Rupert Resources Ltd.

IKKARI PRE-FEASIBILITY STUDY

NI43-101 Technical Report

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Rupert Resources Ltd.

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This technical report pertained to the mineral property termed "The Ikkari project" which is owned by Rupert Resources Ltd. and this report is effective as of February 14, 2025.

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Timothy Daffern. Fellow AusIMM. Fellow IOMMM (QMR). MSME, MCIM. WSP UK Ltd. Date Signed: February 14, 2025	1.9-1.21, 2-4,13,15-22, 24 and 25-27

Certificates for qualified persons are presented in Appendix 1.

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ACRONYMS AND ABBREVIATIONS

Units of measurement	Abbreviation
Approximately	~
Bar (Pressure)	Bar
Centimetre	cm
Cubic centimetre	cm^3
Cubic meters per day	m^3/day
Cubic metre	m^3
Cubic metres per hour	m^3/h
Cubic metres per square metre per day	m^3/m^2.d
Degrees	°
Degrees Centigrade	°C
Dollars per dry metric tonne	$/dmt
Dollars per kilogram	$/kg
Dollars per tonne	$/t
Dry metric tonne	dmt
Easting	mE
Euro	EUR
Euro per hectare	EUR/ha
G-force	G
Giga years	Ga
Gigawatt	GW
Gram	g
Grams per cubic centimetre	g/cm^3
grams per litre	g/l
Grams per tonne	g/t
Greater than	>

Units of measurement	Abbreviation
Hectares	Ha
High voltage	HV
Hour	hr
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per square meter	kg/m^2
Kilograms per tonne	kg/t
Kilograms per year	kg/a
Kilometers	km
Kilometres per hour	km/hr
kiloPascal	kPa
kiloPascal per meter	kPa/m
Kilotonnes per year	kt/a
Kilovolt	kV
Kilovolt ampere	kVA
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per cubic meter	kWh/m^3
Kilowatt hours per tonne	kWh/t
Less than	<
Litres per second	l/s
Megavolt ampere	MVA
MegaVolts Ampere Reactive	MVAr
Megawatt	MW
Meters above sea level	m asl
Meters per month	m/mo
Metre	m

Units of measurement	Abbreviation
Metres below ground level	m bgl
Metres per second	m/s
Micron / micron metres	µm
Milliamp	mA
Milligram	mg
milligrams per cubic metre	mg/m^3
Milligrams per kilogram	mg/kg
Milligrams per litre	mg/l
Millimetre	mm
Million	M
Million cubic metres	Mm^3
Million dollars	M$
Million dollars per year	M$/a
Million ounces	Moz
Million tonnes	Mt
Million tonnes per year	Mt/a
Millisecond	ms
Moles per kilogram	mol/kg
Ounce	oz
Parts per million	ppm
Per tonne	/t
Percent	%
Second	s
Square kilometre	km^2
Square meter per second	m^2/s
Square metre	m^2
Thousand Troy ounces	koz

Units of measurement	Abbreviation
Thousand Troy ounces per year	kozT/a
Tonne	t
Tonnes per Cubic metre	t/m^3
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per hour per square meter	$(t/h)/m^2$
Tonnes per year	t/a
Troy ounce	ozT
Weight for weight	w/w
Weight percentage	wt%
Wet metric tonne	wmt
Year	a

List of Abbreviations	Abbreviation
50.7 mm diamond drill core diameter	NQ2
57.5 mm diamond drill core diameter	WL76
Acid mine drainage	AMD
Adsorption / desorption / recovery circuit	ADR
Airborne Electromagnetic	AEM
Aluminium	Al
American dollars	US $
Antimony	Sb
Arctic drilling company	ADC
Arsenic	As
Atomic Absorption Spectroscopy	AAS
Barium	Ba
Beryllium	Be

List of Abbreviations	Abbreviation
Billion year old	Ga
Bismuth	Bi
Bituminous geomembrane	BGM
Block caving	BC
Bond abrasive index	BAi
Bond Working index	BWi
Cadmium	Cd
Caesium	Cs
Calcium	Ca
Calcium Hydroxide / Lime	Ca(OH)2
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Canadian National Instrument 43 101	NI 43-101
Capital Expenditure	CAPEX
Carbon dioxide	CO2
Carbon in leach	CIL
Carbon in pulp	CIP
Cemented paste Backfill	CPB
Cemented rock fill	CRF
Central Lapland Belt	CLB
Central Lapland Greenstone Belt	CLGB
Cerium	Ce
Certified reference material	CRM
Chalcopyrite	Cp
Characteristic grind	P80
Chromium	Cr
Coarse ore work index	Mia
Cobalt	Co

List of Abbreviations	Abbreviation
Co-disposal facility	CDF
Coefficient of variation	CV
Concentrate	conc
Conceptual hydrogeological model	CHM
Conditional cumulative distribution function	ccdf
Construction Quality Assurance	CQA
Copper	Cu
Crushing ore work index	Mic
Cut-off grade	COG
Differential Global Positioning System	DGPS
Digital terrain model	DTM
Direct current	DC
Dissolved Oxygen	DO
Drop Weight Index	DWI
Drop Weight Test	DWT
Earnings Before Interest, Taxes, Depreciation, and Amortisation	EBITDA
Eartquake Facilities for Earthquake Hazard and Risk	EFEHR
East	E
East-northeast	ENE
East-southeast	ESE
Effective grinding length	EGL
Electromagnetic (survey)	EM
Electron Probe Microanalyser	EPMA
Energy dispersive spectroscopy	EDS
Engineering, Procurement and Construction	EPC
Engineering, Procurement and Construction Management	EPCM
Environmental and social impact assessment	ESIA

List of Abbreviations	Abbreviation
Environmental impact assessment	EIA
Equivalent Linear Overbreak Sloughing	ELOS
European Union	EU
Exhaust air raise	EAR
Finnish Environment Institute	SYKE
First phase of deformation	D1
Food and Agriculture Organisation	FAO
Footwall	FW
Fracture Permeability	KF
Fresh Air Raise	FAR
Gallium	Ga
General and administration	G&A
General Mine Expenses	GME
Geological Survey of Finland	GTK
Geosynthetic clay liner	GCL
Germanium	Ge
Global positioning system	GPS
Gold	Au
Gold per tonne	Au/t
Gravity recoverable gold	GRG
Grinding Solutions Ltd	GSL
Ground Blast Furnace Slag	GBFS
Ground Control Management Plan	GCMP
Hafnium	Hf
Hangingwall	HW
Hazard and Operability Study	HAZOP
Hazard identification	HAZID

List of Abbreviations	Abbreviation
Heating Ventilating and Air Conditioning	HVAC
High Density Polyethylene	HDPE
Horisontal seismic coefficient	K_H
HPGR ore work index	Mih
Human machine interface	HMI
Hydraulic Conductivity	K
Hydraulic radium	HR
Hydrochloric Acid	HCl
Identification	ID
Ikkari Fault intersection zone	IFIZ
Indium	In
Induced polarisation	IP
Inductively Coupled Plasma - Atomic Emission Spectroscopy	ICP-AE
Inductively Coupled Plasma - Mass Spectronomy	ICP-MS
Insulated metal panel	IMP
Integrated Waste Management Facility	IMWF
Internal organisation for standardisation	ISO
Internal organisation for standardisation/International electrotechnical comission	ISO/EIC
Internal Rate of Return	IRR
Inverse Distance squared - Estimation method	ID2
Iron	Fe
Joint Venture	JV
Joint water reduction factor	Jw
Lanthanum	La
Lead	Pb
Lerch Grossman	LG

List of Abbreviations	Abbreviation
Level elevation	L
Life of Mine	LOM
Linear Low Density Polyethylene	LLDPE
Lithium	Li
Load Haul Dump	LHD
Local area network	LAN
Locked cycle tests	LCT
Long hole open stoping	LHOS
Low Density Polyethylene	LDPE
Lower control limit	LCL
Lower warning limit	LWL
Magnesium	Mg
Magnetotelluric	MT
Manganese	Mn
Maximum Design Earthquake	MDE
Mean annual precipitation	MAP
Mean paired relative difference	MPRD
Mercury	Hg
Mesh Of Grind	MOG
Micon International co ltd	Micon
Minable shape optimiser	MSO
Mine Environment Management Ltd	MEM
Mining waste facility area option 1	K1
Mining waste facility area option 2	K2
Mining waste facility area option 3	K3
mix ratio	R
Mixed ultramafic schist	MSCU

List of Abbreviations	Abbreviation
Molybdenum	Mo
Multiple Indicator Kriging	MIK
Nearest Neighbour - Estimation method	NN
Net Present Value	NPV
Net present value at 5% discount rate	$NPV_{5\%}$
Net Smelter Return	NSR
Nickel	Ni
Niobium	Nb
Nitrogen Dioxide	NO_2
North	N
Northeast	NE
North-Northeast	NNE
North-northwest	NNW
Northwest	NW
NPV Scheduler	NVPS
Number	No.
Open pit	OP
Operating Basis Earthquake	OBE
Operating costs	OPEX
Optical ground wire	OPGW
Ordinary Kriging	OK
Original Equipment Manufacturer	OEM
Overall slope angle	OSA
Particle Size Distribution	PSD
Peak Ground Acceleration	PGA
Phosphorous	P
Plant control system	PCS

List of Abbreviations	Abbreviation
Platinum group elements	PGE
Point Load Test	PLT
Pore pressure to effective stress ratio	Ru
Porewater Pressure	PWP
Positive Displacement	PD
Potassium	K
Pre-feasibility Study	PFS
Preliminary economic assessment	PEA
Preventative support maintenance	PSM
Price factors	PF
Probable Maximum Flood	PMF
Process Flow Diagrams	PFD
Pyrite	Py
Pyrrhotite	Po
Qualified Person	QP
Quality assurance and quality control	QA/QC
Quartile	Q
Reasonable Prospects for Eventual Economic Extraction	RPEEE
Refinery charge	RC
Relative level	RL
Relative Standard Deviation	RSD
Remote input/output	RIO
Resource model	RM
Revenue factor	RF
Reverse circulation	RC
Reverso Osmosis	RO
Rhenium	Re

List of Abbreviations	Abbreviation
Rock quality designation	RQD
Rod mill work index	Rwi
Rubidium	Rb
Run of mine	ROM
Rupert Resources Ltd.	Rupert Resources, RR
SAG Mill Comminution (A type of comminution test)	SMC
Scandium	Sc
Scanning Electron Microscopy	SEM
Second phase of deformation	D2
Selective mining unit	SMU
Selenium	Se
Semi-autogenous grinding	SAG
Silicon	Si
Silver	Ag
Sodium	Na
Sodium cyanide	NaCN
Soil Conservation Service	SCS
Solids Loading Rate	SLR
South	S
Southeast	SE
South-Southeast	SSE
South-Southwest	SSW
Southwest	SW
Specific Energy	SCSE
Specific gravity	SG
SRK Consulting Finland Oy	SRK
Storativity	S

List of Abbreviations	Abbreviation
Stress reduction factor	SRF
Strontium	Sr
Structure 1	S1
Structure 2	S2
Subarctic climate	Dfc
Sub-level caving	SLC
Sulphur	S
Sulphur dioxide	SO_2
Sustainability and environmental and social governance	ESG
Tantalum	Ta
Techno economic model	TEM
Tellurium	Te
Tetra Tech Ltd	Tetra Tech
Thallium	Tl
Third phase of deformation	D3
Thorium	Th
Tin	Sn
Titanium	Ti
Total magnetic indensity	TMI
Trademark	TM
Transmissivity	T
Treatment charge / Refining charge	TC/RC
Tungsten	W
Ultrafiltration	UF
Underground	UG
Underground Distribution System	UDS
Uniaxial Compressive Strength	UCS

List of Abbreviations	Abbreviation
Unmanned aerial vehicle	UAV
Upper control limit	UCL
Upper warning limit	UWL
Uranium	U
Vanadium	V
Variable Frequency Drives	VFD
Ventilation On Demand	VOD
Very low frequency radar	VLF-R
Vibrating wire piezometer	VWP
Water treatment plant	WTP
Watershed Simulation and Forecasting System	WSFS
Weak Acid Dissociable Cyanide	CN_{WAD}
West	W
West-northwest	WNW
West-southwest	WSW
Whole Ore Leach	WOL
Wide area network	WAN
Yttrium	Y
Zinc	Zn
Zirconium	Zr

1 SUMMARY

1.1 INTRODUCTION

WSP Finland Oy (WSP) was commissioned by Rupert Resources Ltd. (Rupert Resources) to perform a Preliminary Feasibility Study (PFS) and produce a technical report, prepared in accordance with Canadian National Instrument (NI) 43-101., for the Ikkari Project located in northern Finland (the Project).

The Qualified Persons (QPs) for this Technical Report are Mr Timothy Daffern, C.Eng., B.Eng., FAusIMM, FIMMM, MSMS, MCIM., Mr. Brian Thomas, P.Geo., both are independent QPs, as defined under NI 43-101 and employees of WSP. The Technical Report effective date is 14 February 2025.

1.2 PROPERTY DESCRIPTION AND OWNERSHIP

1.2.1. PROJECT DESCRIPTION AND LOCATION

The Ikkari Gold Deposit is situated within Rupert Resources' "Rupert Lapland Project" exploration licences, located in the province of Lapland, Northern Finland (Figure 1-1).

Figure 1-1 – Location of Rupert Lapland Project, Finland

More locally, the project occurs across an area surrounding the Rajala village in the municipality of Sodankylä. The Ikkari Gold Deposit occurs in the westernmost extents of the Rupert Lapland Project, approximately 30 kilometres (km) northwest of Sodankylä town centre, 10 km north-northwest (NNW) of Jeesiö village and 22 km west-southwest (WSW) of the Pahtavaara Mine, a gold mine currently on care and maintenance within the Rupert Lapland Project tenement package (for coordinates see Table 4-1).

The Ikkari deposit lies on the eastern extreme of the Sirkka Line, a tectonic structure that traverses northern Finland, along which some 25 to 30 gold deposits / occurrences exist. Ikkari is situated at the margins of a low-lying aapa-mire, comprising broad wetlands to the north and west, and is sparsely forested.

The landscape across the Ikkari deposit area is predominantly flat with an elevation of approximately 225 metres (m) above sea level (asl) and rising slightly toward the southeast and the margins of the Iso-Pulkittama hill, which has a maximum elevation of approximately 300 m asl. The overburden cover of glacial till deposits is generally between 10 m to 40 m thick and rock outcrop is very limited across the exploration licence area. In most parts of the deposit area, the ground water table is located close to the ground surface.

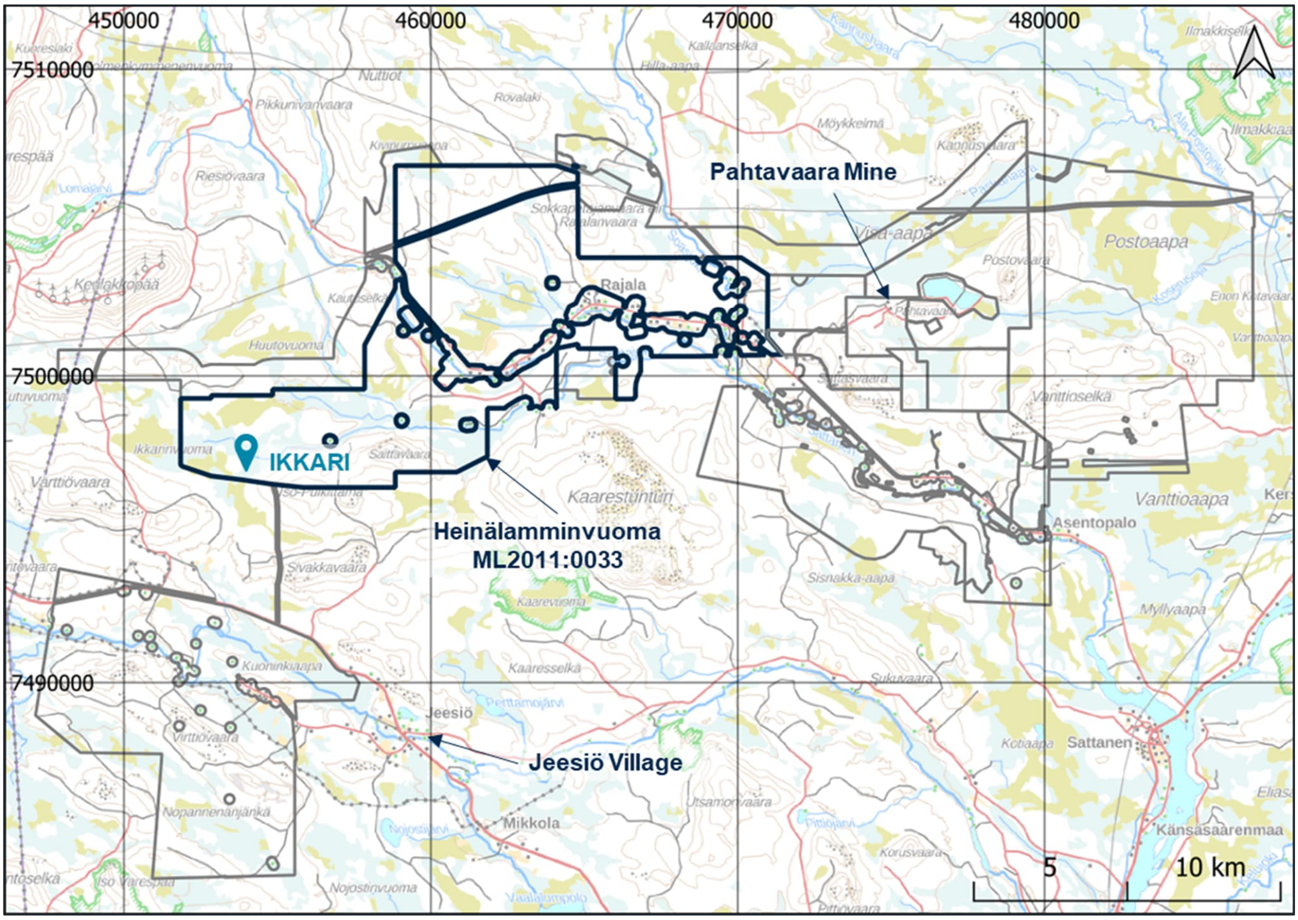
1.2.2. OWNERSHIP

The Rupert Lapland Project area, in which the 100% Rupert Resources owned Ikkari deposit occurs, comprises a contiguous package of mining licences, exploration permits, and exploration permit applications totalling an area of 340.6 square kilometres (km^2) including the Pahtavaara Mine, currently on long term care and maintenance, and it's associated 4.21km^2 mining licence. Additional permits elsewhere in the Central Lapland Belt, contribute to a total of 438 km^2. The Rupert Lapland Project property is subject to a 1.5 percent (%) royalty on revenue, capped at US (United States) $2.0 million (M).

The Ikkari deposit is contained within the existing valid exploration permit Heinälamminvuoma - ML2011:0033, with an area of 84 km^2 (Figure 1-2). Both Rupert Finland Oy and Rupert Exploration Finland Oy are wholly owned subsidiaries of Rupert Resources Ltd., a company incorporated in British Columbia, whose office is at 82 Richmond Street East, Suite 203, Toronto, Ontario, Canada, M5C 1P1.

1.2.3. SITE VISIT

A personal inspection of the project site was conducted by Mr. Brian Thomas, P.Geo., from July 11th to 13th, 2023, and by Mr. Timothy Daffern on 6th February 2025. Other WSP professionals visited the site during their technical investigations, to observe site conditions, review geological data collection and Quality Assurance and Quality Control (QA/QC) procedures and results, confirm drill collar locations, and complete verification sampling and logging of drill core.



Note: The Heinälamminvuoma Licence, where Ikkari is Located, is Shown in Bold.

Figure 1-2 – Location of the Ikkari Gold Deposit

1.3 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

1.3.1. ACCESSIBILITY

Rovaniemi and Kittila Airports offer domestic flights to Helsinki several times daily and international flights during holiday seasons. The drive from Rovaniemi to Sodankylä is approximately 130 km, taking under two hours. To reach the exploration site, accessible year-round, from Sodankylä direction, the route begins with main road 80 towards Kittilä. After Jeesiö village, is a gravel road to Pulkittama for 7.5 km. The Ikkari turn can also be accessed from Kittila in the other direction and the drive time is under 1 hour. Heavy goods can be transported by road from Helsinki, Hamina-Kotka or Kemi ports the latter being 245km south of Sodankylä town. The closest rail heads are at Rovaniemi and Kemijärvi, 130km and 110km respectively south of Sodankylä.

The landscape has been shaped by the last ice age, from 110 000 to 10 000 years ago. The landscape is dominated by low rolling hills and flat lowlands comprised of wetlands (bogs) and lakes. Hills are mostly covered by glacial moraine and sands and are forested, primarily with birch, pine, and spruce which are exploited by the state forestry company. Bedrock outcrops on the hills and along riverbanks is limited to some two percent or less of the project area. The Ikkari gold deposit is located at the margins of low-lying wetland terrain, cut by a small stream, rising towards a boulder-dominated, gentle slope in the south-southeast (SSE). The terrain drains to the typical small Finnish river (creek) Saittajoki River (nominal width 5m) and then into the Sattanen River, and then to the Kemijoki River and the Gulf on Botnia.

1.3.2. CLIMATE

According to Köppen climate classification, northern Finland is classified as Dfc (Continental, Subartic/boreal climate) with no defined dry season. The region has cold winters with the mean temperature of the coldest month below -3°C and short, cool summers with the mean temperature above 10 degrees centigrade (°C) for fewer than 3 months.

Based on data from the Sodankylä, Tähtelä weather station, the average annual temperature from 1991 to 2020 was 0.3°C, rising to 0.8°C in 2023. Summer temperatures range from 10°C to 20°C, and winter temperatures from -2°C to -40°C. Snow covers the ground for 183 days annually, with maximum thickness between 0.6 m and 1.2 m in March. Bogs, lakes, and rivers freeze for 4-5 months, aiding winter exploration. Annual rainfall averages 600 mm, with the wettest period in June-July and the driest in February-April. The climate is influenced by Arctic location and airflow direction, with westerly winds bringing warm weather from the Atlantic and easterly winds causing severe cold in winter and warm days in summer. Weather patterns in northern Finland can change rapidly, especially in winter, due to the collision of cooling sub-tropical and polar air masses, influenced by low-pressure systems from Iceland and high-pressure systems from Siberia and the Azores. The climate is not expected to have any significant impact on the Ikkari operating season, and the operations can be conducted on a year-round basis.

1.3.3. LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The town of Rovaniemi, located 150 km South-Southwest of Ikkari has approximately 60 000 inhabitants. Rovaniemi is the administrative centre of Finnish Lapland. The regional technical centre of the Geological Survey of Finland (GTK) and its analytical laboratory are also located here.

The town of Sodankylä supports the Rupert Lapland exploration permits with accredited sample preparation and fire assay facilities. The region's economy is dominated by small businesses in forestry, agriculture, and manufacturing, with mining as the largest employer. Kevitsa Mine, operated by Boliden, employing an average of 570 people with large numbers of contractors providing services from time to time. There are several hotels, shops, and restaurants which accommodate a growing year-round influx of tourists into Lapland.

Hydroelectric power is relatively inexpensive, and a high voltage power line is located near the Ikkari deposit, with a transformer located 9km form the plant site. Limited infrastructure exists at Ikkari, with recent additions including an access road and powerline. The logistical hub for exploration is located near Sodankylä, where Rupert's management and administration are based.

1.4 HISTORY

Ikkari is an under-cover grass roots discovery made in March 2020. Limited previous exploration activities have been undertaken in the area prior to the work conducted by Rupert Resources during 2019 to present.

The Heinälamminvuoma exploration permit on which the Ikkari Gold Deposit is located, was applied for in 2011 by Lapland Goldminers, the then owners of the operating Pahtavaara Mine. However, no work was completed in the licence area and the exploration permit remained in the application phase. This was the first instance of a private company applying for an exploration permit over the Ikkari deposit. Rupert Resource Ltd. purchased the operation from the administrators of Lapland Goldminers in September 2016.

The Heinälamminvuoma exploration permit has been part of the Rupert Lapland Project area since that time, although very little exploration was undertaken initially and exploration field activities were confined to the easternmost parts of the licence, adjacent to the Pahtavaara Mine itself before 2018.

Geology Survey of Finland have undertaken regional mapping, geophysical surveys and some historic till geochemistry samples in the area.

Considering initially the entire Rupert Lapland exploration licences, the vast majority of historic drilling has been carried out at the Pahtavaara Mine site, and near-mine areas with very little drilling completed elsewhere on the permits. No drilling has been undertaken by previous operators at or near the Ikkari deposit. Historical drilling across the Rupert Lapland Project area has been conducted by GTK, Outokumpu, Terra Mining, Scan Mining, Lapland Goldminers and Anglo American.

1.5 GEOLOGY AND MINERALISATION

1.5.1. DEPOSIT GEOLOGY

Ikkari is located under 10 to 40 m of transported glacial till cover and occupies a complex structural position between thrust imbricated Savukoski Group metavolcanics and metasediments, and synorogenic molasse-type siliciclastic strata of the Kumpu Group. At their most basic level, a 4-fold lithologic subdivision is constructed for the rock types present at Ikkari. (Figure 7-4):

- Dark pyritic shales and siltstones termed the 'black shale' (intruded by gabbro) comprise the northern fault block and form the hangingwall to the mineralisation;
- A central komatiite-dominant zone with complex intercalations of texturally diverse 'felsic' facies;
- A northern, banded 'felsic' facies, intensely albite-altered in places, that pinches out in the eastern part of the deposit; and
- A southern zone comprising dominantly coarse 'felsic' siliciclastics – massive, banded, conglomeratic and typically more quartz-rich than the northern facies but which hosts intercalations of komatiite in decreasing abundance moving southwards.

1.5.2. MINERALISATION

The Ikkari deposit can be described as an orogenic, hydrothermal gold deposit. Gold, which trends at approximately 065° strike, has a strong sub-vertical control. Gold is hosted by disseminated and vein-related pyrite although free gold in the form of ~1 millimetres (mm) gold grains, is also common. Gold is associated with pyrite and occurs either on the surface of pyrite grains or on fractures within the grain.

Mineralisation at Ikkari occurs in several styles, but in all cases, gold distribution is correlated to the abundance of disseminated pyrite and intensity of veining. The style of mineralisation is principally controlled by the host lithology with significant controls on mineralisation localization including:

- Brittle-fracture regime in intensely albite-altered felsic sediments;
- Lithological contacts, particularly sedimentary intercalations within the wider ultramafic package;
- Fold hinges, including short-wavelength parasitic folding; and
- Within and at the margins of hematite-carbonate hydrothermal breccias.

Despite these variations in localization, at the deposit scale, it is considered that all the gold mineralisation is related to the same (oxidized) fluid event that was introduced along a complex brittle-ductile permeability meshwork. Sites of gold deposition are structurally controlled but locally dependent on the availability of a geochemical reductant that allows deposition of pyrite and associated gold. Such iron-rich reductants at Ikkari are likely to include magnetite and chlorite, formed during an earlier iron-metasomatic alteration and/or syngenetic pyrite that may have been present in the intercalated siltstones. The spatial association of high-grade gold zones to apparently later, largely post deformation hematite-carbonate breccias is indicative of a later gold-bearing fluid phase also being present.

1.6 EXPLORATION STATUS

1.6.1. EXPLORATION

Ikkari represents a new discovery that was initially identified through systematic base of till (BOT) sampling beginning in early 2019. In the Ikkari area, a single anomalous BOT sample of 0.2 parts per million (ppm) Au was followed up with infill sampling to a 50-m-x-25-m grid, and a small cluster of anomalous samples up to 1 ppm Au was identified. The first drill hole into this geochemical anomaly (hole 120038) was drilled in April 2020 and assayed 54-m grading 1.5 g/t Au from 25 m, under 13 m of glacial till cover material. Follow-up drill hole intercepts demonstrated very broad mineralized zones with a high-grade component over an initial strike length of greater than (>) 500m.

Exploration continues both within the Heinälamminvuoma exploration permit and on adjacent permits with BOT now supplemented by geophysical techniques ground tested at the Ikkari deposit. Drill testing of both geochemical and geophysical targets continues.

1.6.2. DRILLING

Rupert Resources are the only entity to have drilled in the vicinity of the Ikkari deposit. Yearly drilling by Rupert Resources at the Ikkari Deposit is summarized in Table 1-1. The drilling cut-off for this report was made in June 2023 and 111 896m are considered in the resource.

Table 1-1 – Summary of Ikkari Drilling

Prospect	Year	DH Type	Holes	Metres
Ikkari	2020	Diamond	62	20 320
	2021		75	36 049
	2022		85	35 568
	2023 (EOY 2024)		46	22 069
Total			268	114 006

Notes:

* Including later extensions to drill holes and wedges.

** Including holes such as metallurgical holes not assayed, and therefore not included in the resource estimation (Chapter 14).

Note: Reported as per prospect on coding in database, not all holes are necessarily targeting the same mineralisation occurrence. Errors may occur due to rounding.

1.6.3. SAMPLE PREPARATION, QAQC AND SECURITY

All sample preparation was carried out at independent certified laboratories in Finland, and analyses were carried out at independent certified laboratories in Romania, Ireland, or Finland. No aspect of laboratory sample preparation or analysis was conducted by an employee, officer, director or associate of Rupert Resources.

Rupert Resources has used a combination of duplicates, checks, blanks and standards to ensure suitable quality control of sampling methods and assay testing. The procedures and QA/QC management are consistent with industry practice and are deemed fit for purpose. Results of recent sampling have not identified any issues which materially affect the accuracy, reliability or representativeness of the results.

It is the resource QP's opinion that the sample preparation, analytical, QA/QC and chain of custody procedures used to produce the sample database are consistent with industry practises and Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Mineral Exploration Best Practice Guidelines (November 2018).

1.6.4. DATA VERIFICATION

The mineral resource QP completed a 3-day site visit that included verification logging, sampling and verification of hole collar coordinates for selected holes. All drill logs, assays and hole collar data were found to be consistent with the Rupert Resources database and no material differences were identified.

Additional data verification of the Rupert Resources database was conducted under the supervision of the QP consisting of spot check comparisons of collar coordinates, down-hole surveys, assay and density data against the original data sources with no material differences identified.

It is the QP's opinion that the exploration, drilling and analytical procedures used by Rupert Resources to collect geological data are consistent with industry practises and CIM Mineral Exploration Best Practise Guidelines (November 2018) and that the data is suitable to support the MRE as summarized in this Technical Report.

1.7 DEVELOPMENT AND OPERATIONS STATUS

The Ikkari mineral property has ongoing geological and environmental surveys. There is no industrial activity on site.

1.8 MINERAL RESOURCE ESTIMATE

The MRE for the Ikkari deposit has been prepared in accordance with NI 43-101 and following the requirements of Form 43-101F1. The methodology used to determine the MRE is consistent with the CIM Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines (November 2019) and was classified following CIM Definition Standards for Mineral Resources & Mineral Reserves (May 2014).

Table 1-1 summarises the current Indicated and Inferred Mineral Resources for the Ikkari Project. Mineral Resource Estimates are reported in-situ and inclusive of Mineral Reserves.

There is no certainty that all, or any part, of this Mineral Resource will be converted into Mineral Reserve. Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves.

Table 1-2 – Ikkari Mineral Resource Estimate (Effective Date October 24, 2023)

Resource Category	Mining Method	Cut-Off Grade Au (g/t)	Tonnes (t)	Grade Au (g/t)	Au Content (Troy Ounces)
Indicated	Open Pit	0.4	37308 000	2.21	2649 000
	Underground	0.9	21122 000	2.12	1437 000
Total Indicated	**-**	**-**	**58430 000**	**2.18**	**4087 000**
Inferred	Open Pit	0.4	1271 000	0.81	33 000
	Underground	0.9	2305 000	1.39	103 000
Total Inferred	**-**	**-**	**3576 000**	**1.18**	**136 000**

Notes:

1) Mineral Resource Estimates are reported in-situ and inclusive of Mineral Reserves.
2) Tonnage and ounces are rounded to the nearest 1 000.
3) g/t = grams per tonne, ounces are reported as troy ounces.
4) Totals may not add up correctly due to rounding.
5) Cut-off grade defined by Gold Price, $1700/oz, Metallurgical Recovery 95%, Open Pit Mining Costs $2.9/t, Underground Mining Cost $29/t, Processing Cost $11.30/t, G&A, Rehabilitation & Closure $4.8/t, Royalty 0.75%.
6) Open pit resources constrained within a Whittle Optimized open pit shell using the above assumptions with a 26m offset to the property boundary enforced.
7) Underground resources constrained within the estimation domains to meet the RPEEE criteria for underground mining.

Cut off grades for both the open pit and underground portions of the MRE were calculated based on economic assumptions set out in Table 1-3. Cost data and pit slope criteria were derived from the Ikkari Preliminary Economic Assessment (Tetratech, 2023), filed March 17, 2023.

The Open pit MRE was evaluated for Reasonable Prospects of Eventual Economic Extraction (RPEEE) by reporting blocks above a 0.4 g/t Au cut-off from within a Whittle generated Revenue Factor (RF) 0.95 pit shell based on the assumptions and parameters set out in Table 1-3. In addition to these parameters a 26 m offset from the license boundary to the edge of the pit shell was enforced.

The Underground MRE was reported outside and below the pit shell at a 0.9 g/t UG break-even cut-off grade representing bulk scale Longhole mining. The UG resource is constrained to within the three mineral domain models. Blocks above cut-off outside of the mineral domains did not demonstrate reasonable mining continuity and therefore were excluded from the MRE.

Table 1-3 – Ikkari MRE Economic Cut Off grade and Optimized Shell Parameters

Parameter	Unit	Price/Cost
Gold Price	US dollar / troy ounce	1 700
Metallurgical Recovery (Gold)	Percentage	95
Open pit mining cost	US Dollar / tonne	2.90
Underground mining cost	US Dollar / tonne	29.00
Processing Cost	US Dollar / tonne	11.30
G&A, Rehabilitation & Closure	US Dollar / tonne	4.80
Royalty (state and landowner combined):	Percentage	0.75

1.9 MINERAL RESERVE ESTIMATE

The Mineral Reserve for the Ikkari project was estimated by converting the open pit and underground resource through the application of modifying factors. Indicated Resources have been converted to Mineral Reserves. Inferred Resources have been considered as waste with grades set to zero.

The estimation of Mineral Reserves followed the CIM Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines (November 2019) and is in accordance with CIM Definition Standards for Mineral Resources & Mineral Reserves (May 2014) and Canadian National Instrument 43-101 (NI 43-101).

The Ikkari open pit and underground Mineral Reserve estimate, with an effective date 25 November 2024, was prepared under the supervision of WSP Technical Director Mr. Timothy Daffern, who is the Qualified Person responsible for the Mineral Reserve estimate. The Mineral Reserve estimate is stated in Table 1-4.

Table 1-4 – Ikkari Gold Project Mineral Reserve by Category Effective 25 November 2024

Mining Method	Category	Tonnage [Mt]	Gold Grade [g/t]	Contained Gold [koz]
Open Pit	Proven	0.0	0.0	0
	Probable	35.7	2.2	2 486
Underground	Proven	0.0	0.0	0
	Probable	16.3	1.9	1 007
Total Mineral Reserves		52.0	2.1	3 492

Notes:

1) Tonnages are rounded to the nearest 100,000 and ounces are rounded to the nearest 1,000.
2) Mineral Reserves were estimated using the CIM Standards for Mineral Resources and Reserves, Definitions and Guidelines.

3) The QP for the Mineral Reserve Estimate, as defined by NI 43 101, is Mr. Timothy Daffern, Technical Director with WSP. The effective date of the estimate is November 25, 2024.
4) Mineral Reserves are based on a gold price of US$1,700/oz.
5) Metallurgical recovery is based on a fixed recovery of 95.0%.
6) Open pit Reserves are stated using a 0.34 g/t cut-off. Open pit Reserves are converted from Resources through the process of pit optimisation, mine design, schedule and are supported by a positive cash flow analysis.
7) Open pit Reserves include an allowance for 4% dilution and 4% mining losses applied in the production schedule.
8) Underground Mineral Reserves are stated using a 1.04 g/t cut-off. Underground Reserves are generated through the generation of optimised stopes, design of long hole open stoping, schedule and are supported by a positive cash flow analysis.
9) Underground Mineral Reserves account for planned dilution of 15%, unplanned dilution of 6%, secondary dilution of 3% and with mining losses of 4%.
10) Totals may not sum due to rounding.

1.9.1. MINING METHOD

Ikkari consists of open pit and underground mining. Open pit mining will be performed using truck and shovel configuration. Drilling and blasting are planned on 10m benches, with the open pit extending from the surface (230 mL) to -80 mL.

Two stages are planned, with total open pit inventory at 36 Mt at 2.2 g/t for 2.5 Moz of contained metal. Open pit operations commence in Year -1, with a year of pre-stripping prior to first gold pour. The open pit operations are planned to produce 3.5 Mtpa ore and cease in Year 11.

Long Hole Open Stoping (LHOS) with a combination of paste and waste rock backfill was the selected underground mining method for the Ikkari deposit. Access to the underground mine consists of two declines; one from surface to the east of the open pit and the second from the 40 mL switchback inside the open pit used at the cessation of open pit operations. Ventilation is provided to the underground mine through four shafts; two for exhaust and two for the provision of fresh air and heat when required. Material will be hauled to surface via trucks through the two declines.

Stopes are planned on 30 m vertical intervals and 15 m intervals between stopes. A primary-secondary stope sequence is planned. This entails mining of the primary stopes and leaving at least one stope width between as a supporting pillar. This pillar is referred to as a secondary stope. At the completion of mining and curing of backfill in the primary stope on either side of the pillar, mining of the secondary stope may be commenced.

Development of the main decline access commences in Year 6, with stoping to start at the end of Year 10. The underground mine ends in Year 20. Total underground inventory is 16 Mt at 1.93 g/t for 1.0 Moz of contained metal. The underground operations are planned to produce 2.0 Mtpa of ore.

Total life of mine is expected to be 20 years. A total of 52 Mt of ore is planned to be fed to the plant, with an average grade of 2.1 g/t for 3.5 Moz of contained metal.

1.9.2. ROCK MECHANICS

Open pit stability was analysed with kinematic analyses to determine bench scale susceptibility to structural failures. Based on the stereographic projection analysis of logged joint orientation data, the lithological domains in the open pit area generally only have two relatively steeply dipping joint orientations, of which one lines up with foliation orientation. Bench failure modes are mainly toppling and planar failures (mainly South-East and East walls). Expected failure volumes are small and most of the failures are dominated by strong foliation.

Two-dimensional finite element analysis was performed to analyse large scale stability against failure. The simulation was performed using shear strength reduction method for saturated and drained slope models, separately for each open pit sector. The results demonstrate stable slopes within 45° to 55° overall slope angle, depending on the pit sector. The ramp placement is recommended on the North wall due to lower rock mass quality and vicinity of property boundary on the South side of the planned open pit.

Rock mechanics inputs for underground stope design have been selected to achieve a reliable and robust mine design and sequence using rock mass qualities between the 25th and 50th percentile. Maximum induced stresses on stope surfaces have been assessed with 3D numerical modelling, considering interaction between mining areas and the open pit. Two empirical design methodologies and associated criteria have been used in parallel, resulting in stable stope sizes for primary and secondary stope lines. Different lithologies and sectors within the underground orebody were considered in design recommendations. Dilution from sidewalls has been estimated based on recorded case histories at other mines.

Ground support estimates are provided for mining drives, intersections, and stope backs. They should be seen as input to economic assessments for this PFS and are not designed.

Paste Fill strength requirements for primary and secondary stope lines, as well as stopes above sill pillars are provided. A general triangular retreat shape using a primary and secondary stope arrangement was selected with a mining direction away from the Southern fault zone.

1.9.3. PASTE PLANT PROCESS AND TESTING

Ikkari tailings samples were submitted to WSP's Sudbury, Canada laboratory where test work was performed to determine material characteristics and rheological properties. Paste backfill recipes were also developed for the different target strengths required for the backfill material.

The paste plant will receive filter cake tailings from the filter plant when backfill is required. The filter cake will be mixed with a proportional amount of binder as required by the backfill recipes. Filter cake, binder, and water will be mixed into a homogenous paste backfill material at a specific slump using a twin shaft mixer. The mixer will discharge paste into a piston type positive displacement pump. The pump will transport paste through an underground distribution system to the different levels of the mine, to the location of the stope that requires backfilling.

The paste plant is located adjacent to the filter plant to optimise the transportation of filter cake to the paste process. The paste plant is located 400 m away from a borehole at the edge of the open pit, which is the opening to the underground backfill distribution system. A surface pipeline connects the paste plant to the borehole.

1.10 MINERAL PROCESS AND METALLURGICAL TESTING

Metallurgical testwork was conducted using samples from within the Ikkari deposit. Samples were representative in terms of rock type, composition, and spatial location. The projected metallurgical recovery was determined using results from gravity recovery testwork followed by leaching testwork carried out by Grinding Solutions Ltd.

Metallurgical testwork has confirmed that the expected recovery can be achieved using a conventional beneficiation process consisting of crushing and grinding to 100 µm followed by gravity concentration and carbon-in-leach.

Leach tailings thickening and filtration tests were carried out by Paterson & Cooke. Testwork has demonstrated that the moisture level desired for dry stacking the tailings can be achieved using pressure filtration.

The processing plant flowsheet consists of primary crushing using a jaw crusher, followed by a covered crushed ore stockpile and reclaim tunnel. The grinding circuit is a standard SABC circuit: a SAG mill in closed circuit with a pebble crusher followed by a ball mill and classifying cyclones. A gravity recovery and intensive leach circuit recovers liberated gold from the cyclone underflow. The cyclone overflow is thickened and feeds a carbon-in-leach (CIL) circuit. After desorption in a pressure Zadra elution circuit, gold is recovered via electrowinning and poured into doré bars. Tailings from the CIL circuit are detoxed using a SO2/Air cyanide destruction circuit and pumped to the filtration plant.

The tailings filtration plant is located approximately 350 m away from the processing plant. The future paste plant is planned to be built adjacent to the filtration plant. The tailings from the processing plant are thickened in a high-rate thickener and feed three horizontal pressure filters. The filtered tailings are stored in a building located next to the filtration plant from where it is reclaimed to be used for co-disposal.

Based on the metallurgical testwork results and the proposed flowsheet, the overall projected gold recovery is expected to reach 95.8%.

1.11 PROJECT INFRASTRUCTURE

An overall surface infrastructure to the mine is developed, which is on the gently undulating hill to the south of the Saittajoki river. Terracing (or pads) are provided for the process plant, filter plant, maintenance workshops, administration, water treatment plant. Other assets are specifically sited within the topography at their given location. A network of roads is provided, including the ROM haul road to the ore stockpile and primary crusher, the waste haul road and filtered tailings haul roads. A designated main access road is aligned through the plant site accessing the other main assets including the administration building, process plant, workshops, stores and filtration plant. A network of lighter access roads is provided to access the remaining surface assets. The raw water and treated water storage ponds are to the east of the water treatment plant.

1.12 ENVIRONMENTAL AND PERMITTING

Framework legislation includes several laws, acts, decrees, and permits which all affect Rupert Resources' operations. Applicable codes are for example Mining Act (621/2011), Environmental Impact Assessment Procedure Act (252/2017), Environmental protection Act (527/2014), Water Act (587/2011), Nature Preservation Act (9/2023), Building Act (751/2023) and Land use Act (132/1999).

Permits guiding operations include amongst other: exploration permit, environmental and water permit (with preceding EIA procedure), derogation permit from nature protection provisions, mining and mining safety permit, building permit, permit for handling and storage of dangerous chemicals and permit for storage of explosives. At the current time Rupert Resources Ltd. holds a valid exploration permit for Heinälamminvuoma, ML 2011:0033, setting the current environmental action limits for the project. Rupert Resources Ltd. is working towards submission of the EIA report for the project during H2 2025. The EIA procedure, according to the Environmental Impact Assessment Procedure Act (252/2017), is compulsory before an environmental permit application can be filed within an authority.

1.13 CO-DISPOSAL FACILITY

The mining waste and process tailings are to be co-disposed at one location at the relatively higher ground at Pahkalehto to the North of the Saittajoki river. This co-disposal facility has a designed capacity of 91.5 Mm^3 to a height of approximately 80m. This covers an area of approximately 2.0 Mm^2 and will include a low permeability liner formed on top of overburden from the open pit and excess material from the infrastructure earthworks. Seepage and runoff are diverted into a perimeter channel which diverts flow into a lined runoff collection pond. The design allows for phased development during construction and the first few years of LOM. The average side slope of the facility is 1 (v) to 3 (h), which includes of operational benching. The waste and filtered tailings are to be continuously placed in layers of varying depths depending on the strip ratio and surface area of the facility as it rises. Both the waste and filtered tailings will require compaction during placing. Initial stability assessment is made as well as suggestions for operation and ongoing monitoring. Recommendations for further site investigations and testing are provided.

1.14 SITE WATER MANAGEMENT

To facilitate the mining operation and minimise the risk of polluting the rivulet some 5m in width, locally termed the Saittajoki River, it is proposed to divert both the southern tributary running through the proposed open pit area further west and the main river channel around the mine site to the north into the adjoining sub catchment. To further minimize the amount of surface water runoff that needs treatment it is also important to separate contact and non-contact water on the mine site as far as is practically possible. This practise will need to be extended to the management of snow to allow mine operations to progress unhindered during winter months, where the snow will need to be stockpiled within a contact catchment such that the resulting melt water can be managed appropriately.

Contact water management ponds are required to balance peaks in inflows during wet periods. There are two main contact water ponds proposed, the Co-Disposal Runoff Collection Pond (Co-Disposal Pond) and the Raw Water Pond.

The surge capacity and pump out rate of the two contact water ponds have been sized to manage contributions from storms and/or snow melt to minimise the risk of spilling to the environment as follows:

- Co-Disposal Pond: Surge capacity of 440 000 m^3, peak pump out rate of 200 m3/h; and
- Raw Water Pond: Surge capacity of 600 000 m^3, peak pump out rate of 900 m3/h.

Sediment removal would be required at the inlet to the Raw and Co-Disposal ponds to prevent loss of storage capacity. To minimize the risk of spilling the ponds cannot be used for the long-term storage of water.

A Treated Water Pond has also been included to provide some residence time between the treatment plant and the inlet to the discharge pipeline. This will allow time to take process water quality samples before the treated water is discharged. The Treated Water Pond has been sized with a total volume of 290 000 m^3 to provide 14 days of treated water storage if the discharge pipeline is shut down.

1.15 WATER TREATMENT FOR DISCHARGE

The water treatment requirements were defined based on the site water balance, available water quality data from monitoring and other relevant studies. The definition of acceptable discharge water quality to the environment were based on a review of the current regulatory framework, current permitting practices and known future changes in the national and EU Regulations, which were modelled for the Kitinen River.

Contact mine water and process water from the ore processing plant are to be treated in two separate treatment plants; this is to treat the two different water streams through fit-for-purpose treatment processes.

Contact mine water will be treated such as its effluent water quality complies with environmental discharge water quality. The treated water will first be stored in the treated water pond before being discharge to the Kitinen River via a 37 km pipeline. The plant aims at reducing concentration of suspended solids, metals, nitrogen compounds as well as protecting the discharge pipeline against corrosion and fouling. Depending on the treatment chosen for nitrogen removal, biological or ferric sludge and/or reject from ion exchange process will be produced.

To minimise the requirement for freshwater abstraction, the process water from ore processing plant will be treated such as it can be reclaimed and used as a freshwater source to the ore processing plant. Contact water is to be added to the inlet of this second plant to meet water demand from the ore processing plant. The plant aims at reducing concentration of suspended solids, sulphate, nitrogen compounds, inorganics introduced by the ore processing process. The plant has been designed to be zero liquid discharge, producing a ferric sludge, gypsum slurry and a mixed salt precipitate.

In addition to the mine water treatment requirements, domestic potable and sewerage systems will be provided to serve welfare facilities throughout the site. The potable water will be generated from treated water from the external pit dewatering borehole and will meet Finnish drinking water quality standards. A sewage treatment plant will be provided; the effluent will be discharged via the discharge pipeline.

Biological and ferric sludges will be dewatered before being off-site alongside the mixed salt precipitate. Reject from the ion exchange is to be treated through the zero liquid discharge treatment process. Gypsum slurry will be stored on-site in the Gypsum slurry pond. Four ponds are provided to East of the water treatment plant for disposal of gypsum slurry. These have a capacity of approximately 310 000 m^3.

1.16 MINE CLOSURE

Ikkari has a planned mine life of 20 years, after which the site will enter an active closure period (approximately three years), followed by post-closure monitoring and maintenance. The river diversion will be constructed prior to mining with a nature-based geomorphic design such that it can be monitored throughout the mine life and remain in place at closure.

The waste material co-disposal facility will be rehabilitated at the end of mine and mineral process plant operations.

All major infrastructure will be decommissioned once it is no longer required and will be recycled/repurposed where practical or disposed of in an appropriate landfill facility off-site. The water treatment plants will be some of the last infrastructure to be removed once water quality on site meets permitted discharge criteria, presently assumed to be within five years of the end of active closure. All infrastructure will be removed from the open pit once it is not required, and dewatering wells will be turned off and decommissioned once it is safe to do so; the open pit will be allowed to flood, with water expected to overflow the pit walls within 20-25 years of dewatering wells being turned off. It is presently assumed that the water leaving the open pit some 25 years after closure will be suitable for direct discharge to the environment.

The overarching goal of these closure actions is to reclaim mine-impacted land to support similar land uses to those present prior to mining, albeit in a different arrangement. Accordingly, the post-mining land use vision is focused on re-establishment of pre-mining land uses, mainly locally common habitat (mires and mixed forest), support for local passive recreational enjoyment of nature including snowmobile and hiking trails, and reindeer husbandry.

1.17 COST ESTIMATES

All cost estimates have used the American Association of Cost Estimation (AACE) guidelines. This Pre-Feasibility Study (FEL2, AusIMM) was completed to AACE Class 4 accuracy.

The foreign exchange rates used in this study Euro to USD 1:1.05, Euro to CAD (Canadian dollars) 1:1.49, and GBP to USD 1:1.25.

Capital costs include pre-production and sustaining capital. Pre-production capital designates capital spent until commercial production is reached. This includes capital spent in pre-production years -3, -2 and -1, as well as associated indirect and management costs until the mine ramps up to full production. Pre-production capital totals $575 million.

Sustaining capital is all capital spent after full planned production. This includes the replacement of worn-out or exhausted assets. Capital related to the development of the underground mine are included in the sustaining capital estimate. Sustaining capital totals $571 million.

Table 1-5 – LOM Capital Costs in million U.S. Dollars

Area	Pre-Production Capital	Sustaining Capital
Mining	45	212
Co-Disposal Storage	34	24
Surface Infrastructure	72	3
Water Management	2	2
Concentrator & Filtration Plant	190	2
Closure	0	151

Area	Pre-Production Capital	Sustaining Capital
Water Treatment	134	117
Electrical Engineering	17	2
Indirect	15	0
Contingency	66	59
Total Capital	575	571

Total operating costs are estimated at $46.8/t ore, totalling $2 432 million over the life of the project.

Table 1-6 – LOM Operating Costs

Area	LOM Average ($/t ore)	LOM Total (Million $)
Mining	26.1	1 356
Co-Disposal Storage	2.0	105
Water Management	0.2	10
Concentrator & Filtration Plant	13.4	699
Water Treatment	2.1	108
Site G&A	3.0	154
Total	46.8	2 432

1.18 FINANCIAL SUMMARY

The Ikkari project generates positive post-tax financial results. At a gold price of $2 150/oz, the post-tax NPV5% is $1 680 million, IRR is 38% and the payback from start of production is 2.2 years. The gold price was taken as an average of the long-term bank consensus price as at January 2025.

Table 1-7 – Executive Financial Summary

Total	Unit	Years 1 to 10	LOM
Milled Tonnes	Mt	35	52
Mineral Process Plant Throughput	Mtpa	3.5	2.6
Strip Ratio	W:O	3.8	2.6
Average Processed Grade	g/t	2.13	2.09
Average Metallurgical Recovery	%	95.8%	95.8%
Average Gold Production per Annum	koz	227	167

Total	Unit	Years 1 to 10	LOM
Recovered Gold	koz	2 273	3 346
Initial Capital	$M	(575)	(575)
Sustaining Capital	$M	(261)	(571)
Operating Costs	$M	(1 323)	(2 432)
Selling Expenses	$M	(46)	(67)
Total Cash Cost	$/oz	(603)	(747)
Sustaining Capital	$/oz	(115)	(171)
AISC*	$/oz	(717)	(918)

*Includes selling costs.

Table 1-8 – Financial Results for Varying Gold Prices

Gold Price [$/oz]	NPV [$ 000]	IRR [%]	Payback [Years]
1 500	617 427	21	3.7
1 700	944 489	27	3.1
2 000	1 435 034	35	2.4
2 150	1 680 307	38	2.2
2 650	2 497 882	49	1.7
3 000	3 070 185	55	1.4

1.19 IMPLEMENTATION

A WBS Level 2 implementation schedule (see Section 24.1 or Appendix 2) was prepared considering key industrial components, schedule drivers, and simplification of the critical path. The project timeline spans multiple phases, with a construction phase of approximately 2.5 years and an operational phase extending 20 years.

Key schedule drivers have been identified as follows:

- **Permitting**: The permitting process is a critical path item. This process cannot begin until the Environmental Impact Assessment (EIA) is sufficiently defined. Ongoing engagement with permitting authorities is essential to refine this timeline;
- **Process Plant Construction**: The construction of the Mineral Processing Plant is the focal point for the construction phase. Detailed sequencing of the plant's infrastructure, including the ROM wall and primary crusher building, will be critical to reducing overall construction duration;
- **River Diversion and Pit Dewatering**: The completion of river diversion works is necessary before pit dewatering can commence. The time required for pit dewatering is assumed at 6 to 12 months and will be defined further in future studies;
- **Water Treatment Plant**: The construction of the water treatment plant and discharge pipeline is closely linked to the mineral processing plant. The water treatment plant is expected to take 2 years to build, posing potential schedule risks that will require further analysis; and
- **Site Establishment Material Volume Balance**: The movement of materials, including cut and fill operations, is essential to site development. Optimising material use and managing local material sourcing are critical to avoiding delays and unnecessary costs, especially for the Mineral Process Plant foundation and ROM pad.

The project is assumed to follow an Engineering, Procurement, and Construction Management (EPCM) model, where the contractor oversees engineering, procurement, and construction activities while the client retains project execution control. Alternative contracting models, such as EPC (Engineering, Procurement, and Construction), will be considered for their potential to provide a turnkey solution with fixed timelines and costs.

The selected procurement and contracting strategy will have significant implications on cost and schedule management, risk allocation, and the ability to meet the critical path targets, including the timeline to first gold production.

This Pre-feasibility study outlines a robust framework for the timely and cost-effective implementation of the mining project while highlighting areas for further study and refinement to optimise the project schedule and minimise risks.

1.20 QUALIFIED PERSON CONCLUSIONS AND RECOMMENDATIONS

1.20.1. MINERAL RESOURCE ESTIMATE

It is the QP's opinion that the exploration, drilling and analytical procedures used by Rupert Resources to collect geological data are consistent with industry practises and CIM Mineral Exploration Best Practise Guidelines (November 2018) and that the data is suitable for the reporting of Mineral Resource estimates as summarized in this Technical Report.

This MRE has been prepared in accordance with NI 43-101 following the requirements of Form 43-101F1. The methodology used to determine the MRE is consistent with the CIM Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines (November 2019) and was classified following CIM Definition Standards for Mineral Resources & Mineral Reserves (May 2014).

The QP for this Mineral Resource estimate is Mr. Brian Thomas, P.Geo., an independent QP, as defined under NI43-101 and an employee of WSP Canada Inc. based in Sudbury, Ontario, Canada. The effective date of this Mineral Resource estimate is October 24, 2023.

The QP has taken reasonable steps to make the block model and MRE representative of the project data, but notes that there are risks related to the accuracy of the estimates related to the following:

- The assumptions used by the QP to prepare the data for resource estimation;
- The accuracy of the interpretation of mineralisation;
- Estimation parameters used by the QP;
- Assumptions and methodologies used to estimate SG;
- Orientation of drill holes; and
- Cut-off grade and related assumptions of commodity prices, mining costs and metallurgical recovery.

For these reasons, actual results may differ from the reported MRE.

1.20.2. MINERAL RESERVE ESTIMATE

The Mineral Reserves were estimated in accordance with the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines. The disclosure of the Reserve Estimate uses the NI 43-101 guidelines and has excluded the use of Inferred Mineral Resources.

Reserves are derived from the proposed open pit and underground mining areas. The open pit is based on a conventional truck and shovel operation with conventional grade control processes. Underground mining implements a Long Hole Open Stoping (LHOS) mining method with a hybrid of waste development (sun surface infrastructure) rock or paste backfill manufactured from the mineral process plant tailings.

It is the Mineral Reserve QP's opinion that the Mineral resource Estimate handover to the mining design team was consistent with industry practises and CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines and that the MRE data was suitable for the reporting of Mineral Reserve estimates as summarized in this Technical Report.

The QP for this Mineral Reserve estimate is Mr. Timothy Daffern, B.Eng., C.Eng., ACSM., QMR, FAusIMM, FIMMM, M.CIM., M.SME (USA), an independent QP, as defined under NI 43-101 and an employee of WSP UK. based in England, UK. The effective date of this Mineral Reserve estimate is January 23, 2025.

The QP and the mining design team have taken reasonable steps to ensure the MR is representative of the available date, but notes that there are risks related to the accuracy of the MR estimates related to the following:

- Cut-off grade and related assumptions of commodity prices, mining costs and metallurgical recovery.

For these reasons, actual results may differ from the reported Mineral Reserve statement.

1.21 RISKS

A full risk register is shown in Appendix 3. The key risks and mitigations are:

- Water Management: The works required for dewatering the open pit area, river diversion and site drainage are managed by designed suitable contingent capacity systems for water capture, treatment and discharge to address these risks;

- A review of likelihood of a site event resulted in a 1:1 000 and 1:200 annual exceedance probability of spilling for the Co-Disposal and Raw Water Ponds respectively. This is equivalent to a 1:11 probability of one or more spills occurring over a 20-year life of mine for the Raw Water Pond. The mitigation of any such spill is the significant dilution of contaminants during such a rare spill event due to the large proportion of hydrological water in the system, which would assist in mitigating the environmental impact of such a spill event. At the next project stage, a detailed stochastic water balance and contaminant mass balance will be required to confirm acceptable spill frequencies and pond sizes for permitting.
- Permitting: Obtaining the necessary construction, environmental and operational permits in a manner that minimises the impact on the timeline to first gold pour will require a focus on the EIA, stakeholder engagement and 'right first time' permit application documents. RR have mitigated this risk ith proactive parallel development of the site EIA and community engagement.
- Project Implementation: There are several significant schedule items with the potential to delay the construction which have been assessed, and mitigation is through a thorough critical path analysis, this will be developed to a granular level through detailed planning in the next stage of project development; and
- Closure: Closure and post-industrial activities are linked to potentially long-term environmental impacts and financial commitments. Suitable and sufficient planning for site closure will underpin permitting, mitigate the potential for closure cost escalation and fulfil responsible industrial and mine site obligations.

The Ikkari Project development team have completed extensive optioneering trade-offs and these are included in an optimal form based on the Prefeasibility Study constraints, however, there remain several opportunities going forwards to add value to the project:

- Modification of co-disposal geomorphology and optimising waste rock and tailings layering to reduce mine closure costs; and
- Optimised configuration of site infrastructure, layout and underground access to enable a decreasing cut-off grade strategy in operations.

2 INTRODUCTION

2.1 ISSUER AND TERMS OF REFERENCE

This Technical Report (Pre-Feasibility Study, AACE Class 4) was prepared for Ruper Resources Ltd. and its Finnish subsidiaries. The purpose of this Technical Report is to set out all technical, engineering and scientific works completed on the Ikkari mineral property. The report is a compendium of studies culminating in a Pre-Feasibility Study as defined by the Canadian Institute of Mining and Metallurgy. The mineral resource and mineral reserve estimates were produces in accordance with the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (MRMR Best Practice Guidelines).

This report was prepared by WSP Finland, WSP UK and Ireland, WSP Canada, which are trading names for the WSP plc global entity.

The WSP scope of work was as follows:

1) To update the Mineral Resource Estimate;
2) To produce mining Mineral Reserve Estimate;
3) Estimate all plant, property and equipment necessary for mining, mineral processing suitable and sufficient for the production of Gold Dore which can be refined to a saleable Gold bullion;
4) Develop capital and operating cost estimates for the mine, mineral processing facility, site infrastructure, water treatment, mine wastes and mineral processing tailings; and
5) Coordinate the preparation of this report.

2.2 GENERAL

Rupert Resources Ltd. is a gold exploration and development company listed on the TSX Exchange. The Company is focused on making and advancing discoveries of scale and quality with high margin and low environmental impact potential. The Company's principal focus is Ikkari, a new high quality gold discovery in Northern Finland.

Ikkari is part of the Company's "Rupert Lapland Project," which also includes the Pahtavaara gold mine, mineral process plant, exploration permits and concessions located in the CLB of Northern Finland. The Rupert Lapland Project is located within the CLB, part of the Fennoscandian shield, which hosts 1 700 known incidences of mineralisation in Finland, Sweden, Norway and Russia including around 80 mines.

The town of Sodankylä provides most of the support services for the Rupert Lapland Project including the use of an accredited assay laboratory as well an additional sample preparation laboratory. The municipality of the same name has a population of 8 137 and its industrial base is dominated by small businesses involved in forestry, agriculture and manufacturing. Mining plays an increasingly important role in the local economy with the Kevitsa Mine, owned by Boliden, the largest single employer in the municipality.

The town of Rovaniemi in Finland is located approximately 120 km south-southwest of the Ikkari site. Rovaniemi has a population of approximately 60 000 inhabitants and is the administrative centre of Finnish Lapland.

Following publication of a preliminary economic assessment (PEA), set out in NI-43-101 disclosure format filed on SEDAR on 17th March 2023, WSP were selected to lead the Pre-Feasibility Study ("PFS") solely focussed on the Ikkari project.

The PFS report, was supervised and prepared by Qualified Persons following the guidelines of the NI 43-101 disclosure requirements and in conformity with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Standards on Mineral Resources and Reserves.

2.3 SITE VISITS

A personal inspection of the project site was conducted by Mr. Brian Thomas, P.Geo., from July 11th to 13th, 2023, by Dr. Peter Bolt from August 14th to 18th, 2023 and by Mr. Timothy Daffern, B.Eng (Mining) C. Eng., MBA, FAusIMM, on 6th February 2025. A summary of QP site visit details is provided in Table 2-1. All QPs are employees of WSP.

Mr. Matti Islander visited the site and logging facilities and undertook geotechnical surveys. Mr. Peter Kerry visited the prosed areas for the mining waste facility and plant site on the 29th and 30th of August 2023. This was a walkover survey that included photographs. No samples were taken. Mr. John Preston and Mr. Sam Murray visited the site to undertake a fluvial geomorphology baseline survey on the 24th and 25th October 2023.

Table 2-1 – Site Visit personnel

Name	Company	QP	Site Visit Dates	Site survey
Timothy Daffern	WSP	Yes	6th February 2025	General walk over
Brian Thomas	WSP	Yes	10th - 13th July 2023	Geological survey and general walk over
Dr P. Bolt	Retired WSP	Yes (Retired)	14th - 18th August 2023	General walk over
Matti Islander	WSP	No	14th - 18th August 2023	Geotechnical survey
Peter Kerry	WSP	No	29th – 30th August 2023	General walk over
John Preston	WSP	No	24th – 25th October 2023	Fluvial geomorphology baseline survey
Sam Murray	WSP	No	24th -25th October 2023	Fluvial geomorphology baseline survey

2.4 QUALIFIED PERSONS

The Qualified Persons (QPs) responsible for this report are:

- Mr Brian Thomas, Mineral Resource Estimate;
- Mr Timothy Daffern, Mineral Reserve Estimate;

2.5 AUTHORS

The following professionals have completed the relevant chapters of PFS.

Author	Professional registration	Contributing chapters	Peer reviewed
Timothy Daffern (BEng Mining CEng, MBA)	FAusIMM, MCIMM, FIOMMM, QMR	1, 15, 19, 21, 22	1-26
Brian Thomas (BSc Geol)	P.Geo.	1, 5-12, 14, 25, 26	
Alex Verth (MPE, BSc)	MAusIMM	1, 15, 16, 21, 22, 25, 26	
Tomas Bolsöy (Grad Dip)		16	
Matti Islander (MSc Eng)		15, 16, 21, 24, 27	
Tuomas Rantanen (MSc Eng)		16	
Ko Korenromp (MSc)	P.Eng	1,16, 25, 26	
Thomas Skocir (BEng)		1,16, 21, 25, 26	
Isabelle Larouche (BSc)	P.Eng (OIQ & NAPEG)	1, 13, 17, 21, 25, 26	
Ryan Sweetman (Beng, MSc)	MICE CEng	1,18, 20, 21, 25, 26	
Gareth Digges La Touche (BSc, MSc)	FGS, CGeol, EurGeol	1,18, 20, 21, 25, 26	
Marie Raffin (PhD, MEng, MSc, BSc)		1, 18, 20, 21, 25, 26	
Tomas Rönnbäck		18	
Peter Kerry (BEng, CEng, MICE)		1, 18, 20, 21, 25, 26	
Neeltje Slingerland (PhD, MLA, BSc)	P.Geo (AB, BC), OALA/CSLA	1, 20, 21, 25, 26	
Alex Duff (MSc, BEng)	CEng MIMMM	1, 24, 25, 26	
Talvikki Rundqvist (LLM, MSc Geol)		1, 4, 5, 20, 23	
Rupert Resources Ltd.		4, 5	

2.6 SOURCES OF INFORMATION

As this mineral property is being developed by Rupert Resources plc and its subsidiaries there has been a large number of consulting organisations involved in developing information and data points. The following is a non-exhaustive list of companies which have contributed reports:

Mine Environment Management Ltd., Envineer Oy, Piteau Associates Engineering, Grinding Solutions Ltd, Stress Measurement Company Oy, Geolabs Ltd., Terratec Geophysical Services GmbH, SRK Finland Oy, Afry Finland Oy, Tektonik Consulting Ltd, Know Flow AB.

The Rupert Resources Ltd Preliminary Economic Assessment was a significant source of information for this Technical Report. This Technical Report is based on the following data and pre-existing reports:

- 2023 PEA;
- 2023 Mineral resources estimate;
- Rupert Resources database of surface drill holes that included: Au and multi-element assays, specific gravity (SG) measurements, and descriptions of lithology, structure, mineralisation, alteration;
- Drill hole collar survey data and down-hole survey data;
- Mineralisation models and lithology models;
- Topographic and bedrock surfaces;
- Property boundary;
- QA/QC summary data and graphs;
- Assay certificates;
- Metallurgical test work completed by third parties;
- Rock mechanical laboratory test completed by third parties;
- Environmental studies completed by Rupert Resources and third parties;
- Ground water modelling completed by third parties; and
- Third party reports.

All sources of information for this study are in Chapter 27.

2.7 UNITS OF MEASUREMENT AND CURRENCY

All units of measurement used in this technical report are in metric unless otherwise specified.

All currency is in US dollars unless otherwise noted.

3 RELIANCE ON OTHER EXPERTS

This Technical Report ("TR") has been prepared by WSP for Rupert Resources Ltd. The conclusions, opinions, and estimates contained herein are based on the following:

- Information available to WSP at the time of TR preparation.
- Assumptions, conditions, and qualifications explicitly outlined in this TR.
- Data, reports, and other information supplied by Rupert Resources Ltd. and other third-party sources as noted herein.

This TR includes technical information which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and may introduce a minor margin of error. The Qualified Persons ("QP's") do not consider these margins to be material to the findings or conclusions of the TR.

The QP's who prepared this TR have relied on information provided by the following sources:

- Rupert Resources Ltd.:
  - Information regarding mineral tenure, surface rights, ownership details, agreements, taxation, royalties, environmental obligations, flora and fauna studies, reindeer husbandry and ancient remains, permitting requirements and applicable legislation relevant to the Ikkari Project, provided in items 4, 5, 6 and 20;
  - Employees and technical experts from Rupert Resources Ltd. have provided guidance, interpretation and analysis of reports by third parties, which WSP has relied upon in carrying out computations, design and definition studies;
- Piteau Associates (Tetra Tech Inc.): Information regarding groundwater model development and dewatering evaluation, provided in item 20;
- Know Flow AB (Tomas Bolsöy): Information regarding the mine ventilation, provided in item 16; and
- Ramboll Group: Information on the access road to the mine site, provided in item 18.

The QP's have fully relied on the information provided by these above listed experts and sources. The QP's consider this reliance reasonable, given the expertise and qualifications of the sources. The QP's have not independently verified the information in these specific Items and disclaim responsibility for its accuracy.

Except as required by Canadian securities laws, any use of this TR by third parties is strictly at their own risk.

4 PROPERTY DESCRIPTION AND LOCATIONS

4.1 LOCATION OF IKKARI GOLD DEPOSIT

The Ikkari Gold Deposit is located within Rupert Resources' "Rupert Lapland Project" exploration licences, which occur in the province of Lapland, Northern Finland, as shown in Figure 4-1.

Figure 4-1 – Location of Rupert Lapland Project in Northern Finland

More locally, the project occurs across an area surrounding the Rajala village in the municipality of Sodankylä. The Ikkari Gold Deposit occurs in the westernmost extents of the Rupert Lapland Project, approximately 30 km northwest of Sodankylä town centre in northern Finland, 10 km NNW of Jeesiö village and 22 km west-southwest (WSW) of the Pahtavaara Mine, a gold mine currently on care and maintenance within the Rupert Lapland Project tenement package (for coordinates see Table 4-1), as shown in Figure 4-2 and Figure 4-3.

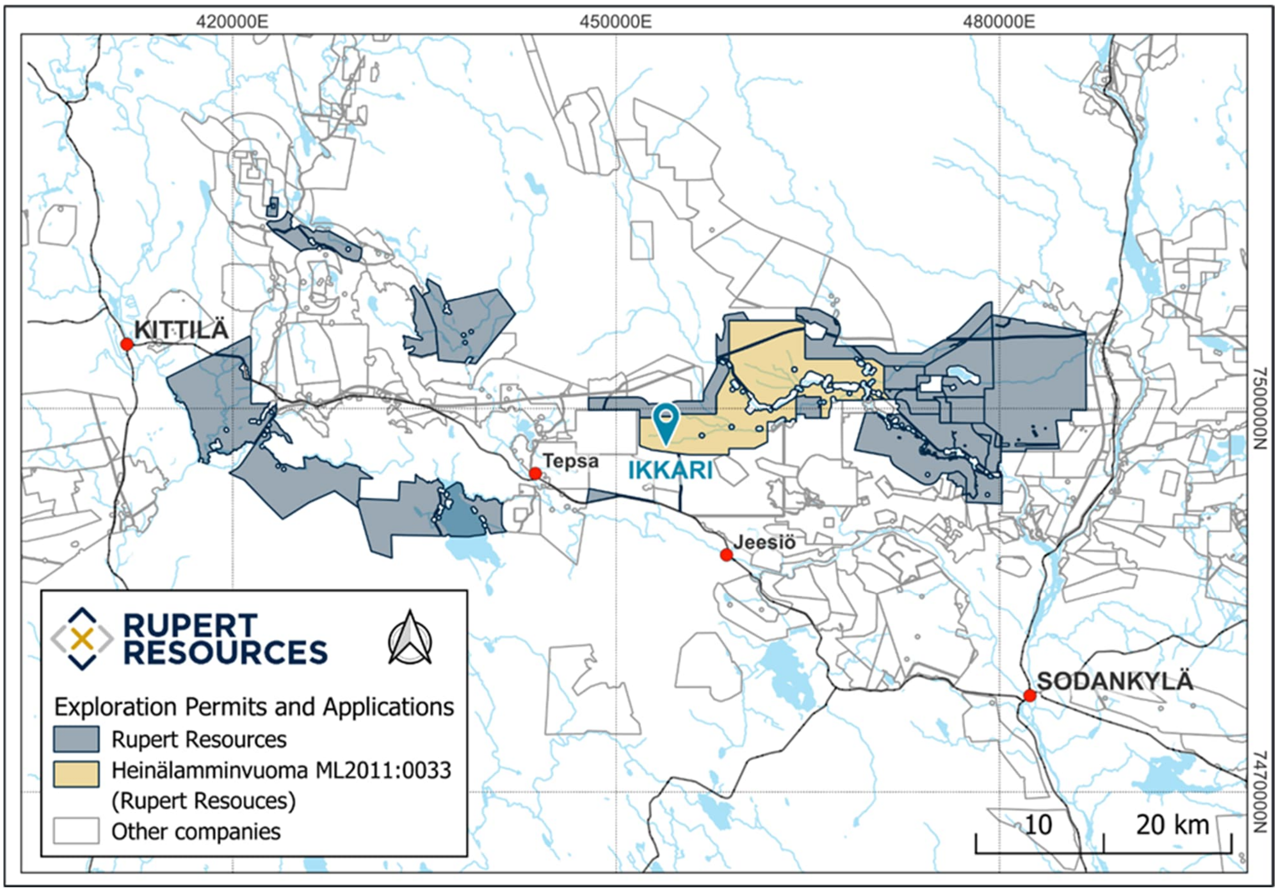


Figure 4-2 – Location of the Rupert Lapland Project Area

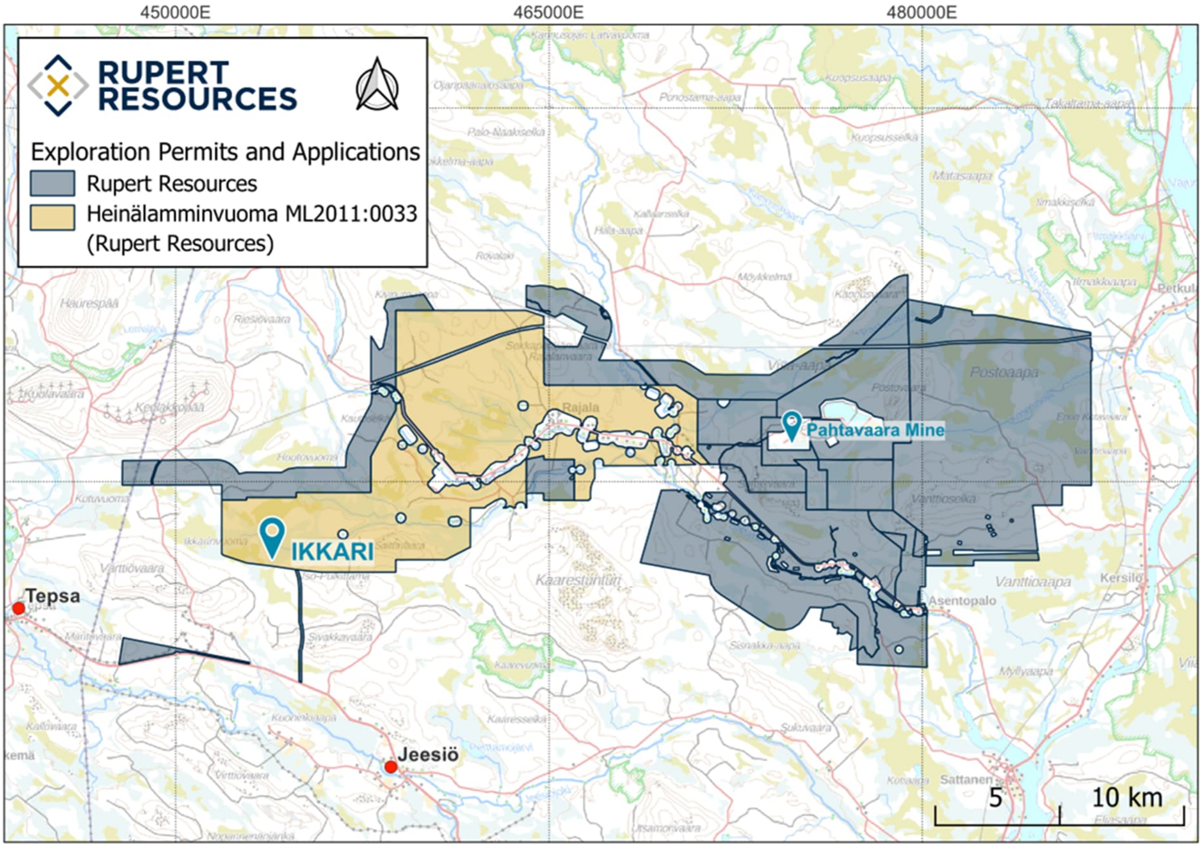


Figure 4-3 – Location of the Ikkari Deposit

Table 4-1 – Deposit Coordinates

Deposit	Reference Grid	Easting	Northing
Ikkari	ETRS-TM35FIN	454100	7496950

The Ikkari deposit lies on the eastern extreme of the Sirkka Line, a tectonic structure that traverses northern Finland, along which some 25 to 30 gold deposits / occurrences exist. Ikkari is situated at the margins of a low-lying aapa-mire, comprising broad wetlands to the north and west, and is sparsely forested.

The landscape across the Ikkari deposit area is predominantly flat with an elevation of approximately 225 m asl and rising slightly towards the southeast and the margins of the Iso-Pulkittama hill, which has a maximum elevation of approximately 300 m asl. The overburden cover of glacial till deposits is generally between 10 m to 40 m thick and rock outcrop is very limited across the exploration licence area. In most parts of the deposit area, the ground water table is typically located close to the ground surface.

4.2 PROPERTY OWNERSHIP

The Rupert Lapland Project area, in which the Ikkari deposit occurs is comprised of a contiguous package of mining licences, exploration permits, and exploration permit applications totalling an area of 340.6 km². These licences are 100% owned by Rupert Resources Ltd. through its local, Finnish subsidiaries. Additional permits elsewhere in the Central Lapland Belt, contribute to a grand total of 490 km² (see Table 4-2 for component parts, expiry and annual fees). The mineral resource at Ikkari is contained within the existing valid exploration permit Heinälamminvuoma - ML2011:0033, with an area of 84 km². Both Rupert Finland Oy and Rupert Exploration Finland Oy are wholly owned subsidiaries of Rupert Resources Ltd., a company incorporated in British Columbia, whose office is at 82 Richmond Street East, Suite 203, Toronto, Ontario, Canada, M5C 1P1.

The rights conveyed to the landholder are defined in the Mining Act of Finland (621/2011) and summarised as in the following items.

4.2.1. MINING PERMIT

The Finnish Mining Act (621/2011) states as follows of a mining activity subject to a permit and its legal effects:

"The establishment of a mine and undertaking of mining activity are subject to a permit (mining permit)." (Mining Act 621/2011, Section 16)

A mining permit entitles the holder to exploit:

1) the mining minerals found in the mining area;
2) the organic and inorganic surface materials, excess rock, and tailings generated as a by-product of mining activities (byproduct of mining activity); and
3) other materials belonging to the bedrock and soil of the mining area, insofar as the use thereof is necessary for the purposes of mining operations in the mining area.

Moreover, the mining permit entitles its holder to perform exploration within the mining area in accordance with the provisions of Section 11, and the more specific conditions specified in the mining permit. (Mining Act 621/2011 Section 17)

According to Finnish legislation, and Mining Act (621/2011, section 168) a decision can be enforced, when other compulsory permit decisions have become legally valid and the collateral according to Mining Act has been provided:

"Measures based on an exploration or gold panning permit may be initiated once the entitling permit decision has become legally valid, and the collateral prescribed in the permit in question has been provided. However, if performance of the measures in question is subject to a permit required under other legislation, the measures may only be initiated once the permit decision in question has become legally valid, or the initiation of activity has been authorised by the authority competent in the matter.

Measures based on a mining permit may be initiated when:

1) the mining permit decision is legally valid;
2) the terms issued in the mining permit concerning initiation of measures have been fulfilled;

3) the redemption decision referred to in section 84 is legally valid, and the final compensation determined for the permit holder in the decision has been paid;
4) collateral has been provided as specified in the mining permit;
5) the permits significant for the measures in question that are required by other legislation are legally valid, or the authority competent in the matter has authorised initiation of activity.

However, construction of a mine and productive activities there shall not be initiated before the mining safety permit has become legally valid. An appeal concerning compensation as ordained in the proceedings establishing a mining area shall not prevent the initiation of measures based on a mining permit and mining safety permit."

However, the Mining Act also states that an authority can, upon the request of an applicant order in the decision of an enforcement of a decision regardless of judicial review:

"On justifiable grounds, the mining authority can, upon the request of the applicant, order in the decision on extending the validity of an exploration permit or gold panning permit or in the decision on an exploration permit, gold panning permit, mining permit or mining safety permit that measures specified in the permit can be undertaken according to the permit decision regardless of any request for a judicial review. The mining authority may also issue an enforcement order for a part of the decision and set a date for the start of enforcement. The provisions of this subsection do not apply to a mining permit concerning the production of uranium or thorium." (Mining Act 621/2011, Section 169, amended 505/2023).

At the current time, the Ikkari deposit ("Heinälamminvuoma") does not hold any legally valid Mining permits, no Mining permit matter is either filed within an authority (Finnish Safety and Chemicals Agency). However, Rupert Resources Ltd. holds a legally valid mining permit for the Pahtavaara deposit, also known as the historically operating Pahtavaara Mine.

4.2.2. EXPLORATION PERMIT

"Pursuant to an exploration permit, the permit holder has the right, on the permit holder's own land and that owned by another landowner, in the area referred to in the permit (exploration area), to explore the structures and composition of geological formations and to conduct other exploration in order to prepare for mining activity and 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.

The holder of the exploration permit may 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 does not authorise exploitation of the deposit." (Section 10).

However, "-- if a mining permit is applied for with respect to a deposit located within an exploration area, the exploration permit holder shall have priority to the mining permit if the permit holder submits an application for a mining permit in accordance with the provisions laid down in section 34 prior to the expiry or cancellation of the exploration permit." (Section 32).

The prerequisites for the granting of the mining permit require the deposit to be "exploitable in terms of size, ore content, and technical characteristics." (Section 47).

The validity of an exploration permit may be extended for a maximum of three years at a time. In total, the permit may remain valid for a maximum of 15 years. (Section 61).

The Heinälamminvuoma exploration permit, where the Ikkari deposit occurs, is currently in year 7 of a possible 15-year validity period. No additional permits are required to continue with the ongoing engineering studies and exploration work as set out in the recommendations for further work, Chapter 26. To exploit the Ikkari deposit Rupert Resources will require a mining permit covering the deposit and associated infrastructure.

4.2.3. RESERVATION

“For the purpose of preparing an application for an ore prospecting permit, an applicant may reserve an area for themself by submitting a notification to the mining authority about the matter (reservation notification). A privilege based on a reservation notification becomes valid once the reservation notification has been submitted in compliance with the provisions laid down in Section 44 of the Mining Act (621/2011) and there is no reason, as specified in the Mining Act, for the rejection of the reservation. The validity of the privilege expires when the decision made by the mining authority on the basis of the reservation notification (reservation decision) expires or is cancelled.” (Section 32).

The reservation does not entitle the applicant to perform exploration. Instead, the reservation grants a privilege regarding the submission of an exploration application. (Section 32).

A reservation decision shall remain valid for a maximum of 12 months after issuing of the reservation notification. (Section 76).

At the current time Rupert Resources and its subsidiaries do not hold any valid reservations.

Table 4-2 – Land Components of the Rupert Lapland Project

Type	Code	Status	Name	Company	Area (km²)	Granted	Expires	Fee Eur/ha
Mining Licence	KL2018:0011, 3921	Valid	Pahtavaara	Rupert Finland Oy	3.86	14/09/1993	N/A	100
	KL2013:0001-01	Valid	Pahtavaara laajennus	Rupert Finland Oy	0.35	12/09/2013	Review after 10 years	100
Sub total					*4.21*			
Exploration Permit	**ML2011:0033-03**	**Valid**	**Heinälamminvuoma**	**Rupert Exploration Finland Oy**	**83.89**	**20/06/2024**	**19/06/2027**	**40**
	ML2019:0024-02	Valid	Pahta NW	Rupert Exploration Finland Oy	37.77	12/06/2024	11/06/2027	30
	ML2019:0023-02	Valid	Satta SE	Rupert Exploration Finland Oy	43.04	26/04/2024	25/06/2027	30
	ML2020:0006-01	Valid	Area 51	Rupert Exploration Finland Oy	65.56	01/09/2021	08/10/2025	20
	ML2020:0007-01	Valid	Liika	Rupert Exploration Finland Oy	0.79	01/09/2021	08/10/2025	20
	ML2022:0058-01	Valid	Kuusajärvi 1	Rupert Exploration Finland Oy	42.25	02/05/2023	08/06/2027	20

Type	Code	Status	Name	Company	Area (km²)	Granted	Expires	Fee Eur/ha
	ML2012:0080-04	Valid	Liikamaa 1-4	Rupert Finland Oy	1.97	11/07/2024	10/07/2025	50
	ML2013:0013-03	Valid	Pahtarimpi 10-11	Rupert Finland Oy	5.46	11/07/2024	10/07/2025	50
	ML2012:0195-03	Valid	Pahtarimpi 2-3	Rupert Finland Oy	1.66	11/07/2024	10/07/2025	50
	ML2013:0014-03	Valid	Paskamaa 1-5	Rupert Finland Oy	4.88	11/07/2024	10/07/2025	50
	ML2011:0034-02	Valid	Paskahaara 1	Rupert Finland Oy	16.77	08/03/2022	14/04/2025	30
	ML2011:0008-03	Valid	Soretiajärvi 3	Rupert Exploration Finland Oy	0.09	20/06/2024	29/07/2025	50
	ML2012:0196-02	Valid	Soretiajärvi 4	Rupert Exploration Finland Oy	0.95	20/06/2024	29/07/2027	50
	ML2017:0080-02	Valid	Liikavaara	Rupert Exploration Finland Oy	3.71	20/06/2024	29/07/2027	30
	ML2017:0079-02	Valid	Rajala	Rupert Exploration Finland Oy	2.94	27/03/2024	03/05/2027	30
	ML2013:0012-02	Valid	Paskamaa 2b-3b	Rupert Finland Oy	0.09	11/07/2024	19/08/2027	30

Type	Code	Status	Name	Company	Area (km²)	Granted	Expires	Fee Eur/ha
	ML2019:0005-02	Valid	Satta	Rupert Finland Oy	4.54	11/07/2024	19/08/2027	40
	ML2024:0086-01	Valid	Säynä	Rupert Exploration Finland Oy	7.3	14/01/2025	13/01/2029	20
Sub total					323.66			
Joint Venture with S2 Resources*	ML2016:0056-01	Valid	Sikavaara E	Sakumpu Exploration Oy	27.45	14/12/2021	20/01/2026	20
	ML2019:0107-01	Valid	Sikavaara W	Sakumpu Exploration Oy	9.49	14/12/2021	20/01/2026	20
Sub total					36.94			
Exploration Permit Extension Application	N/A							
Sub total					0			
Exploration Permit Application	ML2021:0081-01	Application	Rako	Rupert Exploration Finland Oy	0.46	N/A	N/A	20
	ML2021:0113-01	Application	Sattanen West	Rupert Exploration Finland Oy	1.36	N/A	N/A	20

Type	Code	Status	Name	Company	Area (km²)	Granted	Expires	Fee Eur/ha
	ML2022:0025-01	Application	Jeesiö 2	Rupert Exploration Finland Oy	1.63	N/A	N/A	20
	ML2022:0071-01	Application	Kuusajärvi 2	Rupert Exploration Finland Oy	31.98	N/A	N/A	20
	ML2022:0072-01	Application	Kuusajärvi 3	Rupert Exploration Finland Oy	38.19	N/A	N/A	20
Sub total					73.6			
TOTAL					438.4			

Notes: Heinälamminvuomä exploration permit, where Ikkari occurs is highlighted in grey and shown in bold.

Key: EUR/ha = Euros per hectare

*Sikavaara E and Sikavaara W licences are held by Sakumpu Exploration Oy, a 100% held subsidiary of TSX.V listed Valkea Resources Corp. Rupert Resources entered into an option agreement in August 2021 under which Rupert Resources can earn up to 70% interest in the licences over a 6-year period. Rupert Resources is the operator of these licences.

4.3 ANNUAL FEES AND ROYALTIES

Legislation requires holders of exploration and mining permits to make annual payments to landowners on euros per hectare (EUR/ha) basis (see Table 4-3); holders of reservations are required to make annual payments to the state also on a EUR/ha basis. From 2024, a statutory mining royalty of 0.75% is payable on the value of the exploited mineral / metal. This is comprised of 0.15% payable to the landowner and a 0.6% state royalty.

Table 4-3 – Annual Royalty Payments According to Finland Mining Act 2011.

Permit Type	EUR/ha
Reservation	1*
Exploration (years 1 - 4)	20
Exploration (years 5 - 7)	30
Exploration (years 8 - 10)	40
Exploration (years 11 - 15)	50
Mining (if not active)	50 (100)

Note: *Reservation annual payment is payable to the state, not the landowner.

The Pahtavaara Mine is subject to a 1.5% Net Smelter Return (NSR) royalty that is capped at a value of US $2 M.

4.4 ENVIRONMENT

There are no designated protected areas within the exploration permits, exploration permit applications and the Pahtavaara mining licences that comprise the Rupert Lapland Project area. Additionally, there are no designated protected areas within the Ikkari deposit impact zone and as such there is no environmental legislation that adversely impacts the project's reasonable prospects for eventual economic extraction.

Rupert Resources submitted its EIA programme to the ELY Centre in Q1 2023. This was formally accepted by the ELY Centre in Q2 2023. Rupert Resources is currently working towards submission of the EIA report for the Ikkari project during 2025.

Rupert Resources has funded an environmental reclamation bond of EUR 640 000 and a mining bond of EUR 210 000 for the Pahtavaara Gold Mine and a further EUR 49 500 in exploration related bonds covering the Rupert Lapland Project Area.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 PROPERTY ACCESS

The airports of Rovaniemi and Kittilä has several scheduled domestic flights daily to and from Helsinki as well as international flights during the winter and summer holiday seasons. The distance from Rovaniemi to Sodankylä is 130 km by road and takes under two hours to drive. To reach Ikkari from Sodankylä, turn towards Kittilä onto main road 80. Continue to follow road 80 towards Kittilä, 4.5 km after Jeesiö village turn right to Pulkittama. Continue to follow the gravel Pulkittama road for 7.5 km where forest tracks lead directly to the exploration site. Access to the site is possible throughout the year. The Ikkari turn can also be accessed from Kittilä in the other direction and the drive time is under 1 hour. The closest rail heads are at Rovaniemi and Kemijärvi, 130km and 110km respectively south of Sodankylä.

Heavy goods vehicular access is by road from the ports of Helsinki, Hamina-Kotka or Kemi, the latter being the closest one, approximately 245km south of Sodankylä town. Roads are open year-round and regional fabrication and manufacturing vendors are available.

5.2 PHYSIOGRAPHY

The landscape was sculpted by extensive glaciers in the most recent ice age between 110 000 and 10 000 years ago. Following the last glacial period, melting ice sheets resulted in shallow lakes and extensive boggy lowlands. Broad valleys were scoured out in the direction of glacial transport, flanking low-lying hills underlain by resistant rocks. The landscape is dominated by low rolling hills and flat lowlands comprised of wetlands (bogs) and lakes. Hills are mostly covered by glacial moraine and sands and are forested, primarily with birch, pine, and spruce which are exploited by the state forestry company. Bedrock outcrops on the hills and along riverbanks is limited to some two percent or less of the project area. The Ikkari gold deposit is located at the margins of low-lying wetland (bog) terrain, cut by a small stream, rising towards a boulder-dominated, gentle slope in the south-southeast (SSE). The area in general is approximately 225 m asl. This terrain largely drains to the north and then east into the Saittajoki River and then into the Sattanen River and further into the catchment basin of the Kitinen River, and eventually the area drains into the Kemijoki River and the Gulf of Bothnia.

5.3 CLIMATE

According to Köppen climate classification, northern Finland is classified as Dfc (Continental, Subartic/boreal climate) with no defined dry season. The region has cold winters with the mean temperature of the coldest month below -3°C and short, cool summers with the mean temperature above 10 degrees centigrade (°C) for fewer than 3 months.

According to measurement data from the Sodankylä, Tähtelä weather station 40 km Southeast of the Project, the average annual temperature for the period 1991-2020 was 0.3 °C and for 2023 0.8 °C. During the summer months (June to August), temperatures are mostly between 10°C and 20°C, and during the winter months (November to April) between -2°C and -20°C based on 10-year averages from 2005 to 2015 for Sodankylä. Snow covers the terrain on an average of 183 days in the year with a maximum snow thickness varying from 0.6 m to 1.2 m in March. Bogs, lakes and rivers are frozen for four to five months of the year. Exploration work can be conducted during the winter by taking advantage of the frozen bogs for access.

Annual rainfall is around 600 mm with a monthly range between 30 mm (April) to 90 mm (July). The wettest period is June to July and the driest period from February to April. The climate of northern Finland is influenced by its arctic location between the 60th and 70th northern latitude parallels located in the Eurasian continental coastal zone. This region has characteristics of both the maritime and continental climate depending on the direction of airflow. When westerly winds prevail, the weather is warm and clear due to the airflows from the Atlantic Gulf Stream. When airflow is from the east, the Asian continental climate prevails resulting in severe cold in winter and warmer periods in summer.

The mean temperature in northern Finland is several degrees higher than that of other areas in these latitudes such as Siberia and southern Greenland due to the moderating effect of the Atlantic Ocean and the Baltic Sea.

Weather patterns in the project area and in the general region can change quite rapidly, particularly in winter, because northern Finland is in a zone of prevailing westerly winds where cooling sub-tropical and polar air masses collide. The weather systems known to have the greatest influence on the climate are the low-pressure systems originating near Iceland and the high-pressure systems drifting in from Siberia and the Azores.

The climate is not expected to have any significant impact on the Ikkari operating season, and the operations can be conducted on a year-round basis.

5.4 LOCAL RESOURCES AND REGIONAL INFRASTRUCTURE

The town of Rovaniemi in Finland is located some 130 km south-southwest of Ikkari. Rovaniemi has a population of approximately 60 000 inhabitants and is the administrative centre of Finnish Lapland. The regional technical centre of the Geological Survey of Finland (GTK) and its analytical laboratory are also located here.

The town of Sodankylä provides most of the support services for the Rupert Lapland exploration permits, including accredited sample preparation facilities operated by ALS Minerals and Eurofins Labtium as well as a fire assay facility at the Eurofins (Labtium) laboratory. ALS Minerals and Eurofins Labtium are internationally accredited laboratories and are ISO compliant (ISO, 2008 and ISO, 2005). The regional industrial base is currently dominated by small businesses involved in forestry, agriculture and manufacturing though mining is the largest single private employer in Sodankylä with the Kevitsa Mine, operated by Boliden, employing an average of 570 people with large numbers of contractors providing services from time to time. There are several hotels, shops, and restaurants which accommodate a growing year-round influx of tourists into Lapland. The region hosts a skilled work force.

Hydroelectric power in the region is considered comparatively relatively inexpensive for commercial use. A main high voltage electrical power line is present five km north of the Ikkari deposit (Figure 5-1). A substation to this power line is located 9 km from the Ikkari deposit, currently serving a commercial wind farm.

Limited surface infrastructure is currently present at Ikkari. An access road has been constructed from the Pulkittama road and a 20 kV powerline to the site laydown area, servicing two temporary facility buildings, was completed in the last twelve months. The logistical hub for exploration across the Rupert Lapland Project area, including the Ikkari deposit, is located at a purpose-built logging and storage facility 10 km to the south of Sodankylä. Rupert management and administration functions are based at an office in the town of Sodankylä.

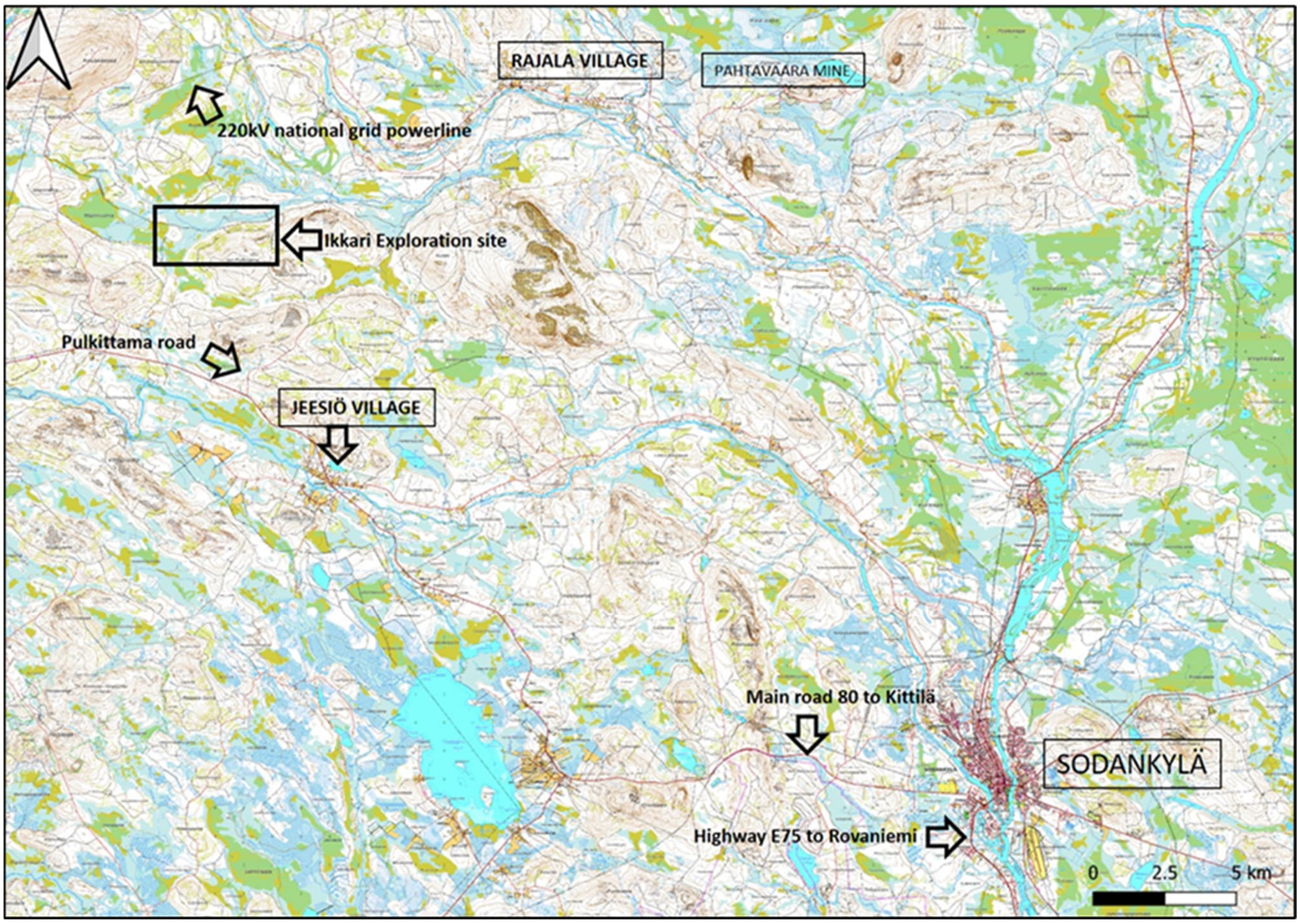


Figure 5-1 – Regional Infrastructure

6 HISTORY

Ikkari is an under-cover grass roots discovery made in March 2020. Limited previous exploration activities have been undertaken in the area prior to the work conducted by Rupert Resources during 2019 to present.

Prior to joining the European Union (EU) in 1995, all exploration in Finland was conducted by the Geological Survey (GTK) and/or Outokumpu, then a state-controlled company. The Heinälamminvuoma exploration permit on which the Ikkari Gold Deposit is located, was applied for in 2011 by Lapland Goldminers, the then owners of the operating Pahtavaara Mine. However, no work was completed in the licence area and the exploration permit remained in the application phase. This was the first instance of a private company applying for an exploration permit over the Ikkari deposit. Lapland Goldminers operated at the Pahtavaara Mine until 2014 when the parent company in Sweden filed for bankruptcy and the operation was placed in care and maintenance. Rupert Resource Ltd purchased the operation from the administrators of Lapland Goldminers in September 2016.

The Heinälamminvuoma exploration permit has been part of the Rupert Lapland Project area since that time, although very little exploration was undertaken initially and exploration field activities were confined to the easternmost parts of the licence, adjacent to the Pahtavaara Mine itself before 2018.

6.1 PREVIOUS MAPPING AND SURFACE SAMPLING

Regional mapping has been undertaken by the GTK, but due to the limited outcrop of the region, the majority of this has been interpreted using regional geophysical surveys. Limited bedrock observations have been undertaken by GTK, largely restricted to higher ground outside of the current exploration permit boundaries.

6.2 PREVIOUS GEOCHEMICAL SURVEYS

Regionally, the Geological Survey of Finland has historically carried out limited outcrop and boulder sampling across the hills to the south and southeast of Ikkari, and Terra Mining (previous owners of the Pahtavaara Mine (1991 to 2000) undertook broad spaced till sampling also across higher ground to the south and east of Ikkari, but no sampling has been undertaken across the Heinälamminvuoma area which hosts the Ikkari deposit.

Previous geochemical sampling within the Heinälamminvuoma exploration licence area comprises only historic (1974 to 1979) till geochemistry in very broad-spaced (>1 km) lines conducted by GTK. These samples were assayed for silver (Ag), lead (Pb), zinc (Zn), copper (Cu), nickel (Ni), cobalt (Co), manganese (Mn), chromium (Cr), vanadium (V), titanium (Ti), potassium (K,) sodium (Na), calcium (Ca), magnesium (Mg), iron (Fe), aluminium (Al) and silicon (Si) and did not include assays for gold. Sample depths appear to have been within the till horizons and did not reach the bedrock contact.

Considering the wider Rupert Lapland Project area, during this sampling campaign copper anomalies were discovered in the Sattasvaara komatiites and subsequent infill sampling, including gold analysis, on a 50-m-x-100 m grid led to the discovery of the Pahtavaara Mine in 1984-1985.

Across the Rupert Lapland Project area historical till sampling comprises 426 737 samples from regional programmes conducted by GTK and previous operators at Pahtavaara (Figure 6-1).

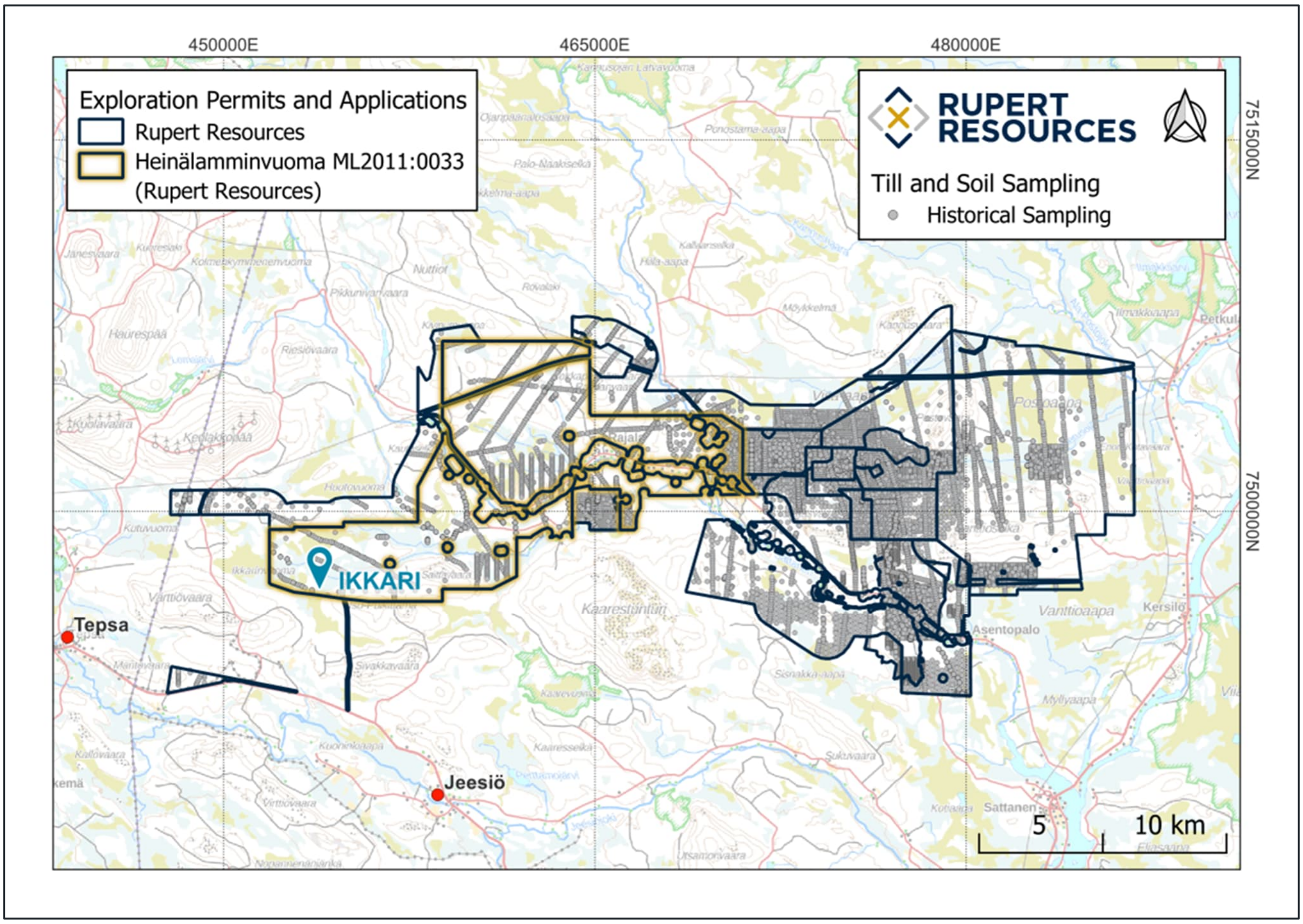


Figure 6-1 – Historical Soil and Base of Till Sampling Across the Rupert Lapland Project Area

6.3 PREVIOUS GEOPHYSICAL SURVEYS

The GTK flew a series of airborne geophysics programmes in the area in the 1970s and 1980s.

Covering the Rupert Lapland Project area, the airborne magnetic, electromagnetic and radiometric surveys were originally flown with a low-level DC-3 system between 1973 and 1979 and then resurveyed in the 1980s using the Twin Otter system. The surveys were flown at a height of 30 m with some blocks flown on N-S lines and others E-W, depending on the geological strike.

The Geological Survey has also conducted more targeted ground magnetic, slingram (electromagnetic [EM]), Induced Polarization (IP) and Very Low Frequency Radar (VLF-R) surveys in the area as well as ground gravity across much of the CLB. Scan Mining analysed the ground geophysics in 2007.

6.4 DRILLING BY PREVIOUS EXPLORERS

Within the Heinälamminvuoma exploration licence area, a total of 2 420 m of historic diamond drilling has been completed within the licence area, from 26 drill holes (Table 6-1). Very limited drilling has been undertaken by any previous explorers and most of these holes are confined to the eastern extent of the licence area (Figure 6-2).

Table 6-1 – Summary of Historic Drill Data for Heinälamminvuoma Exploration Permit Area

Company	DH Type	Holes	Metres
Outokumpu (1989 to 1991)	Diamond	5	584
Geological Survey of Finland (Pre- 1989)	Diamond	21	1 836
Total		26	2 420

No previous drilling has been undertaken at the Ikkari deposit. A review of the drill hole assay database in the region, has indicated that much of the drilling by previous explorers was selectively sampled, with few assays for gold. The only drilling to data on the Heinälamminvuoma exploration perm occurred in the far northeast of the permit.

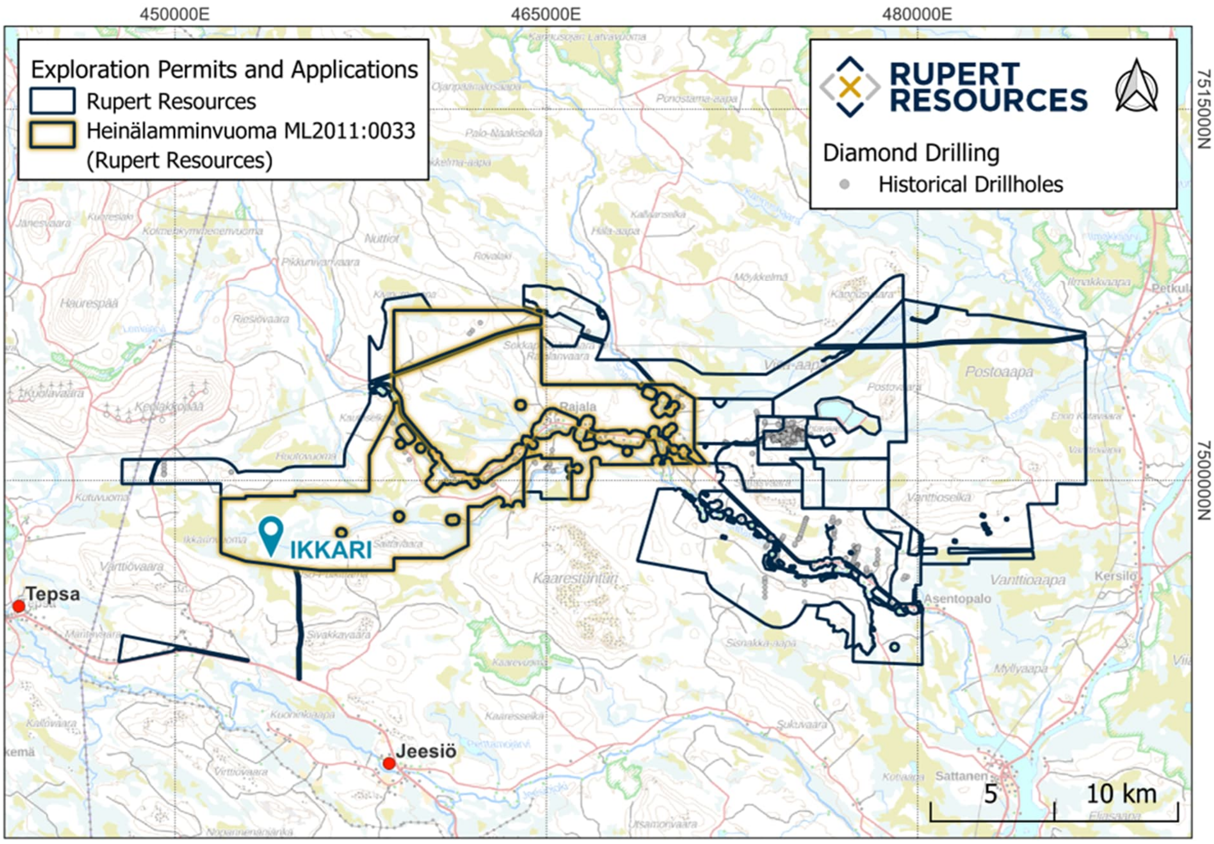


Figure 6-2 – Location of Historical Drilling on the Heinälamminvuoma Exploration Licence

6.5 HISTORICAL RESOURCE AND RESERVE ESTIMATES

Ikkari was discovered by Rupert Resources, therefore there are no historical Mineral Resource or Mineral Reserve estimates.

6.6 PRODUCTION HISTORY

The Ikkari deposit, being an exploration stage project has not had any past mining production.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 REGIONAL GEOLOGICAL SETTING

The Rupert Lapland Project area is located within the CLB, part of the Fennoscandian shield, which hosts 1 700 known incidences of mineralisation in Finland, Sweden, Norway and Russia including approximately 80 mines. The CLB has two gold mines of significance. Currently operating is Agnico Eagle's Kittilä mine, 45 km northwest of the Ikkari deposit, which produced 216 947 oz of gold in 2022 and has a remaining reserve of 3.68 Moz (Agnico Eagle, 2023). The historically producing Pahtavaara mine, 20 km east-northeast of the Ikkari deposit, mined an estimated 441 koz of gold in three periods of ownership between 1996 and 2014 (GTK, Mineral Deposit Report), and hosts an Indicated Mineral Resource of 1.9 Mt at 3.0 g/t for 180 koz together with an Inferred Mineral Resource of 2.2 Mt grading 3.1 g/t Au for 220 koz (estimated by Rupert Resources in 2022). The Heinä Central deposit, 1.5 km north-northeast of the Ikkari deposit with a Mineral Resource of 2.7 Mt at 1.8 g/t Au for 150 koz (estimated by Rupert Resources in 2022) further demonstrates the prolific gold endowment of the CLB.

Copper, along with nickel and Platinum Group Elements (PGEs) are mined from Boliden's Kevitsa mine and reported as part of the Mineral Resource estimate at Anglo American's Sakatti Project located within 45 km northeast and 35 km east from the Ikkari deposit respectively. These two deposits are examples of magmatic sulphide deposits, hosted by an ultramafic intrusive, and are distinct from the styles of mineralisation encountered within the Rupert Lapland Project area to date.

Ikkari was discovered in March 2020 and is a greenfield, undercover, orogenic gold discovery in the Paleoproterozoic CLB, Finland. The Rupert Lapland Project area lies at the eastern extreme of the Sirkka Line (Sirkka shear zone, Eilu et al. 2007), a tectonic structure that traverses northern Finland, along which some 25 to 30 gold deposits exist, either within or related to subsidiary structures along it (Figure 7-1). The shear zone is also associated with intense alteration (albitisation, sericitisation and carbonatisation) as well as anomalous gold along its entire length (Eilu et al., 2007).

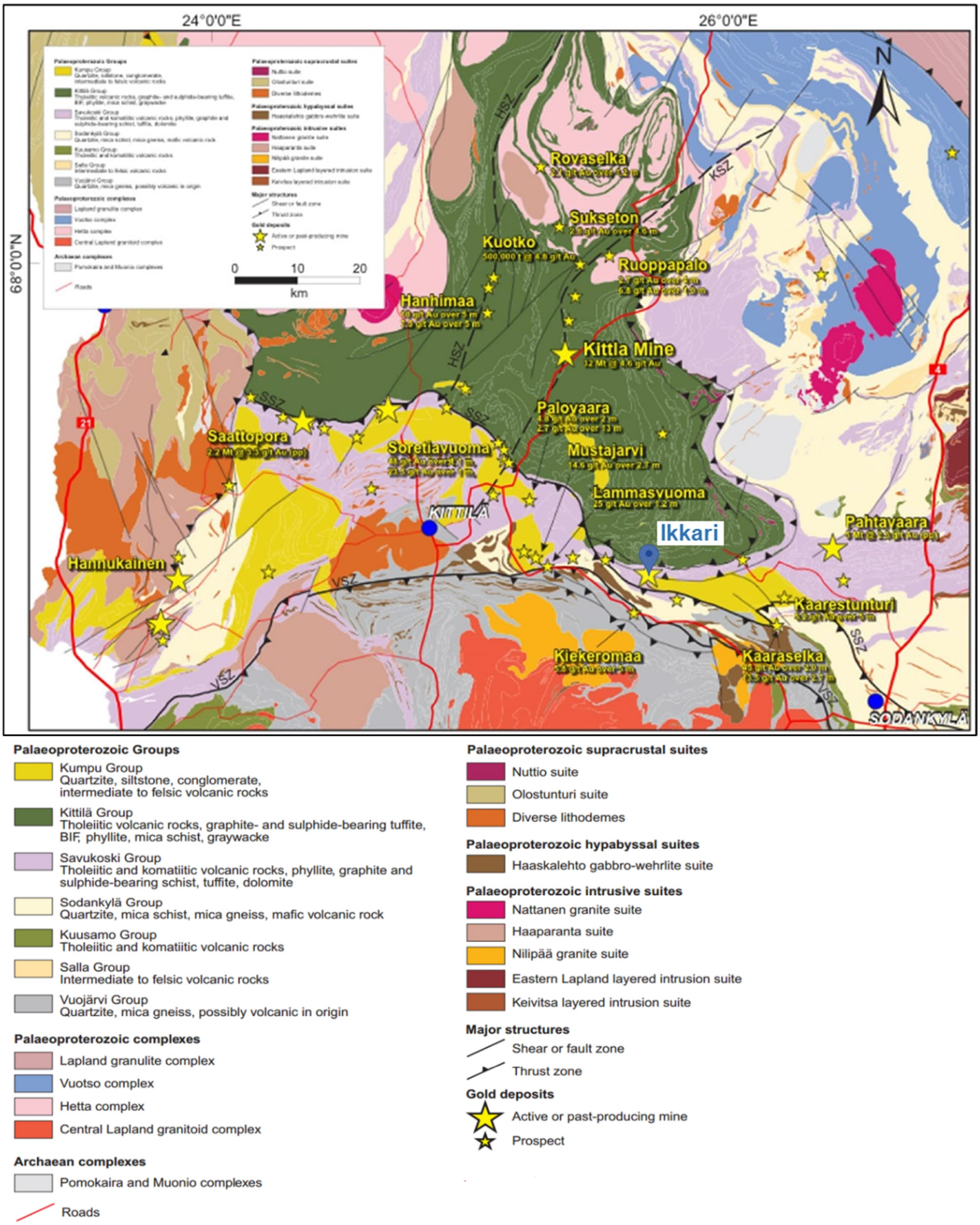


Figure 7-1 – Geological Map of Central Lapland Greenstone Belt

The Rupert Lapland Project exploration permits occur at a significant regional geological domain boundary zone, which trends predominantly east-west through the westernmost extent of the Rupert Lapland Project exploration licences (Figure 7-3). An approximately four-kilometre-wide zone of 2.05 Ga Savukoski Group rocks, comprising fine-grained mafic dominated meta-volcanic and metasedimentary rocks, including phyllite, carbonaceous shale and mafic intrusive rocks, as well as komatiites, occurs between younger (2.00 Ga) Kittilä Group rocks to the north and younger still Kumpu Group rocks (1.88 Ga maximum age) to the south (Figure 7-2). The Kittilä Group is dominantly tholeiitic metabasalts whilst the Kumpu Group is composed of molasse-type fluviatile quartzites, subarkoses and polymictic conglomerates. A stratigraphic column of the region is outlined in Figure 7-2. This zone of Savukoski Group rocks broadly corresponds with the often discussed 'Sirkka Line' structure though the exact nature and location of this is somewhat subjective.

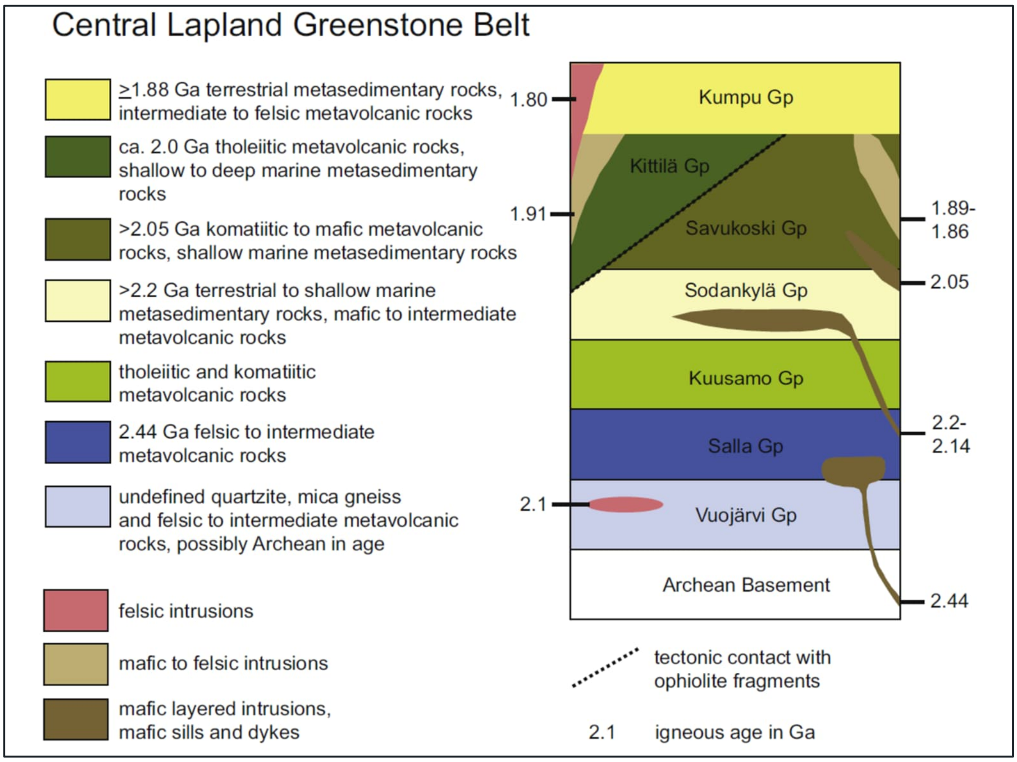


Figure 7-2 – Stratigraphy and Main Igneous Events of the CLB

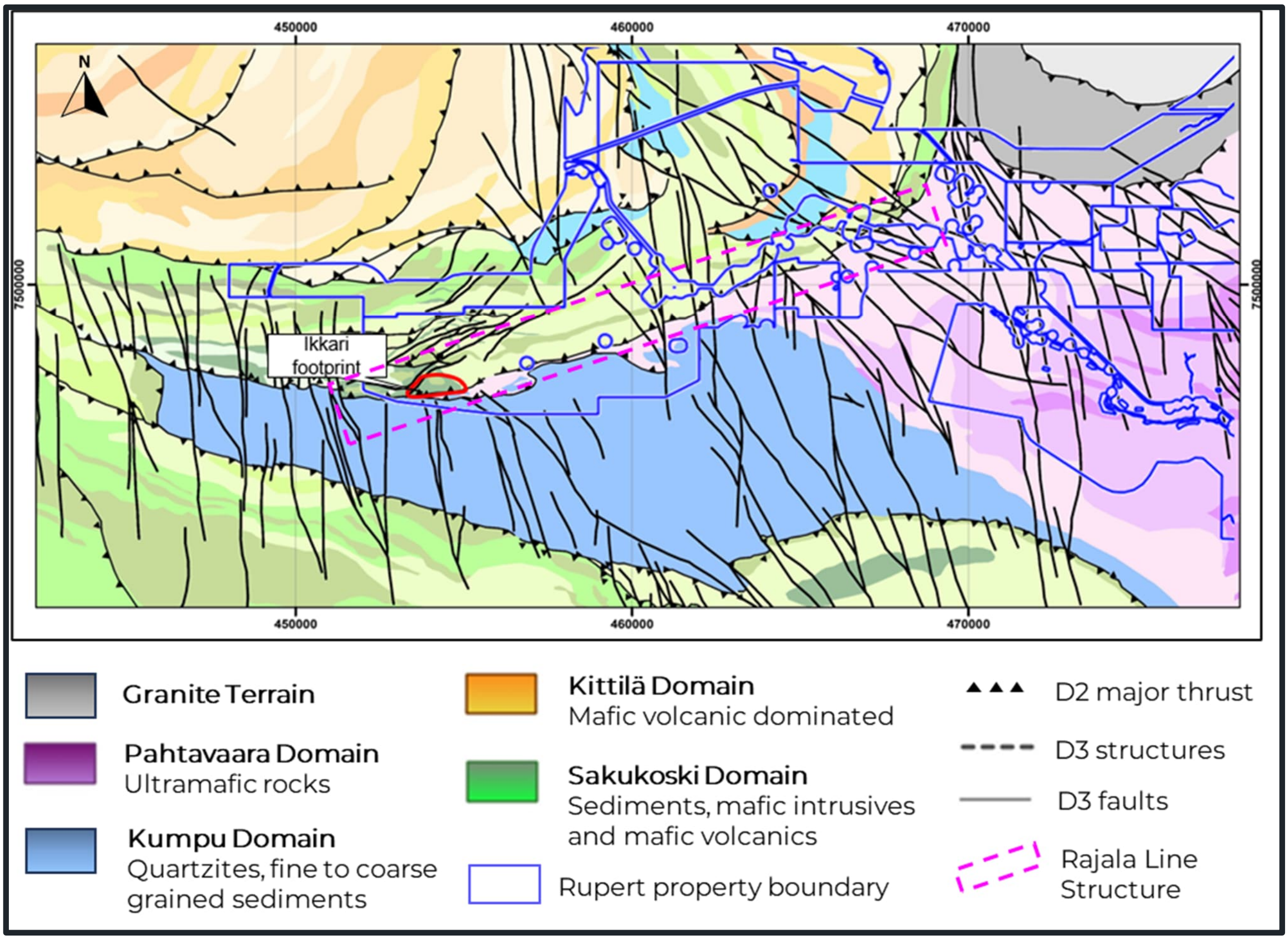


Figure 7-3 – Structural Domain Map of the Ikkari-Pahtavaara District

Regional drilling and mapping by Rupert Resources, indicate that the Savukoski Group 'corridor' across the Heinälamminvuoma permit area is primarily composed of basalts and fine-grained sedimentary rocks cut by a multitude of dominantly mafic intrusions. Relatively early major recumbent NW-SE orientated folds are interpreted to fold the basalts and sediments together during structural thickening producing the layer-parallel foliation and moderately NW plunge fold axis that are typical north of the Rajala Line.

The locally termed "Rajala Line" (Figure 7-3), a 073 trending distinct magnetic and gravity defined lineation sub parallel to the Sirkka Line west of the Rupert Lapland Project, is a 12 to 15 km ribbon of highly deformed and brecciated sedimentary rocks, nominally belonging to the Savukoski Group. The Ikkari deposit is located at the south-eastern extent of this feature though the precise relationship between this distinct geophysical feature and the genesis of the Ikkari deposit is unclear at the present time.

The highest intensity ductile deformation seen to date within the Rupert Lapland Project area occurs along the southern margin of the Rajala Line and is evidenced at both Saitta and Naattuankangas prospects, 4 and 9 km east-northeast of the Ikkari deposit as well as at the Helmi Deposit 1-1.5 km west-southwest of Ikkari (B2Gold-Aurion JV property). This ductile deformation accommodated NNW-SSE compression as the Kittilä and Savukoski Group were thrust towards the south, over the Kumpu Group sedimentary rocks producing tight upright isoclinal folds with shallow plunges.

7.2 DEPOSIT GEOLOGY

It should be noted that outcrop across most of the Heinälamminvuoma permit area and especially in the immediate vicinity of the Ikkari deposit, is virtually non-existent. Transported boulders, particularly of Kumpu Group rocks to the south of Ikkari, are not considered reliable indicators of sub-surface geology. Ikkari is a grassroots discovery, located under 10 to 25 m of transported glacial till cover.

Ikkari occupies a complex structural position between thrust imbricated Savukoski Group metavolcanics and metasediments, and synorogenic molasse-type siliciclastic strata of the Kumpu Group. At their most basic level, a 4-fold lithologic subdivision is constructed for the rock types present at Ikkari (Figure 7-4):

- Dark pyritic shales and siltstones termed the 'black shale' (intruded by gabbro) comprise the northern fault block and form the hangingwall to the mineralisation;
- a central komatiite-dominant zone with complex intercalations of texturally diverse 'felsic' facies;
- a northern, banded 'felsic' facies, intensely albite-altered in places, that pinches out in the eastern part of the deposit; and
- A southern zone comprising dominantly coarse 'felsic' siliciclastics – massive, banded, conglomeratic and typically more quartz-rich than the northern facies but which hosts intercalations of komatiite in decreasing abundance moving southwards.

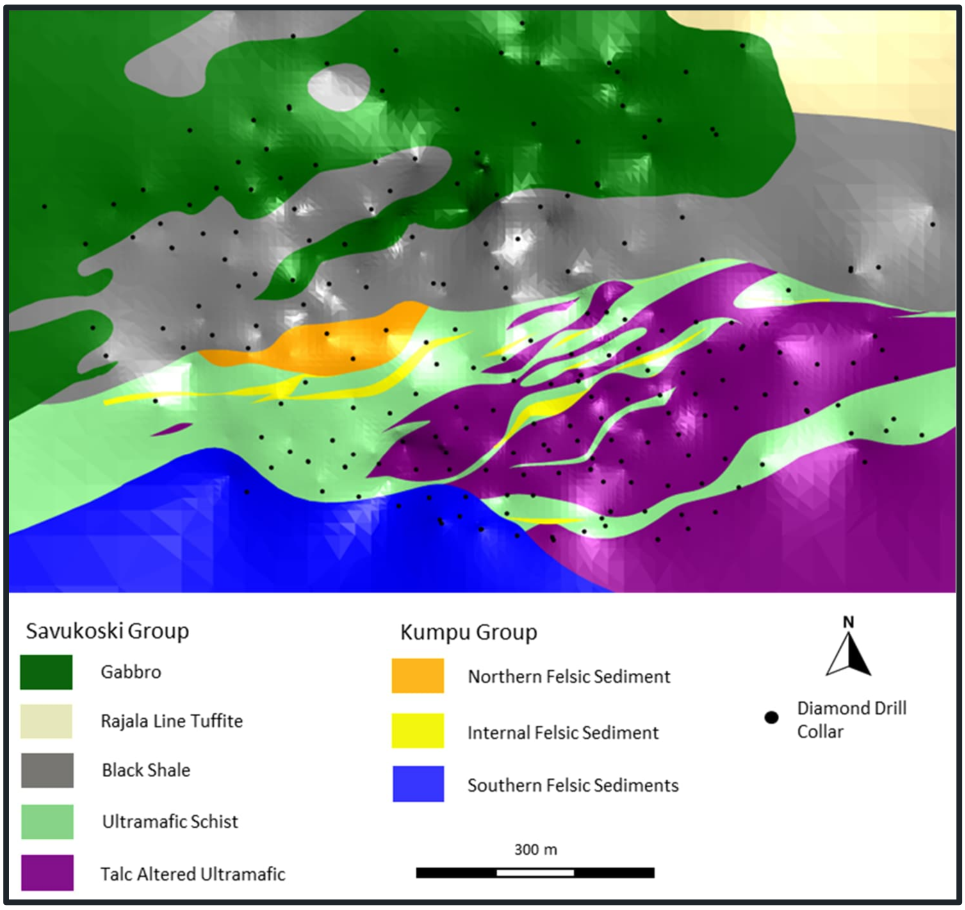


Figure 7-4 – Plan Map of Ikkari Taken From 3D Geological Model with Overburden Removed

At this most basic level these rock types are, to a greater or lesser extent, affected by iron and potassic mesothermal alteration broadly synchronous with the main phase of gold mineralisation. The alteration products are largely dependent on the protolith and the relative location in respect to the mineralisation (Figure 7-5).

Protolith	Dominant Regional Alteration	Distal Mesothermal Alteration	Proximal Mesothermal Alteration
Komatiite	Talc Chlorite Magnetite +/- Biotite Calcite	Chlorite Sericite Siderite Dolomite +/- Magnetite (Logged as MSCU)	Chlorite Siderite - Dolomite Sericite Quartz Pyrite +/- Magnetite
Felsic (Intercalated)	Muscovite Calcite (Rarely Observed)	Albite Dolomite	Albite Quartz Dolomite Pyrite +/- Magnetite, Hematite
Felsic (Northern)	Muscovite Calcite	Albite (Hematitic) Dolomite	Albite Quartz Pyrite Dolomite +/- Magnetite, Hematite
Pyritic Shales and siltstones (Black Shale)	Carbonaceous Pyritic Calcite	Sericite Albite	Albite Quartz Pyrite (Rarely Observed)

Figure 7-5 – Basic Relationship Between Protolith and Alteration Products at Ikkari

7.2.1. ROCK TYPES

Komatiite / Talc-Altered Ultramafic

In more detail, talc altered ultramafic units are dark grey to green-grey schistose extrusive rocks, which can exhibit volcanoclastic textures with lapilli-like deformed clasts where structural intensity is reduced (Figure 7-6). Geochemically, they are komatiitic in composition (> 18% MgO) and are almost completely altered to talc-chlorite composition, but also variably contain serpentine, amphibole and biotite and characteristic narrow, wispy calcite veinlets (Figure 7-7).

These units become more prominent in the southwestern corner of Ikkari, south of an E-W trending fault zone that largely constrains the highly strained domain north of it. Within the highly strained domain this unit still occupies positions distal to the mineralisation and can be seen to form the outline of fold geometries. It represents the regional alteration of Komatiites and is common south of the Rajala Line structure stretching up to and beyond the Pahtavaara mine.

The komatiite/ talc altered Ultramafic sequence, forms an over 100 m-thick continuous unit between the Ikkari mineralisation and footwall quartzites containing only thin and semi-discontinuous altered portions of ultramafic schist.

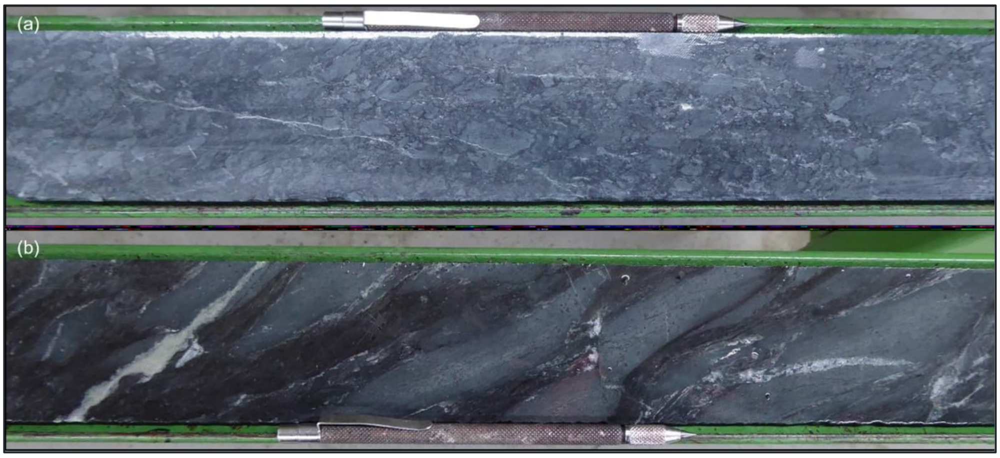


Figure 7-6 – Hole 120061: example of Barren Ultramafic Rocks

Figure 7-7 – Hole 120065: example of Highly Strained, Barren Ultramafic Rocks

Ultramafic Schist and Internal Sediments

Where the komatiitic ultramafic units occur in closer proximity to the mineralisation, intensely altered ultramafic rocks may appear as a more mafic lithology (magnesium replaced by iron). Mineralogically talc is no longer present, and the composition is chlorite-sericite-siderite-magnetite dominated (Figure 7-8). Proximal to the mineralisation, silica and dolomite veining together with pyrite mineralisation become more common also (Figure 7-9). Intermediate stage shear textures such as boudinage veins (Figure 7-10) are more rarely recorded with vein fragments typically dismembered in the foliation as rootless folds.

This unit, derived from the alteration of Komatiites, is logged as metamorphic schist of ultramafic protolith (MSCU) and is represented on maps and sections as the Ultramafic Schist. Alteration, initially iron metasomatism (chlorite-siderite-magnetite) is strongly correlated with the presence of metasedimentary rocks which are present in the ultramafic package as intercalations.

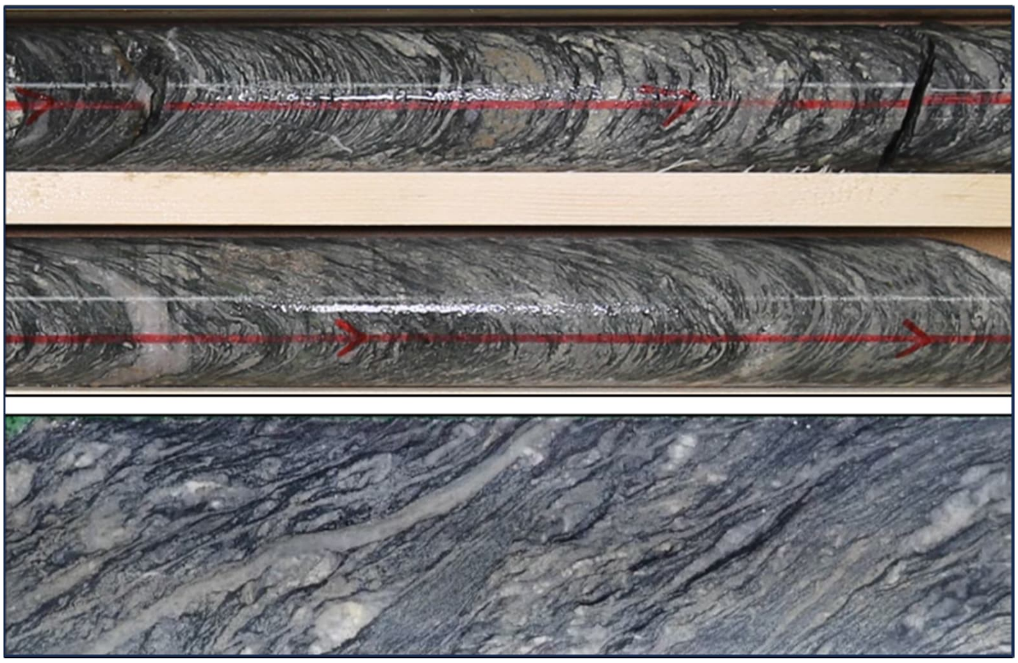


Figure 7-8 – Hole 122005 (top) and Hole 122066 (bottom): example of Altered Mafic-Ultramafic Schistose Rock

Figure 7-9 – Hole 120086: Schistose Rock with Strong Sericite-Silica Overprint

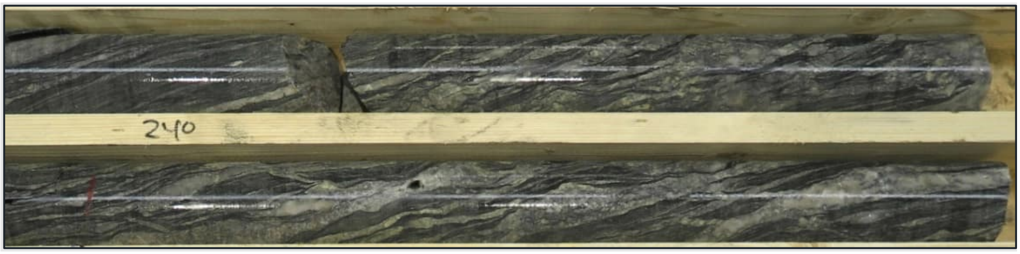


Figure 7-10 – Hole 1120071: Example of Boudinage Quartz-carbonate Veins in Altered Mafic-ultramafic Rock

The mixed ultramafic-sedimentary package (shown on plan map as ultramafic schist and internal sediment) is characterized by highly variable alteration styles, in places intense veining and foliation that frequently overprint texture, making identification of the original lithology difficult.

Sedimentary rocks present range from conglomerates (Figure 7-11) through felsic sandstones and quartzites to siltstones (Figure 7-12), implying that diverse range of sediments have been intercalated within this unit. Downhole widths of sedimentary intercalations range from tens of meters through to sub-centimetre whilst petrological descriptions suggest that millimetre scale clasts of felsic material can also be present in chlorite-sericite matrix.

Where the intercalated sediments occur as cohesive intersections, there is a clear geochemical distinction from the altered ultramafic schists. At the scale of the geochemical sampling, predominantly 1m samples, a unit with a 'mixed' signature is also present and closely correlated with mineralisation. Originally this was attributed to a variable volcanogenic component with these ultramafic schists, nominally more mafic. The current interpretation is that these represent shears active during the intercalation of the sediment within the ultramafic leading to the entrainment of small clasts of sediment with the ultramafic schist leading to the 'mixed' signature.

Where original textures are preserved within the internal sediments, finely laminated dark grey to green-brown silty sediments are common, with occasional coarse grained (up to gravel-sized clasts) units. In places, sedimentary banding is commonly defined (or enhanced) by albite flooding.

The mixed internal sediment and ultramafic schist sequence, hosts ~80% of the mineralisation at Ikkari and forms an over 200 m-thick sequence between the Northern Felsic and/or Black Shale and the lower strained, Komatiites which are dominant further to the south.

Figure 7-11 – Hole 120059: Example of Intercalated Conglomerate Within the Ultramafic Schist

Figure 7-12 – Hole 122005: Example of Intercalated Siltstone

Northern Felsic

Felsic sediments are commonly intensely and pervasively albite-altered, particularly forming a large block of albitized rock in the northwestern extent of the deposit. Albite alteration varies from brown to brick red in colour and original sedimentary textures are obliterated (Figure 7-13). Albite-altered rocks are dominated by brittle fracture, with gold mineralisation associated with pyrite (±magnetite) in veinlets. The northern felsic hosts ~20% of the mineralisation at Ikkari.

Where not flooded by albite and the primary texture preserved, the unit is commonly a fine-grained sandstone to siltstone with weak sericite alteration enhancing bedding (Figure 7-14). It is interpreted that albite preferentially alters the coarser units with sequence leading to an overrepresentation of fine-grained siltstones in the weakly altered portions.

Figure 7-13 – Hole 120072: Example of Strongly Albite Altered Felsic Sedimentary Rock

Figure 7-14 – Hole 122008: Example of Fine Grained Sandstone to Siltstone

Black Shale

Laminated carbonaceous shale (commonly referred to as the black shale) forms the hangingwall (northern margin) to mineralisation in most places (Figure 7-15). It contains significant amounts of syngenetic disseminated pyrite which is often banded, and although graphite content is overall low, graphitic fractures occur in places. The black shale hosts very minor mineralisation that is remobilized from the main felsic and ultramafic hosts rocks at Ikkari.

Figure 7-15 – Example of Laminated Carbonaceous Shale

Gabbro

In the hangingwall of the deposit, a mafic intrusive of gabbroic composition (Figure 7-16) intrudes the carbonaceous shale, including locally, narrow dykes. This unit does not host any mineralisation.

Figure 7-16 – Example of a Very Weakly Foliated Gabbroic Intrusive

Southern Felsic Sedimentary Rocks

To the south of the mineralised zone, and the ultramafic dominated package, the contact with the Southern Felsic Sediments (sometimes referred to as the Kumpu Group quartzites) is poorly defined. The Kumpu Quartzites are coarse-grained, relatively unaltered and weakly strained more than a few meters from the contact. In the southwestern portion of the deposit, near surface, the contact between the Kumpu and the Ultramafic package is clearly faulted however at depth the nature of the contact is debateable, and drill information limited.

At depth in the west of Ikkari drilling beyond the initially inferred contact, in very limited areas, has indicated that the intercalation of komatiitic strata continues albeit with decreasing abundance of ultramafic material to the south. In the east of Ikkari the contact is more well defined and no further ultramafic material is located beyond the contact to low strain quartzites. Minor mineralisation is seen in quartz veinlets at the contact to the quartzites and at one location within the quartzite.

Three separate groups of felsic are modelled based primarily on their spatial location, compositionally and texturally these sediments are commonly indistinguishable. Ages dating of these felsic sediments has shown that all are part of the Kumpu group ~1.88 Ga (Harju, 2022) significantly younger than the 2.05 Ga Savukoski Group komatiites into which the internal sediments are intercalated. This suggests that these younger rocks must have been complexly structurally interposed within the older komatiite units prior to mineralisation.

