NI 43-101 Technical Report

PRELIMINARY ECONOMIC ASSESSMENT on the Mustavaara Vanadium project, Finland

Prepared for Strategic Resources Inc.

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Effective date: 4th May 2021
Execution Date: 10th June 2021

Certificate of Qualified Person – Ville-Matti Seppä, EurGeol

I **Ville-Matti J. Seppä**, EurGeol., do hereby certify that:

1. I am currently employed as a Section Manager of Geology & Mine Design at AFRY Finland Oy. Jaakonkatu 3, FI-01621 Vantaa, Finland.
2. This certificate is to accompany the report "NI 43-101 Technical Report, Preliminary Economic Assessment on the Mustavaara Vanadium project, Finland" for Strategic Resources Inc. with an effective date of 4th May, 2021.
3. I am a graduate from the University of Turku with a M.Sc. Degree in 2009 and have been professionally active since my graduation.
4. I am a European Geologist (#1286) licensed by the European Federation of Geologists.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purpose of NI 43-101.
6. I visited Strategic Resource's Mustavaara property in Finland on June 10th, 2020.
7. I am responsible for Section 1 to 15, 17 to 19 and 21 to 26in the report titled "NI 43-101 Technical Report, Preliminary Economic Assessment on the Mustavaara Vanadium project, Finland" with the effective date of 4th May, 2021, 2021.
8. I am independent of Strategic Resources Inc. applying all of the tests in Section 1.5 of NI 43-101.
9. I have prepared the previous technical report with an effective date of 14th September, 2020 on the property that is subject of this Technical Report.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. As of the date of the technical report, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 10th day of June 2021

"Original Signed"

M.Sc. Ville-Matti Seppä, EurGeol

Certificate of Qualified Person – Pekka Lovén

I, **Pekka Lovén**, MAusIMM(CP), MSc (Mining), do hereby certify that:

1. I am an independent consultant of PL Mineral Reserve Services, Alkutie 10 A1, Helsinki, Finland.
2. This certificate is to accompany the report "NI 43-101 Technical Report, Preliminary Economic Assessment on the Mustavaara Vanadium project, Finland" for Strategic Resources Inc. with an effective date of 4th May, 2021.
3. I graduated with M.Sc. degree in Mining Engineering from the Helsinki University of Technology in 1980. I have worked as a mining engineer for a total of 40 years since my graduation from the university.
4. I am a Member and Chartered Professional with the Australian Institution of Mining and Metallurgy (Member# 301822).
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purpose of NI 43-101.
6. I have visited the Mustavaara property in Finland between the 16th and 17th of November 2011.
7. I am responsible for Section 16 in the report titled "NI 43-101 Technical Report, Preliminary Economic Assessment on the Mustavaara Vanadium project, Finland" with the effective date of 4th May, 2021.
8. I am independent of Strategic Resource Inc.
9. I have prepared the previous technical report with an effective date of 14th September, 2020 on the property that is subject of this Technical Report.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. As of the date of the technical report, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 10th day of June 2021

"Original signed"

Pekka Lovén, MAusIMM

Certificate of Qualified Person – Eemeli Rantala, P.Geo

I **Eemeli Rantala**, P.Geo., do hereby certify that:

1. I am currently employed as a Senior Geologist at AFRY. Frösundaleden 2A 169 99 Stockholm, Sweden.
2. This certificate is to accompany the report "NI 43-101 Technical Report, Preliminary Economic Assessment on the Mustavaara Vanadium project, Finland" for Strategic Resources Inc. with an effective date of 4th May, 2021.
3. I am a graduate from the University of Turku with a B.Sc. Degree (2009) and M.Sc (2011) in Geology. I have been professionally active since 2009.
4. I am a Professional Geoscientist (#169691) registered with APEGBC (Association of Professional Engineers and Geoscientists of British Columbia).
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purpose of NI 43-101.
6. I have not visited Strategic Resource's Mustavaara property in Finland.
7. I am responsible for Section 20 in the report titled titled "NI 43-101 Technical Report, Preliminary Economic Assessment on the Mustavaara Vanadium project, Finland" with the effective date of 4th May, 2021.
8. I am independent of Strategic Resources Inc. applying all of the tests in Section 1.5 of NI 43-101.
9. I do not have any prior involvement to the property that is subject of the technical report.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. As of the date of the technical report, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 10th day of June 2021

"Original Signed"

M.Sc. Eemeli Rantala, P.Geo

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List of Abbreviations

asl	above sea level
AC	Alternating current
ADI	Adriana Resources Inc.
AFRY	AFRY Finland Oy
Al_2O_3	Aluminum oxide
Akkerman	Akkerman Exploration B.V.
BAT	Best Available Techniques
CaO	Calcium oxide
CAPEX	Capital expenditures
CB	Clarification basin
CEO	Chief executive officer
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CLGB	Central Lapland Greenstone Belt
cm	centimetre
CO_2	Carbon dioxide
CP	Chartered Professional
CSA	Canadian Securities Administrators
DC	Direct current
DD	Due diligence
DDT	Dings-Davis Tube
DGPS	Differential Global Positioning System
DTWR	Davis Tube Weight Recovery
E	East
€	Euro
EIA	Environmental Impact Assessment
ELY	Centre for Economic Development, Transport and the Environment
ESIA	Environmental and Social Impact Assessment

EurGEOL	Professional European Geologist with the European Federation of Geologists
Fe	Iron
FeV	Ferrovanadium
FS	Feasibility Study
g	gram
Ga	Giga annum
gps	Global positioning system
GTK	Geological Survey of Finland
ha	hectares
HF	High Frequency
hr	hour
ICP-OES	Inductively Coupled Plasma - Optical Emission Spectrometry
IPPC	Integrated pollution prevention and control
JORC	Joint Ore Reserve Committee
k	thousand
KAIELY	Centre for Economic Development, Transport and the Environment of Kainuu
kg	kilogram
km	kilometer
ktpa	Kilo tonnes per annum
kV	kilovolt
LAB	Eurofins Labtium Oy
lb	Pound
LGB	Lapland Greenstone Belt
LOM	Life of Mine
m	metre
Ma	Mega annum
MAusIMM	Member of the Australian Institute of Mining and Metallurgy
Mefos	Swerea Mefos Research Institute
mm	milimetre
MKOy	Mustavaaran Kaivos Oy
Mt	Million tonnes
N_2	Nitrogen
NI 43-101	National Instrument 43-101
NN	Nearest Neighbor
No	Number
NSR	Net Smelter Return
ODL	Ore disseminated layer
OLL	Ore lower layer
OML	Ore middle layer
OPEX	Operating expenditure
OUL	Ore upper layer
P	Phosphorous
PEA	Preliminary Economic Assessment
%	percent
PFS	Pre-Feasibility Study
PGE	Platinum Group Elements
POPELY	Centre for Economic Development, Transport and the Environment of North Ostrobothnia
PPC	ProspectOre Capital Corp.
ppm	parts per million
PSAVI	Northern Finland Regional Administrative Agency
pXRF	Portable X-ray fluorescence analyzer
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
REACH	Registration, Evaluation, Authorization and Restriction of Chemicals
ROM	Run of Mine
RR	Rautaruukki Oy
RSD	Relative Standard Deviation
§	Section
SAC	Special protection zone

SG	Specific gravity (g/cm^3)
SiO_2	Silicon dioxide
Strategic	Strategic Resources Inc.
t	tonnes
Ti	Titanium
TiO_2	Titanium oxide
TSF	Tailings Storage Facility
tpa	Tonnes per annum
Tukes	Finnish Safety and Chemicals Agency
μm	micrometre
US$	United States Dollar
V	Vanadium
V2O5	Vanadium Pentoxide
V:H	Vertical / Horizontal
VinMC	Vanadium content in the magnetite concentrate
VTM	Vanadiferous titanomagnetite
wt.%	Percentage by weight
WRSF	Waste Rock Storage Facility
XRF	X-Ray Fluorescence
3D	Three dimensional

1 Summary

1.1 Introduction

Strategic Resources Inc. (Strategic) commissioned AFRY Finland Oy (AFRY) to prepare a Preliminary Economic Assessment on the Mustavaara vanadium deposit in compliance with the Canadian Securities National Instrument 43-101 Standards of Disclosure for Mineral Properties and Form 43-101F1.

This PEA study is based on the mineral resources previously defined and disclosed by Strategic Resources Ltd. in September 2020. This study is an evaluation of the project's economics at PEA level.

This Technical Report was authored and supervised by Mr. Ville-Matti Seppä (EurGeol) of AFRY Finland Oy, Mr. Eemeli Rantala of AFRY Sweden ab and Mr. Pekka Lovén of PL Mineral Reserve Services. Authors are independent qualified persons (QP) as defined by Canadian Securities Administrators (CSA) *National Instrument 43-101 – Standards of Disclosure for Mineral Projects* (NI 43-101) and as described in Certificates of Qualified Persons of this Technical Report.

1.2 Location

The Mustavaara Project (Mustavaara or the Project) is located in north-central Finland. The Project is part of the Municipality of Taivalkoski, which is located approximately 650 km north of Helsinki and 180 km northeast of Oulu. The approximate centre of the Project is located at 65°49'N latitude and 28°08'E longitude.

1.3 Ownership and History

Mustavaara was discovered in 1957 when samples sent to Otanmäki Oy for analyses reported vanadium. Fieldwork from 1957 and 1958 lead to a discovery of a vanadium bearing magnetite deposit. Rautaruukki Oy took over the project and continued with more detailed exploration work. They drilled 56 drill holes from 1967 to 1971 and outlined a vanadium bearing ore zone and conducted a "Feasibility Study". Construction on the mine commenced in 1973. The open pit mine and roast-leach processing plant were operational from 1976-1985. The annual production peaked at 1.6 Mt of ore, producing 240,000 t of pelletized magnetite concentrate and 3,000 t of vanadium pentoxide. Operations were suspended in 1985 after a period of very low vanadium prices (Pöyry Finland Oy, 2012).

Akkerman Exploration B.V (Akkerman) was granted the claims in 2006. Since obtaining the claims, no work is known to have taken place. There was a series of option agreements with Adriana Resources Inc. who transferred their rights to ProspectOre Capital Corp. and then to Vanadis Mines Oy. The agreement(s) were finally terminated and Akkerman regained ownership of the project. On May 19th 2011, Akkerman entered into a purchase agreement, whereby 100% of its Mustavaara mineral rights were sold to Mustavaaran Kaivos Oy (MKOy)

(Pöyry Finland Oy, 2012). Shortly after, on May 28th, 2011, MKOy filed an application for a mining license over the Mustavaara mine and surrounding area and started an exploration and due diligence program. In the fall of 2011, MKOy drilled 17 diamond drill holes and collected airborne magnetic data over the property. In 2012, MKOy completed a pre-feasibility study and in 2013 MKOy performed a pit optimization study. MKOy then proceeded to start permitting the mine. In 2016, environmental and water permits were issued to MKOy and appealed. The courts overturned the appeal and granted the permits again on June 14th, 2018. Later that year MKOy changed its name to Ferrovan Oy. The permits remain under MKOy's name but could be transferred to the next company.

On February 10th 2020, Strategic Resources Inc. announced that it had successfully applied for Reservations over the Mustavaara mine area. The company also acquired all of the intellectual property together with the core samples from the 2011 drill campaign and storage facilities associated with Mustavaara from the bankruptcy estate of Ferrovan Oy (Strategic Resources, 2020).

The Mustavaara project consists of three exploration reservations and has a surface extent of 2,660 hectares. All reservations have been approved by Finnish Safety and Chemicals Agency (Tukes) and are valid until February 9th, 2022.

1.4 Geology and Mineralization

The Mustavaara V-Fe-Ti deposit is part of a large, approximately 2.44 Ga old, layered intrusion complex, known as the Tornio-Näränkävaara intrusion belt (Karinen, Hanski & Taipale, 2015). The deposit itself belongs to the Koillismaa Intrusion. The Koillismaa Intrusion is composed of various distinct blocks of sheet-like layered intrusions, which were separated by tectonic movements (Karinen, 2010). The 4 km wide and 20 km long Porttivaara block is the most well-known part of the intrusion and the Mustavaara deposit is located within it (Karinen, Hanski & Taipale, 2015).

Within the Porttivaara block the location, size, and shape of the magnetite-gabbro horizon has been interpreted from airborne and ground magnetic survey data. The gravimetric models show that the magnetite-gabbro horizon extends down to depths of at least 2,000 m (Saviaro, 1976; Ruotsalainen, 1977; Piirainen, et al., 1978).

The layered series of the Porttivaara block is composed mainly of norites, gabbronorites, leucogabbros and anorthosites. The deposit is comprised of four conformable ore layers; disseminated ore and the upper, middle, and lower ore layers, with a total thickness of 80 m. The ore-bearing Mustavaara magnetite-gabbro occurs in the upper part of the layered series, surrounded by anorthosite. Genetically, the vanadium containing magnetite gabbro is considered to be of magmatic origin formed as a segregation from an iron-rich liquid.

1.5 Project Status

The Mustavaara Project is an advanced exploration project that has seen extensive exploration including geophysical surveys and drilling. On top of the drill defined Fe-V-T mineralized zone, a property wide magnetic belt has been identified for further drill testing.

1.6 Mineral Resource Estimates

For the basis of this study, the mineral resource estimate from September 2020 was used. The mineral resource estimate was generated using drill hole sample assay results and the interpretation of a geologic model based on data collected for the JORC mineral resource estimate done by Outotec (Finland) Oy, dated August 30th, 2013. There have not been any new exploration activities concerning the property and the end products (ferrovanadium and pig iron) remain consistent with the 2020 resource report.

The Mineral Resource was calculated in accordance with the Canadian National Instrument for the Standards of Disclosure for Mineral Projects (NI 43-101) requirements.

The Mineral resource estimates at Mustavaara are presented in Table 1-1.

Table 1-1 Mustavaara Estimated Mineral Resources as of the September 14th, 2020 @ 11.0% Magnetite cut-off.

Resource Class	Tonnes	Average Grade				Contained Metal		
		Magnetite	VinMC	Ti	Fe	VinMC	Ti	Fe
	Mt	(%)	(%)	(%)	(%)	(kt)	(kt)	(kt)
Measured Mineral Resource	**64.0**	**15.41**	**0.91**	**3.75**	**63.3**	**90**	**370**	**6 244**
Indicated Mineral Resource	**39.7**	**15.27**	**0.88**	**3.53**	**62.8**	**53**	**214**	**3 805**
Total M&I Mineral Resource	**103.7**	**15.36**	**0.90**	**3.67**	**63.1**	**143**	**584**	**10 049**
Inferred Mineral Resource	**42.2**	**15.11**	**0.92**	**3.75**	**62.3**	**59**	**239**	**3 971**

Note: VinMC refers to vanadium in magnetite concentrate; Ti refers to titanium in magnetite concentrate and Fe to iron in magnetite concentrate.

1.7 Mine Plan

The Mustavaara deposit mine plan was developed based on the assumption that the mine is operated by an owner operated fleet. The selected conventional truck and shovel excavation is a commonly used mining method in Finland and there is a widely available skilled work force. The mining will start by expanding the existing open pit to west and eventually the maximum open pit dimensions are reached by utilizing a one ramp configuration in the open pit. Table 1-2 presents the mine production schedule with a 11% magnetite cut-off.

Table 1-2 presents the designed yearly Life of Mine plan @ 11.0% Magnetite cut-off.

Mining Period	OVB removal M3	Ore kt	Waste kt	Magnetite %	V in situ %	Fe in situ %	VinMC %	Fe %
1	2 896 575	-	-					
2	1 146 942	1 959	4 512	16.23 %	0.15 %	10.5 %	0.93 %	64.59 %
3		3 250	7 482	15.11 %	0.14 %	9.7 %	0.93 %	63.93 %
4		3 250	7 482	15.64 %	0.14 %	10.0 %	0.88 %	63.92 %
5		3 259	7 503	13.70 %	0.13 %	8.9 %	0.96 %	64.83 %
6		3 250	7 482	15.18 %	0.14 %	9.7 %	0.94 %	63.59 %
7		3 250	7 482	15.48 %	0.14 %	9.9 %	0.89 %	64.04 %
8		3 250	6 132	13.70 %	0.13 %	8.8 %	0.93 %	63.95 %
9		3 259	6 149	13.53 %	0.13 %	8.6 %	0.93 %	63.60 %
10		3 250	6 132	16.03 %	0.15 %	10.1 %	0.94 %	63.29 %
11		3 250	6 132	14.01 %	0.12 %	8.7 %	0.86 %	62.43 %
12		3 250	6 132	13.19 %	0.13 %	8.3 %	0.95 %	63.23 %
13		3 259	4 209	15.13 %	0.14 %	9.5 %	0.92 %	62.99 %
14		3 250	4 197	14.58 %	0.13 %	9.0 %	0.86 %	62.06 %
15		3 250	4 197	13.73 %	0.13 %	8.5 %	0.92 %	62.17 %
16		3 250	4 197	13.34 %	0.12 %	8.4 %	0.93 %	63.01 %
17		3 259	4 209	15.12 %	0.14 %	9.6 %	0.93 %	63.26 %
18		3 250	4 197	13.17 %	0.12 %	8.1 %	0.88 %	61.85 %
19		3 250	4 197	13.05 %	0.12 %	8.3 %	0.93 %	63.28 %
20		3 250	4 197	13.11 %	0.12 %	8.4 %	0.94 %	63.85 %
21		3 259	1 517	11.60 %	0.11 %	7.4 %	0.93 %	64.16 %
22		802	-	11.16 %	0.10 %	7.2 %	0.91 %	64.49 %
Totals	4 043 517	64 553	107 742	14.15 %	0.13 %	9.0 %	0.92 %	63.39 %

NOTE the scheduled tonnes and grade do not represent an estimate of Mineral Reserve. VinMC refers to vanadium in magnetite concentrate and Fe to iron in magnetite concentrate.

A detailed hydrological study has not yet been performed and it is recommended to be done in the next project phase. The created life of mine plan (LOM) supports approximately 20 years of production at Mustavaara. The planned annual mining rate is 3.25 Mt of processed material and the total strip ratio during the life of mine is 1.7. Waste rock mining varies between 4.2 to 7.5 Mt during the main production period.

The open pit optimization results that were used as a base for mine design are based on Measured, Indicated and Inferred mineral resources. The Company further cautions that the PEA is preliminary in nature. No detailed mining study has been completed to support reserve estimation. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

1.8 Recovery Methods

Ferrovanadium production process consists of a concentrator plant at Mustavaara and smelter / hydrometallurgical plant in Raahe. Main products are ferrovanadium (FeV80) and pig iron.

Mineral processing pilot plant test work, comminution test work, smelting test work and hydrometallurgical test work have been conducted with Mustavaara ore and concentrate during years 2008-2014. Results from the test work are summarized in chapter 13.5. Test work results indicate that production of ferrovanadium is possible from the Mustavaara ore. Process design is mainly based on assumptions received from the test work.

The concentrator plant process is based on two-stage crushing, three-stage grinding and multi-stage magnetic separation to produce iron/vanadium concentrate. Direct smelting and selective oxidation are used to bring vanadium to a suitable form (vanadium slag) to act as a feed material for the roast-leach process. Pig iron is produced as a by-product of smelting process. The roast-leach process is used to produce vanadium pentoxide (V_2O_5) from vanadium slag. Vanadium pentoxide is fed to the aluminothermic reduction. The vanadium product from the aluminothermic reduction is ferrovanadium (FeV80).

Process design is based on average annual throughput of 3.25 Mtpa ore to the concentrator plant and estimated amount of average annual concentrate production of 505 ktpa with 63.4% Fe grade and 0.92% V grade.

Estimated amount of annual FeV80 production is 4,577 tpa with vanadium recovery from the concentrate to the FeV80 estimated to be 78.9%. Annual pig iron production is estimated to be 329 ktpa. Other possible by-products include TiO_2 slag (83 ktpa), Ca-Al slag (8 ktpa) and sodium sulphate (20 ktpa).

1.9 Infrastructure

The planned waste rock storage facility is located to east of the open pit, partly at the north-east slope of the Mustavaara hill. The area has been partly used for waste rock deposition during the previous mining operation. Due to the historic mining, on the Mustavaara, there is an existing tailings storage facility. The new deposition is planned on top of the existing deposition area. The general layout of Mustavaara mining area is presented in Figure 1-1.

The total installed power of the concentration process main equipment has been estimated at 11.5 MW and total peak power including all processes is estimated at 18 MVA for 487 ktpa concentrate production. The required electric power will be provided through connection to the local 110 kV power grid. The new 110 kW power line covers a distance of approximately 32 km and it will be built to connect the mine site to an existing switchyard at Posio municipality area. The power line route will follow the same path that was used during the former mining operations in Mustavaara area. Water for processing would come from a raw water basin and from the Sirniönlampi lake.

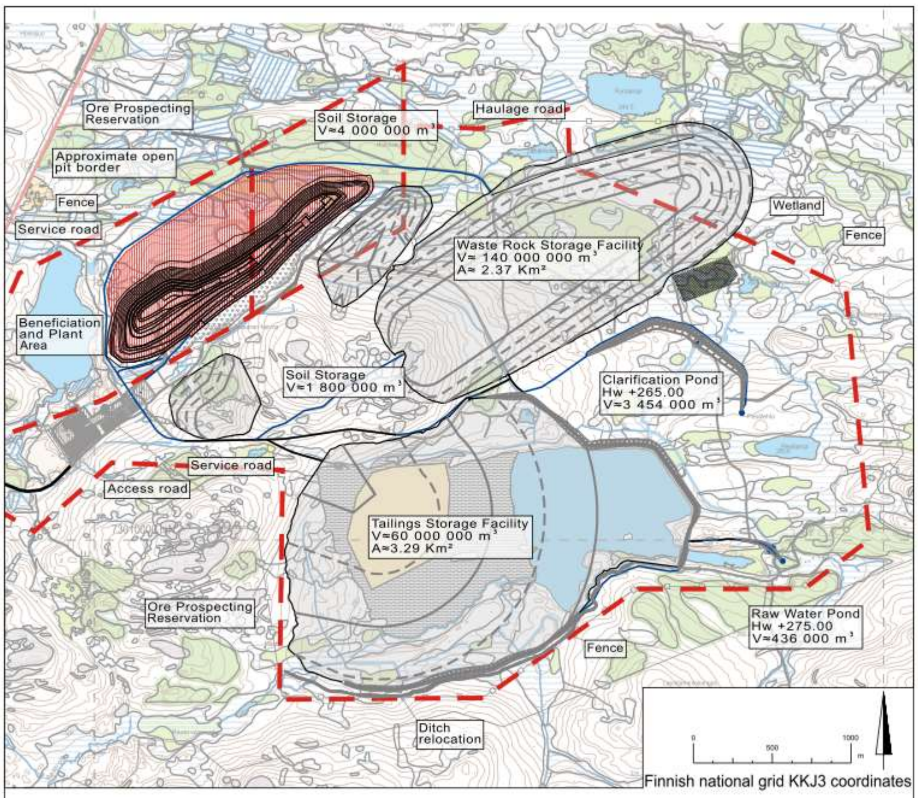

Figure 1-1Mustavaara Mining Area, Site Layout

1.10 Environmental Studies, Permitting and Social or Community Impact

The Mustavaara mine is a brownfield site. The project area has an existing (partly water-filled) pit, existing waste rock storage facilities and an existing tailings storage facility.

Extractive waste materials are not significantly sulphide-bearing and therefore acid rock drainage risk is minimal, but the site causes some metal leaching impacts on the nearest (unclassified) small downstream watercourses. The conclusion is based on extractive waste characterisation from 2009, including assessment of acid generation potential, neutralisation capacity and analysis of total concentrations. There is also monitoring data of downstream watercourses supporting this conclusion. Current impact of the closed mine site is very small on the larger status-classified watercourses further downstream. Additional waste quantities related to future operations are likely to increase these impacts, which must be taken into consideration in planning of the mitigation measures. Complementary geochemical data is needed to proceed to actual planning of the mitigation measures (to be done during PFS stage).

Based on the information available in the reviewed documents, the project is unlikely to significantly impact nearest nature protection areas or groundwater area, but there is one significant protected bird species issue within the project area where mitigation strategies will need to be evaluated.

Key environmental and social issues are related to permitting and protected bird nesting sites. The mine site has an existing environmental and water permit, but for a smaller yearly excavation than discussed in this PEA. Permit doesn´t directly limit the length of the mine life, but the operations described in the permit application are based only on 15 years life of mine. Preparation for amendment permitting will be needed for the longer life of mine and larger extractive waste quantities. This will likely trigger an ESIA process (which takes place prior to the environmental permitting). Amendment permitting requires at least geochemical and hydrogeological assessments (in addition to the technical planning to be carried out at PFS stage). Potential start of production under existing environmental permit could be considered, but it is linked to another permitting matter: according to the permit conditions 8 and 9, no constructions or changes in natural conditions are allowed within 400 m radius from local protected bird nesting sites and during the potential nesting season (15th February – 31st July) no constructions, car transports, noise or emissions are allowed within 1,000 m radius from the nest sites if the birds are actively nesting. These permit terms are assumed to complicate and limit the operations (for example constructions and dam safety control). If LOM (20 years) tailings quantity would be deposited as wet tailings, adding capacity to the old TSF, constructing and operating the TSF would take place too close to the existing nest sites.

Applying for change in permit conditions will be required, if existing TSF and wet tailings deposition will be used. According to the information available in late 2020, nest sites had not been used for years and re-permitting seemed feasible. According to a more recent update received, one of the nests is currently in use, which increases the risk that change of permit conditions might not be successful. Strategic is taking action to mitigate this issue going forward. Other alternatives are another TSF location or another tailings deposition alternative (which requires smaller footprint).

There are no resettlement issues or indigenous people issues within the project area. Stakeholders' concerns are primarily related to the water quality and fishing issues.

The planned smelter site is in an existing industry area in Raahe town, on Bothnian Bay coast. There is no environmental permit for this site, but the land use plan is approved. The site is already strongly impacted by the existing industries and from that perspective stakeholder issues are expected to be minor, assuming stakeholder engagement work is carried out properly.

1.11 Capital and Operating Costs

Development capital of 597 MEUR is estimated for the project including purchase of mine equipment. Sustaining capital expenditure is estimated at 94 MEUR including the closure costs.

Development capital costs are split as follows:

Mine:	28 MEUR
Beneficiation plant:	81 MEUR
Infrastructure and utilities:	43 MEUR
Smelting plant:	321 MEUR
Indirect costs:	70 MEUR
Contingency (10%):	54 MEUR
Total development capital:	**597 MEUR**

Annual total operating cost is estimated to be 140.4 MEUR. Opex and co-product cash costs net of by-products per FeV80 kg are 15.3 EUR/kg and 14.6 EUR/kg respectively. Opex and co-product cash costs net of by-products per pig iron metric tonne are 213.3 EUR/t and 203.6 EUR/t respectively. Cash cost values include royalties and by-product income.

The largest contributor to the opex is the smelting & hydrometallurgical operation, with a 68.5% share of total opex. The second largest contributor is mining operation with a 13.6% share of total opex.

1.12 Economic Analysis

Economic analysis concluded with following results

Pre-tax

NPV(8): 286 MEUR

IRR: 13.9%

Payback: 5.9 years

Post-tax

NPV(8): 190 MEUR

IRR: 12.2%

Payback: 6.4 years

Economic analysis uses pricing of 32 USD/kg V for FeV80 and 450 USD/metric tonne for pig iron. Smelter by-products (CaAl-slag, Ti-slag and $NaSO_4$) are included in the analysis with estimated prices of 150 USD/metric tonne, 9 EUR/metric tonne and 400 EUR/metric tonne, respectively. Including by-products increases post-tax NPV by approximately 31 MEUR. An exchange EUR:USD exchange rate of 1.18 was used in the analysis.

1.13 Conclusions

The remarks and conclusions regarding the Mustavaara project are summarized below:

- The drilling and sampling to date supports the mineral resources estimate and there is sufficient information to be used as a basis for the mineral resource estimate and for this PEA study.
- The drilling pattern and spacing covers the known measured, indicated and inferred mineral resources. A limited amount of new drilling down dip of the historic drilling could upgrade the indicated and inferred resources. The down-dip continuation of the magnetite gabbro remains open and is expected to continue with the same thickness and grade in the same kind of geological framework as with the known mineralization.
- The deposit geology and style of mineralization is well understood and the property has a history of successful mining activities.
- Land use planning for the potential reopening of the mine is at an advanced state and is a major upside for the project, as there would be limited delays to be expected in land planning matters.
- The created mine plan supports ca. 20 year LOM.
- The mineral processing concept is well understood and studied.
- Smelting and hydrometallurgical processing concepts for ferrovanadium (FeV80) production are well known.
- Applying for change in environmental permit conditions is necessary, if existing TSF and wet tailings deposition will be used. Other alternatives

would be another TSF location or completely another tailings deposition alternative (which requires smaller footprint).

Based on the resource and economic models described in this report, it is the QP's opinion that this report is suitable for Preliminary Economic assessment of the Mustavaara project. The PEA results justify the further study of this project and it is possible to advance into a pre-feasibility study. However, the nesting of endangered species in the vicinity of TFS requires option studies to be made.

The PEA study is preliminary in nature, it includes inferred mineral resources that are geologically too speculative to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. No mineral resources described in this PEA have been converted to reserves. Mineral resources that are not mineral reserves have no demonstrated economic viability. There is no certainty that the preliminary economic assessment will be realized.

1.14 Recommendations

Based on the mineral resource estimate and the PEA study results, further study of the Mustavaara deposit and advancement to a Pre-Feasibility study is recommended.

To assist the preparation of the pre-feasibility study, a detailed rock mechanics study is recommended to be completed to confirm the geotechnical parameters for the open pit design. A full hydrological study of the Mustavaara deposit is also needed.

If current process route is selected, further investigation (metallurgical test work and modelling) is recommended to confirm recovery estimates and mass and heat balance. Alternative processing concepts should be studied in more detail to evaluate potential capex/opex savings. Detailed process recommendation list is found in chapter 17.5.

As stated in chapters 18 and 20, any subsequent study phases should include more detailed water quality source-term assessments. Process water quality source terms should be based on water analysis from process metallurgical tests. Furthermore, full re-modelling of site water and loading balance is recommended. The loading balance should be used for predictions that are recommended to be done. Consequently, the mine closure plan needs to be updated.

Additionally, geotechnical, rheological, and geochemical testing is required for tailings samples obtained from the updated process metallurgical tests. In addition to this, geotechnical investigations are needed from the tailings storage facility area, especially from the dam locations.

Applying for change in environmental permit conditions is necessary, if existing TSF and wet tailings deposition will be used. Other alternatives would be another TSF location or completely another tailings deposition method (which requires smaller footprint).

Generally environmental impacts are assessed for a smaller version of the projects that is discussed in this PEA. This means a general need to produce information needed for the updated impact assessments according to the new scale of the project. For example, careful water impact assessment is needed, taking into consideration the increase in LOM waste quantities. Possible additional water treatment or new discharge point in larger river may be required.

Cost estimate for recommended work programs for next phase is presented in Table 26-1. Cost of the Pre-Feasibility study includes items described on Appendix 6.

Table 1-3. Cost estimate for future work programs

Items	Cost Estimate
Rock mechanical study	80 000 €
Full hydrological study	120 000 €
Water quality source-term assessment	50 000 €
Re-modelling of site water and loading balance	20 000 €
Tailings test work	105 000 €
Comminution testing	50 000 €
Metallurgical test work and modelling	150 000 €
Pre-Feasibility study	800 000 €
total	1 375 000 €

Cautionary Note Regarding Forward-looking Information and Statements

Information and statements contained in this Technical Report that are not historical facts are "forward-looking information" or "forward-looking statements" within the meaning of Canadian securities legislation and the U.S. Private Securities Litigation Reform Act of 1995 (hereinafter collectively referred to as "forward-looking statements") that involve risks and uncertainties. Examples of forward-looking statements in this Report include information and statements with respect to: Strategic's plans and expectations for the Mustavaara Project, estimates of mineral resources, and possible related discoveries or extensions of new mineralization or increases or upgrades to reported mineral resources estimates and budgets for recommended work programs.

In certain cases, forward-looking statements can be identified by the use of words such as "budget", "estimates", or variations of such words or state that certain actions, events or results "may", "would", or "occur". These forward-looking statements are based, in part, on assumptions and factors that may change, thus causing actual results or achievements to differ materially from those expressed or implied by the forward-looking statements. Such factors and assumptions include, but are not limited to, assumptions concerning base metal prices; cut-off grades; accuracy of mineral resource estimates and resource modelling; reliability of sampling and assay data; representativeness of mineralization; proposed mine plan and mining method; dilution and extraction recoveries; projected metallurgical recovery rates; infrastructure requirements; capital, operating and sustaining cost estimates; the projected LOM and other design attributes of the project; the NPV and IRR and payback period of capital; capital; future metal prices; the timing of the environmental assessment process; accuracy of metallurgical test work and timely receipt of regulatory approvals.

Forward-looking statements involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements of Strategic to be materially different from any future results, performance or achievements expressed or implied by the forward-looking statements. Such risks and other factors include, among others, fluctuation in the price of base and precious metals; expropriation risks; currency fluctuations; requirements for additional capital; government regulation of mining operations; environmental, safety and regulatory risks; unanticipated reclamation expenses; title disputes or claims; limitations on insurance coverage; changes in project parameters as plans continue to be refined; failure of plant, equipment or processes to operate as anticipated; accidents, labour disputes and other risks of the mining industry; competition inherent in the mining exploration industry; delays in obtaining governmental approvals or financing or in the completion of exploration, development or construction activities, as well as those factors discussed in the sections entitled "Risks and Uncertainties" in Strategic's annual Management's Discussion and Analysis.

Although Strategic and the authors of this Report have attempted to identify important factors that could affect Strategic and may cause actual actions, events or results to differ, perhaps materially, from those described in forward-looking statements, there may be other factors that cause actions, events or results not to be as anticipated, estimated or intended.

There can be no assurance that forward-looking statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements. Accordingly, readers should not place undue reliance on forward-looking statements. The forward-looking statements in this Report are based on beliefs, expectations and opinions as of the effective date of this Report. Strategic and the authors of this Report do not undertake any obligation to update any forward-looking information and statements included herein, except in accordance with applicable securities laws.

2 Introduction

AFRY Finland Oy (AFRY) has been commissioned by Strategic Resources Inc. (Strategic) to prepare a Preliminary Economic Assessment on the Mustavaara vanadium deposit in compliance with the Canadian Securities National Instrument 43-101 Standards of Disclosure for Mineral Properties (NI 43-101) and Form 43-101F1.

This report has an effective date of 4th May, 2021. This technical report is based on the data collected and prepared for the JORC compliant Pre-Feasibility Study (PFS) for the Mustavaara vanadium iron project in 2011 (Pöyry Finland Oy, 2011), on the report "Update Resource estimation and preliminary mining study of the Mustavaara deposit for Mustavaaran Kaivos Oy", (Outotec, 2013) and on the most recent NI 43-101 technical report on the Mustavaara Vanadium project (AFRY 2020).

Ville-Matti Seppä and Pekka Lovén were independent "qualified persons" (QPs) responsible for preparing the mineral resource estimate that is used as basis for this PEA study (AFRY 2020).

The most recent site visit was conducted by Mr. Seppä, who visited the site on June 10th, 2020. The inspection included:

- Visiting the historical open pit area.
- Visiting the tailings area.
- Overall view of the property.
- Inspection of several drill sites.
- Discussions with Jukka Pitkäjärvi, former CEO of Ferrovan Oy.

In addition to the most recent visit, Mr. Seppä has visited the core processing and sample preparation facilities located in Taivalkoski on November 29th, 2017.

AFRY has relied on information provided by Strategic to prepare this report. AFRY has no reason to believe that this information is materially misleading, incomplete or contains material errors. The content of this report as expressed by AFRY is based on the assumption that all the data provided by Strategic is complete and correct to the best of Strategic's knowledge.

All measurement units used in this report are metric, and currency is expressed in the Euro (€) unless stated otherwise. The currency in Finland is the Euro.

3 Reliance on Other Experts

The QPs have relied on additional data from:

- Mineral Deposits database, Geological Survey of Finland
- The Exploration and Mining Registry (permitting), Finnish Safety and Chemicals Agency
- Nesting information, Metsähallitus (government forest council) 9th December 2020

The information, conclusions and recommendations contained in this report are based on:

- The qualified persons' field observations
- Data, reports, and other information supplied by Strategic and other third parties.

For the purpose of the report, Ville-Matti Seppä has relied on the ownership data provided by Strategic and believe that such data and information is complete and correct. The QP has not completed an extensive property title and ownership search on Mustavaara and express no legal opinion on the ownership status of the property.

4 Property Description and Location

The Mustavaara Project is located in north-central Finland, in the municipality of Taivalkoski, in the province of Oulu, 75 km southwest of the city of Kuusamo, on the boundary of the North Ostrobothnia and Lapland provinces (Figure 4-1). The geographic coordinates are 65°49'N latitude and 28°08'E longitude.

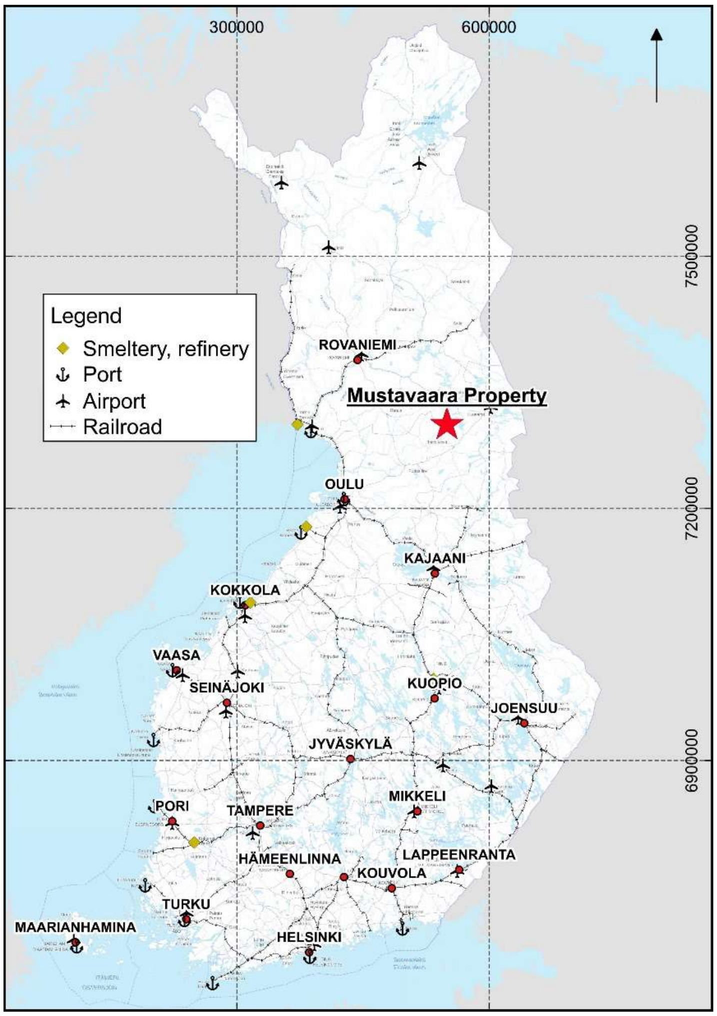

Figure 4-1 Mustavaara property location. Coordinates in ETRS-TM35FIN.

The project includes the historic Mustavaara mine, previously developed and operated by Rautaruukki Oy. The processing plant and all auxiliary buildings and infrastructure were removed from the site by 2002.

4.1 Mineral Rights

The project is comprised of three exploration reservations, and has a surface area of 2,660 hectares (Table 4-1 and Figure 4-2). All reservations have been approved by Finnish Safety and Chemicals Agency (Tukes) and are valid until February 9th, 2022. All of the reservations are held by Strategic through its 100% owned Finnish subsidiary Strategic Explorations Oy.

Table 4-1 Details of exploration reservation. Data from Finnish Safety and Chemicals Agency (Tukes).

Permit Type	Name	Mining Registry Number	Area (hectares)
Exploration Reservation	Kalliolampi 1-4	VA2020:0009	355
Exploration Reservation	Mustavaara West 1-14	VA2020:0011	1,168
Exploration Reservation	Lavotta	VA2020:0010	1,137
Total			**2,660**

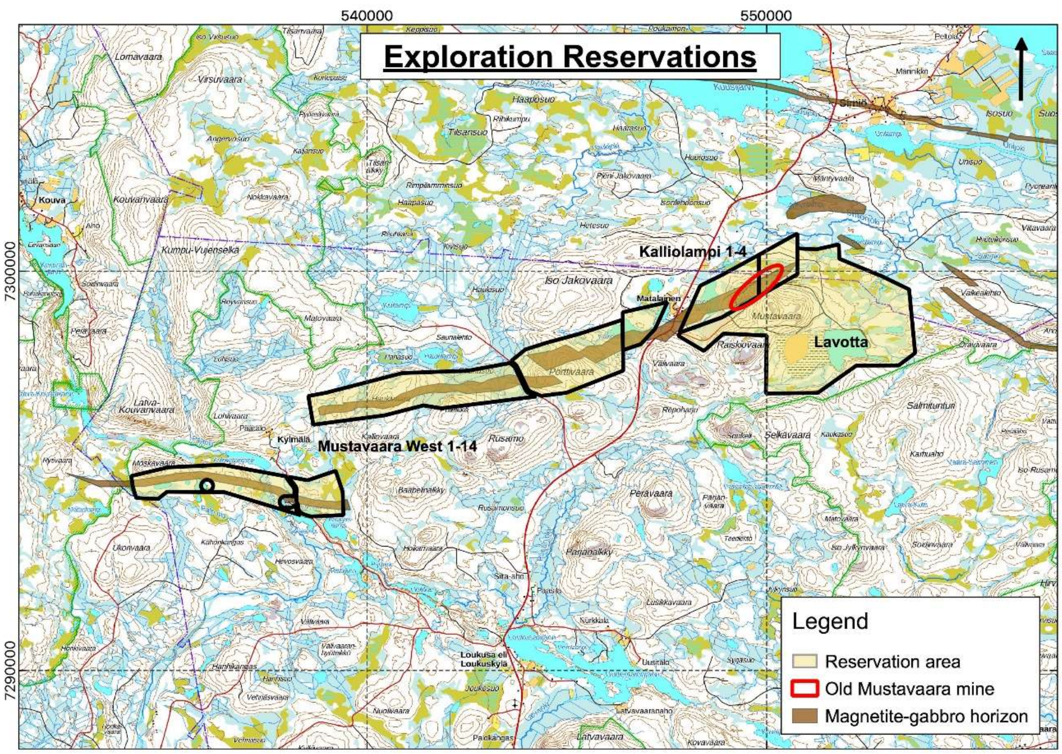

Figure 4-2 Location map of the exploration reservations relative to the local topography, the old mine workings and the position of the magnetite-gabbro horizon

4.2 Preservation Areas

The Mustavaara property is located between areas that belong to Syöte national park, a Natura 2000 area (Figure 4-3).

The Natura 2000 program is European Union's intention to stop the reduction of nature's diversity.

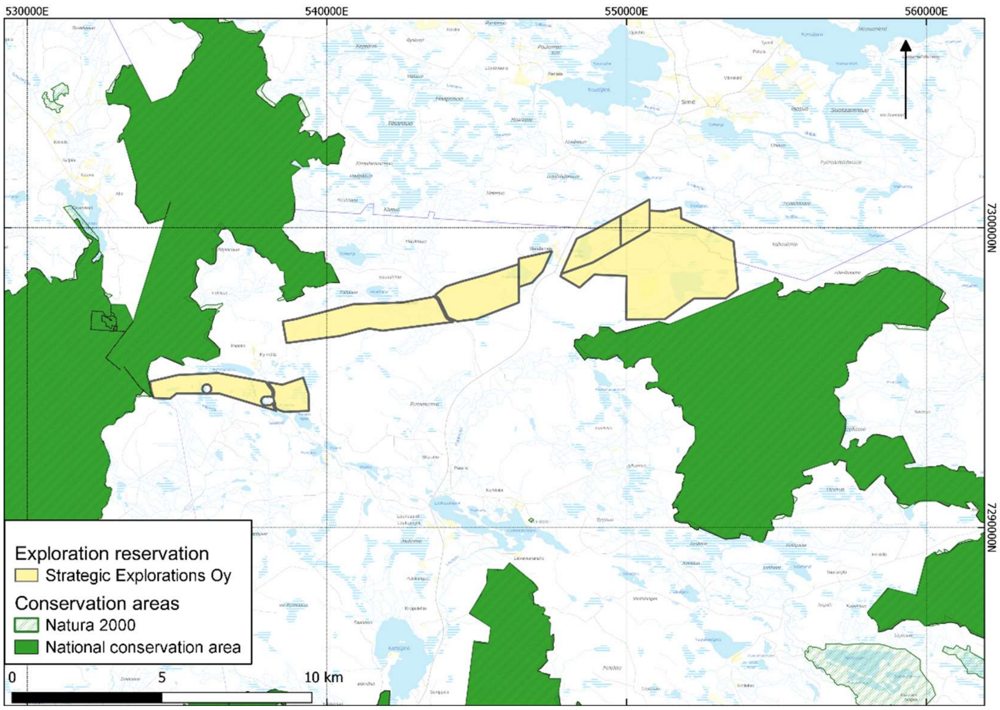

Figure 4-3 Mustavaara reservations and adjacent conservation areas

At Mustavaara, the Natura 2000 area boundary is a few kilometers to the south of the tailings facility area. The laws that govern Natura 2000 do not demand any buffer zones to be set between the conservation area and the land surrounding it. Impacts of projects or plans in or near Natura 2000 sites will be assessed unless it is certain that they will not undermine conservation objectives. The combined effects of different projects are also assessed. Projects can only be approved if the assessment has ensured in advance that they will not have a significant detrimental effect on the conservation objectives of the Natura 2000 sites. The Government may grant a permit for a project that impairs the natural values of a Natura 2000 site if it has to be implemented for an overriding reason in the public interest and there is no alternative solution. In that case, compensatory measures must be taken to maintain the coherence of the network.

At Mustavaara, a Natura 2000 assessment of the effects of the mining activities have been previously carried out and previously proposed work would not have impeded on the Natura 2000 area. The authors recommend reviewing the Natura 2000 assessment against updated work and mining plans in future studies.

4.3 Permits and Compensation Arrangements

The following is a breakdown of the steps needed to advance a project from prospecting to mining according to the Finnish Mining Act (621/2011).

The Environmental Impact Assessment (EIA) procedure and Natura 2000 assessment were approved by the authorities Jan 18th, 2010, POPELY/2/07.04/2010), though EIA-process was done according to the EIA-legislation valid in 2010.

Environmental and water permits issued to Mustavaaran Kaivos Oy (MKOy) were approved on March 16th, 2016 by the Regional State Administrative Agency for Northern Finland (PSAVI). The decision was appealed and the Vaasa administrative court ruled against the appeal making the permits valid on the 14th of June 2018. They remain valid under MKOy as the operator and could be transferred to the next mine operator. The permit contains the water permit, which lapses in July 2022 and the environmental permit, which could lapse as early as July 2023. Strategic has not attempted to transfer the permit as of the effective date of this report.

4.3.1 Reservation

The **Finnish Mining Act** grants reservations that give its holder first refusal to apply for an exploration permit. Rights to a site can be reserved for a maximum period of 2 years. Small-scale prospecting is allowed under the statutory right of public access, subject to the restrictions stipulated in the act, provided that no damage and only minor inconveniences or disturbances are caused.

4.3.2 Exploration Permit

According to the **Finnish Mining Act**, prospecting and advanced exploration are subject to an exploration permit. An exploration permit on a site entitles its holder to the following rights:

- To conduct exploration on the permit holder's own land and that owned by another landowner, or exploration area, in the area referred to in the permit
- To explore the structures and composition of geological formations
- To conduct other exploration in order to prepare for mining activity
- To conduct other exploration in order to locate a deposit and to investigate its quality, extent, and degree of exploitation, as provided for in more detail in the exploration permit
- To build, or transfer to the exploration area, temporary constructions, and equipment necessary for exploration activity, as specified in more detail in the exploration permit

An exploration permit gives its holder first refusal to apply for a mining permit to extract any minerals found within the site. An exploration permit can be granted for a maximum period of 4 years, with an option to extend the permit by 3 years at a time, up to a maximum of 15 years in total.

The permit holder is liable to pay annual compensation to any landowners affected by the permit (known as a 'prospecting fee'). The prospecting fees payable to landowners are as follows:

1) € 20 per hectare per year for the first four years of the exploration permit

2) € 30 per hectare per year for the fifth, sixth, and seventh year of the exploration permit

3) € 40 per hectare per year for the eighth, ninth, and tenth year of the exploration permit

4) € 50 per hectare per year for the eleventh year of the exploration permit and for any subsequent years

The permit holder is also liable to compensate any inconvenience and damage caused in the area by exploration activities based on the **Mining Act**.

4.3.3 Compensation Payable under the Environmental Permit

In addition to the fees payable according to the mining permit, obligations relating to compensation and securities may also be imposed in environmental permits. Typical examples of such obligations include compensation for effects on fishing and waste management securities to ensure that the rehabilitation phase will be completed satisfactorily. Decisions relating to the environmental permit and any compensation payable under the permit rest with the Regional State Administrative Agency for Northern Finland.

4.3.4 Factors Affecting Work on the Property

The permitting authority changed the environmental permit near the Mustavaara tailings area design because there are nesting sites of protected species. No activities within a 400 meter radius around the nest are permitted, which could affect the previous tailings dam design proposed by MKOy. In the springtime, only if nesting is ongoing, no activities within a 1 kilometer radius around nest are allowed. According to the new (9th December 2020) data gained Metsähallitus the nests are active. This will have an effect on the work that can be made on the property during the nesting season.

Apart from the nest, the author is not aware of any other significant factors or risks that would prevent the right or ability to work on the property.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The Mustavaara Project is located in north-central Finland at the border of North Ostrobothnia and Lapland provinces, approximately 650 km north of Helsinki and 180 km northeast of Oulu. Both paved highways and a gravel road lead to the property (Strategic Resources, 2020). The nearest town is Taivalkoski which is located about 35 km southeast of Mustavaara via route 863.

5.2 Climate

The climate in Finland is intermediate and both features of marine and continental climate are typical. The average temperatures at Mustavaara vary from +25 C in the summer to -20 C in the winter. Temperatures rarely go down to -45 C or up to +32 C (Finnish Meteorological Institute, 2012).

The annual precipitation is approximately 500-650 mm. The amount of precipitation increases towards summer, usually July and August are the rainiest months (Finnish Meteorological Institute, 2012).

Wintertime lasts approximately seven months in Central and Northern Lapland, and snow stays on the ground for over half of the year (Finnish Meteorological Institute, 2012).

5.3 Local Resources and Infrastructure

The nearest town to Mustavaara is Taivalkoski which is located about 35 km to southeast. Taivalkoski belongs to the North Ostrobothnia province and can be reached by highways number 5 and 20. Taivalkoski can provide basic goods and services for the early stages of exploration. Neighboring municipalities are Kuusamo, Posio, Pudasjärvi and Suomussalmi (Municipality of Taivalkoski, 2020).

The North Ostrobothnia province has a population of 411,887 and Taivalkoski has 3,976 inhabitants (Central Statistical Office of Finland, 2017). The nearest city is Oulu, approximately 180km southwest of Mustavaara, and has a population of 205,489 (Central Statistical Office of Finland, 2019). The city of Oulu can provide basic goods and services for early stages of exploration and mining. Oulu's airport service daily domestic and international flights closer to the project area. Kuusamo airport, 76 km east of Mustavaara provide daily flights from Helsinki and seasonal international flights.

For international overseas shipments, the ports of Oulu and Raahe are suitable for mine operations. Mustavaara property is connected to Oulu via routes 863, 8610 and highway 20 (179 km) and to Raahe via routes 863, 8610 and highways 20 and E8 (256 km).

5.4 Physiography

Most of Lapland is characterized by lowland topography with the maximum elevations of lowland being 200 m elevation. Lowlands are located mainly from Kemi river regions to Kittilä, Sodankylä and Savukoski (Kujansuu, 2005). The Mustavaara property is located in the Kuusamo region and is approximately 290 m above sea level.

Ecologically, most lakes in Lapland, especially the larger ones, are usually in good or excellent condition. Smaller lakes typically suffer more eutrophication (Finnish Environment Institute, 2013). Surface waters (lakes and rivers), the ecological condition, and groundwater domains adjacent to Mustavaara are shown in Figure 5-1.

