiO_2	Al_2O_3	CaO	MgO	Fe^T	K_2O	MnO	Na_2O	P_2O_5	TiO_2	Cr_2O_3	V_2O_5
Average	1.03	3.50	0.15	1.20	57.62	0.01	0.25	0.01	0.0023	11.91	0.39	1.72
20th Percentile	0.65	3.36	0.09	1.06	57.00	0.01	0.24	0.01	0.0023	11.70	0.22	1.67
80th Percentile	1.22	3.62	0.18	1.35	58.19	0.01	0.26	0.01	0.0023	12.10	0.55	1.76

A grind size of 80 % < 75 µm was ultimately selected as a sensible lower limit for the target grind based on test results related to Fe recovery. A decrease from 96.3 % to 93.8 % was observed for the recovery of Fe to the product as the grind is reduced from 80 % < 212 µm to 80 % < 75 µm.

The final product (concentrate) specification that was selected based on the test results is shown in Table 1-11.

Table 1-11
The recommended concentrate analysis

	SiO_2	Al_2O_3	CaO	MgO	Fe^T	K_2O	MnO	Na_2O	P_2O_5	TiO_2	Cr_2O_3	V_2O_5
Product Specification	1.03	3.50	0.15	1.20	57.62	0.01	0.25	0.01	0.0023	11.91	0.39	1.75

In terms of recovery, the following equation was developed based on the test result data and used to calculate the expected product yield:

$$Yield = 1.8296 * Fe^{Feed} - 9.0556$$

1.9.2 Salt Roast Plant

A Ti-magnetite concentrate sample from the MML was provided to MINTEK (South Africa) to investigate vanadium extraction characteristics based on the alkali salt process. Roasting and leaching tests were conducted on the vanadium-bearing concentrate to determine the extraction efficiencies based on the following varied parameters:

- temperature; and
- roasting time.

A summary of the test results is shown in Table 1-12.

Table 1-12
Roast/leach test summary

Residue	Salt	Temp	Time	Vanadium Extraction (%)
1	Na_2CO_3	900	45	61.52
2	Na_2CO_3	900	120	61.54
3	Na_2CO_3	900	180	60.35
4	Na_2CO_3	1,000	45	64.82
5	Na_2CO_3	1,000	120	69.04
6	Na_2CO_3	1,000	180	66.83
7	Na_2CO_3	1,100	45	78.71
8	Na_2CO_3	1,100	120	Furnace failed
9	Na_2CO_3	1,100	180	Furnace failed
10	Na_2CO_3	1,100	45	83.04
11.1	Na_2CO_3	1,100	120	84.14
11.2	Na_2CO_3	1,100	120	81.03
12	Na_2CO_3	1,100	180	72.19
13	Na_2CO_3	1,200	45	73.75
14	Na_2CO_3	1,200	120	83.65
15	Na_2CO_3	1,200	180	82.67
16	Na_2SO_4	900	120	49.80
17	Na_2SO_4	1,000	120	54.20
18	Na_2SO_4	1,100	120	68.22
19	Na_2SO_4	1,200	120	75.77

Based on the test results, literature sources and industry standards the following parameters were selected for this study:

- a hot zone roasting temperature of 1,150 °C which corresponds to industry benchmarks for a combination of sodium carbonate and sulphate salt feed to the kiln;
- a hot zone roasting time of one hour which corresponds to industry benchmark; and

- a vanadium recovery of 83 % based on the sodium carbonate test results which are more representative for the testwork methods used.

1.10 Recovery Methods

The recovery of the final vanadium product from ore material is achieved through the salt roast process, as is typically employed by a number of existing vanadium producers in South Africa.

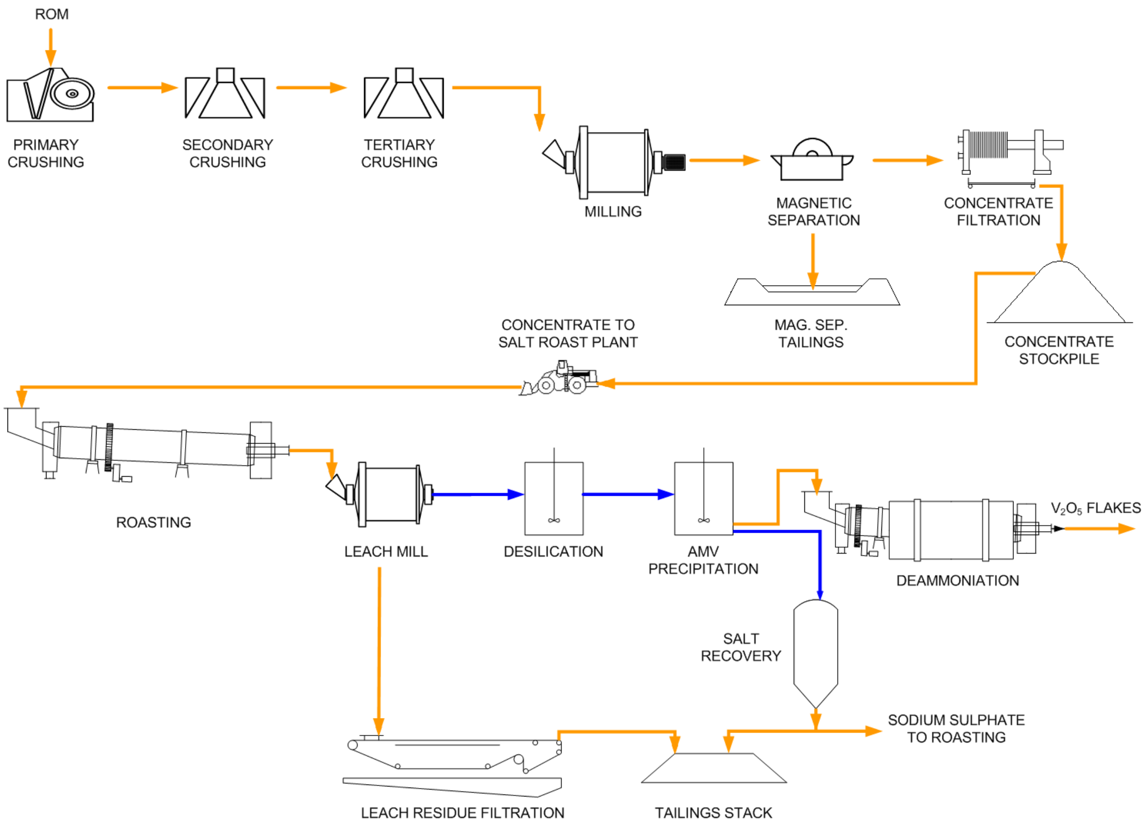
Vanadium-bearing RoM is received from mining operations in the concentrator circuit. The RoM proceeds through three stages of crushing before being milled to the required particle size. A magnetic separation process is used to separate the vanadium-containing magnetic fraction from the non-magnetic waste material, thereby producing a more concentrated, higher metal value material called concentrate. The waste material slurry is pumped to a tailings facility while the concentrate is filtered, stockpiled and then transferred to the salt roast plant for further processing.

The salt roast plant receives concentrate from the concentrator circuit and produces V_2O_5 flake (99.5 wt% purity) as the final product via the alkali salt roast process. The process involves roasting of the concentrate with alkali (sodium) salt, leaching of the resultant material with water, desilication, ammonium metavanadate (AMV) precipitation and deammoniation to produce the final V_2O_5 product.

A schematic representation of the process is shown in Figure 1-4.

Figure 1-4

Schematic representation of the concentrator and salt roast process



A summary of the overall concentrator and salt roast plant design basis parameters is presented in Table 1-13.

Table 1-13

Recovery plant design basis summary

Parameter	Unit	Value
RoM to concentrator	t/a	1,000,000
RoM grade	wt% V_2O_5	1.41
Concentrator mass yield	wt%	67.3
Concentrator recovery	wt% V_2O_5	83.5
Concentrator operating hours (crushing)	hours/annum	4,916
Concentrator operating hours (milling and magnetic separation)	hours/annum	7,790
Concentrate production	tpa	672,600
Concentrate grade	wt% V_2O_5	1.75
Salt roast plant operating hours	hours/annum	7,709
Salt roast plant recovery	%	80.5
V_2O_5 final production	tpa	9,525
V_2O_5 product purity	wt%	99.5

1.11 Project Infrastructure

1.11.1 Bulk Services

It is anticipated that raw water will be provided to site via a new pipeline to be installed from the Flag Boshielo Dam (Olifants River). The Olifants River Water Resources Development Project plans to build a pipeline from the Flag Boshielo Dam to Mokopane to meet the domestic and industrial needs of the area. This pipeline is currently being constructed; however it is anticipated to be able to supply water by 2020. Pamish is liaising with the relevant authorities (Trans-Caledon Tunnel Authority (TCTA)) to register its water requirements. Water for the development of the Project prior to 2020 will be drawn from local water sources.