Breccias

Breccias are common throughout the deposit and occur in most lithology types. Structural relationships indicate at least three main phases of brecciation:

- A polymictic breccia, with coarse fragments, frequently fuchsitic or intensely chlorite-altered, displaying elongation of clasts parallel to dominant (S2) foliation (Figure 7-18). This style of brecciation along with similar textures in conglomeratic sandstones (Figure 7-19) are interpreted to be depositional in origin;

- A relatively early cataclastic tourmaline-welded breccia commonly containing clasts of albite-altered sediments (Figure 7-20). In places these are overprinted by the mesothermal alteration regime and are tentatively interpreted to represent D1 structures related to the structural interposition of sedimentary units with the ultramafic; and
- A late, carbonate-iron-oxide-rich, hydrothermal breccia that contains rounded quartz grains in a fine-grained matrix and is sometimes vuggy (Figure 7-21). With typically narrow (10 to 30 centimetres [cm] wide) cross-cutting geometries that indicate fluidized injection (Figure 7-20), these breccias frequently host disseminated pyrite, and associated gold grades. Breccias appear to have a dominant sub-vertical control, utilise pre-existing lithological contacts and are associated with high-grade gold mineralisation, within and particularly at margins where visible gold is often present.

Figure 7-17 – Hole 120059: Example of Chlorite Alteration and Disseminated Pyrite within Ultramafic Rock

Figure 7-18 – Hole 121160: Example of Fiamme-like clasts in Conglomeratic Sandstone Ultramafic Rock

Figure 7-19 – Hole 122039: Example of Tourmaline Welded Cataclastic Breccia

Figure 7-20 – Hole 120123: Example of Wider, Iron-oxide-rich Breccia in Ultramafic Schist

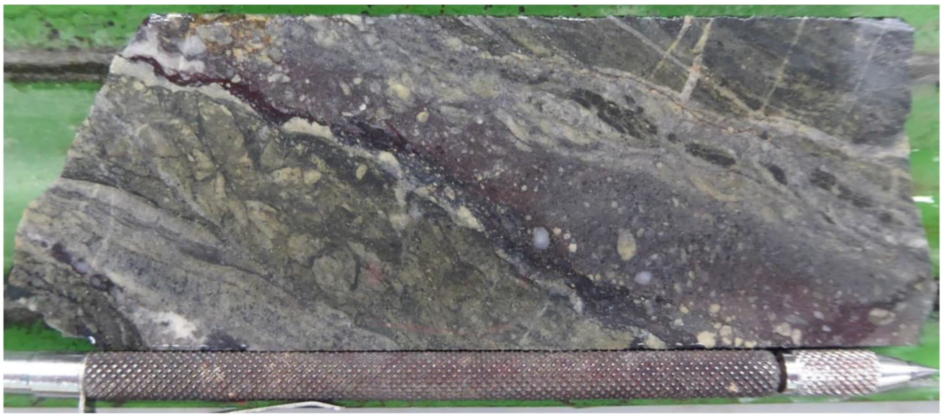


Figure 7-21 – Hole 120123: Example of Narrow, Iron-oxide-rich Breccia in Ultramafic Schist

7.2.2. STRUCTURE

In its simplest terms Ikkari occurs at the structurally modified unconformity between the Savukoski and Kumpu Groups, the Ikkari mineralisation is largely confined to an approximate ENE striking, approximately 200 m wide corridor of structurally interleaved Kumpu Group sediments and komatiite dominated Savukoski Group strata.

A moderately N-dipping cataclastic, tourmaline-bearing shear defines the northern margin of the mineralised, interleaved, corridor, obliquely cutting units in the latter, but sub-paralleling the strike of Savukoski black (carbonaceous) shale-dominated strata to the north.

The southern margin of the high-strain, mineralised corridor is defined by a series of vertical faults that merge at depth and relax towards surface as a flower-like structure. This current expression of this fault structure is talc-chlorite rich fault gouge indicating late-stage brittle deformation and it is likely related to the relative uplift of a block of Kumpu Group Sediments to the south of the structure, which plunges to depth in the west.

Whilst the brittle fault splay defines the southern margin of the high-strain mineralised corridor more weakly deformed talc-altered komatiites and occasional, minor sedimentary intercalations are continuous south of this feature. Mineralisation in this low strain domain is present but often lower grade and discontinuous (Bonson, 2022b.). Further south still, less well constrained by drilling, an outlier of quartzitic Kumpu Group sandstones and conglomerates is at least locally in sheared contact with the komatiitic sequence. Strain intensity and alteration decrease rapidly within the Kumpu Group Sandstones (Figure 7-22).

In the west of Ikkari, at depth, Kumpu Group Sandstones become progressively more dominant though interleaved komatiitic rocks persist in decreasing quantities; it is believed that here the true contact to the homogeneous Kumpu outlier has not been drilled.

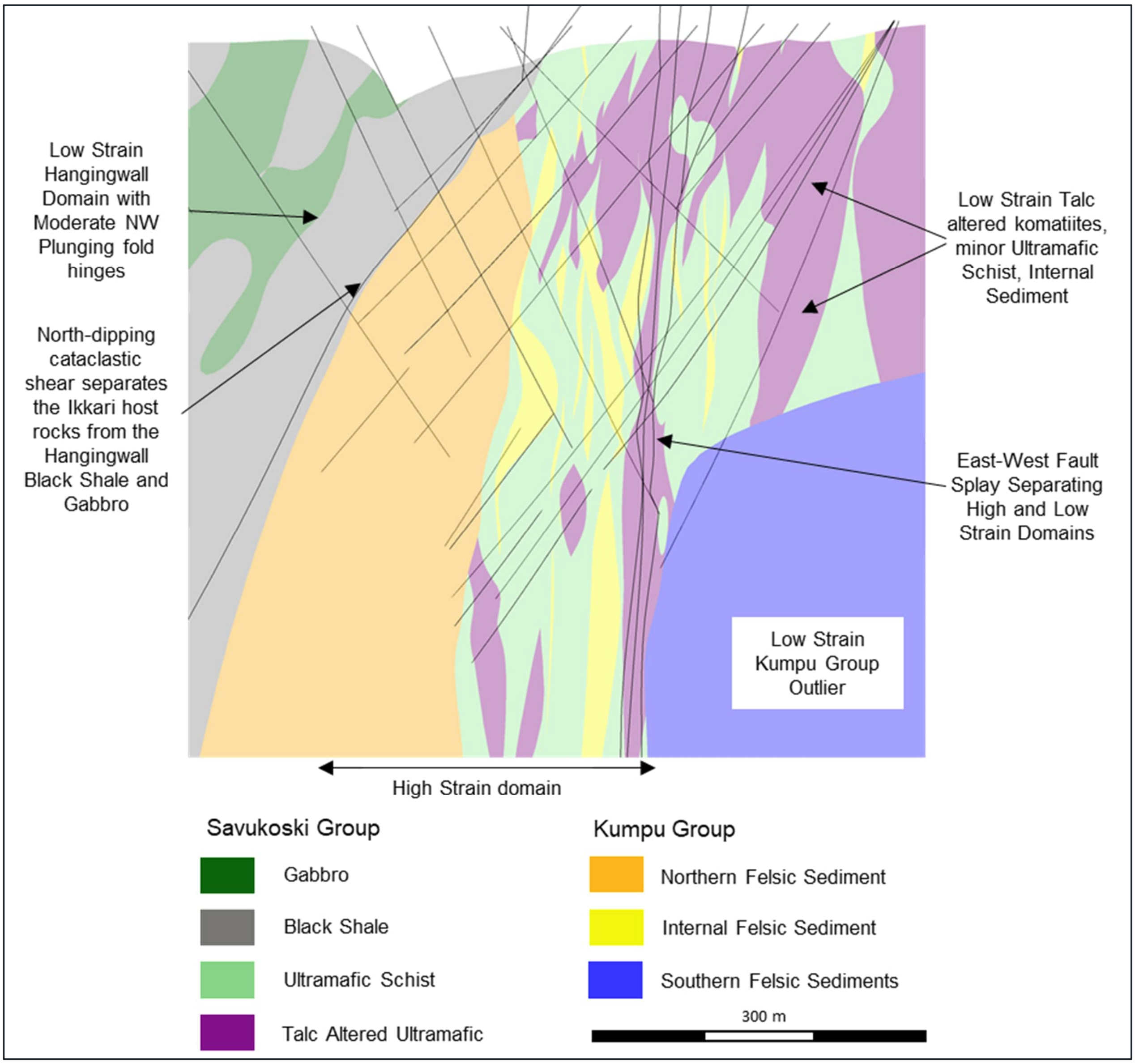


Figure 7-22 – Example Geological Cross Section through Ikkari Looking Toward 065

Considering the high strain domain at Ikkari, that which hosts the vast majority of the mineralisation, structural studies of representative drill hole intersections from Ikkari (Selley, 2021 and Bonson, 2022a) indicate three distinct phases of deformation that are texturally and geometrically analogous to the deformation history recorded throughout the region (Figure 7-23). These phases of deformation have led to the development of a complex meshwork of structures and fractures which have acted as fluid flow pathways at various times. These structural meshworks, and relative timing of iron- and gold-bearing fluids have resulted in the deposition of gold mineralisation, associated with pyrite at structural and geochemical 'trap' sites.

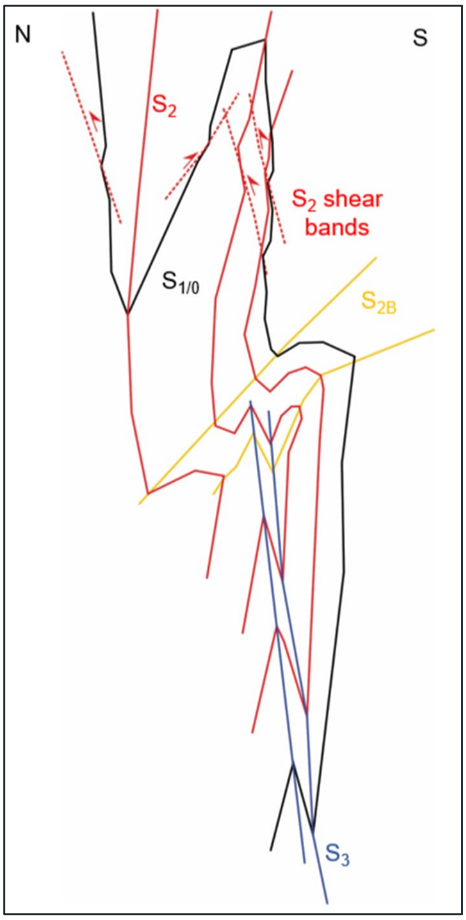


Figure 7-23 – Schematic Representation of the Three Phases of Structural Deformation at the Ikkari Deposit

A first phase of deformation (D_1) records early orogenic, large-scale recumbent folding and thrust stacking, with layer-parallel fabrics developed. Although this deformation is poorly preserved it interpreted to be responsible for the complex interleaving of sediments within komatiitic facies, which appears to have been a necessary 'pre-conditioning' for gold mineralisation throughout the deposit.

A later deformational event (D_2), NW-SE compression of the thrust stack, resulted in the development of tight (meter-scale) upright isoclinal folds with broadly vertical axial planes. This deformation results in the complex geometries of broadly continuous but highly attenuated deformed sediments within the ultramafic rocks (Figure 7-24). The fold plunges recorded in relation to this deformation are shallowly plunging (Figure 7-25) with both NE and SW shallow plunges recorded however, to date, it has not been possible to resolve the different plunge directions in space.

The penetrative S_2 foliation is the dominant fabric identifiable within the mineralised corridor at Ikkari, S_1 fabrics are at most cryptically preserved in fold hinges but more commonly rotated into parallelism with the S_2 foliation on the limbs of S_2 folds making them indistinguishable. The lithological and fold geometry generated during D_2 are the main control to the localization of gold mineralisation at Ikkari with contacts and fold hinges preferentially mineralised.

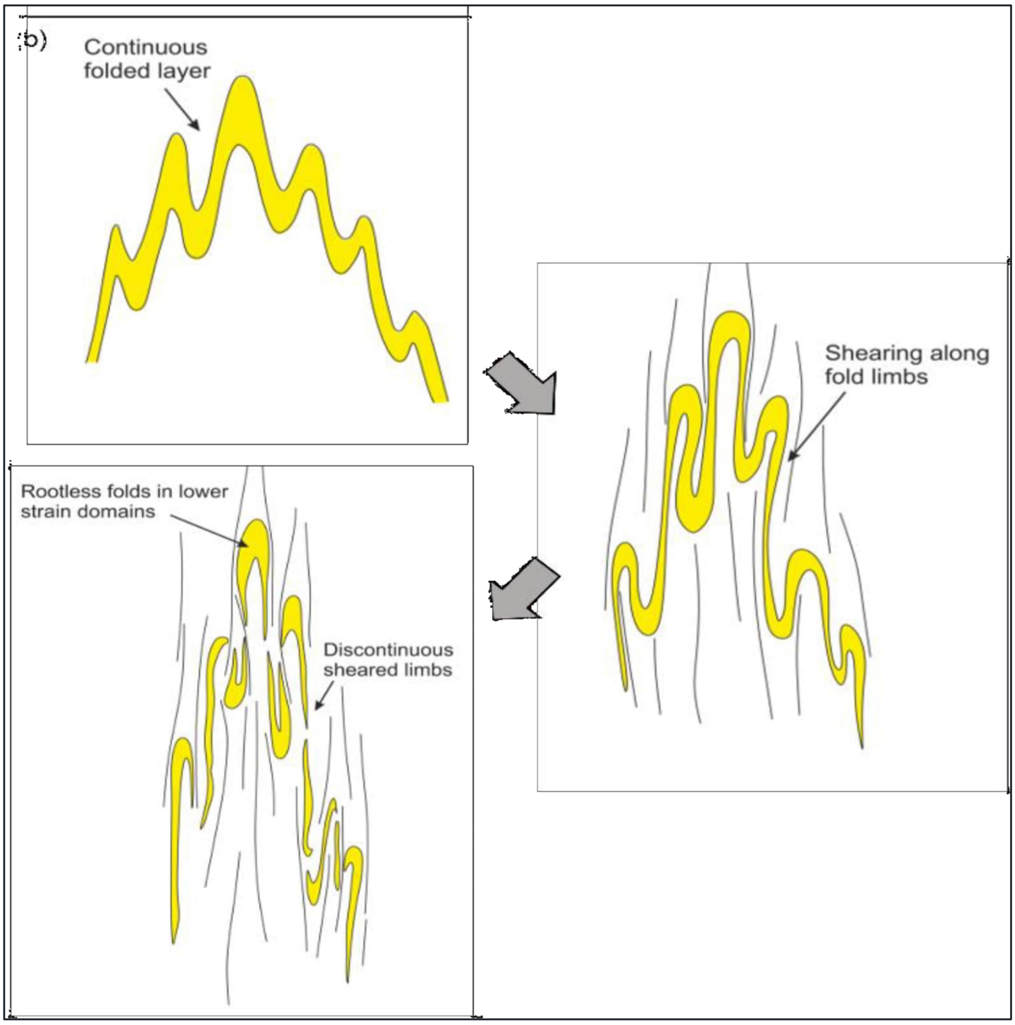


Figure 7-24 – Schematic Representation of Main Deformation Event Recorded at Ikkari

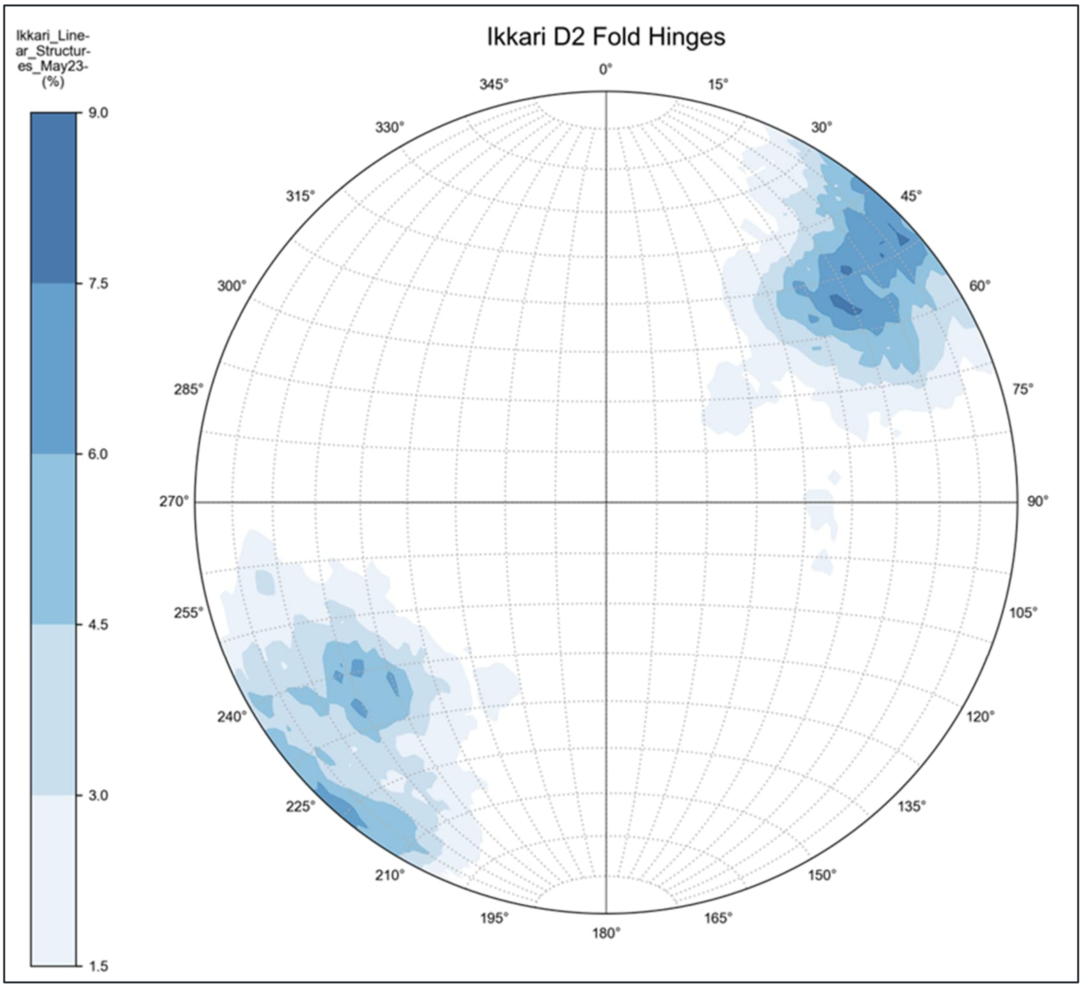


Figure 7-25 – Stereonet of D_2 Fold Plunge Measurements

During progressive deformation, likely due to strain hardening as the tight isoclinal folds have been unable to accommodate further strain a series of WNW-ESE shears are developed with dextral and top to the south sense of movement recorded (Figure 7-26 and Figure 7-27).

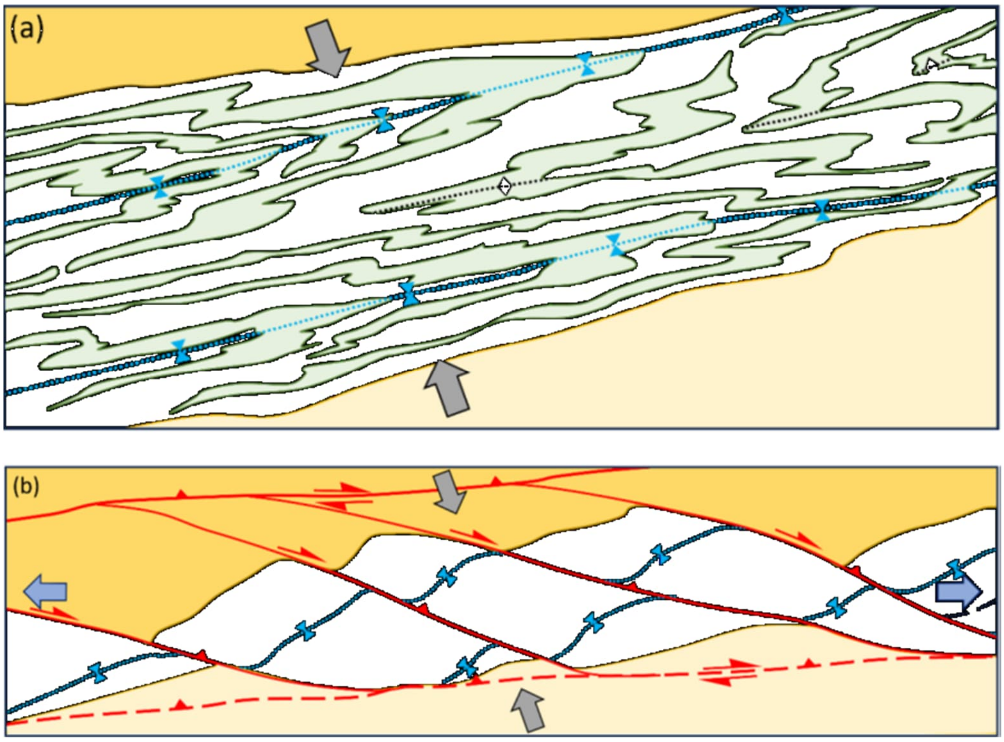


Figure 7-26 –Schematic Representation of The Development of The Isoclinal Folding and Subsequent Compartmentalization

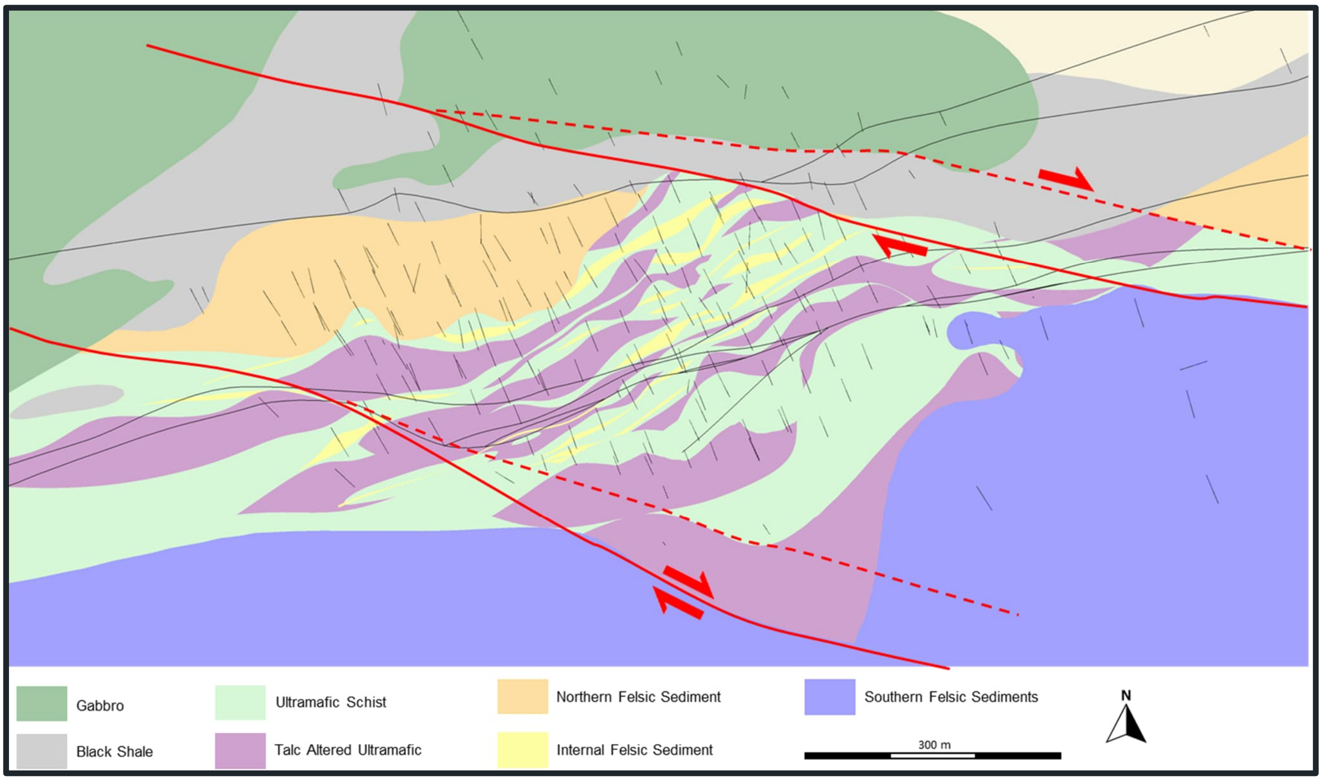


Figure 7-27 – The Expression of the WNW-ESE shears Developed at Ikkari.

A third deformation phase is recorded within the highly deformed corridor at Ikkari though it is currently unclear if it should be correlated to the WNW-ESE shears noted above (Figure 7-27) as a continuum of D_2 or a distinct structural event related to the reactivation of the structure responsible for thrusting the hangingwall sequence southwards onto the Ikkari host sequence. D_3 folds have an average dip / dip direction of 57°/298 and are thus parallel to the Black Shale contact whilst hinges plunge 15-20° steeper than those formed during D_2 (30° → 227°).

A combination of the WNW-ESE developed late during D_2 and D_3 folding is responsible for the anticlockwise rotation of pre-existing fabrics at the western end of the deposit (which corresponds to a reduction in ore volume), and a more subtle anticlockwise deflection at the eastern end of the Northern Felsic Zone, where the ore volume is greatest (Figure 7-28) (Bonson, 2023). They are also responsible for the flexing of the S_2 foliation and therefore the mineralisation such that a northerly dip is prominent close to the Black Shale contact; this is most pronounced in the NE corner of Ikkari where the eastern most shear also cuts across the deposit (Figure 7-29). Structural disks representing foliation trajectories shown in Figure 7-28 and Figure 7-29 are measured on orientated diamond drill core.

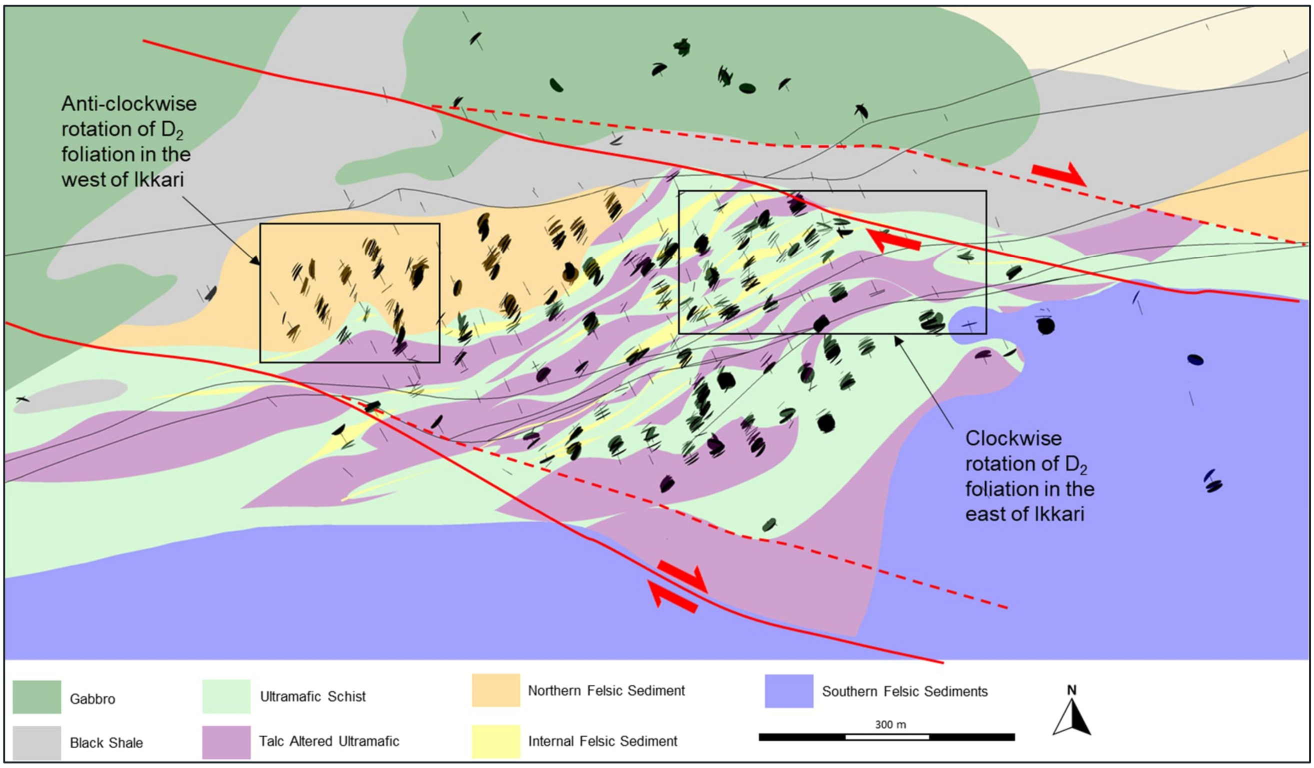


Figure 7-28 – Schematic Representation of the Three Phases of Structural Deformation at the Ikkari Deposit

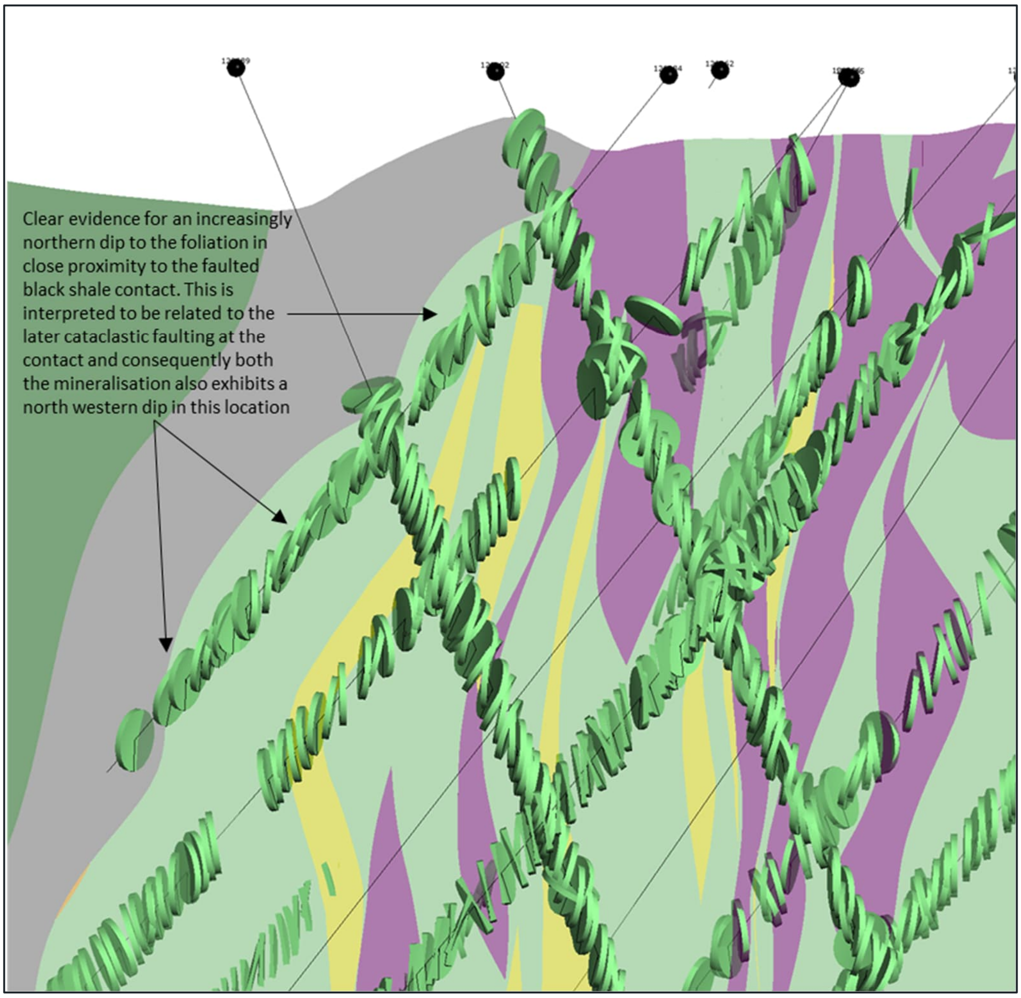


Figure 7-29 – Cross Section from the Central Eastern Portion of the Deposit Showing High Confidence (D_2) Foliation as Discs.

The Ikkari deposit can be described as an orogenic, hydrothermal gold deposit. Modelling of the mineralisation, using over 111 000 m of drilling available, shows the deposit to lie within a mineralised envelope of up to 900 m long, 350 m wide and 750 m deep (Figure 7-30 and Figure 7-31, with 200-m Grid for reference) and that the deposit remains demonstrably open at depth and along strike, at depth.

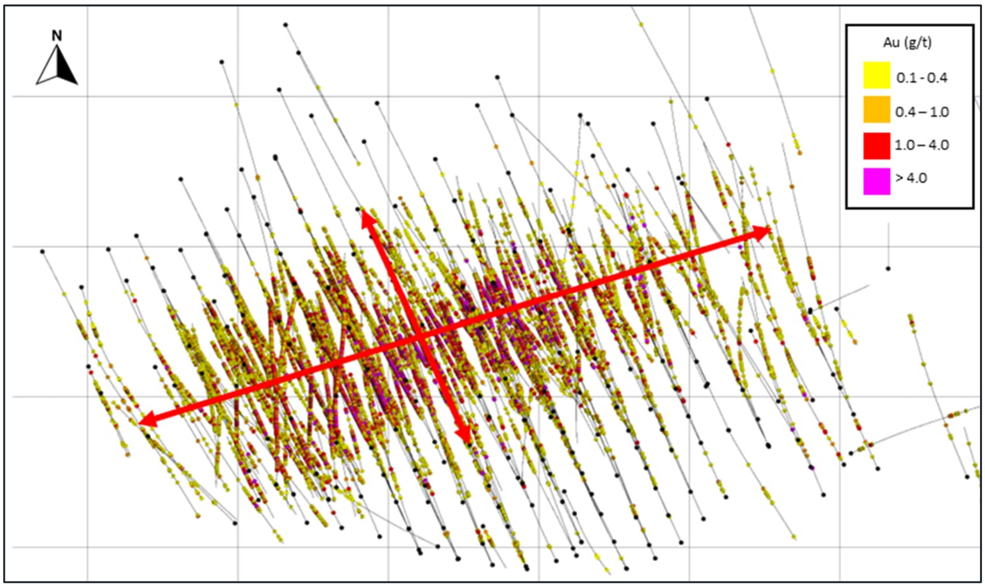


Figure 7-30 –Currently Defined Limits of Ikkari Mineralisation (Plan View)

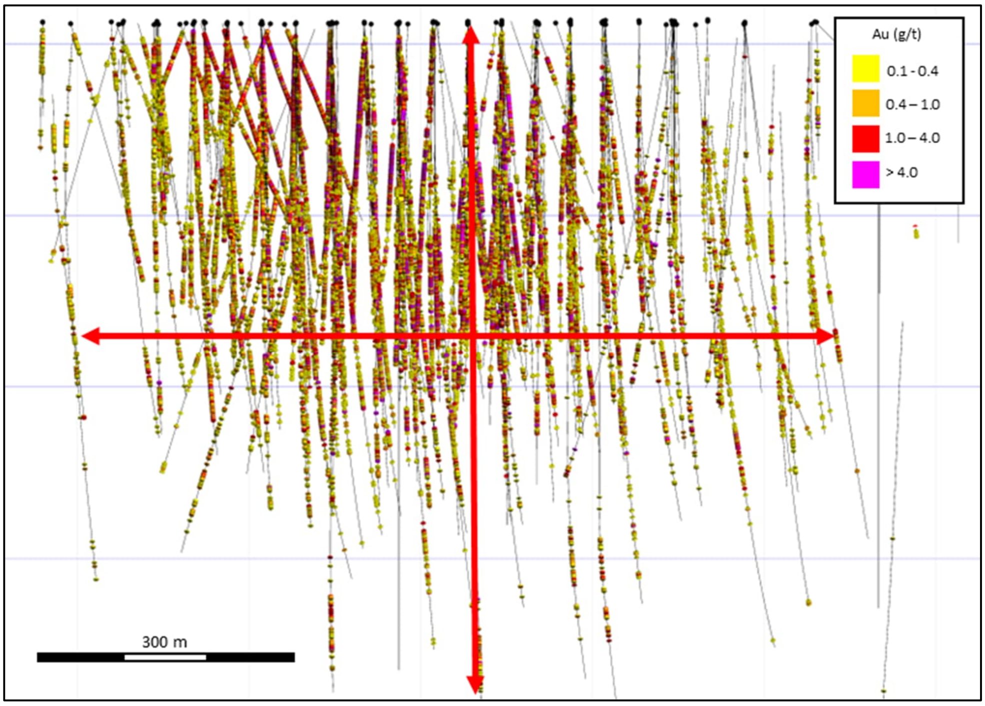


Figure 7-31 – Currently Defined Limits of Ikkari Mineralisation (Looking Northwest toward 335°)

Overall, the mineralisation trends at approximately 065° strike and has a strong sub-vertical control. However, within the mineralised halo different grade zones have varying morphology and plunges on a local scale and these are explored later.

Mineralisation at Ikkari occurs in several styles, but in all cases, gold distribution is correlated to the abundance of disseminated pyrite and intensity of veining, which are in turn considered to be principally controlled by lithological contacts, fold geometry and brittle fracturing. The style of mineralisation is principally controlled by the host lithology with significant controls on mineralisation localization including:

- Brittle-fracture regime in intensely albite-altered felsic sediments that controls veinlets of gold associated with fine-grained pyrite and magnetite (e.g., Figure 7-13). Given that this style of mineralisation is limited to the albite-altered sediments it is most prevalent in the north-western portion of Ikkari where the felsic sediments form a large block. It also occurs in larger felsic intercalations within the komatiite domain;
- Lithological contacts: notably intensely chlorite-sericite-siderite-magnetite-pyrite-(±fuchsite)-altered sediments with felsic sediments, quartzite and conglomerate, and siltstones;
- Complex and concentrated short-wavelength (metre-scale) parasitic folding of narrow felsic and siltstone sedimentary intercalations within intensely chlorite-sericite-magnetite-altered ultramafic rocks that appears to further focus fluid flow and pyrite deposition, particularly at fold hinges. Intense, irregular carbonate-quartz veining is frequently developed in these zones and mineralised higher in grade. (e.g., Figure 7-9); and
- Within and at the margins of hematite-carbonate hydrothermal breccias (Figure 7-20 and Figure 7-21), that have a sub-vertical expression and overprint folding and cross-cut lithological contacts. Where these breccias host intense disseminated pyrite, bonanza gold grades are commonly seen.

Ikkari is unusual among orogenic gold deposits in the width of mineralisation when compared to the strike (Figure 7-30 and Figure 7-31). In typical orogenic gold systems, the strike of mineralisation is an order of magnitude greater than the width, however, at Ikkari the strike length of the mineralisation is only two to three times the width and this is be attributed to multiple, stacked mineralised zones perpendicular to the strike. These stacked zones are interpreted to arise from the structural interleaving of diverse lithologies pre-mineralisation in D_1, with no evidence to support post mineralisation thickening. From the northwest to southeast across Ikkari, at least four subtly different mineralised zones can be described:

- Within the large felsic block to the northwest of Ikkari, a brittle-fracture regime in intensely albite-altered felsic sediments. This coalesces towards surface and exhibits a moderate northern dip in close proximity to the carbonaceous shale. At depth, and in the east, these brittle fracture zones separate into at least two, narrower, vertical trends. These mineralised zones are separated from each other, and the subsequent mineralisation trend to the south, by largely barren sericite and weakly albite-altered felsic sediments (Figure 7-32). In the domaining for resource estimation (Chapter 14) this is termed the “Northern Felsic Domain”;
- At the contact between the northern felsic block and the komatiite domain in the west and then stepping off this contact to the east to be entirely within the komatiite domain, is the next zone of mineralisation. In the west, mineralisation occurs on both sides of the felsic-komatiite contact with the intensely albite-altered felsic sediments hosting an intense silica-pyrite brittle fracture to breccia regime, whilst to the south of the contact and along strike to the east, in the komatiite domain, mineralisation is most commonly related to first intercalated felsic or phyllitic sediment encountered, the contacts of this and fold hinges within (Figure 7-32 and Figure 7-33). In this, the strongest zone of mineralisation, mineralisation is commonly pervasive throughout the intercalated sediment rather than focused on its contacts;

- To the east, away from the large felsic block, barren talc-chlorite-altered komatiite occurs to the north of this mineralisation, separating it from the converging carbonaceous shale. Further east still, this mineralisation trend is terminated by the cross-cutting carbonaceous shale. Where the mineralisation trend occurs in close proximity to the carbonaceous shale it exhibits a northern dip (Figure 7-33) consistent with the shale but elsewhere the dip is more vertical, and the apparent plunge is approximately 30° to the east. In the domaining for resource estimation (Chapter 14) this is termed the “Contact Domain”

- Further south still are several parallel mineralisation trends within the komatiite domain are characterised by a decreasing gold tenor and lateral extent towards the south/southeast. Mineralisation is primarily associated with contacts to intercalated felsic or phyllitic sediments within the komatiites and enhanced at the fold hinges of these intercalations. Mineralisation in this portion of the deposit plunges back to the WSW at approximately 15° which is consistent with the S2 fold hinges measured throughout the komatiite domain. (Figure 7-32 and Figure 7-33); and

  - The opposite plunge of this mineralisation in comparison to the trend north of it, creates diverging mineralisation trends to depth in the west and converging mineralisation trends towards surface in the central-eastern portion of the deposit. To the south of this trend, and where the trends diverge, talc-chlorite-altered barren komatiites separate the mineralisation trends. However, where mineralisation trends are in closer proximity, no talc-altered komatiite is preserved, and weakly mineralised iron-metasomatised chlorite-sericite-siderite assemblages, the distal alteration product of the komatiite domain, separates the mineralisation trends; this is also the case in the poorly mineralised / barren gaps between the mineralisation trends of this type. In the domaining for resource estimation (Chapter 14) this is termed the “Internal UM Domain.”

- To the south of the E-W fault array, within the low-strain talc altered komatiites, laterally discontinuous felsic intercalations host mineralisation at the contacts to the komatiite in a similar style to those described above. However, here the mineralisation is more discontinuous, and the proximal komatiite does not exhibit extensive iron metasomatism as the mineralisation trends further north Figure 7-32 and Figure 7-33).

Although vein arrays and stockwork zones are considered to be linked to the main gold phase, there is little consistent relationship between vein density, vein volume, and gold grade. This is attributed to much of the siderite veining, now transposed into the foliation, being relatively early, likely a product of the iron metasomatism ‘ground preparation’ event along with chlorite and magnetite, that may have been synchronous with the initial structural interleaving of sedimentary and komatiitic units.

There appears to be a closer relationship between gold content, pyrite and late-stage iron-oxides. Magnetite-bearing veins and breccias typically contain elevated gold grades, with associated disseminated pyrite, and where late haematite is (also) present, particularly in coarse breccias comprising haematite-carbonate (+ pyrite) in the matrix, very high grades (>10 g/t Au) are observed. These iron-rich fluids clearly post-date the main deformation event and inject at zones of weakness, particularly lithological contacts and early breccias. Late-stage hematite dominated hydrothermal breccias with a vertical control occur throughout mineralised zones 1 to 3 as described above but are by far the most extensive in zone 2 that hosts the strongest grades in the deposit.

Despite these variations in localisation at the deposit scale, it is considered that all the gold mineralisation is related to the same (oxidised) fluid event that was introduced along a complex brittle-ductile permeability meshwork. Sites of gold deposition are structurally controlled but locally dependent on the availability of a geochemical reductant that allows deposition of pyrite and associated gold. Such iron-rich reductants at Ikkari are likely to include magnetite and chlorite, formed during an earlier iron-metasomatic alteration and/or syngenetic pyrite that may have been present in the intercalated siltstones. The presence of a pre-existing reduced fluid cannot be excluded. The spatial association of high-grade gold zones to apparently later, largely post deformation hematite-carbonate breccias is indicative of a later gold-bearing fluid phase also being present.

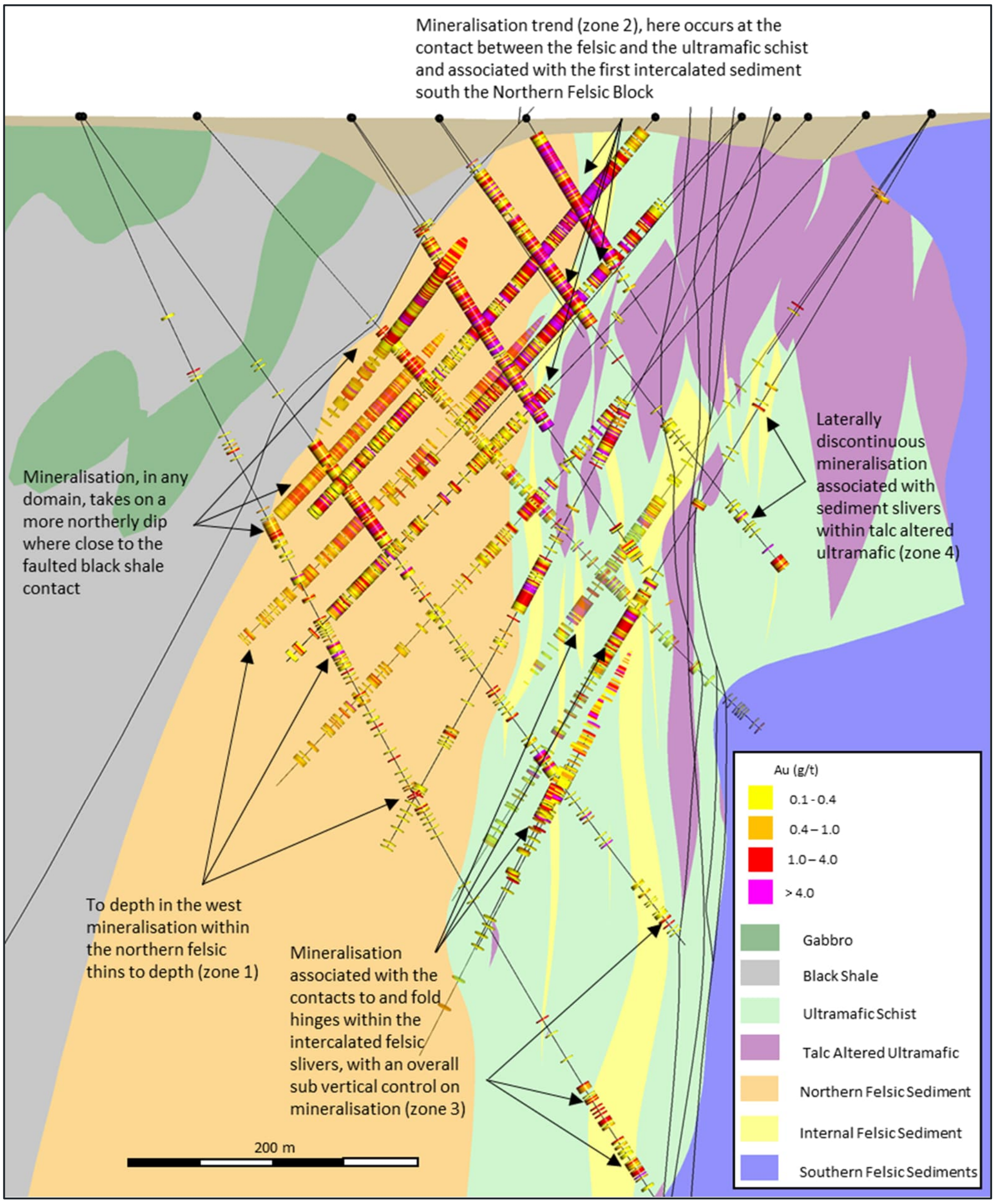


Figure 7-32 – Cross Section from the Central Western Portion of the Deposit

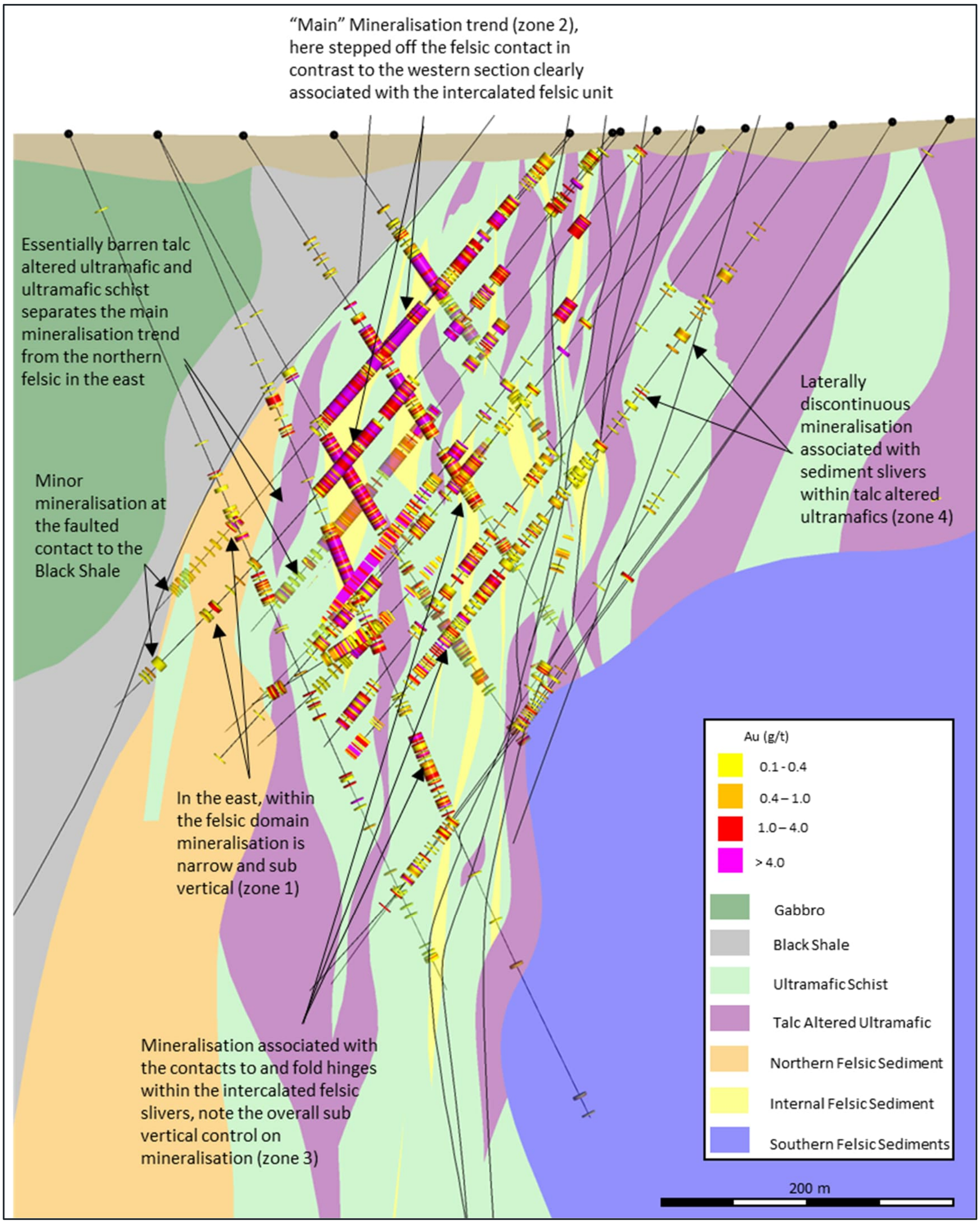


Figure 7-33 – Cross Section from the Central Eastern Portion of the Deposit

Figure 7-34 – Hole 120102: Visible Gold Within Brecciated Carbonate Veining

8 DEPOSIT TYPES

The mineralisation at Ikkari is considered to be orogenic-style with gold mineralisation associated with low sulphidation alteration. Genetic models for orogenic gold deposits have been discussed in several studies (e.g., Groves et al., 1998 and Groves and Santosh, 2015). The key aspects of these models are:

- Metals, complexing agent(s) and fluids transporting the metals are released from the source (or sources) at depth;
- Metal-carrying fluids are focused into shear zones; and
- The auriferous fluids migrate along structures into suitable structural and/or chemical traps where the gold and associated metals are deposited via various physicochemical reactions (Niiranen, et al, 2015), Figure 8-1.

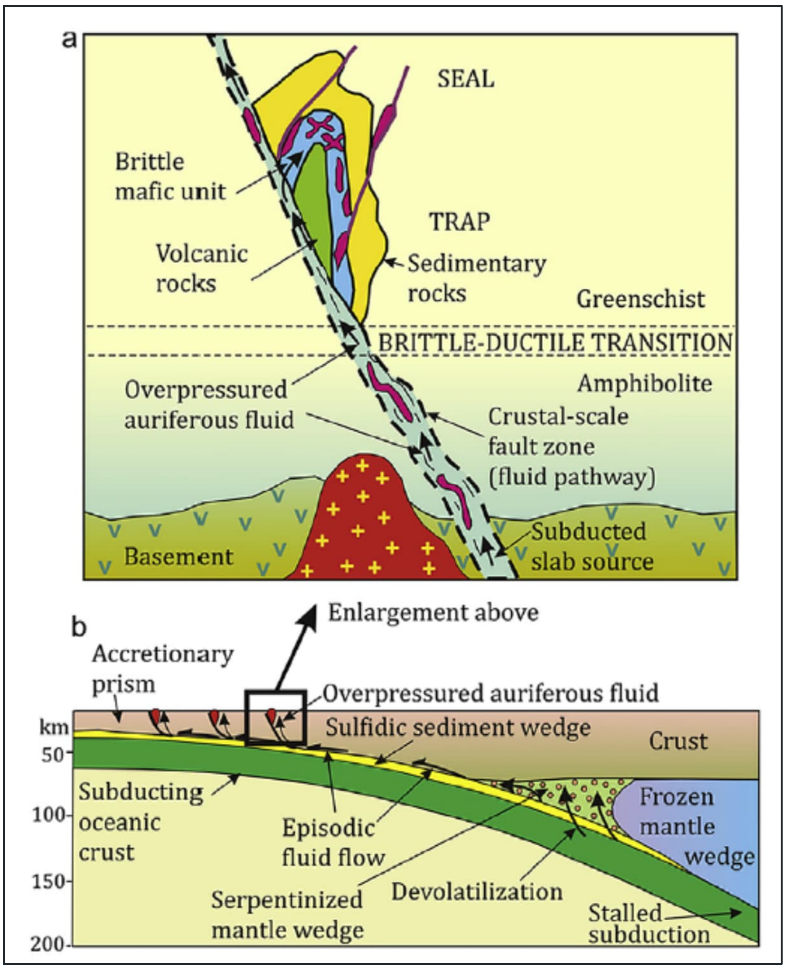


Figure 8-1 – Schematic Representation of a Permissive Scenario for All Orogenic Gold Deposits

A number of orogenic gold deposits are believed to be hosted in the CLB, including the Pahtavaara and Suurikuusikko deposits (Kittilä Mine) (Figure 8-2). Global examples of other orogenic gold deposits include Kalgoorlie (Australia), Val d'Or (Canada) and Ashanti (Ghana) (Groves et al., 1998). Examples of gold deposits associated with atypical metal associations are given in Groves et al., 2003 with base and semi metals, uranium or even rare-earth elements contributing economically important enrichments in some of the deposits. The introduction of fluids from folded and thrusted intracratonic basins, during orogenesis, is considered a key factor in their formation, as well as possible inheritance of base metals from a proto-ore (and subsequent overprint of gold mineralisation) or high salinity fluids released from sedimentary sequences during metamorphism that may introduce base metals into orogenic gold systems (Yardley & Graham, 2002).

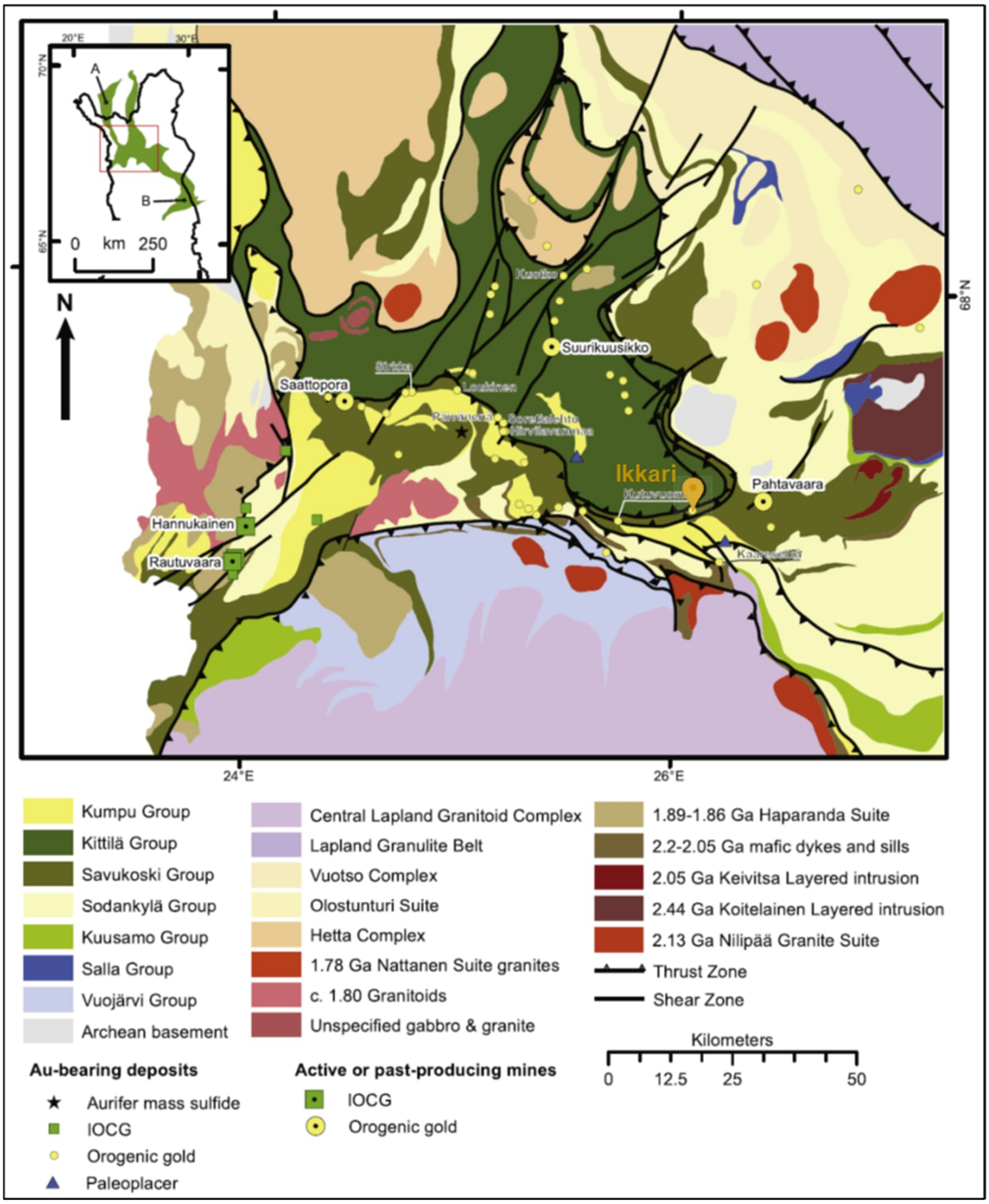


Figure 8-2 – Geology and Gold Deposits of the CLB

At a camp and district scale, known deposits cluster in proximity to transcrustal or other major deformation zones that are formed synchronously with the thickening of the crust during accretionary or collisional tectonic events. In most prospective districts, the deposits were formed at mid-crustal levels, as suggested by the dominant greenschist facies metamorphic assemblages of the host rocks (Niiranen et al., 2015). Within the Rupert Lapland Project land package, including known gold occurrences at Pahtavaara, Koppelokangas and Hookana, gold mineralisation is located close to a number of structures identified on regional geophysics within rocks of the Savukoski Group, and in the westernmost areas of the Rupert Lapland Project area, hosted at the thrusted margin between the Kumpu and Savukoski Groups. Timing relationships between major Groups in the CLB are set out in Figure 8-3 (Wyche et al, 2015).

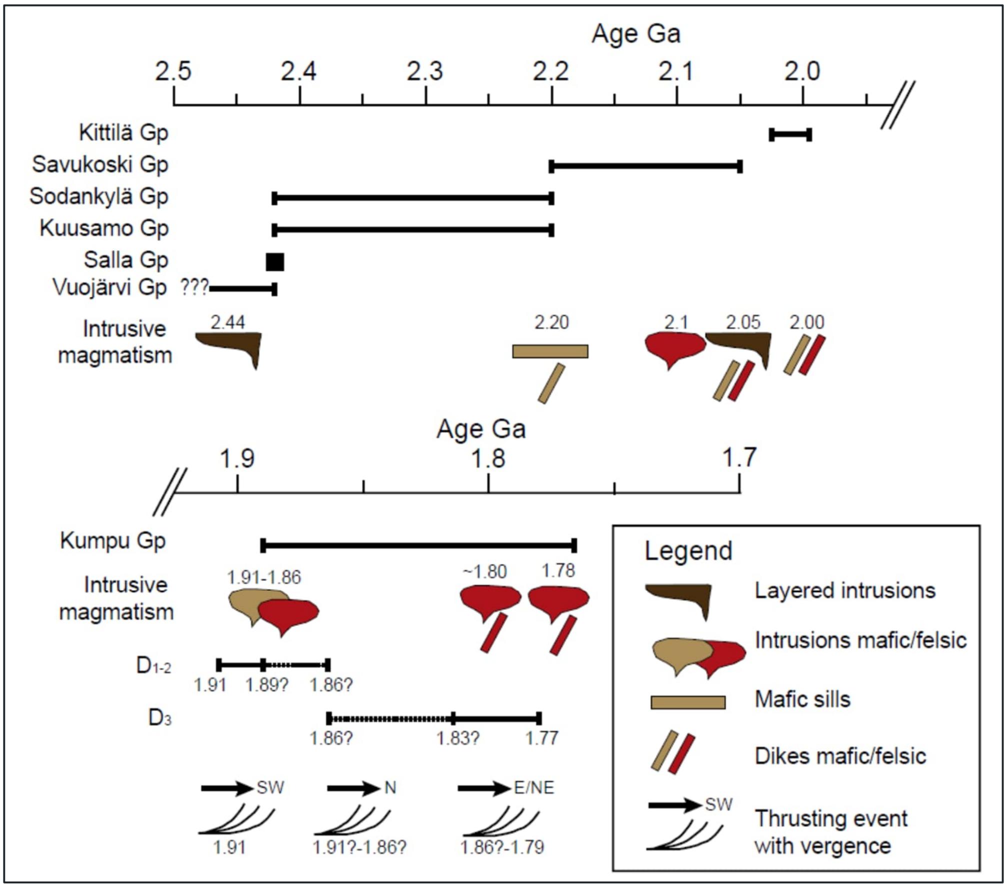


Figure 8-3 – A Schematic Sequence of the Lithostratigraphic Groups, Intrusive Stages and Deformation for the CLB

However, despite obvious structural controls on mineralisation, particularly at Ikkari, where strong WNW-trending foliation is developed related to shearing, there is some indication of a magmatic fluid input where multi-element geochemistry reveals a close association between gold and typically magmatic-related elements such as molybdenum and tungsten. Overprinting alteration events at Ikkari indicate the potential for multifluid sources as a control on gold mineralisation. It is possible that a magmatic fluid from a deeper intrusive source may have been somewhat responsible for the localization of gold mineralisation, especially high-grade gold, in favourable structural sites.

9 EXPLORATION

9.1 PREVIOUS EXPLORATION

Previous exploration on the Heinälamminvuoma exploration licence is limited to very wide spaced geochemical sampling by the GTK discussed in Chapter 6 of this report.

9.2 GEOPHYSICAL SURVEYS BY PREVIOUS OPERATORS

Geophysical surveys are also limited to those performed by the GTK in the 1970's and 80's discussed in Chapter 6 of this report. It should be noted that the products of these surveys, now used in the freely available GTK regional magnetic maps, provide a prolific source of baseline data which Rupert Resources utilized to develop the original exploration concept.

9.3 EXPLORATION UNDERTAKEN BY RUPERT RESOURCES

Focusing only on the work Rupert Resources has undertaken within the Rupert Lapland exploration licences, including the Heinälamminvuoma licence where the Ikkari discoveries is located, the following exploration programmes have been completed.

Exploration programmes commonly refer to "Area 1", a large target area, approximately 8 km by 6 km, in the far southwest corner of the Heinälamminvuoma licence which was defined in 2018 as being the most prospective portion of the tenement package and thus the focus for much for the exploration work. The outline of this broad area is shown in Figure 9-1 to Figure 9-3.

9.3.1. GEOPHYSICS

During May 2018 Rupert Resources conducted a permit-wide aeromagnetic survey using an Unmanned Aerial Vehicle (UAV), which, along with available regional geophysics data, was used as the basis for a regional structural study conducted by structural geology consultant Brett Davis, which highlighted the dominant E-W trending structures across the Heinälamminvuoma permit as being highly prospective for gold exploration (Davis, 2018).

The May 2018 detailed low-altitude magnetic survey represents the most detailed magnetic survey completed to date. This survey extended across the majority of the exploration permit package (Figure 9-1). In addition, a series of ground magnetic programmes were completed during 2020 across selected target areas in Area 1, including Ikkari and Heinä South, a gold occurrence 900 m NW of Ikkari. Ground magnetics were completed with a walking magnetometer + differential Global Positioning System (GPS) with 1 second sampling (GEM GSM-19W).

A ground gravity survey was completed in 2019, across the majority of the Rupert Lapland Project permit area, at a 200-m spaced grid resolution (Figure 9.2), with 3 416 measurements taken.

Since 2016, Rupert Resources has completed 27-line km of IP geophysics on the Rupert Lapland Project area.

A series of ground IP pole-dipole programmes were completed across specific targets in Area 1 during 2020 (Figure 9.3), using a GDD 32cRx receiver, GDD Tx4 5-kilowatt (kW) transmitter, $PbCl_2$ electrodes and stainless steel. Primary voltage was apparent resistivity and chargeability with 20 arithmetic time channels 80 millisecond (ms) each, 240 ms delay.

At Heinä South, 200-point measurements were taken across 8 lines with electrode spacing at 25 m and 50 m (transient time 2 seconds [s], full waveform measurement).

At Ikkari, an initial 200-point programme was completed with 9 initial profiles completed at 100 m line spacing, followed by an extension of the programme towards the east, with an additional 6 lines completed at 200 m line spacing.

At Saitta, a 100-point programme was completed across 2 lines with electrode spacing at 25 m and 50 m.

During spring 2022 a TITAN DCIP/MT survey was completed by Quantec Geoscience, across the Ikkari deposit and the Helmi deposit, a gold deposit discovered by B2 gold approximately 1km to the west of Ikkari. B2Gold was the principal client for the survey however it was performed as a collaboration with both companies receiving the full report and data. Two-dimensional (2D) inversions of the measured direct current (DC), chargeability (IP), and magnetotelluric (MT) data were provided along the N-S orientated survey lines. A total of 19, 2.2 km long lines were surveyed with spacing variable between 200 m and 400 m. Multiple survey lines cross the permit boundary with approximately 60% of the survey line length performed within the Rupert Resources Licence boundary (Figure 9-4).

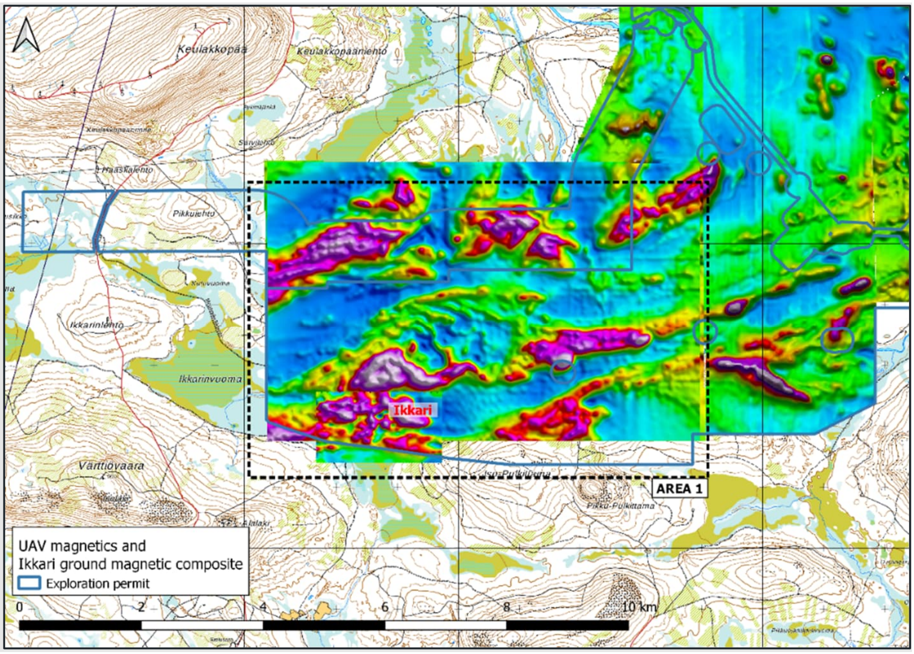


Figure 9-1 – Composite Magnetic Image of Rupert Lapland Exploration Permits

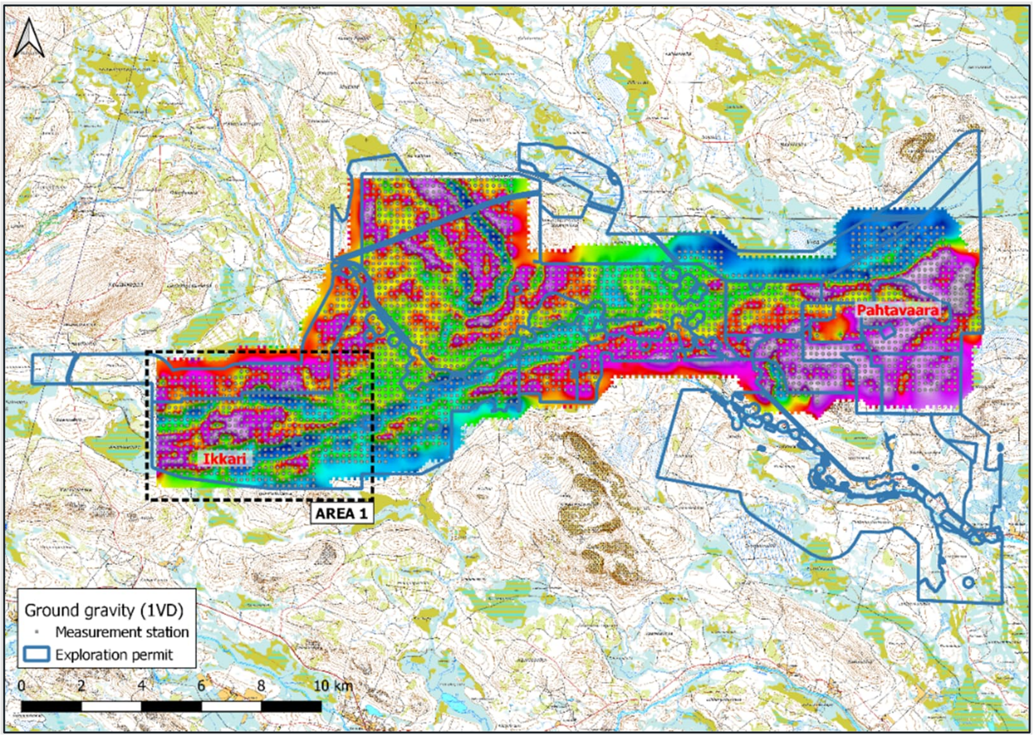


Figure 9-2 – Ground-gravity Programme with Points for Each Measurement Shown

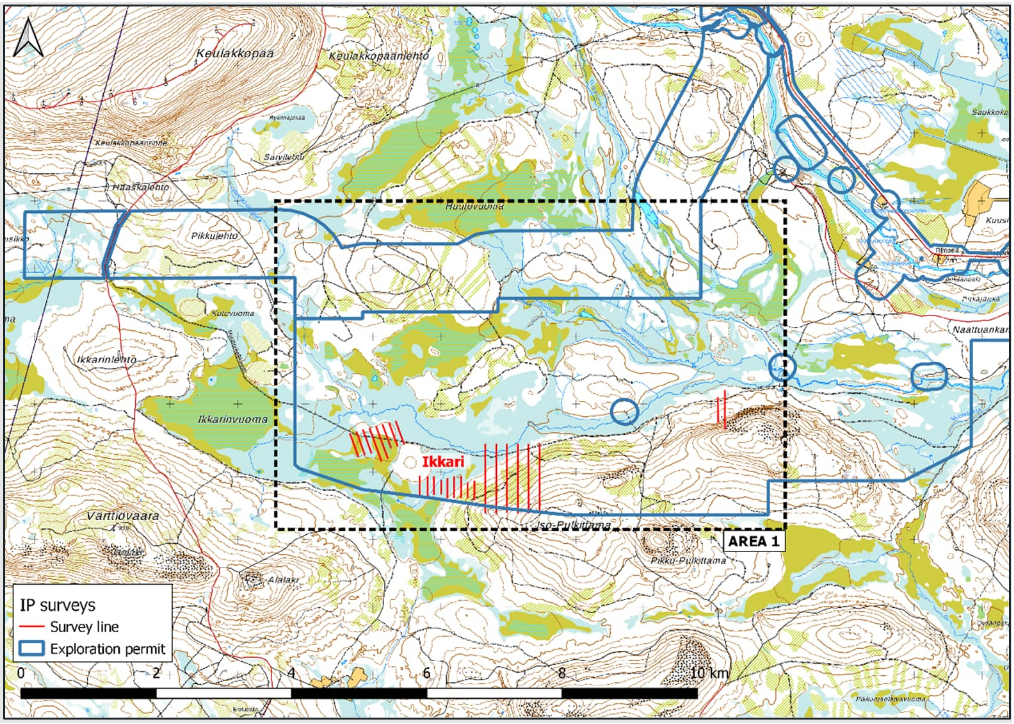


Figure 9-3 – Location of Pole-dipole IP Lines at Target Areas Within Area 1

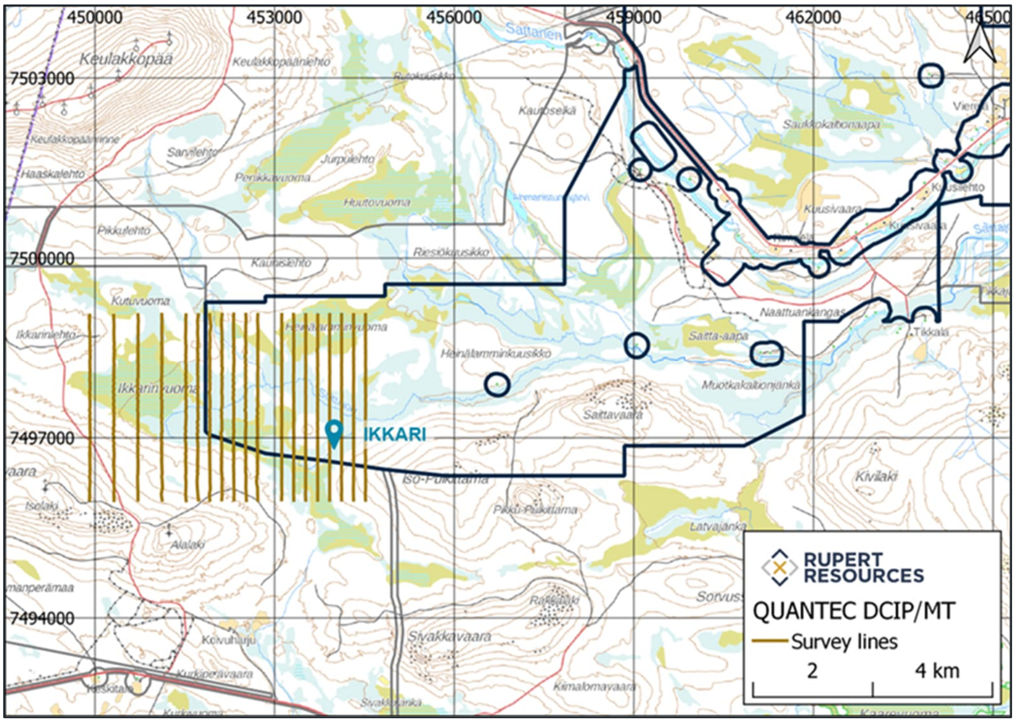


Figure 9-4 – Location of MT-IP Lines at Target Areas Within Area 1

Following successful extrapolation between the MTIP resistivity signature and the known underlying bedrock at Ikkari a further IP survey in 2024 extended the surveyed area >6 km to the east-northeast in with the aim to use resistivity to map the large-scale structural features specifically the geometry of the ultramafic - Kumpu Group quartzite contact at depth. 13 lines for 26 km were surveyed by GRM, which are shown in Figure 9-5.

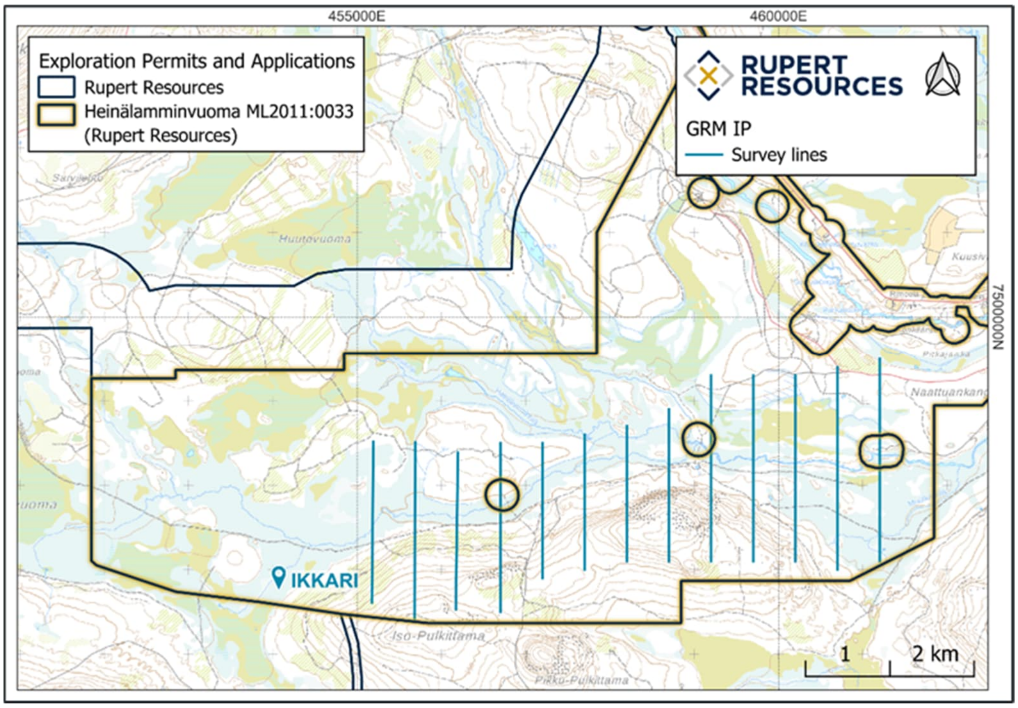


Figure 9-5 – Location of 2024 IP Lines east of Ikkari within Area 1

9.3.2. GEOCHEMISTRY

Initial work by Rupert Resources on the Rupert Lapland Project area, was focused on the area immediately surrounding the Pahtavaara mine. The bedrock mapping and boulder-hunting database of the Heinälamminvuoma permit area contains 1365 rock observations including assayed samples collected by Rupert Resources across the project area. However, in the vicinity of the Ikkari deposit significantly fewer rocks and boulders have been sampled by Rupert Resources, largely due to the lack of outcrop, extensive bogs and thick till cover sequences. However, where accessible, surface geochemical sampling has been undertaken in these areas (Figure 9-6).

In early 2019, Rupert Resources commenced a base of till sampling programme, using a flow-through sampler with a bandwagon mounted rig, across the extent of the Heinälamminvuoma permit aiming to traverse across the key identified structures and identify zones of gold anomalism in base of till soil samples. Infill base of till sampling was completed in areas that displayed anomalism in the first pass 'tram line' traverses (Figure 9-7 and Figure 9-8).

Follow up systematic drill testing of identified base of till gold anomalies was initiated with gold occurrences identified at several locations within the permit. At Ikkari the Initial 'tram line' BOT traverses yielded a single point anomaly of 0.2 ppm Au and this was followed up with closer spaced infill sampling that identified a cluster of >1 ppm Au anomalies. The first drill hole into geochemical anomaly (hole 120038) assayed 54 m grading 1.5 g/t gold from 25 m, including 4.7 g/t over 1 m from 35 m, 5.2 g/t over 2 m from 65 m and 5.7 g/t over 1 m from 71 m.

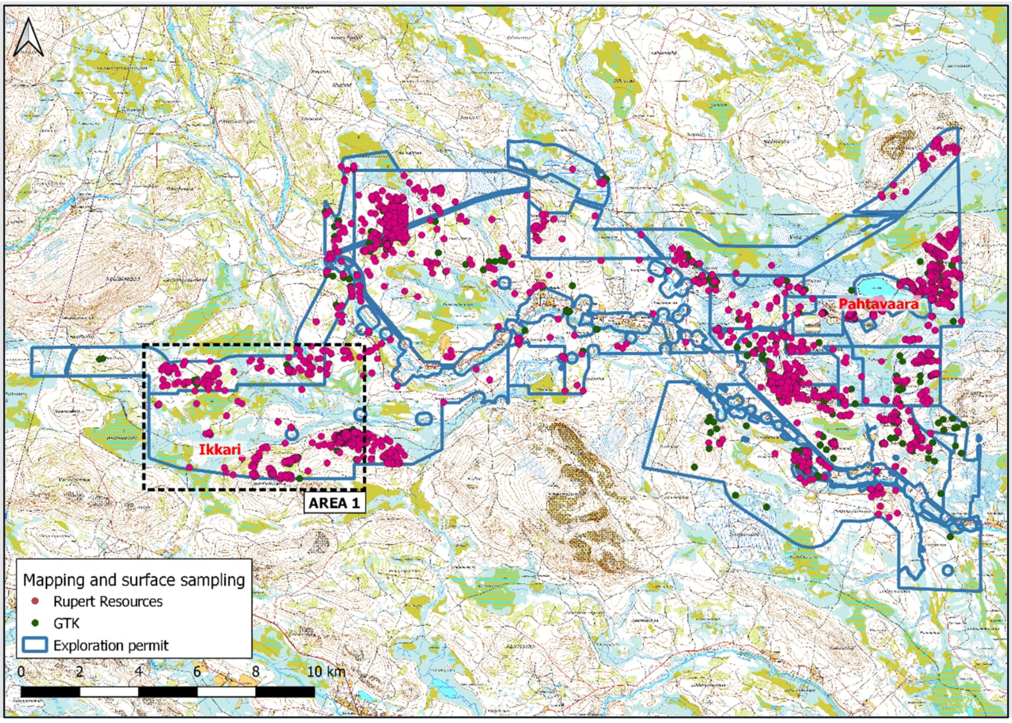


Figure 9-6 – Boulder and Outcrop Observations Undertaken by Rupert Resources

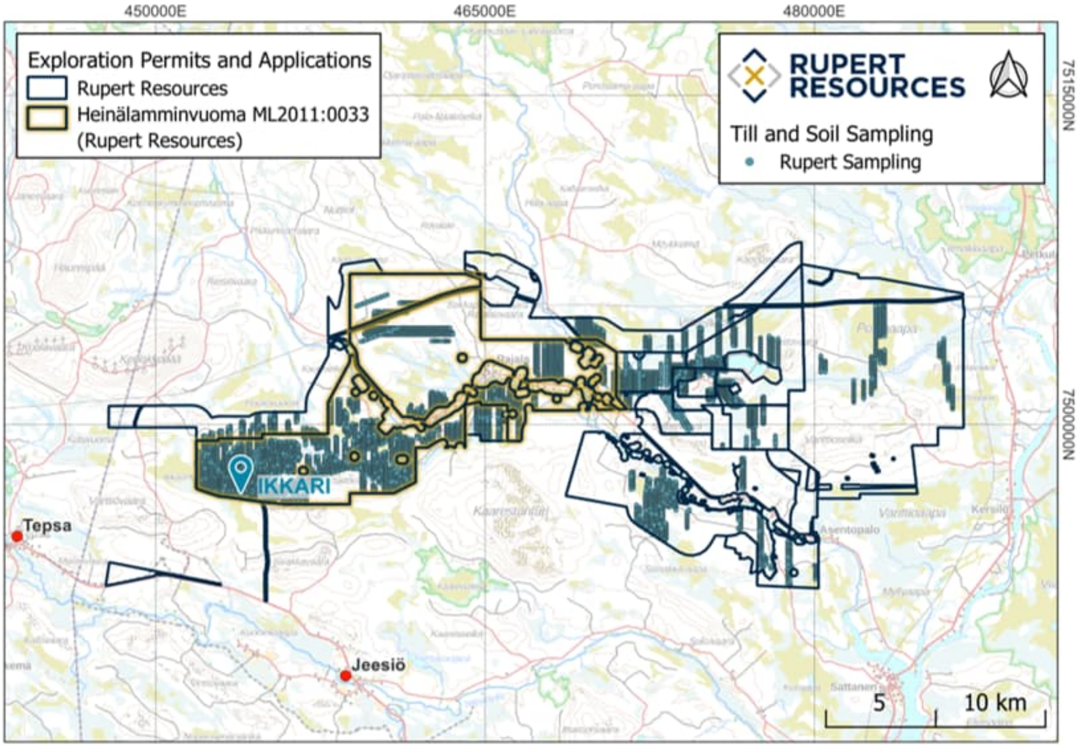


Figure 9-7 – Base of Till Locations Completed by Rupert Resources

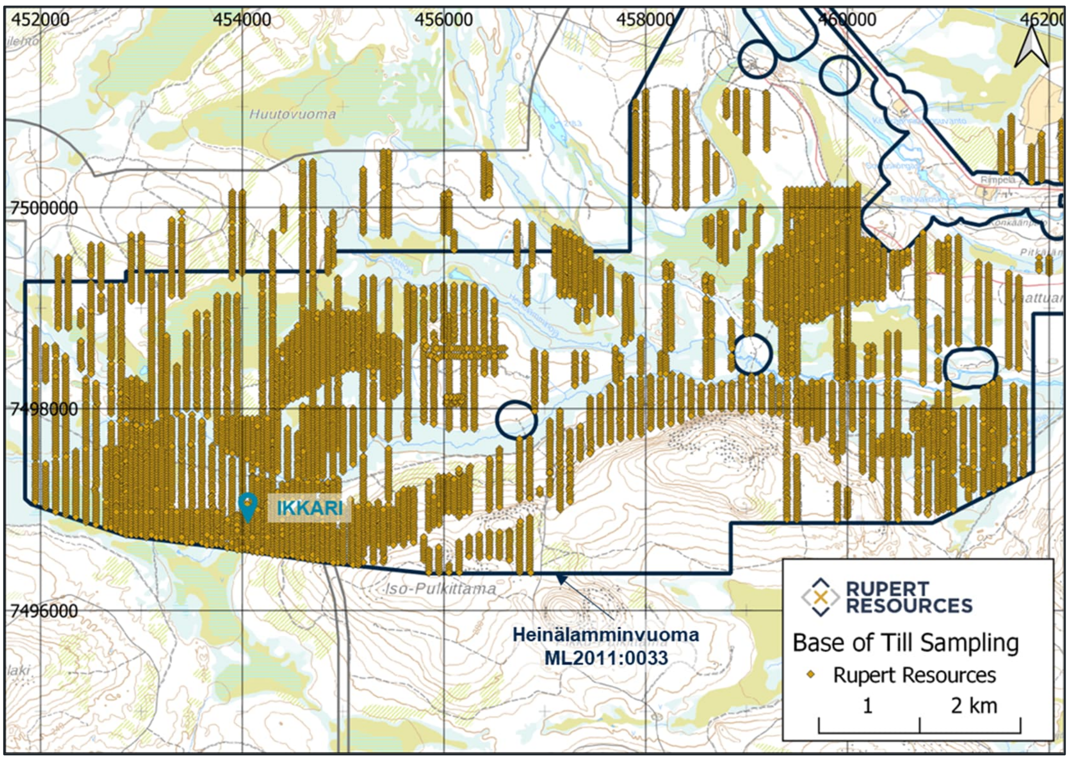


Figure 9-8 – Base of Till Locations Completed by Rupert Resources

10 DRILLING

10.1 DRILLING BY PREVIOUS OPERATORS

Considering initially the entire Rupert Lapland exploration licences, the vast majority of historic drilling has been carried out at the Pahtavaara Mine site, and near-mine areas with very little drilling completed elsewhere on the permits (Figure 10-1). No drilling has been undertaken by previous operators at or near the Ikkari deposit. Historical drilling across the Rupert Lapland Project area has been conducted by GTK, Outokumpu, Terra Mining, Scan Mining, Lapland Goldminers and Anglo American.

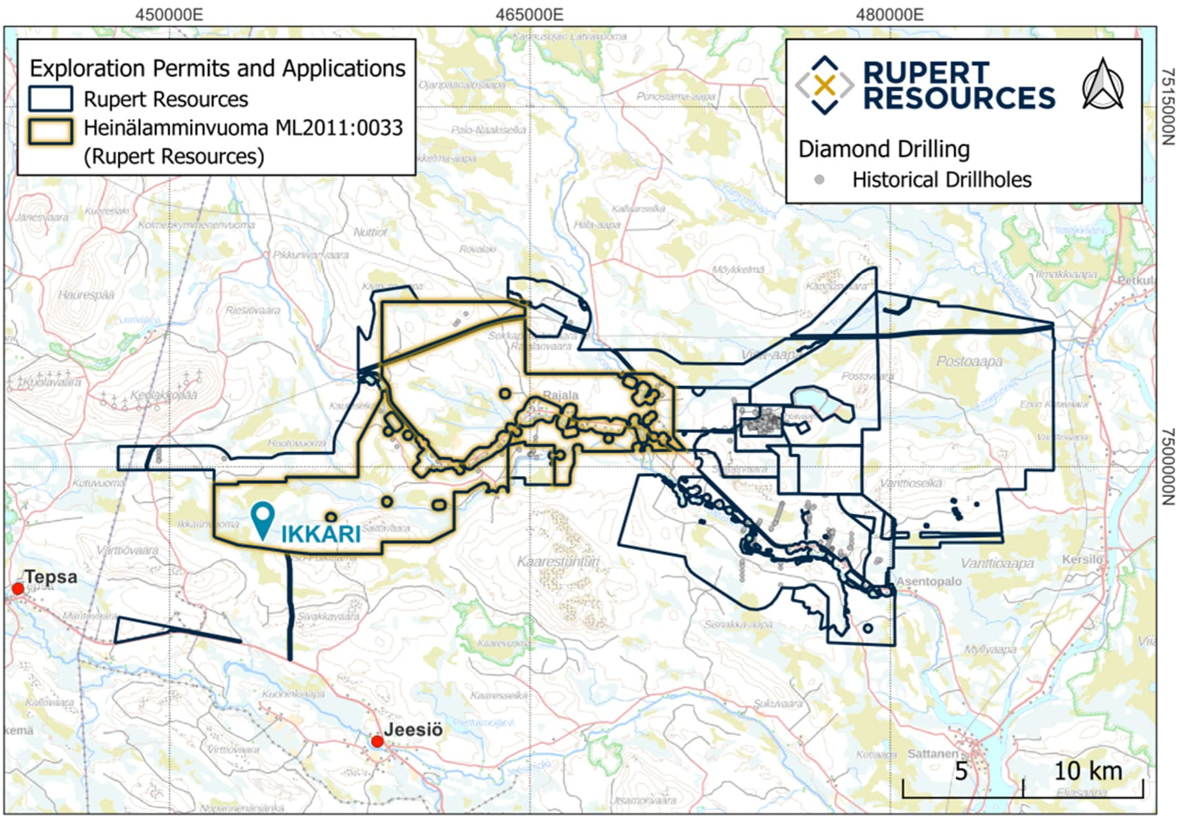


Figure 10-1 – Diamond Drilling on the Rupert Lapland Licence Area by Previous Operators

10.2 DRILLING BY RUPERT RESOURCES

Following an initial period, where Rupert Resources also focussed on the area immediately adjacent to the Pahtavaara Gold Mine (Figure 10-2), on care and maintenance at the time, focus was switched to greenfield exploration in 2018 with a focus on the SW corner of the Heinälamminvuoma permit area in the target area designated “Area 1” (see Figure 10-3).

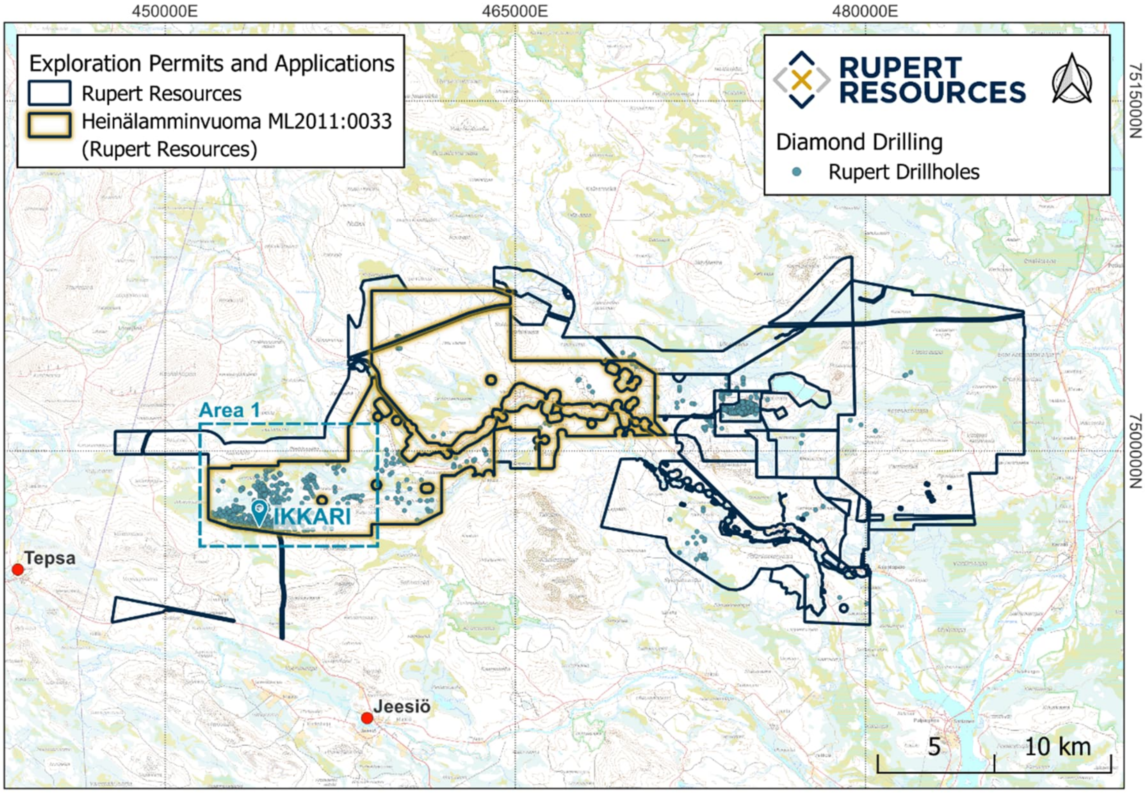


Figure 10-2 – Diamond Drilling on the Rupert Lapland Licence Area by Rupert Resources

Within the Heinälamminvuoma exploration permit area, Rupert Resources has used diamond drilling to predominantly target base of till gold anomalies. In late 2019, following the generation of base of till targets at Area 1, drilling was undertaken at specific prospect locations at Area 1. These drilling statistics are summarized in Table 10-1 and the locations of drilling to date in Area 1 are shown in Figure 10-3.

At Ikkari, an initial two drill holes in early April 2020 (drill holes 120038 and 120042), tested base of till anomalies along the E-W trend, at the possible margin of a magnetic anomaly. Both holes returned gold mineralisation over substantial downhole widths, hosted by sedimentary rocks, and both holes demonstrated strong foliation, shearing, occurrences of visible gold associated with intensive albite-sericite alteration and finely disseminated pyrite throughout.

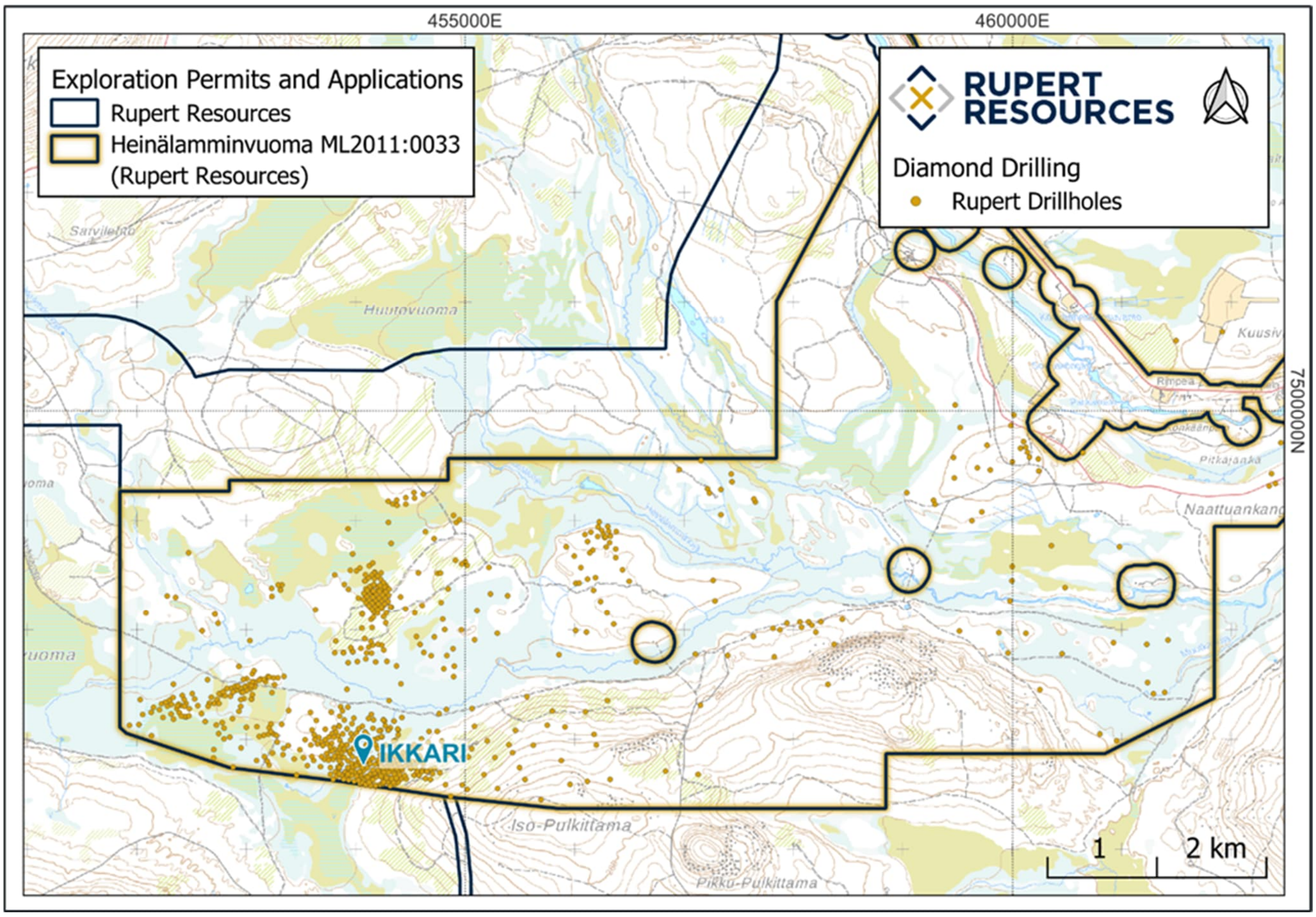


Figure 10-3 – Diamond Drilling on the Heinälamminvuoma Exploration Permit by Rupert Resources

Table 10-1 – Drill Hole Summary for Drilling Undertaken by Rupert Resources

Prospect	Year	DH Type	Holes	Metres	% of Total
Heinä South	2019	Diamond	2	200	0
	2020		22	3 980	2
	2021		28	4 929	3
	2022		32	6 805	4
	2023 (to June)		2	1 123	1
	June 2023 to End 2024		23	5 451	2
Heinä North	2019	Diamond	10	1 612	1
	2020		2	245	0
	2023 (to June)		3	565	0

Prospect	Year	DH Type	Holes	Metres	% of Total
	June 2023 to End 2024		-	-	-
Heinä Central (*inc Ikkari North and sterilisation drilling)	2019	Diamond	19	3 593	2
	2020		10	2 416	1
	2021		39	7 540	4
	2022		39	11 745	6
	2023 (to June)		16	4 935	3
	June 2023 to End 2024		16	3 434	1
Island North (*inc portion of sterilisation drilling)	2019	Diamond	1	152	0
	2020		10	1 791	1
	2021		7	1 405	1
	2023 (to June)		11	2 327	1
	June 2023 to End 2024		13	2 803	1
Saitta (*inc sterilisation drilling	2020	Diamond	11	1 960	1
	2021		2	534	0
	2022		17	3 507	2
	June 2023 to End 2024		7	3 383	1
Ikkari	**2020**	**Diamond**	**62**	**20,320**	**10**
	2021		**75**	**36,049**	**18**
	2022		**85**	**35 568**	**18**
	2023 (to June)		**46**	**22 069**	**11**
	June 2023 to End 2024		**70**	**23 145**	**9**
Others	2019	Diamond	20	2475	1
	2020		2	430	0
	2021		17	2 722	1

Prospect	Year	DH Type	Holes	Metres	% of Total
	2022		65	11 606	6
	2023 (to June)		11	2 190	1
	June 2023 to End 2024		64	13 534	5
Total			667	192 562	100%

Notes:

* Including later extensions to drill holes and wedges.

** Including holes such as metallurgical holes not assayed, and therefore not included in the resource estimation (**Section 14.2**).

Reported as per prospect on coding in database, not all holes are necessarily targeting the same mineralisation occurrence. Errors may occur due to rounding.

Hole 120038 intersected 54 m grading 1.5 g/t Au from 25 m, including 4.7 g/t over 1 m from 35 m, 5.2 g/t over 2 m from 65 m and 5.7 g/t over 1 m from 71 m.

Hole 120042 intersected 137.2 m grading 1.8 g/t Au from 10.8 m, including 7.1 g/t Au over 14 m from 23 m and 10.6 g/t over 3 m from 27 m.

Following these initial results, bold step out drilling was pursued along the interpreted strike, targeting further base of till anomalism and the magnetic anomaly margin. These holes successfully intersected further mineralisation and indicated a potential strike length of 450 m.

Hole 120065 intersected 2.1 g/t Au over 31.0 m from 53 m including 23.7 g/t Au over 1 m. The hole targeted near surface mineralisation and extended the known mineralized strike eastwards. Hole 120067 intersected 1.3 g/t Au over 172.4 m from surface including 12 m at 2.6 g/t Au with the hole ending in mineralisation, extending the known limits 100 m to the north of hole 120042 (1.8 g/t Au over 137.2 m).

These confirmed the presence of a significant mineralized system at Ikkari and further drill testing was prioritized, with some 62 holes for 19 084 m completed during 2020. Wide-spaced drilling traverses were completed between the initial holes in the east and west as well as testing extensions to the trend of base of till anomalies along strike that now extends in excess of 1 km.

With the continued success of the drill programme and the release of the maiden MRE in September 2021 infill drilling on 40 m sections with a 40 m spacing on section commenced immediately, synchronous with further step out drilling to the east, northwest and at depth. An updated resource estimate was published in November 2022 alongside and in support of the PEA of the project, assessed alongside the restart of the Pahtavaara Mine. The November 2022 resource update included 78 holes for 36 398 m in addition to the 36 635 m from 102 holes that were completed at the time of the maiden MRE.

With the successful publication of the PEA and a high-margin operation envisioned, further infill and step-out continued through the 2022-2023 season and with a further 38 742 m from 89 holes completed, including holes targeted primarily for geotechnical, metallurgical and hydrogeological purposes. Principle targets of the 2022-23 drill campaign were to extend mineralisation to the west, at depth, a direction in which the deposit remains open, and to convert all resources above -300m RL to the Indicated Category ahead of a PFS initiating in mid-2023.

The MRE outlined in this technical disclosure was initially published 28 November 2023 and filed on Sedar 12 December 2023. The drilling cut-off for the resource update was June 2023. Subsequent to the initial publication of the MRE and ahead of publication of the PFS a further 70 holes for 23 145m meters have been drilled at Ikkari and the immediate surrounding including exploration holes, geotechnical holes and holes designed to recover further material for metallurgical testing. Less than 10% of these meters occur within the mineralised zone used for MRE in November 2023 and outlined in this report.

10.3 HOLE PLANNING AND SET-UP

Diamond Drilling at Ikkari from 2020 to 2023 was undertaken predominantly by contractors MK Core Drilling, Arctic Drilling Company (ADC), Kati and Comadev. The core diameter used was predominantly NQ2 (50.7 mm core diameter) with WL76 (57.5 mm core diameter) used in some earlier drillholes.

Rupert Resources has an in-house surveyor and field technicians responsible for drill hole setup. Drill holes are planned by the geology team and details passed to the surveyor; this includes the collar coordinate, the coordinates for the planned end of the hole, the azimuth and dip.

The surveyor uses a Differential Global Positioning System (DGPS) to locate the collar location, orients the hole direction from the azimuth determined by the DGPS (according to direction between start and end coordinates).

The collar location is marked by a wooden marker (which has the planned hole number, the coordinates, azimuth and initial dip written on it). The planned azimuth of the hole is also marked with another survey post oriented in the planned drilling direction. An additional ‘marker’ peg is positioned to assist with the drill rig orientation. All orientation ‘pegs’ are annotated to indicate which is the ‘front peg’ (with the – HoleID) and which is the ‘back peg’ (also with the HoleID) ensuring holes are drilled in the correct orientation on the line defined by the pegs.

The drillers use the two orientation guide pegs to set up and orient the drill rig correctly.

10.4 SURVEYING AND ORIENTATIONS

The actual collar position is measured using DGPS total survey equipment once the drill rig as left the drilling location, in all drill holes, casing through the overburden is not removed. The elevation of the drillholes is measured at top of the casing, the same point that the downhole depth is measured from during drilling; this maybe up to 0.5 m above the surface dependent on snow conditions and casing may be cut to ground level during the following summer due to health and safety considerations.

The drilling contractor provides downhole surveys upon completion of the drill hole; intermediate survey may also take place during drilling. Survey tools are dependent on the drilling contractor used. To date Reflexgyro, DeviFlex/DeviGyro, OMNI-IQ, SPRINT-IQ and SPT downhole survey instruments have been used at Ikkari. Considering Reflexgyro, DeviFlex/DeviGyro tools, these are gyro-based tools that measure dip and azimuth every four meters, starting from the bottom of the hole and proceeding upwards to the drill hole collar. The survey data is delivered to the supervising geologist via email as csv- and ds-format using the instrument software or more recently uploaded to the cloud based 'Devi-Cloud' or 'imdexhub-IQ' for download by the supervising geologist or senior database geologist. The azimuth field is re-processed at all depths from the collar when the collar survey is available. In the case of the OMNI-IQ, SPRINT-IQ and SPT tools, these are north seeking gyro, and thus azimuth and dip are measured independently in the drillhole and no post processing with the collar azimuth is required.

10.5 DRY BULK DENSITY COLLECTION

Since initiation of drilling in April 2020, the majority of diamond drill holes have been routinely measured for density. A 10 cm to 15 cm piece of core from every core box, or every 5 m, is weighed first in air, and then in water. These values are recorded in the acQuire database, which calculates the SG using formula SG = ρsubstance / ρH2O, [dry weight/(dry weight-weight in water)].

The logging geologist marks additional measurement points to core boxes in cases of special rock types, for example massive sulphides or breccias.

Given that the rock mass at Ikkari is almost all intact fresh rock containing few voids, and these are avoided during SG determination, the SG is a good match to the bulk density. The density of the lithologies at Ikkari range between 2.5 to 4 g/cm^3 with an average value of 2.86 g/cm^3.

For this resource update the density measurements were estimated into blocks using the extensive database of 11 468 measurements. Density data has interrogated both according to logged lithology (Table 10-2) and the samples position within the geological model (Table 10-3).

Table 10-2 – Ikkari Gold Deposit – Density Statistics (g/cm^3) By Logged Lithology.

Logged Lithology	Median	Mean	1st Quartile	3rd Quartile	Number
IFO (Pre-mineralisation Dyke)	2.81	2.84	2.78	2.89	10
IGB (Gabbro)	2.86	2.85	2.80	2.94	645
IUO (Intrusive Ultramafic)	2.92	2.91	2.87	2.96	112
MQZ (Quartzite)	2.71	2.73	2.68	2.75	440
MSB (Black Shale)	2.78	2.76	2.73	2.83	748
MSCU (Ultramafic Schist)	2.94	2.94	2.88	3.01	3 620
SCO (Conglomerate)	2.71	2.72	2.69	2.74	155
SSI (Siltstone)	2.87	2.88	2.77	2.98	856
SST (Sandstone)	2.75	2.75	2.72	2.78	1 792
UKO (Komatiite)	2.88	2.89	2.86	2.91	2 480
USP (Serpentinite)	2.90	2.88	2.86	2.94	54
VBA (Basalt)	2.89	2.86	2.80	2.93	61
VPI (Ultramafic pillow lava)	2.88	2.88	2.86	2.89	12

Logged Lithology	Median	Mean	1st Quartile	3rd Quartile	Number
VTUI (Ultramafic tuffitic breccia)	2.69	2.67	2.61	2.74	31
VUO (Ultramafic volcanoclastic)	2.90	2.90	2.87	2.93	396
All	**2.87**	**2.87**	**2.78**	**2.87**	**11 468**

Note: Key: g/cm^3 = grams per cubic centimetre

Table 10-3 – Ikkari Gold Deposit – Density Statistics (g/cm^3) By Modelled Lithology

Lithology Series	Median	Mean	1st Quartile	3rd Quartile	Number
Ultramafic (Talc Altered)	2.88	2.89	2.86	2.92	3 238
Northern Felsic Sediments	2.75	2.75	2.72	2.78	1 746
Southern Felsic Sediment	2.75	2.72	2.69	2.75	252
Black shale (MSB)	2.79	2.77	2.74	2.83	765
Ultramafic Schist (MSCU)	2.92	2.93	2.87	3.00	3 691
Gabbro	2.86	2.85	2.79	2.94	661
Internal Felsic	2.85	2.86	2.74	2.98	981

Note: Key: g/cm^3 = grams per cubic centimetre

As would be anticipated, the felsic lithologies, both logged and modelled, in all locations, have a significantly lower density than the ultramafic lithologies. The increased density of the internal felsic, relative to the other felsic units, reflects the greater pyrite and/or hematite/magnetite content commonly found within this well mineralised lithology; it also has the most variance of any domain. The increased density of the Ultramafic Schist (MSCU) over the talc altered ultramafic is also a likely reflection of the pyrite and magnetite addition in the MSCU as a result of mesothermal alteration. Overburden density is not measured by Rupert Resources as it is not consolidated and therefore not amenable to the Archimedes methodology. It hosts no mineralisation and is therefore relevant to pit optimizations and waste stripping only. A density of 1.9 g/cm^3 has been assigned to the overburden based on published literature on the overburden in Lapland, Finland.

10.6 DRILL DATABASE

Data entry in the company database is achieved through a combination of direct entry of data by Rupert Resources personnel and the direct import of third-party data from the raw files which are subsequently archived in a dedicated and secure location on the file server with set naming conventions.

Logging of drillcore is now performed directly into a SQL based relational database management system designed by acQuire. This ‘offline’ data is uploaded to acQuire database utilizing a dedicated import object upon completion of each drillhole.

Multiple validations are built into the database to ensure the integrity of data entered, for all manual data entry fields, except for comments fields, validation look-up tables are used to ensure consistency with company validation codes.

Other validations include but are not limited to:

- Ensuring continuity of downhole data (both logging and sampling);
- Preventing overlapping intervals (both logging and sampling);
- Preventing duplication of data;
- Ensuring downhole data matches the depth of the drillhole;
- Ensuring all the required fields are populated;
- Range violations – flagging data that falls outside the expected range;
- Data type validations such as text, numeric, date etc… (import specific); and
- Validation of format for all import and specifically elements and detection limits for assay data.

The following information is currently stored in the companies acQuire database, the input methodology is given alongside each piece of data:

- Geological logs covering, lithology, alteration, mineralisation, textures and structures; logged directly into the database.;
- Sampling intervals including insertion and QC standards, blanks and duplicates, logged directly into the database.;
- Basic geotechnical logs on all holes (RQD and recovery), logged directly into the database.;
- Detailed geotechnical logs, including point load tests, on selected drillholes, logged directly into the database.;
- Magnetic susceptibility readings, logged directly into the database.;
- SG measurements, logged directly into the database.;
- Collar surveys imported into the database from software export.;
- Downhole surveys directly imported into the database from software export.; and
- Assay data, directly imported to the database from laboratory results.

At the completion of each drillhole the Senior Geologist responsible for database management performs a series of checks on data, this includes but is not limited to checking planned collar location entered by the geologist against surveyed collar location, and checking downhole survey against collar survey. Sampling intervals are checked against logged areas of no recovery to ensure these are honoured in the sampling wherever possible.

Further quality assurance checks are completed by the Senior Geologist responsible for database management at the completion of each drilling season. These include but are not limited to, ensuring agreement between sample intervals without assay data and logged intervals of no recovery and vice versa, ensuring all intervals where logging data occurs has assay data assigned, ensuring all drillholes have collar and downhole survey data. Further to this raw laboratory assays, for several batches, are compared to the database to ensure the continued successful import of assay data.

Upon import and acceptance of assay results the drill hole is then “locked” in the database such that only users with assigned credentials may edit the information related to these drill holes. Each employee has their own login credentials for the database and as such access to unlock drill holes is limited to the Senior Geologist responsible database management, the Exploration Manager, and the Resource Geologist. This ensures the long-term integrity of the database.

The Ikkari database used in this resource evaluation contains 255 diamond drill holes (111 896 m) (Figure 10-3). The difference from the table above (Table 10-1) owes to the exclusion of holes drilled for Metallurgical purposes without assays, the double counting of meters from wedges off drillholes due to the way they are stored in the database and the exclusion of holes flagged as ‘Ikkari’ in the database but at a significant distance from the resource.

The drilling database used in this resource calculation contains 103 839 gold assays and 84 133 multi-element assays. The database also contains 24 456 downhole survey stations and 11 427 SG measurements.

Core recoveries are in general excellent with an average of 98% recovery achieved from bedrock during diamond drilling; this excludes the barren overburden where recovery is not attempted. This average is also negatively affected by poorer recoveries close to surface in the black shale, another predominantly barren unit. Considering only the estimation domains the recovery is >99%. Core recovery achieved is sufficient for resource estimation.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 CHAIN OF CUSTODY, SAMPLE PREPARATION, AND ANALYSES

Chain of custody from drill rig to Rupert Resources facilities is dependent on the number of drill rigs operating at any given time. When two rigs or fewer are operating at Ikkari, the drilling contractor or Rupert Resources employees bring the core to Rupert Resources' facilities, now located 10 km south of Sodankylä. During peak drilling season when more drill rigs are operating, at the end of the shift drill core is transported by the contractor to a prearranged laydown yard, from where it is met by a single, local transport contractor for transportation to Rupert Resources facility. Whichever methodology is in use at the given time, drill core is constantly under supervision of either the drill contractor or the transport contactor until delivery to the Rupert Resources facility and final delivery into the 'core shed'.

The Rupert Resources facility is secured at all times by continuous fencing and a gate that can only be opened through pre-registered mobile phones together with locks on all doors to the internal storage areas including the cold storage area for drill core from previous drilling seasons.

Once inside the core shed, the sample handling team then checks that core samples are in right order, move the core inside the trays against its left border and assembles any broken segments if possible.

After organizing the core boxes and core samples, each piece of the core is taken out from the core box and arranged in the rail of the logging table to draw continuous bottom line on the core, and downhole direction pointed with arrows along the line. A solid line is used to represent core orientated from two or more independent orientation marks, core orientated from only a single orientation measurement is marked with a dashed line. These high and low confidence criteria are reflected in the orientated structural logging. Reflex ACT III orientation tool is used by all drilling contractors to achieve oriented core. Following orientation, the core is measured, and metre intervals are marked on core boxes and on core with black marker pen.

Core logging is performed directly into the acQuire database. Log sheets to be filled include lithology, structural data, magnetic susceptibility, core recovery and rock quality index (RQD) sheets as well as the sample data sheet which includes company quality check samples.

The geotechnical logging includes the magnetic susceptibility and core recovery data. Once the metres are measured and marked correctly onto the core, the magnetic susceptibility of the core is measured. This is done metre by metre, scanning between each metre mark by using a Terraplus KT-10 handheld magnetic susceptibility and conductivity meter. KT-10 has a scanner mode, which automatically calculates the average susceptibility for each scanned interval.

RQD values are measured at each metre interval and written on the left side of each metre line in the core box with pencil. acQuire calculates RQD percentage automatically from given interval length and RQD centimetres.

The acQuire sampling table automatically creates one-metre-long sampling intervals. It also reminds the operator to enter a Quality Control (QC) sample, company blank or commercial standard every tenth sample. Logging geologist inserts one core duplicate per 20 samples and marks it also to the core box. Unique sample numbers are assigned to all samples including the QC samples based on sample books. QC samples also include pulp duplicates. The preparation laboratory has been instructed to insert one pulp duplicate in every 20 samples. Pulp duplicates have the same sample ID number the original sample, with suffix PD. Sampling intervals are marked on the core box (below a certain interval) with a red marker. Places where the sampling intervals begin and end are marked with red arrows (on the core box and on the core) and the sampling number is written with the first six numbers at the top right edge of the core box and the last three numbers under each sample interval on the core box below the core at the beginning of the interval. The QC samples are marked on the core boxes. All sampling documents for a batch of samples, along with sachets containing standards and blanks and corresponding sample tickets are placed in a sealed bag for dispatch along with the batch of samples.

After all the logging and sampling has been undertaken, all the core boxes are photographed. Two photographs are taken: The first of dry core and second of wet core. Core photographs are automatically uploaded to the Imago cloud which is available immediately via an online portal or through links in the modeling software.

Drill core is cut at the Rupert Resources core logging and sampling facility by a Rupert Resources technician. Cutting is done next to the orientation line, and the half with the line remains in the core box. A minority of core has been shipped to ALS for core cutting during the busiest drill periods, where this has occurred the same procedures have been implemented. After the core has been sawn, the samples (half core samples, blanks, core duplicates and standards) are packed in plastic bags tagged with sample tag from the sample book and are packed onto EUR-pallets to be shipped to the laboratory. During packing each sample is weighed and the information is added to the database.

Once a batch has been packed wooden lids are screwed onto the pallet sides and further metal or plastic straps wrapped around the pallet and tightened using a rachet mechanism. Upon arrival of the samples at the laboratory visual checks are performed on the pallet to ensure integrity of the samples.

Geologists are responsible for creating new sample batches and sending the sample submittal form and assay order form to the laboratory. Sample shipment is requested and followed up by the Rupert Resources technician, who handles the contacts with the courier company.

The main laboratory used by Rupert Resources is ALS Minerals at Sodankylä, Finland (prep lab) with gold assays performed at ALS Geochemistry in Rosia Montana, Romania, an ISO 17025 accredited laboratory. Approximately 18% of samples (Table 11-1) have been prepped at ALS Outokumpu, another preparation laboratory in Finland with analysis again performed at ALS Geochemistry, Rosia Montana, Romania. The assay method in use is Au-AA26, Au by fire assay 50 g sample weight and AAS finish (0.01 to 100 ppm). Preparation methods include CRU-31 fine crushing minimum 70% to <2 mm, and PUL-24e, pulverizing the entire sample (max 3 kg) minimum 85% to 75 microns (µm). Samples greater than 3 kg are split prior to pulverizing with method SPL-22.

After pulverizing, a 250 g extra split is packed separately and returned to Rupert Resources for use in umpire lab checks. The remaining pulp material is also returned to Rupert Resources for long term storage. The over limit samples (>100 ppm Au) are automatically re-assayed via fire assay with gravimetric finish, code Au-GRA22. 48 elements, namely silver (Ag), aluminium (Al), arsenic (As), barium (Ba), beryllium (Be), bismuth (Bi), calcium (Ca), cadmium (Cd), cerium (Ce), cobalt (Co), chromium (Cr), caesium (Cs), copper (Cu), iron (Fe), gallium (Ga), germanium (Ge), hafnium (Hf), indium (In), potassium (K), lanthanum (La), lithium (Li), magnesium (Mg), manganese (Mn), molybdenum (Mo), sodium (Na), niobium (Nb), nickel (Ni), phosphorus (P), lead (Pb), rubidium (Rb), rhenium (Re), sulphur (S), antimony (Sb), scandium (Sc), selenium (Se), tin (Sn), strontium (Sr), tantalum (Ta), tellurium (Te), thorium (Th), titanium (Ti), thallium (Tl), uranium (U), vanadium (V), tungsten (W), yttrium (Y), zinc (Zn), zirconium (Zr) have been routinely assayed using method ME-MS61, four acid digestion with Inductively Coupled Plasma Mass Spectrometry (ICP-MS) finish (Ultra Trace Level Method –by HFHNO3-HClO4 acid digestion, Hydrochloric Acid (HCl) leach, and a combination of ICP-MS and Inductively Coupled Plasma – Atomic Emission Spectroscopy [ICP-AE]). Multi-elements are assayed by ALS Geochemistry in Loughrea Ireland. All ALS laboratories are internationally accredited in accordance with ISO 17025 (ISO, 2005).

Samples from some drill holes, amounting to ~7% of all samples, were assayed for gold in Eurofins Labtium Sodankylä (Table 11-1) utilizing their equivalent Au-705P method, gold assay 50 grammes (g) by fire assay with ICP-OES finish. Eurofins Labtium is ISO/IEC 17025 accredited by FINAS, the Finnish accreditation service. Labtium preparation method was agreed to match Rupert Resources' normal procedure at ALS. Jaw crushing of the samples to >60% less than (<)2 mm (method 31) with compressed air cleaning of jaws between samples, pulverizing the whole sample (max 3.5 kg) in one milling (method 50, LM5). After pulverizing the whole pulp is sampled to subsamples for following Fire Assay analysis. The pulp rejects are packed in plastic bags and one sub sample is forwarded to ALS Geochemistry in Loughrea Ireland for Multi Element analysis. The pulverizing puck and the bowl are cleaned by pulverizing barren quartzite.

In 2021 three holes, nine batches were also sent to CRS for preparation. Gold for these batches was assayed by their operational partner the ISO 17025 accredited MSA laboratories in Langley Canada. The preparation method was identical with ALS and Labtium procedures (PRP-999 and PWA-500), assay method was FAS-121, Au (0.005-10 ppm) by trace fire assay (50 g nominal sample weight), aqua regia digest and analysis by AAS. Overlimit assay for assays 10 to 1000 ppm was FAS-425, gravimetric fire assay.

All core is under custody from the drill site to the core processing facility. The Company's QA/QC programme includes the regular insertion of blanks and standards into the sample shipments, as well as duplicate sampling. Standards, blanks and duplicates are inserted at appropriate intervals. Approximately five percent (5%) of the pulps are sent for check assaying at a second lab (umpire split 250 g). Core recovery in the mineralized zones has averaged >99%.

Table 11-1 – Ikkari Assay Samples by Laboratory

Preparatory laboratory	Number of Samples	Assaying Laboratory	Number of Samples
ALS – Sodankylä	74 570	ALS – Romania	93 080
ALS – Outokumpu	18 510		
Eurofins / Labtium	7 747	Eurofins / Labtium	7 747
CRS	3 011	MSA	3 011

Source: Internal Rupert Resources Ltd database, 2023

11.1.1. ASSAY QUALITY CONTROL

For drilling carried out since the beginning of exploration until present the following sets of data have been reviewed and statistically assessed:

- CRM submitted by Rupert Resources to the independent assay laboratories;
- CRM inserted internally by the assay laboratories;
- Sample pairs, including drill core duplicates, pulp duplicates and pulp replicates (lab duplicates) and external duplicates (umpire duplicates); and
- Barren samples (“blanks”) submitted by both Rupert Resources and the assay laboratory.

Standard failures are defined internally as those greater than three times the standard deviation from the certified value. Where this occurs, a re-assay is requested for both the failing standard and the 10 samples either side of the failure; given the insertion rate this covers all samples to the next quality control sample in each direction. Warnings are generated where a standard assays outside of two times the standard deviation from the certified value. Persistent failures, in one direction, between 2-3 times the standard deviation are also re-assayed.

Blanks are monitored for contamination issues with re-assays typically requested for a value over ten (10) times the detection limit is returned. Lower-level contamination issues are routinely discussed with the respective laboratories during regular monthly meetings.

Duplicate data is monitored on a quarterly basis. With the expected differences between duplicate pairs now well established at Ikkari, duplicates are monitored for decreased precision that could result from sample preparation or analytical issues.

11.1.2. QC DATA

QA/QC data from sampling and analyses have been compiled in acQuire 4 relational database. The relevant information has been downloaded for statistical review and analysis. Presented in this report are only the Standards and blanks submitted by Rupert Resources as well as all data pairs, internal QC standards by the laboratories are not presented here.

Blanks:

- Submitted by Rupert Resources to each of the three laboratories (ALS, Eurofins (Labtium), CRS)

CRM (Standards):

- Submitted by Rupert Resources to each of the three laboratories (ALS, Eurofins (Labtium), CRS)

Data Pairs:

- Core duplicates (quarter core pairs);
- Pulp duplicate (duplicate samples taken after pulverized to >85% <75 µm);
- Lab duplicates (duplicates samples taken from within one pulp sachet); and
- Umpire checks (Pulp split sent to second laboratory).

Blanks

Analyses on blanks have been carried out on blank samples submitted by Rupert Resources and on inserted blanks inserted by laboratories, as part of the laboratory QA/QC procedures. The blank material Rupert Resources has been using and continues to use is quartz gravel provided by Sibelco Nordic/Nilsiä kvartsi. Rupert Resources' QA/QC routine with the fire assay method stipulates submitting blanks at the rate of 1 in 20 samples which is the equivalent rate of 1 in 18 primary samples taking both core duplicates and CRMs into account.

Table 11-2 and Figure 11-1 summarise the results of assaying blank samples. For the great majority of analyses, the blanks returned less than detection limit results. A total of 4 blanks have returned assays over three times the detection limit and no systematic contamination from high-grade samples has been noted.

Table 11-2 – Ikkari Gold Deposit Blanks

Standard	Assay Method	Laboratory	Number	Expected Value	Mean	% Bias	% in Tolerance
BLK-CO01	Au-AA26-ppm	ALS	5 403	0.010	0.0058	-42.0	99.96
BLK-CO01	Au-705P-ppm	Labtium	995	0.020	0.0113	-43.4	99.80
BLK-CO01	Au-FAS121-ppm	CRS/MSA	92	0.005	0.0029	-42.5	100

Notes: % bias = (mean assay - certified value) / certified value *100.

Tolerance here defined as three times detection limit.

Source: Internal Rupert Resources Ltd database, 2023.

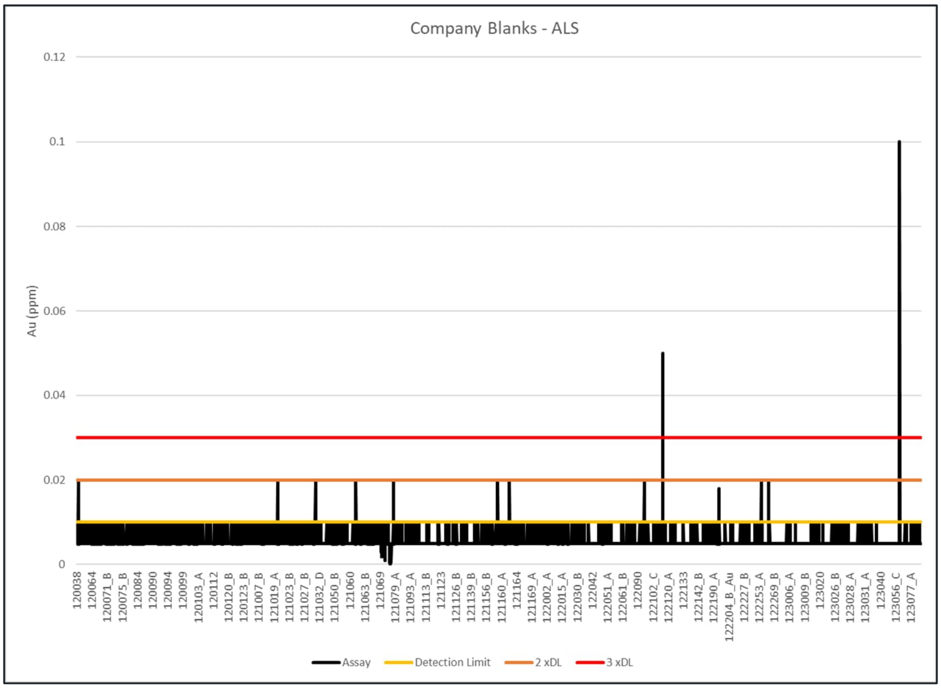


Figure 11-1 – Rupert Resources Blank (BLK-CO01) Performance, ALS

Figure 11-2 – Rupert Resources Blank (BLK-CO01) Performance, Labtium

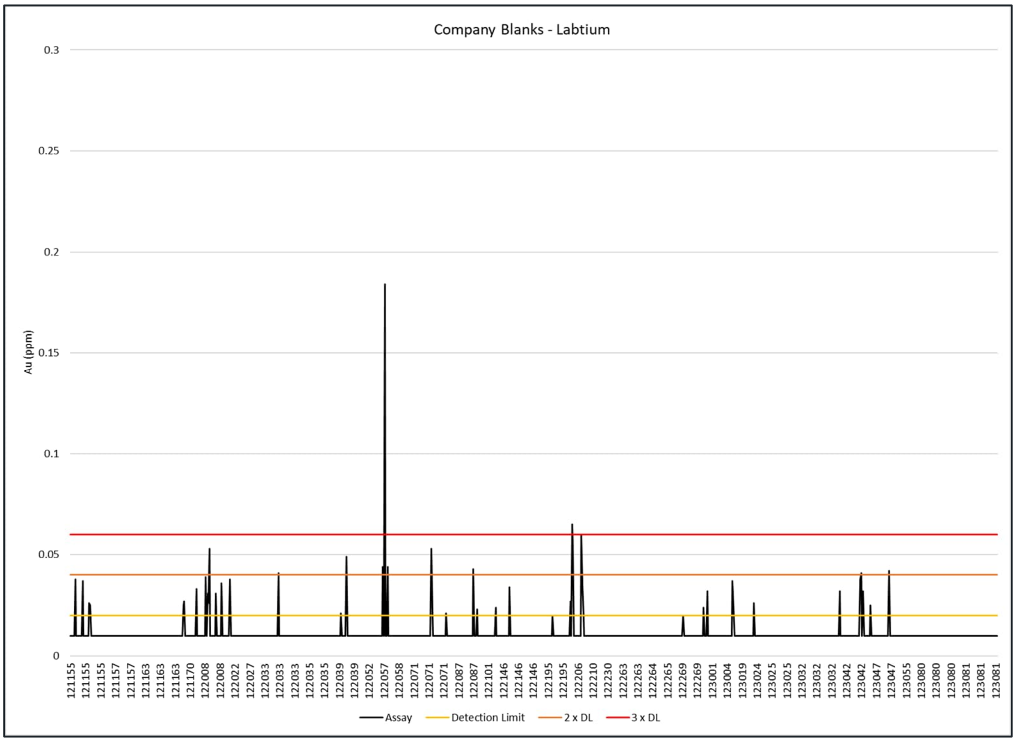


Figure 11-2 – Rupert Resources Blank (BLK-CO01) Performance, Labtium

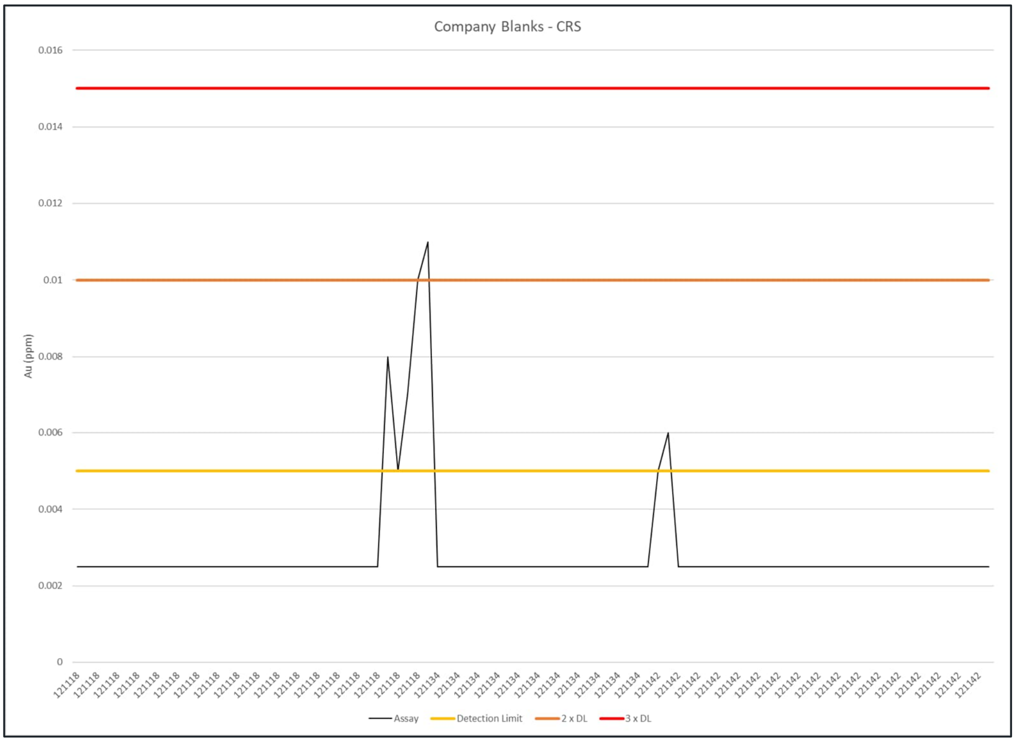


Figure 11-3 – Rupert Resources Blank (BLK-CO01) Performance, CRS

CRM Submitted by Rupert Resources

Rupert Resources routinely submitted accredited CRM at the rate of 1 CRM per 20 samples which is equivalent of 1 in 18 primary samples (5.5%). Rupert Resources has primarily been using gold certified reference materials produced by Geostats Pty Ltd. These CRM's have been selected to represent a range of gold grades covering the vast majority of the grade range experienced at Ikkari. Rupert Resources has also used a minor quantity CRMs prepared by CDN Resource Laboratories Ltd (CDN-GS-3H, CDN-GS-3K, CDN-GS-P7B and CDN-GS-P7H).

Table 11-3 – Ikkari Gold Deposit Standards Submitted to ALS by Rupert Resources

Standard	Assay method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
G312-4	Au-AA26	1	5.3	5.2	-1.89	NA	100
G314-2	Au-AA26	55	0.99	0.98	-1.01	2.53	100
G315-7	Au-AA26	49	0.30	0.29	-2.24	2.58	100
G320-10	Au-AA26	448	0.65	0.64	-1.15	3.51	100
G398-4	Au-AA26	15	0.66	0.65	-2.12	3.07	100
G912-3	Au-AA26	1436	2.09	2.08	-0.51	2.94	100
G915-2	Au-AA26	1486	4.98	5.01	0.62	2.24	100
G915-4	Au-AA26	680	9.16	9.03	-1.43	1.76	100
G915-6	Au-AA26	939	0.67	0.65	-2.61	3.87	100
G917-4	Au-AA26	6	5.10	5.10	0.00	1.49	100
G917-7	Au-AA26	15	4.96	5.01	0.98	2.30	100
GBMS304-3	Au-AA26	32	2.68	2.71	1.19	2.73	100
GBMS304-4	Au-AA26	56	5.67	5.63	-0.65	2.75	100
CDN-GS-3H	Au-AA26	14	3.04	3.02	-0.72	3.74	100
CDN-GS-3K	Au-AA26	11	3.19	3.15	-1.37	2.72	100
CDN-GS-P7B	Au-AA26	1	0.71	0.69	-2.82	NA	100
CDN-GS-P7H	Au-AA26	25	0.80	0.80	0.28	2.27	100

Notes: % bias = (mean assay - certified value) / certified value *100.

% RSD = Standard deviation of assays / certified value *100.

Tolerance here defined as three times standard deviation.

Source: Internal Rupert Resources Ltd. database, 2023.

Table 11-4 – Ikkari Gold Deposit Standards Submitted to Labtium by Rupert Resources

Standard	Assay method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
G320-10	Au-705P	141	0.65	0.66	1.53	2.77	100
G912-3	Au-705P	265	2.09	2.13	1.80	2.15	100
G915-2	Au-705P	217	4.98	5.09	2.30	2.34	100
G915-4	Au-705P	202	9.16	9.26	1.06	2.04	100
G915-6	Au-705P	91	0.67	0.67	0.12	4.91	100
G919-7	Au-705P	23	4.96	5.02	0.71	1.48	100
GBMS304-3	Au-705P	4	2.68	2.72	1.53	3.26	100
GBMS304-4	Au-705P	16	5.67	5.64	-0.45	2.33	100

Notes: % bias = (mean assay - certified value) / certified value *100.
% RSD = Standard deviation of assays / certified value *100.
Tolerance here defined as three times standard deviation.
Source: Internal Rupert Resources Ltd. database, 2023.

Table 11-5 – Ikkari Gold Deposit Standards Submitted to CRS/MSA by Rupert Resources

Standard	Assay method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
G912-3	Au-FAS121	22	2.09	2.04	-2.55	3.75	100
G915-2	Au-FAS121	24	4.98	4.97	-0.18	2.64	100
G915-4	Au-FAS121	24	9.16	8.83	-3.56	2.61	100
G915-6	Au-FAS121	21	0.67	0.63	-6.08	3.45	100

Notes: % bias = (mean assay - certified value) / certified value *100.
% RSD = Standard deviation of assays / certified value *100.
Tolerance here defined as three times standard deviation.
Source: Internal Rupert Resources Ltd. database, 2023.

Control graphs for the most commonly utilised standards at ALS (8), Labtium (6) and CRS (4) are presented below:

- ALS: Figure 11-4 through Figure 11-11;
- Labtium: Figure 11-12 through Figure 11-17; and
- CRS: Figure 11-18 through Figure 11-21.