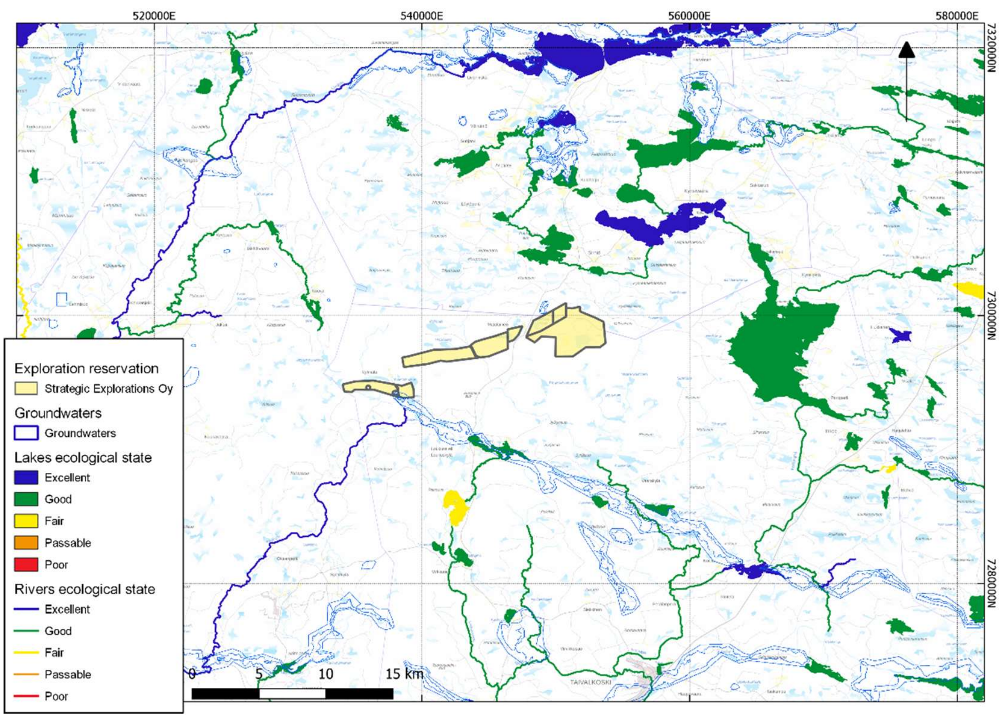

Figure 5-1 Surface waters and groundwater domains. State of lakes and rivers is indicated with different colours. Data from SYKE Vesikartta (2016)

Main parts of Lapland belong to the north boreal vegetation zone. Pine trees form the conifer forest border in the northern forest zone unlike elsewhere in the world where spruce or larch form the forest border (Hyppönen, 2002).

6 History

6.1 Prior Ownership and Historic Results

The Mustavaara deposit was discovered by Otanmäki Oy in the 1960s' when a sample was sent to the company's laboratory for analyses and reported vanadium. A vanadium bearing magnetite deposit was outlined by field work in 1957 and 1958. Rautaruukki Oy, a former Finnish state-owned iron and steel producing company started more detailed investigations in 1961. A diamond drilling program of 56 diamond drill holes carried out between 1967 and 1971 outlined enough high-grade vanadium ore to start a commercial mining operation.

The decision to develop the mine was made in the fall of 1971 and construction began in 1973 with trial runs conducted in 1976. Figure 6-1 shows the processing plant as it was in 1976. An open pit mine and a roast-leach processing operation with a final V_2O_5 product began with a total of 300 personnel. The ultimate pit was designed to be 1,800 m in length with widths varying from 130 to 290 m, varying depths from 50 to 135 m, and with final pit wall slopes at approximately 55°. Annual production reached a peak of 1.6 million tonnes of ore, producing 240,000 tonnes of pelletized magnetite concentrate and a final product of 3,000 tonnes of vanadium pentoxide.

Table 6-1 Annual (1976-1984) Tonnes mined, Fe- and V-compositions of the concentrate in the Mustavaara Mine

Year	Tonnes	Fe (wt%)	V (wt%)
1976	730 000	60.50	0.89
1977	984 000	61.70	0.89
1978	1 256 000	61.60	0.92
1979	1 630 800	61.80	0.90
1980	1 561 600	61.30	0.89
1981	1 584 100	61.60	0.92
1982	1 619 600	61.80	0.92
1983	1 572 000	61.50	0.90
1984	1 490 100	no data	0.93

The mine operated to 1985 but closed after a period of low vanadium prices (US$1.50/ lb V_2O_5).

Figure 6-1 Aerial photo from Rautaruukki processing plant in 1976.

In July 2006, Akkerman Exploration B. V. (Akkerman) signed a 5-year option agreement with Adriana Resources Inc. (ADI), by which ADI could earn a 70% participating interest in the Kalliolampi mineral rights.

On April 6th, 2009, the Mustavaara West claim reservation (which includes the current Mustavaara West 1-14 claim) was granted to Akkerman by the Ministry, covering the western extension of the Mustavaara magnetite-gabbro horizon over a strike length of 15km. In April 2010, Akkerman applied for and was granted 14 exploration claims within the Mustavaara West claim reservation.

During March 2010, ADI transferred its rights under the 2006 Option Agreement with Akkerman to ProspectOre Capital Corp. (PCC). PCC passed its rights to its Finnish subsidiary Vanadis Mines Oy and entered into a Modified Option Agreement with Akkerman. The adverse financial market conditions did not enable Vanadis Mines Oy to raise the necessary funds as required under the modified agreement and consequently the Option Agreement was terminated in August 2010.

During the month of April 2011, Akkerman filed an official request for an extension of the original 5-year term of the Kalliolampi claims.

On May 19th, 2011, Akkerman entered into a purchase agreement, whereby 100% of its Mustavaara mineral rights were sold to Mustavaaran Kaivos Oy (MKOy).

Shortly after, on May 28th, 2011, MKOy filed an application for a Mining District over the Mustavaara mine and surrounding areas with the chief inspector of mines of the Ministry of Labour and the Economy. Total surface extent of the mining concession area applied for was 1,685 hectares.

A map with the outlines of the mining concession is shown in *Figure 6-2*.

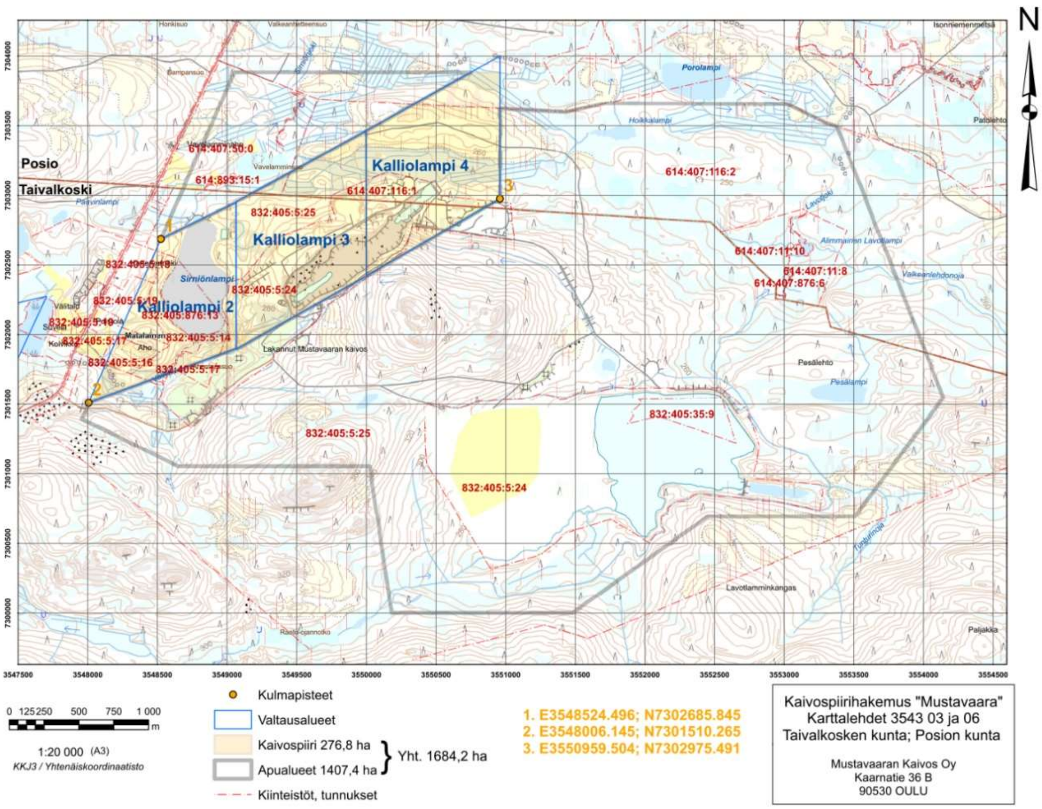

Figure 6-2 Map illustrating the outlines of the Mustavaara Mining Concession. Blue line: claim area, yellow area: mining district, grey line: auxiliary area

In the fall of 2011, MKOy drilled 17 diamond drill holes and modelled airborne magnetic data over the property. In 2012, MKOy completed a pre-feasibility study and in 2013 MKOy completed a pit optimization study. MKOy then proceeded to start permitting the mine. In 2016, environmental and water permits were issued to MKOy and appealed. The courts overturned the appeal and granted the permits again on June 14th, 2018. Later that year MKOy changed its name to Ferrovan Oy. The permits remain valid under MKOy as the operator and could be transferred to the next mine operator. The permit contains the water permit, which lapses in July 2022 and the environmental permit, which could lapse as early as July 2023. Strategic has not attempted to transfer the permit as of the effective date of this report.

On February 10th, 2020, Strategic Resources Inc. announced that it had successfully applied for Reservations over the Mustavaara mine area. The company also acquired all of the intellectual property, 2011 drill core and storage facilities associated with Mustavaara from the bankruptcy estate of Ferrovan Oy (Strategic Resources, 2020).

6.2 Historical Reserves and Resources

The first reserve estimation for Mustavaara was conducted by Rautaruukki Oy based on 57 drill holes at 100 m spaced sections.

These holes were drilled southwards along 100 m spaced cross-lines of a local mine grid with a baseline striking N71°E. Hole dips varied from -44° to -74° and depth of penetration was designed to test the magnetite-gabbro horizon down to a depth of 200 m asl (equivalent to mine level +100), which was the planned depth extent of the open pit.

The Mineral Reserve was estimated by the sectional method. The magnetite contents of the ore were determined by Davis Tube separation and the vanadium grade was determined by wet chemical analyses. The specific gravity (SG) of the orebody and waste rock were determined from laboratory measurements (weight in air and in water). The average SG of the ore was found to be 3.2 and 3.0 for waste rock.

Cut-off grades applied were 11.9% magnetite (as weight percentage recovered in the concentrate) containing 0.75% vanadium on a weighted average basis.

Rautaruukki Oy's historic non 43-101 compliant official mining reserve reported by Heikki Paarma in September 1971, amounted to 38 million tonnes grading 16% ilmenomagnetite. The magnetite concentrate that could be produced from this resource was estimated to contain an average of 0.9% vanadium.

Waste rock to be mined would have amounted to 43.7 million tonnes and overburden to be removed 1.3 million m^3. The open pit was designed with pit slopes of 55°, a total length of 1,500 m, and bench heights of 15 m.

The historical mineral resources and reserves of Mustavaara deposit are listed in Table 6-2, Table 6-3, Table 6-4, Table 6-5, and Table 6-6.

Table 6-2 Rautaruukki Oy 1971 @ 11.9% Magnetite cut-off

	Tonnes (Mt)	Magnetite (%)	VinMC (%)
Reported reserve	**38**	**16**	**0.9**

Note: VinMC refers to vanadium in magnetite concentrate. The table is not NI 43-101 compliant and is included for historic reference only.

The Mineral Resource estimates created by Outotec Finland Oy are based on a 3D model constructed from cross sections taken at 100 m intervals. The cross sections include lithological and assay data. The nominal cut-off grade to be used was determined to be 8% magnetite. The dimensions of the model were extended to cover an additional 20 m in the NW-SE directions and 50 m in the vertical direction according to available drill hole results.

Drill hole samples were composited to 5 m length and the grades were estimated using an Inverse distance squared method with a maximum search distance of 500 metres.

Table 6-3 JORC compliant Mineral resource estimate as of March 10th, 2012 @ 8% Magnetite cut-off (Outotec Finland Oy)

Resource Class	Tonnage Mt	Magnetite%
Indicated Mineral Resource	**109.5**	**14.94**

Note: The table is not NI 43-101 compliant and is included for historic reference only.

Table 6-4 JORC compliant Mustavaara Ore Reserve estimate as of March 10th, 2012 @ 8% Magnetite cut-off (Outotec Finland Oy).

Reserve class	Tonnes (Mt)	Magnetite (%)	VinMC (%)	NSR (€/t)
Probable Ore Reserve	**97**	**13.79**	**0.91**	**62**

Note: VinMC refers to vanadium in magnetite concentrate.

The ore reserves are not additional to the mineral resources in Table 6-3.

The most recent Mineral Resource estimate was prepared by Outotec (Finland) Oy by Markku Meriläinen and Pekka Lovén with an effective date of August 30th, 2013. The resource estimate complies with recommendations in the Australasian Code for Reporting of Mineral Resources and Ore Reserves (Joint Ore Reserve Committee – JORC-code).

Table 6-5 JORC compliant Mineral resource estimates as of May 31st, 2013 @ 8% Magnetite cut-off (Outotec 2013).

Resource Class	Tonnage Mt	Magnetite %
Measured Mineral Resource	**63.5**	**15.13**
Indicated Mineral Resource	**48.1**	**14.70**
Total Mineral Resource	**111.6**	**14.94**

Note: The table is not NI 43-101 compliant and is included for historic reference only.

Table 6-6 JORC compliant Mustavaara Ore Reserve estimates as of May 31st, 2013 @ 8% Magnetite cut-off (Outotec 2013)

Reserve class	Tonnes (Mt)	Magnetite (%)	VinMC (%)	NSR (€/t)
Proven Ore Reserve	**64**	**13.97**	**0.91**	**66.6**
Probable Ore Reserve	**35**	**13.93**	**0.90**	**66.2**
Total Ore Reserve	**99**	**13.96**	**0.91**	**66.4**

Note: VinMC refers to vanadium in magnetite concentrate. The table is not NI 43-101 compliant and is included for historic reference only.

The ore reserves are not additional to the mineral resources in Table 6-5. “VinMC” refers to the vanadium content in the magnetite concentrate produced from the ore.

The authors have not done sufficient work to classify these historic estimates as current mineral resources and mineral reserves. The issuer is not treating the historic estimates as current mineral resources and mineral reserves.

7 Geological Setting and Mineralization

7.1 Regional Geology

The Precambrian East European Craton and Fennoscandian Shield comprises the basement lithlogies of Finland and is one of the oldest parts of the Eurasian continent (Korsman & Koistinen, 1998).

The Northern parts of Finland are composed mainly of Archean and early Proterozoic rocks (Silvennoinen, 1998). The Archean rocks are approximately 3,100 – 2,500 Ma old and the Proterozoic rocks are approximately 1,930 – 1,800 Ma old (Korsman & Koistinen, 1998).

Typical rocks of the Archean period are gneissic rocks, rocks from greenstone belts, mica schists and paragneiss (Luukkonen & Sorjonen-Ward, 1998).

Major parts of Lapland's Paleoproterozoic bedrock are composed of metamorphosed volcanic and sedimentary rocks which form the Central Lapland Greenstone Belt (CLGB), the Peräpohja Schist Belt and the Kuusamo Schist Belt. Volcanic rocks are typical for CLGB. The arc shaped Lapland granulite belt (LGB) is located in the north-eastern part of CLGB. LGB is composed of rocks which were metamorphosed at high pressure and temperature and pushed up in the crust by tectonic movement. Mafic layered intrusions are another typical feature in Northern Finland's Paleoproterozoic rocks. These rocks have crystallized in stable tectonic conditions and they are usually of economic importance. Central Lapland Granitoid Complex is also part of Paleoproterozoic rocks which extend from the boundary of Sweden almost to the boundary of Russia (Silvennoinen, 1998).

7.2 Local Geology

The Mustavaara deposit is a vanadium bearing magnetite gabbro horizon which is part of a large mafic layered intrusive complex, known as the Koillismaa Layered Igneous Complex. The age of the intrusion is approximately 2,400 to 2,500 Ma. This sheet like body intruded into the Archean basement as a single body with a total length of 100 km and a thickness of up to 3 km. This body has separated into at least 5 distinct large blocks due to later folding and faulting. The block containing the Mustavaara deposit is the Porttivaara Block, with a total approximate length of 19 km.

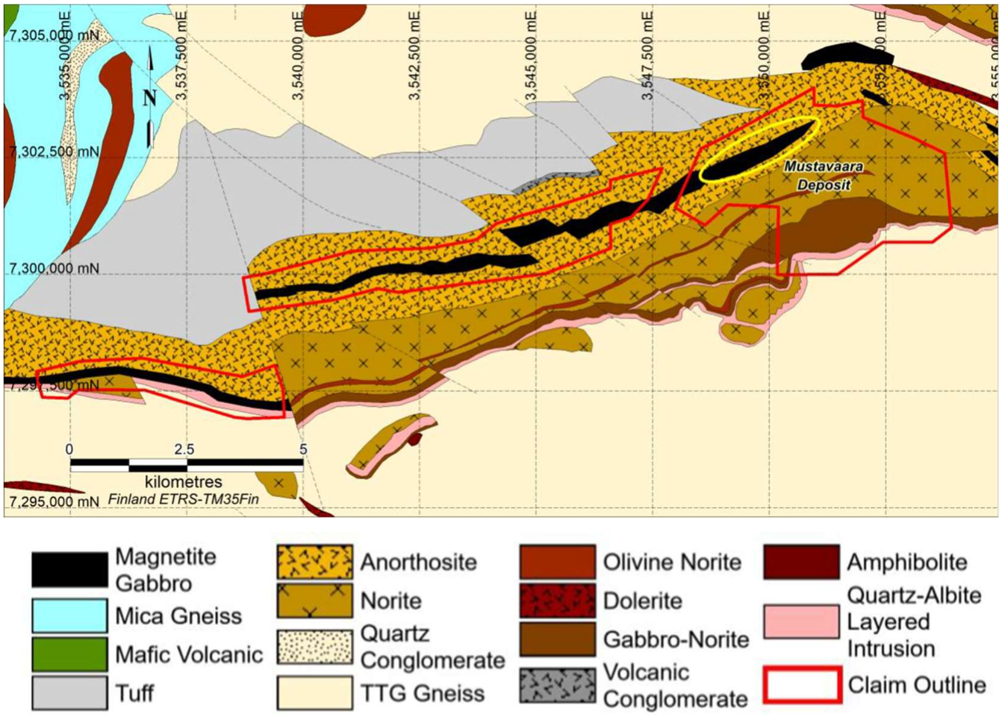

Figure 7-1 Regional geological map showing the Porttivaara block of Koillismaa Layered Intrusion. The historic Mustavaara mine is marked with a yellow circle.

The layered intrusion is divided into a marginal series (50 to 250 m thick) at the base and a thick layered series (up to 2,500 m thick) making up the bulk of the intrusion. The layered series is mainly composed of norites, gabbronorites, leucogabbros and anorthosites. The ore-bearing Mustavaara magnetite-gabbro occurs in the upper part of the layered series, surrounded by anorthosite rocks. The ore-horizon is a sheet-like body that trends east-west over a total strike length of approximately 15km within the Porttivaara Block. The ore-horizon generally dips 35°- 45° to the north. The thickness progressively increases from east to west up to a maximum of 200 m, while the average magnetite concentrate decreases to the west from 20% in the east to less than 10% in the western part.

Genetically, the vanadium rich magnetite gabbro is considered to be of magmatic origin formed as a segregation from an iron-rich liquid.

Lithological classification of the layered intrusion follows the classification used by MKOy. The lithological units presented in the Table 7-1 are included in the data base and are utilized in the geological model.

Table 7-1 Lithological units used in the geological model.

Lithological unit	Notes
Ore Lower Layer (OLL)	Coherent magnetite layer
Ore Middle Layer (OML)	Coherent magnetite layer
Ore Upper Layer (OUL)	Coherent magnetite layer
Ore Disseminated Layer (ODL)	Low grade magnetite layer
Ore	Narrow massive magnetite above the main zone
Internal Rock	Internal anorthosite
Hanging wall Rock	Anorthosite above the coherent magnetite layer
Footwall Rock	Anorthosite below the coherent magnetite layer
Peridotite	
Overburden	

The locations of the drill holes at Mustavaara are presented in Figure 7-2. Figure 7-3 and Figure 7-4 present the plan and cross section views of the deposit geology.

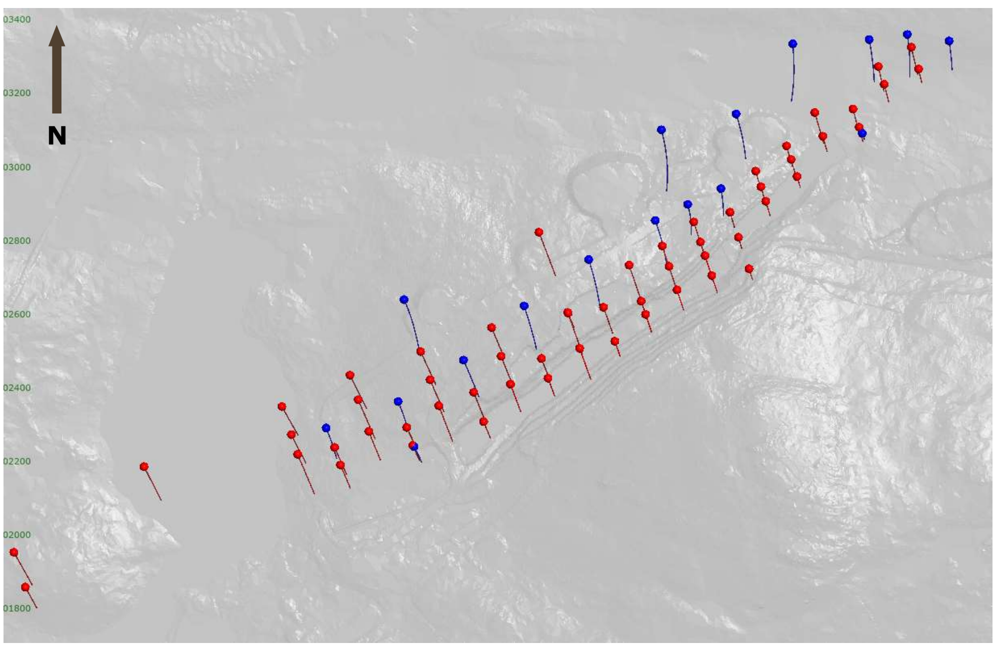

Figure 7-2 Drill hole locations at Mustavaara. Blue= 2011 MKOy drill campaign, Red= 1967-1971 Rautaruukki Oy campaign.

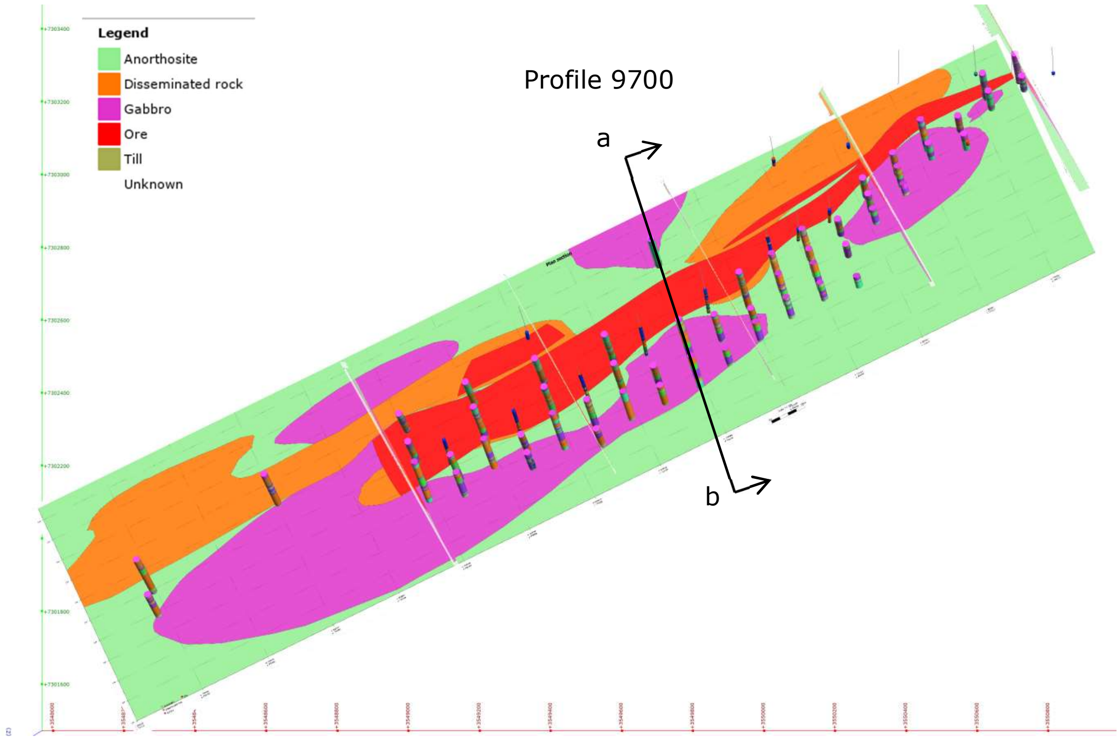

Figure 7-3 Plan view of the deposit geology and drill hole locations

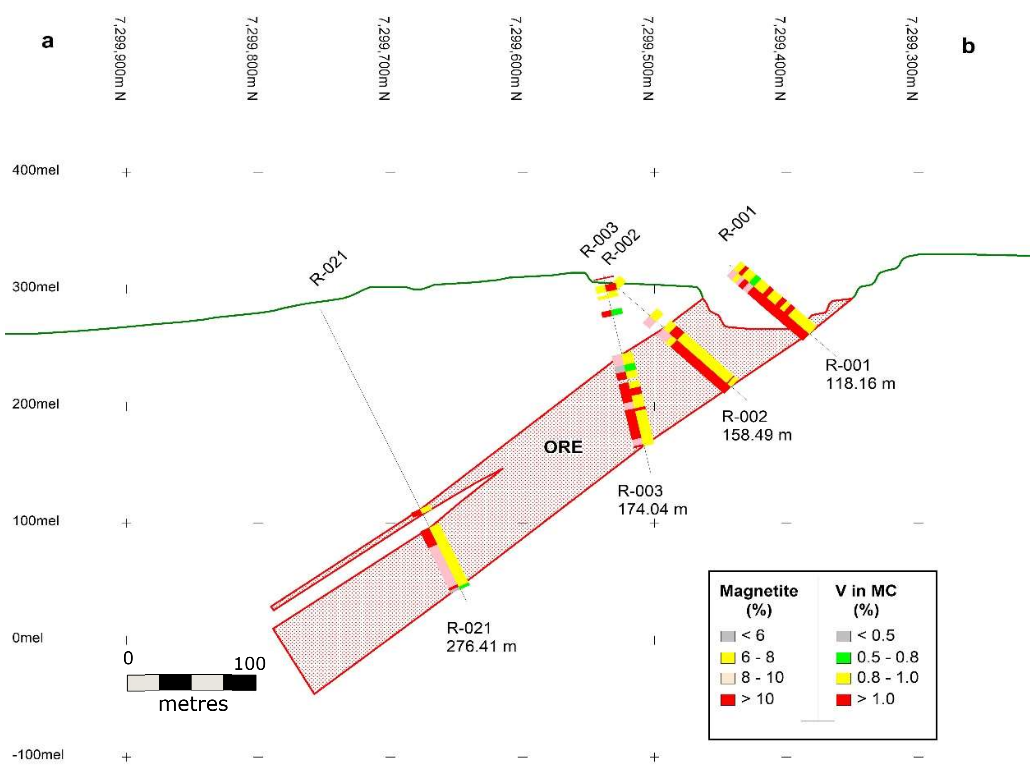

Figure 7-4 Cross section of profile 9700, with an outline of the magnetite horizon, viewing north-east. Note: pre 2011 sampling focused on ore zone only.

7.3 Mineralization

According to Karinen et al. (2015), the Mustavaara vanadiferous titanomagnetite (VTM) deposit was formed by gravity concentration of iron oxide crystals, or by sorting of a magnetite slurry. In the Panzhihua Complex, VTM deposit formation via gravitational accumulation has been suggested as well as fractional crystallisation of Fe-Ti-V oxides (Gao et al. 2019). Repetition of these mechanisms has led to the formation of several stratified layers of magnetite ore. In Mustavaara the magnetite crystals contain thin lamellae of ilmenite and are generally referred to as ilmenomagnetite. Iron content of the ilmenomagnetite concentrate is around 62 - 63% Fe and the average vanadium contents in the magnetite concentrate from the four units is approximately 0.9% V. At Mustavaara the amount of ilmenomagnetite in magnetite gabbro changes in such a regular way that the ilmenomagnetite layer can be divided into four separate layers. The three lowermost layers comprise the main constituent of the Mustavaara ore deposit and they are called: ore lower layer (OLL), ore middle layer (OML) and ore upper layer (OUL). The highest ilmenomagnetite content of 20 – 35 wt.% is in OLL (footwall contact), which forms a narrow and continuous layer (0.2 – 4 m) just above and following the footwall contact. OUL (hanging wall contact) forms a thicker (20 – 40 m) and continuous layer along the hanging wall contact in the main ore layer and has ilmenomagnetite content of 18 – 25 wt.%. In the thickest (10 – 50 m) OML the ilmenomagnetite content is 10 – 15 wt.%.

The fourth layer above the hanging wall contact consists of weakly disseminated ilmenomagnetite and is labelled as the ore disseminated layer (ODL). The ODL is the most inhomogeneous layer, which contains scattered anorthosite, and anorthosite gabbro fragments and compact magnetite veins. The anorthosite gabbro waste rock blocks do not necessarily follow the general layering and are randomly oriented. Upwards, the ilmenomagnetite dissemination gradually decrease and the amount of anorthosite gabbro blocks and layers in the magnetite gabbro increase. This happens until the rock is a heterogeneous anorthosite gabbro containing specks of magnetite gabbro. In this disseminated layer the ilmenomagnetite content usually varies from 2 - 10 wt.%. Figure 7-5 shows an oblique view of Mustavaara deposit.

The magnetite gabbro horizon dips 30 to 40° to the north. In the westernmost part (800 m) the thickness of the coherent ilmenomagnetite ore layer (OLL+OML+OUL) is 60 – 95 metres. Eastward (700 m) the thickness of the ore layer is quite consistent at 40 – 50 metres. In the easternmost part, over a distance of 200 m, the dip starts to steepen from 40 to 70°. Over the same distance the thickness of the ore layer starts to become thinner, from 40 meters to 10 metres thick, until it finally dies out completely.

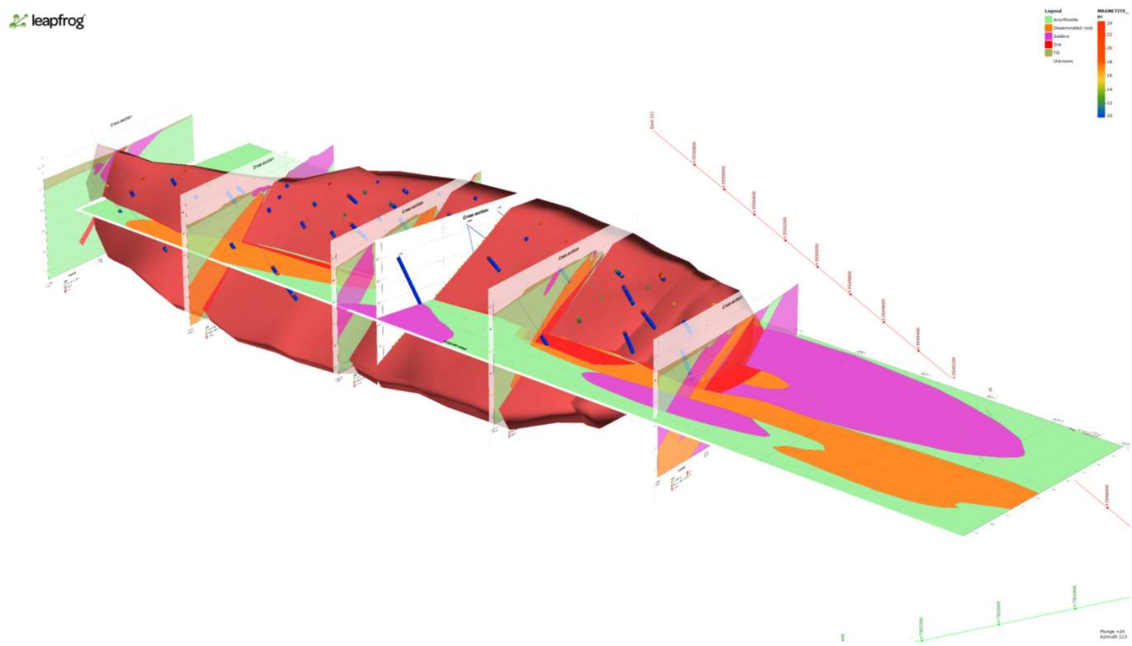

Figure 7-5 Oblique view of Mustavaara deposit, viewing southeast

8 Deposit Types

The Mustavaara deposit is characteristic of a vanadiferous titanomagnetite deposit (VTM), which are typical mafic layered intrusions. Similar VTM type deposits associated with layered mafic intrusive complexes are the Bushveld Complex (South-Africa), the Lac Doré Complex (Canada) and the Panzhihua Complex (China).

All of these deposits have the following similar chemical affinities and host-rock provenance:

- They consist of magmatic accumulations of magnetite and ilmenite and commonly contain on average 0.2 to 1% V_2O_5.
- Host rocks of VTM deposits are mainly mafic and ultramafic igneous rocks.
- The host rocks are typically deep-seated in origin and can occur in tabular bodies that are thick and laterally extensive, or as smaller lens-shaped bodies.
- The textures and mineralogy of VTM ores are very similar in that they typically form discrete layers between 0.1 and 10 m in thickness, although they can be thicker, and their oxide layers are laterally extensive with sharp lower boundaries with their host silicate rocks and gradational upper contacts.
- VTM ores can either be massive with greater than 80% titanomagnetite or disseminated with about 50% titanomagnetite and lesser amounts of clinopyroxene, olivine and plagioclase in both.
- VTM deposits can also be enriched with chromium, copper, nickel and platinum group elements.

VTM deposits associated with titaniferous magnetite layers are found from the fractionated upper portions of a layered series of igneous complexes. Typically, these deposits are subdivided into either magnetite-dominant (typically hosted by gabbroic rocks in layered intrusions, like Mustavaara) or ilmenite dominant deposits (typically hosted by massif-type anorthosites) (Gross, 1996).

Mustavaara magnetite gabbro lies in the upper part of the Porttivaara intrusion (part of Koillismaa igneous complex) where it forms a coherent layer following the general layering of the intrusion. The magnetite gabbro is plagioclase-augite-ilmenomagnetite adcumulate with sharp lower contact and gradual upper contact which is characteristic for VTM deposits.

Toplis and Corgne (2002) proved that vanadium-rich magnetite crystallizes in a rather narrow range of fO_2 conditions during fractional crystallization of basaltic melts. Furthermore, Balan et al. (2006) attested that Mustavaara ilmenomagnetite was crystallized in similar fO_2 conditions and has alike V^{4+}/V^{3+} ratio with Bushveld and Skaergaard (Greenland) vanadium-rich magnetite crystals.

9 Exploration

Exploration on the Mustavaara deposit started in 1957 with a few rock samples. Since then, there has been 9,991 m of drilling in 73 drill holes, outcrop mapping, ground and airborne geophysical surveys, and historic mining. The continuation of the magnetite gabbro beyond the former Mustavaara mine is known from the exploration work carried out by Rautaruukki Oy and MKOy. A national high-altitude airborne geophysical survey covered the area sometime between 1951 and 1972 which first confirmed the magnetite rich horizon. More detailed magnetic data modelling and ground surveys were later commissioned by MKOy in 2011, resulting in the confirmation that the magnetite gabbro extends over at least 15km to the west. Geophysical interpretations suggest progressive thickening of the horizon from 60 m around the open pit to more than 200 m in the west. The magnetite grades range from 15% in the east and appear to decrease to less than 10% in the west.

9.1 Geophysics, Strike Extensions

The location, size and shape of the magnetite-gabbro horizon is clearly reflected in airborne and ground magnetic survey data (Figure 9-1 and Figure 9-3). In 2011, MKOy commissioned GTK (Salmirinne 2011) to conduct some further data modelling on airborne magnetic data obtained from a GTK survey in 1998. The line spacing of the GTK airborne geophysical survey flown in 1998 was 200 m, with a terrain clearance of 30 m in both N-S and E-W directions. The main goals were to define location, depth and orientation of the almost 19 km long magnetite-gabbro reef in the Porttivaara layered intrusion and to locate sub-areas with higher concentrations of magnetite.

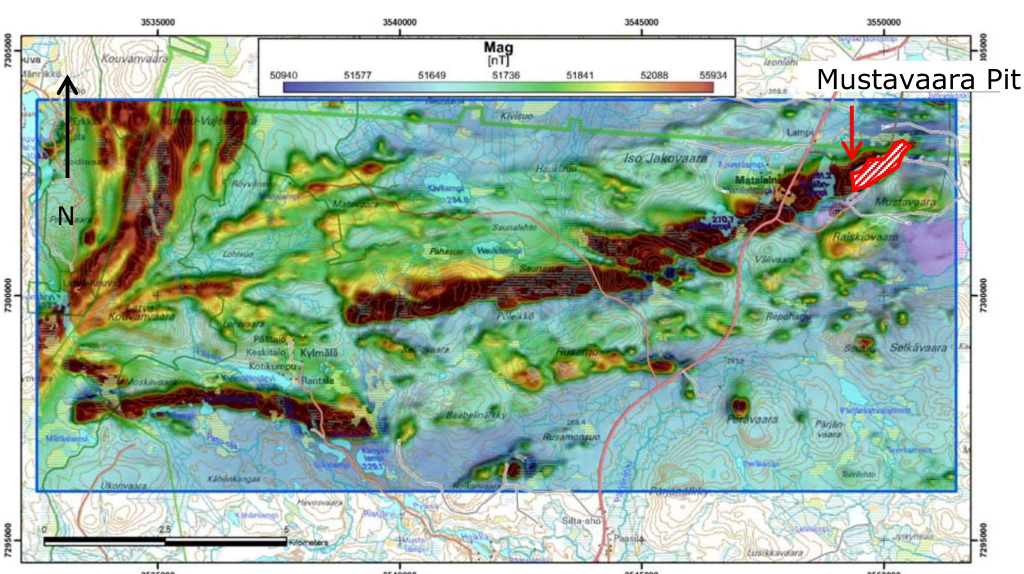

Figure 9-1 Aeromagnetic map of Porttivaara layered intrusion area showing the anomalous magnetite gabbro horizon

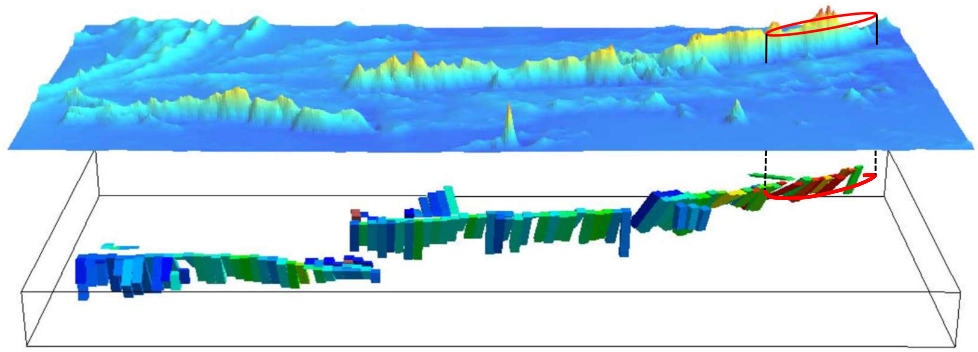

Figure 9-2 Modelled airborne magnetic data in 3D, reflecting varying magnetic intensity along strike and interpreted dips. Mustavaara pit area marked with red ellipse.

From the modelled data it can be seen that the magnetite-gabbro horizon terminates a few hundred metres to the east of the current open pit. West of the historic open pit however, the magnetite-gabbro extends over a total distance of at least 15 km. The western extension is referred to as Mustavaara West.

As a result of the modelling, the GTK geophysicist identified 3 sub-areas in Mustavaara West considered to have the best exploration potential for magnetite-gabbro. The highest priority was given to a 3 km long area extending directly west of the existing open pit.

In August 2011, this priority area was covered with a new ground magnetic survey. A line spacing of 50 m and a point spacing of 5 m was used. A total of 48,976-line kilometres and 9,674 points were surveyed. The colour shaded magnetic total field is represented in Figure 9-3.

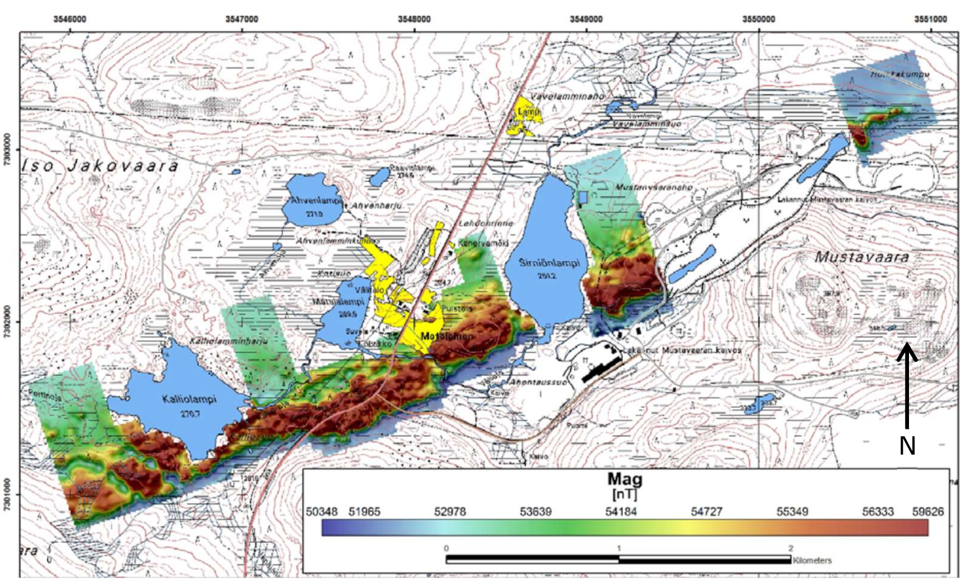

Figure 9-3 Total magnetic field, reflecting the extension of the ore horizon west of the open pit.

In the 1960s and the 1970s both Outokumpu Oy and Rautaruukki Oy carried out minimal exploration drilling at Mustavaara West. They concluded that magnetite grades were lower in the west, while the horizon became much thicker in western direction, up to 200 metres.

To date, only part of the historic Mustavaara West exploration data has been reviewed. The author is not aware of the extent of the historic exploration results: surface sampling or geological mappings etc. Some indicative intersections from Rautaruukki Oy are summarized in the table below:

Table 9-1. Drilling intersections from Rautaruukki Oy's exploration work.

Profile	Hole No	Width	% Magnetite	% V_2O_5
8900	RN 15	35	16.7	1.5
		3	21.9	1.6
8500	RN 36	77	10.2	1.5
	RN 38	137	10.5	1.4
6100	RN 3	85	8.9	1.3
	RN 4	147	10.0	1.3

10 Drilling

10.1 1967-1971 Drilling

The drilling conducted by Rautaruukki Oy between the years of 1967 and 1971 contains a total of 6,822.7 metres of drilling in 56 holes. Records of these holes exist in Finish reports and are available from MKOy's database. Additionally, approximately half of the drill hole cores are stored and available for review in GTK's Finnish Geologic Core Library. The drill locations of these holes were reported in relation to a local mine grid (Pöyry 2012). To date, no reference points of this grid have been located in the field although the grid position has been extrapolated using the topographic data of the historic mine reserve estimates to orient the mine grid. The drill hole locations have been estimated to within 10 m of their true location. The database and drill hole information does not contain a record of the drill core size but according to the available core photos and the other drilling at the same time, a wireline system WL-46 was used resulting in 28.8 mm core sample diameters. Drill core handling procedures applied to the 1967 to 1971 drilling programs were not documented in the reports provided to the authors. Locations of the drill holes are presented in Figures 7-2 and 7-3.

10.2 2011 Drilling

Mustavaaran Kaivos Oy (MKOy) conducted a diamond drill core program consisting of 17 holes totalling 3,088.50 m in the Mustavaara deposit area from September to November of 2011. Details of the program are shown in Table 10-1. The drilling was carried out by the drill contractor of Nivalan Timanttikairaus with a NQ2 size wireline system (50.7 mm core size). Holes were measured for magnetic susceptibility at 10 cm interval and surveyed with a Reflex Maxibor II system for deviation. Two drill holes (MV-59-2011 and MV-69-2011) do not have a susceptibility survey because they had extremely fractured rocks and one hole (MV-69-2011) does not have deviation measurements because it was a shallow hole. The locations and azimuths of the drill collar casings were measured by the surveying company Rovamitta Oy with the use of a Differential GPS system and a tachymeter.

Drill cores were transported by MKOy staff from the drill site to the core logging and storage facility in Taivalkoski. Logging and sampling was conducted by MKOy staff geologist along with a consulting geologist both experienced in the rock and ore types of this deposit. The information collected included geological contacts, rock descriptions, fracture information and alteration within the described intervals. Special emphasize was placed on geological contacts of the magnetite gabbro and its ore subunits for which the data of the susceptibility survey was used. The drill holes were logged in Finnish and reports remain available in the database. Drill core recovery was excellent.

The drill core was sampled in intervals up to 3 m with breaks in sample intervals based on geological contacts of different lithologies and in the case of the ore horizon, on ore subunits. Before splitting, the drill core was photographed to document the core and to include the markings for analytical intervals in the core boxes.

Table 10-1 Details of the drilling of year 2011 by MKOy

DDH	X	Y	Z (m)	Azimuth	Dip	Length (m)	Maxi -bor	Susceptibility
MV-57-2011	7302242.78	3549348.53	282.94	158.65°	57°	86.70	X	X
MV-58-2011	7302362.62	3549306.46	288.96	163.40°	60°	167.00	X	X
MV-59-2011	7302292.01	3549113.07	271.32	163.06°	60°	164.70	X	
MV-60-2011	7302634.07	3549321.86	282.94	161.74°	60°	290.60	X	X
MV-61-2011	7302473.07	3549481.58	304.66	159.76°	55°	184.60	X	X
MV-62-2011	7302617.03	3549643.06	307.79	163.31°	55°	195.30	X	X
MV-63-2011	7302740.79	3549814.69	300.45	163.19°	50°	199.80	X	X
MV-64-2011	7302844.92	3549991.76	285.29	159.00°	50°	179.30	X	X
MV-65-2011	7302887.93	3550078.78	280.69	164.03°	60°	150.60	X	X
MV-66-2011	7302929.91	3550167.14	278.11	167.56°	63°	152.50	X	X
MV-67-2011	7303087.51	3550008.20	262.25	164.55°	60°	299.70	X	X
MV-68-2011	7303129.71	3550208.01	259.12	159.70°	60°	228.30	X	X
MV-69-2011	7303078.19	3550545.18	258.51	156.16°	58°	31.70		
MV-70-2011	7303317.15	3550359.50	257.81	172.31°	55°	244.90	X	X
MV-71-2011	7303328.43	3550563.56	256.28	164.41°	58°	205.60	X	X
MV-72-2011	7303341.83	3550665.50	260.23	167.38°	58°	193.70	X	X
MV-73-2011	7303324.61	3550777.16	260.38	165.76°	50°	113.50	X	X

X = latitudinal location in Finnish KKJ grid (zone 3), Y= longitudinal location in Finnish KKJ grid (zone 3), Z = altitude asl, Azimuth = measured azimuth, dip = planned dip.

The drill program was designed to outline and infill the down-dip continuation of the magnetite gabbro to at least 50 m down from the topographic level +200 used in the historic reserve and resource estimates by Rautaruukki Oy. Drill sites were planned into the old mine grid.

The main objective of the 2011 drilling program was to confirm enough tonnage for a minimum mine life of 20 years. The second objective was to check the accuracy of Rautaruukki Oy's historic diamond drill holes for their locations and metal grades. Two of the diamond drill holes in the MKOy program were designed to twin historic holes with the remainder testing down dip extension.

10.3 Quality Assurance and Quality Control Results

10.3.1 Location of Drill Sites

Due to the uncertainties of the location of the historic drilling site coordinates, the twined hole locations were the best estimated location to within 10 m. The drilling sites of MKOy were originally positioned using a handheld GPS and properly surveyed after drilling with a differential GPS. The final location of MV-57-2011 was within less than 5 m of the estimated location for historical hole R-053, and up to 20 m between drill holes MV-69-2011 and historical hole R-032. Of the two twined holes drilled by MKOy, hole MV-57-2011 matched well with its rock intervals and assay results against hole R-053 drilled by Rautaruukki Oy (Figure 10-1). The second twined hole, MV-69-2011, did not match as well with hole R-032.

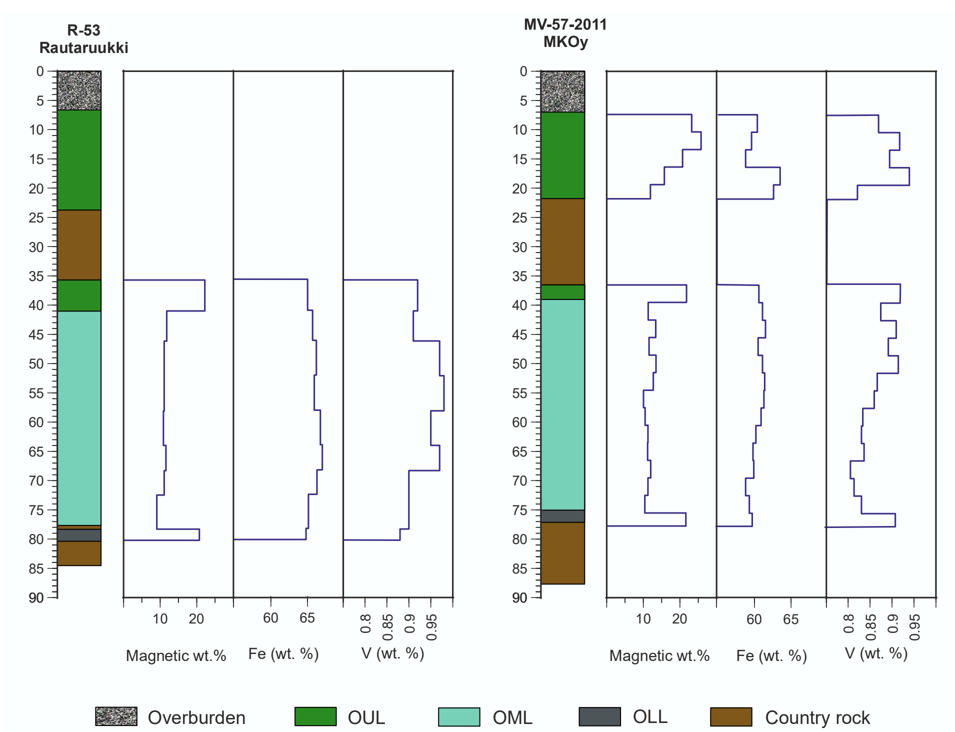

Figure 10-1 Stratigraphic Section of drill holes R-053 (Rautaruukki Oy) and MV-57-2011 (MKOy) showing geology, variation in the amount of magnetic minerals (Magnetic wt.%), and contents of vanadium and iron in concentrate. Contents of magnetite, vanadium and iron were not measured in the upper part of hole R-53.

11 Sample Preparation, Analyses and Security

11.1 Rautaruukki Oy

Detailed reports from the 1967 to 1971 drill programs are lacking and therefore not a lot is known about Rautaruukki Oy's procedures. Once the mine was operational, and possibly prior to mining the company did operate their own sample and analytical laboratory on site. From the assay results and core that remains available for review at GTK's National Drill Core Archive, we know that Rautaruukki Oy selectively sampled ore bearing intervals up to 3 m long and consistently ended the sample interval at geologic breaks in the run. Rautaruukki Oy also ran their samples through Davis Tube Separation or some other magnetic concentrator as the values obtained for vanadium, and titanium are recorded to be values in concentrate. The total amount of samples analysed with surveying records for the Rautaruukki Oy program was 599 samples.

11.2 MKOy, Chain of Custody and Sample Preparation

For the 2011 drill program, the drill core and samples were handled with normal security measures throughout the handling process. The drill core was picked up by the geologist from the drill site and moved to their secure logging and processing facility. Samples were marked in up to 3 m intervals with emphasis on geological contacts of different lithologies and in the case of the ore horizon, on ore subunits. Before splitting, the drill core boxes were photographed so that the markings for analytical intervals were visible and documented.

The core was split using a Cedima CTS-175 rock saw operated by an experienced contracted sawyer and then sent to Eurofins Labtium Oy (Labtium) laboratory in Rovaniemi for crushing, milling and assay.

During MKOy's QA/QC check of the historic data, the resampling of the old drill cores was done by consulting geologist Markku Iljina. Rautaruukki Oy sample intervals were marked out on the historic core and the remaining core was quartered. The samples were sawed by GTK staff and sent to Labtium for assay. The total amount of historical samples rerun was 18 and the total amount of samples in the 2011 drill campaign was 436 bringing the total MKOy samples to 454.