The power supply has been identified as a main Eskom line approximately 10 km from the mine with the existing Eskom servitude on the public road being identified as the route for the incoming line. Back up from a minimum 5 MVA permanent on site diesel generator will be ensured to reduce the risks associated with any potential load shedding.

1.11.2 Mining

In order to enable mining of Pit 1 and Pit 2, the following infrastructure will be provided:

- bulk water supply and electrical reticulation:
    - bulk water is stored at the receiving area from the local bulk water suppliers
    - electricity from the Eskom Incomer yard to the concentrator, salt roast plant and bulk material handling point of distribution;
- waste water treatment, carried out at a water treatment facility in the concentrator area;
- clean and dirty water runoff channels located at strategic points in and around the mining areas at points determined via interaction with environmental specialists;
- mining contractor laydown area, including operational facilities,
- requisite fencing of the mining and plant operations; and
- pollution control dams including piping and pumping systems to treatment facility located within the process plant.

The supply of bulk electricity and water to the Project has been based on point sources located 10 km from the mine fence based on preliminary discussions with the relevant service providers and have been costed accordingly.

1.11.3 Residue Disposal Facility and Stockpiles

Residue disposal facility (RDF) and Stockpile facilities have been allocated as described below.

The RDF comprises the tailings from the magnetic separation (Magsep) process and includes:

- a Class C lined Tailings Storage Facility (TSF). A Class C liner is a requirement for all Type 3 classified waste (e.g. Magsep tailings). The liner consists of a 300 mm thick compacted clay layer overlain by a high-density polyethylene (HDPE) geomembrane with and a leakage detection system installed beneath the liner. Type 3 wastes refers to products containing chemical substance of above LCT0 but below or equal to LCT1 limits;

- a concrete Return Water Sump (RWS); and
- a Class C lined Storm Water Dam (SWD).

The liner requirements for the RDF's and stockpiles are defined by the following Government Notices (GN):

- R. 634 National Environmental Management: Waste Act (59/2008): Waste Classification and Management Regulations (2013)
- R. 635 National norms and standards for the assessment of waste for landfill disposal (2013)
- R. 636 National norms and standards for disposal of waste to landfill (2013).

The waste assessment norms and standards is dictated in GN R. 635 based according to the Leachable Concentration Threshold (LCT, expressed as mg/l of waste) and the Total Concentration Threshold (TCT, expressed as mg/kg of waste). These refer to the concentration threshold of particular elements or chemical substances found within the waste.

The Magsep RDF is positioned approximately 3 km from the proposed pit locations, with the TSF being a self-raised facility having a final height of 30 m. For the purpose of the PFS, the size of the starter wall was based on a rate of rise of 1.5 metres per annum (mpa) and designed to provide at least 1 m of freeboard based on the expected grading of the Magsep TSF. Drainage of the supernatant pond will be done by means of a penstock system which decants into a silt trap and ultimately to the RWS. Spillage from the RWS is discharged into the SWD in the event of a large rain event. The SWD is designed to provide sufficient capacity to store run-off from the RDF in a 1:50 year storm event. The Magsep TSF has purposefully been placed away from the pit areas to avoid impact from the blast vibrations.

The calcine, lime cake and sodium sulphate RDF, known as the calcine RDF, comprises:

- a three-phased Class A lined Dry Stack Facility. A Class A liner is a legal requirement for all Type 1 Waste (i.e. the calcine and lime cake and sodium sulphate). It is a "double" liner system comprising of two layers of clay, and two HDPE geomembranes, all underlain by a leakage detection system. Type 1 wastes are classified as products of chemical substance of above LCT2 but below or equal to LCT3 limits; or above TCT 1 and below or equal to TCT2 levels ;
- a concrete RWS; and
- a Class A lined SWD.

The EIA for Pamish has been conducted and has presented guidelines for all of the Pamish tailings and stockpile infrastructure. This includes both the Low and Lower Grade stockpiles and the Magsep RDF. The material from each pit's hanging walls and the MML parting will be stockpiled in phased Class C lined facilities, one facility per pit. These stockpiles have been placed as close to the respective pits as possible to reduce hauling distance. The Low and Lower Grade material from each pit (hanging wall and MML parting) are placed together into a combined stockpile. In doing so, the total footprint area will be minimised, thereby minimising the capital expenditure. The hanging wall stockpile at Pit 1 has a final height of 66 m and 45 m at Pit 2, both lower than the maximum height suggested by the EIA of 70 m.

The EIA for SaltCo has still to be conducted. This EIA will address all of the associated tailings for SaltCo, i.e. the calcine, lime cake and sodium sulphate RDF. The calcine DSF has been combined with the lime cake and sodium sulphate tailings stream in order to reduce the overall footprint and to reduce the cost of lining. The facility has also been phased to reduce the start-up capital expenditure. Based on the assumption that the calcine, lime cake and sodium sulphate products will be deposited by means of trucking from the plant, the facility has been placed as close as possible to the plant (2.2 km to the southeast) to reduce haulage cost. Both materials provide a potential environmental hazard and the RDF has thus been designed such that continuous capping of the exposed material will take place. A SWD has been included to contain run-off from the RDF in the event of a storm event of the 1:50 year intensity. The facility has been restricted to a height of 35 m and will be subject to the findings of the EIA which was still to be conducted at the effective date of this Report. As a result of the limited available footprint area, the calcine RDF may impact an indicated wetland area, the results of which should be established in the EIA for the facility.

1.12 Environmental Aspects

The Project site is surrounded by several communities and farmers who could potentially be impacted through the development of the Project.

Pamish is in the process of obtaining the necessary environmental authorisations and aims to comply with all relevant legal requirements. Key tasks that need to be completed to secure the Mining Right include the following:

- in terms of the requirements of the Mineral and Petroleum Resources Development Act, 2002 as amended (Act No. 28 of 2002) (MPRDA), an application for a Mining Right (MRA) was submitted to the DMR for the Project through the South African Mineral Resource Administration System (SAMRAD) online portal under Reference LP 30/5/1/1/2/10102 MR;
- Environmental Impact Assessment (EIA) and Environmental Management Plan (EMP) in accordance with the MPRDA in support of the MRA and environmental authorisation in accordance with the National Environmental Management Act, 1998 (Act No. 107 of 1998) (NEMA);
- Public Participation Process in accordance with the EIA 2014 Regulations (GN R982 published in Government Gazette 38282 of 4 December 2014);
- specialist investigations in support of the EIA Report and EMP;
- Integrated Water Use Licence Application (IWULA) in compliance with the National Water Act, 1998 (Act No. 36 of 1998) (NWA);
- approval from the South African Heritage Resources Development Agency (SAHRA) in terms of the National Heritage Resource Act, 1999 (Act No. 25 of 1999) (NHRA); and
- Waste Management Licence (WML) in compliance with the National Environmental Management: Waste Act, 2008 (Act No. 59 of 2008) (NEM:WA).

The purpose of the EIA process is to ensure that potential environmental and social impacts associated with construction, operation and closure of a project are identified, assessed and appropriately managed. There are two primary phases of an EIA process, namely the scoping phase and the impact assessment phase. Identification of potential impacts occurs during the scoping phase, whilst the assessment and mitigation of those impacts occurs during the impact assessment phase.

Based on the proposed scope of work, the total liability cost has been calculated at US$ 15,425,748.

A Public Participation Process was completed during which landowners, communities, traditional council and other Interested and Affected Parties (I&APs) were consulted. Table 1-14 shows the communities that were consulted during the EIA phase of the MRA process.

Table 1-14
Consultation meetings held during the EIA phase of the MRA process

Date	Time	Venue	Communities Invited
28-Sep-15	14:00 - 16:00	Ditlotswane Primary School	Ditlotswane Village and Mmotong
	17:00 - 19:00	Dipilikomong	Pudiakgopa, Mautjane, Kwenaite, Mahlaba ,Basogadi, Matlhaba, Mathoathoasa and Bakenburg Traditional Council
29-Sep-15	17:00 - 19:00	Lephadimisha Senior Secondary School	Claremont, Goodhope and Taolome
30-Sep-15	17:00 - 19:00	Leyden Community Hall	Kaditswene and Leyden
01-Oct-15	10:00 - 12:00	Oasis Lodge	Key Stakeholder Meeting
	17:00 - 19:00	Nkgakgautha Primary School	Rooiwal and Malokongskop
02-Oct-15	10:00 - 12:00	Mpedi Farm Lodge	Landowners/ Farmer Focus Group Meeting
	17:00 - 19:00	Mphaka Primary	Sepharane, Mapela
03-Oct-15	10:00 - 12:00	Bakenberg Stadium	Public Meeting

The specialist studies that were undertaken as part of the EIA process are listed below.