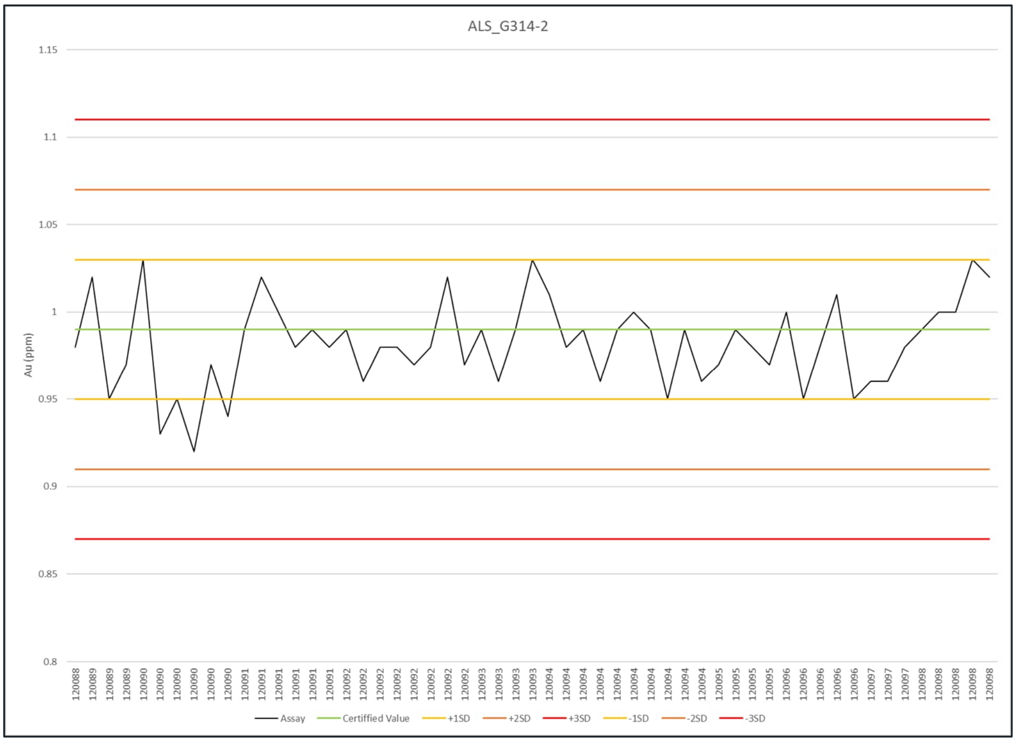


Figure 11-4 – Rupert Resources CRM's Performance in ALS, G314-3

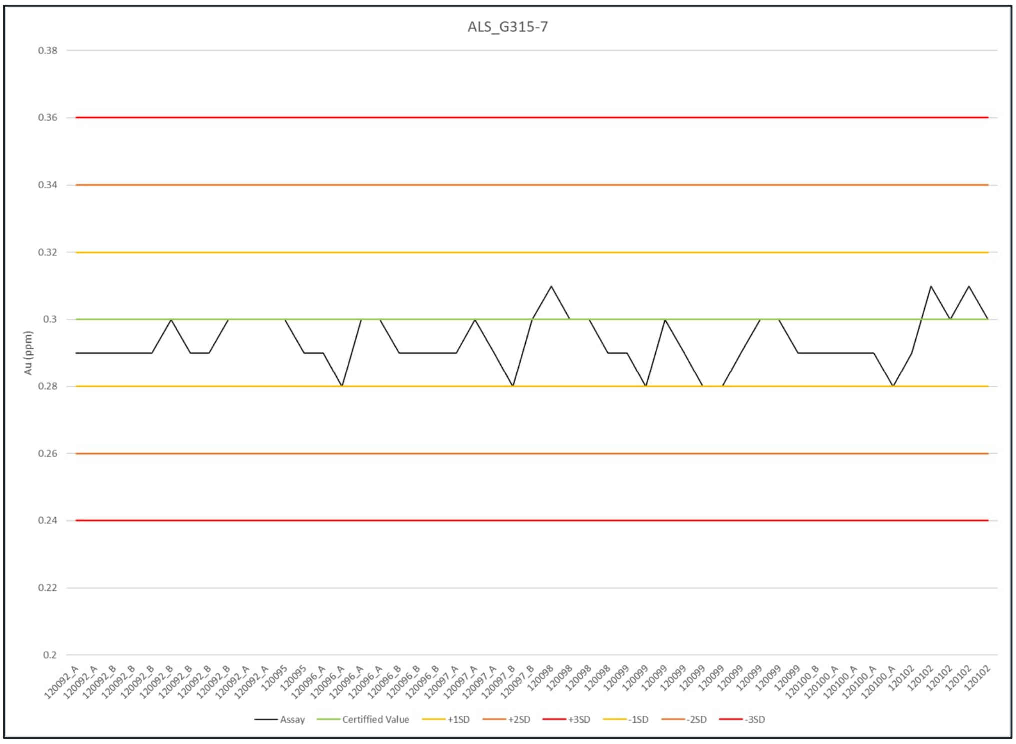


Figure 11-5 – Rupert Resources CRM's Performance in ALS, G315-7

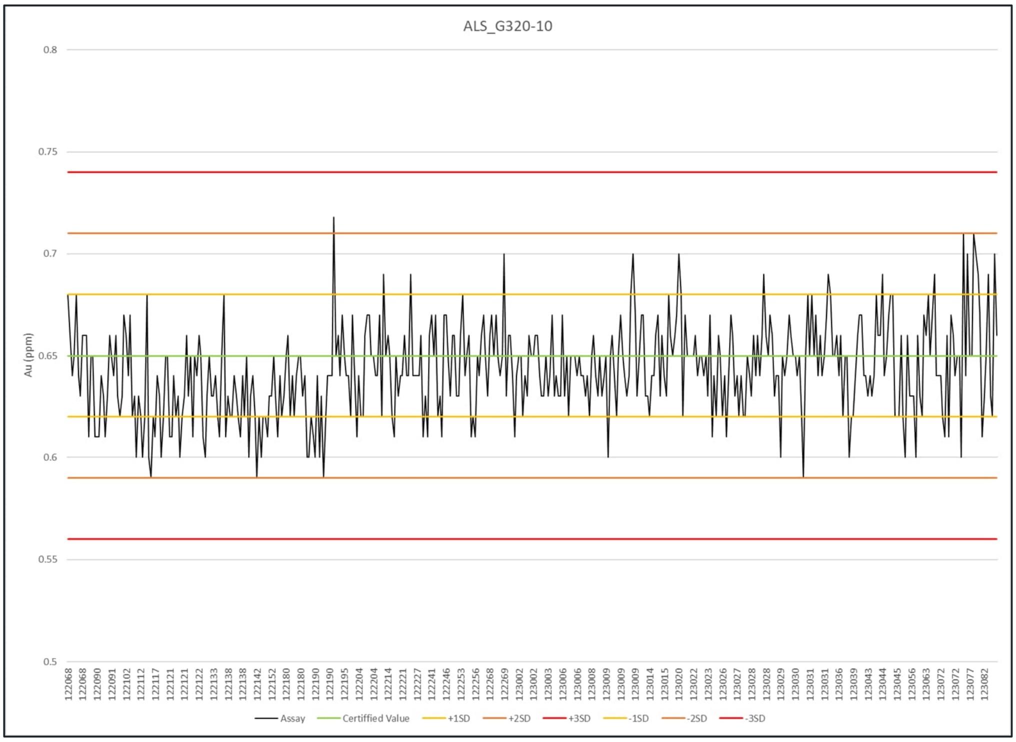


Figure 11-6 – Rupert Resources CRM's Performance in ALS, G320-10

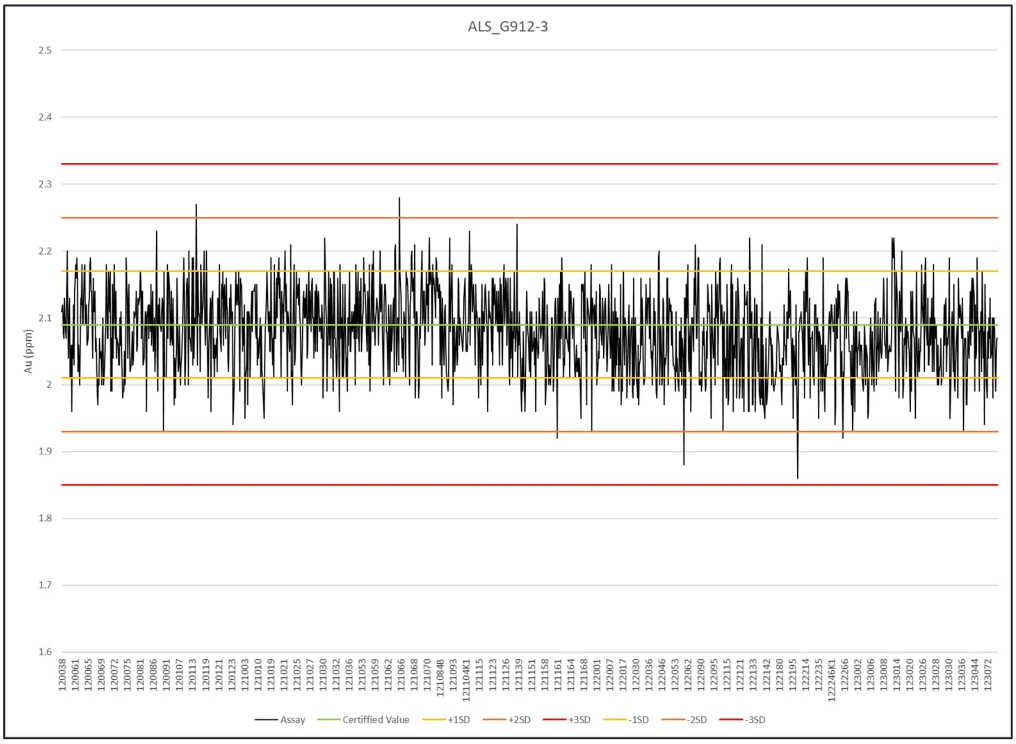


Figure 11-7 – Rupert Resources CRM's Performance in ALS, G912-3

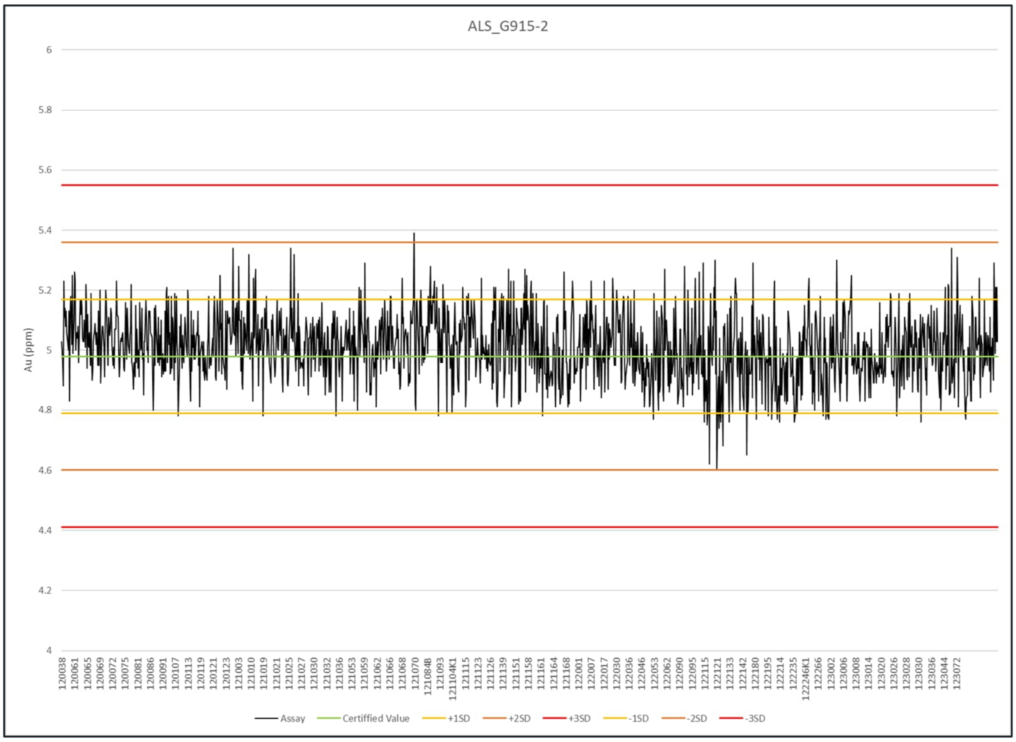


Figure 11-8 – Rupert Resources CRM's Performance in ALS, G915-2

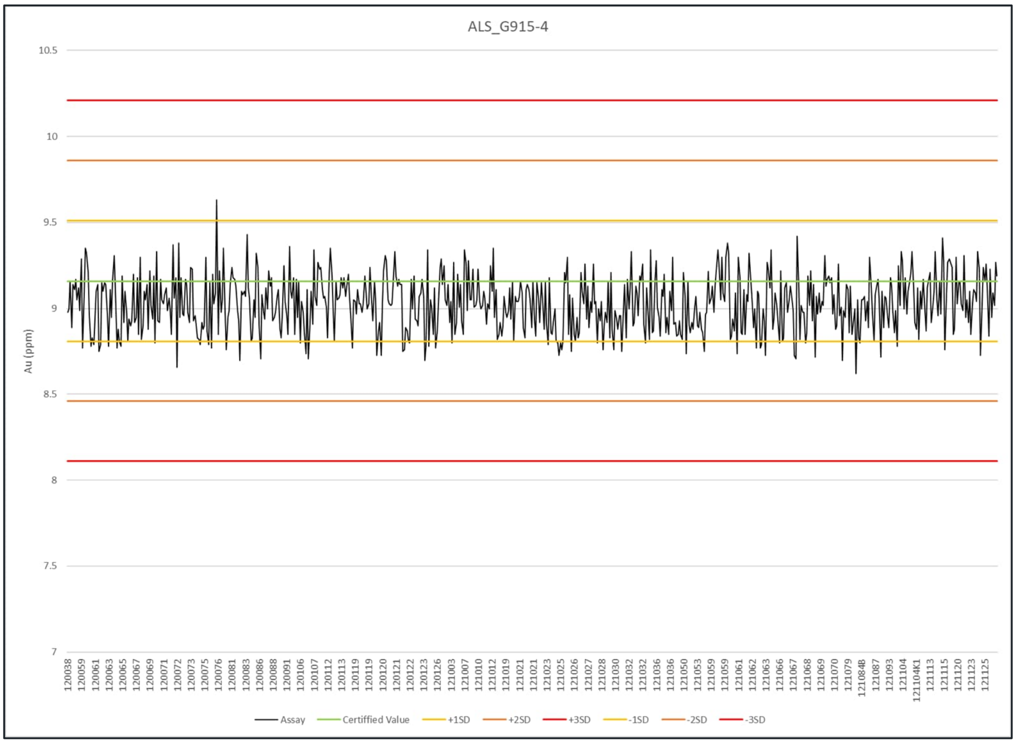


Figure 11-9 – Rupert Resources CRM's Performance in ALS, G915-4

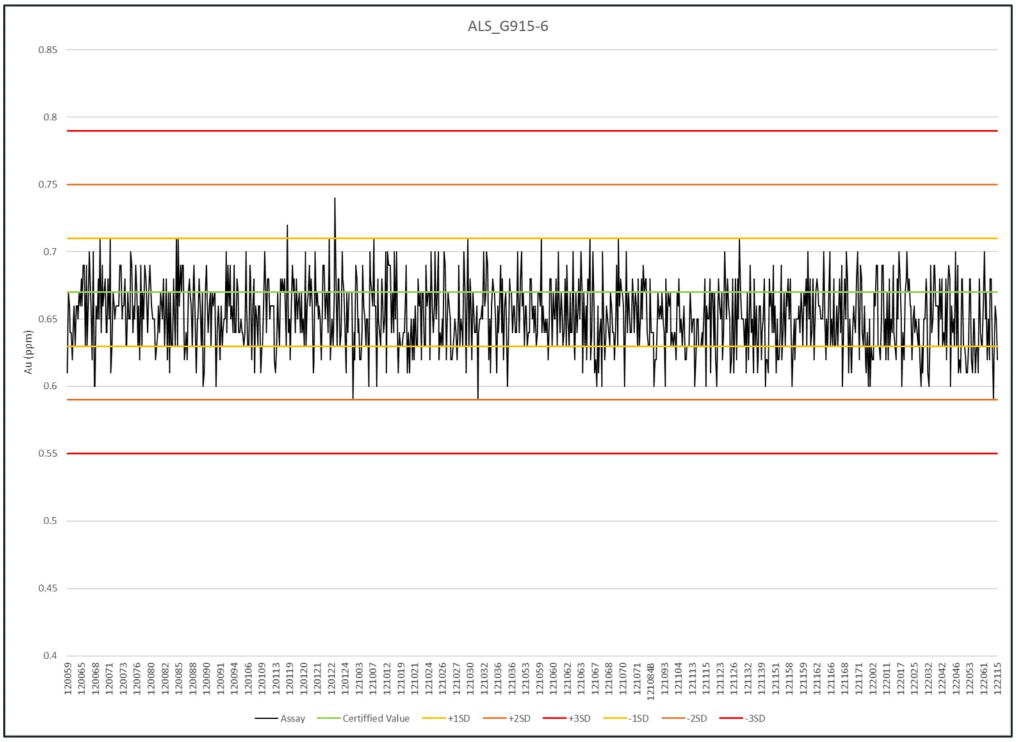


Figure 11-10 – Rupert Resources CRM's Performance in ALS, G915-6

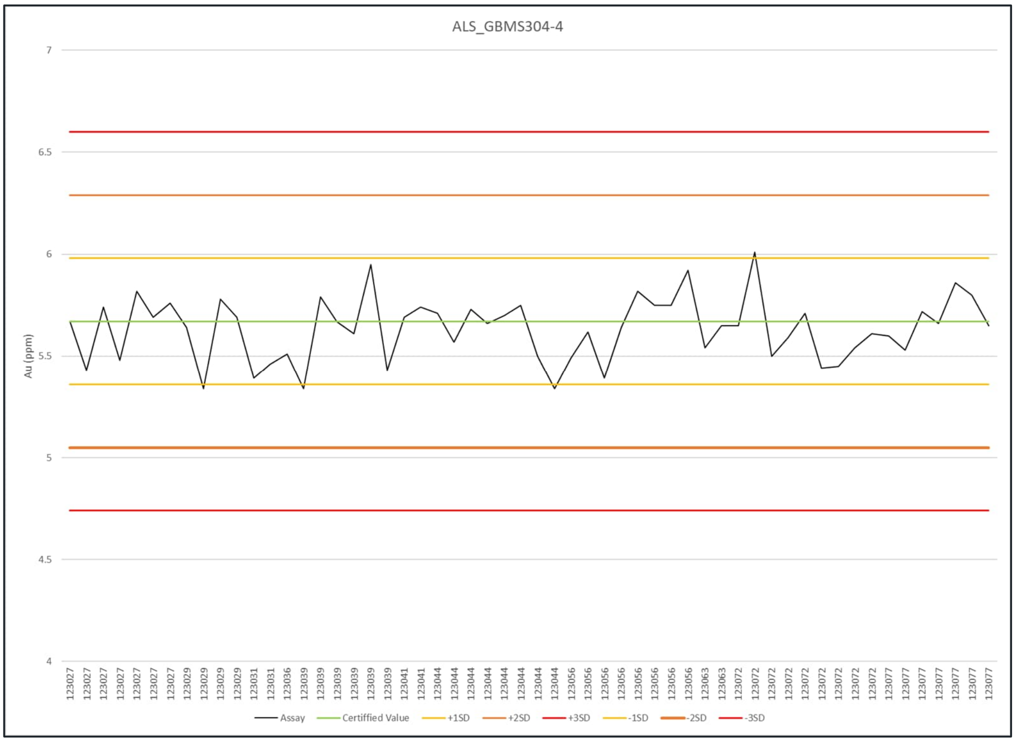


Figure 11-11 – Rupert Resources CRM's Performance in ALS, GBMS304-4

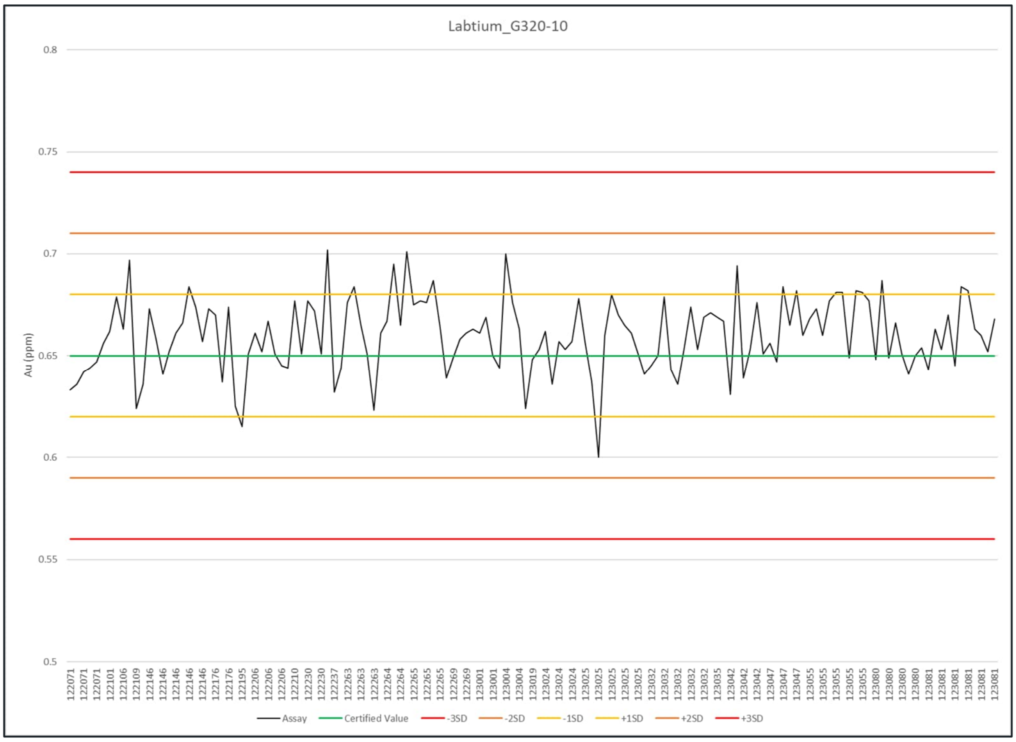


Figure 11-12 – Rupert Resources CRM's Performance in Labtium/Eurofins, G320-10

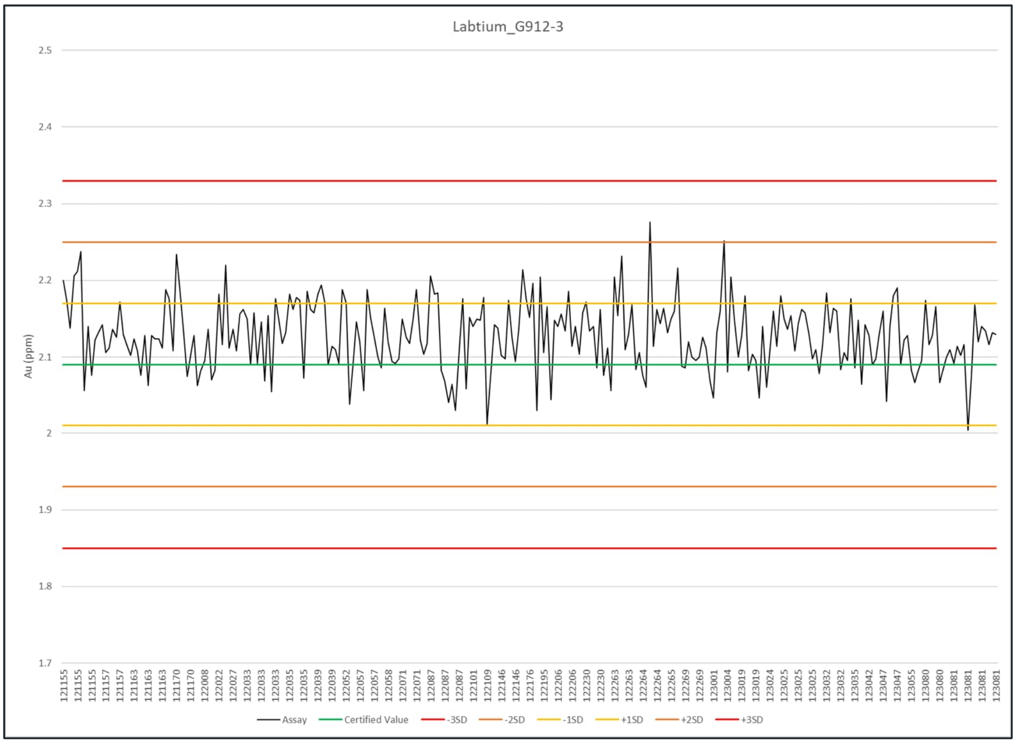


Figure 11-13 – Rupert Resources CRM's Performance in Labtium/Eurofins, G912-3

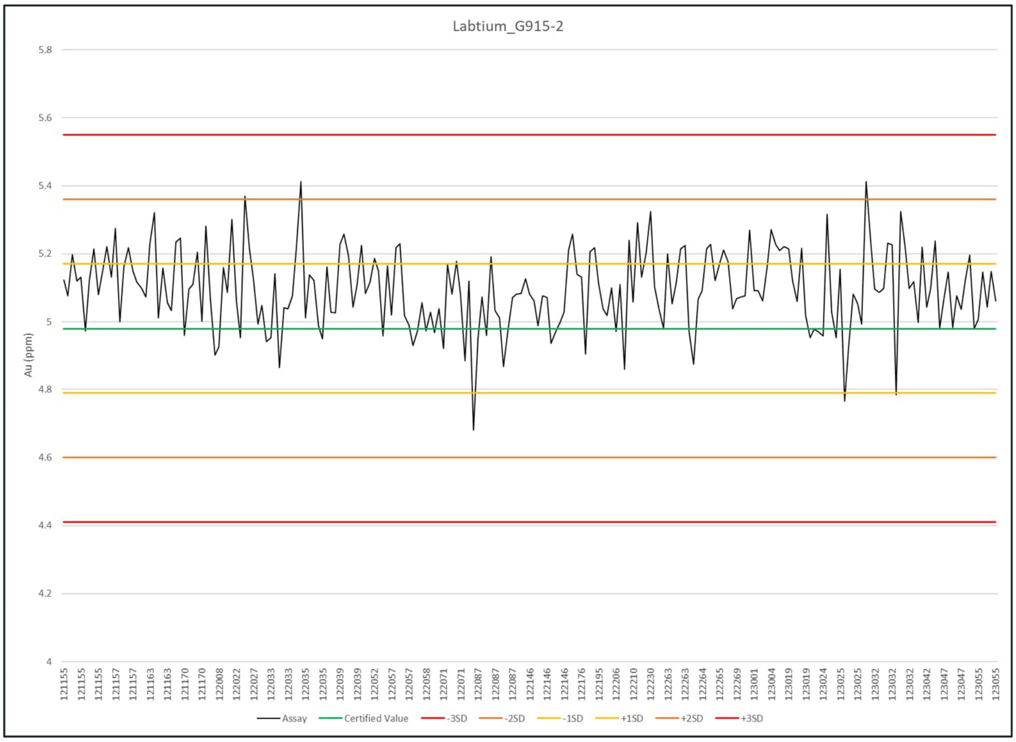


Figure 11-14 – Rupert Resources CRM's Performance in Labtium/Eurofins, G915-2

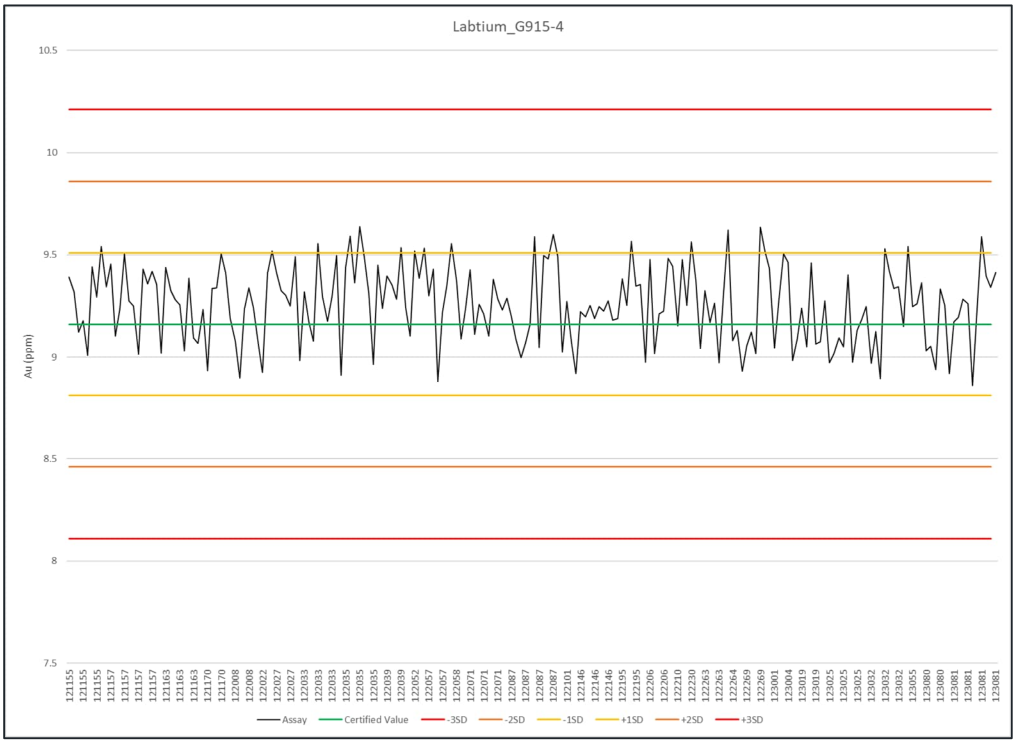


Figure 11-15 – Rupert Resources CRM's Performance in Labtium/Eurofins, G915-4

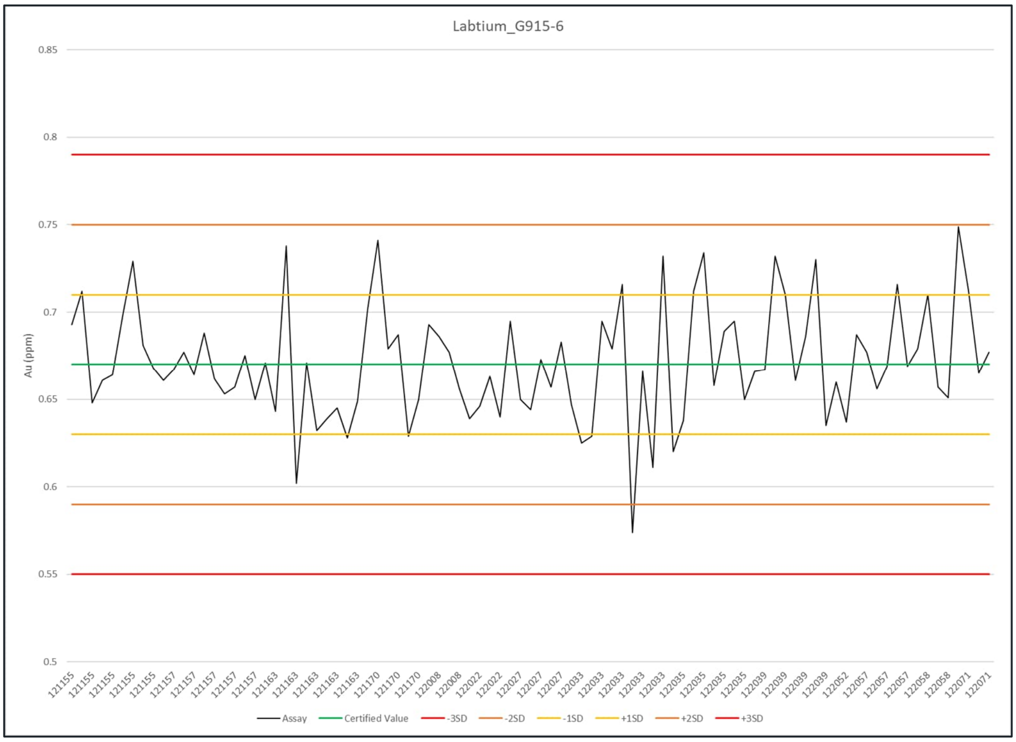


Figure 11-16 – Rupert Resources CRM's Performance in Labtium/Eurofins, G915-6

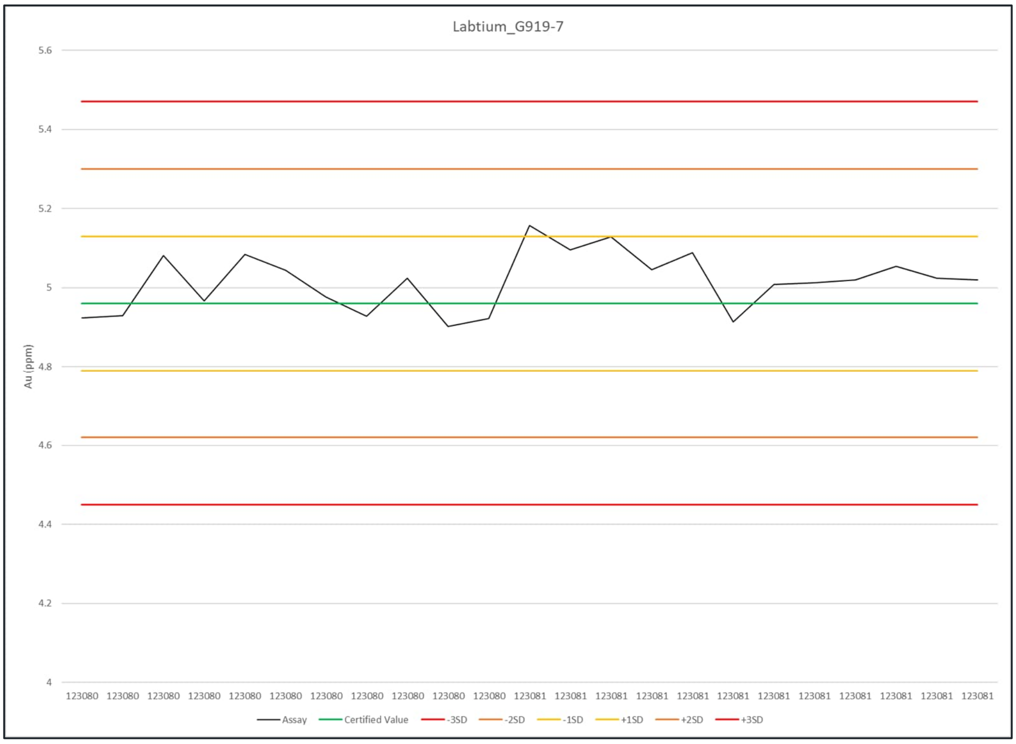


Figure 11-17 – Rupert Resources CRM's Performance in Labtium/Eurofins, G919-7

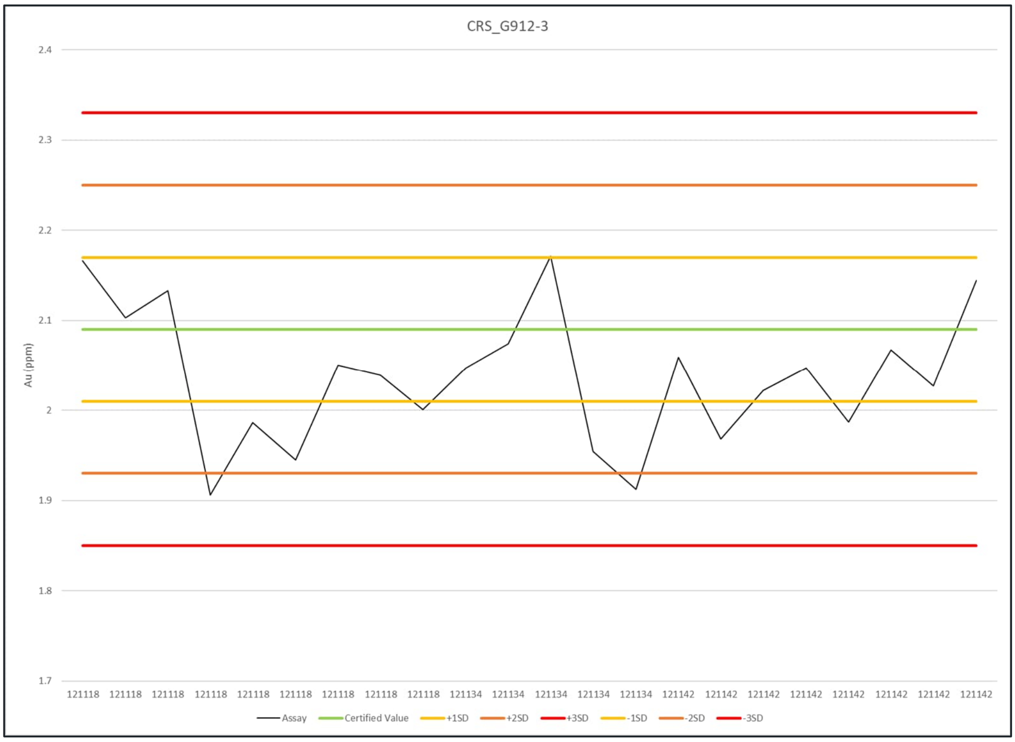


Figure 11-18 – Rupert Resources CRM's Performance in CRS/MSA, G912-3

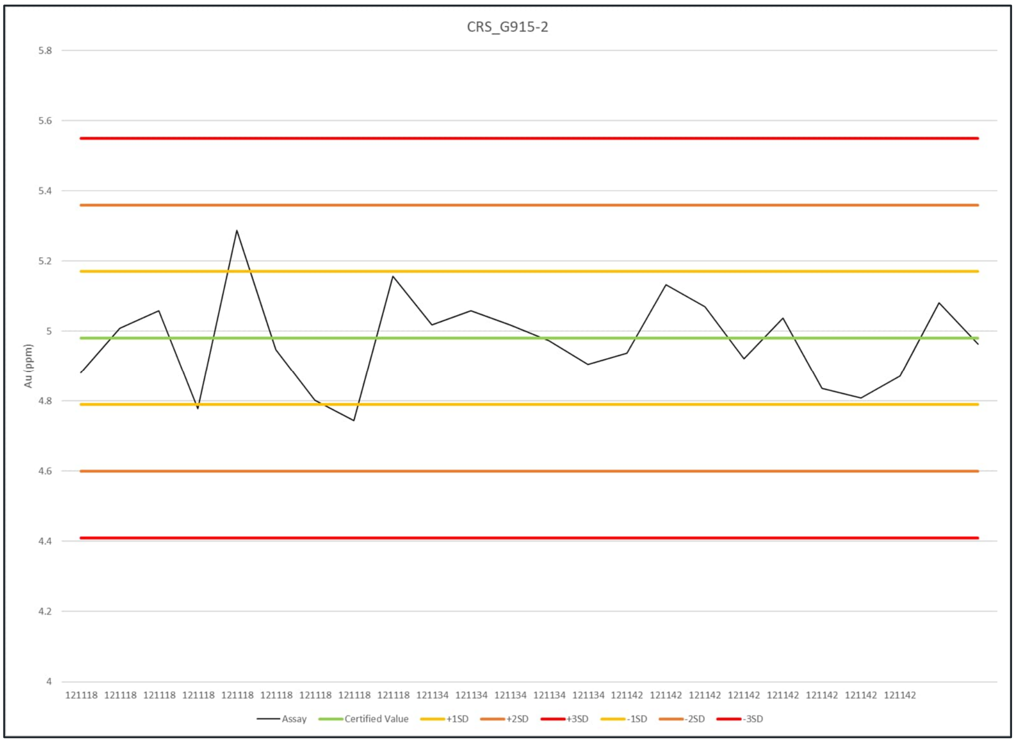


Figure 11-19 – Rupert Resources CRM's Performance in CRS/MSA, G915-2

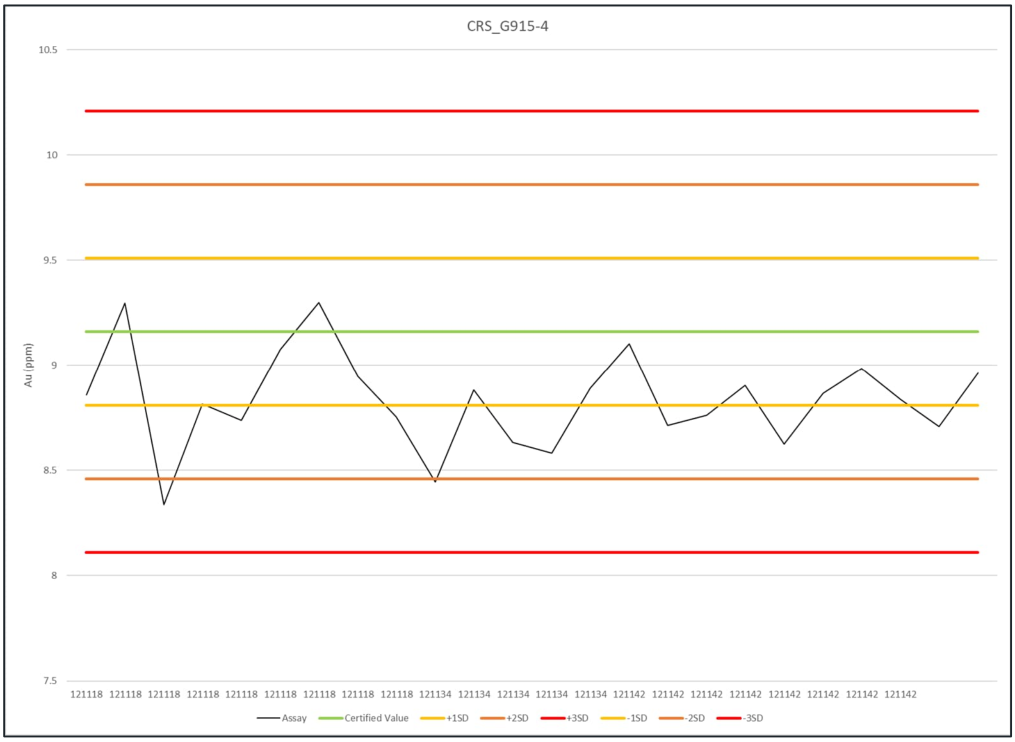


Figure 11-20 – Rupert Resources CRM's Performance in CRS/MSA, G915-4

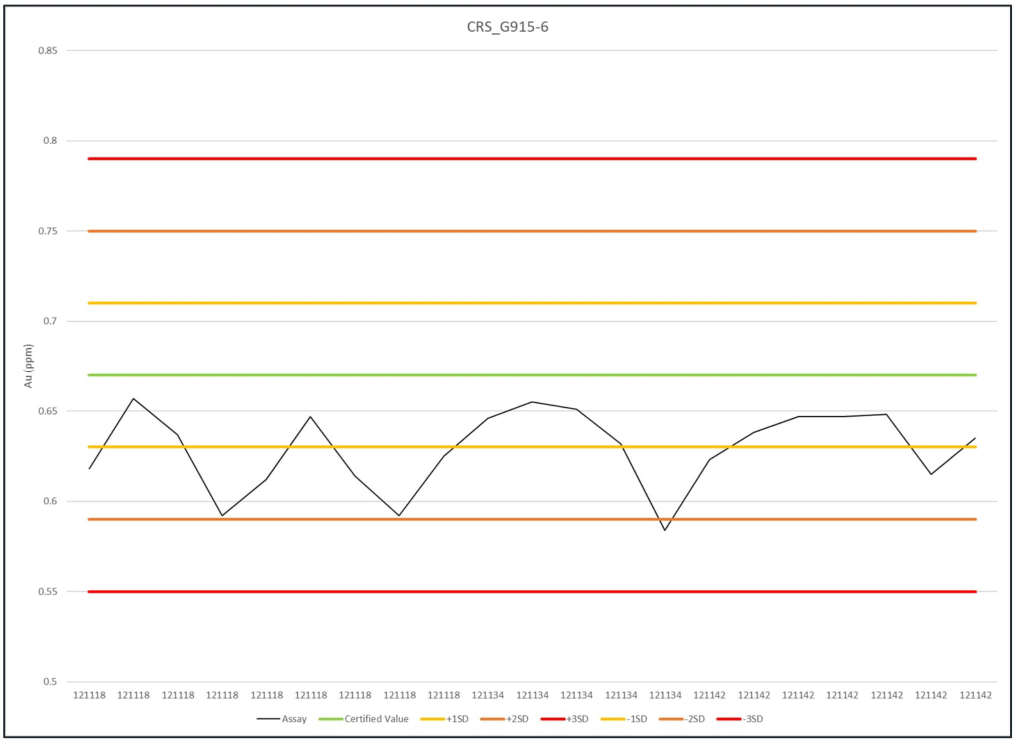


Figure 11-21 – Rupert Resources CRM's Performance in CRS/MSA, G915-6

Comparison of Common CRM

All CRM's Rupert Resources have been using since July 2018 perform very well with used fire assay methods in all laboratories, the main laboratory ALS minerals as well as in MSA labs and Eurofins Labtium. Rupert Resources' policy of re-assaying CRMs (and the surrounding primary samples) when assays occur outside of three standard deviations results in a very narrow spread of results for all CRMs.

For all CRMs, ALS demonstrates a very slight negative bias when compared with the certified values whereas Eurofins Labtium demonstrates a very slight positive bias relative to the certified value; CRS, with its limited dataset demonstrates a larger negative bias.

Considering the variability, the same standards (G915-6) demonstrate the highest variability at both ALS and Eurofins Labtium indicative of slightly less homogenized reference material. G915-4 and G919-7 produce the lowest Relative Standard Deviation (RSD) at both ALS and Eurofins Labtium indicative of very homogenous reference material. This together with the similarities between the laboratories in RSD suggest both laboratories have similar levels of precision, something that is further addressed when discussing the data pairs.

Data Pairs

Rupert Resources' QA/QC routine with the fire assay method includes submitting core duplicates, pulp duplicates, and umpire checks, each 5% of the samples.

Available data pairs have been reviewed, subdivided by the assay laboratory. The different types of data pairs comprise the following:

- Field duplicates (quarter core pairs);
- Lab duplicates (two samples taken after pulverizing sample material >85% <75 µm); and
- Pulp duplicates (duplicates samples taken from within one pulp sachet).
  - *Pulp duplicates not performed at CRS Laboratory.
  - Umpire checks (Pulp split sent to second laboratory) are detailed in section Umpire Checks.

Table 11-6 – Ikkari Gold Deposit Data Pairs

Duplicate Type	Laboratory	Total Number of Pairs	Au Original Mean (g/t)	Au Check Mean (g/t)	Corr. Coeff.
Field duplicate	ALS_All	5 308	0.35	0.33	0.73
Pulp duplicate	ALS_SO	5 149	0.48	0.48	0.99
Pulp duplicate	ALS_OT	103	0.17	0.19	0.99
Lab duplicate	ALS_All	5 039	1.12	1.14	0.99
Field duplicate	Labtium	979	0.40	0.44	0.70
Pulp duplicate	Labtium	973	0.34	0.34	0.99
Lab duplicate	Labtium	438	1.27	1.26	0.99

Duplicate Type	Laboratory	Total Number of Pairs	Au Original Mean (g/t)	Au Check Mean (g/t)	Corr. Coeff.
Field duplicate	CRS/MSA	94	0.06	0.09	0.61
Lab duplicate	CRS/MSA	47	0.61	0.65	0.99

ALS Sodankylä and ALS Outokumpu are preparation laboratories only with assaying from both completed at ALS Romania. Laboratory duplicates and core duplicates have therefore been grouped for both preparation laboratories.

The paired assay data has been assessed using the following techniques and plots:

- MPRD by Mean Grade;
- Correlation Plot; and
- Quantile-Quantile Plot.

The contents of the following figures are set out below:

- Figure 11-22 Sample Pair Statistical Analysis: Samples Submitted to ALS: Field Duplicates;
- Figure 11-23 Sample Pair Statistical Analysis: Samples Submitted to ALS Sodankylä: Pulp Duplicates;
- Figure 11-24 Sample Pair Statistical Analysis: Samples Submitted to ALS Outokumpu: Pulp Duplicates;
- Figure 11-25 Sample Pair Statistical Analysis: Samples Submitted to ALS : Laboratory Duplicates;
- Figure 11-26 Sample Pair Statistical Analysis: Samples Submitted to Labtium: Field Duplicates;
- Figure 11-27 Sample Pair Statistical Analysis: Samples Submitted to Labtium: Pulp Duplicates;
- Figure 11-28 Sample Pair Statistical Analysis: Samples Submitted to Labtium: Laboratory Duplicates;
- Figure 11-29 Sample Pair Statistical Analysis: Samples Submitted to MSA/CRS: Field Duplicates; and
- Figure 11-30 Sample Pair Statistical Analysis: Samples Submitted to CRS/MSA: Laboratory Duplicates.
  - No Pulp Duplicates were performed on samples submitted to CRS/MSA.
  - *ALS Sodankyla and ALS Outokumpu.

Figure 11-22 – Sample Pair Statistical Analysis: Samples Submitted to ALS: Field Duplicates

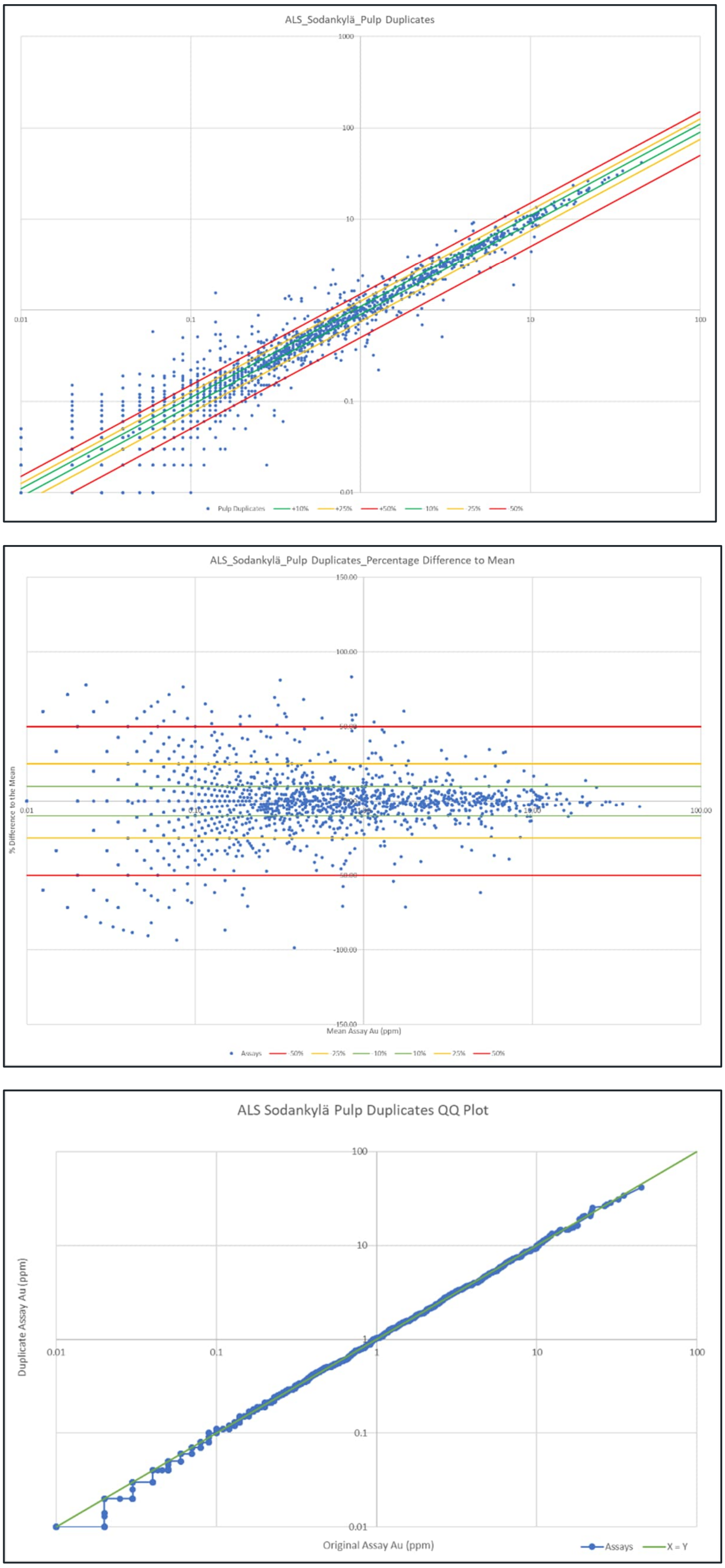
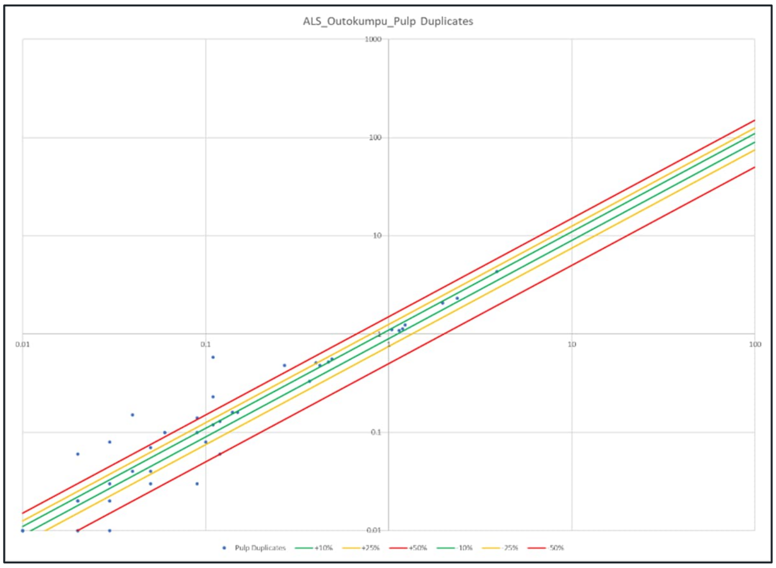
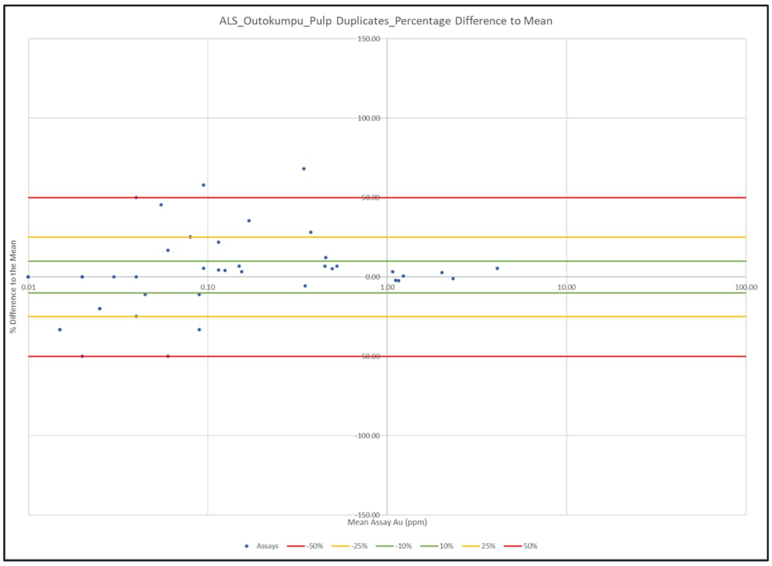


Figure 11-23 – Sample Pair Statistical Analysis: Samples Submitted to ALS Sodankylä: Pulp Duplicates





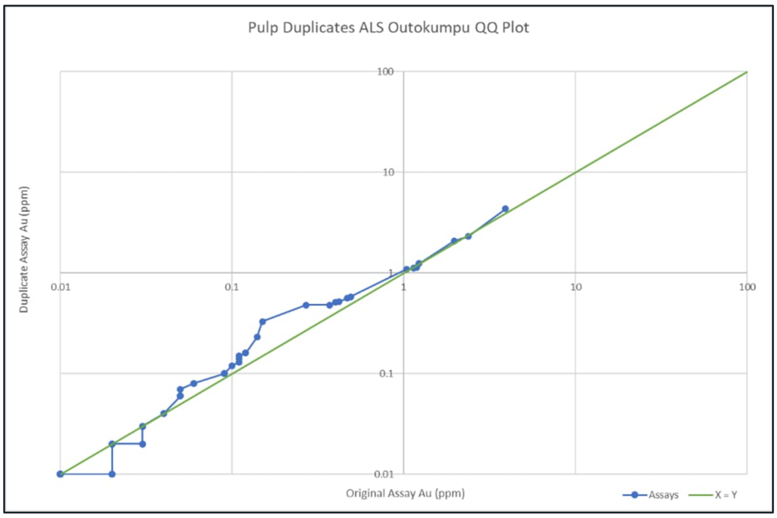


Figure 11-24 – Sample Pair Statistical Analysis: Samples Submitted to ALS Outokumpu: Pulp Duplicates

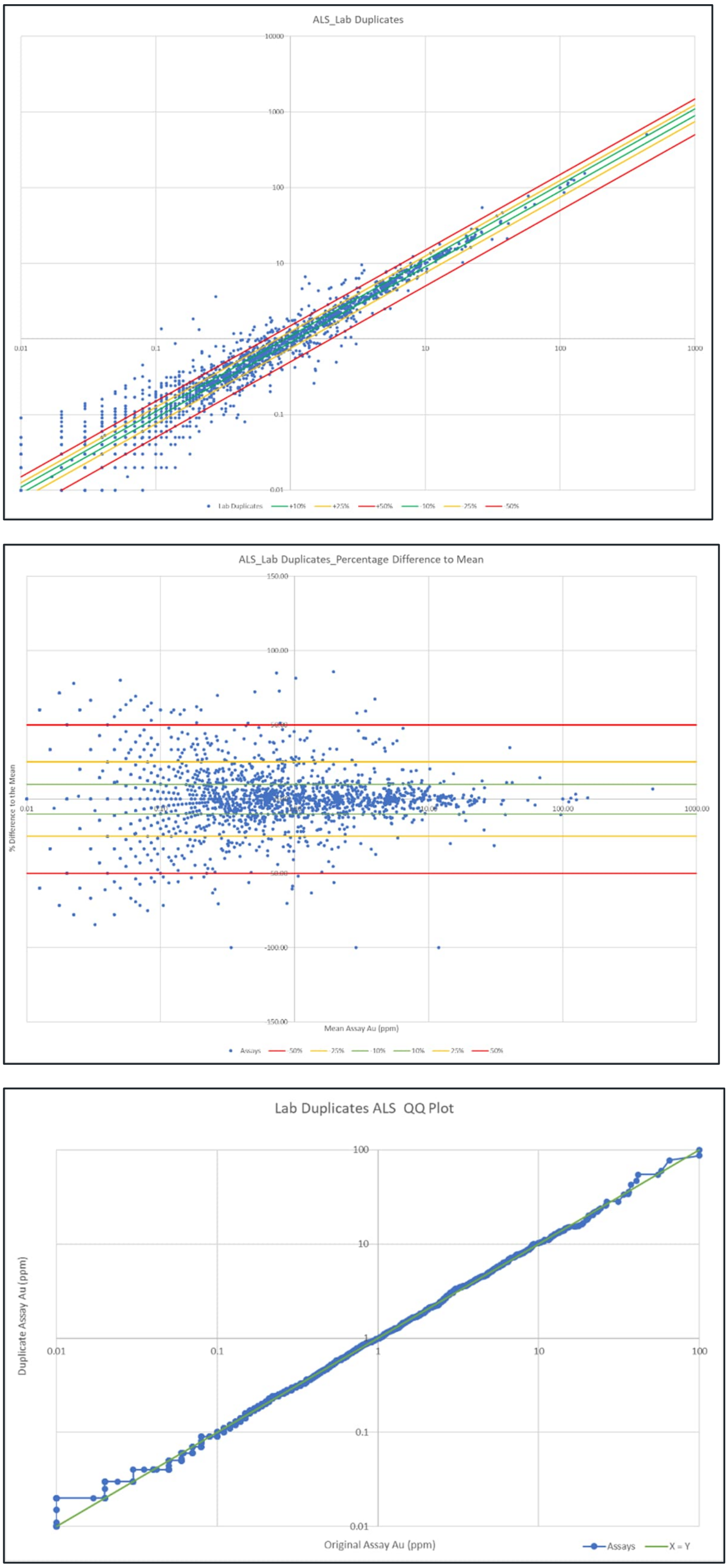


Figure 11-25 – Sample Pair Statistical Analysis: Samples Submitted to ALS: Laboratory Duplicates

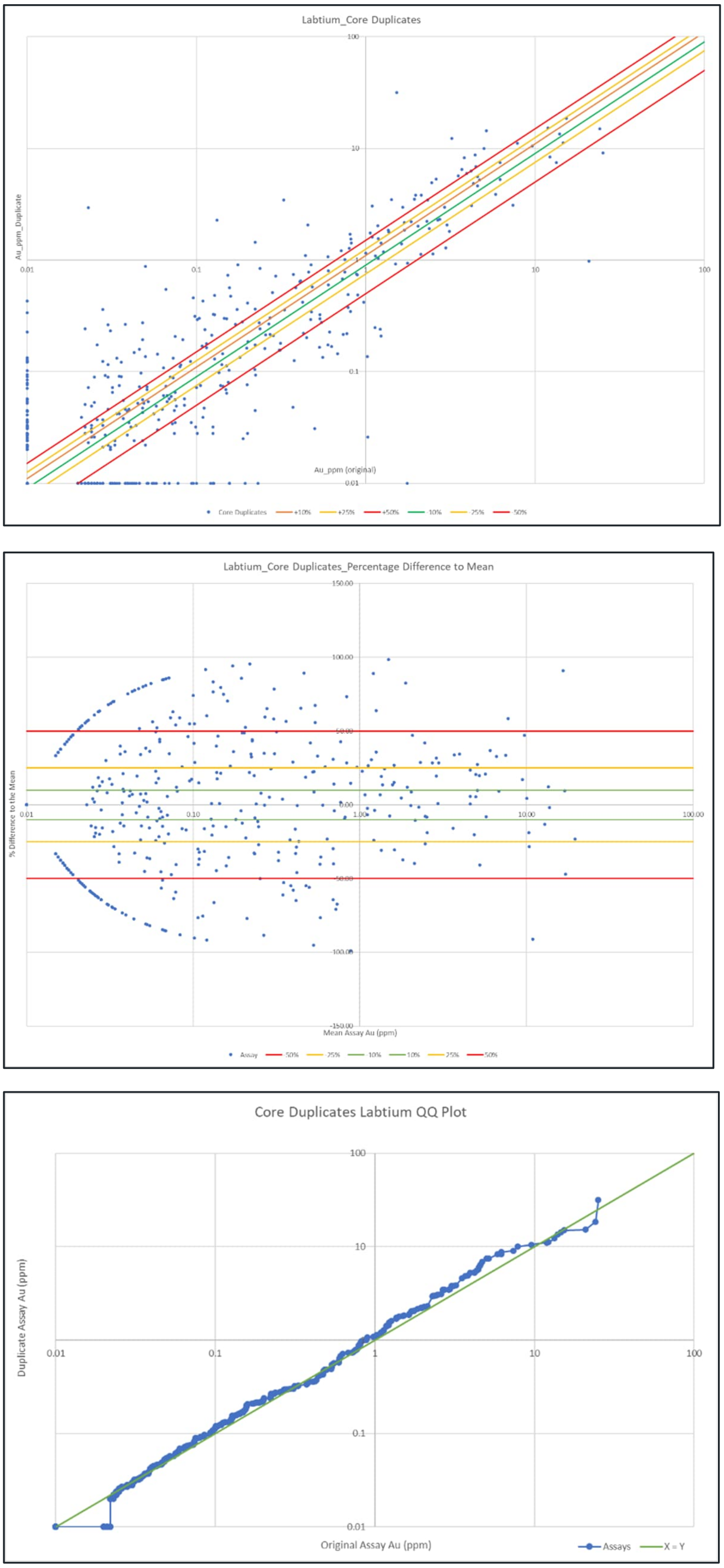


Figure 11-26 – Sample Pair Statistical Analysis: Samples Submitted to Labtium: Field Duplicates

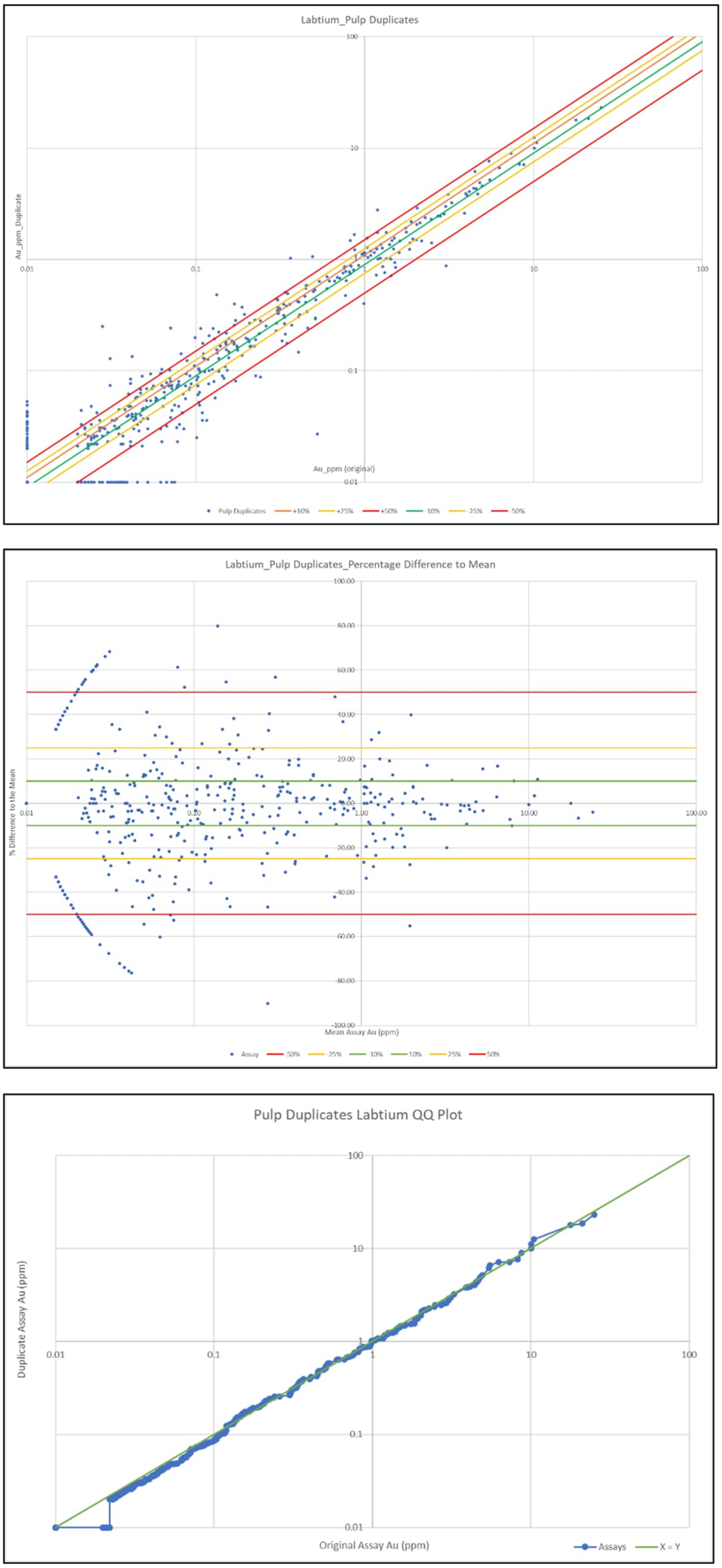


Figure 11-27 – Sample Pair Statistical Analysis: Samples Submitted to Labtium: Pulp Duplicates

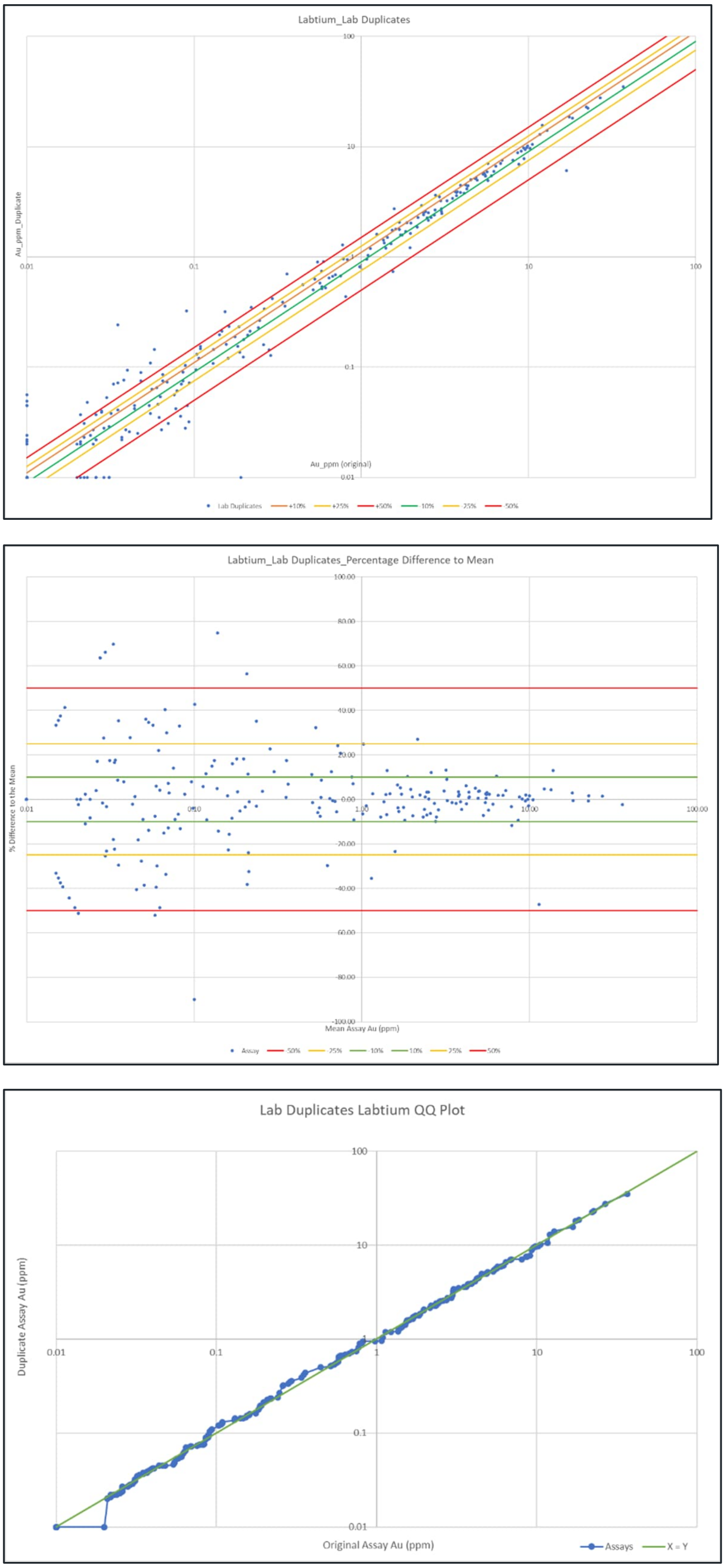


Figure 11-28 – Sample Pair Statistical Analysis: Samples Submitted to Labtium: Laboratory Duplicates

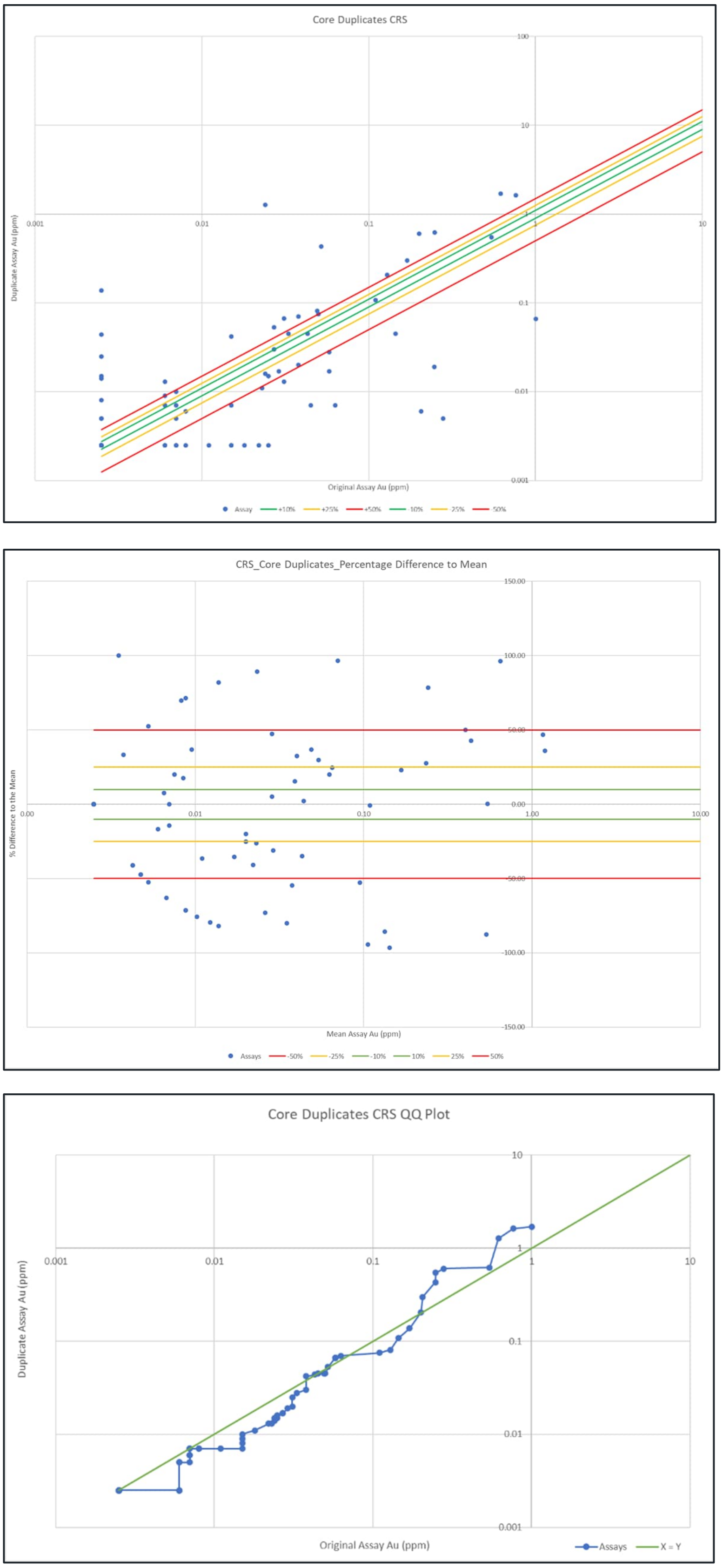
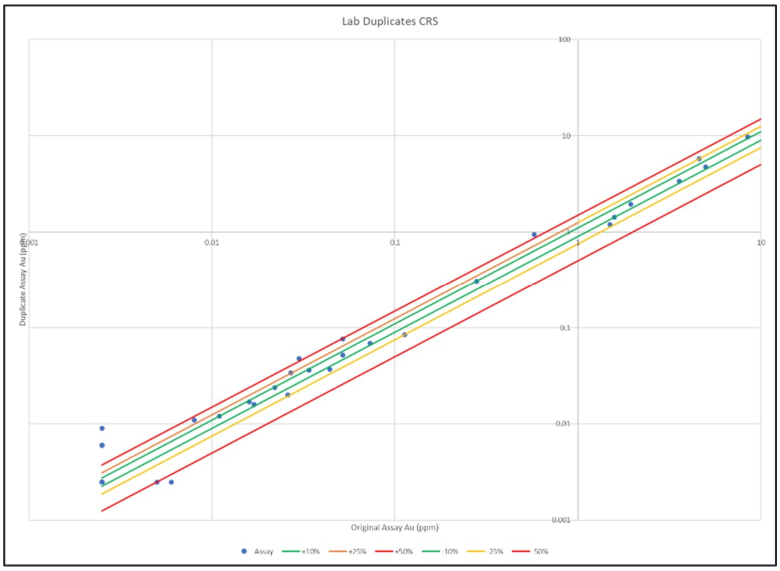
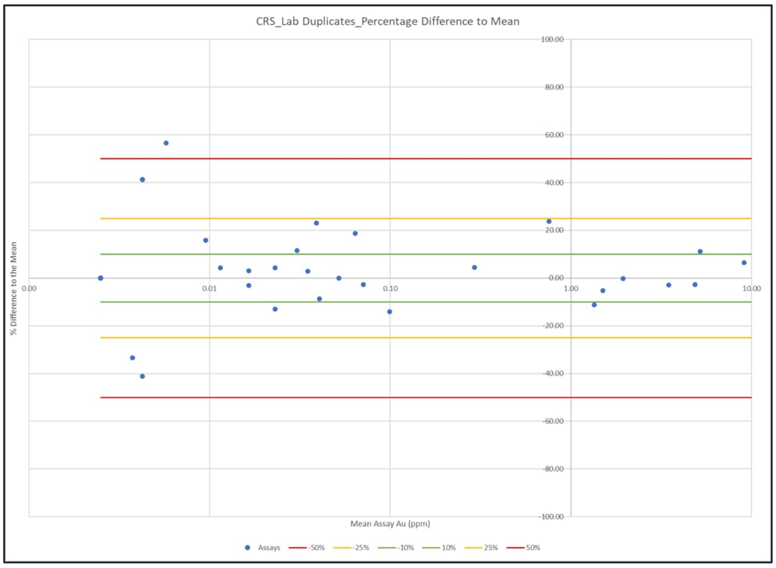


Figure 11-29 – Sample Pair Statistical Analysis: Samples Submitted to MSA/CRS: Field Duplicates





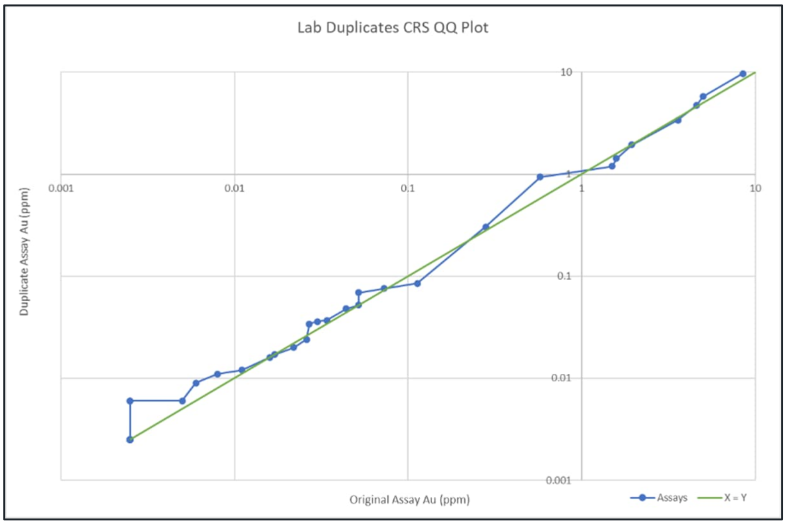


Figure 11-30 – Sample Pair Statistical Analysis: Samples Submitted to CRS/MSA: Laboratory Duplicates

Umpire Checks

ALS Minerals laboratory has been instructed to make a 250 g extra split at pulverizing stage, to be sent to second laboratory for umpire check. Five percent of all samples have been sent to a different external Laboratory for check (umpire) assay with weighting applied such that samples greater than 0.1 ppm Au, of economic interest, are oversampled.

The majority of ALS umpire checks are sent to Eurofins Labtium with a minority sent to CRS; all Labtium umpire checks are sent to ALS.

Umpire checks statistics are displayed in Table 11-7 with samples originally assayed at ALS shown in

Figure 11-31 and those originally assayed at Labtium shown in Figure 11-32.

Table 11-7 – Ikkari Gold Deposit Umpire Checks Data Pairs

Original laboratory	Umpire Laboratory	Total Number of Pairs	Au Original Mean (g/t)	Au Check Mean (g/t)	Corr. Coeff.
ALS	Labtium	3 359	3.36	3.36	0.92
ALS	MSA	421	3.99	3.94	0.99
Labtium	ALS	1 120	1.29	1.26	0.91

The paired assay data has been assessed using the following techniques and plots:

- MPRD by Mean Grade;
- Correlation Plot; and
- Quantile-Quantile Plot.

The contents of the following figures are set out below:

- Figure 11-31: Sample Pair Statistical Analysis: Samples Submitted to ALS Originally: External/Umpire Duplicates to CRS and Eurofins Labtium;
- Figure 11-32: Sample Pair Statistical Analysis: Samples Submitted to Eurofins Labtium Originally: External/Umpire Duplicates to ALS; and
- Figure 11-33: Comparison of Absolute Mean Percentage Difference Between the Umpire/External Check Assays and the Pulp Duplicates – the Equivalent Stage Internal Assay check.

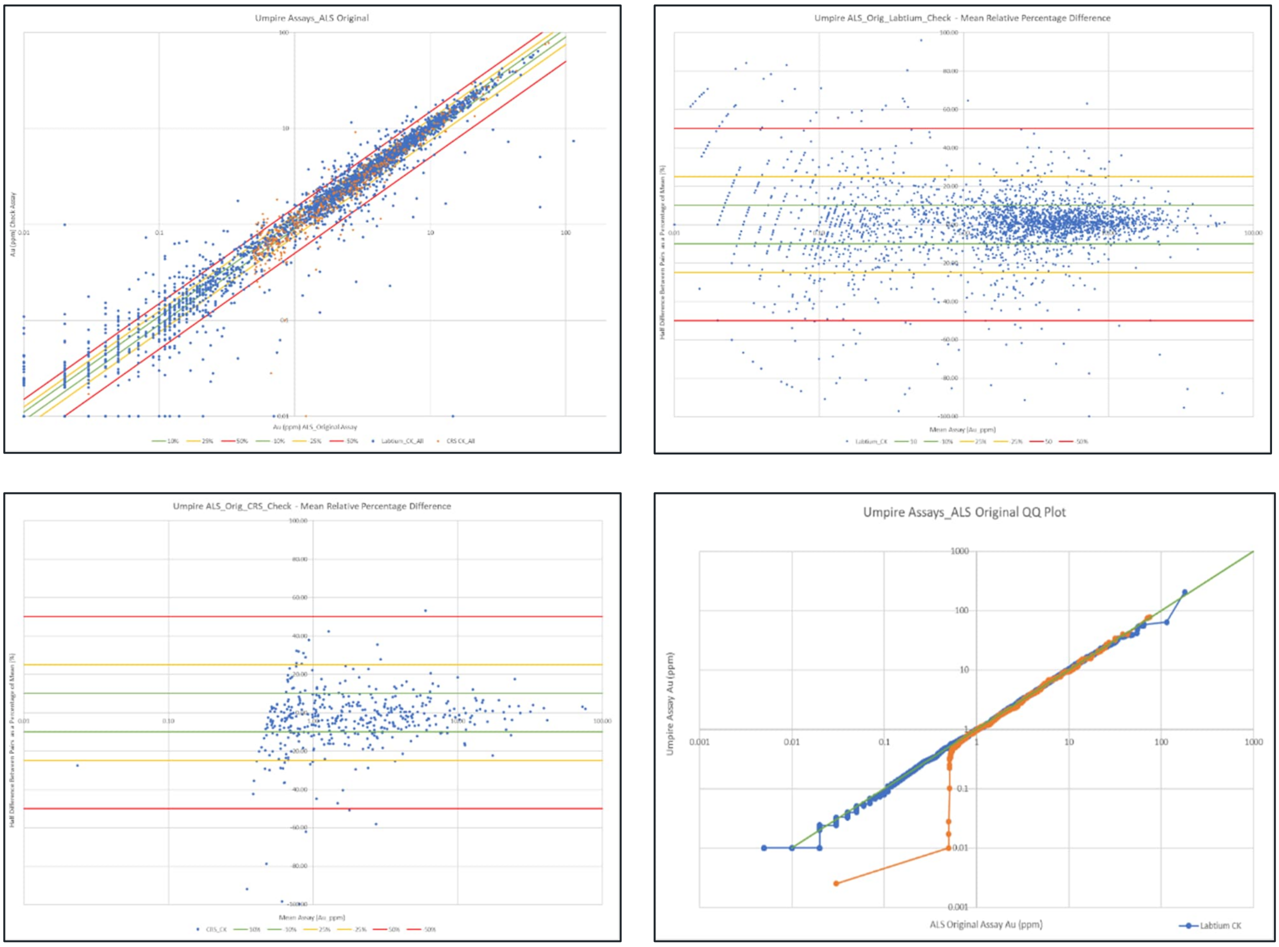
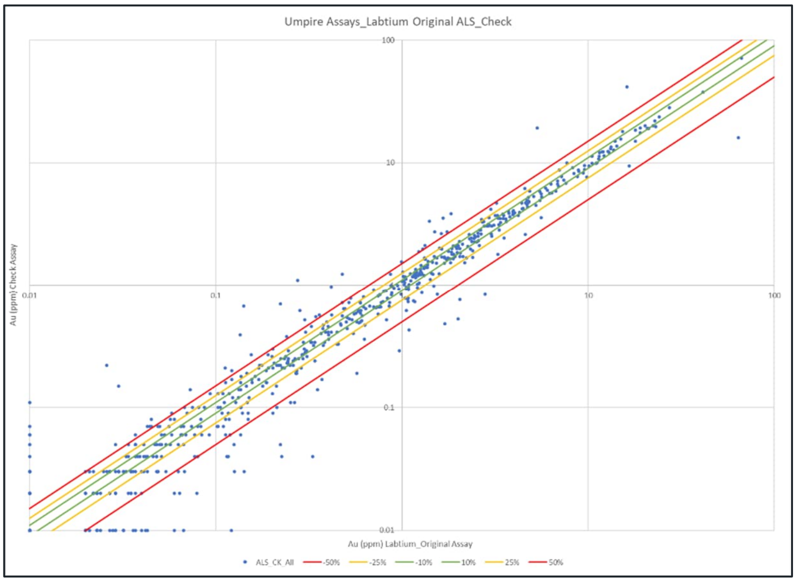
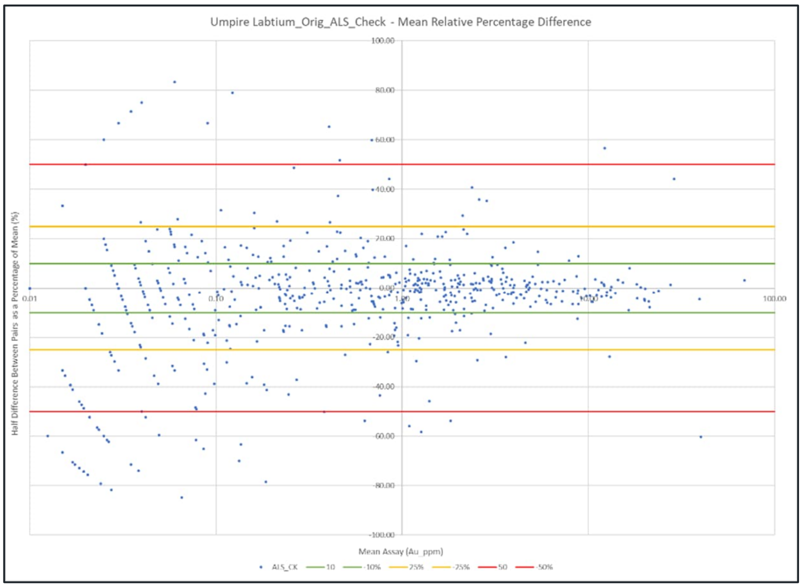


Figure 11-31 – Sample Pair Statistical Analysis: Samples Submitted to ALS Originally: External/Umpire Duplicates to CRS and Eurofins Labtium





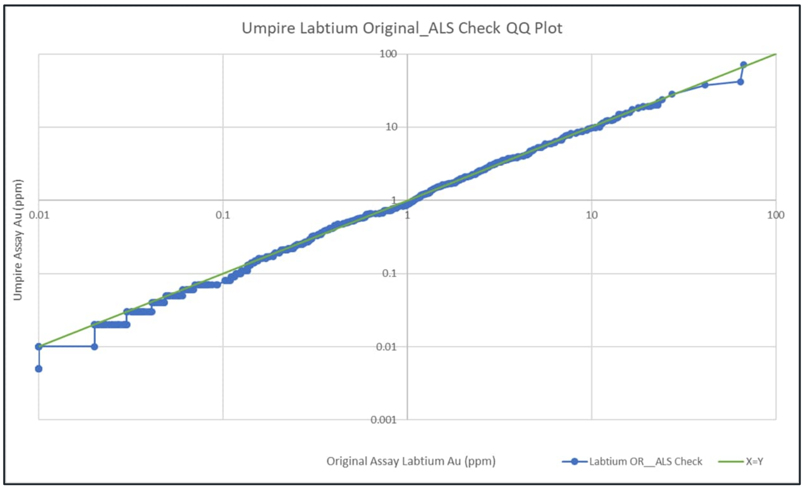


Figure 11-32 – Sample Pair Statistical Analysis: Samples Submitted to Eurofins Labtium Originally: External/Umpire Duplicates to ALS

Since umpire check assays are performed on a separate pulp spilt than the original analysis, they are equivalent to the pulp duplicates analysed as part of the normal QC process. Comparison of the absolute mean percentage difference for both the umpire check assays and pulp duplicates shows that differences between duplicates pairs for both are minor with umpire assays showing slightly reduced differences versus the pulp duplicates (Figure 11-33).

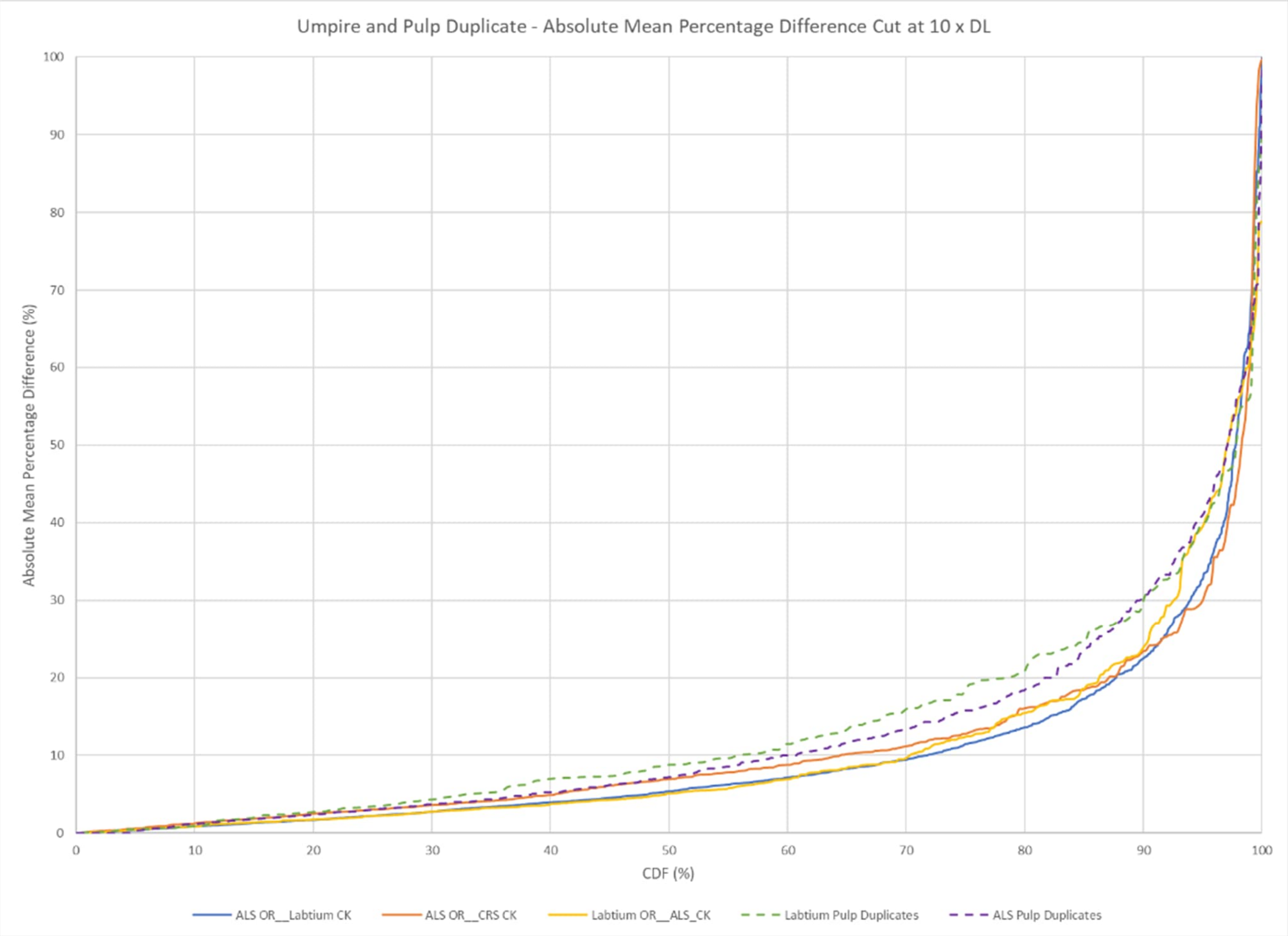


Figure 11-33 – Comparison of Absolute Mean Percentage Difference Between the Umpire/External Check Assays and the Pulp Duplicates – the Equivalent Stage Internal Assay check

11.2 CONCLUSIONS

All sample preparation was carried out at independent laboratories in Finland, and analyses were carried out at independent laboratories in Romania, Ireland, or Finland. No aspect of laboratory sample preparation or analysis was conducted by an employee, officer, director or associate of either Rupert Resources.

Rupert Resources has used a combination of duplicates, checks, blanks and standards to ensure suitable quality control of sampling methods and assay testing. The procedures and QA/QC management are consistent with industry practice and are deemed fit for purpose. Results of recent sampling have not identified any issues which materially affect the accuracy, reliability or representativeness of the results.

It is the resource QP's opinion that the sample preparation, analytical, QA/QC and chain of custody procedures used to produce the sample database are consistent with industry practises and CIM Mineral Exploration Best Practice Guidelines (November 2018).

12 DATA VERIFICATION

This Item summarizes the data verification conducted by the QP which consisted of a personal site inspection, verification logging and sampling, verification of drill hole collar locations, independent checks on the assay data and database validation including spot checks of the assay data compared to laboratory certificates and checks on collar locations, downhole surveys and interval data, among others.

12.1 SITE INSPECTION

Site inspection was completed by Dr Peter Bolt, PhD, from 14 to 18 August 2023, Mr. Timothy Daffern 6 February 2025, and Mr Brian Thomas, P.Geo., an independent QP, as defined under NI43-101 and an employee of WSP, from 11 to 13 July 2023.

The site inspection covered only the Ikkari deposit and excluded the other Rupert Resources deposits in the region (Pahtavaara, Heinä Central) that comprise the Rupert Lapland project.

Dr Peter Bolt examined the project setting, inspected the project site, storage facilities for drill core and reviewed the geotechnical data and logging procedures. The principal environmental concern discussed was related to future water discharges from site.

Mr Timothy Daffern traversed the entire proposed site as well as current site sub surface drilling operations at two sites, the planned co-disposal topography, the proposed mineral process plant area and the planned open pit mining area. During the visit Mr Daffern also inspected the two minor streams that traverse the site along with the installed monitoring stations.

Mr Brian Thomas site inspection included the following items:

The site inspection included the following items:

- Review of geology, mineralisation and structural controls on mineralisation;
- Review of current interpreted geological models;
- Review of drilling, logging, sampling, analytical and QA/QC procedures;
- Review of site security and chain of custody of samples from the drill to the lab;
- Independent verification logging and sampling of selected drill holes;
- Inspection of the project site and verification of drill hole collar locations; and
- Inspection of storage facilities for drill core and pulp samples.

Rupert Resources provided access to all data requested and there were no restrictions or limitations imposed on the QP.

12.2 IKKARI PROJECT SITE

The QP went to the project site to observe current conditions and verify the location of randomly selected drill hole collars. As the project is still in the exploration phase, there is little development at the site which mainly consists of storage containers and a large lay down area for drill equipment as shown in Figure 12-1 and Figure 12-2. Waypoints were taken at 7 drill hole collar locations with a handheld GPS and imported into Google Earth to confirm their locations (Figure 12-2).

Figure 12-1 – Ikkari Project Site

Figure 12-2 – Verified Collar Locations (Image from Google Earth)

Collar locations were flagged with pickets, and casings were capped with the hole number stamped onto the caps as seen in Figure 12-3.

Figure 12-3 – Example Drill Hole Collar

Collar coordinates measured at the site were compared to the Rupert Resources database and found to be consistent within the 3 – 6 m accuracy of the handheld GPS used as summarized in Table 12-1.

Table 12-1 – Collar Verification Summary

	Rupert Resources Survey Coordinates			WSP Verification Coordinates		
Hole ID	Easting	Northing	Elevation	Easting	Northing	Elevation
120061	454287	7496721	228	454289	7496721	232
121032	454182	7496575	229	454185	7496571	224
121125	454244	7496726	227	454247	7496722	227
121168	454278	7496750	227	454281	7496751	225
122195	454223	7496582	230	454225	7496579	229
122214	454231	7496660	228	454233	7496658	228
122246	454300	7496607	232	454306	7496605	229

Notes: Rupert Resources survey coordinates rounded to the nearest metre.
WSP handheld GPS accurate to approximately 3 - 6 m.

12.3 VERIFICATION LOGGING AND SAMPLING

Intervals from four holes were selected for verification logging and sampling during the site visit including holes 122001 (364 m – 369 m), 121068 (369 m – 376 m), 121158 (293 m – 298 m) and 123042 (171 m – 180 m) which represented different mineralized lithologies and holes drilled from opposing orientations. A total of 26 quarter sawn core samples were taken along with the submission of control samples consisting of 3 standards and 2 pulp duplicates and 1 blank. Samples were placed in plastic bags and closed with sample tags attached and submitted to Eurofins laboratory located in Sodankylä for fire assay analysis consistent with the methodology used by Rupert Resources.

There was one very high-grade Rupert Resources sample that wasn't reproduced (74.80 g/t vs 8.19) but aside from that outlier, the verification assay results were generally consistent with Rupert Resources original assays and no bias was observed. Variability of assay results from field duplicates is common in gold deposits due to the nature of the gold distribution and differences in sample volumes between the original half core samples and quarter core verification samples. A scatterplot was generated to graphically compare the verification results to the original assays, as shown in Figure 12-4

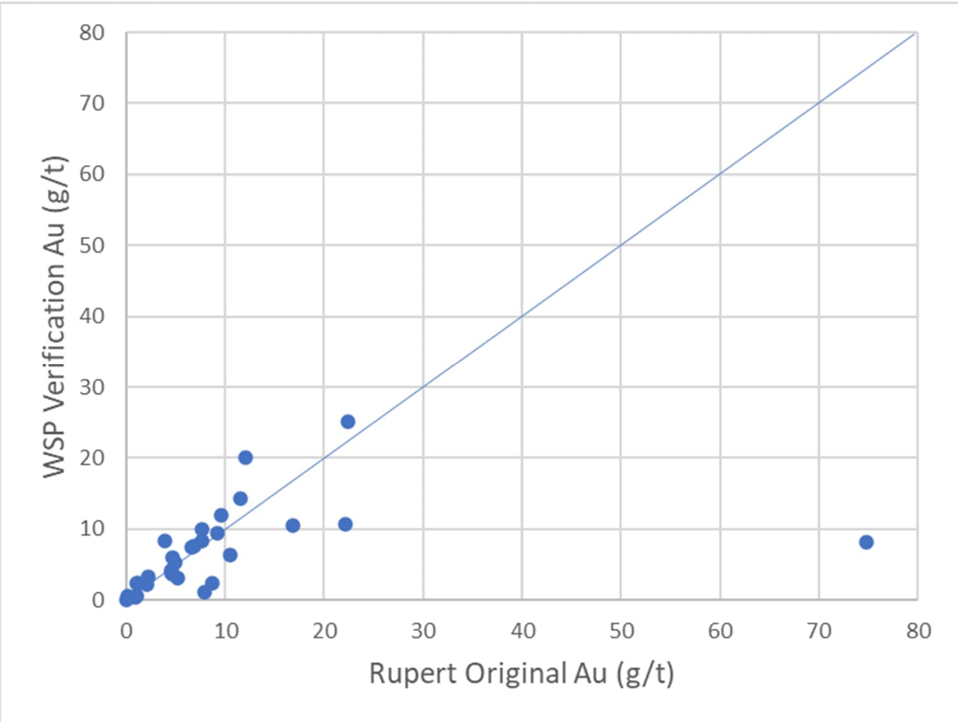


Figure 12-4 – Scatterplot Comparison of Rupert Resources vs WSP Verification Assays

12.4 DATABASE VERIFICATION

The drillhole database was verified based on the following spot checks and analysis of data:

- Analysis of collar coordinates incl. spot check comparisons against original survey pickups;
- Analysis of downhole surveys incl. spot check comparisons against original downhole measurements;

- Analysis of assay results incl. spot check comparisons of Au assays against laboratory certificates and out-of-range values;
- Analysis of density values; and
- Analysis of interval data for overlaps and gaps.

No material issues were identified during the database verification process.

12.5 CHAIN OF CUSTODY

The chain of custody procedures for drill core and samples was reviewed with no material concerns identified, with the risk for potential sample manipulation considered to be low. The core is stored and processed in a secure, modern facility with restricted gate access as shown in Figure 12-5. Although Rupert Resources does not use security tags for their sample shipments, they are shipped in secure wooden bins with plywood lids screwed in place and samples are shipped, generally short distances by a third-party contractor.

Figure 12-5 – Rupert Resources Core Logging and Storage Facility, Sodankylä, Finland

12.6 CONCLUSIONS AND RECOMMENDATIONS

After completion of the site visit and data verification, Brian Thomas the Qualified Person for MRE concludes that the exploration, drilling and analytical procedures used by Rupert Resources to collect geological data are consistent with industry practises and CIM Mineral Exploration Best Practise Guidelines (November 2018) and that the data is suitable to support the reporting of the MRE as summarized in this Technical Report.

After completion of the site visit, QP MR, Mr Daffern concludes that the data, information and work practises by Rupert Resources and its contractors meet the standards required by the mineral property development industry.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

This chapter presents mineralogical characterization and metallurgical tests results from laboratory work conducted on samples from the Ikkari deposit. PEA and PFS metallurgical test results as well as a DFS testwork memo are detailed in the following previously published reports:

PEA:

- Research report from samples AEM 001-AEM020 Ikkari Au prospect, Kari Kojonen FT, spring 2021;
- 21-1882 Ikkari Deposit Gold Recovery Testing - Rupert Resources, Grinding Solutions, May 2021;
- 22-1967 Rupert Resources Phase II - Ikkari Gold Recovery Optimisation Testing1, Grinding Solutions, February 2022; and
- 22-2061 Rupert Resources - Pre-aerated Cyanide Leach Testing, Grinding Solutions, May 2022.

PFS:

- 24-2176 Pre-feasibility Level Metallurgical Study on the Ikkari Gold Project, Finland - Rupert Resources, Grinding Solutions, March 2024;
- 24-2244 PFS - Whole Ore Leach & Thickening Filtration Testing - Rupert Resources, Grinding Solutions, September 2024;
- 51-0523-00-TW-REP-0001 Rev A Ikkari Dewatering Testwork: Phase 1 Laboratory Report, Paterson & Cooke, June 2024; and
- 51-0523-00-TW-REP-0002 Rev A Ikkari Dewatering Testwork: Phase 2 Laboratory Report, Paterson & Cooke, August 2024.

DFS:

- 25-2266 Rupert Resources – Ikkari DFS Testing Memo, Grinding Solutions, December 2024.

The professionally qualified person reviewed the above metallurgical testwork reports to prepare this technical report. Earlier work completed by ALS is not described in this section but has also been consulted. The professional specialist was not responsible for the samples selection and did not oversee the test programs. However, the samples are deemed to be representative of the Ikkari deposit.

13.2 MINERALOGICAL TESTWORK

In the spring of 2021, Kari Kojonen FT issued a mineralogical examination report for the work conducted on two samples from the Ikkari deposit. Six polished sections and 18 polished thin sections were prepared from six drill cores for microscopical, SEM/EDS and electron microprobe point analysis and element distribution maps. The mineralogical study was completed using a polarizing microscope in reflected and transmitted light.

The major minerals in the samples were observed to be pyrite, magnetite, ilmenite, rutile with minor amounts of chalcopyrite, sphalerite, galena, and native Au alloy as 5-100 µm inclusions in pyrite and the gangue minerals. Native gold is present in the samples for the most part in connection with pyrite, on its surface or in inclusions and on fracture surfaces. In addition, native gold is in the grain boundaries of gangue minerals. The average of the gold analysis in the samples is 10.8 g/t Au. The occurrence of pyrite and native gold refers to epigenetic gold in shear zones.

Other ore minerals in the samples include magnetite, ilmenite, rutile and titanomagnetite. Accessory minerals include monazite, xenotime, zircon, brannerite, and apatite. The main minerals in the samples are sericite, carbonate, quartz, biotite and chlorite.

Based on elemental distribution images, pyrite shows compositional zoning growth and is also heterogeneous in terms of Ni and Co concentrations. The average concentrations of pyrite are Co 1.07% and Ni 0.27% (EDS); Co 0.90% and Ni 0.20% (EPMA) and magnetite Co 0.44% and Ni 0.33% (EDS).

13.3 IKKARI PEA METALLURGICAL TESTWORK

The Ikkari PEA metallurgical testwork was carried out in two phases by Grinding Solutions (2022a, 2022b and 2022c).

The report from Phase 1 of the testing was issued on May 19th, 2021. This program focused on tests related to head grade analysis, comminution (SMC, Bond abrasion, Bond rod and ball mill indices), gravity recovery, cyanidation, flotation and thickening.

Phase 2 focused on optimising the gold recovery. The report was issued on February 9th, 2022. The focus was on reaffirming the processing methodology on newly submitted sample and optimising parameters such as primary grind size, flotation reagents scheme and cyanidation conditions. Cyanide detoxification tests were also performed.

As flotation is not considered for the Ikkari process design within the PFS study, information related to these tests has been omitted from this report. Also excluded, are results from tests completed on flotation concentrate or flotation tailings (thickening, environmental testing, etc.).

Based on information provided by Rupert Resources, all the samples tested for the Phase 1 testwork program at Grinding Solutions Limited (UK based technical consultancy) were taken from within the boundaries of the projected open pit area. For the Phase 2 program, most of the samples also originated from the projected open pit area, with the others taken at depth under the open pit.

13.3.1. HEAD GRADE

Direct assays of the sample used for the first phase of tests indicated that the gold content ranged from 4.76 g/t Au to 9.54 g/t Au. During testing of the sample, the back calculated Au head grade ranged between 3.6 g/t Au and 4.2 g/t Au. It is considered that the result showing 9.54 g/t Au was an outlier. The silver content of the sample was shown to be 0.4 g/t. The sulphide speciation showed that the sample contained 1.88% of sulphide sulphur. The carbon speciation indicated that only 0.03% of the carbon was organic.

The sample used for the second phase of tests contained 3.14 g/t Au and 0.5 g/t Ag. Sulphide content was shown to be 1.3% and organic carbon content was low at 0.04%. Cd, Hg, U, and Th levels are all shown to be below levels of detection. It was noted during the testing programme that the weighted average of the back calculated gold grade was lower (1.81 g/t Au) than indicated in the head assay. This value was taken as the correct gold head grade.

13.3.2. COMMINUTION

SMC Testwork

The results of the SMC testwork are shown in Table 13-1 and Table 13-2. The results are compared to the SMC hardness classification (Table 13-3), showing that the material is harder than medium and abrasive.

Table 13-1 – SMC Test Results

Sample	DWI (KWh/m^3)	DWI %	Mi Parameters (kWh/t)			SG
			Mia	Mih	Mic	
Rupert Resources	7.30	58.00	19.40	14.60	7.50	2.90

Table 13-2 – Parameters Derived from the SMC Results

Sample	A	b	A*b	t_a	SCSE (kWh/t)
Rupert Resources	60.8	0.65	39.5	0.35	10.38

Table 13-3 – Hardness Classification for the SMC Results

DWT Relative Values		Very Hard--------------------------------Medium---------------------------------Very Soft						
A*b	Impact	<30	30-38	38-43	43-56	56-67	67-127	>127
t_a	Abrasion	<0.24	0.24-0.35	0.35-0.41	0.41-0.54	0.54-0.65	0.65-1.38	>1.38

Abrasion Testwork

The Bond abrasion index of the submitted sample was shown to be 0.59, which categorises the material as abrasive.

Bond Rod Mill Grindability

A sample was submitted for Bond rod mill work index testing at a closing size of 1180 µm. The material had a RWI of 11.8 kWh/t, which is considered marginally soft.

Bond Ball Mill Grindability

A sample was submitted for Bond ball mill work index testing at a closing size of 150 µm. The material showed a BWI of 15.5 kWh/t, which is considered of average hardness.

13.3.3. GRAVITY RECOVERY

Gravity Release Analysis

A 1 kg sample pre-ground to -1 mm was sized into 6 size fractions for separate gravity release tests.

The gravity release tests show that gravity concentration is viable for grinds below 600 µm, with the best results attained at size fractions below 300 µm. Below 300 µm, a mass pull of approximately 13% would provide a gold recovery of about 74% to a gold grade of around 32 g/t.

Figure 13-1 shows a steep cumulative gold recovery for particles below 600 µm indicating that gravity recovery is a viable concentration and recovery method.

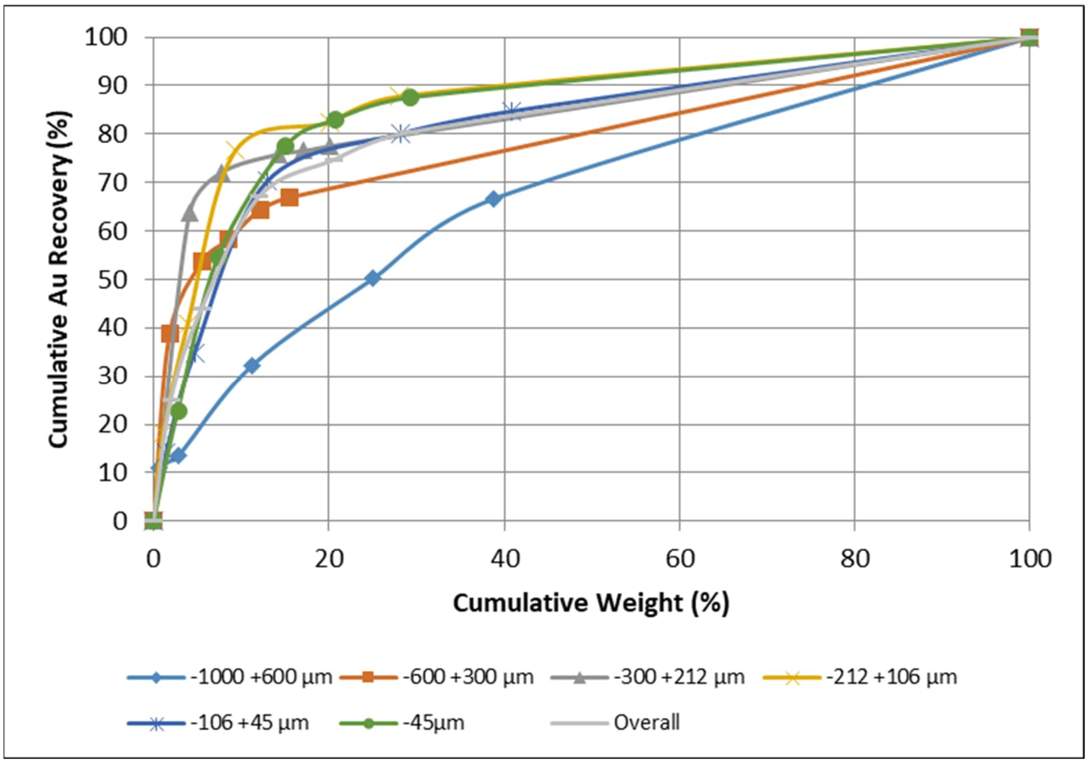


Figure 13-1 – Mass Pull vs Au Recovery Curves for Gravity Release Analysis

Gravity Recoverable Gold and E-GRG Test

Gravity recoverable gold ("GRG") testing showed that 65.2% of the gold was recovered by the gravity concentrators. Mozley™ panning demonstrated that the gold content could be cleaned further. However, no free gold was observed on the table.

E-GRG testing was carried out in the second phase of testing. E-GRG tests on the feed sample demonstrated a GRG recovery of 47.0% at a mass pull of 1.15% over three recovery stages (509 µm, 270 µm and 185 µm). The head grade was back calculated and found to be 1.7 g/t Au.

The E-GRG tests showed that gold recovery generally occurs for particles 300 µm. The P_{80} of the recovered gold was 167 µm, the P_{50} was 84 µm and the P_{20} was 36 µm. Compared with the database, this GRG gold grain size distribution is considered to be moderately coarse.

13.3.4. WHOLE ORE LEACHING

In the first phase of testing, a series of whole ore cyanidation tests were performed at different grind sizes to assess the amenability of the ore to direct cyanidation. The tests were carried out at a cyanide concentration of 1 g/l.

The gold and silver extraction kinetics results are shown in Figure 13-2 and Figure 13-3.

The results show gold extraction between 94.8% and 98.8% for all grind sizes. Gold leaching was complete after 24 hours. Tests performed on samples 53 µm and finer have higher gold recoveries with 98.8% at 38 µm and 98.5% at 53 µm.

Silver recoveries were lower, ranging between 48.5% and 81.7%.

The cyanide consumption ranged between 0.3 and 0.5 kg/t of feed showing a slight increase with the finer grind sizes. Lime consumption varied between 0.3 to 0.6 kg/t of feed again showing an increase in finer grind sizes.

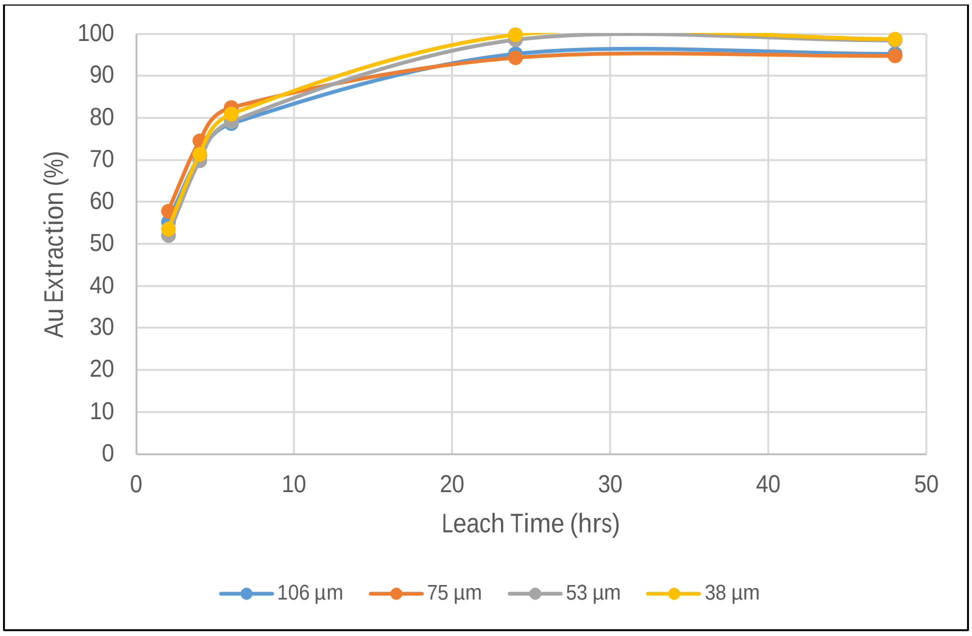


Figure 13-2 – Kinetic Extraction Plots for Au for the Mesh of Grind Cyanidation Tests

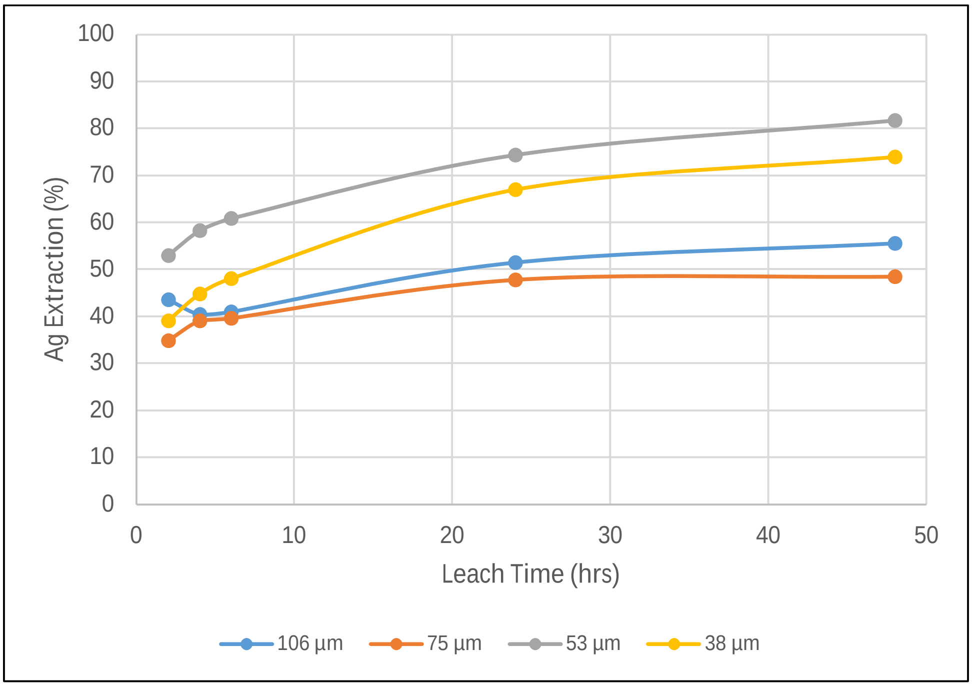


Figure 13-3 – Kinetic Extraction Plots for Ag for the Mesh of Grind Cyanidation Tests

13.4 IKKARI PFS METALLURGICAL TESTWORK – FLOWSHEET INVESTIGATION (PHASE 1)

The Ikkari PFS metallurgical testwork was carried out in two phases by Grinding Solutions Ltd (2024a and 2024b). The reports were issued on March 2nd and September 13th, 2024.

The first phase of testing was designed primarily to investigate the flowsheet selected from the Ikkari PEA study consisting of milling with gravity recovery on cyclone underflow, followed by flotation of gravity tailings and leaching of generated flotation concentrate. The second phase of the testing was aimed at providing metallurgical data of whole ore cyanidation after the extraction of gravity recoverable gold.

This program included tests related to head characterization, comminution (SMC, Geopyörä, French crushability and abrasiveness, Bond abrasion, Bond rod and ball mill indices), gravity recovery, cyanidation, flotation, rheology and oxygen uptake rate as well as environmental testing (waste compliance, acid base accounting, net acid generation potential).

As flotation is not included in the Ikkari processing plant design for this PFS study, information related to these tests has been omitted from this report.

This section summarises the results of Phase 1 of the PFS testwork program.

Based on information provided by Rupert Resources, all the samples tested for the Phase 1 testwork program at Grinding Solutions Ltd., were taken from within the boundaries of the projected open pit area.

13.4.1. SAMPLES RECEIVED AND HEAD CHARACTERIZATION

Rupert Resources provided two master composites for the Phase 1 test program, one Felsic and one Ultramafic, and fifteen variability samples.

Chemical Head Analysis

Direct assays of the composites indicated gold grades at 4.73 g/t Au and 4.97 g/t Au for the Felsic and Ultramafic composites respectively. Silver grade was under the detection limit of 0.1 g/t for both composites. Both composites returned negligible quantities of organic carbon at 0.04% and 0.02% respectively for the Felsic and Ultramafic composites. The ultramafic composite was shown to possess a higher concentration of sulphide mineralisation with sulphide sulphur showing as 1.33% versus 0.67% for the Felsic composite.

All the fifteen variability samples were analysed. The results show that the gold grade ranged from 0.05 to 7.91 g/t Au averaging 2.55 g/t Au. While silver grade was 0.4 g/t Ag or lower. All sample contained low base metal contents.

Mineralogical Characterization

Mineralogical study and gold deportment analysis were realized on both composites.

Modal mineralogy indicates that the Felsic composite has a particle P_{80} of 232 µm and consists of quartz (41.5%), plagioclase (31.3%), muscovite mica (7.47%), and magnesite (6.87%). Pyrite represents 1.37% mass of the sample and is 86.9 % liberated (summation of liberated, free, and pure) at this grind size. The Ultramafic composite has a particle P_{80} of 179 µm and consists of quartz

(34%), magnesite (26.9%), lesser quantities of muscovite (7.96%), chlorite (6.16%), and biotite (5.55%). Pyrite represents 2.86% mass of the sample and is 90.0% liberated at this grind size.

For both composites, pyrite is generally most prevalent in the 100 µm and coarser size classes.

Mineralogical gold deportment analysis shows that the majority of the gold in both composites occurs as visible or microscopic gold at 98.6% and 99.0% respectively for Felsic and Ultramafic. The remaining (1.4% and 1.0% respectively) occurs as sub-microscopic gold. Moreover, all of the microscopic or visible gold content in both composites occurs as native gold.

Bulk modal mineralogical analysis was performed on the variability samples, mineral abundancies the samples differ. Pyrite ranges from 0.24% to 8.86%, magnesite from 2.3% to 43.3%, Fe Oxides from 1.9% to 11.1%, chlorites from 1.9% to 16.4%, quartz from 19.5% to 67.2%, muscovite from 0.3% to 20.8%, and plagioclase from 0.2% to 53.8%. The particle P_{80} varies between 616 and 790 µm. The variability samples liberation and association of pyrite showed that free and liberated pyrite account for 33.9% to 78.4%, while the non-liberated pyrite occurs as complex particles (17.3% to 64.7%), quartz/feldspars (0.1% to 10.6%), Fe-Oxides (0.01% to 8.28%), and minor associated particles with other minerals (<6% each).

13.4.2. COMMINUTION

SMC testwork

The Felsic and Ultramafic composites were submitted for SMC testing which is a simplified version of the JK Drop Weight test. It provides the same basic ore comminution parameters but requires less material. The result is used to determine the JK Drop Weight Index (DWI), which is a measure of the strength of the rock when broken under impact conditions. Table 13-4 presents the results for the composites. When referring to the hardness classification for the SMC results (Table 13-3) the composites can be considered hard and abrasive.

Table 13-4 – SMC Test Results for Felsic and Ultra Mafic Composites

Job	Serial	Description	A	b	A*b	t_a	DWI (kWh/m^3)	SCSE (kWh/t)
24-2176	1128	Felsic	93.5	0.31	28.99	0.27	9.6	11.76
24-2176	2001	Ultra Mafic	71.4	0.5	35.70	0.31	8.29	11.05

SMC testing was also conducted on composites generated from the individual variability samples:

1) Ultramafic Dominated (ID No. 8001) – assumed softest, least abrasive;
2) Mixed Ultramafic (ID No. 8101) – Felsic Material – Intermediate; and
3) Felsic Dominated (ID No. 8201) – assumed hardest, most abrasive

Table 13-5 summarizes the results. Details of the variability blend composites can be found in the previously published report.

Table 13-5 – SMC Test Results for Variability Blend Composites

Job	Serial	A	b	A*b	t_a	DWI (kWh/m^3)	SCSE (kWh/t)
24-2176	8001	67.1	0.57	38.25	0.34	7.67	10.63
24-2176	8101	67.8	0.5	33.90	0.30	8.65	11.32
24-2176	8201	96.4	0.32	30.85	0.29	8.9	11.42

Geopyörä Testing

The test uses small discrete samples from full or halved one-meter section of drill cores or less than a kilogram of crushed rocks. The data measured by Geopyörä can be used to estimate standard comminution parameters such as A*b, DWI (Drop Weight Index) and BWI (Bond Ball Mill Work Index) as well as rock mechanical properties. Table 13-6 presents the results obtained for the main comminution parameters derived from Geopyörä testing for both composites.

Table 13-6 – Calculated Comminution Parameters from Geopyörä Testing for Felsic and Ultra Mafic Composites

Sample	Description	t10 High (%)	t10 Low (%)	-150 µm High (%)	-150 µm Low (%)	Axb	SG (t/m^3)	DWI (kWh/m^3)	BWI (kWh/t)	PLT Is (MPa)	UCS (MPa)
24_2176_1129_1	Felsic Comp	16.8	7.6	2.9	1.3	23.9	2.7	11.1	18.4	13	259.8
24_2176_1129_2	Felsic Comp Dup	17.1	7.5	2.7	1.3	23.1	2.7	11.3	18.8	9.8	194.9
24_2176_2002_1	Ultra Mafic Comp	21.4	5.1	4.3	1	29.4	2.8	9.4	16.4	9.6	192.5
24_2176_2002_2	Ultra Mafic Comp Dup	22.4	5.6	4.8	1.2	34.7	2.8	8	15.2	8.6	171.2

Each of the individual variability samples and the variability blended composites was submitted for Geopyörä testing. Table 13-7 presents the ranges obtained for the main comminution parameters derived from Geopyörä testing.

Table 13-7 – Calculated Comminution Parameter Ranges from Geopyörä Testing for Variability Samples

Parameter	Units	Low	High
A*b	-	23.1	37.3
SG	-	2.6	2.9
DWI	kWh/m^3	7.6	11.3

Parameter	Units	Low	High
BWI	kWh/t	14.9	18.8
PLT	MPa	8.6	14.7
UCS	MPa	171.2	322.6

French Crushability and Abrasiveness Testing

French crushability and abrasiveness testing was performed to provide crusher sizing data and an indication of mill liner and grinding media wear rates experienced when processing material from the deposits. Both composites are considered to be of medium crushability, with the Felsic sample considered to be abrasive whilst the Ultramafic is medium abrasive, Table 13-8 summarizes the results.

Table 13-8 – Results of French Crushability and Abrasiveness Testing on Felsic and Ultramafic Composites

Parameter	Units	Felsic 24-2176-1130	Ultramafic 24-2176-2003
Solid Density	t/m^3	2.85	3.01
Crushability	%	30	36
Abrasiveness	g/t	1 340	900
Dust	% -125 µm after French tests	24	28.6

Each of the individual variability samples and the variability sample blend composites were submitted for French crushability and abrasion testing. The results are presented in Table 13-9.

Table 13-9 – Ranges for French Crushability and Abrasiveness Tests for Variability Samples

Parameter	Units	Low	High
Solid Density	t/m^3	2.73	3.03
Crushability	%	30	40
Abrasiveness	g/t	20	1 660
Dust	% -125 µm after French tests	21.5	29.3

The results show that the crushability of the sample ranged from 30 to 40 which is classified as medium to difficult crushability. The results of the abrasiveness portion of the test shows that the results range between 20 and 1 660. This classifies the variability samples as between non-abrasive and very abrasive.

Bond Abrasion Index Testing

Four Bond abrasion indices were performed on both composites: wet rod and ball mill, dry ball mill and crushers. The results of the Bond Abrasion Index tests are summarised below in Table 13-10 and confirm the abrasiveness of the material.

Table 13-10 – Bond Abrasion Index Results for Felsic and Ultramafic Composites

				Wear rates g/kWh						
				Wet Rod Mill		Wet Ball Mill		Dry Ball Mill		Crushers
Job	Serial	Description	Bond Abrasion Index	Rod Media	Steel Liners	Ball Media	Steel Liners	Ball Media	Steel Liners	Liners
24-2176	1131	Felsic	0.6309	143.86	13.72	135.29	10.20	18.01	1.80	35.09
24-2176	2004	Ultra Mafic	0.3264	125.31	11.19	108.03	8.31	12.96	1.29	22.53

Bond Ai units are (g/t).

Each of the variability blend composites was submitted for Bond Abrasion index testing. The results are shown in Table 13-11.

Table 13-11 – Bond Abrasion Index Results for variability Blend Composites

			Wear rates g/kWh						
			Wet Rod Mill		Wet Ball Mill		Dry Ball Mill		Crushers
Job	Serial	Bond Abrasion Index	Rod Media	Steel Liners	Ball Media	Steel Liners	Ball Media	Steel Liners	Liners
24-2176	8002	0.4479	133.97	12.35	120.43	9.17	15.18	1.52	27.54
24-2176	8102	0.3672	128.48	11.61	112.5	8.62	13.74	1.37	24.21
24-2176	8202	0.5798	141.36	13.37	131.48	9.94	17.27	1.73	32.98

Bond Ai units are (g/t).

Bond Rod Mill Grindability

Both composites were submitted for Bond rod mill work index testing at a closing size of 1 180 µm. Felsic and Ultramafic composites had a RWI of 18.08 and 15.53 kWh/t respectively, which is considered hard.

Bond Ball Mill Grindability

Master composites and variability blend composites were submitted for Bond ball mill work index testing at a closing size of 150 µm. Table 13-12 presents the results. All composites were considered hard except the ultramafic dominated that is considered as medium hard.

Table 13-12 – Bond Ball Mill Work Index Results for Felsic and Ultramafic composites and Variability Blend Composites

Composite	BWi (kWh/t)
Felsic	18.9
Ultramafic	14.4
Ultramafic Dominated	13.0
Mixed Ultramafic	15.1
Felsic Dominated	16.4

13.4.3. GRAVITY RECOVERY - E-GRG TEST

E-GRG testing was carried out on both composites.

E-GRG tests on the Felsic composite demonstrated a GRG recovery of 58.0% at a mass pull of 0.98% over three recovery stages (280 µm, 220 µm and 117 µm). The head grade was back calculated and found to be 4.4 g/t Au.

For the Ultramafic composite the GRG recovery was 51.5% at a mass pull of 2.26% over three recovery stages (552 µm, 154 µm and 135 µm). The head grade was back calculated and found to be 5.5 g/t Au.

The sizing data of the recovered gold content shows this to be of moderate fineness.

13.4.4. WHOLE ORE LEACHING

Whole ore leach testing was realized on the master composites and the variability samples. The tests were all carried out at a cyanide concentration of 1 g/l. For the variability samples, a grind size of P_{80} 100 µm was used while the grind size was varied for both master composites to establish a relationship between grind size and direct cyanidation Au extraction.

The gold extraction kinetics results are shown in Figure 13-4 and Figure 13-5 for Felsic and Ultramafic composite respectively.

For Felsic composite, the results show that high levels of extraction are shown for all grind sizes tested, ranging from 96.8% to 99.0%. Cyanide consumption ranges between 0.24 kg/t and 0.49 kg/t NaCN. However, the results of 0.24 kg/t and 0.31 kg/t appear to be outliers, with the true range being between 0.41 kg/t and 0.49 kg/t NaCN. Lime consumption is shown to be between 0.42 kg/t and 1.62 kg/t $Ca(OH)_2$.

For Ultramafic composite, the results show that high levels of extraction are shown for all grind sizes tested, ranging from 95.7% to 98.1%. Cyanide consumption ranges between 0.25 kg/t and 0.58 kg/t NaCN. Lime consumption is shown to be between 1.12 kg/t and 2.57 kg/t $Ca(OH)_2$.

For both composites, the kinetics of the leach show that the extraction of gold was at completion by 24 hours.

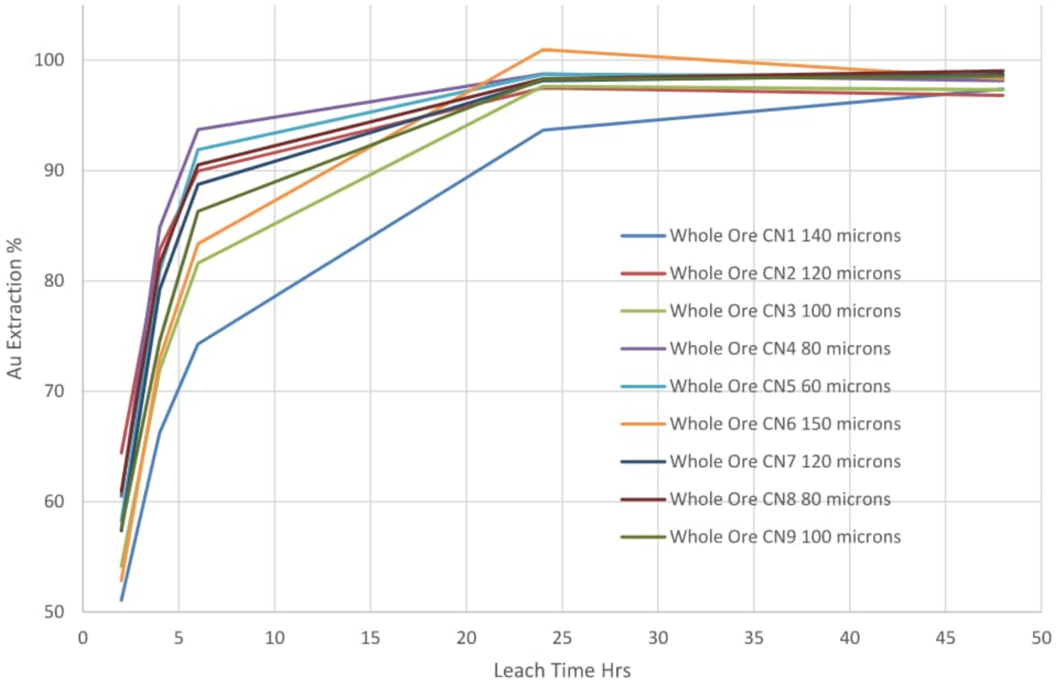


Figure 13-4 – Gold Leach Extraction Kinetics for Felsic Whole Ore Leach Tests

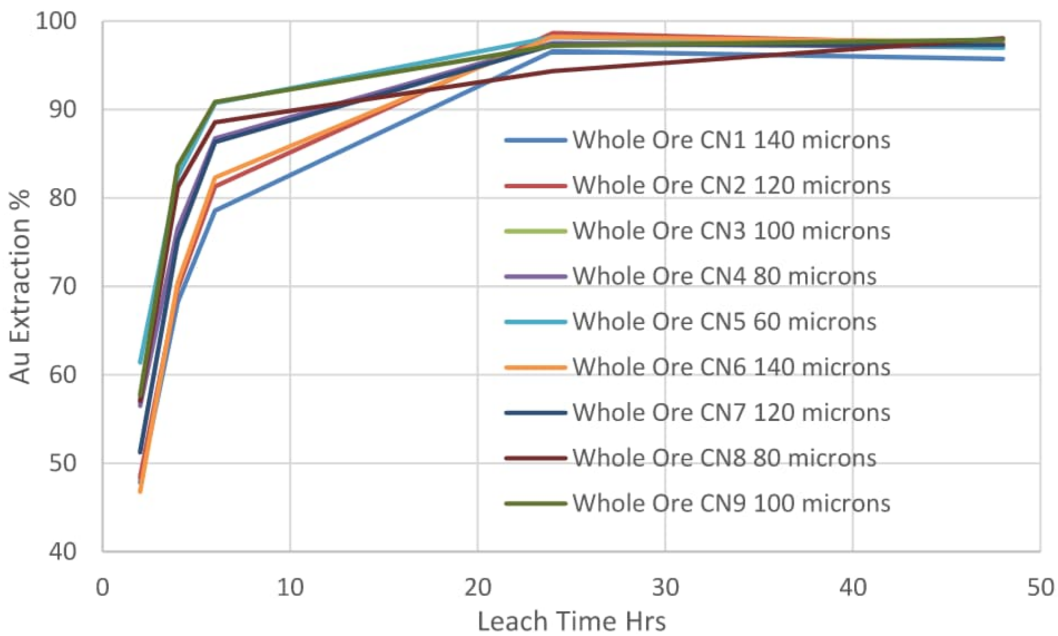


Figure 13-5 – Gold Leach Extraction Kinetics for Ultramafic Whole Ore Leach Tests

Table 13-13 presents the whole ore leach test conditions and summarizes the results for the variability samples. The results show that gold extraction from the samples ranged between 93.39% and 98.76%, averaging 96.81%. The average leach profile of the samples is shown in Figure 13-6, the result shows that leaching reaches completion in 24 hours.

Cyanide consumptions ranged between 0.22 kg/t and 0.67 kg/t NaCN averaging at 0.41 kg/t NaCN. Lime consumption is shown to be between 0.34 kg/t and 3.86 kg/t $Ca(OH)_2$.

Table 13-13 – Test Conditions and Summary Results of Variability Whole Ore Leach Tests

Variability sample		Operating Conditions			Extraction %			Tails ppm			Back Calc ppm			Kg/t	
		NaCN g/l	pH	% Solids	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	NaCN	$Ca(OH)_2$
24-2176 6100	Felsic LG 1	1	10.5-11	40%	93..39	13.14	36.39	0.064	0.380	6.090	0.97	0.44	9.57	0.32	0.66
24-2176 6200	Felsic MG 2	1	10.5-11	40%	97.66	12.90	32.96	0.045	0.266	4.250	1.92	0.32	6.34	0.22	0.57
24-2176 6300	Felsic HG 3	1	10.5-11	40%	98.57	25.58	34.95	0.076	0.213	3.840	5.33	0.29	5.90	0.30	0.74
24-2176 6400	Ultramafic (Pure) LG 4	1	10.5-11	40%	98.06	3.12	25.58	0.017	0.504	4.580	0.93	0.52	6.15	0.39	1.28
24-2176 6500	Ultramafic (Pure) MG 5	1	10.5-11	40%	98.39	6.75	35.70	0.046	0.488	2.570	2.85	0.52	4.00	0.48	1.23
24-2176 6600	Ultramafic (Pure) HG 6	1	10.5-11	40%	97.63	11.06	34.16	0.134	0.428	2.790	5.66	0.48	4.24	0.50	1.02
24-2176 6700	Ultramafic (With Intercalat) LG 7	1	10.5-11	40%	93.55	4.37	37.25	0.070	0.458	3.000	1.10	0.48	4.78	0.43	1.19
24-2176 6800	Ultramafic (With Intercalat) MG 8	1	10.5-11	40%	83.05	53.90	6.91	0.013	0.071	227.800	0.08	0.15	244.70	0.37	0.51
24-2176 6900	Ultramafic (With Intercalat) HG 9	1	10.5-11	40%	97.64	16.93	12.85	0.121	0.353	18.500	5.13	0.43	21.23	0.37	0.97
24-2176 7000	Ultramafic Partially Oxidiced LG 10	1	10.5-11	40%	95.38	6.66	20.89	0.041	0.396	8.500	0.89	0.42	10.74	0.45	3.86
24-2176 7100	Ultramafic Partially Oxidiced MG 11	1	10.5-11	40%	97.64	26.38	23.15	0.026	0.203	28.400	1.13	0.28	36.95	0.30	1.06
24-2176 7200	Intercalted Silts LG 12	1	10.5-11	40%	98.76	8.71	25.24	0.029	0.266	3.840	2.34	0.29	5.14	0.52	0.82

Variability sample		Operating Conditions			Extraction %			Tails ppm			Back Calc ppm			Kg/t	
		NaCN g/l	pH	% Solids	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	NaCN	$Ca(OH)_2$
24-2176 7300	Intercalted Silts MG 13	1	10.5-11	40%	94.84	6.16	14.80	0.094	0.429	49.800	1.83	0.46	58.45	0.39	0.75
24-2176 7400	Intercalted Silts HG 14	1	10.5-11	40%	97.46	14.53	24.88	0.083.	0.191	3.730	3.32	0.23	4.97	0.44	0.56
24-2176 7500	BXH Rich HG 15	1	10.5-11	40%	96.36	8.47	31.79	0.344	0.466	2.460	9.46	0.51	3.61	0.43	1.42
24-2176 7600	Ultra Mafic IC	1	10.5-11	40%	96.76	2.72	16.90	0.115	1.360	7.110	3.56	1.40	8.56	0.67	0.34

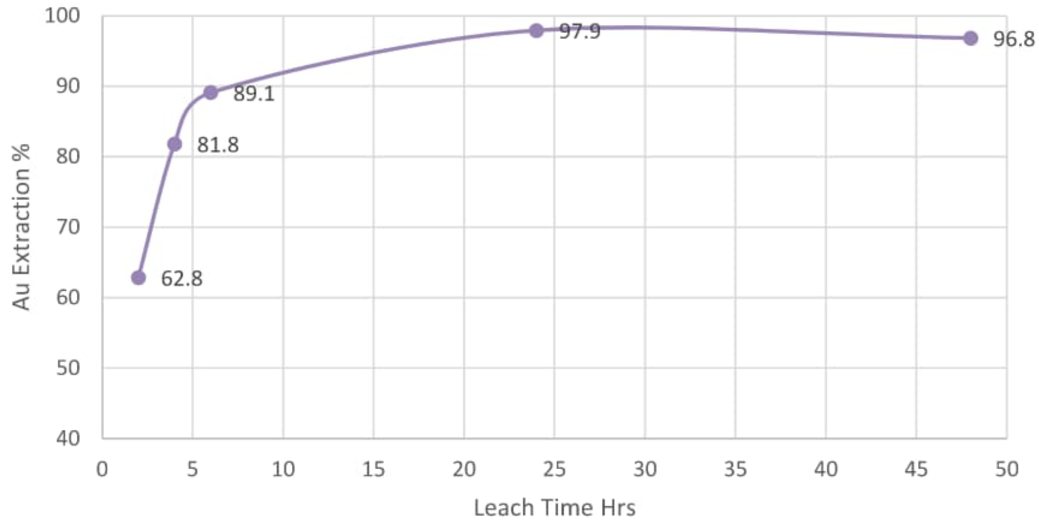


Figure 13-6 – Average Gold Leach Extraction Kinetics Profile from Variability Whole Ore Leach Tests

13.5 PFS IKKARI METALLURGICAL TESTWORK – WHOLE ORE LEACHING TESTING (PHASE 2)

The Ikkari PFS metallurgical testwork was carried out in two phases by Grinding Solutions Ltd (2024a and 2024b). The reports were issued on March 2nd and September 13th, 2024.

The first phase of testing was designed primarily to investigate the flowsheet selected from the PEA study consisting of milling with gravity recovery on cyclone underflow, followed by flotation of gravity tailings and leaching of generated flotation concentrate. The second phase of testing was aiming at providing metallurgical data of whole ore cyanidation after the extraction of gravity recoverable gold.

This program included tests related to head characterization, gravity recovery, cyanidation, and cyanide destruction. Thickening and filtration test charge was also generated from detoxified whole ore leach material from fresh feed and not gravity tailings.

This section summarises the testwork results of the second phase used in the PFS study.

Based on information provided by Rupert Resources, all the samples tested for the Phase 2 testwork program at Grinding Solutions were taken from within the boundaries of the projected open pit area.

13.5.1. SAMPLES RECEIVED AND HEAD CHARACTERIZATION

Rupert Resources provided samples and instruction for the formation of one composite for the Phase 2 test program.

The generated composite contained 2.00 g/t Au. This value was obtained by back calculation from the GRG production.

13.5.2. GRAVITY RECOVERY – GRG

Since the final plant would likely include a gravity circuit, a bulk gravity pass was performed to provide a realistic leach feed in terms of gold content. Laboratory gravity recovery gold tests (3-stage approach) is known to recover near all of the gravity recoverable gold, but the operation of a plant gravity circuit would not be as efficient. To avoid over extracting the gold, a single stage pass was completed on a 250 µm ground sample.

The GRG pass recovered 32.6% of the gold and the mass pull was 0.2%. The gravity tailings graded at 1.35 g/t Au. The back calculated head grade was 2.00 g/t Au.

13.5.3. GRAVITY TAILINGS LEACHING

The gravity tailings obtained from single pass gravity concentration were homogenized and used as feed for different leach tests.

Effect of Grind Size in CIL Cyanidation Testing

Gravity tailings was ground to P_{80} between 75 to 150 µm and submitted to CIL 24-h tests with a cyanide concentration of 1 g/l on a 45% solids pulp. Results are presented in Table 13-14.

Table 13-14 – Results for Grind Size Variation Carbon in Leach Tests

		Adsorption % on Carbon	Tails ppm	Back Calc ppm	Kg/t	
Sheet	Test	Au	Au	Au	NaCN	$Ca(OH)_2$
CIL 150	1	85.8	0.183	1.29	0.00	0.32
CIL 125	2	90.7	0.126	1.36	0.06	0.30
CIL 100	3	93.4	0.075	1.14	0.08	0.41
CIL 75	4	94.6	0.077	1.42	0.08	0.75

The results show that gold extraction increases with decreasing grind size. Highest extraction observed was 94.6 % Au at the finest grind size of 75 µm. A grind size of 100 µm was selected for the PFS to account for the likely detrimental effects of finer particle sizes on downstream processes such as solid liquid separation.

Cyanide consumption was less than 0.08 kg/t NaCN. Lime consumption is shown to be between 0.3 kg/t and 0.75 kg/t $Ca(OH)_2$ with a significant increase at the finest grind size tested.

Effect of Pregnant Robbing (Carbon in Pulp)

A leach test was conducted to determine whether there were any pregnant leach robbing effects as it had been previously observed on flotation concentrate material. The test was carried out at a cyanide concentration of 1 g/l on a 45% solids pulp. As can be seen in Figure 13-7, gold extraction reached 95.1 % after 48 hours of leaching. After 24h, gold extraction in CIL for the 100 µm sample (93.4%) was slightly higher than with CIP (91.8%). Exact causes for the difference haven't been identified but pregnant leach robbing could have taken place with this material.

Cyanide consumption for this test was very low at 0.11 kg/t NaCN and lime consumption was 0.47 kg/t $Ca(OH)_2$.