11.3 Laboratory Assay Preparations and Protocols

For MKOy's 2011 drill program, the core samples were shipped to Labtium for preparation and analysis. Labtium is an ISO/IEC 17025:2005 accredited laboratory. Part of the assay procedure was done by GTK's Mineral Technic Laboratory (Mintec) mineral processing laboratory in Outokumpu Figure 11-1.

The core samples, each weighing up to 3 kg, were sent for geochemical analyses to Labtium, Rovaniemi. Once in the lab, the samples were dried at 70° C if they needed drying, after which they were crushed and split in a rotary splitter to form a subsample and a reject (Labtium codes 10, 32, 34 and 38). The coarse subsample (< 2 mm particle size) was pulverized with a carbon steel bowl (< 0.2% Fe, no base metal contamination) in an LM2 pulverizing mill (>90% < 100µm grain size, Labtium code 52). The specific gravity for hole MV-

70-2011 was measured (Labtium code 880F). The sample analysis procedure is outlined in Figure 11-1.

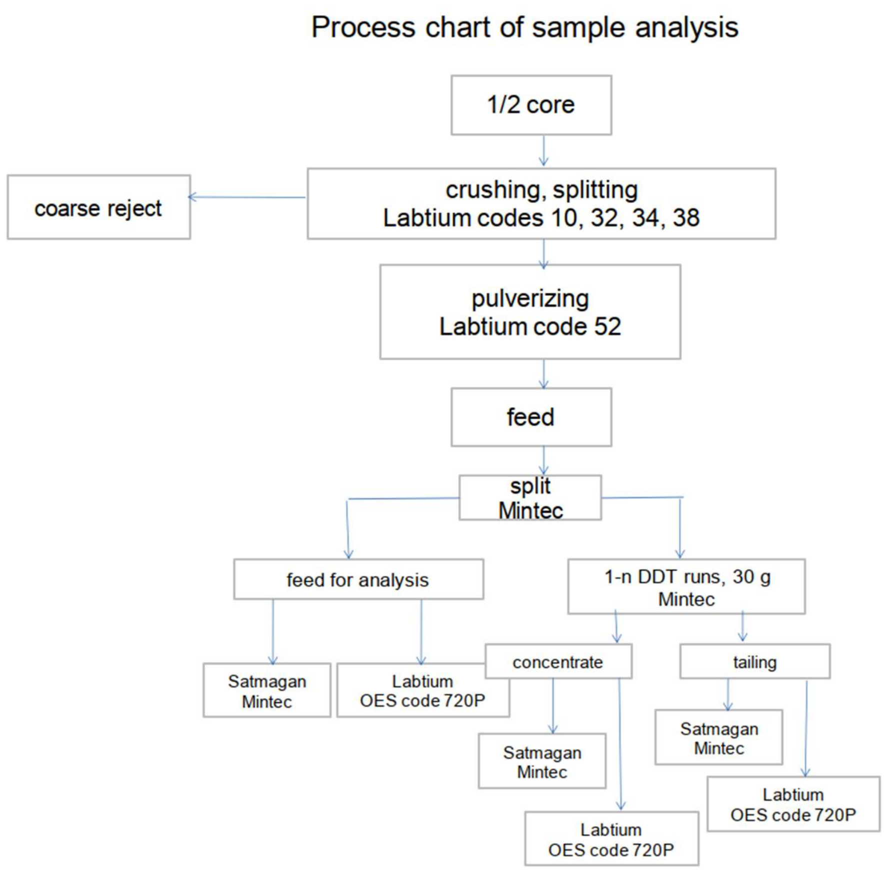

Figure 11-1 Process chart of sample analysis

Dings-Davis Tube

Dings-Davis Tube (DDT) testing is considered to be a simulation of industrial wet magnetic separation techniques and gives a "probable" concentrate grade at any given grind size. The quality of DDT concentrates are process sensitive and dependent on the sample (feed) size, magnetic field strength, tube tilt angle, wash water flow and tube oscillation, among other parameters.

Magnetic separation using approximately 30 g of the pulverized sample known as the "feed" was performed using DDT machines for wet fractionations of small samples yielding a strongly magnetic subsample known as the "concentrate" and a nonmagnetic subsample known as the "tail". The separation was carried out by Mintec of GTK, using a laboratory testing magnet (Type TM) of the KHD Humbold Wedag AG. In addition, Satmagan analyser for determining magnetic value in the feed, concentrate and tailing was used in the same laboratory. The Satmagan of Mintec is calibrated to indicate by its value the amount of magnetic minerals in wt. % in a sample.

Prior to the DDT runs, the wet suspensions of feed samples were dispersed ultrasonically for 1 minute. The DDT run duration was usually about 10 minutes per sample where during the first minute the feed sample was water flushed to a separation tube of the DDT machine through a funnel. The conditions of DDT runs at Mintec were:

- Tube sliding speed frequency setting "90", this is equivalent to 112 rpm
- Magnetic flux density setting "70" (magnetic flux density approximately 0.25 Teslas in the middle and approximately 0.75 Teslas on the wall of the separation tube)
- The tilt angle of the separation tube was constant at 45°
- Pumping speed of flushing water was constant at approximately 1.0 litre/minute

The pre-treatment method for geochemical analyses by Labtium was sodium peroxide fusion for 0.2 g of the feed, concentrate and tailing. The multi-element analysis of 27 elements was performed with inductively coupled plasma optical emission spectrometry (ICP-OES) at the geochemical laboratory of Labtium (Labtium code 720P). According to Labtium, the sodium peroxide fusion and multi-element analysis by ICP-OES is close to total analysis. The ICP-OES analysis by Labtium is covered by the scope of accreditation according to ISO/IEC 17025 from the Finnish Accreditation Service FINAS. The detection range of the ICP-OES (Labtium code 720P) for iron is 0.01 - 70.0% and for vanadium is 0.005 - 5.0%.

The remaining half-split cores, coarse and pulverized sample rejects were returned to MKOy and are currently stored in a storage warehouse for Strategic in Taivalkoski.

11.4 Quality Assurance and Control

11.4.1 Assay Result Repeatability

For the 2011 drill campaign samples and for the due diligence on the historic samples Labtium completed duplicate sample checks. For the repeatability assessment every 20th sample was duplicated by Labtium. The quality of the assay duplicates has been illustrated in Figure 11-2.

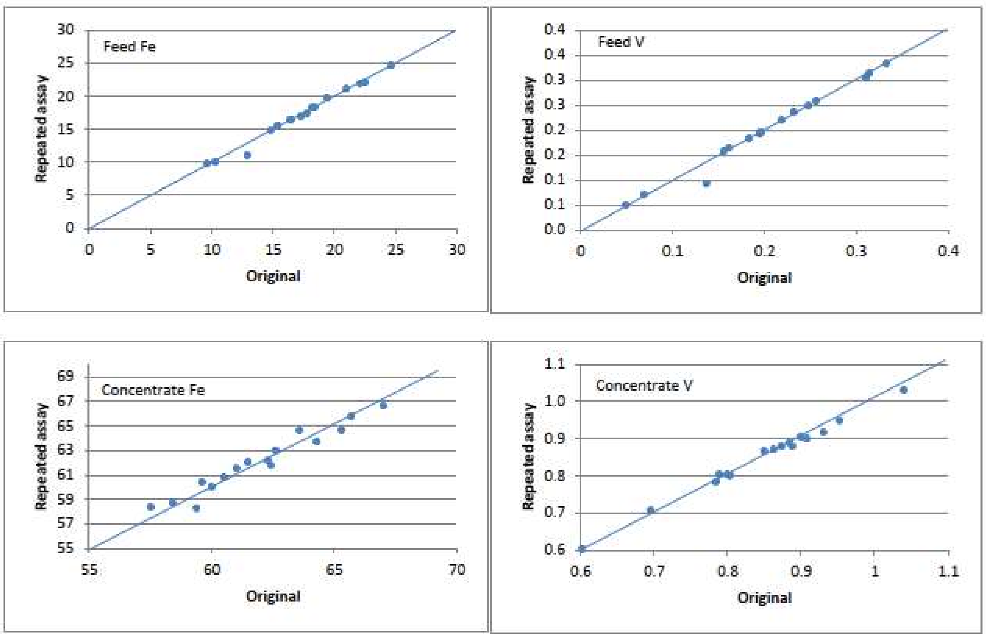

Figure 11-2 Assay duplicates versus original assay results for feed and concentrate fractions. Concentrations in wt.% and 1:1 trendline is shown.

The visual analysis of Figure 11-2 shows high repeatability and is supported by the calculated Relative Standard Deviation (RSD), which is generally <<1%. The Fe content of concentrate fraction showed the widest scattering, with an average RSD of 0.59% and the maximum RSD of 1.32%. RSD of less than 3% in assay duplicates is considered to indicate good laboratory performance.

11.4.2 DDT Run and Satmagan Precision and Accuracy

The quality of DDT runs and Satmagan measurements was tested by doing separate runs for duplicate samples (Figure 11-3) and comparing measured and calculated Satmagan values of the feed. The calculated values were derived from the Satmagan values of the concentrate and tailing fractions and their respective weights (Figure 11-4).

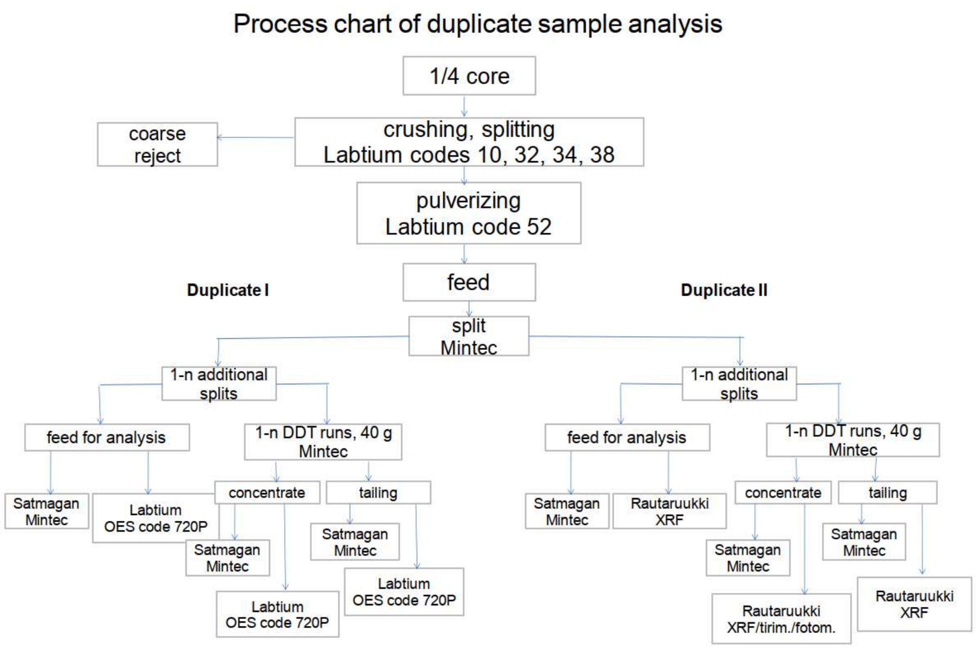

Figure 11-3 Process chart for duplicate sample handling and analysis.

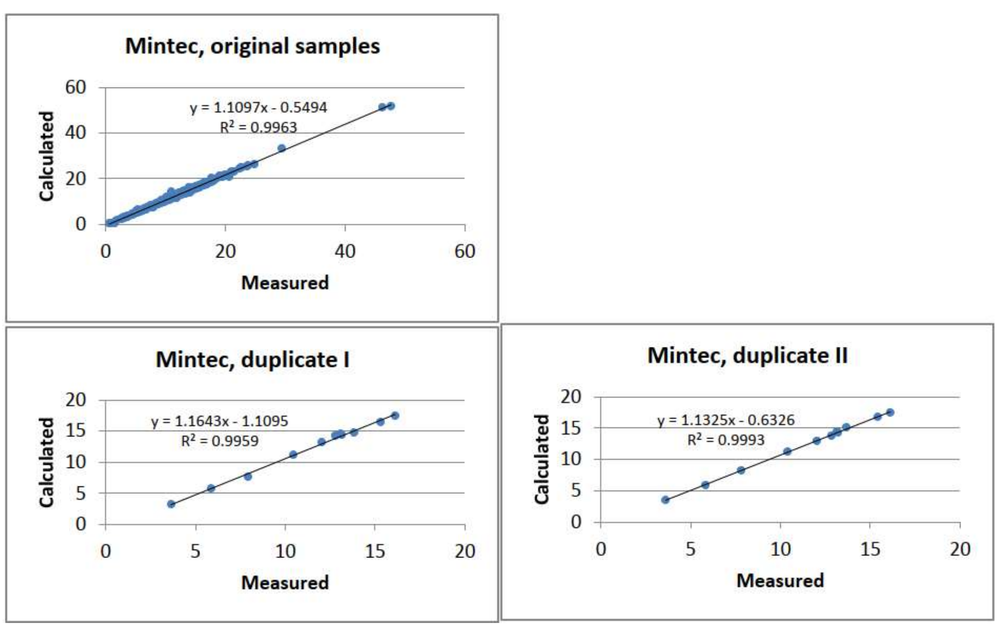

Figure 11-4 Satmagan values (wt.% of magnetic minerals) of the feed calculated from Satmagan values of the concentrate and tailing and their respective calculated weights versus measured values.

The DDT runs made at Mintec showed a good correlation with a slight tendency for the calculated values to increase very slightly above the measured magnetite content of the feed at higher grades.

Satmagan measurements

The reproducibility of the Satmagan samples was studied by comparing both, the original feed and concentrate values with duplicate I, and duplicate I and II sample sets (Figure 11-5).

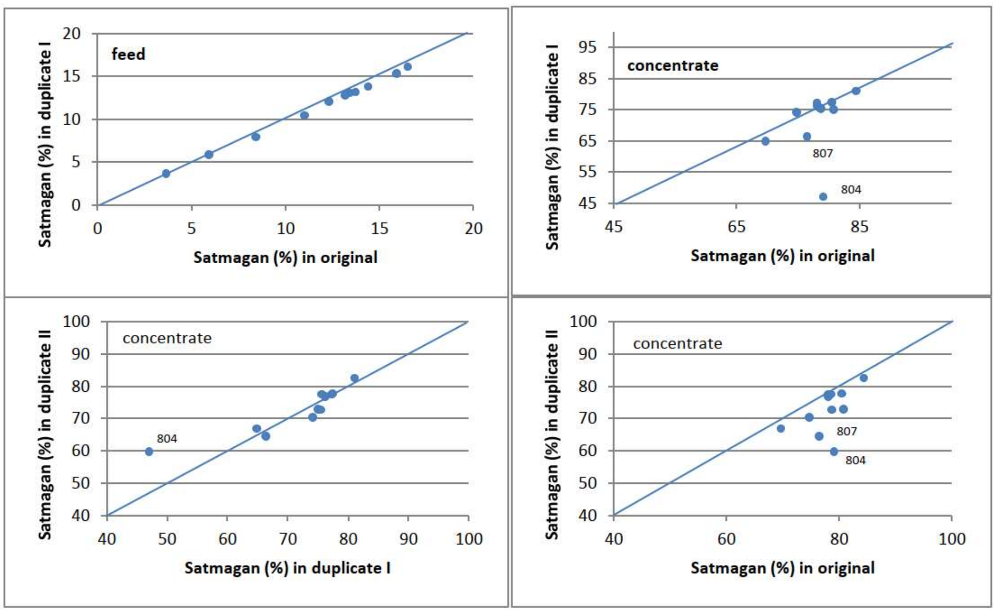

Figure 11-5 Comparison of the Satmagan values of the feed and concentrate against the original, duplicate I and II samples.

The comparison between the feed samples of the original and duplicate I showed a good correlation as did the concentrate duplicate I versus the duplicate II. However, the comparison between the concentrate original and the duplicates I and II showed a wider scattering. This scattering was studied in more detail by calculating the RSD, which showed values of less than 10%, except for one outlying sample (Table 11-1).

Table 11-1 Satmagan readings of the concentrate fraction of original, duplicate I and II sample sets with calculated Relative Standard Deviation (RSD).

Sample ID	Original	Duplicate I	Duplicate II	RSD
800	69.71	64.87	66.89	3.6
801	78.10	76.22	76.85	1.2
802	80.84	75.04	72.86	5.4
803	78.12	77.30	77.49	0.6
804	79.12	47.04	59.68	26.1
805	78.57	75.68	77.54	1.9
806	84.45	81.06	82.61	2.1
807	76.52	66.42	64.55	9.3

808	74.76	74.16	70.38	3.2
809	78.72	75.46	72.71	4.0
810	80.51	77.47	77.72	2.1

Common bounding values of RSD for inter-laboratory reproducibility tests are considered good if they are <5%, satisfactory if they are <10% and unsatisfactory if they are >10%. Satmagan is a less precise measurement technique and the apparent scattering may be attributable to a higher variability within this population. Taking this into account, the Satmagan measured reproducibility was generally good.

DDT runs and mass recovery

An essential part of the resource estimate is the percentage of magnetic concentrate recovered in the DDT run. Davis Tube weight recovery (DTWR) results of the original and duplicate I and II samples were compared in Figure 11-6. The DTWR represents the percentage of the concentrate (by weight) from the sum of the concentrate and tail.

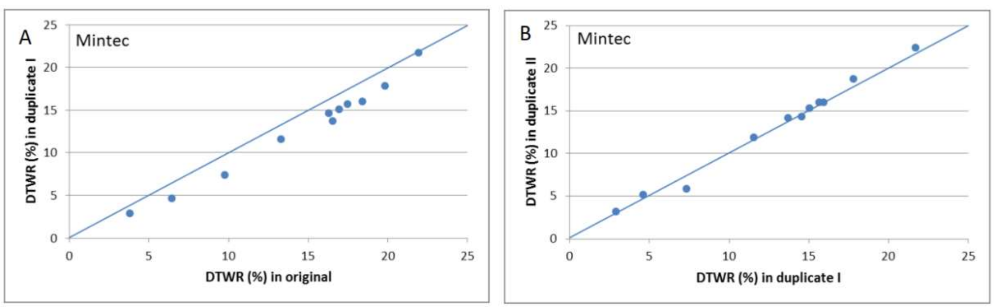

Figure 11-6 DTWR correlation in DDT runs from 11 samples.

The duplicate I and II samples showed very good correlation with each other (Figure 11-6B). The correlation between the original and duplicate I samples was good (Figure 11-6A although, the original samples showed systematically greater values than the duplicate I samples). Calculated RSD between original and duplicate I averaged 11.7% (0.9 - 23.2%), which has been considered acceptable. Nevertheless, the systematic difference in the increased original values should be recognised. The reason for this systematic difference may be attributed to differing sample load weights (30 g for the original versus 40 g for the duplicate).

11.4.3 Assay Precision and Accuracy

The precision of Labtium laboratory has been statistically and visually assessed through bivariate plots of iron- and vanadium-contents for the original and the duplicate samples representing feed, concentrate and tail materials (Figure 11-7 and Figure 11-8). The iron and vanadium feed samples both had good precision. For iron-content the correlation coefficient was 0.88 (Pearson's coefficient) with 12.50 - 22.30 wt. % Fe in the original analyses and 12.80 - 22.10 wt. % Fe in the duplicate analyses. The vanadium-content in the feed samples showed the correlation coefficient to be 0.92 with 0.13 - 0.27 wt. % V in the original samples and 0.14 - 0.28 wt. % V in the duplicate samples.

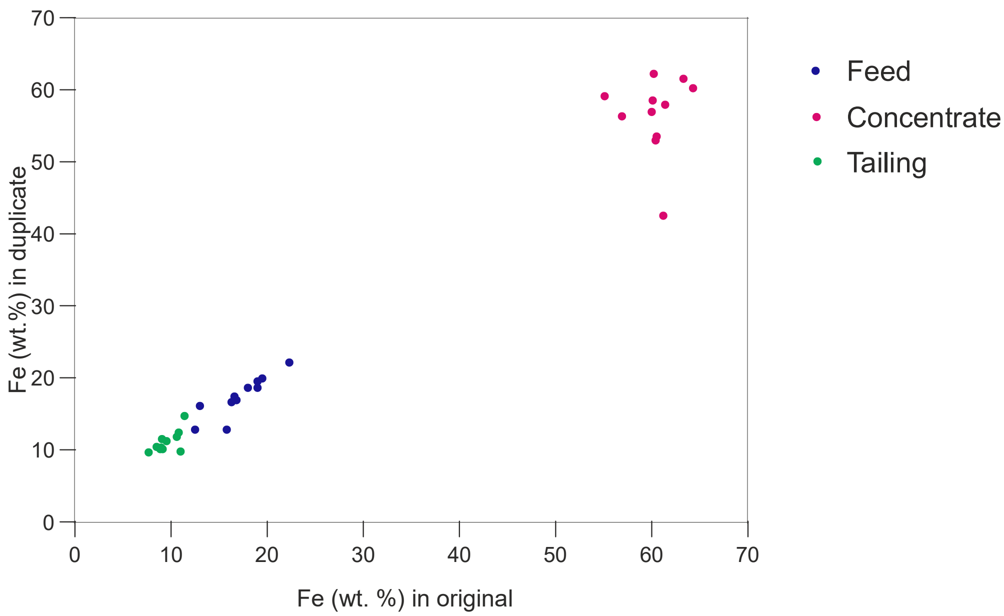

Figure 11-7 Correlation between Fe contents in original and duplicate samples (n = 11) analysed by Labtium

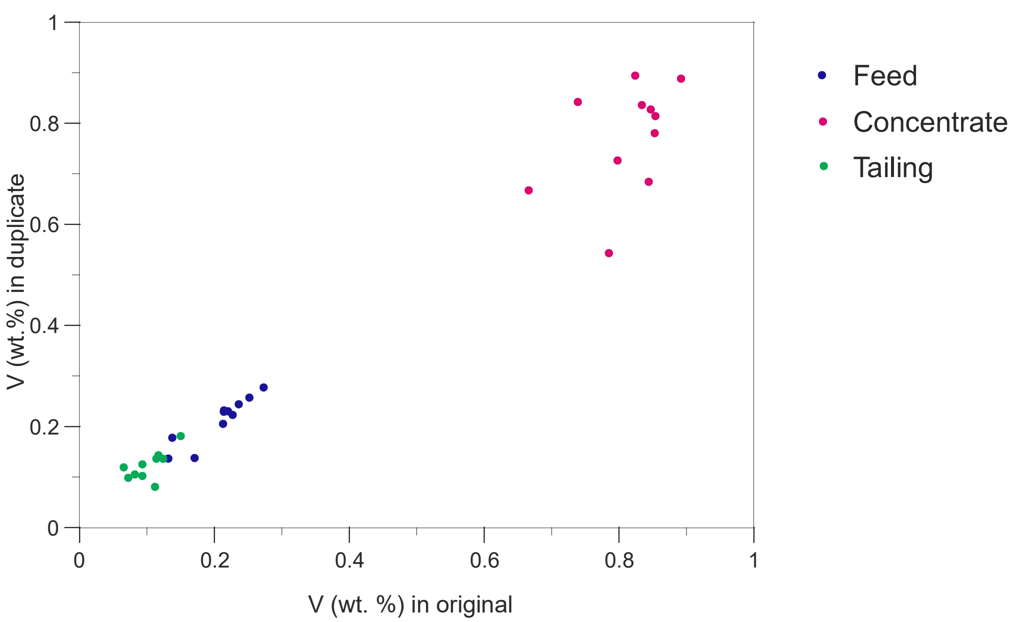

Figure 11-8 Correlation between V contents in original and duplicate samples (n = 11) analysed by Labtium

The concentrate and tail samples appeared to have significant differences between the contents of iron and vanadium in the original samples compared to the duplicate samples. The iron content in the concentrate samples had a correlation coefficient of 0.03 (meaning no correlation) where the variation was 55.10 - 64.30 wt. % Fe in the original samples and 42.50 - 62.20 wt. % Fe in the duplicate samples. Vanadium content showed similar features. The correlation coefficient for vanadium in the original and their duplicate concentrate samples was 0.45 with 0.67 - 0.89 wt. % V in the original and 0.54 - 0.89 wt. % V in the duplicate samples. The results representing the tail display similar features to the concentrate. The correlation coefficient of iron was 0.67 with 7.67 - 11.40 wt. % Fe in the original and 9.65 - 14.70 wt. % Fe in the duplicate samples. The correlation of vanadium content of original and duplicate samples representing the tail was 0.69 with 0.07 - 0.15 wt. % V in the original and 0.08 - 0.18 wt. % V in the duplicate samples.

The results indicate that the precision of the Labtium laboratory was good. The duplicate rock samples were collected from the drill core and therefore the variation in the composition of feed samples is a sum of rock composition and analytical error, but with a minor error in precision.

In the case of sample compositions after grinding and DDT runs, i.e. the compositions of concentrate and tail samples, it became clear that the principal error came from either the grinding and/or the DDT runs. This is because these procedures were performed separately with different analytical submissions for the original samples and their duplicate samples from ¼ resampled drill core.

The correlation between specific gravity (g/cm^3) and mass fraction of magnetic phases in rock (magnetic mass %) was estimated (n = 60) from drill holes R-002, R-003 and MV-70-2011 (Figure 11-9). The equation is defined with best fitting linear trendline. The specific gravity was measured for the sampled intervals of the drill hole MV-70-2011. The specific gravity was measured to be 3.2 g/cm^3 for the ore.

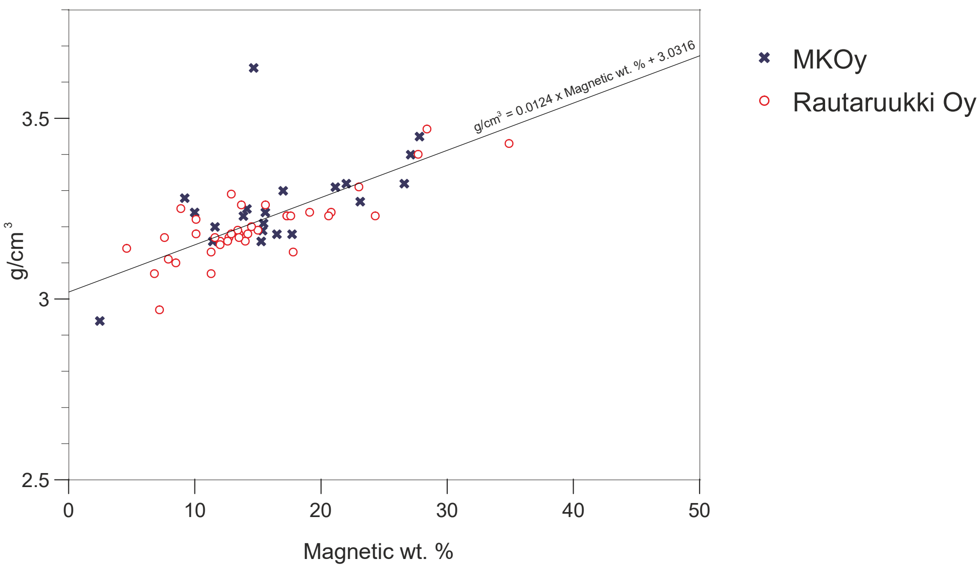

Figure 11-9 The correlation between specific gravity (g/cm^3) and mass fraction of magnetic phases in rock

Accuracy of the assays

To test the accuracy of Labtium's ICP-OES run samples, fine rejects of the feed, concentrate and tail were sent for secondary analyses to Rautaruukki laboratory in Raahe where the samples were analysed utilizing XRF-techniques. Because the analysed material for these analyses were fine rejects, the analyses were independent of grinding and DDT run variability. The accuracy of these duplicates compared to their original results has been plotted for iron and vanadium contents, representing the feed, concentrate, and tail samples (Figure 11-10 and Figure 11-11).

On the basis of the XRF analysis by Rautaruukki laboratory, the accuracy of the ICP-OES method of Labtium was good. According to the best fitting trendline the FeLab/FeRR ratio was 1.05 and the VLab/VRR ratio was 0.90 (RR stands for Rautaruukki Laboratory and Lab for Labtium).

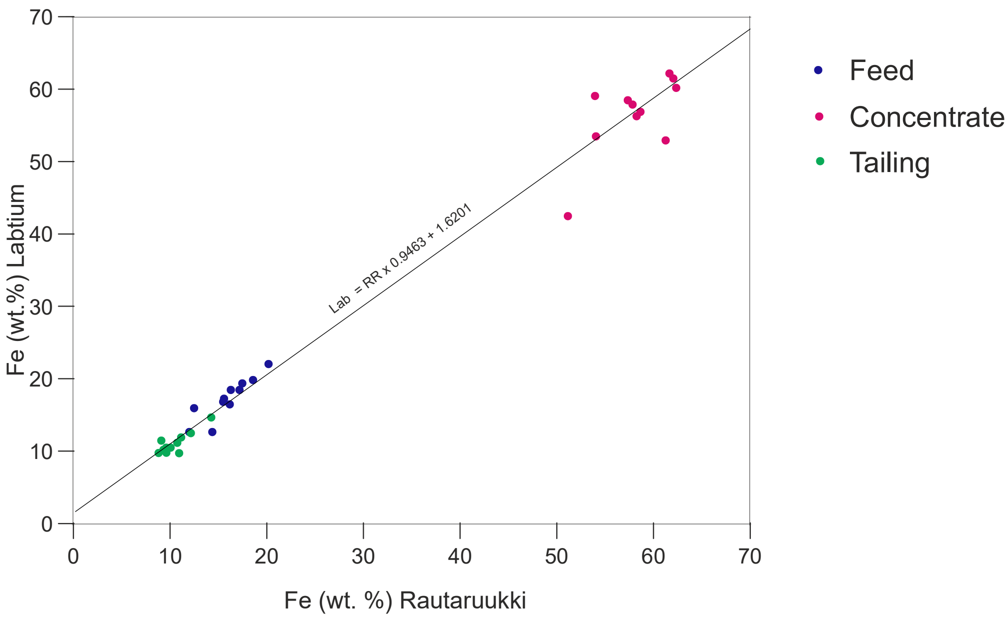

Figure 11-10 Correlation between the Fe contents in duplicate sample analyses determined with different analytical methods. The equation defined the best fitting linear correlation.

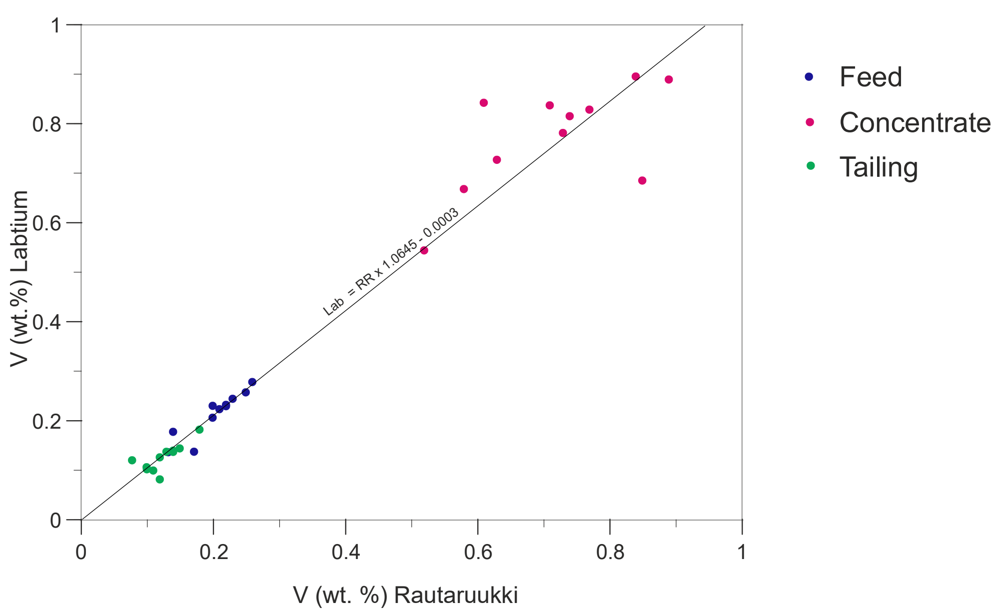

Figure 11-11 Correlation between V contents in the duplicate sample analyses determined with different analytical methods. The equation defines the best fitting linear correlation.

11.4.4 Grinding and DDT Runs

The precision test of the ICP-OES method of Labtium showed that the concentrate and tail samples had poor correlations with their original and duplicate sample compositions. The source of the error was either grinding, DDT method, or both.

Grinding may have affected the DDT result and thereby changing the assay results of a sample. This would have been because the grinding procedure on rocks of different physical properties, such as varying textures and mineralogy, may have responded differently to pulverization.

The particle size data of the 5 concentrate samples indicated that in spite of different physical properties of the Mustavaara ore (Table 11-2), the grinding by Labtium worked well. The grain size distributions of the concentrate samples are similar. In all samples more than 90% of grains were less than 100 μm in size (Figure 11-12).

Table 11-2 Descriptions of the concentrate samples used in the grinding test

Sample	Drill Core	Subzone	Magnetic mass%*	Satmagan in tail**	Fe in conc. (wt. %)	V in conc. (wt. %)
001	MV-57-2011	OUL	23.19	1.60	60.40	0.87
211	MV-62-2011	OML	15.57	1.22	59.40	0.87
212	MV-62-2011	Ore	30.72	1.45	58.10	0.83
213	MV-62-2011	OML	12.13	0.89	60.60	0.81
239	MV-63-2011	OUL	18.82	2.41	58.30	0.80

*mass fraction of magnetic phases in feed sample according to DDT run.
** mass fraction of magnetic phases in tailing sample according to Satmagan.

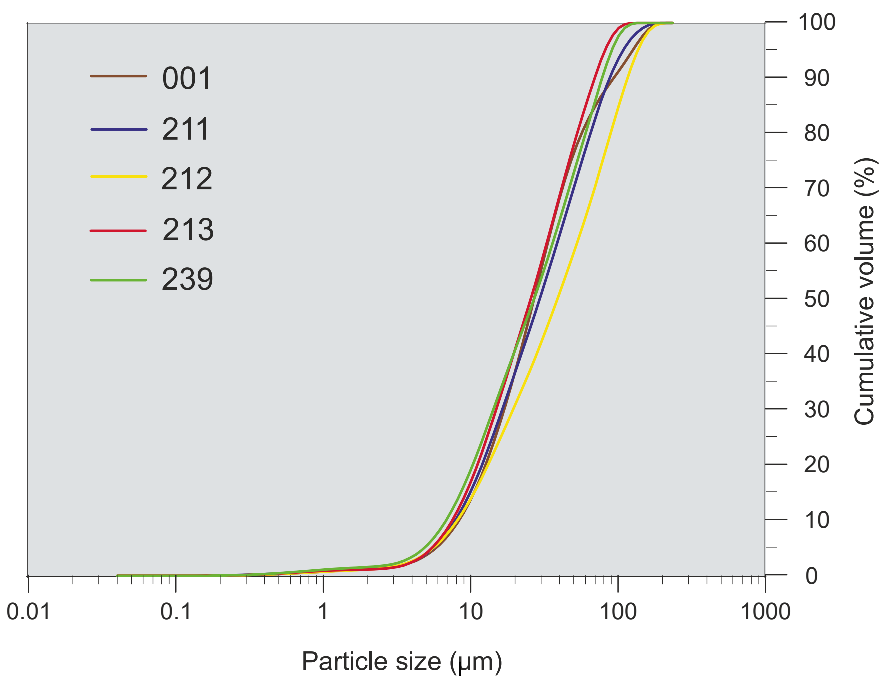

Figure 11-12 Cumulative grain size distribution of concentrate samples chosen for particle analysis

The DDT appears to have been the most critical parameter in view of quality assurance and control. This has been illustrated with the determinations of mass fraction of magnetic phases (Magnetic wt. %) and Satmagan readings of tail samples (Satmagan_tail) (Figure 11-13). The correlation between these two parameters indicates that during runs Mintec's DDT machine failed to collect all magnetic phases to concentrate, which was clearly related to the amount of magnetic phases in the feed sample.

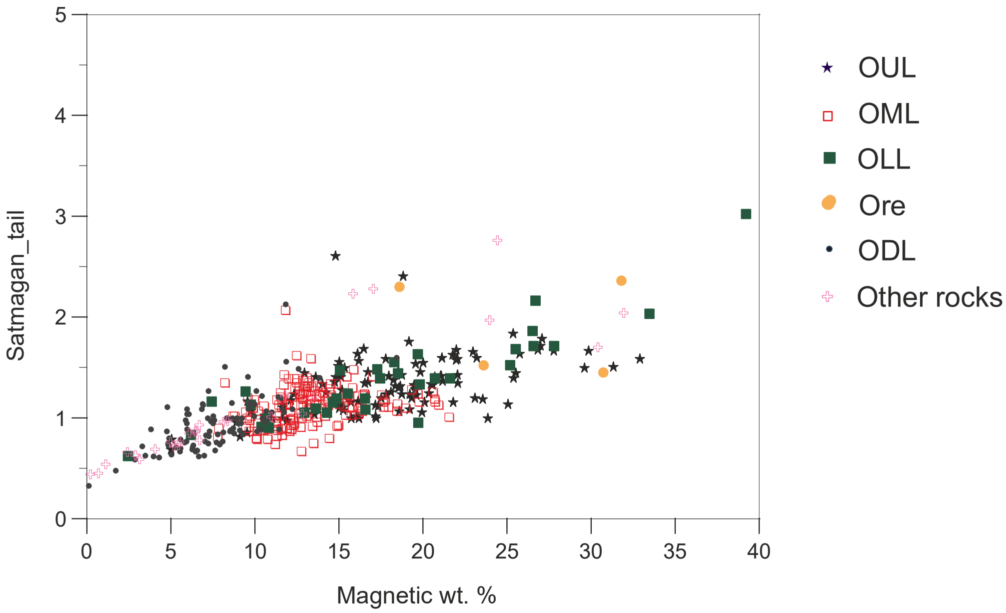

Figure 11-13 Relation between Satmagan values of the tail samples (Satmagan_tail) with mass fraction of magnetic phases (Magnetic wt. %) from the drill core collected by MKOy. Note that the Satmagan value indicates the amount of magnetic minerals (in wt. %) in the tails, whilst the mass fraction of magnetic phases is the volume of the concentrate in the total weight of the rock, was determined using the DDT run results. Classification was made on the basis of lithology.

11.5 Results of 2011 Drilling

The drillers placed the drill core into wooden boxes. Wooden tags marked with downhole depth were placed in the box. The core was strapped together with a lid placed on the uppermost box and loaded into a vehicle. The core was then transported to MKOy's logging and processing facility. At the drill site, a Reflex Maxibor II downhole survey was done once the hole was completed to determine drill hole deviation. All of the holes showed a small amount of right hand deviation, with two holes (MV-67-2011 and MV-70-2011) showing significant downhole deviations of 15-20°. All of the drill holes in the 2011 program intersected the ore zone, which has been confirmed with the susceptibility surveys of each hole.

The assay results of the MKOy drill program showed that the ICP-OES multi-element analyses correlated well with the earlier analytical results from the historic drill programs conducted by Rautaruukki Oy. In the concentrate samples, the analyses by MKOy have similar vanadium and iron values as with the Rautaruukki Oy holes, but the MKOy holes show a wider spread towards lower values (Figure 11-14). The increase in lower vanadium and iron values in the data was related to MKOy's processing procedures and them choosing to sample the entire hole and not just the ore horizon as done with Rautaruukki Oy's program (Figure 11-15).

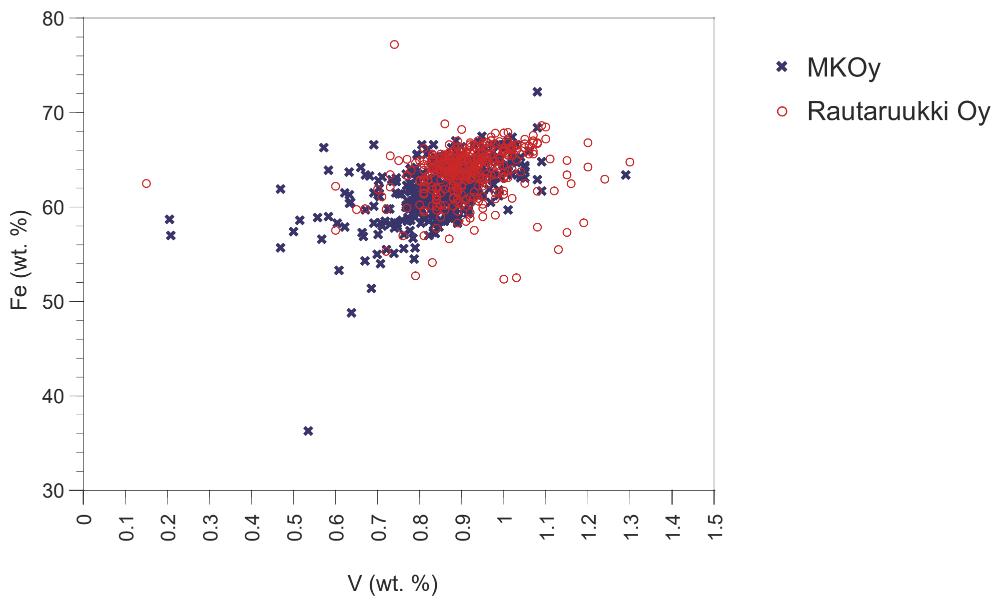

Figure 11-14 Fe vs. V diagram illustrating the composition of the magnetic fraction (concentrate) in the samples from Rautaruukki Oy and MKOy

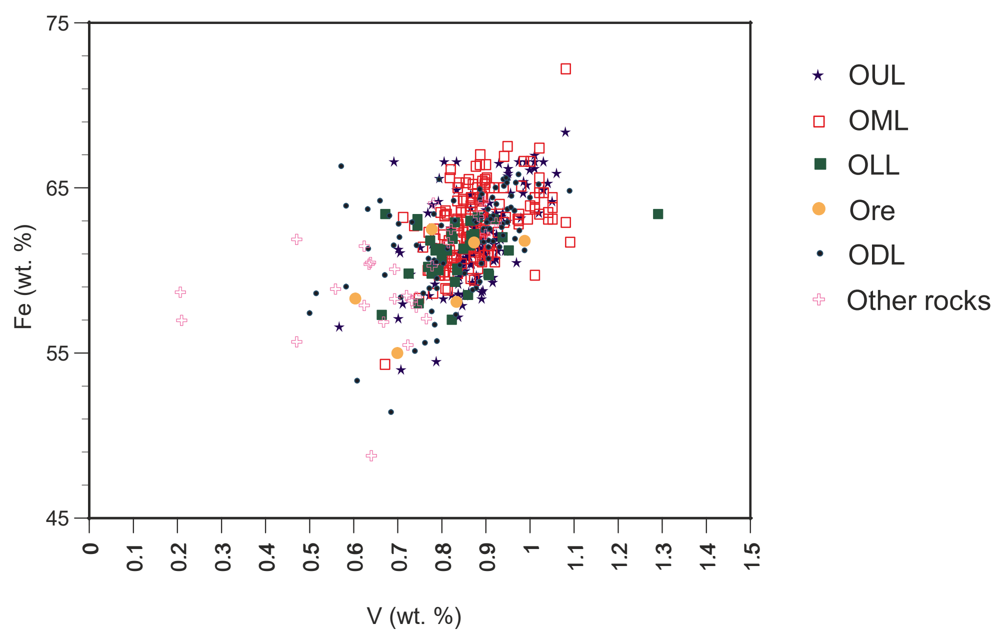

Figure 11-15 Fe vs. V diagram illustrating the composition of the magnetic fraction (concentrate) in the samples of MKOy. Classification is made on the basis of rock type

Figure 11-16 compares the Mintec Satmagan and DDT results from the 2011 drill holes with the results from the historic drill holes completed by Rautaruukki Oy. As with the analytical results, the 2011 drill holes appear to be similar with the historic data, while also showing a wider spread with increased lower values due to increased assaying of unmineralized material. In all of the ore types there appeared to be a negative correlation with the Satmagan values of concentrate and mass fraction of magnetic phases. This was a textural feature of the ore, that indicated the samples containing the highest amounts of oxides showed the highest amounts of ilmenite lamellae in oxide grains.

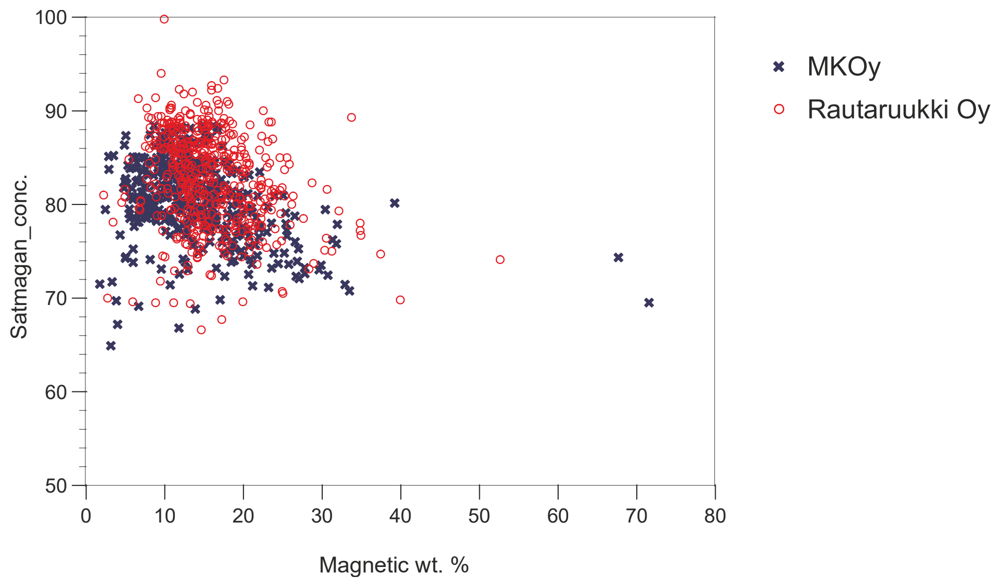

Figure 11-16 The correlation between Satmagan values of concentrate (Satmagan_conc.) with mass fraction of magnetic phases (Magnetic wt. %) in the samples collected from the drill cores of Rautaruukki Oy and MKOy. Note that the Satmagan value indicates the amount of magnetic minerals (in wt. %) in concentrate, whilst the mass fraction of magnetic phases is the weight volume of concentrate in a rock sample, determined using DDT run results.

11.6 Due Diligence Check

Strategic conducted their own due diligence (DD) checks in December of 2019 and January of 2020 prior to the purchase of the property. As part of the DD checks, pulps from the 2011 concentrate samples were recovered from Ferrovan Oy's core storage warehouse. 151 of the historic pulp concentrates were analysed with a handheld XRF. Of those samples, 50 were selected at random for additional DD checks and were sent to ALS Laboratory for Davis Tube analyses. The XRF pulp concentrate values were compared to the original database values and shown in Figure 11-17. The ALS pulp concentrate Davis Tube reruns were compared to the database values and shown in Figure 11-18.

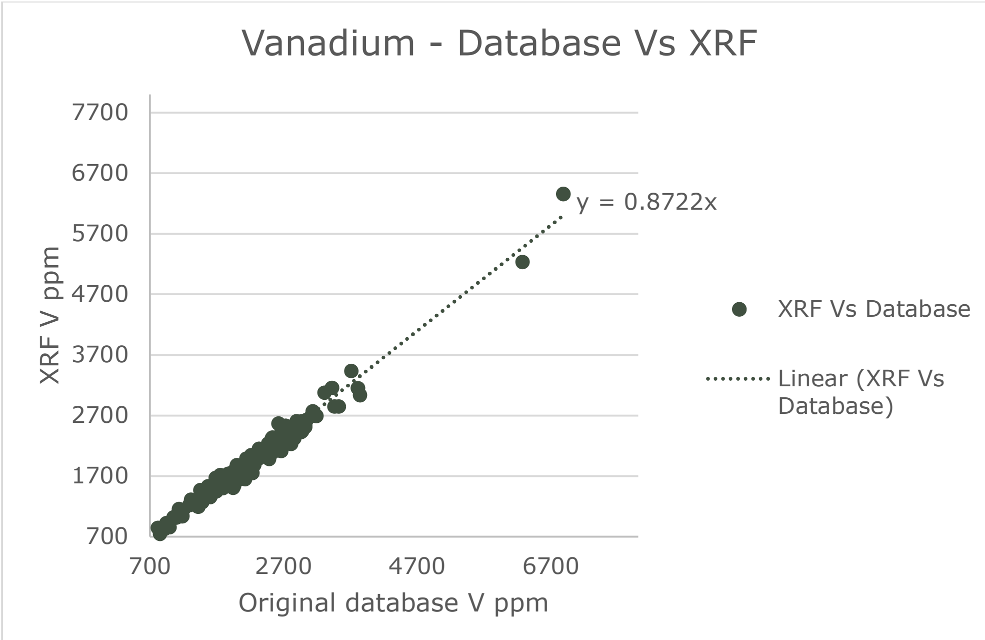

Figure 11-17 Pulp concentrate –XRF pulp concentrate values versus original database values.

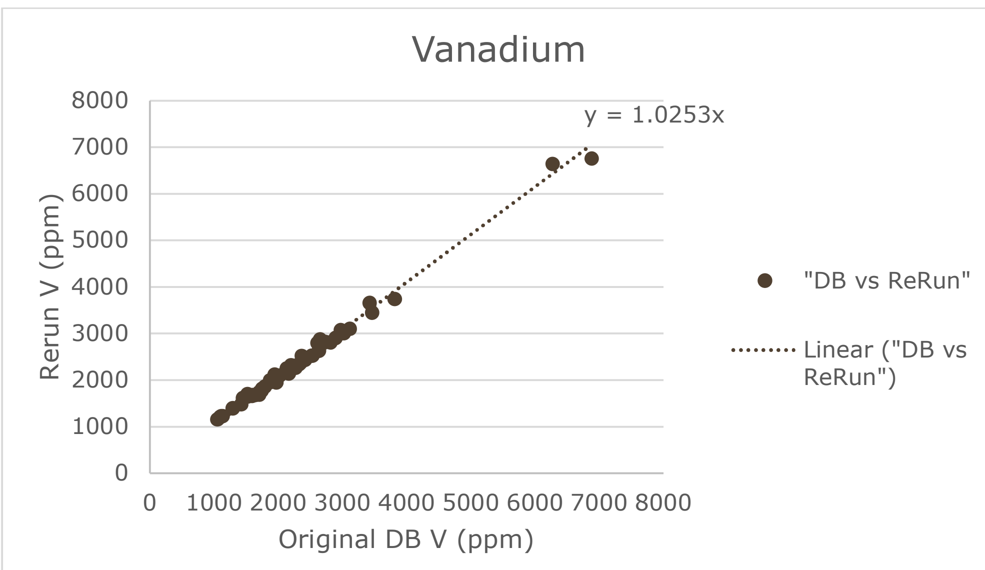

Figure 11-18 Pulp concentrate – ALS pulp concentrate Davis Tube reruns versus original database values.

The second part of the DD checks included XRF measurements on 8 historic Rautaruukki Oy drill holes, photographing 10 drill holes and sampling 13 intervals from 2 historic drill holes from GTK's National Drill Core Archive. It was noted that limited material remains available for resampling as much of the core has been resampled by previous owners of the property. These quartered core samples were sent to ALS laboratory for Davis Tube reassay analysis.

Figure 11-19 shows the reassayed historic core samples versus the original database values. It was concluded that there was an acceptable correlation between the original concentrate values and the pulp reassayed values. The resampled core versus the historic database did show a little bit more variation but it was also considered acceptable. Variation in the core resampling was to be expected considering the age of the core and the amount of times it has been moved and looked at by previous geologists. There is also generally greater sample variations when a sample has been recut, crushed and pulverized since the mineralogy may have changed with subtle variations in this new section of ¼ core.

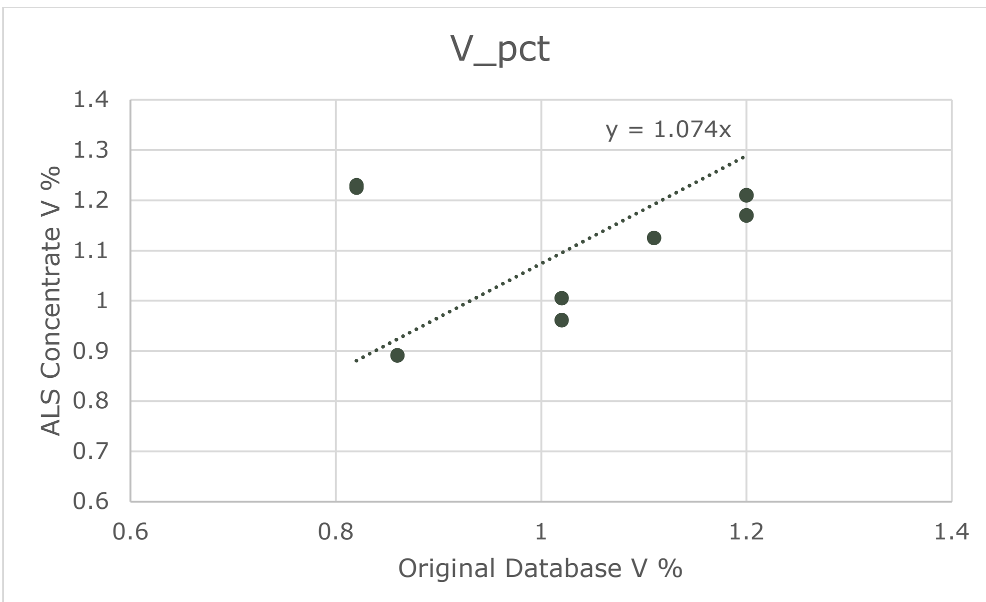

Figure 11-19 Concentrate values from resampled core versus the original database concentrate values.