- Visual Assessment;
- Greenhous Gas and Climate Change Assessment;
- Soil Assessment;
- Surface Water Assessment;
- Groundwater Assessment;
- Geochemistry and Waste Classification;
- Air Quality Assessment;
- Fauna and Flora Assessment (including avi-fauna);
- Wetland Assessment;
- Aquatic Ecology Assessment;
- Macro Economic Assessment;
- Social Assessment;
- Community Health Assessment;
- Traffic and Transport Assessment;
- Noise Assessment;
- Blast and Vibration Assessment;

- Cultural Heritage Assessment;
- Closure Cost Assessment; and
- Rehabilitation Plan.

Key environmental authorisations to be completed for the proposed salt roast plant and associated infrastructure include:

- environmental authorisation for Listed Activities as per NEMA EIA 2014 Regulations;
- WML in compliance with the National Environmental Management Waste Act 1998 (Act No. 59 of 2008) (NEM:WA);
- an Atmospheric Emissions Licence (AEL) application and authorisation from the Waterberg District Municipality as per the requirements of the National Environmental Management: Air Quality Act, 2004 (Act No. 39 of 2004) (NEM:AQA); and
- Water Use Licence Application (WULA) in compliance with the National Water Act, 1998 (Act No. 36 of 1998) (NWA).

1.13 Market Studies and Contracts

1.13.1 Properties and uses of Vanadium

Vanadium is a grey, soft, ductile high value metal whose main application is in the steel industry. As a steel alloying element, vanadium offers an excellent combination of strength, corrosion resistance, weldability and fabricability. It improves the tensile strength of steel, making it an effective alloy for strengthening construction steels (rebar). Vanadium also has a grain refining and dispersion hardening effect in tempering steels, which provides for corrosion and abrasion resistance to steel alloys, making them suitable for use in extreme temperature environments. Its high strength to weight ratio makes it an important component in the manufacturing of automotive and aviation vehicles where fuel efficiency is an important factor. Other uses include application in vanadium-containing titanium alloys, and various chemical uses, the most significant of which is its use in utility scale energy storage battery systems.

1.13.2 Consumption

According to Roskill, global consumption of vanadium in final products in 2014 was 92,700 tonnes. Consumption is concentrated applications-wise and geographically. At a 90 % consumption rate, the steel sector is the single largest driver of vanadium demand on account of its share of vanadium consumption and growing intensity of use of vanadium. Most of this is based in China, which accounts for 46 % of global vanadium usage.

The growing intensity of use of vanadium in steel has seen the demand growth of vanadium outpace the crude steel production. Between the years 2006 to 2014, vanadium consumption grew at a compound annual growth rate (CAGR) of 7.4 %, while crude steel production grew approximately 3.6 % per annum. An increase in the intensity of use of vanadium in emerging markets to close the gap with global averages is likely to contribute to demand growth, particularly in China where greater enforcement of legislation seeks to phase out the use of lower strength steel and replace it with higher strength steel (Grade III rebar). Consequently, Chinese steel intensity of use increased by 8 % between 2006 and 2014. Continued enforcement of new standards is expected to continue to increase vanadium demand growth, even while steel production is expected to grow at a subdued 1 % CAGR through to 2025.

In the automotive sector, the regulation-driven push for greater fuel efficiency and emission controls will continue to drive the growth of high-strength, low-weight steel alloys, of which vanadium is an important component. The aerospace industry, which currently accounts for approximately 7 % of vanadium consumption, is also a demand growth contributor through the increasing use of titanium-vanadium alloys in new aircraft models.

The growing energy storage market could present a step change opportunity for vanadium demand outlook through vanadium-based, utility-scale vanadium redox flow battery (VRFB) technology. The energy storage market is expected to top US$ 300 billion by 2030, according to various market forecast reports, with vanadium-based energy storage systems estimated to capture a significant share of this market. Although VRFBs' need for more space and use of a liquid electrolyte make them a poor fit for electronics and cars, their scalability, quick recharge rate and nearly unlimited ability to recharge without performance degradation are touted as making them attractive in utility-scale applications. The industry estimates that the successful commercialisation of VRFBs could provide a market for up to 10,000 tpa of vanadium pentoxide.

1.13.3 Supply

Vanadium supply is also concentrated both geographically and in terms type of production. Approximately 80 % to 90 % of the global supply of vanadium is from three countries, namely China (53 %), South Africa (20 %) and Russia (17 %). The majority of the vanadium produced (approximately 64 %) is from vanadium slag produced as a co-product in melting vanadium-bearing magnetite iron ores (co-producers) during the steel production process. The slag is further processed via a roast-leach process in countries with ferrovanadium conversion facilities (e.g., Czech Republic, South Korea and Japan). A significant amount of vanadium (approximately 20 %) is produced directly from vanadium-bearing magnetite ores with sufficiently high vanadium grades (primary producers), mostly located in South Africa, via a roast-leach process. About 12 % of global vanadium supply is produced from secondary sources.

Recent developments in the steel sector, which accounts for over 60 % of vanadium production and is itself dominated by Chinese production, have, and are expected to continue to have, a material impact on the vanadium outlook.

At the time of writing, there were eight steel mills in China producing vanadium slag. The vanadium slag producers produce a combined 40 Mt steel each year (less than 5 % of Chinese steelmaking capacity), but supply approximately 50 % of global vanadium feedstock supply. Should just one of these mills cease production, it could reduce global vanadium output by over 10 %. Notwithstanding the significant excess vanadium slag making capacity associated with steel plants, vanadium production from co-production is under threat, primarily due to constrained steel economics (exacerbated by low steel prices and high mining input costs of their captive low grade iron ore mines) and the more attractive alternative of substituting the low grade, high cost (>US$

90 per tonne) vanadium-bearing magnetite iron ore with higher grade, cheaper (<US$ 40 per tonne) seaborne haematite (non-vanadium-bearing) iron ore, available mostly to coastal steel plants. Chengde Steel, located close to a port, is one of the largest vanadium producers in China. During 2015, Chengde Steel began substituting domestically produced iron ore with seaborne iron ore, resulting in an approximately 15 % drop in their vanadium production levels. For vanadium co-producing steel plants which are not located close to a port, the scope for substitution barely exists given the additional costs associated with transportation of the ore from the coast. As a result, the Panzhihua Group, which is located inland in Sichuan Province, put two blast furnaces on care and maintenance during 2015, resulting in a decrease in its vanadium production by about 25 % or 5,500 tonnes contained vanadium annualised. Other steel mills in the southwest Sichuan region (Chuan Wei, Kunming, and Desheng) are also slowing their vanadium production.

On April 14, 2015 Evraz Highveld Steel and Vanadium Limited, a South African based integrated operation comprising a vanadiferous magnetite mine and a steel plant producing steel and vanadium-bearing slag, announced that it was going into business rescue and subsequently stopped production. This resulted in a decrease of as much as 11 % in the global vanadium feedstock capacity.

Thus, while capacity for final vanadium products exists in the market, it can only be utilised, mostly, with vanadium feedstock from co-product steel producers, most of which is uneconomic and thus unavailable as described above. In addition, several planned new vanadium projects which were expected to add to global capacity are unlikely to come on-stream in the current economic climate.

1.13.4 Supply and demand balance

The demand and supply of vanadium have largely been in balance in 2014/15. The growing intensity of use of vanadium can however be expected to reduce available vanadium supply significantly more than a reduction in steel production is expected to reduce vanadium consumption, driving the market towards a vanadium supply deficit. According to TTP Squared, the deficit arising from declining supply and a relatively flat demand could be as much as 18,000 tonnes contained vanadium outside China, more than the anticipated excess of approximately 5,000 tonnes contained vanadium from China in 2016. This suggests a decrease in inventories of approximately 13,000 tonnes vanadium in 2016, or 14 % of the annual supply/demand. The elimination of excess vanadium inventories is expected to return fundamentals to drive vanadium prices going forward, the most significant of which is the cash cost of production.

1.13.5 Vanadium cost curve

Overall vanadium production costs vary across the various types of producers. Co-production represents the lowest-cost production source, estimated at approximately US$ 4.50/lb V_2O_5. The low cost assumes that there is, in effect, no cost for the production of vanadium slag (which is produced as a co-product in the steel making process) and that the cost of production is for refining only. However, while co-production presents the lowest cost production source, its viability is determined not by the economics of processing the slag but by the economics of the steel making that produces the slag.