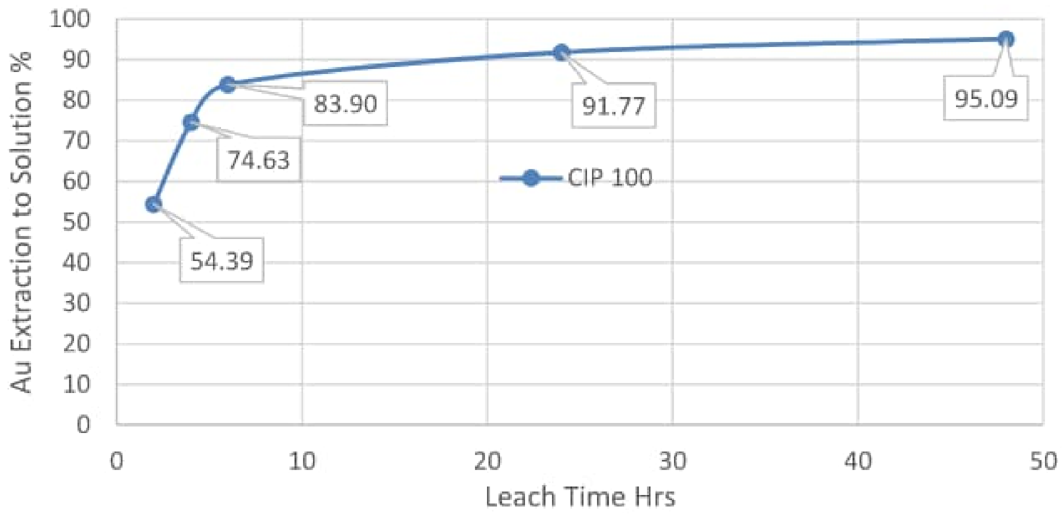


Figure 13-7 – Kinetic Gold Extraction Profile for Carbon in Pulp Leach Test

Effect of Cyanide Dosage in CIL Tests

CIL tests were conducted at a grind size of 100 µm for 24 h on a 40% solids pulp to investigate the impact of cyanide dosage on the recovery of gold to activated carbon. The cyanide concentration varied between 0.25 g/l to 5.0 g/l. Results, presented in Table 13-15, show a minimal effect of cyanide dosage on the recovery of gold within the tested range of cyanide dosages.

Cyanide consumptions were negligible, and lime consumption ranged between 0.26 kg/t and 0.38 kg/t $Ca(OH)_2$.

Table 13-15 – Results for Cyanide Dosage Variation Carbon in Leach Tests

		Adsorption % on Carbon	Tails ppm	Back Calc ppm	Kg/t	
Sheet	Test	Au	Au	Au	NaCN	$Ca(OH)_2$
CIL 5 g/l	6	94.04	0.081	1.36	BDL	0.30
CIL 1 g/l	7	92.46	0.104	1.38	BDL	0.26
CIL 0.75 g/l	8	93.39	0.085	1.29	0.14	0.30
CIL 0.50 g/l	9	94.80	0.061	1.17	BDL	0.38
CIL 0.25 g/l	10	92.79	0.098	1.36	BDL	0.38

BDL: Below detection limit.

CIL – Leach Kinetics

CIL tests were conducted at a grind size of 100 µm to establish the kinetics of gold extraction in a CIL system. The tests were carried out at a cyanide concentration of 0.5 g/l on a 45% solids pulp and 5 g/l carbon concentration. The results are shown in Table 13-16 and Figure 13-8.

Table 13-16 – Results for Leach Retention Time Variation Carbon in Leach Tests

Sheet	Test	Adsorption % on Carbon	Tails ppm	Back Calc ppm	Kg/t	
		Au	Au	Au	NaCN	$Ca(OH)_2$
48 hr CIL 0.5 g/l	11	94.1	0.076	1.29	0.13	0.37
36 hr CIL 0.5 g/l	12	92.1	0.108	1.37	BDL	0.38
24 hr CIL 0.5 g/l	13	95.0	0.075	1.50	BDL	0.36
18 hr CIL 0.5 g/l	14	91.9	0.115	1.43	BDL	0.37

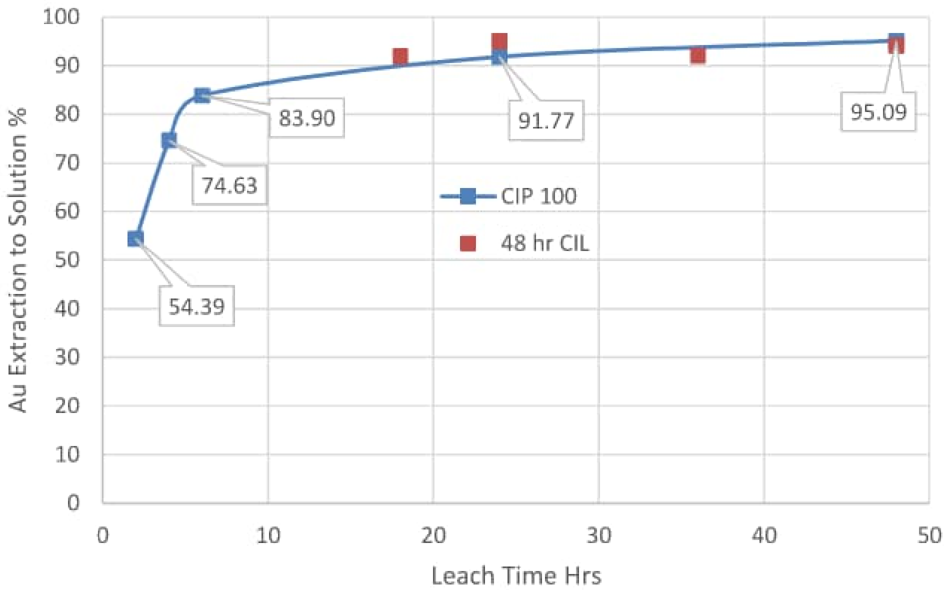


Figure 13-8 – Kinetic Gold Extraction Profile for Carbon in Leach Tests (Red Squares) Compared to CIP test (Blue Line)

CIP kinetic test was performed at a grind size of 100 µm. The gold recovery was 91.8% after 24 hours and 95.1% after 48 hours. CIL kinetic tests were also performed at the same grind size. The gold recovery to the carbon phase showed a recovery of 95.0% after 24 hours and 94.1% after 48 hours.

Following the review of this data, a leach time of 24 hours was selected in a CIL configuration.

Cyanide consumptions during the tests was up to 0.13 kg/t NaCN whilst lime consumption ranged between 0.36 kg/t and 0.38 kg/t $Ca(OH)_2$.

Effect of Lead Nitrate Addition

A single CIL test (Test 15) was performed with the addition of 200 g/t lead nitrate. The test was carried out at a grind size of 100 µm for 24 h at a cyanide concentration of 0.5 g/l on a 45% solids pulp. Results, presented in Table 13-17, show that the gold recovered after 24 hours was 93.8%

which is similar to other tests in absence of lead nitrate addition. Cyanide consumption was negligible, and lime consumption was 0.42 kg/t $Ca(OH)_2$. On review of the results, it was decided that lead nitrate would not be added to further tests.

Table 13-17 – Results for Lead Nitrate Addition (200 g/t) Carbon in Leach Test

		Adsorption % on Carbon	Tails ppm	Back Calc ppm	Kg/t	
Sheet	Test	Au	Au	Au	NaCN	$Ca(OH)_2$
24 hr CIL 0.5 g/lIPBNO3	15	93.8	0.092	1.49	BDL	0.42

Confirmation CIL Test

A confirmation CIL test was conducted at the final chosen conditions: grind size of 100 µm for 24 h at a cyanide concentration of 0.5 g/l on a 45% solids pulp. Results, presented in Table 13-18, show a 94.03% gold recovery to the activated carbon which agrees with previous results. Cyanide consumption was shown to be negligible whilst lime consumption was shown to be 0.39 kg/t $Ca(OH)_2$.

The back calculated gold content for the test was shown to be 1.22 g/t Au comparing against the measured from the gravity tailings of 1.35 g/t Au.

Table 13-18 – Results for Confirmatory Carbon in Leach Test

		Adsorption % on Carbon	Tails ppm	Back Calc ppm	Kg/t	
Sheet	Test	Au	Au	Au	NaCN	$Ca(OH)_2$
CIL 100 (3)	19	94.03	0.073	1.22	0.07	0.39

CIL Results Comparison at Final Chosen Conditions

Table 13-19 presents the results of all tests conducted at the chosen leach conditions. An average gold recovery to activated carbon of 94.94% was obtained. The average back calculated head grade for these tests was 1.34 g/t Au and for all tests conducted was also 1.34 g/t Au comparing to the measured of 1.35 g/t Au. All tests except for test 16 were performed on 1 kg sample charges.

Table 13-19 – Results for CIL Tests Conducted at the Chosen Conditions (Grind 100 µm, NaCN 0.5 g/l, 24 h Leach Time)

		Adsorption % on Carbon	Tails ppm	Back Calc ppm	Kg/t	
Sheet	Test	Au	Au	Au	NaCN	$Ca(OH)_2$
CIL 0.50 g/l	9	94.80	0.061	1.17	BDL	0.38
24 hr CIL 0.5 g/l	13	95.0	0.075	1.50	BDL	0.36

Sheet	Test	Adsorption % on Carbon	Tails ppm	Back Calc ppm	Kg/t	
		Au	Au	Au	NaCN	$Ca(OH)_2$
24 hr CIL 0.5 g/LIPBNO3	15	93.8	0.092	1.49	BDL	0.42
4 kg Kin CND Leach	16	97.03	0.039	1.31	0.43	0.19
CIL 100 (3)	19	94.03	0.073	1.22	0.07	0.39

Cyanide consumption during testing was generally very low. However, when conducting a bulk leach on the gravity tailings in preparation for cyanide detoxification testing, a cyanide consumption of 0.43 kg/t NaCN was observed.

13.5.4. SO_2 AIR CYANIDE DESTRUCTION TESTING

A 4 kg batch of ore was milled and leached at the final chosen conditions (100 µm, 24 h, 0.5 g/l NaCN, 45% solids) to provide pulp for kinetic cyanide destruction testing. The results of the leach show a 97.03% gold recovery (Test 16, Table 13-19).

The pulp generated from the 4-kg cyanide leach test was used for SO_2 Air cyanide destruction testing. A clarified cyanide destruction feed sample was submitted for weak acid dissociable cyanide (CN_{WAD}) determination following a picric acid and visible spectrum photometer method. A 150 mg/l CN_{WAD} solution was obtained. The targeted residual CN_{WAD} after cyanide destruction process was less than 1 mg/l CN_{WAD}.

A retention time of 40 minutes was used for the test with a final target SO_2 addition rate of 7.83 g / g CN_{WAD} used. Copper addition was made at a rate of 0.09 kg/t to maintain a copper in solution concentration of approximately 75 mg/l. The results are shown below in Table 13-20 and illustrated in Figure 13-9 for the continuous test.

Table 13-20 – Results of SO_2 Air Cyanide Destruction Testing - Batch and Continuous

Mode	Retention Time Hrs	g SO2 / g CN_{WAD}	CN_{WAD} mg/l
Batch	0.00	0.00	150
	0.17	1.96	0.502
	0.33	3.91	0.418
	0.50	5.87	0.368
	0.67	7.83	0.364
Continuous	0	-	0.36
	0.5	7.83	0.54
	1	7.83	0.26

Mode	Retention Time Hrs	g SO2 / g CN_{WAD}	CN_{WAD} mg/l
	1.5	7.83	0.34
	2	7.83	0.40
	2.5	7.83	0.85
	2.67	7.83	0.60

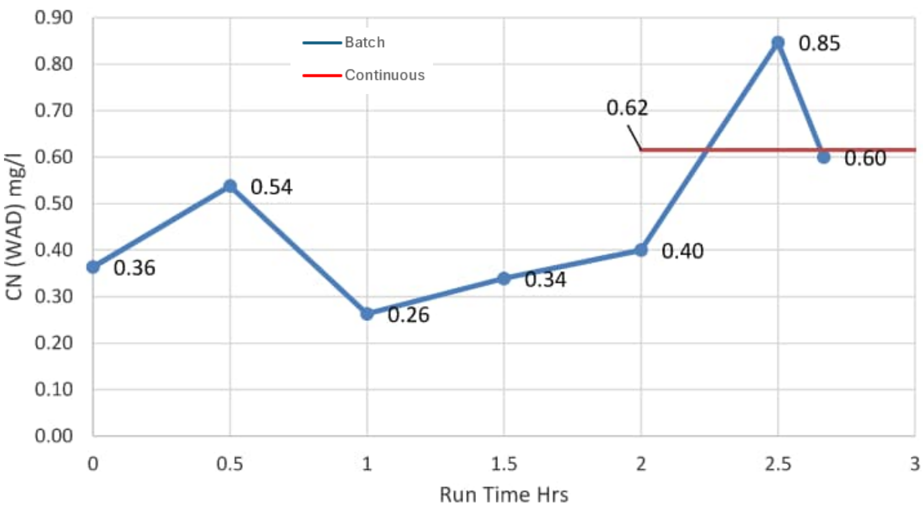


Figure 13-9 – Results of SO2 Air Cyanide Destruction Testing – Continuous

The batch testing results show that the target CN_{WAD} values of below 1 mg/l were achieved very early during the test. This result suggests that lower addition rates of SO_2 could be used to reach the target. However, when in continuous operation, some short circuiting of feed might occur, and it is expected that the discharge at the same reagent feed dosages and retention times would be higher than for a closed batch test.

The continuous test was run with the same reagent addition rates. The continuous test results show that the achieved CN_{WAD} discharge levels were successfully below the requested limit of 1 mg/l CN_{WAD}. The average CN_{WAD} value for the remaining test duration from the point beyond the displacement of three reactor volumes (3 X 40 min = 2 h) was 0.62 mg/l CN_{WAD}. This suggests that lower reagent levels could be used, and future testing would confirm this observation.

13.5.5. THICKENING AND FILTRATION TEST CHARGE GENERATION

To provide material for thickening and filtration testing, 60 kg of detoxified whole ore leached material was generated. The material used for this test was fresh ore feed and not gravity tailings. The test was performed as a bulk agitated leach using an overhead stirrer. The bulk leaching was done under the final chosen conditions: 100 µm, 24 h, 0.5 g/l NaCN, 45% solids.

The pulp was submitted to Paterson and Cooke (UK) Limited to complete the thickening and filtration testing. The results are summarized below.

A representative solids sample was submitted for PSD and indicated a P_{80} of 86.3 µm.

13.6 PFS IKKARI METALLURGICAL TESTWORK – WHOLE ORE DEWATERING TESTING

The dewatering testwork on the Ikkari cyanide detoxified tailings from whole ore leach was carried out in two phases by Paterson & Cooke (UK) Ltd (2024a and 2024b). The reports were issued on June 24th and August 19th, 2024.

The first phase consisted of a full set of dewatering testing on the PFS testwork generated sample including thickening and chamber pressure filtration.

The second phase consists of thickening and pressure filtration characterisation testing on four grind sizes (expected P_{80} 150 µm, 125 µm, 100 µm and 75 µm). These tests were performed to gather data for further grind size optimization. A preferred grind size was then selected for further thickening and chamber pressure filtration testing. This was done to reduce the amount of material required and to accommodate the testing schedule.

13.6.1. PHASE 1 LABORATORY REPORT

Material Characterisation

PFS tailings sample arrived in the form of settled slurry and were first subjected to material characterisation. Water was decanted from the sample and subjected to analysis. As additional process water was unavailable for the testwork, a synthetic process water was produced.

The particle size distribution of the sample was obtained by laser diffraction (P_{80}, P_{50}, and P_{20} determined to be 60 µm, 23 µm, and 8 µm). Solids density of 2.98 t/m3 was measured using a Helium-Pycnometer.

Water density, dissolved salt content, and zero free water solids mass concentration were measured and recorded for the decanted water samples. Samples were also subjected to ICP elemental analysis and Species Photometry.

Samples were also submitted to Petrolab Ltd. for bulk minerology analysis and false light imagery. At the time of this report, results have not yet been made available.

Thickening – Colloidal Stability

A colloidal stability test was performed to determine if a clear supernatant is developed over 24-hours. The sample settled within the first hour with a clear overflow indicating that the sample is not colloidally stable. Therefore, an anionic flocculant is considered sufficient for producing a suitably clear overflow.

Thickening – Flocculant Screening

Six anionic flocculants of various molecular weights and ionic charge were tested. Settling rate comparison of different flocculant products are shown in Figure 13-10. BASF M919 was selected as the flocculant to use for the rest of the thickening testwork.

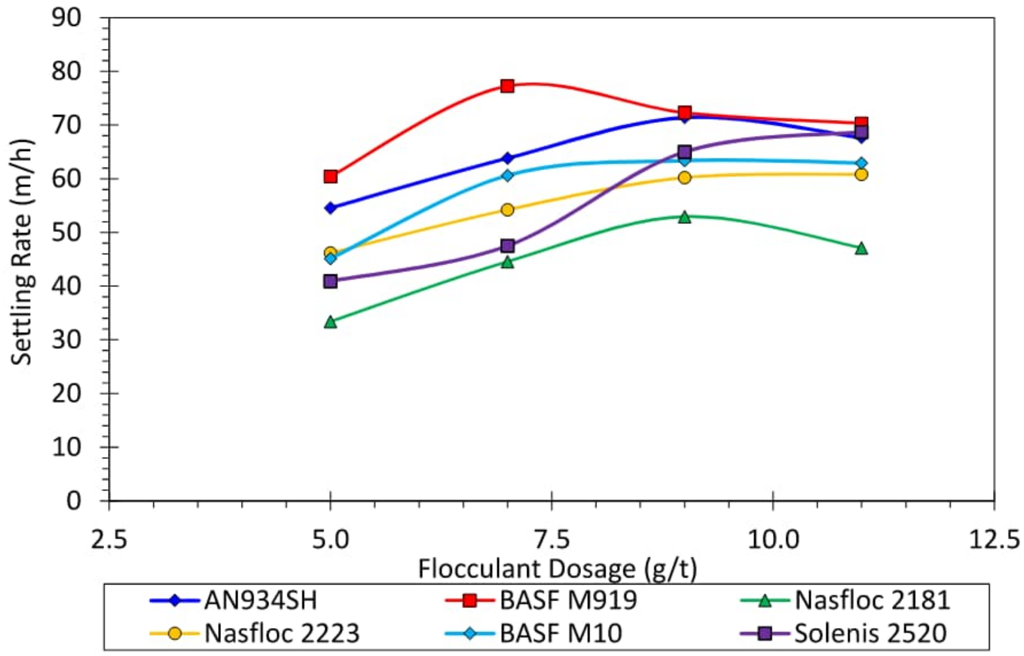


Figure 13-10 – Flocculant Screening on PFS Tailings Samples

Thickening – Static Testwork (Optimum Feed Solids concentration)

Optimal feed solids concentration to the thickener was also determined to achieve optimum settling. Tests are performed by measuring the free settling rate of various feed solids concentrations (6.0% to 16.0%) and flocculant dosages (4.0 g/t to 12.0 g/t). These are converted into free settling solids flux, allowing for a direct comparison of mass throughput. BASF M919 was selected from the flocculant screening test. It is noted that the values obtained figures based on settling rate alone and should not be used for anything other than comparison between feed solids mass concentration in this testing. Based on the results, a feed density of 10% was targeted for the dynamic testwork.

Thickening – Dynamic Testwork

High-rate thickening tests were conducted at a fixed solids loading rate (SLR) of 0.2 (t/h)/m^2 with flocculant dosage range of 10 g/t to 30 g/t. The clearest overflow and the densest underflow were achieved at a flocculant dosage of 25 g/t.

Tests were also conducted at a fixed flocculant dosage of 25 g/t with solids loading rate range of 0.2 (t/h)/m^2 to 0.9 (t/h)/m^2. Solids loading rate of 0.6 (t/h)/m^2 was found to be optimal for the high-rate thickener without impacting the density of the underflow. Note that this is not considering a safety factor for up-scale in sizing.

Thickening – Consolidation and Yield Stress Testwork

Dynamic batch consolidation testwork was carried out to determine the impact of thickener bed retention time on the underflow density production. From the dynamic thickener solids loading rate testwork, a targeted solids rate of 0.6 (t/h)/m^2 with flocculant dosage of 25 g/t was used.

Unsheared and partially sheared vane yield stress measurements on the thickener underflows were taken immediately after discharge. A summary of the thickener work summary is shown in Table 13-21.

Table 13-21 – Dynamic High-Rate Thickening Results Summary

Description	PFS Tails – S100
Optimum Feed Solids Concentration (%m)	10
Flocculant Type	BASF M919
Flocculant Dosage (g/t)	25
Solids Loading Rate ((t/h)/m^2)	0.6
Overflow Concentration (mg/l)	38
Dynamic Underflow Concentration (%m)	65.4
Dynamic Underflow Un-Sheared Yield Stress (Pa)	40.2
1 hr Consolidation Underflow Concentration (%m)	72.9
1 hr Consolidation Underflow Un-Sheared Yield Stress (Pa)	137.3
3 hr Consolidation Underflow Concentration (%m)	74.4
3 hr Consolidation Underflow Un-Sheared Yield Stress (Pa)	170.3
Underflow Solids Range (%m)	61.3-74.4
Underflow Un-Sheared Yield Stress Range (Pa)	16.7-170.3

Pressure Filtration – Filter Cloth Selection

Testwork was performed to determine the filter cloth used to develop detailed test data from chamber filtration. Four clothes were tested under the same conditions. The cloth used for the remaining filtration tests was selected based on filtrate quality and cake release.

Pressure Filtration – Chamber Filtration (without membrane squeeze)

Tests were conducted on 20-, 40-, and 60-mm chambers. Most of the filter feed concentration was targeted as 65% obtained from the thickening testwork with one test at 72%. The summary of the filtration test is shown in Table 13-22.

Table 13-22 – Chamber Press Filtration Results (without membrane squeeze)

Test Run	1	2	3	4	5	6	9
Chamber Thickness (mm)	20	40	60	20	40	60	40
Chamber Diameter (mm)	235						
Feed Solids Mass Concentration (%m)	65.3						72.3
Form Pressure (kPa)	600			1 000			600

Test Run	1	2	3	4	5	6	9
Dry Pressure (kPa)	450			900			450
Cake Form Point (seconds)	34	44	66	30	40	56	36
Cake Dry Time (minutes)	3.90	5.10	5.78	2.73	3.90	4.88	4.35
Cake Thickness (mm)	20.5	41.0	62.8	20.7	43.0	63.3	41.4
Cake Form Solids Mass Concentration (%m)	72.4	77.1	78.1	70.9	77.1	78.2	76.2
Cake Final Solids Mass Concentration (%m)	89.5	87.9	85.8	90.7	89.3	87.8	87.6
Final Wet Cake Density (kg/m^3)	1 636	1 910	1 959	1 835	1 843	1 952	1 897
Final Dry Cake Density (kg/m^3)	1 465	1 680	1 681	1 664	1 647	1 714	1 661
Cake Loading Per Plate (kg/m^2)	30.0	68.9	105.6	34.4	70.8	108.5	68.8

Pressure Filtration – Chamber Filtration (with membrane squeeze)

Filter cake was formed from the pressure and cake thickness testing. After, membrane squeeze, and extended air blow were applied. Results showed that target final cake solids mass concentration can be achieved without the membrane squeeze. The summary of the filtration test is shown in Table 13-23.

Table 13-23 – Chamber Press Filtration Results (with membrane squeeze)

Test Run	1	2
Chamber Thickness (mm)	60	
Chamber Diameter (mm)	235	
Feed Solids Mass Concentration (%m)	65.3	
Form Pressure (kPa)	1 000	600
Membrane Squeeze Pressure (%m)	1 500	1 000
Dry Pressure (kPa)	900	450
Cake Form Point (seconds)	44	85
Membrane Squeeze Time (Minutes)	3.32	4.10
Dry Time (minutes)	3.22	3.38
Cake Thickness (mm)	40.2	40.8
Cake Form Solids Mass Concentration (%m)	80.0	85.7

Test Run	1	2
Cake Final Solids Mass Concentration (%m)	88.8	89.1
Final Wet Cake Density (kg/m^3)	1 986	1 916
Final Dry Cake Density (kg/m^3)	1 762	1 706
Cake Loading Per Plate (kg/m^2)	70.8	69.6

13.6.2. PHASE 2 LABORATORY REPORT

Material Characterisation

The samples to be tested on four different grind sizes arrived in the form of settled slurry and was first subjected to material characterisation similarly to Phase 1. The particle size distributions of the samples were obtained by laser diffraction.

The samples expected to be P_{80} of 150 µm, 125 µm, 100 µm, and 75 µm were determined to be P_{80} of 142 µm, 110 µm, 89 µm, and 62 µm. Solids density of 2.92 t/m^3 for all four samples were measured using a helium pycnometer. The results for the 100 µm test were used to support the preliminary thickener and filtration sizing for the PFS study.

Each of the samples were also subjected to Maximum Bed Packing Concentration tests. Samples were left to settle over 24-hours in a cylinder after which maximum bed packing concentration was determined by applying a hydraulic pressure of over 400 kPa across the settled bed with water draining through the solids.

Similarly to Phase 1, water analysis was performed and recorded for the decanted water samples.

Samples were also submitted to Petrolab Ltd. for bulk minerology analysis and false light imagery. Magnesian siderite, dolomite and pyrite were mostly found in the samples with trace amounts of calcite. The major gangue phases are quartz, mica and clay groups. Chlorite and plagioclase were also present in all samples.

Thickening – Colloidal Stability

A colloidal stability test was performed on all four grind size samples to determine if a clear supernatant is developed over 24-hours. In all four samples, settling occurred within the first hour producing a clear overflow indicated the sample is not colloidally stable. Therefore, an anionic flocculant is considered sufficient for producing a suitably clear overflow.

Thickening – Flocculant Screening

Six anionic flocculants of various molecular weights and ionic charge typically used on tailings slurry dewatering in thickeners were tested on the 150 µm sample. Settling rate comparison of different flocculant products are shown in Figure 13-11. BASF M919 was selected as the flocculant to use for the rest of the thickening testwork.

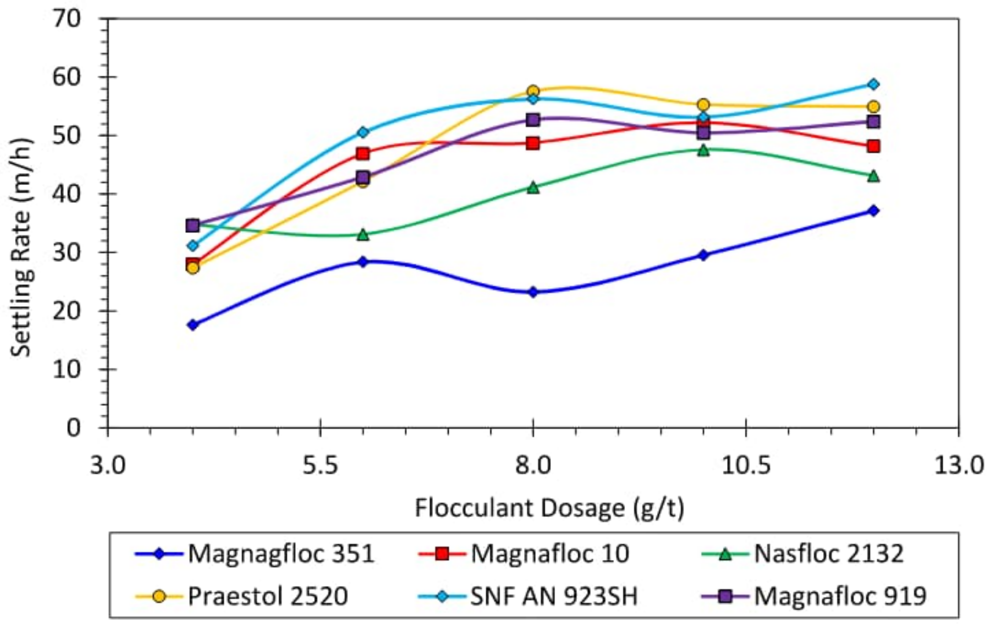


Figure 13-11 – Flocculant Screening on 150 µm Sample

Thickening – Static Testwork (Optimum Feed Solids Concentration)

Optimum feed solids concentration was determined by performing tests where free settling rates of various feed solid concentrations (6.0% to 16.0%) were measured at various flocculant dosages (3.0 g/t to 10 g/t). These are converted into free settling solids flux, allowing for a direct comparison of mass throughput. BASF M919 was selected from the flocculant screening test. Based on the results, the optimum feed densities of 12%, 12%, 10%, and 8% were targeted for grind sizes 150 µm, 125 µm, 100 µm, and 75 µm.

Thickening – Dynamic Testwork

High-rate thickening tests were conducted at a fixed solids loading rate (SLR) for each grind size. These were selected based on settling rates obtained from the static test. Optimal flocculant dosages were determined by taking overflow samples for each grind size test and analysing for solids concentration. Underflow samples were collected and analysed for solids concentration and vane yield stress. A comparison of the testwork result for each grind size is shown in Table 13-24.

Table 13-24 – Dynamic High-Rate Thickening Summary (4 grind sizes)

Description	150 µm – S200	125 µm – S300	100 µm – S400	75 µm – S500
Optimum Feed Solids Concentration (%m)	12	12	10	8
Flocculant Type	BASF Magnafloc 919			
Flocculant Dosage (g/t)	10	20	25	25
Solids Loading Rate ((t/h)/m^2)	0.75	0.75	0.6	0.5
Overflow Concentration (mg/l)	31.7	33.3	53.3	103.3

Description	150 µm – S200	125 µm – S300	100 µm – S400	75 µm – S500
Dynamic Underflow Concentration (%m)	63.4	63.0	62.4	51.4
Dynamic Underflow Un-Sheared Yield Stress (Pa)	9.0	9.3	18.0	13.4

Rheology

Testing was performed on each of the grind sizes as the product from the thickener underflow, flocculated and fully sheared.

The results of the Boger slump testing are shown graphically in Figure 13-12

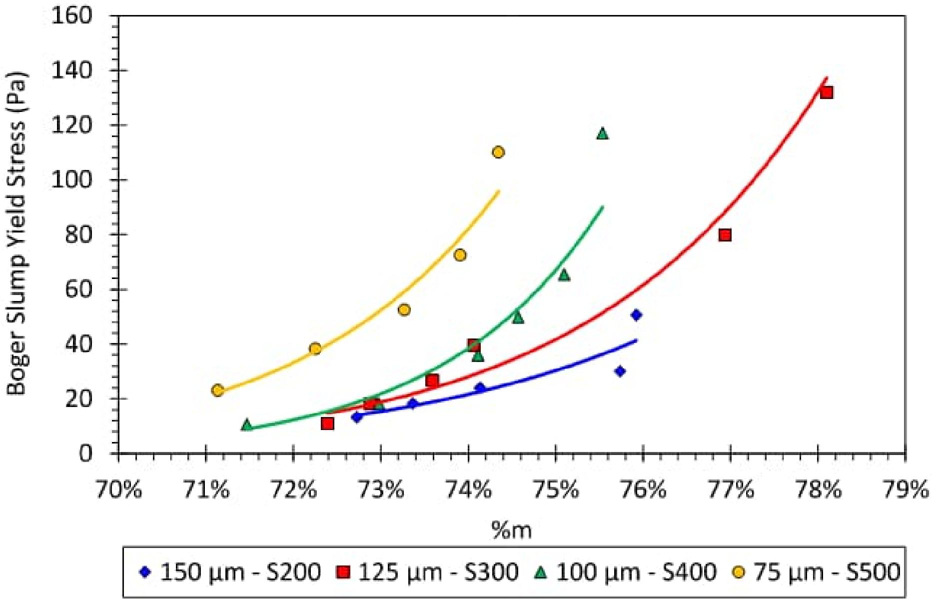


Figure 13-12 – Boger Slump Yield Stress

Samples were also subjected to rotational rheology testing using an Anton Parr viscometer where rheograms at varying solids concentrations were generated to characterise the slurry rheology. All data were corrected for end effects and undeveloped flow. The Bingham plastic model characterised by the plastic viscosity and Bingham yield stress was applied to the rheogram data. A summary graph of the yield stress and the plastic viscosity as a function of the tailings mass solids concentration for all grind sizes are shown in Figure 13-13 and Figure 13-14.

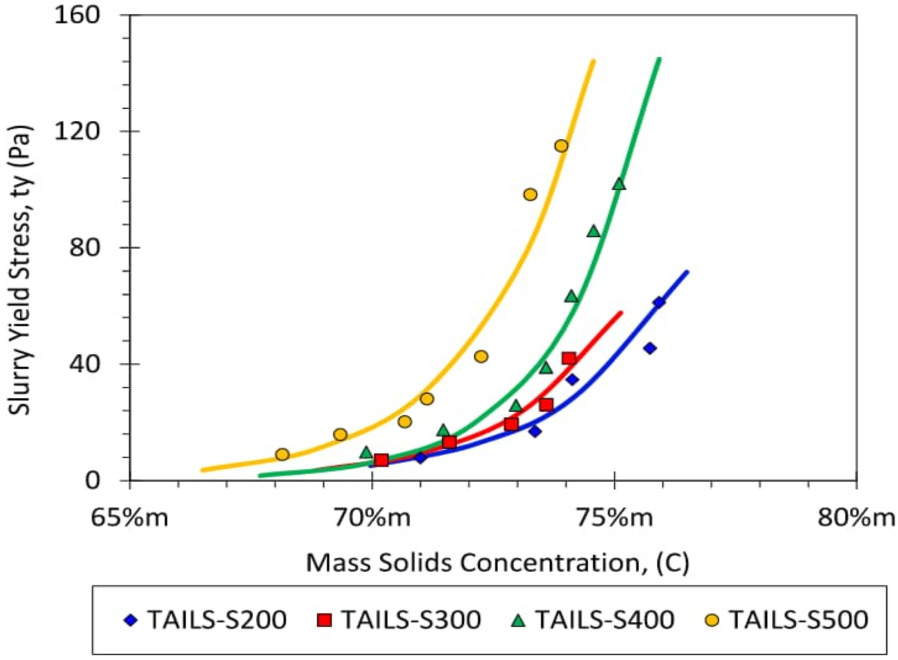


Figure 13-13 – Grind Size Rotational Rheology Yield Stress

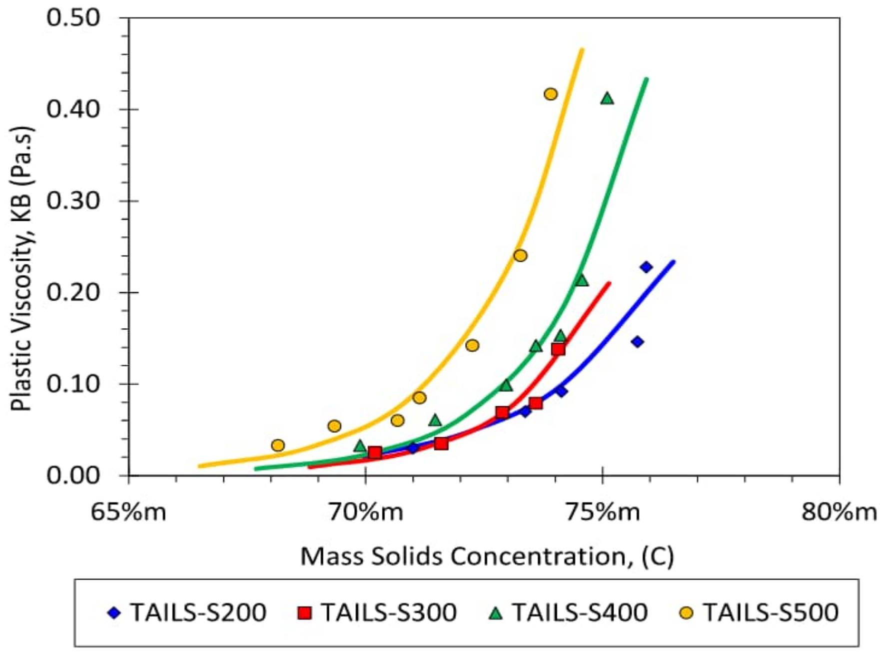


Figure 13-14 – Grind Size Rotational Rheology Plastic Viscosity

Pressure Filtration

Comparative pressure filtration testing was performed on all four grind sizes. Similarly to Phase 1, filter cloth to be used for testing was selected based on the lowest suspended solids and cleanest release while producing an acceptable cake thickness.

The concentration of the slurry feed for each grind size was determined by the thickening test as shown in Table 13-24. Samples were subjected to series of tests which adjusts only the thickness of the filter cake while maintaining all the others. Three different filter cakes were formed at targeted thickness which covered the range of typically available recess chamber depths. Moisture content of the filter cakes was also measured. This test is summarized in Table 13-25. A relationship between the moisture content of the cake and drying coefficient was also developed. The drying coefficient is determined by cake loading/air blow time.

Table 13-25 – Grind Size Filtration Results (variable recess chamber depths)

Description	150 µm – S200	125 µm – S300	100 µm – S400	75 µm – S500
Cake Form Solids Mass Concentration Range (%m)	77.4% - 80.3%	76.5% - 79.8%	75.5% - 78.5%	73.5% - 77.2%
Cake Final Solids Mass Concentration Range (%m)	87.1% - 88.0%	86.6% - 88.4%	86.3% - 87.0%	85.6% - 87.1%
Final Wet Cake Density Range (kg/m^3)	2 056 – 2 061	2 037 – 2 061	1 938 – 2 016	1 883 – 1 965
Final Dry Cake Density Range (kg/m^3)	1 792 – 1 814	1 785 – 1 800	1 687 – 1 747	1 641 – 1 682
Cake Loading Per Plate Range (kg/m^2)	44.6 – 90.6	44.6 – 89.8	43.5 – 87.1	42.2 – 84.6
Average Filtrate Suspended Solids (g/l)	0.66	1.00	0.96	0.58

Filtration test on all four grind sizes using a 50 mm chamber is summarized in Table 13-26.

Table 13-26 – Grind Size Filtration Results (50 mm chamber)

Description	150 µm – S200	125 µm – S300	100 µm – S400	75 µm – S500
Cake Thickness	50.1	50.3	50.8	50.3
50 mm Chamber Cake Form Time (s)	37	34	27	61
50 mm Chamber Cake Form Solids Mass (%m)	80.3%	79.8%	78.5%	77.2%
50 mm Chamber Cake Final Solids Mass (%m)	87.8%	86.6%	86.3%	85.6%
50 mm Chamber Final Wet Cake Density (kg/m^3)	2 061	2 061	1 987	1 965
50 mm Chamber Final Dry Cake Density (kg/m^3)	1 809	1 785	1 715	1 682

Description	150 µm – S200	125 µm – S300	100 µm – S400	75 µm – S500
50 mm Chamber Cake Loading (kg/m^2)	90.6	89.8	87.1	84.6
50 mm Chamber Filtrate Suspended Solids	0.66	1.00	0.96	0.58

13.7 DFS IKKARI METALLURGICAL TESTWORK

Over the course of the PFS, additional testwork has commenced to inform a future Definitive Feasibility Study (DFS). A memo summarizing the work thus far was issued by Grinding Solutions (2024c) on December 2nd, 2024. The work performed at the time included grind calibrations on ore samples, bulk gravity pass, CIP and CIL cyanidation tests to confirm remaining testwork configuration, and mesh of grind cyanidation tests on the selected CIL configuration. For the current PFS, the results were mainly used to confirm lime and cyanide consumption values.

Based on information provided by Rupert Resources, the samples tested to date for the DFS testwork program at Grinding Solutions were taken from within the boundaries of the projected open pit area.

13.7.1. SAMPLE HEAD ASSAY

Head assay of the samples used in the testwork were back calculated from chemical analysis of bulk gravity concentrate and tailings produced during the bulk gravity pass test. The back calculated gold head grade was 2.3 g/t Au. Silver grade was less than 0.6 g/t Ag. Total carbon was 3.10%, however there were no results for organic carbon available at this time.

13.7.2. BULK GRAVITY PASS TEST

Testwork was performed on -3.35 mm crushed composite samples that were split into charges and ground for 20 minutes which produced a P_{80} size of 279.6 µm. This was determined from grind calibrations performed on the crushed samples. Figure 13-15 shows the results of the grind calibration. The samples were then passed through the Falcon gravity concentrator. The test showed a gold concentrate grade of 292.91 g/t Au with a recovery of 38.4%. The back calculated gold head grade was 2.04 g/t Au. The size-by-size gold deportment to concentrate and tails is shown in Figure 13-16.

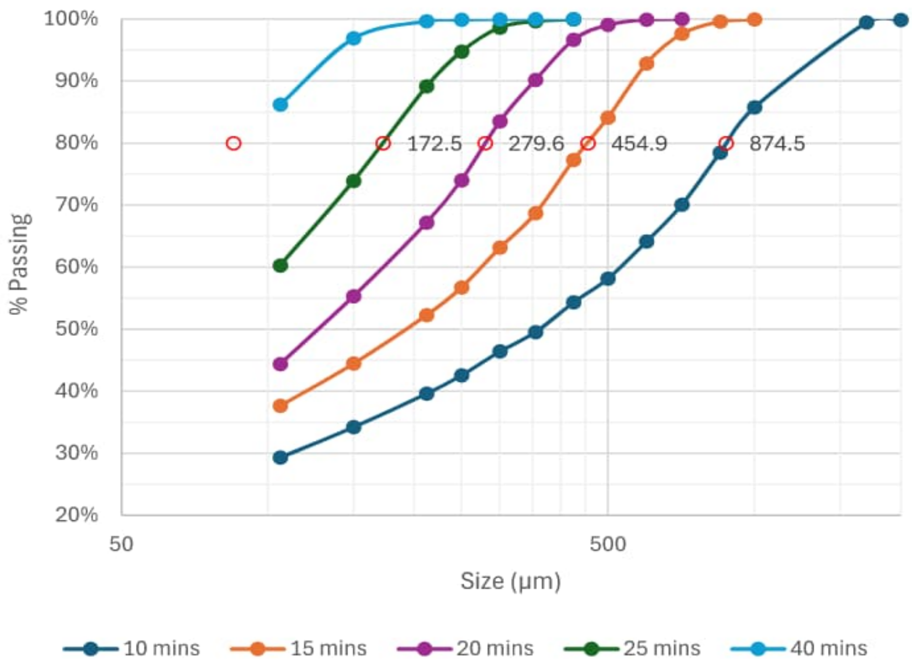


Figure 13-15 – Grind Calibration on-3.35mm Ore Sample

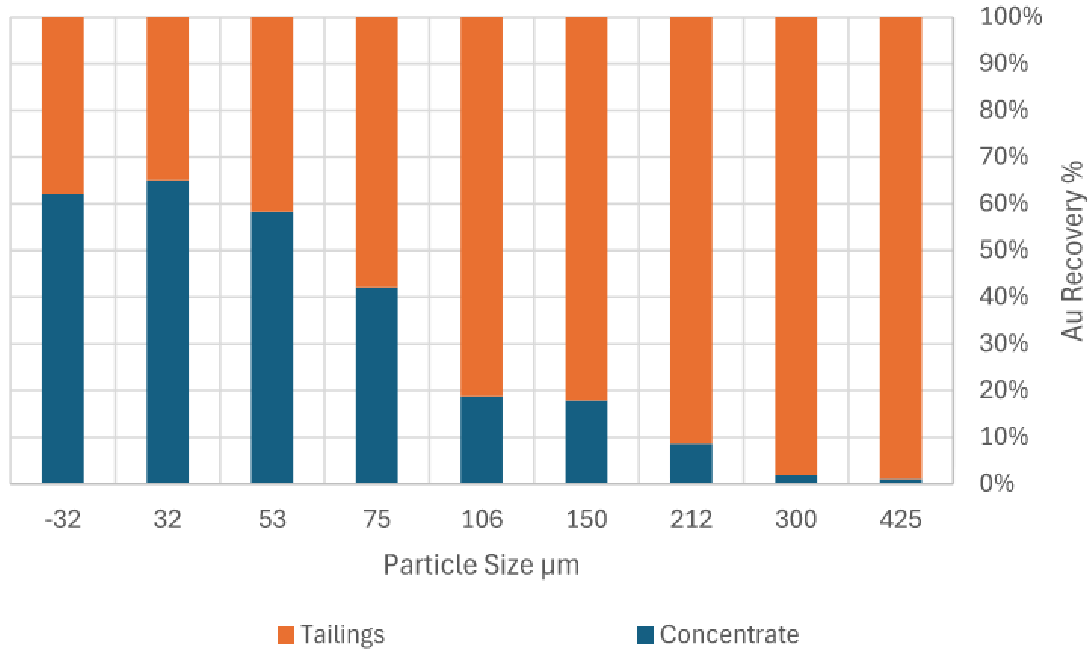


Figure 13-16 – Size-by-Size Gold Deportment to Gravity Products for Bulk Gravity Pass

13.7.3. CIL AND CIP CYANIDATION TESTS

Cyanidation tests were performed for a 48-hour leach period to compare CIL and CIP results and determine the configuration for the remaining testwork. Summary of the testwork is shown in Table 13-27. Gold recovery to carbon during the CIL tests averaged 93.9% whereas the recovery for the CIP tests averaged 93.3%. CIP leach kinetics test results showed that gold leaching continued after 24-hours. Cyanide consumptions for CIP tests averaged 0.09 kg/t NaCN. Cyanide consumption for

CIL tests averaged 0.19 kg/t NaCN which may be an indication of free cyanide adsorbing onto the activated carbon.

Table 13-27 – Summary Results of CIP and CIL Leach Tests

Operating Conditions	Extraction %			Adsorption % on Carbon			Tails ppm			Back Calc ppm			Kg/t	
Sample code	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	NaCN	$Ca(OH)_2$
1 CIP 100	91.47	2.08	20.25				0.134	0.616	14.380	1.57	0.63	18.03	0.06	0.68
2 CIP 100	94.69	2.37	21.86				0.085	0.533	14.540	1.60	0.55	18.61	0.08	0.63
3 CIP 100	93.65	3.15	21.77				0.100	0.515	14.380	1.58	0.53	18.38	0.07	0.60
4 CIP 100	93.47	2.36	22.92				0.098	0.535	14.450	1.50	0.55	18.75	0.16	0.54
1 CIL 100				95.02	2.24	21.49	0.070	0.531	15.110	1.41	0.540	19.25	0.15	0.55
2 CIL 100				94.26	2.35	22.56	0.077	0.527	15.080	1.34	0.54	19.47	0.21	0.55
3 CIL 100				92.60	2.57	22.92	0.106	0.517	14.640	1.43	0.53	18.99	0.17	0.60
4 CIL 100				93.87	2.25	23.30	0.084	0.537	14.160	1.37	0.55	18.46	0.24	0.56

13.7.4. MESH OF GRIND CYANIDATION TESTS

A range of grind sizes between 125 µm and 60 µm were subjected to CIL cyanidation tests. The tests were carried out at a cyanide concentration of 0.5 g/L, carbon concentration of 5 g/L, and slurry percent solids of 45% for a 48-hour leach period.

Gold recovery to carbon increased by approximately 4% as the grind size decreased from 125 µm to 60 µm. The gold extraction to carbon for the mesh of grind tests (MOG) are shown in Figure 13-17.

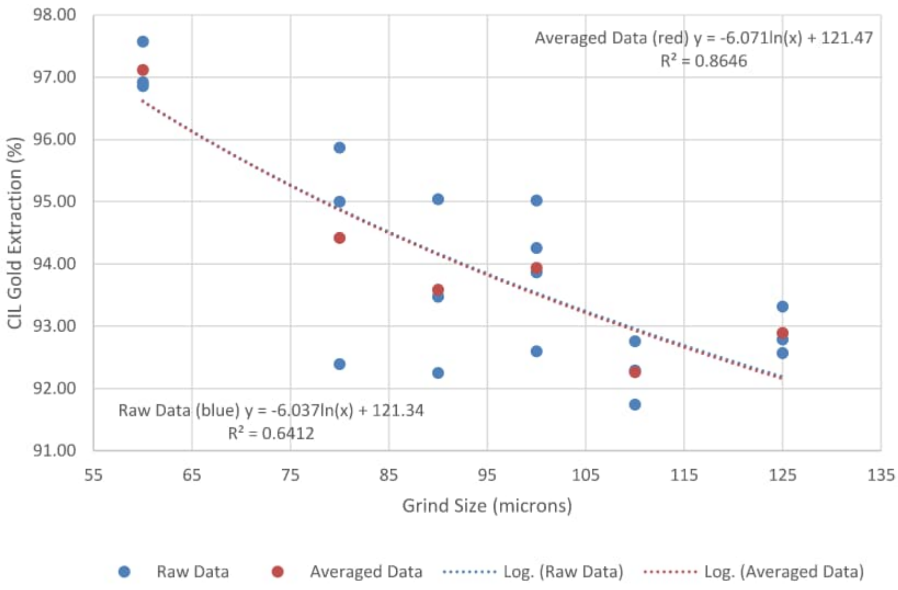


Figure 13-17 – CIL Gold Recovery to Carbon MOG Leach Tests

Cyanide and lime consumptions increased with decreasing grind size. Figure 13-18 shows that cyanide consumption increased from 0.12 kg/t to 0.19 kg/t NaCN as grind sizes decreased from 125 µm to 60 µm. A cyanide consumption of 0.15 kg/t was determined using the regression equation for a grind size of 100 µm. Figure 13-19 shows that lime consumption also increased from 0.59 kg/t $Ca(OH)_2$ to 0.74 kg/t $Ca(OH)_2$ as grind size decreased from 125 µm to 60 µm. A $Ca(OH)_2$ consumption of 0.6 kg/t was determined using the regression equation for a grind size of 100.

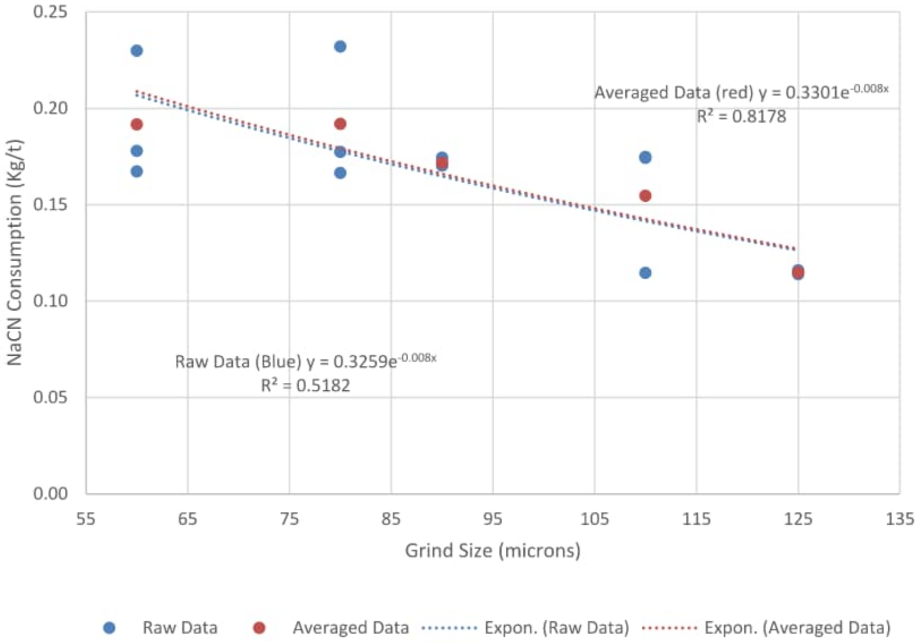


Figure 13-18 – Cyanide Consumption for MOG Leach Tests

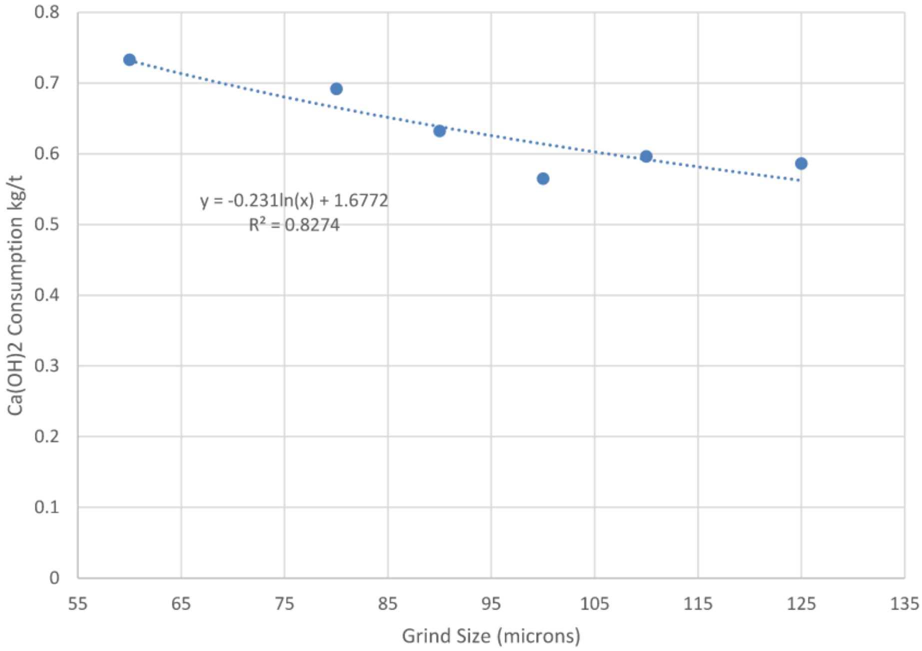


Figure 13-19 – Lime Consumption for MOG Leach Tests

14 MINERAL RESOURCE ESTIMATES

The MRE and other information in this Item are forward-looking information. The factors that could cause actual results to differ materially from the forward-looking information may include any significant differences from one or more of the following material factors or assumptions that were applied in drawing the conclusions or making the estimates, forecasts or projections set forth in this Item, including: the natural geological variability of the deposit, accuracy of assay database, the assumptions used by the QP to prepare the data for resource estimation, the interpretation of the controlling structural environment and mineral domain models, the selection of grade interpolation method, sample search and estimation parameters used for grade interpolation, treatment of high-grade outlier sample data, continuity of mineralisation and factors used to determine reasonable prospects for economic extraction.

14.1 INTRODUCTION

The Mineral resources have been estimated and reported in accordance with NI 43-101 following the requirements of Form 43-101F1 by Mr. Brian Thomas, P.Geo., an independent QP, as defined under NI 43-101 and an employee of WSP Canada Inc. based in Sudbury, Ontario, Canada. The methodology used to determine the MRE is consistent with the CIM Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines (November 2019) and was classified following CIM Definition Standards for Mineral Resources & Mineral Reserves (May 2014).

This chapter is based on the latest mineral resource report. NI 43-101 Technical Report Rupert Resources Ltd. Updated Mineral Resource Estimate for the Ikkari project – Finland, effective date 12 December 2023. The updated Ikkari MRE was publicly disclosed on November 28, 2023, in the news release titled "Rupert Resources Reports Updated MRE for Ikkari of Over Four Million Ounces Gold in Indicated Category and Provides Details of Winter 2023/2024 Drilling Targets"

The MRE outlined in the following sections, was based on geological models and drill hole data provided by Rupert Resources and was estimated using a 3D block modelling approach, based on Ordinary Kriging (OK), in Datamine Studio RM (Datamine) software.

14.2 DRILL HOLE DATA

The MRE is based upon data provided from recent surface diamond drilling, completed by Rupert Resources between 2020 and 2023. The final drill hole database consisted of 255 drill holes, totalling approximately 111 896 m of core, 103 839 gold assays and 11 436 SG measurements and was made available for modelling on August 28, 2023.

Table 14-1 – Overview of Ikkari Drilling database.

Year	Nº Drill Holes	Length (m)
2020	62	20 320
2021	75	36 049
2022	78	34 085
2023	40	21 442
Total	**255**	**111 896**

For the purposes of modelling, Rupert Resources provided only drill holes located within the Ikkari project area so no sub-domaining of the data was required prior to MRE.

The database was analysed for interval errors and out of range values and was reviewed in 3D space to validate the hole locations and de-surveyed hole traces with no significant issues identified. Further to this, the database was validated for potential errors with the collar locations, downhole surveys, assay and density entries, core recovery and logged structural data. The QP concluded that the drill hole database was robust in its construction and suitable for use in MRE as described in Chapter 12.

The Rupert Resources drill hole data is supported by a QA/QC process as described previously in Chapter 11. The QP has also completed independent sample verification and check logging as summarized in Chapter 12 and has not identified any material flaws in the drill hole data or data collection procedures. Rupert Resources' data collection procedures were found to be consistent with industry practice. All drilling is recent and has been completed by Rupert Resources and there is no historical (legacy) data.

14.3 GEOLOGICAL DOMAINING

Rupert Resources modelled the deposit lithology, bedrock and topographic surfaces as well as three mineralized domains consisting of the Northern Felsic, Contact, and Internal Siderite domains, as outlined in Figure 14-2 and Figure 14-3. The domain models were based on a combination of lithology and mineralisation generally above an approximate 0.3 g/t cut-off. The domain models were constructed by Rupert Resources using Leapfrog Geo software and reviewed by the QP relative to the drill hole data.

Mineralisation domains were constrained by the bedrock-overburden contact and to the north by the contact with the black shale lithology unit, consistent with the grade distribution at this contact (Figure 14-1). Continuity of grade at the Northern Felsic - Ultramafic contact was also investigated (Figure 14-1). With no significant change in the grade distribution at the contact this was not treated as a boundary for the purpose of mineral domaining.

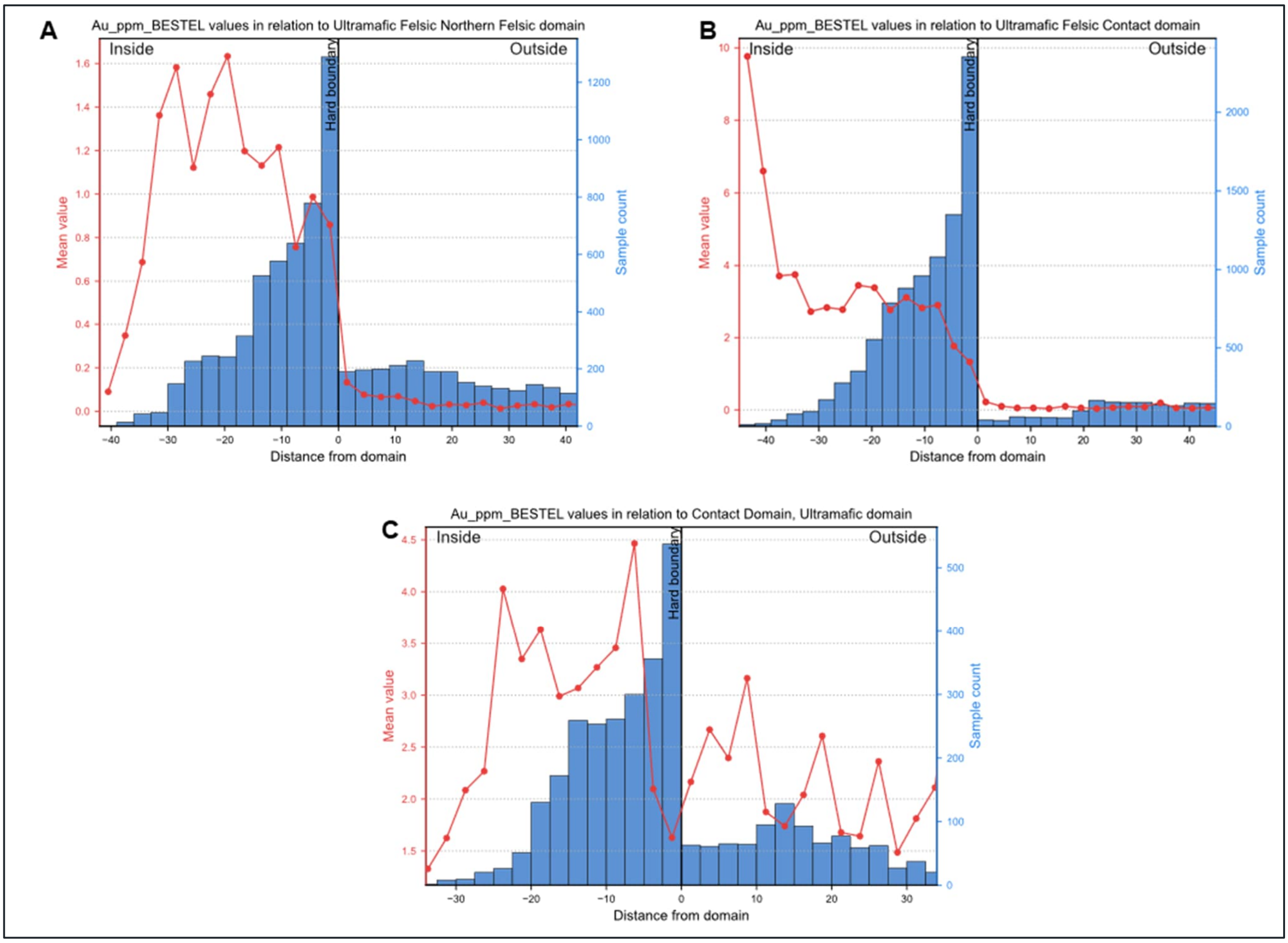


Figure 14-1 – Contact plots of lithology domains

Small discontinuous lenses were removed and the models were verified for common issues such as duplicate vertices, duplicate faces, open edges and crossovers. Model volumes were queried and summarized in Table 14-2.

The QP notes that there is mineralisation remaining outside of the modelled domains that could not be interpreted into continuous mineral domain volumes with any high degree of confidence. This mineralisation was estimated within the surrounding background material. The background material was broken into two separate areas including the North, representative of the hangingwall black shale and gabbro units north of the Northern Felsic domain, and the South, representative of the footwall rocks that host the main mineralisation. No bounding structures were identified that either offset or cut off mineralisation.

It's the QP's opinion that the mineral domain models are representative of the current drill hole data observed for the Ikkari deposit and are suitable for use in the determination of this MRE.

The QP notes that there are risks associated with any geological interpretation and that it is subject to change with new data and geological understanding over time.

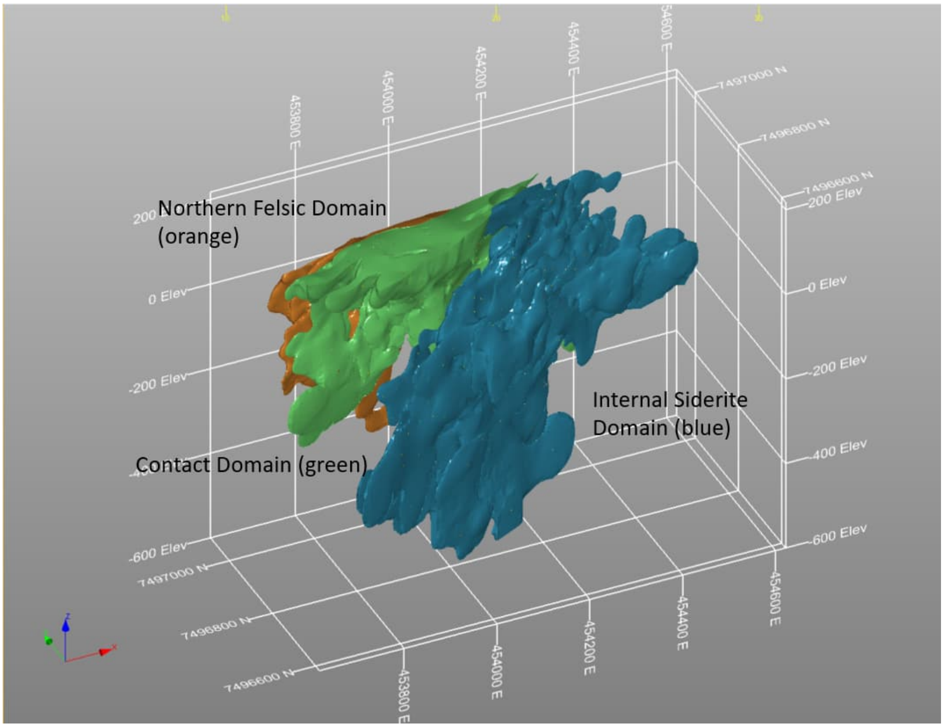


Figure 14-2 – Ikkari Mineral Domains (Oblique View Facing Northeast)

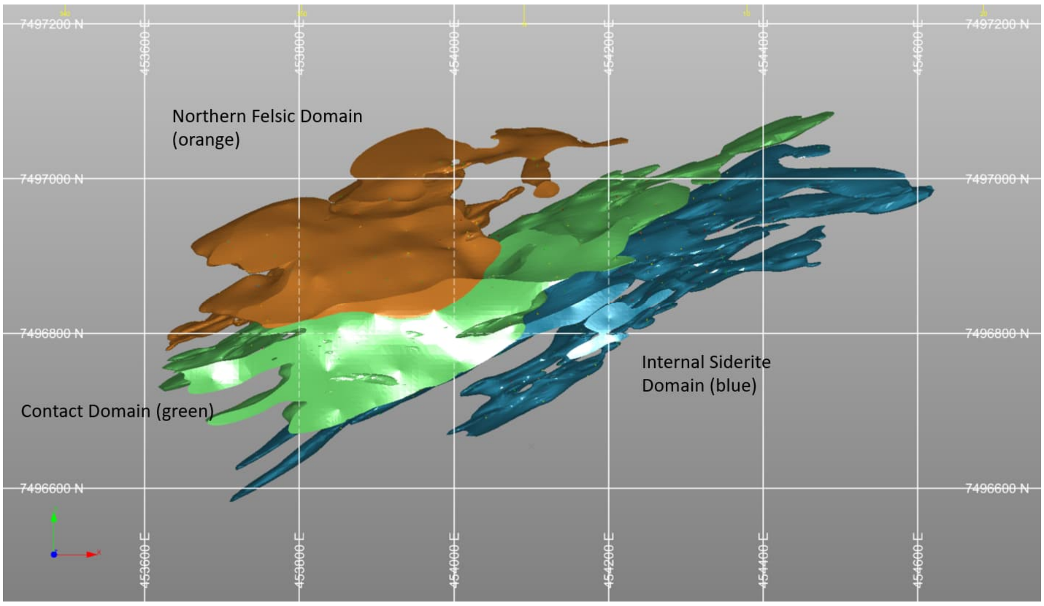


Figure 14-3 – Ikkari Mineral Domains (plan view)

Table 14-2 – Mineral Domain Volumes

Mineral Domain	Volume (m^3)
Northern Felsic	7 339 035
Contact	9 977 062

Mineral Domain	Volume (m^3)
Internal Siderite	14 617 334

14.4 EXPLORATORY DATA ANALYSIS

Exploratory data analysis was conducted on the Ikkari gold and density data within each mineral domain in order to understand the grade distribution, validate the data for out-of-range values, assess sample lengths and identify high-grade outlier values in order to inform decisions relative to the estimation methodology such as interpolation method, sample composite length and outlier control strategies.

14.4.1. DESCRIPTIVE STATISTICS

Table 14-3 summarizes the descriptive statistics by domain for raw sample grades, capped sample grades and capped composite grades. Comparison of the mean capped sample grades to the composite grades confirmed there were no changes to the mean grades during compositing. All grade domain populations were found to be positively skewed with some high-grade outlier values.

Table 14-3 – Comparison of Au Sample Statistics

Domain	Sample type	# of Samples	Min (g/t)	Max (g/t)	Mean (g/t)	Variance	Std. Deviation	CV
Contact	Raw	8 590	0.00	465.30	2.54	44.08	6.64	2.62
	Capped	8 590	0.00	30.00	2.41	18.72	4.33	1.79
	Composite	3 435	0.01	29.12	2.40	11.26	3.35	1.40
Internal siderite	Raw	9 668	0.00	165.00	1.76	24.51	4.95	2.82
	Capped	9 668	0.00	50.00	1.72	17.27	4.16	2.42
	Composite	3 809	0.01	35.11	1.72	8.52	2.92	1.70
Northern felsic	Raw	4 980	0.01	55.04	1.09	4.78	2.19	2.01
	Capped	4 980	0.01	30.00	1.09	4.37	2.09	1.93
	Composite	1 989	0.01	15.32	1.09	2.04	1.43	1.32
Background North	Raw	15 384	0.00	65.86	0.04	0.41	0.64	15.27
	Capped	15 384	0.00	5.50	0.03	0.04	0.20	5.76
	Composite	6 272	0.00	4.36	0.03	0.02	0.14	4.11
Background South	Raw	65 279	0.00	438.00	0.06	3.52	1.88	29.45
	Capped	65 279	0.00	15.00	0.05	0.12	0.34	6.43
	Composite	26 167	0.00	12.00	0.05	0.06	0.24	4.47

14.4.2. COMPOSITING

A composite length of 2.5 m was chosen based on the block model Selective Mining Unit (SMU) dimensions of 10 x 5 x 5 m. All raw sample intervals were composited to a mean length of 2.5 m. As the composite length was variable, the composite lengths ranged between a minimum of 1 m to a maximum of 3 m. The global mean Au grades and total sample lengths were compared to ensure that no significant number of samples were lost during the compositing process.

14.4.3. OUTLIER ANALYSIS

The raw samples within each mineral domain were assessed for high-grade outlier values based on XY scatterplots, cumulative probability plots, box plots, review of the raw ranked Au grades and descriptive statistics including Coefficient of Variation (CV). The CV values were observed to be relatively low in the 3 mineral domains and higher in the background domains due to their unconstrained nature. Lower CV values are considered an indicator that outlier grades will have less potential to bias block estimates. Based on these assessments top-cut values were used to cap (limit outlier grades to the top-cut value) the Au grades of the raw samples as summarized in Table 14-4.

Table 14-4 – Summary of Outlier Controls

Domain	Top-Cut (g/t)	Number of Samples
Northern Felsic	30	1
Contact	30	16
Internal Siderite	50	11
Background North	5.5	10
Background South	15	11

Due to the unconstrained nature of the background domains, an additional distance restriction constraint of 10 m along strike, 10 m down dip and 5 m across strike was applied to all samples greater than 3 g/t in order to prevent excessive grade spreading in the block model estimates.

14.4.4. SPECIFIC GRAVITY

Specific Gravity (SG) measurements were taken by Rupert Resources from 10 – 15 cm core samples using the weight in air versus the weight in water method (Archimedes) based on the following formula:

$$SG = \frac{Sample\ Weight\ in\ Air}{Sample\ Weight\ in\ Air - Sample\ Weight\ in\ Water}$$

A full description of the SG measurement process is outlined in Chapter 11. SG measurements were assessed for out-of-range values with measurements less than 2.5 being discarded and measurements greater than 3.5 being capped at 3.5. Table 14-5 summarizes the SG data used in the block model estimate by domain.

Table 14-5 – Summary of SG Data by Domain.

Domain	Sample type	# of Samples	Min (g/cm^3)	Max (g/cm^3)	Mean (g/cm^3)	Variance	Std. Deviation	CV
Contact	Raw	925	2.06	6.82	2.88	0.05	0.21	0.07
	Capped	925	2.50	3.50	2.87	0.03	0.16	0.06
Internal siderite	Raw	1 164	2.18	4.44	2.95	0.02	0.15	0.05
	Capped	1 164	2.50	3.50	2.95	0.02	0.14	0.05
Northern felsic	Raw	422	2.55	3.83	2.76	0.01	0.09	0.03
	Capped	422	2.55	3.50	2.76	0.01	0.07	0.03
Background North	Raw	1 476	2.01	3.89	2.80	0.03	0.16	0.06
	Capped	1 476	2.50	3.50	2.81	0.02	0.14	0.05
Background South	Raw	7 419	1.79	5.69	2.87	0.01	0.12	0.04
	Capped	7 419	2.50	3.50	2.87	0.01	0.11	0.04

14.5 BLOCK MODEL AND MINERAL RESOURCE ESTIMATION

14.5.1. ASSESSMENT OF SPATIAL GRADE CONTINUITY

Spatial continuity of Au grade was assessed using a combination of variogram maps and directional variograms. This analysis provided input on the orientations and interpreted distance of grade continuity in each of the mineral domains. The variogram analysis was found to be consistent with geological orientations observed in the deposit and those modeled by Rupert Resources in the mineral domain models. This analysis was used as the basis for the search ellipse distances defined in the sample selection strategy as summarized in Section 14.5.4 and used for the purpose of assigning Kriging weights to the composite samples for grade estimation using OK.Table 14-6 summarizes the variogram model parameters for the two-structured spherical models. Variogram ellipses were generated for validation purposes and compared in 3D against the composite data to confirm reasonable alignment with observed grade trends.

Table 14-6 – Variogram Model Parameters

Domain	VANGLE1	VANGLE2	VANGLE3	VAXIS1	VAXIS2	VAXIS3	NUGGET	ST1	ST1PAR1	ST1PAR2	ST1PAR3	ST1PAR4	ST2	ST2PAR1	ST2PAR2	ST2PAR3	STA2PAR4	TOTAL SILL
Contact	-25.00	10.00	77.70	3.00	2.00	1.00	0.298	1.00	14.20	7.10	9.20	0.34	1.00	80.10	60.30	29.90	0.10	0.74
Internal siderite	-27.34	14.77	79.66	3.00	2.00	1.00	0.297	1.00	32.60	6.20	23.30	0.49	1.00	80.00	59.80	38.80	0.08	0.87
Northern felsic	-25.00	10.00	77.70	3.00	2.00	1.00	0.298	1.00	14.20	7.10	9.20	0.34	1.00	80.10	60.30	29.90	0.10	0.74
Background North	-10.00	0.00	55.00	3.00	2.00	1.00	0.145	1.00	13.40	3.10	24.60	0.20	1.00	100.00	59.90	39.60	0.10	0.45
Background South	-25.00	15.00	73.60	3.00	2.00	1.00	0.253	1.00	9.20	16.90	15.60	0.17	1.00	100.20	79.90	40.10	0.13	0.55

14.5.2. BLOCK MODEL DEFINITION

The Ikkari block model specifications are summarized in Table 14-7. Block shape and sizes are typically a function of the geometry of the deposit, density of sample data, and expected SMU. On this basis, a -25 degree rotation was applied to orient the X-axis along the strike direction of mineralisation and a parent block size of 10 m (X-axis along strike) by 5 m (Y-axis across strike) by 5 m (Z-axis down-dip) was chosen to represent the selective mining unit considered for the base case open pit and underground longhole mining scenarios.

Table 14-7 – Block Model Definition

Direction	Minimum	Maximum	Block Size (m)	No. Blocks
Easting	453 700	455 040	10	134
Northing	7 496 200	7 497 100	5	180
Elevation	- 620	280	5	180

The final model was later expanded and moved to a new prototype for mine planning purposes with the expanded model specifications outlined on Table 14-8. The new origin positions were calculated to align with blocks in the original model prototype.

Table 14-8 – Expanded Block Model Definition

Direction	Minimum	Maximum	Block Size (m)	No. Blocks
Easting	453 603.26209	455 343.26209	10	174
Northing	7 495 934.21470	7 497 234.21470	5	260
Elevation	- 620.0	280.0	5	180

The mineral domain envelopes were filled with regularized blocks (no block splitting used) and block volumes were then compared to the mineral domain wireframe volumes to confirm there were no significant volume discrepancies. Block volumes for all zones were found to be within reasonable tolerance limits of the mineral domain volumes.

14.5.3. INTERPOLATION METHODS

OK was the final Au and SG interpolation method chosen as the basis of the Ikkari MRE. This method assigns estimation weights to the samples within the search volume relative to the distance and direction of the sample data from the centre of each block. Samples located closest to the block centroid in directions of preferred grade continuity receive higher estimation weights as defined by the modeled variogram parameters.

Inverse Distance squared (ID^2) and Nearest Neighbour (NN) interpolation methods were also used for global comparison and validation purposes but were not used for final resource reporting.

14.5.4. SAMPLE SELECTION STRATEGY

A 3 pass, elliptical search strategy was used to interpolate block grades with the first pass search distances based on half the variogram range and representing the areas having the highest drill hole density. The second pass was based on the full variogram range, and the third pass was twice the variogram range. Sample selection criteria for Au were calibrated based on a change of support smoothing ratio evaluation for each mineral domain and therefore have minor differences as summarized in Table 14-9.

Table 14-9 – Sample Selection Criteria Used for Au Grade Estimation

Domain	Pass	Along Strike (m)	Down Dip (m)	Across Strike (m)	Min No. of Samples	Max No. of samples	Max No. Samples per Hole	Min. No. of Holes
Contact	Pass 1	40	30	10	8	18	3	3
	Pass 2	80	60	20	8	18	3	3
	Pass 3	160	120	40	3	8	3	1
Internal Siderite	Pass 1	40	30	10	5	8	3	2
	Pass 2	80	60	20	5	8	3	2
	Pass 3	160	120	40	3	8	3	1
Northern Felsic	Pass 1	40	30	10	6	12	3	2
	Pass 2	80	60	20	6	12	3	2
	Pass 3	160	120	40	3	8	3	1
Background North and South	Pass 1	100	80	7.50	5	8	3	2
	Pass 2	150	120	11.25	5	8	3	2
	Pass 3	200	160	15	3	8	3	1

Smoothing ratios are based on the ratio between the theoretical expected model variance, and the actual OK model variance. The theoretical variance is calculated based on the declustered sample variance, the variogram model, block size, and F-Function.

A smoothing ratio of 1 represents the ideal scenario where the expected variance equals the model variance, and ratios between 0.8 to 1.2 are considered to be within acceptable tolerances that would not require any corrective actions. Ratios less than 0.8 are considered “under-smoothed” (lower tonnes and higher grade) and over 1.2 are considered “over smoothed” (higher tonnes and lower grade). Smoothing ratios generally greater than 2 need to be reviewed for any potential issues such as biased drill hole support and could require corrective actions as the proportion of tonnes and grade above the selective mining cut-off may not be representative of what can be achieved during mining. Corrective actions would include options such as adjusting various estimation parameters or conducting a variance correction on the model. Table 14-10 summarizes the smoothing ratios assessed for each domain relative to the selected search strategy.

Table 14-10 – Summary of Smoothing Ratios by Domain

Domain	OK Smoothing Ratio
Contact	0.88
Internal Siderite	0.97
Northern Felsic	1.00
Background North	0.87
Background South	1.15

Dynamic Anisotropy was used to account for minor variations in deposit orientation. Dynamic Anisotropy is a Datamine process used to adjust search orientations based on the shape of a controlling surface, which in this case was a centre line surface through the middle of each mineral domain. Search orientations, defined by dip and dip direction, were estimated into the blocks based on the trends implicit to the mineral domain envelopes which were used to control the search ellipse orientation for each block during estimation.

14.5.5. MODEL VALIDATION

The block model validation process included visual comparisons between block estimates and composite grades in plan, section, and long section along with a global comparison of mean grades and swath plots. Block estimates were visually compared to the drill hole composite data to check agreement.

Figure 14-4and Figure 14-5 provide comparisons of the composite samples and block model Au estimates in plan and cross-section views. No material grade bias issues were identified, and the block grades compared well to the composite data.

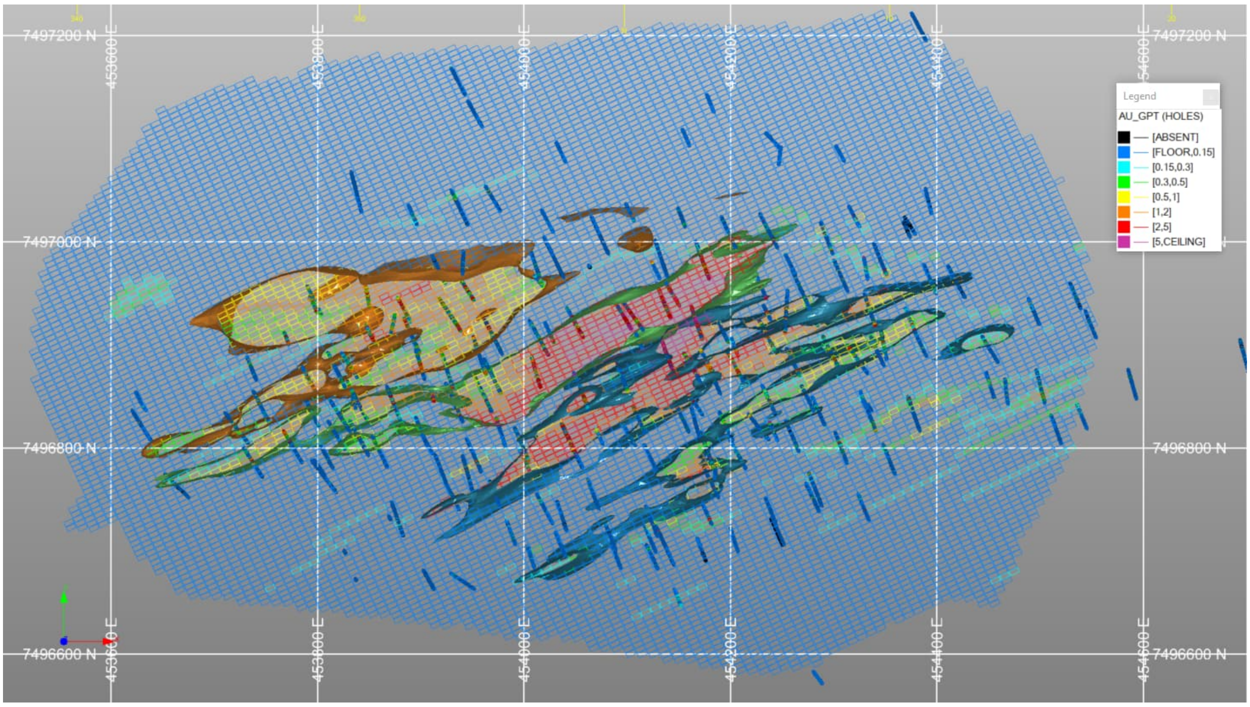


Figure 14-4 – Plan View Comparison of Block Grades vs Composite Grades (0 Elevation)

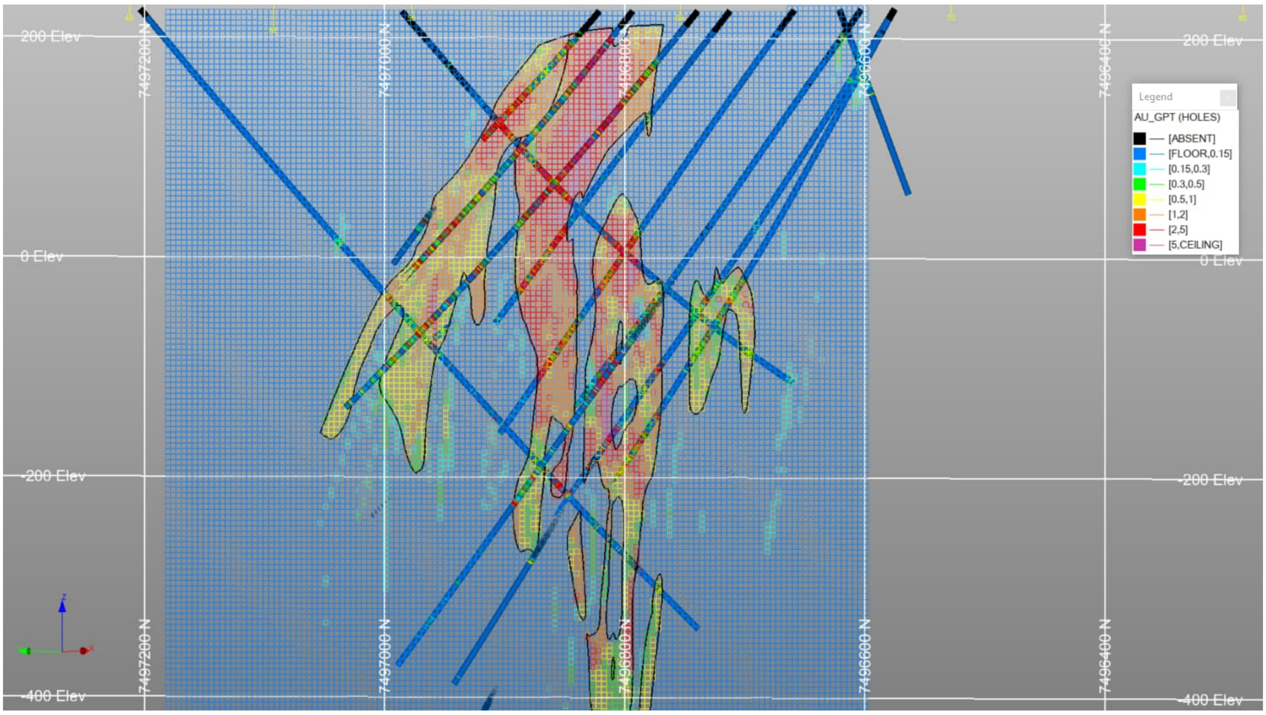


Figure 14-5 – Example Cross-Section Comparison of Block Grades vs Composite Grades (Facing North-East)

Global statistical comparisons between the composite samples, NN estimates, ID^2 estimates and the final OK estimates were compared to assess for global bias, where the NN model estimates represent de-clustered composite data based on a 5 m composite length. A longer composite length was used for the NN model to be more representative of the block size so that more samples would be used and fewer samples excluded from the global estimate since NN is based on only the nearest sample. Clustering of the drill hole data can result in differences between the global means of the composites and NN estimates. Comparison of the global NN grades were found to compare favourably with the global OK estimates indicating that no material global bias was observed in the model. The results of the global bias assessment are summarized in Table 14-11.

Table 14-11 – Statistical Comparison of Global Mean Au Grades

Domain	Composite Mean (g/t)	Global NN Mean (g/t)	Global ID^2 Mean (g/t)	Global OK Mean (g/t)	NN-ID Relative Difference (%)	NN-OK Relative Difference (%)
Contact	2.40	2.33	2.32	2.29	-0.31	-1.66
Internal Siderite	1.72	1.45	1.43	1.42	-1.98	-2.46
Northern Felsic	1.08	1.03	1.03	1.04	-0.03	0.75
Background North	0.03	0.03	0.03	0.03	-0.72	-0.58
Background South	0.05	0.05	0.05	0.05	-0.53	-0.04

Notes: The comparison is for all blocks (global) in the model irrespective of classification. Relative difference calculated between OK mean and NN mean Au grades.

Swath plots of Au grades were generated from 40 m swaths for the 3 main mineral domains throughout the final model to evaluate for local grade bias issues. Figure 14-6, Figure 14-7 and Figure 14-8 highlight the grade comparisons for each interpolation method in each axis direction. The swath plots compare the OK, and ID^2 model grades to the NN model grades (de-clustered composite grades) in order to identify potential local grade bias in the model. Review of all the swath plots did not identify any significant bias in the model that is material to the MRE as there was general agreement between the de-clustered composites (NN model) and the final model grades with minor variances noted around the margins of the deposit.

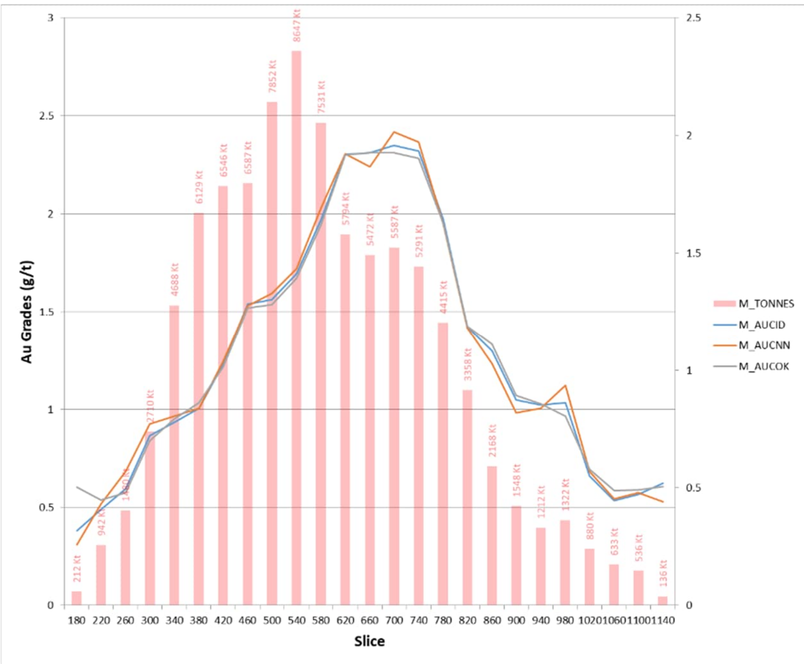


Figure 14-6 – Swath Plot of the Ikkari Block Model (X-Axis)

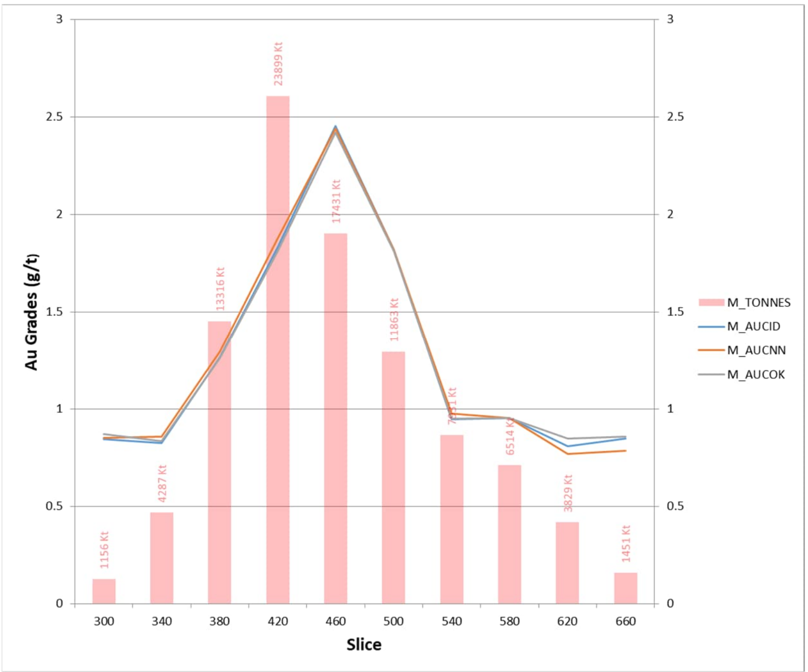


Figure 14-7 – Swath Plot of the Ikkari Block Model (Y-Axis)

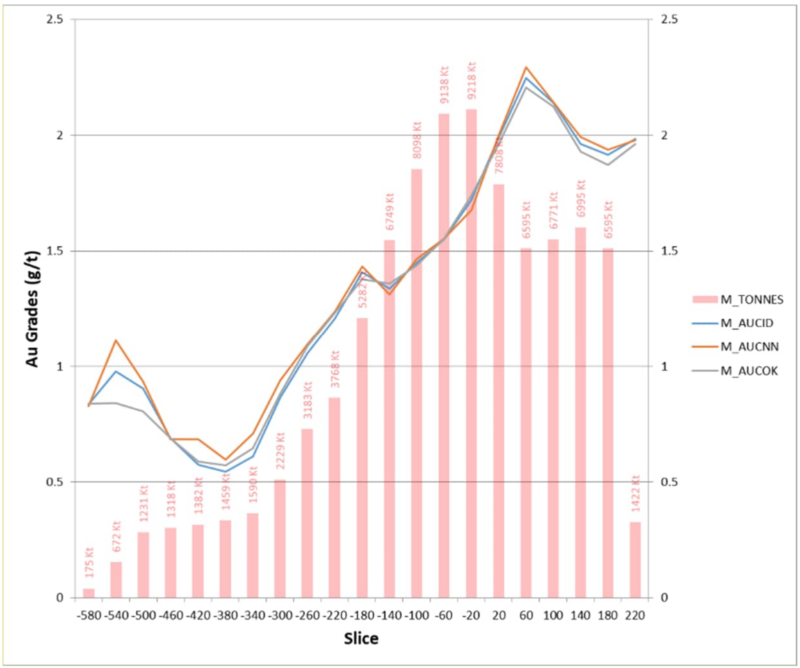


Figure 14-8 – Swath Plot of the Ikkari Block Model (Z-Axis)

14.5.6. RESOURCE CLASSIFICATION

The MRE was classified following the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). Resource classifications were assigned to broad regions of the block model based on QP confidence and judgement related to drill hole spacing, geological understanding, continuity of mineralisation in conjunction with data quality and block model representativeness. Indicated Mineral Resources were defined at an approximate 40 m drill spacing or less and estimated within the first or second pass and confined to the mineral domain models (i.e., background domains excluded).

Inferred Mineral Resources were defined between 40 m and 80-m drill spacing. Final Inferred Mineral Resources for the UG MRE were confined to within the mineral domain model volumes as the mineralized material outside of the domain models, in the background domains, did not demonstrate adequate continuity to support RPEEE.

Measured Mineral Resources were not defined due to insufficient drill spacing relative to the deposit type.

Figure 14-9 and Figure 14-10 outline the locations of Indicated and Inferred Mineral resources in the Ikkari Deposit.

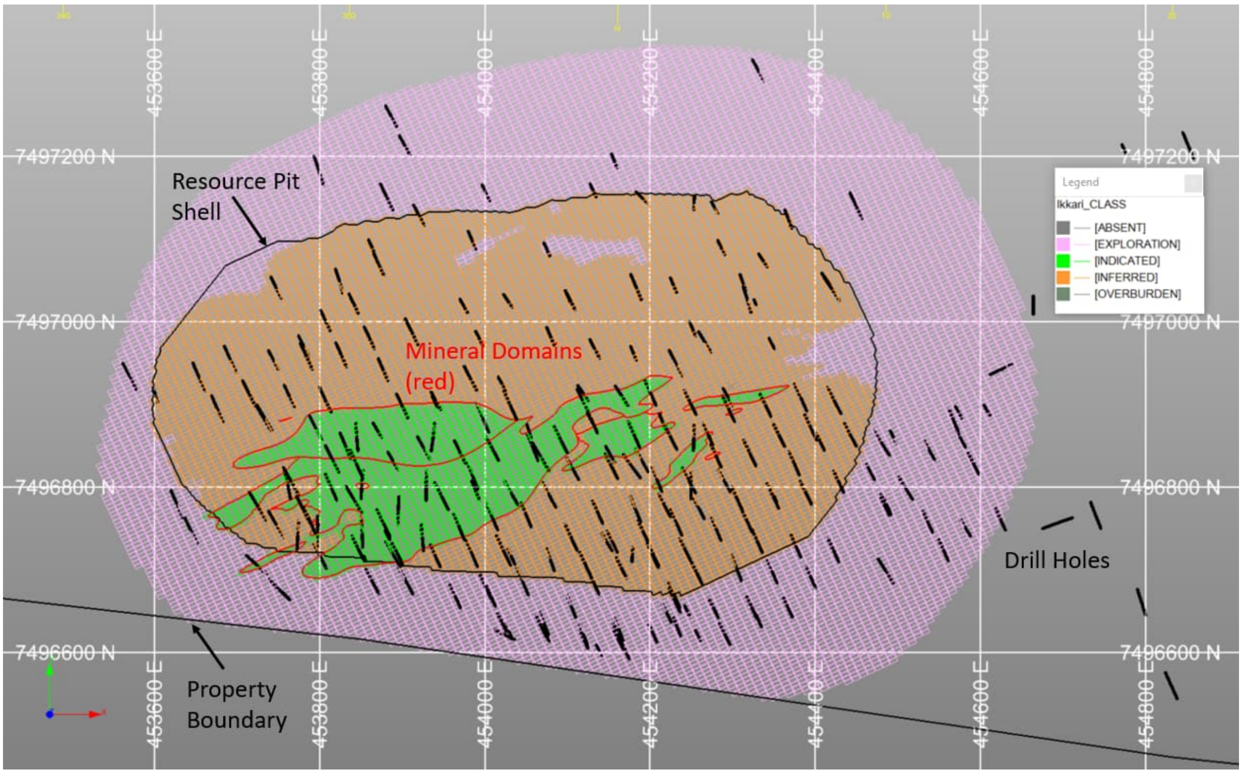


Figure 14-9 – Ikkari Mineral Resource Classification in the Open Pit Area (Plan View, 150 m Elev.)

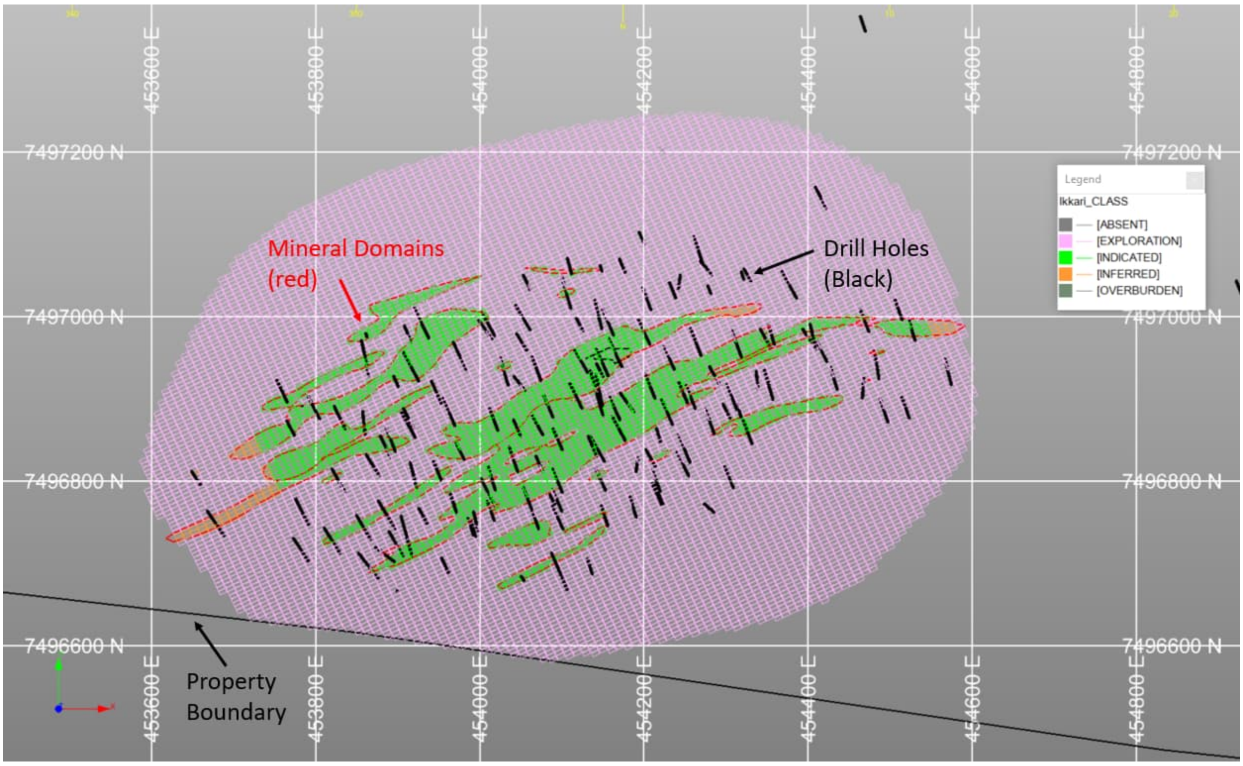


Figure 14-10 – Ikkari Mineral Resource Classification in the Underground Area (Plan View, -100 m Elev.)

14.5.7. REASONABLE PROSPECTS FOR EVENTUAL ECONOMIC EXTRACTION (RPEEE)

Open Pit

Mineral Resources were reported above a 0.4 g/t break-even cut-off grade and constrained within a Whittle resource pit shell based on a revenue factor of 0.95. A 26 m buffer between the south edge of the resource pit shell and the license boundary was imposed. Pit slope angles were determined for five different pit sectors based on SRK's Geotechnical report on Ikkari dated June 2023 (SRK, 2023b). The pit shell and cut-off grade determination were supported by the following economic assumptions (costs stated in $US dollars):

- Gold Price: $1 700 / oz;
- Metallurgical Gold Recovery: 95%;
- Mining dilution 5%;
- Mining recovery 95%;
- Open pit ore mining cost: $2.9/t;
- Open pit waste mining cost $2.2/t;
- Additional haulage cost of $0.05/t/10-meter bench height;
- Processing Cost: $11.30/t;
- G&A, Rehabilitation & Closure: $4.8/t;
- Royalty (state and landowner combined): 0.75%;
- Gold payable 99.92%;
- Treatment charge $2.5/oz; and
- Pit slope angles from 3rd party geotechnical report prepared for Rupert Resources dated June 2023:
  - North pit sector 50.8°;
  - East pit sector 44.8°;
  - South-Southeast pit sector 43.2°;
  - Southwest pit sector 44.9°; and
  - West pit sector 49.0°.

Figure 14-11 and Figure 14-12outline mineral resource blocks greater than 0.4 g/t within the pit shell (blue). The QP notes that mining dilution and recovery factors were used for the purpose of generating the resource pit shell only and were not applied to the MRE.

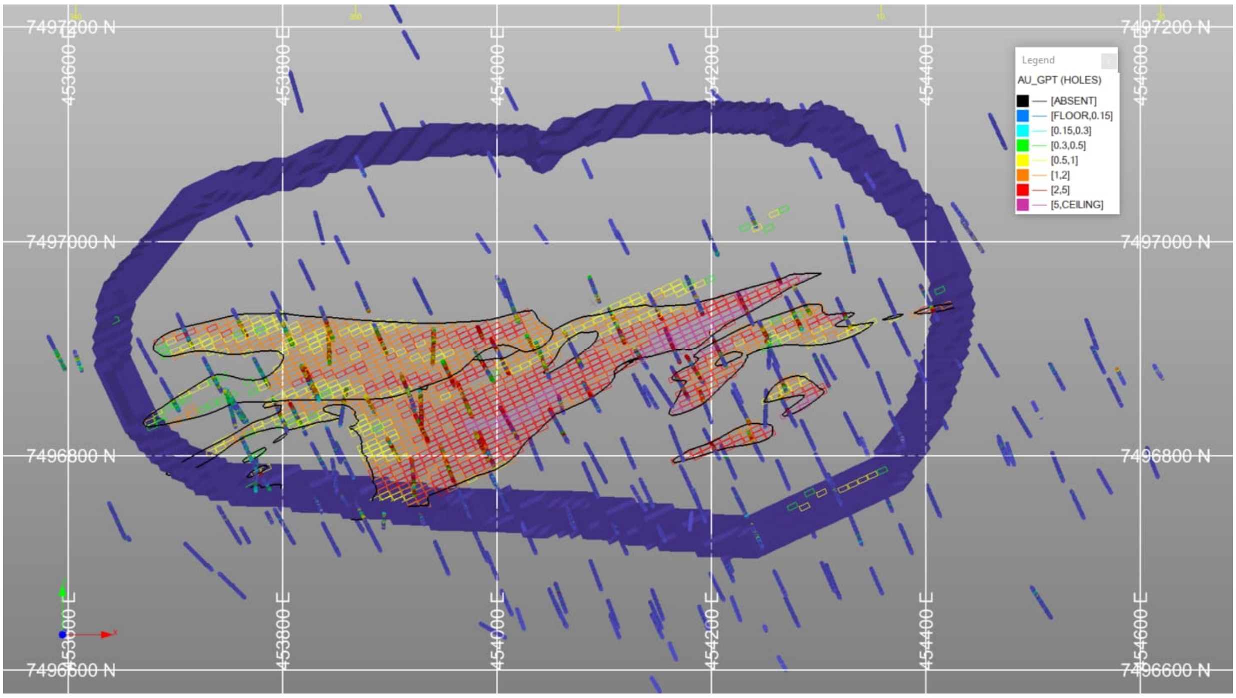


Figure 14-11 – Plan View of the Resource Pit Shell (100-m Elevation)

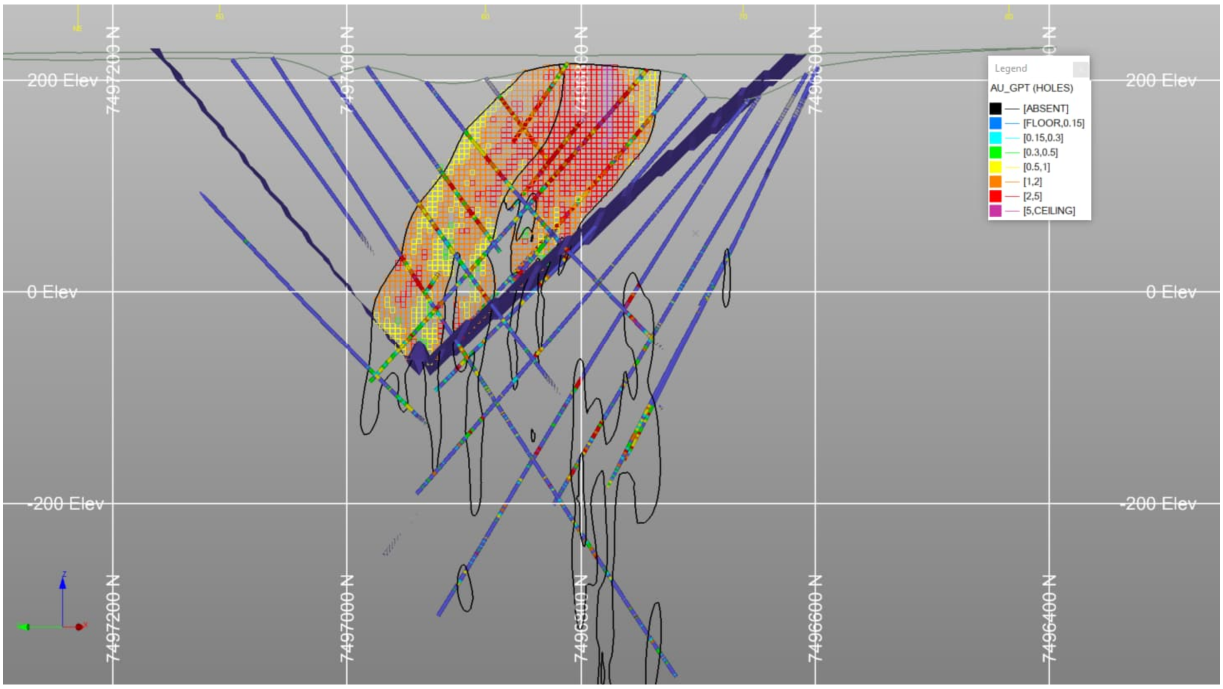


Figure 14-12 – Example Cross-Section View of Resource Blocks within the Pit Shell (Facing North-East)

Underground

The UG MRE was reported outside and below the pit shell at a 0.9 g/t UG break-even cut-off grade representing bulk scale Longhole Open Stope mining. Resource blocks were evaluated for reasonable mining continuity and a decision was made to constrain the UG resource to within the

3 mineral domain models. Blocks above cut-off outside of the mineral domains did not demonstrate reasonable mining continuity and therefore were excluded from the MRE.

The calculation of the UG mining cut-off is supported by the following economic assumptions (costs stated in $US dollars).

- Gold Price: $1 700 / oz;
- Metallurgical Gold Recovery: 95%;
- Underground mining cost: $29/t;
- Processing Cost: $11.30/t;
- G&A, Rehabilitation & Closure: $4.8/t; and
- Royalty (state and landowner combined): 0.75%.

Further UG constraining envelopes were evaluated but they did not account for some changes in orientation and no material difference was observed from reporting inside the mineral domains, therefore underground constraining grade envelopes were not used for reporting the MRE. Blocks above the cut-off grade, constrained within the mineral domain models, have been reviewed for continuity. In the opinion of the QP, they are of sufficient spatial continuity that these blocks meet the RPEEE test for underground mining. Figure 14-13 provides an example cross-section view of UG resource blocks.

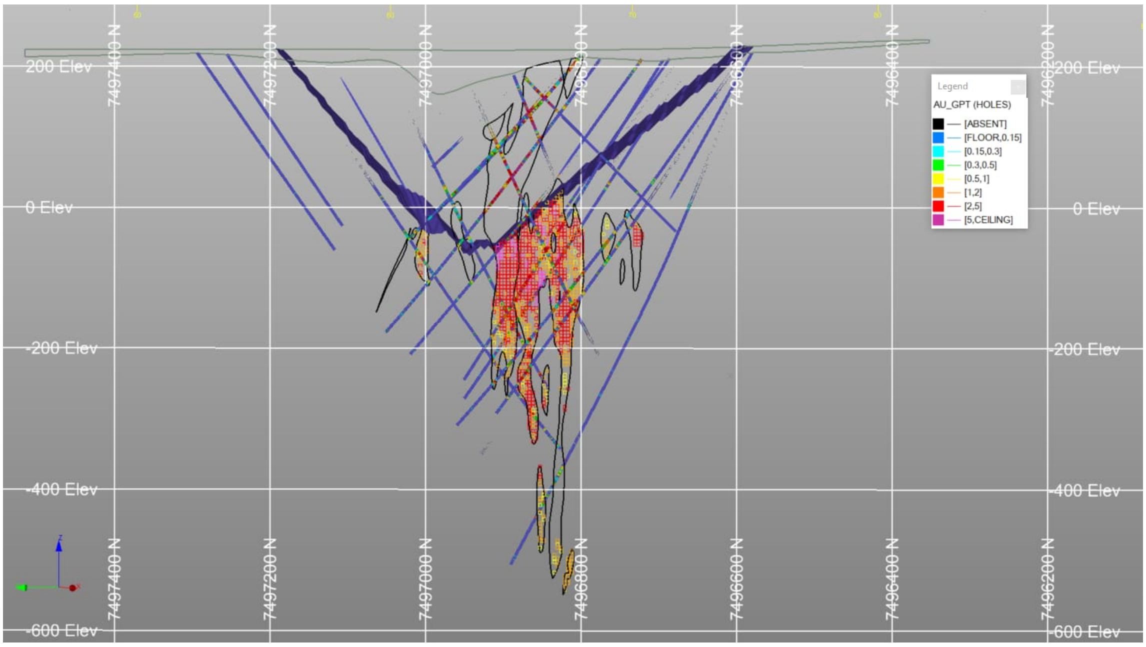


Figure 14-13 – Example Cross-Section of Resource Blocks Constrained to within the Mineral Domains

Combined OP and UG Mineral Resource Blocks

Figure 14-14 illustrates the OP and UG resource blocks combined above their respective cut-off grades as described in the previous sections (resource pit shell shown in blue).

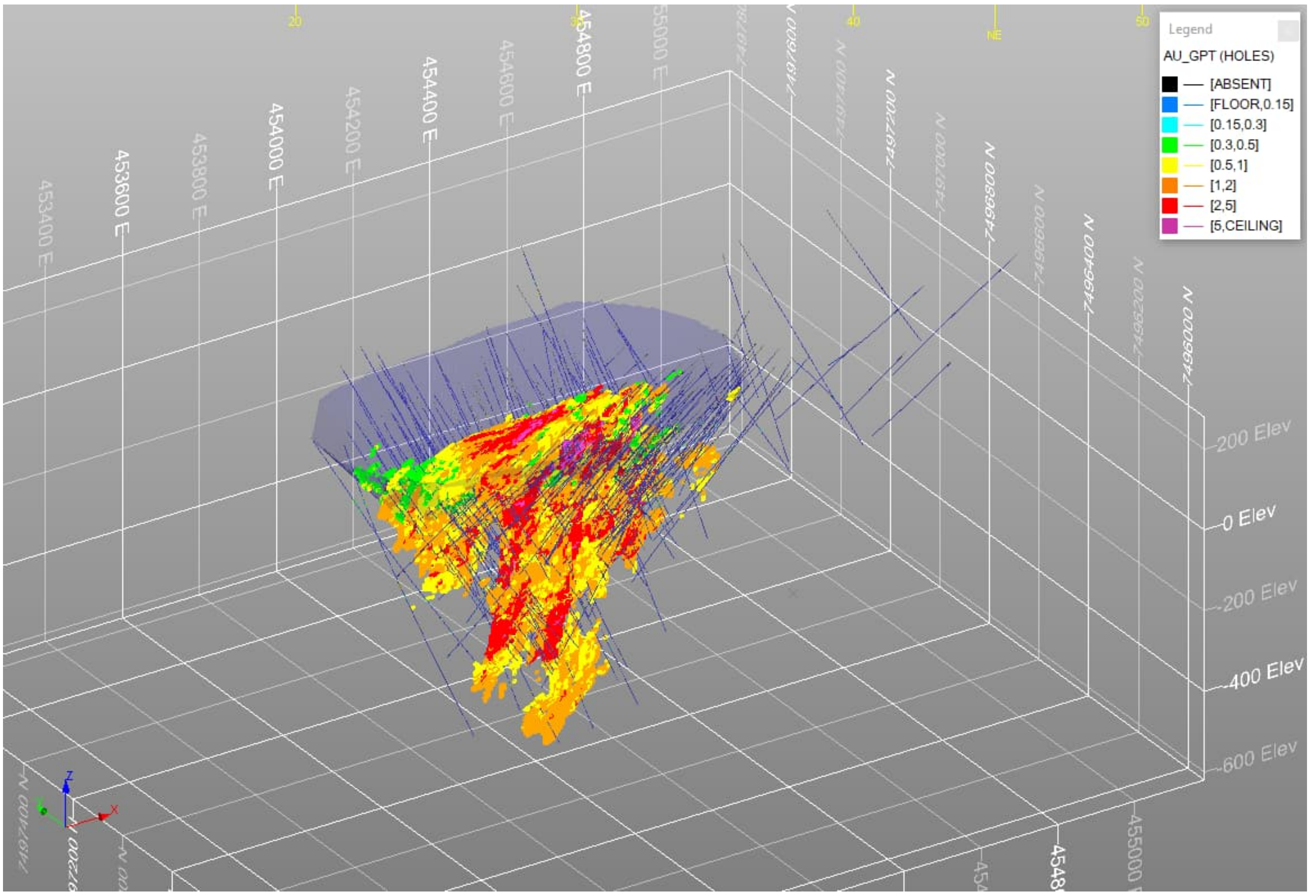


Figure 14-14 – Oblique View Facing NE of Combined OP and UG Mineral Resource Blocks

14.5.8. MINERAL RESOURCE STATEMENT

Mineral Resources are not Mineral Reserves, and do not demonstrate economic viability. There is no certainty that all, or any part, of this Mineral Resource will be converted into Mineral Reserve. Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves.

Table 14-12 summarizes the in-situ Indicated and Inferred Mineral Resources for the Ikkari Project, and Table 14-13, and Table 14-14 demonstrate the tonnage and grade sensitivity relative to other potential OP and UG mining cut-offs (base case scenarios highlighted in bold). Mineral Resources are reported inclusive of Mineral Reserves.

The OP MRE was evaluated for RPEEE by reporting blocks above a 0.4 g/t Au cut-off from within a Whittle generated pit shell based on the assumptions and parameters described in section 14.5.7.

The UG MRE was constrained to the three mineral domains as they demonstrated reasonable continuity for the base case scenario of bulk tonnage longhole mining. Blocks above the cut-off grade in the Background domains were excluded from the reported underground resource estimate as they did not demonstrate adequate continuity for mining. Test case mining envelopes were generated to confirm that there was no material difference between constraining the resource to the mineral domain models.

Table 14-12 – Ikkari Mineral Resource Estimate (Effective Date October 24, 2023)

Resource Category	Mining method	Cut-Off Grade Au (g/t)	Tonnes (t)	Grade Au (g/t)	Au Content (Troy Ounces)
Indicated	Open pit	0.4	37 308 000	2.21	2 649 000
	Underground	0.9	21 122 000	2.12	1 437 000
Total indicated	**-**	**-**	**58 430 000**	**2.18**	**4 087 000**
Inferred	Open pit	0.4	1 271 000	0.81	33 000
	Underground	0.9	2 305 000	1.39	103 000
Total inferred	**-**	**-**	**3 576 000**	**1.18**	**136 000**

Notes:

1) Mineral Resources are reported in-situ and inclusive of Mineral Reserves
2) Tonnage and ounces are rounded to the nearest 1 000.
3) g/t = grams per tonne, ounces are reported as troy ounces.
4) Totals may not add up correctly due to rounding.
5) Cut-off grade defined by Gold Price, $1700/oz, Metallurgical Recovery 95%, Open Pit Mining Costs $2.9/t, Underground Mining Cost $29/t, Processing Cost $11.30/t, G&A, Rehabilitation & Closure $4.8/t, Royalty 0.75%.
6) Open pit resources constrained within a Whittle Optimized open pit shell using the above assumptions with a 26m offset to the property boundary enforced.
7) Underground resources constrained within the estimation domains to meet the RPEEE criteria for underground mining.

Table 14-13 and Table 14-14 demonstrate OP and UG Mineral Resource sensitivities. Estimates reported below the base case mining scenario cut-offs for open pit and underground mining are shown for informational purposes and do not demonstrate RPEEE.

Table 14-13 – Ikkari Open Pit Cut-off Sensitivity Comparison

Resource Category	Au Cut-Off (g/t)	Tonnes (t)	Grade Au (g/t)	Au Content (Troy Ozs)
INDICATED	0.30	38 385 000	2.16	2 662 000
INDICATED	0.35	37 866 000	2.18	2 656 000
INDICATED	**0.40**	**37 308 000**	**2.21**	**2 649 000**
INDICATED	0.45	36 618 000	2.24	2 640 000
INDICATED	0.50	35 944 000	2.28	2 630 000
INFERRED	0.30	1 883 000	0.66	40 000
INFERRED	0.35	1 510 000	0.74	36 000
INFERRED	**0.40**	**1 271 000**	**0.81**	**33 000**
INFERRED	0.45	1 059 000	0.88	30 000

Resource Category	Au Cut-Off (g/t)	Tonnes (t)	Grade Au (g/t)	Au Content (Troy Ozs)
INFERRED	0.50	913 000	0.95	28 000

Notes:

1) Base case scenario highlighted in bold.
2) Au cut-offs listed below the base case scenario do not demonstrate RPEEE and are shown for informational purposes only.
3) Tonnage and Au content estimates are rounded to the nearest 1 000.
4) g/t – grams per tonne.
5) Ounces are reported as troy ounces.
6) Totals may not add up correctly due to rounding.

Table 14-14 – Ikkari Underground Cut-off Sensitivity Comparison

Resource Category	Au Cut-Off (g/t)	Tonnes (t)	Grade Au (g/t)	Au Content (Troy Ozs)
INDICATED	0.8	23 174 000	2.00	1 493 000
INDICATED	**0.9**	**21 122 000**	**2.12**	**1 437 000**
INDICATED	1	19 212 000	2.23	1 379 000
INDICATED	1.1	17 556 000	2.34	1 323 000
INDICATED	1.2	16 158 000	2.45	1 272 000
INFERRED	0.8	3 118 000	1.25	125 000
INFERRED	**0.9**	**2 305 000**	**1.39**	**103 000**
INFERRED	1	1 747 000	1.53	86 000
INFERRED	1.1	1 273 000	1.71	70 000
INFERRED	1.2	1 015 000	1.85	60 000

Notes:

1) Base case scenario highlighted in bold.
2) Au cut-offs listed below the base case scenario do not demonstrate RPEEE and are shown for informational purposes only.
3) Tonnage and Au content estimates are rounded to the nearest 1 000.
4) g/t – grams per tonne.
5) Ounces are reported as troy ounces.
6) Totals may not add up correctly due to rounding.

Grade-tonnage curves were also generated for the Indicated category to evaluate sensitivities for OP and UG estimates as shown in Figure 14-15 and Figure 14-16.

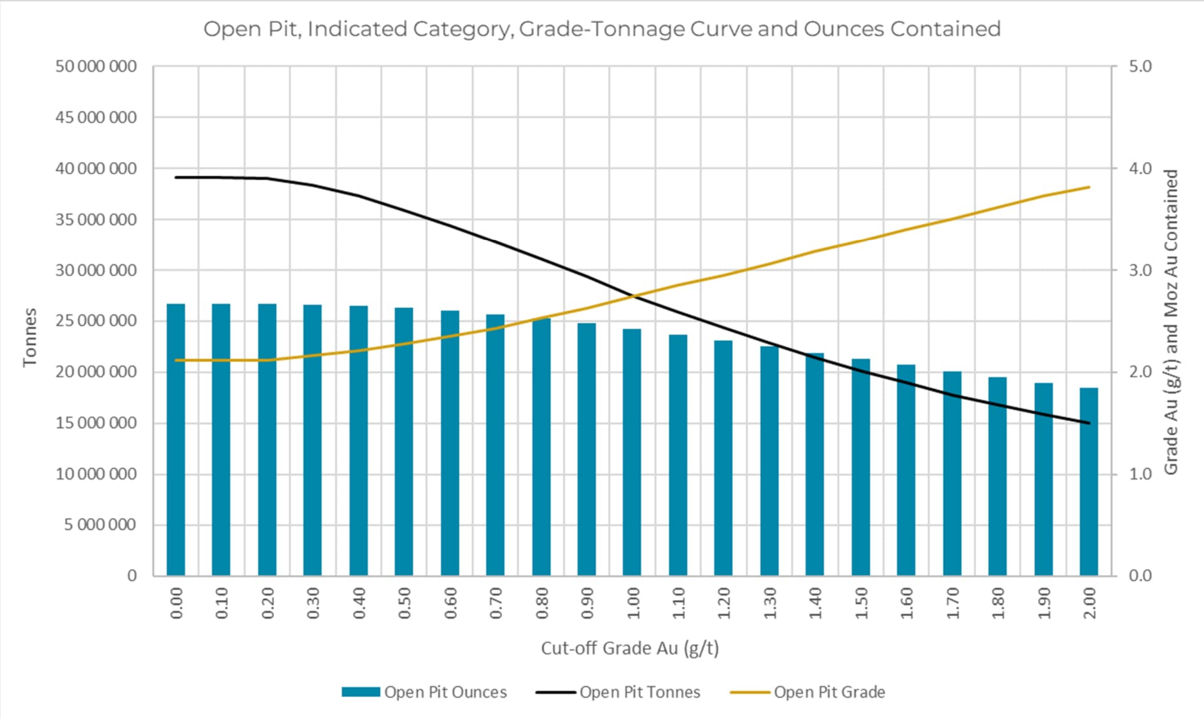


Figure 14-15 – Grade-Tonnage Curve for OP Indicated Category

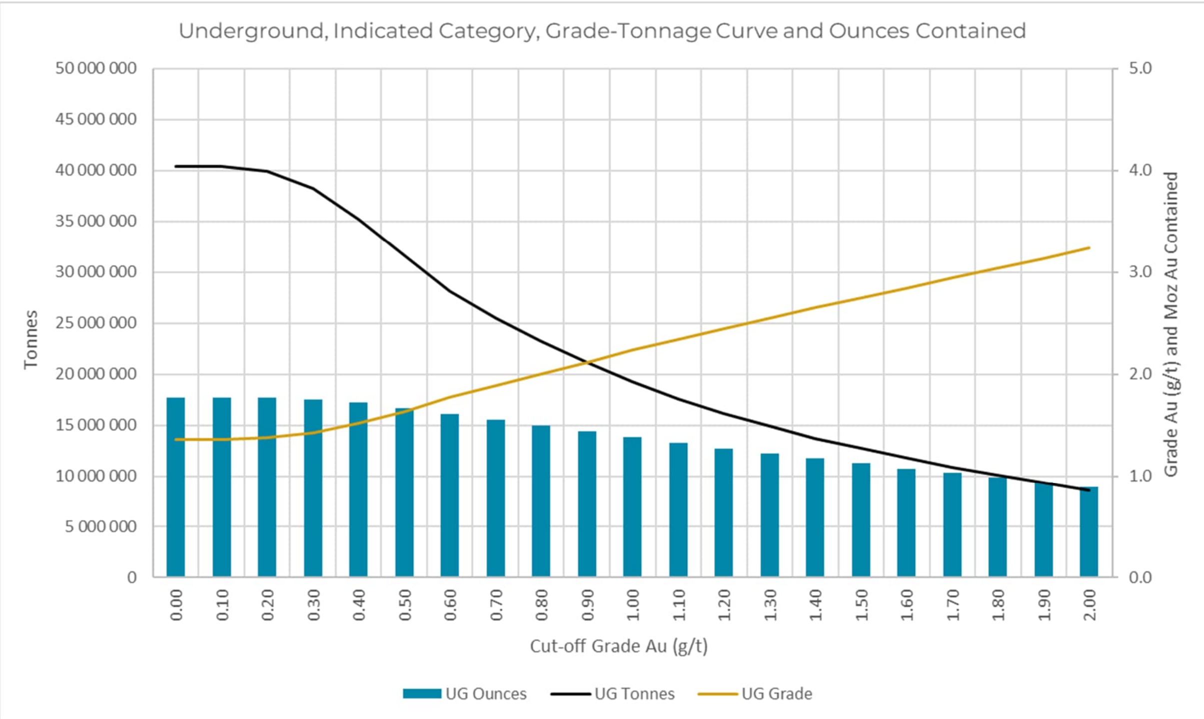


Figure 14-16 – Grade-Tonnage Curve for UG Indicated Category

A comparison was completed to evaluate changes between the 2022 and 2023 MRE, as summarized in Table 14-15.

Table 14-15 – Summary of Changes to the Ikkari Mineral Resource Estimate from 2022 to 2023

	2022 Resource Estimates			2023 Resource Estimates			Changes to the Resource Estimate		
Category	Tonnes (000)	Au Grade (g/t)	Au Content (000s Troy Ozs)	Tonnes (000)	Au Grade (g/t)	Au Content (000s Troy Ozs)	Tonnes (000)	Au Grade (g/t)	Au content (000s Troy Ozs)
Indicated OP	30 000	2.5	2 400	37 308	2.21	2 649	7 308	- 0.29	249
Indicated UG	16 500	2.4	1 280	21 122	2.12	1 437	4 622	- 0.28	157
Inferred OP	3 100	1.5	48	1 271	0.81	33	- 1 829	- 0.69	- 15
Inferred UG	8 700	2.0	550	2 305	1.39	103	- 6 395	- 0.61	- 447

There were significant changes between 2022 and 2023 that resulted in material differences to the stated MRE, as summarized in the following list:

1) Rupert Resources conducted infill drilling to an average drill spacing of approximately 40 m which resulted in the conversion of lower grade Inferred Mineral Resources to the Indicated Mineral Resource category;
2) The resource cut-off grades for the OP and UG resources were each reduced by 0.1 g/t to 0.4 g/t and 0.9 g/t respectively resulting in an increase in tonnage and a decrease in grade;
3) The resource estimation methodology was changed significantly but did not result in material changes to the gold content; and
4) The resource is now constrained with an optimized resource pit shell whereas the 2022 resource was constrained within a designed open pit.

14.5.9. RISKS AND OPPORTUNITIES

The QP has summarized the following risks and opportunities related to this MRE:

- Mineral domain and lithological models were interpreted from drill hole data and may not accurately represent the geology or account for the full scale of geological variability due to the complex structurally deformed nature of the deposit. Geological models generally change and evolve and improve over time as new information becomes available;
- Orientations of some of the drill holes may not represent a true cross-section and are possibly oriented sub-parallel to the down dip direction locally which may result in some local grade bias in the block model;

- The sample database contains high-grade outlier values which can have a material impact on the MRE. The QP has taken steps to reduce the impact of this data but there remains some uncertainty regarding the impact on the overall quantity of metal in the deposit;
- Many different grade estimation methodologies can be used to support a MRE and variations in the approach and estimation parameters used can have a material impact on the resource estimate. Different approaches may affect the degree of grade smoothing which can have a material impact when reporting mineral resources above a grade cut-off. The QP has made efforts to achieve the expected level of smoothing to match the change of support from 2.5 m composites to the SMU block size, but the process is not an exact science and is dependent on the quality of the variogram and mineral domain models;
- The density measurements are not as closely spaced as the grade data and may present a relatively minor risk to the accuracy of the tonnage and metal content of the MRE;
- Changes in metal prices and mining costs can vary significantly over short periods of time which has the potential to materially impact the MRE;
- The metallurgical recovery assumed for the MRE is based on test work completed to date and may not reflect actual recoveries achieved during future mining;
- The exclusion of mineralisation in the background domains from the UG MRE presents an opportunity to increase UG resources through continued exploration and infill drilling; and
- Further infill drilling will provide an opportunity to increase resource confidence and may support the conversion of Indicated resources to the Measured category.

The QP is unaware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or any other potential factors that could materially impact the Ikkari MRE provided in this Technical Report.

Since initial publication of this MRE (28 November 2023, filed on Sedar 12 December 2023) and its inclusion within the Ikkari PFS technical disclosure here Rupert Resources have conducted further drilling at the Ikkari deposit principally for further geotechnical and metallurgical characterisation as well as exploration drilling within the wider project area. Of this drilling approximately 2 250 m occurs within the estimation domains utilised for this MRE. Results from this drilling have been reviewed and in the opinion of the QP do not represent a material change from the MRE presented here.

15 MINERAL RESERVE ESTIMATES

15.1 INTRODUCTION

The Mineral Reserve estimate was prepared in accordance with the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (MRMR Best Practice Guidelines) prepared by the Canadian Institute of Mining, Metallurgy and Petroleum's (CIM) Mineral Resources and Mineral Reserves Committee (CIM MRMR Committee) to update an earlier version that was accepted by CIM Council on November 23, 2003 (CIM, 2003). These 2019 MRMR Best Practice Guidelines supersede and replace the November 23, 2003, version of the MRMR Guidelines.

The disclosure of the Reserve estimate uses the NI 43-101 guidelines and has excluded the use of Inferred Mineral Resources. Mineral Reserves are generated from both the proposed open pit and underground mining areas. The open pit is based on a conventional truck and shovel operation. Underground mining implements a Long Hole Open Stoping mining method (LHOS) with either waste rock or paste backfill.

The Mineral Reserve is estimated at 52 Mt at an average grade of 2.1 g/t for 3.5 million ounces of gold as summarised in Table 15-1 and was used in the preparation of production schedules and cashflow analysis.

Mineral Reserves were estimated based on the Resource contained within the open pit and underground mine designs, with allowances for mining losses and dilution. The Ikkari Mineral Resource Estimate contains Indicated and Inferred resources only with no component in the Measured category. Only Indicated Mineral Resources were used in the open pit and underground optimisation and design process, resulting in the Mineral Reserve comprising of Probable Reserves only.

Table 15-1 – Ikkari Gold Project Mineral Reserve by Category

Mining Method	Category	Tonnage [Mt]	Gold Grade [g/t]	Contained Gold [koz]
Open Pit	Proven	0.0	0.0	0
	Probable	35.7	2.2	2 486
Underground	Proven	0.0	0.0	0
	Probable	16.3	1.9	1 007
Total Mineral Reserves		**52.0**	**2.1**	**3 492**

15.2 RESOURCE MODEL

The Ikkari Gold Project Mineral Reserve Estimate was based on the Indicated Mineral Resource material contained in the Resource block model prepared by WSP effective October 24, 2023. The model contains attributes for gold grade, classification, and density with block sizes at 10m (X) by 5m (Y) by 5m (Z).

15.3 MINING MODEL

The mining model used for optimisations, designs and scheduling was unchanged from the resource model provided with the selective mining unit (SMU) for open pit mining corresponding to the block size in the resource model.

Additional attributes were included in the mining model to incorporate geotechnical domains, the proposed underground stope geometry (strike lengths), dilution factors and net smelter return (NSR).

15.4 PROCESSING RECOVERY

At the time of the studies into excavation geometry optimisation, results of the PFS metallurgical test work were not available. After reviewing previous test work a fixed processing recovery of 95% was implemented. The PFS metallurgical test work indicated an overall gold recovery of 95.8%, aligning with the assumed recovery implemented for the optimisations.

15.5 PRODUCT SELLING COSTS

A refining and treatment charge of $2.50/oz was applied. This was based on quotes obtained by Rupert Resources during the Preliminary Economic Assessment (PEA) (Tetratech, 2023). A gold payment factor of 99.92% was applied, based on quotes obtained during the PEA, and is in line with industry norms.

In line with Finnish mining law, a royalty of 0.75% of revenue was applied to the excavation cost estimates and optimisation computations, comprised of 0.6% state royalty and 0.15% landowner royalty. The value of the royalty is a percentage of the revenue generated based on the contained metal delivered to the processing plant.

15.6 OPEN PIT MINING

15.6.1. DILUTION AND RECOVERY/LOSS

The dilution factor for the open pit mine design to modify the Resource model for conversion to Reserves was based on the correlation between the SMU and Resource model blocks being of the same dimensions. The principal dilution parameter is the external dilution which was estimated by implementing a 1m dilution skin around the block. The overall weighted average dilution was derived to be 3.8%. The full results of the dilution analysis are shown in Table 15-2. It was decided to simplify the dilution by rounding this up to a 4% dilution modifying factor was applied universally to all mining blocks during the open pit scheduling.

Table 15-2 – Dilution by Ore Bearing Domain

Solid Model	Tonnage [Mt]	Au Grade [g/t]	Au Content [koz]
Contact Domain	20.1	2.6	1 674
Dilution Skin	21.4	2.5	1 716
Difference	**6.0%**	**-3.6%**	**2.4%**
Internal Siderite Domain	7.5	2.1	496

Solid Model	Tonnage [Mt]	Au Grade [g/t]	Au Content [koz]
Dilution Skin 2	8.5	1.9	527
Difference	**11.4%**	**-6.2%**	**5.8%**
Northern Felsic Domain	13.0	1.1	475
Dilution Skin 3	13.7	1.1	490
Difference	**5.3%**	**-2.7%**	**3.0%**
Total Domains	40.6	2.0	2 600
Total with Dilution Skins	43.6	1.9	2 700
Total	**6.9%**	**-3.8%**	**3.3%**

*Results may not add up correctly due to rounding.

The ore loss was estimated to be 3.6% ore loss, which was rounded to 4.0%. This was applied to all blocks as a Reserve modifying factor and utilised in the mine schedule to account for ore loss. An example section of the block model, where the blocks are coloured according to the partial percentages, is shown in Figure 15-1 with the orebody model outline in white.

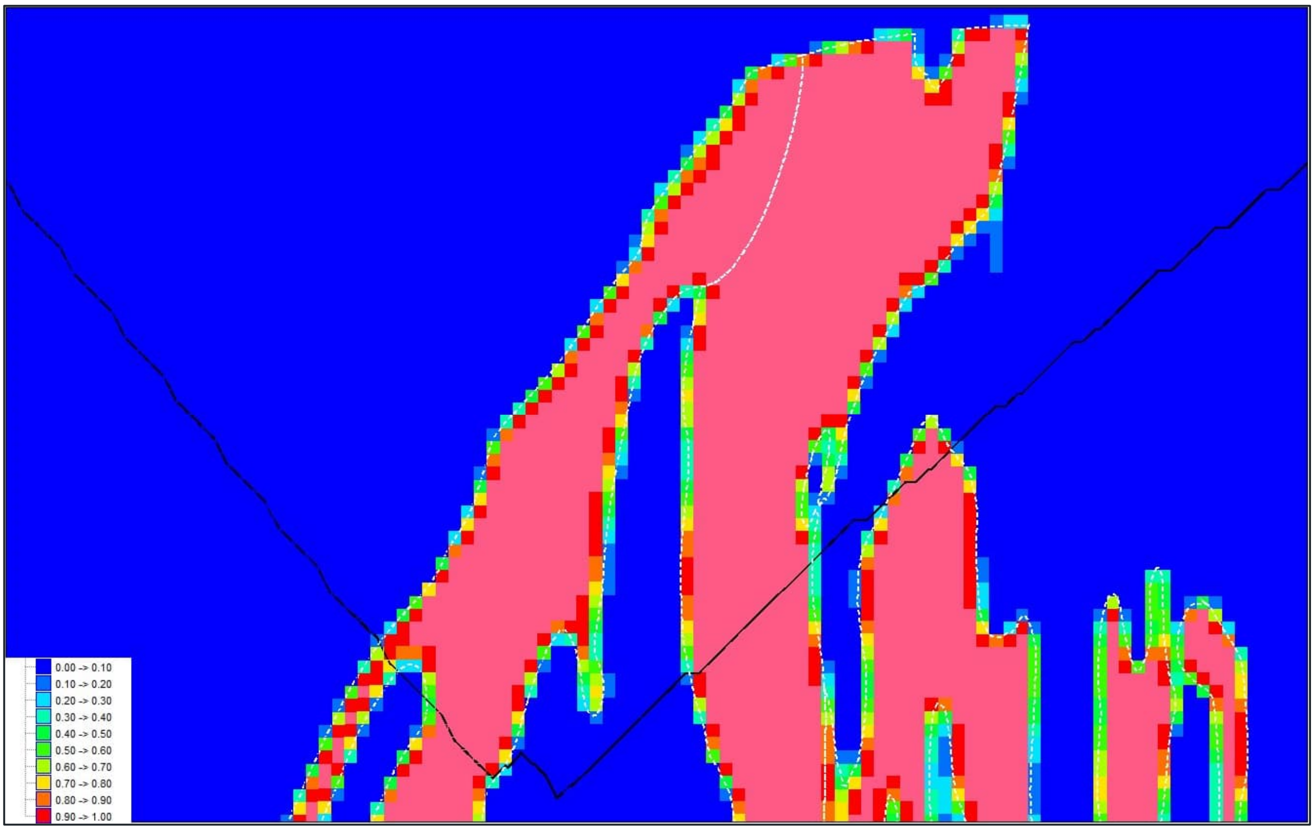


Figure 15-1 – Block Model Section with Blocks Coloured According to Partial Percentages

15.6.2. CUT-OFF GRADE

The open pit break-even cut-off grade (COG) was evaluated by comparing the mining, processing, and selling costs to the revenue generated from gold sales. A COG of 0.34 g/t was implemented for the open pit optimisation, with parameters shown in Table 15-3. The parameters for the open pit optimisation computations and analysis were derived from the previous PEA study, industry knowledge and technical assessment of the mineral property setting.

In line with long term consensus gold price at the time (December 2023), 1 700 $/oz was used in the optimisation.

Table 15-3 – Open Pit Optimisation COG Parameters

Parameter	Value	Unit
Gold Price	1 700	$/oz
Processing Recovery	95.00	%
Payability	99.92	%
Refining & Treatment	2.50	$/oz
Royalty	0.75	% of ROM Gold Content
NSR	**51.42**	**$/t ROM**
Production Rate	3.5	Mtpa
Mining Cost – Ore	2.40	$/t
Mining Cost – Waste	2.20	$/t
Grade Control	0.50	$/t ore
Mine Rehabilitation	1.65	$/t ore
Closure Cost	0.80	$/t ore
Processing Cost	11.32	$/t ore
General & Administration	2.35	$/t ore
NSR Cut-Off	**19.02**	**$/t ROM**
COG	**0.34**	**g/t**

15.6.3. OPEN PIT OPTIMISATION

Open pit optimisation analysis was performed using Geovia's Whittle software. Only Indicated Resources were considered. A 20m offset from Rupert's property boundary was implemented at the open pit crest to allow for bunding, a road and primary drainage culvert between the area.

The overall slope angles for the open pit optimisation were derived within the WSP's rock engineering stability study. The open pit slope angles were bounded in geometric sectors which were coded into the mining model for the optimisation. The geotechnical sectors are shown in Figure 15-2 and corresponding overall slope angles are shown in Table 15-4.

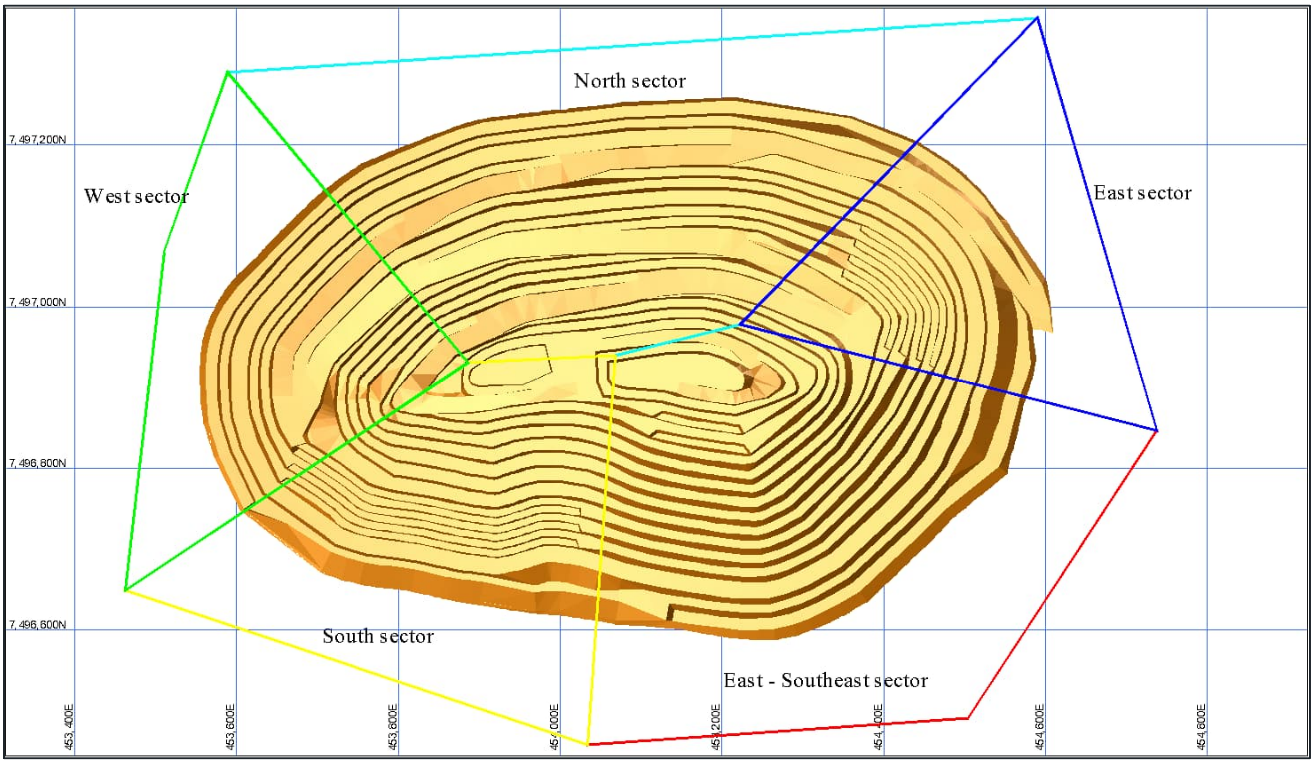


Figure 15-2 – Open Pit Geotechnical Sectors

Table 15-4 – Pit Optimisation Overall Slope Angle Parameters

Pit Sector	Unit	Value
North	Deg.	51.0
East	Deg.	45.7
Southeast	Deg.	51.4
Southwest	Deg.	48.5
West	Deg.	48.4
Overburden	Deg.	24.4

The optimisation analysis output is shown in Figure 15-3. A revenue factor of 0.8 was selected as the basis for the open pit Reserve design. A total of 39.6 Mt at 2.20 g/t and 2.8 Moz is defined at the RF shell of 0.8. The stripping ratio for the optimal pit shell is 3.4 (Waste t to Ore t).