12 Data Verification

12.1 Database Validation

The drill core database was checked for errors, no duplicates, unmatched values or overlapping samples were found.

12.2 Down-Hole Survey Validation

Only the new (2011 drill campaign) drill holes have been downhole surveyed for azimuth and dip. The data was validated by checking the consistency of consecutive survey results.

12.3 Assay Verification

The collar, geology, survey and assay files were provided in Excel®. All From-To data are either zero or a positive value. No intervals exceeded the total depth of its drill hole. Intervals with no assay data were listed as -1.0 in the database. Those negative values were changed to zero. The core recover for the 2011 drill program was excellent. There is no indication that grade is related to core recovery.

12.4 Geologic Data Verification and Interpretation

The author has compared the lithological drill core loggings against the drill core photos taken during the due diligence work.

12.5 QA/QC Protocol

Quality control and quality assurance work is documented by independent consultant Markku Iljina from GeoConsulting as part of the mineral resource estimation work done in 2013. For this report the author has reviewed this information and has found the data to be adequate.

12.6 Conclusion

After reviewing the available data the author considers the drill hole data to be suitable for estimation and reporting of the Mineral Resource estimate.

13 Mineral Processing and Metallurgical Testing

13.1 Pyrometallurgical Test Work Results

The Swerea Mefos research institute (Mefos) located in Luleå, Sweden has done research on behalf of MKOy regarding ferrovanadium production for Mustavaara. The following topics have been studied and reported on during the years 2008 to 2009.

- Titanium-removal from pig iron obtained from reduction of Mustavaara iron concentrate - summary of results from trials in the Mefos 150 kg induction furnace, 16.2.2009.
- A novel process for vanadium and iron recovery from the Mustavaara iron-titanium-vanadium concentrate - summary of results from reduction and oxidation tests in a 150 kg induction furnace, 5.9.2008.
- Summary of results from reduction trials of Mustavaara magnetite ore for vanadium-recovery, 28.4.2008.

The most relevant reports and results from pyrometallurgical tests have been summarized below from Appendix VII in the Pre-feasibility study completed by Pöyry Finland Oy in 2012.

13.1.1 Summary of Results from Reduction Trials of Mustavaara Magnetite Ore for Vanadium Recovery, 28.4.2008

Anthracite, slag formers and concentrate were placed inside a crucible and were melted in the Mefos Tamman furnace in an inert nitrogen atmosphere at 1,650 °C. Slag and metals were weighed and sampled. The determination of the mixture ratios was based on the theoretical calculations of the anthracite needed and practical experiences on the reduction kinetics and slag chemistry. A total of four tests were carried out, the mixtures differed in the amount of reductant in order to control the reduction degree of V_2O_5 and TiO_2, crucible material and slag former additions. Lime was added in the aim of obtaining a CaO/SiO_2-ratio of 1.20. Fluorspar was added in two of the tests to lower the viscosity and thereby improving metal and slag separation.

- A vanadium recovery of 84% in the metal was obtained
- An iron recovery of almost 100% in the metal was obtained
- A titanium recovery of 99% in the slag was obtained

13.1.2 A Novel Process for Vanadium and Iron Recovery from the Mustavaara Iron-Titanium-Vanadium Concentrate - Summary of Results from Reduction and Oxidation Test in a 150 kg Induction Furnace, 5.9.2008

The process concept based on pyrometallurgical treatment of Mustavaara iron-titanium-vanadium concentrate for co-production of iron and ferrovanadium consisted of three steps:

Step 1: Total reduction of the ore concentrate making a metal with about 1.3% vanadium

Step 2: Selective oxidation of the metal from Step 1 to enrich vanadium in a slag phase aiming at a vanadium/iron ratio greater than 1

Step 3: Final reduction of the vanadium slag for production of ferrovanadium

Step 3 is already practiced commercially and not considered in this study. Step 1 was tested in 3 kg scales at Mefos.

The original Mustavaara concentrate for production of the metal needed was used in Step 2. About 82 kg of iron melt was successfully produced from the 150 kg Mustavaara iron concentrate using anthracite as reductant. The reduction work was performed in the Mefos 150 kg induction furnace with a graphite crucible. The highest iron and vanadium recovery achieved was 99.6% and 93.3% respectively producing a metal with 1.23% vanadium.

- By using iron oxide combined with a synthetic slag containing Al_2O_3-CaO, liquid slag containing up to 13% vanadium could be obtained at approximately 1,550 °C. The targeted vanadium/iron ratio >1 in the slag was easily achieved.
- The vanadium content in the metal phase was lowered from 1.19% to 0.24% corresponding to a vanadium recovery of 80%. This could however easily be increased to >90% by further oxidizing the vanadium to less than 0.1%.

It was also found that for reduction heats with high vanadium yields the titanium-level was also high and the slag became more viscous.

13.1.3 Titanium Removal from Pig Iron Obtained from Reduction of Mustavaara Iron Concentrate - Summary of Results from Trials in the Mefos 150 kg Induction Furnace, 16.2.2009

Since the iron concentrate from Mustavaara would have high contents of titanium oxide, some of the titanium will also be reduced to the metal phase together with iron and vanadium. This was discovered in the reduction tests previously mentioned. To obtain a high vanadium recovery the typical pig iron produced will contain about 1.2% vanadium, 0.6 - 1% titanium, 4 – 5% carbon and 0.5 - 0.8% silica. The high Ti content is a potential problem for the vanadium recovery step (Step 2).

With this background it was decided by Akkerman Exploration B.V. and Adriana Resources Inc. to test a titanium removal concept proposed by Mefos in the autumn of 2008. The primary purpose of this study was to find a suitable method for efficient titanium removal with the lowest possible loss of vanadium using oxygen and nitrogen-based reagents. Both preliminary thermodynamic calculations and trials in the Mefos 150 kg scale induction furnace have been conducted to prove the concept.

The main results and conclusions are summarized below:

- The test results from the 150 kg scale tests have proved that the proposed concept for titanium removal is technically feasible. The titanium content could be efficiently lowered from over 1% down to below 0.1% (>90% titanium removal) and at the same time keeping the vanadium content stable at about 1.1% in the metal.
- All tested reagents including magnetite fines (Fe_3O_4), CO_2, N_2 and FeSiN, have shown good capability for titanium removal. The degree of titanium removal from the oxygen-based reagents was up to 96% and for the nitrogen based reagents was up to about 80% without any additional optimization of the process.
- The chosen slag formers based on the CaO-SiO_2-MgO system seemed to be efficient with good slag fluidity and high titanium capacity. A slag with up to 30% $Ti0_2$ has been achieved with low vanadium contents, about 1% or lower. The titanium oxide content could probably be increased to 50%.
- In accordance to the theoretical study the oxygen-based reagents were more efficient than the nitrogen based reagents.

Based on the experimental results of this series of test work it has been suggested to use iron oxide for the purpose of titanium removal as it would be simple and efficient. The best choice for this would be to use the Mustavaara iron concentrate as it would be the most cost efficient.

13.1.4 Smelting Reduction Trials with a 3 MW DC Furnace (2012)

23 metric tonne of Mustavaara concentrate was treated in a 5-day pilot campaign using the 3 MW DC furnace in Swerea MEFOS. Anthracite was used for reduction of iron oxide and vanadium oxide from the ore. In addition, 2.5 metric tonne of LD slag was treated as vanadium source and slag former. In total 26 metric tonne of hot metal and 3.9 metric tonne of titanium slag were produced. A vanadium recovery of about 90% can be anticipated while maintaining good reduction selectivity prior to both silicon and titanium. The overall iron recovery was 98 to 99%.

13.2 2011 Comminution Test Work Summary

Comminution test work has been undertaken by ALS Ammtec during the year 2011. The comminution test work was carried out on ore samples from Mustavaara mine. The test work included the following procedures (results summarized in table 13-1):

- Bond impact crushing work index determination
- JK drop-weight test work
- SMC test work
- SAG Design test work
- Bond abrasion index determination
- Bond rod mill work index determination
- Bond ball mill work index determination

Table 13-1. Summary of comminution test work for main ore composites.

Test	Value	Unit	Description
Bond Impact crushing work index determination (Cwi)	19.8	kWh/t	
JK drop-weight test work (A*b)	40.315		Moderate hard range resistance to impact breakage
JK drop-weight test work (ta)	0.15		Very hard abrasion range
SMC test work (A*b)	39.2		Moderately hard
SAG Design test work (SAGW)	10.79	kWh/t	Hard
Bond abrasion index determination (Ai)	0.5585	g	
Bond rod mill work index determination (Rwi)	18.7	kWh/t	Hard
Bond ball mill work index determination (Bwi, closing screen 53 µm)	20.4	kWh/t	Very hard

Comminution indices indicate that Mustavaara ore falls generally in the range of moderately hard – very hard in the various tests.

13.3 2012 GTK Mintec Pilot Plant Test Work

A metallurgical test work on Mustavaara ore was carried out at the Geological Survey of Finland (GTK) Mintec pilot plant in Outokumpu in February – April 2012. For the continuous production run, Mustavaaran Kaivos Oy delivered a 700 t feed ore sample from Mustavaara Fe-V-Ti open pit. The grades of the feed sample were 17.4% for Fe, 0.216% for V and 5.08% for TiO2. Pilot plant flowsheet comprised crushing, homogenization of the feed sample and four-stage milling and low-intensity magnetic separation (LIMS). 105 t of concentrate sample grading 61.3% of Fe, 0.85% of V, 3.06% of SiO2 and 7.35% of TiO2 was produced. Average magnetite recovery of 96.8% to concentrate was achieved in the pilot tests.

13.4 Roasting, Leaching and Precipitation Test Work with Mustavaara Vanadium Slag (2013-2014)

Hydrometallurgical test work on Mustavaara ore was carried out at the Geological Survey of Finland (GTK) during 2013 and 2014. The main results of the roasting, leaching and precipitation tests with the vanadium slag were:

Vanadium could be successfully extracted from V-slag using sodium carbonate roasting at 800 ºC and water leaching methods. The highest vanadium recovery was 96%. The differences on recoveries between different size fractions seemed to be negligible.

Almost complete vanadium recovery was achieved in the two stage precipitation tests: Precipitation of silicates by hydrated aluminium sulphate and then precipitation of ammonium metavanadate by ammonium sulphate. The recoveries were over 97%.

The chemical analyses of final precipitates showed that the high-quality ammonium metavanadate can be produced using above mentioned two stage precipitation procedure.

13.5 Summarizing Conclusions

- Continuous pilot plant test work with Mustavaara ore has been conducted in 2012 producing 105 t of concentrate grading 61.3% Fe and 0.85% V.
- Comminution indices of Mustavaara ore fall in the range of moderately hard – very hard.
- Hydrometallurgical test work including roasting, leaching and precipitation tests has shown that high quality metavanadate can be produced.
- Vanadium, iron and titanium recoveries of 84%, almost 100% and 99% respectively can be obtained through conventional smelting.
- Reduction and oxidation tests yielded the highest iron and vanadium recovery of 99.6% and 93.3% respectively producing a metal with 1.23% vanadium.
- Smelting tests at a 150 kg scale have confirmed that using oxygen and nitrogen-based reagents, it is possible for titanium to go from over 1% to below 0.1% while keeping the vanadium content stable at about 1.1% in the metal. Oxygen based reagents proved more effective removing up to 96% of the titanium.

14 Mineral Resource Estimates

The Mineral Resource estimate that is used as a basis for this study has been prepared by Ville-Matti Seppä and Pekka Lovén (AFRY 2020). Since the release of the last resource estimate, there has been no material change and there has not been any new exploration activities concerning the property and the end products (ferrovanadium and pig iron) have remained the same.

The QP is not aware of any known environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other similar factors that could materially affect the stated Mineral Resource estimate.

The following chapters are results of AFRY (2020) work unless otherwise stated.

14.1 Data

The Mustavaara database was provided by Strategic in the form of Excel® spreadsheets containing collar locations, down-hole survey information, geologic data and assay results along with digital copies of drill core photos, historic reports and drill logs. The resource database contains drill hole data from 73 drill holes with a total length of 9,911.2 m. A total of 1,156 intervals are included in the database with 1,036 intervals that have assay data and 120 intervals with no assay data. Half of the 120 intervals with no data were in unmineralized sections either in the hanging wall or footwall of the deposit with 58 samples that were possibly believed to be mineralized but did not have enough magnetic content in the sample to return a value from the assay lab. The drill core samples have been assayed for (ilmeno)magnetite, VinMC (Vanadium in magnetite concentrate), V_2O_5_eq (VinMC converted to vanadium pentoxide equivalent), titanium (in magnetite concentrate) and iron (in magnetite concentrate). The magnetic susceptibility has also been included in the assay database. Individual sample lengths vary from 0.23 m to 15.82 m with an average of 4.90 m for the 1957 - 1976 program and 2.71 m for the 2011 program. Ilmenomagnetite is labelled as magnetite within the database and also in related resource tables. The database contains calculated density values which were used for ore, while waste rock was assigned a value of 3.0 g/cm^3. Downhole survey data was only available for holes in the 2011 drill hole program.

Geologic information was gathered during the various drill phases and include six main rock units with five ore subunits.

14.2 Drill Hole Compositing

The resource estimate was based on resource intersections defined using the wireframes of the mineralized zones. Intersection data was used to extract samples for statistical analysis and for compositing the data for grade interpolation. Drill hole sample composites were generated in order to standardize the data for further statistical evaluation which would eliminate any adverse effects related to sample length. The average length of the composite was defined as 4 m according to the average assay interval for samples above 5% magnetite. Basic statistics related to the composites used in grade estimates are presented in Table 14-1. The data set shows low Coefficient of Variation which indicates low variability of the data.

Table 14-1 Basic statistics of the composited data used in the grade estimations

Variable	**4 m composites**			
	Ore			
	Magnetite_%	**VINMC_%**	**Ti_%**	**FE_%**
Number of samples	741	741	496	497
Minimum value	5.04	0.55	1.34	55.55
Maximum value	39.21	1.29	9.12	68.57
Mean	15.69	0.90	3.91	63.43
Median	15.07	0.90	3.71	63.72
Geometric Mean	15.17	0.89	3.70	63.39
Variance	17.40	0.01	1.81	5.36
Standard Deviation	4.17	0.09	1.35	2.31
Coefficient of variation	0.27	0.10	0.34	0.04

14.3 Resource Modelling and Block Model

The resource outline for the 3D model was constructed using cross sections taken at 100 m intervals. The nominal cut-off grade to be used in creating the 3d solid was determined to be 8% magnetite. The cut-off value was reconsidered while calculating the resource estimation. An 11% magnetite cut-off value was selected for the mineral resource estimate based on break even review of the assumed operating expenses and revenues. In this PEA study the assumed costs and prices are re-evaluated and presented in chapters 16.14.

Compared to the 2013 geologic model, the dimensions for the current geological model were extended to cover an additional 50 to 100 m along the depth of the ore body. Figure 14-1 illustrates the plan view of the Mustavaara ore body.

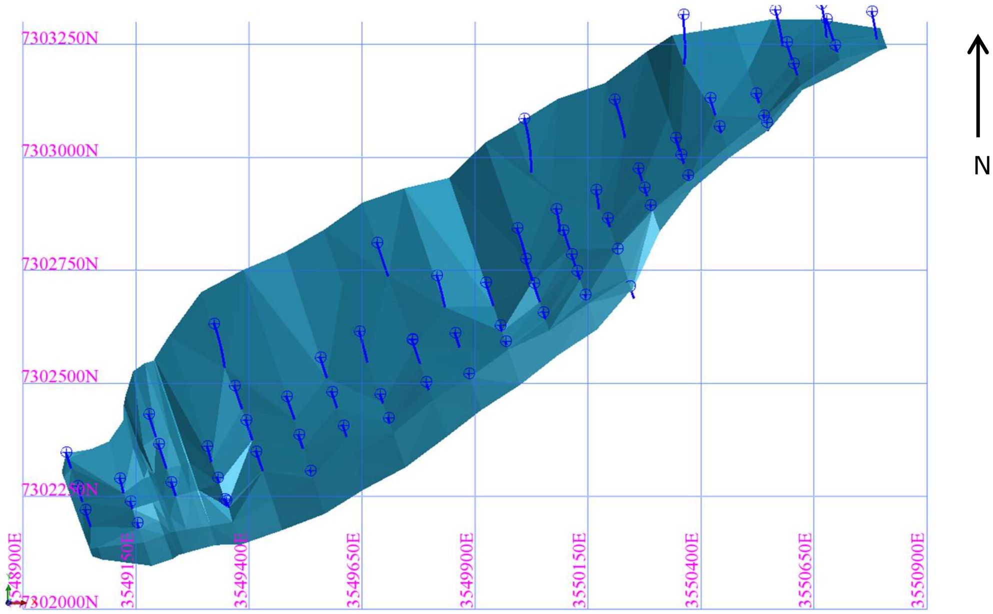

Figure 14-1 Plan view of the Mustavaara 3-D ore body model with drill hole traces

The resource block model was created using GEOVIA Surpac™ software. The block sizes for the resource model were selected to measure 20 m x 20 m x 12.5 m, based partly on the basis of drilling density and partly on the smallest mining unit. The summary of the block model parameters are given in the

Table 14-2.

Table 14-2 Mustavaara block model parameters

Type	**Y**	**X**	**Z**	
Minimum Coordinates	7302500	3548250	-100	
Maximum Coordinates	7305500	3549750	350	
User Block Size	20	20	12.5	
Minimum Block Size	10	10	6.25	
Rotation	55	0	0	
Attribute Name	Type	Decimals	Background	Description
an_dist_to_nearest_sample	Real	3	-99	Anisotropic distance to nearest sample
avg_true_dist_to_samples	Real	3	-99	Average true distance to samples
avgdst	Float	1	-1	Average search distance
density	Float	2	3	Density
dst2ns	Float	1	-1	Distance to nearest sample
fe_pc	Float	1	0	Fe in magnetite concentrate
magnetite	Float	2	0	Magnetite%
magn_nn	Float	2	0	Magnetite%, Nearest neighbor estimation
material	Integer	-	1	1=waste, 2=ore, 3=ovb,4=sirnionlampi, 5=air
ns	Integer	-	0	Number of samples used in estimate
num_of_dh	Integer	-	-99	Number of drill holes used in estimate
resource_class	Integer	-	0	1=measured, 2=indicated, 3=inferred
ti_pc	Float	2	0	Titanium in magnetite concentrate
vinmc_pc	Float	2	0	Vanadium in magnetite concentrate
vinmc_pc_nn	Float	2	0	Vanadium in magnetite concentrate, Nearest neighbor estimation

14.4 Grade Interpolation

4 m downhole composites were generated from the assay data prior to grade interpolation with consideration of the mineralized lens boundaries. The lengths of the composites were defined by the average length of the samples inside the mineralized envelopes. Interpolation of the magnetite and VinMC grades within the blocks was achieved by using the Inverse Distance squared method.

Three rounds of estimations were run with varying search radiuses that were based on geo-statistical results. Search ellipsoid parameters are presented below in

Table 14-3.

Table 14-3 Anisotropy Ellipse Parameters

Parameter Value	Value

Bearing	**55.6284**
Plunge	**0.0000**
Dip	**43.3694**
Anisotropy factors	
Parameter	**Value**
-------------------------	
major / semi-major	**2.576**
major / minor	**7.701**
Search ranges	
Round 1	**500**
Round 2	**250**
Round 3	**125**

A maximum search distance of 500 m was used to fill the blocks within the wireframes. The search ellipsoid was oriented according to the main continuity directions of the ore lenses. Block grades were estimated using a minimum of 5 and a maximum of 20 composites with respect to the search distances.

14.5 Validation

Validation of the block model was performed visually against the drill hole data in cross section views (Figure 14-2 and Figure 14-3). The block model was also validated in the domain level by comparing the mean values of the composited and estimated data (Table 14-4). These reviews did not reveal any inconsistencies between block model results and drill hole assays.

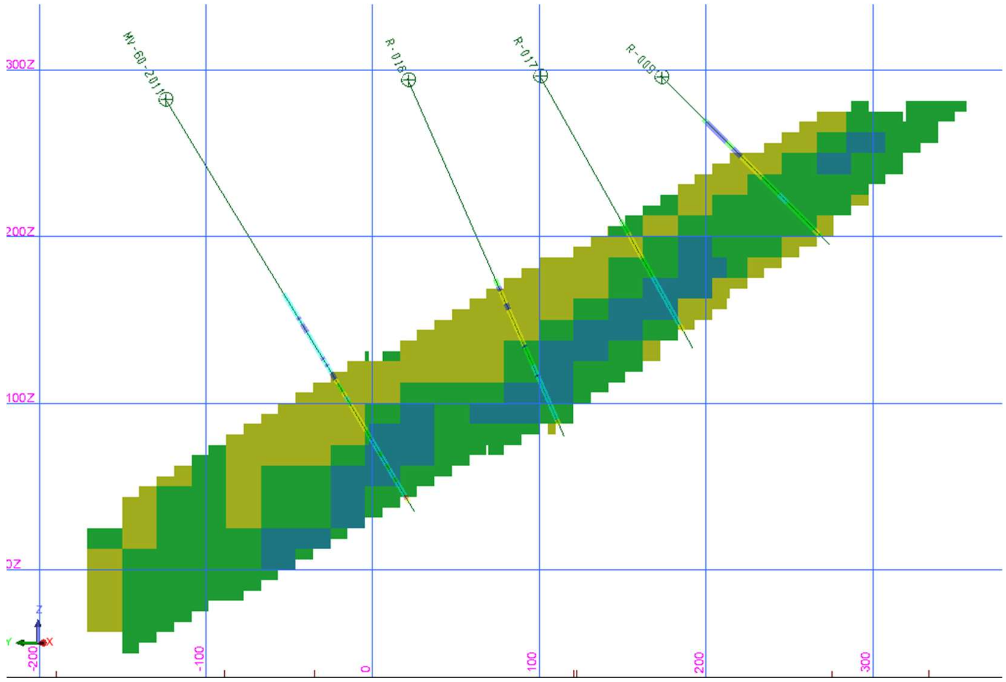

Figure 14-2 Section 9,700E viewing East displaying magnetite grades in blocks and drill holes. Turqoise indicates grades between 6 – 12%, green between 12 – 15% and yellow 15 – 25%.

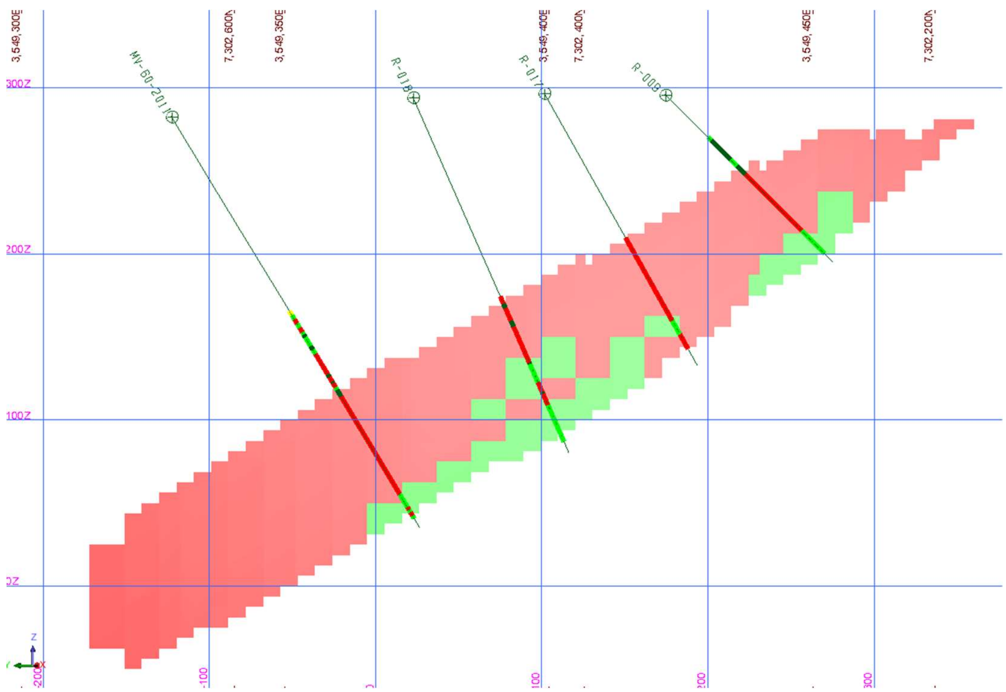

Figure 14-3 Section 9,700E viewing East displaying vanadium in magnetite concentrate grades in blocks and drill holes. Green indicates grades between 0.7 – 0.9% and red between 0.9 – 1.3%.

Table 14-4 Basic statistics of the block model and composites used to estimate the block grades.

	4 m composites		Blockmodel	
	Ore		Indicated Resource	
Variable	**Magnetite%**	**V in Mag. Conc.**	**Magnetite%**	**V in Mag. Conc.**
Number of samples	741	741	11,876	11,876
Minimum value	5.04	0.55	8.90	0.63
Maximum value	39.21	1.29	25.57	1.17
Mean	15.69	0.9	15.39	0.90
Median	15.07	0.9	15.14	0.90
Geometric Mean	15.17	0.89	15.15	0.90
Variance	17.4	0.01	7.48	0.00
Standard Deviation	4.17	0.09	2.74	0.07
Coefficient of variation	0.27	0.1	0.18	0.07

According to the basic statistics there was an acceptable variation between the estimated values and the composited values.

When comparing the volume of the geological 3D solids against the block model cells, a good congruence between the volumes can be seen. Figure 14-4 illustrates an oblique view of the Mustavaara 3D ore solid and the block model.

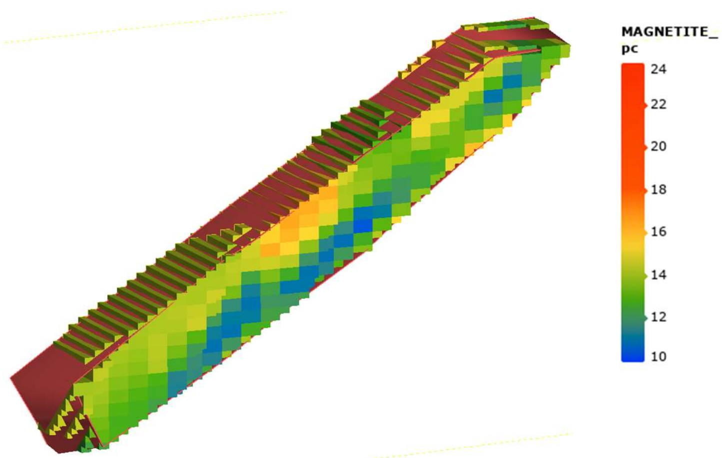

Figure 14-4 Volume comparison of 3D solid vs block model

The total volume difference between the 3D solid and the block model is only 0.36% (Table 14-5) and can be concluded that the volume difference is in a good range.

Table 14-5 Volumes of the 3D solid and the reported block model cells

Volume of 3D solid	**50 417 238 m^3**
Volume of reported block model cells	**50 596 875 m^3**
% difference	**0.36%**

The nearest neighbour (NN) method was a fast way to do a global validation of the resource model and it was used for the initial check in block model validation. Table 14-6 shows the comparison between the inverse distance estimation method and the NN method. The inverse distance method and nearest NN produced fairly similar grades.

Table 14-6 Comparison between estimation methods

		Magnetite%		VinMC	
Resource Class	Tonnes	Inverce distance	Nearest neighbor	Inverce distance	Nearest neighbor
	Mt	%	%	%	%
Measured	64.0	15.41	15.32	0.91	0.91
Indicated	39.7	15.27	15.49	0.88	0.87
Total M&I	103.7	15.36	15.39	0.90	0.89
Inferred	42.2	15.11	14.95	0.92	0.88

Swath plot analysis showed good correlation between the composited magnetite and VinMC grades versus the estimated grades from the block model. Swath plot analyses for magnetite grade are presented in Figure 14-5 and in Figure 14-6 and for VinMC grades in Figure 14-7 and Figure 14-8.

Figure 14-5 Swath plot analysis, Easting. Blue= Magnetite grade from composite file, Green= Magnetite grade from block model

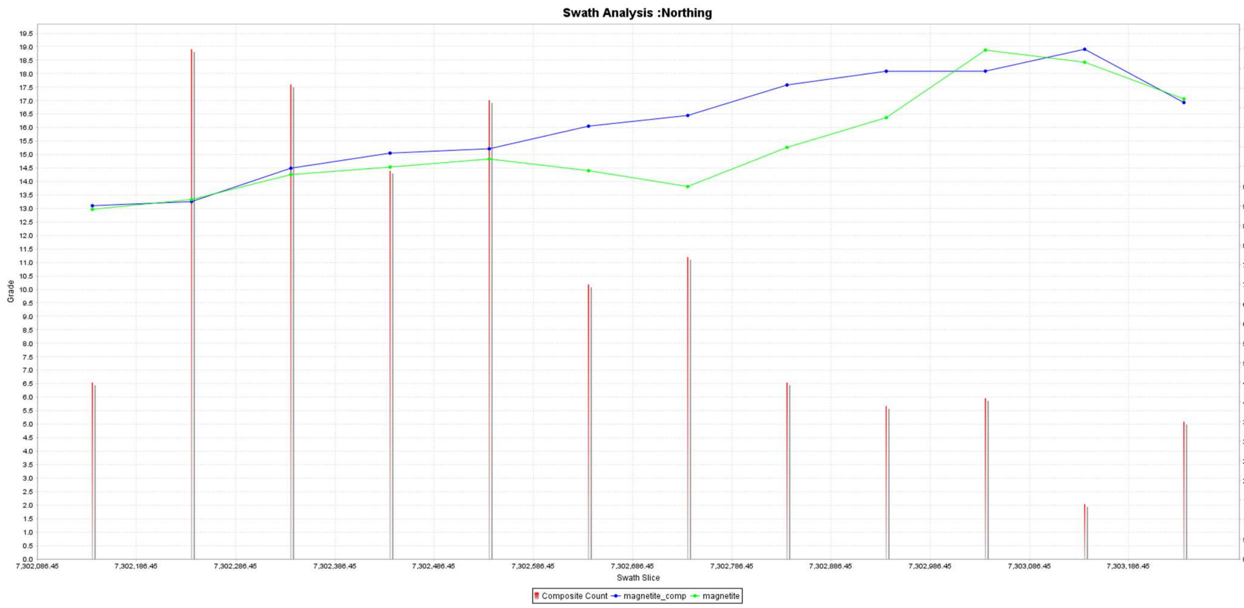

Figure 14-6 Swath plot analysis, Northing. Blue= Magnetite grade from composite file, Green= Magnetite grade from block model

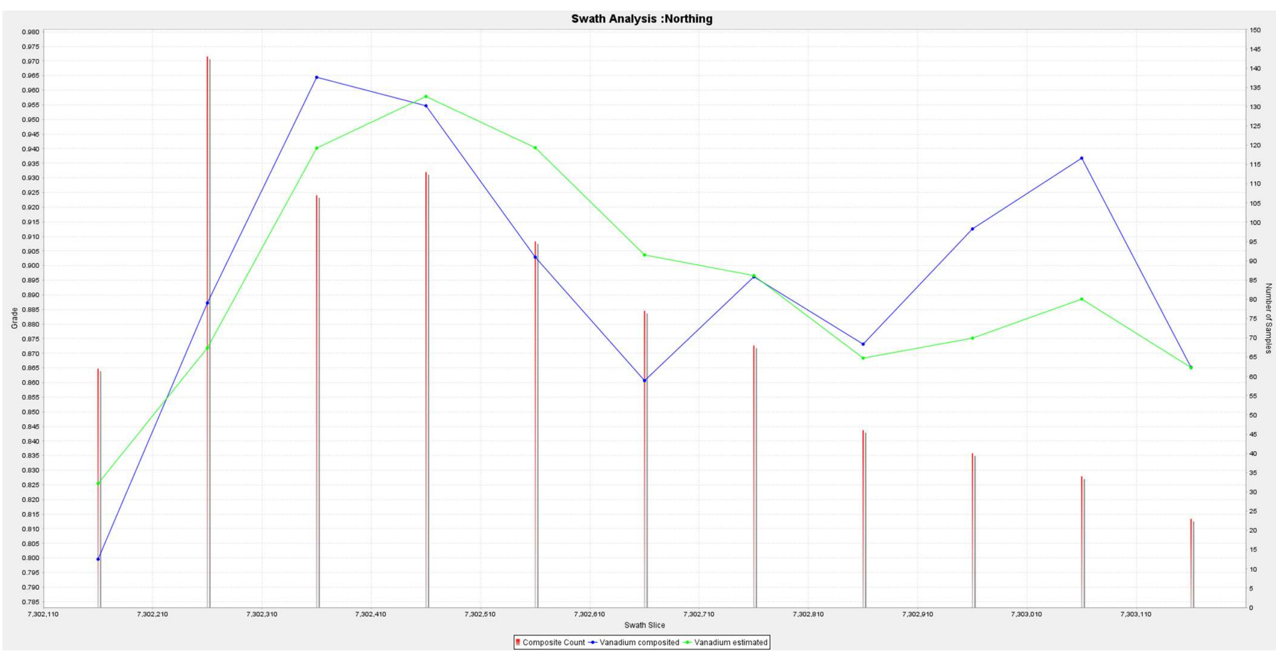

Figure 14-7 Swath plot analysis, Northing. Blue=VinMC grade from composite file, Green= VinMC grade from block model

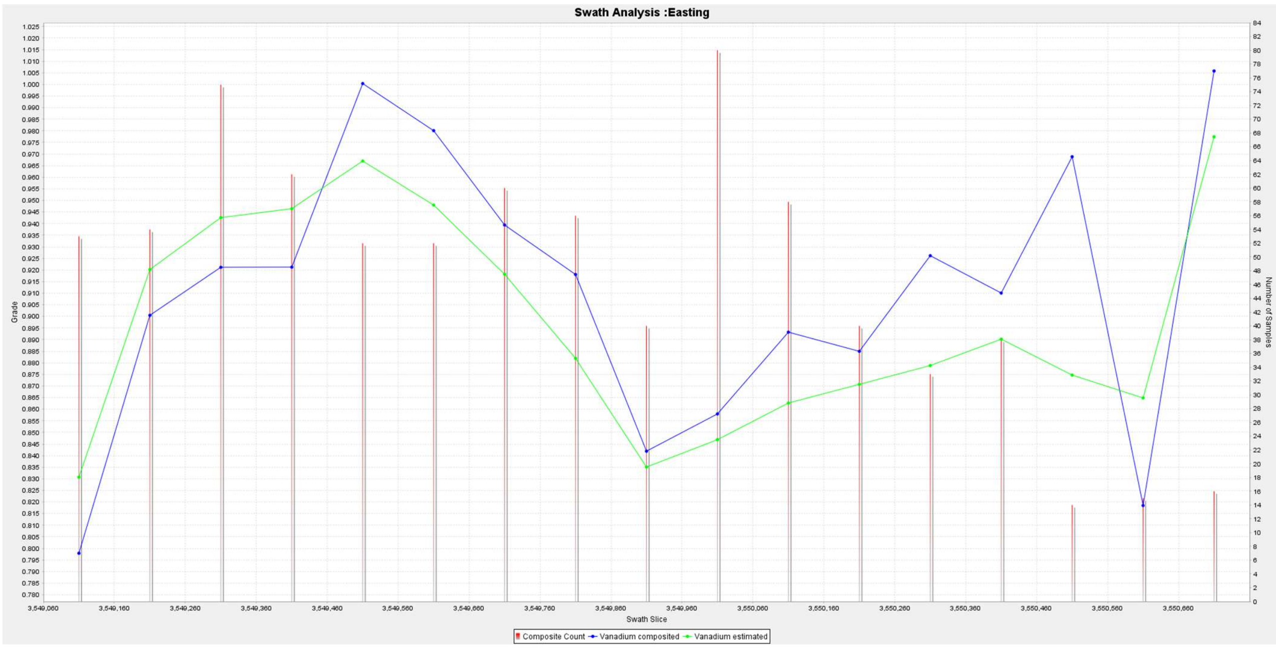

Figure 14-8 Swath plot analysis, Easting. Blue=VinMC grade from composite file, Green= VinMC grade from block model

14.6 Mineral Resource Classification

According to the Outotec Finland Oy (2012) report, the Mustavaara main magnetite layers (OLL, OML and OUL) are classified according to the current data as Measured and Indicated Mineral Resources. The classification is based on a very simple and well understood geological framework, the drilling density, detailed magnetic survey data, and confirmed continuities of magnetite gabbro layers in the historic 1,800 m long, 50 to 135 m deep open pit. The statistical and geostatistical analysis has shown that the magnetite and therefore vanadium content of the host rocks show geostatistical ranges greatly in excess of the current drill spacing.

The Measured Mineral Resource estimate has classified the mineralization using a drill spacing of 100 m by 50 m. The mineralization has a down-dip continuation from the bottom of the old open pit.

The Indicated Mineral Resource has classified the mineralization, which continues directly downward from the measured resource, using a drill spacing of 100 – 200 m by 100 m.

The Inferred Mineral Resource has classified the mineralization, which is projected to continue directly downward from the indicated resource, and continues 100 to 150 m downward from the deepest drill hole.

The geological framework controlling the uppermost measured mineralization continues unchanged to the indicated mineralization. To the west, the indicated classification has ended at the first indication of an internal anorthositic waste block. The volume of the anorthositic waste block cannot be determined with the current drill spacing density in that area. In the east the magnetite gabbro layer is known to thin and eventually die out. To the east, the indicated classification includes known mineralization with a thickness of at least 5 m.

The uppermost disseminated magnetite layer (ODL) has not been added to the Mineral Resource because of its variable nature. The true width, volume and exact location of anorthosite waste blocks do not follow a predictable pattern and remain more or less open even with increased drilling density.

The westward continuation of the magnetite gabbro can be considered good exploration potential. The downward continuation of the magnetite gabbro remains open and is estimated to continue with the same thickness and grade in the same kind of geological framework as with the known mineralization. Future drilling downdip of known mineralization can generate additional indicated and inferred resources. Figure 14-9 illustrates the Mustavaara mineral resource classes.

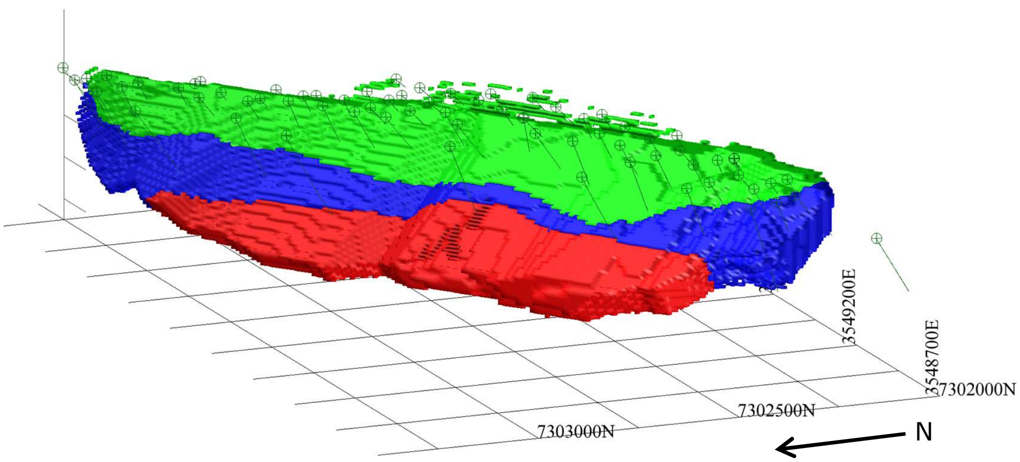

Figure 14-9 Oblique view of Mustavaara mineral resources, green = measured resources, blue = indicated resources, red = inferred resources

14.7 Mineral Resources

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) define a mineral resource as:

"[A] concentration or occurrence of solid material of economic interest, in or on the Earth's crust in such form, grade or quality and quantity, that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."

The "reasonable prospects for eventual economic extraction" requirement generally implies that quantity and grade estimates meet certain economic thresholds and that mineral resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recovery. For this PEA study the operational cut-off grade is calculated and presented in chapter 16.14.

Table 14-7 below summarizes the mineral resources using a magnetite cut-off grade of 11%. The author is not aware of any factors related to environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other relevant factors which could materially affect the mineral resource estimate contained in this Report.

Table 14-7 Mustavaara Mineral Resources as of the September 14th, 2020 @ 11% Magnetite cut-off

Resource Class		Average Grade				Contained Metal		
	Tonnes Mt	Magnetite (%)	VinMC (%)	Ti (%)	Fe (%)	VinMC (kt)	Ti (kt)	Fe (kt)
Measured Mineral Resource	**64.0**	**15.41**	**0.91**	**3.75**	**63.3**	**90**	**370**	**6 244**
Indicated Mineral Resource	**39.7**	**15.27**	**0.88**	**3.53**	**62.8**	**53**	**214**	**3 805**
Total M&I Mineral Resource	**103.7**	**15.36**	**0.90**	**3.67**	**63.1**	**143**	**584**	**10 049**

Note: VinMC refers to vanadium in magnetite concentrate, Ti refers to titanium in magnetite concentrate and Fe to iron in magnetite concentrate.

The depth extent of the mineral resources of the ore body is classified as Inferred Mineral Resources (Table 14-8). Although there is evidence that imply the geological and grade continuity in the depth extents of the ore body there is not sufficient data to categorise it into Indicated resources. Most of the inferred mineral resources can be upgraded to indicated mineral resource with diamond drilling.

Table 14-8 Mustavaara Inferred Mineral Resources as of the September 14th, 2020 @ 11% Magnetite cut-off

Resource Class		Average Grade				Contained Metal		
	Tonnes Mt	Magnetite (%)	VinMC (%)	Ti (%)	Fe (%)	VinMC (kt)	Ti (kt)	Fe (kt)
Inferred Mineral Resource	**42.2**	**15.11**	**0.92**	**3.75**	**62.3**	**59**	**239**	**3 971**

Note: VinMC refers to vanadium in magnetite concentrate; Ti refers to titanium in magnetite concentrate and Fe to iron in magnetite concentrate.

14.8 Sensitivity of Mineral Resources

The relationship between the magnetite cut-off grade and the resource tonnage is shown in Figure 14-10. The effects of selected cut-off grade to Measured and Indicated Mineral resource are shown in

Table 14-9. In Table 14-10 the sensitivity Inferred Mineral resources is shown against varying cut-off grades.

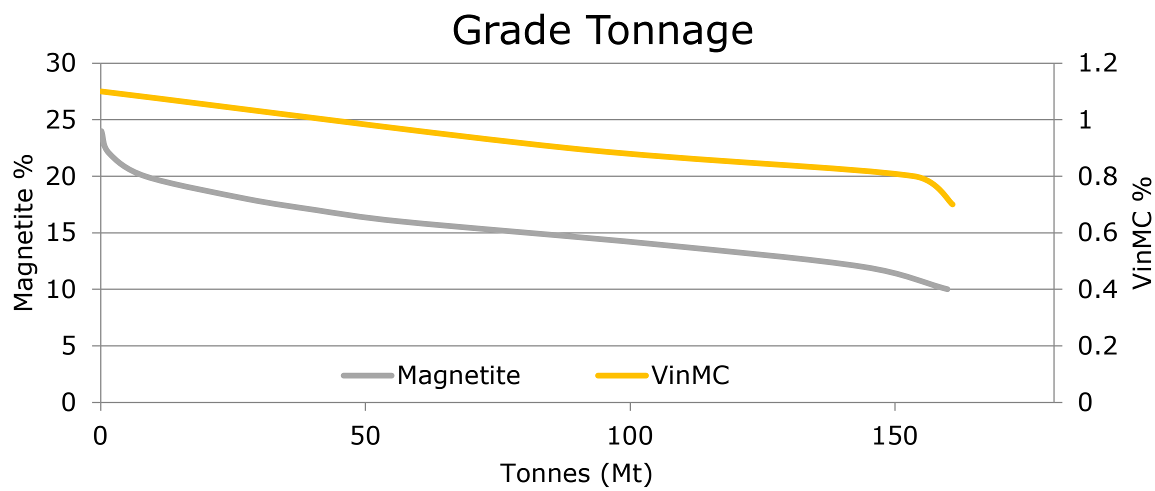

Figure 14-10 Mustavaara Grade-Tonnage curve

Table 14-9 Sensitivity of Measured + Indicated Mineral Resource to varying cut-off grades

		Average Grade				Contained Metal		
Cut-off	Tonnes	Magnetite	VinMC	Ti	Fe	VinMC	Ti	Fe
	Mt	(%)	(%)	(%)	(%)	(kt)	(kt)	(kt)
6	150	15.14	0.9	3.68	62.9	205	838	14,323
8	107	15.17	0.9	3.64	63.2	146	593	10,281
10	106	15.26	0.9	3.65	63.2	146	590	10,219
11	**104**	**15.36**	**0.90**	**3.67**	**63.1**	**143**	**584**	**10,049**
12	95	15.71	0.9	3.72	63	134	555	9,394
14	67	16.81	0.9	3.8	62.9	102	430	7,115
16	39	18.11	0.9	3.91	62.6	64	277	4,436
18	18	19.46	0.9	4.11	62.3	32	144	2,181

Table 14-10 Sensitivity of Inferred Mineral Resource to varying cut-off grades

		Average Grade				Contained Metal		
Cut off	Tonnes	Magnetite	VinMC	Ti	Fe	VinMC	Ti	Fe
	Mt	(%)	(%)	(%)	(%)	(kt)	(kt)	(kt)
6	43	15.00	0.92	3.76	62.3	60	244	4,045
8	43	15.00	0.92	3.76	62.3	60	244	4,045
10	43	15.00	0.92	3.76	62.3	60	244	4,044
11	**42**	**15.11**	**0.92**	**3.75**	**62.3**	**59**	**239**	**3,971**
12	38	15.46	0.92	3.79	62.2	55	225	3,687
14	28	16.41	0.91	4.00	62.2	41	181	2,820
16	11	18.59	0.85	4.87	61.8	17	98	1,245
18	6	19.95	0.86	5.00	62.2	10	59	737

15 Mineral Reserve Estimates

This section is not applicable to this Report. The project has no declared Mineral Reserves as per CIM definitions.

16 Mining Methods

The mine design and mine planning for Mustavaara project is based on the Mineral resource estimate described in section 14. Design criteria for the open pit design and open pit optimization were obtained from the WSP (2013) study report. Hanging wall slope design parameters were reviewed by Wyllie & Norrish Rock Engineers Inc. (2021). The open pit optimization results are based on Measured, Indicated and Inferred mineral resources. The used end-product prices are received from Strategic.

The Company further cautions that the PEA is preliminary in nature. No detailed mining study has been completed to support mineral reserves. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

16.1 Overview

The following mining methods were evaluated against the Mustavaara Vanadium deposit to determine the most suitable mining method:

- Strip mining
- Terrace mining
- Truck and shovel operation
- Underground mining

Conventional Truck and Shovel Operation

A truck and shovel operation refers to the use of large, generally rigid ore body, off-highway haul trucks being loaded by large shovels or excavators. This combination of mining equipment is proven technology and being used in the majority of open pit mines in Finland and throughout the world. Conventional Truck and Shovel Operation is seen as the most suitable open pit mining method and was the selected method for this study.

Following advantages are seen in the selected mining methods.

The key points of truck and shovel operation are:

- The truck and shovel combination is a known and proven mining technology capable of handling most rock types in Finland.
- The haulage and loading equipment can handle both free-dig and blasted material.
- The targeted production rate of 3,250 Kt/a of ore can be easily reached with most of the available mining truck options.
- Use of contractors is very common in open pit blasting, loading and haulage in Finnish open pit mining projects. And there are several reputable contractors available. Skilled work force is also widely available is owner operated mining is selected.

Based on the open pit optimization a mine design was created to be mined using ~~a~~ conventional truck and shovel operation. The mining is designed to be started from the old open pit and eventually expanding to West / North West from the current open pit. The existing ramp will be used and continued to northbound creating the new ramp to the pit bottom of the new maximum open pit design.

The designed open pit is 1,935 meters long (SW–NE) and 435 meters wide (NW–SE). The maximum depth of the pit is 199 meters, calculated from the top of the existing ramp.

The stripping ratio of waste to ore for the Mustavaara open pit for the duration of the life of mine is 1.7 to 1.0. The yearly throughput of the processing plant is designed to be 3,250,000 tonnes of ore. This means a daily rate of 10,400 tonnes of mined ore. The mine is assumed to operate 313 days and the processing plant is assumed to operate with 90% of availability meaning ca.330 days per year.

16.2 Mustavaara Deposit Geotechnical Evaluation

16.2.1 Joint Set Analysis

According to WSP (2013) report a joint set analysis was carried out by studying 3D photographs and mapping altogether 737 joints. In this study no major faults were observed, but few fractured zones were found. Joints were divided into six different joint set groups J1-J6, J1 being the most common. J1 was interpreted as dominant and systematic set and was found from the whole mapped area. J1 dips steeply (81°) towards west (263°). All joint sets are summarized in the

Table 16-1 and map of the studied area with observations show in Figure 16-1.

Table 16-1 Summary of joint sets at Mustavaara from WSP (2013) report.

Joint set	Dip	Dip direction	Trace color	No. of joints
J1 (major)	81° (57°-90°)	263° (270°/090°)	Red	351
J2 (minor)	62° (54°-70°)	158° (140°-170°)	Green	38
J3 (minor)	72° (53°-88°)	197° (181°-212°)	Blue	65
J4 (minor)	42° (35°-52°)	128° (110°-145°)	Yellow	29
J5 (minor)	35° (25°-50°)	331° (350°-310°)	Magenta	29
J6 (minor)	67° (57°-77°)	079° (067°-094°)	Orange	54
Random	-	-	Grey	171

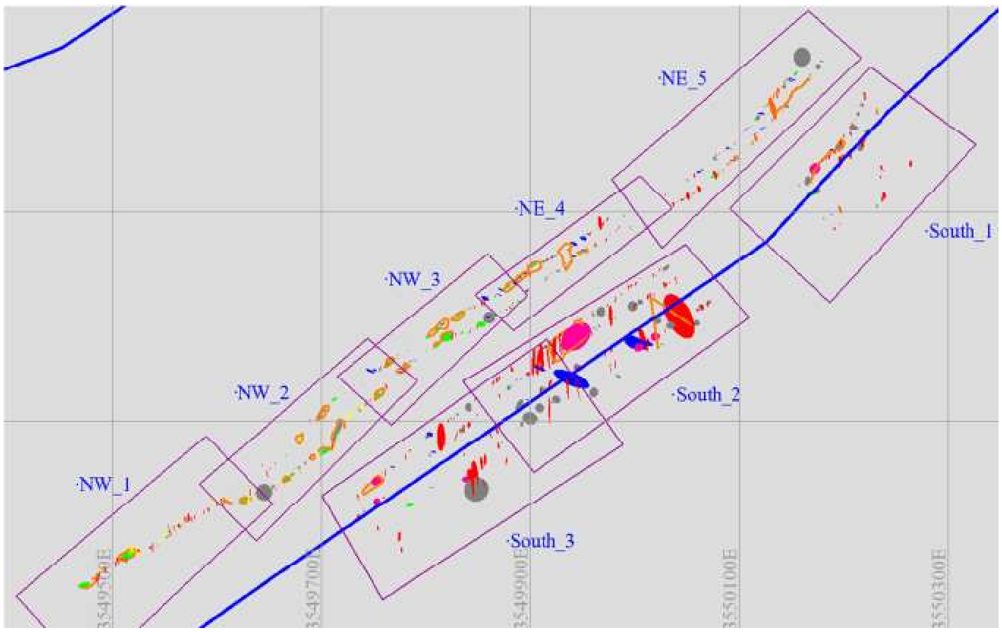

Figure 16-1 Photogrammetry mapped joints and digitized features (orange polylines) from WSP (2013) report.

J1 joint set is the only major joint set studied, all other joint set J2-J6 are not forming wide and coherent sets. Distribution of different joint sets and the variation of their dip and dip direction can be observed from Figure 16-2.

More detailed joint set analysis of different sectors can be found in the WSP report (2013).