Primary production from vanadiferous titano-magnetite (VTM) mines (similar ores used by co-producers) requires mining and production of concentrate and, as a result, costs are slightly higher than those of integrated producers. Depending on the grades of the primary deposits, Roskill estimates costs in the order of US$ 6.00/lb V_2O_5. Secondary production, which is often linked to current vanadium market prices and is therefore exposed to market fluctuations, is considered to have the highest production costs.

Based on today's consumption level, the cash cost of the least economic units necessary to meet demand is roughly US$ 6.00/lb V_2O_5.

1.13.6 Vanadium price outlook

Vanadium prices have been following a slight downward trend since 2010 due to a combination of oversupply and low demand, particularly in China. This was further impacted by a material fall in 2015 in the thin vanadium spot market, which is key in determining contract pricing.

At prices below US$ 21/kg, however, some secondary production is estimated to no longer be economic. In combination with the low iron ore prices which are contributing to a reduction in co-production, it is expected that inventories may start falling resulting in the possibility of a small deficit in supply which could lead to a partial recovery in prices.

Roskill suggests that a recovery in prices to about US$ 21 to US$ 24/kg is likely over the period 2015-2017. In the longer term, this price level is expected to offer insufficient incentive to encourage the development of new supply or increases in secondary output. Assuming ongoing growth in demand, a further recovery (in real terms) to about US$ 24 to US$27/kg could be expected. In nominal terms, the effects of inflation and an eventual recovery in energy prices will likely lead to higher US$ prices.

A price of US$ 7.50/lb (US$ 16.53/kg) for V_2O_5 flakes at >98 % purity is assumed for the Project, with an anticipated initial production in 2019. This approximates the 10 year historical average of US$ 7.63/lb (Jun 2005 – May 2015), and is approximately 17 % higher than the 15 year historical average of US$ 6.39/lb (Jun 2000 – May 2015).

1.14 Capital and Operating Cost Estimates

1.14.1 Mining and Shared Infrastructure

The capital expenditure (Capex) for mining and infrastructure was estimated based on a quantitative assessment of the volume (m^3), area (m^2), length (m) or quantity (number) of units required per item to sustain the planned capacity of the operation and associated mining activities. A unit cost per item (US$/unit) was factored from current estimates in the Consultants database and escalated to the base date of evaluation.

The Capex and operating cost (Opex) summaries are summarised in Table 1-15 to Table 1-18.

Table 1-15
Mining Capex breakdown

Factor	Unit	Value (US$)
Earth Moving Equipment	US$ Real 2015	1,877,120
Loading and Unloading	US$ Real 2015	41,578
Development (Capitalised)	US$ Real 2015	1,337,160
Mining Equipment fleet site establishment / de-establishment	US$ Real 2015	167,149
Total Mining Capital	**US$ Real 2015**	**3,423,007**
This excludes a total contingency of		*342,301*

Note: *slight discrepancies in totals may occur due to rounding*

The Capex required for the establishment of the shared infrastructure for the mine is set out in Table 1-16.

Table 1-16
Shared infrastructure Capex

Factor	Unit	Result (US$)
Access control	US$ Real 2015	398,629
Drains	US$ Real 2015	57,083
Fencing	US$ Real 2015	1,572,531
Haulage Roads	US$ Real 2015	11,539,482
Offices	US$ Real 2015	1,775,121
Pollution Control Dams	US$ Real 2015	1,971,082
Power Supply	US$ Real 2015	2,830,053
Return Water	US$ Real 2015	2,786,694
Stores	US$ Real 2015	1,694,889
Waste Management	US$ Real 2015	1,248,602
Water Supply	US$ Real 2015	2,193,679
Workshops	US$ Real 2015	225,567
Total Infrastructure Capital	**US$ Real 2015**	***$28,293,410***
This excludes a total contingency of		*2,829,341*

Note: *slight discrepancies in totals may occur due to rounding*

Table 1-17 Mining unit costs per phase		
Metric	**Unit**	**Result (US$)**
Avg. Mining Cost (Yr 1 to Yr 5)	US$/t RoM	8.96
Avg. Mining Cost (Yr 6 +)	US$/t RoM	16.22
LoM Avg. Mining Cost	**US$/t RoM**	**15.01**

Table 1-18 Opex summary			
Factor	**F/V[1] Ratio**	**US$ Exposure (%)**	**Result (US$)**
Mining	15:85	50 %[2]	15.01
Infrastructure Cost	90:10	20 %	0.72
Avg. LoM Operating Cost	**18:82**	**49 %**	**15.83**

Note: *[1] F/V – fixed variable*

[2] Based on the proportion of the cost relating to Diesel, Explosives and Tyres

1.14.2 Mineral Processing

1.14.2.1 Recovery Area Capex

The Capex estimate includes all of the direct and indirect costs associated with the capital portion of the proposed concentrator and salt roast plant sections.

The proposed concentrator and salt roast plant Capex is summarised in Table 1-19.

Table 1-19 Capex summary by area	
	Total (US$)
TOTAL DIRECT CAPEX	143,800,000
Concentrator	27,700,000
Salt Roast Plant	116,100,000
TOTAL INDIRECT CAPEX	38,600,000
Concentrator	6,900,000
Salt Roast Plant	31,700,000
TOTAL CONTINGENCY	35,200,000
GRAND TOTAL	**217,600,000**
Owners Cost	*4,300,000*
Total Including Owner's Cost	***222,000,000***

1.14.2.2 Recovery Area Opex

The annual Opex estimate for the proposed concentrator and salt roast plant sections includes the fixed and variable costs required for 9,525 tpa of vanadium pentoxide (V_2O_5) production.

Table 1-20 and Table 1-21 summarise the breakdown of operating cost items for the concentrator and the salt roast plant respectively.

Table 1-20
Concentrator Opex summary

Cost Item	Total Cost per Annum (US$)	US$/t Concentrate
Reagents	5,922,000	7.90
Consumables	593,000	0.79
Water	107,000	0.14
Power	2,744,000	3.67
Labour	450,000	0.60
Maintenance Materials	611,000	0.82
TOTAL	10,427,000	14.00
Unit Cost per Tonne Final Product		
Opex Unit Cost (US$/tonne V_2O_5)	US$ 1,100	

Note: *slight discrepancies in totals may occur due to rounding*

Table 1-21
Salt roast plant Opex summary

Cost Item	Total Cost per Annum (US$)	US$/t Concentrate
Reagents	22,150,000	29.60
Consumables	2,215,000	2.96
Water	234,000	0.31
Power	4,802,000	6.43
Labour	4,880,000	6.53
Maintenance Materials	2,555,000	3.42
TOTAL	36,836,000	49.30
Unit Cost per Tonne Final Product		
OPEX Unit Cost (US$ / ton V_2O_5)	US$ 3,870	

Note: *slight discrepancies in totals may occur due to rounding*

1.14.3 Residue disposal facilities and stockpiles

Capex estimates to within +25/-15 % and operating expense estimates within an accuracy of ±25 % have been undertaken. Closure and rehabilitation costs for each of the RDF's and stockpiles have also been included to an accuracy of ±35 %.

Table 1-22, Table 1-23 and Table 1-24 summarise the Capex, Opex and closure costs associated with the RDF's and stockpiles respectively.

Table 1-22
Residue Disposal Facility and Stockpile Capex summary

Cost Item	Total Cost (US$)
Magsep RDF	9,222,245
Calcine RDF	30,918,852
Hanging Wall Stockpile - Pit 1	21,528,113
Hanging Wall Stockpile - Pit 2	12,565,675
Professional Services (Definitive Feasibility Study)	274,510
Total	**74,509,395**

Table 1-23
Residue Disposal Facility and Stockpile Opex summary

Cost Item	Total Cost Per Annum (US$)
Magsep RDF	203,137
Calcine RDF	156,078
Professional Services	31,373
Total (Per Annum)	**390,588**
Total (Life of Mine)	**11,717,647**

Table 1-24
Residue Disposal Facility and Stockpile Closure Cost summary

Cost Item	Total Cost (US$)
Magsep RDF	1,127,392
Calcine RDF	2,132,980
Hanging Wall Stockpile - Pit 1	4,378,390
Hanging Wall Stockpile - Pit 2	2,495,159
Total	**10,133,921**

Rehabilitation, closure and aftercare costs (to an accuracy of ±35 %) of the RDFs and two stockpiles have been estimated at US$ 10.13 million with US$ 1.47 million of this cost incurred over the operational phase of the mine, US$ 7.99 million incurred over a period of one year during mine closure, and US$ 0.70 million is incurred over a period of approximately five years following the mine closure.