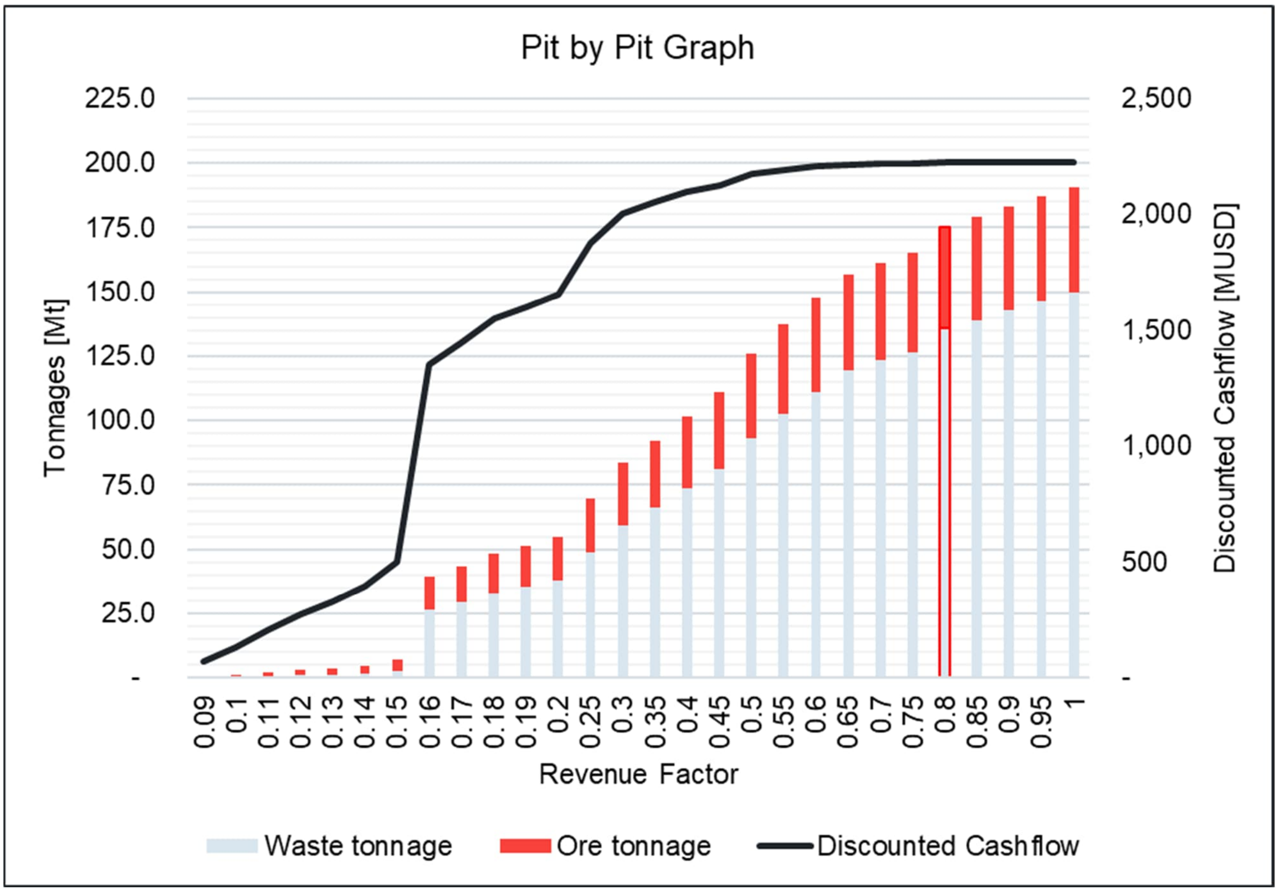


Figure 15-3 – Pit by Pit Optimisation Results

15.6.4. RESERVE OPEN PIT DESIGN

Two principal design criteria were applied to the open pit:

- The Southern wall of the open pit was to be maintained at its OSA, without ramps, to maximise the recovery of ore from the open pit; and
- Two operational pushback stages were designed to enable early gold production and reduce waste stripping in the early years whilst still maintaining practical mining areas.

Stage 1 pushback targets the centre of the pit and extends to the final Southern wall. The ramp entrance was placed on the Eastern side to allow close access to the processing plant and waste storage facility. The ramp was placed on the Northern wall for the first pushback to maximise Resource recovery. A total of 61.5 Mt will be mined, with 16.9 Mt of ore at 2.5 g/t and 1.3 Moz gold. Stage 1 pit design is shown in Figure 15-4.

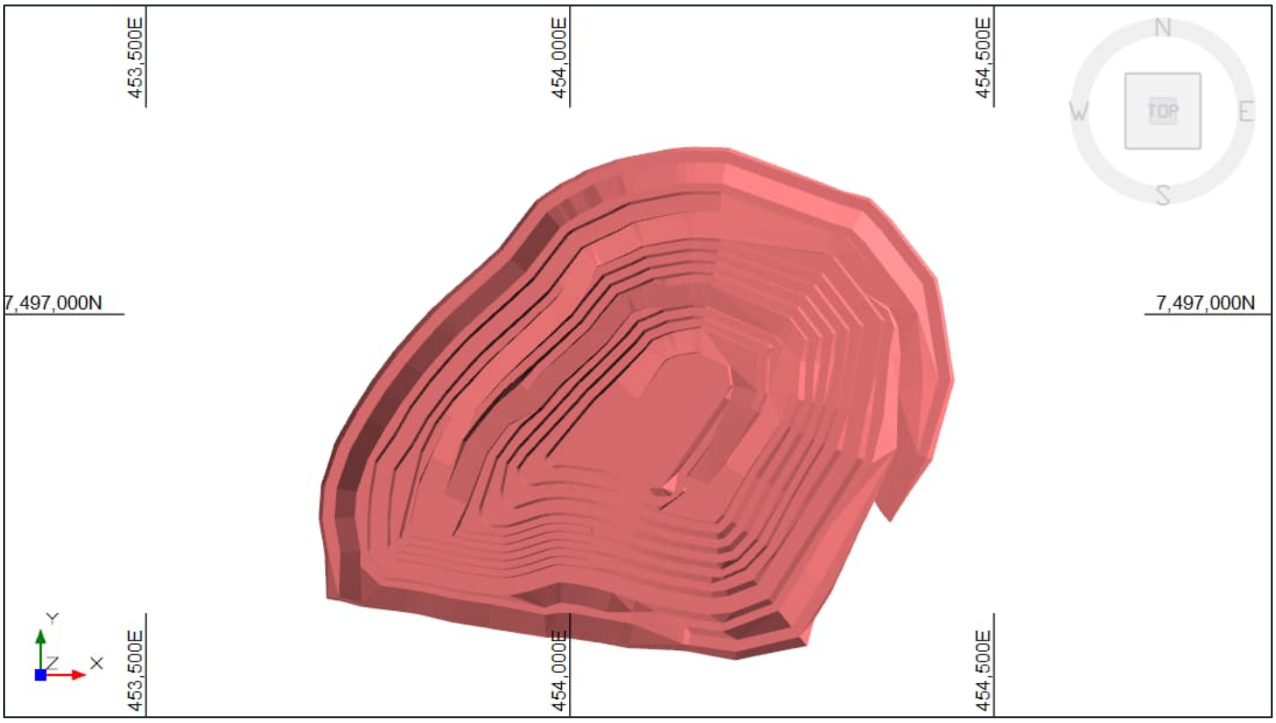


Figure 15-4 – Stage 1 Pit Design

Stage 2 comprises of the final pit design for the Ikkari Project. Stage 2 also maintains the ramp access on the Northern side. The pit expands North, West and East from the Stage 1 pit. A total of 18.8 Mt at 1.9 g/t and 1.1 Moz of gold will be mined from Stage 2. 86.4 Mt of waste will be mined. Stage 2 is shown in Figure 15-5. A total of 35.8 Mt of ore will be mined at 2.2 g/t and 2.5 Moz gold will be mined in Stages 1 and 2.

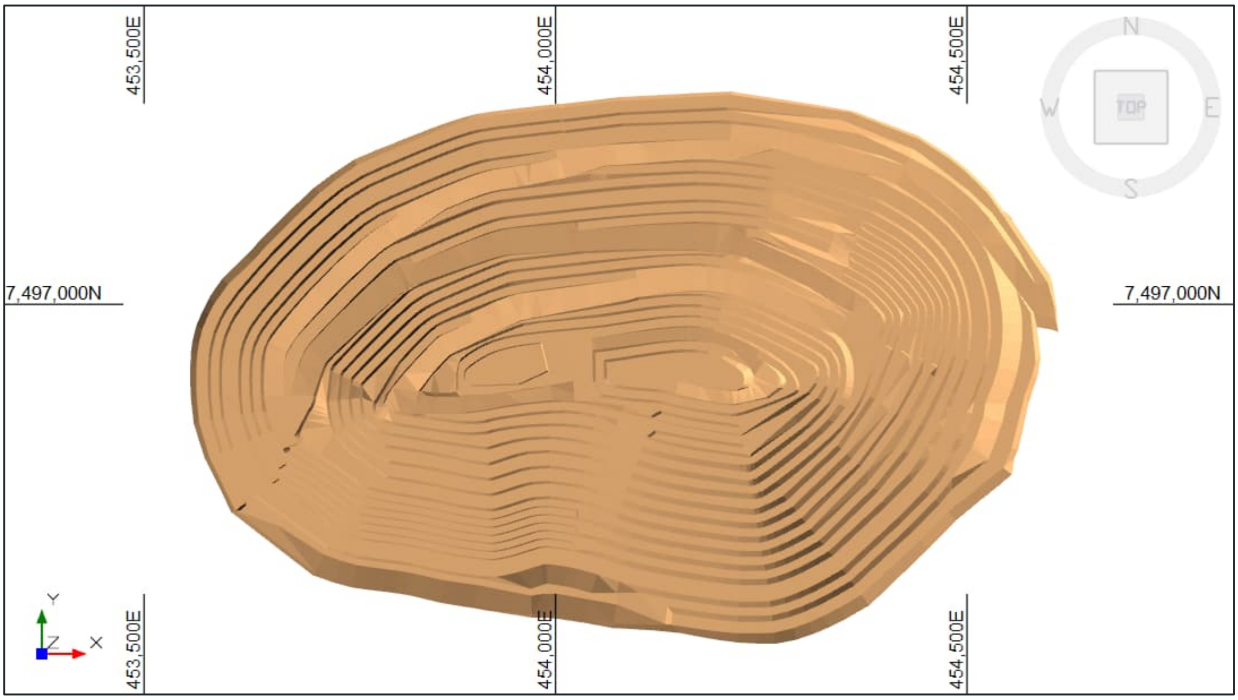


Figure 15-5 – Stage 2 (Final) Pit design

Waste material will be stored at a co-disposal facility North of the open pit mine. A total of 131 Mt of waste will be mined. Open pit material inventories by stage are shown in Table 15-5.

Table 15-5 – Open Pit Stage Inventories

Stage	Ore [Mt]	Au Grade [g/t]	Au Metal [koz]	Waste [Mt]	Strip Ratio
1	16.9	2.5	1 353	44.6	2.6
2	18.8	1.9	1 132	86.4	4.6
Total	**35.7**	**2.2**	**2 486**	**131.0**	**3.6**

15.7 UNDERGROUND MINING

Underground Mineral Resources were converted to Mineral Reserves by accounting for conventional modifying factors. Optimised stope shapes were generated and used as the basis for detailed design. The mine plan and design informed the underground production schedule and subsequent cost model of required capital investment and stope grade operational income.

The modifying factors set out below were used for the generation of optimised stopes within Datamine's Mineable Shape Optimiser (MSO).

15.7.1. STOPE DIMENSIONS

Stope dimensions were based on geotechnical assessments carried out by WSP and are outlined in Chapter 16 of this report. A stope height of 30 m and stope width of 15 m was implemented throughout the underground mine

Stope strike length varied depending on the geometry of the geological and mineral resource model, summarised in Table 15-6.

Table 15-6 – Maximum Stope Strike Length by Geological Domain and Depth

Geological Domain	Depth from Surface [m]	Maximum Stope Strike Length [m]
Internal Felsic	0-300	50
	300-420	36
	>420	28
Mixed Ultramafic Schist	0-300	34
	300-420	20
	>420	16
Northern Felsic	0-300	20
Ultramafic	0-300	24
	300-420	14
	>420	11

15.7.2. DILUTION

Dilution is below COG material mixed with ore during the mining extraction phase and fed to the processing plant, with discrimination between ore and waste coming from the COG imposed during operational grade control. Waste was estimated by quantifying material below the COG within the mining domains.

WSP specified that planned dilution as rock below the COG within the initial stope outline. Inferred material within the stope was treated as zero-grade waste with zero economic value, in line with CIM guidelines. Internal dilution varies for each stope. Planned dilution was estimated at 15% waste rock to ore.

WSP specified unplanned dilution as rock originating from outside the stope limits. An Equivalent Linear Overbreak Sloughing (ELOS) was applied to estimate external dilution on the hanging-wall and footwall. ELOS was estimated by depth and geological domains as shown in Table 15-7. Unplanned dilution due to ELOS was estimated at 6%.

The planned and unplanned dilution concept is shown in Figure 15-6.

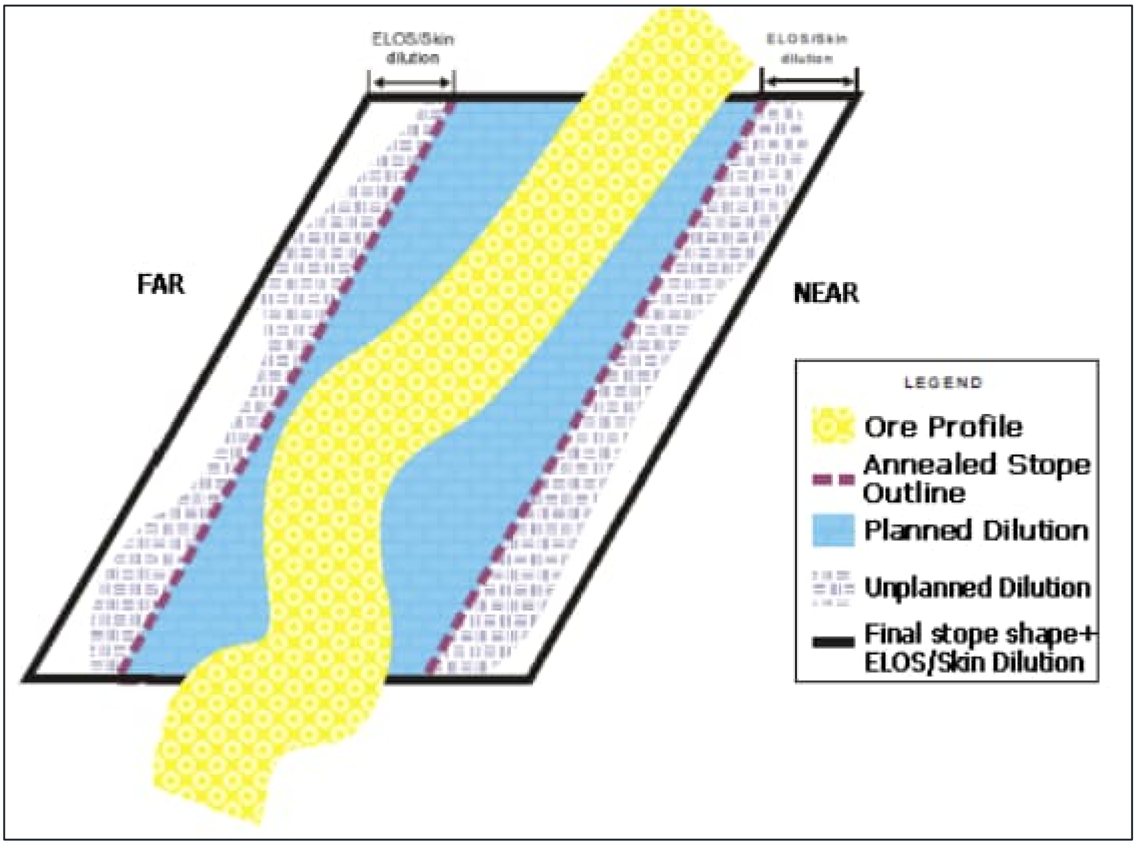


Figure 15-6 – Planned and Unplanned Dilution (Datamine, 2024)

Table 15-7 – Unplanned Overbreak by Geological Domain and Depth for Hangingwall and Footwall

Geological Domain	Depth from Surface [m]	Unplanned Overbreak [m]
Internal Felsic	0-300	0.7
	300-420	0.8
	>420	0.9
Mixed Ultramafic Schist	0-300	1.0
	300-420	1.3
	>420	1.6
Northern Felsic	0-300	0.9
Ultramafic	0-300	1.4
	300-420	1.7
	>420	2.2

Stopes with sidewalls in ore had no additional unplanned dilution applied. The potential overbreak from blasting of primary stopes into secondary stopes occurs mostly internal to the orebody and WSP has computed no impact on forecasted grade and tonnage over time in determining the Reserves.

Stopes with sidewalls in waste or sidewalls, floor or end-wall in paste fill had dilution applied using the parameters outlined in Table 15-8. Dilution occurs because a portion of the backfilled material can become entrained with the newly blasted material located in the sidewalls, end-wall or floors. This is referred to as secondary dilution.

Table 15-8 – Parameters used for Secondary Dilution Estimate

Waste	Parameter
Sidewall Overbreak	0.5 m
Density	2.8 t/m^3
Paste	**Parameter**
Sidewall Overbreak	0.5 m
Floor Overbreak	0.3 m
End Wall Overbreak	0.4 m
Density	2.1 t/m^3

Stopes were separated by sidewall and end-wall interface material to account for the additional unplanned dilution as shown in Table 15-9. Each stope was assigned a "Type" and assigned a level of secondary dilution. This is in addition to the planned dilution and hangingwall and footwall overbreak which is accounted for in the MSO shapes. This process saw an additional 3% material added to dilution.

Table 15-9 – Unplanned Dilution for Different Waste and Paste Exposure

Type	Description	Unplanned Dilution [%]
1	Primary stope surrounded by ore only.	0.0
2	Primary stope with paste end wall exposure and ore elsewhere.	0.6
3	Primary stope with paste floor exposure and ore elsewhere.	0.7
4	Primary stope with paste end wall, paste floor and ore elsewhere.	1.3
5	Primary stope with one sidewall in waste and ore elsewhere.	3.3
6	Primary stope with paste floor, one sidewall in waste and ore elsewhere.	4.0
7	Secondary stope with sidewalls in paste and ore elsewhere.	4.9
8	Secondary stope with sidewalls in paste and paste floor.	5.6
9	Secondary stope with one sidewall in paste and one sidewall in waste.	5.7
10	Secondary stope with sidewalls in paste, paste floor and paste end-wall.	6.2
11	Secondary stope with a sidewall in paste, a sidewall in waste and paste end-wall.	6.3
12	Secondary stope with a sidewall in paste, a sidewall in waste and a paste floor.	6.4
13	Primary stope or secondary stope surrounded by waste	6.5

15.7.3. MINING LOSSES

A 96% modifying factor for stope material recovery was applied to the stopes in the mining schedule to account for material remaining in the stope after blasting and excavation. Stopes directly under the open pit have had a further 90% recovery factor applied to account for any operational factors in recovery, resulting in an overall recovery factory of 86%. These stope recovery factors are comparable to other underground mining operations with similar rock characteristics, mineralisation geometry, mining method and stope geometries.

The underground Mineral Reserves do not include material excavated from the open pit. Consideration was given to extracting mineralisation adjacent to the open pit. A 10 m offset from the open pit walls to possible underground stopes was applied. Mineralisation under the pit floor has not had the 10 m offset applied, as it probable that that this material can be recovered successfully. Material contained within the Southern fault zone was also not considered for optimisation and

Mineral Reserve estimation due to underground mining creating unstable open pit slope stability ground conditions hindering operational recovery. Figure 15-7 displays material excluded from optimisation and Reserve consideration. Blocks coloured blue are not considered in the optimisation process.

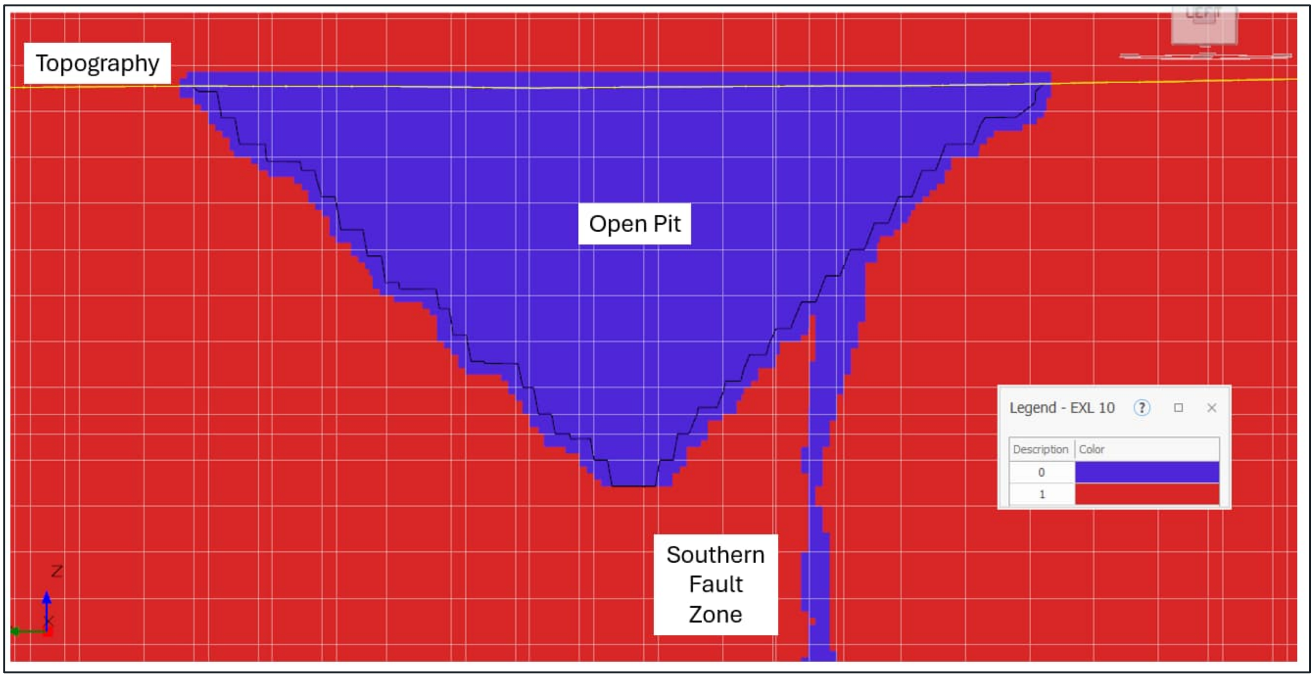


Figure 15-7 – Cross-Section of Mining Model and Open Pit showing Excluded Blocks (Coded 0 and Coloured Blue)

15.7.4. INFERRED & WASTE MATERIAL

Inferred Resources were treated as zero grade and zero economic value material during underground optimisation, design, schedule, and economic analysis as per CIM guidelines. Inferred material captured within the stopes contributes to dilution.

Inferred and waste material within the mineable shapes is captured by the mining method employed and cannot be separated out as waste material, therefore forming part of the feed to the plant.

15.7.5. CUT-OFF GRADE

The underground mining break-even COG was evaluated by comparing the mining, processing and on-site general and administration costs to the revenue generated from gold sales. A COG of 1.04 g/t Au was implemented for the underground optimisation, with parameters shown in Table 15-10.

Mining capital and operating cost estimates were derived from first principles using machinery performance, operational regime and OEM cost factors built up for the chosen stope geometries.

The parameters for the underground optimisation computations and analysis were derived from the previous PEA study (Tetratech, 2023), industry knowledge and technical assessment of the mineral property setting. In line with long term consensus gold price at the time (December 2023) 1 700 $/oz was used in the optimisation.

Table 15-10 – Underground Optimisation COG Parameters

Parameter	Value	Unit
Gold Price	1 700	$/oz
Processing Recovery	95.00	%
Smelter Payability	99.92	%
Refining and Treatment	2.50	$/oz
Royalty	0.75	% of ROM Gold Content
NSR	**51.42**	**$/t**
Production Rate	2	Mtpa
Mining Cost	39.60	$/t
Processing Cost	11.32	$/t
General and Administration	2.35	$/t
Costs	**53.27**	**$/t**
COG	**1.04**	**g/t**

15.7.6. UNDERGROUND OPTIMISATION

Datamine MSO was used to generate mineable shapes above the COG. Parameters outlined in Table 15-6, Table 15-7 and Table 15-10 were used for the optimisation. Additional MSO parameters are outlined in Table 15-11.

Table 15-11 – MSO Parameters

Parameter	Value	Unit
Framework Type	Slice Method, XZ.	-
Stope Vertical Height	30 & 20	m
Stope Width	15	m
Minimum Stope Length	5	m
Maximum Stope Length	As per Table 15-6	-
Minimum Waste Pillar	0	m
Dilution	As per Table 15-7	-
Stope Dip Angle	Near 70/110/10 Far 60/120/10	Min/Max/Change °

Parameter	Value	Unit
Stope Strike Angle	-10/10/20	Min/Max/Change °
Materials Excluded	Waste Inferred Open Pit Southern Fault Zone	-
Maximum Waste Percentage	30 30 0 0	Waste % Inferred % Open Pit % Southern Fault Zone %

Results from the optimisation studies were peer reviewed and validated. Isolated stopes were assessed against capital development cost requirements to determine economic viability and were removed if they were not economic. Manual stopes were designed and added where possible. Manual stopes followed the same geometric constraints as the 30 m height stopes.

MSO and manual stope results are shown in Table 15-12.

Table 15-12 – MSO Results

Run	Tonnes [Mt]	Au Grade [g/t]	Au Metal [koz]
30m Height	15.9	2.0	1 010
20m Height	0.4	2.0	27
Manual Stopes	0.1	2.1	5
Total	**16.4**	**2.0**	**1 043**

15.7.7. MINERAL RESERVE UNDERGROUND MINE DESIGN

The Mineral Reserve underground mine design was used as the basis for the production schedule and cash flow modelling. The underground mine design is shown in Figure 15-8.

Two surface declines were designed to access the stopes. Initial access is located to the East of the open pit providing access to the Northern side of the orebody. A second decline from inside the open pit is used to access the Southern side of the orebody. Material will be hauled via truck to surface using these declines.

Ventilation exhaust and fresh air raises are located East, North and West of the open pit and provide adequate ventilation and heating to the underground mine. There will be two fresh air intake raises and two exhaust raises at the mine.

Stopes will be accessed with overcut and undercut ore drives. These drives were declined towards the footwall drive to promote water drainage. The footwall drives were also declined towards the level access which then links into a sump.

Primary stopes will be paste filled. Secondary stopes will be rock filled where possible. Paste filling of secondary stopes will be required when adjacent stopes are to be blasted or required for support purposes.

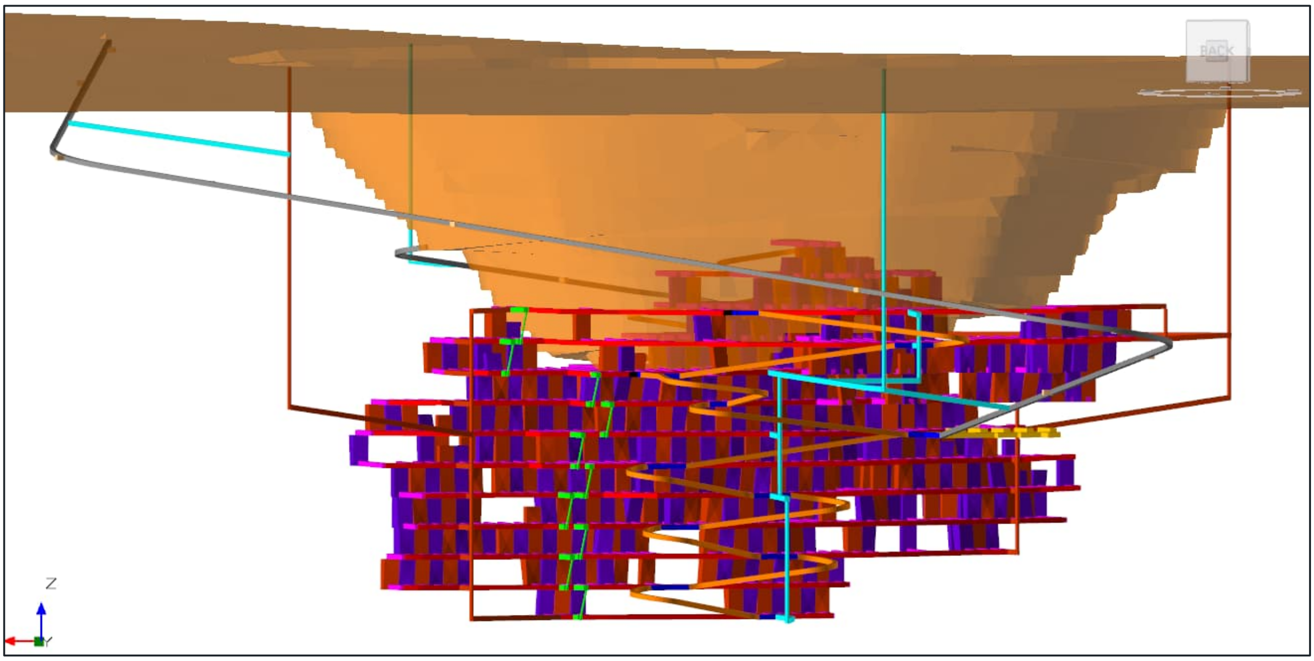


Figure 15-8 – Underground Reserve Mine Design with Depleted Open Pit Looking North-West

15.8 OPEN PIT AND UNDERGROUND MINERAL RESERVE ESTIMATE

The Mineral Reserve Estimate was determined in accordance with the 2019 Edition of the CIM Standing Committee on Reserves Definitions, adopted by CIM Council.

CIM defines a Mineral Reserve as:

- “Mineral Reserves are estimates of the tonnage and grade or quality of material contained in a Mineral Resource that can be economically mined and processed. To be considered a Mineral Reserve, modifying factors must be applied to the Mineral Resource estimate as part of the preparation of a prefeasibility study (PFS) or a feasibility study (FS) as outlined in the CIM Definition Standards. The estimated amount of saleable material contained in the final product must demonstrate a positive Net Present Value (NPV) using an appropriate discount rate and must demonstrate that eventual extraction could be reasonably justified.”

Mineral Reserves were estimated based on the Resource contained within open pit and underground mine designs with allowances for mining losses and dilution. Only Indicated Mineral Resources were used in the open pit and underground optimisation and design process, resulting in the Mineral Reserve comprising of Probable Reserves only.

Mineral Reserves total 52Mt at an average grade of 2.1 g/t for 3.5 Moz of gold as summarised in Table 15-13 and was used in the preparation of production schedules and cashflow analysis.

The Mineral Reserves shown in Table 15-13.

Table 15-13 – Ikkari Gold Project Mineral Reserve by Category Effective 25 November 2024

Mining Method	Category	Tonnage [Mt]	Gold Grade [g/t]	Contained Gold [koz]
Open Pit	Proven	0.0	0.0	0
	Probable	35.7	2.2	2 486
Underground	Proven	0.0	0.0	0
	Probable	16.3	1.9	1 007
Total Mineral Reserves		**52.0**	**2.1**	**3 492**

Notes:

1) Tonnages are rounded to the nearest 100,000 and ounces are rounded to the nearest 1,000.

2) Mineral Reserves were estimated using the CIM Standards for Mineral Resources and Reserves, Definitions and Guidelines.

3) The QP for the Mineral Reserve Estimate, as defined by NI 43 101, is Mr. Timothy Daffern, Technical Director with WSP. The effective date of the estimate is November 25, 2024.

4) Mineral Reserves are based on a gold price of US$1,700/oz.

5) Metallurgical recovery is based on a fixed recovery of 95.0%.

6) Open pit Reserves are stated using a 0.34 g/t cut-off. Open pit Reserves are converted from Resources through the process of pit optimisation, mine design, schedule and are supported by a positive cash flow analysis.

7) Open pit Reserves include an allowance for 4% dilution and 4% mining losses applied in the production schedule.

8) Underground Mineral Reserves are stated using a 1.04 g/t cut-off. Underground Reserves are generated through the generation of optimised stopes, design of long hole open stoping, schedule and are supported by a positive cash flow analysis.

9) Underground Mineral Reserves account for planned dilution of 15%, unplanned dilution of 6%, secondary dilution of 3% and with mining losses of 4%.

10) Totals may not sum due to rounding.

15.9 COMPARISON TO MINERAL RESOURCES

A comparison of the conversion of Mineral Resources to Mineral Reserves is shown in Table 15-14.

Table 15-14 – Mineral Resources to Mineral Reserves Conversion

Mining Method	Category	Tonnage [Mt]	Gold Grade [g/t]	Contained Gold [koz]
Open Pit	Mineral Resources	37.3	2.2	2 650
	Mineral Reserves	35.7	2.2	2 486
Underground	Mineral Resources	21.1	2.1	1 437
	Mineral Reserves	16.3	1.9	1 007
Total	**Mineral Resources**	**58.4**	**2.2**	**4 087**

Mining Method	Category	Tonnage [Mt]	Gold Grade [g/t]	Contained Gold [koz]
	Mineral Reserves	**52.0**	**2.1**	**3 492**

15.10 FACTORS THAT MAY AFFECT MINERAL RESERVES

Factors that may affect the Mineral Reserves estimate include:

- Resource model assumptions;
- Cut-off grade assumptions;
- Geotechnical and hydrogeological factors affecting stope design;
- Long-term consumables price assumptions;
- Metallurgical recovery assumptions;
- Mining recovery assumptions based on overall mine layout and basis of mining methodology;
- Unplanned dilution assumptions; and
- Long-term commodity price assumptions.

16 MINING METHODS

16.1 INTRODUCTION

The Ikkari project is a combined open pit and underground gold mine with the total life of mine (LOM) at 20 years. LHOS was implemented for the underground mine design and schedule. Summaries of the LOM designs and schedules for each operation, along with data and assumptions, are presented below.

16.2 OPEN PIT MINING

16.2.1. INTRODUCTION

The topography in the Ikkari area is mostly flat, with an elevation of 225 m above sea level (elevation), rising towards southeast up to 300 m elevation.

Most of the open pit area comprises a wet swamp. The overburden cover in the open pit area consists of a 0 m to 2 m thick peat layer above a 5 m to 65 m thick glacial till layer.

16.2.2. OPEN PIT MINING

The orebody extends to the bedrock surface. Beneath the overburden, there is hard rock that will require blasting for it to be mined. A traditional truck and shovel configuration has been selected for open pit operations.

Drilling and blasting are planned on 10 m bench heights. Double benching will be utilised for permanent and semi-permanent pit walls in areas where the rock mass quality is sufficient. Some method of controlled blasting is likely required to improve the safety of the permanent and semi-permanent pit wall slopes; however, this has not been included for in this study.

Loading is mainly designed with 10 m bench height, but in areas where more selectivity is required to reduce dilution, loading will be done in two 5 m flitches.

16.2.3. HYDROLOGIC CONSIDERATIONS

The natural hydrogeological regime comprises four main hydrostratigraphic units, namely the peat at surface, underlain by shallow glacial sediments and the weathered and fresh bedrock at depth. Each has significantly different properties to store and allow the transmission of groundwater.

Within the Ikkari shear zone, the bedrock is deeply weathered, significantly folded and fractured. Groundwater movement inside the shear zone is chiefly controlled by faulting and the deeper weathering associated with these major regional fault zones (Piteau, 2024). Major structures previously identified as the Gauge and Black Shale fault zones provide preferential pathways and linkages for groundwater movement inside and along the margins of the shear zone.

Geological interpretation by RR and Piteau (2024) suggests that the Eastern and Western faults, as well as the East-West Fault, their intersections with each other, the Gauge and Black Shale faults, referred to as the Ikkari Fault Intersection Zone (IFIZ), plays a significant role in the groundwater movement at depth at Ikkari.

Away from the IFIZ, regional shallow permeability is directly connected to the IFIZ, via regionally extensive but thinner weathered and fractured bedrock zone that directly overlies the fresh bedrock. Near surface, the overlying peat and shallow glacial sediments, where present and sufficiently thick, form a predominantly confining layer to the weathered bedrock aquifer zone.

Beyond the shear zone, the Kittilä Group mafic bedrock to the north, and the Kumpu Group metasediments to the south, display very low K values and groundwater movement occurs primarily in the thinner weathered and fractured bedrock zone and possibly in, yet, unidentified fault zones, (Piteau, 2024).

Groundwater flow is driven by the elevated heads created by direct precipitation recharge along the elevated ridges in the south and south-east of the Ikkari ore deposit as well as the higher ground further up the Saittajoki catchments towards the west. The overall groundwater flow being to the east along the Saittajoki Stream catchment.

Tests undertaken during several testing campaigns, including 113 packer tests in 15 deep exploration drill holes (SRK, 2023c) suggest a range of hydraulic conductivity (K) values between 1.14 $x10^{-9}$ m/s to 6.24 $x10^{-6}$ m/s and mean of $5.88x10^{-7}$ m/s, suggesting no significant difference along a 1.5 km section of the Saittajoki valley within the shear zone. Dynamic impellor flow logging runs and analyses provided fracture permeability (KF) ranges between 9.47 $x10^{-5}$ and 9.73 $x10^{-3}$ m/s, with a geomean KF of $9.49x10^{-4}$ m/s.

Analyses of cross -test pumping provided geomean values for T and K in the general rock mass of 3.2 $x10^{-3}$ m^2/s and 1 $x10^{-5}$ m/s respectively nearest the pumping well and 1.1 $x10^{-3}$ m^2/s and 5.2 $x10^{-6}$ m/s respectively in the outer zone, typically 100 m or more from the well. The geomean of Storativity (S) in the region nearest the well is 2.3 $x10^{-4}$ and for the outer zone it is 1.4 $x10^{-4}$ (SRK, 2023c).

Piteau (2024) reviewed the work undertaken by SRK up to 2023 and, updated the conceptual and numerical hydrogeological models for Ikkari. WSP has reviewed this latest numerical groundwater modelling by Piteau in 2024 as part of the PFS studies.

The Piteau model has indicated that dewatering wells placed strategically around the developing pit would capture over 90% of the groundwater inflow when the pit is in operation (Years 1-11). An annualised peak dewatering rate from the peripheral borehole dewatering rates is reached in Year 1 of the open pit operation at approximately 16 500 m^3/day (191 l/s).

Piteau (2024) reports that remaining pit groundwater inflow rates remain insignificant throughout the open pit and underground operations, while the peripheral dewatering wells reduce to approximately 8 300 m^3/day (96 l/s) by the end of the life of mine. Underground groundwater inflows are estimated to increase from Year 6 after underground development commences, to approximately 7 800 m^3/day (90 l/s) by Year 10, requiring a total dewatering effort of approximately 17 500 m^3/day.(203 l/s) at that time.

Perimeter drains will be included around the Open Pit to prevent runoff to the pit from the external catchments. Water captured in the perimeter drains will be considered non-contact water and will be directed to the diverted tributary on the south side of the pit, and to the non-contact mine site drainage on the north side of the pit.

The internal pit water management will include bench drains, draining to an in-pit sump. From the sump, pumps will discharge contact water to the Raw Water Pond. During the winter months in-pit sump pumping is expected to be low, due to precipitation being mainly in the form of snow, which will need to be managed in-pit until it thaws. Groundwater pit inflow is also expected to be significantly reduced by the freezing atmospheric conditions experienced in the Open Pit until the beginning of the warmer months (SRK, 2023c). The Open Pit pumping capacity must, therefore, be sized for the expected thaw volume peaks during spring, summer rainfall and any groundwater inflows which are not managed through the peripheral pit dewatering well system (Piteau, 2024). Open pit dewatering via the peripheral pit dewatering wells is expected to continue beyond the life of the open pit, into the remaining operational period of the underground mine.

Depending on the final pit depth, the in-pit dewatering may consist of multiple pumping stages to reach the pit crest. The peak pump rate from the pit sump will need to manage peak runoff rates during the spring thaw, along with the maximum expected ground water inflows, such that flooding of the base of the pit is kept below a reasonable frequency. A peak pump rate of 810 m^3/h has been allowed for at this stage, capable of managing a 1:20 annual exceedance probability runoff event.

Refer to Chapter 20 of the PFS Report for a description of the wider mine water management approach and a monthly water balance.

16.2.4. GEOTECHNICAL CONSIDERATIONS

Open pit stability was analysed with kinematic analyses to determine bench scale susceptibility to structural failures. Based on the stereographic projection analysis of logged joint orientation data, the lithological domains in the open pit area generally only have two relatively steeply dipping joint orientations, of which one lines up with foliation orientation. Bench failure modes are mainly toppling and planar failures (mainly Southeast and East walls). Expected failure volumes are small and most of the failures are dominated by strong foliation.

Two-dimensional finite element analysis was performed to analyse large scale stability against failure. The simulation was performed using shear strength reduction method for saturated and drained slope models, separately for each open pit sector. The results demonstrate stable slopes within 45° to 55° overall slope angles, depending on the pit sector. Three-dimensional analysis is recommended in the next study phase to further optimize the pit angles.

The ramp placement is recommended on the North wall due to the close vicinity of the property boundary on the South side of the planned open pit, that constrains the achievable pushback towards the South.

Figure 16-1 displays the pit sectors that were determined based on the geotechnical study for the PFS. Table 16-1 shows the geotechnical design parameters for each pit sector and lithology domain.

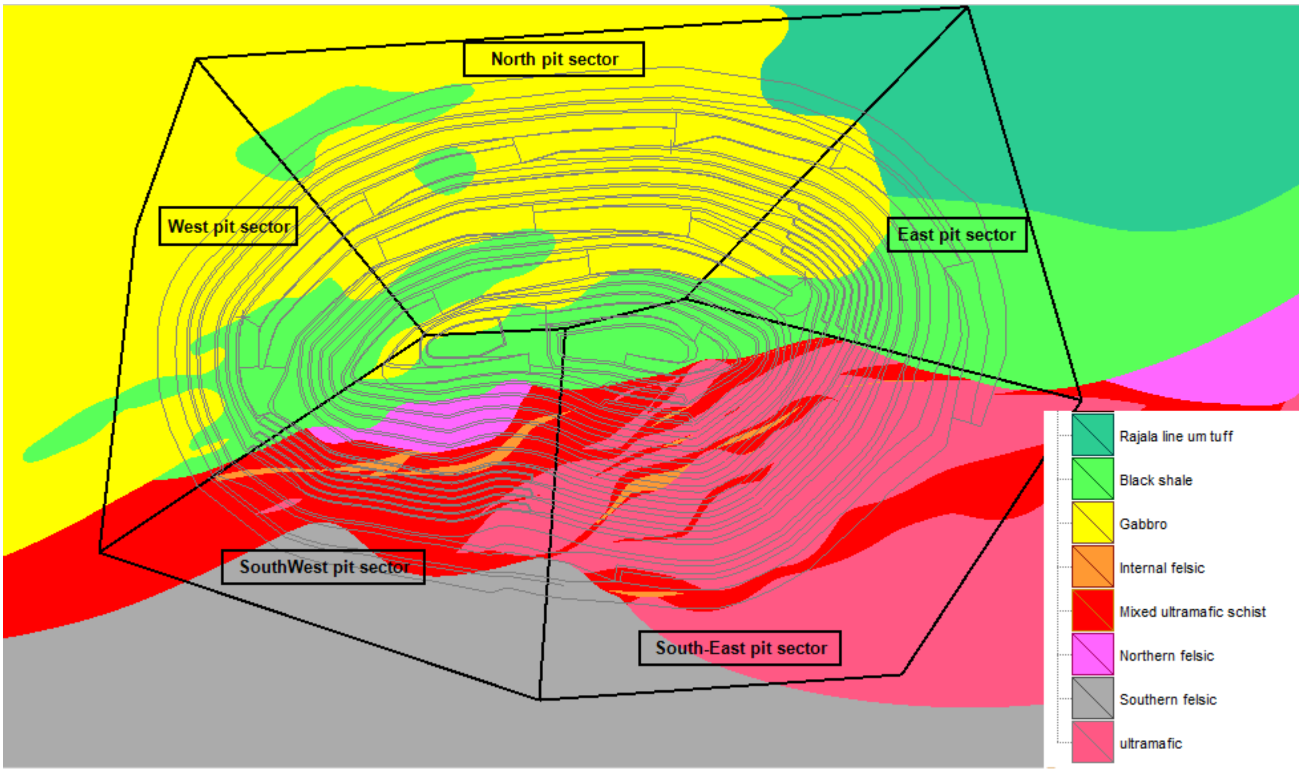


Figure 16-1 – Open Pit Sectors Coloured by Lithology Domains

Table 16-1 – Open pit design parameters

Pit Sector	Domain	Bench height [m]	Bench face angle [°]	Berm width [m]	Ramp width [m]	OSA [°]	IRA [°]
North	Gabbro	20	80	10	25	55	55
	Black shale	20	80	10	25	55	55
East	Gabbro	20	80	10	25	45	45
	Black shale	10*	80	12	25	45	45
South-East	Ultramafic	20/10*	70	10	n/a	50	50/45**
	MSCU & IF	20/10*	80	10	n/a	50	50/45**
South-West	Ultramafic	20/10*	80	10	n/a	50/45**	50/45***
	MSCU & IF	20/10*	80	10	n/a	50/45**	50/45***
	Northern felsic	20	80	10	n/a	50/45**	50/45**
West	Northern felsic	20	80	10	25	55	55
	Black shale	20	80	14	25	55	55

Pit Sector	Domain	Bench height [m]	Bench face angle [°]	Berm width [m]	Ramp width [m]	OSA [°]	IRA [°]
	Gabbro	20	80	10	25	55	55
Overburden	-	-	-	-	-	33	-

* Bench height 10m within fault domains.

**45° within fault domains.

*** 45° for the first 60m vertical.

16.2.5. MINE DESIGN PARAMETERS

The mine design criteria and parameters for computation are based on data gathered during previous PEA studies, recommendations from geotechnical analysis, first principles estimations and WSP industry experience and are summarised in the sections below.

Ore and waste material properties utilised in the mine design are outlined below:

- Ore in-situ density: 2.8 to 3.0 t/m^3.
- Waste in-situ density: 2.6 to 2.9 t/m^3.
- Swell = 40%.

16.2.6. OPEN PIT DESIGN

Stage 1 and 2 pits were designed in accordance with Table 16-1.

Ramp Placement

The ramp placement for the open pit design was to maximise the overall slope angle of the south wall, that is constrained by the permit boundary, to maximise the gold recovered from the open pit. The ramp starts from the Northeast corner of the open pit to minimise the distance from the ramp exit to the processing plant and waste rock co-disposal impoundment. The ramp includes two switchbacks that have been designed with a 60 m width. Figure 16-2 shows the estimated switchback dimensions to fit passing trucks. The switchback dimensioning was done with the largest truck size in consideration in the equipment selection. The switchbacks were located on the east and west ends of the pit. The designed ramp width was 25 m, which is suitable for two-way traffic for 140 t trucks.

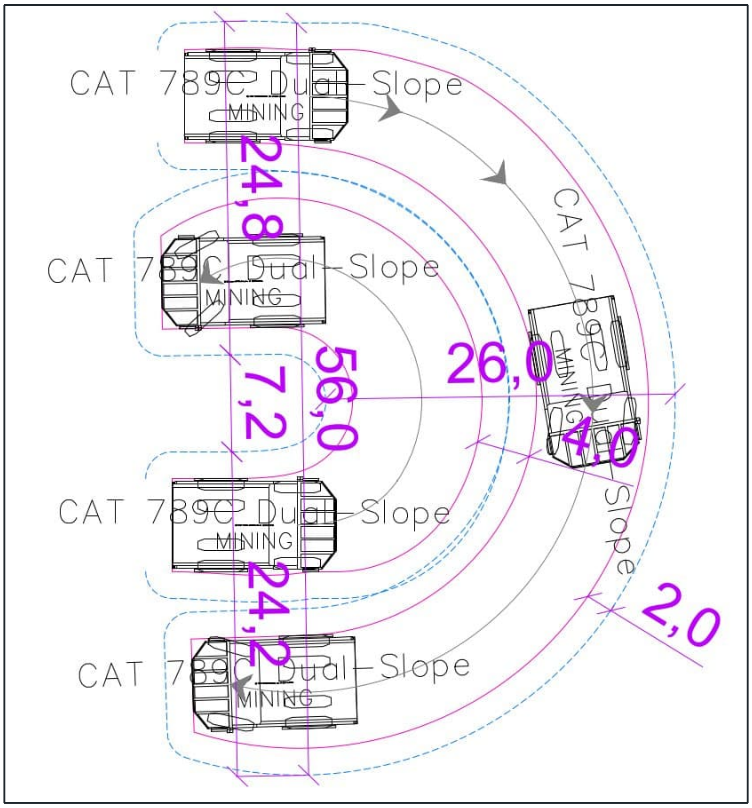


Figure 16-2 – Ramp Switchback Dimensioning (Units in Metres)

Two and three pit pushback stages were evaluated to guide the scheduling of the open pit mining. Two stages were found to be the most practical mining schedule. Although, three pit stages would provide more control on the amount of waste stripping especially during the first years, three stages were found to produce an impractical open pit mining sequence.

Stage 1 Pit Design

The Stage 1 design is based on the RF 0.17 pit shell. This pit shell was selected because it is a sufficient size to function as an initial pushback. Larger RF shells did not leave sufficient mining widths to the final pit, particularly on the Northern side.

The stage 1 design was extended South to include the final pit wall. The stage 1 design and RF 0.17 pit shell is shown in Figure 16-3. The stage 1 pit design is 180 m deep, 640 m long and 620 m wide. Figure 16-4 displays the stage 1 design in plan view, and Figure 16-5 displays the stage 1 design with the stage 2 design.

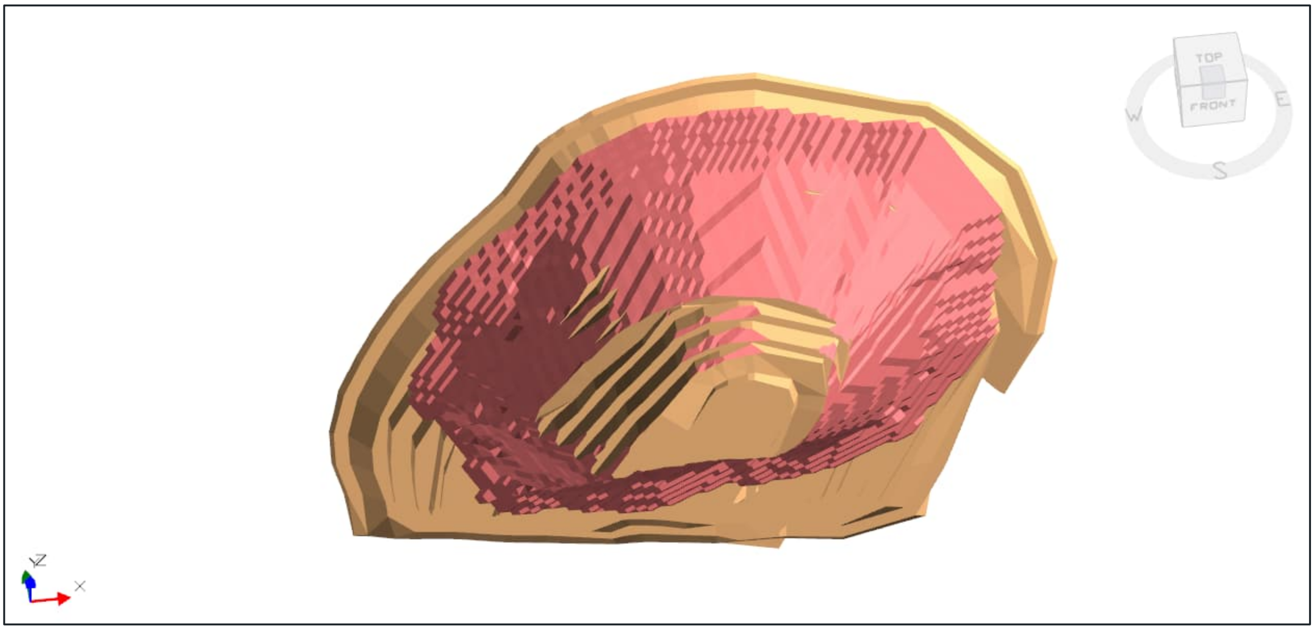


Figure 16-3 – Stage 1 Pit Design (Gold) compared to RF 0.17 Pit Shell (Red)

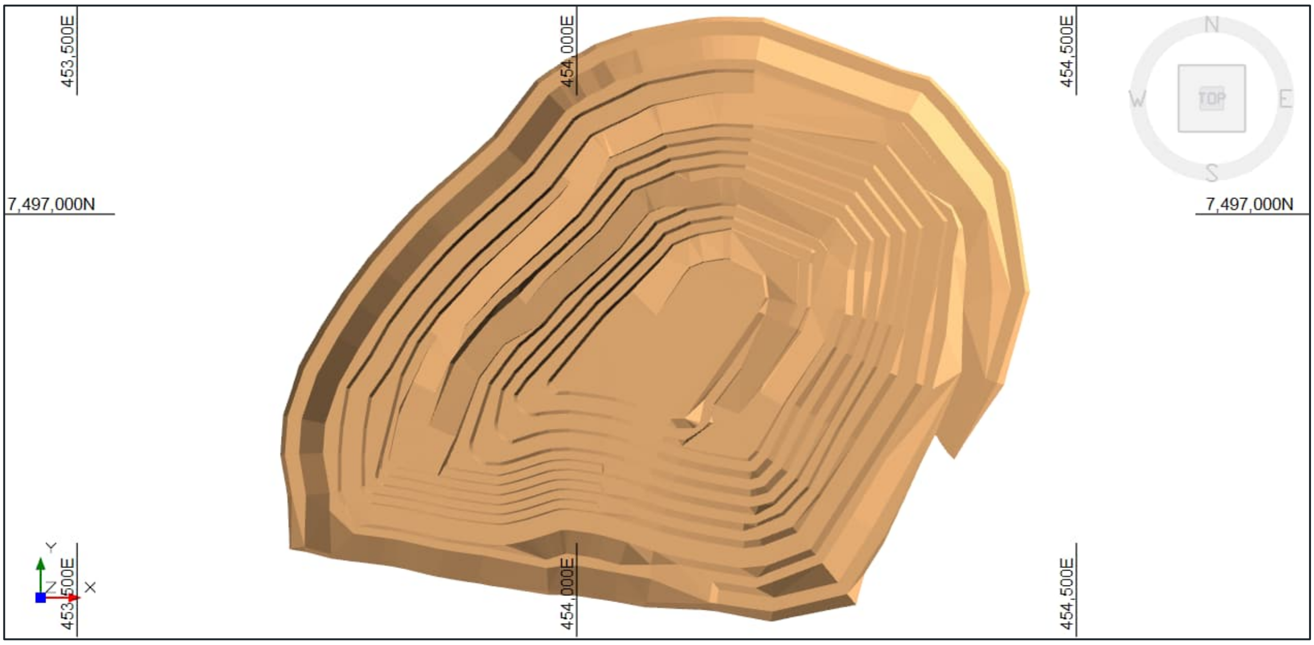


Figure 16-4 – Stage 1 Pit Design in Plan View

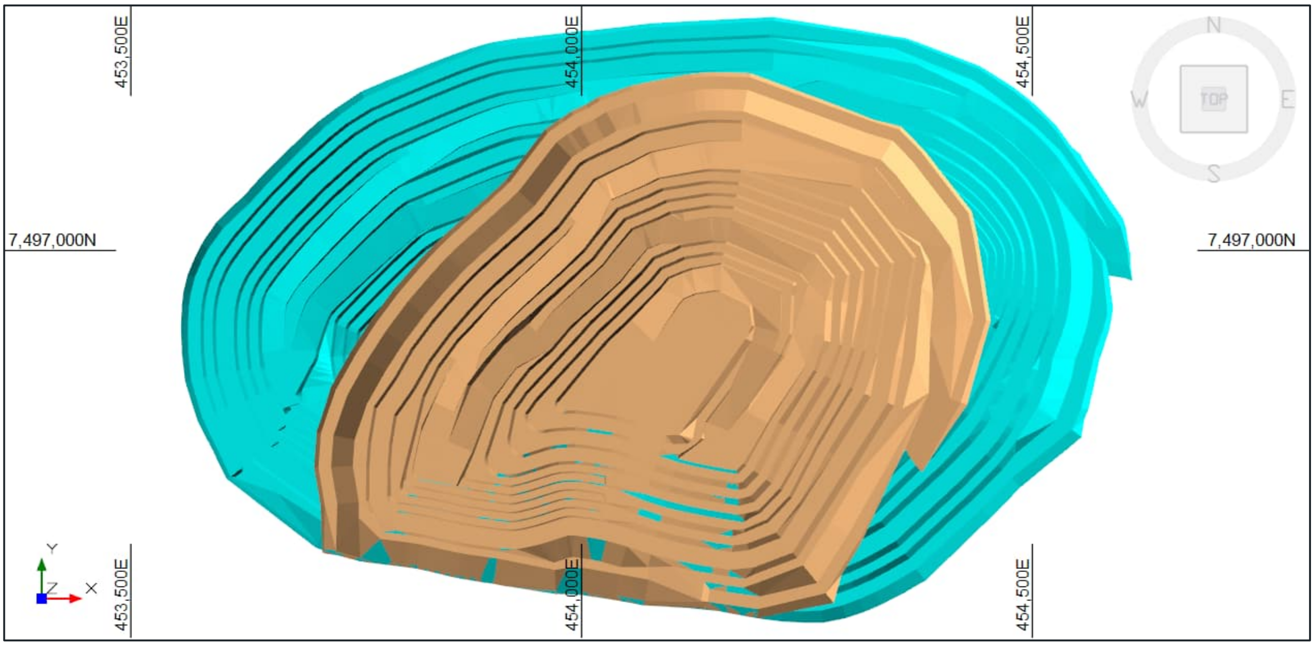


Figure 16-5 – Stage 1 (Gold) and Stage 2 (Blue) Pit Designs in Plan View

Stage 2 Pit Design

The pit shell with a RF of 0.80 was selected as the basis for the stage 2 (final) pit design. The stage 2 design follows the pit shell very closely especially on the South wall to maintain the 20 m offset to the permit boundary. The RF 0.80 pit shell and stage 2 pit design are show in Figure 16-6 and Figure 16-7.

The stage 2 pit design reaches down to -80 m elevation. The open pit base could be extended further downward by including more waste on the Northern side of the pit. The stripping ratio for this extension would be approximately 1:10 which is close to the inflexion point stripping ratio where underground mining becomes more economically positive than open pit mining. Thus, a decision was made to leave that ore to be mined by underground mining methods.

The stage 2 pit design is 300 m deep, 1.03 km long and 660 m wide. The pit design is shown in Figure 16-8. There is a bulge on the south pit wall, which results from a thicker area of overburden.

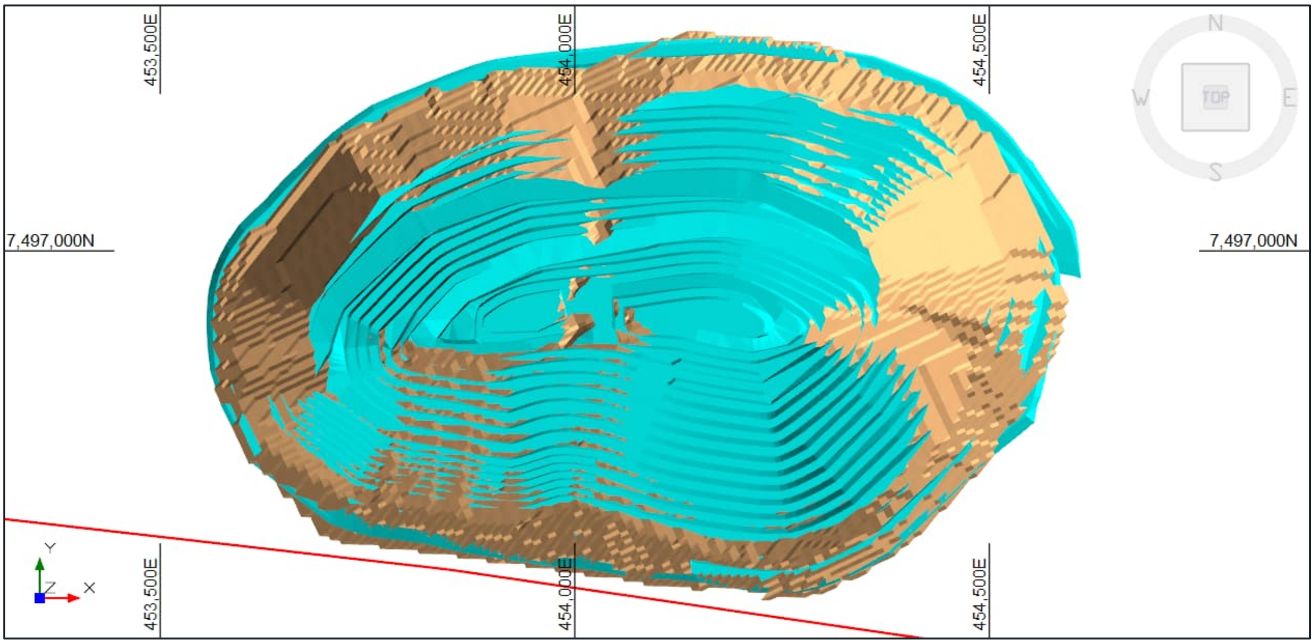


Figure 16-6 – Stage 2 Pit Design (Blue) with RF 0.80 Pit Shell (Gold) and Mining Licence (Red)

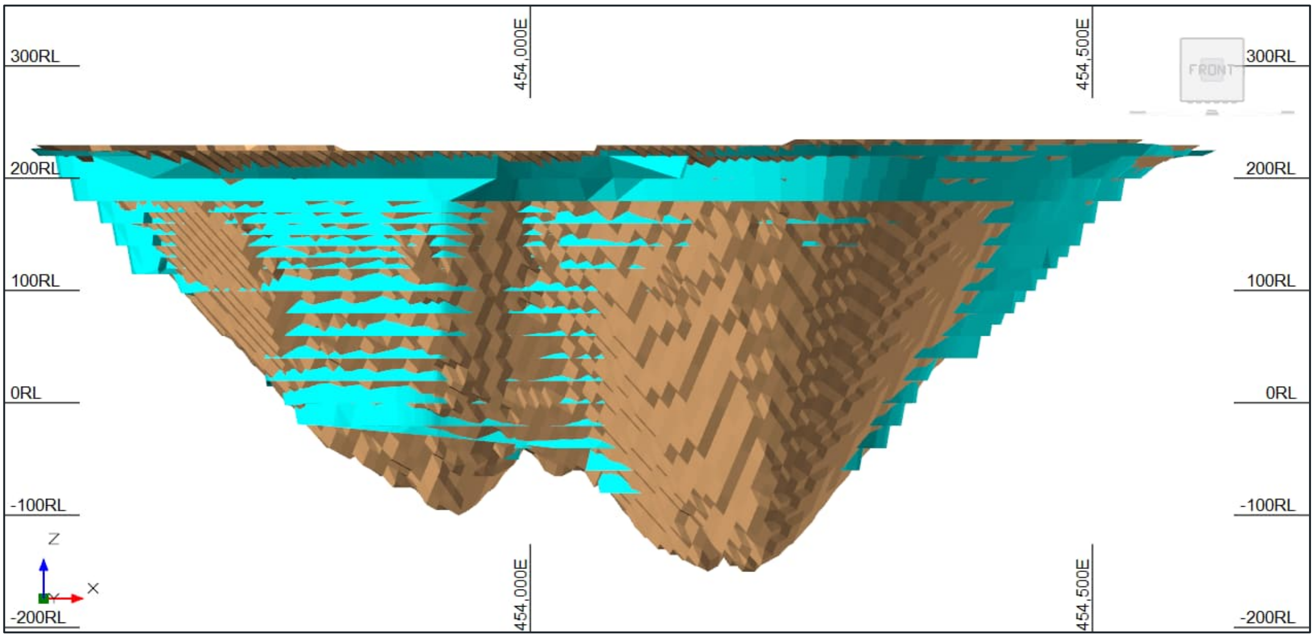


Figure 16-7 – Stage 2 Pit Design (Blue) and RF 0.80 Pit Shell (Gold)

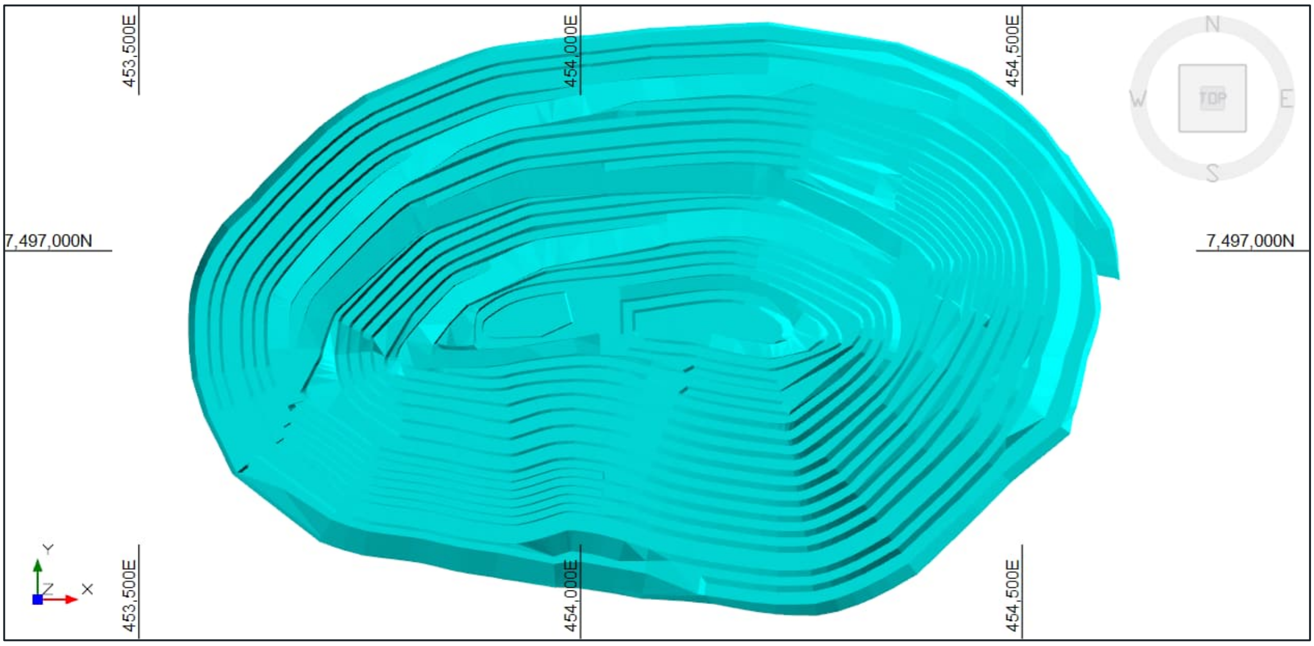


Figure 16-8 – Stage 2 Pit Design in Plan View

16.2.7. OPEN PIT SCHEDULE

The open pit main scheduling targets were to provide:

- Maximising NPV;
- Consistent tonnage and Au grade fed to the processing plant; and
- Balanced waste mining schedule.

Initial strategic scheduling was done with Geovia's Whittle software to find near optimal material movement rates. The final open pit schedule was generated using Deswik's mine scheduling software.

Pre-stripping performed in the years prior to mining ore for plant feed removes the overburden from the open pit. The pre-stripping activities requires a minimum of 12 Mt of movement which is aligned to site needs for construction of roads, co-disposal facility foundation, RoM pad and river diversion. The 3.5 Mt/a ore feed is achieved in Year 1 and is sustained for 10 years. The material movement schedule is shown in Figure 16-9.

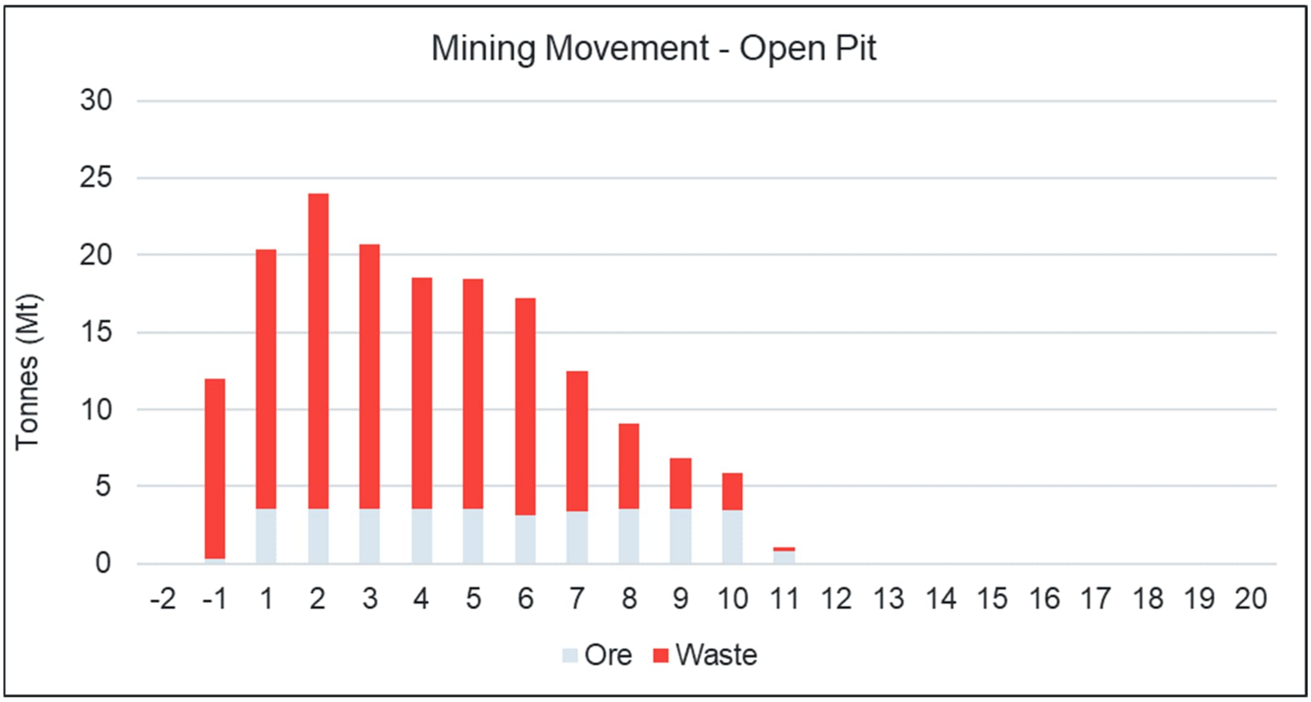


Figure 16-9 – Open Pit Mining Movement

Grade fluctuates over the course of the open pit schedule as shown in Figure 16-10. Stockpiling and blending were utilised in the schedule to ensure consistent feed and grade to the plant, as outlined in Section 16.4.

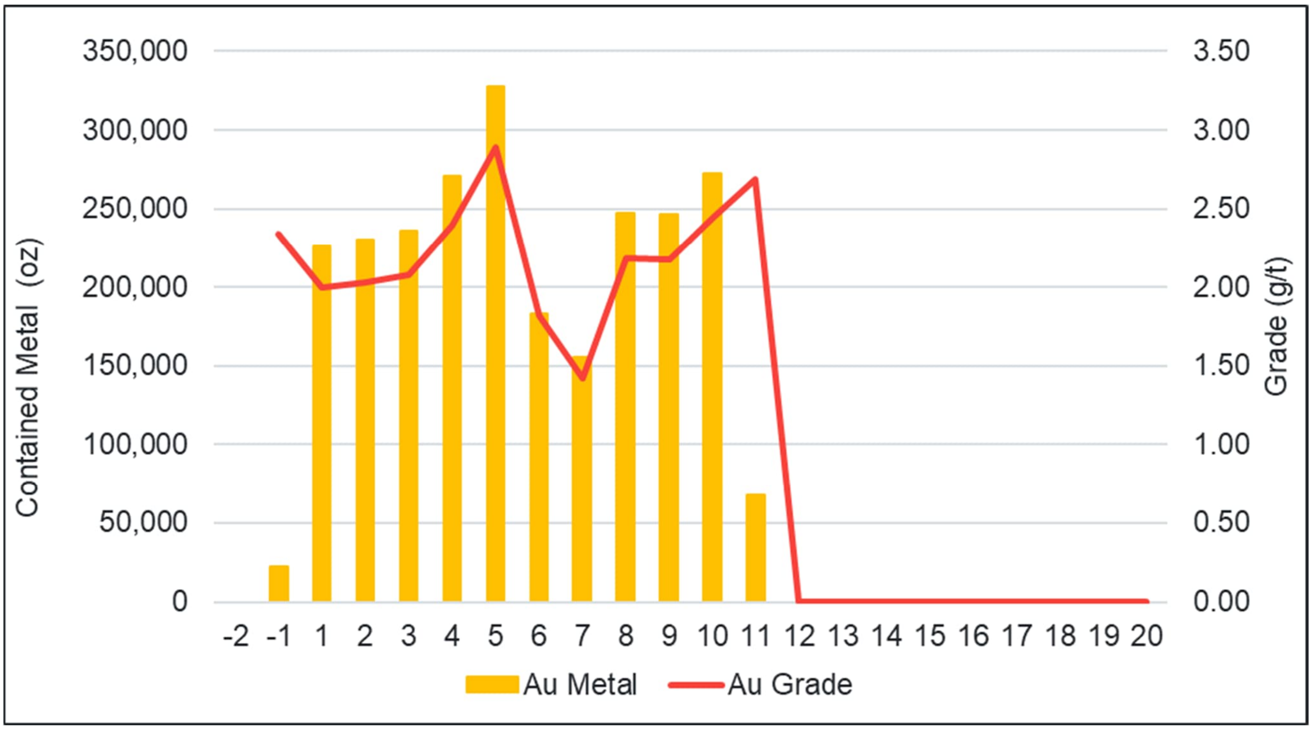


Figure 16-10 – Open Pit Mined Grade and Contained Metal

16.2.8. OPEN PIT MINING EQUIPMENT

Open pit mining operations are planned to be undertaken by a contractor. As part of the PFS, budget quotations were received by European mining contractors. Additional equipment required will be provided by the owner. Equipment to be used for the open pit operations are shown in the table below.

Table 16-2 – Open Pit Equipment and Numbers

Equipment	Model	Number
Mining Contractor		
Shovel	CAT 6030	2
Wheel Loader	CAT 993	1
Wheel Loader	CAT 992	1
Haul Truck	CAT 785	14
Bulldozer	CAT D10 T	1
Bulldozer	CAT 9 T	2
Motor Grader	CAT 18 M	1
Wheel Dozer	CAT 844	2
Drill Rig – Ore	EPIROC DML	3
Drill Rig – Waste	SANDVIK PANTERA 1500	2
Water Truck	CAT 777	1
Owner		
Rockbreaker	KOMATSU PC 390	1
Light Vehicles	TOYOTA HILUX	7
Mobile Lighting Plants	ATLAS COPCO HiLIGHT	10
Fork Lift	MANITOU	1
Crane	LIEBHERR LTM 1030	1
Service Truck	KOMATSU HD 605	1
Fuel Truck	KOMATSU HD 605	1

Equipment numbers for the mining contractor and owner over the LOM are shown in the figures below.

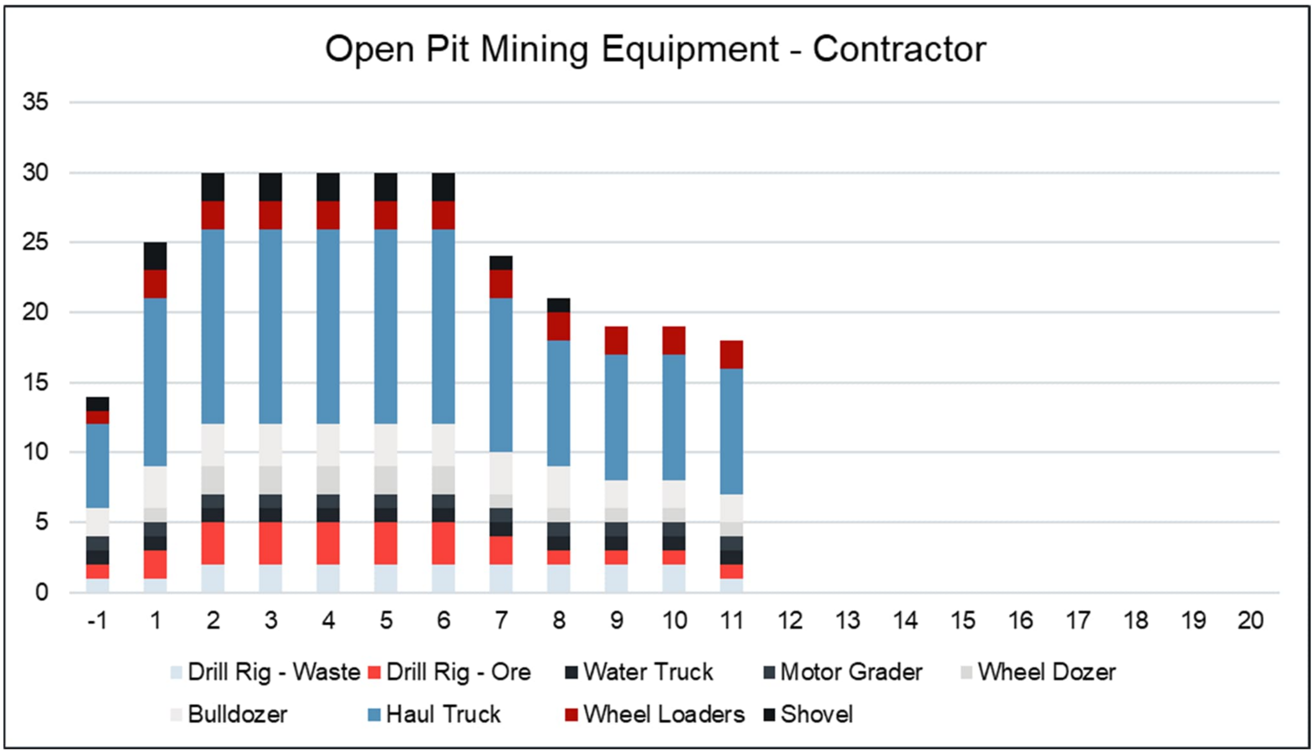


Figure 16-11 – Open Pit Contractor Equipment

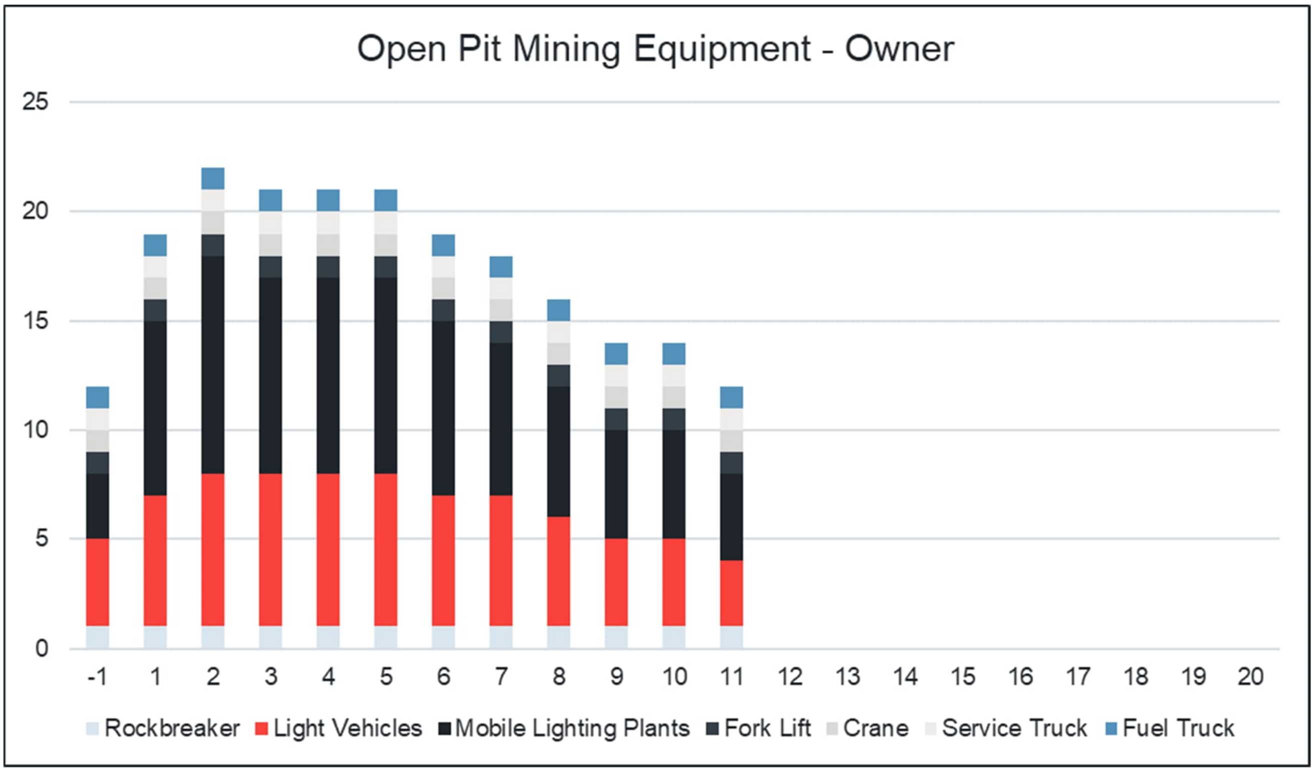


Figure 16-12 – Open Pit Owner Equipment

A top hammer drill rig, with appropriate smaller diameter holes, and higher penetration rate was selected for the drilling of ore. This drill rig is also more agile which is required in the first production years when the undulating bedrock surface is mined.

A top hammer drill rig with larger hole diameter was selected for waste drilling, so that larger blast pattern burden and spacing can be used and where rock mass characteristics allows. The characteristics of the rock mass for optimal fragmentation by blasting has been assessed at PFS level.

No charging and blasting initiation equipment have been included in the equipment listing as this is planned to be supplied by a regional specialist blasting contractor. Cost estimates were received from explosives manufacturers and suppliers to site. The drill and blast parameters are listed in Table 16-3.

Table 16-3 – Open Pit Drill and Blast Parameters

Parameter	Value	Unit
Blast Hole Size (Ore)	115	mm
Average Hole Length (Ore)	11	m
Average Charge Length (Ore)	1.5	m
Burden (Ore)	3.0	m
Spacing (Ore)	3.5	m
Powder Factor (Ore)	0.4	kg/t
Waste Blast Hole Size (Waste)	171	mm
Average Hole Length (Waste)	11	m
Average Charge Length (Waste)	2.0	m
Burden (Waste)	4.8	m
Spacing (Waste)	6.2	m
Powder Factor (Waste)	0.3	kg/t

A reverse circulation (RC) drill rig has been selected for grade control drilling in the open pit. Grade control drilling is planned on a 10m x 10m x 10m grid with 45° angled holes over the entire deposit with spacing decreasing to 10m x 10m x 5m at the ore-waste contacts. The total LOM grade control drilling meters were estimated at 321 000m round to 330 000m to allow some contingency.

Productivity Estimates

Equipment productivity estimates have been built up by computation of the truck, shovel cycle times based on WSP data base, OEM information and first principal equipment performance derivation encompassing the open pit geometry, regional climate and operational regime. These are summarised in Table 16-4.

Table 16-4 – Operational Parameters for Open Pit Productivity

Operational Parameters	Value	Units
Working Days per Year	360	days
Shift Length	12	hours
No. of Shifts	2	#
Available Time per Shift	11	hours
Annual Effective Work Hours	7,920	hours
Annual Leave	18	days
Sick Leave	5	days
Absenteeism	3	days
Training	5	days
Allowance	9	%

Equipment Maintenance

Mobile equipment will be maintained in a site workshop for maintenance and repairs. Regular maintenance work on drill rigs and shovels will be made in the pit and will only be taken to the workshop for more extensive maintenance work and overhauls. The number of overhauls and replacement units have been estimated based on WSP Data base, OEM estimates and first principal computations.

16.2.9. OPEN PIT PERSONNEL

Open pit mine personnel have been calculated based on the required equipment numbers for production personnel and based on required administration, technical, managerial and engineering roles. During peak open pit production, it is estimated that at peak operations approximately 285 people will be required. The open pit production personnel numbers per year are shown in Figure 16-13.

The open pit mine production will be operated 24 hours year-round. The production will be operated with five teams as shown in Table 16-5. The production team rotation is based on Finnish working hour regulations, with an appropriate allowance for training, annual vacations and sick leave (Table 16-4).

Table 16-5 – Open Pit Shift Schedule

Work Day	Team 1	Team 2	Team 3	Team 4	Team 5
Monday	6-18		-	-	18-6
Tuesday	6-18		-	-	18-6
Wednesday	18-6	6-18	-	-	

Work Day	Team 1	Team 2	Team 3	Team 4	Team 5
Thursday	18-6	6-18	-	-	
Friday	-	18-6	6-18	-	
Saturday	-	18-6	6-18		
Sunday	-	-	18-6	6-18	
Monday	-	-	18-6	6-18	
Tuesday	-	-			6-18
Wednesday	-	-			6-18

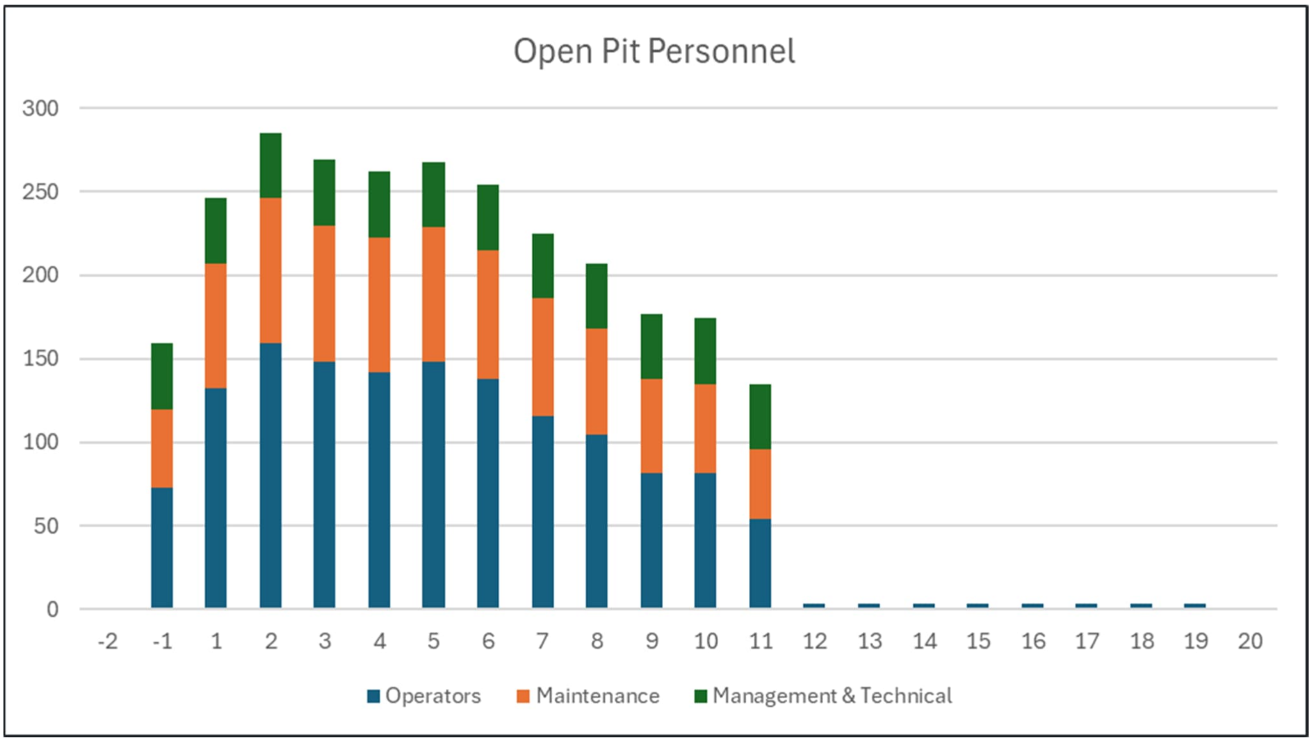


Figure 16-13 – Open Pit Production Personnel

16.3 UNDERGROUND MINING

The Ikkari Underground analysis evaluates the development of the Ikkari Underground Mineral Reserve. Access to the orebody is initially through a decline with portal access on surface. Access is also established later in the LoM schedule through a decline from the open pit. The decline intersects the mid-point of the orebody, where key mine and ventilation infrastructure will be developed. Four shafts are required, two for exhaust and two for the provision of fresh air. A 2.0 Mt/a production rate is achieved in the underground mine.

16.3.1. MINING METHOD SELECTION

At the commencement of the study two mining methods were assessed. These mining methods were derived from the PEA (Tetratec, 2023) and environmental studies. The process of selecting the underground mining method(s) to apply to the Resource considered several modifying factors, including:

- Deposit geometry and size;
- Geotechnical and hydrological parameters;
- Production rate;
- Open pit to underground transition; and
- Proximity to adjacent property boundary.

In the PEA (Tetratec, 2023), a Sub Level Caving (SLC) mining method at a 3 Mt/a production rate was selected primarily based on achieving maximum NPV and Internal Rate of Return (IRR). Review of the SLC mining method showed it was not a favourable method due to:

- The effect of subsidence and its interaction with the adjacent property resulting in a significant loss to the mineable inventory. As shown in Figure 16-14, material below the subsidence planes would likely cause ground movement in the adjacent property not owned by Rupert Resources;
- The requirement for a crown pillar, if underground mining at the same time as the open pit. Reducing the mineable inventory further; and
- Variability in mineralisation and lack of selectivity meaning large quantities of internal waste would need to be caved and mined.

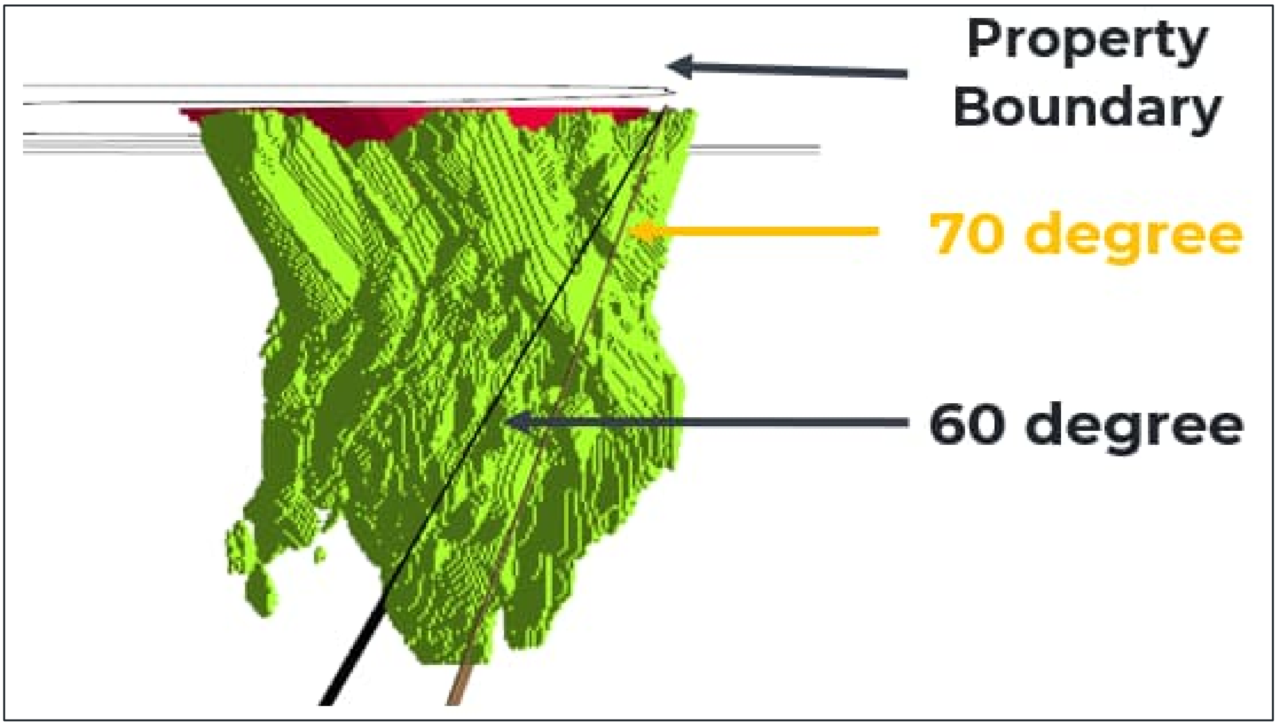


Figure 16-14 – Subsidence Plane Projected from Property Boundary at 60 and 70 Degrees with Underground Resource Model (Light Green)

The mining method of LHOS with backfill was evaluated. A transverse stope and footwall development arrangement was implemented due to orebody width reaching up to 140 m.

Transverse stoping extracts ore in blocks perpendicular to the strike of the orebody. Access is established in sublevel tunnels above and below the stope block, perpendicular to the orebody strike. Top access is used for production drilling of the block, whilst the bottom drive is used for loading using Load-haul-dump units (LHDs).

LHOS has additional benefits by:

- Allowing the simultaneous development of the open pit and underground deposit which optimises the transition phase;
- Eliminating the requirement for an open pit crown pillar by implementing an underground bottom-up mining direction which maintains a suitable crown pillar with the open pit until open pit mining has ceased;
- Maximising the mineable inventory through the backfilling of mined stopes to avoid subsidence and surface disturbance; and
- Providing the selectivity required with the variability in mineralisation and grade.

16.3.2. GEOTECHNICAL CONSIDERATIONS

For the underground mine stope dimensioning and scheduling it is necessary to understand the development of stresses around the excavations throughout the life of mine. A 3D elastic boundary element analysis was performed with EX3 (computerised geotechnical software) to estimate the mining-induced stresses, considering interaction between underground mining areas and the open pit. The mine schedule was emulated by calculating the stresses in one-year intervals, adding more stopes in each period as scheduled. Maximum induced stresses on stope surfaces have been assessed and were used as input to empirical stope design workflows. Mining induced stresses on underground infrastructure were evaluated with a combination of 3D stress analysis and analytical analyses to estimate ground support requirements.

Analysis of stresses on stope surfaces were done for various sectors (Northern Felsic Orebody, Main Orebody, and Main Orebody Near Pit) and for three zones within the Main Orebody (top, middle and bottom zone). Figure 16-15 and Figure 16-16 visualise how these sectors have been defined.

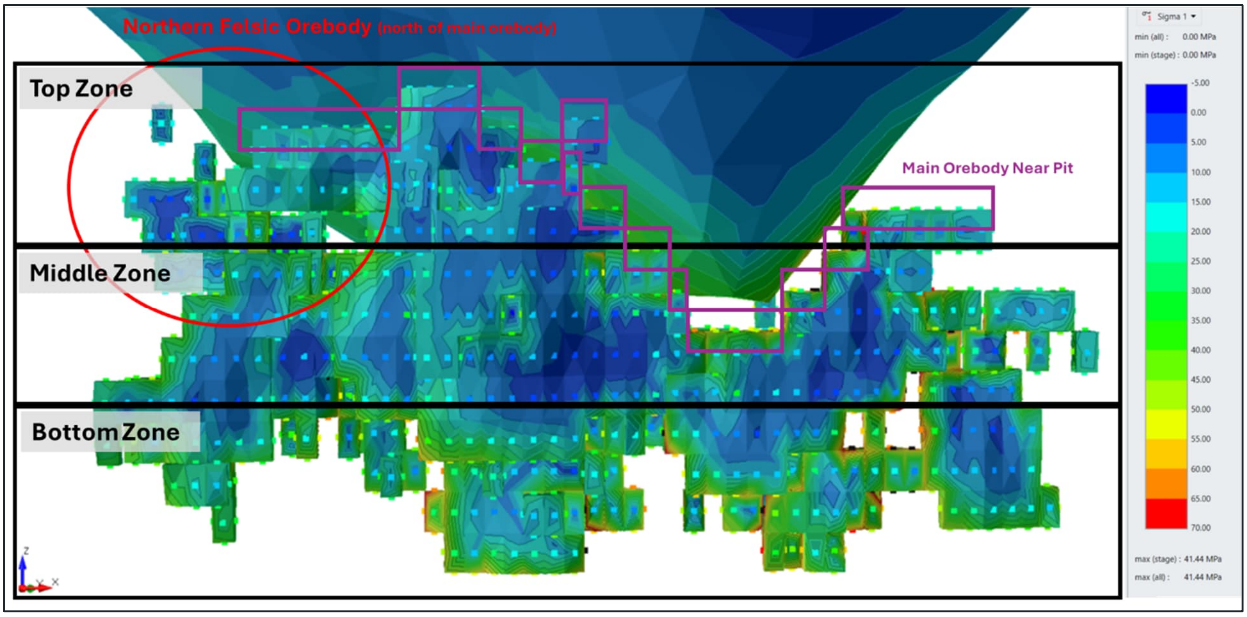


Figure 16-15 – Definition of Sectors and Zones looking North-West

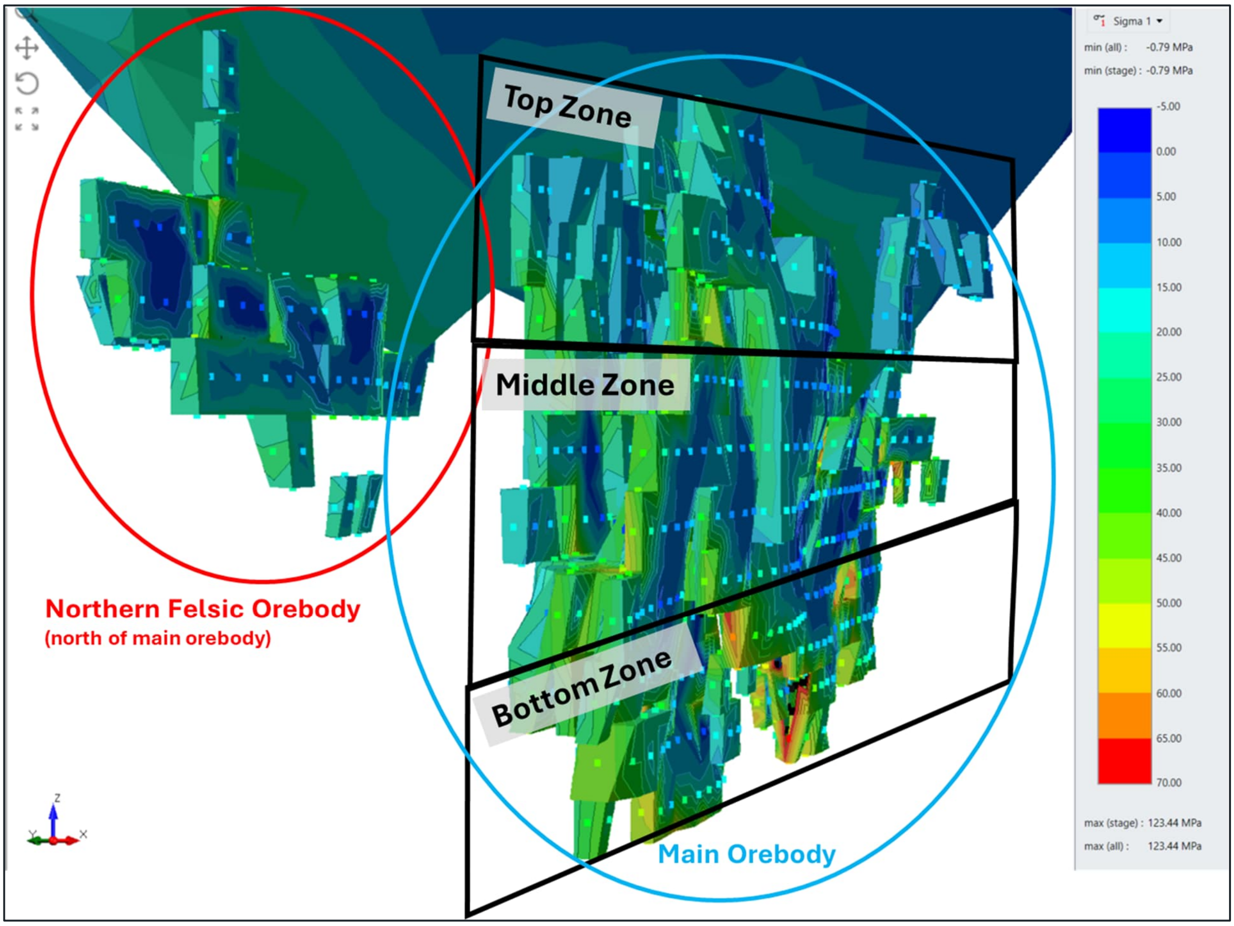


Figure 16-16 – Perspective View with Definition of Sectors and Zones

The advantages and disadvantages of various stope sequences have been evaluated. A general triangular retreat shape using a primary and secondary stope arrangement was selected with a mining direction away from the Southern fault zone.

The rock mechanics inputs for underground stope design have been selected to achieve a reliable and robust mine design and extraction sequence, using rock mass quality values (Q') between the 25th and 50th percentile. Two empirical design methodologies and associated criteria have been used to determine stable stope sizes for primary and secondary stope lines:

- Unsupported stope walls and unsupported stope backs need to be in or above the unsupported transition zone from Potvin (1988), as well as meeting or exceeding the 70% isoprobability line as defined by Mawdesley (2001).
- The supported transition line as defined by Nickson (1992) has been selected as design limit for supported stope backs.

Table 16-6 shows the maximum primary stope lengths and the minimum required cable bolt length and density for primary and secondary stope backs in various rock types and depth ranges across the orebody. It was determined that 50m long stopes on secondary lines are feasible if cable bolt length and density in the stope back are increased to account for the increased hydraulic radius of the stope hanging wall (back). Cable bolts should be fully grouted and plated, have a capacity of 25 t, and not installed at an angle that is more than 45° from vertical.

The final brow of each stope is subject to wear & tear during production and requires additional reinforcement. If cable bolts are already installed (because the ore drive used to serve as an overcut for the stopes below and has now transitioned into an undercut for the stopes above) no action is required. Cable bolts for bottom levels in the mine or domains that do not require cables in stope backs will require 8 x 6 m single cables (25 t) at the final brow of each stope.

Table 16-6 – Stopes sizes, cable bolt length, and cable bolt densities for stope backs

Rock types	Sector Name	Zone	Stope Height (m)	Stope Width (m)	Maximum Stope Length Primaries	Cable bolt length primaries (m)	Cable bolt density in stope back primaries (bolts/m^2)	Cable bolt length secondaries of 15 x 50 (m)	Cable bolt density in roof secondaries for 50 m strike length (bolts/m^2)
Internal felsic	Main Orebody Near Pit	Top	30	15	50	No cable bolts in stope back		No cable bolts in stope back	
Internal felsic	Main Orebody	Top	30	15	72	No cable bolts in stope back		No cable bolts in stope back	
Internal felsic	Main Orebody Near Pit	Middle	30	15	36	11.0	0.18	11.9	0.18
Internal felsic	Main Orebody	Middle	30	15	36	11.0	0.1	11.9	0.1
Internal felsic	Main Orebody	Bottom	30	15	28	10.3	0.1	11.9	0.1
Mixed Ultramafic Schist	Main Orebody Near Pit	Top	30	15	34	10.9	0.1	11.9	0.18
Mixed Ultramafic Schist	Main Orebody	Top	30	15	34	No cable bolts in stope back		11.9	0.1
Mixed Ultramafic Schist	Main Orebody Near Pit	Middle	30	15	20	9.2	0.22	11.9	0.22
Mixed Ultramafic Schist	Main Orebody	Middle	30	15	20	9.2	0.18	11.9	0.18
Mixed Ultramafic Schist	Main Orebody	Bottom	30	15	16	8.4	0.18	11.9	0.22
Northern Felsic	Northern Felsic Orebody	Top	30	15	20	No cable bolts in stope back		11.9	0.1
Ultramafic	Main Orebody Near Pit	Top	30	15	24	9.8	0.18	11.9	0.22
Ultramafic	Main Orebody	Top	30	15	24	9.8	0.1	11.9	0.18
Ultramafic	Main Orebody Near Pit	Middle	30	15	14	8.0	0.22	11.9	0.24
Ultramafic	Main Orebody	Middle	30	15	14	8.0	0.22	11.9	0.22
Ultramafic	Main Orebody	Bottom	30	15	11	7.1	0.22	11.9	0.24

Dilution from stope sidewalls has been estimated based on recorded case histories at other mines, captured in the ELOS parameter, which is the volume of rock failed from the stope hanging wall (HW) divided by the HW area, thus representing an average depth of failure over the HW surface. ELOS values per rock type, sector and zone are summarised in Table 16-7.

Table 16-7 – Equivalent Linear Overbreak Slough Assessment

Rock types	Sector Name	Zone	ELOS
Internal felsic	Main Orebody Near Pit	Top	0.7 m
Internal felsic	Main Orebody	Top	0.7 m
Internal felsic	Main Orebody Near Pit	Middle	0.8 m
Internal felsic	Main Orebody	Middle	0.8 m
Internal felsic	Main Orebody	Bottom	0.9 m
Mixed Ultramafic Schist	Main Orebody Near Pit	Top	1.0 m
Mixed Ultramafic Schist	Main Orebody	Top	1.0 m
Mixed Ultramafic Schist	Main Orebody Near Pit	Middle	1.3 m
Mixed Ultramafic Schist	Main Orebody	Middle	1.3 m
Mixed Ultramafic Schist	Main Orebody	Bottom	1.6 m
Northern Felsic	Northern Felsic Orebody	Top	0.9 m
Ultramafic	Main Orebody Near Pit	Top	1.4 m
Ultramafic	Main Orebody	Top	1.4 m
Ultramafic	Main Orebody Near Pit	Middle	1.7 m
Ultramafic	Main Orebody	Middle	1.7 m
Ultramafic	Main Orebody	Bottom	2.2 m

Table 16-8 summarises paste backfill strength requirements for primary and secondary stope lines in a mine design without sill pillars. It is based on the load of its own weight when adjacent stopes are open, exposing the paste backfill wall. It is assumed that no load (surcharge) is transferred onto the paste walls when they are temporarily exposed. Small surcharges will be covered by a safety factor in the design and even larger additional loads would unlikely result in large-scale instability but rather as sloughing of paste into the non-entry stopes which can be managed operationally.

Target paste fill strength in the plant should be higher to account for strength loss during transportation and to ensure there is an acceptable small number of batches that would fall below the criterium, resulting in a backfill product that consistently and reliably meets strength requirements. It is advised to design a paste fill recipe that meets or exceeds the strength requirements in Table 16-8 in at least 84% of the cases (mean – standard deviation). This approach is sufficiently conservative to prevent a large increase in operational costs due to higher cement usage in more detailed cost estimates within the Feasibility Study stage or during Detailed Design. Secondary stope lengths should be limited to 50m strike length.

Table 16-8 – Summary of Paste Backfill Strength requirements which need to be met in at least 84% of the cases

Primary Stopes	640 kPa
Secondary Stopes	320 kPa

Hadjigeorgiou and Potvin (2016) compared ground support standards from 45 Ground Control Management Plans (GCMPs) in North America and Australia and presented guidelines for preliminary design of reinforcement and surface support for drives in metalliferous mines at the early stages of mine design. Table 16-9 shows 5 different ground support classes for ranges of Q values, based on their guidance. They should be seen as input to economic assessments for this Project and are not designed.

Table 16-9 – Empirical Ground Support Classes

Support Class	Q	Minimum Bolt Density (bolts/m^2)	Reinforced Shotcrete Thickness (mm)	Wall Support Coverage
1	0.001-0.2	0.65	100	To floor
2	0.2-1	0.65	75	To floor
3	1-4	0.50	50	Mid-drift
4	4-10	0.45	50	Shoulder
5	10-400	0.40	50	Shoulder

The relative application of ground support classes per drive type and year were estimated as percentages. These percentages are based on preliminary infrastructure designs and P25, P50, and P75 (Percentiles) Q' values for relevant geotechnical domains. The Q' values were used together with the Joint Water Reduction Factor (Jw; based on expected hydrogeological conditions) and the Stress Reduction Factor (SRF_{74}; based on mining-induced stress from EX3 along infrastructure design polylines) to estimate a Q value ($Q = Q' * (J_w/SRF)$).

For ramps & declines, footwall drives, and ore drives that are not highly stressed, the following ground support elements can be applied on the walls and back:

- Use of 2.4 m long, 20 mm diameter, fully resin grouted rebar with spherical seats as per Table 16-9; and
- Use of #6 gauge welded wire mesh or fibre reinforced shotcrete as per Table 16-9.

Installation of spiling bolts in front of the working face to avoid overbreak or cave in is required in ground support class 1 ($Q < 0.2$). Yielding rock bolts and cable bolt anchors may need to be included in ground control categories for highly stressed ground conditions. This could be applicable to 7% of declines & ramps and 15% of footwall drives based on numerical modelling. Dynamic ground support design will need to be completed during detailed design and updated based on encountered operational ground conditions. The 3-way and 4-way tunnel; intersections require six and nine twin strand cable bolts (2 x 25 t) of 6 m length, respectively.

A Preventative Support Maintenance (PSM) program can prevent complete rehabilitation of underground excavations. PSM is less time consuming and costly than complete rehabilitation where ground support is stripped out and tunnels / drifts must be re-supported. Mining-induced stress changes from EX3 have been used to estimate changes in brittle rock mass damage and associated PSM and rehabilitation requirements per drive type and year of the LOM for costing purposes of ground support.

Just-in-time development is recommended to reduce costs related to preventative maintenance of ground support and rehabilitation.

16.3.3. HYDROGEOLOGICAL CONSIDERATIONS

The physical hydrogeological considerations in the underground are similar to the deep open pit considerations, with groundwater movement restricted primarily to fault zones within the IFIZ of Ikkari shear zone (Piteau, 2024). As the underground mine is, however, not directly exposed to rain, snow and freezing conditions applicable to the open pit, these are significantly less relevant to the underground mine, until the very late stages of the underground operations when stoping operations break through to the open pit and lead to a direct connection to the underground mine. Continued dewatering of the open pit for as long as possible from the open pit sump is required and once this becomes inaccessible due to potentially unstable pit floor conditions and increased leakage from the pit sump, this water may need to be appropriately managed in the underground through increased pumping capacity or possibly allowing lower mined-out areas of the mine to flood towards the end of LOM in a controlled manner.

As the underground mine develops from surface from Year 6, cover drilling will be required to determine precise locations of water bearing fault structures. Although the location of the surface portal should generally be dewatered, shallow groundwater associated with weathered bedrock and fluvio-glacial sediments zone may be encountered.

At greater depth, underground development may intersect fault zones and geological structures that may still contain significant water, which must be either grouted or allowed to drain, depending on the pressures and flows intersected in individual cover drill holes at the time.

At depths below the peripheral wells in the underground development, the risk of higher inflows increases further and will require further drilling safety precautions to guard against high pressure water intersections, sufficient grouting capacity as well underground pump availability and capacity to deal with the increasing water volume intersections in declining development ends.

Piteau (2024) estimate that underground inflows are expected to peak in Year 10 of the overall LOM, at approximately 7 800 m^3/day (90l/s), when mine development reaches its deepest and levels. Following the closure of the open pit mine, peripheral pit dewatering wells will need to continue to operate but dropping to approximately 55% of the overall dewatering effort required by

Year 12. Underground mine dewatering decreases slightly after Year 10 to approximately 6 300 m^3/day (73 l/s) by the end of LOM.

16.3.4. MINE DESIGN PARAMETERS

Development Design

Design criteria and assumptions are based on data gathered during previous studies, recommendations from geotechnical analysis, first principles estimations and industry experience and are summarised in the sections below.

Ore and waste material properties utilised in the mine design are outlined below:

- Ore in-situ density: 2.8 to 3.0 t/m^3;
- Waste in-situ density: 2.6 to 2.9 t/m^3; and
- Swell = 40%

Design features for the lateral development include:

- Ramps level off at each sublevel over a 40 m distance to reduce risk of rollovers; and
- Footwall and ore drive development inclined towards sumps located at the level access to the allow for water to drain.

Following the geotechnical considerations outlined and productivity requirements the mines underground lateral and vertical development sizes are listed in Table 16-10. All lateral development follows the same profile except for the maintenance workshop.

Table 16-10 – Mine Design Parameters - Development

Lateral Development	Section Profile
Lateral Development	5.0 mW x 5.15 mH Arched (Figure 16-17)
General Workshop Bay	6.0 mW x 5.15 mH Arched
Maintenance Bay	8.0 mW x 9.0 mH Arched
Service Bay	7.0 mW x 9.0 mH Arched
Maximum Ramp Gradient	1 in 7
Minimum Curve Radius	25 m
Vertical Development	**Section Profile (Diameter)**
North Fresh Air Raise (FAR)	4.0 m
South FAR	2.5 m
Exhaust Air Raise (EAR)	3.0 m
Escape Raise	2.0 m

A powder factor of 1.4 kg/t was implemented for lateral development. Table 16-11 shows the assumed drill and blast parameters. The planned drill design for lateral development is shown in Figure 16-17.

Table 16-11 – Development Drill and Load Parameters

Parameter	Value	Unit
Blast Hole Size	48	mm
Number of Blastholes	54	#
Reamer Size	125	mm
Number of Reamer holes	6	#
Target Powder Factor	1.4	kg explosive/broken t
Average Drill Length	4.6	m

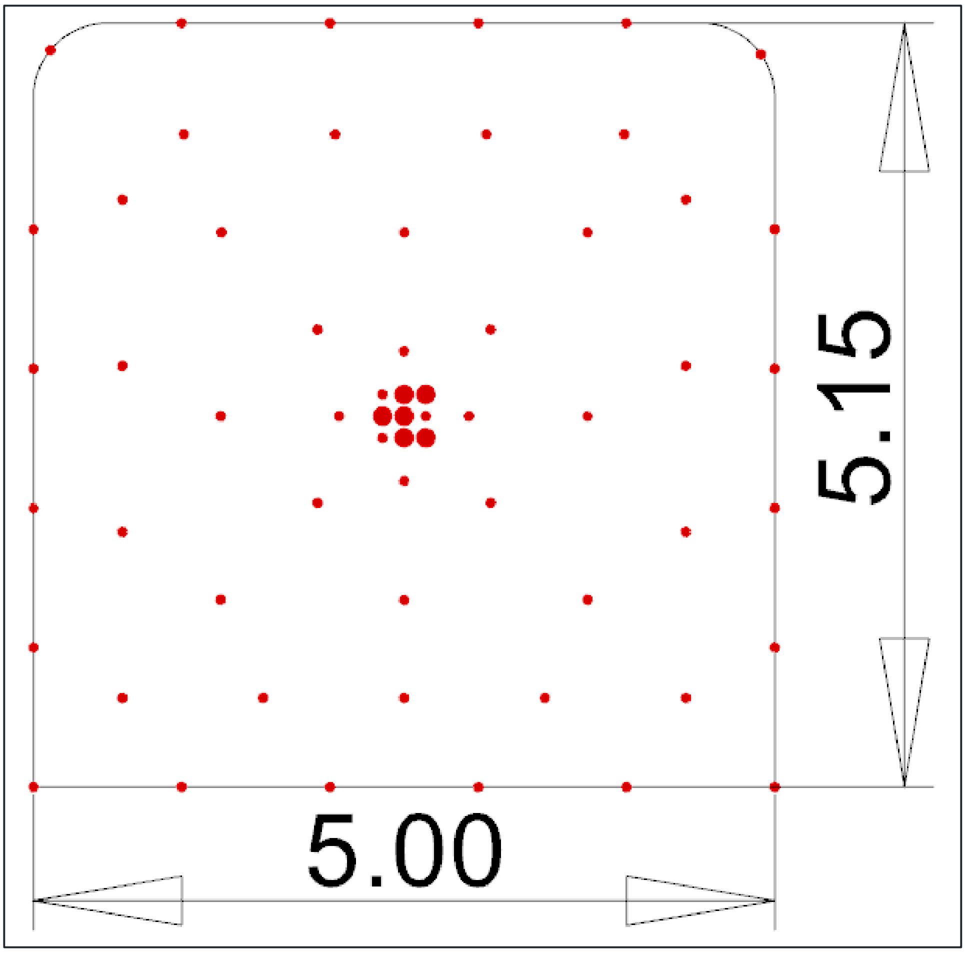


Figure 16-17 – Planned Development Drill Design

Ground Stability and Reinforcement Support

A total of five different ground support classes for variable rock mass quality conditions were produced, as outlined in Section 16.3.2.

A combination of rock bolts and reinforced shotcrete is common in Finland and is envisioned for implementation in the Ikkari underground mine. A minimum bolt density (bolts/m^2) and reinforced shotcrete thickness was provided to estimate ground support requirements as shown in Table 16-9. Provisions have also been made for ongoing rehabilitation required. Cable bolts will be required for support of three-way and four-way intersections as well as in stope hanging-wall backs.

16.3.5. STOPE LAYOUT AND DESIGN

The design implements a stope width of 15 m with a vertical height of 30 m. A small number of stopes were designed at 20 m to improve Resource recovery. Stope lengths vary depending on depth and lithology as per Table 15-6. Average stope length is 19 m and the minimum mineable stope length is 5 m. Stopes were designed with a minimum footwall angle of 70° and hanging-wall angle of 60°. Figure 16-18 displays a typical stope design.

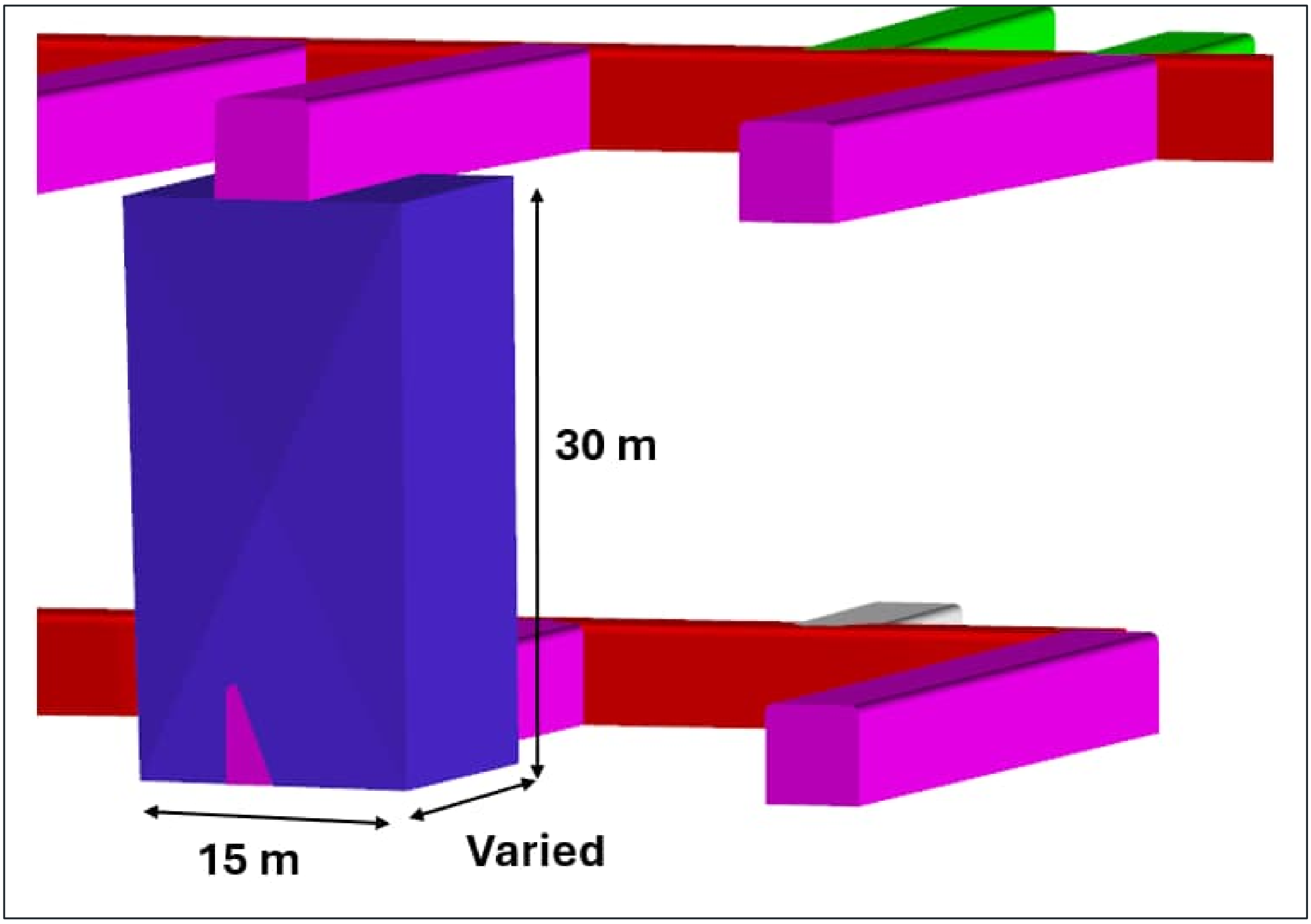


Figure 16-18 – Typical Transverse Stope Design

A primary and secondary stope design was implemented. This entails mining of the primary stopes and leaving at least one stope width between as a supporting pillar. This pillar is referred to as a secondary stope (Figure 16-19). At the completion of mining and curing of backfill in the primary stope on either side of the pillar, mining of the secondary stope may be commenced.

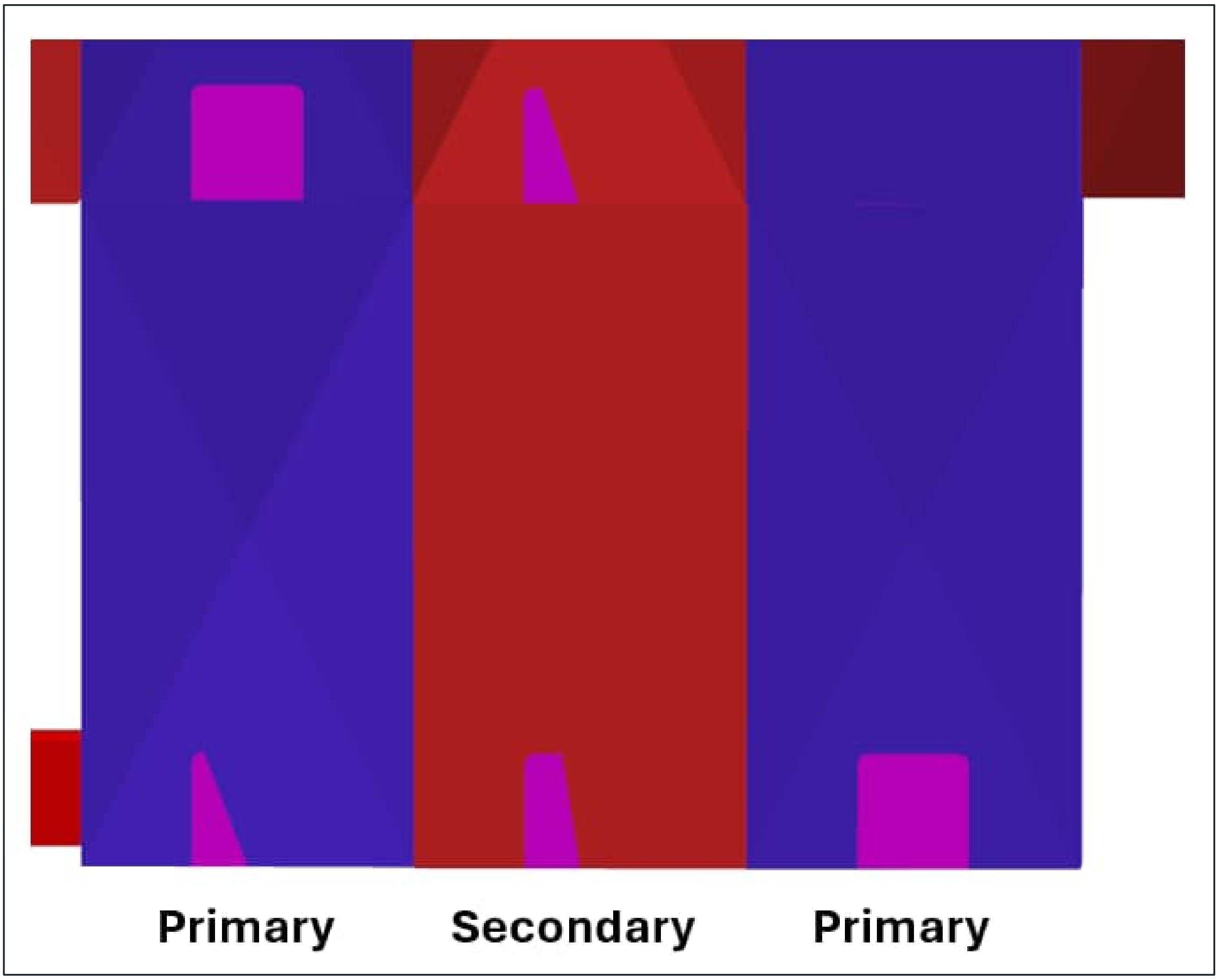


Figure 16-19 – Primary/Secondary Mine Sequence

LHOS is most usually a bottom-up mining method. The lowest stopes are removed first. Initial access to the stope from a top drift to allow for down hole fan pattern drilling. Production drill and load parameters are outlined in Table 16-12. The proposed drilling configuration is shown in Figure 16-20. It is estimated that 10,100 t of explosives will be used over the LOM with a powder factor of 0.6 kg/t for stoping.

Table 16-12 – Production Drill and Load Parameters

Parameter	Value	Unit
Blast Hole Size	89	mm
Vertical Level Spacing	30	m
Average Drill Length	19	m
Average Charge Length	15.2	m

Parameter	Value	Unit
Burden	2.8	m
Spacing	2.6	m
Primers	2	per hole
Explosives	Emulsion	-
Target Powder Factor	0.6	kg explosive/broken t

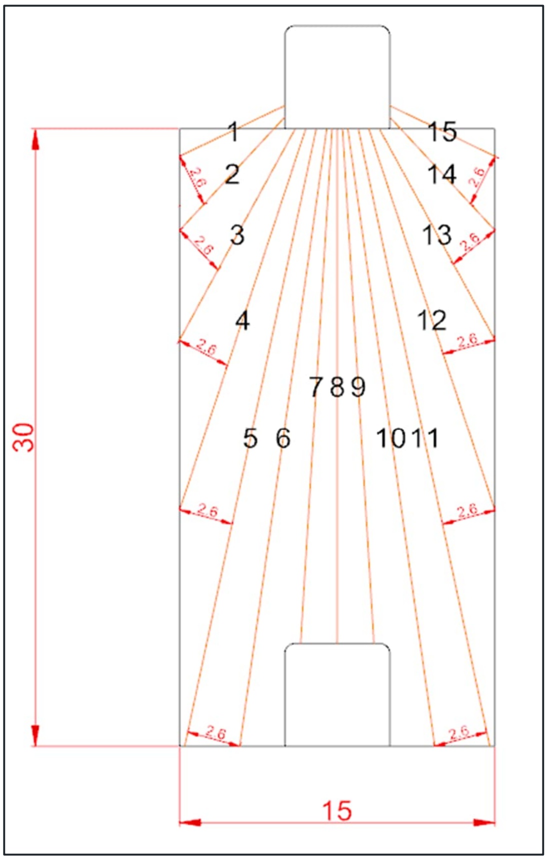


Figure 16-20 – Planned LHOS Drill Configuration

Once the footwall (stope top and bottom) development is in place, a 2 x 2 m slot void is created prior to production blasting (Figure 16-21). The slot void provides expansion (relief) space for the blasted rock in the stope. The proposed slot raise comprises of a 760 mm diameter reamed hole and 12 x 76 mm blastholes that enlarge the reamed hole to 2 x 2 m. It is proposed that the slot raise blasting is done in two lifts.

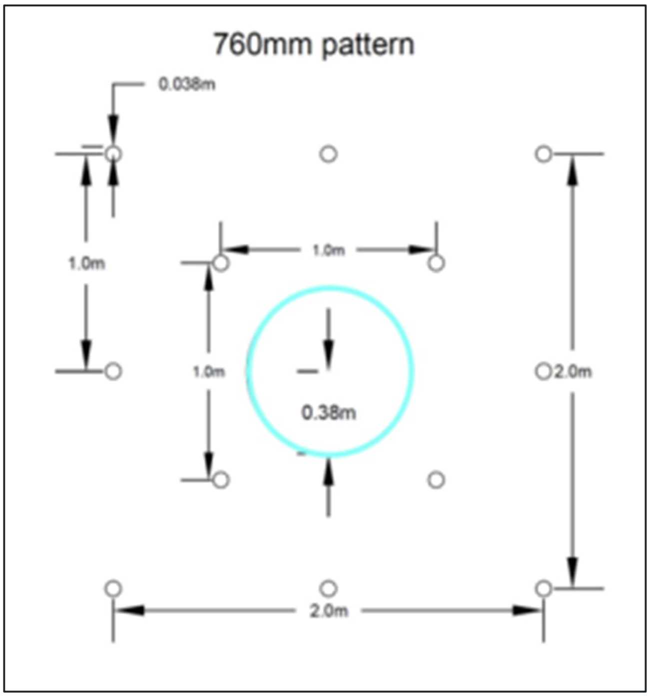


Figure 16-21 – Planned Slot Raise Drill Configuration (After Dong, 2019)

16.3.6. DILUTION AND RECOVERY FACTORS

A 100% recovery and 5% overbreak allowance was applied to underground lateral development.

Stope dilution and recovery factors are outlined in Section 15.7.2 and 15.7.3 of this report. A summary of total dilution and loss is shown in Table 16-13.

Table 16-13 – Stope Dilution and Recovery Summary

Parameter	Value	Unit
Planned Dilution	15	%
Unplanned Dilution	6	%
Secondary Dilution	3	%
Stope Recovery	96	%
Stope recovery – Under Open Pit	86	%

16.3.7. MINE LAYOUT AND DESIGN

Access to the underground mine was designed as follows (Figure 16-22):

- North ramp access with a portal located on the surface (220 m elevation) to the East of the open pit to provide access for mining equipment, personnel, services, and ventilation. This is the initial and primary access to the underground mine; and
- South ramp access with a portal located on the 40m elevation switchback inside the open pit. This will be developed after the open pit mining is completed.

Material from the underground mine will be hauled to the RoM pad via trucks using these declines.

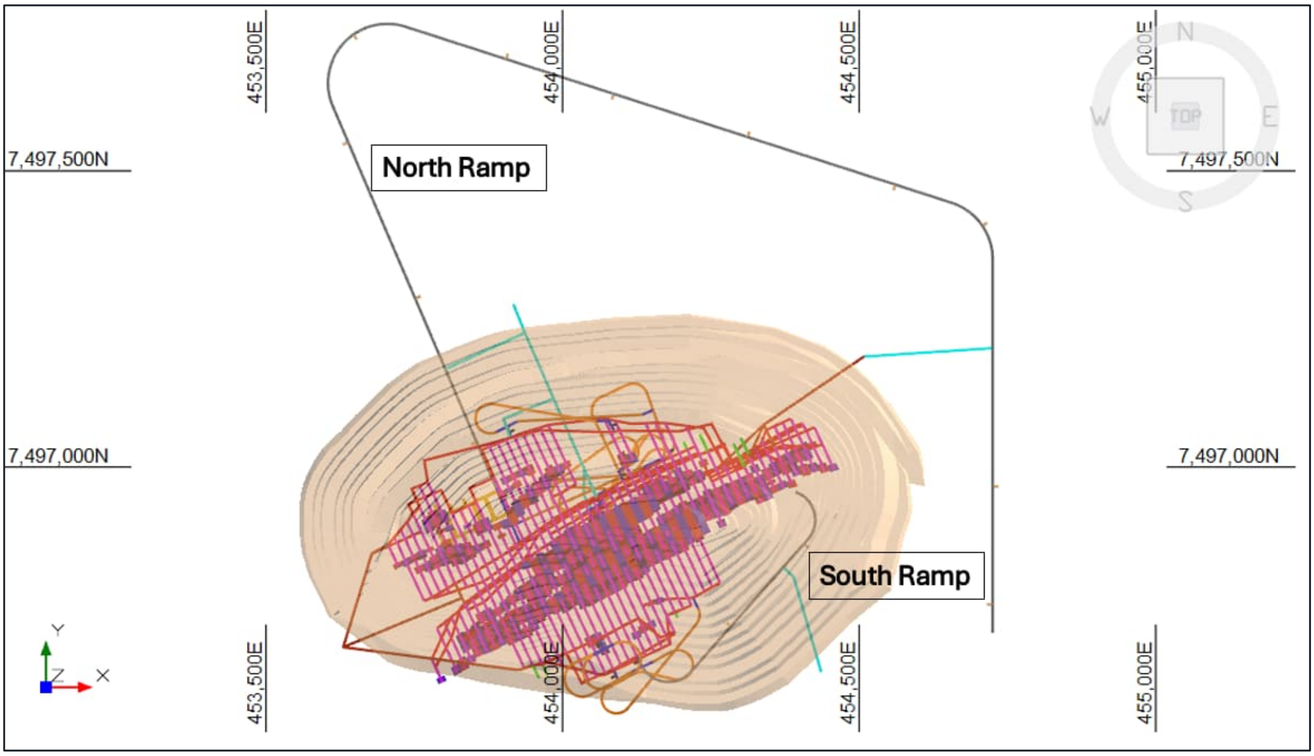


Figure 16-22 – Surface Access Location in Plan View

The North ramp ties into the -140 m elevation where the underground maintenance workshops and offices will be located. A central decline was designed to access the production sub-levels, with level access, footwall drives, and ore drives developed at each sublevel. A single fresh air raise (FAR) supplies ventilation to the Northern side with two exhaust air raises (EAR) located on the Western and Eastern side of the underground mine. The Northern side of the underground design is shown in Figure 16-23.

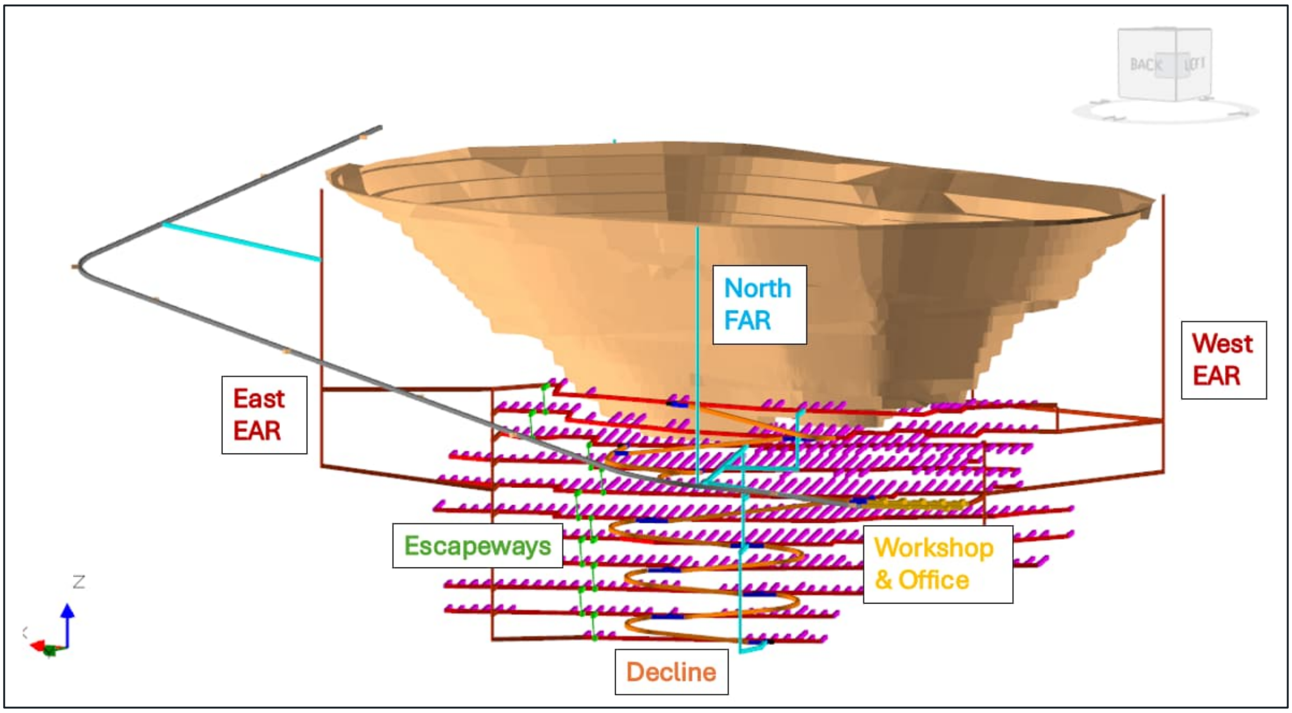


Figure 16-23 – Northern Side of Underground Mine Design looking South

On the Southern side, the ramp access from the open pit ties into the -20m elevation. Like the northern side a central decline is designed to access the production sub-levels with level access, footwall drives and ore drives developed at each sublevel. A single FAR supplies ventilation to the Southern side with the existing EARs used to exhaust ventilation. The Southern side of the underground design is shown in Figure 16-24.

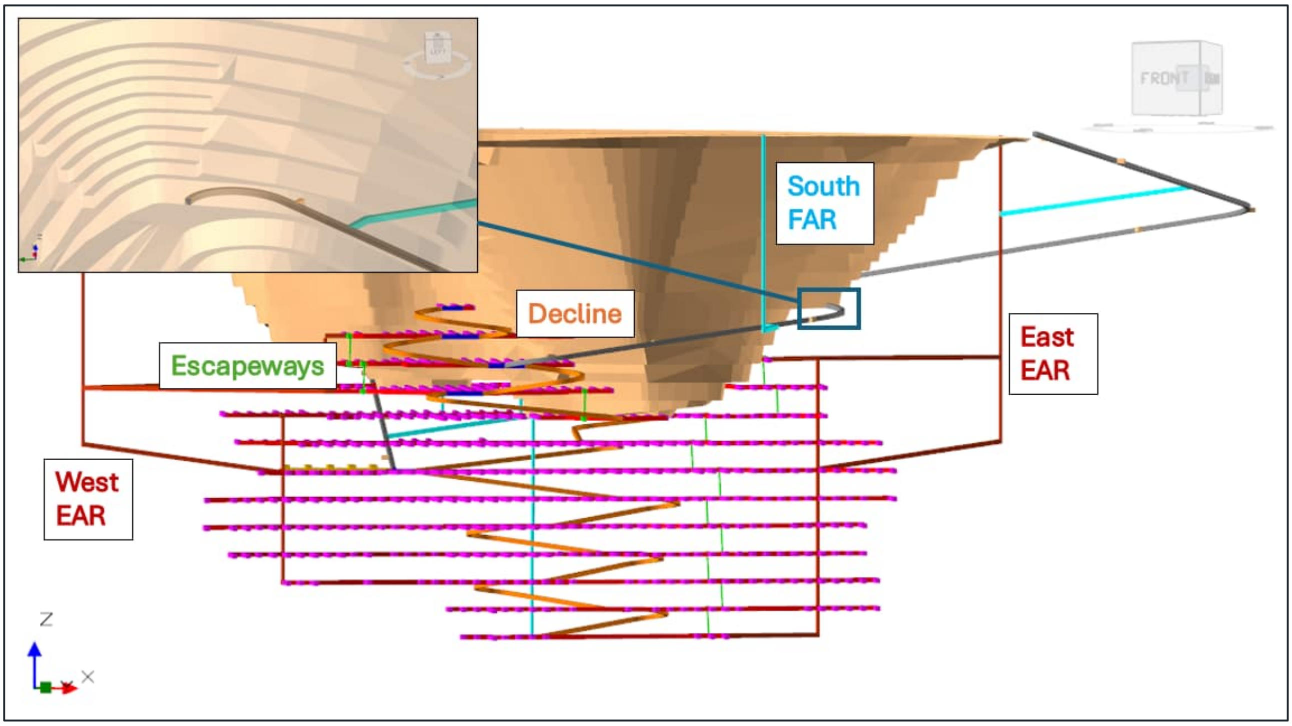


Figure 16-24 – Southern Side of Underground Mine Design looking North

16.3.8. UNDERGROUND MINE SCHEDULE

The underground mine development commences in Year 6 with the development of the North ramp access. The ramp will proceed down to the -140 m elevation where the primary aim is to establish the ventilation infrastructure and commence the central decline down to -320 m elevation.

Stoping commences in Year 10 in a bottom-up sequence, with the mine ending in Year 20. At its peak, the underground mine will produce just over 2 Mt/a.

Development Rates

The development cycle incorporates drilling, blasting, mucking, hauling, ground support and services.

Average advance per round is planned at 4.5 m. A twin-boom jumbo drill will be used to advance development with 48 mm blastholes.

There are a total of five different ground support classes defined for variable rock mass conditions. It is common in Finnish mines to use a combination of bolts and shotcrete. A similar approach has been envisioned for Ikkari.

Cable bolts will be required at 3-way and 4-way intersections, as well as in stope backs. Cable bolt requirements in stope backs vary by lithology.

Lateral and vertical development rates are outlined in Table 16-14. The development rate for the surface access activities was increased to account for its high priority and dedicated single heading. Ore and waste development have the same rates applied.

Table 16-14 – Development Rates for Underground Production Schedule

Activity	Rate
Access Ramps	110 m/mo
Workshop	40 m/mo
Other Lateral Development	75 m/mo
Vertical Development	30 m/mo

Development Schedule

Figure 16-25 shows the annual development metres for the Ikkari underground mine. Development peaks at just over 10,000 m per annum between Years 11 to 13 before ramping down. For Years 5 to 9, underground mining efforts focus on the North ramp access and ventilation infrastructure. The focus then shifts to development of the ore drives to commence production. Development of the South ramp access from the open pit commences in Year 13. Table 16-15 shows the development schedule by activity type over the LOM.