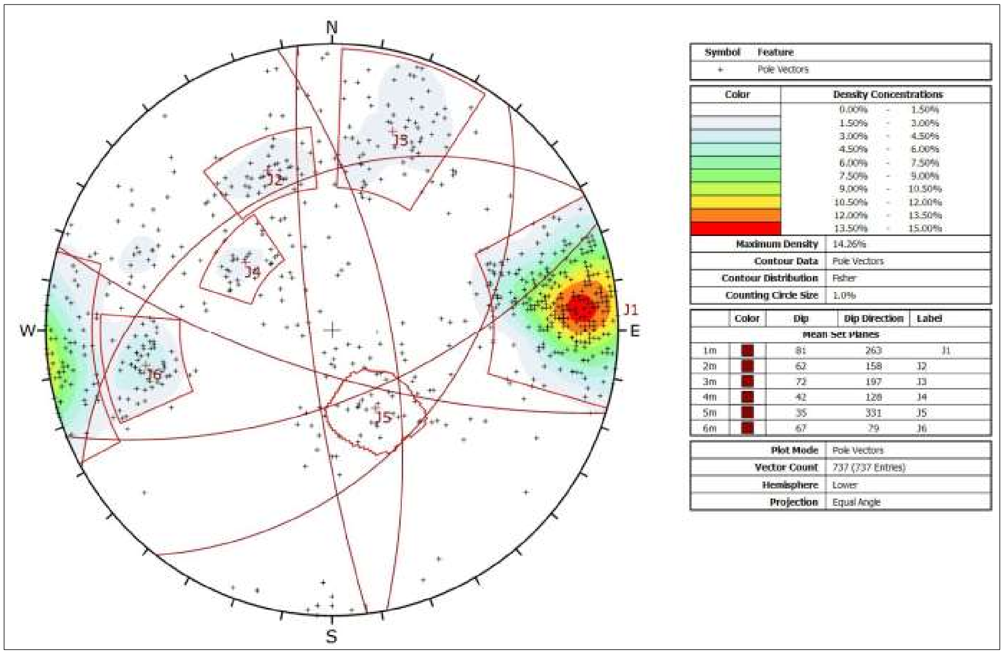

Figure 16-2 Studied joint set windows and mean planes of interpreted joint sets from WSP (2013) report.

16.2.2 Rock Quality Mapping From Drill Cores

Open pit rock quality mapping was conducted in 2013 by WSP.

According to WSP the rock quality was determined basically good to very good based on Q'-logging of cores including logging based on photographs. Core logging is unreliable inside ore, because they were sawn in half. Q'-classes and logging meters of all logged cores are illustrated in Figure 16-3.

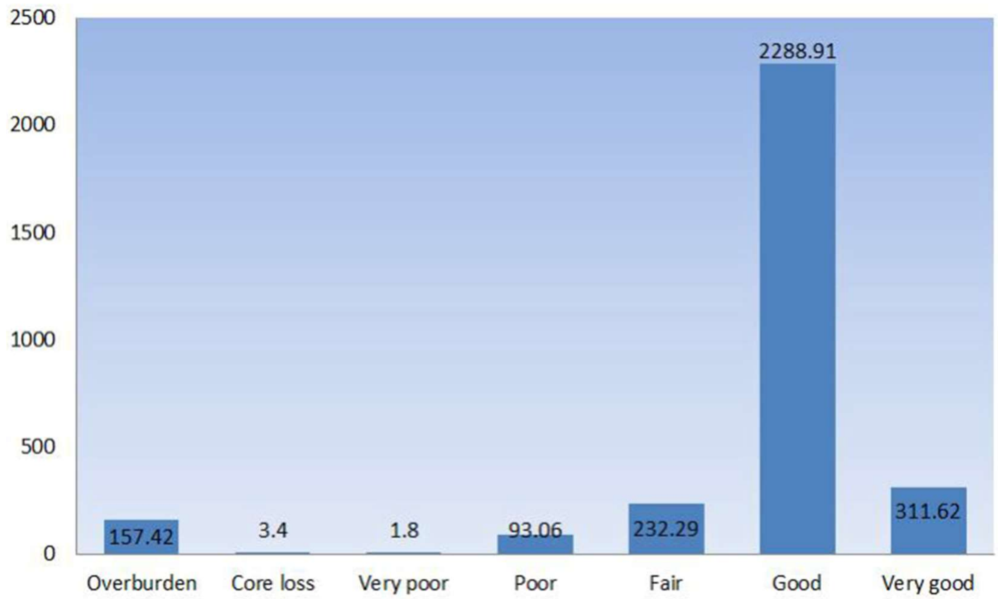

Figure 16-3 Q'-class based on all of the logged drill holes, including the photo logged holes.

WSP notes that occasional fractured zones were observed from the three logged drill cores. The drill cores had tight iron oxide or light mineral filling. No fracture or shear zones were observed which could have been related to either of the ore contacts. Joints were mostly very rough and joint alteration was low. In most cases joints were not slippery despite the filling. No considerable correlation between rock quality and lithology was observed, although fine grained gabbro had a less frequent jointing than coarse grained rock. Detailed logging notes of the three logged drill cores are presented in WSP (2013) report.

Observations are similar when taking into account the photo logged cores. Occasional narrow, fractured zones were logged almost in every core and often they were located at the surface right after overburden. Q'-class of the northeast corner indicates more fractured zones.

WSP also made few observations from the older exploration campaigns logging notes. They noted that there is an exceptionally wide fracture zone in hanging wall contact of the ore body. In addition, the north-east corner of the pit has >20 m thick overburden associated with >15 m of weathered bedrock.

16.2.3 Rock Mass Classification

WSP carried out a geotechnical mapping from the pit walls during a site visit in 2013. Due to very strong weathering of the pit walls rock mass classification was very roughly estimated using the parameters from Rock Mass Rating (after Bieniawski 1989) and Q'-classification (after Barton et al 1974). The evaluation was also made based on 3D photographs.

A rough estimation of Q'-classification based on site visit observations, 3D photographs and logged drill cores is presented in the Table 16-2. Jn parameter was determined based on the stereographic projections and their joint sets and

Jr was evaluated based on logged drill cores and from 3D photographs. Jr is determined to be a little worse in NW 2 and NW 3 because the amount of large slickenside joint surfaces is more abundant there. Ja was estimated an average of 2 based on logged drill cores. In general the rock mass quality is determined as good. NW 2 and NW 3 are classified fair, but they are very close to good.

Table 16-2 Rough estimates of Q'-parameters and values from the open pit mappings. Rock quality of the current Mustavaara open pit is in generally good. WSP report (2013).

Sector	RQD	Jn	Jr	Ja	Q	Q-class
NW 1	85	4	3	2	31.88	Good
NW 2	85	9	2	2	9.44	Fair
NW 3	85	9	2	2	9.44	Fair
NE 4	85	6	3	2	21.25	Good
NE 5	85	9	3	2	14.17	Good
South 1	85	6	3	2	21.25	Good
South 2	85	6	3	2	21.25	Good
South 3	85	9	3	2	14.17	Good

The most prominent joint set J1 was mapped in more detail by WSP (2013) and some parameters and properties considering the Rock Mass Rating (RMR) system and the Q-classification are listed below:

J1 has a high (10-20 m) to very high (>20 m) continuity length (RMR), because it clearly intersects up to two benches, especially seen in the south wall.

Spacing of the J1 joints varies between few centimetres to several meters (RMR). Spacing is observed to be more frequent at some places.

Separation of the joints is <5mm (RMR). The open pit walls are very strongly weathered which naturally leads to wider separation at the surface, but it should not be confused to real aperture.

Joint roughness rating varies from slickensided to rough, but mainly J1 joints are seen stepped and quite rough → Jr = 3 (Q').

Joint filling and alteration could not be examined properly due to weathered and hazardous rock faces at the site. Based on the logging results (MV-61-2011, MV-63-2011 and MV-67-2011) common joint fillings are iron oxides, clay minerals (kaolin) and carbonates with thickness <1mm → Ja value varies between 1 to 4, but is commonly 2-3 (Q').

Observed parameters of the other joints (WSP 2013):

Joint set J2 has slickensided (Jr = 1.5-2) and quite large open joint planes, which are observed to be occurring mostly in north side of the pit. However, slope faces are very strongly weathered and open joint surfaces were exposed to that too. It may have smoothen joint roughness for example from original Jr = 3 to Jr = 1.5-2. Trend of the slope face in north wall and J2 is nearly the same. Surface areas of the J2 planes are observed to be fairly wide at some places, for example at NW 1 there is couple of J2 planes that have an area of 10m x 10m (Figure 16). Slope face direction ~150°± 25° in the North has caused most of the sliding along the plane on joint planes J2 and J4 + random. J5 forms similar sliding surfaces in the footwall side (mostly South 2 and South 3) as J2 in the hanging wall.

16.2.4 Major Structures and Fracture Zones

No large fracture zones were observed by WSP in 2013. Observations were made from aerial photographs, laser scanning data and 3D photographs.

WSP found several continuous fracture zones in the middle of the open pit. Two fracture zones were observed from both pit walls and are oriented approximately along the main joint set J1 and are not considered to have critical direction for mine design.

These fracture zones intersect with each other to create two large wedge-type formations observed from the south wall, which can be clearly distinguished as critical zones because of their highly continuous and altered joints.

North-east area below wetland is potential problem area for stability because of possibly worse rock quality but especially because of challenging ground and surface water conditions.

16.2.5 Planar, Wedge and Toppling Failure Analyses

Planar, wedge and toppling failure analyses were made from the geotechnical data by WSP in 2013. More detailed descriptions about the methods can be found from the WSP (2013) report.

Planar failure risks are occurring throughout the whole open pit area. Wedge failures are also common but the risk percentages are theoretically evaluated lower. The highest risks of having planar failures are in the north-western slope face at the directions 150° and 125° through the joint sets J2 and J4.

WSP 2013 report points out that potential wedge failures are also occurring throughout the whole open pit, but mainly at low risks (theoretical risk) in the north-western and northern slope faces. As mentioned in chapter 16.2.3, J1 has high (10-20m) - very high (>20m) continuity length and may therefore form quite hazardous wedges with some random joints. For example J1 with J5 (and nearby joints of J5) are quite likely to form wedges in the footwall. Wedge sliding may occur if the intersection point of two planes is placed on a critical area. However, this is theoretical possibility and wedge cannot be determined absolutely only from stereonet. Other factors that affects to wedge forming process are for example water pressure, shear strength of joints and continuity/geometry on joints that form the wedge.

Systematic toppling failure was not observed in the Mustavaara open pit.

16.2.6 Rock Mass Strength

In 2012 9 samples were tested according to ISRM suggested methods (2007) by Aalto University Rock engineering laboratory with MTS 815 rock mechanics test system. Tests were performed on the 19th and 20th October 2012 in laboratory dry air. Performed tests included uniaxial compressive strength and indirect tensile strength tests (Brazilian test).

According to laboratory tests, the intact rock strength is very high. Results indicate that the rock samples can be divided to two groups by grain size. Fine grained samples 1, 6 and 9 have very high strength values, average UCS being 403 MPa and average tensile strength being 20 MPa. Coarse grained samples have average UCS of 192 MPa and average tensile strength of 13.7 MPa.

Average strength of the hanging wall side samples seems to be higher than foot wall side samples but due to the number of samples, no definitive conclusions should be made. None of the measured parameters seem to correlate with depth.

Based on the lab tests, drill core loggings and open pit face mapping WSP calculated rock mass strength parameters in 2013. These parameters were calculated for minimum and maximum values of estimated GSI value range and are presented in Table 16-3. Uniaxial compressive strength of the rock mass varies from 6.7 to 23.8 MPa. Tensile strength varies from 0.3 to 1.3 MPa and deformation modulus from 10.5 to 23.2 GPa.

Table 16-3 Hoek-Brown strength criterion parameters and calculated rock mass parameters.

Parameter	GSI 60	GSI 75
Hoek-Brown Criterion		
m_b	0.8040	2.3470
S	0.0013	0.0155
A	0.5030	0.5010
Rock mass parameters		
σ_t	-0.3 MPa	-1.3 MPa
σ_c	6.7 MPa	23.8 MPa
σ_{cm}	23.2 MPa	42.7 MPa
E_m	10.5 GPa	23.2 GPa

16.2.7 Rock Slope Stability Calculations

According to WSP (2013) report, the modelling results indicate that with determined rock strength parameters overall slope stability should not be an issue and possible failures are most likely to be structurally controlled local failures.

The worst case scenario would be steep, fully saturated slope. Results from the simulations of fully saturated slopes with 55° and 60° overall angles are presented in Figure 16-4 and Figure 16-5. Critical SRF number for these two cases varies from 2.32 for 55° slope to 2.15 for 60° slope.

Excavating the pit and pumping the ground water will naturally lower the ground water pressure around the slope, and thus fully saturated conditions usually occur only due to heavy downfall or extreme ground water conditions. In Figure 16-6 is presented simulation results for 55° angle slope, where ground water is located at surface 2 times the height of the slope behind the toe. Critical

SRF number is 2.69, which clearly shows the significance of proper dewatering of the slope.

The SRF value, or the safety factor, is well above 3 for gentler overall slope angles, especially in favourable ground water conditions. Estimated rock mass strength is very good, and as more information are available, it is recommended to both re-evaluate the rock mass strength parameters and perform revised calculations for overall slope stability.

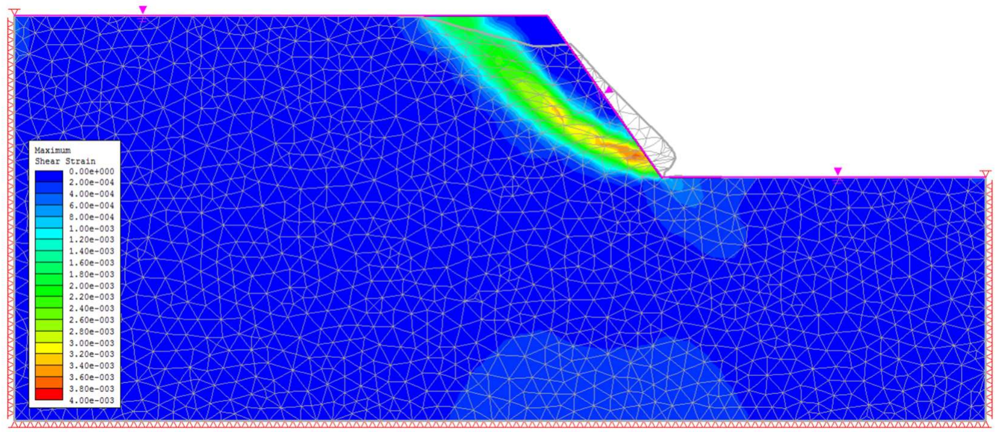

Figure 16-4 Slope angle 55°, fully saturated. Critical SRF 2.32. Contours for maximum shear strain and deformed mesh give indication of location of the failure circle.

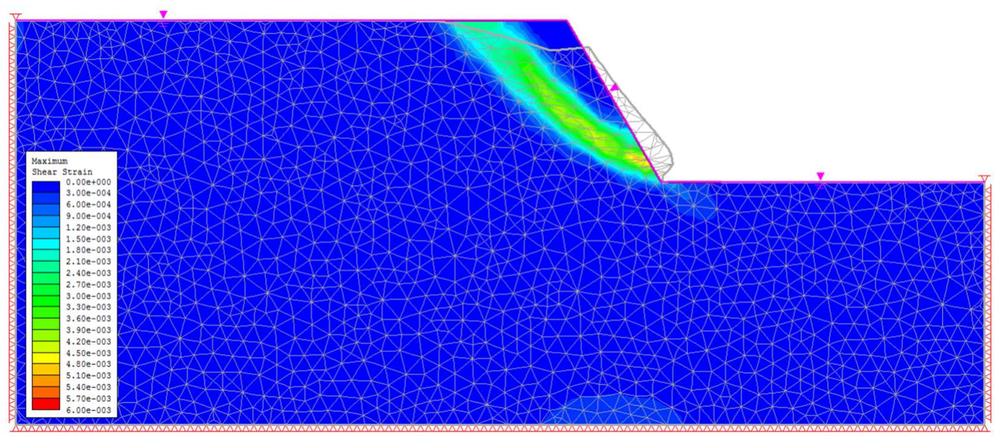

Figure 16-5 Slope angle 60°, fully saturated. Critical SRF 2.15. Contours for maximum shear strain and deformed mesh give indication of location of the failure circle.

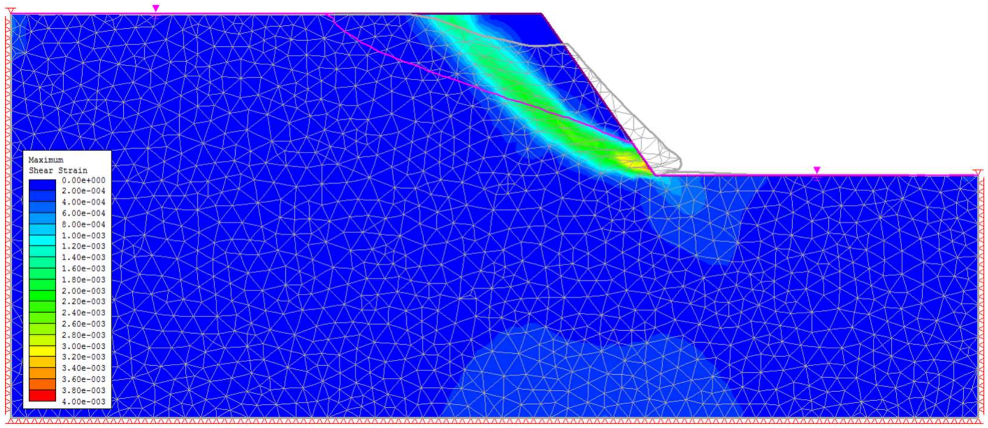

Figure 16-6 Slope angle 55°, ground water level at surface at 2x the height of the slope from toe. Critical SRF 2.69. Contours for maximum shear strain and deformed mesh give indication of location of the failure circle.

16.3 Mustavaara Deposit Hydrogeological Evaluation

According to WSP Open pit Stability Report (2013) there are two main concerns about surface water. The first and most concerning matter is wet land area in north-east which the designed preliminary ultimate pit will cross (Figure 16-7). The second is Sirniönlampi (pond) which is located relatively close to the final ultimate pit boundary (Figure 16-7). Both cases need to be studied in more detail by hydrogeological study.

Unfortunately, old drill holes closest to the Sirniölampi east side have not been geotechnically logged but based on WSP Report (2013), tectonic weakness zones are found close to the ore contacts. Closest geotechnically logged hole is MV-59-2011 which was photologged by WSP. Overburden of 5 m and weathering surface down to 12 m is observed from MV-59-2011. There is a longer fracture zone at 70 m depth along the drillhole, but this is in ore – hanging wall contact and direction is likely along the orebody. Major joint set J1 direction observed in existing pit is not particularly worse concerning the Sirniönlampi. Both joints sets J4 and J2 have direction that intersects designed pit and Sirniönlampi. Especially J2 is continuous and it has been observed in south corner of existing pit.

Old drill holes drilled from and near the Sirniölampi were examined by WSP to determine if they could potentially create hydrological channels to the planned open pit. Old drillhole R-013 drilled from Sirniönlampi (or near it) does not penetrate preliminary ultimate pit design, however the direction of hole is towards it.

In wetland areas in north-east overburden is deep > 20m and this is followed by weathered rock zone > 15 m. The rock has also more fractured zones in that area. These will have big effect for water inflow into pit. Major joint set J1 direction is unfavourable concerning surface water inflow into pit.

AFRY agrees with WSP that in the next study phase, when scheduling pit phases it should be considered to delay pit extension to wet land area. It will be important to plan proper dewatering of wetland area around pit and also inside the pit to lower water pressure. In the worst case the pit have to be designed shorter to avoid wetland area.

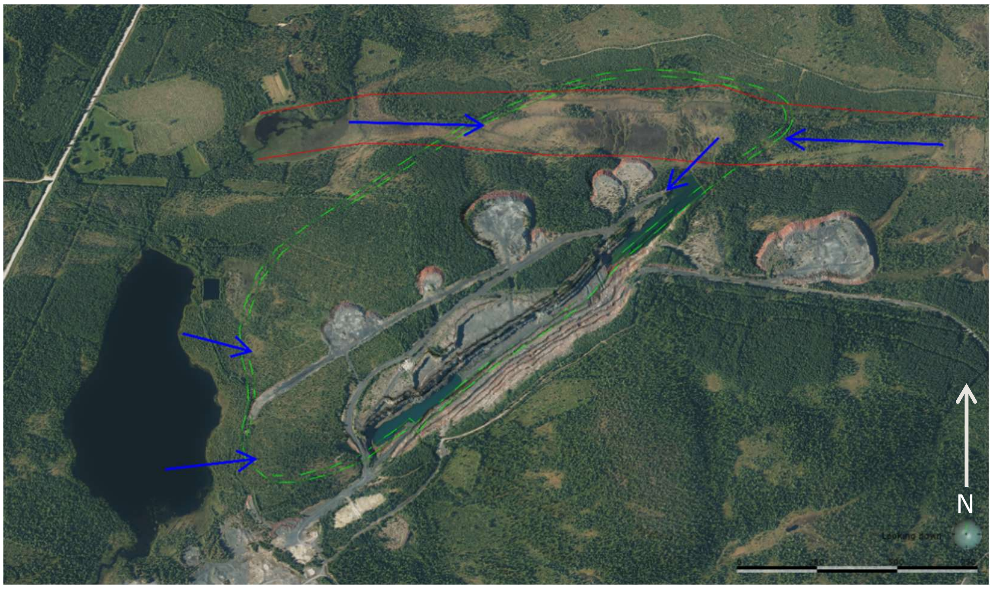

Figure 16-7 Aerial photo of Mustavaara area. Sirniölampi at the west and wetlands at the north. Maximum pit outline marked with dashed green line.

16.4 Design parameter recommendations from previous studies

The design parameter recommendations have been gathered from two previous studies. WSP's Open pit stability study (2013) and Wyllie & Norrish's Hanging wall slope design guidance study (2021) are referenced below.

The design parameters from WSP open pit stability study (2013) are summarized and presented in Table 16-4 and Table 16-5.

The design parameters from Wyllie & Norrish Hanging wall slope design guidance study are summarized and presented in Table 16-6.

Table 16-4 WSP 2013 Open pit stability study, recommended design parameters (1/2)

WSP 2013 Open Pit Stability Study		
Recommended design parameters (1/2)		
Bench height	12.5	m
Foot wall (single benching)	12.5	m
Hanging wall (double benching 2x12.5m)	25	m
Safety level width	min. 8	m
For bench height 12.5m	8	m
For bench height 25m	12.5	m
Scaling and cleaning safety levels		
Scaling of all loose rock pieces from rock faces		
Safety levels to be cleaned from fallen rocks		
Bench angle		
Foot wall	75°-85°	
Hanging wall (planar failure risk, joint sets J2/J4)	65°-70°	
This will decrease overall slope angle from preliminary 50°		
Bench angle as steep as possible		
If bench angle >75° rocks will stop at next level toe		
Mineralized material contact		
Pit benches will be mined into waste rock clearly away from mineralized material contact in footwall because some weakness zones are found at the contact. Near parallel dipping joint set J5 in the contact might also cause planar failures. There are also fracture zones at the hanging wall contact, which should be taken into account in mining.		
Ramp stability		
Ramp width (preliminary 22m)	25 - 28	M
Hanging wall side (temporary ramp)		
Footwall side (permanent ramp)		
Decrease batter angle to 65°		
Reduce risk for planar failure caused by joint set J2		
Follow and map long continuous joints and structures, which could form unstable wedges with joint set J1. These joints include especially joint directions 350° (±45°) dipping 40°-80°.		
Pre-reinforcement, cable bolts for areas with local instability		
Use catch ditches in the toe of benches		

Table 16-5 WSP 2013 Open pit stability study, recommended design parameters (2/2)

WSP 2013 Open Pit Stability Study		
Recommended design parameters (2/2)		
Rock supporting		
Rock bolts for unstable wedges	50°-70°	dip
Double twisted steel net for local rock faces		
Cement grouted double strand cable bolts with face plates		
Pit walls above haulage ramp possibly should be systemically cable bolted to ensure safety and to maximize final overall slope angles, at least at hanging wall side.		
Blast quality		
Controlled blasting to improve stability of final faces		
Pre-shearing		
Cushion blasting		
Overall slope angle		
Hanging wall		
Overall slope angle	<45°	
with batter angle	65°	
Overall slope angle (recommended to start with)	<48°	
with batter angle	70°	
Footwall		
Overall slope angle	30°-45°	
with batter angle	80°	
Pit ends		
Overall slope angle	41°-44°	
with batter angle	65°-70°	
Monitoring stability		
Simple pit slope monitoring (beginning phase)		
Laser scanning		
3D photogrammetry		
Documentation		

Wyllie & Norrish state that *"accepting WSP's conclusion that overall slope stability, based on rock mass strength, is well within design tolerance, it is recommended that the bench geometry and the inter ramp slope angles for preliminary hanging wall design should be within the option range"* presented in Table 16-6.

Table 16-6 Wyllie & Norrish 2021 Preliminary hangingwall slope design guidance study, recommended design parameters (1/1)

Wyllie & Norrish 2021 Preliminary Hangingwall Slope Design Guidance
Recommended design parameters (1/1)

Parameter	**Abbrev.**	Option 1	Option 2
Bench height (m)	BH	25	25
Bench face angle (deg)	BFA	70	72
Catch bench width (m)[1]	CBW	9.7	9.7
Inter ramp angle (deg)	IRA	53.0	54.5

[1] Minimum CBW defined by Ryan & Pryor (2000) at 9.5m for 25m bench height.

16.5 Open Pit Optimization

Open pit optimisation was used to evaluate the ultimate open pit size for the PEA mine plan. Two options were evaluated with different end products. The first option produces FeV80 and Pig Iron and the second studied option V_2O_5 and Pig Iron. The resulted open pit geometry was used in the engineering design of the open pit shape. After the open pit optimization process an evaluation was also made to utilize an owner operated fleet in order to improve the project economics.

The open pit optimisation was achieved using the Deswik GO software (Version 2020.1). Deswik GO calculates the discounted cumulative cash flow indicating the net present value of the open pit (NPV) by using the Direct Block Scheduling algorithm. Direct Block Scheduling testing (Direct Multi-Period Scheduling methodology) which schedules mine production considering the correct discount factor of each mining block, resulting in the final pit. Each block is analysed individually in order to define the best target period.

<u>The input factors used in the optimisation process incudes:</u>

- Overall slope angles
- Geological block model
- Mining costs including variation by mining bench height
- Mineral processing costs
- Mineral processing and mining recoveries
- Mining dilution
- Product revenues

- Discount rate

The optimisation method is applied to the block model, and Deswik GO progressively constructs a list of related blocks that should or should not be mined. The final block list defines an ultimate pit outline that has the highest possible NPV value, while considering the required economical parameters, pit slope angles and other physical constraints.

Deswik GO produces pit shells that are used for mine design. In practical designs ramps, berms and batters will change the shape of the optimized Pit shell. This will make changes to the reported amounts of processed material depending on how accurately the original pit shell was used in the practical pit design. After the optimisation procedure the open pit is designed according to the geotechnical parameters.

16.5.1 Optimisation Parameters

The optimisation parameters were adjusted according to the data provided by Client or estimated by AFRY based on the data made available. The optimisation parameters included the Mineral Resource estimation block model, all necessary operational costs, time costs and processing costs of the final end product. All parameter categories are explained in the following chapters.

The summary of the optimisation parameters is presented in the

Table 16-7.

Table 16-7 Open Pit Optimisation Parameters

Open pit optimization parameters	
Currency conversions	
Exchange rate EUR-USD	1.1
Time costs	
Costs in REAL/NOMINAL	REAL
Fixed annual write-offs, overheads (concentrate)	-
Annual production cost increment	-
Annual discount Rate	7%
Blasting unit costs	
Ore	1.14 EUR/t
Waste	0.82 EUR/t
Loading and hauling unit costs	
Ore loading and haulage to surface	1.54 EUR/t
Waste rock loading and haulage to surface	1.43 EUR/t
Mining level supplement /20m	0.17 EUR/t
Material hauling inside mining concession	0.24 EUR/t/km
Mineral processing costs	
Concentrator	3.8 EUR/t
Smelting FeV80	23.1 EUR/t
Smelting Pig Iron (additional to FeV80)	0.76 EUR/t
Admin & Logistics	
Admin	1.26 EUR/t
Logistics	2.3 EUR/t
Selling price	
FeV80	25.5 USD/kg
Pig Iron	340 USD/t
Mining parameters	
Overall slope angle	See Table 16-11
Mining dilution	8.0%
Mining recovery	95%
Cut-Off (Magnetite)	Determined block by block basis
Processing parameters	
Input ore processing capacity (Ore t)	3,250 Kt/a
Producing magnetite Concentrate (Fe_3O_4)	98.00%
Metal Plant recovery for FeV80	79.00%
Metal Plant recovery for V_2O_5	80.21%
Iron Production Fe-recovery:	98.00%

16.5.2 Geological Block Model

The used block model is described in chapter 14.3 was used along with modifying factors to develop the component of the mineral resource considered for processing in this PEA Mine Plan.

16.5.3 Time Costs

To estimate the time value of money annual discount rate of 7% was used in the optimisation procedure. The selected discount rate reflects well to the current economic situation and insignificant investment risk status in Finland.

Generally, the annual discount rates vary in Finnish mining feasibility studies in the range of 5 - 10% if discounting is considered possible to be used in the estimations.

No annual production increments were used for production costs. The optimisations were performed in Real costs.

16.5.4 Processing Plant Capacity

The annual throughput feed to the processing plant was set to be 3,250,000 t/a.

16.5.5 Processing Recovery

For the open pit optimisation purposes the mineral processing recovery of the FeV80 and Pig Iron were calculated using the following recovery rates:

- Producing magnetite Concentrate (Fe_3O_4): 98% recovery
- Metal Plant recovery for FeV80: 79%
- Metal Plant recovery for V_2O_5: 80.2%
- Iron Production Fe-recovery: 98%

16.5.6 Mining and Transportation Costs

All mining operation cost estimates used in the open pit optimisation are based on contractor given estimates. The costs include blasting and loading of the ore and waste rock, transportation to the ore pad or waste rock storage. The contractor costs include all personnel, fuel and maintenance costs.

All of the contractor costs estimates correspond to AFRY´s internal database of similar operative costs (using contractor) in Finland.

The operative costs for the optimisation are calculated inside the geological block model in Surpac. The costs increase by depth according to the mining level supplement Table 16-8. Once the block model has been imported to the Deswik Go the operative costs are calculated to the cash flow during the optimisation procedure. The operative costs are presented above in

Table 16-7.

Table 16-8 Mining costs by level in Mustavaara

Depth	Mining level	Ore mining opex EUR/t	Waste mining opex EUR/t
25	285	1.97	2.01
50	260	2.05	2.09
75	235	2.14	2.18
100	210	2.22	2.26
125	185	2.31	2.35
150	160	2.39	2.43
175	135	2.48	2.52
200	110	2.56	2.60
225	85	2.65	2.69
250	60	2.73	2.77
275	35	2.82	2.86
300	10	2.90	2.94

After the optimisation study was finished a trade-off study was made concerning the use of contractor vs use of owner operated equipment fleet and manpower for mining. As the mining costs have great importance to the estimated total operating costs, it is advised to pay attention to the competitive bidding of the mining contractor or to the selecting and optimizing the mining fleet in the future study phases. Small changes in the mining costs have no great impact to the open pit size selection but there is an upside potential for making the mining operations more profitable.

16.5.7 Processing Costs

The processing costs were calculated by AFRY based on energy and material consumption of the processes. The calculations done in 2012 study were used as basis from where the costs and consumption estimates were updated.

The total processing cost used in optimization is 27.66 €/ tonne of processed ore €/t for the FeV80 option and 26.06 €/t for V_2O_5 option. The used costs are presented in

Table 16-9.

The finalised processing cost are slightly different in the final financial model. Small variations in the processing cost were tested in the open pit optimisation and will not affect the open pit size.

Table 16-9 Processing OPEX

Area	€ / tonnes of ore
Concentrating	3.8
Smelting	
FeV80	23.1
V_2O_5	21.5
Additional cost to Produce Pig Iron	0.76
Logistics	2.3
Administrative	1.26
TOTAL FeV80	27.66
TOTAL V_2O_5	26.06

16.5.8 Open Pit Constraints and Mining Limits

The final optimisation was run with a physical constraint to keep a 100 meters distance to Sirniölampi (pond) shoreline. The constraining was done by writing an extremely high mining opex to geological block model this created a hard boundary for the optimization.

Maximum ore mining was set to 3,250,000 tonnes of ore /a. Waste rock mining was not limited to any value. This was done in order to avoid ore mining to be limited by waste rock mining.

16.5.9 Mining Recovery and Dilution

The mining recovery refers to the ore that is lost (hauled to the waste rock storage facility) during the selective ore mining. The average mining recovery factor of 95% was applied to the optimisation. Usually a range from 3 to 10% is considered to be a reasonable estimation for the ore loss in an open pit operation.

The mining dilution occurs during the blasting and excavation processes where ore and waste material are mixed. The additional waste rock materials are not desirable, as low-grade ore or waste material adversely affect the output of the processing system. Mining dilution increases the quantity of ROM ore to be mined and simultaneously reduces the mill feed grade. For the optimization an 8% dilution was used.

16.5.10 Product Prices

FeV80 price of 25.5 USD/kg was used in optimization and Pig iron price of 340 USD/kg. In the V_2O_5 scenario a 11.4 USD/kg price was used. USD prices were converted to Euros by using an exchange rate of 1.1 USD/Euro. Higher product price forecasts were used in the cash flow calculations. The cashflow calculation also uses an exchange rate of 1.18 USD/Euro.

16.5.11 FeV 80 Equivalent

For the optimisation purpose a FeV80 equivalent was calculated for the end product.

- *FeV 80 eq (kg) = FeV 80 (kg) + (Pig Iron Factor * Pig Iron (t) *1000)*
- *Pig Iron Factor = (Pig Iron price * Pig Iron [tonnes]/Fev80 price/1000)/Pig Iron (t)*

16.6 Optimisation Results

In the first phase the optimisation process produces a series of nested pit shells. The cash flow for each shell is calculated using the input selling prices and costs and thereafter provides an indication of the economic changes for the various pit shells. The first optimisation phase produces the pit shell with highest NPV. The selected pit shell is then divided into mining phases in order to create most profitable realistic mining scenario available.

16.6.1 FeV80 Scenario

The Mustavaara open pit optimisation results are presented in Figure 16-8. Phase optimised open pit can support approximately 20 years of operation with estimated discounted cash flow of 225 M€ (Pre-tax). The summary of open pit optimisation is presented in Table 16-10. The Mustavaara open pit optimisation FeV80 scenario indicates 65 Mt of processed material at total strip ratio of 1.9. The optimised Mustavaara open pit shell is presented in the Figure 16-9.

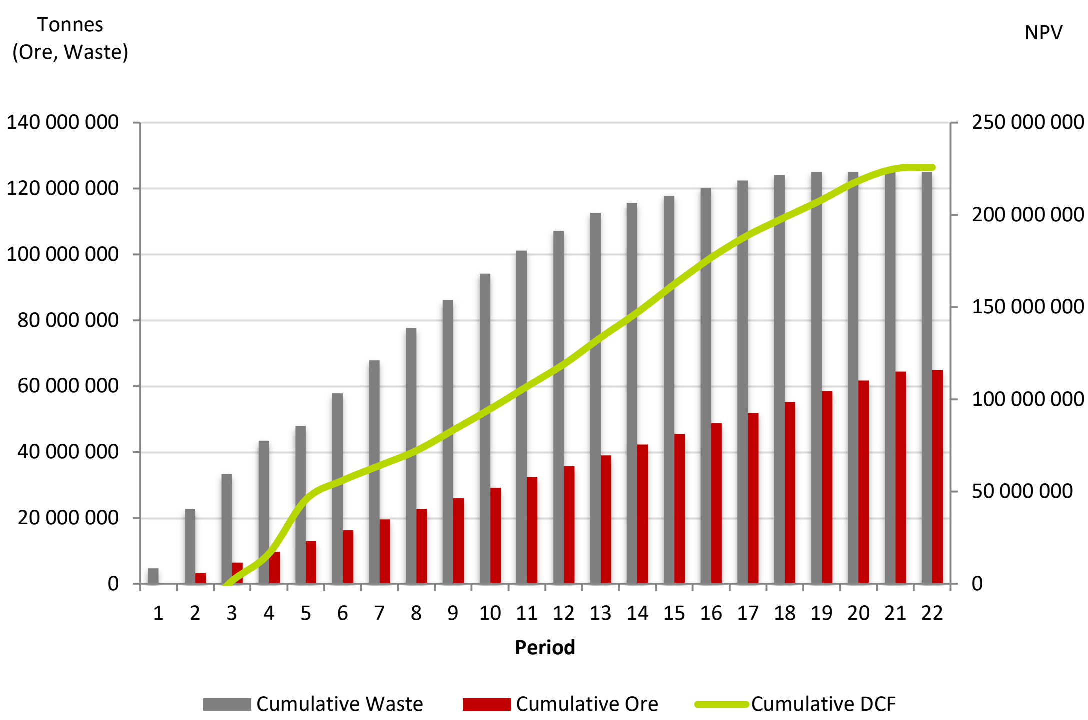

Figure 16-8 Mustavaara open pit optimisation results

Table 16-10 Mustavaara Open Pit Optimisation results

Tonnes of Processed Material	Tonnes of Waste	Strip ratio	Total tonnes	LOM	NPV
65 000 000	125 110 000	1.93	190 110 000	22	225 800 000

Dimensions for the optimised pit shell size are 1950 m (NE–SW), 485 m (NW–SE) and 225 m deep. The geometry of the optimised pit shell is suitable for 3,250 kt/a ore production.

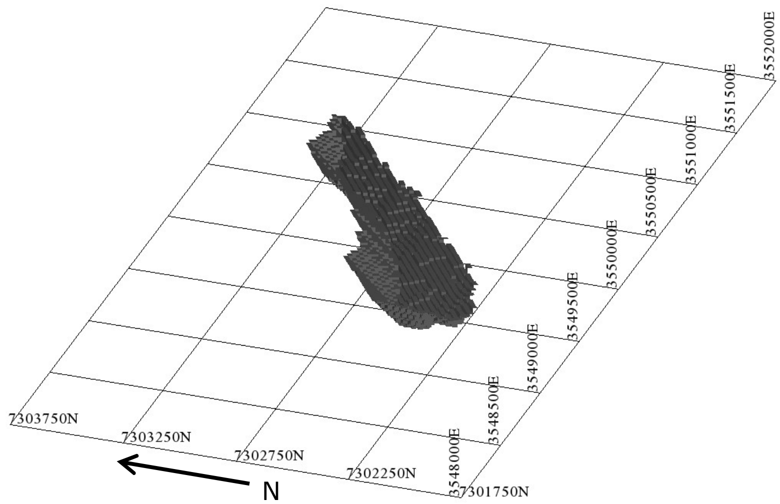

Figure 16-9 Perspective view of the Selected Optimal Pit (Looking towards East)

16.6.2 V_2O_5 Scenario

The Mustavaara open pit optimization was also run with V_2O_5 end product. Using the parameters presented in Paragraph 16.5.1 the optimization procedure did not find the operation to be feasible. Deswik GO optimization produced only a scenario with NPV of 21 M€ (Pre-tax). The total processed material was 5.4 M tonnes.

The V_2O_5 Scenario was abandoned because of the unprofitable open pit optimization.

16.7 Open Pit Design

This section details the assumptions and methodology used in the open pit design process for Strategic Resources Ltd Mustavaara deposit. The used slope parameters and abbreviations are clarified in Figure 16-10.

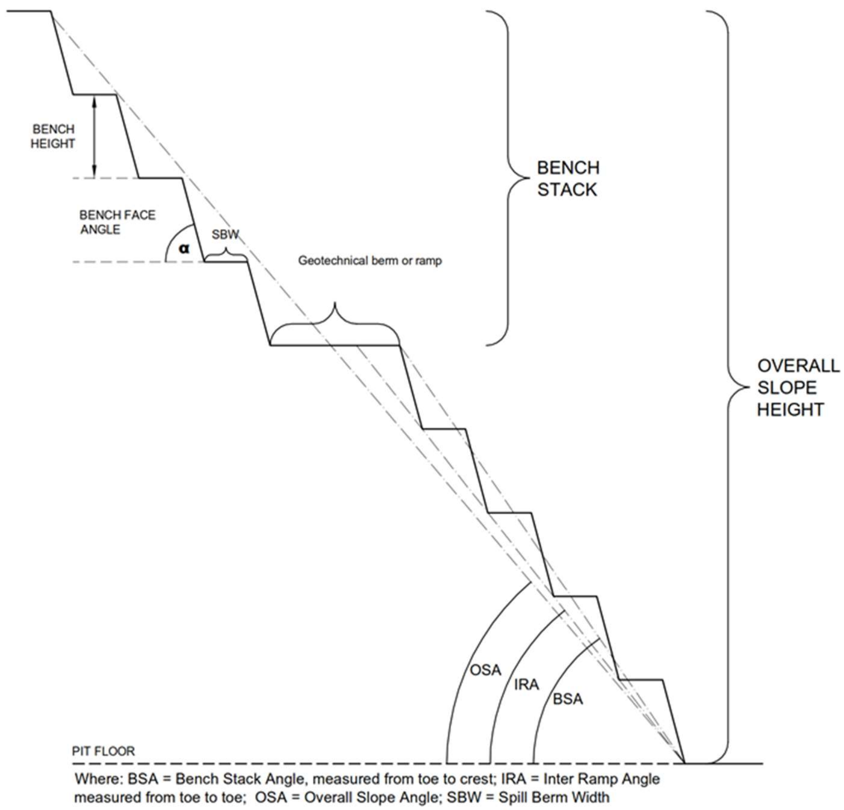

Figure 16-10 Open pit slope sections and naming acording to AFRY.

16.7.1 Open Pit Design Parameters

Table 16-11 lists the used design parameters that were used when create the final pit design. In detailed pit design small variation in design parameters is acceptable in order to create a functional open pit design.

The pit design parameters apart from the hanging wall slope design were obtained from the WSP 2013 Open Pit Stability Study. The ramp width has been selected to be 25 meters which allows for drain ditches and safety berms to be constructed and a safe two-way traffic using the Caterpillar 777 haul truck. The hanging wall slope design recommendations were provided by Wyllie & Norrish Rock Engineers Inc.

Table 16-11 Open Pit design criteria

Bench Height	25			
Ramp gradient	1:10			
Ramp width	25			
Sectors				
from	to	Batter Angle	Berm Width	OSA
270	5	70	9.7	50
5	45	70	12.5	45
45	170	80	12.5	44
170	200	80	16	48
200	270	70	9.7	44

AFRY recommends that the open pit design parameters would be checked with more detail to confirm optimal split drill angles, bench widths and bench heights.

16.7.2 Open Pit Geometry

The focus of mine design is to demonstrate that the Mustavaara open pit has reasonable prospects of eventual economic extraction. The final pit was selected in an effort to improve project NPV and ROI. The maximum NPV did not give the result the best ROI. Thus almost 10 years shorter LOM plan and pit shell was selected.

Secondary target of the open pit design was to minimise the mined waste rock amounts and to decrease the mining footprint of the Mustavaara open pit.

The open pit production is planned to be started by expanding the old open pit to NW. Figure 16-11 shows the maximum pit geometry and overall slope angles. The pit location related to Sirniölampi and surrounding environment is presented in Figure 16-12. The main haulage ramp (ramp) in the final open pit runs in the eastern wall and it turns back to south at the north side of the main pit. The ramp is located in the foot wall when going north and on the hanging wall at the very end of the open pit.

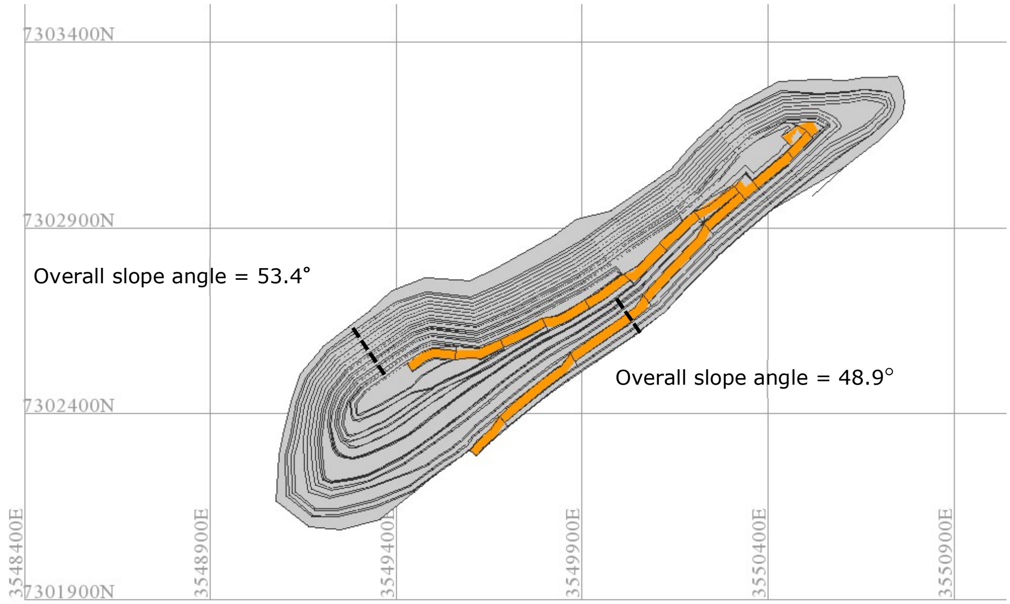

Figure 16-11 Mustavaara maximum pit

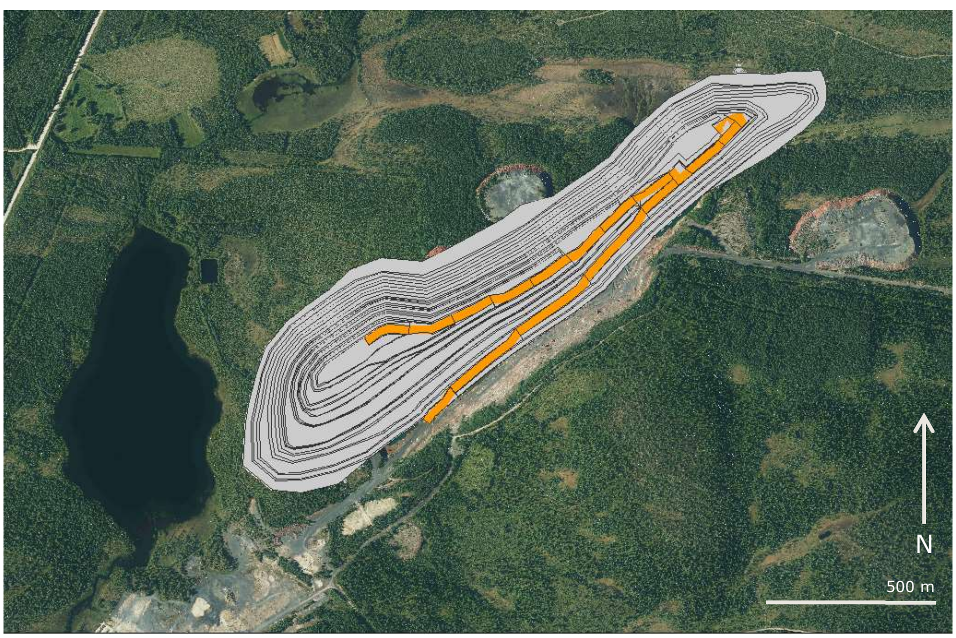

Figure 16-12 Illustration of the designed maximum pit relation to Sirniölampi

The ramp is designed to be 25 meters wide in order to enable two-way traffic and leaving enough room for safety barriers at least 3/4 of the height of the largest tire on any vehicle hauling on the road. The ramp includes also a ditch to collect waters and snow. The designed 25-meter ramp is proven to support the selected caterpillar 777 haul truck selection. The ramp is designed for all traffic to and from the open pit.

No secondary ramp is designed to the pit in order to keep the hanging wall overall slope angle as steep as possible.

16.8 Dewatering

Gravitational water flow control using ditches are used to collect inflow water from the pit into water sumps. Auxiliary submersible pumps are used to transfer the collected water into the main pumping stations from where the water is pumped into the surface. The open pit is surrounded by a ditch that prevents any excess inflow into the pit. All dewatering waters are pumped into the clarification basin.

A more detailed dewatering plan with required pump capacities is needed once a hydrological model for this project is available. The mine dewatering system needs to handle all inflow sources of water that can reach the mine.

16.9 Rock Supporting

No detailed rock supporting plan has created for Mustavaara project. According to WSP (2013) study, two types of rock support can be considered for Mustavaara pit. Unstable rock wedges are considered to be bolted and small local scale rock falls are recommended to be prevented by using double twisted steel nets.

16.10 Operating Hours

The mine operating hours are calculated for 350 production days per year, 24 hours per day and seven days per week. The rotation will be in two 12-hour shifts per day. Fifteen days of production are assumed to be lost per year due to bad weather and breakdowns. Furthermore, one hour will be lost per shift due to mealtimes and breaks.

There will be five shift crews rotating all together. Two crews will be working whilst three crews are rostered off. One crew will work one week of dayshift and then change to one week of nightshift. Hereafter the crew will have three weeks off.

16.11 Mining Equipment

16.11.1 General

Conventional open pit mining will be adopted as a standard approach. All proposed equipment to be used are diesel-powered open pit mining equipment. Certain items of the equipment, such as the excavators and drill rigs are suited for electrical power, and this option would need to be investigated at the time of any subsequent engineering work.

The loading and hauling fleet is the key capacity driver for mine production.

The main fleet will consist of the following units:

- Excavator: 230t and 12m^3 capacity (Cat 6020B)
- Haul truck: 90t capacity (Cat 777)
- Production drill rig: 172–254 mm diameter (Sandvik D55SP)
- Pre-split drill rig: 110–178 mm diameter (FlexiROC D60)

The main fleet operations will take place with the following assumed parameters:

- Operating hours per day: 22
- Operating days per year: 350
- Overall job efficiency: 90%
- Mechanical availability: 87%
- Operator efficiency and equipment utilization: 92%

No separate earthmoving fleet is assumed. Earthmoving has been evaluated using assumed mining equipment.

16.11.2 Excavators

The performance of the excavators is summarised in Table 16-12 . These are based on standard performance figures. They assume that a truck is always available for loading. These figures are used to estimate fleet numbers. The selected excavator will be a Cat 6020 backhoe.

Table 16-12 Excavator performance summary

Excavator Cat 6020B		
Measure		
Bucket capacity	12	m^3
Theoretical bucket payload	22.7	t
Fill factor	80 %	
Actual bucket payload	18.1	t
Overall job efficiency	90 %	
Mechanical availability	87 %	
Operator efficiency and equipment utilization:	92 %	
Number of passes	5	
Minutes per truck load	2.5	min
Maximum trucks per hour	17	
Tonnes per hour	1 568	t/h
Tonnes per day	34 505	t/d
Maximum tonnes per year	12 076 876	t/a

16.11.3 Haul Trucks

The mining haul trucks will be approximately 90-t rigid body trucks for moving ore and waste. The Cat 777 was chosen for the operation.

Table 16-13 below summarises the average hauling performance from open pit to crusher and waste rock storage facility. The average distance of haulage one way is calculated for 4,04km. Additionally it is assumed that loaded and empty hauling speeds are averaged to 24 km/h. Specific haulage profiles have not been used.

Table 16-13 Haul truck performance summary

Haul truck Cat 777		
Measure		
Truck capacity	48	m^3
Actual truck payload	91	t
Truck speed	24	km/h
Haulage distance one way (LOM average)	4.04	km
Cylcle		
Load	2.5	min
Haul	10	min
Tip	1.5	min
Return	10	min
Wait	0	min
Total cycle	24.2	min

16.11.4 Production Drill Rigs

Mine production requires drilling and blasting. It is proposed that 12.5m benches are appropriate for the mining operations. Sandvik D55SP was chosen as the preferred production drill rig and Epiroc FlexiROC D60 was chosen as the preferred pre-split drill rig. Calculations have been performed for the total amount of drill rigs required for the operations.

Using the parameters, together with standard drill performance figures, the following drilling performance can be expected as shown in Table 16-14. These figures are used to estimate the equipment fleet numbers.

Table 16-14 Drill performance parameters

Drill rig Sandvik D55SP / FlexiROC D60		
Measure		
Hole size	172 - 254 / 110 - 178	mm
Bench height	12.5	m
Hole length	14.4	m
Drill rate	27.5	m/h
Single hole	31	min
Drill move	5	min
Overall job efficiency	90 %	
Mechanical availability	87 %	
Operator efficiency and equipment utilization	92 %	
Holes per hour	1.2	
Holes per day	26.1	
Drill meters per t	0.028	m/t
Drill meters per day per rig	376	m/d/rig
Drill meters per year per rig	131 635	m/a/rig
Tonnes per year per rig	4 786 734	t/a/rig

16.11.5 Support Fleet

In addition to the main mining fleet, there will be a fleet of support vehicles.

Pre-split drill:
To ensure good ground conditions and optimum slope design, pre-splitting of the pit walls will be needed to ensure a high-quality face condition. A diesel-hydraulic unit will be required to drill small diameter holes on the final walls for specialised pre-split blasting. The Epiroc FlexiROC D60 drill rig was chosen for this task.

Tracked dozers:
Used for road construction and waste rock hauling operations. A possible option for this task is the Cat D10. Any brand name used in this report is purely for sizing and specification purposes and is not a recommendation to purchase. Two machines will be required during full production.