The total life of mine cost associated with the Mokopane Vanadium PFS design of the RDFs and Stockpiles over the duration of the Project life (Study/Design Phase to Post Closure) is estimated at US$ 96.36 million.

1.14.4 Environmental Studies

1.14.4.1 Environmental Monitoring Costs per Annum

Environmental monitoring costs have been calculated based on the assumption that Consultants will be appointed to undertake the monitoring. Monitoring equipment costs have not been included (Table 1-25).

Table 1-25
Estimated environmental monitoring costs per annum

Aspect	Cost per Annum (US$)
Aquatic and Wetland (bi-annual basis)	7,059
Air quality (monthly basis)	10,196
Noise (quarterly basis)	20,387
Surface Water	
Construction Phase (bi-weekly)	21,412
Operational Phase (quarterly)	3,294
Groundwater (quarterly basis)	9,412
Annual Total	**71,760**

Note: *slight discrepancies in totals may occur due to rounding*

Table 1-26 gives a breakdown of the costs associated with the additional studies and licences required for the salt roast plant and for water supply.

Table 1-26
Estimated costs for additional environmental studies or licence applications required

Study/Application	Associated Cost (US$)
Salt Roast Plant EIA/EMP	274,510
Salt Roast Plant WULA	15,686
Air Emissions Licence	4,314
Groundwater Supply Study	
Geophysics	3,922
Drilling (per borehole)	4,706
Aquifer Testing (per borehole)	3,137
Groundwater Monitoring Network Establishment – drilling of six boreholes	28,235
Kinetic Tests	23,529
Environmental Input for Definitive Feasibility Study	11,765
TOTAL (excluding drilling and aquifer testing)	**369,804**

Note: *slight discrepancies in totals may occur due to rounding*

1.15 Financial Valuation

A business case optimisation exercise concluded that an integrated mining/concentrator/salt roast plant business case was optimal for the Project and should be pursued as the base case. The economic analysis for the Project was completed for a 1,000 ktpa run of mine (RoM) and concentrator plant capacity which is equivalent to a production of 673 ktpa of vanadium-containing magnetite concentrate, which in turn is fed into a Salt Roast Plant to produce 9.53 ktpa of vanadium pentoxide (V_2O_5) flakes (98 % contained V_2O_5).

The business case metrics are set out in Table 1-27.

Table 1-27
Summary of salient technical metrics for the Project

RoM			Beneficiation – Concentrate Plant		Beneficiated Product - Concentrate		Salt Roast Plant	
RoM Ore (ktpa)	Fe (%)	V_2O_5 (%)	Mass Yield (%)	V_2O_5 Recovery (%)	Treatment Plant Feed (kt)	Conc Grade V_2O_5 (%)	V_2O_5 Recovery (%)	V_2O_5 Produced (ktpa)
1,000	42	1.41	67.26	83.50	672.33	1.75	80.52	9.53

The Project will be developed under a "two operating company model". The Mine/Concentrator will be owned by Pamish, the company which is the current holder of the Prospecting Rights for the Project and which has submitted the Mining Right Application to the Department of Mineral Resources (DMR). The Salt Roast Plant will be owned by a company still to be formed (SaltCo).

The valuation of the Project was completed using an ungeared real terms discounted cash flow (DCF) financial model. The valuation date is 1 September 2015. All monetary inputs are in 1 September 2015 money terms. The real discount rate (hurdle rate of return) used to calculate the net present value (NPV) was set at 9 % real. The long term V_2O_5 (98 %) price assumed is US$ 7.5/lb or US$ 16.53/kg.

The following development timetable milestones have been assumed in generating the DCF financial model:

- commencement of the Definitive Feasibility Study (DFS) – March 2016, after a period of time for Bushveld to raise the necessary funding for the DFS;
- completion of the DFS, including Project fund raising – February 2017;
- commencement of construction – March 2017;
- commencement of Pamish operations – March 2018; and
- commencement of SaltCo operations – March 2019.

Production ramp-up for both companies is assumed to be 69 % in year 1, 99 % in year 2 and 100 % in year 3.

The techno-economic parameters were sourced from capital expenditure and cash operating cost estimates which were generated by the Project Consultants. The summary of Project establishment capital expenditure is set out in Table 1-28. Note that Capex estimates beyond financial year (FY) 2021 are incorporated into the ongoing Capex provisions for Pamish and SaltCo. This is particularly true for the "Waste & Rock Dumps" category, a significant portion of which is scheduled to be spent beyond FY2021.

Table 1-28
Capital expenditure schedule – includes contingencies FY2016 to FY2021

	FY2016 (US$)	FY2017 (US$)	FY2018 (US$)	FY2019 (US$)	FY2020 (US$)	FY2021 (US$)	Total (US$)
Studies	1.74	5.99	0.00	0.00	0.00	0.00	7.73
Mining	0.00	0.00	3.67	0.00	0.00	0.00	3.67
Beneficiation & Owners Cost	0.00	0.00	125.87	95.37	0.90	0.00	222.14
On-site Infrastructure	0.00	0.00	20.45	-1.51	-0.01	8.69	27.61
Bulk Services	0.00	0.00	2.85	1.03	0.01	0.00	3.89
Corporate	0.00	0.00	0.66	0.00	0.00	0.00	0.66
Environmental	0.00	0.00	0.53	0.06	0.00	0.00	0.60
Waste & Rock Dumps	0.00	0.00	23.86	7.88	0.07	0.00	31.81
Total Establishment Capex	1.74	5.99	177.89	102.83	0.97	8.69	298.11
Capital Intensity - US$/ton V_2O_5							**31,284**
Capital Intensity - US$/Kg V_2O_5							**31.28**
Capital Intensity - US$/lb V_2O_5							**14.19**

Sustaining capital expenditure has been set at an average of 6.1 % of on site consolidated cash costs per annum. This is equivalent to 1.3 % of establishment capital expenditure per annum (in real terms).

A summary of operating cost metrics is set out in the Table 1-29 below. The US$ cost per pound of V_2O_5 compares favourably to the Project's industry peers.

Table 1-29
Salient cash operating metrics of the Project

Metric	Units	Results
Working Cost Make-up for V_2O_5 for 2022		
RoM Production	ktpa	1000.00
Concentrate Production	ktpa	672.60
Total Effective V_2O_5 Production	ktpa	9.53
Total Effective V_2O_5 Production	million lbs pa	21.0
Total Mining Cash Cost (including Royalty)	US$ million	13.31
Total Concentrate Cash Cost	US$ million	9.54
Total SRP Cash Cost	US$ million	33.69
Total Overheads & Logistics Cash Cost	US$ million	12.29
Make-up Costs per lb V_2O_5		
Mining (including Royalty)	US$/lb	0.63
Concentrator	US$/lb	0.45
SRP	US$/lb	1.60
Overheads & Logistics	US$/lb	0.58
Total	**US$/lb**	**3.28**
Make-up Costs per RoM Tonne		
Mining (including Royalty)	US$/t	13.31
Concentrator	US$/t	9.54
SRP	US$/t	33.69
Overheads & Logistics	US$/t	12.29
Total	**US$/t RoM**	**68.82**
Make-up Costs per tonne of concentrate		
Mining (including Royalty)	US$/t	19.79
Concentrator	US$/t	14.18
SRP	US$/t	50.09
Overheads & Logistics	US$/t	18.27
Total	**US$/t Conc**	**102.33**

Note: *SRP – Salt Roast Plant*

A long term Rand/US$ exchange rate has been set at ZAR 12.75. At the time when the techno-economic parameters were generated, particularly the capital expenditure and operating cost estimates, the consensus Rand/US$ exchange rate was set at ZAR R11.7. Accordingly, amendments to the capital expenditure and operating costs estimates have been made. The Rand/US$ exchange rate has weakend further and consequently a sensitivity to measure the effect on the Project of this further Rand weakness has been calculated. The salient financial metrics for the Project are set out in Table 1-30.

Table 1-30
Salient financial metrics

Metric	Units	Results	
Long Term Exchange Rate	ZAR/US$	12.75	
V_2O_5 Price	US$/kg real	16.53	
	US$/lb real	7.50	
Total Capex	US$ million	298.11	
		Pre-tax	**Post-tax**
NPV	US$ million	418.04	259.35
IRR	%	24.80	20.36
Effective Equity Return	%	45.60	36.70

The calculated pre-tax NPV for the Project is US$ 418.04 million (post-tax US$ 259.35 million) at a 9 % real discount rate. The pre-tax IRR is 24.80 % real (post-tax 20.36 % real). The effective annual return on equity, assuming a 50 % debt to equity ratio is 36.70 % in real terms (post-tax).