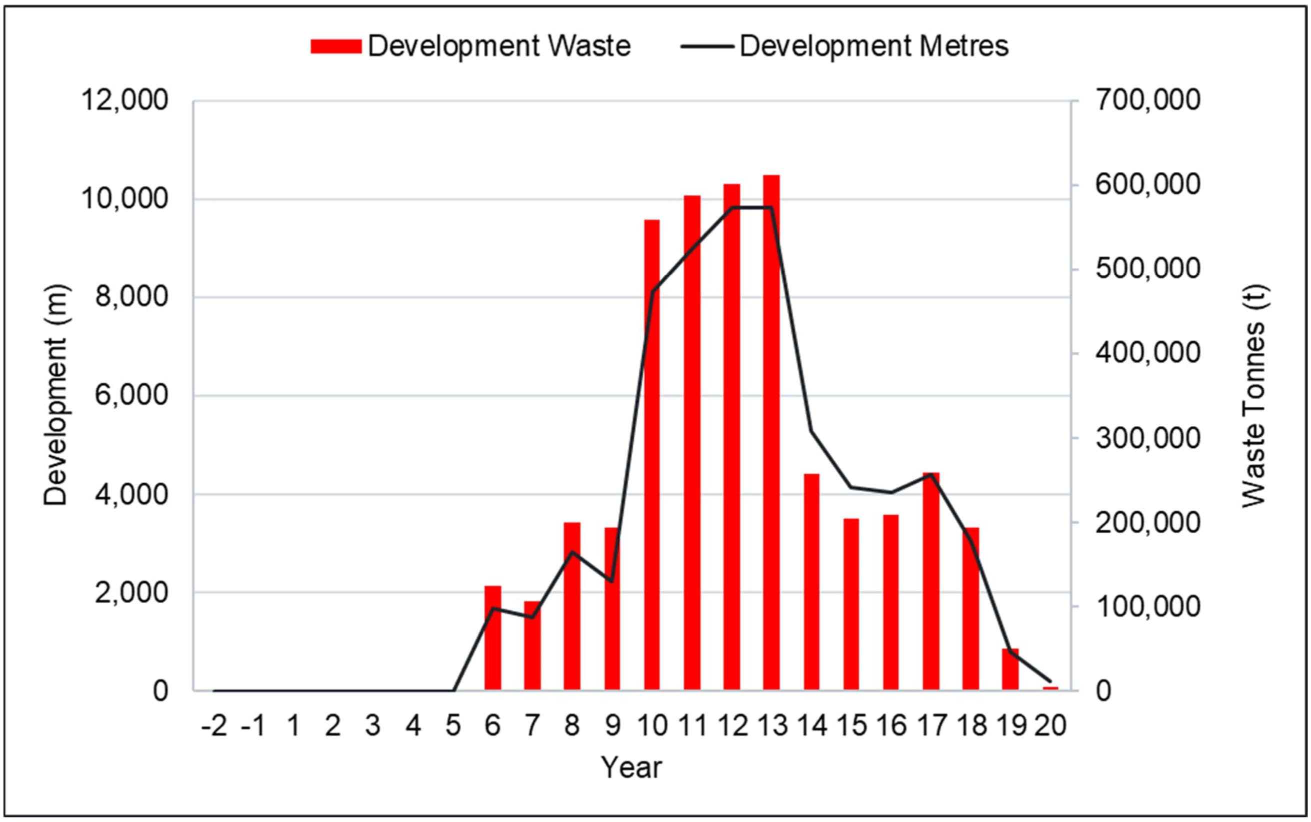


Figure 16-25 – Ikkari Underground Development Schedule

Table 16-15 – Life of Mine Development Schedule

Year	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Lateral (m)	**64 792**	**0**	**0**	**0**	**0**	**0**	**1 592**	**1 491**	**2 245**	**1 788**	**7 741**	**8 840**	**9 588**	**9 606**	**5 277**	**4 138**	**4 040**	**4 397**	**3 044**	**799**	**205**
Ore Drive (m)	43 937	0	0	0	0	0	0	0	0	0	3 527	5 585	6 526	7 062	5 106	3 926	4 015	4 217	2 970	799	205
Footwall (m)	9 478	0	0	0	0	0	0	0	728	25	3 520	2 019	1 753	886	55	212	25	180	74	0	0
Surface Access (m)	3 175	0	0	0	0	0	1 322	1 318	40	0	0	0	496	0	0	0	0	0	0	0	0
Level Access (m)	904	0	0	0	0	0	0	0	201	172	108	141	101	180	0	0	0	0	0	0	0
Decline (m)	3 358	0	0	0	0	0	0	0	608	699	174	793	244	839	0	0	0	0	0	0	0
Sump (m)	150	0	0	0	0	0	0	0	0	10	60	30	20	30	0	0	0	0	0	0	0
Stockpile (m)	350	0	0	0	0	0	270	60	0	0	0	0	20	0	0	0	0	0	0	0	0
Ventilation (m)	2 940	0	0	0	0	0	0	112	668	382	351	272	428	609	116	0	0	0	0	0	0
Workshop (m)	499	0	0	0	0	0	0	0	728	25	3 520	2 019	1 753	886	55	212	25	180	74	0	0
Vertical (m)	**2 074**	**0**	**0**	**0**	**0**	**0**	**89**	**0**	**588**	**427**	**359**	**154**	**239**	**217**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Raisebore (m)	1 260	0	0	0	0	0	89	0	468	427	0	0	118	157	0	0	0	0	0	0	0
Rhino (m)	815	0	0	0	0	0	0	0	120	0	359	154	121	60	0	0	0	0	0	0	0
Total (m)	**66 866**	**0**	**0**	**0**	**0**	**0**	**1 681**	**1 491**	**2 833**	**2 215**	**8 100**	**8 995**	**9 827**	**9 824**	**5 277**	**4 138**	**4 040**	**4 397**	**3 044**	**799**	**205**

LHOS Stope Cycle

A day delay between each stope activity was implemented to account for operational disruptions. Each stope within the schedule has a stoping rate applied based on individual activity rates. A standard 24,000 t sized stope cycle is shown in Table 16-16 with an average stope cycle of 65 days or 376 t/d.

The cycle time does not include ore development or pre-support as these activities are scheduled to be completed before production commences.

Table 16-16 – LHOS Cycle for Typical Stope

Activity	Slot	Drill	Load	Muck	Backfill	Cure	Other	Total	Average Tonnes	Average Performance
Duration	3 d	13 d	1 d	9 d	6 d	21 d	5 d	65 d	24 000	376 t/d

The Ikkari underground mine has a relatively high production rate which requires several stopes in cycle at once. The stope cycles include long hole drilling, loading, blasting, mucking and backfilling. The annual number of stopes required in cycle are shown in Figure 16-26.

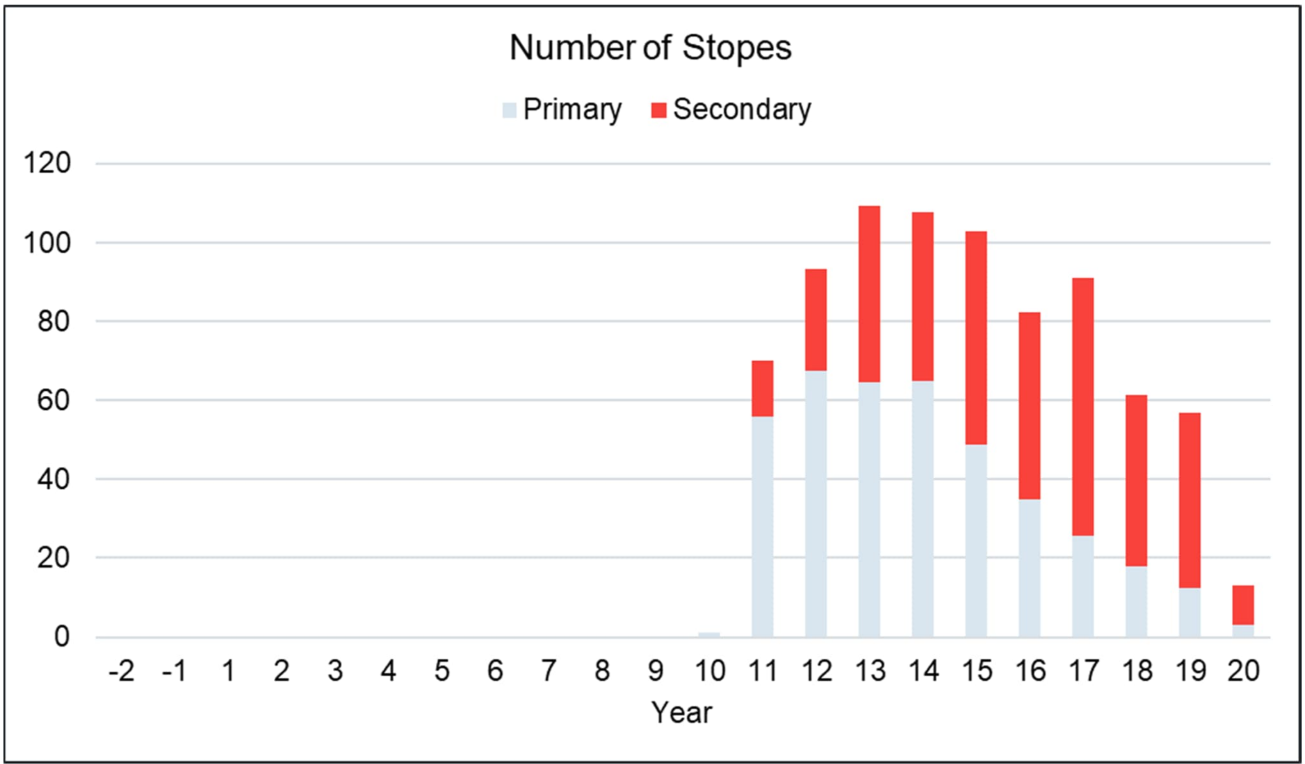


Figure 16-26 – Ikkari Annual Number of Stopes

Mining Sequence

A primary and secondary stope sequence was implemented. This method allowed for a high degree of flexibility and productivity, as well as allowing for the use of unconsolidated backfill within the secondary stopes.

A bottom-up sequence was implemented, with stoping commencing at the bottom of the mine. This was implemented to achieve the optimal production profile. Starting from the bottom of the mine also allowed for a smooth transition between open pit and underground mining.

The extraction sequence is shown in Figure 16-27, with initial mining commencing in the centre with a secondary stope between. The numbers on the figure represent the mining stages, assuming secondary stopes are mined after primaries are paste filled and cured.

Stopes on the same ore drive retreat towards the footwall drive and need to be cured prior to mining above.

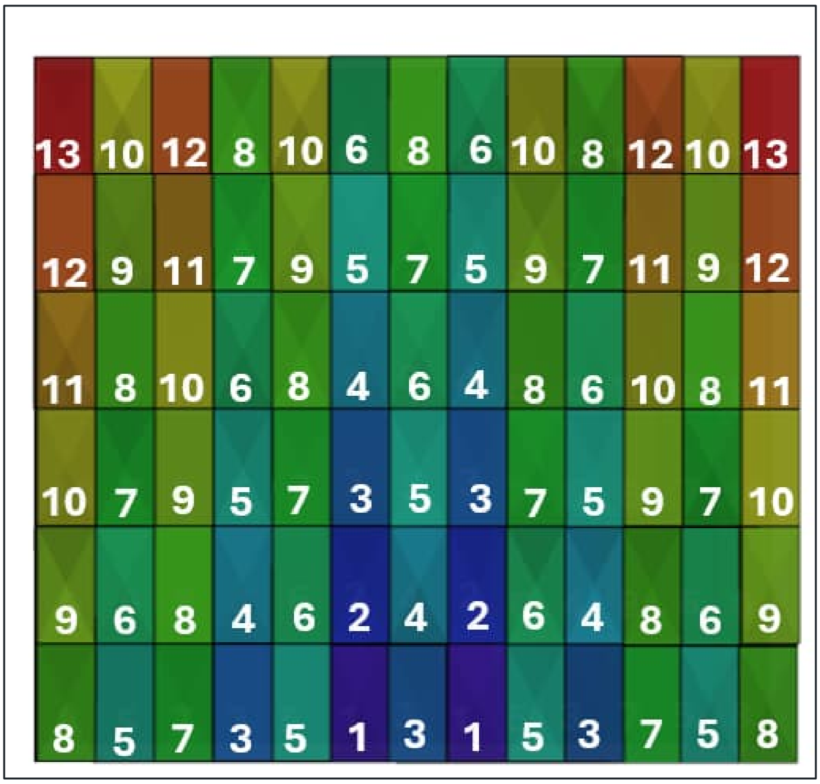


Figure 16-27 – LHOS Mine Sequence Schematic

Five separating workings areas were designed; four in the main ore body and one in the Northern Felsic zone, separate to the main orebody (Figure 16-28).

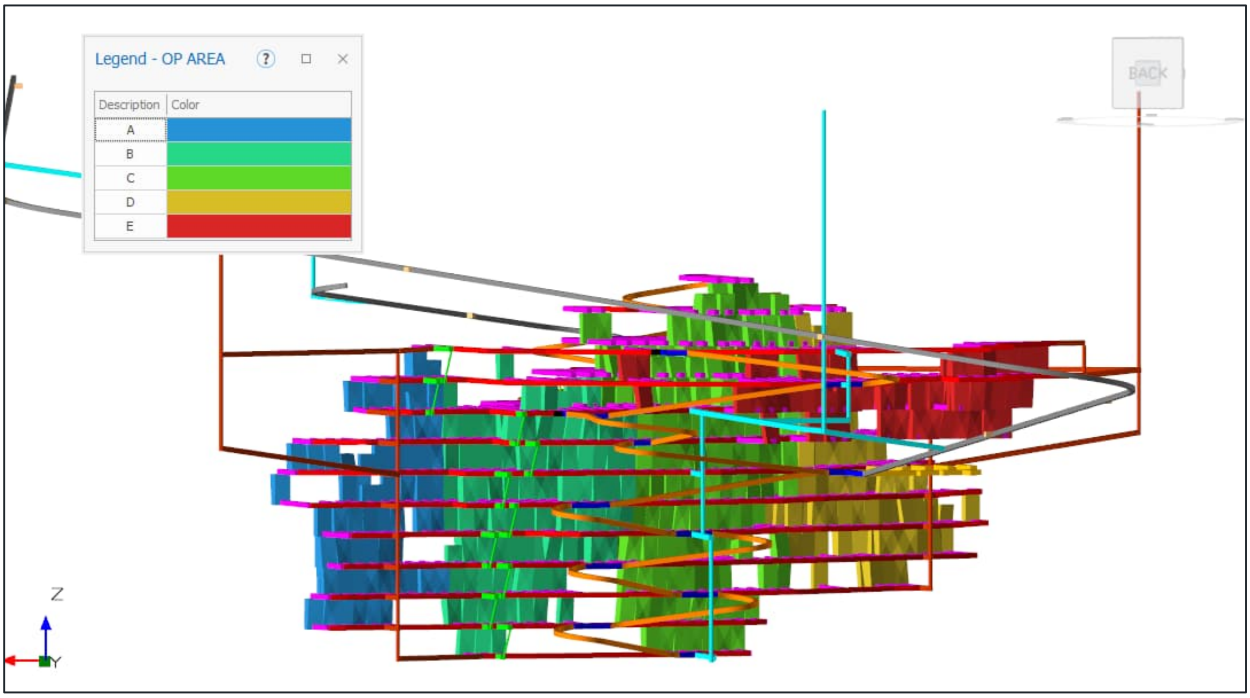


Figure 16-28 – Production Areas for Underground Mine. Colours refer to Operating Area

The four connected operating areas progress holistically allowing for no rib pillars between them. An overall pyramidal shape is developed to control and mitigate stresses and to allow for a high production rate (Figure 16-29).

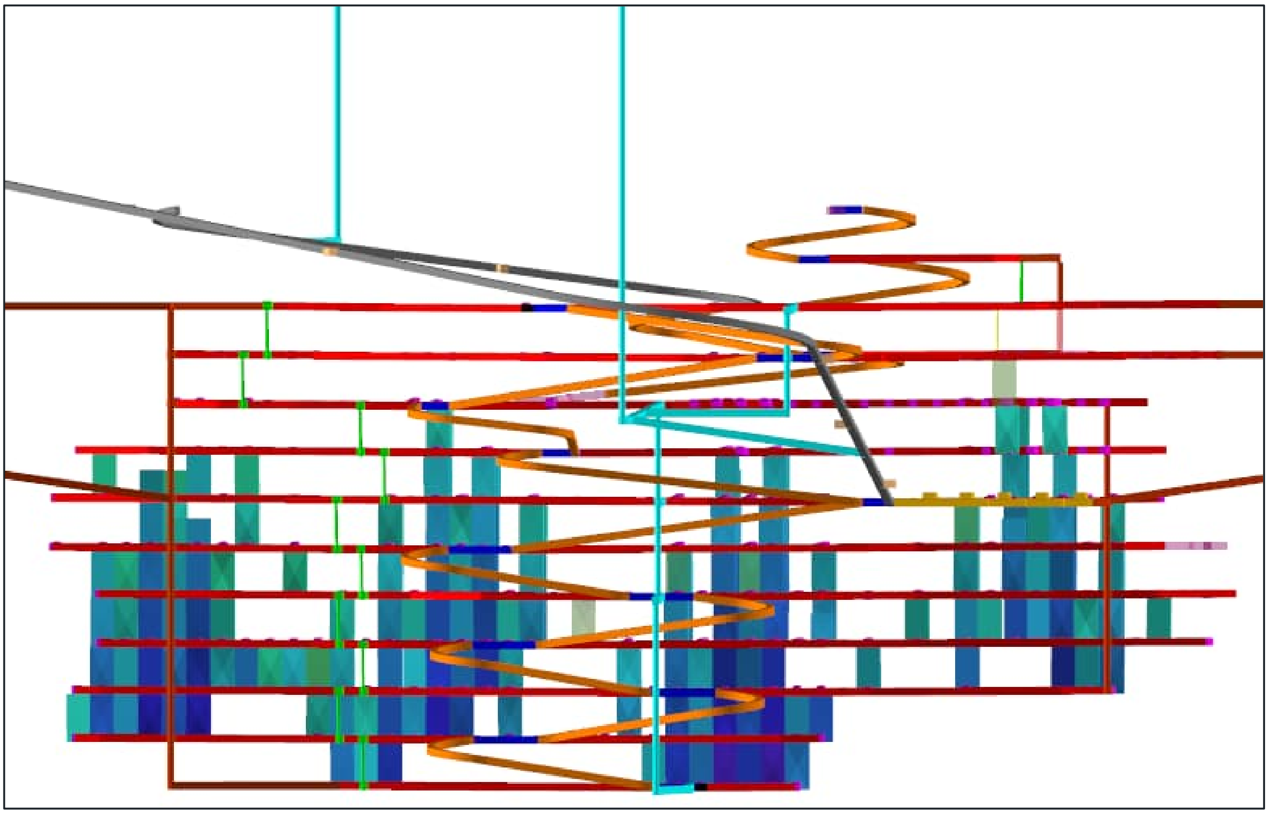


Figure 16-29 – Snapshot of Underground Mining Sequence showing Pyramid Shape

Production Schedule

A total of 16.3 Mt of ore is mined over the LOM at a gold grade of 1.93 g/t containing 1.1 Moz. The annual production schedule is shown in Figure 16-30. A small amount of development ore is mined in Year 8. Production ramp-up occurs between Year 10 and 12. Peak production lasts for about 5 years before ramping down as the number of available stopes reduce.

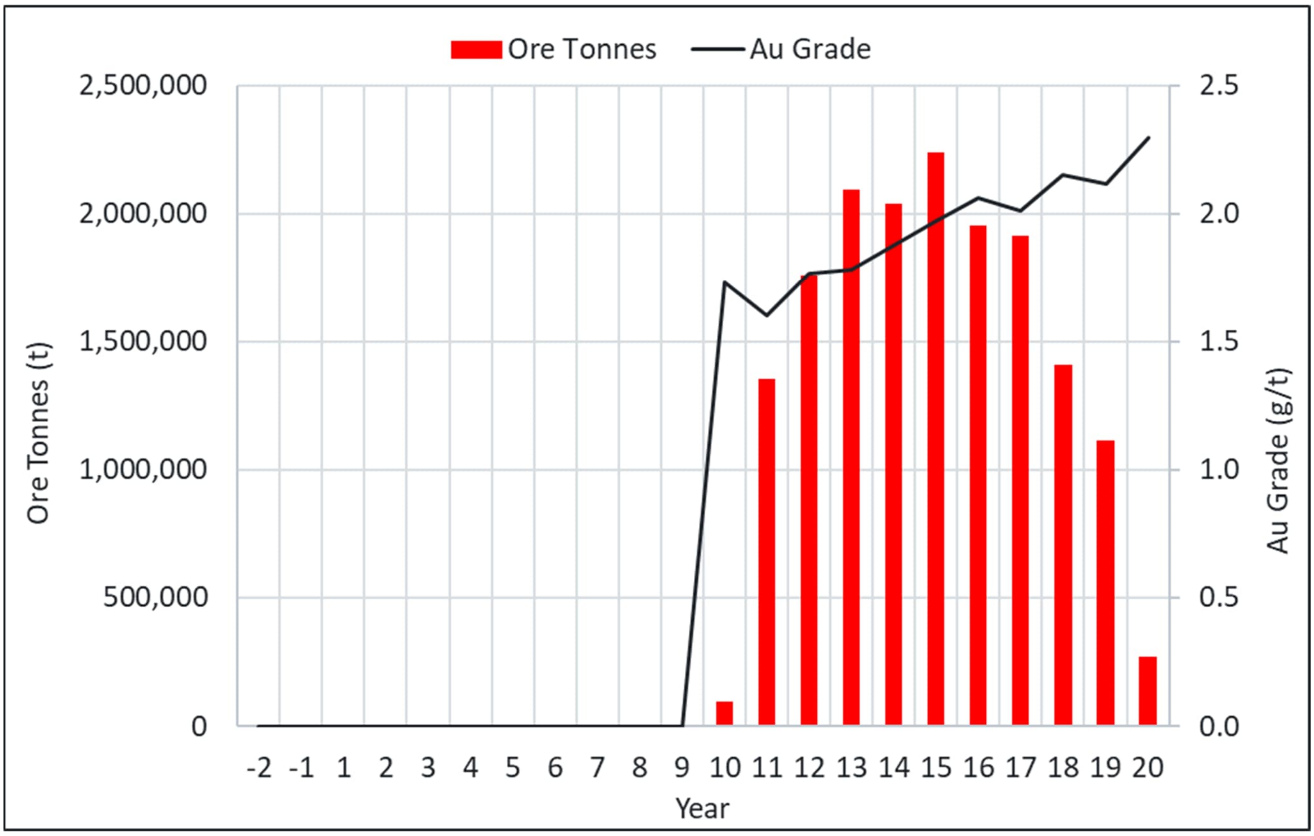


Figure 16-30 – Annual Underground Production Schedule

16.3.9. BACKFILL

The LHOS mining method requires backfill as a support medium. Two backfill products will be used:

- Cemented Paste Fill (CPF); and
- Waste Rock from mine development.

CPF will be used for backfilling primary stopes and some secondary stopes. Most secondary stopes will be backfilled with waste rock where possible. The backfill schedule and quantities are shown in Figure 16-31.

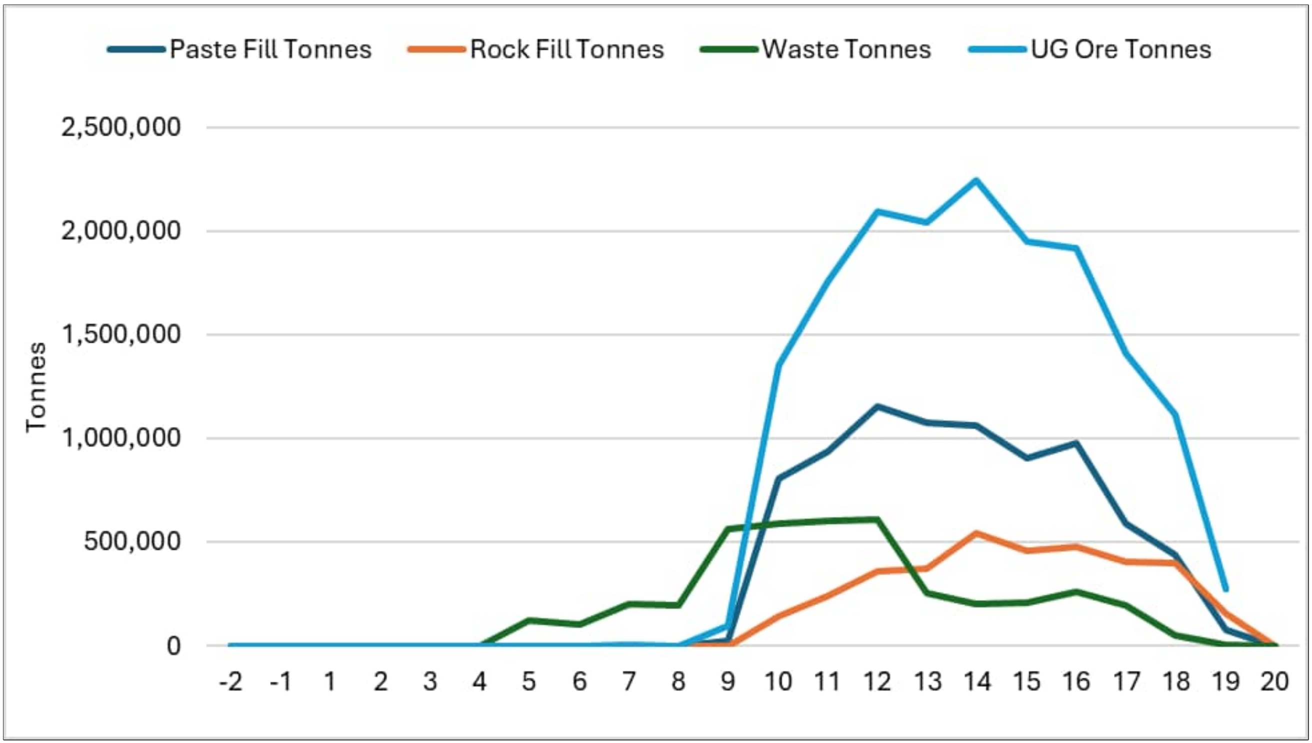


Figure 16-31 – Annual Backfill Schedule

Backfill Design Criteria

The design criteria for the Cemented Paste Backfill (CPB) plant can be found in Table 16-17. When backfill is required in the underground mine, the filter plant will divert the production of filter cake to the paste plant.

The paste plant availability will match that of the mill availability. Due to the difference in densities between the mined ore and the paste backfill, the CPB plant can backfill voids at a higher rate than they are mined. The calculated CPB plant utilisation rate is 56% to fill the mined voids. The resulting idle time will be used for equipment cleaning, performing regular maintenance in the paste plant, and installing UDS piping and barricades in the underground mine.

WSP defines availability as the amount of time that a CPB could provide once regular service, and maintenance are considered and is based on the type of equipment being used and the redundancies built into the system. Utilization on the other hand is the amount of time that the system is required to operate to service the needs of the end user, the mine. Utilization must be less than availability.

Table 16-17 – CPB Plant Design Criteria

Parameter	Value
Annual Nominal Throughput – Underground	2 000 000 t/a
Tailings S.G.	3.09
Voids Created Underground Daily	1 773 m³/day

Parameter	Value
Nominal Daily Required Backfill Rate	1 365 m³/day
Maximum Daily Paste Production Rate	3 616 m³/day
Paste Plant Availability	92%
Paste Plant Utilisation	56%

Backfill Plant

A paste backfill plant located on the surface adjacent to the filtered tailings plant will produce CPB. When CPB is required, filter cake from the filtration plant will be diverted to the CPB plant. Binder will be mixed into the filter cake and the slump of the mixture adjusted to the required target value using water.

Due to the location of the CPB plant relative to the underground mine, a surface pipeline is required to transport the paste to the location of the boreholes at the edge of the open pit. A positive displacement (PD) pump will transfer the paste through the surface pipeline to the underground distribution system (UDS) piping network via the boreholes located on the surface.

Tailings Sample and Laboratory Testing

The initial process design for the mill had all the tailings reporting to flotation, with the float portion reporting to a leaching circuit. This generated two tailings streams, floatation and leach tails. Both these samples were to be combined to create the CPB. Samples generated during pilot testing based on the described process, were shipped to WSP's Sudbury, Canada laboratory for testing.

After tailings samples were generated, the mill process circuit was changed to a whole ore leach process. A new whole ore leach tailings sample could not be generated within the timeframe of this study, so the leach portion of the sample that had already been received by the laboratory would be tested. While the existing sample was not completely representative of the whole ore leach tailings, it would provide some indication of the material properties, and the behaviour of the tailings when mixed with different binder types.

The following test work was performed on the leach portion of the sample:

- Material characterisation;
- Rheology; and
- Unconfined Compressive Strength (UCS) testing.

<u>Particle Size Distribution</u>

The particle size distribution (PSD) of the received sample was measured and had a D_{80} value of 37 µm. Subsequent testing on the whole ore leach sample by others showed that it has a D_{80} value closer to 100 µm.

A finer D_{80} PSD can result in higher binder addition requirement for a given target strength. A finer D_{80} PSD will also result in higher pipeline friction values, which is used to determine the pressure losses of the paste in the UDS system and PD pump sizing. Future testing will be required to confirm the whole ore leach tailings properties.

Paste Backfill Properties

Based on the material testing results, the paste backfill properties are listed in Table 16-18.

The weight percent solids range for the Ikkari paste backfill defined in Table 16-18, is the theoretical range at which the paste plant will operate. Slump values higher than 254 mm may result in particle segregation in the paste and potentially to plugged pipelines. Slumps lower than 178 mm will have increasingly higher friction factors and may not be pumpable over the distances required or require excessively high pumping pressures.

Additional limitations may be put on the slump range due to the higher pipeline friction factor of the paste at lower slump values. This a function of distances and elevations within the UDS relative to the paste plant, and the requirements to reach all areas that need backfill underground. This is discussed further in Section Underground Distribution System (UDS) Flow Model.

Table 16-18 – Paste Backfill Material Properties

Parameter	Value
Solids Content at 178 mm Slump	77.9 wt%
Solids Content at 254 mm Slump	76.1 wt%
Specific Gravity of Paste at 178 mm Slump	2.11
Specific Gravity of Paste at 254 mm Slump	2.06

Unconfined Compressive Strength Testing

A baseline test was performed on samples of leach tailings blended with Oiva brand cement (produced by Finnsementti Oy), and ground blast furnace slag (GBFS).

A 7 wt% binder content was used for different blends of binder and tailings; straight Oiva cement, a 50/50 blend of GBFS and Oiva cement, and a 90/10 blend of GBFS and Oiva cement. The blends were cast into cylinders and cured in a high humidity environment for 28 days. The cylinders were removed from their moulds, placed in a load frame, and compressed to failure to determine the amount of strength each binder type produced. The results are presented in Table 16-19.

Straight Oiva cement, and a 50/50 blend of GBFS and Oiva cement produced roughly the same amount of strength after 28 days curing. The 90/10 blend of GBFS and Oiva cement produced 60% more strength than the other two binders for the same time frame.

While there is a marked benefit to using a 90/10, GBFS / Oiva cement blend versus a straight Oiva cement in the paste recipes, the supply and pricing of GBFS to the site has not been established. If future testing confirms the same results for whole ore leach tailings samples, even at a higher binder price, there may be a cost benefit to using GBFS / Oiva blends in the backfill recipe.

Based on the final underground backfill strength requirements for primary and secondary stopes (Table 16-8), an average binder content of 4.0 wt% of straight Oiva cement was used for this study for the mix designs.

Table 16-19 – 28-day UCS Strength for Different Binder Types

Binder	Wt% Binder	Curing Days	Strength (kPa)
Oiva Cement	7	28	1 956
50/50 (GBFS/Oiva Cement)	7	28	2 066
90/10 (GBFS/Oiva Cement	7	28	3 139

Process Overview

The location of the paste plant is between the filter plant and the filter cake storage building and stockpile.

Filter Cake Conveying

When paste production is required, the filter plant discharge conveyor will divert filter cake to a live bottom feeder located outside of the filter plant. The live bottom feeder will pre-condition the filter cake by breaking up lumps and producing a consistent size filter cake feed into the paste plant at a controlled rate.

The discharge from the live bottom feeder reports to an inclined belt conveyor that will transport the filter cake into a continuous discharge twin shaft mixer. A weigh scale on the belt conveyor will provide a mass flow rate signal of material being fed into the mixer.

Binder System

The paste plant will be equipped with a two-compartment binder bin. The bin will receive bulk delivery of binder from trucks and is sized to cumulatively hold approximately two and a half days of binder requirement at the average binder content of 4.0 wt% in the paste.

The binder from each bin compartment will discharge into a loss-in-weight feeder through fast acting dome valves. From the loss-in-weight feeder, the binder will be metered into a series of screw conveyors, and finally into the continuous mixer.

The target mass flowrate of binder required will be calculated based on the set ratio of tailings to binder, and the mass flowrate of dry tailings solids measured at the inclined conveyor. A rotary valve located at the bottom of the loss-in-weight feeder will have its speed modulated to discharge binder at the required mass flowrate.

Mixing System

The continuous mixer will blend filter cake, binder, and water to bring the paste to the target mix design and consistency. This required consistency or slump will be correlated to the strength which is based on the desired mix design. The operator will input the target slump into the control system. The mixer power draw is correlated to the paste slump range during commissioning. The system will increase or decrease the amount of dilution water being added such that the measured mixer power draw approaches the target power draw.

The blended paste will be discharged continuously through an overflow chute to the one of the two paste hoppers (one operating, one standby). The overflow chutes are equipped with gates that allow the operator to select which hopper will be in use. Each paste hopper will provide feed to one of two positive displacement pumps located directly underneath.

A high-pressure wash pump will be used to clean out the mixer manually after each paste production pour, or as required.

Positive Displacement Pumps

The location of the paste plant adjacent to the filtered tailings plant precludes the use of gravity flow of paste to the underground. Paste will be transported to the underground through a pipeline from the paste plant to boreholes located near the eastern edge of the pit.

Redundancy in case of component failure, is designed into the paste delivery system through the inclusion of operating and standby components.

Two PD pumps (one operating, one standby) will be used to pump paste through the surface pipelines (one operating, one standby) to the underground via boreholes (one operating, one standby).

The continuous discharge from the mixer flows into the active paste hopper which is located directly above the feed nozzle of the positive displacement pumps.

Paste Plant Ancillary Equipment

The paste plant will also include the following equipment that support the paste production:

- Air compressor and compressed air receiver;
- Instrument air dryer and receiver;
- Process Water Tank;
- Process Water Pumps;
- High Pressure Flush Pump for clearing paste lines on surface and underground;
- Dust Collectors for the Binder System; and
- Plant sump and sump pumps to collect spilled material and wash water.

Underground Distribution System (UDS) Flow Model

A hydraulic flow model analysis of the UDS was performed to assess the viability of the proposed system to deliver backfill to the stopes. The basis of a flow model analysis is determining a planned route for the UDS pipe and apply a calculated pipeline friction factor to the designed flow rate of paste to determine the fluid pressure required. In addition, the flow model analysis identifies any portions of the distribution system which may potentially experience over pressurization beyond the allowable design pressure of the pipe materials selected for the system.

For the Ikkari paste system there are three primary considerations which need to be analysed to ensure that the system is viable for delivery of paste to all underground stopes. The UDS system was analysed for both 178 mm and 254 mm slump mix designs. These considerations are:

- That the PD pump size selected for the plant can supply sufficient pressure to reach the worst-case pumping scenario;
- That the surface run of pipe from the plant to the surface borehole collar does not reduce pressure at the borehole collar below what is required to operate the UDS; and
- That the pressure in the system will not exceed the pipe design pressure of 120 bar.

Typically paste systems operate at a flow velocity between 1 and 2 m/s. Slower fluid velocities are better because they produce less wear on the pipe and typically require less pumping pressure to achieve desired results. For the Ikkari backfill plant this means that the system is to be designed using a combination of 250 and 200 NB pipe of various materials depending where in the system they are installed. The mainline piping system will be 250 NB Sch 80 pipe with some portions being 200 NB Sch 80. The main level pipe branching out from the mainline was selected to be 250 NB Sch 40 to ensure that system pressures are kept within reasonable limits.

After the pipe sizes were selected, calculations were performed using the rheology data determined during the lab testing, to obtain the following friction factors presented in Table 16-20. These factors were then applied to the flow model to determine system pressures and the overall design of the system.

Table 16-20 – Paste Distribution System Friction Factors

Line Type	Piping Type	Allowable Operating Pressure [MPa]	Paste Velocity [m/s]	Friction Factor 175mm Slump [kPa/m]	Friction Factor 250mm Slump [kPa/m]
1	200NB Mainline Sch80	12.0	1.50	9.94	5.22
2	250NB Mainline Sch80	12.0	0.96	7.73	4.10
3	250NB Stope HDPE DR9	12.0	1.30	8.99	4.82
4	250NB Mainline Sch40	12.0	0.87	7.41	3.87

The distance from the pump discharge to the borehole collar required nominal friction factors. WSP typically uses 90 bar as a maximum design output pressure from the piston pump.

Piston pumps can typically provide up to 120 bar of pressure however, the system was designed to the lower limit of 90 bar. This differential allows sufficient safety in the design to provide capacity to push through blockages, plant output upsets, flow variation, and other unforeseen scenarios. When considering the upper pumping limit of 90 bar, this leaves an additional pressure capacity of 59.5 and 74 bar for the paste to traverse the UDS.

WSP analysed the UDS for each level in the mine design from the proposed breakthrough of the surface borehole to underground, through to the discharge point into the furthest stope. Two most relevant scenarios are presented below for discussion.

The analysis of the various flow models revealed that a worst-case pumping scenario occurs on the 50 level. Calculated pumping pressures on this level are high enough to show that utilization of a 250 mm slump paste will be required to deliver backfill to the furthest stope. As a result, WSP determined what the furthest safe extent to pump 175 mm slump paste would be, and then verified that a 250 mm slump paste would be capable of reaching the furthest extents of the system. Figure 16-32 shows the farthest pumping scenario for 175 mm slump material at approximately 1900 m of linear pipe, and Figure 16-33 was used to determine the maximum pressure required for the 250 mm pumping scenario of approximately 2300 m of linear pipe.

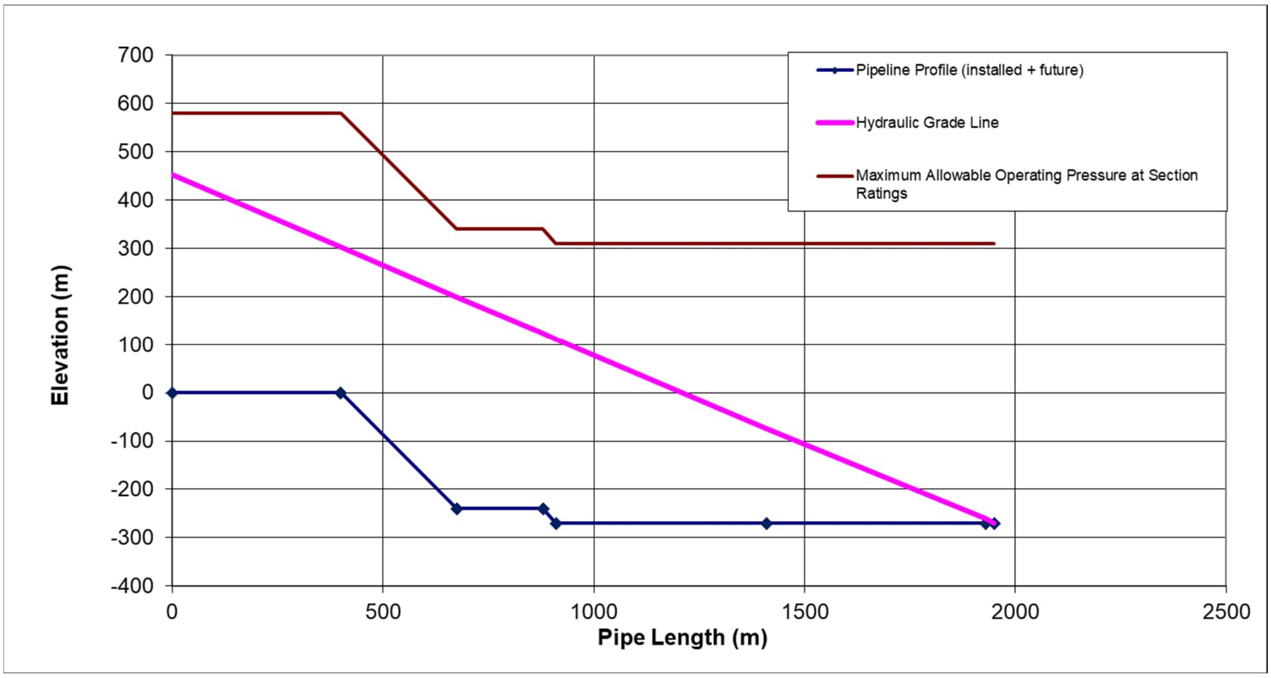


Figure 16-32 – UDS Flow Model to Furthest Possible Stope on 50L with 175 mm Slump

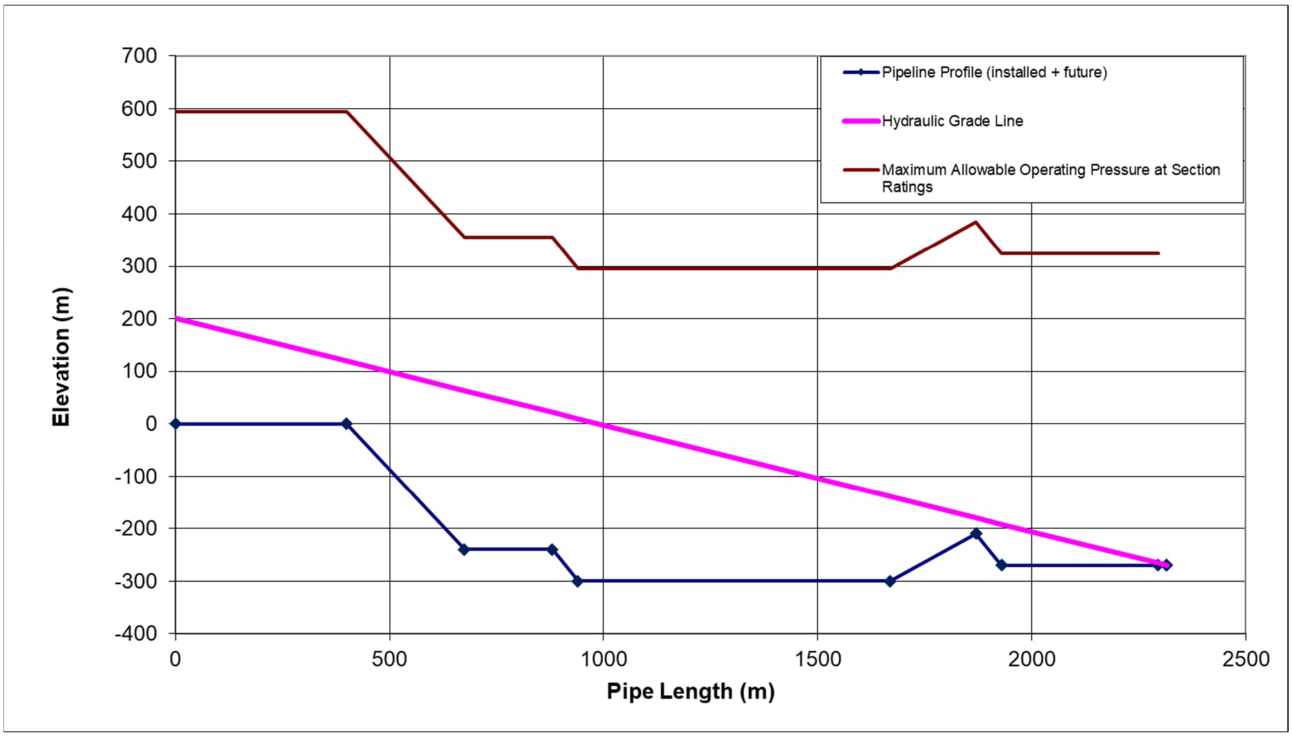


Figure 16-33 – UDS Flow Model to Furthest Possible Stope on 50L with 250 mm Slump

The flow model results show that the system will be viable as designed with a maximum pumping pressure of 93 bar for the 175 mm scenario which can reach a stope located 1040 m horizontally from the borehole breakthrough on that level. In the 250 mm slump scenario the analysis shows that paste can reach the furthest stope with a pumping pressure of 40 bar.

This analysis was repeated for every level of the mine and was used to determine where higher slump paste would be required for filling stopes. These higher slumps require larger amounts of binder to reach the required backfill strengths due to the higher water to cement ratio in the mix.

16.3.10. MINE VENTILATION

EOL Vent Mining AB were tasked by WSP to participate as ventilation engineers for the study. The following sections are the outcome of their work.

Design Criteria

Finland has no legislation regarding air quantity demands according to power rating of equipment in underground equipment. Instead, hygienic threshold levels are not to be exceeded. The ventilation work has been completed with a focus on the nitrogen dioxide (NO_2) levels using diesel-powered equipment.

New thresholds will come into effect for Finnish mines from 21st February 2026 (European Commission, 2019). These have been used for the ventilation modelling. The new criteria are shown in Table 16-21. It is assumed that working time adjacent to diesel machinery is 8 hours out of a 10-hour paid shift.

Table 16-21 – Hygienic Thresholds for Diesel Exhausts

Chemical	8 Hours (ppm)	15 min (ppm)
Nitrogen Dioxide	0.5	1
Carbon Monoxide	20	100
Sulphur Dioxide	0.5	1

Dust levels were also considered and are important to measure once in operation, particularly if rock contains silica. The maximum level for non-organic dust is shown in Table 16-22. Silica is present in the rock; however, it was deemed not to be a dimensioning issue for ventilation.

Table 16-22 – Hygienic Threshold for Dust, 8 Hours

Non-Organic Dust	Inhalable (mg/m^3)	Respirable (mg/m^3)
Dust, Without Silica	5	2.5
Dust, Containing Silica	-	0.1

Notes: Inhalable < 50 µm, Respirable < 5 µm.

Simulations were completed using Ventsim software. These simulations showed that the hygienic thresholds were not exceeded during the LOM for the underground operation.

Ventilation Layout

Figure 16-23 and Figure 16-24 show the locations of the ventilation raises for the underground mine. Two fresh air raises (FAR), and two exhaust air raises (EAR) are planned. All primary fans are planned to be located at surface, except the primary fan for the South FAR.

North FAR (4 m diameter)

Fresh air won't be distributed using auxiliary fans underground. Instead, the raise will have bulkheads at the bottom of the shaft (-320 m elevation) and a regulator on the -130 m elevation. The fresh air will be distributed through the internal ramp. The connection at -130 m elevation provides fresh air to the surface access ramp. The connection at -320 m elevation provides fresh air to the production levels.

The North FAR is planned to be installed by the end of Year 8. It will deliver 250 m^3/s of air capacity.

Fans recommended for the North FAR are four vertically installed 1.8 m diameter fans, with a power rating of 200 kW each. These fans will be housed with air heaters.

South FAR (2.5 m diameter)

The South FAR connects in the southern surface access. Fans will be placed in the bulkhead between the bottom of the FAR and the ramps. Air heating will be installed on the surface.

The South FAR is planned to be installed by the middle of Year 13. It will provide 50 m^3/s of air capacity.

East and West EAR (3.0 m diameter)

EAR will be placed on the eastern and western sides of the open pit. The EAR connect into each side of the footwall drive. Bulkheads will divide the production levels from the raises. On the surface, two parallel, 1.8 m diameter, 132 kW power rated fans will be installed in each EAR.

The West EAR is planned to be installed by the middle of Year 9 and the East EAR by the end of Year 9.

Auxiliary Ventilation System

In each level, an auxiliary exhaust fan will be placed in the EAR shaft bulkhead. A 1.0m diameter, 11 kW power rated fan with a damper will be installed. The production level ventilation system is shown in Figure 16-34.

In the footwall drift, opposite each ore drive, a small auxiliary fan, 0.63 m diameter, 11 kW power rated with 800 mm duct will be installed. The duct will be installed into the corresponding ore production drive (crosscut).

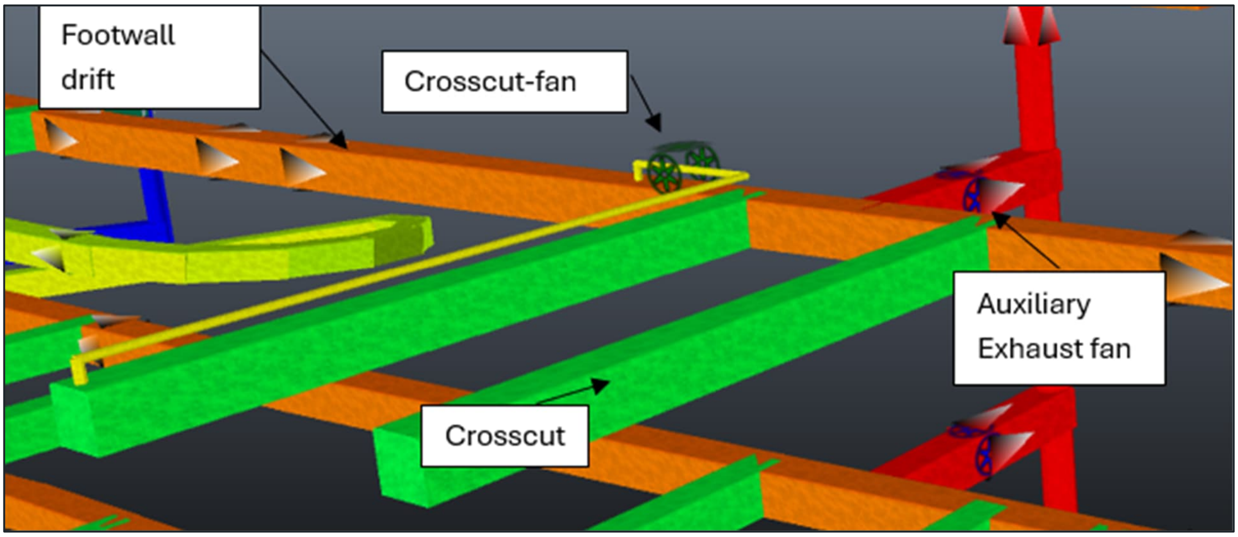


Figure 16-34 – Production Level Ventilation System

Ventilation Control

Primary and auxiliary fans will require variable frequency drives (VFD) connected to a control system to allow for ventilation on demand (VOD). Gas sensors and flow meters will also be required.

Development Ventilation

The ventilation system above reflects the situation when all ventilation infrastructure is in place. Prior to the North FAR being installed, the East FAR will be used as a fresh air source. This is to ventilate the North ramp. A 1.0 m diameter duct with a 0.9 m diameter, 37 kW powered fan will be used.

Once the North ramp has been developed, the upper section (before the temporary FAR) can be ventilated from surface through two parallel 1.0 m ducts and 1.0 m diameter, 45 kW powered fans.

From the temporary FAR to the North FAR, two parallel 1.0 m ducts and two 1.0 m diameter, 45 kW powered fans in series.

Heating

Intake air will need to be heated during winter months when the outside temperature drops below 0°C. The Table 16-23 shows the required heating capacity and an estimate of energy consumption. The dimensioning outside temperature was set to -30°C.

Table 16-23 – Heating Requirements

Parameter	North FAR	South FAR
Quantity, Maximum Capacity (m^3/s)	250	50
Dimensioning Outside Temperature (°C)	-30	-30
Minimum Intake Temperature (°C)	1	1
Delta Temperature (°C)	31	31
Heating Maximum Capacity (MW)	10	2

Electric heaters will be used to heat both the North and South FAR.

16.3.11. UNDERGROUND MINING EQUIPMENT

It is planned for development to be conducted by an underground mining contractor, whilst production will be done by RR. Equipment estimates have been made for both development and production activities.

Underground mining activities will be undertaken with a diesel-powered fleet. Equipment numbers were based on productivity estimates and the number of active working areas at any given time. The underground effective working time parameters are outlined in Table 16-24.

Table 16-24 – Underground Working Time Parameters

Parameter	Value	Units
Operating Days per Year	360	Days
Shifts per Day	2	#
Hours per Shift	10	Hours
Shift Change/Travel Time	1	Hours
Lunch Break	0.5	Hours
Safety Talks/Inspections	0.5	Hours
Available Working Time	8	Hours
Efficiency	75	%
Daily Operating Hours	12	Hours
Annual Operating Hours	4 320	Hours
Annual Leave	18	days
Sick Leave	5	days
Absenteeism	3	days
Training	5	days
Allowance	9	%

The estimated underground equipment fleet is shown in Table 16-25. The peak numbers refer to maximum number of units in any year over the LOM.

Table 16-25 – Mobile Equipment Requirements

Equipment Type	Peak Number
Development	
Bolter	3
Cable bolter	3
Emulsion Loader	2
Flatbed Truck	3
Grader	2
Jumbo Drill	4
LHD, 17t	1
Light Vehicle	8
Scaler	2
Scissor Lift	2
Service Truck	2
Shotcrete Sprayer	2
Transmixer	2
Truck	3
Production	
LHD, 17t	5
Longhole Drill	4
Emulsion Loader	4
Mobile Rock Breaker	1
Truck, 51t	10
Slot Drill	2

Figure 16-35 and Figure 16-36 provides a summary of the development and production equipment by year.

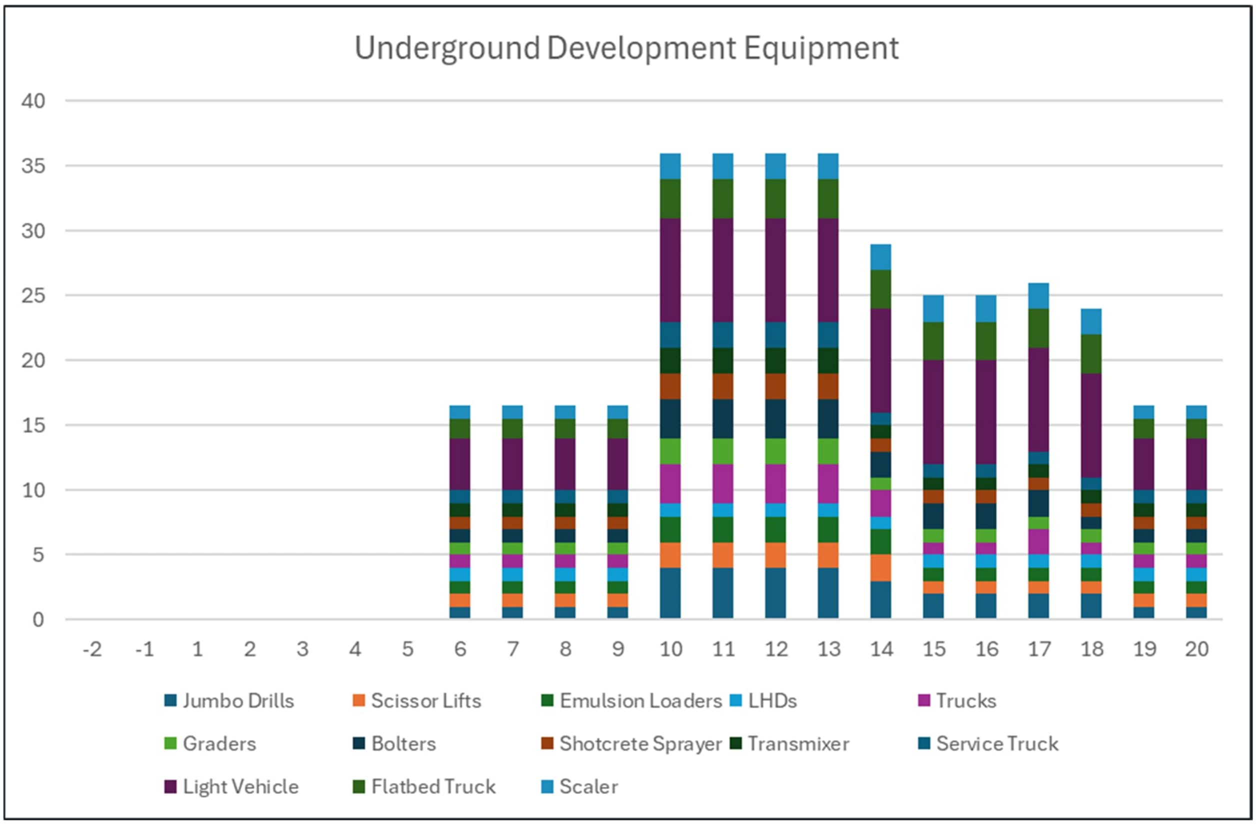


Figure 16-35 – Development Mobile Fleet by Year

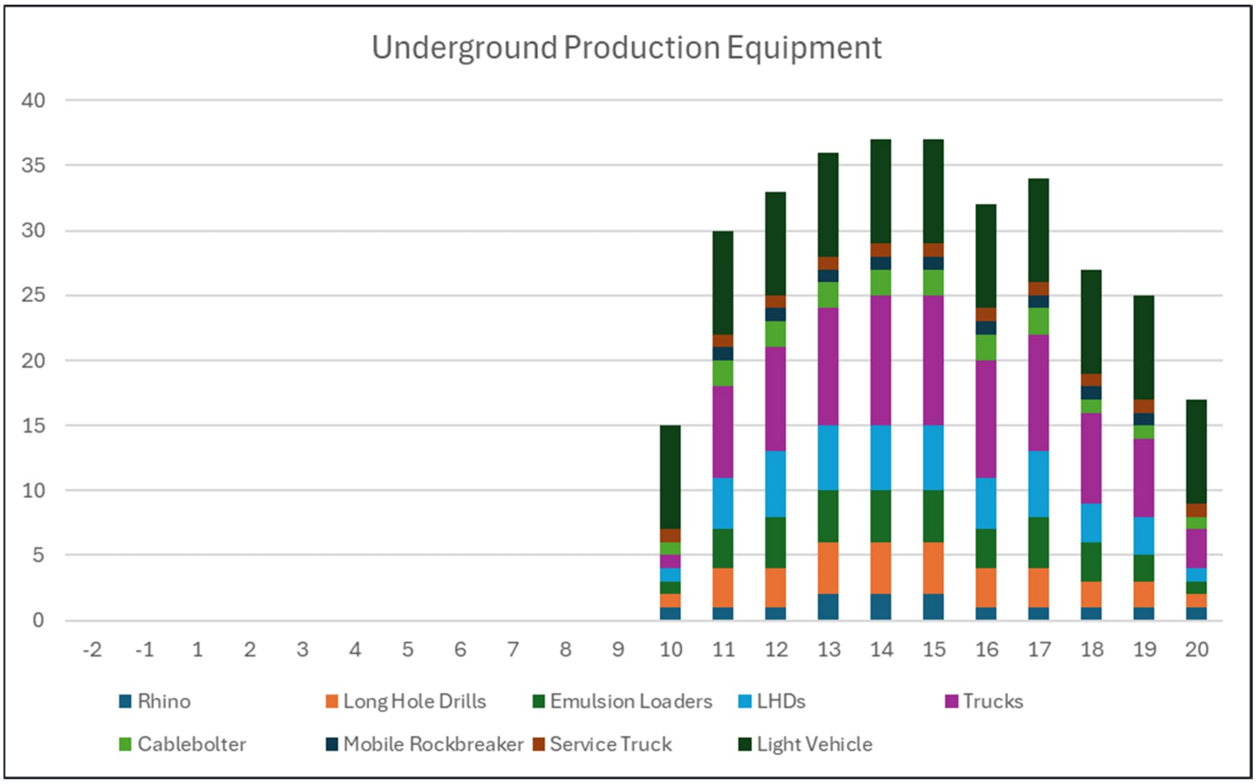


Figure 16-36 – Production Mobile Fleet by Year

16.3.12. UNDERGROUND PERSONNEL

It is planned for development to be conducted by an underground mining contractor, whilst production will be done by Rupert. Personnel estimates have been made for both development and production activities with appropriate allowances incorporated (Table 16-24).

The work schedule assumes 360 days of 20 hours per day operation. It is assumed that 5 days are lost due to inclement weather. Underground mining personnel will work on two 10-hour shifts per day.

The underground work roster is shown in Table 16-26 below.

Table 16-26 – Underground Working Roster

Work Day	Team 1	Team 2	Team 3	Team 4
Monday	7-17	19-5	-	-
Tuesday	7-17	19-5	-	-
Wednesday	7-17	19-5	-	-
Thursday	7-17	19-5	-	-
Friday	7-17	19-5	-	-
Saturday	-	-	7-17	19-5
Sunday	-	-	7-17	19-5
Monday	-	-	7-17	19-5
Tuesday	-	-	7-17	19-5
Wednesday	-	-	7-17	19-5
Thursday	19-5	7-17	-	-
Friday	19-5	7-17	-	-
Saturday	19-5	7-17	-	-
Sunday	19-5	7-17	-	-
Monday	19-5	7-17	-	-
Tuesday	-	-	19-5	7-17
Wednesday	-	-	19-5	7-17
Thursday	-	-	19-5	7-17
Friday	-	-	19-5	7-17
Saturday	-	-	19-5	7-17

Labour requirements over the LOM for development and production are summarised in Figure 16-37.

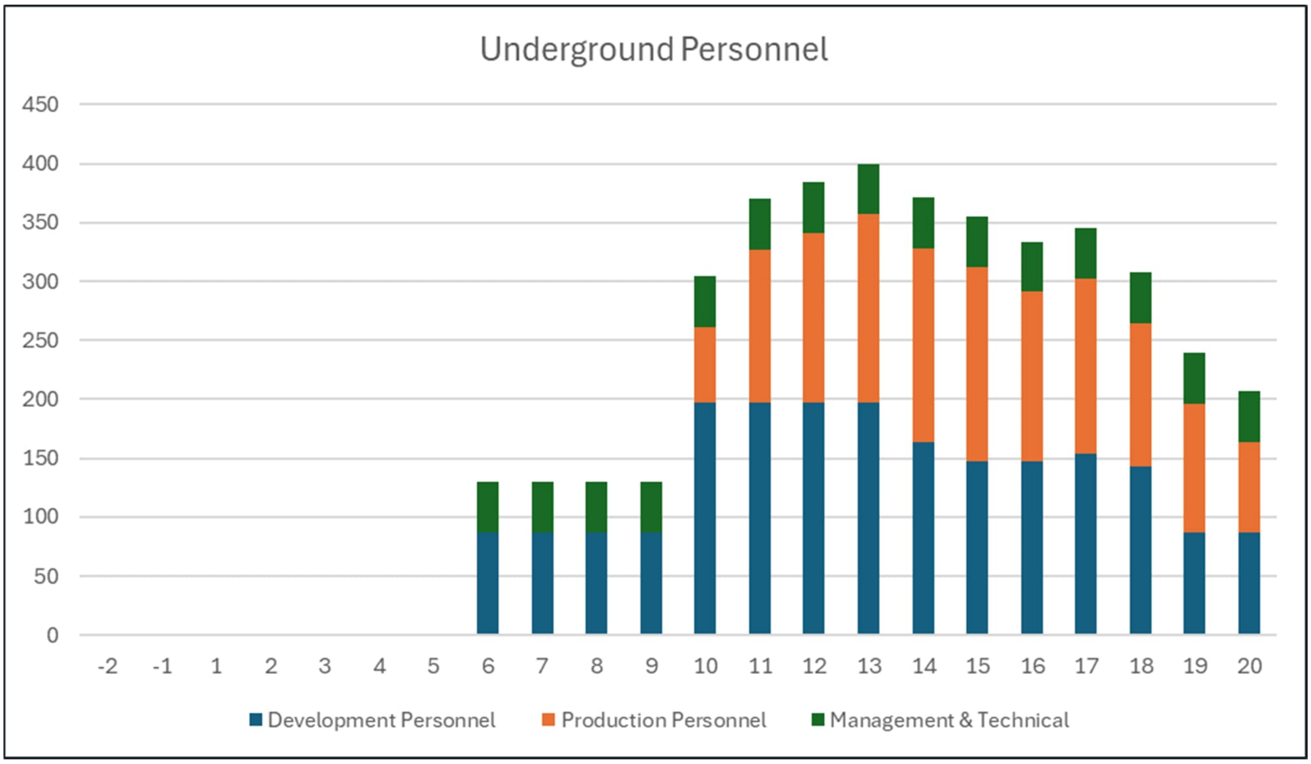


Figure 16-37 – Underground Mine Personnel by Year

16.4 PLANT FEED SCHEDULE

A 3.5 Mt/a production rate can be sustained through open pit mining. Underground mining at Ikkari can only sustain 2 Mt/a, hence a reduction in processing throughput occurs in Year 11 to Year 20 where mining transitions underground. The combined mining schedule and plant feed schedule by year is shown in the figures below.

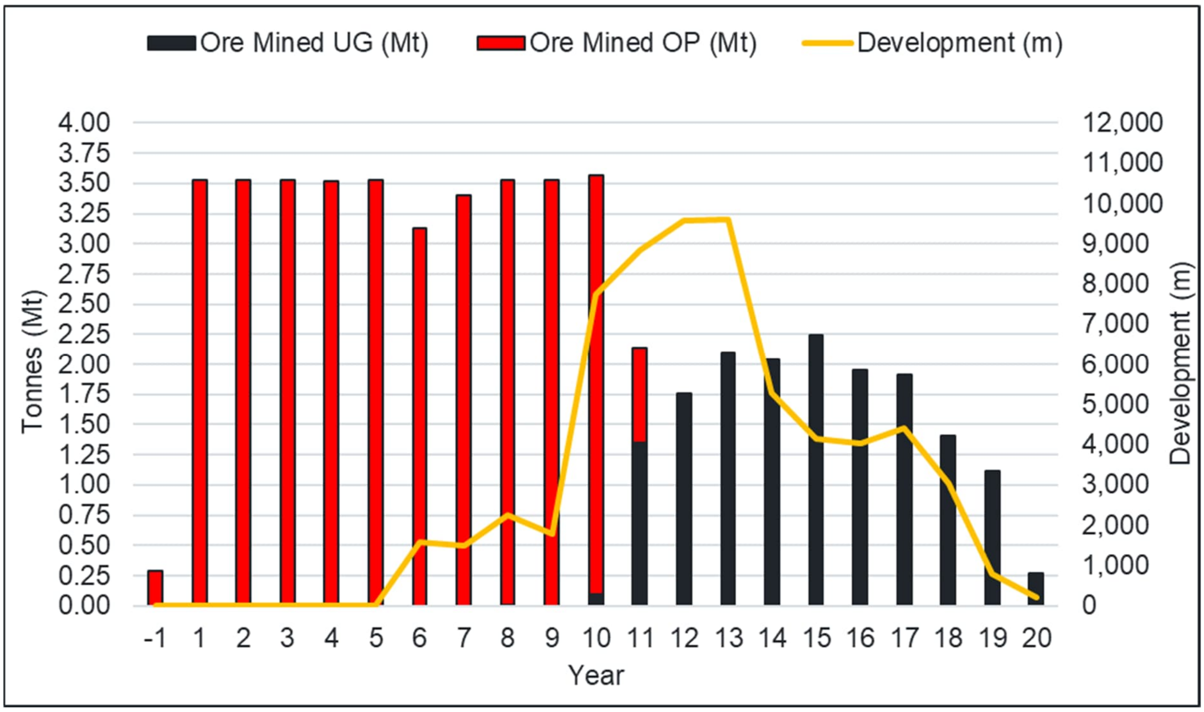


Figure 16-38 – Combined Mining Schedule by Year

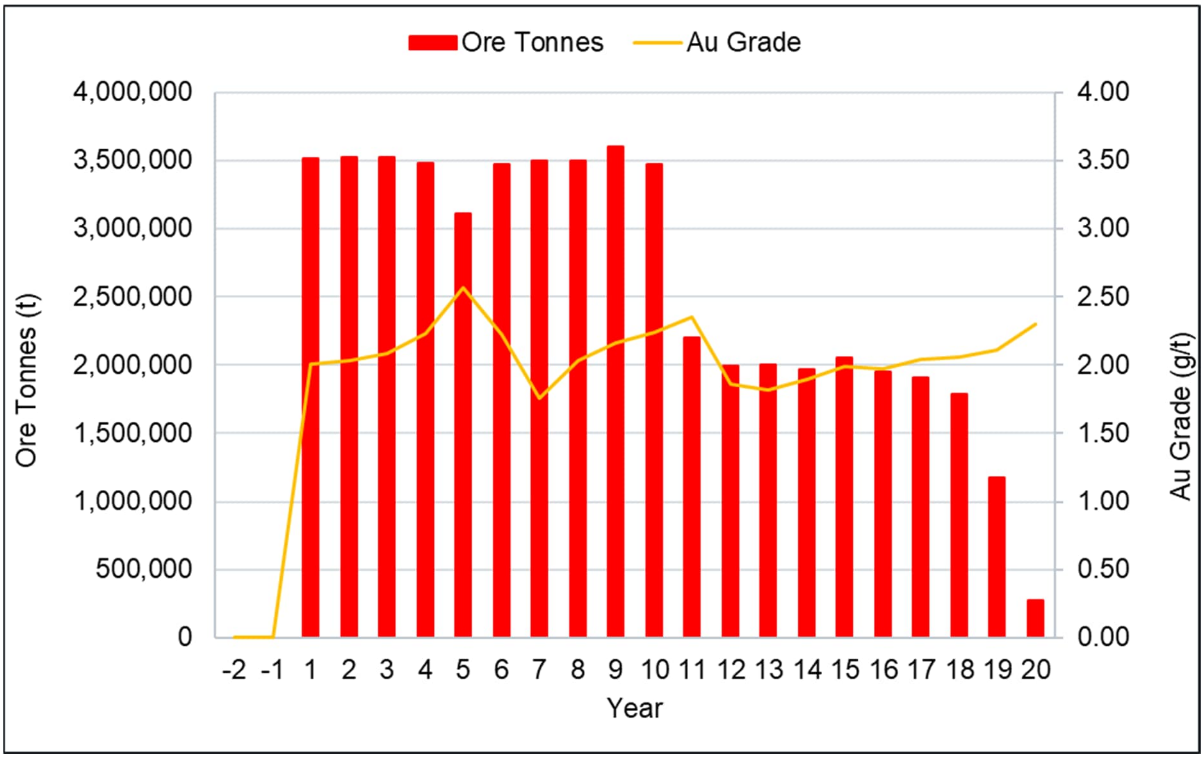


Figure 16-39 – Plant Feed Tonnes and Grade by Year

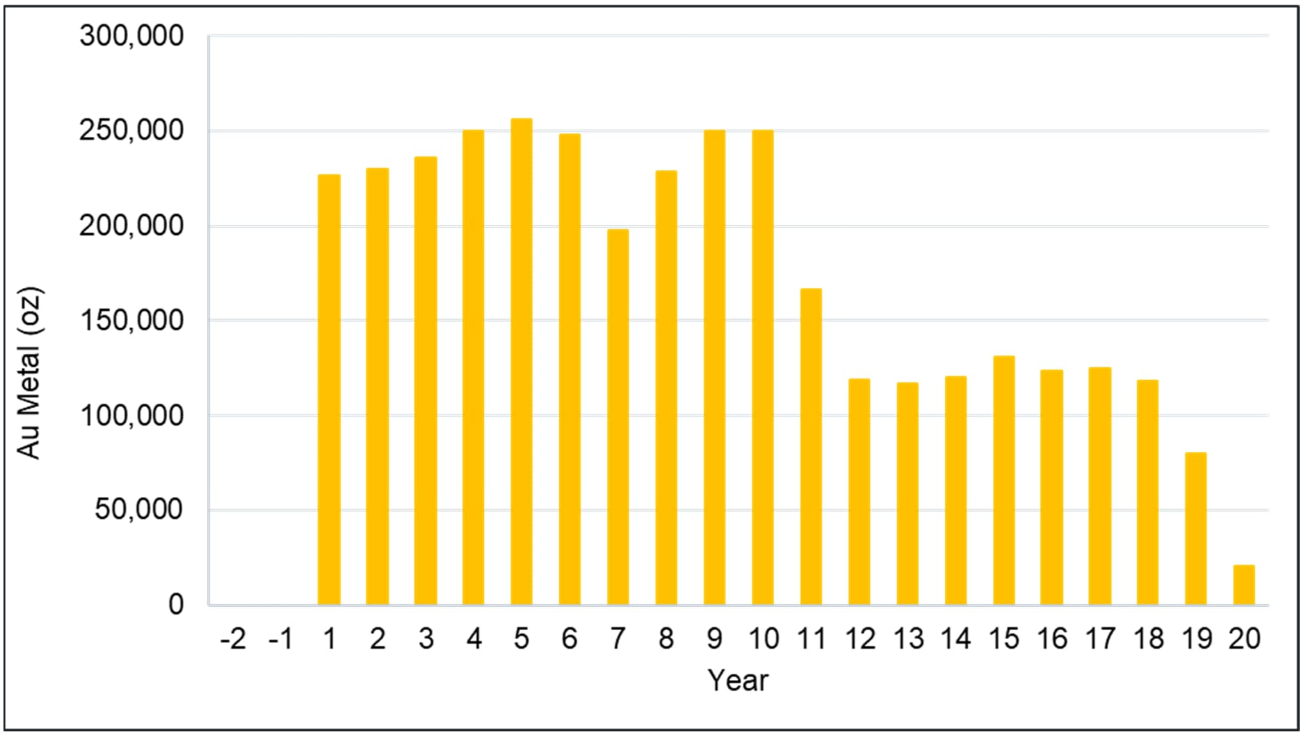


Figure 16-40 – Plant Feed Metal by Year

The processing plant at Ikkari is fed through ex-pit pit material and rehandle. Three stockpiles were used in the LOM and plant feed schedule. Grade requirements for each are shown in Table 16-27. The maximum stockpile amount during the LOM is 0.7 Mt in Year 5 (Figure 16-41).

Table 16-27 – Stockpile Criteria

Stockpile	Au Grade Min [g/t]	Au Grade Max [g/t]
High Grade	2.50	-
Medium Grade	1.50	2.50
Low Grade	0.34	1.50

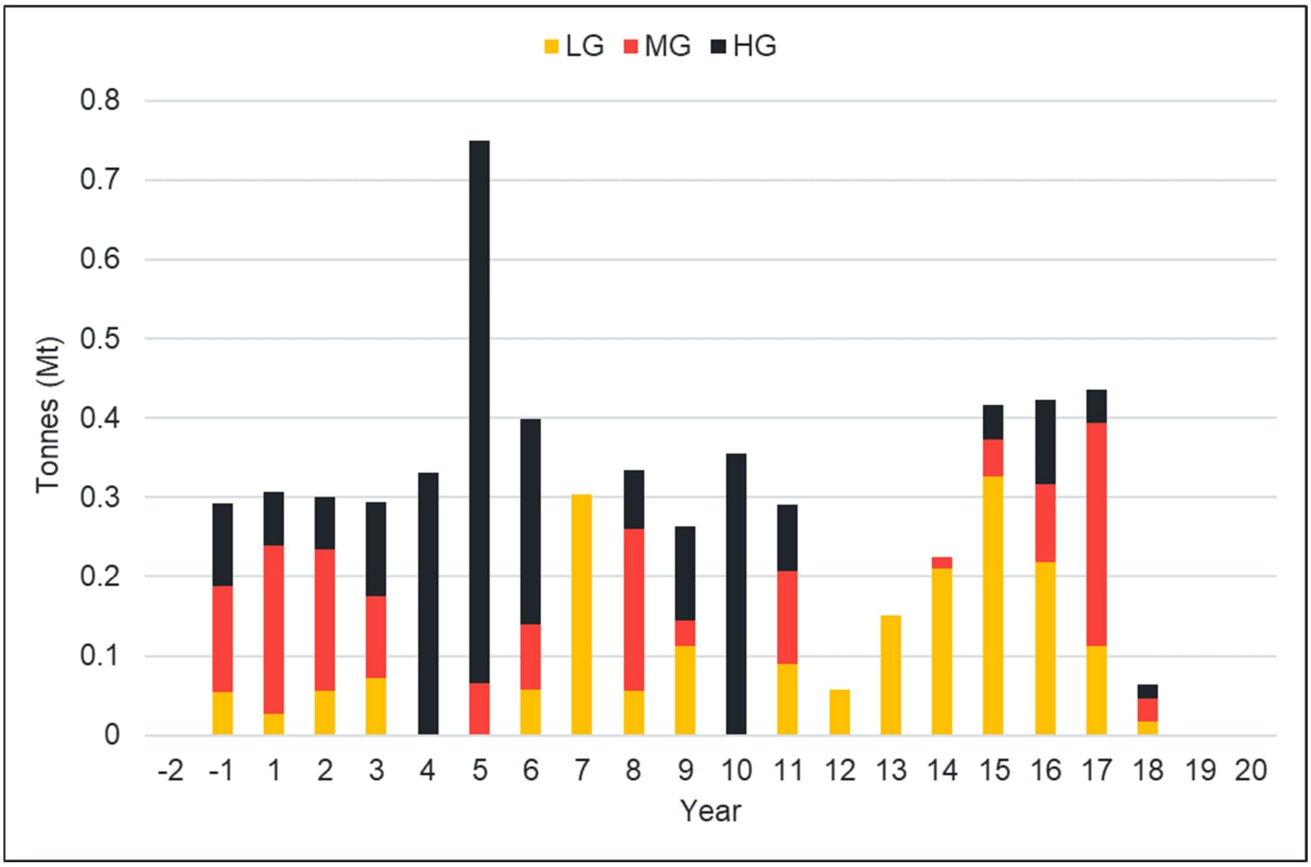


Figure 16-41 – Annual Stockpile Balance

The large amount of stockpiling required in Year 5 is due to the high ROM grade in that year. Plant feed limitations on grade and metal were placed based on RR design specifications, resulting in the higher grade needing to be stockpiled. This higher-grade material does become useful later in the schedule when mining progresses through lower grade areas.

16.5 MINE INFRASTRUCTURE

16.5.1. REFUGE STATIONS AND EMERGENCY EGRESS

Emergency egress is established through the escapeway raises connected to each level (Figure 16-42). These raises have been designed on both the North and South side of the mine RR defined the use of ladderways within the escapeway raises to allow personnel to move between levels.

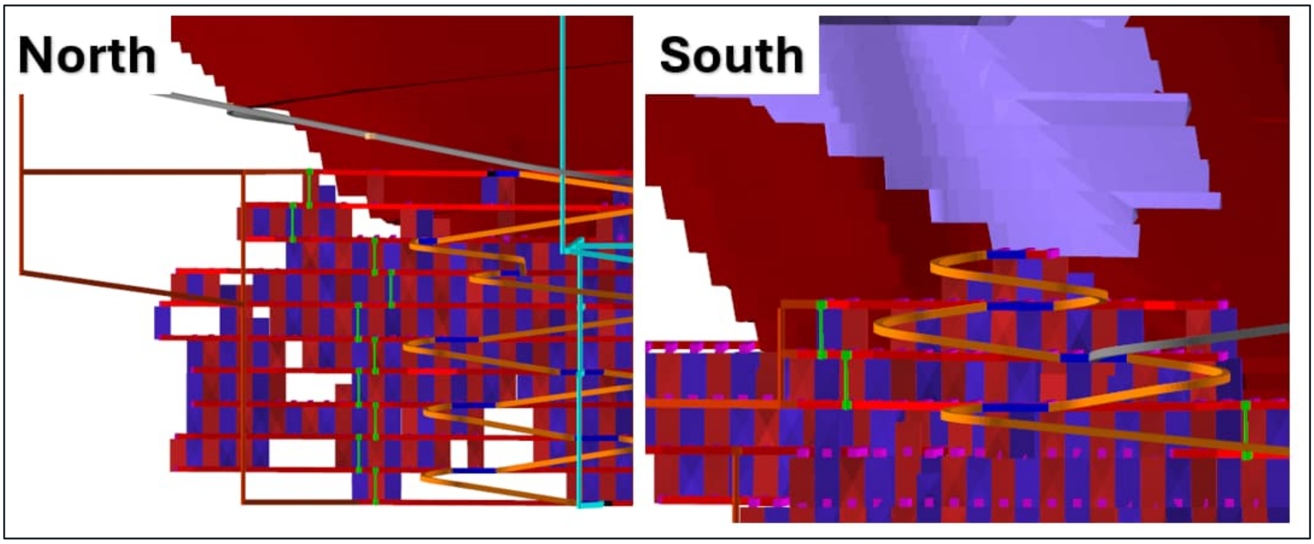


Figure 16-42 – Underground Mine Escapeways (Green)

Mobile refuge chambers are planned to be procured for the mine to ensure they are less than 1 000 m from any workplace. Self-sustaining mobile refuge chambers will be used. These provide all basic life support systems. The main workshop, located on the -140 m level, will have a lunchroom area and an additional refuge chamber.

16.5.2. MAINTENANCE WORKSHOPS

A workshop will be placed on surface with five heavy machinery and five light vehicle bays. This will care for open pit equipment and underground equipment requiring major maintenance works. An underground maintenance workshop will be constructed on the -140 m level (Figure 16-43) which will be used for general maintenance.

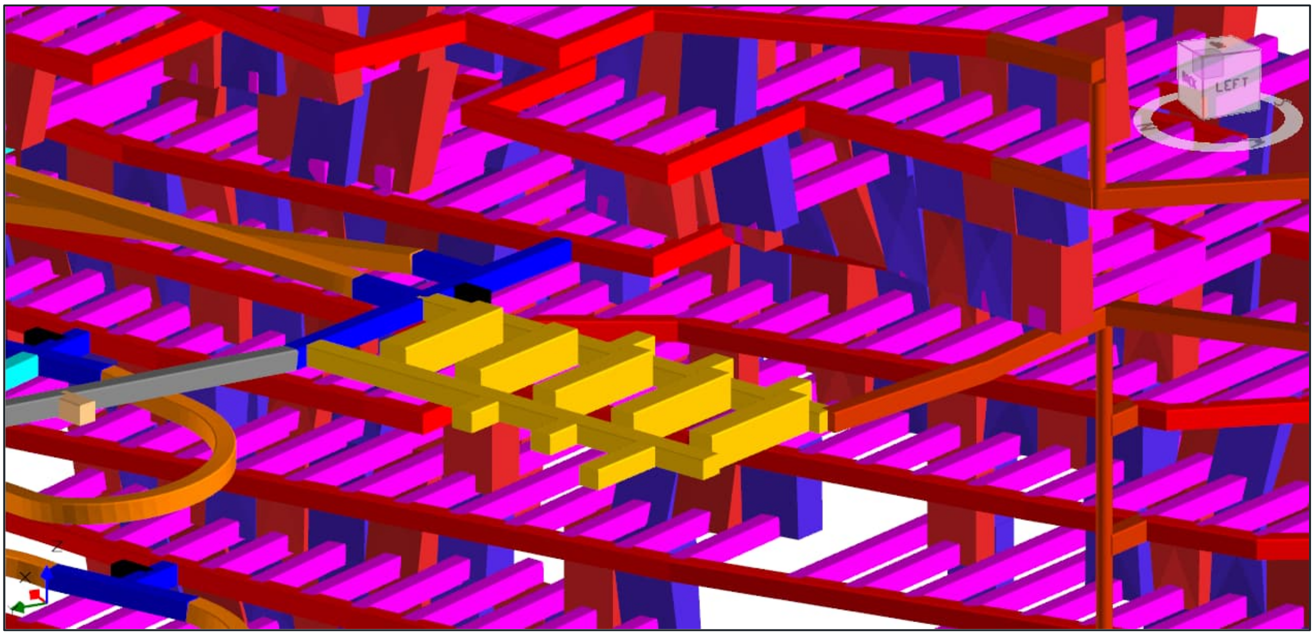


Figure 16-43 – Underground Maintenance Workshop on -140m Level

16.5.3. POWER SUPPLY

Power will be sourced from the national grid infrastructure located 6 km north of Ikkari. A 220 kV substation is available, which will be extended to supply the site through a 110 kV connection.

The receiving substation at Ikkari is to be placed south-east of the mine to and is defined as able to receive up to 25 MW.

16.5.4. WATER SUPPLY

The site will have a positive water balance with no fresh water intake. A potable water treatment plant has been designed which will utilise treated water from site dewatering.

16.5.5. DEWATERING

Surface water in the mining areas will be managed through a series of pumps and sumps located within the open pit. Skid and feed pumps will collect water at the lowest mining elevation and discharge to the appropriate outlet.

Underground water inflows are estimated between 80 to 90 L/s as outlined in Section 16.3.3. Development tunnels are graded to assist and enable drainage. Declines will have face and sidewall sumps with pumps for nuisance inflows. On each level, sumps will be placed off the level access, each containing 6 kW pumps suspended under a steel frame connecting into the main dewatering pipelines. The main underground dewatering scheme will consist of a three stage pumps running from the bottom to the top of the underground mine. Each stage collects water from the immediate level and above.

16.5.6. COMPRESSED AIR

Most equipment open pit and underground has compressed air built in. Mobile compressors will be used for compressed air supplies where required.

16.5.7. COMMUNICATIONS

Leaky feeder and Wi-Fi communications will be installed throughout all main lateral underground development. Every person underground will carry handheld radios to communicate underground and to surface.

16.5.8. EXPLOSIVES STORAGE FACILITIES

Blasting products will be stored above ground. Separate storage facilities will be made for bulk explosive products and detonators. Underground storage has also been planned for shorter-term, operational storage, again separated by emulsion and detonators.

17 RECOVERY METHODS

The flowsheet for the Ikkari Project processing plant was established on the basis of laboratory scale testwork, mainly performed at the Grinding Solutions Ltd. laboratory (GSL) located in the UK. The metallurgical testwork programs were carried out using composite samples prepared from representative drill cores obtained from the Ikkari deposit. The testwork results are summarized in Chapter 13.

The comminution circuit consists of a single stage crushing circuit, followed by a grinding circuit containing a semi-autogenous mill (SAG) in closed circuit with a pebble crusher followed by ball mill in closed circuit with hydrocyclones (SABC circuit).

A gravity circuit, followed by intensive cyanidation, recovers free gold from the hydrocyclone underflow, while the cyclone overflow is transferred to a carbon-in-leach (CIL) circuit. Gold is recovered in a pressure Zadra elution circuit followed by electrowinning and a gold refinery to produce doré. A cyanide destruction circuit (SO_2/Air) is also included to treat CIL tailings before being sent to a separate tailings filtration plant. The filtration plant is equipped with pressure filters to produce dewatered tailings destined for co-disposal.

The mineral processing facility and the filtration plant have both been designed using standard and widely accepted technologies.

A schematic process flow diagram of the process plant is presented in Figure 17-1.

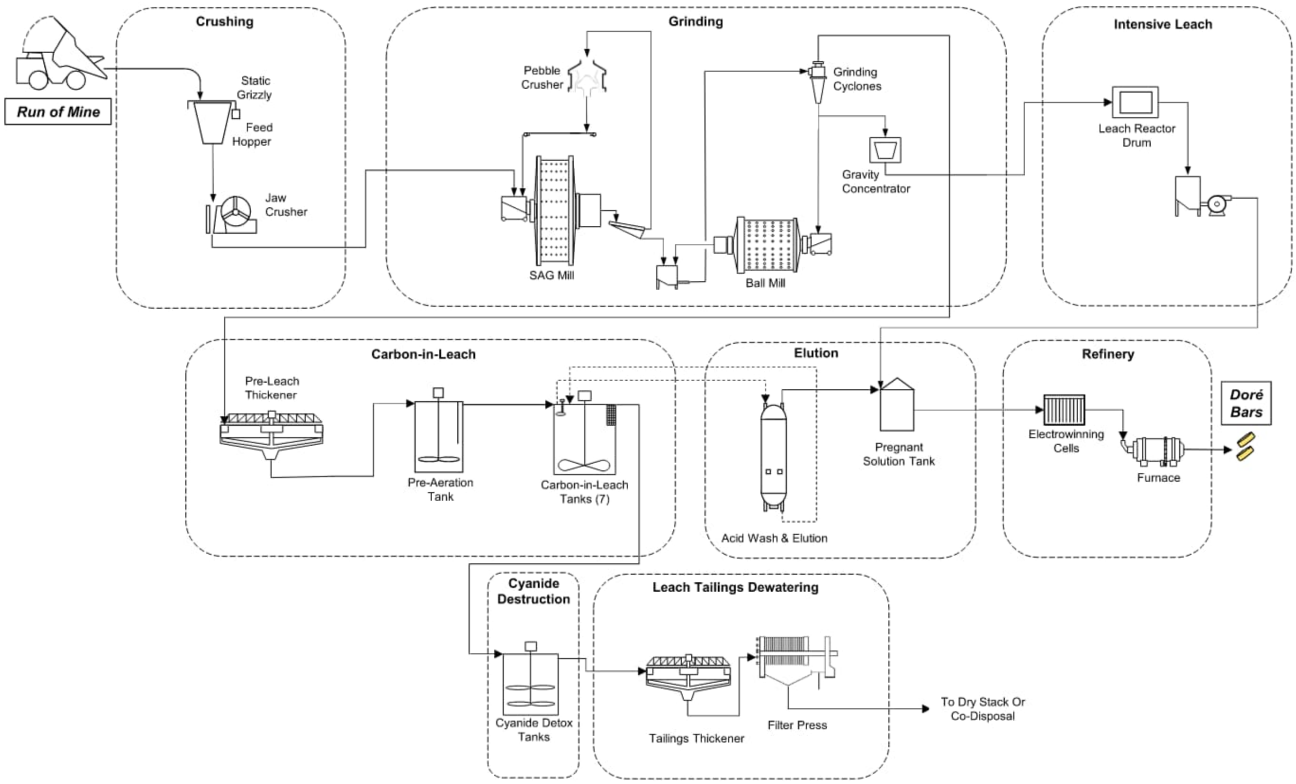


Figure 17-1 – Simplified process flow diagram

17.1 PROCESS PLANT DESIGN CRITERIA

The processing plant is designed to process ore at an average throughput of 9 589 tonnes per day (t/d) or 3,500,000 tonnes per annum (t/a), equivalent to milling rates of 434.3 tonnes per hour (t/h) with plant availability of 92%. The design criteria to determine the sizing of the equipment are based on a maximal daily throughput of 11 986 t/d which includes a 25% design factor.

The processing plant described in this chapter is based on the requirements for the first ten years of operation of the mine, when the open pit is in production. When mining reaches the underground portion of the orebody, it is expected that the annual throughput to the plant will decrease to about 2.0 Mtpa. Provisions have been made in the design, and sustaining capital has been estimated to account for the changes required to the plant for when the throughput is reduced to 2.0 Mtpa. A summary of the proposed modifications is available at the end of this chapter.

The tailings filtration plant is designed to receive the processing plant tailings which are produced at a nominal throughput of 434.3 t/h. The filtration plant design includes a 25% design factor, allowing the plant to process up to 542.9 t/h of solids.

Table 17-1 presents an overview of the main process design criteria used as a basis for the design. The values presented were derived from testwork data, WSP's database or based on Rupert's requirements.

The plant overall gold recovery is calculated to be 95.8% based on 29.3% obtained in the gravity circuit and 94.9% in the CIL circuit. Minimal gold losses in the intensive leaching, elution, stripping and refining circuits have also been accounted for in the overall recovery calculation.

Table 17-1 – Summary of key process design criteria

Description	Unit	Value
Plant throughput - annual	Mt/a	3.5
Plant throughput - nominal	t/d	434.3
Feed composition – open pit		
Felsic	%	37
Ultramafic	%	63
Average Au feed grade (nominal)	g/t	2.28
Solids specific gravity		2.90
Bond work index – open pit	kWh/t	16.1
SMC Axb – open pit		33.2
Crushing plant utilization	%	75
Process and filtration plant utilization	%	92
Au recovery by gravity circuit	%	29.3

Description	Unit	Value
Grind size to leaching, P_{80}	µm	100
Leaching retention time	h	24
Au recovery by CIL	%	94.9
Carbon stripping, regeneration capacity	t/d	12
Overall Au recovery	%	95.8
Residual CN_{WAD} concentration at plant discharge (max)	mg/l	1.0
Tailings production rate – nominal	t/d	434.3

17.2 PROCESS PLANT FACILITIES DESCRIPTION

17.2.1. CRUSHING, STORAGE AND RECLAIM

Run-of-mine (ROM) material transported from the open pit mine consists of felsic and ultramafic ore. A static grizzly (1 000 mm) mounted above the crushing circuit feed bin and a rock breaker are installed. Material is discharged from the bin to a vibrating grizzly feeder below where the oversize material is directed to an open circuit jaw crusher to reduce the material to a P_{80} of about 155 mm. The crushed product and grizzly feeder undersize material are collected on a sacrificial conveyor and then to the crushed ore stockpile feed conveyor. A dust collection system captures the dust created by the crushing operations. A conveyor tramp steel magnet is mounted above the sacrificial conveyor to protect the downstream equipment.

The crushed ore stockpile is designed with a live capacity equivalent to approximately 12 hours of production which corresponds to about 6 000 t, and an overall capacity of about 15 000 t. Ore is reclaimed from the stockpile through the reclaim tunnel. Three reclaim feeders discharge the crushed ore onto a belt conveyor that feeds the SAG mill. The SAG mill feed conveyor is fitted with a weightometer. A dust collection system captures the dust created in the reclaim tunnel.

17.2.2. GRINDING CIRCUIT AND GRAVITY RECOVERY

The grinding circuit is a SABC circuit, comprised of a SAG mill followed by a single ball mill. The SAG mill operates in closed-circuit with a pebble crusher, while the ball mill operates in closed-circuit with hydrocyclones. The product particle size exiting the grinding circuit as cyclone overflow is a P_{80} of 100 µm.

SAG Mill Circuit

The crushed rock is conveyed to the SAG mill feed chute via SAG mill feed conveyor. Process water is added to the mill feed chute to achieve a slurry density of about 75% (w/w) solids within the mill. A SAG mill size of 8.50 m in diameter x 4.25 m effective grinding length (EGL) with a total installed power of 6 400 kW was selected for primary grinding. The mill is operated with a charge of 125 mm diameter steel balls.

The SAG mill product discharges through a trommel screen where the oversize is conveyed to the pebble crusher and the undersize discharges into the cyclone feed pump box. A self-cleaning tramp metal belt magnet is mounted above the pebble recycle conveyor. The pebble crushing circuit processes the equivalent of 20% of the new SAG mill feed. The crushed pebbles are recirculated to the SAG mill feed conveyor. The circuit is also equipped with a pebble diverter gate should the requirement to bypass the pebble crusher arise. In this case, material from the trommel screen oversize is conveyed directly back to the SAG mill feed conveyor.

Ball Mill Circuit

A ball mill size of 6.00 m in diameter x 10.50 m EGL fitted with a trommel screen, and a total installed power of 6 800 kW is selected for secondary grinding. Process water is added to the mill feed chute to achieve a slurry density of about 72% (w/w) within the mill. The mill is operated with a charge of 50 mm diameter steel balls.

The cyclone feed pump box receives slurry from the SAG mill discharge trommel screen undersize and ball mill discharge. The hydrocyclone cluster is fed by a variable speed centrifugal pump connected to the cyclone feed pump box. Process water is added to the cyclone feed pump box to control the slurry density. The design circulating load from the hydrocyclones to the ball mill is 300% of the SAG mill fresh feed. The overflow is sent to the vibrating trash screen ahead of the pre-leach thickener and CIL circuit. The oversize material in the underflow is returned to the ball mill with a portion sent to the gravity circuit.

Gravity Circuit

The underflow from three out of eight operating hydrocyclones (37.5%) feeds two gravity scalping screens via a distributor. The coarse oversize material from the scalping screen is sent to the cyclone feed pump box. The undersized materials feeds two gravity concentrators arranged in parallel. A batch intensive cyanidation system is used to process the gold concentrate from both gravity concentrators. The gravity concentrator tailings return to the cyclone feed pump box.

The pregnant gravity solution from the intensive cyanidation system is pumped to the electrowinning circuit located in the gold room. The intensive leach tailings return to the cyclone feed pump box.

17.2.3. CARBON-IN-LEACH

Pre-Leach Thickening

The slurry from the hydrocyclones overflow pass through a trash screen before feeding the pre-leach thickener feed box. The pre-leach high-rate thickener diameter is 34 m. The underflow from the pre-leach thickener at 45% solids (w/w) is pumped to the CIL circuit. The thickener overflow water is sent to the process water tank.

CIL

The pre-leach thickener underflow slurry is pumped to the CIL circuit consisting of one pre-aeration tank followed by seven CIL tanks operating in series. Each tank is 16 m in diameter, 16.5 m in height, and mechanically agitated. The total live volume of each of the CIL tank is 3,181 m^3. Slurry is first sent to the pre-aeration tank to be aerated with sparged plant air. Lime is added to the tank to maintain a pH of approximately 10.5. Leaching is performed in the CIL tanks using sodium cyanide, and lime is added to control the pH. All CIL tanks are also sparged with compressed air to keep the dissolved oxygen concentration at sufficient levels for gold leaching.

The circuit allows for 24-hour residence time at a design feed rate. Each tank is interconnected with launders to allow slurry to flow sequentially. Inter-tank screens are used to retain the carbon within the upstream tank. Reactivated and fresh barren carbon is introduced to CIL tank 7. The carbon advances countercurrent to the slurry flow periodically via carbon advance pumps until the loaded carbon reaches CIL tank 1. The loaded carbon is then pumped to the loaded carbon pump box from which it is pumped to the loaded carbon screen in the elution circuit. Tailings slurry from CIL tank 7 flows by gravity to the carbon safety screen to recover any remaining carbon in the event of damage to the interstage screen. Tailings is then discharged to the cyanide destruction circuit.

17.2.4. ELUTION AND GOLD RECOVERY CIRCUIT

The gold recovery circuit is based on the processing of 12 t/d of loaded carbon with a pressure Zadra process.

Carbon Elution

The loaded carbon batch from CIL tank 1 is transferred once daily to the acid wash vessel through the loaded carbon screen. The slurry recovered at the screen undersize is returned to the CIL tank 1.

A dilute solution of hydrochloric acid is circulated through the acid wash vessel to remove contaminants in a closed circuit with the acid wash circulation pump and dilute acid tank. Once the washing step is complete, the carbon is rinsed with treated water before transferring to the carbon strip vessel.

A hot strip solution consisting of dilute sodium hydroxide and sodium cyanide is introduced to the carbon strip vessel at an elevated pressure. Each stripping cycle operates approximately for 12 hours. The elution circuit operates in a continuous closed loop with the electrowinning cells and associated equipment. Barren strip solution is reused from one cycle to another with a periodic bleed to the CIL circuit to control impurity buildups. Transport water flows to the strip vessel and the stripped carbon transfer pump sends the stripped carbon to a dewatering screen. The undersize of the screen reports to the fine carbon collection tank and the oversize reports to the carbon transfer dewatering screen prior to the carbon regeneration kiln.

Carbon Regeneration and Fines Handling

A carbon regeneration kiln reactivates the stripped carbon to an activity close to its original level. The regeneration kiln operates at a nominal temperature of around 850°C. Reactivated carbon is quenched with treated water upon exiting the kiln. Fresh carbon is also added to the reactivated quench tank as required. It is then pumped to the carbon sizing screen to remove the fines. The adequately sized carbon is recovered at the screen oversize and is returned to CIL tank 7, while the screen undersize reports to the carbon fines tank. The carbon fines tank also receives the undersize from the carbon transfer dewatering screen and fines from the kiln screw feeder. It is then pumped to the carbon fines filter press for dewatering. The filter press cake is bagged to be transported off-site once sufficient inventory has built up. The carbon fines filter press filtrate returns to the carbon fines tank.

Electrowinning and Refining

The electrowinning is done “in-line” with the stripping circuit and split between two electrowinning cells. Gold contained in the pregnant solution from the intensive leach circuit and the Zadra stripping circuit is deposited onto stainless steel cathodes and the solution exiting the cells is pumped to the barren strip solution tank. The gold bearing sludge from the cathodes is washed off and the gold sludge filter press removes excess moisture. The gold sludge is then dried in an oven and mixed with fluxes before subjected to the induction smelting furnace. Doré is poured from the furnace into a cascade of moulds. The refining equipment is designed to handle the gold from the stripping circuit and from the gravity recovery system.

17.2.5. CYANIDE DESTRUCTION CIRCUIT

Tailings from the CIL circuit is sent to a carbon safety screen where the oversize is reported to a carbon bag and the undersize is reported to the CIL tailings pump box. Tailings are then pumped to the cyanide destruction feed box at about 44.3% solids (w/w) which also receives copper sulphate, sodium metabisulphite and lime. Cyanide destruction is completed using the Inco SO_2/air process. The process occurs in two tanks operating in series, providing a total retention time of 2 hours. Each tank is 11.5 m in diameter, 12.3 m in height, and mechanically agitated. Plant air is injected through cone spargers located at the bottom of the tank to oxidize the cyanide species present. The cyanide destruction tailings then flow to the cyanide destruction pump box and are to the tailings filtration plant.

17.2.6. TAILINGS FILTRATION PLANT

The tailings filtration plant is located in a nearby separate building approximately 400 m from the process plant. It is used to dewater the detoxified tailings to a moisture level suitable for co-deposition.

Leach Tailings Thickening and Water Management

Cyanide destruction tailings pass through the leach tailings trash screen before feeding the leach tailings thickener feed box to remove any tramp material that could damage the pressure filters. The leach tailings thickener diameter is 34 m. The underflow from the thickener at around 60% solids (w/w) is pumped to two leach tailings filter feed tanks installed in parallel. The thickener overflow water is pumped to the process water tank, water treatment plant (WTP) and the clarifier. The clarifier produces water that is usable for gland water and filters washing requirements within the filtration plant. The clarifier underflow is returned to the leach tailings thickener.

The tailings filtration plant has its dedicated flocculant mixing system. Flocculant is added to the tailings thickener and the clarifier to assist in solids sedimentation.

Filtration

Two filter presses are in operation to meet the filtration plant required capacity while one is on standby. Each with approximately 1150 m^2 of filtration area, are used to produce a cake at a density of 87.5% solids (w/w). The operating filters each have their own dedicated filter feed tank which also have the option to feed the spare. Cakes from the filters are then discharged onto belt conveyors and directed to the filtered tailings storage.

For the first ten years of open pit mining, the filtered tailings is deposited to a co-disposal facility. After underground production begins, a portion of the filtered tailings is used for paste back filling.

Clarified water provides core and cloth wash water for the filter press wash cycle. The filtrate, core and cloth wash waters from all three filters are collected into a common agitated filtrate tank which are then pumped back to the leach tailings trash screen.

17.2.7. PROCESS AND FILTRATION PLANTS SERVICES REQUIREMENT

Water Requirements

A process water tank is installed in the processing plant. It collects water from the pre-leach thickener overflow as well as process water from the filtration plant. Treated water is used as a make-up to meet the process water requirements.

In the filtration plant, a water tank collects the leach tailings thickener overflow. A portion of process water from this tank is pumped to the clarifier in order to produce water suitable for the tailings filters washing steps. The remaining water from the leach tailings thickener overflow is split between the water treatment plant and the process water tank located in the processing plant. When the paste plant is operational, water from the leach tailings thickener overflow will also be provided to meet the paste preparation water demand.

Treated water is collected in the treated water tank located in the processing plant. Treated water is mainly used as gland water and for reagents preparation.

Air Requirement

Compressors and a receiver are planned to handle process plant air requirements within the plant. Dryers are installed to meet services and instrumentation air demand. A set of one air compressor and air receiver are used to supply cyanide destruction. The crushing circuit has a dedicated compressed air system.

The filtration plant is serviced by separate compressors located in the filtration plant.

17.2.8. REAGENTS SYSTEMS

The reagents preparation area includes receiving, mixing, and metering systems for the reagents required within the process and filtration plant. Reagents tanks are located within designated berms so liquids can be contained in an event of a spill. The area is easily accessible by delivery trucks.

A summary of the reagents required in the process and filtration plants along with expected mixing requirements are shown in Table 17-2.

Table 17-2 – Summary of Reagents

Description	Delivery	Preparation and Dosing
Lime (CaO)	Solid	Lime silo, slaking system, distribution tank, distribution pumps – pressurized distribution loop
Sodium metabisulfite (SMBS)	Solid	Mixing tank, distribution tank, metering pump
Copper sulphate pentahydrate ($CuSO_4.5H_2O$)	Solid	Mixing tank, distribution tank, metering pump
Flocculant	Solid	Eductor, mixing tank, inline mixer, metering pump
Sodium Cyanide (NaCN)	Solid pellets	Mixing tank, distribution tank, distribution pump – pressurized distribution loop
Sodium Hydroxide (NaOH)	Liquid	Storage tank, distribution pump

Description	Delivery	Preparation and Dosing
Hydrochloric Acid (HCl)	Liquid	Storage tank, distribution pump
Antiscalant	Liquid	Tote, metering pump
Leach aid	Solid	Bucket directly introduced to intensive leach
Fluxes	Solid	Bags directly introduced to flux bins

17.3 PLANT DESIGN FOR 2.0 MT/AOPERATION

The plant is designed for the first years of operation when the throughput to the mill will be around 3.5 Mt/a. During the second half of the LOM when mining operations will move underground, the throughput will decrease to 2.0 Mt/a.

An analysis of the equipment selection in all the process areas has been completed to identify the key changes needed to the plant in order to continue operating adequately at the lower feed rate.

No design choices were made upfront for the 2.0 Mt/a scenario except for the CIL circuit configuration. The tanks sizing has been determined in order to operate at least four CIL tanks at 2.0 Mt/a (seven are in use at 3.5 Mt/a) while maintaining the 24h residence time.

The comminution circuit will require changes in operating parameters. The jaw crusher closed side setting could be increased, and the crushing circuit operating hours might be reduced. It is expected that the SAG and ball mills will operate with a lower ball charge and at lower speeds. The final grind size will likely be reduced to a P_{80} around 75 microns.

In other areas of the process, it is expected that equipment will be bypassed if necessary or longer residence times will be observed. Some units might be put on stand-by (, gravity concentrator, filter, etc.).

More critically, some slurry pumps and slurry piping will require replacements to ensure proper operation and critical velocities are maintained. An efficient execution strategy should reduce the plant downtime associated with these changes.

18 PROJECT INFRASTRUCTURE

18.1 EXISTING INFRASTRUCTURE

A summary of local existing infrastructure is provided below, the nearest sizeable town being the town of Sodankylä.

18.1.1. POWER

An existing overhead powerline is located on site, the current transformer building has an 800 kVA transformer, and it is possible to increase it to a 1000 MVA transformer. If necessary, the electricity supply can also be taken from the currently existing power line/transformer for another additional transformer (1 MVA). The current consumption power of the connection is 44 kVA.

18.1.2. ROAD ACCESS

The site is currently accessed along forestry tracks, which allows for heavy axal wheeled road vehicles to the proposed plant site and water treatment pond areas and northwards across the Saittajoki river to the high ground at Pahkalehto, where the co-disposal facility is located. Access to the West of this is on lower quality tracks.

Surrounding these at the lower elevations is marsh which is “easy to traverse” on foot according to the Finland Survey mapping legend. If access is required for the river diversion during summer months, when the ground is unfrozen a range of temporary load spreading formations can be used, avoiding the requirement for soft ground replacement and bulk earthworks exchange.

Further upgrading of the surface to the roads may be required using locally excavated silts and gravels depending on the permitted level of usage required prior to construction of the new main access off the main road 80 public highway between Sodankylä and Kittilä.

18.2 ON-SITE INFRASTRUCTURE AND PROPOSED LAYOUT

18.2.1. DESIGN APPROACH AND PFS ASSUMPTIONS

The following design basis for PFS assumptions was used in surface layout of the mine infrastructure:

- The main plant site and facilities are located at Industrial area -option 1 (Tehdasalue – vaihtoehto 1);
- Provision of a compact and efficient site area is required which will assist with the permitting of the project;
- The river diversion which runs to the North of the co-disposal facility; and
- Consideration for the mine closure.

The following constraints to local assets are considered whist establishing the surface infrastructure layout:

- The siting of all infrastructure within the property boundary;
- Avoiding the low land in proximity to the Saittajoki river;
- Avoiding restriction areas related to protected predatory bird nests to the East;

- The Southern edge of the open pit with a 20m offset to the property boundary to accommodate drainage, de-watering wells, single lane access road and safety bund;
- The other outer edges of the open pit;
- The location of footprints for stockpiles at the ROM pad area; and
- Portal access for the development of the underground mine implemented at a later stage of life of mine.
- Location of known mineral deposits and occurrences, the subject of ongoing exploration activities.

The following interdependencies of infrastructure assets are also considered for the surface infrastructure:

- Proximity of the ROM pad to both the entrance of the open pit and UG portal;
- Proximity of ore stockpiles at the ROM pad to the primary crusher bin;
- Alignment of the process plant layout within the hillside to minimise cut and fill earthworks
- Proximity of the filter plant to both the process plant and co-disposal facility;
- Location of paste plant feeding into the underground mine;
- The slurry pipework connection of process main alignment with filtration plant;
- Siting of other buildings within proximity to the process plant, administration, canteens, welfare and HME workshop;
- Siting of other processes including the water treatment plant, associated ponds and pumping facilities which are currently assumed to be fed under gravity;
- The route of the treated water pipeline;
- A location required for on-site disposal of sludge from the water treatment process;
- The proximity of the water treatment plant to the process plant;
- The route of the powerline connection from the Finland national grid infrastructure and the location of the 110KV outdoor sub-station (switches and transformers) and 20kV building;
- Separation of heavy mobile equipment (HME) and light mobile equipment (LME) traffic where practicable;
- Constructability of pads (or terracing) within the hillside;
- Road network connectivity for HME to ROM pad area, co-disposal and workshop area facility whilst considering width of plant;
- Proximity of the HME and LME workshops and parts warehouse;
- Access for HME to the fuel storage area;
- Proximity of administration to all other buildings;
- Providing warehouse, stores and core logging facilities in three adjoining buildings which can be expanded in phases;
- A central location of mine rescue / ambulance and fire truck within the mine site area;

- Access to a helicopter pad;
- Road network connectivity for LME to the other mine infrastructure assets whilst considering width of plant, especially for deliveries of large items, and frequency of usage for either two-way or one-way with passing bays, whist also considering width of plant and usage;
- Gradients and turning circles of roads;
- Sufficient car parking;
- Fencing of the mine working areas;
- Gated control at the main site entrance and exit for personnel and deliveries;
- Surface drainage which needs to gravitate;
- The location of overhead power lines into the 110KV outdoor sub-station;
- The location of underground power cables to infrastructure assets; and
- Locations of telecommunication towers (if required) and telemetry.
- The designated location for temporary storage of:
  - peat and topsoil;
  - overburden (moraine/glacial soils) for use as engineering fill; and
  - material deemed to be unsuitable for use as engineering fill.

18.2.2. PROPOSED LAYOUT

The developed layout of the overall mining area is shown in Figure 18-1. The details of the plant site layout are shown in Figure 18-2. The proposed locations are described in section 18.3.

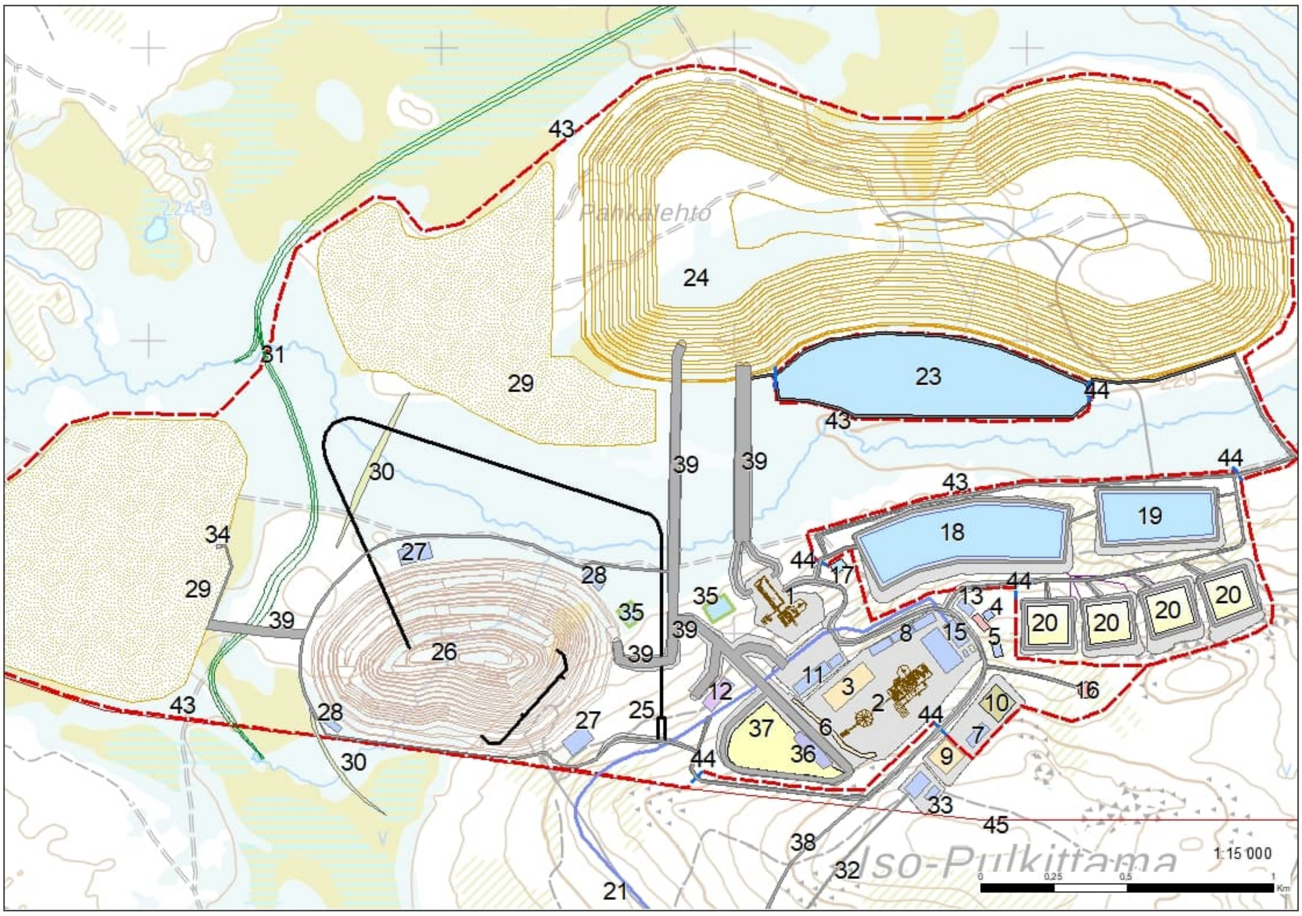


Figure 18-1 – Plan of overall mine layout

Table 18-1 – Overall Layout Map's Item Number Descriptions

Map item number and description	
1: Filtration and paste plant	24: Co-disposal facility
2: Process plant	25: Underground access portal
3: Laydown area	26: Open pit
4: Heating facility	27: Air intake
5: Additional fuel station	28: Exhaust building
6: ROM wall	29: Peat storage areas
7: Administration building	30: Embankment
8: Warehouse, store and logging facilities	31: River diversion
9: Civil carpark	32: Power line,
10: LME carpark	33: 110 kV and 20 kV buildings

Map item number and description	
11: HME and LME maintenance workshops	34: Explosive storage
12: Fuel station	35: Groundwater holding ponds
13: Fire protection water pumphouse,	36: ROM Pad
14: Mine rescue, ambulance and fire truck facility	37: Ore stockpile
15: Water treatment plant	38: Main access road
16: Helicopter landing pad	39: Haul road
17: Raw water intake	40: Access road option 1
18: Raw water pond	43: Fence
19: Treated water pond	44: Gates
20: Sludge disposal area	45: Property boundary
21: Treated water pipeline	
22: Sediment control dam	
23: Runoff collection pond	

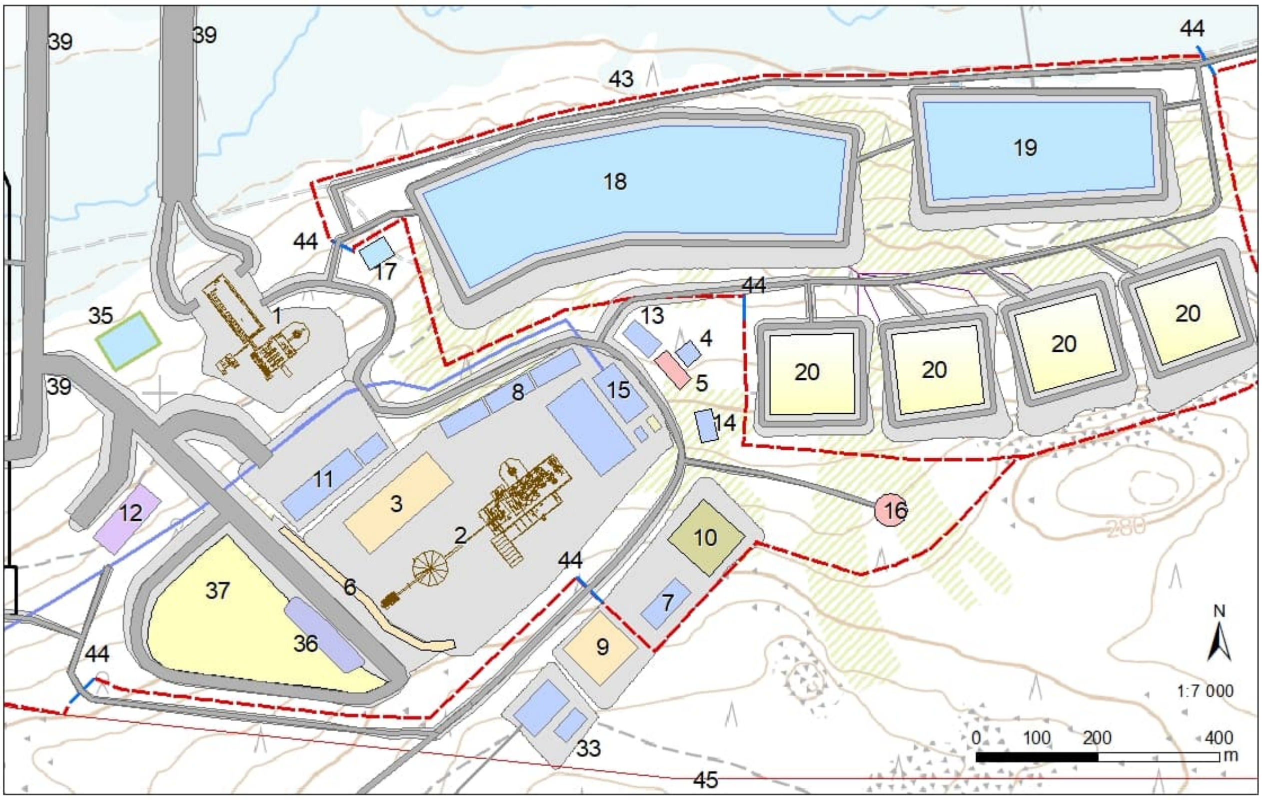


Figure 18-2 – Plan of plant site area

18.2.3. PLANT SITE LOCATION

The land available for the plant site and associated assets is on hill South of the Saittajoki river and to the east of the open pit and west of the ecological constraints. This has an area of approximately 1.6 km^2 ranging from approximately 225 m up to 290 m elevation, close to the summit of Iso-Pulkittama. The planned plant site area is open forested area, with also some boulder fields. Access to the site can be gained along existing forestry access tracks. A photograph of the typical site area for the planned plant location is shown in Figure 18-3.

Figure 18-3 – Typical site area of the planned plant location

18.3 DEVELOPMENT LOCATIONS

18.3.1. ROM PAD AREA

The ROM pad has a designed elevation of 282.5 m, to provide a 17.5 m height above ground floor level of the primary crusher building as shown in Figure 18-4.

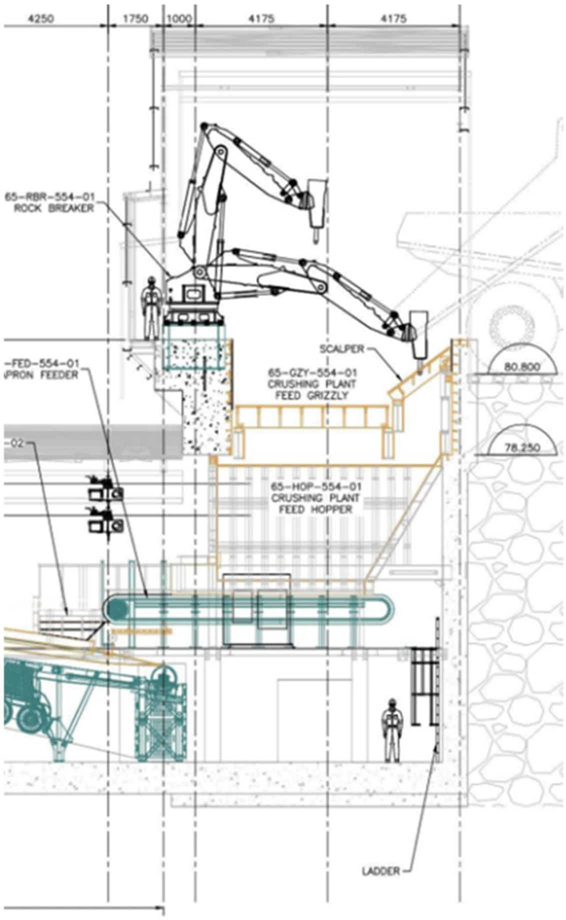


Figure 18-4 – Cross section of primary crusher building

The ore stockpile is to be crescent shaped at the ROM working face to allow for different grades to be fed into the process plant. The ore stockpile is required to have a capacity of 749 254 t of storage in Year 5. This is approximately 400 000 m³, which with a height of 10.5 m requires a footprint of approximately 42 200 m².

The ROM pad is lined with the liner extending up to the edge of the ROM wall. Geochemical testing will need to inform the permitting decision for this requirement. This will need to be sufficient depth below the stockpile to avoid damage and will require an area of approximately 70 000 m². A drainage system will also be required to transfer seepage collected from above the liner to the raw water pond.

Access for haul trucks is provided around the stockpile to allow for flexibility with the off-loading of ore.

18.3.2. ROM RETAINING WALL

The maximum height of the ROM wall is 17.5 m, located adjacent to the primary crusher bin.

The ROM stockpile restricts the alignment of the ROM road. The wall will therefore need to extend Northwards to support the haul road up to the ROM pad. This will avoid the width required for a cut slope thereby providing space for the Laydown Area and HME Workshop below.

The wall will also need to extend Southwards to retain the hill near to the property boundary and avoid the requirement for a cut slope to the Southwest of crusher building.

Initial estimates allow for a minimum height of 5 m at either end, below which a slope is width of 10 m wide in plan for a 1:2 gradient is required. The height increases with the rise of the haul road up to the maximum height behind the primary crusher building and then descends on the Southern side to height required to support the cut below the main site access road. This is estimated to be constructed with in-situ reinforced concrete having a wall width of 1 m and foundation base depth of 1 m. The base width is estimated at two thirds of the wall height. An additional 10% is also applied for counterforts. Detail design of the reinforcement, and both sliding and overall stability assessment for the configuration is also required at a more detailed stage of study. These analyses will depend on the founding conditions. Future design may consider the wall construction being, where practicable, integral with the primary crusher building foundations.

Alternative solutions could be considered using pre-cast panels and reinforced earth ties. However this would need to allow for the construction of the liner beneath the ore stockpile, which would need to be sited a few meters below the ROM elevation whilst avoiding risk of rupture from heavy plant operating on the ROM.

A guard rail is required at the top of the retaining wall to prevent persons from falling from height.

18.3.3. PROCESS PAD

The pad has a designed elevation of 265 m with an approximate length 600 m and width 300 m. It accommodates the process plant which is in a 50 degrees North direction across the hillside. This allows the process plant to be at the same elevation and avoids main items such as the SAG and ball mills being sited on made ground. Adjustment of the pad elevation may be required to suit the foundation conditions. The outer limits of the plan shape are formed from the ROM wall to the Southwest and the alignment of the main site access road and associated earthwork slopes around the perimeter.

The current arrangement has an elevation difference which would assist with energy and pumping required.

Other assets and buildings located at the process pad include:

- The water treatment plant and associated storage tanks;
- Warehouse, Stores and Logging facilities constructed in a modular alignment to allow for staged expansion; and
- A laydown area for construction and maintenance.

18.3.4. HME AND LME WORKSHOPS

Both the HME and LME maintenance workshops are located to the Northwest of the Process Pad. These two buildings are sited at an elevation of 255 m, which is approximate to the existing ground level at the Southeast edge. The footprint is approximately 250 m long and 125 m wide. This is to provide space for the building and turning area for haul trucks. It is currently assumed that the mining haul truck fleet will also park at the Northern edge of this pad. Access is gained via a junction off the haul road to the ROM.

There is a 10 m high difference with the neighboring process pad. With a maximum slope of 1v:2h this will require a horizontal distance between the pads of at least 20 m. The pad is on made up ground and consideration will be required for the design of the foundation to these relatively light structures. Vehicular access for the LME can be gained off the main site access road which has a switch back at this point.

18.3.5. FILTRATION AND PASTE PLANTS PAD

The Filtration and Paste Plant Pad is at an elevation of 235 m. The pad is shaped to also accommodate the paste building which will be constructed for the underground mine. The location was selected because it minimises haul distance for filtered tailings to the co-disposal facility.

This avoids both the filtration and paste plant being located on made ground. Adjustment of the pad elevation may be required to suit the foundations. The stockpile and loading area are currently sited on made ground. Access is gained via the main site access road which terminates at this location.

It has also been suggested that the filtration plant could be located to the West of the co-disposal facility, however the current arrangement provides for a more compact arrangement of facilities South of the river Saittajoki.

18.3.6. FUEL STATION AND 110 KV AND 20 KV BUILDING

The fuel station is located off the ROM haul road to the Southwest. Access for deliveries is via a specific gated entrance close inside the property boundary. This is at approximately 245 m elevation. Consideration will need to be given for the formation of neighboring earthworks for the haul roads to and behind the ROM road, and the fill slope East of the open pit.

The 110 kV electrical equipment and 20 kV building are also located close beside the property boundary and adjacent to the main site access road. The pad elevation is 290 m.

18.3.7. INCOMING POWERLINE

The alignment of the powerline up to Site Boundary and into the 110 kV building is shown in Figure 18-5. The powerline route from the Finnish national grid connection point up to the site boundary is shown on in Figure 18-6.

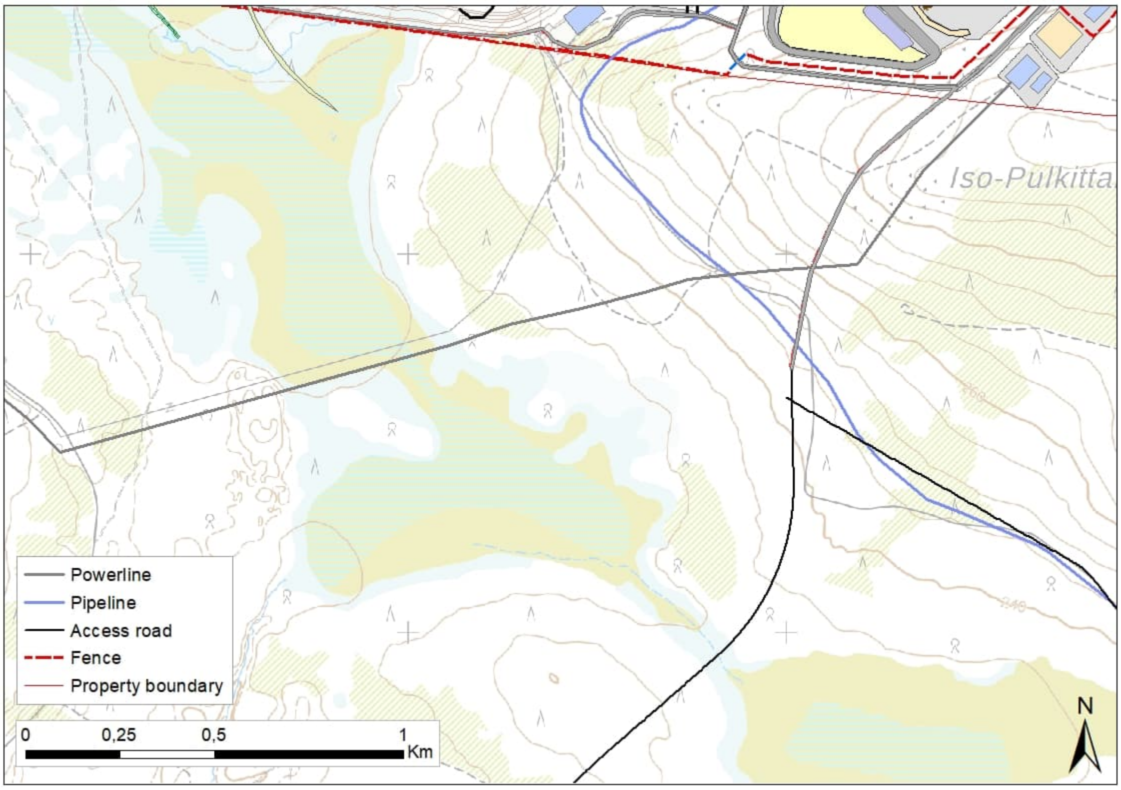


Figure 18-5 – Alignment of Powerline up to Site Boundary

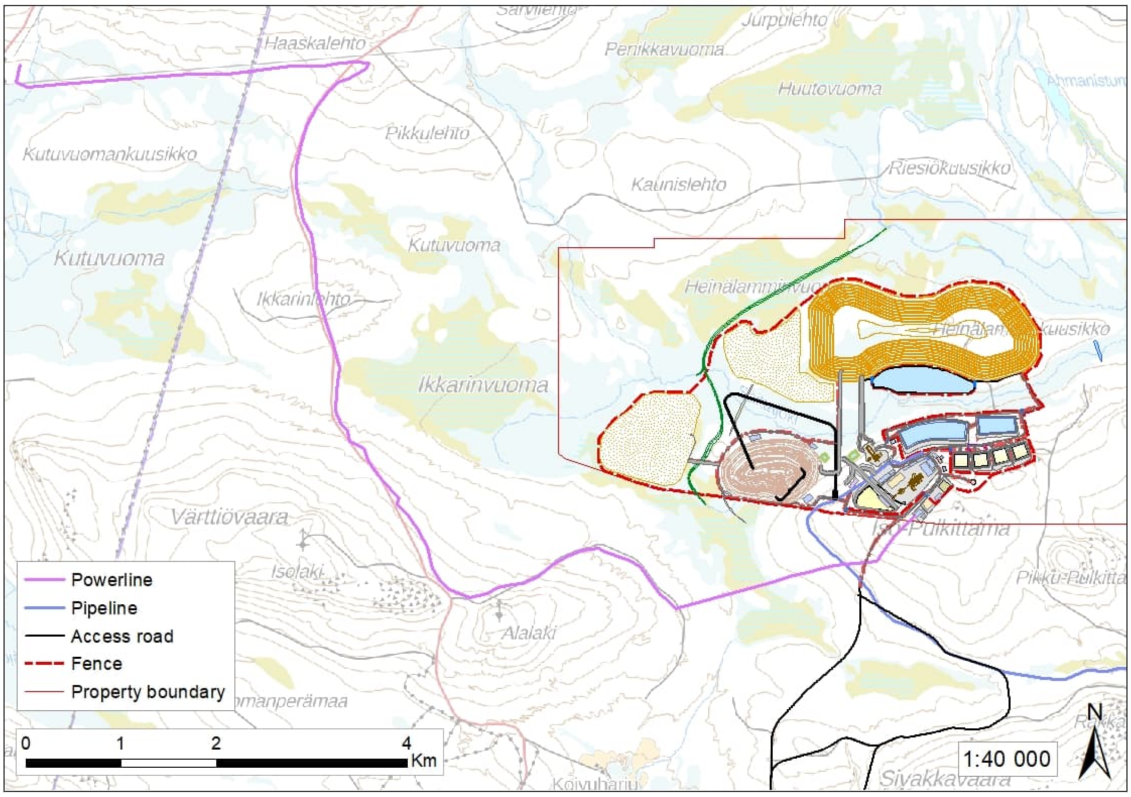


Figure 18-6 – Powerline route from Finnish national grid connection point up to the site boundary.

18.3.8. ADMINISTRATION BUILDING

The administration building is located to the Southeast of the main site access road close to the site entrance. Gated access is provided between this and the civil (public) car park. The LME car park is located to the Northeast and within the controlled mining area. The pad elevation is 280 m.

18.3.9. MINE RESCUE FACILITY, AMBULANCE, FIRE STATION AND HELICOPTER PAD

The mine rescue facility, ambulance and fire station are located off the main site access road at approximately 260 m elevation. This could, if required, be located on the other side of the access road to the helicopter landing pad, to be closer to the administration building.

Access to the helicopter pad is along a designated road approximately 300 m to the East of the mine rescue facility, also at approximately 260 m elevation on the outer curve of the main access road. The helicopter pad is to be in an area clear of tall vegetation up to 100 m diameter and will require a designed hard standing of approximately 15 m diameter, depending on size of aircraft designation.

18.3.10. HEATING AND ADDITIONAL FUEL STATION, FIRE PROTECTION WATER PUMPHOUSE

The heating and additional Fuel Station is off the main site access at approximately 255 m elevation on the outer curve of the main access road.

The fire protection water pumphouse is located at approximately 255 m elevation on the outer curve of the main access road. This is close to the main plant site area.

18.3.11. RAW WATER AND TREATED WATER PONDS, SLUDGE DISPOSAL AREA

The raw water pond has a designed capacity of 600 000 m^3 and the crest elevation with surrounding access track is at 230 m. The treated water pond has a designed capacity of 290 000 m^3 and the crest elevation with surrounding access track is at 225 m. These two ponds are cut into the hillside with material won from this being used for the confining embankments. A 1 m freeboard is allowed in both ponds.

The sludge disposal area is constructed as four lined ponds (150 m x 150 m) allowing 4 m of solids storage capacity with an allowance of 1 m for freeboard in each to provide estimated total capacity of 308 000 m^3. These cascade Eastwards, with transfer to the SE corner of the raw water pond cut slope. Staged construction is also possible with expansion of the capacity depending on the actual volumes of sludge being generated.

18.3.12. ROADS

Basis of Road Design

The roads within the mining area are categorized for usage as shown in Table 18-2. This also shows design parameters for the running widths, shoulder widths, drainage reserve as well as pavement layers (wearing course, base course and sub-base). The road network layout is shown in Figure 18-1.

Table 18-2 – Design Parameters for Roads

Road / Access Track	Running Width (m)	Shoulder Width (m)	Drainage Reserve (m)	Total Top Width (m)	Sub-Base Course Depth (mm)	Base Course Depth (mm)	Wearing Course Depth (mm)
Mining haul road - 2 way	27.0	0.0	5.0	32.0	300	800	450
Mining haul road - 1 way	13.5	0.0	5.0	18.5	300	550	450
Filtered tailings haul road - 2 way	24.2	0.0	3.0	27.2	300	800	450
Filtered tailings haul road - 1 way	12.1	0.0	3.0	15.1	300	550	450
Main site access road - 2 way	7.5	1.5	1.2	10.2	300	300	300
General access track - 2 way	6.0	1.0	1.1	8.1	-	150	150
General access track - 1 way	5.5	0.0	0	5.5	-	150	150

It is the intention to separate LME and HME traffic where possible. However, this is not achievable with perimeter access road to the North of the open pit, unless a bridge or tunnel crossing is provided.

Where possible longitudinal gradients are designed to be flatter than 1:12.

Curvatures are developed for PFS level of study depending on the road usage. The main site access road has snaking arrangement and swept path analysis may need to be considered for further design development to allow for long articulated vehicles delivering heavy and large items during construction, maintenance and demolition.

Junctions are located where there are good visibility distances.

Safety berms are to be provided at edges of roads to contain vehicles where there are either steep/high drops at the edge or assets. These will need to be sized for vehicle usage of the road.

The new main access road off the Sodankyla to Kittilä highway into the plant site is discussed in Section 18.4.2.

Haul Roads

The haul road for the open pit exits Southwards and there is a “U” turn where it must cross over the alignment of the future development of the underground access. Consideration will need to be given to the structural support at this location, whilst also planning the works to minimise impact on underground access at a later stage in the life of mine. The junction with the waste haul road and the ROM haul road has a surface elevation at approximately 225 m, which is close to ground level. The gradient up to the ROM pad is approximately 1:12. There is also a single lane haul road around the ore stockpile. The haul road to the co-disposal facility is also at approximately 225 m surface elevation as it crosses the valley.

An alternative location for the haul road out of the open pit at the Northwest corner would improve the layout and reduce haul distances.

There is also a designated one-way haul road for transporting filtered tailings to the co-disposal facility and then a return one-way road to the filter plant.

Main Plant Site Access Road, Access Tracks

The main site access road joins the property boundary at the South edge close to the ore stockpile at the ROM pad. From here the road descends Northeastwards to the plant site pad area, which is above the South of the cut profile. At the pad elevation there is a bend Northwards, to allow for access onto the pad. From here, it circles Westwards below the North fill profile towards the HME and LME workshops pad. A switchback is required on the descent from this to the filter and paste plant pad.

Access tracks are either two way or one way with passing bays. It is expected that these will be used by 4 x 4 LME site vehicles. Tracked machine use would be occasional.

Perimeter Road Around the Open Pit, Road South of the Open Pit

A design should be undertaken to check that the perimeter track around the open pit, constructed with compacted layers of engineering fill, can retain lateral loading from adjacent peat which is expected to be 1 -2 m.

The in-situ conditions of the foundation layer beneath the embankment will also need to be investigated.

The perimeter access road to the South of the open pit is located within a 20 m wide margin next to the property boundary as shown in Figure 18-7.

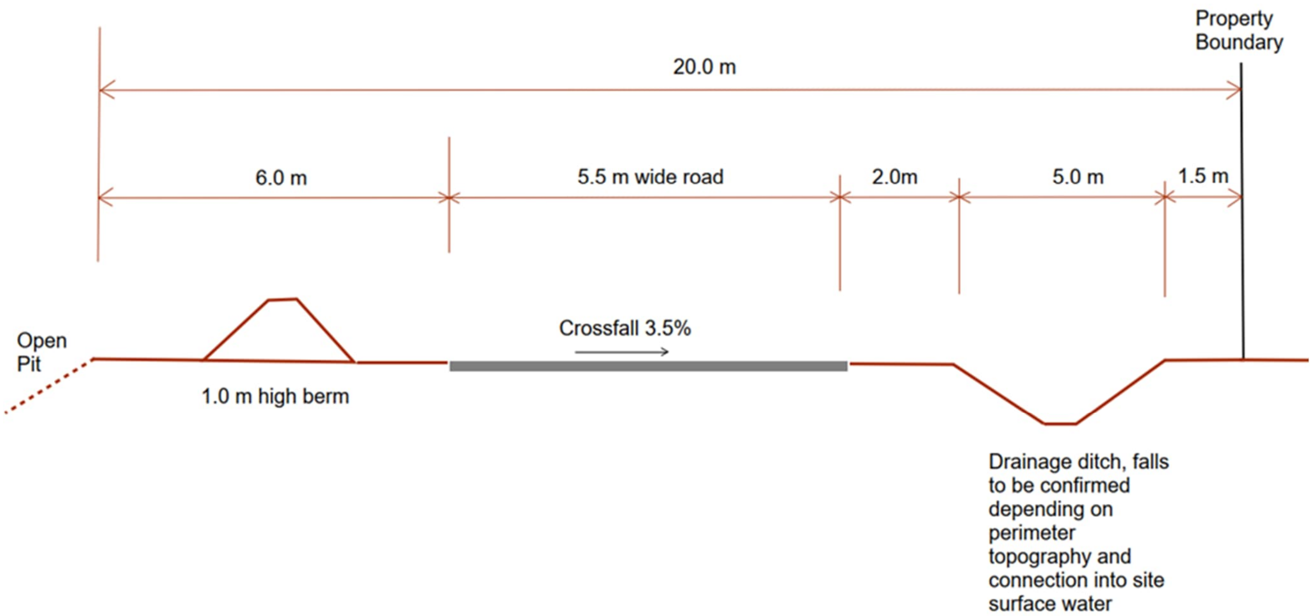


Figure 18-7 – Perimeter Access track South of the Open Pit

This 20 m margin allows for:

- 5.5 m single lane access road for LME only;
- space either side of this for adjusting alignment of road with respect to the design outline of the open pit;
- 1 m high safety berm adjacent to the open pit;
- behind this a mining/infrastructure working tolerance of 1.0 m;
- 2.0 m margin into which the dewatering wells could be installed, protection from driving into the ditch can also be provided within this;
- 1.5 m working margin next to the property boundary; and
- 5 m wide margin for a V ditch, or culvert depending on the topography and design invert levels, either of which will need to connect into the surface water drainage and/or river diversion.

Fencing

Fencing around the mine site area with gated access control at the Administration building is provided for. This is for controlling persons entering and leaving the mining area. It is estimated to be 2.25 m high, and without barbed wire.

18.4 INFRASTRUCTURE TIES-INS

To accommodate the mine development, the following infrastructure ties-ins were identified:

- Connection to the national grid infrastructure;
- River diversion;
- Enhanced road access to site; and

- Water discharge infrastructure.

18.4.1. ELECTRICAL POWER

The primary investigation shows that the 220 kV national grid infrastructure is available approximately 6 km north of the site location. Discussions with Fingrid have also provided some preliminary information regarding the possibilities of using the available infrastructure in the vicinity. As a wind power plant has been recently built further north of the 220 kV transmission line and connected to the grid, from Fingrid's point of view, the available substation seems a preferable Point of Connection (PoC). This has been further discussed by Rupert Resources and Fingrid in the beginning of 2024 and a cost estimation was provided by Fingrid, which is included in the CAPEX cost estimate.

The plan is to use/extend the available substation in the area giving access to the power demand at PoC at a voltage level of 110 kV for supplying to the site. The technical investigation of the power station 220/110 kV is not included in the scope of this study. This is expected to be provided by Fingrid.

A few possible transmission routes for the incoming 110 kV power line to the site have been investigated the placement of mining facilities, accessibility to the site and possible barriers. The result is presented in the offsite infrastructure.

The receiving substation at the site is to be placed south-east of the mining facilities accepting 110kV incoming power transmission line for a total demand of about 25 MW. A primary estimation of the receiving substation, associated equipment and occupied footprint is provided in section Surface infrastructure.

A switchgear room and a power factor correction station are to be placed in the vicinity of the receiving substation supplying the site facilities with power distribution at 20 kV and compensating the reactive power consumption in the site before connection to the grid.

The power distribution at the site supplies the power to every single consumption point through 20 kV cables laid directly in the ground. At the consumption point, locally placed substations and step-down transformers are to be provided.

It is to be noted that the suggested schematic for the power distribution system, at the site level, is without having a full redundancy as agreed by Rupert Resources. However, some initial redundancy is predicted for vital consumption points. The necessity of redundancy can be discussed and reconsidered in the next steps and detailed design.

Furthermore, there is an external power line at medium voltage (20 kV) on the site, which is planned to serve as auxiliary power in the future and as a temporary electrical connection during construction.

Surface Infrastructure

The receiving station is to be placed on the southeast side of the mining area and to be supplied by a single 110 kV incoming transmission line to the site. The receiving station consists of two power transformers at 110/20 kV with a rated capacity of 25 MVA each. Each transformer is assumed to be loaded at 50% of its nominal capacity during the normal operation of the mine. In case of failure for one of the transformers, the other one can be temporarily fully loaded and keep the mining operation undisturbed. The redundancy plan at the receiving station only covers the transformer level and does not extend to the upper side (incoming 110 kV). In case of having full redundancy even for the transmission line, the 110 kV receiving infrastructure needs to be revised. Figure 18-8 shows a general schematic for the receiving substation considering the required equipment and sufficient clearance area. The existing external power line available on the site could be used to supply auxiliary power in the receiving substation as well.