Wheel dozers:
These are to be used for general cleaning work around the blasthole drill sites and around the excavators.

One will be sufficient. Their extra speed compared to the tracked dozers makes them eminently suitable for a multitude of tasks. Cat 854 has been used for the calculations.

Graders:
Used for road cleaning and grading, and general drainage and ditching work. In open pit mines, the condition of the haul roads is key to a successful venture, so good grading is always important. A fleet of two will be required to ensure constant coverage. Cat 16M would be appropriate.

Wheel loader:
A single unit can be used for re-handling work at the stockpiles. Additionally, it will serve as a backup ore loader if a main excavator is not operating for any reason. A Cat 994 has been used.

Water truck:
During the summer season, dust will be a problem on the mine roads. This affects both the visibility of equipment operators and dust will also affect the area around the site. To keep dust allayed, two water trucks will be used to spray the roads during the operating shifts. A Cat 773G size of machine is proposed.

Additional trucks:
Mechanics will need a mobile service truck to attend to the servicing of tracked equipment which includes the dozers and the drill rigs. Additionally, a fuel truck and a lube truck are required for site support activities.

Personnel vehicles:
Personnel mobility is key in open pit operation, and therefore a fleet of up to 7 pick-up type vehicles is included and may include personnel buses and a medium weight truck with an integrated handling system for carrying larger spares to the site.

Mobile rock breaker:
An excavator mounted hydraulic breaker will be provided for dealing with any oversize material in the pit. Cat 336 size of machine is proposed.

16.11.6 Equipment Schedule

Table 16-15 summarises the equipment requirements for the duration of the mine life.

Table 16-15 Mining fleet requirements

Function	Model type	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Excavators	CAT 6020B	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine trucks	CAT 777	3	6	10	10	10	11	11	10	10	11	11	11	9	9	10	10	10	10	10	11	6	2
Production / Pre-split drill rigs	Sandvik D55SP / FlexiROC D60		3	4	4	4	4	4	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2
Wheel loaders	CAT 994	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tracked dozers	CAT D10	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Wheeled dozers	CAT 854	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Graders	CAT 16M	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Fuel trucks	CAT ADT740	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lube trucks	CAT ADT740	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water trucks	CAT 773G	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Secondary breaker	CAT 336	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Front-end loaders	CAT 980M	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Stemming loaders	CAT 930M	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Light vehicles	Twin cab 4x4	4	6	7	7	7	7	7	6	6	7	7	7	6	6	6	6	6	6	6	7	5	4

16.12 Personnel

The minimum requirements of personnel for mine site activities and support functions are listed in Table 16-16. The labour costs are derived from benchmarking similar operations in Finland.

Table 16-16 Mine staff requirements and costs

Mine staff requirements	**(Persons)**	Cost / person €/year	By-exp. 35%	Total	Total Cost €/year
Mine superintendent	1	109 620	38 367	147 987	147 987
Chief Mine engineer	1	81 900	28 665	110 565	110 565
Mine Planning Engineer	1	75 600	26 460	102 060	102 060
Shift foreman / Contractor supervisor	2	63 000	22 050	85 050	170 100
Maintenance crew	4	44 100	15 435	59 535	238 140
Senior geologist	1	81 900	28 665	110 565	110 565
Shift geologist	2	63 000	22 050	85 050	170 100
Senior surveyor	1	55 440	19 404	74 844	74 844
Survey assistant	1	50 400	17 640	68 040	68 040
Equipment operators (peak year, 5x crew rotation)	130	37 040	12 960	50 000	6 500 000
Total	144				7 692 401

The equipment operators are calculated for the peak year equipment capacity. The peak year equipment capacity is 26. While one additional excavator, two additional mine trucks, and one additional production drill rig are on standby in case of unexpected breakdowns.

Two crews of 26 operators each, will be working whilst three crews are rostered off. One crew will work one week of dayshift and then change to one week of nightshift. Hereafter the crew will have three weeks off.

The cost of all equipment operators is included in the mining operating cost and is presented in Table 16-16 just as reference to give scale on the cost comparison for required personnel.

16.13 Production Schedule

At full capacity, the planned mill feed rate will be 3.25 million tonnes per year. The mining operation starts with overburden removal one year prior to rock excavations. On the first actual mining the planned mining rate is lower enabling an easy ramp up period for the open pit operation. 1,959 kt of material is planned to be mined for concentrator feed. Waste rock mining in the ramp up period is designed to be 4,512 kt. With the 3.25 million tonnes production rate the Life of Mine (LOM) will be approximately 20 years. Table 16-17 presents the LOM plan for this PEA study. Strip ratio for the designed open pit is 1.7.

Table 16-17 Life of Mine plan @ 11.0% Magnetite cut-off.

Mining Period	OVB removal	Ore	Waste	Magnetite	V in situ	Fe in situ	VinMC	Fe
	M3	kt	kt	%	%	%	%	%
1	2 896 575	-	-					
2	1 146 942	1 959	4 512	16.23 %	0.15 %	10.5 %	0.93 %	64.59 %
3		3 250	7 482	15.11 %	0.14 %	9.7 %	0.93 %	63.93 %
4		3 250	7 482	15.64 %	0.14 %	10.0 %	0.88 %	63.92 %
5		3 259	7 503	13.70 %	0.13 %	8.9 %	0.96 %	64.83 %
6		3 250	7 482	15.18 %	0.14 %	9.7 %	0.94 %	63.59 %
7		3 250	7 482	15.48 %	0.14 %	9.9 %	0.89 %	64.04 %
8		3 250	6 132	13.70 %	0.13 %	8.8 %	0.93 %	63.95 %
9		3 259	6 149	13.53 %	0.13 %	8.6 %	0.93 %	63.60 %
10		3 250	6 132	16.03 %	0.15 %	10.1 %	0.94 %	63.29 %
11		3 250	6 132	14.01 %	0.12 %	8.7 %	0.86 %	62.43 %
12		3 250	6 132	13.19 %	0.13 %	8.3 %	0.95 %	63.23 %
13		3 259	4 209	15.13 %	0.14 %	9.5 %	0.92 %	62.99 %
14		3 250	4 197	14.58 %	0.13 %	9.0 %	0.86 %	62.06 %
15		3 250	4 197	13.73 %	0.13 %	8.5 %	0.92 %	62.17 %
16		3 250	4 197	13.34 %	0.12 %	8.4 %	0.93 %	63.01 %
17		3 259	4 209	15.12 %	0.14 %	9.6 %	0.93 %	63.26 %
18		3 250	4 197	13.17 %	0.12 %	8.1 %	0.88 %	61.85 %
19		3 250	4 197	13.05 %	0.12 %	8.3 %	0.93 %	63.28 %
20		3 250	4 197	13.11 %	0.12 %	8.4 %	0.94 %	63.85 %
21		3 259	1 517	11.60 %	0.11 %	7.4 %	0.93 %	64.16 %
22		802	-	11.16 %	0.10 %	7.2 %	0.91 %	64.49 %
Totals	4 043 517	64 553	107 742	14.15 %	0.13 %	9.0 %	0.92 %	63.39 %

NOTE the scheduled tonnes and grade do not represent an estimate of Mineral Reserve. VinMC refers to vanadium in magnetite concentrate and Fe to iron in magnetite concentrate.

16.14 Break Even Cut-off Grade

The cut-off grade used for the September 2020 Mineral Resource estimates is 11% Magnetite based on the breakeven calculation which used the assumed costs for mining and processing. For this PEA study the break-even cut-off was calculated based on the updated cost estimates for using a contractor-based excavation costs and also using an owner operated fleet.

To ensure that the mineral resource estimate can be considered for eventual potential economic extraction, the following economic and technical constraints have been used:

- Total Operating Cost (contractor price) €39.30/t
- Ferrovanadium Price US$ 25.5/kg
- Pig Iron Price US$ 340/t
- Exchange rate 1.18 (US$/€)
- Vanadium grade 0.92%
- Iron grade 63.6%
- Pit slope angle varying

Cut-off value for magnetite was estimated by using a Net Smelter Return (NSR) calculation. The NSR was calculated from the magnetite and represents the combined metal values for iron and vanadium in the ore to produce pig iron and ferrovanadium.

The metal prices were provided by Strategic Resources Ltd., metallurgical performance was based on the evaluation work done for this PEA study. Processing and mining costs were estimated based on material consumption calculations, contractor prices for mining and client owned equipment costs for mining. AFRY confirmed the mining costs by comparing them to in-house prices from similar sized mining operations.

Table 16-18 summarizes the assumed costs, product prices and recoveries that were used in the cut-off calculation.

Table 16-18 Assumed economic and technical parameters for the cut-off grade calculation

Operating Cost			**Contractor**	**Own**
	Waste mining	€/t ore	5.59	3.99
	Ore mining	€/t ore	2.47	2.18
	Mine overheads	€/t ore	0.02	0.02
	Concentrator	€/t ore	3.80	3.80
	Met. Plant	€/t ore	23.86	23.86
	Product freight	€/t ore	2.30	2.30
	Administration	€/t ore	1.26	1.26
Total opex			39.30	37.41
Product prices				
Pig Iron		Us$/t	340.0	
FeV80		Us$/kg	25.5	
Exchange rate		Us$/€	1.18	
Magnetite Concentrate				
Rec to concentrate		%	98.0%	
Fe		%	63.6%	
V		%	0.92%	
Metallurgical Plant				
	Pig Iron			
	Fe rec	%	98.0%	
	FeV			
	V rec	%	79.0%	

The NSR values (€/metric tonne) for a processed material were calculated using varying magnetite grades. The NSR was then compared against the operating cost (OPEX) broken down in Table 16-18 to see what the break-even value and the related cut-off grade should be. Using the assumed metal prices, operating costs and metallurgical recoveries the previously used 11% Magnetite cut-off grade is estimated to still valid (Figure 16-13). It should be noted that the NSR calculation is based on assumed economic and technical parameters presented in Table 16-18.

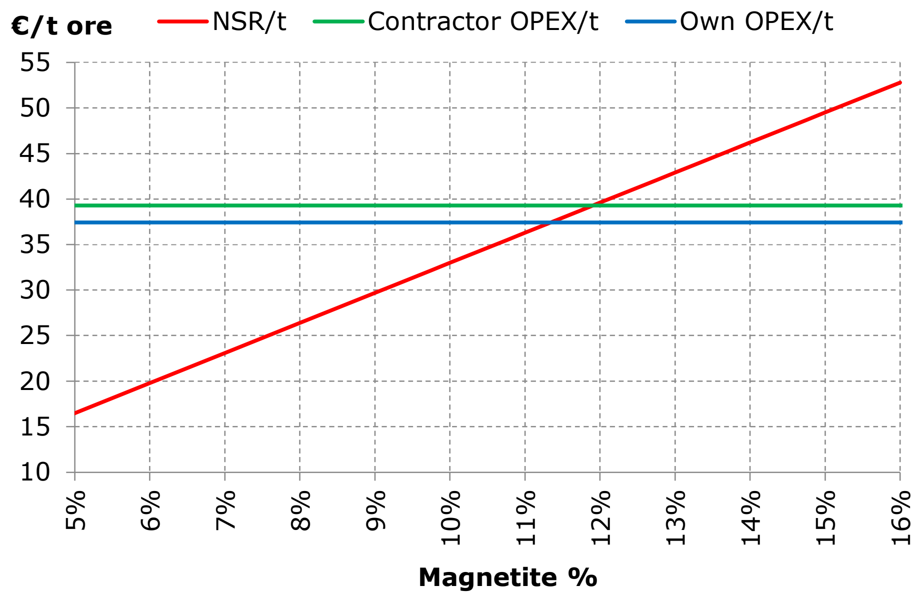

Figure 16-13 Cut-off breakeven calculation

The effects of varying vanadium grade to NSR are presented in Figure 16-14. The NSR calculations are based on assumed economic and technical parameters presented in Table 16-18.

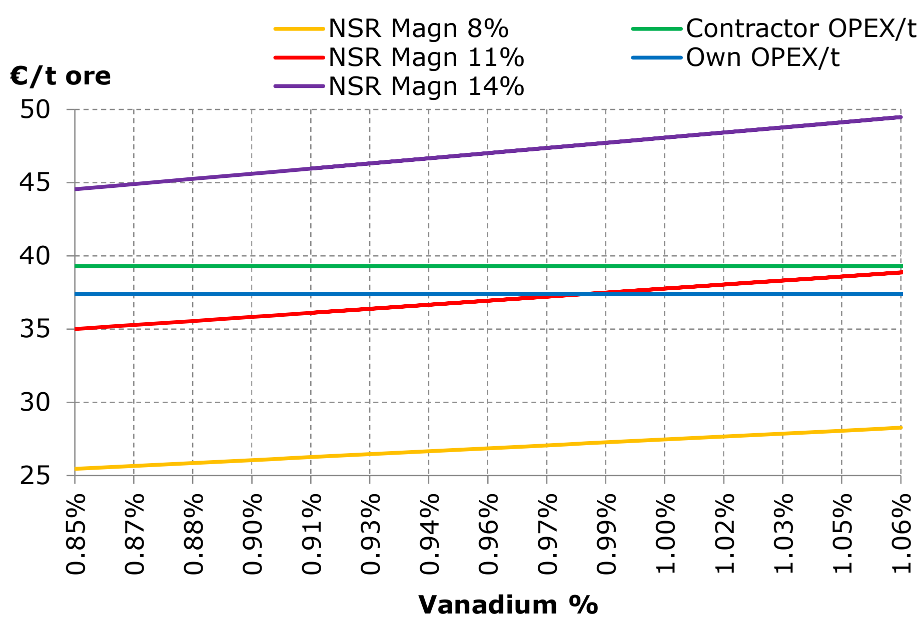

Figure 16-14 Vanadium grade sensitivity analysis

It should be noted that the vanadium grade has a much smaller effect on the NSR value than the magnetite grade and the varying vanadium grade does not affect the profitability with the selected 11% magnetite cut-off.

16.15 Total Material Movements

The total material movement (waste and ore) from the open pit is shown in the Figure 16-15. The annual total excavation volumes are kept at relatively steady level for periods of five to eight years. This enables the competitive bidding of mining contractors at steady intervals and contractors can more easily plan their equipment needs or alternatively the owner operated fleet can be optimized in a way that the fleet is effectively used. First year of the mining operation is planned to be ramp up phase with 50% of full production rate.

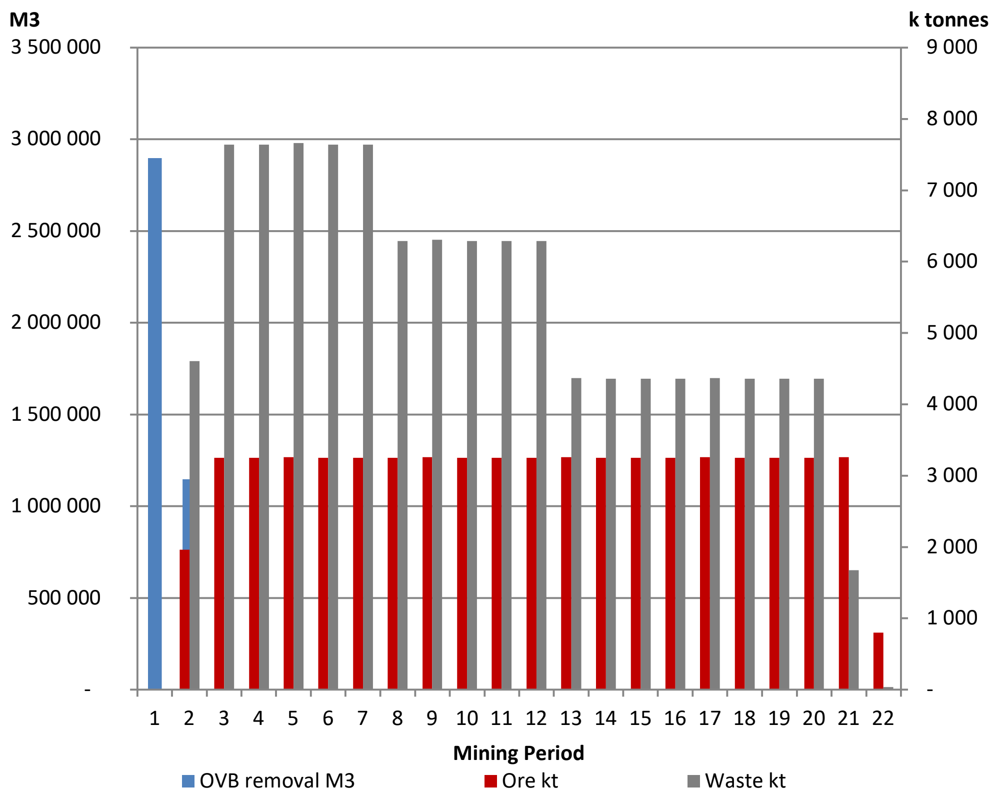

Figure 16-15 Annual total material movement

The annual feed grades stay relatively stable throughout the life of mine with magnetite feed grade varying between 12.41 to 15.91%. VinMC varies between 0.86% to 0.95% and Fe between 61.52 to 64.82%. Figure 16-16 presents the annual feed grades to concentrator plant.

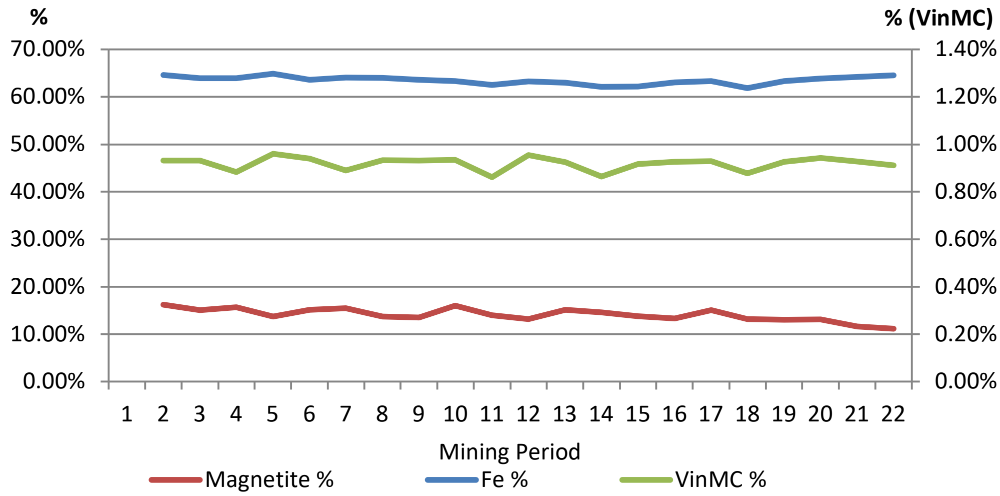

Figure 16-16 Concentrator annual feed grades

Below in

Table 16-9 is presented the excavated material from the open pit. 88% of the processed plant feed material in the mine plan is measured category material and the rest is indicated.

Table 16-19 Total material removed from the open pit

	Tonnes (Mt)	Magnetite	VinMC	Fe
Processed material	65	14.2%	0.92%	63.4%
Waste (Mt)	108			
Strip ratio (w/o)	1.7			

NOTE the tabulated tonnes and grade do not represent an estimate of Mineral Reserve.

17 Recovery Methods

17.1 Process Description

Ferrovanadium production process consists of concentrator plant in Mustavaara and smelter / hydrometallurgical plants in Raahe. Concentrator plant process is based on two-stage crushing, three-stage grinding and multi-stage magnetic separation to produce iron/vanadium concentrate. Direct smelting and selective oxidation are used to bring vanadium to suitable form (vanadium slag) to act as a feed material to roast-leach process. Pig iron is produced as a by-product of smelting process. Roast-leach process is used to produce vanadium pentoxide (V_2O_5) from vanadium slag. Vanadium pentoxide is fed to the aluminothermic reduction. Vanadium product from aluminothermic reduction is ferrovanadium (FeV80). Process is presented in the Figure 17-1.

Process design is based on annual throughput of 3.25 Mtpa ore to the concentrator plant. Estimated amount of annual concentrate production is 505 ktpa with 63.4% Fe grade and 0.92% V grade. Magnetite recovery to the concentrate is estimated to be 98%.

Estimated amount of annual FeV80 production is 4,577 tpa with vanadium recovery to the FeV80 estimated to be 78.9%. Annual pig iron production is estimated to be 329 ktpa. Other possible by-products include TiO_2 slag (83 ktpa), Ca-Al slag (8 ktpa) and Sodium Sulphate (20 ktpa).

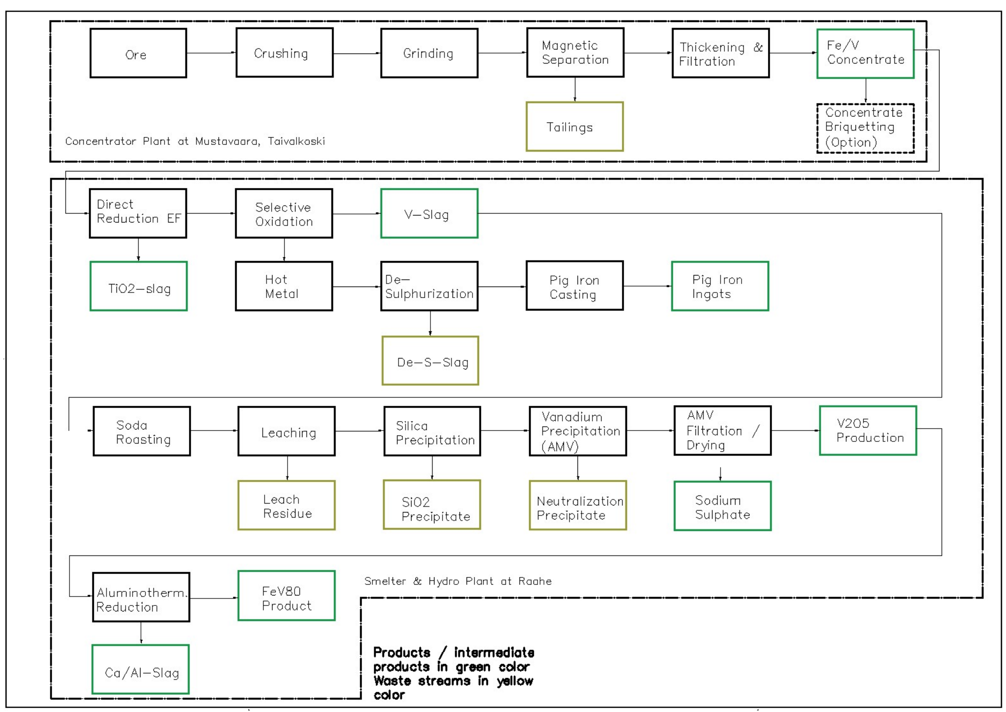

Figure 17-1 Process Block Diagram

17.2 Concentrator Plant

Concentrator Plant is operated 7 days a week in three shifts (estimated plant availability 90%), except crushing which is operated 5 days a week in two shifts. Concentrator plant flowsheet is presented in Figure 17-2.

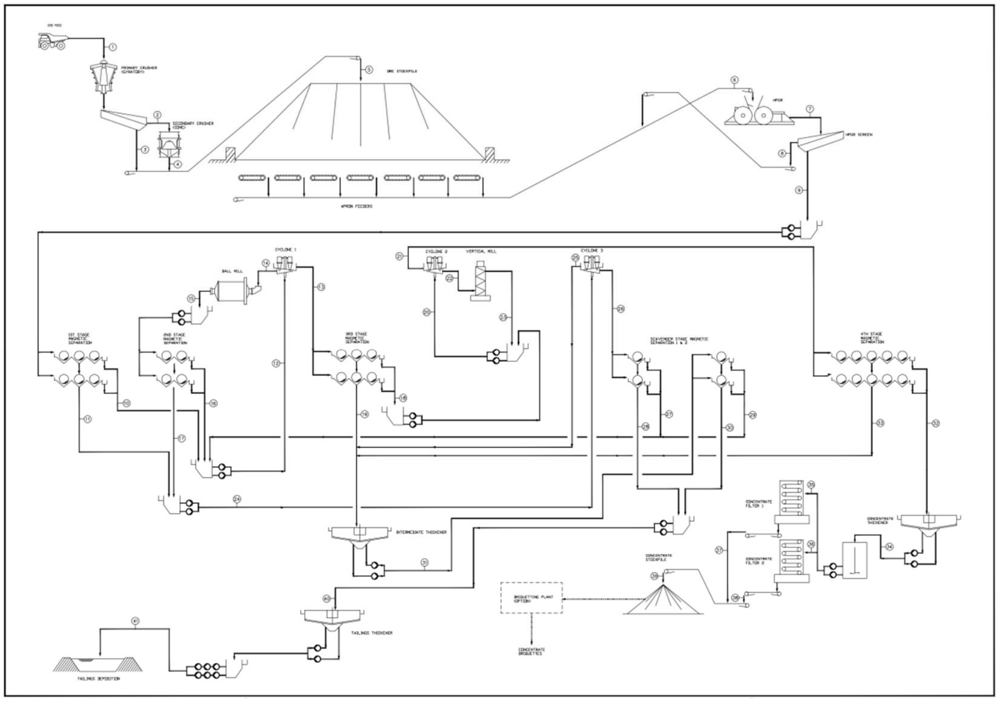

Figure 17-2 Concentrator Plant Flowsheet

17.2.1 Crushing & Primary Grinding

Gyratory crusher (375 kW) is used as a primary crusher. Ore is transported by trucks from the pit to the crusher. Target product particle size after primary crushing is approximately 150 mm (P80). Crushed ore is fed to the screen. Screen overflow is fed to the secondary crusher (cone crusher). Screen underflow and cone crusher product are fed to the conveyor belt and transported to the 30,000 tonne stockpile. Apron feeders are used to feed the ore from the stockpile to the grinding circuit.

HPGR (high pressure grinding roll, 1658 kW) is used in primary grinding. Target product particle size after HPGR circuit is approximately 1 mm (P80). HPGR is operated in a closed circuit with a vibrating screen. Screen oversize is returned to the mill. Screen undersize is fed to the first stage magnetic separation.

17.2.2 Magnetic Separation and Secondary/Tertiary Grinding

Magnetic separation circuit consists of five magnetic separation stages (1,2,3,4 and scavenger separation stages) together with secondary and tertiary grinding stages.

First magnetic separation stage consists of two parallel three drum concurrent LIMS separators. Concentrate is fed to the secondary grinding circuit, tailings to the third hydrocyclone classification.

2,000 kW ball mill is used as a secondary mill. Secondary mill is used to grind first, second and scavenger stage magnetic separation concentrates. Secondary milling is done in closed circuit with pre-classification. Hydrocyclone underflow is fed to the mill, overflow is fed to the third stage magnetic separation. Target product particle size after secondary grinding is 200 µm (P80).

Second magnetic separation stage consists of two parallel double drum concurrent LIMS separators. Concentrate is fed to the secondary grinding circuit, tailings to the hydrocyclone classification.

Third magnetic separation stage consists of three parallel three drum concurrent LIMS separators. Concentrate is fed to the tertiary grinding circuit, tailings to the intermediate thickener.

2,250 kW vertical mill is used as a tertiary mill. Tertiary mill is used to grind third stage magnetic separation concentrate. Tertiary milling is done in closed circuit. Hydrocylone underflow is fed to the mill, overflow is fed to the fourth stage magnetic separation. Mill discharge is fed back to the hydrocyclone. Target product particle size after tertiary grinding is 37 µm (P80).

Fourth magnetic separation stage consists of two parallel six drum concurrent LIMS separators. Concentrate is fed to the concentrate thickener (final concentrate), tailings to the intermediate thickener.

Scavenger magnetic separation stage consists of two single-drum concurrent LIMS separators. Scavenger concentrates are returned to the secondary grinding circuit, tailings are fed to the tailings thickener.

17.2.3 Concentrate Dewatering & Handling

Concentrate from the fourth magnetic separation stage is thickened with a thickener before being fed to two parallel concentrate pressure filters. Concentrate filter cake has an estimated moisture content of 8%. Filter is transported to the concentrate storage with a conveyor belt. Concentrate is loaded to the trucks from the stockpile by wheel loader.

In case concentrate is sold to markets (instead of smelting process) some sort of agglomeration of the concentrate is recommended for easier handling. Agglomeration of the concentrate can be carried out with briquetting. Briquetting plant consists of dosing system for concentrate, binder(s) and other possible additives. After dosing briquetting batch is mixed in order to get homogenous briquetting mix. Briquetting is carried out in roller type presses. After briquetting formed briquettes are screened and under size fraction is returned to mixing. Screen overflow is then dried and cured in curing kiln in order to make briquettes more durable.

17.2.4 Tailings Handling

Intermediate thickener is used to thicken third and fourth stage magnetic separation tailings and hydrocyclone 3 overflow.

Scavenger magnetic separation tailings are fed to the tailings thickener. Solids percentage of thickened tailings is approximately 60%. Tailings are pumped to the tailings pond with centrifugal pumps.

17.2.5 Utilities

Chemicals used in the concentrator plant are lime in tailings neutralization (pH control) and flocculant in thickeners to improve solids settling. Lime will be delivered to the plant in powder form. Flocculant is delivered to the plant in bags/big bags.

Raw water, process water and recirculation water are used in the plant. Plant and instrumentation air networks are used to provide air in the plant.

17.3 Smelting Plant

Smelting and hydrometallurgical plant flowsheet is presented in Figure 17-3.

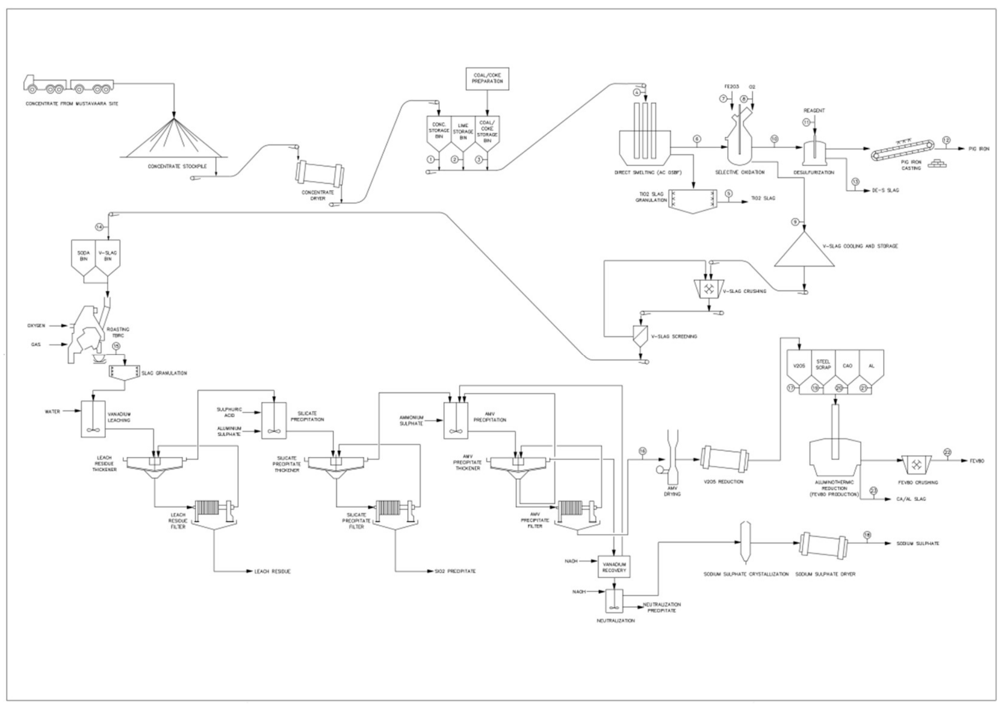

Figure 17-3 Smelting / hydrometallurgical Plant Flowsheet

17.3.1 Raw Material Handling

The smelting plant raw material consist of concentrate from Mustavaara, anthracite which is used as a reductant, lime for fluxing and iron ore pellets which are used as an oxidizer in the converter process. Shipped raw materials are stored prior to feeding them into the raw material handling process.

First raw materials are dried in rotary dryers in order to reduce energy consumption in downstream processes and to ensure accurate dosing. There is a separate dryer for oxide materials (concentrate, lime and iron oxide) and dedicated dryer for the anthracite reductant. After drying the materials are stored in dedicated day bins from which those are dosed and transported to smelting and oxidation feed bins.

17.3.2 Direct Smelting Reduction

Purpose of direct smelting reduction is to reduce iron and vanadium in concentrate into liquid metallic phase which then can be processed further.

Concentrate, lime and anthracite are transported from the dosing to the smelter feed bins in the top of the smelter building. The charge materials are fed into the furnace via several feed chutes which enables even distribution of the charge. Furnace feed system should enable precise feeding of the charge in order to control feed/power –ratio of the furnace.

Smelting and reduction of the Mustavaara concentrate is carried out in open slag bath type AC furnace (OSBF). Open slag bath furnace is one of the only type furnaces which is able to handle fine feed such as Mustavaara concentrate. Also, one limiting factor for the furnace type selection is the high required power input (85 - 90 MW). High required power derives from the high specific energy consumption of the direct smelting which is according to mass and energy balance 1,550 kWh/t_{feed}.

Reduction of the concentrate occurs in the temperature of ~1,600 °C, in that temperature the carbon in anthracite reductant reacts with the oxides and forms metallic elements and CO-gas. The main reduction reactions that are occurring in the furnace are presented below.

Metallic elements descend to the bottom of the furnace and form uniform metal phase. Unreduced gangue from the concentrate and anthracite form oxide slag on the top of the metal phase. Slag is fluxed with lime in order to get basicity (CaO/SiO_2) of the slag to 1.2. The slag basicity level reduces the activity of the silicon in the melt which reduces silicon reduction in to the metal phase.

Heat required of the reduction reactions comes from electrical power that is transferred from the furnace transformer, via a conduction path from the water-cooled bus tubes, through flexible copper connections to the lower electrode and contact shoes and into the baked electrode tip.

Electrodes in the OSBF are Söderberg-type which are made from lengths of steel casing sections that are added to the top of the electrode column and electrode paste is added inside the casing. When the electrode paste travels downwards, towards the furnace, the paste melts and bakes and forms uniform electrode.

Metal and slag tapping is carried in approximately in two-hour intervals. Vanadium rich hot metal is collected into the ladle from which is transported to further processing. Slag is granulated with high pressure water spray, which cools down and disperses the slag flow into small droplets. Granulated slag can be used as a construction material. Analysis for furnace hot metal and slag is presented in Table 17-1.

Table 17-1 Analysis of Furnace Hot Metal and TiO2 Slag

Hot Metal		**TiO_2 Slag**	
Component	**w-%**	**Component**	**w-%**
Fe	93.45	CaO	26.97
C	4.40	MgO	3.76
V	1.30	SiO2	22.47
Mn	0.13	Al2O3	8.21
Si	0.25	MnO	0.27
P	0.004	V2O5	0.47
S	0.26	Na2O	0.00
Ti	0.20	TiO2	37.14
		P2O5	0.00
		FeO	0.70

In addition to hot metal and slag, carbon monoxide containing off-gas (CO-gas) is major product of the furnace. The hot off gases are washed and cooled down with water scrubbers. After scrubbing CO-gas is pressurized and utilized as a fuel in the other parts of the smelting and hydro plant.

17.3.3 Selective Oxidation

Purpose of selective oxidation phase is to oxidize the vanadium in the hot metal into slag which can be then further processed. Oxidation is carried out in converter which is refractory lined vessel which can be tilted for charging and tapping.

Vanadium rich hot metal from the smelting reduction is transported by ladles into the converter. In the selective oxidation aim is to oxidize as much vanadium as possible without oxidizing excess amount of other elements in the hot metal. After charging the hot metal into the converter oxidation commences. Oxygen required in the oxidation reaction is provided in the form of iron oxide pellets and gaseous oxygen. Also, CO_2 can be used as an oxidizer. Iron oxide pellets are charged from the top of the converter from the charging opening and the gaseous oxidizers from the submerged tuyeres from the bottom part of the converter vessel. Gases also enhance the oxidation reactions due to stirring.

Oxidation reactions are controlled mainly with temperature because a lower temperature promotes metallic element oxidation rather than carbon oxidation. Temperature is controlled with iron oxide pellets and oxygen. Iron oxide pellets lower the temperature of the melt whereas the oxygen increases the temperature.

After the oxidation converter is tilted to tapping position and the liquid hot metal is tapped to ladle for further processing. After hot metal tapping the converter is turned upside down and the solid slag is extracted to the slag pot from the mouth of the converter. Slag is then cooled and transported for further processing. Analysis of Oxidation Hot Metal and V Slag is presented in Table 17-2

Table 17-2 Analysis of Oxidation Hot Metal and V Slag

Hot Metal Component	**w-%**	**V Slag Component**	**w-%**
Fe	95.69	CaO	0.48
C	4.00	SiO_2	11.45
Mn	0.01	Al_2O_3	0.48
P	0.00	MnO	2.79
S	0.27	V_2O_3	32.97
V	0.04	P_2O_5	0.01
		TiO_2	5.91
		FeO	44.55
		MgO	1.37

17.3.4 Pig Iron Handling

The hot metal from the smelting and oxidation process contains significant amount of sulfur which originates from the anthracite reductant. For the hot metal desulfurization is needed in order to valorise the hot metal into saleable product. Typical sulfur content required for pig iron is <0.01%. Desulphurization is carried out after the oxidation in order to reduce vanadium losses to the desulphurization slag.

Desulphurization is carried out in the ladle station where the desulphurization reagents are injected from submerged lance to the hot metal. Reagents are magnesium and calcium-based powders which react with the sulphur in the hot metal forming calcium and magnesium sulphides according to reactions below.

FeSi may be needed in order to reduce dissolved oxygen in the melt, lowering dissolved oxygen enhances the sulphur removal conditions. Also, reactions are enhanced with bottom stirring. Silicon reacts with dissolved oxygen and forms silicon oxide according to reaction below.

After the injection and stirring, the ladle is moved to de-slagging station where the slag is raked of from the top of the metal. When the slag is raked ladle is transported to pig iron casting machine which casts the pig iron into ingots. Estimated Fe content of pig iron is 95.9%.

17.4 Hydrometallurgical Plant and FeV80 Production

Purpose of hydrometallurgical plant is to produce vanadium pentoxide (V_2O_5) from vanadium slag to act as ferrovanadium (FeV80) production process feed.

17.4.1 Roasting

V-slag from selective oxidation is first crushed, cooled and screened. Crushed V-slag is fed to the roasting furnace. Sodium carbonate is fed to the roasting furnace to modify vanadium into a water-soluble form (sodium metavanadate, $NaVO_3$) and reduce the required melting temperature. Roasting process temperature is approximately 1,200 °C. Molten slag is collected into slag pots and fed to the slag granulation. Formed slag granules are fed to the vanadium water leaching.

17.4.2 Vanadium Leaching and Solid/Liquid Separation

Granules containing sodium metavanadate are fed to a series of agitated hot water leaching tanks. Solution passes through each tank until the sufficient dissolution is achieved. Number and volume of leaching tanks is selected according to required leaching residence time. Leaching is done in closed tanks in atmospheric pressure conditions. pH level in sodium vanadate leaching is basic (in the range of 11-13) and temperature approximately 70–80 °C. Remaining heat from roasting process provides part of required heating to the leaching process.

Non-soluble residue from the vanadium leaching is thickened in a leach residue thickener to approximately 40% solids. Underflow from the thickener is pumped to the leach residue filter. Overflow from the thickener continues to the silicate precipitation circuit.

Thickened residue is filtered in a pressure filter. Cake washing is used in the filtration process to enhance the recovery of vanadium. Residue filter cake is collected and transported to the hazardous waste disposal area by truck. Filtrate from the leach residue filter is returned to the leach residue thickener.

17.4.3 Silicate Precipitation and Solid Liquid/Separation

PLS (pregnant leach solution) from vanadium leaching proceeds to the silicate removal circuit. Silica is precipitated in a series of precipitation tanks with aluminium sulphate. pH level in sodium vanadate leaching is approximately 8 and temperature approximately 50–70 °C. Sulphuric acid is used in a pH control. Number and volume of precipitation tanks is selected according to required precipitation residence time. Precipitation is done in closed tanks in atmospheric pressure conditions.

Silicate precipitate from the precipitation circuit is thickened in a silicate precipitate thickener. Underflow from the thickener is pumped to the silicate precipitation filter. Overflow from the thickener proceeds to the AMV precipitation circuit.

Thickened silicate precipitate is filtered in a pressure filter. Cake washing is used in the filtration process to enhance the recovery of vanadium. Residue filter cake is collected and transported to the hazardous waste disposal area by truck. Filtrate from the leach residue filter is returned to the silicate precipitation thickener.

17.4.4 AMV Precipitation and Solid/Liquid Separation

PLS from the silicate precipitation circuit proceeds to the AMV precipitation circuit. AMV (ammonium metavanadate, NH_4VO_3) is precipitated in a series of precipitation tanks with ammonium sulphate. pH level in AMV precipitation is approximately 8 and temperature approximately 50–70 °C. Number and volume of precipitation tanks is selected according to required precipitation residence time. Precipitation is done in closed tanks in atmospheric pressure conditions.

AMV precipitate from the precipitation circuit is thickened in a AMV precipitate thickener to approximately 30% solids. Underflow from the thickener is pumped to the AMV precipitation filter. Part of the underflow can be returned to the AMV precipitation to act as seed material. Overflow from the thickener continues to the vanadium recovery circuit.

Thickened AMV precipitate is filtered in a belt or pressure filter. Filter cake is fed to the AMD drying process. Cake washing is used in the filtration process to ensure purification and moisture content of the cake proceeding to the AMV drying. Estimated moisture content of AMV filter cake is approximately 25%. Filtrate from the AMV filter is returned to the AMV precipitate thickener.

17.4.5 AMV Drying and V2O5 Reduction

AMV filter cake is dried in a flash dryer. Flash dryer utilizes hot gas to dry AMV moisture content down from 25% to <0.1%. Dried AMV is discharged to the V_2O_5 reduction furnace feed hopper. AMV is reduced in rotary tube furnace at approximately 900–1000 °C to V_2O_5. V_2O_5 is cooled down and conveyed to the FeV production area.

17.4.6 Vanadium Recovery, Neutralization and Sodium Sulphate Production

AMV precipitation thickener overflow is fed to the vanadium recovery process. Vanadium recovery utilizes ion exchange process to recover remaining vanadium. Recovered vanadium is returned to the AMV precipitation circuit.

Neutralization process utilizes sodium hydroxide (caustic soda) to neutralize the solution from vanadium recovery. Neutralization precipitate is collected and transported to the hazardous waste disposal area with trucks. Remaining solution after neutralization precipitation is fed to the sodium sulphate production.

Sodium sulphate can be potentially produced as a by-product. Solution from the neutralization is cooled and sodium sulphate is crystallized to salt including crystal water. Crystallized sodium sulphate is dried to sodium sulphate flakes.

17.4.7 Hydrometallurgical Plant Utilities

Hydrometallurgical plant and FeV production require at least the following utilities:

- Water (fresh/cooling/sealing/potable)
- Plant and instrumentation air
- Steam for water leaching
- CO gas
- Aluminium and ammonium sulphates in precipitation circuits
- Sulphuric acid in pH control
- Sodium hydroxide (caustic soda) in neutralization
- Coagulant and/or flocculant in thickeners

17.4.8 FeV Production

Purpose of the ferrovanadium production is to produce ferrovanadium alloy by aluminothermic reduction from the vanadium pentoxide. Other raw materials include aluminium, which is used as reductant, iron scrap which forms FeV alloy with reduced vanadium and lime which is used as a flux.

Reduction is carried out in DC furnace because aluminothermic reaction requires initial heat to start. In the reaction metallic aluminium reacts with vanadium pentoxide forming metallic vanadium and aluminium oxide. Metallic vanadium and molten iron descend to bottom of the furnace forming FeV alloy and oxide elements form Ca-Al slag in top of the metal phase.

FeV production is batch wise operation which starts by charging initial charge mix to the furnace. Then power is switch on and the heat from the electricity starts the reaction. If the furnace is still hot from previous batch, reaction can start spontaneously. After the reaction is started more raw materials are charged to the furnace until desired batch size is achieved.

After the feeding has stopped the furnace is kept hot in order to promote alloy settling. After certain time the furnace is tapped into pot where the settling continues, and the cooling happens. After the alloy and slag has cooled pot is tipped and both alloy and slag are crushed and transported to customers. Estimated vanadium grade of FeV80 product is 80%.

17.5 Process Recommendations and Alternatives

17.5.1 Concept Development and Modelling

Mineral processing (concentrator plant) section of the Mustavaara ore processing is well understood and based on well-known unit operations. Further development of concept to optimize and confirm opex and capex estimates (e.g. comminution concept mentioned in chapter 17.5.2) is recommended.

Proposed pyro- and hydrometallurgical process has been developed using assumptions from the previous studies and existing references. Process modelling using the most recent concentrate composition should be done to create more accurate mass and heat balance. Also, further mineral processing and metallurgical test work to confirm recovery estimates is recommended for the PFS level study.

17.5.2 Comminution Circuit Development

Current comminution concept is based on use of HPGR (high pressure grinding roll) after primary and secondary crusher. Screened HPGR product is fed to the magnetic separation and ball mill circuit. HPGR based process was selected and equipment sizing done in the PEA phase based on the existing comminution test work results and discussions with equipment vendors. Further development of the comminution circuit together with vendors is recommended for the following project phases. HPGR and vertical mill test work are required to prove the suitability of comminution concept to the Mustavaara ore. Equipment vendors have a capability to conduct both HPGR and vertical mills test work. Risk in the HPGR based comminution is that an additional grinding stage (e.g. ball mill) and classification stages might be required if HPGR product is not fine enough for the following process stages.

17.5.3 Utilization of Secondary Vanadium Sources (Slags)

Utilization of vanadium containing slags from steel production have been previously investigated to be used alongside Mustavaara concentrate as a vanadium source. Potentially available vanadium-containing steel making slags in Finland & Sweden have generally higher vanadium content than Mustavaara concentrate. Availability and potential mixing of slags with concentrate could increase ferrovanadium production significantly and therefore improve project economics.

Based on the test work, approximately 20 – 30% of slag could be used alongside with concentrate without any significant changes to process flowsheet and equipment dimensioning. Slag would also remove the need for lime because vanadium containing slags are typically high in CaO. Additional de-phosphorization process could be needed if the slag is to be utilized.

17.5.4 Pre-reduction of Concentrate

There is possibility to lower the smelting energy and reductant consumption with pre-reduction of the concentrate. Based on the earlier test work the possible smelting energy and reductant reduction would be ~30–40%. The pre-reduction has a minor effect to project NPV because it increases CAPEX but on the other hand lowers OPEX. It is estimated that pre-reduction equipment would increase initial CAPEX by 100 – 200 M€. Pre-reduction is not included in the PEA base case.

Main positive upside of the pre-reduction is that it would lower the risk resulting from energy and reductant price fluctuations. The effect of pre-reduction to overall process and project economics should be further investigated.

17.5.5 Alternative Intermediate and Final Products

One possibility to reduce OPEX would be to produce V_2O_3 instead of V_2O_5 and use that as a feed stock for FeV production. For risk mitigation purposes V_2O_5 was selected as an intermediate product because it is a saleable product contrary to V_2O_3.

Using V_2O_3 as a FeV production feed stock it is estimated that lime and aluminium consumption would be 40% lower compared to V_2O_5 use. Also, specific energy consumption would be lower.

Production of low-grade ferrovanadium (FeV50) has been considered in previous studies, but with current market situation it's not seen as a viable option.

17.5.6 Hydrometallurgical Treatment of Concentrate

Alternative process for the treatment of concentrate would be direct feeding of concentrate into the hydrometallurgical roast-leach process without smelting. This type of direct roast leach process is currently in use in Largo Resources' Maracas Ménchen mine and in Bushweld Minerals' Vametco plant. Also, historically it has been utilized in Rautaruukki's Otanmäki mine.

This option would increase the feed tonnage to the hydrometallurgical process by factor of 27 (19 ktpa vs 505 ktpa). However, as the vanadium content in the feed wouldn't increase in the same factor compared to the feed tonnage, unit process sizing after roasting & leaching wouldn't increase by same factor.

Pre-requisite for this process path would be lower calcium (CaO) content of the concentrate. With currently estimated calcium content of the Mustavaara concentrate it is probable that concentrate roasting process would create unsoluble calcium vanadates. These unsoluble compounds reduce the vanadium recovery in the roast-leach process significantly. Methods for reaching lower CaO content in the concentrator plant should be included in the further evaluation of this option.

17.5.7 Selective Oxidation in Ladle

Selective oxidation could possibly be conducted in ladle instead of converter which is the base case of the PEA. Converter was selected because it provides controlled process with enough stirring. Also tapping from the converter reduces hot metal losses to slag significantly compared to slag raking from the ladle.

Oxidation in the ladle would bring some savings in the CAPEX because then converter and its auxiliaries would not be needed. Oxidation could be conducted at same ladle station as the de-sulphurization. However, the effectiveness and suitability of the ladle oxidation needs to be further investigated.

18 Project Infrastructure

18.1 Mustavaara Mine Site

18.1.1 Introduction

The Mustavaara site Project Infrastructure is mainly based on concepts presented in earlier works, in particular the Pöyry PFS revision B dated 15 March 2012 and mine environmental permit application (16ICI4126.40.LUP) dated 19 December 2012. From dam construction perspective, new pre-engineering work was conducted by AFRY during the PEA.

The earlier work covers processing plant, all buildings, tailings and waste rock deposition solutions and structures, site water management systems, utility supply, and site power supply. Although the reports are eight years old at the time of the writing of this present report, the content of the former PFS report and the environmental permit application are, from most parts, considered to be still valid and appropriate to be used as basis for the current PEA.

The environmental permit for mining operations has been received in March 2016 (EVP (Nro 32/2016/1), 2016). The environmental permit regulates the Project environmental emissions and discharges, and guidelines the construction of environmental protection structures. The former PFS report and environmental permit application have been reviewed for:

- compliance with environmental permit regulations
- possible changes in circumstances that would invalidate any of the content
- costing methodologies

The LOM in the environmental permit application and decision is only fifteen (15) years and the LOM mine waste volumes for tailings and waste rock are in accordance to that (this applies to 2012 PFS as well). For purposes of this PEA, the LOM waste volumes have been re-calculated and corresponding facilities have been re-designed.

The used unit prices are based on the latest AFRY project cost references.

18.1.2 Site Layout

The site layout prepared for Pöyry (2012) study and for the environmental permit application, served as basis for the general site layout development.

The site layout is presented below in Figure 18-1. Additionally, Appendix 1 (document number 10002) provides the detailed layout drawing.

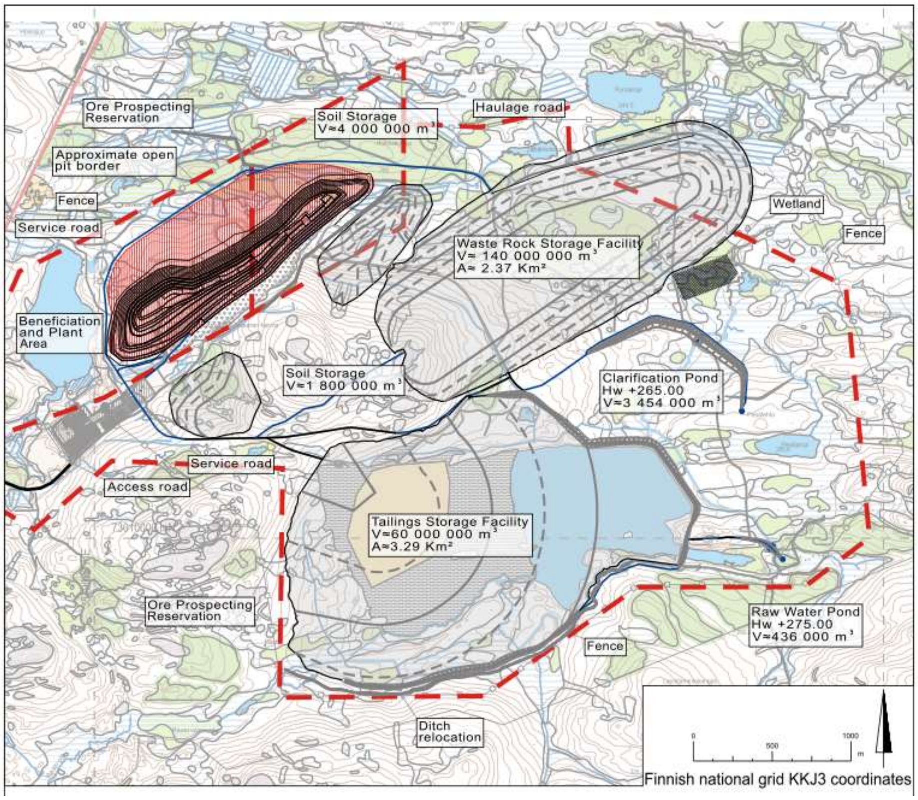

Figure 18-1 Mustavaara Mining Area, Site Layout

18.1.3 Plant Layout and Buildings

The Mustavaara area processing plant layout has been developed to cover all the activities and facilities required for an operating mine (Figure 18-2). Plant layout is mainly based on the solutions done in the previous study. A more detailed drawing can be found in Appendix 2 (document number 10000).