Sensitivities of the calculated NPV to variations in salient metrics have been calculated and are set out in Table 1-31 (pre-tax) and Table 1-32 (post-tax).

Table 1-31

Sensitivities of the Consolidated Project Pre-tax NPV (Real US$ million) to changes in key metrics

Salt Roast Recovery			Total Capex			ZAR/US$ Exchange Rate			Real Discount Rate		V_2O_5 Flake Price			Working Costs			Concentrator Recovery		
%	Recovery	NPV US$ million	%	Capex US$ million	NPV US$ million	%	ZAR/US$	NPV US$ million	%	NPV US$ million	%	Long Term Price US$/lb	NPV US$ million	%	US$/lb V_2O_5	NPV US$ million	%	Recovery	NPV US$ million
-10	72.5 %	**301.9**	10	327.9	**393.8**	-10	11.48	**362.3**	12	**252.4**	-10	6.75	**298.7**	10	3.60	**315.7**	-5	79.3 %	**356.5**
-5	76.5 %	**360.0**	5	313.0	**405.9**	-5	12.11	**391.6**	10	**353.4**	-5	7.13	**358.4**	5	3.44	**366.8**	-2.5	81.4 %	**387.3**
0	80.5 %	**418.0**	0	298.1	**418.0**	0	12.75	**418.0**	9	**418.0**	0	7.50	**418.0**	0	3.28	**418.0**	0	83.5 %	**418.0**
5	84.5 %	**476.1**	-5	283.2	**430.2**	5	13.39	**441.9**	8	**495.0**	5	7.88	**477.7**	-5	3.11	**469.2**	-2.5	85.6 %	**448.8**
10	88.6 %	**534.2**	-10	268.3	**442.3**	10	14.03	**463.7**	7	**587.2**	10	8.25	**537.4**	-10	2.95	**520.4**	5	87.7 %	**479.6**
15	92.6 %	**592.3**	-15	253.4	**454.4**	15	14.66	**483.5**	6	**698.4**	15	8.63	**597.1**	-15	2.78	**571.6**	10	91.9 %	**541.1**

Table 1-32

Sensitivities of the Consolidated Project Post-tax NPV (Real US$ million) to changes in key metrics

Salt Roast Recovery			Total Capex			ZAR/US$ Exchange Rate			Real Discount Rate		V_2O_5 Flake Price			Working Costs			Concentrator Recovery		
%	Recovery	NPV US$ million	%	Capex US$ million	NPV US$ million	%	ZAR/US$	NPV US$ million	%	NPV US$ million	%	Long Term Price US$/lb	NPV US$ million	%	US$/lb V_2O_5	NPV US$ million	%	Recovery	NPV US$ million
-10	72.5 %	**175.2**	10	327.9	**239.5**	-10	11.48	**219.1**	12	**143.7**	-10	6.75	**172.9**	10	3.60	**186.7**	-5.0	79.3 %	**219.3**
-5	76.5 %	**217.4**	5	313.0	**249.5**	-5	12.11	**240.3**	10	**214.2**	-5	7.13	**216.3**	5	3.44	**223.1**	-2.5	81.4 %	**239.3**
0	80.5 %	**259.3**	0	298.1	**259.3**	0	12.75	**259.3**	9	**259.3**	0	7.50	**259.3**	0	3.28	**259.3**	0.0	83.5 %	**259.3**
5	84.5 %	**301.2**	-5	283.2	**269.1**	5	13.39	**276.5**	8	**313.0**	5	7.88	**302.3**	-5	3.11	**295.6**	2.5	85.6 %	**279.2**
10	88.6 %	**343.0**	-10	268.3	**278.9**	10	14.03	**292.1**	7	**377.3**	10	8.25	**345.3**	-10	2.95	**331.9**	5.0	87.7 %	**299.1**
15	92.6 %	**384.8**	-15	253.4	**288.7**	15	14.66	**306.4**	6	**454.8**	15	8.63	**388.3**	-15	2.78	**368.2**	10.0	91.9 %	**339.2**

The Project is most sensitive to commodity price fluctuation and recoveries in the Concentrator and the Salt Roast Plant and least sensitive to capital expenditure. The NPV for a Rand/US$ exchange rate of ZAR 14.66 to US$ 1 (a 15 % weakening), increases the pre-tax NPV to US$ 83.5 million real (post-tax US$ 306.4 million real) and the real pre-tax IRR to 28.10 % (post-tax 23.00 % real) The greatest risk to the financial viability of the Project is therefore fluctuations in commodity prices.

The calculated financial results of the Project, based on the assumed techno-economic parameters, are robust. The average earnings before interest, taxes, depreciation and amortisation (EBITDA) margin for the Project is above 50 %, giving the Project a significant buffer in times of low commodity prices. This will ensure the Project's debt to equity ratio can be maximised.

During the DFS, the Project team must prepare geared financial models to test the maximum debt to equity ratio which the Project can bear. The higher this ratio, the greater the effective equity return will be to shareholders of the Project. The Project team should also investigate all opportunities to optimise the Project's tax regime, seek Government development grants and maximise funding from development agencies, all of which would have the effect of increasing the ungeared NPV and IRR, thereby maximising the compounded annual return to the holders of equity.

On the basis of these results, it is recommended from a financial perspective that the Project is pursued and work on the DFS is commenced.

1.16 Interpretation and Conclusions

1.16.1 Mining

The Project is a high grade vanadium deposit that is readily mineable and can be easily accessed with the minimum of infrastructure requirements. The proposed rate of extraction is not excessive and well within the capabilities of typical earthmoving equipment. The open pit mining method to be used is well understood by mining contractors who are experienced in operating in similar geological conditions and thus the design of new or significantly modified mining techniques or specialised equipment is not required.

Should the Project advance to a DFS, the mining contracting companies will be requested to tender actual costs with the understanding that the final two selected companies be actively involved in the mine design and production scheduling phase.

1.16.2 Mineral processing, metallurgical testwork and recovery methods

The salt roast process is the vanadium industry standard for the processing of titano-magnetite ores from the Bushveld Complex in South Africa. As a result, the salt roast process was pre-selected for the PFS, having been proven on a commercial scale by existing facilities over many decades of operation. While alternate technologies were not evaluated as part of the current study, various high level trade-off and optimisation studies were undertaken to improve both the operability and cost effectiveness of the selected process. This resulted in a number of high level process decisions being made which include the following:

- the selection of a single rotary kiln as opposed to multiple kilns to minimise ancillary equipment cost and operational complexity;
- the use of Circulating Dry (Lime) Scrubber (CDS) technology for kiln off-gas SO_2 abatement – the technology was determined to be the lowest cost, least complex and lowest footprint technology and also provided a cost effective solution for the evaporation of process plant bleed streams; and
- the use of ammonium metavanadate (AMV) as opposed to ammonium polyvanadate (APV) precipitation since AMV is the industry standard for the production of high purity vanadium pentoxide and allows for less complex operability.

The process design criteria were selected based on a combination of previous testwork data and industry standards. High level benchmarking of the mass and energy balance and equipment size calculations has shown the proposed plant design to correspond to existing operations at the selected plant production.

1.16.3 Residue disposal facilities and stockpiles

The Magsep tailings, calcine tailings, lime cake and sodium sulphate and the hanging wall material require storage within the mining license area. A self-raised TSF has been proposed for storage of the Magsep tailings. The calcine tailings, lime cake and sodium sulphate will be trucked and dry stacked at a separate dry stack facility. Two stockpiles are required for the hanging wall, each to be placed in close proximity to the mining pits to limit haulage distance.

The storm water and return water infrastructure has been appropriately sized for both the calcine DSF and the Magsep TSF.

1.16.4 Environmental studies, permitting and social or community impact

Impact identification is performed by determining the potential source, possible pathways and receptors. In essence, the potential for any change to a resource or receptor (i.e. environmental aspect) brought about by the presence of a Project component or by a Project-related activity has been identified as a potential impact. The potential impacts associated with the proposed Project activities were considered and the potential perceived impacts by I&APs were identified during the consultation process. The significance, probability and duration of the potential impacts were assessed, once the detailed specialist studies were completed.

The most significant negative impacts identified are:

- loss of land capability and land use;
- loss of terrestrial habitat;
- loss of wetland habitat and functionality;
- the change in the sense of place and visual aesthetic of the Project Area due to the change in land use and development of structures such as the open pits;
- indirect impacts on heritage sites;

- groundwater contamination due to poor quality seepage from the waste rock dumps, residue and product storage facilities and operational areas (workshops, stores etc.);
- blasting related impacts (ground vibration, air blast, noxious fumes and fly rock) on people, structures (houses) and animals; and
- increase in ambient noise levels at sensitive receptors in the area.