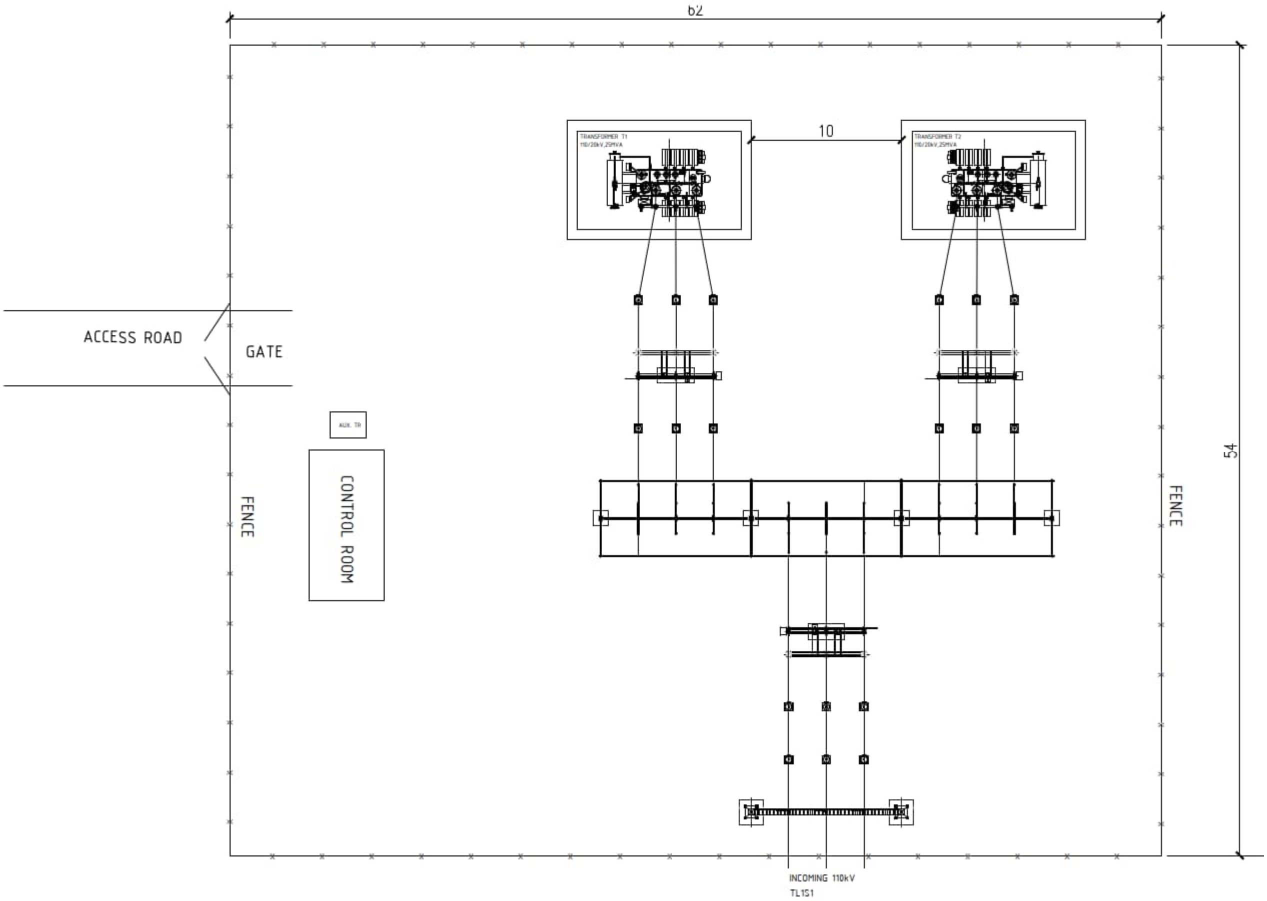


Figure 18-8 – Receiving 110/20 kV substation layout

Switchgear Room

A switchgear room is to be situated near the receiving substation getting supplied by the transformers at 20 kV (so called main switchgear room). Switchgears A and B, each powered individually by their respective transformers. In case of failure in one of the transformers, a bus-coupler couples the busbars A and B. The switchgear room would be a two-stories building; the ground floor is to house incoming and outgoing electrical cables; the second floor includes a switch room, control and battery rooms. Figure 18-9 shows a general layout of the main switchgear room.

A secondary switchgear room is available near the filter plant to supply facilities around, mainly the consumption points which are further away from the main switchgear.

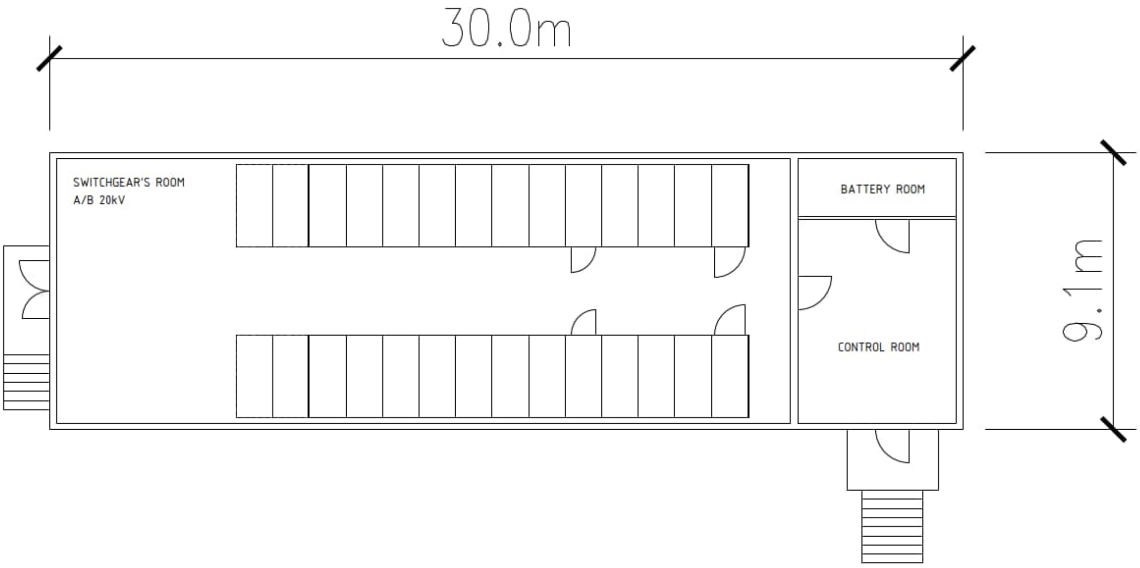


Figure 18-9 – Layout of the Main Switchgear room

Power factor correction station

The primary list of the loads shows that the total reactive power consumed by the loads is approximately 6 MVAr. To enhance the power factor and avoid extra cost/penalties received from the grid owner, the site is to be equipped with reactive compensation, i.e., capacitor bank, damping reactors and filters. Various solutions exist for reactive power compensation. However, to choose the proper solution, further investigation of the types of loads, their potential impact, and harmonic distortion is required.

At this stage, only preliminary calculations have been conducted, and a basic reactive compensation component has been considered. The placement of the power factor correction station will be near the main switchgear room and will be connected to busbar A and B, accordingly. Figure 18-10 shows a fixed reactive power compensation system from Hitachi Energy, SIKAP 18 capacitor bank.

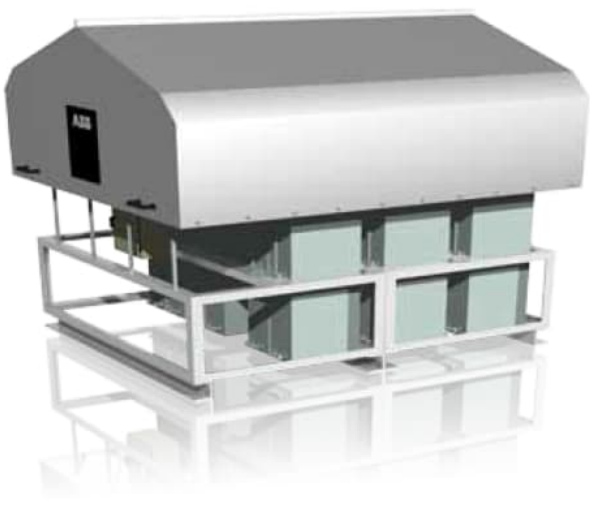


Figure 18-10 – Metal enclosed capacitor bank SIKAP 18-Hitachi energy

Power distribution

The power distribution on the site is considered through 20 kV cables directly laid underground, and supplied from the main and the secondary switchgear rooms. The secondary switchgear room will be positioned closer to the loads situated in the central and western parts of the mining area. Each major consumption point has its own series-connected secondary substation, supplying electrical demand at lower voltage to nearby facilities.

Power substations by the loads

Each consumption point will be equipped with its own 20/0.4 kV series-connected secondary substation, comprising step-down transformer(s) and low voltage switchgear(s). The configuration of the series-connected secondary substations, whether containerized or built structures, will vary based on factors such as size, number of transformers, and site positioning.

Underground Infrastructure

Given the lack of detailed information on the type and power requirements of the underground infrastructure, certain assumptions have been made at this stage. The total power demand at underground facilities is assumed to be approximately 2500 kVA. All facilities and equipment will be powered through a maximum of three locally situated substations, housed within containers. These containers will be positioned near the equipment, with radial connections established between the substations. Should redundancy be necessary, provisions can be made for the containers to be supplied from multiple directions. No information at this stage is available for possible cabling routes to the underground facilities either.

Offsite Infrastructure

The intended overhead line has a voltage level of 110 kV and is supposed to connect to the substation at the mining area as well as an already existing substation. The nearest substation is placed northwest of the mining area and is a connection for a wind farm as well as a 220 kV overhead line.

The start for the intended overhead line is at the previous mentioned substation, then crosses the existing 220 kV overhead line, and is then placed parallel to the existing overhead line for about 3.5 kilometres in an eastern direction. An angle point is added to stand the line parallel to the road Värttiövaarantie in southeast direction.

Two minor roads are crossed by the overhead line along the path until it crosses the road Värttiövaarantie until another large angle is added to continue the line to the mining area. The line passes through a wetland until it arrives at the mining areas intended substation.

Based on the available orthophoto, elevation model as well as map from National land survey of Finland, NLS, most of the poles are assumed to be placed in soil such as moraine. The wetland which is about one kilometre southwest of the mining area is avoided by using slightly taller poles since it is 250 metres wide, compared to the assumed distance of 150 metres between each pole. Most of the poles are placed in forest land.

The intended 110 kV overhead line is illustrated in Figure 18-6 together with the substation area, existing 220 kV overhead line as well as roads in the area. The intended line is approximately 13.6 kilometres.

18.4.2. NEW ROAD ACCESS TO MINE

It is planned to develop a new 7.5 m wide bituminous surface road accessing the mine. Three alignments have been identified, T1_VE1, T1_VE2 and T1_VE3, which take into consideration both construction and operating costs as well as environmental and social considerations. (Ramboll, 2024). T1_VE1 has been used as the base case for the PFS.

Two alternatives have been selected in the current planning for PFS. In both alternatives, the road is planned to have a junction 37.3 km northwest of Sodankylä off the main state-owned road, Kt80. The length for the preferred route options are 3.7 km for T1_VE1, and 6.3 km for T1_VE2 as shown in Figure 18-11 below; (Ramboll, 2024). T1_VE1 was used in the financial model.

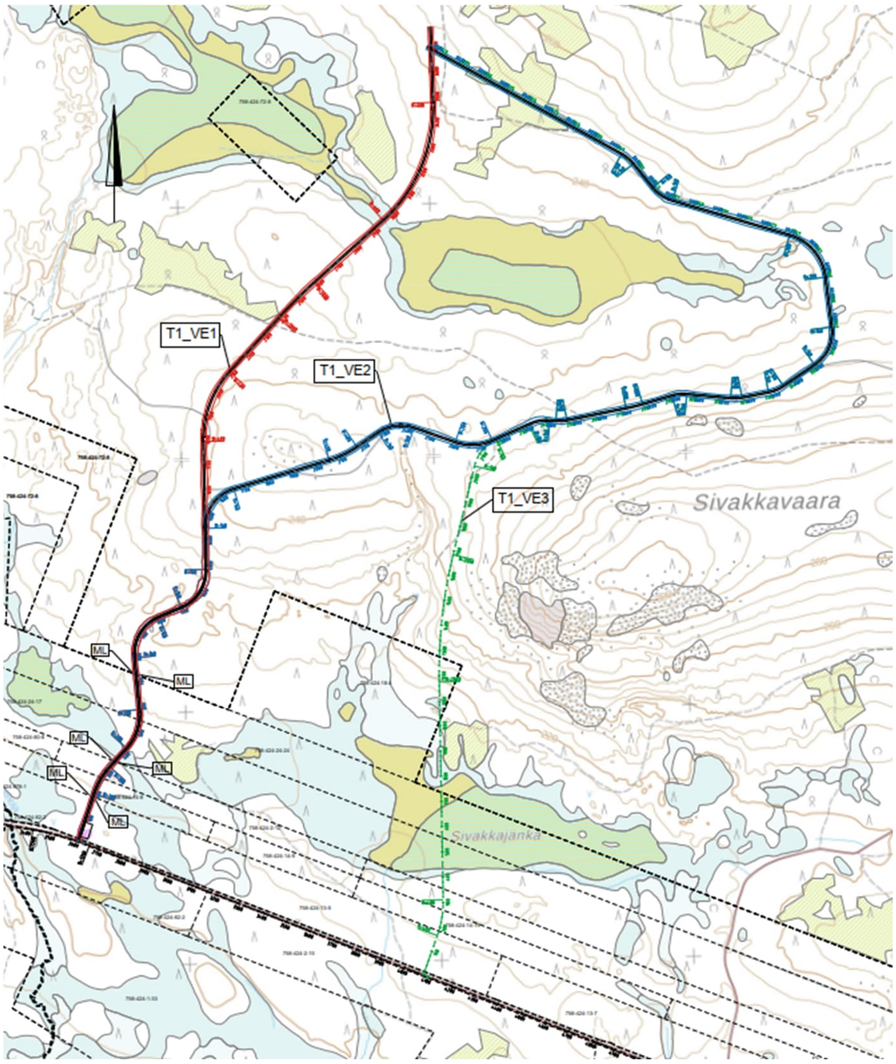


Figure 18-11 – New access road options

The southern intersection for both options is planned to be at a designated point of curvature on the existing main road, to comply with the visibility requirements according to the Decree of the Ministry of Transport and Communications on visibility areas 65/2011. The joining visibility is 200 m and stopping visibility is 120 m towards the main road, without requiring any changes to the current elevation. A bypass area is to be constructed at the intersection with main road Kt 80, and the intersection area will be illuminated in both the main road and private road directions. Visibility clearing will be required in the intersection area to achieve the sighting distances according to Decree of the Ministry of Transport and Communications on visibility areas, 65/2011. (Ramboll, 2024).

Further details are shown in Figure 18-12 below.

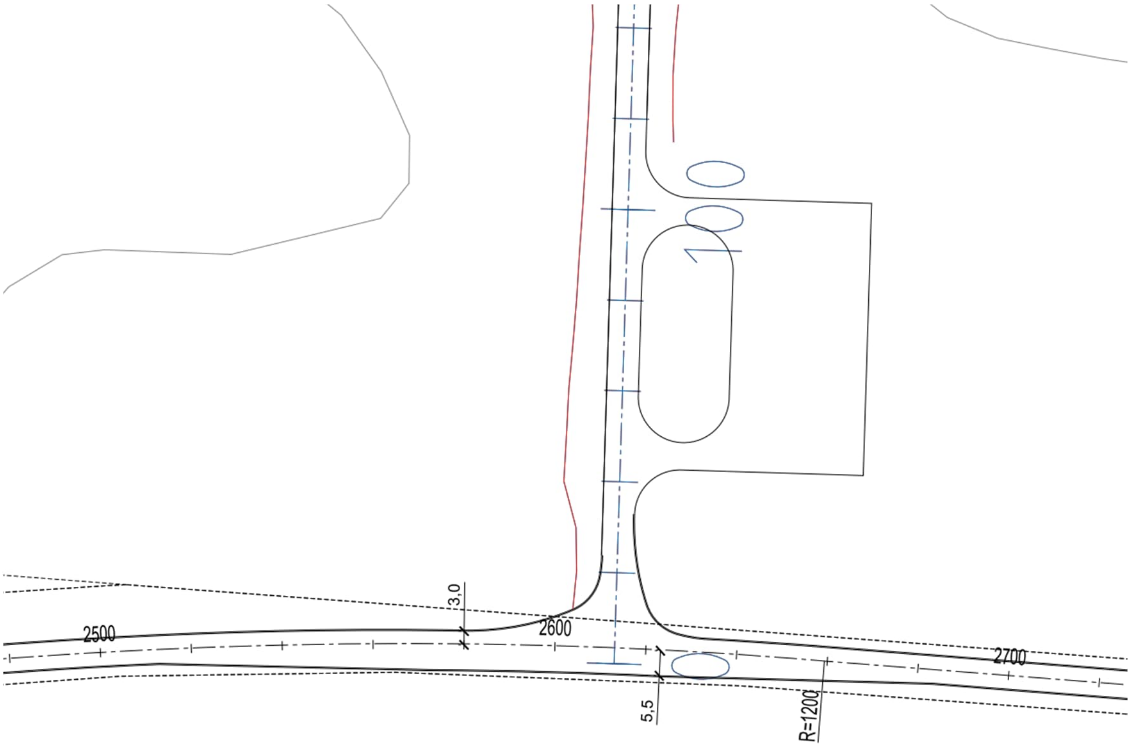


Figure 18-12 – New access road junction

18.4.3. SAITTAJOKI RIVER DIVERSION

The project is in the Saittajoki River valley, where both the tributary and the main river channel are planned to be diverted around the mine site to minimize pollution risks. Three diversion options were considered, with Option 3 selected as the best approach. This option involves diverting the Saittajoki River 2.5 km north into the Heinälamminoja Stream, aiming to prevent river pollution and allow natural river channel development. However, it poses risks such as permitting challenges and environmental impacts on associated habitats and sediment transport. Initial estimates include constructing earth embankments and designing the diverted channels to match existing flows, with minimal intervention proposed to form the new channel. The river diversion design will be further developed with detailed surveys and hydraulic modelling.

More details are provided in Chapter 20.

18.4.4. WATER DISCHARGE INFRASTRUCTURE

The mine water management study has assessed the mine to be water positive. Consequently, the excess water is to be discharged into the environment.

External pit dewatering boreholes

Based on testwork to date, the PFS assumes that the water from external pit dewatering is of suitable water quality to be discharged into the Saittajoki River. The conveyance of the water and point of discharge are to be identified at a later design phase.

Mine water

Any excess treated water (including contact water, and sewagewater from the welfare facilities) is discharged to the environment. Various discharge locations are appraised as part of the Environmental Impact Assessment.

Basis of design

The PFS assumes that the treated water is discharged in the Kitinen River, upstream of the Kelukoski dam.

The pipeline route from the Ikkari site to the river should be optimised, taking into consideration:

- the topography of the area - pumping requirement should be minimised to reduce operating carbon footprint;
- the type of superficial deposit (peat, rocky land, pebbles and clay, etc) – for constructability easiness and long-term stability of the pipe;
- the ecological constraints (Endangered habitat, protected fauna and flora, deforestation and forest law) and nature conservation areas – to minimise environmental impacts;
- The presence of groundwater resource;
- Flood zone;
- Existing mines; and
- Cultural heritage sites

To minimise greenfield land disruption, and to provide easy access for maintenance, the pipeline route should follow existing roads or forest tracks, wherever possible. Consideration of the reindeer herder's community should be considered such as the pipeline construction and operation has minimal impact on the operation of the reindeer herder's cooperative.

The pipeline should be designed to accommodate a peak flow of nominal 900 m^3/h, 24 hours a day, seven days a week. The pipeline should be protected against frost.

The design of the discharge infrastructure should maximise mixing at point of discharge.

Pipe material is to be HDPE.

Illustrative discharge pipeline infrastructure

The optimised route of the pipeline has a length of 37 km. The route layout is providing in Figure 18-13.

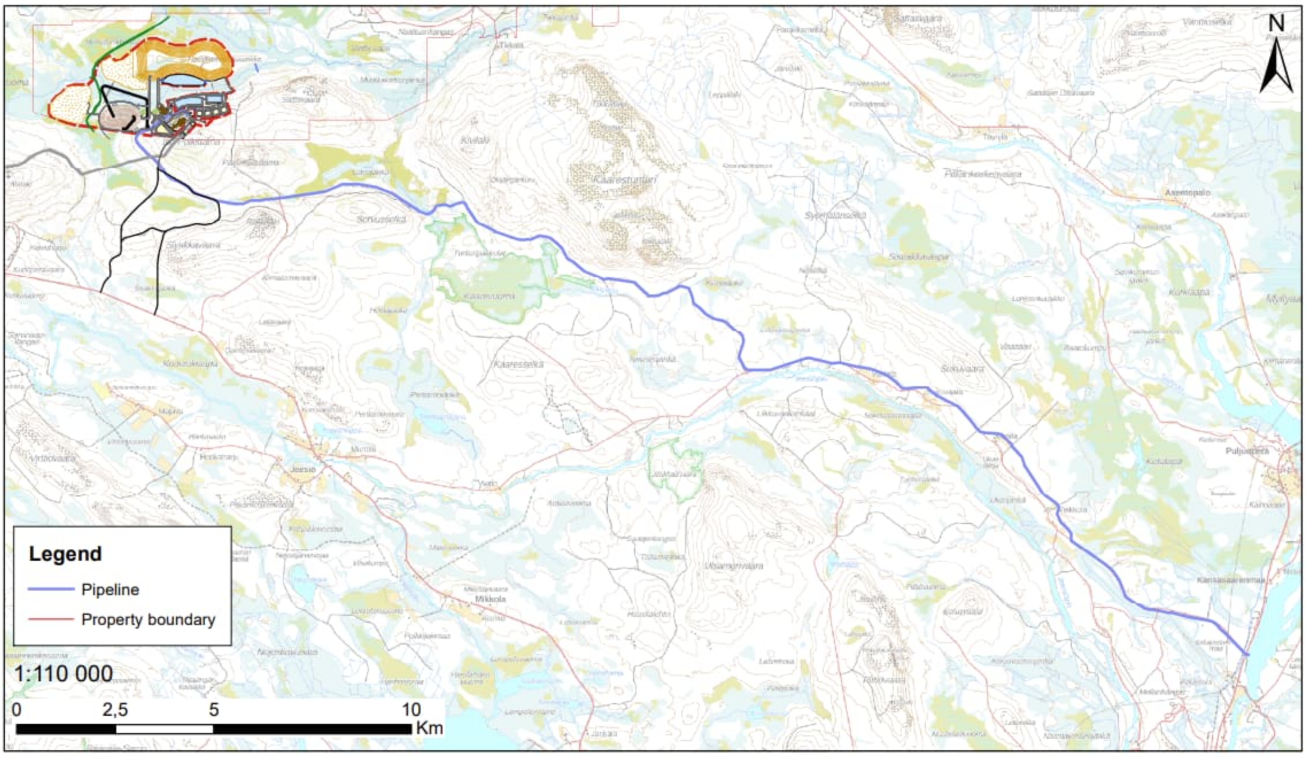


Figure 18-13 – Discharge pipeline layout

A hydraulic model was carried out to define the minimum pipe diameter and pressure rating for the discharge of 900 m^3/h by gravity. Outcomes from the model, alongside the review of the European standard for HDPE pipe diameter and wall thickness chart, identified that a 630 mm outer diameter with a pressure rating of 10 bar is required (SDR 17 630 OD HDPE (PE 100)).

To protect the pipe from frost, the pipe is to be buried underground with a minimum of 1.3 m of clear cover and the use of a frost insulation sheet (Figure 18-14). A 1.5 m cover is proposed at this stage of the design to mitigate the risk of pipe buckling. More detailed study will be caried out at detailed design to assess whether the pipe cover can be decreased to 1.3 m.

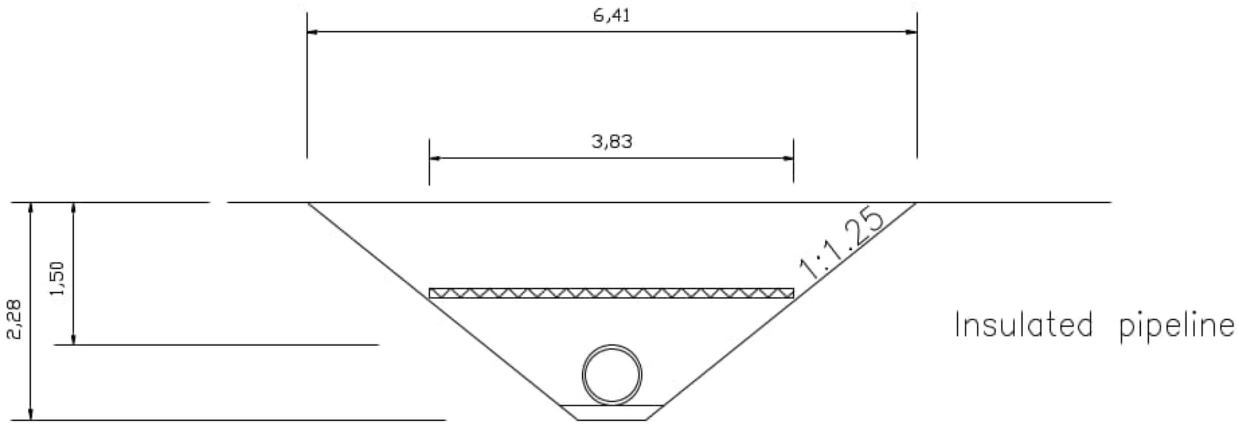


Figure 18-14 – Illustrative pipe cross-section

It has to be noted that due to the terrain along the pipe alignment, certain sections of the pipeline will be pressurized, with negative pressure presumed at local high points and bends. To avoid build-up of air at these critical locations, relief valves are required to remove vacuum and prevent backflows.

On gravity/non-pressurised sections, manholes are to be located on bends greater than 30 degrees, and points where gradient changes. A sufficient number of inspection and maintenance chamber for access and isolation should be placed between manholes to facilitate maintenance and repair.

The discharge pipe will be continuously monitored for flow and pressure at strategic points as well as for a selection of water quality parameters. These parameters are to be defined through the permitting process.

Illustrative outfall infrastructure

The outfall structure (Figure 18-15) consists of a break pressure tank, into which the water is discharged, and an outflow pipe from the tank to the Kitinen River. The break pressure tank is located on the side of the river embankment. The outflow pipe crosses the embankment downward following the embankment slope to then become parallel to the riverbed. Diffusers are attached alongside the horizontal outfall pipe to distribute the flow of water over a wider area, reducing the velocity and energy of the discharge. Scour assessment should be performed in the later stages of design.

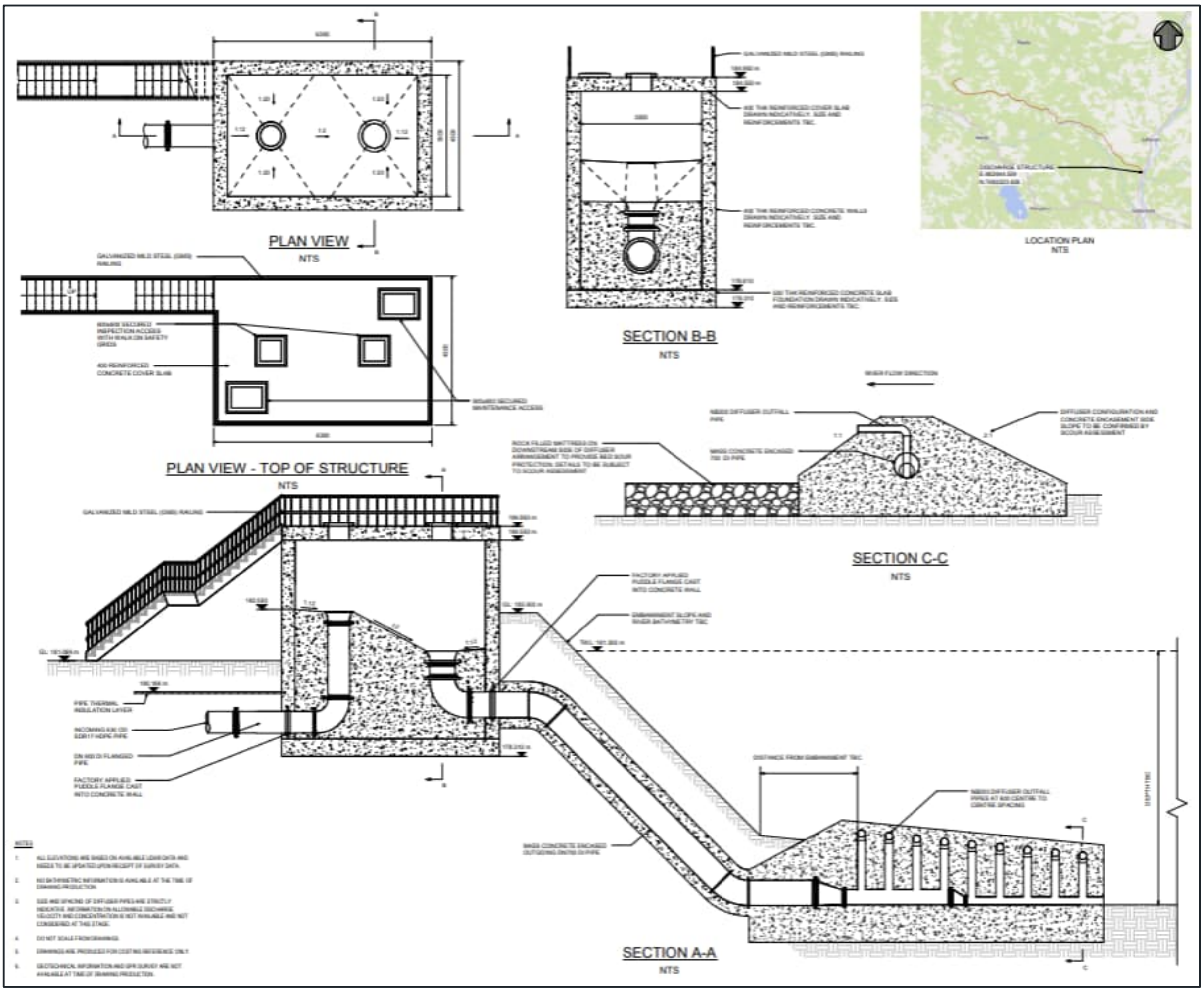


Figure 18-15 – Illustrative outfall infrastructure

18.5 SCHEDULE OF BUILDINGS

Schedule of buildings in the plant site area is provided in Table 18-3 below.

Table 18-3 – Schedule of Buildings

Ref.	Description	Requirements
B1	Mineral processing plant	Architect design not needed at PFS stage Assumed fire egress not needed in PFS stage Requires external heating system
B2	Filtration Plant	Architect design not needed at PFS stage Assumed Fire egress not needed in PFS stage Requires external heating system
B3	Main Administration and Facilities building	Private Offices Open Offices Conference Room Canteen Kitchen Coffee Room Laboratory Control Room Bomb Shelter Changing Rooms 2 No. Saunas Building Technology HVAC Rooms General moving space
B4	Heating Building	4.5 MW System Building
B5	HME and LME Maintenance Building	140m x 40m for haul trucks (HME) and 40 x 30 for light vehicles (LME)
B7	Water Treatment Plant	Potable water treatment plant – 23.35m x 13.9m Water treatment plant 1 – 92.6m x 44m Water treatment plant 2 – 146.26m x74m A lime storage area – 25m x 15m Underground sewage treatment plant - 14.7m x 8.5m
B8	Raw water intake pumping facility	Below ground
B9	20kV building and 110kV outdoor substation	2 transformers Fencing around 110kV outdoor substation

Ref.	Description	Requirements
B10	Fire Protection Water Pump House	Emergency pumping facility and distribution network, with abstraction from treated water pond with a reserved capacity to be determined 15 m x 15 m building
B11	Refuse re-cycling/disposal facility	Contained in Administration facility
B12	Explosives Storage Facility	Modular units
B13	Fuel Station	2 pumps. One with industrial (red) diesel and another for normal taxed diesel.
B14	Mine Rescue Building	Clinic, paramedic, 1 room needed for rescue equipment and special clothing.

19 MARKET STUDIES AND CONTRACTS

The operational income for Rupert Resources from the development of Ikkari enterprise will be wholly from the sale of gold bullion. The market for gold is mature with many reputable refiners and brokers located globally. The advantage of gold, like other precious metals, is that virtually all production can be sold on the open market. As such, market studies and entry strategies are not required.

Metallurgical process studies confirm that the Ikkari enterprise will produce doré bars of a specification comparable with existing gold mines.

Demand for Gold is strong with prices in the range of over $2900 per ounce at the time of publishing. In an approach consistent with the 2022 PEA, the gold price applied the base case financial analysis in this study is $2150 based on the long-term consensus forecasts from over 20 investment banks. Mineral Reserves were calculated using a gold price of $1700 USD per ounce.

The Ikkari site will produce gold doré bars which will generally contain 90% gold content. These gold doré bars will be shipped to a gold refinery. The refinery will transform the gold doré bars into Gold Bullion with 999.9 fine gold investment grade. The refinery will sell the Gold Bullion to the open market using the daily Gold price as set by the New York Stock Exchange. The refinery will establish a commercial contract with Rupert Resources (Finland) to enable repatriation of funds at point of sale.

Rupert Resources has not entered into any forward contracts or hedging agreements at the time of publication of this study.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL SUMMARY

20.1.1. CURRENT ENVIRONMENTAL ACTION LIMITS

Framework legislation includes several laws, acts, decrees, and permits which all effect Rupert Resource's Ikkari project.

Crucial laws and regulations include amongst other:

- Mining legislation including the Mining Act (621/2011, amended 2023);
- Environmental Protection Act (527/2014) including the Environmental Protection Decree (713/2014);
- the Water Act (587/2011);
- Environmental Impact Assessment Procedure Act (252/2017);
- Dam Safety Act (494/2009);
- Chemical legislation including Chemical Act (599/2013) and Act on the Safe Handling and Storage of Dangerous Chemicals and Explosives (390/2005);
- Waste Act (646/2011) and Waste Decree (179/2012), Government Decree on Extractive Waste (190/2013 as amended);
- Nature Conservation Act (9/2023);
- Fire safety legislation, including Rescue Act (379/2011);
- Construction Act (751/2023) and Land use Act (132/1999, amended 2023);
- Forest Act (1093/1996);
- Reindeer Husbandry Act (848/1990);
- Ancient Remains Act (295/1963);
- Radiation Act (859/2018)
- Contaminated soils decree (214/2007)
- Air Pollution Control Decree (79/2017);
- Decree on the Safe Production and Handling and Storage of Explosives (1101/2015);
- Council Directive 92/43/EEC of 21 May 1992 on the conservation of natural habitats and of wild fauna and flora (Habitats directive); and
- Directive 2009/147/EC of the European Parliament and of the Council of 30 November 2009 on the conservation of wild birds (Birds directive).

Permits guiding operations include amongst other:

- Environmental and water permit (with preceding EIA procedure);

- Derogation permit from nature protection provisions;
- Mining permit;
- Mining safety permit;
- Construction permit;
- Permit for handling and storage of dangerous chemicals;
- Permit for storage of explosives; and
- Exploration permit.

At the current time the Ikkari deposit is located within the Heinälamminvuoma exploration permit (ML 2011:0033), there are no other valid permits covering the deposit or the immediate surrounding area. According to Mining Act (621/2011), section 11, the exploration permit's holder shall limit exploration and other use of the exploration area to measures necessary for the purposes of research activity. The measures shall be planned so as not to cause an infringement of public or private interests that is avoidable by reasonable means. In the exploration permit ML 2011:0033 it is stated that following exploration activities and methods are allowed in the area: geophysical and geochemical or comparable research methods, mechanized soil and bedrock sampling (till, trench, and point samples, drilling) and exploration test pits.

According to Mining Act (621/2011), section 51, regulations, set by TUKES (Finnish Safety and Chemicals Agency), need to be included in an exploration permit. According to the exploration permit regulations (ML 2011:0033), different measures have been set to protect groundwater zones (for location see figure Figure 20-1) and springs (Section 10 of the Forest Act 1093/1996, Section 11 of Chapter 2 of the Water Act 587/2011). It is noteworthy that according to the Environmental protection Act (527/2014), section 17, groundwater pollution is prohibited, which imposes additional controls on exploration in the groundwater zones (exploration permit ML 2011:0033).

Furthermore, the exploration permit regulations (ML 2011:0033), are taking in account that within the area occur endangered, protected species and cultural heritage sites, such as serpentine rocks, boulders and gravel, predatory birds, otter and ancient remains. These nature values impose additional controls for the exploration operations in specifically given regulations in the exploration permit ML 2011:0033. (Section 65, 70, 74, 78 of the Nature Conservation Act 9/2023, Annexes II and IVb of the Habitats Directive 92/43/EEC). The ancient remains are located on the map in Figure 20-2. It is significant that the exploration permit area exceeds the planned mine site area (Figure 20-1).

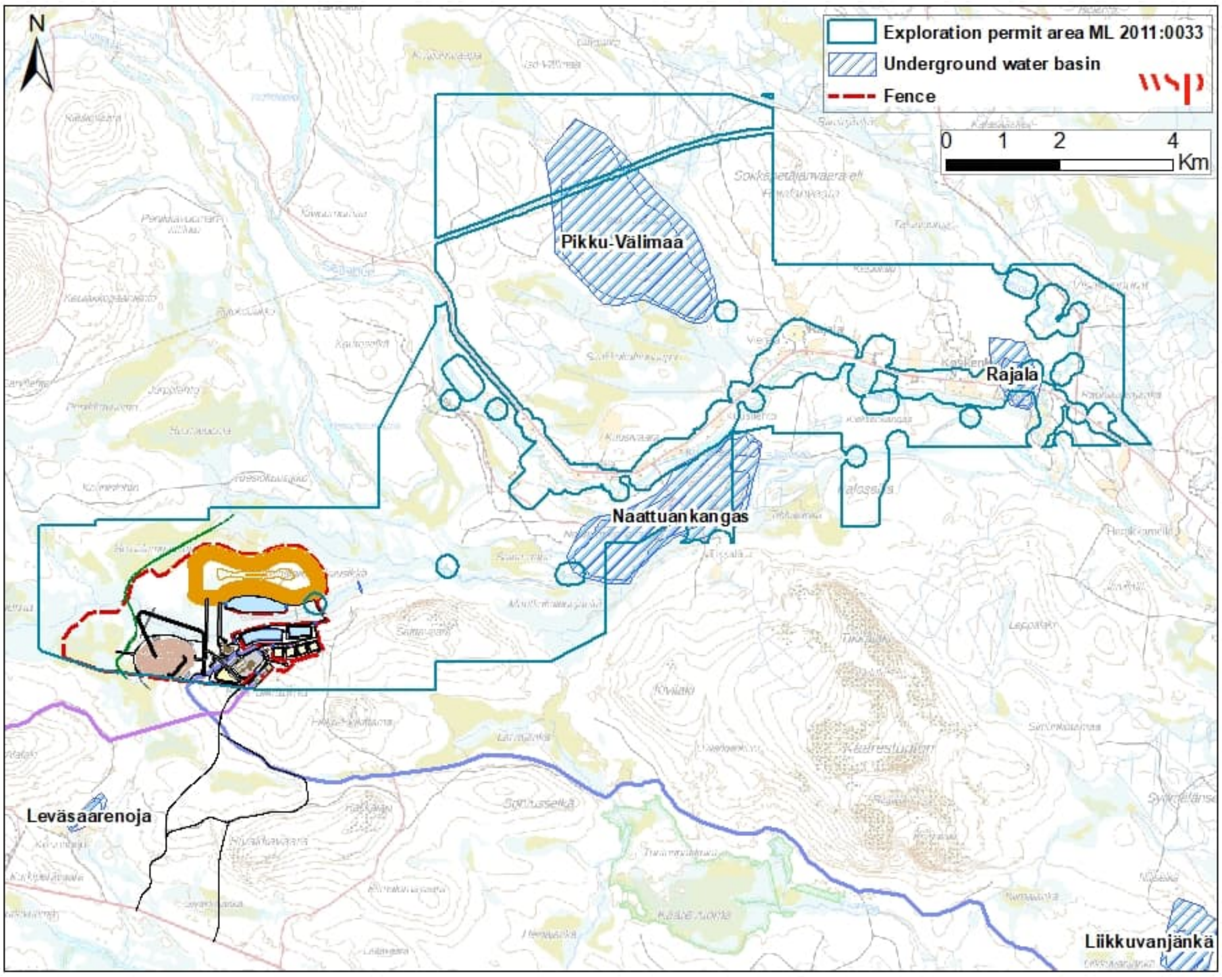


Figure 20-1 – Exploration permit area ML 2011:0033 and the planned mining facility

20.1.2. ANCIENT REMAINS AND REINDEER HUSBANDRY

There are a few known remains in the project area, to the NW of the planned open pit. They are related to 20th century logging and log floating industry. An ancient remain is protected according to the Ancient Remains Act (295/1963).

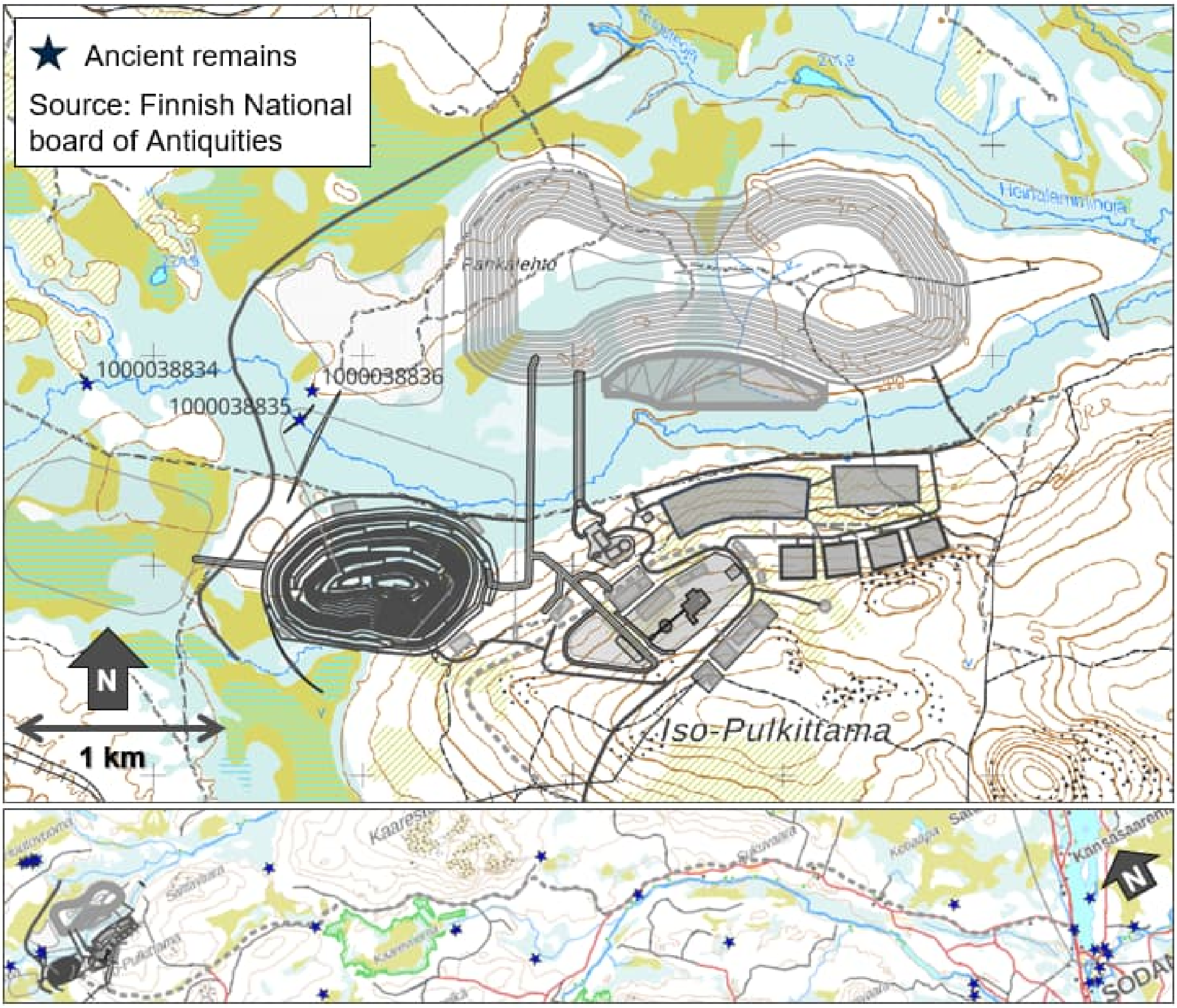


Figure 20-2 – Ancient remains in the area

The right for reindeer husbandry is provided for by a special law, the Reindeer Husbandry Act (848/1990). It regulates the right of free reindeer grazing in the reindeer husbandry area in Northern Finland. The Project area is in the Sattasniemi Reindeer Herding Cooperative which is in the area specially intended for reindeer husbandry. The section 50 in the Mining Act (621/2011) stipulates that exploration-, mining- and gold panning permits cannot be granted if the permitted operation causes significant harm on the reindeer husbandry. However, according to the same section, 50, in the Mining Act, the permit can be granted if the hindrance can be removed with a permit order. Rupert has continuous discussions ongoing with the local Reindeer Herding Cooperatives where possible project impacts and mitigation methods are discussed. Mitigations are needed to assure that the Project is not causing significant negative impacts on the reindeer husbandry. Recommended and common practice is a mutual agreement with the Sattasniemi cooperative defining the mitigation measures and compensation scheme. New agreements will have to be done for the operation phase, as the impacts will differ from those of the exploration phase.

20.1.3. FLORA AND FAUNA

The project area is located within the Northern Boreal Perä-Pohjola (4b) forest vegetation zone and the Central and Northern Perä-Pohjola aapa mire (4c) mire vegetation zone. The project area is situated between the upland forests and hills, in a region where the Saittajoki river collects most of its water on its way east towards Sattanen. Noteworthy open mire areas are Ikkarinvuoma and Heinälamminvuoma. The forests in the area are mostly typical conifer-dominated commercial forests. The most natural forests in the area are mainly found in the islands of mires in the middle of the Ikkarinvuoma and Heinälamminvuoma open mires, in a few places, from the edges of the spruce mires along the streams and from the rocky northern slope of Saittavaara. One of the characteristics of the area is the abundance of springs. (Envineer 2023c, pp. 187-188)

The vegetation and habitat types of the area's mires are diverse. Heinälamminvuoma is mostly of the Peräpohjola aapa mire type. The Pikkulehdonoja stream flows through the area from the northwest, along which there are natural grass and herb spruce mires and rich fens. Almost the entire shoreline has occurrences of *Ranunculus lapponicus* (a protected species listed in Annexes II and IV of the Habitats Directive). Ikkarinvuoma is in its western part a vast and mostly wet, treeless flark fen. The most notable site is a spring pond located at the northern edge of the western part of Ikkarinvuoma and its surrounding area, where, among other things, *Hamatocaulis vernicosus* (listed in Annex II of the EU Habitats Directive) can be found. In terms of moss species, the area has abundant occurrences of *Hamatocaulis vernicosus* and *Scorpidium scorpioides*. Particularly notable occurrences include the *Hamatocaulis vernicosus* in the northern part of Ikkarinvuoma, and the *Scorpidium scorpioides* on the western side of Heinälammi. (Envineer 2023c, pp. 190-191)

The typical features of the birdlife in Sodankylä are the northern character of the species and the abundance of mire birds. The most characteristic species of the area's mires and their forest islands include the bean goose (*Anser fabalis*), willow ptarmigan (*Lagopus lagopus*), and capercaillie (*Tetrao urogallus*), diurnal raptors, northern hawk-owl (*Surnia ulula*), crane (*Grus grus*), many waders (such as the wood sandpiper, *Tringa glareola*), common snipe (*Gallinago gallinago*), jack snipe (*Lymnocryptes minimus*), red-necked phalarope (*Phalaropus lobatus*), and golden plover (*Pluvialis apricaria*)), three-toed woodpecker (*Picoides tridactylus*), and certain passerines (such as the yellow wagtail, *Motacilla flava*), Siberian tit (*Poecile cinctus*), Siberian jay (*Perisoreus infaustus*), and the little bunting (*Emberiza pusilla*) and rustic bunting, *Emberiza rustica*. The birdlife of the project area can be described as typical for the Central Lapland region. (Envineer 2023c, pp. 195-196)

The project area hosts a regionally typical mammal fauna. Based on snow track surveys and data from the Finnish Biodiversity Information Facility (laji.fi), the area is inhabited by moose (*Alces alces*), mountain hares (*Lepus timidus*), foxes (*Vulpes vulpes*), stoats (*Mustela erminea*), and squirrels (*Sciurus vulgaris*), as well as other smaller mammals. Due to their distribution, all large predators found in Finland may occur in the vicinity of the project area. Species listed in Annex IV(a) of the Habitats Directive are animal species considered important by the community and require strict protection. The destruction and deterioration of the breeding and resting places of these species are prohibited under the Nature Conservation Act (Section 73, 9/2023). The area has known observations of the otter amongst other, which is strictly protected by these legislations. (Envineer 2023c, pp. 197-199). In an individual case, a Centre for Economic Development, Transport and the Environment may authorise a derogation from a prohibition of a conserved habitat type, or strictly protected habitat type, if the conservation objectives of the habitat type are not considerably jeopardised or if the conservation of the habitat type prevents the implementation of a project or plan of very high public interest and there is no technically or economically feasible alternative to this project or plan (Nature Conservation Act, Section 66). The authority might also grant a permit to deviate from the provisions on the protection of species, Section 83 of the Nature Conservation Act, if this does not have adverse impacts on the preservation or attainment of a favourable conservation status of the species.

Lately, the Project area and possible discharge routes have been mapped for vegetation and nature types in summer seasons 2019-2024 by Ramboll Oy, Envineer Oy and Luontoselvitys Kangas (see section 20.2.). A variety of animal and bird surveys have been conducted for the Project area with emphasis on European Union’s Directive species. Envineer Oy has during 2021, 2022 and 2023 mapped mammals, birds (including migratory birds, night active predators and day active predators), otters, bats, dragonflies, diving beetles and moor frogs (see section 20.2). Aquatic surveys conducted so far are fishery study by Envineer Oy, benthic invertebrates by Latvasilmu osk, diatoms by KVVY Tutkimus Oy and river pearl mussel studies by Eurofins Ahma Oy and Alleco Oy, Eurofins Oy also conducted a water vegetation survey in the Project area (see section 20.2). The Saittajoki River has a trout population which may impact on the project plans in the choosing of certain routing or riverbed structures, and bridges.

The EIA report, which Rupert Resources Ltd. is currently working towards submission of the report for the Ikkari project during H2 2025, will examine the ecosystem of the area as well as endangered and protected species and cultural heritage sites in detail, as well as impacts of the project on the reindeer economy and livelihood.

20.2 EXPECTED MATERIAL ENVIRONMENTAL ISSUES AND ENVIRONMENTAL STUDIES DONE

There are no designated protected areas in immediate vicinity of the Ikkari deposit. Kaaresvuoma nature conservation area (ESA302828, SSO120578) is the closest nature conservation area to Ikkari and is situated 8 km east of the Ikkari deposit. Tollovuoma-Silmäsvuoma-Nunarvuoma (SAC/SPA, FI1300608, 9 673 hectares [ha]) is both a Natura 2000 area and a nature conservation area, located more than 10 km west of the Ikkari deposit. Joukaisvaara nature conservation area (ESA302827) is located 15 km southeast of the deposit. The Ikkari project area contains production forests, mires, swamps and springs, small streams, and headwaters of small rivers. Rupert Resources is currently working towards submission of the EIA report for the Ikkari project during H2 2025. The following nature and environmental studies have been conducted:

Nature Surveys

- Alleco. 2024. *Jokihelmisimpukkakartoitus suunnitellulla Ikkarin kaivosalueella* 2024. River pearl mussel survey in the Ikkari mining area 2024. (October 23, 2024);
- Envineer. 2024. Draft. *Ikkarin luontoselvitykset* 2021-2023, Ikkari nature surveys 2021-2023 Includes: mammals, birds, otters, bats, dragonfly, diver beetle, moor frog, mosses, vegetation and nature types, springs, and the following water surveys: fish, diatoms, benthic invertebrates and water vegetation. Both for the mine area and for the discharge pipeline alternatives anno 2023 (March 27, 2024);
- *To be reported in the report above*: Nature survey results for the discharge pipeline alternatives anno 2024;

- Eurofins Ahma. 2018. *Pahtavaaran malminetsintäalueiden esiselvitys*. Desktop review of nature values and habitats of exploration area (February 20, 2018);
- Eurofins Ahma. 2019a. *Pahtavaaran malminetsintäalueen linnustoselvitys* 2019. *Heinälamminvuoman osa-alue* 1. Bird survey 2019 of the Pahtavaara mineral exploration area, Heinälamminvuoma part 1. Eurofins Ahma Oy, September 18, 2019 (Finnish) and December 10, 2019 (English). A transect line survey for breeding birds in the Ikkari area;
- Eurofins Ahma. 2019b. *Pahtavaaran malminetsintäalueiden esiselvitys* 2019. Desktop review 2019 of nature values and habitats of exploration area (September 30, 2019);
- Eurofins Ahma. 2019c. Moor frog survey 2019 of Pahtavaara mineral exploration area. Eurofins Ahma, December 10, 2019;
- Eurofins Ahma. 2021a. *Saitta-Sattanen-Jeesiöjoki Raakkuselvitys* 2021. A desktop study of freshwater pearl mussels (February 12, 2021);
- Eurofins Ahma. 2022. *Saittajoki-Sattanen-Jeesiöjoki päivitetty Raakkuselvitys* 2021. Saittajoki-Sattanen-Jeesiöjoki updated river pearl mussel survey 2021 (March 28, 2022);
- Eurofins Ahma. 2023a. *Leväsaarenojan raakkukartoitukset* 2022. Leväsaarenoja river pearl mussel survey 2022 (May 12, 2023);
- KVVY Tutkimus. 2023. *Piilevätutkimus vuonna* 2022. Diatom survey in 2022 (June 16, 2023)
- Latvasilmu Osk. 2022. *Sodankylän Ikkarin alueen virtavesien pohjaeläimistöselvitys vuonna* 2022. Benthic invertebrate study in rivers in the Sodankylä Ikkari area. (November 28, 2022).
- Luontoselvitys Kangas. 2023. *Ikkarin sammalkartoitus vuonna* 2022. Moss survey in Ikkari 2022. (January 19, 2023); and
- Ramboll. 2020. *Kasvillisuusselvitys. Saitta-aavan, Muotkakaltiojängän, Heinälamminvuoman ja Ikkarinvuoman alueen luontoselvitys*. Vegetation Survey of the area of Saitta-aapa, Muotkakaltionjänkä, Heinälamminvuoma and Ikkarinvuoma (January 31, 2020).

Water Studies

- Envineer. 2021. *Kemijoen Kitisen vesistöreitin nykytilan selvitys Pahtavaaran ja Ikkarin hankkeiden vaikutusalueilla*. Status of the water systems within the impact area of Pahtavaara and Ikkari (November 12, 2021);
- Envineer. 2022a. Preliminary review of route alternatives for Ikkari discharge water (May 10, 2022);
- Eurofins Ahma, 2021b. *Ikkarin malminetsintäalueen vesistön perustilaselvitys*. Baseline survey of the waterways in the Ikkari exploration area. This report describes water quality results taken in 2017-2021. (July 15, 2022);
- Eurofins Ahma. 2023b. *Ikkarin pohjavesitarkkailu* 2022. Ikkari groundwater survey 2022. (June 13, 2023);
- Eurofins Ahma. 2024a. *Ikkarin vesistöjen perustilaselvitys* 2022-2023. Baseline survey of the waterways in the Ikkari area. This report describes water quality results taken in 2022-2023. (March 2, 2024);

- Eurofins Ahma. 2024b. *Ikkarin pohjavesitarkkailu* 2023. Ikkari groundwater survey 2023. (February 9, 2024);
- Eurofins Ahma. 2024c. *Ikkarin sedimenttitarkkailu vuonna* 2023. Ikkari sediment investigations 2023. (March 4, 2024); and
- *To be reported*: Ikkari surface water and groundwater 2024 monitoring results.

Overburden Surveys

- AFRY. 2024a. Ikkari overburden investigations 2023-2024, reported in Ikkari investigation report (October 10, 2024);
- AFRY. 2024b. *Ikkari, maaperän geokemiallinen selvitys* 2023. Ikkari, geochemical investigation of the overburden 2023. (October 24, 2024);
- Geolite. 2022. Ikkari peat layer and peat quality studies 2022, reported in: Peat mapping survey report, SIA Geolite, 2023;
- Geovisor. 2024. Electrical Resistivity Tomography (ERT) at Ikkari, Sodankylä in Summer/Autumn 2023 and Summer 2024;
- GTK. 2023. Investigations of the surficial geology at the Ikkari planned mining area. GTK, November 11, 2023; and
- GTK. 2024a. *Työpöytätarkastelu Ikkarin suunnitellun kaivosalueen ympäristön käyttökelpoisista moreenialueista*. Desktop survey on possible moraine quarry areas in the Ikkari planned mine area. (May 15, 2024).

Hydrogeological Studies

- Piteau. 2024. Prefeasibility study 3D numerical groundwater model development and dewatering evaluation (Piteau Associates, November 2024);
- SRK. 2021. Phase 1 Review of data to support the hydrogeological study of the Ikkari gold and satellite deposits, Northern Finland (SRK Consulting, December 2021);
- SRK. 2022. Phase 2 Hydrogeological field study report for the Ikkari Au and satellite deposits, Northern Finland (SRK Consulting, June 2022);
- SRK. 2023a. Phase 3 Hydrogeological field study report for the Ikkari Au and satellite deposits, northern Finland (SRK Consulting, May 2023); and
- SRK. 2023c. Phase 5 Hydrogeological study of the Ikkari gold and satellite deposits, Northern Finland (SRK Consulting October 2023).

Mine Waste

- Finnish Institute of Occupational Health. 2024. *Pöly- ja kuituselvitys sivukiven koemurskauksessa 18.9.2024*. Dust- and fiber study during crushing of waste rock September 18th 2024. Statement by Finnish Institute of Occupational Health, November 7, 2024;
- GTK. 2024b. *Sivukivien murskaus pölytutkimuksia varten*. Waste rock crushing for dust investigation purpose. (November 19, 2024);
- Mine Environment Management. 2022. Geochemistry data review, waste characterisation and gap analysis. (November 2, 2022);

- Mine Environment Management. 2023. Detailed geochemistry and waste characterisation report. (October 2, 2023);
- Mine Environment Management. 2024a. Carbonation rate testing results, memorandum. Mine (March 20, 2024);
- Mine Environment Management. 2024b. Ikkari geochemistry and waste characterisation report part 2. (August 2024);
- Mine Environment Management. 2024c. Ikkari mine waste facility assessment: semi-quantitative assessment of water quality impacts. (November 15, 2024);
- Mine Environment Management. 2024c. Whole ore leach tailings grind size comparison (DRAFT). (November 21, 2024);
- WSP. 2024. Dust monitoring of ore test crushing (DRAFT). (September 2024); and
- *To be reported*: Source term modelling. Mine Environment Management Ltd.
- *To be reported*: Mine Environment Management. 2025. Detailed geochemistry and waste characterisation report update.

Social, Cultural and Other Surveys

- Envineer. 2022b. Climate change model for Ikkari Gold Mine. Envineer, May 31, 2022;
- Envineer. 2023a. Kairakoneen melumittaus. Noise measurements of a drill rig. (July 31, 2023);
- Envineer. 2023b. Ympäristömelumittaukset. Noise measurements. Envineer, September 5, 2023;
- Eurofins Ahma. 2024d. *Ikkarin pölylaskeuman tarkkailu 2022-2023*. Dust monitoring report for Ikkari, 2022-2023, Eurofins Ahma 2024;
- Kalliotekniikka Consulting Engineer. 2023. *Tärinämittausraportti, Ikkari, Sodankylä*. Vibration measurement report, Ikkari Sodankylä. (November 14, 2023);
- Mikroliitti. 2022. Archaeological survey for Ikkari and Pahtavaara area 2022 (Mikroliitti);
- Mikroliitti. 2023. Archaeological survey for the Ikkari discharge pipeline alternatives 2023 (Mikroliitti);
- Mikroliitti. 2024. Archaeological survey for the Ikkari discharge pipeline alternatives 2024 (Mikroliitti);
- *To be reported*: A reindeer herding baseline study as part of the ongoing EIA work (Alfred Colpaert); and
- *To be reported*: As part of the ongoing EIA work, Ramboll Oy has initiated social studies, questionnaires and interviews with locals.

20.3 WASTE AND TAILINGS DISPOSAL

20.3.1. PROCESS PLANT TAILINGS AND MINING WASTE

The co-disposal facility will need to store two streams of waste which are:

1) Whole ore leach tailings from a filter press; and
2) Waste rock from the open pit and underground mine.

The mining schedule was used to determine both the waste streams and the total tonnage of waste to be transferred to the co-disposal facility is 133.2 Mt, of which 17.0 Mt is overburden. An estimated 1.6 Mt is to be returned to backfilling underground, which equates to a waste rock estimate of 114.6 Mt. For an assumed density of 2.19 t/m^3 of placed and compacted rockfill, the void requirement is 52.3 Mm^3.

A total of 52.0 Mt of ore is to be processed, of which 7.6 Mt is to be used as paste backfill, which results in 44.4 Mt of tailings going to the co-disposal facility. For an assumed density of 1.7 t/m^3 of placed and compacted filtered tailings, this amounts to a void capacity of 26.1 Mm^3. The combined total of both waste and filtered tailings is 79.2 Mm^3.

The gold production life for the mine is 20 years.

The PFS storage volume requirements are summarised in the Table 20-1 below, estimates in a yearly basis are provided in Table 20-3.

Table 20-1 – Estimate of Tailings and Waste Rock Volumes of Mining Waste

Waste Stream	Volume for Life of Mine (Mm^3)
Filtered WOL tailings	26.12
Waste rock *(excluding overburden, including temporary storage for backfill)*	53.06
Total Co-disposal	79.18

The design capacity of the co-disposal facility is set at 91.5 Mm^3. This is based on a previous iteration of the mine schedule. This accommodates a 12.3 Mm^3 (13.5%) reserve capacity at PFS level. Further detailed estimates of tailings and waste rock are provided in Table 20-3.

20.3.2. COMBINING MINE WASTE

Co-Disposal

Mine waste streams are typically separated according to their particle size due to their origin in the mining process and are conventionally disposed in separate locations within a waste rock dump or tailings storage facility. Co-disposal combines waste streams in a variety of ways and allows for disposal at the same location.

This includes a range of blended forms which are often categorised as follows;

1) Waste rock inclusion within a tailings disposal facility;
2) Tailings disposal facility inclusion within a waste rock dump;
3) Tailings storage in cells constructed of, or encapsulated within waste rock;
4) Waste rock piles encapsulation with tailings;
5) Layered co-disposal of tailings and waste rock;
6) Fully mixed placement (“co-mingling”) of tailings and waste rock; and
7) Pumped co-disposal of coarse and fine tailings (Wickland, et.al., 2006).

20.3.3. CO-MINGLING OR CO-LAYERING

With filtered tailings produced from the processing of ore for placement into a composite stack together with the waste from the mine, either options 5 or 6 above are applicable. This is because the filtered tailings have a sufficiently high solids content targeted at 87.5% (weight solids/weight of solids plus water) for placement and compaction within the co-disposal facility. This allows for higher densities at the time of placement to be achieved. The other options assume either slurry, thickened or paste tailings which have a lower tailings solids content.

For layering co-disposal, the waste rock and filtered tailings are placed separately in layers.

For co-mingling, the waste rock and tailings are blended and then placed. The effectiveness of the blend can depend on the ratio of mixing of tailings and waste as well as the rock porosity. Figure 20-3 (Wickland et al. 2006) below shows the optimum "just-filled" point when the optimal interstitial contact between the rock and the tailings are also fully occupying the void space, with minimal air present within the mix. This is also referred to as the transition between rock dominated and fines dominated behaviour.

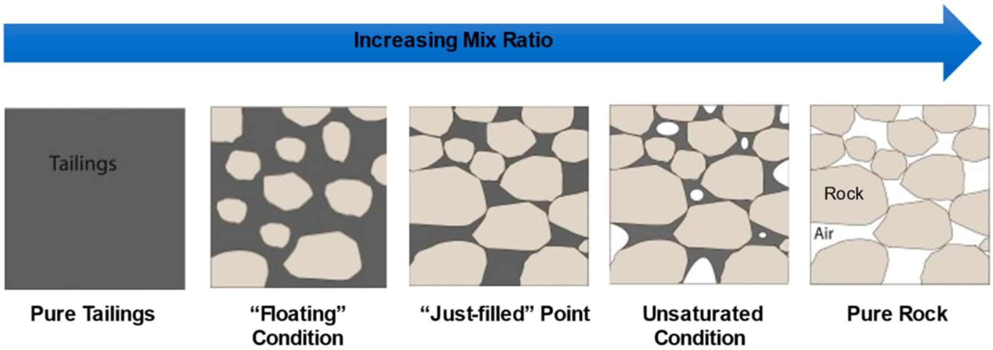


Figure 20-3 – Diagram of Blending during Co-mingling for Ranges of Mix Ratios

The mix ratio (R) is defined as the dry mass of waste rock solids to the dry mass of tailings solids. For the precious metal tailings at Ikkari, this is effectively the same as the strip ratio of ore and mine waste when working in the open pit. For comingling to be effective, the tailings would need to be blended and placed in this range. Also, minimal waste is available for co-disposal when working underground.

Figure 20-4 below shows a ternary diagram indicating the ranges of mixing for various solids content (Burden and Ward, 2023). The percent solids content (mass of solids / total mass) and tailings solids content (mass of tailings solids / tailings solids and water or weight solids / weight slurry) are combined with the mix ratio to show the structure and behaviour of composite tailings (which have both a fines and sand content). The transition zone has a waste rock skeleton porosity between 40% of 50%, which is used to define the optimum mix ratio.

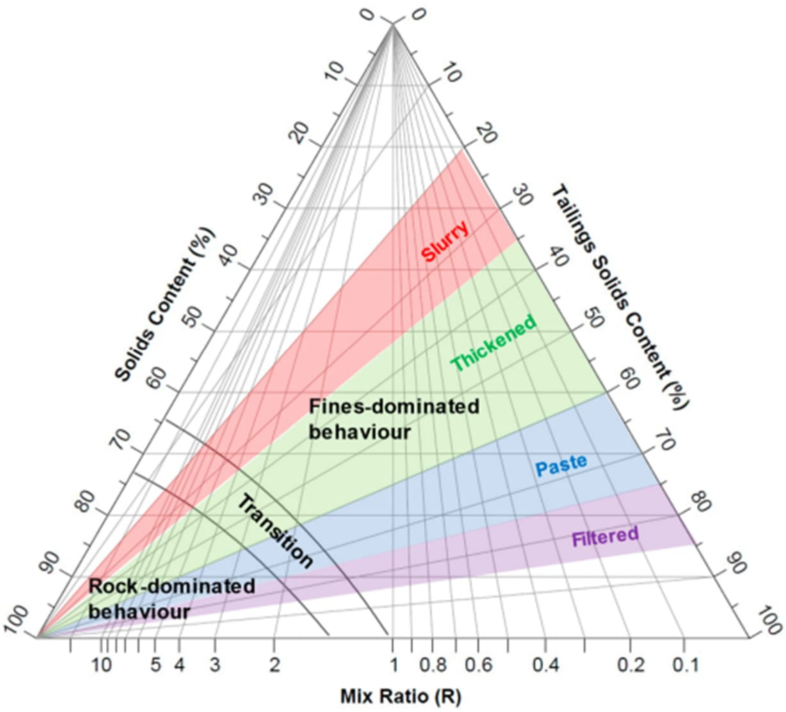


Figure 20-4 – Diagram of Optimum Mix Ratios and Various Tailings Solids Content

For filtered tailings, it is observed that there is a limited operational scope within the transition zone. It is therefore not considered practical at PFS level to use co-mingling for the deposition of the combined waste streams.

The reasons for this are summarised as follows:

- Expected strip ratio not comparable with R between 1.25 and 1.5 for filtered tailings (yearly ranges of this are provided in Table 20-3);
- The fines to sand content of the gold tailings is not yet fully confirmed;
- A waste rock stream with suitable size would be required for blending with the filtered tailings which would require blast design and controlled selection of material from the open pit;
- Additional challenges associated with meeting specific co-mingling requirements when operating 24/7 all year round within the Arctic;
- Additional challenges in the event of shutdown on either the mining operation or the process plant;
- Varied particle size and rock porosity of the waste;
- More challenging to provide construction quality assurance of placed co-mingled waste and tailings and define strength parameters;
- Drainage of comingled layers would be less effective than waste layers; and
- Confining embankments may also be required using the excess waste.

It is therefore recommended to place the filtered tailings and waste rock in separate layers when forming a co-disposal stack. Each of these will need to be placed, spread and then compacted to achieve density and strength. This process of co-layering is indicated in Figure 20-5 below.

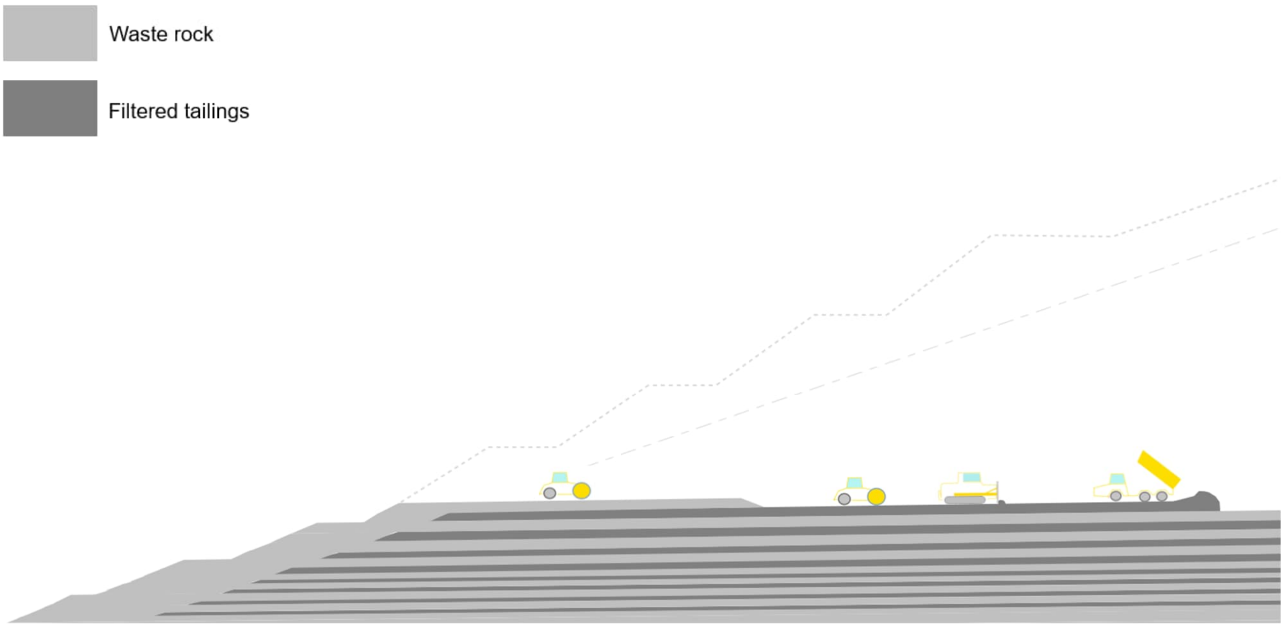


Figure 20-5 – Diagram showing co-layering

20.3.4. DISPOSAL LOCATION

Topography

Figure 20-6 indicates that much of this site for the co-disposal facility is above the 220 m contour on two distinct higher elevation sections, one of which is named Pahkalehto. Surrounding these at the lower elevation is marsh which is “easy to traverse” according to the mapping legend and typically relates to frozen winter conditions. Access to the area can be gained along forestry access tracks from the South along unpaved roads using vehicles less than 30 t gross weight.

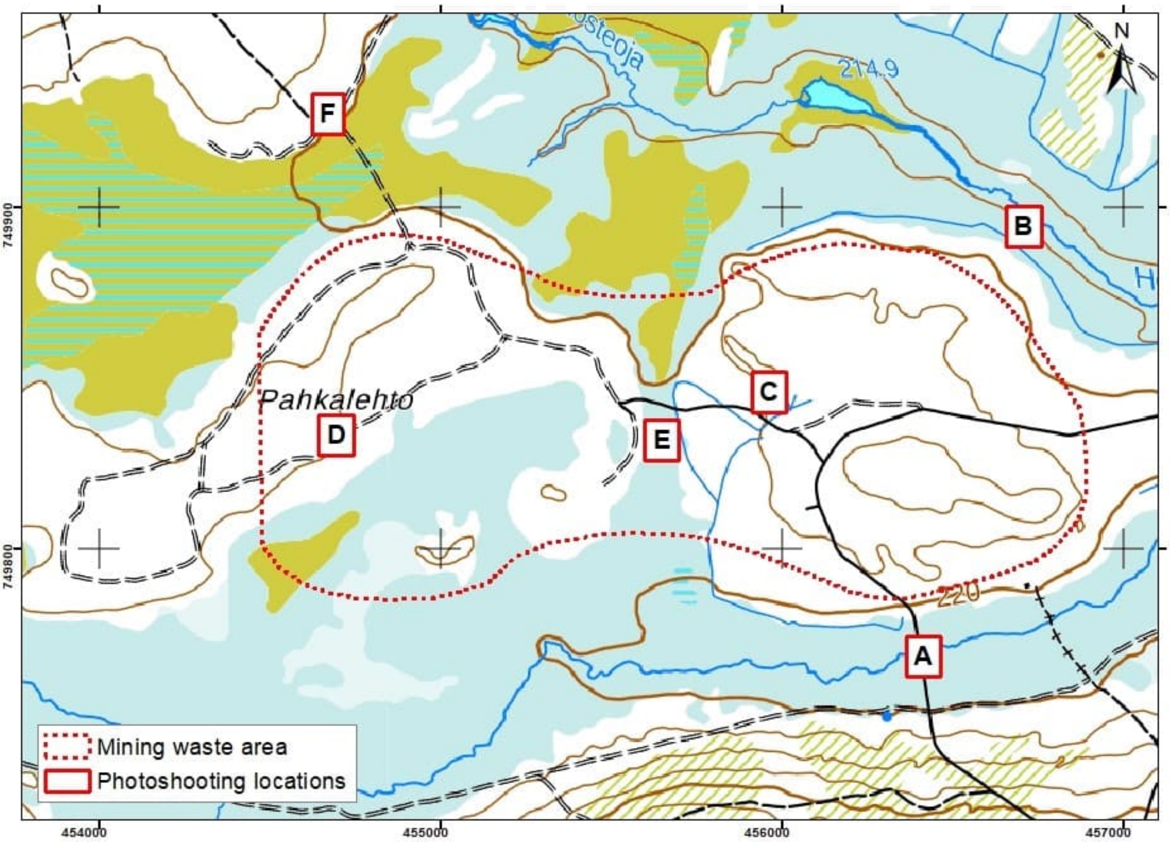


Figure 20-6 – Photoshoot locations and planned Mining waste area

Photographs of Site Area

Photos were taken during the site visit from 30th to 31st August 2023 at various locations as indicated on Figure 20-6 are shown below (Figure 20-7 to Figure 20-14).

Figure 20-7 – K1 Point A Looking North

Figure 20-8 – K1 Point B Looking North

Figure 20-9 – K1 Point C Looking South

Figure 20-10 – K1 Point D Looking Southwest

Figure 20-11 – K1 Point E Looking Southeast

Figure 20-12 – K1 Point E Looking East

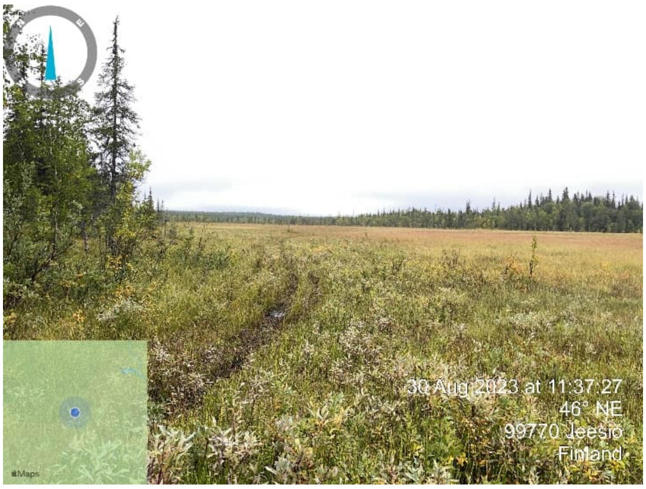


Figure 20-13 – K1 Point F Looking Northeast

Figure 20-14 – K1 Point F Looking East

20.3.5. CLIMATIC DATA

Due to its location, between the 60th and 70th northern parallels, the Ikkari Project Site is classified as continental, subarctic with cool summers and all year long precipitation. A further significant influence is the prevailing air currents which when westerly tends to bring warmer and clearer maritime weather and when easterly can account for more severe continental conditions both warm in summer and severely cold in winter.

Average temperature readings over a ten year period taken from the nearest meteorological station at Sodankylä, indicate mean temperatures in summer (June to August) ranging from 10°C to 15°C, while winter months (November to April) range from -5°C to -20°C, as shown in Figure 20-15.

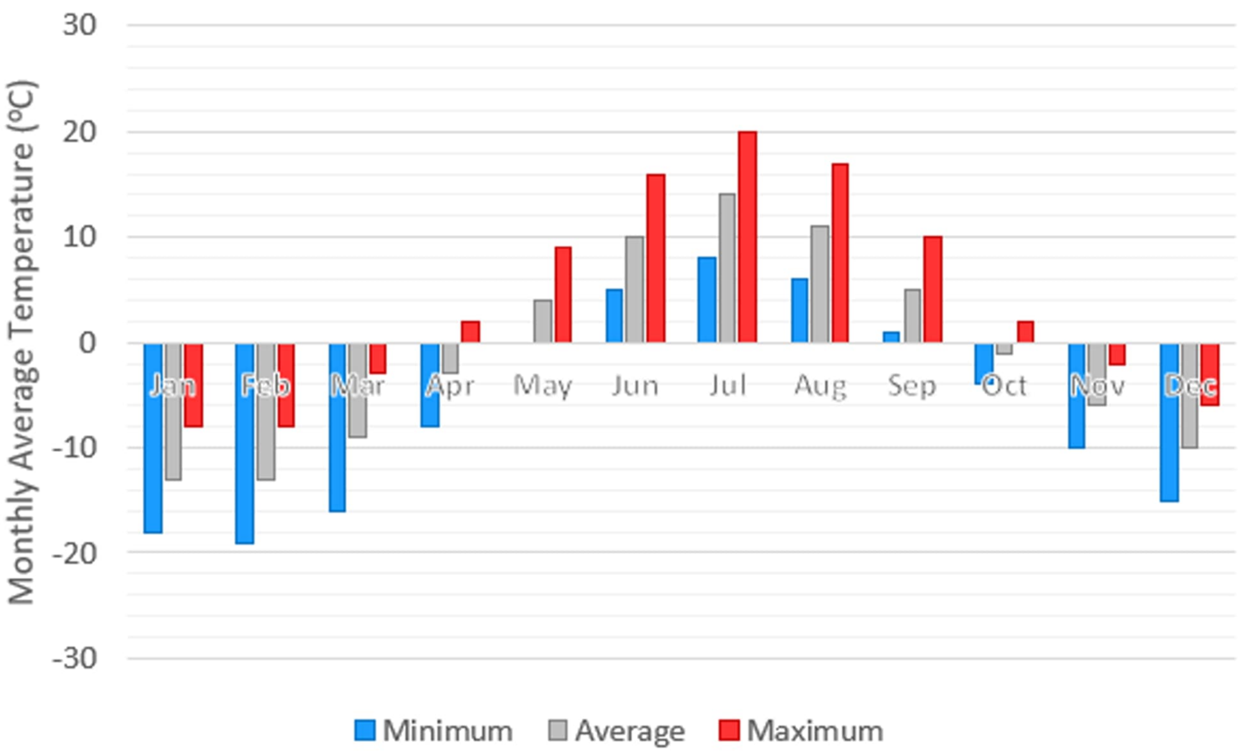


Figure 20-15 – Average temperatures recorded at Sodankylä

Snow covers the terrain on an average of 183 days in the year with maximum snow thickness varying between 0.6 m to 1.2 m in March (SRK 2023a). Surface waters are frozen for four to five months of the year. Precipitation occurs as snowfall during the months of September to May. From June to August rainfall predominates. The snow melts annually in April and May, creating significant water run-off during these early spring months.

Table 20-2 – Average Monthly Snowmelt Plus Rainfall Data (1962 – 2021, WSFS Model)

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Rainfall + Snowmelt (mm)	1.1	1.4	5.1	61.8	170.9	65.0	74.8	64.3	54.1	37.7	16.3	3.9	556.4
Number of days either occurred	1	1	4	15	26	23	25	26	24	19	9	3	177

As shown in Table 20-2, the modelled snowmelt plus rainfall data shows high values during spring thaw and low values during winter months, when most of the precipitation occurs as snowfall and due to the low temperatures limited snowmelt occurs.

The WSFS simulated combined rainfall and snowmelt time series has been adopted for assessing monthly average contributions to the mine water balance. It has also been used to size the raw water storage ponds on the site to limit the frequency of spills to the environment.

20.3.6. PRELIMINARY DESIGN OF FACILITY

Basis of Design

The following aspects are taken into consideration during the design of the facility;

- Storage capacity for life of mine;
- Stability of containment of mine waste;
- Safety and practicality of operation;
- Containment of seepage and surface runoff;
- Constructability with staged development;
- Ease of operation;
- Minimising environmental impact; and
- Ability to rehabilitate on closure.

It is anticipated that this ore production and waste output will vary as the mine plan changes. The co-disposal facility has a range of benefits including:

- Flexibility for raising and expanding the capacity;
- Possibility for early rehabilitation of the sides;
- Top shaping for closure profile;
- A level of flexibility of mining and strip ratio;
- Variability of placed densities for filtered tailings and waste; and
- Access for maintenance.

The current design provides an extra 13.5% capacity to allow for this as discussed in Section 20.3.1.

20.3.7. OUTLINE AND SHAPE OF FACILITY

The outline of the footprint is shown on Figure 20-16.

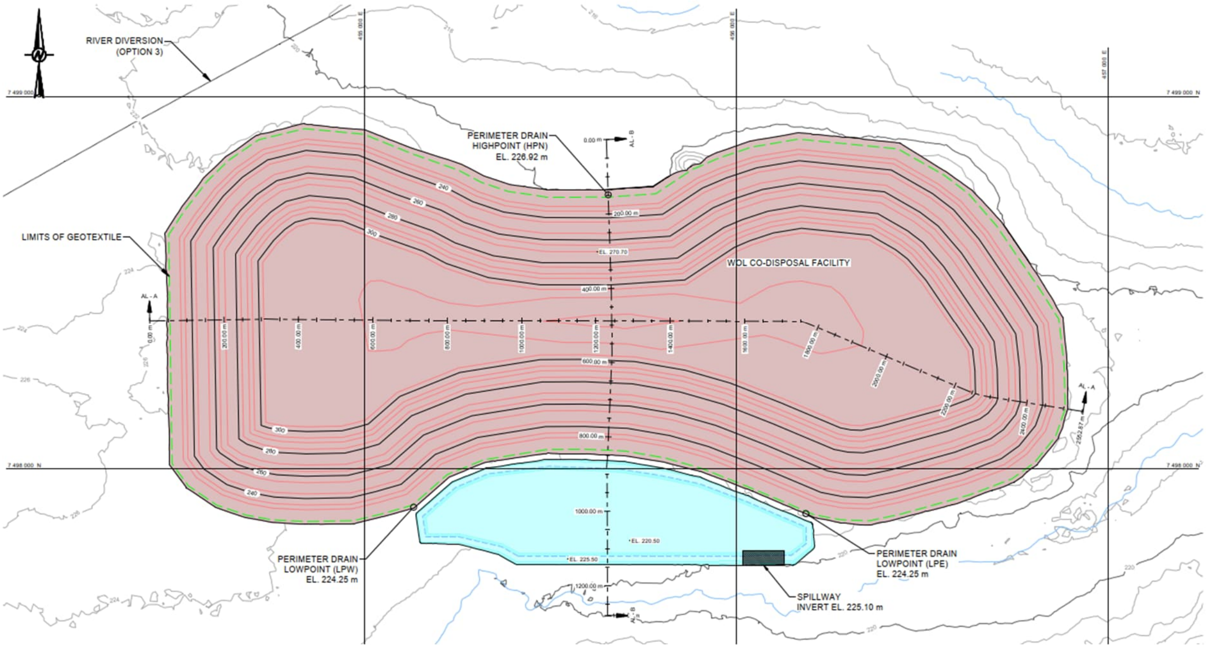


Figure 20-16 – Developed Plan of WOL Co-Disposal Facility

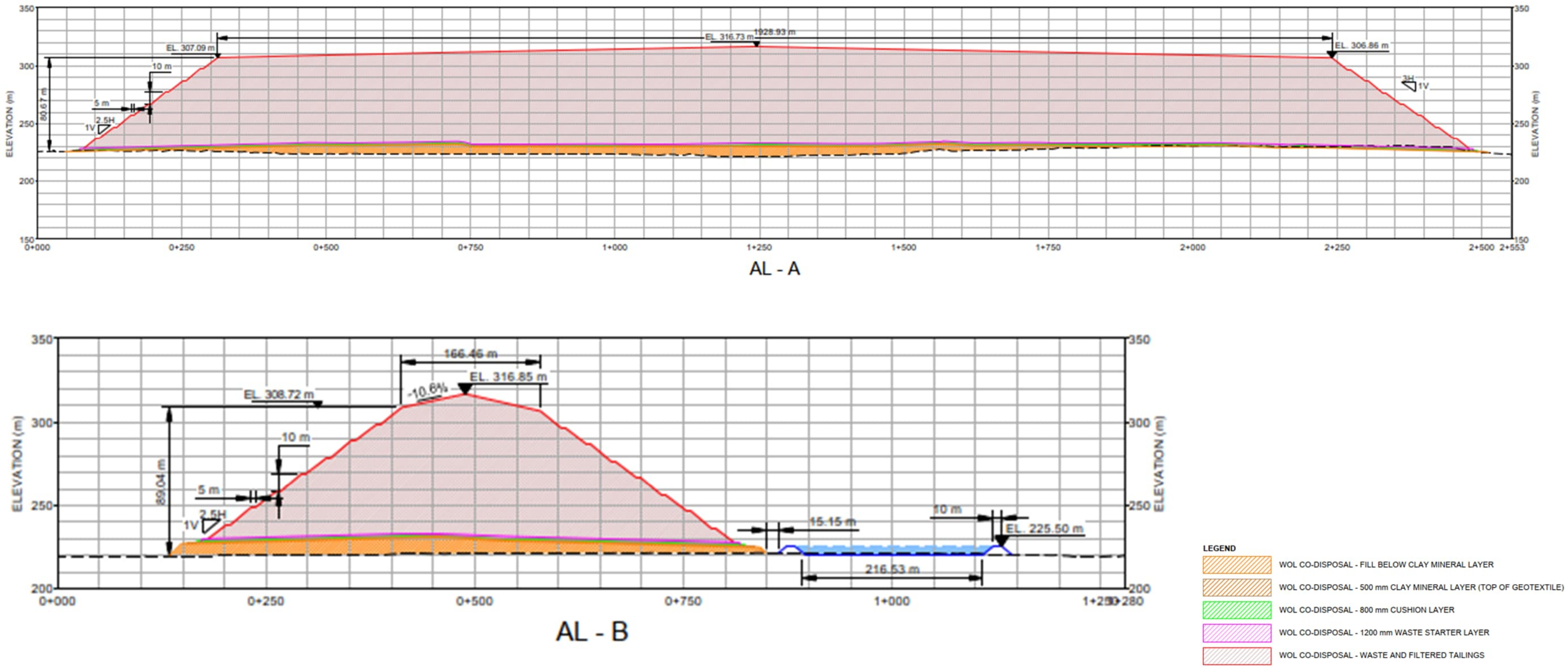
This makes use of the two higher topographies at this location. It avoids the Heina Central deposit to the West which is not considered in the PFS. Whilst the deposit is not fully defined, the footprint does not extend westwards beyond 454 450 mE. It also avoids the softer ground between the higher topographies with a reduction in width to the North and South on the assumption that the peat will need to be removed before overburden being placed to form the foundation to the co-disposal facility. Space to the South is also provided for accommodating the runoff collection pond which is located North of the Saittajoki stream. Curvature is applied to the perimeter to provide a more sympathetic geomorphology of the facility once raised and to minimise material handling on closure.

Three-Dimensional Modelling

Civil 3D modelling (computerised software package) indicates that a height of approximately 80 m would be required for this configuration. With an average base elevation assigned at 227 m, this would have a top elevation of 307 m. This is modelled with 10 m lifts having sides at 1 (vertical): 2.5 (horizontal) and with 5 m wide berms, giving an average side slope of 1 (vertical) : 3 (horizontal).

This criterion is based on the Best Available Techniques (BAT) (EU Directive 2006/21/EC, 2018).

The plan and elevation are shown in Figure 20-17 (Civil 3D model).



Note: Vertical scale exaggerated by factor of 2 with horizontal scale.

Figure 20-17 – Developed Sections of Co-disposal Facility

This is modelled with a flatter top surface, which could be further modified to accommodate design, operational and closure requirements as discussed in Section 20.3.15. Also, a flatter side slope profile may be required at the top for storing the tailings when mining underground when there is no waste for the co-disposal layers. The top surface of the filtered tailings will be profiled to control surface runoff. If necessary, the gradient would need to be controlled to minimise collecting catchments and small flat area 'berms' could be introduced to slow down the velocity of any channelling runoff.

Further design will include requirements for the introduction of an access road on the side of the facility which will rise from the entry point in the SW corner of the facility where it is the shortest distance from the open pit. This would typically have a longitudinal gradient of 10% and either travel Northwards or Eastwards, depending on operational and closure preference.

Depth Capacity

The layering of waste and filtered tailings is placed within the area contained, which reduces as the stack rises. The depth capacity curve up to 91.5 Mm3 is shown in Figure 20-18 below.

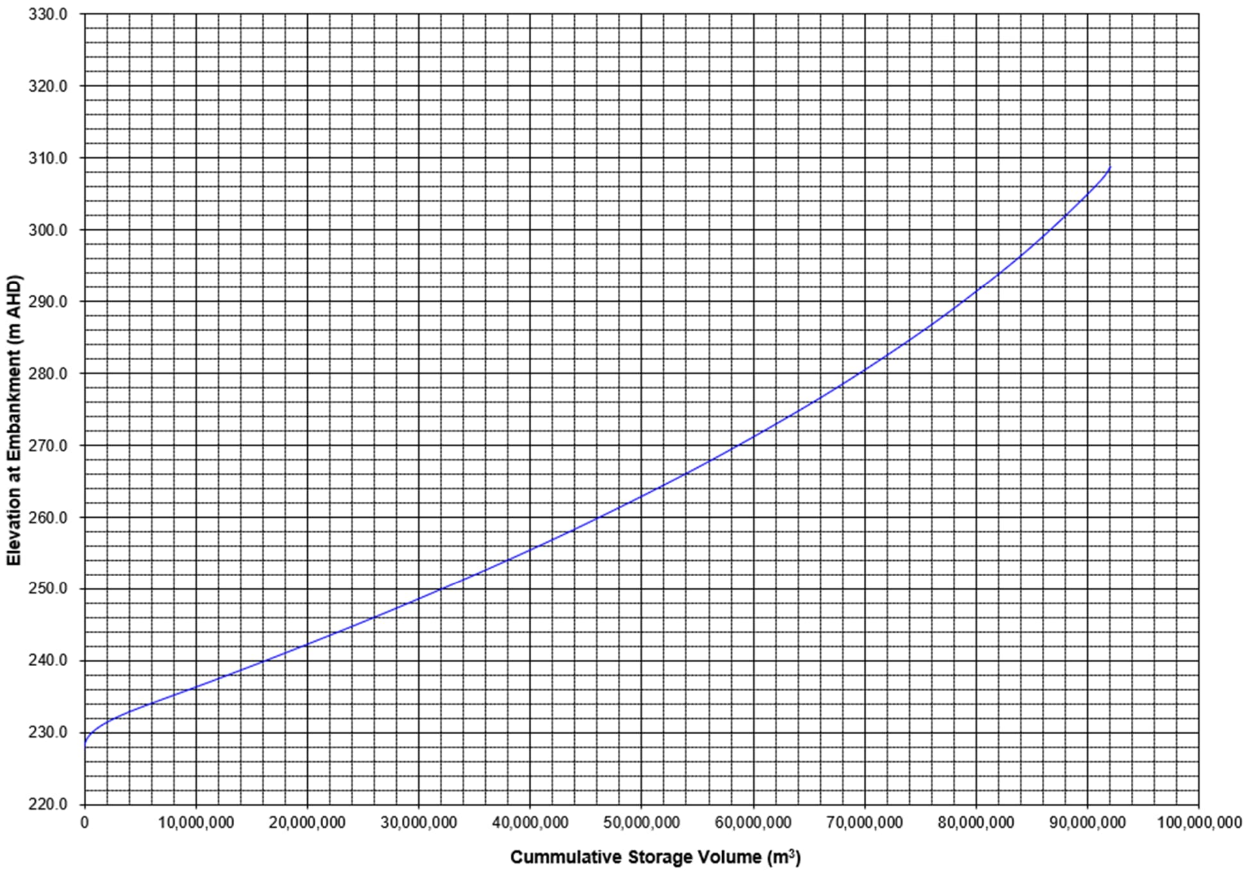


Figure 20-18 – Depth Capacity Curve for WOL Co-disposal Facility

20.3.8. OPERATION OF CO-DISPOSAL FACILITY

Continuous Filling with Filtered Tailings and Waste rock

The filling of waste rock and filtered tailings will need to be combined into one continuous operation. Both the filtered tailings and waste rock will need to be placed in a series of layers when filling the facility.

Table 20-3 provides the estimate of ore and waste excluding overburden to be stored in the co-disposal facility on a yearly basis. This data is used to determine the height of the waste and tailings for each year, whilst also allowing for a 40 m (horizontal width) waste rock wedge on the outside face of the facility. The initial estimate of tailings and waste depths also on a yearly basis are shown in Table 20-4.

Table 20-3 Mining Estimate of Ore Production and Waste Output on an Annual Basis

Year	-1	1	2	3	4	5	6	7	8	9	10
Ore Tonnes (t)	0	3 512 000	3 525 000	3 525 000	3 484 641	3 108 246	3 473 705	3 500 000	3 500 000	3 600 000	3 472 773
Waste Tonnes (t)	778 983	11 993 503	19 264 726	17 133 657	15 037 764	14 949 939	14 245 938	9 190 937	5 799 009	3 514 555	2 991 108
Strip Ratio	-	3.4	5.5	4.9	4.3	4.8	4.1	2.6	1.7	1.0	0.9
Year	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**Total**
Ore Tonnes (t)	2 203 986	1 991 547	2 000 000	1 967 182	2 049 999	1 946 432	1 903 341	1 783 493	1 178 972	271 925	51 998 241
Waste Tonnes (t)	694 397	361 371	251 128	0	0	0	0	0	0	0	116 207 016
Strip Ratio	0.3	0.2	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	2.2

Notes:

Waste tonnes exclude overburden material.

Table 20-4 – Initial Estimate of Annual Tailings and Waste Depths

Year	-1	1	2	3	4	5	6	7	8	9	10
Utilisation %	50	100	100	100	100	100	100	100	100	100	100
Height of tailings (m)	-	1.46	1.55	1.71	1.85	1.88	2.32	2.63	2.85	3.25	3.42

Year	-1	1	2	3	4	5	6	7	8	9	10
Height of waste layers (m)	0.44	3.08	5.22	4.99	4.65	5.33	5.51	3.71	2.25	1.14	0.89
Combined Height (m)	0.44	4.53	6.77	6.69	6.49	7.21	7.83	6.34	5.10	4.39	4.31
Year	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	20	
Utilisation %	100	100	100	75	75	75	75	75	50	25	
Height of tailings (m)	1.46	0.99	0.84	1.04	1.17	1.24	1.13	2.16	1.37	0.72	
Height of waste layers (m)	0.05	0.09	0.05	-	-	-	-	-	-	-	
Combined Height (m)	1.51	1.09	0.89	1.04	1.17	1.24	1.13	2.16	1.37	0.72	

Notes:

Based on mine operation schedule (derived in Deswik) estimation, (WSP, 2024)

An estimated 1 639 171 tonnes of waste to be returned underground for backfilling.

Density of placed and compacted waste rock used for estimation is 2.19 t/m3 and for filtered tailings is 1.70 t/m3, based on PEA.

This considers that only 50% of the co-disposal will be available in Year 1, to allow for staged construction of the facility. It assumes that this will be fully available for Year 2.

Also, as the mining works progress underground after year 14 the utilised surface area is reduced to 75% to provide more manageable height lifts. This is then further reduced to 50% after year 18 to enable shaping of the top profile for closure, and 25% in year 21. Non utilised areas can then be filled in subsequent years to maintain optimal capacity.

This considers a wedge of waste rock around the external perimeter with a horizontal width of 40 m for the first eight 10 m high lifts. At the top there is an extra wide berm at approximately 35 m width and a reduced permitter wedge of waste rock 3 m wide, when there is also less waste because of working underground.

The opportunity for stockpiling geochemically suitable rock for future use should also be considered.

The initial year of ore processing is in Year 1 with a steady production rate at approximately 3.5 Mt/a until Year 10. Most of the waste is produced between Years -1 to 8 with a maximum of 19.3 Mt/a in Year 2, this then declines to 9.2 Mt in Year 7. After Year 7 the waste production steadily reduces from 5.8 Mt/a to 0.25 Mt/a when the mining operation transfers from the open pit to underground. The strip ratio accordingly ranges from an initial 3.4 (waste/ore by weight) in the first year of production, and up to 5.5 in Year 2 and is also approximately 4.1 to 4.9 in Years 3 to 6, before decreasing to 0.9 in Year 10. This then further reduces with underground workings when there is less waste extracted, giving a strip ratio as low as 0.3 to 0.1 before the waste rock is returned as backfill.

The initial height of tailings in Year 1 is 1.46 m and then during Year 2 to 10 the yearly height of tailings ranges from 1.55 m to 3.42 m, with further reduction to 1.46 in Year 11. Then the height ranges from 0.84 m to 2.16 m.

Layering of Tailings and Waste

It is anticipated that the tailings will need to be placed in layers with compacted depths ranging from 250 mm to 350 mm. Multiple layers can be combined with these ranges to meet the required yearly depth of placing. The potential for increasing the layer height can be demonstrated during compaction trials as discussed in Section Compaction Trials.

The height of waste ranges from 6.64 m in Year 1 and downwards to 1.14 m in Year 10. Then the height ranges from 0.9 m to 0.05 between year 11 and 14. The particle size distribution curve for the waste rock which is based on the PEA (Tetratech, 2023) is shown in Figure 20-19.

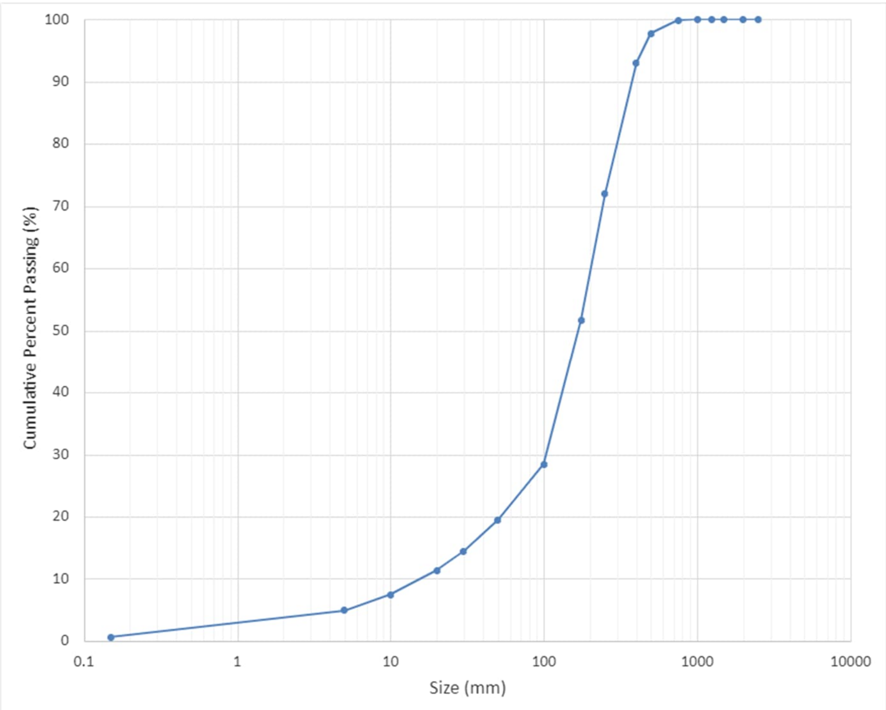


Figure 20-19 – Particle size distribution (PSD) of waste rock

This indicates that the maximum size of rock is approximately 800 mm, with the majority below 500 mm (D98 ~ 500 mm). On the basis that the maximum particle (rock) size when placing and compacting is no greater than 2/3 the depth of layer, a minimum placement depth of 750 mm is recommended to avoid the requirement for breaking oversize rock.

The oversize will depend on the layer depth as shown in Table 20-5.

Table 20-5 – Maximum Rock Size for Layer Depth Proposed with Estimate of Oversize

Maximum Rock size (mm)	Layer Depth (mm)	Within size (%)	Oversize (%)
400	600	93.1	6.9
500	750	97.8	2.2
600	900	98.6	1.4
700	1 050	99.5	0.5
750	1 125	99.9	0.1

The oversize can either be cleared from the main stack area or locally broken with a rock hammer on a tracked machine.

Typically, a maximum layer depth of approximately 1 000 mm is anticipated to ensure effective compaction through the entire layer.

The layer depths can likewise be varied to meet the required yearly depth of placing. Estimates of layer depths for selected years are provided in the Table 20-6.

Table 20-6 – Estimates of Layer Depths for Tailings and Waste within Co-disposal Facility

Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Yearly total height of tailings (m)	-	1.46	1.55	1.71	1.85	1.88	2.32	2.63	2.85	3.25	3.42	1.46	0.99	0.84	1.04	1.17	1.24	1.13	2.16	1.37	0.72
Number of layers of tailings	-	5	5	5	6	6	7	8	9	10	10	5	3	3	3	4	4	3	6	6	3
Equivalent height of each layer of tailings (mm)	-	291	310	341	308	313	332	329	316	325	342	291	331	279	347	293	310	377	361	228	240
Yearly total height of waste behind outer wedge (m)	0.44	3.08	5.22	4.99	4.65	5.33	5.51	3.71	2.25	1.14	0.89	0.05	0.09	0.05	-	-	-	-	-	-	-
Number of layers of waste	1	5	6	6	6	6	6	5	3	2	1	1	1	1							
Equivalent height of each layer of waste (mm)	442	616	869	831	774	889	919	741	750	569	892	49	94	48							
Sequence of placing filtered tailings (FT) and waste (W)	W	W-FT-W-FT-W-FT-W-FT-W-FT	W-FT-W-FT-W-FT-W-FT-W-FT-W	W-FT-W-FT-W-FT-W-FT-W-FT-W	FT-W-FT-W-FT-W-FT-W-FT-W-FT-W	FT-W-FT-W-FT-W-FT-W-FT-W-FT-W	FT-W-FT-W-FT-W-FT-W-FT-W-FT-W-FT	W-FT-FT-W-FT-FT-W-FT-FT-W-FT-FT-W	FT-FT-FT-W-FT-FT-FT-W-FT-FT-FT-W	FT-FT-FT-FT-W-FT-FT-FT-W-FT-FT-FT	FT-FT-FT-FT-FT-W-FT-FT-FT-FT-FT	FT-FT-FT-FT-FT	FT-FT-FT	FT-FT-FT	FT-FT-FT	FT-FT-FT-FT	FT-FT-FT-FT	FT-FT-FT	FT-FT-FT-FT-FT-FT	FT-FT-FT-FT-FT-FT	FT-FT-FT-
Maximum depth of tailings between layer of waste (mm)	-	291	310	341	308	313	332	658	949	1 300	1 711	1 456	994	838	1 042	1 172	1 239	1 130	2 164	1 369	240

The two streams of both filtered tailings and waste will need to be combined to form “sandwiching”. This will allow for dissipation of any excess porewater pressure (PWP) in the tailings. In addition to this, it is expected that doming of the tailings and waste rock surface will be required to allow for flow of both PWP and rainfall runoff to the outer edges of the stack when it can then be transferred into the perimeter channel.

Where the balance of waste streams requires two or more layers of filtered tailings to be placed one above the other or when there is limited availability of rock from underground, then additional drainage measures may need to be incorporated to further assist with the dissipation of PWP by shortening the drainage pathways and an assisting with consolidation of the tailings. These may include rock drainage zones or proprietary drainage systems.

Based on the mine schedule in Table 20-3 and the PSD in Figure 20-19, the waste rock layer depths are greater than ~750 mm during Years 2 to 8, with less than 0.1% oversize as indicated in Table 20-5. In Year 9, the waste layer depth is estimated to be 569 mm with up to ~1.6% oversize. During Year 10, this estimated depth is 892 mm. For Year 11 and beyond, the workings are underground when additional drainage measures will need to be introduced because the rock mined is being used for backfilling.

The sequencing of filtered tailings will also need to consider adjoining layers between each year.

A more detailed estimate of layer depths can be further developed on a quarterly basis once the mine plan is developed and estimates of processed ore throughput, that excludes stockpiling on the ROM pad, and waste rock are identified.

It is anticipated that the waste rock type and associated characteristics will vary considerably. The Southern area of the proposed open pit area consists of lithological domains of mixed ultramafic schist, ultramafic and internal felsic. These are reckoned to be weaker strength rock type domains compared to the Northern side of the open pit, which is dominated by gabbro and black schist domains. A plan view and vertical section of the lithology domains are presented in Figure 20-20.

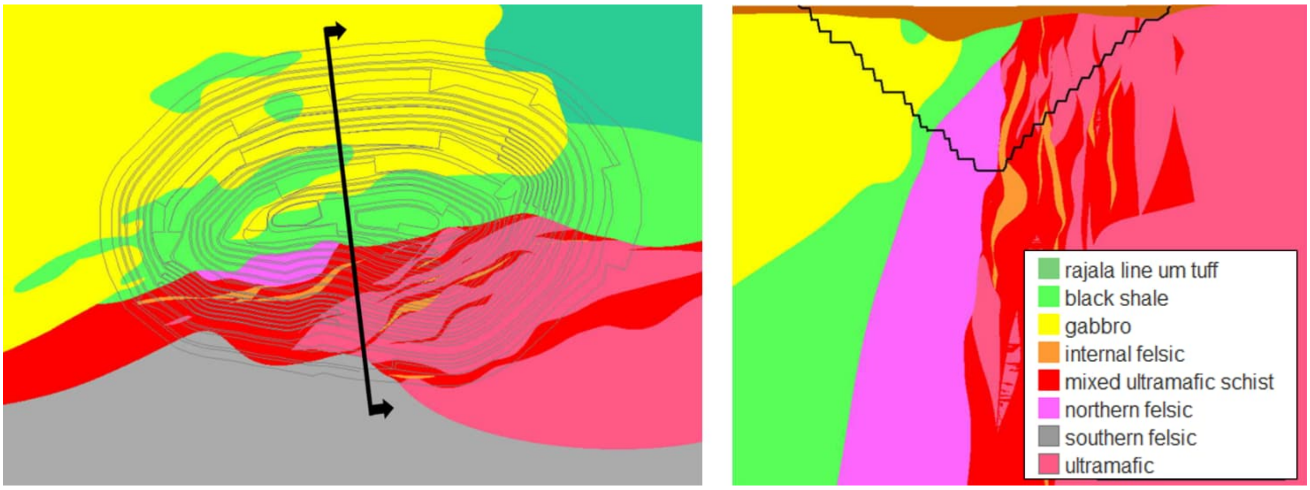


Figure 20-20 – Plan and Section of Open Pit Lithology

Compaction Trials

Optimal construction depths of filtered tailings and waste will need to be confirmed with compaction trials to determine the layer thickness and compaction from plant (number of passes) which is typically vibrating rollers. The compaction achieved in the field would need to be referenced to laboratory tests such as the proctor test which has two variations known as either Standard or Modified based on compaction energy applied to the sample at a range of moisture contents. The optimum moisture content for each energy application is then determined to provide an assurance of the compacted density being achieved. The moisture content of the filtration process would need to be compatible with that being targeted in the field for the filtered tailings. A target solids content for the filtered tailings 87.5% (weight solids/ weight slurry) is being assumed for the design of the filter plant, which will result in a geotechnical moisture content of 14.3% (weight of water/weight of solids). Optimum compaction for the waste would also need to be considered. In general, heavier plant applies more energy to the compaction process which results in a higher density at a lower moisture content for the maximum dry density as indicated in Figure 20-21. This is demonstrated by moving from a Standard Proctor test to a Modified Proctor test.

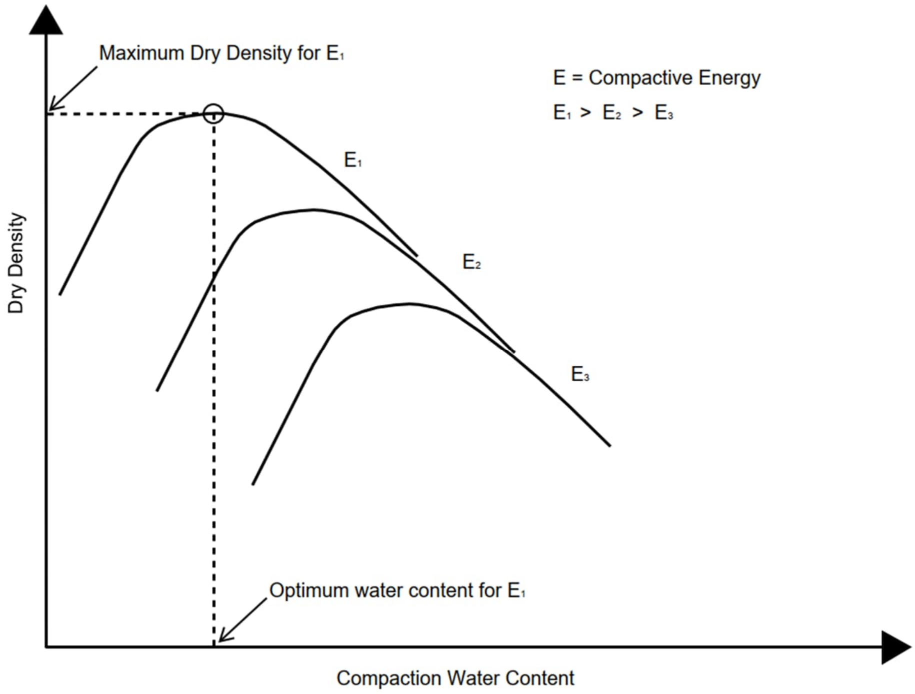


Figure 20-21 – Maximum Dry Density for Ranges of Compactive Energy

Operational Considerations

The operation includes working in winter when there are extreme cold temperatures and limited daylight/twilight, which would require specific safety and operational considerations. The filling depths could also be adjusted for winter and summer working. During winter the filtered tailings would need to be kept from freezing during transportation, and both spread and compacted immediately before freezing under extreme cold conditions. Waste should also be placed over the exposed tailings surface as soon as practicably to minimise migration of airborne emissions (dust). It is considered that only the outer layer of the facility will be impacted from seasonal freezing and thawing.

The layers would be profiled to allow for drainage towards the perimeter of the facility which is shaped to have a 1:1 000 fall around the perimeter and minimum fall of approximately 1:100 from the centre outwards to avoid ponding of surface water.

Surface runoff is collected and transferred with open channel drains constructed within the horizontal berms that would cascade down to the co-disposal runoff collection pond on the south side of the facility. Silt collection sumps can also be included within these pathways to minimise the suspended solids reporting to the collection pond. These would be located at locations which intercept silt near to the sources of runoff that have a high silt load. They may be of various sizes (typically 4 m x 2 m and 1 m deep) and lined with geotextile to contain the silt. This would be removed during periodical maintenance depending on the stacking arrangement and traffic routing of haul trucks.

Waste Rock to be Returned Underground

It is currently estimated that approximately 748 000 m^3 of waste rock will need to be returned underground.

There are various options considered for accommodating this requirement which include;

1) Providing a temporary storage area outside of the facility possibly to the East, which would only be either inert or non-acid generating rock; and
2) Providing a wider outer wedge of waste rock at a location on the stack, to allow the waste rock to be removed at a later stage. The impact on slope stability and also ability for early closure would need to be assessed.

Instrumentation and Topographical Survey

The following instrumentation is typically used to monitor the overall stability of the facility:

- Vibrating wire piezometers for identifying PWP;
- Standpipes for identifying any presence of standing water;
- Total earth pressure cells to measure weight of material placed above a given point;
- Temperature sensors for seasonal variation and also potential detection of seepage within the stack and formation of ice lenses;
- Inclinometers for identifying any lateral movement in the facility (in outer berms); and
- Survey monuments for identifying any vertical displacement (settlement).

In addition to these instruments, routine topographical surveys are taken to measure the volume of material placed, which is correlated with tonnages of waste to estimate densities being achieved.

Seepage Flows

Consideration will need to be given to the risk of seepage of surface runoff downwards through the stack and mitigation against risk of piping action causing the fines in the tailings to migrate into the voids of the underlying waste layers and impacting on the drainage capacity through these. Surface sealing of the waste may be needed (depending on the particle size distribution) to ensure large voids are covered, especially when there is localised uniformity of rock size. Additional compaction requirements may be necessary to provide assurance of the low permeability of the upper layers ensuring that seepage flows migrate laterally after closure. The impacts of both drainage seepage chemistry and chemical alteration will also need to be confirmed.

20.3.9. DEVELOPMENT OF FACILITY

Surface Profile of Facility

The base of the co-disposal facility is shaped to collect seepage and runoff and transfer this to the collection pond to the South. This is shown in Figure 20-22.

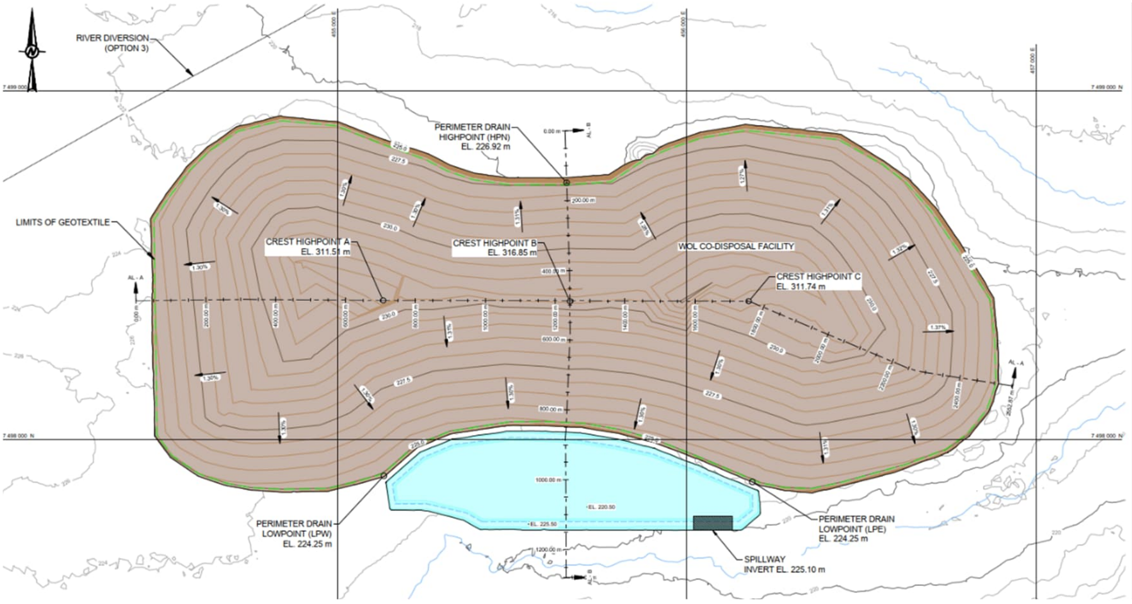


Figure 20-22 – Surface Profile of Co-disposal Facility

There is a central ridge provided in an East - West direction (A 311.51 m, B 316.85 m and C 311.74 m) which allows seepage and runoff to flow outwards to a channel around the perimeter. This has a high point (226.92 m) to the North allowing flows on the East side to travel in a clockwise direction and flows to the West, anticlockwise. There is also a highpoint to the South to allow flows into the East and West inlets of the runoff collection pond both at 224.25 m.

As the Northwest corner is approximately 50 m from Saittajoki stream, underdrainage may be required for control of ground water. Additional erosion protection on the external face of the embankment may also be required here.

Further development of profile contours and site elevations is anticipated in subsequent feasibility design phases.

Perimeter Arrangement

A typical cross-section of the perimeter to the facility is shown in Figure 20-23.

This includes the following components:

- 6 m wide perimeter access track with cross fall (currently designed outwards) and earthwork side slope;
- Containment and demarcation berm;
- Runoff collection channel;
- Margin for respective layering; and
- Toe access track.

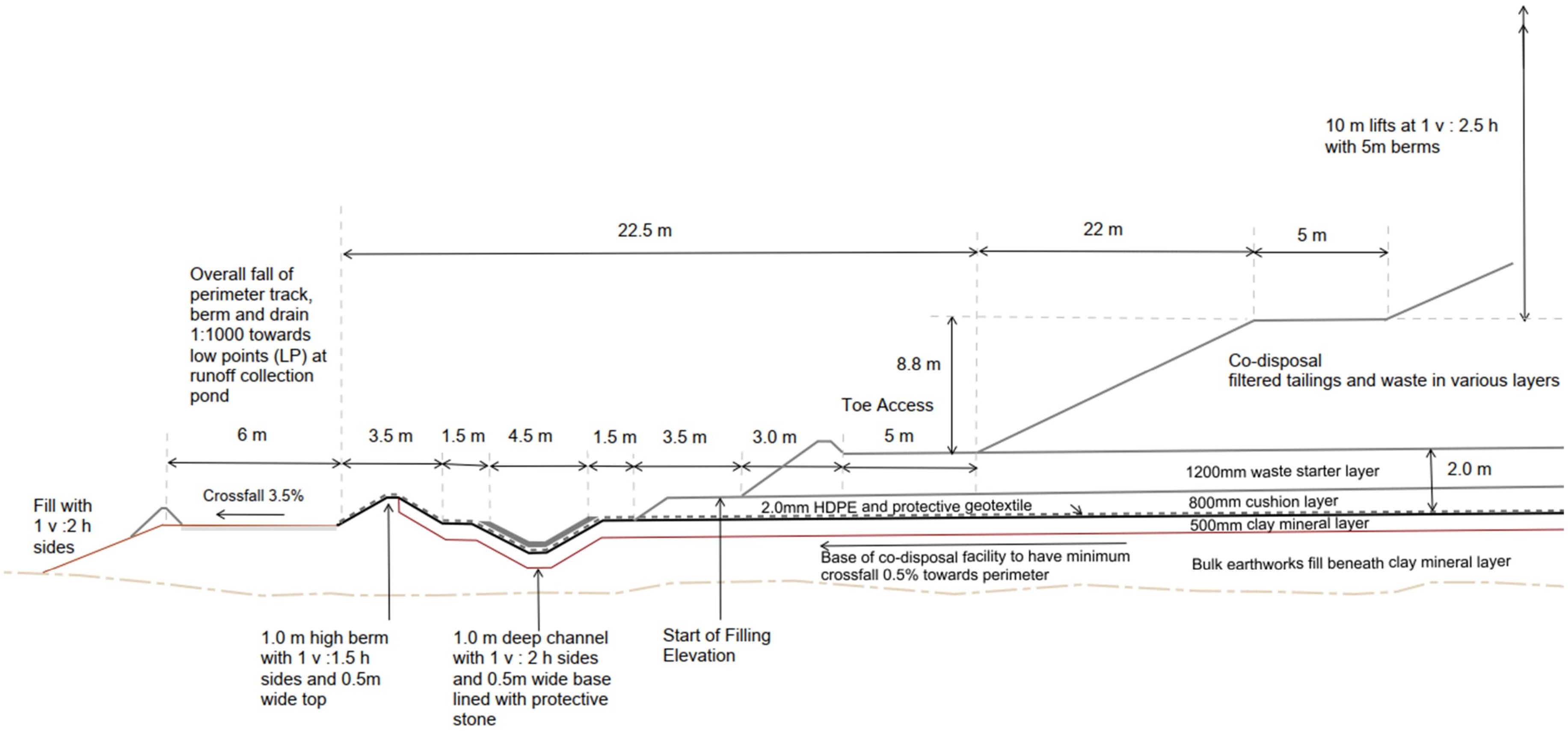


Figure 20-23 – Cross Section of Co-Disposal Facility Perimeter

Perimeter drains are covered with protective stone to minimise risk of damage when clearing snow and ice from the channel in winter.

Co-Disposal Runoff Collection Pond

Engineering fill can be used to form the embankments containing the co-disposal runoff collection pond. Both the internal and external slopes are at 1 v :2.5 h and the crests have a width of 10 m.

The co-disposal runoff collection pond has a spillway to allow for discharge under extreme flood conditions. The spillway will need to be sized to convey a PMF event. However, it is the intention that this would only be used under extreme scenarios, which is further discussed under the Water Management Section of the PFS, and that the pond would be maintained below a specified minimum operating level with an allowance for wave run-up being maintained.

The design has a crest elevation of 225.5 m. There is a spillway in the Southeast corner with an invert of 225.10 m and width of approximately 30 m. A freeboard depth of 0.8 m below the spillway invert is allowed. The base of the pond has an elevation of 220.5 m. The capacity at the spillway invert is 808,000 m^3.

Water from this pond would be returned to the raw water pond for subsequent water treatment.

Further details are shown in Figure 20-17.

20.3.10. LOW PERMEABILITY LINER SYSTEM

At PFS level it is the assumption that the co-disposal facility includes a liner throughout at the base of the stack and within perimeter channels that flow to the pond. The liner is to mitigate against yet to be completed geochemistry studies. The pond will also be lined. Water is abstracted from this pond and treated before returning to the environment. There exists an opportunity to balance further washing of the tailings within the mineral process plant and the extent of the liner which sits beneath both the co-disposal facility and pond.

It is proposed that the impermeable components of the liner system will include both compacted low permeability clay and an HDPE membrane to provide a low permeability barrier which diminishes seepage of contact flows into the environment. This system, however, may be subject to change following discussion with the permitting authorities.

For the clay layer and high-density polyethylene (HDPE) membrane liner system to function effectively, additional elements are incorporated into the design. A layer of geotextile is provided above the geomembrane for protection and above this a cushion layer of sandy gravel is placed. Additional proprietary drainage measures can be incorporated above the cushion layer to improve the performance of the liner system below. The availability of clay will need to be investigated at the Feasibility Study design stage, if necessary, an alternative liner system such as LLDPE with GCL/bentonite could be used. The closure and water management permitting recommends the low permeability clay for longevity.

A starter layer of selected waste is then placed onto the cushion layer before the mine waste and filtered tailings are co-disposed. The primary function of this cushion layer is to provide protection, however, as there will be relatively high permeabilities with this layer it will assist in reducing the hydraulic potential (or gradient) over the liner.

The following technical aspects will need to be taken into consideration during the design and specification of the integral liner system:

- Assessment of the underlying soils, incorporating the requisite amount of SI data, when loaded from the stack;
- Estimation of vertical displacement (settlement) of the underlying soils;
- Profiling of the liner, considering elevations and cross falls;
- Control of any groundwater within the foundation of the co-disposal facility;
- Leak detection system;
- Compaction and surface preparation of the sub-grade;
- Depth of mineral layer (typically 300 to 500 mm);
- Particle size distribution and proportion of clay fraction in the mineral layer;
- Compaction density and impermeability of the mineral layer (typically k < 1x10-9 m/s);
- Preparation of the top surface of the mineral layer to receive the HDPE membrane;
- Thickness, surface texture and permeability of the HDPE membrane (typically 1.5 mm, smooth and k < 1x10-14 m/s);
- Placing and welding requirements for the HDPE membrane;
- Weight of geotextile (g/m2);
- Placing and overlapping requirements for the geotextile;
- Particle size distribution, strength and angularity of stone within the cushion layer;
- Depth of cushion layer (estimated 800 mm) for distribution of loading from plant during construction of waste starter layer;
- Depth of waste starter layer (estimated 1200 mm) for distribution of traffic loading from proposed mine fleet during initial raising of the co-disposal facility; and the maximum particle size in the waste starter layer;
- Construction quality assurance (CQA) process will need to include construction of the liner system; and
- Alternatives to HDPE membrane could typically include low density polyethylene (LDPE), bituminous geomembrane (BGM) or geosynthetic clay liner (GCL).

The facility is located on fill excavated from the overburden within the open pit boundary and placed to form the shape of the base to the facility. The detail design of the facility will need to consider both the underlying soils and placed overburden in the form of engineering fill. The soils will need to have sufficient strength and resistance to settlement to withstand the loading from the stacking of the facility.

20.3.11. PHASING OF CONSTRUCTION

The co-disposal facility may be segregated into cells and eight such divisions are shown in Figure 20-24. These are defined by high points, low points and the ridge to give proportioned areas. These will need to be sequenced to maintain flows along the perimeter channels whilst allowing for expansion upstream. Sufficient capacity of the runoff collection pond will need to be provided in advance of operation of the first cell. The timeline for the development of other cells is further discussed in Section 20.3.15.

Access into these may be gained via a two-lane haul road running along the Southeast corner of P1 and then into each respective cell as the formation of the facility progresses.

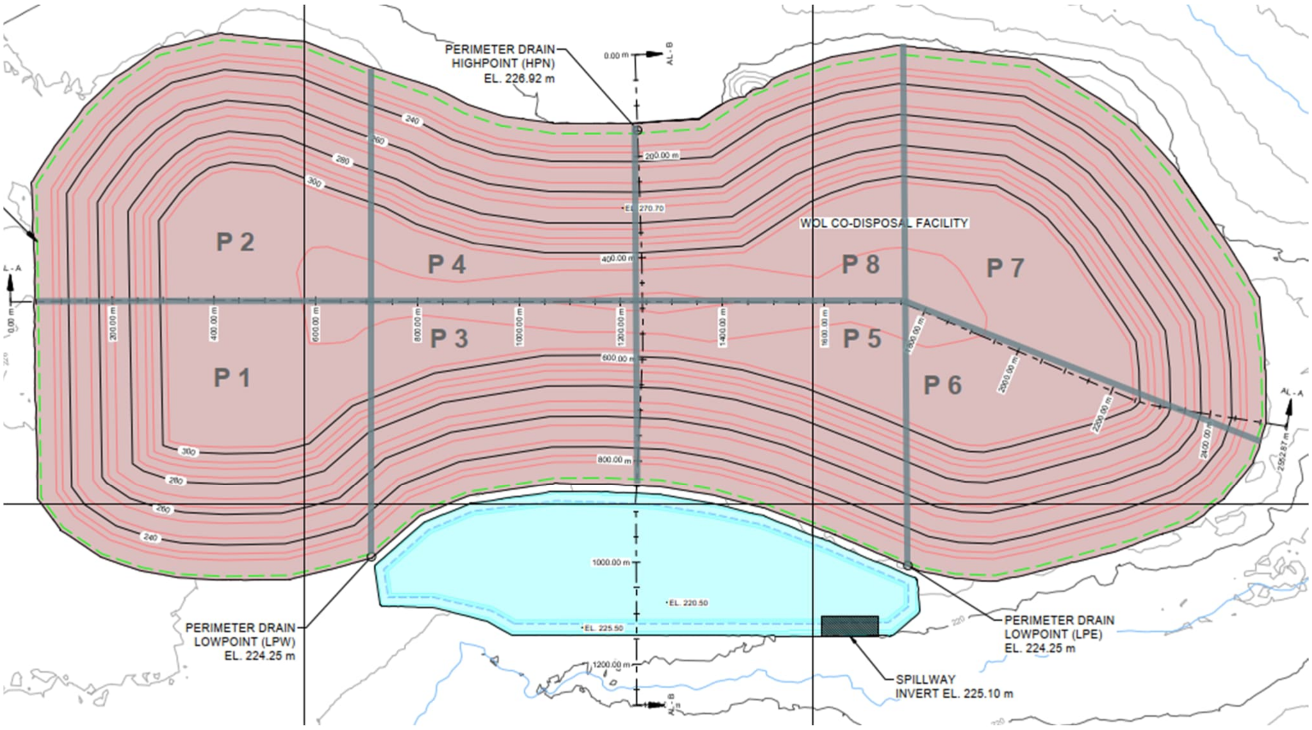


Figure 20-24 – Phased Development of Co-disposal Facility

Prior to ore production, it is anticipated that there will be considerable overburden from the open pit which would need to be stored prior to co-disposal. It is anticipated that topsoil and/or peat will need to be stripped and stockpiled nearby. This will then require inspection and testing of in-situ soils, followed by placement, spreading and compaction of overburden, which will need to be capable of forming a foundation onto which an impermeable liner could be placed to avoid differential settlement and rupture from loading during the raising of the facility. Testing of the overburden for its engineering properties will be required to assess this. Also, regrading of the existing topography may also be considered to provide cut/fill balance of engineering fill materials in conjunction with the other surface earthworks. Additionally, it is anticipated that construction materials such as sands and gravels can be sourced from the high ground.

It is anticipated that construction will need to take place during the summer months, with potential for some activity also during the full winter freeze, whilst avoiding autumn and spring. The schedule should allow for at least two years construction in advance of the co-disposal and runoff collection pond facility needing to be operational. This will include use of suitable material from the overburden of the open pit for inclusion within the foundations to the facility.

The Geochemistry of rock needs to be confirmed, and if feasible, consideration can also be given to placing waste rock within the foundations, beneath the liner.

20.3.12. RAISING OF THE CO-DISPOSAL FACILITY

Figure 20-25 gives an indication of the rate of rise of the facility.

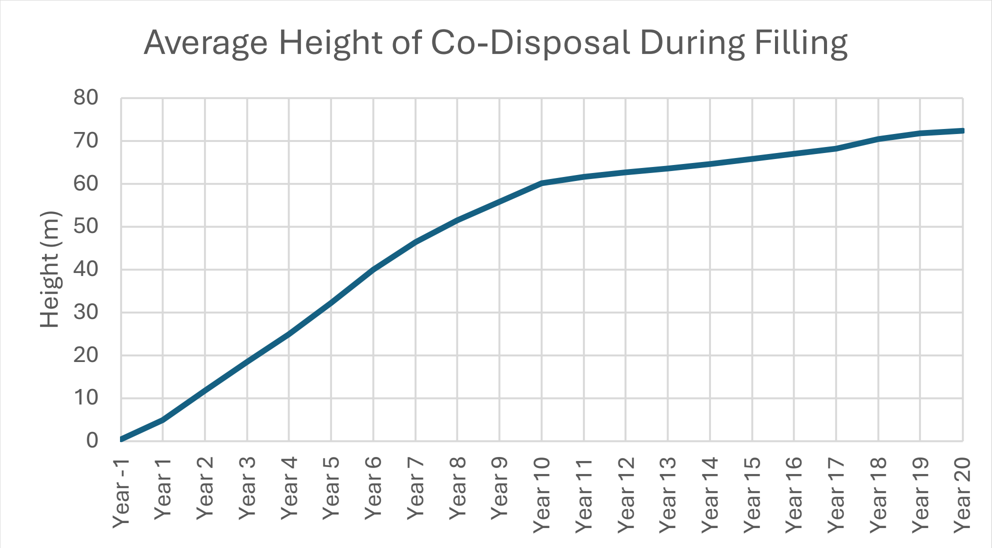


Figure 20-25 – Height of Co-Disposal Facility During Life of Mine

The rate of rise reduces considerably after Year 10 when mining works transfer from the open pit to underground.

20.3.13. DISPOSAL OF SLURRY OR OUT-OF-SPECIFICATION FILTERED TAILINGS

There is currently no contingency for an emergency disposal facility for the storage of slurry tailings in the event on non-performance of the filter plant.

Also, consideration will need to be given to the temporary storage of tailings that are not reaching the target 87.5% (weight solids) on the co-disposal facility. Re-working of these tailings will be required to allow these to be spread and compacted at the optimum moisture content.

In addition to these constraints, there is only 18 hours of storage capacity in the filter building.

20.3.14. CLOSURE ASPECTS

The design of the co-disposal facility will need to consider the requirements for closure.

The study recognises that the post mining landscape will differ from the pre-development landscape. It is the intention to reclaim the land such that it will support similar land uses to those present prior to mining, albeit in a different arrangement. Accordingly, the post-mining land use goal is to replace, to the extent possible, pre-mining ecological and socio-economic functionality.

The supporting reclamation objectives for Ikkari are as follows:

- long term physical and chemical stability of drainage courses, landforms and features;
- water quality that meets standards for discharge to the surrounding environment;
- self-sustaining, locally common vegetation that supports the targeted post-mining land uses; and
- reflection of community and stakeholder values in post-mining land uses to the extent practicable.

The opportunity for early closure of the side slopes to the facility should also be investigated, this needs to allow for:

- rock to be returned to underground works as discussed in Section 20.3.8; and
- control of waste and filtered tailings layers as discussed in Section 20.15.7.

20.3.15. SLOPE STABILITY ANALYSES

Developed Model

The developed model of the co-disposal tailing storage facility used for slope stability analysis consists of waste rock on the perimeter with an individual rise of slope 1V: 2.5H and berms of 5 m width. The overall slope of the facility being approximately 1V: 3H. The vertical distance between the berms is 10 m and the slope thickness of the waste rock is approximately 14.0 m. However, the topmost raise consists of 3.0 m thick waste rock. The waste rock and tailings are stacked alternatively within the storage facility. The cross-sectional geometry of the co-disposal storage facility in the east-west direction (A-A') and north-south direction (B-B') are shown in Figure 20-26 and Figure 20-27 respectively.

A sensitivity analysis was undertaken to demonstrate the impact from having thinner or deeper layers of tailings with respect to the adjoining waste rock. Two scenarios were investigated as follows:

1) tailings depth of 0.35 m and waste rock of 0.85 m; and
2) tailings depth of 0.85 m and waste rock of 0.35 m.

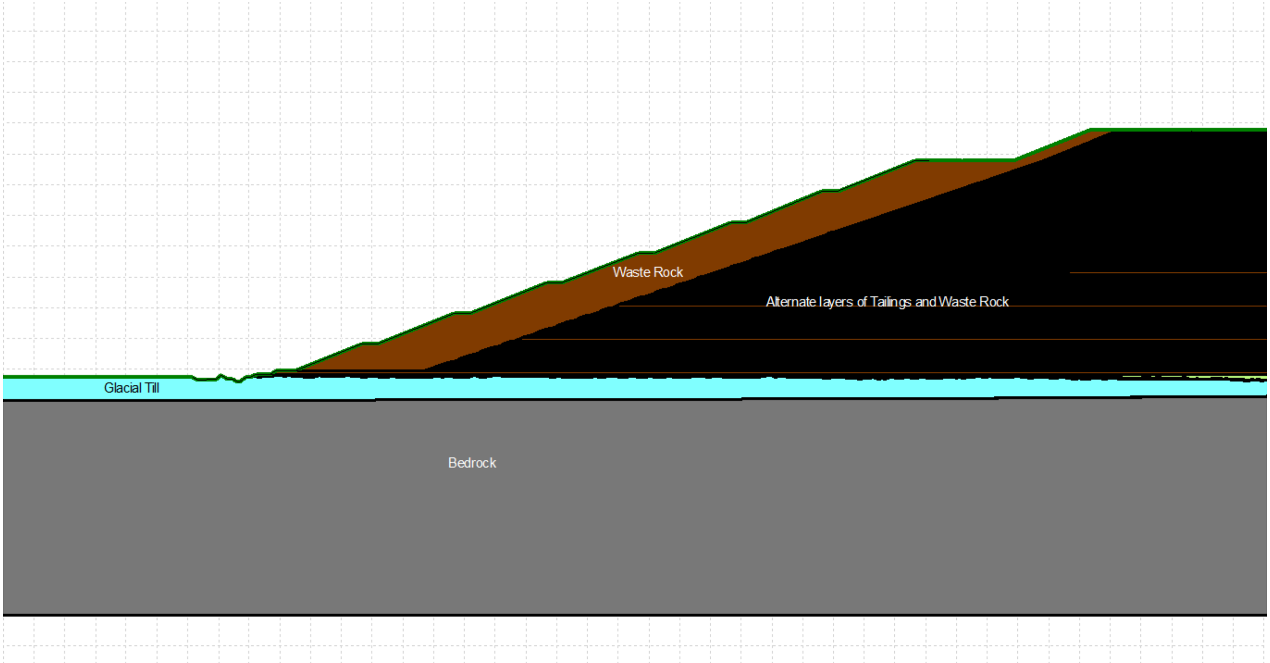


Figure 20-26 – Cross Section of Co-Disposal Facility Along Section A-A West Looking North

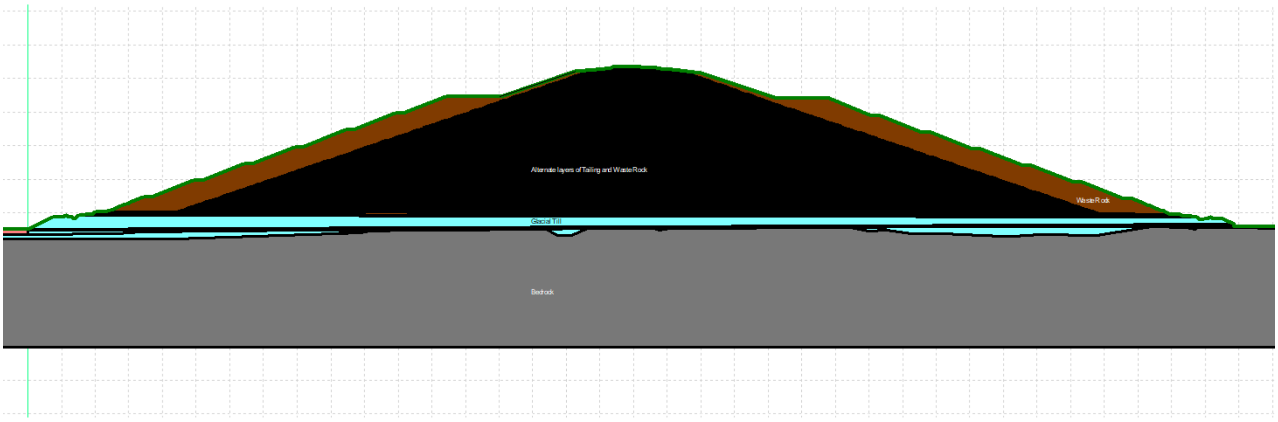


Figure 20-27 – Cross Section of Co-Disposal Facility Along Section B-B Looking East

Types of Analyses

Following types of analyses were carried out using GeoStudio:

1) Static loading conditions; and
   a. Drained analyses for different pore pressure conditions; and
   b. Undrained analyses for peak and residual strength of tailings.
2) Pseudo static analysis for both drained and undrained conditions.
   a. Drained analyses for different pore pressure conditions; and
   b. Undrained analyses for peak strength of tailings. (ANCOLD, 2012). (CDA, 2013 & 2014).

A sensitivity analyses was carried out for the above analysis types by varying the thickness of the filtered tailings and waste rock to assess the impact this might have on the stability of the stack. This considered deep seated failure slip circles and not shallow failures within the temporary 1V : 2.5H berms.

The groundwater table was assumed to be at the ground surface level, as a worst condition. The pore pressure to effective stress co-efficient, Ru for a worst condition was determined to be 0.18 for the material below the ground. This value of Ru was used for all the analysis types stated above.

Along the section A-A’, the analysis was carried out for the western and eastern slope due to varied ground conditions. The glacial till at the eastern embankment was approximately 27.0 m deep as compared to 8.0 m on the western side of the facility.

Characteristic Properties of Materials

The material characteristic properties considered for the analyses are as shown in Table 20-7 as follows:

Table 20-7 – Material Characteristic Properties

Material Type	Model	Unit Weight, γ (kN/m³)	Effective Friction, φ (º)	Effective Cohesion, c' (kPa)	τ/σ Ratio
Waste Rock	Mohr-Coulomb	20	40	0	N/A
HDPE Liner Bedding Layer	Mohr-Coulomb	17.5	37	0	N/A
Tailing (Peak Strength)	Shear/Normal Function	20	N/A	N/A	0.30
Tailing (Residual Strength)	Shear/Normal Function	20	N/A	N/A	0.18
Tailings (Drained)	Mohr-Coulomb	20	30	0	N/A
Glacial Till	Mohr-Coulomb	21	32	10	N/A
Bedrock	Bedrock (Impenetrable)	N/A	N/A	N/A	N/A

Pore Pressure

Three types of drained analysis were carried out with regards to water pressure, which is listed as follows:

1) No water content in the TSF;
2) The pore pressure to effective stress ratio, R_u = 0.1; and
3) The pore pressure to effective stress ratio, R_u = 0.2.

Seismic Load For Pseudo Static Analyses

Although Ikkari is not in a seismically active area this was considered in the analyses. According to the Canadian Dam Association Dam Safety Guidelines (CDA, 2013 and 2014), for dams with a consequence classification of HIGH for flood and earthquake hazards, the peak ground acceleration (PGA) for construction and operation (operating basis earthquake) and post closure (maximum design earthquake) is as mentioned below:

- Operating Basis Earthquake (OBE) – 1/2 475 year return period seismic event; and
- Post closure/Passive care phase, Maximum Design Earthquake (MDE) – ½ between 1/2 475 year and 1/10 000 year return period seismic event.

The OBE was obtained from the Earthquake Facilities for Earthquake Hazard and Risk (EFEHR) website. The mean value for Ikkari site noted was between 0.0001 to 0.05 g, while the 95% fractile noted was 0.1 to 0.15 g. Considering the Ikkari site to be of low seismicity, an OBE of 0.05 g was chosen.

To obtain the 1/10 000 year seismic event parameters, the parameters for the 1/475 year and 1/2 475 year return period events were plotted on a log-log scale as shown in Figure 20-28, and the values extrapolated to estimate the parameters for the 1/10 000 year seismic event. This procedure is as per Natural Resources Canada Seismic Hazard calculator (NRCAN, 2016) for estimating low probability return period seismic events. The graph below in Figure 20-28 shows the extrapolation method which resulted in a PGA value of 0.11g as 1/10 000 year earthquake event.