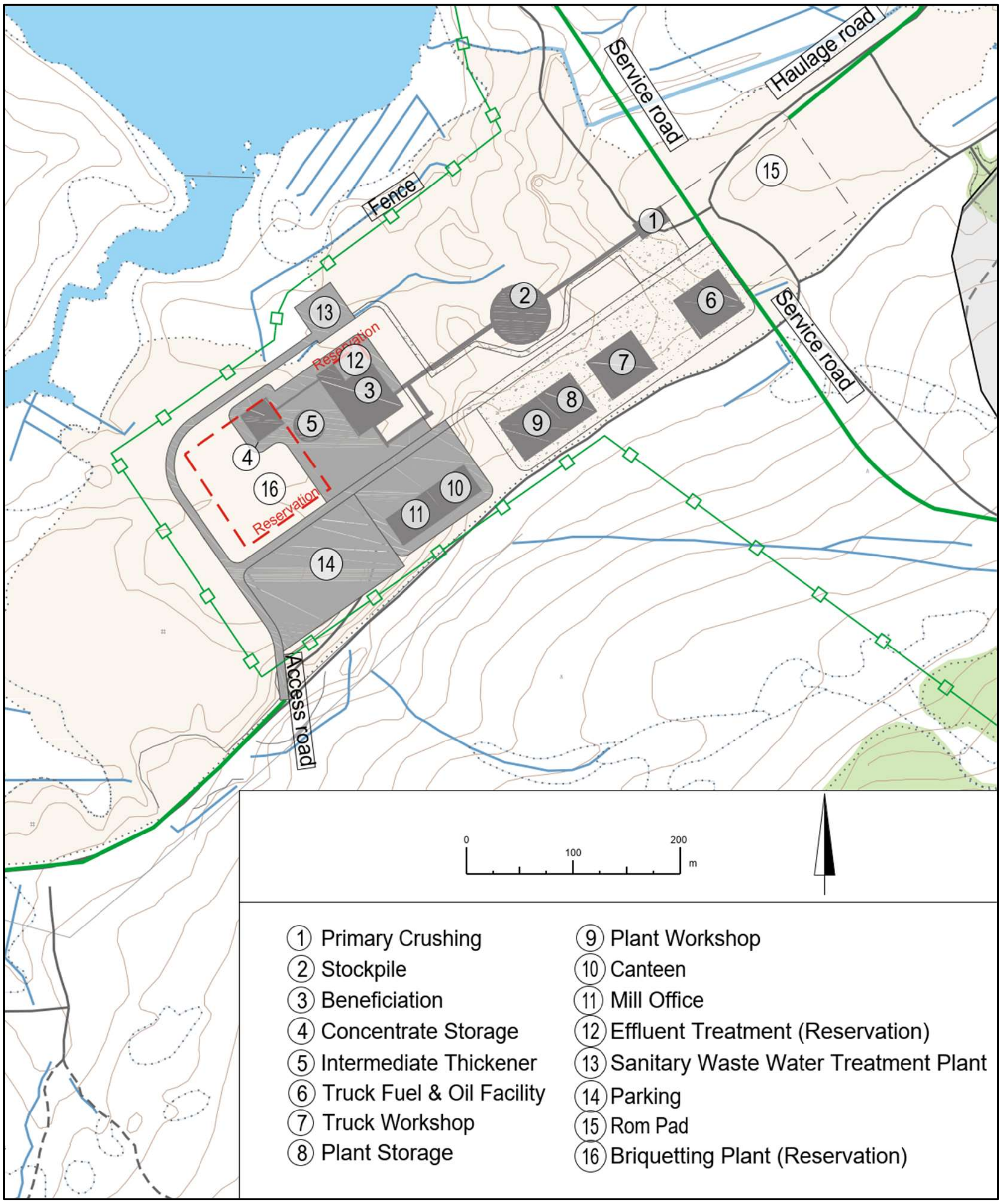

Figure 18-2 Plant Layout

The concentrating area buildings and site facilities are listed in Table 18-1.

Table 18-1 List of Buildings and Site Facilities

Item #	Building / Facility	Area [m²]
1	Primary Crushing	450
2	Stockpile	2,400
3	Beneficiation Building	2,900
4	Concentrate Storage	900
5	Intermediate Thickener	700
6	Truck Fuel & Oil Facility	2,100
7	Truck Workshop	2,100
8	Plant Storage	1,260
9	Plant Workshop	2,100
10	Canteen	1,200
11	Mill Office	1,500
12	Effluent Treatment Reservation	-
13	Sanitary Waste Water Treatment	450
14	Parking	9,600
15	Rom Pad	12,000
16	Briquetting Plant Reservation	-

According to the Finnish legislation, the buildings must be designed to conform to the requirements for permanent buildings since the life span of the operation is more than five years. Buildings must comply with the requirements of the Finnish construction legislation and recommendations, and the design and construction must adhere to the Finnish national construction standards.

The planned site facilities, their structural solutions, building foundation principles, and costing, as described on the previous PFS, are found appropriate for the purposes of the PEA.

18.1.4 Site Service Utilities

Potable water (drinking water) is supplied from a local external supplier. Potable water tank and distribution on the site are well described and costed on the previous study.

All buildings will be connected to the plant’s fire water network. The fire water network is fed from the raw water tank.

Sanitary water is collected and supplied to a sanitary water treatment unit, owned by the mining company. In the Pöyry 2012 study the planned technology of the sanitary water treatment unit has been a biological-chemical batch process. The solution is appropriate for the purposes of the PEA, but it is recommended, that the future Project development phases would re-consider upgrading the technology to continuous process (for example bio-rotor process) or even outsourcing the service supply.

The site facilities are planned to be connected to a district heating system that is sourced from a site heating plant. The heating plant consists of a 3 MW solid fuel boiler (wood chips or peat) and two back-up and peak boilers. The back-up and peak boilers, having capacities of 4 MW and 2 MW, use oil as fuel. The solution and costing presented in previous study is considered to be appropriate for the purposes of the PEA.

A fuel station, which is divided to serve both heavy mining vehicles and light vehicles, is planned on site. Area of the fuel station is in total 750 m². Runoff water from the fuel storage area shall be led and treated in an oil separation system. The solution and costing was checked and considered appropriate for the purposes of the PEA.

Process chemicals will be stored in a reagent storage in connection with the main process building. Explosives will be stored in a separate explosives storage, which is located outside the processing plant area.

18.1.5 Site Infrastructure

18.1.5.1 Deposition Areas

The planned Waste Rock Storage Facility (WRSF) is located to east of the open pit, partly at the north-east slope of the Mustavaara hill. The area has been partly used for waste rock deposition during the previous mining operation. Planned WRSF footprint is 2.37 km². Environmental permit (EVP (Nro 32/2016/1), 2016) condition 43 regulates, that the highest WRSF deposition level is +360 in N60 height system. Also the final slope of the WRSF is regulated to 1V:3H or gentler.

The environmental permit regulates, that all overburden material, that is removed from the construction sites, has to be stored and used later on closure structures.

18.1.5.2 Roads

The existing site access road and existing site internal service roads will be improved by adding a crushed rock layer on top of the existing road structures. In addition, asphalt pavement will be added to the access road surface. The length of the improved access road section is 1,800 m.

Additionally, new service road lines will be built to access the dams, WRSF, and the north side of the open pit. The road structure will be made of blasted rock and crushed rock. Road surfacing material shall be crushed rock. The total length of the new road alignments is 7,600 m.

Ore and waste rock haulage roads will be constructed to WRSF and to the ROM-Pad from the open pit. The road structure consist of blasted stone fill and crushed rock surfacing. The total length of the haul roads measured from the ultimate pit outline is 2,100 m.

18.1.5.3 Plant Area and ROM-Pad

Earthworks cost in the mill site are calculated based on yard and building coverage areas. Yard area unit costs consist of excavation, crushed rock fillings, surface leveling as well as storm water wells and pipes. Traffic area will be partly paved with asphalt. From building areas, all frost susceptible soil will be removed up to frost depth and replaced with crushed rock fill. The total plant area footprint is 11.36 ha. ROM-Pad, which area is 1.5 ha, consist of lined basal structure and waste rock and crushed rock filling.

18.1.5.4 Gates and Fences

Fence will be constructed around the whole mining operation area. Total length of the fence line is 16,000 metres. The costing of gates and fences is updated to this PEA.

18.1.5.5 Area Lighting

Cost for area lighting is included in the previous PFS cost estimate. The cost allocation is considered appropriate for the purposes of the PEA.

18.1.6 Tailings Disposal and Management

18.1.6.1 Current Tailings Area

Due to the historic mining, on the Mustavaara site is located an existing tailings storage facility (TSF). The new deposition is planned on top of the existing deposition area. Currently, approximately 1/5 of the tailings area is used for agriculture (Figure 18-3).

Figure 18-3 Viewing East Towards the Current Tailings Area

18.1.6.2 Tailings Management and Disposal Solutions

Thickened tailings (60% w/w consistency) will be deposited from the western ridge of TSF using one-point discharge method. Dams are needed to the south, east, and north part edges of the facility to prevent tailings propagation outside from the area. The estimated tailings average beach slope angle is 3%.

The dam structure consists of supporting embankment, a sealing layer, a filter layer, and a slope erosion protection layer. Supporting embankment and slope protection layers will be made with blasted rock (waste rock) from the open pit. The sealing layer will be made from glacial till material borrowed from the overburden removal of the open pit. The filter layer will be made from crushed aggregate from site (crushed waste rock). The total length of perimeter dams is approximately 3,700 m and the total volume of dams is approximately 600,000 m³.

Supernatant water from TSF will be discharged to the Clarification Pond through overflow structures located in the dam section between the ponds.

Based on the mine environmental permit requirement, there is no need for an environmental basal structure beneath the tailings fill. Thus the majority of TSF earthwork cost consist of dam construction costs.

Nesting sites of protected species in the south-west part of the Raiskiovaara hill may have a major impact on the TSF design and operation. Operations in the proximity of the nesting site are regulated on the environmental permit conditions 8 and 9. On the PEA plans, the chosen mitigation measure is amendment permitting to change the existing permit conditions. Thus the tailings deposition plan includes an environmental risk, which is described in section 20.13.1.

18.1.7 Water Ponds

As described on the former PFS, the main water storages on the mine site are the Clarification Pond and the Raw Water Pond. They both require construction of earth-fill dams. The previous PFS and the environmental permit application (2012) describe the dam dimensions slightly differently.

For the purposes of this PEA, dam slope inclinations have been re-checked and the dam volumes have been re-estimated.

The main figures of the water storage dams are summarized in the Table 18-2.

Table 18-2 Main Water Storage Dams as Described in the Environmental Permit Application (2012)

	Clarification Basin Dam	Raw Water Basin Dam
Dam crest	+267.7	+277.7
HW level	+265	+275
Pond volume at HW-level	3.4 Mm³	440,000 m³
Dam type	Earthfill dam consisting of supporting embankment, till	Homogeneous till dam

	sealing layer, filter structures and erosion protection layer	
Main dimensions of the dam	L = 1,400 m H_{max} = 20 m	L = 300 m H_{max} = 5.5 m
Dam volume	800,000 m³	18,500 m³

18.1.8 Site Water Management

18.1.8.1 Water Management Requirement from the Environmental Permit

The environmental permit for mining operations has been received in March 2016 (EVP (Nro 32/2016/1), 2016). The environmental permit regulates the Project site water management by regulating the discharges to environment. A summary table of the relevant permit conditions is presented in Table 18-3.

Table 18-3 Summary Table of the Relevant Environmental Permit Conditions Regarding Site Water Management

Permit Condition(s)	Topic	Main Stipulations
13	Construction phase water management and construction sequencing related regulations	Construction phase discharges to natural water courses: maximum solids concentration 30 mg/l
14	General water management principles	- Clean and contaminated water are to be separated - Water recycle rate in the process should be maximized
15	Discharge water quality regulations	See below
17	Sanitary water treatment plant regulations	Required removal efficiencies: - BOD7/ATU 90% - Total P 90% and ≤ 0.8 mg/l Treated sanitary waste water is to be led to the TSF
18, 19	Operational phase water management regulations	- Mine has to have a water balance model - Water treatment capacity has to be dimensioned for 1/20 years repeating storm conditions - Water recycle rate in the process should be maximized
59	Process raw water intake from the Sirniönlampi lake	Raw water intake should happen mainly from the Clarification Basin or the Raw Water Basin. Secondarily, the Sirniönlampi lake can also be used for raw water abstraction at maximum abstraction rate of 215 m³/h.
61	Raw Water Basin excess water flow direction	Raw Water Basin excess water flow direction is to the Pesälampi lake

The permit regulations for water discharge to the Lavotjoki river are as follows (permit condition 15):

pH	5.0 – 9.0	individual sample
Vanadium	0.2 mg/l	monthly flow weighted average
Copper	0.1 mg/l	monthly flow weighted average
Solids (fixed residue)	10 mg/l	quarterly flow weighted average

Additionally, target value for total nitrogen is 14 mg/l as monthly flow weighted average.

Total annual loading of vanadium is restricted to 130 kg/a and total annual loading of copper to 110 kg/a. Target value for total nitrogen loading is 22 000 kg/a.

Permit condition 15 talks about discharges to the Lavotjoki river but in practice the permit condition requirements are to be met in the water discharged from the Clarification Basin.

18.1.8.2 Site Water Balance

The former PFS report (2012), as well as the environmental permit application, included descriptions of site water balance modelling principles and results. The water balance model has been generated as deterministic model in Microsoft Excel. The modelling has been performed on annual basis, but only for the 15 years' LOM footprints.

For the purposes of this PEA, the previous water balance models were reviewed, combined, and updated as appropriate. The site water balance was modelled for the 20 years' LOM scenario. The updated model outcome is described in Figure 18-4.

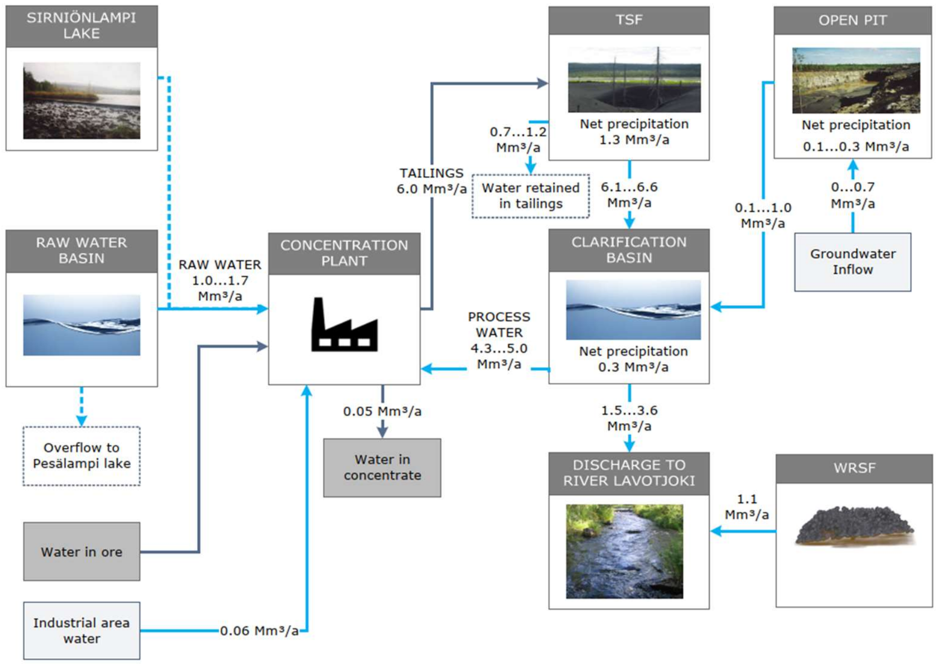

Figure 18-4 Conceptual Water Cycle Graph and Preliminary Annual Site Water Balance

The most critical recommendations for the future PFS work include review of groundwater inflow component, review of process components, re-assessment of tailings deposition solution, and generation of the model on monthly basis for the entire LOM. The current groundwater inflow component on the water balance model is from the environmental permit application and it appears, that there is not hydrogeological model behind the estimate. Additionally, as later described in section 18.1.8.4, the water balance model should be enhanced to include the water quality components and thus to serve as loading balance, as well. The loading balance will be, on future phases, important to understand, as the environmental permit does not restrict the discharge volumes as such but the loading to discharge water stream.

18.1.8.3 Raw Water Abstraction and Water Discharge

Raw water abstraction has on the previous PFS been planned from the Raw Water Basin and from the Sirniönlampi lake. Discharge water course has on the previous PFS been the Lavotjoki river. Excess water from the mine water cycle is discharged from the Clarification Basin.

Raw water abstraction and excess water discharge solutions are in compliance with the environmental permit. AFRY however recommends, that despite the solution compliance to the environmental permit, the system dimensioning flow rates should be checked in the future PFS work.

Furthermore, a careful discharge water course impact assessment is still needed, as the ecological status of nearby lakes and rivers cannot be deteriorated[1] (see chapter 20.13 *Environmental and Social Risks*). Enough initial data for this is typically available as a result of PFS stage. The assessment should follow the loading balance development. If the impact assessment should indicate deterioration of the ecological status of a water body or jeopardize the attainment of its status objectives, additional water treatment or new discharge point in larger river could be required.

Raw Water Basin overflow discharge to the Pesälampi lake is a new requirement from the environmental permit. However, the cost implication of this new requirement is not significant and it is recommended that this item is added to the cost estimate in the future PFS work.

18.1.8.4 Water Quality and Water Treatment

The previous PFS does not discuss the water qualities at any extent. The effluent water treatment solution is a passive treatment that consists of sedimentation on the Clarification Basin and wetland treatment before discharge to the Lavotjoki river.

The environmental permit application (2012) discusses the water quality aspects and declares the following:

- Process water quality estimate is estimated based on pilot plant water analyses and measured water quality data from the existing tailings area downstream water
- Pilot plant water analyses indicated high concentrations of for example aluminum, cobalt, copper, iron, manganese, nickel, vanadium and zinc but the soluble portion of these components were low

On the permit application is stated that the need for an active water treatment unit is seen unlikely but a provision is made.

In Finland, the environmental permitting authorities have during the past years been recommending active, plant-like water treatment applications for mine effluent water treatment. Furthermore, it seems that the process water quality is not properly declared and analyzed on the previous studies and there is no comprehensive loading balance at place. It is recommended, that re-analyzing of process water quality shall be included in the future process test work. Based on updated and thorough water quality source term assessments, re-considerations of process internal water cycle and water treatment are recommended for the future PFS work.

From costing point of view, primarily based on the fact that the current environmental permit does not, eventually, require other water treatment processes than described in the previous PFS, the 2012 PFS water treatment costing solution is considered appropriate for the purposes of the PEA.

[1] According to Directive 2000/60/EC of the European Parliament and of the Council (Water Framework Directive)

18.1.8.5 Water Management Structures

Site water management related water ponds are described in section 18.1.7.

Site water management related pumping systems (pumps and long-distance water transportation pipelines) are well described in the former PFS. The system dimensioning was not reviewed in detail for the purposes of this PEA. However, the dimensioning appears to be in the right order of magnitude and all major structures are included. Thus, the solutions and costing of water management structures are considered appropriate for the purposes of the PEA. The main water management related structures are summarized in the list below (see Table 18-4). The permit decision raw water abstraction solution deviates from the PFS solution, which is incorporated on the item list.

Table 18-4 List of Site Water Management Related Pumping Systems as Described in the 2012 PFS, environmental permit deviations incorporated

Item Description	Related Structures
Process water reclaim from TSF Clarification Basin to process water tank (PWT)	- Pumping station at Clarification Basin (CBPS1) - 2 x 200 kW pumps (PEA phase comment: 3 x 75 kW pumps would rather be suggested) - Pressure pipeline from CBPS1 to process water tank; L=5,300 m; HDPE PN 10; OD 630 mm
Process water tank (PWT) pumping systems (from PWT to process)	- Tank volume 25,000 m³ - 3 x 300 kW pumps
Raw water pumping station	- 2 x 45 kW pumps - Pressure pipeline from the pumping station to process water tank; L=6,000 m; HDPE PN 10; OD 200 mm
Pit dewatering pumping station	- 2 x 70 kW pumps - Pressure pipeline from pit to Clarification Basin; L = 2,650 m; HDPE PN 10; OD 315 mm

18.1.9 Mine Power Supply and Distribution

On the previous PFS, the total installed power of the concentration process main equipment has been 11.5 MW and total peak power including all processes is estimated at 18 MVA for 505 ktt/a production. The required electric power will be provided through connection to the local 110 kV power grid. The new 110 kW power line covers a distance of approximately 32 km and it will be built to connect the mine site to an existing switchyard at Posio municipality area. The power line route will follow the same path that was used during the former mining operations in Mustavaara area.

In addition to power supply, the previous PFS has included plans for power transmission and distribution. The previous plan and costing were found appropriate for the PEA purposes. It is recommended, that the solutions are reviewed and updated as part of any future PFS work.

18.2 Smelter Site

The smelter production plant is planned to be located in SSAB's industrial area in Raahe, Finland. The smelter site Project Infrastructure is fully based on Ferrovan Oy's former study phases. Ferrovan Oy's latest metal production plant development phase was the Early Works phase during Q4/2018-Q1/2019. During the Early Works phase, project design and cost estimation was developed to accuracy level of +/- 5% (CAPEX estimate dated 8.2.2019).

The Early Works phase project infrastructure covers buildings, plant infrastructure, raw water supply, water treatment, water discharges, and all necessary utility supplies. The content of the former study phases and Early Works phase project budget is considered to be valid and appropriate for the current PEA. Additionally, the work is far more detailed than the work done for this PEA study. At the time of this report Strategic Resources do not have an agreement use the smelter site. However, City of Raahe is prepared to offer a long-term rental agreement to smelter operation. Needed permits are described in detail in chapter 20.3.

The capital costs presented in the Early Works phase cost estimate (8.2.2019) are considered appropriate for the purposes of the PEA as such.

The smelter area plant layout is presented below in Figure 18-5. Additionally, Appendix 3 (document number 10001) provides the detailed layout drawing.

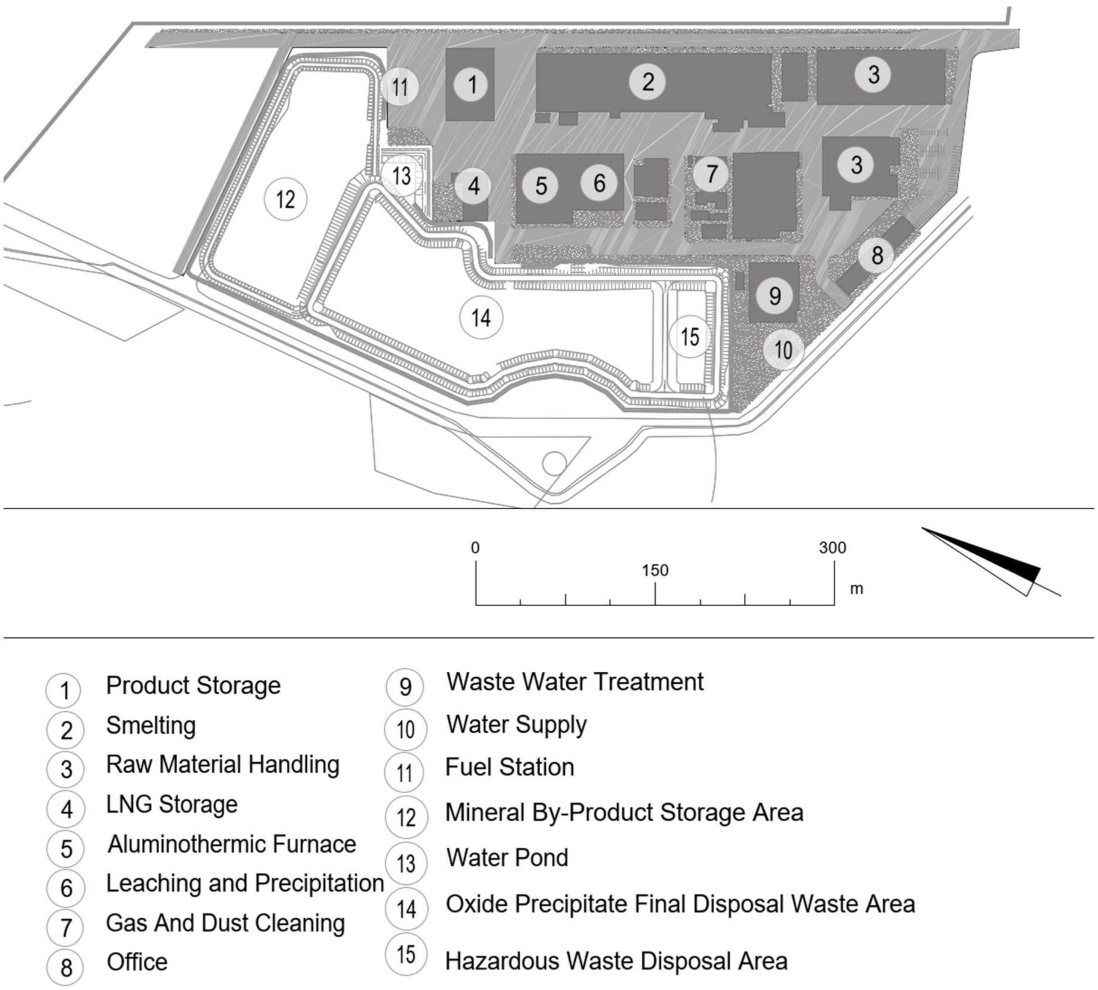

Figure 18-5 Smelter area plant layout

19 Market Studies and Contracts

No contracts are in place for products.

19.1 Market Studies

Information in this chapter is mainly derived from sources available to general public.

19.2 Mustavaara Ferrovanadium and Pig Iron

Implementing the smelter and hydrometallurgical plant enables production of ferrovanadium (FeV80, min. 78% V) and pig iron bars from the concentrate with following presented in Table 19-1.

Table 19-1. Potential Mustavaara ferrovanadium and pig iron compositions.

	Fe	C	Mn	P	S	V	Al
FeV80	19					80	1
Pig iron	95.9	4.05	0.01	<0.01	<0.01	0.04	

19.2.1 Ferrovanadium

On average over the LOM, Mustavaara is expected to produce 3,500 metric tonne of pure vanadium per year in form of FeV80. For comparison, global vanadium production has grown from ~76,000 metric tonne of pure vanadium to ~102,000 metric tonne in 2011 to 2019 according to Vanadium market summary conducted by TTP Squared. In the same period, vanadium consumption has increased from ~72 000 metric tonne to ~102 000 metric tonne . Consumption in China and Taiwan only increased from 30 000 metric tonne to 54 000 metric tonne within the period of 2011 to 2019. According to TTP Squared this growth is largely attributed to increased use of vanadium in Chinese rebar applications. In 2019 84% of all vanadium consumption in China was in rebar.

According to TTP Squared research, world excluding China has consistently lower production than consumption of vanadium which leads to requirement of Chinese exports to maintain balance in the market. From 2016 to 2019 Chinese consumption of vanadium has increased from 35 000 metric tonne of vanadium to 54 000 tonne. This has an impact on the availability of the Chinese exports to other markets (Perles, Terry; TTP Squared, Inc., 2020).

For purposes of this PEA study, FeV80 price of 32 USD/kg have been used. FeV80 Europe spot price history from December 2016 to March 2021 is presented in Figure 19-1. Over this period, the mean price is approximately 44 USD/kg and median 31 USD/kg. Prior significant vanadium price increase in 2017 and 2018 global vanadium production decreased from approximately 90 000 metric tonne in 2014 to 76 000 metric tonne in 2016. Global vanadium consumption in 2016 was 81 000 metric tonne and increased to 95 000 metric tonne in 2018 while annual production lacked behind by several thousand metric tonne as presented in Figure 19-2.

Figure 19-1 FeV80 Europe spot from Dec 2016 to Mar 2021 (data from investing.com).

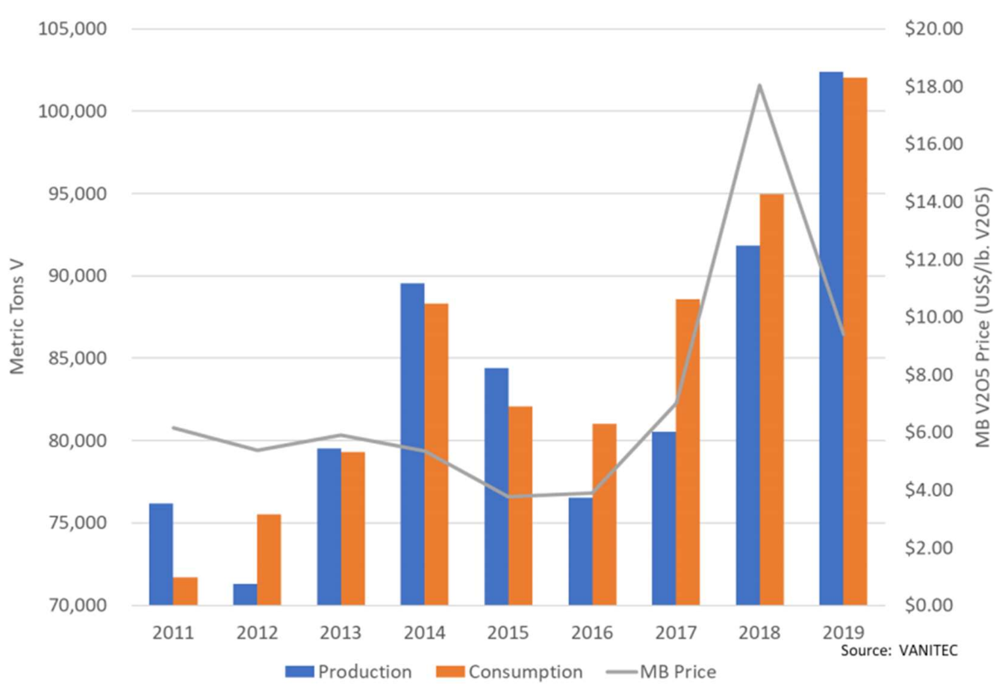

Figure 19-2. Global annual vanadium production & consumption (Perles, Terry; TTP Squared, Inc., 2020)

19.2.2 Pig Iron

Mustavaara is estimated to produce approximately 318,000 metric tonne of pig iron per year over the LOM.

Opinion for pig iron pricing was requested from Finnish trading house Meca-trade Oy by the Owner's team. Based on the pig iron composition presented in Table 19-1, Meca-trade estimated a price range of 360-400 EUR/metric tonne. US$450 /metric tonne. Penalty for higher phosphorus content is estimated at 30-50 EUR/metric tonne should P-% increase to 0.05, which is not the case for Mustavaara's product. Consequently, higher P content is estimated to make marketing more difficult.

Mustavaara pig iron is suitable material for foundries due to its low phosphorus content to be used in casting. Pig iron bars can be used for cooling the melt during BOF process in steel mills, yet this application typically favors scrap usage due to lower carbon content.

19.3 Other By-Products

Three saleable by-products are identified: TiO_2-slag, sodium sulphate and Ca-Al-slag. TiO_2-slag can potentially be used in construction applications as a replacement for other materials, for example in road construction as a fill material. Sodium sulphate is used in pharmaceutical industry as a drug or for therapeutic use. Ca-Al-slag is used in steel industry.

These can be considered as potential upsides for revenue generation after more detailed research.

19.4 Outlook

The 14th Five-year plan is currently being prepared by Chinese government and according to S&P Global questionnaire respondents some 68% expect it would help the steel industry via consolidation and urbanization (development of infrastructure) of central and western China. Consolidation of the steel industry is seen to reduce the capacity and increase pricing power of the larger companies according to S&P Global's sources (Hao;Cao;& Bartholomew, 2020).

Infrastructure development involves rebar and therefore depending on the extent of construction activities, rebar consumption in China can be expected to increase in the coming years. According to Chinese standard, rebar grades 3,4 and 5 require 0.03%, 0.06% and >0.1% of vanadium, respectively (Roskill, 2018). According to TTP Squared, >10mm diameter rebar production in China in 2019 was approximately 213 million metric tonne of which 60 million metric tonne were inferior quenched & tempered (Q&T) steel subject to be replaced with Nb or V bearing steel. Of the remaining 153 million metric tonne of rebar, 33 million metric tonne were based on Nb alloyed steel and 120 million metric tonne on V alloyed steel. TTP Squared states that elimination of Q&T steel could lead to an additional 12 000 – 20 000 metric tonne of vanadium consumption depending on relative prices of V and Nb (Perles, Terry; TTP Squared, Inc., 2020).

Additionally, the 14th five-year plan is focused on transfer from fossil fuel-based energy generation to renewable sources. Reuters article states that based on Wood Mackenzie evaluation, in order for China to reach its goal in energy transition, solar, wind and storage capacities must increase eleven-fold to 5 040 gigawatts (GW) by 2050 (Reuters, 2020). Vanadium redox batteries are one potential solution for storage solution and systems are being implemented in several countries (Colthorpe, 2019).

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Introduction

NI 43-101 requirements include discussing the reasonably available information on environmental, permitting, and social or community factors related to the project. This means considering and, where relevant, including

a) a summary of the results of any environmental studies and a discussion of any known environmental issues that could materially impact the issuer's ability to extract the mineral resources or mineral reserves;
b) requirements and plans for waste and tailings disposal, site monitoring, and water management both during operations and post mine closure;
c) project permitting requirements, the status of any permit applications, and any known requirements to post performance or reclamation bonds;
d) a discussion of any potential social or community related requirements and plans for the project and the status of any negotiations or agreements with local communities; and
e) a discussion of mine closure (remediation and reclamation) requirements and costs.

20.2 Environmental and Social Setting

The locations of the smelter and the mine site are presented below (Figure 20-1). Project environmental and social setting is summarized in Table 20-1 and Table 20-2.

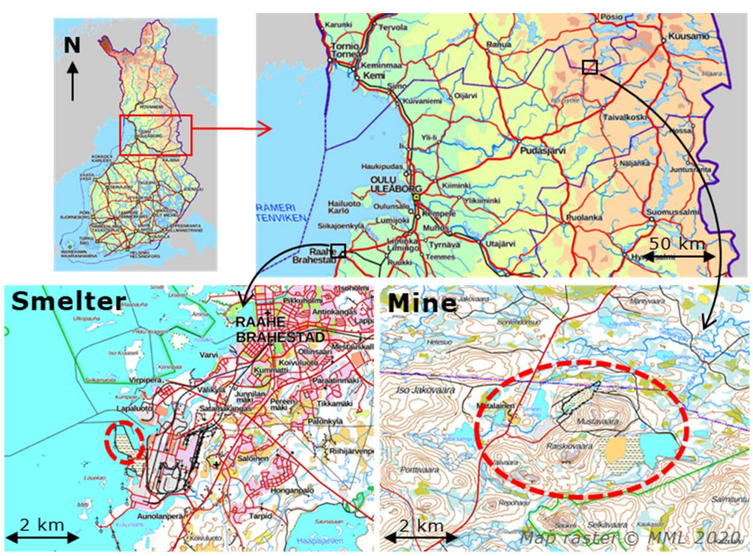

Figure 20-1. The location of the smelter and the mine site. The smelter would be located in Raahe and the Mine site on the border between Taivalkoski and Posio municipalities.

Table 20-1. Environmental and Social Setting Summary, Mustavaara Mine Site

Geographical setting	The Project location is in northern Finland at the border of North Ostrobothnia and Lapland provinces, approximately 650 km north from Helsinki (capital of Finland) and 180 km northeast from city of Oulu.
Climate	The climate in Finland is intermediate and both features of marine and continental climate are typical. Site climate is relatively humid (rain and low evaporation).
Catchment area	Iijoki-river water system; Kostonreitti water system (61.1) and Sirniönjoki-river (61.654)
Surface water status	Some of the rivers and lakes near (downstream) the project area are classified in good or excellent ecological status according to the Decree on Water Resources Management (1040/2006). The first watercourses are lacking official ecological classification, but there is some ecological data indicating that they have a good or excellent status, despite the vanadium content (due to the previous mining activities) impacting the water chemistry.
Groundwater	There is an important groundwater area for water supply and a groundwater intake near the mine site. Even thoug it is close to the mine site, it is located in a different sub-catchment area.
Biodiversity: Nature protection areas	Nearest Natura 2000 -area (Salmitunturi - Rääpysjärvi SAC/SPA, FI1105405) is located only few hundred meters from the project area. Syöte national park (west from the Project area) is close to the exploration reservation areas. (Figure 20-2). In the project area there are few objects protected according to forestry and water legislation (Act 1096/1996 and Act 264/1961), including creeks, ponds and springs.
Biodiversity: Flora and fauna	The project is located in northern boreal coniferous forest zone and belongs into the southern parts of the Perä-Pohjola forest vegetation zone. According to the mine site environmental permit documentation, inside of the project area there are some threatened plants, but no objects of the strictest protection measures. Probably most significant plant species is Leathery grape-fern (Sceptridium multifidum), which is Near Threatened (NT) and Regionally Threatened (RT). The avifauna of the area is an issue to be observed: for example 4 EU Bird Directive –species nesting within the area and six species that are Finland's special responsibility species. There are also two bird species with high protection status.
Landscape	Site is approximately 290 m above sea level. Landscape is dominated by hills and forest areas.
Cultural heritage and monuments of antiquity	There are some known ancient remains in the vicinity of the Project area, but not within the area to be built.
Residential and holiday houses	Less than 5 houses are located within distance of 1 to 2 kilometres from the Project area.

Social structure of the area	Population of Taivalkoski includes approximately 4000 inhabitants. Project area is located ~25 km north from Taivalkoski municipality centre, partly in Posio municipality. There are few houses north-west from the site (distance approximately one kilometre). Nearest village (Sirniö) is located 4 km to north from the Project area.
Resettlement issues	There are no known resettlement issues as there are no residential or holiday houses within the project area.
Indigenous people	Taivalkoski and Posio municipalities are not included to the Sami people homeland area.
Reindeer herding	The Project area is located in the reindeer herding area of Taivalkoski reindeer owners' association.

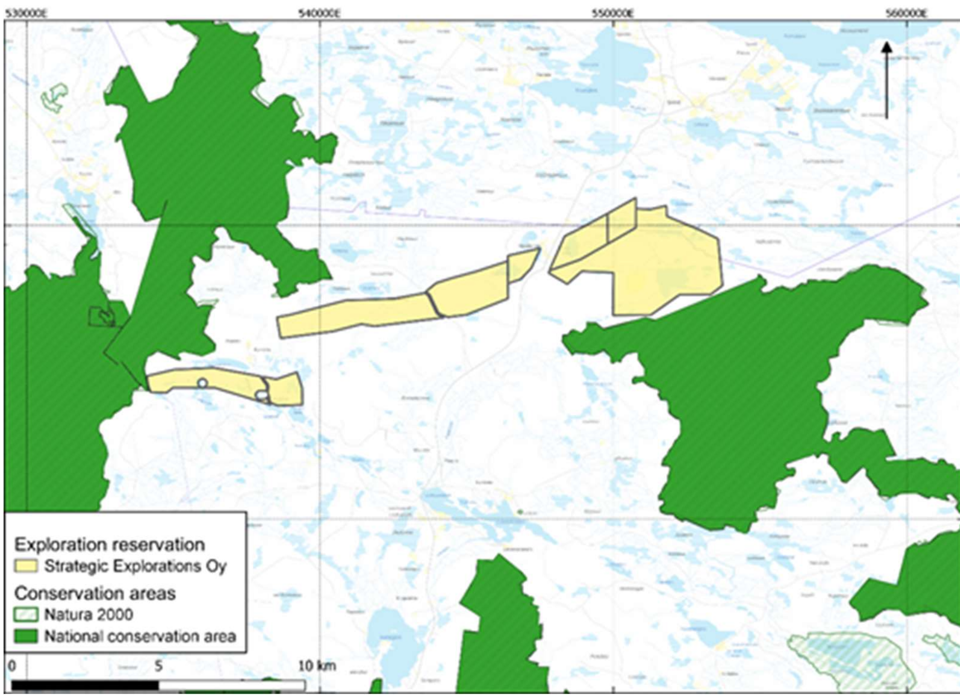

Figure 20-2. Mustavaara exploration reservations and adjacent conservation areas

Table 20-2. Environmental and Social Setting Summary, Raahe Smelter Site

Geographical setting	The smelter site is located by the Gulf of Bothnia, in Raahe City, in the North Ostrobothnia region, Northern Finland. The site location is within the Raahe Port area.
Climate	The climate in Finland is intermediate .The annual average temperature in 2015 was 5.6°C. The Monthly average temperature varies between 16.1°C (August) and -6°C (January).
Catchment area	The site is located on an earth-fill area next to the deep-water harbour, surrounded by sea water (Gulf of Bothnia).
Surface water status	The ecological status of the water zone closest to the shore is satisfactory, and a few kilometres out, good. Nutrient content in water is rather high, causing eutrophication. There are several emission sources in the nearby area; biggest is the SSAB steel factory.
Hydrogeology	The area is located on an earth-fill next to the deep-water harbour. The nearest classified groundwater area is located more than 6 kilometres North-East of the area.
Biodiversity: Nature protection areas	The Raahe archipelago is included in a nature protection area which is also a Natura 2000 area. This area is located North of the site, one kilometre away at its closest.
Biodiversity: Flora and fauna	As the area is an earth-fill, there are no major nature values.
Landscape	The site is located at the shore, between the relatively flat landscape on land, and the open sea. The nearest environment is dominated by the harbour and the steel factory.
Cultural heritage and monuments of antiquity	There are no valuable objects in the project area or nearby area.
Social structure of the area	Raahe is a town with 24,000 inhabitants and situates 5 kilometers to north-east from smelter site.
Residential and holiday houses	There are no residential buildings in the nearby area, the closest residential area is Lapaluoto, 1.5 km to the North-East. The closest summer cottages are located South of the site, 1.3 km away.
Resettlement issues	There are no known resettlements issues as there are no residential or holiday houses within the project area.
Indigenous people	Raahe town is not included to the Sami people homeland area.
Reindeer herding	The area is not located within the reindeer herding area.

20.3 Regulatory Framework and Approvals

20.3.1 General Information Concerning the Regulatory Framework

Most of the critical environmental regulation applied within the mining and metals industry in Finland are national implementations of EU regulations. Examples of important directives in this industry are:

- Directive 2006/21/EC of the European Parliament and of the Council of 15 March 2006 on the management of waste from extractive industries and amending Directive 2004/35/EC
- Directive 2000/60/EC of the European Parliament and of the Council establishing a framework for the Community action in the field of water policy

20.3.2 Finnish Permitting Process – Environmental and Water Permits

In Finland, environmental permits are required for all activities involving the risk of pollution of air and water or contamination of soil. One important condition for a permit is the use of Best Available Techniques (BAT). Environmental permit application must include extractive waste management plan and suggestion for financial guarantee.

Environmental permit applications must be submitted to the relevant authority, as defined in the Environmental Protection Act (527/2014) and Decree (713/2014). For Mustavaara project, the relevant permit authority is PSAVI (Northern Finland Regional Administrative Agency).

After filing a permit application, the authority will publish the application to allow the relevant other authorities and anyone affected by the plans to comment and make proposals concerning the requirements for the permit. Permit decisions may be appealed to the Administrative Court of Vaasa and subsequently to the Supreme Administrative Court.

Water permits are required when the planned operation or activity may alter the position of the groundwater table or the groundwater quality. Also, changes related to the water flow or shorelines are subject to authorization. Permits according to the Water Act (587/2011) and the Water Decree (1560/2011) are applied in the environmental permit application context. For Mustavaara water permits, the permit authority is PSAVI.

20.3.3 ESIA Process

An ESIA (environmental and social impact assessment) procedure for mining projects in Finland is required prior to the permitting. ESIA status of Mustavaara project is described in the following chapter 20.3.7. The Finnish ESIA-procedure (for operations of this scale and type) is not integrated in the permitting process. The procedure includes the ESIA program stage and the actual ESIA. The purpose of the ESIA procedure is to assess the environmental and social impact but also to share information and add interaction with different stakeholders. An ESIA does not lead to a permit decision, but a finished and approved ESIA process enables the submission of an environmental permit application. The coordination authority for the ESIA procedure in the Mustavaara project is PP-ELY (Northern Ostrobothnia Centre for Economic Development, Transport, and the Environment).

20.3.4 Mineral Rights and Mine Safety Permitting

Mineral rights, including decisions concerning mining permits, are regulated under the Mining Act (621/2011). Tukes, the Finnish Safety and Chemicals Agency, is the responsible authority.

Mining Act also requires completion of ESIA before the mining permit can be granted. The ESIA report and coordinating authority statement can be attached to both the mining permit application and the environmental & water management permit application.

Ownership of the mining area land is not required by law, but ownership may simplify many issues related to compensations and liabilities.

To build a mine and start up the actual mining operations, a mine safety permit is required. According to the Mining Act 621/2011, this is called a mine safety permit (section 12 of the Mining Act) and is mostly related to work safety items. The application requires, e.g., a mine general plan and an internal rescue plan. This permit governs all safety-related issues and is granted by Tukes.

20.3.5 Other Permits

Permits can be required also according to the Building and land-use Act (132/1999). Additional deviation permits according to the Nature Protection Act or Natura 2000 assessments may be required in case of the presence of sensitive species or habitats. Dam safety is regulated by the Dam Safety Act (2009/494); the building permit for a dam requires a statement from the dam safety authority KAIELY (Kainuu Centre for Economic Development, Transport, and the Environment), risk assessment and a dam safety monitoring plan. A negotiation duty concerning reindeer herding is implied when the project impacts on government-owned land within the reindeer-keeping area (Reindeer Keeping Act 848/1990).

Tukes handles also applications for the utilization and storage of industrial chemicals according to the Industrial Chemical Decree (59/1999), and it is the authority for chemical registrations, labeling, and packing according to the REACH-decree and CLP-decree. The use and storage of explosives, lifting equipment, and electrical work require permits from Tukes.

20.3.6 Land Use Plans

Recently, due to the current Mining Act and Environmental Protection Act, the importance of land use planning procedures has increased considerably. For example, a mining permit cannot be granted in case the land use plan of the area is inadequately defined. The Mining Act stipulates that any mining activity shall be based on a legally binding plan in accordance with the Land Use and

Building Act or, considering the impacts of mining activity, the matter shall be otherwise sufficiently explored in cooperation with the local authority. In general, mines must be included in both regional and municipal land use plans.

Land use planning and construction are regulated by the Land Use and Building Act (132/1999) and Decree (895/1999). The land use planning system consists of the national land use guidelines and three planning levels: the regional land use plan, the local master plan, and the local detailed plan. The principle of the land use planning system follows the descending hierarchy towards more specific plans.

More detailed land use plans must follow the guidance of the regional plan which also heavily influences the prerequisites for granting the environmental permit and mining permit. The need to renew local master plans and local detailed plans depends on the state of existing regional land use plans, the location of the mining project, local conditions, and the scale and impacts of the project including associated buildings. New master plans or detailed local plans may also be required whenever it is important to harmonize the mining activity with other local activities. Planning permission for ore processing facilities, smelter and other significant buildings must be applied based on the detailed local plans.

The local master plan is a municipality's general land use plan. Master planning defines the principles of targeted development and prescribes the preparation of local detailed plans for the area. The relevant municipalities in Mustavaara project are Taivalkoski and Posio.

Land-use planning status of the project is described in the following chapter 20.3.7.

20.3.7 Project Approvals Status

Important items to be considered for project permitting are presented in the table (Table 20-3). As production capacity in this PEA (2020) is larger than the permitted production capacity, it is assumed that amendment permitting is needed to cover the total production assessed in this PEA. This probably also triggers a new ESIA-procedure.

Amendment permitting would be needed also due to other current permit conditions. Permit condition 8 stipulates that due to nesting sites of a protected species, no constructions or changes in natural conditions are allowed within 400 m radius from protected bird nesting sites (3 sites, south-west from planned operations). According to the permit condition 9, during the potential nesting season (15th February – 31st July), no constructions or car transports are allowed within 1,000 m radius from the nest sites. Working within the planned operation area under these permit conditions is could be restrictive. (Notice, protected species nesting information cannot be added to this type of document in more detail than presented here.)

Nesting sites discussed above have been assumed been abandoned for years, but according to new data gained from Metsähallitus (government forest council) 9th December 2020, nesting sites have been active since 2011 . This increases the project's risks related to the current tailings plan. Strategic is actively working on mitigation plans.

Amendment permitting/ESIA costs are included to the PEA economic model.

Table 20-3. Project Permitting.

Authority and *Legislative References*	**Specification**	**Mustavaara Project**
ESIA/EIA process		
The coordination authority in this project is North Ostrobothnia Centre for Economic Development, Transport, and the Environment (ELY Centre). *EIA Act (252/2017) and the EIA Decree (277/2017).*	An ESIA (environmental and social impact assessment) procedure in Finland is required prior to the permitting. The procedure includes the ESIA programme stage and the actual ESIA. The purpose of the ESIA procedure is to assess the environmental and social impacts of project alternatives but also to share information and add interaction with different stakeholders. An ESIA does not lead to a permit decision, but a finished and approved ESIA process enables the submission of an environmental permit application.	Mustavaara mining project ESIA process started 25th of May 2008 and ended 18th of January 2010. Capacity: Ore extraction 3 Mt and 6 Mt waste rock in year. Smelter project ESIA process started 4th of December 2012 and ended 20th of December 2016. Capacity/products: ferrovanadium 6 700 t and pig iron 60 000 t and recycled sludges 288 000 t in the year.
Environmental permit and water permit		
The relevant permitting authority is Northern Finland Regional Administrative Agency (PSAVI). *Environmental Protection Act (527/2014) and*	In Finland, environmental permits are required for all activities involving the risk of pollution of air and water or contamination of soil. Permit decisions can be appealed to the Administrative Court of Vaasa and subsequently to the Supreme Administrative Court.	The environmental and water permit for Mustavaaran Kaivos Oy Mustavaara proposed mine (32/2016/1) was issued on the 16th of March 2016 by PSAVI. The decision was appealed on June 14th, 2018, and the Vaasa administrative court ruled against the appeal. It is possible to transfer environmental and water permit to new mine operator. If construction at Mustavaara has not commenced before the

Decree (713/2014) *Water Act (587/2011) and the Water Decree (1560/2011)*	Water permits are required when the planned operation or activity may alter the position of the groundwater table or the groundwater quality. Also, changes related to the water flow or shoreline are subject to authorization. Water permit application and handling is normally integrated into the environmental permitting process	14th of July 2022, the water permit and parts of the environmental permit will expire. If construction has not started before the 14th of July 2023, the environmental permit could be discontinued. It is possible to apply for an extension of up to 3 years as according to the Environmental Act 91 § to permits water permit parts. It is recommended that the mine project operator has a meeting with the supervising and permitting authorities before any actions. Permit extension schedule and needed actions can be planned after meeting. Capacity in the permit is 12.25 Mt/a, combined ore and waster rock extraction. The smelter doesn't yet have any environmental permits.
Mineral rights		
Mineral rights, including decisions concerning mining permits, are regulated under *the Mining Act (621/2011)*, and Tukes, the Finnish Safety and Chemicals Agency, is the responsible authority.	Establishment of a mine and undertaking of mining activity are subject to a permit (mining permit). Permit application requires information concerning the mineral deposit, planned operations, feasibility and finance.	Strategic Explorations Oy (subsidiary) has three valid exploration reservations (valid until February 9th, 2022). Exploration reservation means priority for applying an exploration permit.
Land Use Planning and Building Permits		
Land Use and Building Act (132/1999) and Land Use and Building Decree (895/1999)	Land use planning in Finland normally takes place on three different levels: • Regional Planning on the state or regional level • Master planning on the municipal level • Detailed planning on the local level	• Regional plan for Mustavaara area has been approved by the Council of Oulu Region. Smelter location is within the area designated for industrial operations. • Master plan for Mustavaara area has been approved by Taivalkoski and Posio Municipality (2017).

	Land use planning procedures are needed to obtain building permits for project infrastructure, but also actual operations require land-use designation(s) that allows mining/industry.	• Master plan for the smelter has been approved by the city of Raahe. • Detailed plan for Mustavaara area has been approved by Taivalkoski Municipality. • Detailed plan for the smelter has been approved by the City of Raahe (2017). Building permit for ore processing facilities, smelter and other significant buildings must be applied based on the detailed local plans.
Mine powerline building permit		
Based on the *Act on Electric Market, 18 §,* the Energy Market Authority grants building permits for constructing power lines for 110 kV and higher.	The permit application shall include sufficient environmental baseline descriptions and impact assessments. Typically, this capacity of power lines do not require a separate EIA procedure but this needs to be confirmed with the local ELY prior to submitting the building permit application. An important aspect of the permit is that the granted permit only justifies the building works. Rights for land use under the power line route has to be applied for in a separate process with the landowner(s). This can involve redemption of land or agreement for right to use the land. A granted permit is typically valid for 5 years. If the power line is not finished within the timeframe the permit must be renewed.	The power line from the historic mining operation has been decommissioned but the route reservation still exists in the regional land use plans of Western Lapland and Northern Ostrobothnia (Länsi-Lappi and Pohjois-Pohjanmaa). This remarkably simplifies the permitting procedure as there is no need for further land use planning.
Other permits		
Additional deviation permits according to the Nature Protection Act or Natura 2000 assessments may be required in case of the presence of sensitive species or habitats. Dam safety is regulated by the Dam Safety Act (2009/494); the building permit for a dam requires a statement from the dam safety authority KAIELY (Kainuu Centre for Economic Development, Transport, and the Environment), risk assessment and a dam safety monitoring plan. A negotiation duty concerning reindeer herding is implied when the project		

impacts on government-owned land within the reindeer-keeping area (Reindeer Keeping Act 848/1990).
Tukes handles also applications for the utilization and storage of industrial chemicals according to the Industrial Chemical Decree (59/1999), and it is the authority for chemical registrations, labeling, and packing according to the REACH-decree and CLP-decree. The use and storage of explosives, lifting equipment, and electrical work require permits from Tukes.