The Project risks that were identified are listed below:

- Hydrocarbon spill from vehicles and machinery or hazardous materials or waste storage facilities;
- Spills/leaks from pipelines, tailings dam, hazardous materials or waste storage facilities;
- Mine flooding due to no dewatering;
- Fire and explosions;
- Resistance from Traditional Council and communities against the Project;
- Increased closure liability from the abandoned granite mine which is situated adjacent to the Project Area; and
- Delay due to the accidental uncovering or finding of unmarked grave(s).

The final EIA Report for the MPRDA and NEMA authorisation process for Pamish was submitted to the DMR for consideration on 22 October 2015. According to the EIA 2014 Regulations, the DMR needs to make a decision within 107 days, upon receipt of the final EIA Report. The DMR should therefore make a decision by end of January 2016 (assuming the DMR remains within legislative timeframes).

Once established, SaltCo will need to apply for the relevant environmental authorisations for activities associated with the salt roast plant

1.16.5 Financial valuation

The financial results of the Project are robust, based on the techno-economic parameters set out above. This is underlined by the short discounted and undiscounted payback periods for the Project. The average earnings before interest, taxes, depreciation and amortisation (EBITDA) margin for the consolidated Project is above 50 %, giving the Project a significant operating profit buffer in times of low commodity prices and will ensure the Project's debt to equity ratio can be maximised.

1.16.5.1 Risk and Opportunities

The Project is most sensitive to commodity price fluctuations and recovery in the Salt Roast Plant, and least sensitive to capital expenditure. The greatest risk to the financial viability of the Project is therefore fluctuations in commodity prices.

Once the Project is fully operational and a discount rate more commensurate with an operational WACC is applied, significant NPV will be added to the shareholders of the Project (at a real discount rate of 6 % the NPV of the Project is US$ 698.4 million on a pre-tax basis and US$ 454.8 million on a post-tax basis. A Rand/US$ exchange rate of ZAR 14.03 to the US$ (a 10 % weakening from the assumed long term rate of R12.75/US$), increases the real NPV to

US$ 463.7 million and the real IRR to 27.1 % on a pre-tax basis and US$ 292.1 million and 22.2 % on a post-tax basis.

During the DFS, the Project team must prepare geared financial models to test the maximum debt to equity ratio the Project can bear. The higher this ratio the greater the effective equity return will be to shareholders of the Project. The Project team should also investigate all opportunities to optimise the Project's tax regime, seek Government development grants and maximise funding from development agencies, all of which would have the effect of increasing the ungeared NPV and IRR, thereby maximising the compounded annual return to the holders of equity.

1.17 Recommendations

1.17.1 Geology and Mineral Resource Estimate

Since the PFS was commissioned, BML has carried out additional exploration work on the MML, MML HW and the A-B Zone VTM mineralisation. Should a decision be made to proceed with the DFS, the findings of this additional exploration work should be incorporated into an updated Mineral Resource Estimate.

1.17.2 Mining

In order to increase the level of confidence beyond the current levels, it is recommended the following be undertaken:

- the running of numerous models, using suitable mine planning software, to determine the optimal mining shell,
- the use of criteria for ore/waste determination in the planning and design of the proposed mining operation, including an optimised grade/quality cut-off waste rock handling policy;
- the definition of the final mineable Ore Reserves; from the final optimal open pit design;
- the optimisation of the LoM production rate detailing the basis of the selected production rate; and
- a detailed mine schedule reflecting the production phases and selected mining sequences, the design methodology adopted and the processes used to optimise the sequence and schedule for the entire deposit.

The benefits from undertaking the above modelling will be that a greater level of accuracy will be attained with regards to the depletion of the two. The mining operations could then be optimised to enable the delay of capital expenditure for the opening of Pit 2.

The potential for the concentrator to process the Zone A, B, C and MML parting stockpiled during the first 30 years of mining also exists. This will require a separate evaluation. As the feedstock ore will be placed close to the RoM tip, the cost of reclaiming the stockpiled material will be relatively low compared with primary mining.

1.17.3 Mineral processing, metallurgical testwork and recovery methods

The following section describes the design opportunities to be investigated in the next study phase (DFS) and provides recommendations for testwork activities.

1.17.3.1 Trade-off studies for recovery plant

Concentrator

There are a number of plant design related trade-off studies required including:

- stockpile design - a philosophy relying primarily on mobile equipment versus one with automated feed systems; and
- RoM particle size distribution (PSD): Obtaining a finer RoM PSD from blasting versus increasing the crushing requirements.

There are also a number of technical processing options and details related to material properties that require further investigation:

- comminution options:
    - conventional three stage crushing and ball milling;
    - primary crushing followed by SAG milling and ball milling;
    - primary crushing followed by HPGR and ball milling;
- beneficiation options:
    - pre-concentration prior to ball milling;
    - scavenging;
- dewatering:
    - dewatering rates of magnetic and non-magnetic fractions;
    - the potential addition of paste thickening;
- material handling
    - bulk material handling properties; and
    - rheological studies of the magnetic and non-magnetic slurries

Salt Roast Plant

The potential trade-off studies for the salt roast plant include:

- AMV vs APV precipitation – while a preliminary analysis was completed as part of the current PFS, a more detailed trade-off study will be possible in the next project phase (DFS) based on additional recommended testwork which is described below;
- salt roast kiln reagents - trade-off the capital savings of not including an SO_2 scrubber system in the kiln off-gas handling against the increase in operating cost and waste disposal cost associated with the exclusive use of sodium carbonate in the salt roast kiln;

- calcine cooling - trade-off the capital savings of not including a calcine cooler and PLS evaporator against the increase in operating cost of coal and increased capital cost for mill off-gas handling associated with the removal of calcine cooling.

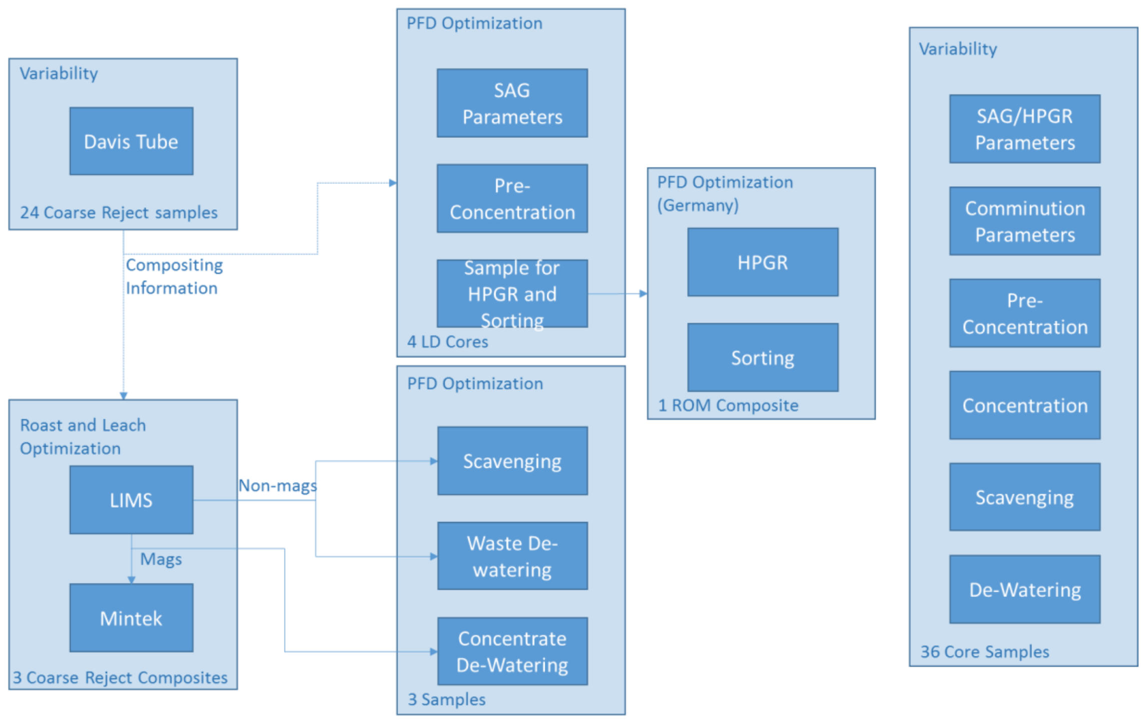
1.17.3.2 Testwork studies for recovery plant

The following section describes the proposed testwork campaign to be completed as part of the next project phase (DFS).

Concentrator Testwork

The recommended test work for the concentrator is summarised in Figure 1-5.

Figure 1-5
A high level overview of the recommended testwork



Salt Roast Plant Testwork

The recommended testwork for the salt roast plant is been summarised in Table 1-33.

Table 1-33
Recommended testwork

Unit Operation	Recommended Test Parameter	Comment
Comminution	Determine optimum feed concentrate particle size	The feed concentrate particle size must be investigated in conjunction with the salt roasting kiln and leach milling tests to determine the particle size that allows vanadium recovery to be maximised while minimising the comminution requirements.
Salt Roasting	Flux to Concentrate Ratio	The optimum flux addition that maximises vanadium recovery while minimising reagent requirements.
	Flux Composition	A 60:40 sodium sulphate: sodium carbonate ratio was used in the PFS. This ratio should be used as the base case for all residence time / temperature investigations. However, tests on alternate ratios should also be conducted as well as a 100 % sodium carbonate case which will be used for trade-off studies.
	Residence Time	Determine impact of residence time on vanadium recovery.
	Operating Temperature	Determine impact of hot zone temperature on vanadium recovery.
	Cooling Rate	Confirmation of cooling times required.
	Alumina dosing	Investigate the potential for alumina dosing during roasting which could prevent silica solubilisation downstream.
Leach Mill	Grindability	Determine optimum solids concentration in mill.
	Leach Kinetics	Perform leach cycle test to determine leach kinetics (optimum leaching time).
	Settling Rates/Filtration Rates	To confirm equipment design parameters.
Desilication	Desilication Chemistry	Confirm alumina requirements for precipitation reaction.
	Desilication Kinetics	Confirm residence time.
	Desilication rise rate and filtration rates	To confirm equipment design parameters.
AMV Precipitation	AMV Chemistry	Confirm reagent addition parameters for precipitation reaction.
	Temperature	Intermediate cooling required to prevent premature precipitation of sodium salt.
	pH	Test precipitation efficiency at different pH values.
	AMV Kinetics	Confirm residence time.
	AMV final product	Produce small quantity of final product to prove product quality.
APV Precipitation	APV Chemistry	Confirm reagent addition parameters for precipitation reaction.
	Temperature	Determine optimum reaction temperature
	pH	Test precipitation efficiency at different pH values.

Unit Operation	Recommended Test Parameter	Comment
	APV Kinetics	Confirm residence time.
	APV final product	Produce small quantity of final product to prove product quality.

1.17.4 Mine infrastructure

Finalisation of the delivery of bulk services with quantities and defined delivery dates is required as this will have a significant effect on the Project, with late or no delivery being a potential risk.

The use of treated waste and mine water should be studied further to minimise the dependency on the bulk water supplier.

Detail designs will be required for the roads in the following phase of work to ascertain the final vertical and horizontal alignment; this will have a bearing on the costing of this infrastructure.

Further work is required to ascertain the most cost effective manner in which engineering materials are to be obtained, whether it is through blending, from commercial sources or from borrow pits.

Further work is required to determine the actual volumes of each type of hazardous waste expected on site. The availability of waste facilities and contractors and the costs associated with the removal from site of these materials should be further investigated.

1.17.5 Residue disposal facilities and stockpiles

For the DFS Phase of the project it is recommended that the following be included:

- validation of the suitability of the RDFs and Stockpile sites by the EIA and other specialist studies;
- geotechnical investigation of the RDFs and Stockpile sites in order to confirm the type, extent and characteristics of the in situ materials and available construction materials;
- confirmation of the physical characteristics of the Magsep and Calcine tailings products based on laboratory testing of a representative sample;
- seepage and slope stability analyses based on the geotechnical characteristics of the Magsep and Calcine tailings and construction or in situ materials to confirm the stability of the facility and the optimisation of the toe drainage design;
- a further surface hydrology study of the area is required so as to ascertain any possible flood lines affecting the TSF and WRD sites;
- an overall water balance estimation should be undertaken for both the Magsep and Calcine RDFs to optimise the size of the SWD;
- liaison with the appropriate authorities to reclassify the two Stockpiles as requiring a Class D liner (a 150 mm base preparation layer comprising ripped and recompacted material on which the stockpile is placed) and not a Class C liner;
- the identification of borrow pit areas within the Project site for the sourcing of material for the construction of the clay liner and starter walls;
- possible further optimisation of each facility's preparatory works in terms of layout, footprint extent, phasing of the capital costs, etc.;

- the possibility of placing the Calcine or Magsep RDF over the wetland situated between them so as to improve both facilities layout and related capital and operating costs; and
- the compilation of a more detailed schedule of quantities describing the proposed preparatory works and the pricing of the schedules to a greater accuracy level.

1.17.5.1 Risks identified for the residue disposal facilities and hanging wall stockpile

Possible Project risks with regards to the RDFs and Stockpile facilities are:

- the Magsep TSF is positioned upstream of Pit 2. In the case of TSF failure, the open pit may be inundated with tailings. This could result in loss of life, contamination of workings, decreased operations capacity and could possibly result in expensive clean-up costs for the mine. This risk would need to be addressed, minimised and mitigated by the implementation of a Code of Practise and Emergency Preparedness Plan for the TSF;
- the PFS assumes a typical spigot self-build depositional strategy for the Magsep TSF. The geotechnical characteristics of the tailings will need to be assessed in the next phase of work to confirm if this depositional strategy is feasible. Should there be an insufficient percentage of coarse tailings, the depositional strategy would need to be reassessed;
- no stability analyses were undertaken for any of the facilities. Should stability be an issue, the configuration of RDFs would need to be reassessed;
- it is assumed that the clays found on site will be suitable for use as the clay section in the liner requirements (both Class A and Class C liners). This will need to be confirmed with geotechnical test work in the next phase of work. Should the clay prove to be unsuitable, this could result in increased capital costs;
- the Environmental Impact Assessment (EIA) of the RDFs and Stockpile sites may yield findings that would require any of the facilities to be relocated (e.g. rare species of flora found in the TSF basin area, unacceptable noise and/or dust impacts on surrounding communities etc.). The relocation of any of the facilities would mean the reassessment of the facility's configuration and may have a cost implication to the project;
- for this study, the Calcine, Lime Cake and Sodium Sulphate (LCSS) residue products are co-disposed in the same facility. The viability of this would need to be confirmed in the following phase of work, particularly if the marketability of the calcine material is investigated;
- the effectiveness of capping the LCSS with calcine so as to reduce the rate at which the Sodium Sulphate dissolves would need to be assessed. Should it not prove effective, this could result in higher operating costs for the Calcine RDF;
- the capacity to support vehicles (trafficability) on the calcine needs to be assessed and confirmed. Should a wearing course need to be introduced over each lift in the Calcine DSF, a significant increase in operating costs would be incurred; and
- the construction rates used in pricing the capital works need to be confirmed. In the next phase of the work local contractors would need to be engaged to obtain indicative construction rates for the various items on the bill of quantities.

1.17.6 Environmental studies, permitting and social or community impact

Pamish is in the process of obtaining the necessary environmental, social and community authorisations and aims to comply with all relevant legal requirements. Key tasks that should continue for the mining operation include the following:

- EIA and EMP in accordance with the MPRDA in support of the MRA and environmental authorisation (NEMA);
- Public Participation Process in accordance with the NEMA EIA 2014 Regulations;
- Specialist investigations in support of the EIA and EMP;
- IWULA in compliance with the NWA;
- approval from SAHRA in terms of the NHRA; and
- WML in compliance with the NEM:WA.

Key tasks to be completed by SaltCo for the salt roast plant include:

- environmental authorisation for Listed Activities as per NEMA Regulations (2014);
- WML in compliance with the National Environmental Management Waste Act 1998 (Act No. 59 of 2008) (NEM:WA);
- an Atmospheric Emissions Licence (AEL) application and authorisation from the Waterberg District Municipality as per the requirements of the National Environmental Management: Air Quality Act, 2004 (Act No. 39 of 2004) (NEM:AQA); and
- Water Use Licence Application (WULA) in compliance with the National Water Act, 1998 (Act No. 36 of 1998) (NWA).Based on the proposed scope of work, the total liability cost has been calculated at ZAR 196,678,285.

1.17.7 Financial valuation

The following recommendations are made in terms of this PFS report:

- on the basis of the financial results set out above, it is recommended that the Project is pursued and work on the DFS is commenced;
- that during the DFS, the Project team undertakes the following from a financial perspective:
    - prepare more detailed capital and operating cost estimates in line with DFS standards;
    - prepare geared financial models to test the maximum debt to equity ratio the Project can bear;
    - investigate all opportunities to optimise the Project's tax regime;
    - seek Government development grants; and
    - maximise funding from development agencies.