Table 20-4. Upcoming Changes in Regulation

Legislative	**Update/Change**	**Schedule**	**Notes**
the Mining Act (621/2011)	The duration of the exploration permit, environmental protection, taxation and collateral issues.	Probably completion 2022.	Change process delayed.
the Environmental Protection Act (527/2014)	One of the upcoming legislation changes is update of Environmental Protection Act and new decree concerning better utilization of excess soils coming from construction sites. Also, there is under development a secondary responsibility system for environmental damages for situations where initial responsible is in bankrupt or unknown.	Complete in the year 2022. Complete in the year 2021.	Developments in Finnish: Hallituksen esitys laiksi ympäristönsuojelulain muuttamisesta ja Valtioneuvoston asetus rakentamisen maa-ainesten hyödyntämisestä ja Ympäristövahinkojen toissijaisten vastuujärjestelmien kehittämisen lainsäädäntöhanke (TOVA-lainsäädäntöhanke)
the Nature Conservation Act (1096/1996)	The aim is to update the legislation technically and to clarify the Act. Also, natural resources will be examined so that the legislation would prevent the fragmentation of habitats more effectively	To be completed in 2020.	
the Land Use and Building Act (132/1999)	The aim is to simplify the land use planning system, support citizens' opportunities to influence the planning and decision-making processes in their own living environment.	To be completed in 2022.	

The impacts of the reforms concerning mining regulation are still uncertain. More precise information on the content and impacts of the reforms will be available in 2021 at the earliest. It is possible that the changes in taxation, nature conservation, collateral, and environmental permitting will increase the costs of the mining industry. Land use planning processes may become more agile, but it would also require better resourcing of administrative courts dealing with appeal procedures.

20.4 Good Practice to be Applied

20.4.1 Best Available Techniques (BAT)

Most of the Best Available Techniques reference documents are directed to the Industrial Emissions Directive (IE-directive) plants. According to directive 2010/75/EU (annex 1), the IE-directive is not applied on mining, but it is applied on smelters. Therefore, most BAT-references mentioned below are used only in the smelter-context.

At the Mustavaara project, the following documents should take into account:

- Reference Document on the General Principles of Monitoring (2003)

Mine:

- Best Available Techniques Reference Document for the Management of Waste from Extractive Industries (2018)

Smelter:

- Best Available Techniques (BAT) Reference Document for Iron and Steel Production (2012)
- Best Available Techniques (BAT) Reference Document for the Non-Ferrous Metals Industries (2017)
- Reference Document on Best Available Techniques on Emission from Storage (2006)
- Reference Document on the application of Best Available Techniques to Industrial Cooling Systems (2001)
- Reference Document on Best Available Techniques for Energy Efficiency (2009)

20.4.2 ICMM Good Practice in Mine Closure

The International Council on Mining and Metals (ICMM) has launched a guide for Integrated Mine Closure in 2019. Integrated mine closure is a dynamic and iterative process that takes into account environmental, social and economic considerations at an early stage of mine development. Closure planning is cyclic, as information is updated from different sources. Environmental impacts and residual risks of the suggested mine closure approach must be on an acceptable level. In the end, the monitoring is the key to make sure that the closure goal is achieved.

20.5 Site specific Environmental Requirements for Waste Management and Monitoring

20.5.1 Mustavaara Mine Site

According to extractive waste characterisation (Pöyry 2012) the dominating minerals in (old, already deposited) tailings are plagioclase, actinolite, hornblende and epidote. Other minerals present are chlorite, quartz, augite, ilmenite and biotite. There are also very small quantities of for example apatite and sulphides. Most distinctive metal concentration present in the tailings is copper.

Assumption is that majority of metals/metalloids in tailings are in silicates or oxides and therefore not especially mobile. No detailed assessment is available concerning the presence of elements in different minerals. In the reviewed documents, leaching tests include only deionized water leach. Therefore, the details concerning the forms of metal/metalloid presence are not yet fully confirmed.

According to extractive waste characterisation (Pöyry 2012) the dominating rock types in (old, already deposited) waste rock are anorthosite and anorthosite-gabbro. In hanging wall there are small quantities of magnetite. Most significant minerals are plagioclase, augite, epidote, uralite and chlorite. Sulphide concentration is low (0.04%) and therefore metal leaching related to sulphide oxidation is likely to be very low.

Water monitoring from August 2012 was reviewed as an example. There are slightly elevated concentrations of some elements, mostly copper and antimony in Sirniönlampi lake near the pit, Sirniönjoki river (starting from Sirniönlampi lake) and Lavotjoki river (downstream from TSF). There are also elevated concentrations of these elements in pit water and TSF water. Sources of metal leaching are probably tailings and pit walls and in some extent waste rock.

No data related to presence of potential fibrous or radioactive minerals has been reviewed.

Receptor watercourses are very small and therefore relatively easy to impact. Nutrition and solids loads to watercourses are potential risks in this type of environments. Nitrogen load from explosives use is an issue to be taken to consideration at the next study steps. There is some phosphorus-containing extractive waste material on the site already, but that is not strongly reflected in the site waters or the receptors (in 2012 monitoring). If the mine opens again, increasing extractive waste quantities and mine dewatering can generate increasing load of for example nutrition and copper. During the next study phases, more detailed water quality assessments and load predictions are recommended. These are needed as inputs for site load balance and for assessment of water treatment requirements. Monitoring program should reflect these identified geochemical risks.

20.5.2 Raahe Smelter

The smelter site location is within the Raahe port and industrial area. The site is located on an earth-fill area next to the deep-water harbour, surrounded by sea water (Gulf of Bothnia).

There are several operators in the nearby area that have cumulative environmental impacts: the SSAB steel factory and the sewage treatment plant of Raahe. Also, the discharge pipe from the Laivakangas mine (currently not operational) has its outlet in the area.

Water impacts are mitigated by maximized process water recycling and chemical water treatment. Sulfate load (marine) is reduced by production of sodium sulphate.

Key factors in the discharge are solids, vanadium, sodium, chloride, nitrogen (ammonium-N), sulphate, nickel, lead, cadmium, chrome, arsenic, zinc and copper.

There are prelimianry plans for the smelter waste depostion. Deposition is planned to take place on the earth fill land areas (sea shore fill).

20.6 Project Alternatives

Processing alternatives and the scale of operations have been studied at the different development phases. The operation of the mine is closely linked to the further processing of metals.

During the EIA-program phase, the cities Oulu and Raahe were considered for the location of smelter. After EIA-program, the location of Raahe was considered to be more favourable, and in EIA-report only Raahe was included.

20.7 Management Approach

The environmental, social, and governance policies shall be developed and implemented as the project proceeds towards permitting, development and construction. The focus must be on managing the identified key environmental and social issues.

20.8 Stakeholder Issues and Stakeholder Engagement

20.8.1 Stakeholder Engagement

Stakeholder engagement is the process of involving people or parties who may be affected by the mining and smelter operations. Stakeholder engagement and participation are essential parts of the EIA and permitting procedure. Thorough management of the stakeholder issues and active communication are fundamental in promoting the acceptance of the project among the local residents and other interest groups. During the EIA-phase, all the essential stakeholders have been identified: municipalities, reindeer herders, residents of the closure villages/towns, groundwater users and fishing communities.

20.8.2 Stakeholder Concerns

In the Mustavaara mine site environmental permitting process following issues have been the main concerns of the various stakeholder groups:

- Impacts in surface water quality
- Impacts on fishing
- Impacts on groundwater supply of nearest villages/houses

20.8.3 Contracts and Compensations

No valid agreement with a reindeer herding cooperative is in place currently. Fish compensation is included to the current mine site environmental permit conditions (permit condition 70). Fish compensation is included to the OPEX.

20.8.4 Community Health and Safety

Community health and safety issues in Mustavaara mine site area are largely related to increasing traffic. The main community health and safety concerns (presented by the stakeholders) are related to water and groundwater quality, including the drinking water supply. Also, in the smelter site traffic would increase, but generally the area is already an industrial area. Additional impacts on air and sea water will have to be more closely discussed in the next study phases, from community perspective.

20.9 Labour Conditions

In the Finnish labour market, the organisation level is high on both the employee and the employer side. Collective bargaining has a relatively important role in labour regulation. The basis of the regulation is in comprehensive and detailed labour legislation. Employment legislation covers for example contract issues, probation, severance pay, notice, hours of work, paid leave, maternity leave and maternity protection, sick leave, minimum age and protection of young workers, equality and trade union freedom.

20.10 Resource Efficiency

The resource efficiency in mining industry means maximising the value from an ore deposit and simultaneously reducing impacts and emissions. Legislation and BAT conclusions are used to ensure that resource efficiency is taken into account in the Mustavaara project planning.

Following objectives have been communicated in the reviewed material:

- The project will adopt new equipment and technologies to decrease energy consumption. The process will be optimized so that the ore recovery is efficient.
- The target is to minimize the amount of consumed fresh water. If possible, mine water is used in the process. Water management pumps are optimized for the best pump efficiency to reduce energy consumption and pumping costs. If the flow rates are constantly below the dimensioning values, pump efficiency decreases.

- Peat and till are separated from the waste rock so it can be used in the mine closure for the rehabilitation of waste rock storage facility. Waste rock is used for the dam and road construction if possible. The amount of waste rock is minimized through mine planning.

An energy efficiency study has not yet been carried out. For example, mine site internal transports and concentrate transport from mine to the smelter are key parts to be studied. Also, part of the concentrator and smelter processes are rather energy-demanding and key parts for future energy efficiency assessments.

Permit Condition 58 of the current environmental permit (mine site) requires taking energy efficiency into consideration in choice of machinery and equipment. Measures to support energy efficiency must be reported in environmental protection annual reporting.

20.11 Monitoring

Monitoring includes, discharge, water and groundwater quality monitoring (and groundwater level monitoring). Watercourse impact monitoring includes also monitoring of fish, macroinvertebrates, diatoms, phytoplankton and water moss. Air and terrestrial monitoring includes dust fallout, sediment studies, terrestrial biology and noise. Also, extractive waste quality monitoring is carried out. Monitoring costs are included to the PEA OPEX and post-closure monitoring into the financial guarantee (CAPEX).

20.12 Mine Closure

20.12.1 Existing Mine Closure Plans

According to the current permit conditions, tailings area cover must be 0.5 m and waste rock area cover 0.3 m. Cover requirements have not been re-evaluated. Current data set does not enable detailed post-closure source term and impact assessments to confirm the adequacy of the planned closure measures. On the other hand, according to the available information sulphide content in Mustavaara extractive wastes is rather low and for example prevention of oxidation is not likely to be necessary.

For the purposes of the closure costs assessments in this PEA it has been assumed that also clarification pond will require cover material. To provide stability, cover material is assumed to be mix of use of waste rock and local moraine. Another assumption is that overburden stored on the site is enough to provide all the required over materials. End of operation moraine mass balance has not been generated to confirm this assumption. If moraine has to be extracted further away from the mine site, cover material transport costs may increase and add to the closure costs. Potential pit re-sloping (above and near the final water surface) is not included to the current closure cost assessment.

20.12.2 Updating Mine Closure Plan

Using only high-level closure assessments at PEA stage is a common practise. The following study phases should include an iterative closure process. In such process information basis is systematically developed to reduce critical uncertainties and to enable gradual improvement of the closure plan. This cycle starts in early project development stages and continues over the whole life cycle of the mine. The closure plan needs to be confirmed by assessment of post closure impacts and assessment of closure and post closure risks. If impacts (or risks) are at an unacceptable level, review and partial re-planning of closure measures is required.

20.12.3 Mustavaara Post-closure Conceptualisation (outlines)

Key objects remaining on the site after closure are the extractive waste facilities and pit (pit lake). Infrastructure is largely demolished, but road network serves later land-uses.

Open pit will be filled with water and sloped to a safe angle above the water table (and below water table near surface). Salinity stratification is assumed to take place in the pit lake and the primary contact with environment would probably take place via the upper parts of the pit lakes as according to the RQD-data most of the narrow fractured zones are located near surface and no large fracture zones are found.

Site infrastructure will be demolished (unless subsequent use is discovered), but roads will remain to serve post-closure land-uses, like forestry and reindeer-keeping. Water management and treatment systems from operational time serve until active closure implementation is completed.

Post closure seepage flow direction from the pit area is towards Sirniönlampi lake and Sirniönjoki river. Tailings storage facility flow direction is towards Lavotjoki river. Further downstream the post closure receptor is Unijoki river.

20.12.4 Financial Guarantee Requirements

According to the existing environmental permit the operator must set 372,000 € deposit before mining begins for discharging, treatment, monitoring and to maintain these operations for contaminated waters. Also 3,385,200 € guarantee deposit is needed for current wide tailings area closure. These costs are based on currently permitted waste facility dimensions and unit costs. As 20 years of operation will require larger waste facility capacities, also guarantee will be reassessed in amendment permitting. Concerning financial guarantee, also VAT (24%) must be added to the costs. Covering and sloping costs (Table 20-5) and units costs (Table 20-6) are presented below. In addition to these costs, financial guarantee includes monitoring and maintenance costs. These are taken into consideration in the PEA economic model. Post closure monitoring and maintenance costs for mine site are 19,680 €/a for 30 years after closure.

Table 20-5. Cover and sloping costs (PEA 2020), excluding VAT (24%)

	Shaping, €	Cover, €	Total, €
Tailings	616,875	8,225,000	8,841,875
Sediment pond		2,500,000	2,500,000
WRD		3,555,000	3,555,000
WRD top	465 000		465,000
WRD slopes	2 376 000		2,376,000
Total			17,737,875

Table 20-6. Cover and sloping unit costs (PEA 2020), VAT (24%)

Unit costs	€	unit
shaping the surface WRD	0.5	m^2
shaping the surface TSF*	0.75	m^2
moraine from the site	5	m^3
moraine or waste rock from the site	5	m^3

20.13 Environmental and Social Risks to the Project

20.13.1 Identification of Environmental and Social Project Risks

Key environmental and social project risks are related to current environmental permit conditions and nature values. Environmental and social risks are presented below (Table 20-7).

As the project risk assessment below forms the “looking inwards” risk assessment approach, also “looking outwards” risk assessment approach is a general requirement the mining and metals industry in Finland. Mustavaara mine site environmental permit condition 57 requires environmental risk assessment and updating the assessment. Authority has to approve the risk assessment professionals selected by the company.

Table 20-7. Environmental and social project risks.

Risk	Recommended mitigation
Mine site: **Production and waste capacities in permit decision 32/2016/1 PS-AVI).** Current Mustavaara mine site environmental permit is issued for 15 years of production waste quantities and related waste facility surface areas. Amendment permit handling is needed. This is assumed to trigger also a new ESIA procedure.	Early initiation of ESIA and amendment permitting handling. This includes early initiation of technical planning work and required environmental assessments (for example source term assessments, nature and water impact assessments and closure planning with post-closure impact assessments).

Risk: delay in start of the operations (or start of operations without permit for the whole LOM, notice also another linked risk below)	
Mine site: **Permit conditions 8 and 9, current environmental and water permit (permit decision 32/2016/1 PS-AVI).** • Permit condition 8 stipulates that due to nesting sites of a protected species, no constructions or changes in natural conditions are allowed within 400 m radius from protected bird nesting sites (3 sites, south-west from planned operations). • According to the permit condition 9, during the potential nesting season (15th February – 31st July), no constructions, car transports, noise or emissions are allowed within 1000 m radius from the nest sites. Working within the planned operation area under these permit conditions is rather challenging. At least amendment permit and potentially also significant change of tailings deposition plans are needed. **Risk:** Project delay due to re-permitting or new planning requirements that add costs.	In the current plans mitigation measure is amendment permitting to change the existing permit conditions. This approach was selected before the knowledge of nest site becoming active again was gained. Possibilities for new permit conditions via amendment permitting can still be explored, but challenges have increased due to this knowledge. Especially permit condition 9 requires changing, as for example dam safety monitoring and waste facility operations in general require vehicle access to the area not allowed during the nesting period. Compensation measures like artificial nests and satellite monitoring of these birds could be proposed as a part of the amendment permitting approach. It might be possible to operate the mine site at least over the early years despite permit condition 8, but whole LOM operation likely requires alterations in tailings storage facility plans. Advisory discussion with permit and control authorities is recommended, concerning the approach to be selected. The most extreme mitigation measure would be finding a completely new site or selecting another deposition method for tailings storage (in order to survive with a smaller footprint).
Mine site: **Ecological status of nearby lakes and rivers.** EU member states are required to refuse environmental permits for any project that may cause deterioration of the status of a water body or jeopardize the attainment of its status objectives. Expansion of project (new LOM waste quantities etc.) cause impacts that conflict with the status objectives. **Risk:** Permitting difficulties (permit denied or delayed) or need for additional mitigation measures (costs).	Careful water impact assessment is needed, taking into consideration the increase in LOM waste quantities. Possible additional water treatment or new discharge point in larger river may be required.

Mine site: **Leathery grape-fern (Sceptridium multifidum)** This plant is Near Threatened (NT) and Regionally Threatened (RT). **Risk:** Expansion of project (new LOM waste quantities etc.) impact negatively on the protected species.	To be taken care in sites layout and access road planning. Possible transplant plan.
Mine site: **Data gaps concerning waste characterization and extractive waste facility source terms.** Planned basal and cover structures will have to be reviewed after complementary waste characterization data and source term assessments (see chapter 20.5.1) are available. This issue may also materialize as poorer water quality and larger loads than previously assessed. **Risk:** re-planning basal and cover structures is needed, potential additional cost. Another potential risk is additional water management costs.	Initiation of complementary waste characterization and source term assessments to serve the nest feasibility study phases. Complementary information is needed about the form of presence of critical elements and waste long term behavior. New source term assessments are needed after complementary characterization. On the other hand, source term assessment update is always needed when waste quantities or waste facility dimensions are changed.
Mine site: **Mine dewatering drawdown impact on surrounding area groundwater levels.** This may have impacts on survival of habitats or species with protection value. (Uncertainty concerning previous hydrogeological assessment methods, but also new pit dimensions.) **Risk:** permitting difficulties or delay	Hydrogeological testing and dewatering /drawdown assessment is recommended to increase understanding on the risk. If necessary, mitigation measures can be studies if relevance of the impact or major risk becomes confirmed.
Smelter: **Poorer quality of the discharge water in case of no market for the sodium sulphate product.** **Risk:** permitting challenges (delay, mitigation costs)	Risk assessments concerning the side product markets and alternative water treatment studies in case of higher sulphate concentrations. To be included to the trade-off studies at the next study phases.
Smelter: Safety and environmental risks related to rising of the Sea level, and waves.	Location of critic items. Risks taken into cognizance in planning of the base structures and dams. No direct access for swells to the dam structures.

20.13.2 Recommendations for the Next Study Phases

Following issues are recommended to be taken especially well into consideration during the next study phases:

- Preparation for amendment permitting concerning the longer LOM /larger extractive waste quantity. Amendment permitting requires geochemical assessments (points below) and technical planning. These would be convenient to carry out in pre-feasibility context.
- Investigating potential for changing permit conditions 8 and 9. This includes advisory meeting with permitting and control authorities PS-AVI and PP-ELY. Alternative sites /deposition technics for tailings storage may also have to be investigated, to avoid conflicting with requirements related to protected bird species nesting sites.
- Complementary waste characterizations (long-term behavior, form of presence for harmful elements. For example, sequential leaching can be a relevant alternative, if including deionized water leach, salt leach (BaCl), hydroxyl amyl leach and hydrogen peroxide leach ("NAG-leachate").
- Source term assessments for new extractive waste facility dimensions (and waste quantities). Source term assessments should take into consideration also the long-term behavior of the waste and field liquid/solid ratio. Previous water quality and load assessments ore based on deionized water leach and laboratory liquid/solid – ratio.
- Hydrogeological testing and dewatering /drawdown assessment are recommended to increase understanding on the risk of drawdown impacts on habitats/flora/fauna.
- More detailed investigation of side product markets and preparation of plan B for management of smelter sulphate loads.

Generally environmental impacts are assessed for a smaller version of the projects that is discussed in this PEA. This means updated impact assessments according to the new scale of the project are needed.

21 Capital and Operating Costs

21.1 Capital costs

21.1.1 Basis of estimate

The capital cost estimate was prepared by AFRY with expected accuracy range of ±50% which is in AACE Class 4 range. Base pricing is in the fourth quarter of 2020 Euros. No allowances for escalation or inflation beyond this time have been applied.

The estimate includes direct and indirect costs such as EPCM (Engineering, Procurement and Construction Management) and construction period temporary facilities and services as well as owner's cost and contingency for overall project.

The estimate covers following areas:

- Mine (pre-development and supporting infrastructure)
- Beneficiation plant (3.25 Mtpa material input)
- Tailings storage facility (TSF)
- Waste rock storage facility (WRSF)
- Smelting plant (505 ktpa concentrate input)
- On-site infrastructure for beneficiation and smelting plants

Following engineering material prepared during the study was used as a basis for estimate development:

- Conceptual mine, beneficiation and smelting plant design criterion
- Conceptual process flowsheets
- Conceptual mine plan
- Conceptual earthworks quantities derived from preliminary layouts
- Conceptual general site layout

Additionally, cost data from previous Mustavaara-related projects has been utilized in high proportions. Previous projects include pre-feasibility study conducted by Pöyry in 2012/2013 ("PFS 2013") and Ferrovan Early works study from 2018/2019 ("Ferrovan"). Mine and beneficiation plant costs are derived from PFS 2013 and smelting plant costs from Ferrovan. Smelting plant main equipment costs were factored with a rule of six tenths based on the PEA capacity requirements. Original main equipment prices are based on vendor budgetary and firm quotations.

Costs derived from previous projects were assessed by AFRY and a conclusion was established that, for PEA purposes, costs for items in the scope are on acceptable level after applying an escalation and updating the costs of main equipment.

Cost items from previous studies are mostly used as-is and therefore details behind cost estimate summary are on more detailed level that should be used in PEA.

Following assumptions were considered:

- All equipment and material will be new
- Implementation work will be continuous
- Project will be executed through EPCM (Engineering, procurement, construction management) contract.

Following costs are excluded from the estimate:

- land acquisition
- financing costs and interest during construction
- exchange rate fluctuations
- changes in legislation
- insurances
- working capital (included in cashflow model)
- pre-operating general expenses
- changes in design criteria or scope

21.1.2 Labour assumptions

As most of the costs are derived from previous studies, the same methodology is used in this PEA. Construction labour costs are included in the material pricing. Other installation activities are factored from equipment and material prices.

21.1.3 Material costs

All materials required for construction are included in the capital cost estimate. Materials costs include freight and installation. Material quantities and pricing is based on previous studies.

Dams for TSF and water ponds were re-evaluated and priced according to prepared material take-offs.

21.1.4 Contingency

Flat contingency of 10% was applied for the whole project. This contingency was used in 2013 PFS and Ferrovan. Subsequently, as engineering was more matured in these studies, using same level of contingency was considered applicable in this PEA.

21.1.5 Capital costs summary

The total estimated capital cost for the project is approximately 691 MEUR (million Euros). Development capital (years -3 to -1) is approximately 597 MEUR and sustaining capital is approximately 94 MEUR.

Development capital costs are split to mine, beneficiation plant, their respective infra and utilities and smelting plant as follows:

Table 21-1. Capital cost summary.

Capital costs summary	Cost (million euros)
Direct costs	
- Mine	28.1
- Beneficiation plant	80.5
- Mine and ben. plant infra & utilities	43.0
- Smelting plant	321.3
Direct costs total	**473.0**
Indirect costs	
- Mine & beneficiation plant	21.8
- Smelting plant	47.7
Indirect costs total	**69.5**
Development capital contingency	
- 10% contingency	54.2
Development capital total	**596.7**
Sustaining capital costs	94.2
Capital costs total	**690.9**

Indirect costs include EPCM, construction period temporary facilities and services and owner's costs. Percentage factors used are based on AFRY's experience from similar projects. Smelting plant indirect costs are based on assessment conducted in Ferrovan.

Sustaining capital costs include mine site overburden removal, mining equipment purchases, dam enhancements and closure costs. According to Finnish legislation, a security needs to be deposited for purpose of covering the closure costs should the operating company be unable to conduct required closure activities. Approximately 22 MEUR is allocated for the closure costs in sustaining capital. No salvage credits were included. Security deposit is covered in cashflow model for this same amount split equally for LOM. For sake of clarity, deposit payback is allocated in full for year 21.

21.2 Operating Cost Estimate

21.2.1 General

Operating cost estimate is based on following assumptions:

- Mining and concentrator plant operations are based in Mustavaara
- Smelting and hydrometallurgical operations (ferrovanadium production) based in Raahe
- Concentrate is transported from Mustavaara to Raahe by truck
- Consumables transportation to Mustavaara/Raahe sites by truck
- Opex estimates are based on 3.25 Mtpa ore per year, 505 ktpa concentrate per year, 4,577 tpa ferrovanadium per year and 329 ktpa pig iron per year.

Operating cost has been structured to the following five cost centers:

- Administration
- Mining

- Concentration
- Smelting & hydrometallurgical plant
- Logistics

Operating cost is calculated as an annual average, without taking account the annual variations during the life of mine.

21.2.2 Total Operating Cost

Total operating cost estimate per cost center is presented in Table 21-2. Total cash cost includes royalties and income from by-products. Total all-in sustaining cash cost includes royalties, income from by-products, sustaining capex and closure cost.

Table 21-2 Operating cost summary.

OPERATING COST SUMMARY			
			Opex (Eur/a)
Administration (Head office)		**Eur/a**	**-4 022 429**
Mining costs		**Eur/a**	**-19 135 209**
Concentration costs		**Eur/a**	**-12 234 718**
Smelting & Hydromet. Plant costs		**Eur/a**	**-96 196 185**
Logistic costs		**Eur/a**	**-8 853 401**
Other costs	Exploration costs (for LOM)	**Eur/a**	**0**
	Sust. capex & closure cost	**Eur/a**	**-4 653 193**
	Royalties	**Eur/a**	**-59 056**
By-product income	Total	**Eur/a**	**+6 425 054**
Total Opex		**Eur/a**	**-140 441 941**
Total Opex (to concentrate)		**Eur/a**	**-44 245 756**
Total Cash Cost, incl. royalties and by-product income		**Eur/a**	**-134 075 943**
Total All-In Sustaining Cash Cost), incl. royalties, by-product income, sust. capex and closure cost		**Eur/a**	**-138 729 136**

Total operating cost is 140.4 MEur/a (without sustaining capex). Total cash cost including royalties and by-product income is 134.1 MEur/a. All-in sustaining cash cost including royalties, by-product income, sustaining capex and closure cost is 138.7 MEur/a. Total operating cost for concentrate production is 44.2 MEur/a.

Smelting plant is the largest contributor to operating costs and represents 68.5% of total operating cost. Second largest cost center is mining with 13.6% share. Operating cost split to cost centers is presented in below (

Table 21-3).

Table 21-3 Operating cost split to cost centers.

Cost Center	Opex Eur/a	Share of Total Opex (%)
Administration (Head office)	-4 022 429	2.9
Mining costs	-19 135 209	13.6
Concentration costs	-12 234 718	8.7
Smelting & hydromet. costs	-96 196 185	68.5
Logistic costs	-8 853 401	6.3
Total Opex (Eur/a)	-140 441 941	100.0

21.2.3 Operating Cost per Unit

Operating cost structure per unit (kg or metric tonne) ore, concentrate, ferrovanadium and pig iron tonnes is presented in

Table 21-4. Opex allocation to ferrovanadium and pig iron products is 50% for each. Opex is calculated per kg of FeV80 and per metric tonne of pig iron, concentrate and ore. Cash cost per unit of production includes royalties and income from by-products. All-in sustaining cash cost per unit of production includes royalties, income from by-products, sustaining capex and closure cost.

Table 21-4 Operating cost per unit of Ore, Concentrate, Ferrovanadium and Pig Iron.

Opex	**Eur/unit**	
per FeV80 (kg)	-15.3	Allocation 50/50 to FeV80 and pig iron
per Pig Iron (metric tonne)	-213.3	Allocation 50/50 to FeV80 and pig iron
per Conc. (metric tonne)	-278.2	
per Ore (metric tonne)	-44.1	
per Conc. (metric tonne, cost to concentrate)	-87.7	
per Ore (metric tonne, cost to concentrate)	-10.0	
Cash Cost (incl. royalties and by-product income)	**Eur/unit**	
per FeV80 (kg)	-14.6	Allocation 50/50 to FeV80 and pig iron
per Pig Iron (metric tonne)	-203.6	Allocation 50/50 to FeV80 and pig iron
per Conc. (metric tonne)	-265.6	
per Ore (metric tonne)	-42.1	
All-In Sustaining Cash Cost (incl. royalties, by-product income, sust. capex and closure cost)	**Eur/unit**	
per FeV80 (kg)	-15.2	Allocation 50/50 to FeV80 and pig iron
per Pig Iron (metric tonne)	-210.7	Allocation 50/50 to FeV80 and pig iron
per Conc. (metric tonne)	-274.8	
per Ore (metric tonne)	-43.5	

Operating cost split to cost centers per unit is presented in .

Table 21-5.

Table 21-5. Operating cost split to cost centers.

Admin Opex	**Eur/t**	
per FeV80 (kg)	-0.44	Allocation 50/50 to FeV80 and pig iron
per Pig Iron (metric tonne)	-6.11	Allocation 50/50 to FeV80 and pig iron
per Concentrate (metric tonne)	-7.97	
per Ore (metric tonne)	-1.26	
Mining Opex	**Eur/t**	
per FeV80 (kg)	-2.09	Allocation 50/50 to FeV80 and pig iron
per Pig Iron (metric tonne)	-29.06	Allocation 50/50 to FeV80 and pig iron
per Concentrate (metric tonne)	-37.91	
per Ore (metric tonne)	-6.00	
Concentrator Plant Opex	**Eur/t**	
per FeV80 (kg)	-1.34	Allocation 50/50 to FeV80 and pig iron
per Pig Iron (metric tonne)	-18.58	Allocation 50/50 to FeV80 and pig iron
per Concentrate (metric tonne)	-24.24	
per Ore (metric tonne)	-3.84	
Smelting & Hydromet. Plant Opex	**Eur/t**	
per FeV80 (kg)	-10.51	Allocation 50/50 to FeV80 and pig iron
per Pig Iron (metric tonne)	-146.07	Allocation 50/50 to FeV80 and pig iron
per Concentrate (metric tonne)	-190.58	
per Ore (metric tonne)	-29.31	
per Ore metric tonne (pig iron)	-0.86	
Logistics Opex	**Eur/t**	
per FeV80 (metric tonne)	-0.97	Allocation 50/50 to FeV80 and pig iron
per Pig Iron (metric tonne)	-13.44	Allocation 50/50 to FeV80 and pig iron
per Concentrate (metric tonne)	-17.54	
per Ore (metric tonne)	-2.78	

21.2.4 Basis of Opex Estimate

- Energy (electricity) price used is 41 Eur/MWh. Price is based on long-term average in Nordic energy market and AFRY reference projects.

- Natural gas price used is based on price from Finnish Energy Authority (Energiavirasto) and is 30 Eur/MWh.
- CO_2 emission cost is based on European carbon market price from Q4/2020 and is 25 Eur/t.
- Chemical and other consumables (grinding media, liners, etc.) costs are based on AFRY database from reference projects.
- Logistics costs are based on distance tables between plants and AFRY data base from reference projects.
- Salary costs are based on AFRY database from reference projects.
- Other costs (insurances, licenses etc.) are based on AFRY reference projects.

21.2.5 Mining Equipment Operating Cost Trade-Off Study

A trade-off study has been made comparing the operating cost of contractor owned mining equipment including personnel with the operating cost of client owned mining equipment and personnel. The trade-off study results are presented in Table 21-6 and

Table 21-7.

Table 21-6 Mining equipment operating cost trade-off for contractor owned equipment and client owned equipment – Part 1/2

Contractor equipment	Ore	Waste	**Own equipment**	Ore	Waste
Year	€/t	€/t	Year	€/t	€/t
1	-	4.34	1	-	2.93
2	2.22	3.29	2	2.16	2.08
3	2.25	3.05	3	1.88	1.93
4	2.28	3.06	4	1.89	1.93
5	2.29	3.08	5	1.90	1.92
6	2.33	3.10	6	2.00	2.01
7	2.36	3.12	7	2.01	2.00
8	2.37	3.12	8	1.98	2.12
9	2.39	3.14	9	1.98	2.11
10	2.44	3.17	10	2.09	2.23
11	2.45	3.19	11	2.10	2.22
12	2.46	3.20	12	2.11	2.22
13	2.50	3.22	13	2.19	2.61
14	2.53	3.23	14	2.20	2.62
15	2.54	3.30	15	2.30	2.77
16	2.55	3.31	16	2.30	2.77
17	2.60	3.32	17	2.30	2.76
18	2.62	3.37	18	2.31	2.76
19	2.64	3.40	19	2.31	2.76
20	2.70	3.48	20	2.42	2.92
21	2.76	3.61	21	2.63	3.20
22	2.82	-	22	4.51	-

Weighted average	2.47	3.23	Weighted average	2.18	2.30

The trade-off study shows a 122M€ reduction in operating costs for client owned mining equipment compared to contractor owned mining equipment (

Table *21-7*). However, this difference will reduce once the capital cost of mining equipment is added to the client owned mining equipment.

Table 21-7 Mining equipment operating cost trade-off for contractor owned equipment and client owned equipment – Part 2/2

Contractor equipment	Ore	Waste	**Own equipment**	Ore	Waste
Year	€	€	Year	€	€
1-22	159 708 943 €	360 908 942 €	1-22	140 878 638 €	257 480 375 €
Total	520 617 885 €		Total	398 359 013 €	

21.2.6 Mining Equipment Capital Cost

The total mining equipment capital cost during the life of mine is presented in Table 21-8. The total capital cost during the life of mine is roughly 65M€.

Table 21-8 Mining equipment capital cost, annual investment and life of mine

Own equipment

Yearly investment	
-1	10 839 641 €
1	7 852 346 €
2	4 559 288 €
3	
4	
5	2 347 387 €
6	711 450 €
7	1 441 691 €
8	3 389 930 €
9	1 576 000 €
10	9 347 950 €
11	2 986 416 €
12	1 516 644 €
13	
14	2 150 013 €
15	5 737 317 €
16	2 287 450 €
17	
18	
19	50 000 €
20	7 072 984 €
21	1 291 691 €

22	
Total investment	65 158 201 €

Taking this into consideration together with the client owned equipment operating cost, owning the equipment results in a total of 57M€ reduction compared to contractor owned mining equipment. On a yearly scale, for a mine life of 22 years, the annual reduction in mining equipment Opex and Capex is roughly 2.59M€.

22 Economic Analysis

The economic analysis contained in this report is based, in part, on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that economic forecasts on which this PEA is based will be realized.

22.1 Methodology Used

Discounted cash flow ("DCF") was used for financial analysis. Net annual cash flows were estimated projecting yearly revenues and subtracting projected yearly outflows such as capital and operating costs, taxes and environmental monitoring expenses. These annual cash flows were discounted back to the first period of capital expenditure and summarized to determine the NPV for the project. Discount rate of 8% was used. Additionally, IRR rate and payback period were estimated. IRR expresses the discount rate that yields NPV of zero. Payback period expresses the time from start of production to time at which all initial and sustaining capital expenditures have been covered.

All monetary values are expressed in Q4/2020 Euros.

22.2 Financial Model Input Parameters

Financial model considers FeV80 and pig iron production from Mustavaara. Mining quantities are based on a conceptual mine plan and production recovery rates based on test work conducted during 2013 PFS.

Revenue generation is mainly based on FeV80 price of 32 USD/kg and pig iron price of 450 USD/metric tonne. In addition, smelter by-products (Ca-Al slag, Ti-slag and $NaSO_4$) are included in the model as revenue generators with prices of 400 EUR/metric tonne, 9 EUR/metric tonne and 150 USD/metric tonne for Ca-Al slag, Ti-slag and $NaSO_4$, respectively. Exchange rate of EUR/USD = 1.18 is used. Products are assumed to be sold on FOB Raahe basis due to immediate vicinity of Port of Raahe to smelting plant. Operating costs (for mining, beneficiation and smelting, including concentrate freight from Mustavaara to Raahe) are derived from PEA OPEX calculation. Development and sustaining capital figures are outlined in chapter 21.1 with the exception of environmental monitoring costs that are included only in the cash flow model.

Development capital has been allocated to years -3 to -1 with split of 20%, 40% and 40%, respectively. Sustaining capital cost allocation is based on previous studies.

A 20% tax rate has been used in the cash flow model (current corporate tax according to Finnish legislation). The corporate tax was lowered from 24.5% to 20% in 2014. Depreciation has been applied according to Finnish law for machinery and property. Accelerated depreciation for machinery has been used. With accelerated depreciation the rate is 50% of the net expenditure of the machinery. For buildings, structures and general project costs other than machinery, a rate of 7% has been used.

The working capital estimate considers payment terms of 60 days for the products sold and 30 days payment terms for operating cash costs.

22.3 Inflation

No escalation or inflation has been applied.

22.4 Closure Costs and Salvage Value

Closure deposit is presented as an annual flat expense. In year 21, closure costs lump sum is consumed and in the same year the annual deposit payments are returned.

Salvage value has not been estimated and is therefore excluded.

22.5 Financing Costs and Interest

Model is based on 100% equity financing. No interest or financing costs are included.

22.6 Economic Analysis

The mine is estimated to have LOM revenue of 5.0 billion Euros using product prices presented in chapter 22.3. Breakdown of revenue is presented in Figure 22-1.

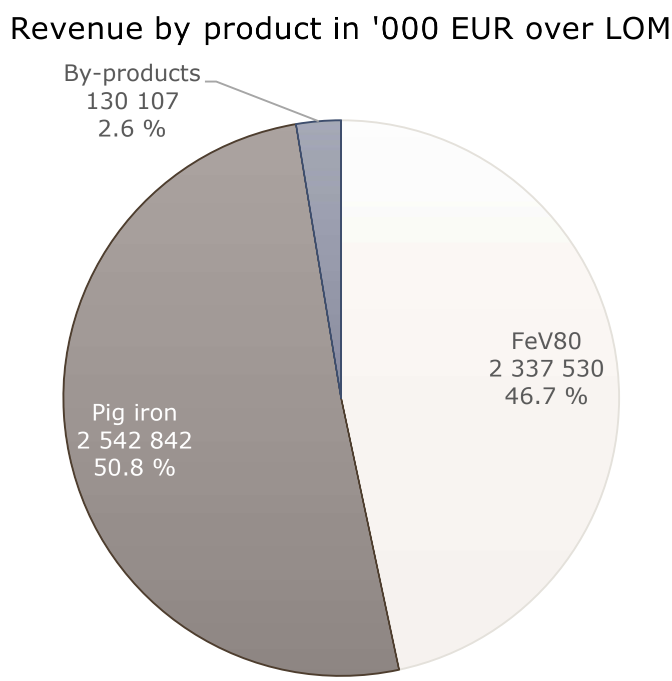

Figure 22-1. Project total revenue.

Total LOM operating costs total to 2.8 billion Euros. Split of operating costs is presented in Figure 22-2. Smelting plant operating costs constitute over half of the total operating costs.

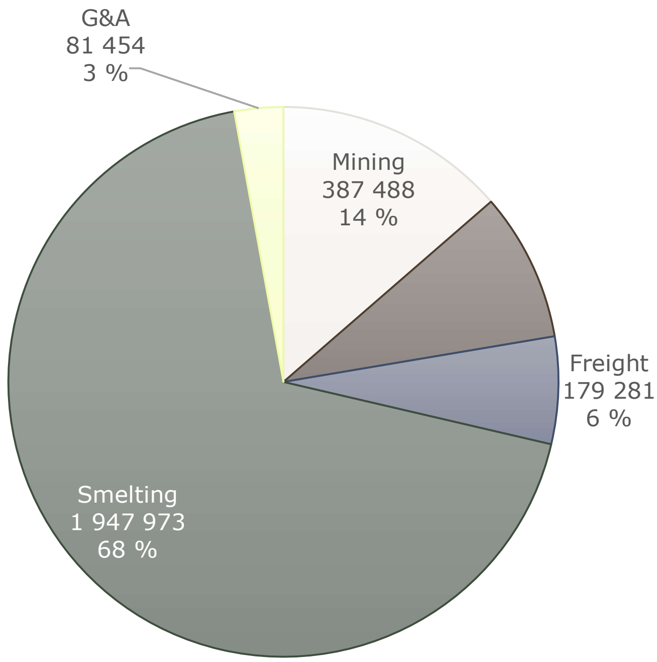

Figure 22-2. Total operating costs.

PEA cash flow model estimates following financial metrics:

Pre-tax values	**Value**	**Unit**
Free cash flow	1 474 405	‘000 EUR
NPV (at 8%)	286 416	‘000 EUR
Payback period	5.9	Years
IRR	13.9	%
Post-tax values		
Free cash flow	1 177 198	‘000 EUR
NPV (at 8%)	189 812	‘000 EUR
Payback period	6.4	Years
IRR	12.2	%

Cash flow profiles showing pre-tax and post-tax cash flows and cumulative cash flows are presented below (Figure 22-2 and Figure 22-3).

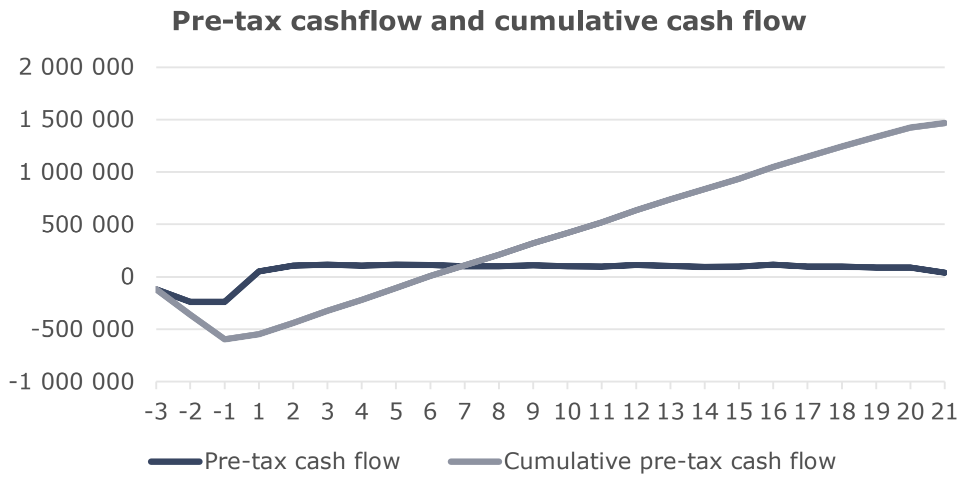

Figure 22-3. Pre-tax cash flow.

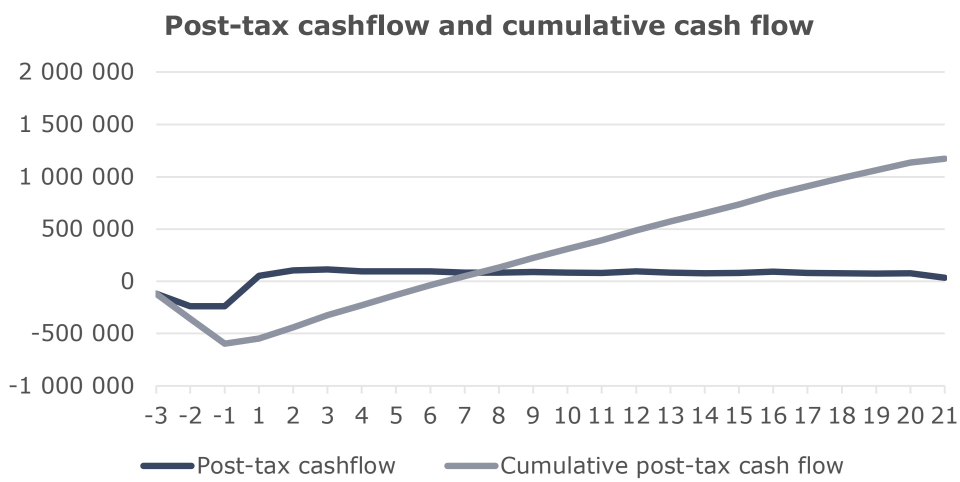

Figure 22-4. Post-tax cash flow.

Full LOM cash flow model is presented in appendix 5.

22.7 Sensitivity Analysis

Sensitivity analysis was conducted on post-tax NPV(8%) with certain opex items, main product prices and development capital. Sensitivity graph is presented in Figure 22-5.

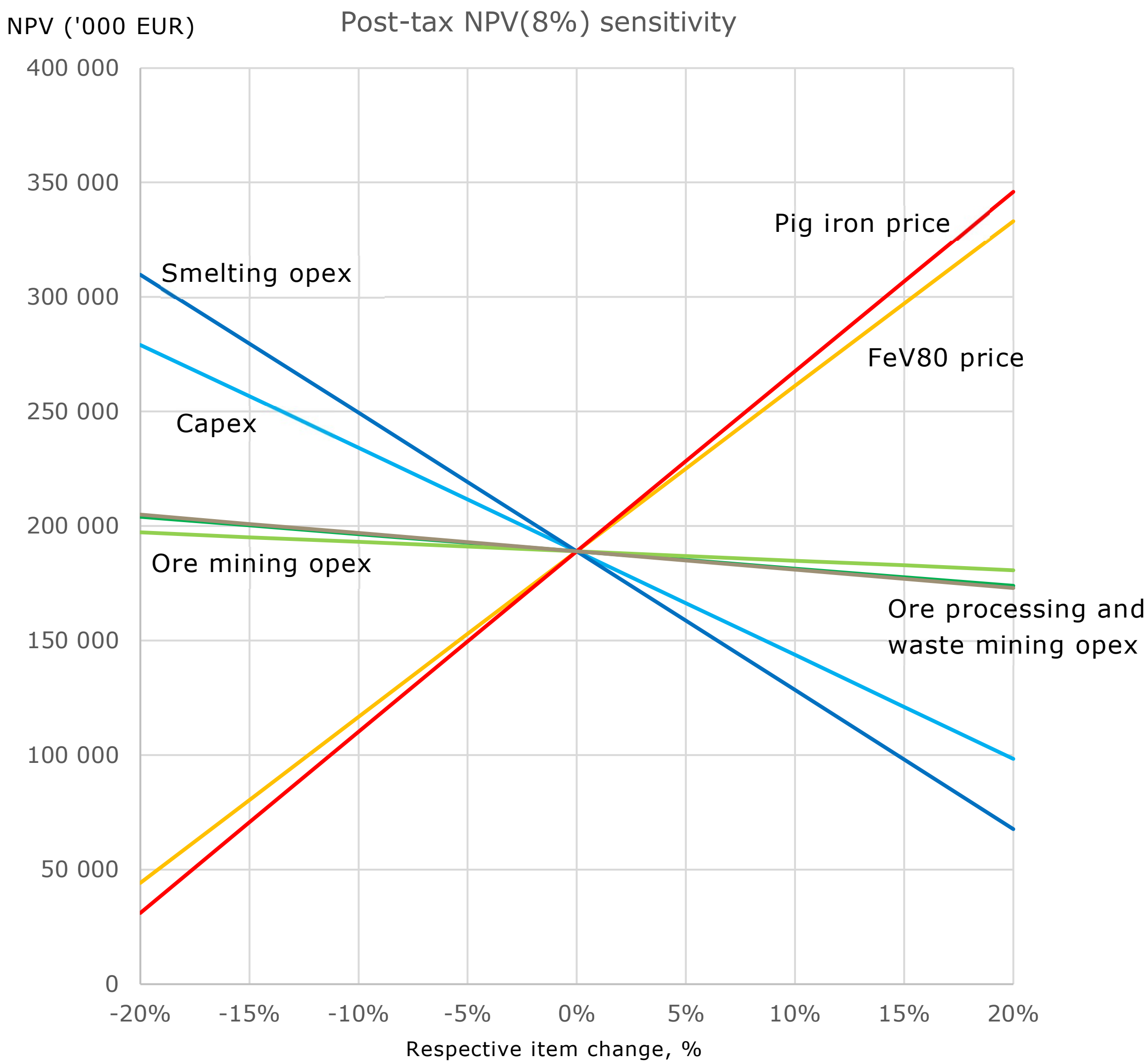

Figure 22-5. Post-tax NPV sensitivity graph.

As can be seen from figures above, project is most sensitive to pig iron and FeV80 prices.

23 Adjacent Properties

Although there are a few vanadium occurrences in the area, there are no nearby published mineral reserves or resources. No information from any adjacent properties has been used in the estimate of the mineral resources at Mustavaara.

24 Other Relevant Data and Information

There are no other relevant data or information.

25 Interpretation and Conclusions

The remarks and conclusions regarding the Mustavaara project are summarized below:

- The drilling and sampling to date supports the mineral resources estimate and there is sufficient information to be used as a basis for the mineral resource estimate and for this PEA study.
- The drilling pattern and spacing covers the known measured, indicated and inferred mineral resources. A limited amount of new drilling down dip of the historic drilling could upgrade the indicated and inferred resources. The down-dip continuation of the magnetite gabbro remains open and is expected to continue with the same thickness and grade in the same kind of geological framework as with the known mineralization.
- The deposit geology and style of mineralization is well understood and the property has a history of successful mining activities.
- Land use planning for the potential reopening of the mine is at an advanced state and is a major upside for the project, as there would be limited delays to be expected in land planning matters.
- The created mine plan supports ca. 20 year LOM.
- The mineral processing concept is well understood and studied.
- Smelting and hydrometallurgical processing concepts for ferrovanadium (FeV80) production are well known.
- Applying for change in environmental permit conditions is necessary, if existing TSF and wet tailings deposition will be used. Other alternatives would be another TSF location or completely another tailings deposition alternative (which requires smaller footprint).

Based on the resource and economic models described in this report, it is the QP's opinion that this report is suitable for Preliminary Economic assessment of the Mustavaara project. The PEA results justify the further study of this project and it is possible to advance into a pre-feasibility study. However, the nesting of endangered species in the vicinity of TFS requires option studies to be made.

The PEA study is preliminary in nature, it includes inferred mineral resources that are geologically too speculative to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. No mineral resources described in this PEA have been converted to reserves. Mineral resources that are not mineral reserves have no demonstrated economic viability. There is no certainty that the preliminary economic assessment will be realized.

26 Recommendations

Based on the mineral resource estimate and the PEA study results, further study of the Mustavaara deposit and advancement to a Pre-Feasibility study is recommended.

To assist the preparation of the pre-feasibility study, a detailed rock mechanics study is recommended to be completed to confirm the geotechnical parameters for the open pit design. A full hydrological study of the Mustavaara deposit is also needed.

If current process route is selected, further investigation (metallurgical test work and modelling) is recommended to confirm recovery estimates and mass and heat balance. Alternative processing concepts should be studied in more detail to evaluate potential capex/opex savings. Detailed process recommendation list is found in chapter 17.5.

As stated in chapters 18 and 20, any subsequent study phases should include more detailed water quality source-term assessments. Process water quality source terms should be based on water analysis from process metallurgical tests. Furthermore, full re-modelling of site water and loading balance is recommended. The loading balance should be used for predictions that are recommended to be done. Consequently, the mine closure plan needs to be updated.

Additionally, geotechnical, rheological, and geochemical testing is required for tailings samples obtained from the updated process metallurgical tests. In addition to this, geotechnical investigations are needed from the tailings storage facility area, especially from the dam locations.

Applying for change in environmental permit conditions is necessary, if existing TSF and wet tailings deposition will be used. Other alternatives would be another TSF location or completely another tailings deposition method (which requires smaller footprint).

Generally environmental impacts are assessed for a smaller version of the projects that is discussed in this PEA. This means a general need to produce information needed for the updated impact assessments according to the new scale of the project. For example, careful water impact assessment is needed, taking into consideration the increase in LOM waste quantities. Possible additional water treatment or new discharge point in larger river may be required.

Cost estimate for recommended work programs for next phase is presented in Table 26-1. Cost of the Pre-Feasibility study includes items described on Appendix 6.

Table 26-1 Cost estimate for future work programs

Items	Cost Estimate
Rock mechanical study	80 000 €
Full hydrological study	120 000 €
Water quality source-term assessment	50 000 €
Re-modelling of site water and loading balance	20 000 €
Tailings test work	105 000 €
Comminution testing	50 000 €
Metallurgical test work and modelling	150 000 €
Pre-Feasibility study	800 000 €
total	1 375 000 €

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