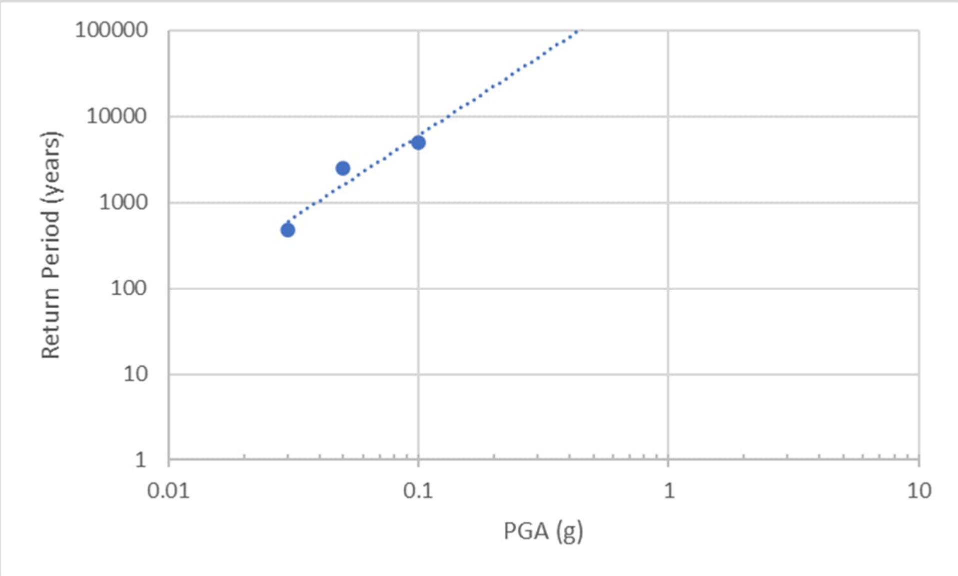


Figure 20-28 – Extrapolation of PGA for 1/10 000 Year Seismic Event

The MDE was then calculated as 0.08 g.

Seismic loading was modelled by performing pseudo static analyses for the MDE, as required by CDA guidelines. Pseudo static analyses apply a horizontal force (seismic coefficient) to the model to simulate earthquake loading. The horizontal seismic coefficients used in the seismic stability analysis were estimated using the formula developed by Melo and Sharma (2004), KH = 0.5 x PGA which resulted in a coefficient of 0.04 g for MDE loading.

Results of Analyses

The results from the stability analyses are as shown in Table 20-8.

Table 20-8 – Summary of Factor of Safety for Various Conditions

Slope Location	Drained/Undrained Condition	Case ID	Analysis Condition	Required minimum FoS	FoS from Analysis
Western Slope along Section A-A	Thickness of layers: Tailings = 0.35 m, Waste Rock = 0.85 m				
	Drained	1	Static (dry TSF)	1.5	2.3
		2	Static with R_u=0.1	1.5	2.1
		3	Static with R_u=0.2	1.5	1.9

Slope Location	Drained/Undrained Condition	Case ID	Analysis Condition	Required minimum FoS	FoS from Analysis
		4	Pseudo Static (dry TSF)	1.0	2.0
		5	Pseudo Static with R_u=0.1	1.0	1.8
		6	Pseudo Static with R_u=0.2	1.0	1.7
	Undrained	7	Pseudo Static (dry TSF)	1.5	2.0
		8	Static Peak Strength	1.2 to 1.3	2.3
		9	Static Residual Strength	1.5	2.2
	Thickness of layers: Tailings = 0.85 m, Waste Rock = 0.35 m				
	Drained	10	Static (dry TSF)	1.5	2.1
		11	Static with R_u=0.1	1.5	1.9
		12	Static with R_u=0.2	1.5	1.7
		13	Pseudo Static (dry TSF)	1.0	1.8
		14	Pseudo Static with R_u=0.1	1.0	1.7
		15	Pseudo Static with R_u=0.2	1.0	1.5
	Undrained	16	Pseudo Static (dry TSF)	1.5	1.4
		17	Static Peak Strength	1.2 to 1.3	1.5
		18	Static Residual Strength	1.5	1.2
Eastern Slope along Section A-A	Thickness of layers: Tailings = 0.35 m, Waste Rock = 0.85 m				
	Drained	19	Static (dry TSF)	1.5	2.5
		20	Static with R_u=0.1	1.5	2.4
		21	Static with R_u=0.2	1.5	2.2
		22	Pseudo Static (dry TSF)	1.0	2.2
		23	Pseudo Static with R_u=0.1	1.0	2.1
		24	Pseudo Static with R_u=0.2	1.0	1.9
	Undrained 7 above	25	Pseudo Static (dry TSF)	1.5	2.1
		26	Static Peak Strength	1.2 to 1.3	2.4
		27	Static Residual Strength	1.5	2.2

Slope Location	Drained/Undrained Condition	Case ID	Analysis Condition	Required minimum FoS	FoS from Analysis
	Thickness of layers: Tailings = 0.85 m, Waste Rock = 0.35 m				
	Drained	28	Static (dry TSF)	1.5	2.4
		29	Static with R_u=0.1	1.5	2.3
		30	Static with R_u=0.2	1.5	2.0
		31	Pseudo Static (dry TSF)	1.0	2.1
		32	Pseudo Static with R_u=0.1	1.0	2.0
		33	Pseudo Static with R_u=0.2	1.0	1.7
	Undrained	34	Pseudo Static (dry TSF)	1.5	1.5
		35	Static Peak Strength	1.2 to 1.3	1.7
		36	Static Residual Strength	1.5	1.4
Northern Slope along Section B-B	Thickness of layers: Tailings = 0.35 m, Waste Rock = 0.85 m				
	Drained	37	Static (dry TSF)	1.5	2.2
		38	Static with R_u=0.1	1.5	2.1
		39	Static with R_u=0.2	1.5	1.9
		40	Pseudo Static (dry TSF)	1.0	1.9
		41	Pseudo Static with R_u=0.1	1.0	1.8
		42	Pseudo Static with R_u=0.2	1.0	1.7
	Undrained	43	Pseudo Static (dry TSF)	1.5	1.8
		44	Static Peak Strength	1.2 to 1.3	2.1
		45	Static Residual Strength	1.5	2.0
	Thickness of layers: Tailings = 0.85 m, Waste Rock = 0.35 m				
	Drained	46	Static (dry TSF)	1.5	2.1
		47	Static with R_u=0.1	1.5	1.9
		48	Static with R_u=0.2	1.5	1.7
		49	Pseudo Static (dry TSF)	1.0	1.8
		50	Pseudo Static with R_u=0.1	1.0	1.7

Slope Location	Drained/Undrained Condition	Case ID	Analysis Condition	Required minimum FoS	FoS from Analysis
		51	Pseudo Static with R_u=0.2	1.0	1.5
	Undrained	52	Pseudo Static (dry TSF)	1.5	1.4
		53	Static Peak Strength	1.2 to 1.3	1.6
		54	Static Residual Strength	1.5	1.1

Discussion on Results

All the analyses for the co-disposal facility modelled scenario of tailings depth of 0.35 m and waste rock of 0.85 m have factors of safety greater than the minimum required.

For the scenario of tailings depth of 0.85 m and waste rock of 0.35 m, a range of FoS have been obtained which span from being less than the requirement to greater than the requirement. Overall, it can only be concluded that the factors of safety for this configuration are less than the requirement for some of the undrained cases, indicating there is a significant risk of failure even where some of the risks are mitigated at detailed design stage.

To avoid this risk, waste and tailings needs to be layered to specified depths which can be further determined using stability analysis during later design stages. The current mass balance indicates that there is sufficient waste rock to ensure that a safe configuration can easily be achieved for the years up to Year 9, when the volume of waste exceeds the volume of tailings. During these early years, at the lower levels, suitable layers for the given waste streams will need to be specified, when working in the upper sections of the open pit. This is on the basis that the full area of the facility would be used requiring that all phases be developed within the first 1 to 2 years of production.

For a tailings depth of 0.85 m and waste rock of 0.35 m, this would equate to a strip ratio of 0.53. For the more favourable scenario with a tailing material depth of 0.35 m and waste rock of 0.85 this amounts to 3.18. Table 20-3 gives the variance of strip ratio on a yearly basis with a LOM average of 3.0.

Annual volumes are also provided in the Table 20-9 below.

Table 20-9 – Annual Volumes of Tailings and Waste Rock During Operation of the Open Pit

Year	-1	1	2	3	4	5	6	7	8	9	10
Volume of Tailings (Mm^3)	0.00	2.07	2.07	2.07	2.05	1.83	2.04	2.06	2.06	2.12	2.03
Volume of waste mined (Mm^3)	0.36	5.48	8.80	7.82	6.86	6.83	6.50	4.20	2.65	1.60	1.37
Total Volume (Mm^3)	0.4	7.5	10.9	9.9	8.9	8.7	8.5	6.3	4.7	3.7	3.4

The volume of tailings starts to exceed the volume of waste rock in Year 9, whilst working in the base of the open pit; at this point of study, it is estimated that the stack will be ~56 m high. From this point onwards, additional drainage measures may need to be provided between the layers of tailings for the remaining additional height of 17 m up to approximately 73 m, as indicated in Figure 20-25. Other options that can be considered as alternatives to the installation of drainage measures would include the stockpiling of waste rock at an alternative location, possibly within the open pit depending on the open pit footprint and using this material at a later stage. It is likely that a final solution would include using both these options which allows the amount of double handling to be minimised while also maximising the overall strength of the combined materials within the facility.

The rate of rise in Year 9, 10 and 11 is 4.39 m, 4.31 m and 1.51 m respectively. Thereafter it averages at approximately 1.2 m per year for the next 9 years towards the end of the enterprise life of the mine.

Further stability analyses can be undertaken to determine the maximum depth of tailings between rock layers which could then be specified to provide further flexibility during operation of the stack once the mine plan is fully recognised.

The possibility of setting aside any available benign and non-acid generating waste rock excavated from earlier years for subsequent placing of tailings after Year 8 could also be investigated in further studies. This could provide both improved drainage and also co-disposal facility capping for dust suppression. Further to this any sands and gravels arising from the construction works should be set aside for such use.

Likewise, testing and assessment can be made to determine characteristic of the filtered tailings and waste rock as discussed in Section 20.3.17.

Section 20.3.18 provides examples of similar projects with publication of shear strength parameters.

In addition, further stabilising measures can be provided which can include:

- sizing of the width of the outer rock wedge;
- adjustment of the top shoulder berm; and
- adjustment of the top slope shoulder angle.

20.3.16. RISK

Risks during development, operation and closure of the facility were assessed at PFS level and those higher and above are referred to in the risk register Appendix 3.

20.3.17. FURTHER SITE INVESTIGATION

Table 20-10 gives an indication of the geotechnical testing required for subsequent feasibility study and detailed designs to validate assumptions made at PFS level. This does not include for geochemical testing.

Table 20-10 – Indicative Recommendations for Geotechnical Testing at Feasibility and Detail Design

Material	Material Type	Parameter	Tests	Feasibility Design	Detailed Design	Comments
In-situ Rock beneath co-disposal facility		Density & Strength (c, φ and E)	Point load	✓	✓	
			Unconfined Compressive Strength (UCS)	✓	✓	
			Core Recovery Parameters - TCR, SCR and RQD	✓	✓	
In-situ Soil beneath co-disposal facility	Peat					No tests recommended as Peat will be excavated and replaced
	Moraine/ Glacial Till	Density	Nuclear density test	✓		
		Classification	PSD	✓		
			Atterberg Limits - Liquid Limit, Plastic Limit	✓		
			Moisture Content	✓		
		Strength	Standard Penetration Test (SPT)	✓		
		Compaction (Maximum Dry Density and Optimum Moisture Content)	Proctor Test		✓	Required if embankments are planned to be built using Glacial Till

Material	Material Type	Parameter	Tests	Feasibility Design	Detailed Design	Comments
		Undrained Strength	Triaxial Test (UU test) or Uniaxial Unconfined Compression Test (cohesive soil), Field Shear Vane Test		✓	
		Drained Strength	Shear Box Test (coarse soil)		✓	
Tailings		Density, State Parameter, Strength	Static Cone Penetration Test (CPTu)			Not relevant as they are conducted on deposited tailings
		Seismic shear wave velocity	CPTu Seismic test			Not relevant as they are conducted on deposited tailings
		Shear wave velocity	Triaxial compression tests with bender elements	✓	✓	
		G/Gmax, PWP ratio, Damping ratio	Cyclic DSS Test		✓	
		Classification	Moisture Content	✓		
			Specific Gravity	✓		
			Density - dry density and bulk density	✓		
			Void ratio, porosity	✓		
			Atterberg Limits - Liquid Limit, Plastic Limit	✓		
			PSD	✓		

Material	Material Type	Parameter	Tests	Feasibility Design	Detailed Design	Comments
		Compressibility - Dry Density, Coefficient of consolidation, Coefficient of volume compressibility and Compressibility Index	Oedometer test		✓	
		Compaction (Maximum Dry Density and Optimum Moisture Content)	Proctor test		✓	
		Strength and Critical State Locus	Triaxial test (CIU and CID) or DSS		✓	
		Hydraulic conductivity	Permeability test			
		NorSand and Critical State Parameters			✓	
			CBR		✓	
Waste Rock		Classification	PSD	✓	✓	
		Density & Strength (c, φ and E)	Point load	✓	✓	
			Unconfined Compressive Strength (UCS)		✓	
			CBR		✓	
Ground water			Standpipes	✓		

20.3.18. FILTERED TAILINGS AND CO-DISPOSAL APPLICATIONS

Filtered Tailings at the Meliadine Gold Mine, Nunavut, Canada

Filtered tailings are being placed at the Meliadine Gold Mine at Nunavut, Canada. The mine is in an area of continuous permafrost with a mean air temperature of -10°C. Tailings from the process plant are filter pressed to produce a stackable filter cake at a design solids content of 85%. The filter cake is trucked to the tailings storage facility, where it is end-dumped, spread, and compacted in a sequence of layers. Figure 20-29 and Figure 20-30 show the operation of the dry stack (Goldup, 2019).

Figure 20-29 – Filtered Tailings Placement, Spreading and Compaction

Figure 20-30 – An Aerial View of Filtered Tailings

Mine Waste Case Examples of Stacked Tailings and Co-Disposal

This 2017 paper lists projects using “dry” methods for disposal of tailings. Table 20-11 gives various types of filter processes and the corresponding climate. Table 20-12 provides details of co-disposal. The list is not intended to be comprehensive, and is based on literature review, and on WSP studies where permissions were granted. Several initiatives by mineral resource companies were known to the authors at the time of writing and are not included here.

Table 20-11 – Filtered Tailings Disposal Examples

Project and Location	Implementation Date	Filter Type	Deposition Strategy	Climate
Mandalay, Guatemala	Design	Pending	Pending	Wet
Platreef, South Africa	Design	Thickener to Vacuum disc filter	Conveyors and stackers. Initial deposition will be trucks for drainage areas and compacted areas	Summer rainfall , heavy thunder-showers
Peabody Wilpinjong, NSW, Australia	2015	Pressure filter	Filter cake mixed with coal rejects and trucked, and end dumped into mined out voids	Semi-arid
OCP, Morocco	2015 2016	Pressure filter	Conveyor, stacker and reclaimer	Dry
La Coipa, Chile	1990	Thickener and Belt filter	Conveyed, stacked and dozer spread	Dry (Desert)
Cobre Las Cruces, Spain	2013, 2015, 2016	Started with Belt filter and changed to Pressure filter	Stacking by truck	Moderate precipitation, warm
Jinfeng, China	2017	Pressure filter	Trucked, tipped spread and compacted in 1 m lifts. Initially compaction was by smooth drum roll- er but subsequently changed to a tyned roller	Wet
White Mountain, China	2017	Pressure filter	Delivered to the TSF by conveyor. The tailings are deposited over the TSF surface via a string of secondary conveyors and spread using high mobility excavators. The tailings do not dry back much and there is no further compaction	Cold
Green's Creek, USA	Early 2000's	Pressure filter	Cake drops into a concrete vault and a loader loads trucks. The flow is split here into the backfill feed or the	Wet and cold

Project and Location	Implementation Date	Filter Type	Deposition Strategy	Climate
			surface disposal feed. The surface disposal feed is trucked, spread into cells and roller compacted	
Tambomayo, Peru	2017	Thickener and Pressure filter	Trucked and dried for 7-10 days, deposited and compacted	Dry
Confidential client, Mexico	Design	Thickener and Pressure filter	Conveyed, stacked and compacted	Dry
Cerro Lindo, Peru	2010	Belt filter	Trucked, windrowed for 2 days then spread and compacted with vibratory rollers	Dry
Raglan, Canada	2000-2001	Thickener to Pressure filter	Trucked to the TSF and dozed and compacted	Moderate precipitation, cold
El Sauzal, Mexico	Closure (operated for ~10 years)	Thickener to Pressure filter	Tailings conveyed to drying area and spread with a dozer. Once dry they are pushed over the edge of the drying embankment and loaded into trucks. Truck trans- ported to dry stack area and compacted. Separated into structural zone and non-structured zone (50 cm lifts vs 5 m lifts)	Dry
Molycorp, USA	2014	Pressure filter	Blended bentonite and tailings paste or non-blended tailings filter cake. System is split into surface disposal and backfill.	Arid
Pogo, USA	2006	Pressure filter	Cake is trucked and compacted in lifts in the shell of the TSF area for structural stability; Rest is dumped inside the TSF shell. Filtered tailings are placed in 30 cm lifts on surface.	Cold and wet, Arctic with permafrost
Confidential Client, USA	2014	Pressure filter	Overland conveyor to load out and then dumped and spread in TSF in cells. Lifts in outer shell were 30 cm lifts and interior were 1 m	Cold (full four seasons)
Kupol (Kinross), Russia	2017	Pressure filter	Truck haul, dozing and compaction.	Arctic, extensive permafrost

Table 20-12 – Co-disposal Examples

Project / Location	Description
Implemented	
Jeebropilly Colliery, Gordonstone Colliery, Burton, Charbon, Coppabella, Cumnock, Hail Creek, Kestrel, Moorevale, Moranbah North, North Goonyella, Stratford,	Coarse and fine coal rejects are mixed and pumped to an impoundment
Argyle Diamond Mine, Australia	Coarse and fine waste products are transported separately and mixed together at the disposal site (slimes and tailings)
Mt. Thorley, Australia	Dewatered tailings are added to coarse rejects on a conveyor, then transported in trucks for dumping at the same time as mining spoil
Daggafontein, South Africa	Tailings and waste rock are mixed and used as a closure cover on a tailings storage facility
Greens Creek Mine, USA	Filtered tailings and mine rock end are dumped at the waste site at approximately a 1:1 ratio, and then compacted with a vibratory roller
Agua Blanca, Spain	Thickened tailings are discharged directly onto layers of waste rock within a lined tailings impoundment
Dunka Mine, USA Kidston Gold Mine, Australia	Waste rock deposited in a tailings impoundment on mine closure Waste rock and thickened tailings were placed in an open pit from opposite sides of the pit rim
Tarong Coal Mine, Australia	A void was filled with rejects and pumped tailings
Neves Corvo Mine, Portugal	Mine rock used to construct storage cells for tailings paste over an existing conventional tailings impoundment
Illawarra Coalfields, Australia	Cells constructed in a waste rock dump filled with tailings
Oak Mine, South Africa	Waste rock used to build paddock type cells into which tailings are disposed
Snap Lake Mine, Canada	Processed kimberlite is placed in unlined storage cells composed of grits and waste rock
Proposed or Considered	
NICO Project, Canada	Thickened tailings and mine rock to be placed in alternating 5 m thick layers with perimeter embankments for containment
Esquel Gold Mine, Argentina	Tailings to be disposed with waste rock or leach ore
Nunavik Nickel Mine, Canada	Thickened tailings to be deposited in lined waste rock containment cells
Shakespeare, Canada Krumovgrad Gold, Bulgaria	Thickened tailings to be placed in waste rock cells

Project / Location	Description
Esquel Gold Mine, Argentina	Paste tailings to be placed in cells constructed from mine rock
Sites with Published Trials	
Brukunga Remediation Project, Australia	Tailings and mine rock mixed with limestone
Ulan Coal Mines, Australia Douglas Colliery,	Coarse rejects pushed onto wet coal tailings and mechanically mixed
South Africa	Coal tailings slurry poured over 0.3 m thick layers of coarse rejects
Cerro De Maimon, Dominican Republic	Layered co-disposal of mine rock placed over desiccated thickened tailings in 1.5 m to 2.0 m lifts
Copper Cliff, Canada	Tailings, slag and waste rock were mixed to form PasteRock and then placed in lined test cells
Porgera Gold Mine, Papua New Guinea	Fully mixed tailings and waste rock placed in 6 m high columns

Filtered Tailings Plant Design at the Ada Tepe Mine

In 2021, a feasibility study for producing filtered tailings within a limited footprint was undertaken at the Ada Tep Gold Mine, Krumovgrad, which is an example of a permitted IMWF at a European gold mine and evidence of regulatory and basis of design suitability for Ikkari. This was to improve the operability of the IMWF, construction of which had commenced some two years earlier. This consisted of assessing the area required for the filtration plant, overall design efficacy, considering the filterability of the tailings, transportation of the tailings, examining the filtered tailings storage requirements and modifying the integrated mine waste facility deposition strategy to suit filtered tailings instead of thickened tailings.

The study considers the general benefits of filtered tailings as well as some of the challenges in implementing such a system. In addition to the key design considerations for the filtration plant, outlining the specific operational benefits and identifying recoverable costs for this site are discussed. (Diaz, et.al., 2023).

20.4 SITE MONITORING

Environmental baseline data collection at Ikkari began 2017 with water sampling of main streams and rivers. The sampling programme has been broadened and comprises currently 37 surface water locations both upstream and downstream of the Ikkari area. A wide range of water analyses (total of 63 parameters) are carried out 6 times per year on all surface water samples. Groundwater monitoring includes sampling of 28 shallow wells, three deep drill hole sampling and five springs four times a year. In addition, deep groundwater sampling has been undertaken in two winter campaigns as part of the hydrogeological study. Three piezometric clusters were installed at or close to the deposit in spring 2023 with three standpipes each, one in peat, one in till and one in bedrock. A total of 22 level loggers measure groundwater head, of which 8 in deep drill holes and the rest in shallow (<50 meter) holes.

Continuous environmental monitoring stations have been installed in the area:

- 1 to measure flow and 12 to measure flow, temperature, turbidity and electrical conductivity in water courses above and below the Ikkari project area and adjacent streams;
- Seven to measure pore water pressure in drill holes in and around the Ikkari deposit;
- A weather station measuring temperature, wind direction and speed, air moisture, air pressure, rainfall and solar radiation next to the deposit on hill Iso-Pulkittama; and
- Snow depth is measured manually on a daily basis in the winter.
- Monitoring of dust: since November 2022, it currently includes 20 collectors, analysed once a month. A total of 29 parameters are analysed monthly during the first year, after the first year metals analyses are conducted quarterly. Snow samples have been collected near the deposition collectors during two winters, which are analysed with 10 additional parameters.
- Monitoring of bioindicators in the vicinity of Ikkari began in 2024. It is planned to continue the baseline monitoring of the area in 2025, and then again during the construction phase. The bioindicator monitoring includes the observation of lichens, mosses, ants, berries, and mushrooms, and is conducted at 23 different points.
- Monitoring of wells used for drinking water is included in groundwater monitoring. This has been carried out for two years, twice a year, during the spring flood and the dry season. Drinking water monitoring includes 7 wells, from which the same parameters as in groundwater sampling are analysed, and in addition, the total concentrations of elements for which there are quality recommendations and requirements defined by the drinking water regulation are determined.

Samples are collected and are sent for analysis by Eurofins Ltd. The results are submitted to the open environmental information system maintained by SYKE.

20.5 MINE WATER MANAGEMENT

The following sections describe the proposed water management infrastructure at the site. These proposals will be further developed in following stages of design.

20.5.1. WATERCOURSE DIVERSIONS

The planned Ikkari mining and mineral processing project is to be located within the Saittajoki River valley. The mine site is located in the upper reaches of the Saittajoki River, which has a catchment area of 30.6 km^2. A minor tributary to the Saittajoki River passes over the Ikkari ore body where it is proposed to develop an open pit.

To facilitate the mining operation and minimise the risk of polluting the Saittajoki River, it is proposed to divert both the tributary and the main river channel around the mine site. This decision has been informed by an initial baseline survey of the watercourses.

The Saittajoki River is proposed to be diverted over 2.5 km to the north of the mine site into the adjacent Heinalamminoja Stream, which itself joins the Saittajoki River downstream of the mine site (typical Saittajoki River channel is presented in Figure 20-31, while diversion alignment is presented in Figure 20-32).

Figure 20-31 – Typical Saittajoki River channel

The proposed river diversion removes the risk of the Saittajoki River becoming polluted if routed through the mine site. An appropriately designed diversion will also allow a natural river channel to develop in perpetuity, as opposed to the requirement of a heavily modified channel passing through the mine site. The main risk associated with the proposed diversion is associated with the permitting of a major river diversion, the existing river of which is classified and monitored under the European Union Water Framework Directive. Diverting the Saittajoki will markedly reduce the flow in around 6 km of the existing channel upstream of the confluence with the Heinälamminoja river, permanently impacting associated habitats and sediment transport. While diverting this flow into the Heinälamminoja river may itself provide enhanced wetland habitat. A detailed habitat and geomorphological surveys will be required to inform an assessment of the environmental impacts of the proposed diversion.

To provide initial quantities for including in the PFS capital cost estimate the diversion is taken to be an excavated channel. This will provide a conservative estimate of the quantities as it is proposed to use a 'Stage Zero' approach in which much of the river channel will be allowed to form its own channel requiring minimal intervention. To prevent high flows on the Saittajoki flooding the downstream mine site it is proposed to construct an earth embankment across the Saittajoki flood plain downstream of the channel diversion. This will be founded below the peat layer, with the crest a nominal 3 m above the lowest existing ground level. A similar 1.5 m high embankment is also proposed to separate the pit from the diverted tributary. Depending on the results of further ground water and seepage modelling seepage cut offs may be required beneath the embankment foundations.

The diverted river channels have been sized to match the existing, with a check that they can carry the 1:100 annual exceedance probability catchment runoff considering the channel gradient. These flows are estimated as follows:

- Diverted tributary 1:100 AEP flow: 1.9 m^3/s; and
- Inlet to Saittajoki River diversion 1:100 AEP flow: 7.1 m^3/s.

Where the tributary diversion passes through high ground to the west of the pit there will be a need for a 5 m deep cut over a length of 400 m. Moving the diversion further west is not an option due to the presence of a known gold deposit under exploration.

At the next stage of the mine development, the river diversion design will be developed further to maximise the extent of natural channel. This will be informed by a detailed geomorphological survey of the existing Saittajoki River and the Heinälamminoja Stream, topographic survey information and hydraulic modelling.

20.5.2. SURFACE WATER MANAGEMENT PLAN

The surface water management catchments and infrastructure are schematised in Figure 20-32 below. To minimize the amount of surface water runoff that needs treatment it is necessary to separate non-contact and contact water on the mine. Where contact water has come into contact with contaminants, and so is referred to as 'contact' water, or is carrying inert sediment from disturbed areas, referred to as 'sediment control areas'. Non-contact water is runoff from undisturbed natural catchments, including discharge from the external pit dewatering boreholes. In this regard, where the terrain allows, runoff from non-contact catchments will be diverted in open channels around contact catchments, while that from contact and sediment control catchments will be captured and treated or used as process water.

All mine contact water, which includes runoff and seepage from the co-disposal facility, temporary waste and ore rock storage facilities, and mine dewatering flows will be collected in lined ponds and either treated for discharge to the environment or used in the site operational process.

The Process Plant buildings and surrounding areas are taken to be sediment control catchments, as tailings and ore bearing rocks are assumed not to be present externally to these buildings. At this stage it is proposed to direct the runoff from these areas to sediment traps before discharge to the environment. As a result, these areas are referred to as 'sediment control' catchments in the key on Figure 20-32. On further development of the mine infrastructure layout this assumption will need to be reviewed. A view will need to be taken on whether to treat the haul roads as contact catchments (rather than sediment control areas) depending on the risk and quantity of ore and tailings that could spill while being transported. As such, for the PFS level study haul roads are treated as 'sediment control' catchments and not 'contact water' catchments.

20.5.3. SNOWFALL MANAGEMENT

The average temperature at the site during the winter months from November to April ranges from -5°C to -20°C. For the purpose of this water management plan, during this period all precipitation is taken to be snowfall and will not produce runoff until the spring thaw during April and May.

Snow will need to be cleared from areas such as the Co-disposal Facility, Open Pit, Run of Mine Pad (ROM Pad) and roads etc. to allow mine operations to progress unhindered. To minimise pollution risk, snow falling on contact catchments will need to be stockpiled within a contact catchment such that the resulting melt water can be managed appropriately.

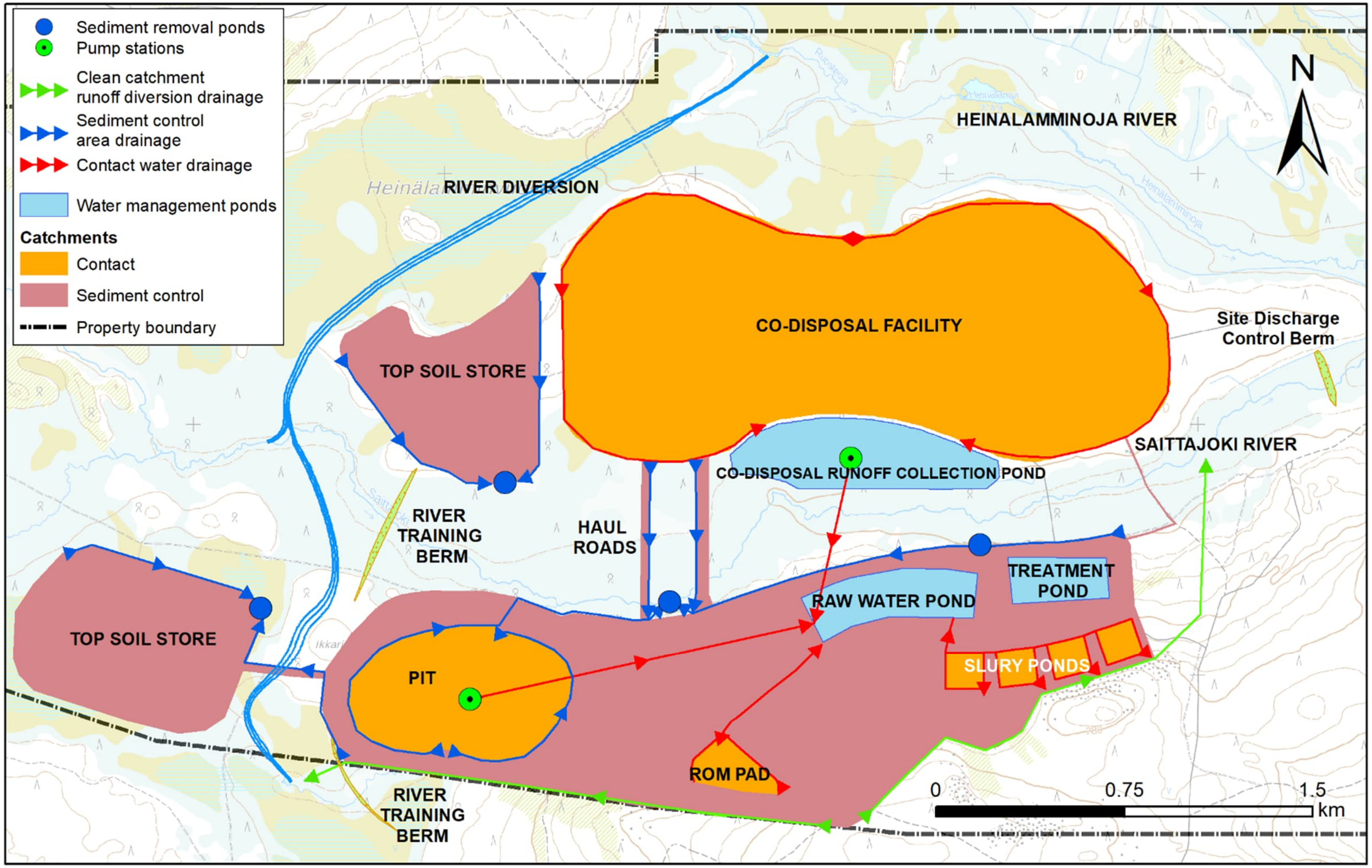


Figure 20-32 – Surface Water Management Infrastructure

20.5.4. SURFACE WATER DRAINAGE

Figure 20-32 shows the preliminary surface water drainage infrastructure at the site. This includes drains to divert non-contact water runoff around the mine site, drains to capture contact water, sediment traps and water treatment ponds. The individual items are described in the following sections.

Surface water drainage from non-contact and sediment control catchments

The channels labelled 'diversion drainage' and 'sediment control area drainage', in Figure 20-32 above, will either divert non-contact runoff water around the mine, or direct runoff from sediment control areas to the Saittajoki river via sediment traps. This will minimize the amount of surface water runoff that needs to be treated. In the case of runoff from the 'sediment control' areas, this will need to pass through sediment traps before being discharged to limit suspended solids entering the Saittajoki river. Where there is a risk of oil or fuel spills, such as in carparks, runoff would need to pass through oil interceptors before discharge to the environment. Local bunding around fuel tanks, generators and chemical storage will be required to capture any spills.

The non-contact drains are anticipated to be nominally sized gravel lined open channels a minimum of 1 m deep, with 1 m base widths and 1:2 side slopes. Additional scour protection or drop-down structures would be required where flow velocities are high due to steep channel gradients. HDPE corrugated culverts would be required where the drainage passes under access roads.

Sediment traps would be formed from an earth embankment with a weir allowing adequate residence time for water in the pond for an agreed fraction of suspended solids removal. Access to the sediment traps for equipment will need to be provided for the periodic removal of settled sediment.

The final sizing of drainage channels and sediment traps will be undertaken under the next phase of the project.

Contact surface water drainage

The channels labelled 'contact water drainage' in Figure 20-32 above will carry runoff from contact catchments to either the co-disposal Runoff Collection Pond or Raw Water Pond for treatment or use as process water for the mine. This drainage will be similar to the non-contact water drainage described above but will need to be lined with either concrete or a protected HDPE liner. Concrete channel linings are preferable as they are less prone to damage, especially if drains need to be cleared of sediment or ice. Joints between concrete panels will need to include water bars to limit seepage.

Co-disposal facility water management

The co-disposal facility will be one of the highest risk sources of contamination at the site. The co-disposal Facility will be used to dispose of WOL tailings and waste rock, having a basal liner to prevent seepage to groundwater.

The natural catchment to the north of the co-disposal facility drains away from the facility to the north. Runoff from the co-disposal facility will be collected and conveyed by gravity along perimeter channels and into the co-disposal Runoff Collection Pond (co-disposal Pond) on the southern side of the facility. Both the perimeter channels and pond need will need to be lined.

Contact Water management ponds

Contact water management ponds are required to balance peaks in flows during wet periods. There are two main contact water ponds proposed, the co-disposal Collection Pond and the Raw Water Pond, as shown in Figure 20-32 above.

The storage capacity of the two contact water ponds has been sized with the same continuous daily rainfall runoff models used to establish the average monthly wet and dry surface water contributions to the water balance. The contact water management system is schematised in the water balance block flow diagram set out under Section 20.5.6. This was simulated by connecting the runoff models to storage units and pumps to represent the management of water in the ponds. The main purpose of the ponds is to provide surge capacity for changes to inflows above the nominal operating flows. The ponds are modelled on the assumption that where the surge capacity starts to fill it is to be pumped down at the maximum pump rate until the water level returns to the nominal operating level. This ensures that there is adequate surge capacity available to manage surges in raw water contributions due to storms or periods of snow melt. To achieve the stated probability of spilling the pond surge capacity cannot be used for the long-term storage of water.

The risk of an un-scheduled shut down of the water treatment plant (WTP 1) was included in the model as a probability of the Raw Water Pond pump not operating. The following failure risks were agreed with the water treatment plant designers:

- 0.2% probability of failure on any given day (equivalent to a 1:2 annual exceedance probability failure risk); and
- 80% probability of remaining shut down if the treatment plant was shut down the previous day (resulting in an average shut down period of 5 days).

The raw water management model was run over a historic 60-year simulated rainfall / snowmelt time series, giving 60 years of annual maximum storage volumes in the ponds. This process was repeated for different pump rates from the ponds. The resulting annual maximum storage volumes were then statistically analysed to give an estimate of the storage volume associated with a given annual exceedance probability of spilling to the environment.

Due to the nature of hydrological inputs, it is not possible to size a pond that has a zero risk of spilling. It is also necessary to limit the peak rate at which water is removed from the ponds to that which can be handled by a reasonably sized water treatment plant. It was therefore necessary to compromise on spill frequency to keep the size of the ponds below a practical limit, considering site constraints. The optimum combination of overall raw water treatment rate and pond size resulted in a 1:200 annual exceedance probability spill risk from the Raw Water Pond. This is equivalent to a 1:11 probability of one or more spills occurring over a 20-year life of mine.

It should be noted that there would be significant dilution of contaminants during such a rare spill event due to the large proportion of hydrological water in the system. At the next project stage, a detailed stochastic water balance and contaminant mass balance will be required to confirm acceptable spill frequencies and pond sizes for permitting.

The resulting parameters of the raw water ponds are set out in Table 20-13.

Table 20-13 – Parameters of the Contact Water Ponds

Contact Water Ponds	Storage Capacity (m^3)	Peak Pump Rate from Pond (m^3/h)	Estimated Spill Frequency (1:X AEP)
Co-Disposal Runoff Collection Pond	440 000	200	1:1 000
Raw Water Pond	600 000	900	1:200

The raw water ponds will need to be lined to prevent seepage into the ground water and include a spillway for the safe release of excess water. As the ponds will receive direct runoff from the co-disposal facility and ROM Pad, there will be a need for sediment removal at the inlet to the ponds to prevent loss of storage capacity. This can be achieved with a separate pond having adequate residence time to remove the majority of the sediment. These will need to be sized in the next development project study stage but are expected to be around 20 m by 10 m in area, and 2 m deep. It will be necessary to remove sediment from these ponds from time to time, and so they would need to be concrete lined for robustness and incorporate a bypass flow channel.

The Treated Water Pond has also been included to provide some residence time between the treatment plant and the inlet to the discharge pipeline. This will allow time to take process water quality samples before the treated water is discharged. The Treated Water Pond has been sized with a total volume of 290 000 m^3 to provide a minimum of 14 days of treated water storage if the discharge pipeline is shut down or/and if the treated water quality is out of specification. This pond will be lined and include a spillway to safely discharge excess water.

Provision has been made for a berm across the Saittajoki River at the point it crosses the downstream site boundary. Inclusion of a flow control device in this berm will allow the temporary retention of runoff from the mine site in the event it is found to be contaminated. The need for this berm and the nature of the flow control device will be explored further in the next project phase.

20.5.5. OPEN PIT AND UNDERGROUND MINE WATER MANAGEMENT

The ground water table will need to be drawn down in advance of the open pit development to manage seepage inflows. This will be achieved by a system of external open pit dewatering boreholes described in more detail under Section 20.5.6 on the Water Balance. Based on groundwater quality results to date it has been assumed for the PFS that the groundwater quality from the peripheral pit dewatering boreholes will meet environmental discharge permit limits and so can be discharged directly to the Saittajoki River. In the event that any of the borehole discharges do not meet permit limits the water will be diverted to two 20 000 m^3 lined ponds. These ponds will give up to 2 days of storage during the highest expected flow rate from the de-watering boreholes of 688 m^3/h, expected to occur during the initial dewatering period (Water Balance Year 1). Longer term management of contaminated ground water would need additional water treatment capacity, which has not been accounted for in the PFS. Further work is required to understand whether ground water can be discharged to the environment.

The maximum de-watering borehole discharge rate is also considered to be insignificant when considering its impact on high flows in the river. The exact location of the discharge point will need to be determined in the next phase of the project and could be either the section of the Saittajoki River channel passing through the mine site, or the upstream river diversion channel. This decision will be informed by a better understanding of the impact of the mine ground water cone of depression on the base flow in the Saittajoki River diversion, and the environmental benefits to the section of the Saittajoki River passing through the mine site of a consistent base flow.

Perimeter drains will be included around the open pit to prevent runoff to the pit from the external catchments. Water captured in the perimeter drains will be considered non-contact water and will be directed to the diverted tributary on the south side of the pit, and to the non-contact mine site drainage on the north side of the pit.

During the winter months in-pit sump pumping is expected to be low, due to precipitation being mainly in the form of snow, which will need to be managed in-pit until it thaws. Groundwater pit inflow is also expected to be significantly reduced due to dewatering by perimeter dewatering wells (Piteau, 2024). Pit pumping capacity must, therefore, be sized for the expected thaw volume peaks during spring, summer rainfall and any groundwater inflows which are not managed through the peripheral pit dewatering well system (Piteau, 2024). Open pit dewatering via the peripheral pit dewatering wells is expected to continue beyond the life of the open pit, into the remaining operational period of the underground mine.

The internal pit water management will include bench drains, draining to an in-pit sump. From the sump, pumps will discharge contact water to the Raw Water Pond. The in-pit dewatering may, as required, consist of multiple pumping stages to reach the pit crest. The peak pump rate from the pit sump will need to manage peak runoff rates during the spring thaw, along with sny minor ground water inflows, such that flooding of the base of the pit is kept below a reasonable frequency. A peak pump rate of 810 m^3/h has been assumed at this stage, capable of managing a 1:20 annual exceedance probability runoff event.

The management of groundwater from the underground mine essentially comprises a system of interconnected underground mining stopes and development tunnels graded to drain towards settling ponds, sumps and pumps located in the deepest part of the mine. These collect, settle and then pump the groundwater inflow together with collected service water towards the surface via a system of pipes, supported by number of intermediate pumping stations.

20.5.6. WATER BALANCE

A high-level water balance diagram has been developed at four snapshots during the life of mine (Years 1, 8, 10 and 20), with the resulting diagrams presented in Figure 20-33 to Figure 20-36 below. The ground water and surface water inputs to the water balance are discussed in detail below. The water balance has been developed in conjunction with the process and water treatment teams, for which a description of the associated water management is also included in Section 20.5.7. The water balance is based on historic site climate record and does not account for future climate change impacts. Climate change impacts should be explored in later project stages.

The water balance suggests that the overall mine is water positive requiring excess treated water to be discharged to the environment. For the PFS, the excess treated water is to be discharged in the River Kitinen upstream of the Kelukoski dam. Alternative locations are being appraised through the ongoing Environmental Impact Assessment.

Groundwater

The variability in groundwater inflows reporting to the pit sump has been presented in terms of minimum, average and maximum flows for each pit stage to align with the surface water inputs, although seasonal fluctuations in this inflow is not expected to be significant due to the peripheral dewatering well actively intercepting seasonal groundwater infiltration.

The estimated groundwater inputs to the water balance have been estimated from the (Piteau, 2024) numerical groundwater model for the following:

- The perimeter pit dewatering wells;
- The pit wall and floor inflows; and
- The underground mine ingress.

It is expected that 16 peripheral pit dewatering wells will be required, ranging in depth by approximately 100 to 250 m. Peripheral pit dewatering will, however, require higher dewatering efforts from these wells than predicted for passive pit inflow as this will create a greater dewatering footprint. This requires a significant initial dewatering rate of up to 16 500 m^3/day, which would effectively stop most, if not all the passive groundwater inflow into the pit. The peripheral dewatering well pumping rate is expected to decrease significantly to approximately 8 300 m^3/day by the end of the life of mine. Early (in time) installation of the peripheral pit dewatering well system would be a good management decision to manage the high initial pit dewatering volumes that may otherwise be required to keep the water level below the pit floor as the pit deepens.

The underground mine groundwater inflows will be assisted at higher elevations by the continued operation of the peripheral pit dewatering wells, possibly supplemented by deeper replacement dewatering wells, where needed or feasible. Underground groundwater inflows will be collected passively by a system of collection drains in development drives and crosscuts, draining towards a collection sump and pump system at the deepest part of the mine.

The groundwater will be pumped to surface via a system of pipes assisted by intermediate pump stations. Further consideration should be given to separating better quality groundwater, in shallower sections of the mine, from deeper groundwater if this is found feasible from a treatment cost perspective.

The overall groundwater inflow in the underground mine is expected to peak in Year 10 of the life of mine, three years after the underground development commences, at approximately 7 800 m^3/day, and then decrease to approximately 6 300 m^3/day by the end of life of mine.

Surface Water

The surface water inflows originating from precipitation have been presented in terms of:

- Average dry month, corresponding to February;
- Average monthly value for whole year; and
- Average Wet month, corresponding to May.

For surface water inputs to the water balance, monthly runoff estimates from the site catchments were estimated using an adapted United States Soil Conservation Service (SCS) rainfall runoff model and a combined daily soil water balance based on FAO Irrigation and Drainage Paper, No.56, Crop Evapotranspiration.

Simulated daily precipitation, snow melt and reference evapotranspiration values for the site were taken from the Finnish Environment Institute (SYKE) Watershed Simulation and Forecasting System (WSFS) model covering the period 1962 to 2022. The simulated data is shown to be a good match to gauged precipitation data. For the purposes of estimating runoff, a combined rainfall and snowmelt time series was applied to the daily soil water balance model, thereby treating snowmelt as rainfall.

The runoff model was only applied to the mine catchments defined as 'contact' areas, from which runoff would either be used in the process or treated, and as a result contribute to the mine water balance. The contact area catchments are defined in Figure 20-32 as the pit, Run of Mine (ROM) Pad, co-disposal facility and contact water management ponds. The runoff model parameters used for these catchments are set out in Table 20-14 below.

When assessing catchment runoff from working areas (such as the pit, ROM pad and top surface of co-disposal facility) it is assumed that any accumulating snow is cleared and stockpiled until the spring thaw and that surface water drainage channels are mostly frozen during winter. This assumption will result in conservative runoff values for spring months when the greatest contribution to the water balance is expected. A more detailed model of the management of snow in the working areas will need to be developed for the next project stage.

It has been assumed that all water management ponds on the mine will freeze over during the winter on which a snowpack will develop. Operational water will however continue to circulate within the ponds beneath the ice cover. This ice and snowpack are taken to melt in the spring thaw and contribute directly to the volume of water in the ponds. As a result, the snowmelt and rainfall time series is used to estimate the hydrological inputs to the ponds. Evaporation from the ponds has been estimated using the simulated evaporation time series from the WSFS model. For the purposes of this PFS study the simulated evaporation values are taken to be equivalent to open water evaporation – for which any discrepancy will not be significant due to the low evaporation of the local climate.

Table 20-14 – Contact Surface Runoff Characteristics

Parameter	Pit & Rom Pad	Co-disposal	Ponds
Type of surface	Compacted soil (clay)	Tailings (silt)	Assessed as direct rainfall and snowmelt
Curve Number	90	90	-
Topsoil depth subject to evaporation (mm)	300	300	
Vegetation rooting depth (mm)	No vegetation	No vegetation	-
Field capacity (m^3/m^3)	0.4	0.3	-
Wilting point (m^3/m^3)	0.12	0.17	-

Table 20-15 shows the resulting simulated monthly average surface water contributions from the contact catchments, including direct precipitation on the water management ponds. The overall site water balance has been developed for four separate stages of mining development. However, the surface water inputs have been estimated based on the final mining development extent.

The runoff model also provides an estimate of the seepage from the co-disposal facility. This is estimated to be an annual average of 409 000 m^3. However, due to freezing conditions during winter this seepage is only expected to report to the co-disposal pond over a period of 7 months, from April to October. This gives an average seepage rate of 88 m^3/h.

Table 20-15 – Monthly Average Snowmelt / Rainfall Contributions to Water Balance

Month	Monthly Average Runoff (m^3)				Monthly Average Rainfall / Snowmelt less Evaporation (m^3)	Total
	ROM Pad	Pit	WTP Slurry Pond	Co-disposal	Water Storage Ponds	
January	9	61	11	250	495	826
February (Dry Month)	**9**	**64**	**13**	**264**	**594**	**944**
March	36	242	46	992	2 161	3 477
April	1 387	9 404	560	38 771	25 122	75 243
May (Wet Month)	**5 550**	**37 644**	**1 538**	**154 069**	**59 994**	**258 794**
June	861	5 842	585	21 550	5 379	34 217
July	853	5 783	673	22 026	10 477	39 812
August	801	5 432	578	21 444	14 173	42 428
September	831	5 638	486	22 712	17 414	47 083
October	643	4 360	339	17 602	14 479	37 423
November	298	2 023	147	8 380	6 800	17 648
December	39	264	35	1 085	1 639	3 063
Annual Average	**11 317**	**76 759**	**5 011**	**309 145**	**158 726**	**560 957**
Monthly Average	**943**	**6 397**	**418**	**25 762**	**13 227**	**46 746**

Welfare water services

A domestic water supply will be generated on-site by treating water from the external pit dewatering borehole. Potable water treatment facilities will be provided to ensure groundwater source is suitable for consumption.

A sewage treatment plant will be located adjacent to the plant area.

Process Water

Process water is recovered from the pre-leach thickener located at the processing plant and the leach tailings thickener located at the filtration plant. While a good proportion of the process water is recycled within the process plant, a portion is sent to a clarifier for use by the tailings filters and another portion is sent to the process water treatment plant. Treated water is used as the source of make-up water to meet process water demand. Treated water is also used as gland water and for reagents preparation.

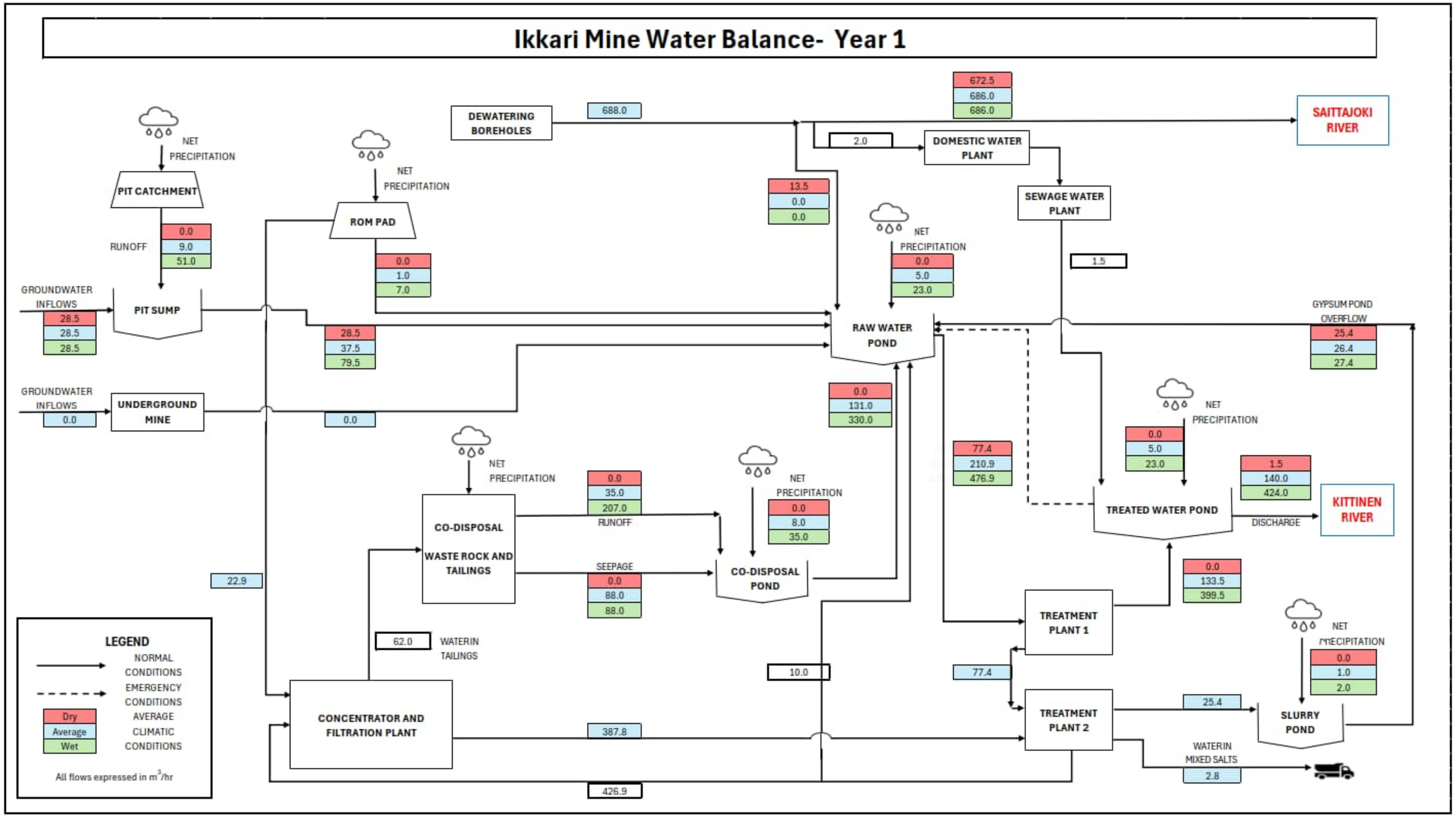


Figure 20-33 – Mine Water Balance - Year 1

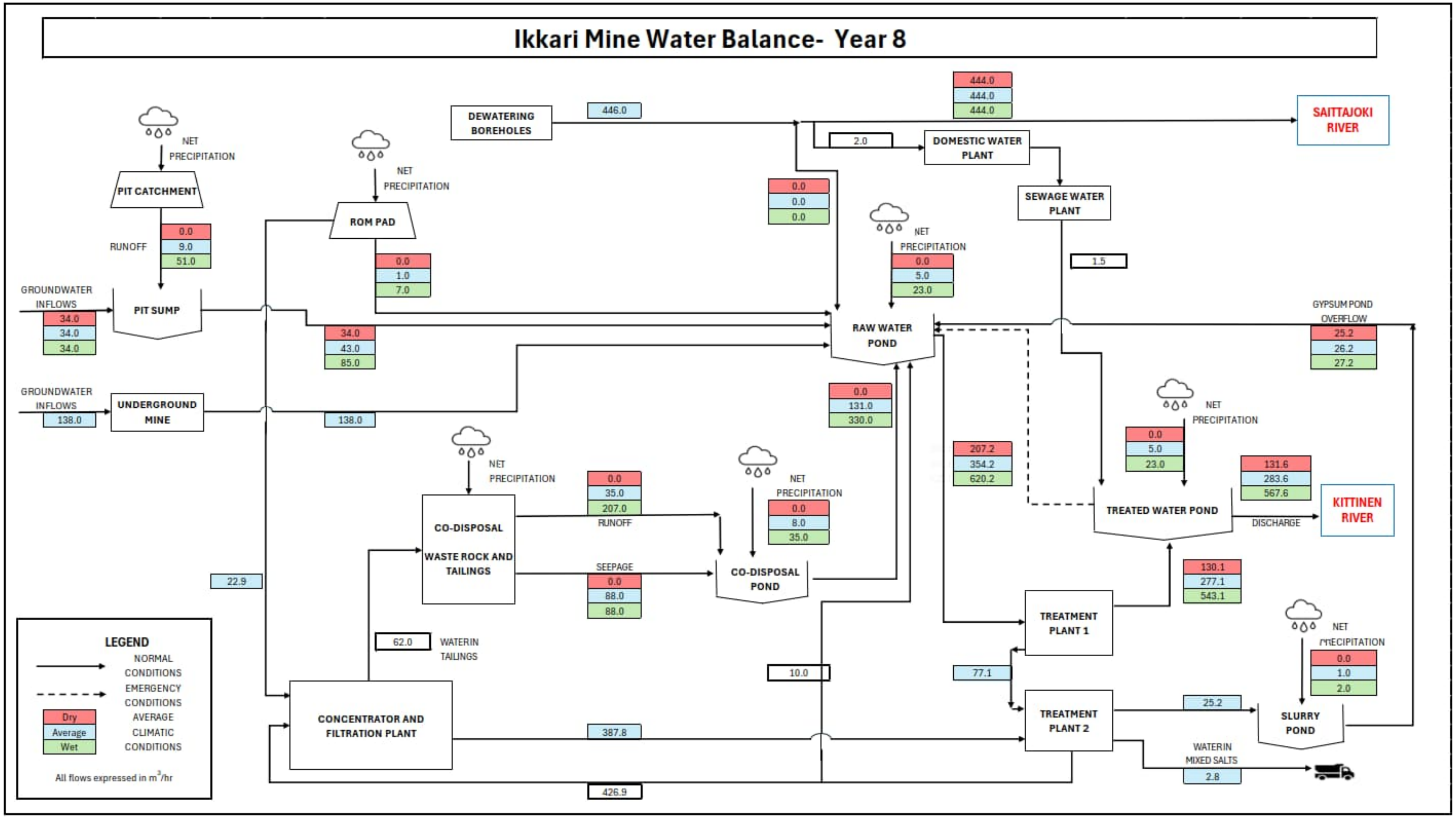


Figure 20-34 – Mine Water Balance - Year 8

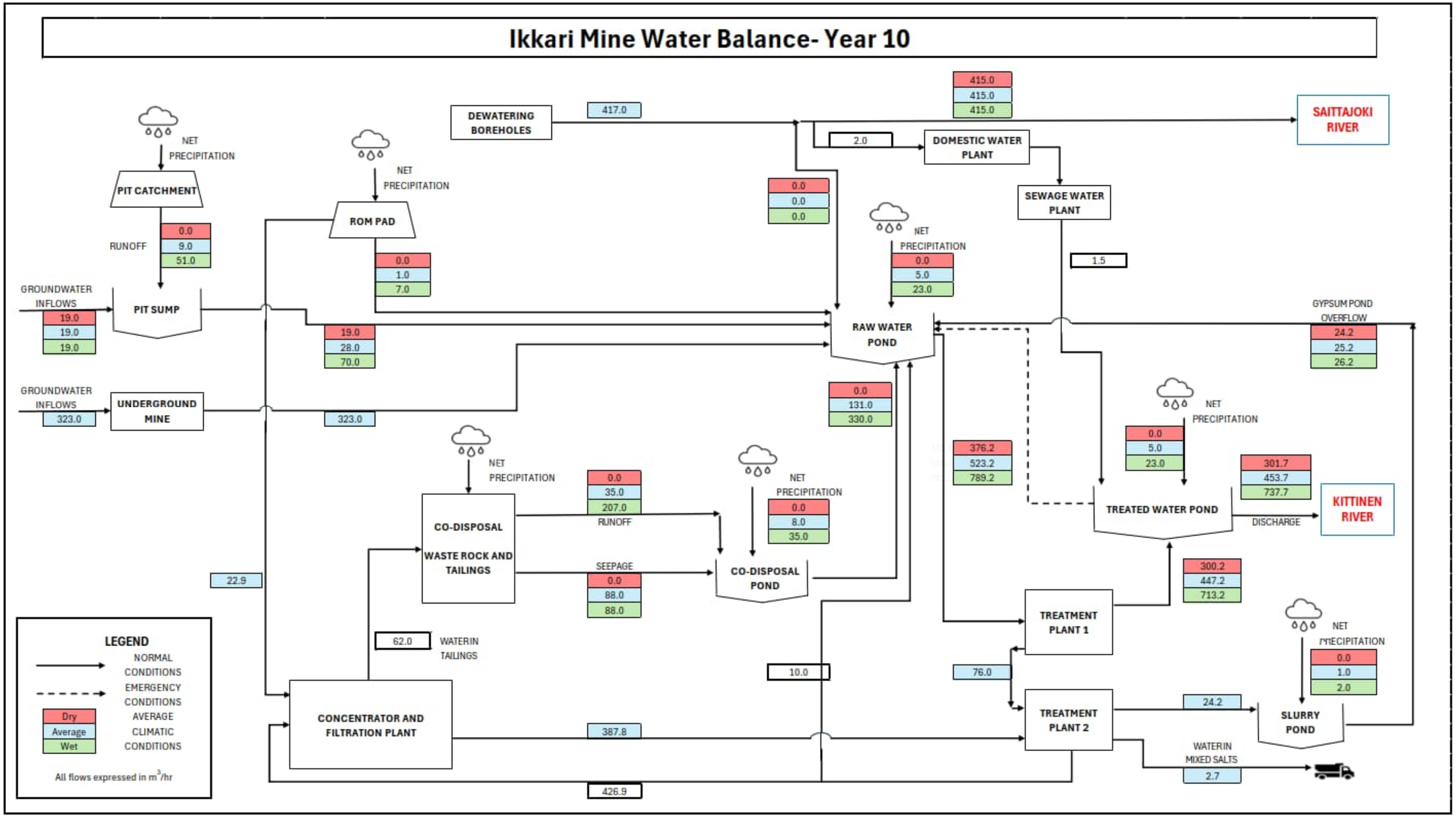


Figure 20-35 – Mine Water Balance - Year 10

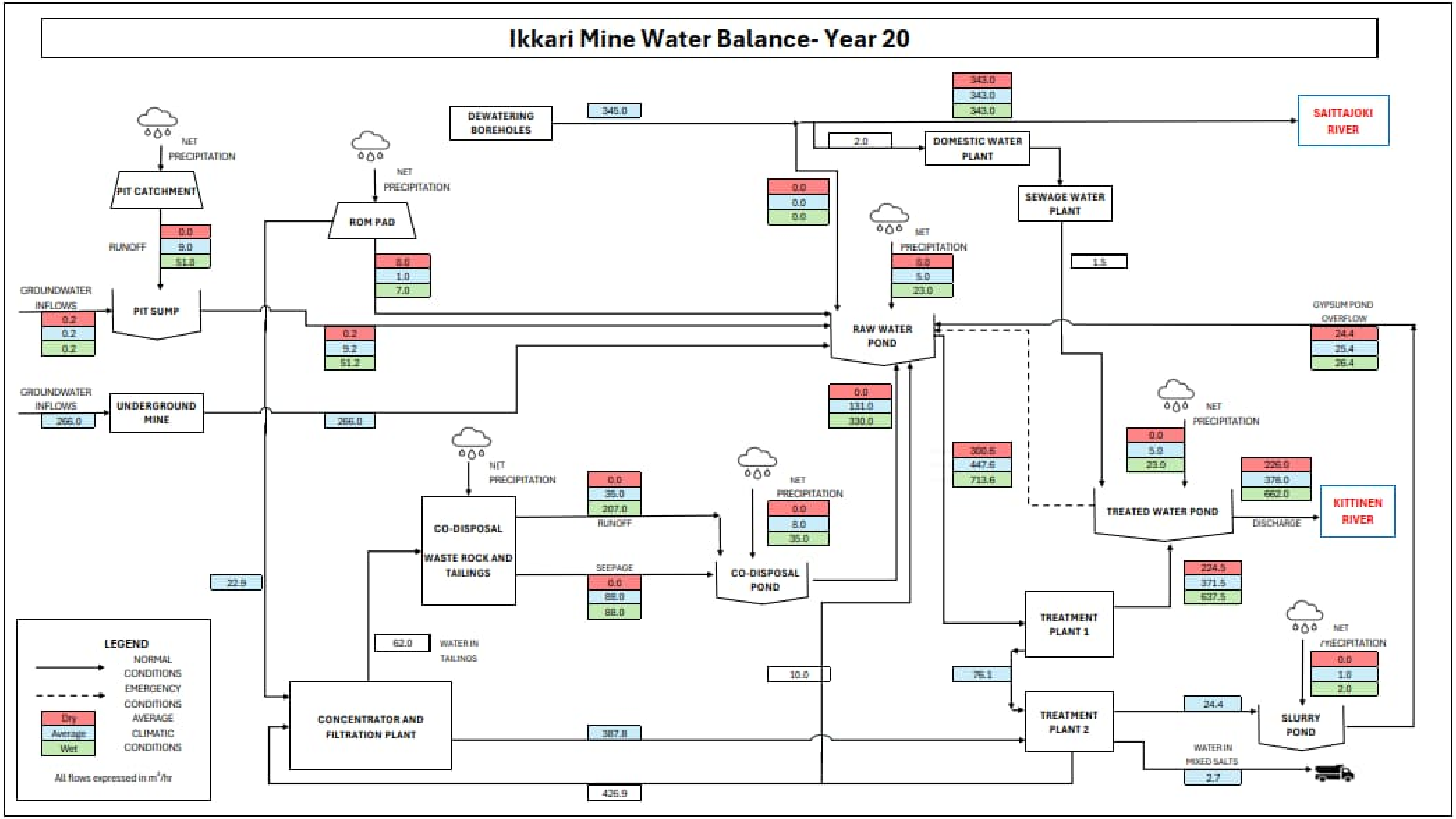


Figure 20-36 – Mine Water Balance - Year 20

20.5.7. WATER TREATMENT AND CONVEYANCE INFRASTRUCTURE

The purpose of this section is to describe the proposed water treatment and conveyance infrastructure required to:

- Treat contact mine water, water exiting the processing plant and domestic water so they can be discharged appropriately into the environment;
- Provide a water supply to the processing plant; and
- Provide a potable water supply to welfare facilities.

Basis of design

The following basis of design were used to define the conceptual design of the water treatment plants.

Treated water quality requirements

The design of the water treatment infrastructure should ensure that it mitigates any environmental and human health risks associated with the delivery of the scheme, complying with the requirements stated in the Finnish Laws (and relevant European Union (EU) Directives). Relevant to this section in particular, this includes compliance with:

- Environmental Regulations: Water Act (587/2011), Environmental Protection Act (527/2014), Government Decree on substances dangerous and harmful to aquatic environment (1022/2006); and
- Public health Regulations: Health Protection Act (763/1994), Radiation Act (48/2).

It has to be noted that the Revised EU Industrial and Livestock Rearing Emissions Directive (IED 2.0) has widened its scope and now includes the mining of ores, including gold. The IED 2.0 has more stringent requirements with regards to water management.

Review of the current regulatory framework, current permitting practices and known future changes in the national and EU Regulations has highlighted the following requirements:

- The scheme should achieve a set of emission limit values/ water quality limits. For the environmental discharge, compliance is to be achieved for each of the water streams. Environmental water quality targets derived from the previously mentioned review are summarised in Table 20-16. Table 20-16 also includes discharge standards set after consultation with Rupert Resources and Envineer, although these standards are not set in the relevant laws; and
- Water abstraction from the environment should be minimised – the water treatment infrastructure should maximise internal water recycling.

To maximise internal water recycling, the quality of the treated water should be fit-for-purpose meeting water quality requirement to supply the ore processing plant. For the PFS, it is assumed that the reclaimed water quality should be similar to the groundwater quality, which would be used otherwise.

Table 20-16 – Environmental water quality targets

Pollutant	Discharge standard (mg/l)	Annual load
Sulphate (as SO_4)	2 000	23 568 t/year
Total nitrogen (as N)	14	129 t/year
Phosphorus (as P)	-	4 734 kg/year
Copper	0.10	-
Nickel	0.15	1 138 kg/year
Arsenic	0.20	-
Cadmium	0.01	88 kg/year
Lead	0.0012	307 kg/year
Uranium	0.01	-
pH	6.0 to 9.5	-

Raw water quality modelling

Based on the water balance presented in Section 20.5.6, a raw water quality model was developed to define the water quality of the contact water (water exiting the raw water dam) and the process water (water exiting the concentrator). The model provides a contaminants mass balance throughout the water management infrastructure. The model was run for Year 1, 8, 10 and 21 for average flow.

It is assumed that groundwater from the dewatering boreholes will provide a source for the potable water supply. Consequently, water quality data from groundwater monitoring (SRK, 2023d) were used to define the water quality at the inlet of the potable water treatment plant.

It is assumed that the sewage produced by welfare facilities is of similar quality than domestic sewage.

Contact water

During operation, contact water originates from pit and underground mine dewatering, the co-disposal pond and the ROM pad. Water quality for each of the sources was derived as followed:

- Pit dewatering – Water is a blend of groundwater and rainwater. The average bedrock groundwater (ore body) quality has been used as the quality for groundwater (SRK, 2023d). It is assumed that rainfall contains zero concentration of all parameters. Allowance is made for increased concentration of ammonia and nitrate to account for residual explosives as well as increased concentration of other compounds to account for contact with pit walls and base.
- Underground mine dewatering – The average bedrock groundwater (ore body) quality has been used (SRK, 2023d). Allowance is made for increased concentration of ammonia and nitrate to account for residual explosives as well as increased concentration of other compounds to account for contact with underground mine walls.

- Co-disposal pond – The co-disposal facility comprising tailings and waste rock in an approximate ration of 1:3 over the life of the mine. The quality of water emanating from the co-disposal facility is blended in proportion to the average annual flows of seepage and runoff. The 'seepage water quality' was estimated on the basis that a portion of rainwater is contact with tailings water entrained in the fines, and a portion of the rainwater is in contact with waste rock within the co-disposal facility. The extent of dissolution of minerals and the water quality was estimated through laboratory bench-scale tests (MEM, 2024c). It is assumed that rainfall contains zero concentration of all parameters. The 'co-disposal runoff quality' was estimated that a portion of the rainfall may be in contact with waste rock or ore. The extent to which the water is contaminated was estimated based on the waste rock geochemistry (MEM, 2023).
- ROM pad – Water is a blend of seepage from the ROM pad facility and rainwater. The seepage water quality was estimated based on the column test presented in (name of the report) (MEM, 2023) and propensity for contaminants to diffuse into the feed water to the ore processing plant. It is assumed that rainfall contains zero concentration of all parameters. Allowance is made for increased concentration of ammonia and nitrate to account for residual explosives.

All contact water streams are blended into the raw water pond.

During active closure, it is assumed that contact water from the co-disposal pond and ROM pad will require treatment, with the contact water quality improving as the closure plan is implementing (MEM, 2024c). It is assumed that no treatment will be required post-active closure period.

Process water

As part of the metallurgical test work (as reported in Chapter 13), whole ore leach testing was carried out (Grinding Solutions Ltd., 2024b). This testing allowed to estimate the process water quality exiting the processing plant.

Resilience and redundancy

The design of the water treatment infrastructure and conveyance should minimise downtime of the mine operation.

Design constraints

Design constraints identified includes:

- Air temperature – As reported in section 20.3.5, temperature range from -5°C and -20°C between November and April. It is thus assumed that all water treatment infrastructure should be either in building, underground or/and trace heated.
- Water temperature – It is assumed that the water temperature of the contact water can be as low as 0.5°C (during snow melt in April and May and during winter months)
- Flow variability – Contact water flows vary over the season and over the operation of the mine (see Figure 20-33 to Figure 20-36). The treatment process should be able to cope with this variation.
- Waste production – It is assumed that the quantity of solid waste produced by the treatment process should be minimised, allowing any waste streams to be either stored on the mine site or removed by road truck tanker and disposed off-site safely through a third party.

Water Treatment Infrastructure Requirements

Mine water treatment

Contact mine water and process water from the ore processing plant are to be treated separately in two separate treatment plants; this is to treat the two different water streams through fit-for-purpose treatment processes.

Contact water treatment plant

The contact water treatment plant, also referred as water treatment plant 1 (WTP1), will treat water from the raw water pond, which is fed from the pit and underground mine dewatering, the ROM pad runoff as well as collection of co-disposal runoff and seepage.

The contaminants mass balance has identified the water quality parameters that require to be removed through treatment to meet permitting limits prior to discharge to the environment. This includes nitrogen compounds and probable trace concentration of uranium, for which treatment optioneering is presented below.

Nitrogen compounds

The presence of nitrate and ammonia results from the estimated ammonium nitrate emulsion loss into the water system. Two treatment options are in consideration at this stage of design:

- Ion exchange: The removal of ammonium and nitrate is achieved through a two-step ion exchange process, with the ammonium removed using a strongly acid cation exchange resin bed and nitrate removed using a strongly basic anion exchange resin bed. The ion exchange beds require to be regenerated using respectively hydrochloric acid for the cation exchange resin and sodium chloride for the anion exchange resin. Consequently, a concentrated regenerant stream is produced.; and
- Biological process (e.g. Moving Bed Biofilm Reactor): In order to remove both ammonia and nitrate, both nitrifying and denitrifying processes are required. The contact water does not contain sufficient level of organic carbon and phosphorus; an external carbon source such as methanol and phosphoric acid will require to be dosed to reach sufficient ammonia and nitrate removal. A biological sludge will be produced. The low water temperature and the variation in flow may pose a challenge during the operation of this technology.

For the purpose of the PFS costing, ion exchange was included until a refined water balance is provided and a trade-off study is completed. The trade-off study will compare capital and operational costs to assess which process is economically and technically favourable.

Uranium

Uranium, which is a naturally occurring elements, can be treated through various treatments such as ion exchange and reverse osmosis. Ion exchange is preferred for the WTP1 for the following reason:

- The treated water is to be discharged to the environment. A reverse osmosis would overtreat the water, removing more salt than required. Consequently, further conditioning, such as pH correction and remineralisation, would be required before discharge to the environment. The concentrate from a reverse osmosis will contain a large quantity of salts which will require additional treatment, which would result in increased enterprise capital and operating costs.; and

- Uranium selective ion exchange can be used. Due to the low concentrations of uranium in the contact water, it is estimated that the ion resin can be replaced and disposed of rather than regenerating it. In this case no ion exchange reject is produced. Alternatively, Uranium can be removed by the same ion exchange resin used for nitrate removal, in which case the uranium will be contained within the regenerant stream and disposed or managed together.

Pipeline protection

To prevent discoloration and odour at discharge point into the Kitinen River, suspended solids and dissolved compounds such as nutrient and metals require to be removed to low concentrations. This is to:

- Reduce the potential for solids to sediment in the pipeline at low flow – the sediments could be flushed out at higher flow and discoloured water would reach the discharge point. The sediments could also reduce the pipe capacity.
- Avoid formation of biofouling on the pipe wall - The presence of organic and inorganic matters increases the propensity of biofouling formation. The accumulation of microorganism can lead to the production of compounds that affect the smell of the water and can results in discoloration of the water when it detaches from the pipe (often associated with change of flow).

This is proposed to be achieved through a coagulation, flocculation and clarification process followed by a filtration process. This has the benefits to serve as a pre-treatment to the ion exchange processes.

Should the biological process be chosen at a later design stage, the coagulant would be added in the return activated sludge from the clarifier, increasing metals and inorganic removal through the biological process. An ultrafiltration membrane steps would be required to pre-treat water before the ion exchange bed.

Sludge management

The sludge dewatering facility of WTP1 is to treat sludge from WTP1, WTP2 and WTP0. The sludge is to be dewatered before being disposed of safely through a third party.

PFS costed solution

For the purpose of the PFS, the following treatment train was selected:

- Coagulation, flocculation and clarification through lamella clarifiers;
- Rapid gravity filter;
- Two-step ion exchange; and
- Sludge dewatering.

Process water treatment plant

Process water treatment plant, also referred as water treatment plant 2 (WTP2), will treat water generated by the ore processing plant to sufficient water quality so it can be reclaimed and reused as a freshwater source to the ore processing plant. It has to be noted that the inlet of the WTP2 will be topped-up with feed water from WTP1 in order to ensure that the freshwater supply from WTP2 meets the demand from the ore processing plant.

The contaminants mass balance has identified the water quality parameters that require to be removed through treatment to meet the process water feed quality requirements. This includes sulphate, nitrogen compounds and salinity.

To maximise the water recovery whilst minimising waste stream generation, WTP2 will provide a zero-liquid discharge solution.

Pre-treatment

Water is to be pre-treated through a coagulation, flocculation and clarification process followed by rapid gravity filtration. The pre-treatment aims at protecting the downstream processes removing suspended solids and metals and organics that may affect their performance.

Sulphate removal through gypsum formation

The water treated through the rapid gravity filter is further filtered through a first step of ultrafiltration (UF) membrane and reverse osmosis (RO) membrane processes. The RO permeate is used as a freshwater source to the ore processing plant. The RO reject is dosed with lime which reacts with sulphate and precipitates as gypsum. The treated RO reject is clarified from the gypsum and further treated through UF and RO membrane.

The RO permeate from this second step is of marginally lower quality than that of the first stage, and is thus recirculated to the inlet of the first UF step to provide a second pass treatment, and increase the overall WTP2 water recovery rate. The RO reject from the second step is also dosed with lime where gypsum is formed before clarification.

Nitrogen compounds and salinity removal

Through the treatment described above, nitrogen compounds and various salts have been concentrated in the second stage RO reject. A crystalliser will be used to separate a mixed salt from the water. The mixed salt precipitate will be disposed of safely through a third party. The crystalliser will also be fed with the ion exchange regenerant stream from WTP1.

Gypsum slurry management

The gypsum slurry produced from the clarification process will be dewatered and washed through counter-current wash of the dewatered solids on the belt filter. A portion of the crystalliser condensate will be used for the counter-current wash there ensuring that the entrainment of dissolved and unstable salts is minimised. To ease transportation, the washed and dewatered gypsum cake will be re-slurried using the remaining crystalliser condensate and pumped to the gypsum slurry pond. The supernatant water from the gypsum slurry pond is to be returned to the raw water pond.

Sludge management

Sludge resulting from the coagulation process will be pumped to the sludge dewatering facilities of WTP1.

PFS costed solution

For the purpose of the PFS, the following treatment train was selected:

- Coagulation, flocculation and clarification through lamella clarifiers;
- Rapid gravity filter;

- Stage 1and Stage 2 Ultrafiltration and Reverse Osmosis;
- Stage 1 and Stage 2 Lime dosing and gypsum precipitation clarifier;
- Crystalliser;
- Chemical storage and make-up; and
- Gypsum sludge dewatering and counter current washing.

Welfare water services

Potable water treatment plant

The potable water treatment plant, also referred as water treatment plant 0 (WTP0), will treat groundwater abstracted from the external pit dewatering boreholes to achieve drinking water quality.

A water quality risk assessment was carried out to identified parameters required to be removed through treatment. The approach follows the methodology of the WHO Water Safety Plan Manual System Assessment (Module 2 and 3 and part of Module 4). A semi-qualitative risks matrix was used to derive public health risk scores. Iron and manganese, based on available water quality, are the two parameters requiring treatment.

The groundwater is first aerated to promote oxidation and precipitation of iron and manganese. The water then undergoes coagulation and flocculation, with floc settlement achieved thanks to a lamella clarifier. pH adjustment may be required to enhance the coagulation and flocculation process. The clarified water is then filtered through a greensand filter allowing further manganese removal. Finally, the filtered water is disinfected using sodium hypochlorite. Sludge from the clarifier and backwash water will be dewatered in the WTP1 sludge dewatering plant.

Sewage treatment plant

This facility will consist of a packaged sewage treatment unit and will remain on site to provide long-term sewage treatment requirements during operations. The sewage treatment plant is based on activated sludge process designed to remove organics and nitrogen compounds biologically (nitrification and denitrification). A coagulant is added to enhance phosphorus removal. A sand filter is used as a tertiary treatment to reduce solids concentration. The resulting sludge will be disposed of appropriately. Any water that is discharged to the environment will be treated to meet the minimum local water quality requirements.

20.6 PERMIT REQUIREMENTS, STATUS OF PERMIT APPLICATIONS AND BOND REQUIREMENTS

20.6.1. APPLICABLE CODES

Mining Code

Mining and exploration projects in Finland are subject to the Finland Mining Act (621/2011). The Act underwent some changes in 2023, the changes are incorporated in this text. The General Provisions of this act are described as follows:

The objective of this Act is to promote mining and organise the use of areas required for it, and exploration, in a socially, economically, and ecologically sustainable manner. In order to fulfil the purpose of the Act, the securing of public and private interests is required, with particular attention to:

1) The preconditions for engaging in mining activity;

2) The legal status of landowners and private parties sustaining damage; and

3) The impacts of activities on the environment and land use, and the economic and sustainable use of natural resources.

A further objective of the Act is to provide the municipalities and individuals with opportunity to influence decision-making. Furthermore, an objective of the Act is to promote the safety of mines and to prevent, decrease and avert any inconvenience and damage incurred in the activities referred to in this Act, and to ensure liability for damages for the party causing the inconvenience or damage.

Environmental Code

The Mining Act (621/2011) also refers to other legislation for “decisions on permit issues or other matters hereunder and other activities in accordance with this Act shall comply with the provisions of the Nature Conservation Act (9/2023), the Environmental Protection Act (527/2014), the Act on the Protection of Wilderness Reserves (62/1991), the Land Use Act (132/1999), the Building Act (751/2023), the Water Act (587/2011), the Reindeer Husbandry Act (848/1990), the Radiation Act (859/2018), the Nuclear Energy Act (990/1987), the Antiquities Act (295/1963), the Off-Road Traffic Act (1710/1995), the Dam Safety Act (494/2009), the Administrative Act (434/2003), the Act on Electronic Transactions in Official Activities (13/2003), the Act on Sámi Assemblies, the Sámi Language Act (1086/2003), the Language Act (423/2003) the Act on Monitoring Foreign Business Acquisitions (172/2012), the Act on the Requirement of Permits for Certain Real Estate Acquisitions (470/2019), the law on the State's Right of First Refusal in Certain Areas (469/2019) and in the law on the Redemption of Immovable Property and Special Rights to Ensure National Security (468/2019), as well as elsewhere in the law.

20.6.2. REGULATIONS

Regulations are specified for exploration (Section 51) and mining (Section 52) permits in the Mining Act (621/2011).

Section 51 - Regulations to be included in an exploration permit

The exploration permit shall specify provisions for the location and borders of the exploration area. The exploration permit shall include the necessary provisions for securing public and private interests concerning the following:

1) The times and methods of exploration surveys and the equipment and constructions related to exploration;

2) Measures to diminish harm caused to reindeer herding in the area specially intended for reindeer herding, and to other traditional livelihood for the Sámi in the Sámi residential area;

3) Ensure that activity under the permit will not endanger the Sámi people’s rights to maintain and develop their language, culture, and traditional livelihoods in the Sámi residential area, or the rights of the Skolt’s in accordance with the Skolt act in the Skolt area;

4) Obligation to report about exploration activities and results;

5) Post investigation measures and the final deadline for submission of notification concerning these measures;

6) The waste management plan for extractive waste and compliance therewith;

7) The obligation to report on the exploration work to the appropriate authority overseeing public interests within its line of duty;
8) The schedule for decreasing the size of the exploration area;
9) Collateral in accordance with chapter 10;
10) measures to ensure that ore prospecting and other use of the exploration area does not cause harm to human health or danger to public safety, significant harm to other business activities, significant changes in natural conditions, significant damage to rare or valuable natural occurrences, significant landscape damage or other significant harmful environmental impact;
11) Other terms concerning exploration and use of the exploration area in order to ensure that the activity does not result in any consequence prohibited by this act; and
12) Other specifications that are necessary in view of public and private interests and pertaining to the implementation of the conditions of the permit.

Good practices for reducing the environmental impacts caused by ore exploration must be taken into account when issuing permit regulations, regarding the timing and methods of exploration surveys, as well as equipment and structures related to ore prospecting.

More detailed regulations on the regulations issued in the exploration permit can be issued by a decree of the Government.

Section 52 - Regulations to be included in a mining permit

A mining permit shall give provisions for the location and borders of the mining area to be formed and the auxiliary area to the mine, taking the provisions laid down in sections 19 and 47, and the content of the rights of use and other special rights pertaining to the auxiliary area to the mine, into consideration. However, the permit authority may implement such changes in the location and borders of the mining area or auxiliary area to a mine presented in the application as are necessary in consideration of the provisions laid down in this Act. The mining permit shall specify a term within which the mining permit holder shall engage in mining activity or other such preparatory activity that indicates that the permit holder is seriously aiming to initiate actual mining operations. The time limit may be, at maximum, 10 years after the permit becomes legally valid. The mining permit shall include the necessary provisions for securing public and private interests concerning the following:

1) Avoidance or limiting of detrimental impacts of mining activity and addressing of elements necessary to ensure people's health and public safety;
2) Measures for ensuring that mining activities do not entail obvious wasting of mining minerals or endanger or hamper potential future use of the mine and excavation work;
3) The obligation to report on the extent of exploitation of the deposit and results;
4) Measures to diminish harm caused to reindeer herding in the area specially intended for reindeer herding, and to other Sámi traditional livelihoods in the Sámi residential area;
5) Ensuring that activity under the permit will not endanger the Sámi's rights to maintain and develop their language, culture and traditional livelihoods within the Sámi residential area, or the rights of the Skolts in accordance with the Skolt Act in the Skolt area;
6) Collateral, in accordance with Chapter 10, associated with mine-closure alongside other obligations related to termination of mining activities and those after termination;

7) The deadline to be set for submission of any further specifications related to verifying the permit regulations;

8) Material on other aspects of activity under the mining permit in order to ensure that the activity does not result in any consequence prohibited by this Act:

   a. On the preservation and renewal of trees and other vegetation and on new plantings during mining operations;

   b. On the placement of operations in the mining area, taking into account the effects on biodiversity and other environmental effects;

   c. Measures to prevent the occurrence of significant harmful environmental effects;

   d. Measures to prevent a significant deterioration of the settlement or economic conditions of the locality; and

   e. On phased closure of the mine.

9) Other specifications that are necessary in view of public and private interests and pertaining to the implementation of the conditions of the permit.

More detailed regulations on the regulations issued in the mining permit can be issued by a government decree.

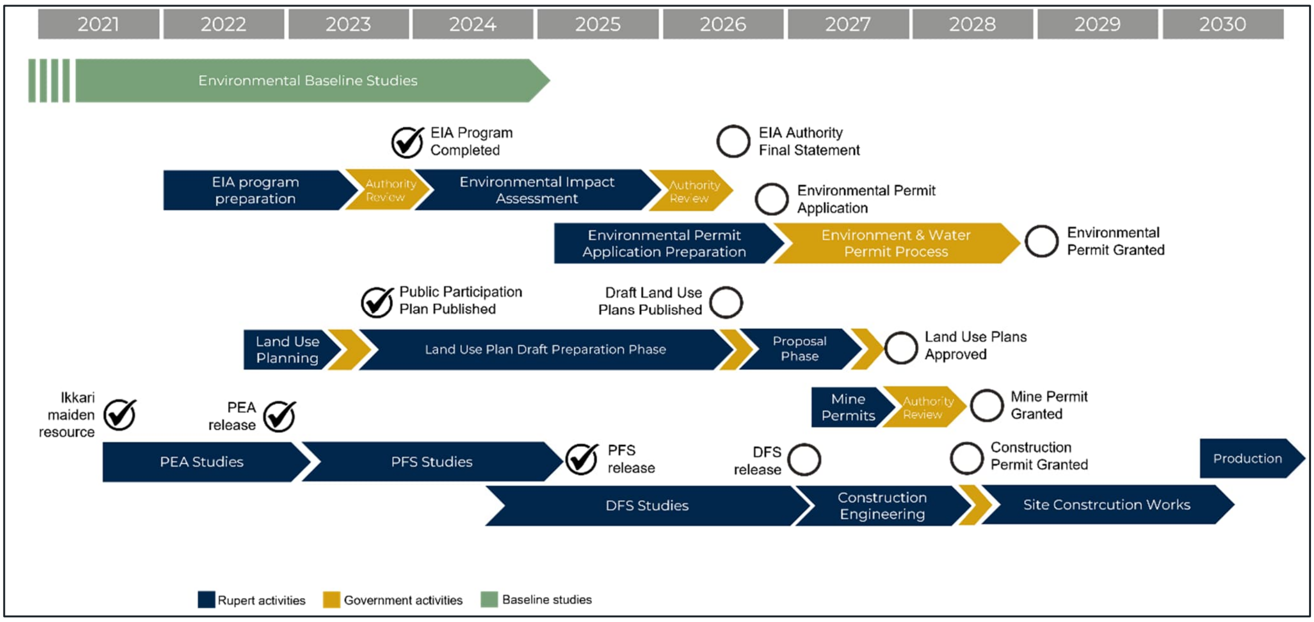


Figure 20-37 – Permitting Pathway in Finland - Company Sourced

20.6.3. ENVIRONMENTAL PROTECTION POLICIES AND STRATEGIES

Rupert Resources has a corporate social responsibility policy, environmental policy, community policy and health and safety policy that have been designed provide a risk management framework for the Project. These documents are available on the Company website. There are no Natura areas or national protected areas on Rupert Resources' current exploration land package.

20.6.4. RURAL AND LAND DEVELOPMENT POLICIES AND STRATEGIES

The mining area is part of the Northern Lapland provincial plan, which was ratified by the Government on December 27, 2007. The Ikkari project requires three stages of land use planning. The processes have been initiated in 2022 and work will continue for at least three years.

20.6.5. INTERNATIONAL AGREEMENTS, PROTOCOLS AND CONVENTIONS

Rupert Resources' activities are currently confined to Finland where local legislation is considered to meet or exceed international best practice.

20.6.6. BOND ARRANGEMENTS

The closure bond guarantee has been treated according to the following logic.

1) Closure costs are recorded to the financial model as and when they are expected to occur, including progressive reclamation activities during project life, with bullet payment at the end of mine life for any post-operational closure provisions;
2) The cost of the guarantee is included as a line item in "G&A" and should build up as the "damage" requiring reclamation occurs, which is therefore linked to surface area primarily, but not exclusively; and
3) The annual cost of the guarantee is then gradually wound down as the reclamation closure activities are implemented, to reach zero at the end of mine life plus first year of dedicated closure works, whereupon the guarantee is replaced by final bond.

20.7 SOCIAL AND COMMUNITY RELATED REQUIREMENTS

Reindeer herding is a traditional livelihood in the Northern half of Finland. The reindeer herding area consists of 54 cooperatives. The 20 northernmost cooperatives form an area specifically intended for reindeer herding, where other land use may not be used in a manner that may significantly hinder reindeer herding. The 13 northernmost cooperatives are also the Sámi residential area. Ikkari is located within the area specially intended for reindeer herding but not within the Sámi residential area. The Rupert Resources exploration permits fall mostly within the Sattasniemi Reindeer Herding Cooperative. Rupert Resources has regular interaction with Sattasniemi reindeer herders and annual meetings to discuss matters concerning the interaction between exploration and reindeer herding and to coordinate each other's activities in the area. As part of the ongoing EIA work, Rupert Resources has initiated dialogues with three other reindeer herding cooperatives adjacent to the Sattasniemi cooperative to the West, South and Southwest, where impacts from the project may occur in the future.

The nearest reindeer farm, and closest inhabited house, is located some 3.5 km to the southwest of the Ikkari deposit. Since reindeers are grazing freely, animals are pasturing across the whole exploration area. The project area is mainly used in wintertime, and a transportation route splits the project area from South to North.

Rupert Resources has organized regular village meetings since 2017 for all the closest villages. Five different village meetings were held in fall 2021 and more than 100 inhabitants attended the meetings. Meetings included a general presentation of Rupert Resources activities in the region and an engaged question and answer session, including open conversations with company members. During spring 2022, Rupert Resources arranged a local stakeholder feedback survey for exploration areas nearest landowners and inhabitants. This was renewed in autumn 2023 and will be repeated on annual basis.

Also, the company has taken part in the “*Experienced impacts of Mining in Sodankylä*” follow-up study since 2018. A Study has been arranged every other year for all Sodankylä inhabitants. As part of the EIA process, Rupert Resources has established several ways to initiate dialogues with stakeholders:

- A stakeholder steering committee for the Ikkari EIA process, where authorities and local stakeholders can give their feedback and comments to the ongoing EIA process.
- Small group discussions have been held twice in 2023 and plans are to continue in 2024 as long as EIA work is ongoing. The small groups are thematized: reindeer herding, inhabitants, municipality and livelihoods, recreational use and nature protection, land and water area owners.
- Interviews with key persons, implemented by Ramboll Finland Oy.
- Open coffee events once a month in the Sodankylä town centre, where anyone can come and discuss Rupert Resources activities in the area.

20.8 MINE CLOSURE

The overarching approach to mine closure and reclamation planning is guided by Ikkari’s post-closure land use vision and reclamation objectives. The post-mining land use vision is one of re-established pre-mining land uses, mainly locally common habitat (mires and mixed forest), support for local passive recreational enjoyment of nature including snowmobile and hiking trails, and reindeer husbandry. Supporting reclamation objectives are as follows:

- long-term physical (geotechnical) stability, chemical stability, and the erosional dynamic equilibrium of watercourses and landforms;
- water quality that meets accepted standards for safe discharge to the surrounding environment;
- self-sustaining, locally common vegetation that supports the targeted post-mining land uses; and
- reflection of community and stakeholder values in post-mining land uses to the extent practicable.

The successful achievement of these reclamation objectives and post-closure land use vision begins with the overall mine site layout design. The site layout proposed for Ikkari reduces impact on the environment by avoiding disturbance within sensitive environmental areas and by minimizing the overall disturbance footprint. Features that will remain in the landscape in perpetuity such as the river diversion, water treatment sludge disposal facilities (sludge landforms), and co-disposal facility (co-disposal landform) are designed such that they will be reclaimed in a phased progressive manner. Reclaiming disturbed areas as soon as is practical allows for continuous performance monitoring during mine operations, rapid maintenance or repair, and refinement of closure designs and implementation techniques over time based upon each new lesson learned from past reclamation works completed.

The mine closure plan and designs are based upon existing leading industry practices that have been layered to create a solution that is both robust and flexible. For example, the co-disposal facility has been designed to provide physical and chemical stability to the final landform while reducing the overall disturbance footprint required for waste storage in perpetuity. The co-disposal facility will be lined, progressively re-graded and covered, and to further reduce potential for degradation of downstream environments, the landform will be designed in a geomorphic manner. The anticipated outcome of this geomorphic design approach is that the co-disposal landform will perform similarly to local natural landforms in terms of erosion and deposition, vegetation (distribution, quality, and quantity), and land use capability (slopes that are common to both recreational and reindeer herding land), while also visually blending into the natural environment. This exemplifies how Ikkari's post-closure land use vision and reclamation objectives will be implemented.

The closure plan is based on the current approved permits and information available. Overall, it is considered to be a conceptual plan that will evolve rapidly as additional information and data is gathered, particularly during the detailed engineering phase and the first several years of mining during which reclamation trials are planned to support progressive reclamation activities.

The closure bond guarantee is described in section 20.6.6.

20.9 HEALTH AND SAFETY ISSUES

The design of the proposed Ikkari mineral property (mine, mineral processing plant, necessary infrastructure and utilities) has been completed using the legal and professional requirements of the UK Construction (Design and Management) Regulations 2015 (CDM), ICMM guidance on mine waste materials (GIMST), ICMM guidance on the critical controls for fatal hazards and all relevant occupational ad workplace laws, regulations of Finland, UK and Canada where WSP design engineers reside.

WSP and Rupert Resources have collaborated to ensure all parties involved ensure the adequate consideration of health and safety, that both organisations are competent and adequately resourced, the avoidance or mitigation of risks where reasonably practicable in all facets of the planned industrial activity has been undertaken, that adequate information is provided to others parties to enable them to manage residual risks, and that there has been adequate communication and co-operation in respect of health and safety.

21 CAPITAL AND OPERATING COSTS

21.1 INTRODUCTION

The capital and operating costs for Ikkari have been separated into cost areas based on project work breakdown structure (WBS). The breakdown is as follows:

- Mining;
- Co-Disposal Facility;
- Surface Infrastructure;
- Water Management;
- Concentrator and Filtration Plant;
- Closure;
- Water Treatment;
- Electrical Engineering; and
- Contingency.

21.2 COST AREAS ESTIMATES

Costs included in the different WBS areas are described below.

21.2.1. MINING

Capital costs associated with open pit and underground mining include:

- **Mining Equipment**
- **Surface Infrastructure**
  - Surface paste plant;
  - Open pit dewatering equipment;
  - Open pit explosives storage; and
  - Portal construction.
- **Underground Infrastructure**
  - Underground dewatering equipment;
  - Underground workshops, offices and refuges; and
  - Underground explosives storage.
- **Mine Ventilation**
  - Fresh air and exhaust air raises and housings including heating requirements.
- **Capitalised Waste**
  - Open pit overburden material; and

- Underground declines and workshop development.

- **Equipment & Software**
    - Technical equipment and software;
    - Personal protective equipment; and
    - Fleet management systems.
- **Sustaining Capital**
    - Includes capital expenditure spent after full planned production has been reached;
    - Mining fleet replacements;
    - Mobile equipment, equipment and software, and surface infrastructure spare parts; and
    - Paste plant replacements.
- **Contingency**
    - Additional contingency of 15% on initial capital costs

Operating costs included in the mining WBS are:

- **Labour**
    - Includes operators, supervisors, maintenance and technical services.
- **Consumables and Materials;**
    - Includes explosives, paste fill, ground support and equipment parts.
- **Maintenance**
    - Includes main components, service parts, other required parts, lubricants and all necessary labour for maintenance.
- **Diesel, Power and Water**
    - Consumption estimated using productivity estimates for equipment, dewatering, backfill and underground mining operational ventilation.

Operating costs for mining include all costs associated with open pit mining, underground development, stoping and backfill, mine services, mine management and technical services.

21.2.2. CO-DISPOSAL STORAGE

The estimation of the co-disposal capital costs includes for quantification of:

- topsoil and/or peat stripping and transporting to nearby stockpile(s);
- bulk earthworks using overburden won from the open pit to form the base of the co-disposal facility;
- formation of perimeter channels;
- construction of a perimeter access track;
- lining of the base with compacted low permeability clay and HDPE; and

- protection of the lining layers.

The estimation of the runoff collection pond capital includes for quantification of:

- topsoil and/or peat stripping and transporting to nearby stockpile(s);
- bulk earthworks using imported engineering fill to form the embankments to the confining collection pond;
- lining of the base with compacted low permeability clay and HDPE;
- protection of the lining layers; and
- downstream slope protection at the spillway.

The estimation of the co-disposal OPEX includes for quantification of:

- disposal of waste end tipped from the mining fleet with spreading and compaction;
- disposal filtered tailings with transportation from the filter plant, placing, spreading and compaction;
- excavating drainage channels and sumps;
- install geotextile in sumps;
- supplying and installing instrumentation; and
- topographic surveys.

21.2.3. SURFACE INFRASTRUCTURE

The estimation of the surface infrastructure capital costs includes for quantification of:

- Topsoil and/or peat stripping and transporting to nearby stockpile(s);
- Bulk earthworks;
- Formation of pads;
- Run-of-Mine wall;
- Lining beneath ore stockpile;
- Road construction;
- Surface buildings, including administration;
- Site fencing and access control;
- Culverts for drainage;
- River diversion works; and
- Sediment control dams, and all necessary water ponds.

21.2.4. WATER MANAGEMENT

Capital and operating costs for water management account for:

- Perimeter pit dewatering wells drilling and surface works construction; and
- Perimeter pit well pumping equipment, piping and electrics.

21.2.5. CONCENTRATOR & FILTRATION PLANT

Capital costs associated with the crushing circuit, crushed ore stockpile, processing plant and filtration plant include the following (non-exhaustive list):

- **Process and mechanical**
  - Main process equipment (crushers, mills, tanks, agitators, elution column, etc.);
  - Ancillary and secondary process equipment (pumps, chutes, etc.);
  - Material handling equipment (conveyors, feeders, etc.);
  - Reagents preparation equipment;
  - Services (cranes, sump pumps, compressors, etc.);
  - Equipment installation labour costs.
- **Civil and structural**
  - Site preparation;
  - Concrete;
  - Structural steel,
  - Architectural components;
  - Civil and structural labour costs.
- Electrical and instrumentation
  - Cable trays;
  - Grounding equipment;
  - Fire detection equipment;
  - Lighting and services;
  - Instruments;
  - Cables;
  - Electrical equipment and instruments installation labour costs.
- **Sustaining capital**
  - Includes capital expenditure spent after full planned production has been reached;
    - Plant modifications required for 2.0 Mt/a throughput (equipment and labour)
- **Contingency**
  - Additional contingency of 15% on initial capital costs

Operating costs included in the processing and filtration WBS are:

- **Labour**
  - Includes operators, supervisors and maintenance personnel.

- **Reagents and grinding media**
  - Includes all mill reagents (pH control, leaching, cyanide destruction, flocculation, etc.) and grinding media for SAG and ball mill.
- **Power**
  - Includes electrical power for operating processing and filtration plant equipment.
- **Consumables, wear parts and maintenance**
  - Includes wear and replacement parts, consumables and maintenance supplies.
- **Mobile equipment**
  - Includes mobile equipment fuel and maintenance for filtered tailings handling, reagents transport, etc.
- **Laboratory**
  - Includes external laboratory testing and analyses as well as laboratory supplies replacement.

21.2.6. CLOSURE

Costs have been developed based on a detailed build-up of tasks and activities associated with each of the specific site components. Thirteen major cost categories were used:

- Open Pit and Underground Mining Workings;
- Co-disposal Facility;
- Gypsum Slurry Ponds;
- Reclamation Material, Ore Stockpiles and Borrow Area;
- Water Retention Structures;
- Buildings and Infrastructure;
- Roads and Linear Disturbances;
- Site Wide Water Management;
- General Surface Reclamation;
- Success Criteria, Monitoring and Maintenance;
- Closure and Post Closure Management;
- Mobilisation and Demobilisation; and
- Closure Planning and Studies.

21.2.7. WATER TREATMENT

The estimation of the water treatment capital costs includes:

- Contact mine water treatment plant;
- Process water treatment plant;
- Potable water treatment plant;

- Sewage treatment plant; and
- Discharge pipeline.

The estimation of the water treatment operating costs includes:

- Power and chemicals; and
- Mechanical and electrical replacement as part of maintenance.

21.2.8. ELECTRICAL ENGINEERING

Electrical engineering costs include:

- Finngrid connection fee;
- Overhead power line (110 kV);
- Receiving substation (110/20 kV);
- Switchgear and reactive compensation (24 kV);
- Series-connected secondary substation; and
- Associated cables and infrastructure.

21.2.9. SITE GENERAL & ADMINISTRATIVE

Site general and administrative operating costs comprise of:

- **Administrative Salaries and Wages**
  - Personnel includes site management, administration, finance, environment safety, information technology, community affairs, security, medical and warehouse/supply.
- **Supplies and Services**
  - Include auditing, bank charges, cleaning, first aid, insurance legal fees, maintenance for administration facilities, office supplies, and consultants.
- **Closure Bond**
  - An allowance for the cost of closure bond to be paid has been made.

21.3 COST BASIS AND ESTIMATION

The base date for the capital estimate was mid-2024. Costs obtained prior to this were escalated accordingly. Costs are expressed in U.S. dollars (USD) and are reported on a dry short ton basis, unless otherwise stated. Costs were converted to USD based on the exchange rates shown in Table 21-1. The cost estimate is a project basis costing and does not include the cost of financing or escalation in costs.

Costs were prepared and estimated by WSP to an AACE Estimate Class 4 with accuracy ranging +/- 15-30%.

Table 21-1 – Exchange Rates

Exchange Rate	Rate
EUR/USD	1.05
EUR/CAD	1.49
GBP/USD	1.25

21.3.1. MINING

Mine capital and operating costs have been estimated using multiple methods:

- Contractor mining costs including open pit mining and underground development were sourced from contractor bid analysis. WSP was able to obtain multiple budget estimates based on a schedule of quantities, successfully implementing contractor costs into the Project. Contractor open pit mining and underground mine development have been implemented for Ikkari, which results in lower capital outlay for mining equipment;
- Vendor quotes were used to cost mining equipment and consumables including fleet, dewatering, explosives, diesel, power, ground support and ventilation. Required equipment numbers were estimated based on production requirements and cycle estimates; and
- Bottom-up and first principles in combination with the LOM schedule were used to estimate mine operating costs.

21.3.2. CO-DISPOSAL STORAGE

The physical elements were measured using Civil-3D software and used as an output to the design process. Unit rates were applied to the quantities to determine the amounts for each line item. These were sourced from the Finnish FORE ROLA (EG Finland, 2024), which is a cost estimation tool for infrastructure and building projects and based on completed projects in Finland. The FORE database is maintained by a group of construction industry experts.

The database consists of unit prices that usually include materials, transportation and work. Next to every item there is a description of what it contains more specifically.

FORE considers different factors that affect the prices. These factors include location, scale and conditions of the site. In this project, the factors were:

- Location 1.04 (Northern Lapland);
- Scale 0.98 (1.00 would be considered a “normal sized” project); and
- Conditions 1.00 (difficulty of conditions with 1.00 being normal conditions).

In this project, some of the unit prices were not directly available in FORE and had to be estimated using built-up rates for preparation of surfaces for lining.

Unit rates were applied to the quantification with derived unit rates were applied to:

- Disposal of waste; and
- Disposal of filtered tailings.

21.3.3. SURFACE INFRASTRUCTURE

The earthwork elements were measured using Civil-3D software which was also used for the measurement of primary dimension to other components and as an output to the design process.

The cut and fill slopes are assumed to be 1V: 2H for the PFS study, these will need to be confirmed with further site investigation and subsequent slope stability analyses.

This estimate excludes:

- Mobilisation and preliminary items such as clearing of vegetation and deforestation prior to the peat stripping;
- Construction site establishment; and
- Construction de-mobilisation.

Unit rates were applied to the quantities to determine the amounts for each line item. These were sourced from the Finnish FORE ROLA (EG Finland, 2024).

The database consists of unit prices that usually include materials, transportation and work. Next to every item there is a description of what it contains more specifically. The database also has information of the CO_2 emissions for each item, if required.

FORE considers different factors that affect the prices. These factors include location, scale and conditions of the site. In this project, the factors were:

- Location 1.04 (Northern Lapland);
- Scale 0.98 (1.00 would be considered a “normal sized” project); and
- Conditions 1.00 (difficulty of conditions with 1.00 being normal conditions), except for the river diversion where 1.25 is applied.

The price of miscellaneous items was estimated through the application of $/m^2 based on industry standard rates.

21.3.4. WATER MANAGEMENT

The basis of the capital cost estimate for the peripheral pit dewatering (via boreholes) is as follows:

- Costs are based on budget quotes or previous project experience costs;
- Detailed engineering studies and procurements costs of 30% were allowed for;
- Where applicable a 5% contractor markup was applied; and
- All equipment costs are based on new purchased equipment.

Piteau (2024) propose that 16 boreholes would be needed to dewater the Ikkari open pit and underground operations. It is, however, considered necessary to provide for sufficient capital to drill additional boreholes in anticipation of already experienced difficult drilling conditions and the likely underperformance of individual constructed dewatering boreholes, which would require the drilling and equipping of boreholes at further locations to make up for such insufficiencies.

Standby boreholes should also be drilled near strategic high yielding dewatering boreholes should these fail or become obsolete due to mining expansion. For these purposes this costing has allowed for 24 dewatering boreholes, eight (8) of which to be drilled to 100 m depth and sixteen (16) to 250 m depth.

21.3.5. CONCENTRATOR & FILTRATION PLANT

Mechanical Equipment and Piping

For the processing and filtration plants, budgetary quotes were obtained from reputable equipment suppliers for the main process equipment. About a dozen equipment packages were established, covering most of the pricier items and approximately 60% of the overall equipment CAPEX. The cost of the remaining secondary equipment was estimated using WSP's internal database.

For each package, equipment datasheets were prepared based on the information contained in the process design criteria and the equipment list. On average, requests for quotations were sent to two or three suppliers. Once received, quotations were analysed from a technical perspective to ensure compliance. A high-level commercial review was conducted and WSP provided recommendations for the equipment selection, which were then discussed with RR.

Freight, handling and duties costs were estimated using WSP's nominal factors.

To estimate the equipment installation costs, installation manhours were determined for each equipment. Labor rates were established across the project and confirmed by RR.

Larger bore piping was sized and quantities were estimated based on the P&IDs and approximate routing inferred from the 3D model. WSP's internal cost database was used to determine the unit cost of each meter of piping for each pipe type. Piping installation costs were evaluated based on WSP's standard engineering methods.

Electrical, Instrumentation and Automation

Based on the equipment list and preliminary P&IDs, the electrical and instrumentation deliverables were prepared, and quantities were determined. WSP's internal database was used to estimate the overall cost of the electrical equipment and instruments.

Civil and Structural

The 3D model was used as a basis to estimate the quantities of concrete and structural steel required for the processing and filtration plants. Concrete and steel unit costs established for the overall project were used to estimate the CAPEX.

21.3.6. CLOSURE

The closure cost estimate was prepared following a AACE Class 5 cost estimating methodology. The development of the closure cost estimate generally followed a deterministic estimating methodology where the properties are known and can be fully or partially determined.

The following bullets outline the general estimating steps and methodology used:

- Preparation of a WBS list of cost categories and sub-categories based on scope of work and reclamation requirements with estimated quantities and closure schedules obtained from the conceptual design drawings and take-offs;
- Development of select unit rate using HCSS HeavyBid estimating software;

- Assignment of resources to major cost and work breakdown items with anticipated production rates developed based on the estimator's judgement, past project reference data, CAT performance handbook and other productivity sources;
- Benchmarking of the developed unit rates against the suite of contractor rates from WSP's mine closure database; and
- Costs for the work tasks have primarily been developed as a first principles bottom-up, crew-based, detailed cost model. The crews required to perform the majority of the various work items were built-up using a first principles approach, to develop hourly or unit rates. Inclusion of support equipment within a crew has been determined on a per activity basis.

21.3.7. WATER TREATMENT

Mechanical Equipment and Piping

For the water treatment plants, costs for the main process equipment were derived from budgetary quotes from reputable equipment suppliers or estimated using WSP's internal database.

The cost of the remaining secondary equipment was estimated at 15% of the total mechanical CAPEX.

Based on experience, the contribution of mechanical and piping costs was estimated at 45% of the total CAPEX.

Electrical and Instrumentation

Based on WSP experience, the electrical cost contribution to the total CAPEX is estimated at 18%.

Civil and Structural

Based on WSP experience, the civil and structural cost contribution for the water treatment to the total CAPEX is estimated at 7%. Furthermore, it is assumed that mechanical equipment and storage would be skid mounted, requiring a low requirement in civil work.

As per the co-disposal storage facilities, the costs associated with the pipeline construction was defined using the Finnish FORE ROLA (EG Finland, 2024) database.

Other costs

Site equipment installation and preliminary and general costs were estimated based on experience at 20% and 10% of the total CAPEX respectively.

21.3.8. ELECTRICAL ENGINEERING

Electrical engineering costs were based on site and operational requirements. Costs were sourced from budget quotations.

21.3.9. SITE GENERAL & ADMINISTRATIVE

Site G&A costs are comprised of administrative salaries, wages, supplies and services.

- Administrative salaries and wages are based on Finland industry labour benchmarks. Personnel numbers have been estimated based on site requirements;
- Supplies and services required for site have been based on expert knowledge, experience and benchmarking; and

- A bond guarantee is paid annually at 1.5% of the total closure cost. At the end of the LOM, the guarantee is replaced with a closure bond, equal to the cost of closure. The 1.5% figure was obtained from a major international bank.

21.3.10. CONTINGENCY

Contingency is estimated by discipline and applied to capital costs for the project. A risk-based approach assessing accuracy in quantity, price, confidence, method measured and design to estimate the level of contingency was applied. The total contingency for the Ikkari financial model is 11% of total capital.

21.4 CAPITAL COST SUMMARY

Capital costs include pre-production and sustaining capital. Pre-production capital designates capital spent until commercial production is reached. This includes capital spent in pre-production years -3, -2 and -1, as well as associated indirect and management costs until the mine ramps up to full production.

Sustaining capital is all capital spent after full planned production. This includes the replacement of worn-out or exhausted assets. Capital related to the development of the underground mine are included in the sustaining capital estimate.

A summary of capital costs over the LOM are shown in Table 21-2.

Table 21-2 – LOM Capital Costs in million U.S. Dollars

Area	Pre-Production Capital	Sustaining Capital
Mining	45	212
Co-Disposal Storage	34	24
Surface Infrastructure	72	3
Water Management	3	2
Concentrator & Filtration Plant	190	2
Closure	0	151
Water Treatment	134	117
Electrical Engineering	17	2
Indirect	15	0
Contingency	70	55
Total Capital	575	571

Totals may vary due to rounding.

21.5 OPERATING COST SUMMARY

Operating costs have been estimated at $46.8 per tonne of ore. A summary of operating costs is shown in Table 21-3. The basis of cost estimation includes vendor quotations, costed proposals from European based mining contractors and WSP engineering computations.

Table 21-3 – LOM Operating Costs

Area	LOM Average [$/t ore]	LOM Total [Million $]
Mining	26.1	1 356
Co-Disposal Storage	2.0	105
Water Management	0.2	10
Concentrator & Filtration Plant	13.4	699
Water Treatment	2.1	108
Site G&A	3.0	154
Total	46.8	2 432

Totals may vary due to rounding.

Open pit and underground mining operating and unit costs are separated in the table below. Waste and total rock tonnages exclude overburden material which has been capitalised.

Table 21-4 – LOM Mining Operating Costs

Mining Area	LOM Average [$/t rock]	LOM Total [Million $]
Open Pit	4.1	608
Underground	36.6	747
Total	26.1	1 356

Totals may vary due to rounding

Unit rates exclude overburden material.

Open pit mining costs are summarised in Table 21-5.

Table 21-5 – Open Pit Mining Unit Costs

Open Pit Mining Area	Unit	Cost
Ore Mining	$/t ore	4.0
Waste Mining	$/t waste	3.7
Grade Control	$/t ore	0.3
Rehandling	$/t rehandled	0.5

Open Pit Mining Area	Unit	Cost
Mining Services	$/t rock	0.1
Management & Technical	$/t rock	0.1
Total	$/t rock	4.1

Totals may vary due to rounding

Unit rates exclude overburden material.

Tonnes include material moved in open pit only.

Underground mining costs are summarised in Table 21-6.

Table 21-6 – Underground Mining Unit Costs

Underground Mining Area	Unit	Cost
Lateral Development	$/m	4 887
Vertical Development	$/m	4 860
Stope Mining	$/t ore	22.4
Mine Services	$/t rock	2.2
Management & Technical	$/t rock	2.5
Total	$/t rock	36.6

Totals may vary due to rounding

Tonnes include material mined underground only.

Processing unit costs vary with the different production rates. The table below summarises the unit costs for 3.5Mtpa and 2.0Mtpa.

Table 21-7 – Processing Unit Rates

Processing Rate	LOM Average [$/t ore]	LOM Total [Million $]
3.5 Mtpa	11.9	414
2.0 Mtpa	15.6	285
Total	13.5	699

Totals may vary due to rounding.

22 ECONOMIC ANALYSES

It is WSP's opinion that costs estimated in the previous section and considered for this study meets the requirements of accuracy and contingency required for a pre-feasibility study for the techno-economics required to support declaration of Mineral Reserves.

A discounted cashflow (DCF) approach was used to confirm the Mineral reserves utilising annual revenues, capital costs, operating costs, royalties, taxes, depreciation and closure costs. Resulting net annual cashflows are discounted to determine NPV at a selected discount rate. The internal rate of return (IRR) is calculated as the discount rate that yields a zero NPV. Payback period is estimated from production start.

A single mine production scenario was evaluated in the DCF model and sensitivity analysis was carried out for gold price, capital and operating costs.

22.1 KEY ASSUMPTIONS

Assumptions used in the economic model are described below:

- Real model expressed in 2024 U.S. dollars;
- No inflation has been included in the model;
- Working capital is assumed at 5% of net revenue;
- Total project life of 20 years and 2.5 years of pre-production development;
- A processing recovery of 95.8% was applied to contained metal;
- Payability of 99.92% was applied to processed metal to calculate payable gold;
- Commodity price assumption was provided by Rupert Resources, sourced from consensus reporting. A gold selling price of $1 700/oz was used for the Mineral Reserve estimate and confirmed to be economic viable. The PFS base case gold price is set at $2 150/oz and based on long-term consensus by a group of bank analysts;
- The 20-year mine life includes 52 Mt of ROM feed produced from the open pit and underground mine at a grade of 2.09 g/t and 3.5 Moz of contained gold;
- A discount rate of 5% was used to account for the cost of capital and project risk;
- Rupert Resources will be required to pay land and state royalties on the value of the gold delivered to the processing plant according to Finnish regulation. A 0.15% landowner and 0.60% state royalty are required to be paid;
- A 1.5% production royalty, capped at $2 000 000 in relation to the Pahtavaara land purchase;
- A treatment and refining charge of $2.50/oz was applied as a selling cost;
- A corporate tax rate of 20% on taxable income was implemented, in line with Finnish regulation. Taxable income is calculated as the gross revenue less royalties, operation costs, depreciation and carried losses;
- An operating carried net operating loss for the project of $150 million was assumed. These are historical operating losses that the owner incurred in previous years and aims to claim as tax reductions; and

- Depreciation is calculated using a straight-line method over a 10-year period for depreciable tangible assets.

22.2 FINANCIAL RESULTS AND CASHFLOWS

The Project discounted cashflow summary results for Ikkari is shown in Table 22-1. The results of the financial analysis at a gold price of $2 150/oz (PFS base case) shows an after-tax $NPV_{5\%}$ of $1 680 million and an after-tax IRR of 38% and a payback period of approximately two years. Project economics were based on the LOM schedule outlined in Chapter 16 and contain only Mineral Reserves.

Table 22-1 – Ikkari Project Economics Summary

Description	LOM Total @ $2150/oz ($M)	$/oz Payable
LOM ROM Ore Tonnes	52 Mt	-
LOM Au Grade	2.09 g/t	-
LOM Contained Gold	3.5 Moz	-
LOM Payable Gold	3.3 Moz	-
Gross Revenue	**7 188**	**2 150**
Selling Costs	(67)	(19)
Net Revenue	**7 121**	**2 131**
Mining Cost	(1 356)	(406)
Processing Cost	(699)	(209)
Site G&A Cost	(154)	(46)
C1 Cost	**(2 208)**	**(661)**
Water Management Costs	(10)	(3)
Co-Disposal Storage Costs	(105)	(31)
Water Treatment Costs	(108)	(32)
C3 Cost	**(2 432)**	**(727)**
Sustaining Capital	(571)	(171)
All-In Sustaining Costs*	**(3 070)**	**(918)**
Initial Capital	(575)	(172)
Tax Payable	(724)	(217)
All-In Costs	**(4 369)**	**(1 306)**

Description	LOM Total @ $2150/oz ($M)	$/oz Payable
Free Cashflow	**2 819**	**-**
After-Tax $NPV_{5\%}$	**1 680**	**-**
After-Tax IRR	**38%**	**-**
Payback Period from Production	**2.2 years**	**-**

Totals may vary due to rounding.

*Includes selling costs.

The PFS financial results (at $2 150/oz) are summarised below in Table 22-2.

Table 22-2 – Ikkari PFS Results Summary

Description	Unit	Years 1 to 10	LOM
Milled Tonnes	Mt	35	52
Mill Throughput	Mtpa	3.5	2.6
Strip Ratio	W:O	3.8	2.6
Average Processed Grade	g/t	2.13	2.09
Average Metallurgical Recovery	%	95.8%	95.8%
Average Gold Production	Koz	227	167
Recovered Gold	koz	2 273	3 346
Total Cash Cost	$/oz	(603)	(747)
Sustaining Capital	$/oz	(115)	(171)
AISC*	$/oz	(717)	(918)

*Includes selling costs.

Key financial results are shown in the figures below.

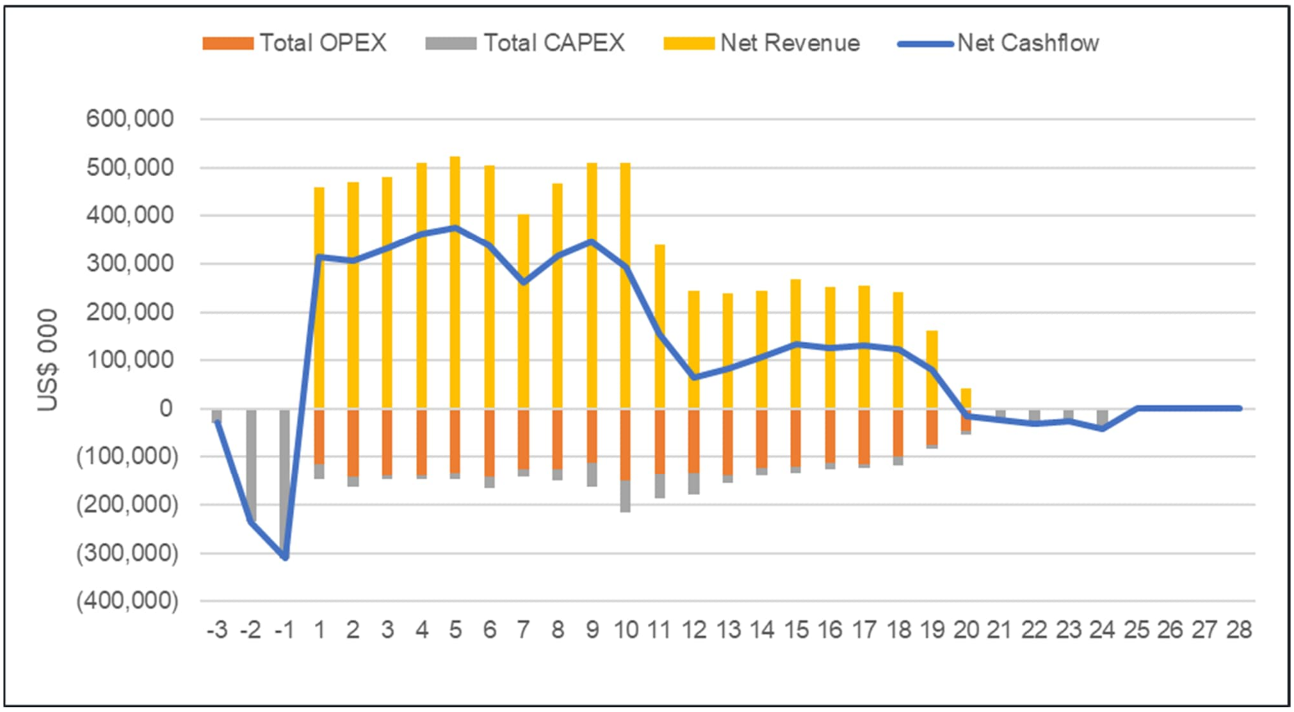


Figure 22-1 – Project Cashflows

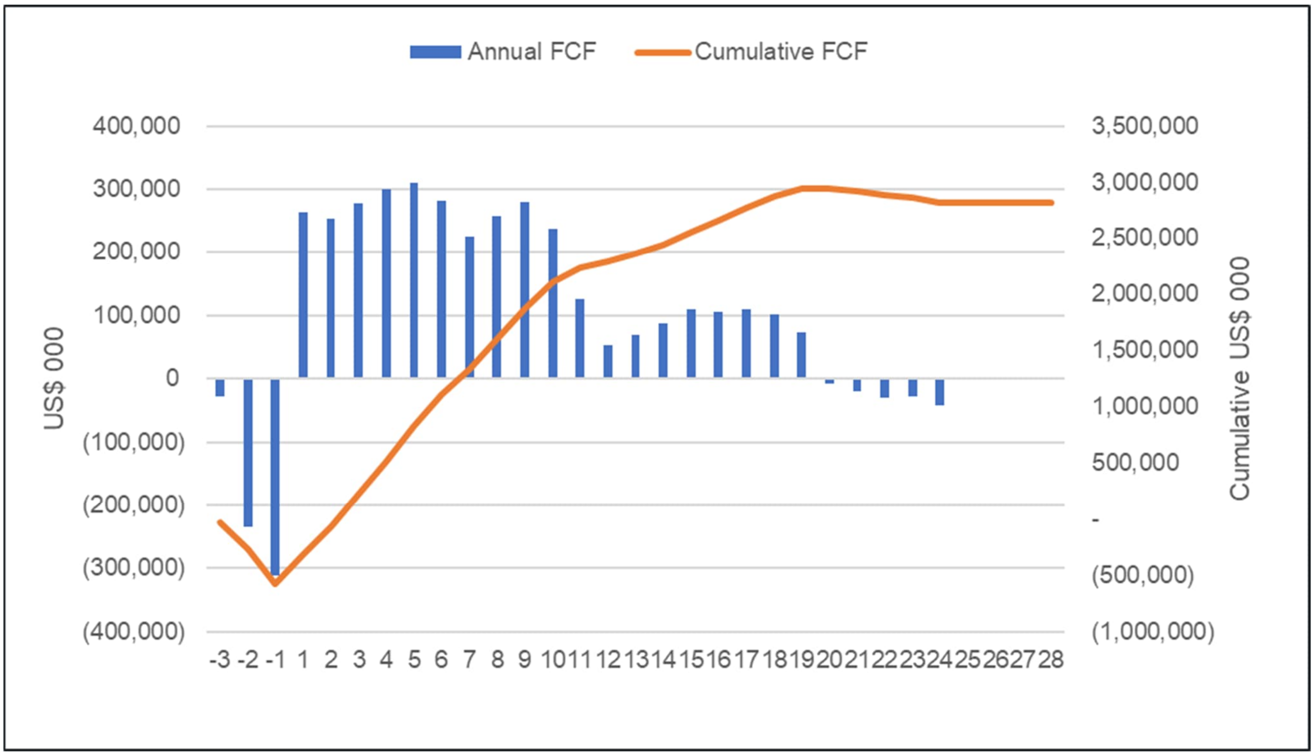


Figure 22-2 – Free Cashflows

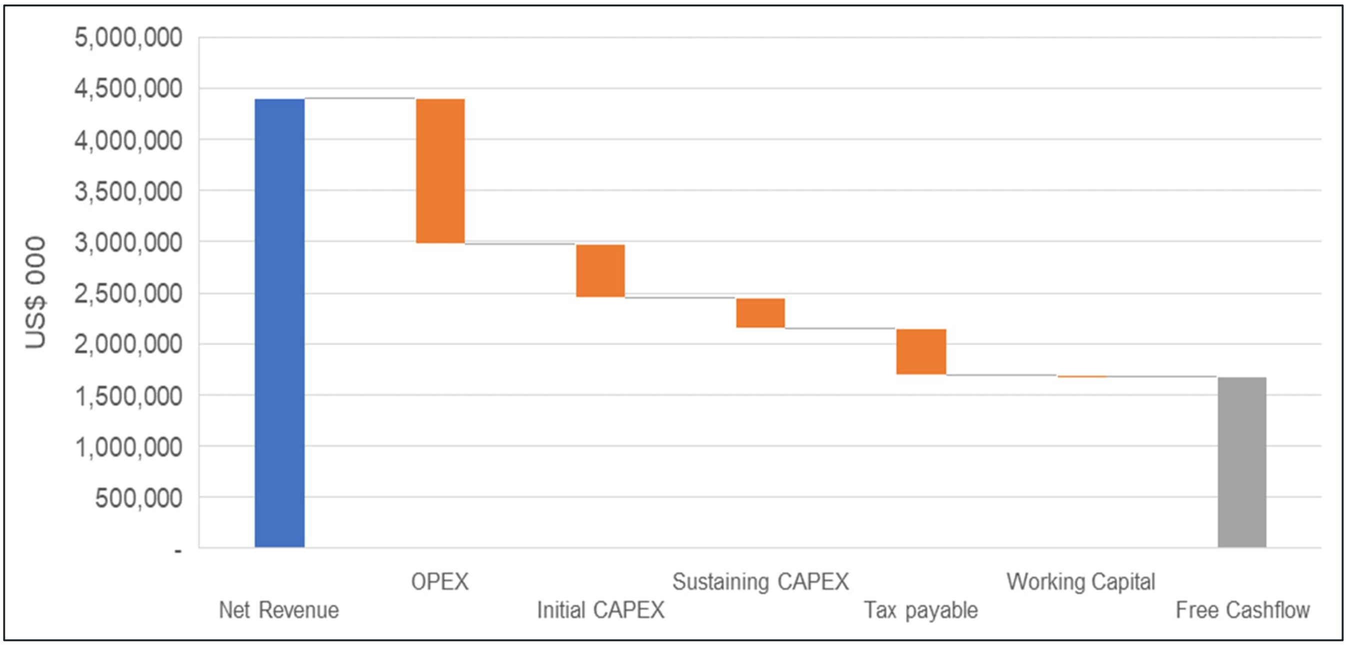


Figure 22-3 – NPV Waterfall

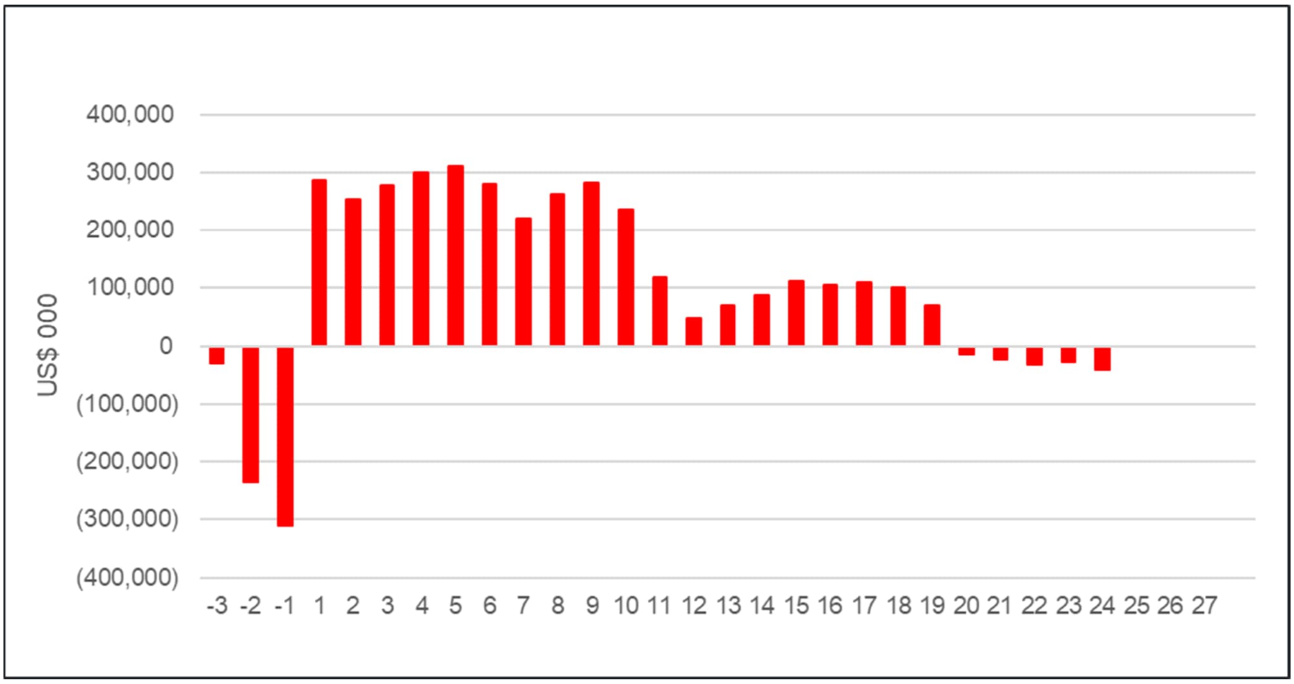


Figure 22-4 – Post-Tax Cashflows

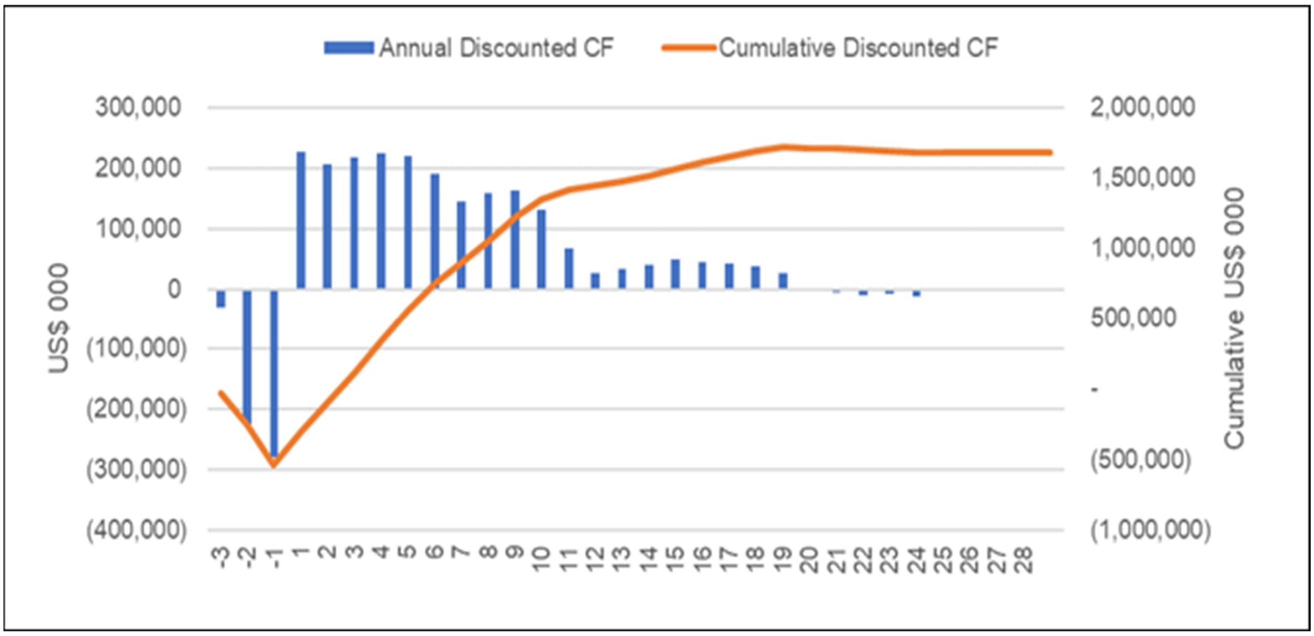


Figure 22-5 – Discounted Cashflows

Annual project cashflows are shown in Table 22-3 and
Table 22-4.

Table 22-3 – Ikkari Project Cashflow at $2 150/oz Summary Part 1

Description	Units	Total	-3	-2	-1	1	2	3	4	5	6	7	8
Production													
ROM Ore	kt	**51 998**	0	0	0	3 512	3 525	3 525	3 485	3 108	3 474	3 500	3 500
ROM Grade	g/t	**2.09**	0	0	0	2.01	2.03	2.08	2.23	2.56	2.22	1.76	2.03
Contained Gold	koz	**3 492**	0	0	0	227	230	236	250	256	248	198	229
Payable Gold	koz	**3 343**	0	0	0	216	220	225	238	244	236	188	218
Revenue													
Gross Revenue	$ 000	**7 187 705**	0	0	0	466 866	473 698	485 250	514 297	527 033	509 870	406 491	470 698
Royalties	$ 000	**(56 316)**	0	0	0	(3 658)	(3 711)	(3 802)	(4 030)	(4 129)	(3 995)	(3 185)	(3 688)
Other Selling Costs	$ 000	**(8 358)**	0	0	0	(543)	(551)	(564)	(598)	(613)	(593)	(473)	(547)
Net Revenue	$ 000	**7 123 031**	0	0	0	462 665	469 436	480 884	509 670	522 291	505 283	402 834	466 463
Operating Costs													
Open Pit Mining	$ 000	**(608 454)**	0	0	0	(51 082)	(74 292)	(73 461)	(72 983)	(73 015)	(70 648)	(56 855)	(47 484)
Underground Mining	$ 000	**(747 246)**	0	0	0	0	0	0	0	0	(5 409)	(4 559)	(13 808)
Co-Disposal Storage	$ 000	**(104 708)**	0	0	0	(8 146)	(11 171)	(10 694)	(9 983)	(9 419)	(9 715)	(8 183)	(7 125)
Water Management	$ 000	**(10 332)**	0	0	0	(517)	(517)	(517)	(517)	(517)	(517)	(517)	(517)
Concentrator & Filtration	$ 000	**(699 213)**	0	0	0	(41 765)	(41 876)	(41 876)	(41 531)	(38 314)	(41 437)	(41 662)	(41 662)
Closure	$ 000	**0**	0	0	0	0	0	0	0	0	0	0	0
Water Treatment	$ 000	**(108 127)**	0	0	0	(5 750)	(5 824)	(5 824)	(5 786)	(5 430)	(5 776)	(6 297)	(6 297)
Electrical Engineering	$ 000	**0**	0	0	0	0	0	0	0	0	0	0	0
Site G&A	$ 000	**(153 732)**	0	0	0	(7 522)	(7 522)	(7 522)	(7 522)	(7 522)	(7 522)	(7 522)	(7 522)
Total Operating Cost	$ 000	**(2 428 179)**	0	0	0	(114,780)	(141,202)	(139,894)	(138,322)	(134,217)	(141,023)	(125,596)	(124,415)
Operating Cashflow	$ 000	**4 689 220**	0	0	0	345 884	328 234	340 990	371 347	388 074	364 260	277 238	342 048
Capital Costs													
Initial	$ 000	**(575 363)**	(29 685)	(236 040)	(309 639)	0	0	0	0	0	0	0	0
Sustaining	$ 000	**(571 185)**	0	0	0	(32 386)	(23 366)	(6 027)	(7 349)	(12 469)	(24 736)	(14 008)	(24 463)
Net Cashflow	$ 000	**3 542 672**	(29 685)	(236 040)	(309 639)	313 498	304 869	334 963	363 998	375 605	339 524	263 230	317 585
Tax													

Description	Units	Total	-3	-2	-1	1	2	3	4	5	6	7	8
Depreciation	$ 000	**(1 146 548)**	0	0	0	(57 536)	(60 775)	(63 111)	(63 714)	(64 449)	(65 696)	(68 170)	(69 571)
Operating Profit	$ 000	**3 542 672**	0	0	0	288,348	267,459	277,879	307,633	323,625	298,564	209,068	272,478
Tax Payable	$ 000	**(724 028)**	0	0	0	(27 670)	(53 492)	(55 576)	(61 527)	(64 725)	(59 713)	(41 814)	(54 496)
Value Analysis													
Free Cashflow	$ 000	**2 818 644**	(29 685)	(236 040)	(309 639)	262 796	250 938	278 814	301 032	310 249	280 661	226 539	259 908
Cumulative Free Cashflow	$ 000	**2 818 644**	(29 685)	(265 725)	(575 363)	(312 568)	(61 630)	217 185	518 217	828 466	1 109 127	1 335 666	1 595 574
Discount Rate	**%**	**5**											
NPV	**$ 000**	**1 680 307**											
IRR	**%**	**38**											
Payback	**Years**	**2.2**											

Table 22-4 – Ikkari Project Cashflow at $2 150/oz Summary Part 2

Description	Units	Total	9	10	11	12	13	14	15	16	17	18	19	20>
Production														
ROM Ore	kt	**51 998**	3 600	3 473	2 204	1 992	2 000	1 967	2 050	1 946	1 903	1 783	1 179	272
ROM Grade	g/t	**2.09**	2.16	2.24	2.35	1.86	1.81	1.9	1.99	1.97	2.04	2.06	2.11	2.3
Contained Gold	koz	**3 492**	250	250	167	119	117	120	131	123	125	118	80	20
Payable Gold	koz	**3 343**	239	239	159	114	111	114	125	118	119	113	76	19
Payback	Years	2.2												
Gross Revenue	$ 000	**7 187 705**	514 513	514 513	342 892	245 109	240 121	246 834	269 695	253 622	257 243	243 326	164 277	41 356
Royalties	$ 000	**(56 316)**	(4 031)	(4 031)	(2 687)	(1 920)	(1 881)	(1 934)	(2 113)	(1 987)	(2 016)	(1 906)	(1 287)	(324)
Other Selling Costs	$ 000	**(8 358)**	(598)	(598)	(399)	(285)	(279)	(287)	(314)	(295)	(299)	(283)	(191)	(48)
Net Revenue	$ 000	**7 123 031**	509 884	509 884	339 807	242 903	237 960	244 613	267 269	251 340	254 929	241 137	162 799	40 984
Operating Costs														
Open Pit Mining	$ 000	**(608 454)**	(40 145)	(38 226)	(9 798)	(88)	(22)	(49)	(61)	(85)	(59)	(85)	(13)	0
Underground Mining	$ 000	**(747 246)**	(8 959)	(50 553)	(76 838)	(88 164)	(94 220)	(78 479)	(74 084)	(67 147)	(70 810)	(54 248)	(39 666)	(20 303)
Co-Disposal Storage	$ 000	**(104 708)**	(6 560)	(6 179)	(2 604)	(2 020)	(1 699)	(1 680)	(1 824)	(1 883)	(1 721)	(2 076)	(1 414)	(612)
Water Management	$ 000	**(10 332)**	(517)	(517)	(517)	(517)	(517)	(517)	(517)	(517)	(517)	(517)	(517)	(517)

Description	Units	Total	9	10	11	12	13	14	15	16	17	18	19	20>
Concentrator & Filtration	$ 000	**(699 213)**	(31 134)	(30 816)	(31 619)	(30 615)	(30 197)	(29 036)	(23 177)	(14 386)	(31 134)	(30 816)	(31 619)	(30 615)
Closure	$ 000	**0**	0	0	0	0	0	0	0	0	0	0	0	0
Water Treatment	$ 000	**(108 127)**	(6 392)	(6 327)	(4 928)	(4 727)	(4 735)	(4 704)	(4 783)	(4 685)	(4 644)	(4 530)	(3 959)	(6 727)
Electrical Engineering	$ 000	**0**	0	0	0	0	0	0	0	0	0	0	0	0
Site G&A	$ 000	**(153 732)**	(7 522)	(7 522)	(7 522)	(7 522)	(7 522)	(7 522)	(7 522)	(7 522)	(7 522)	(7 522)	(7 472)	(10 869)
Total Operating Cost	$ 000	**(2 428 179)**	(112 611)	(150 753)	(135 318)	(134 090)	(139 849)	(123 767)	(120 409)	(112 453)	(115 469)	(98 015)	(76 217)	(53 414)
Operating Cashflow	$ 000	**4 689 220**	397 273	359 130	204 489	108 814	98 112	120 846	146 860	138 887	139 459	143 122	86 583	(12 430)
Capital Costs														
Initial	$ 000	**(575 363)**	0	0	0	0	0	0	0	0	0	0	0	0
Sustaining	$ 000	**(571 185)**	(49 869)	(66 009)	(50 855)	(45 337)	(15 650)	(15 126)	(12 540)	(13 203)	(8 523)	(20 136)	(6 028)	(123 105)
Net Cashflow	$ 000	**3 542 672**	347 404	293 121	153 634	63 477	82 462	105 720	134 320	125 684	130 936	122 987	80 554	(135 535)
Tax														
Depreciation	$ 000	**(1 146 548)**	(72 017)	(77 004)	(26 068)	(27 915)	(30 112)	(31 075)	(31 852)	(31 859)	(30 706)	(30 157)	(29 725)	(215 036)
Operating Profit	$ 000	**3 542 672**	325 256	282 126	178 421	80 898	68 000	89 771	115 008	107 028	108 754	112 965	56 858	(227 466)
Tax Payable	$ 000	**(724 028)**	(65 051)	(56 425)	(35 684)	(16 180)	(13 600)	(17 954)	(23 002)	(21 406)	(21 751)	(22 593)	(11 372)	0
Value Analysis														
Free Cashflow	$ 000	**2 818 644**	280 182	236 696	126 454	52 143	69 109	87 433	110 185	105 075	109 006	101 083	73 100	(127 395)
Cumulative Free Cashflow	$ 000	**2 818 644**	1 875 756	2 112 452	2 238 906	2 291 048	2 360 157	2 447 590	2 557 776	2 662 851	2 771 857	2 872 940	2 946 039	2 818 644
Discount Rate	**%**	**5**												
NPV	**$ 000**	**1 680 307**												
IRR	**%**	**38**												
Payback	**Years**	**2.2**												

22.3 SENSITIVITY ANALYSIS

Sensitivity analysis was completed to identify key variables that have a significant impact on the Project returns. The sensitivity analysis independently varied the following parameters:

- Gold selling price;
- Operating costs; and
- Capital costs.

Each parameter was varied by -30% to +30% and the resulting NPV and IRR was charted. Results at the PFS base case are shown in Figure 22-6 and Figure 22-7. As shown in the figure, project NPV and IRR is most sensitive to gold price.

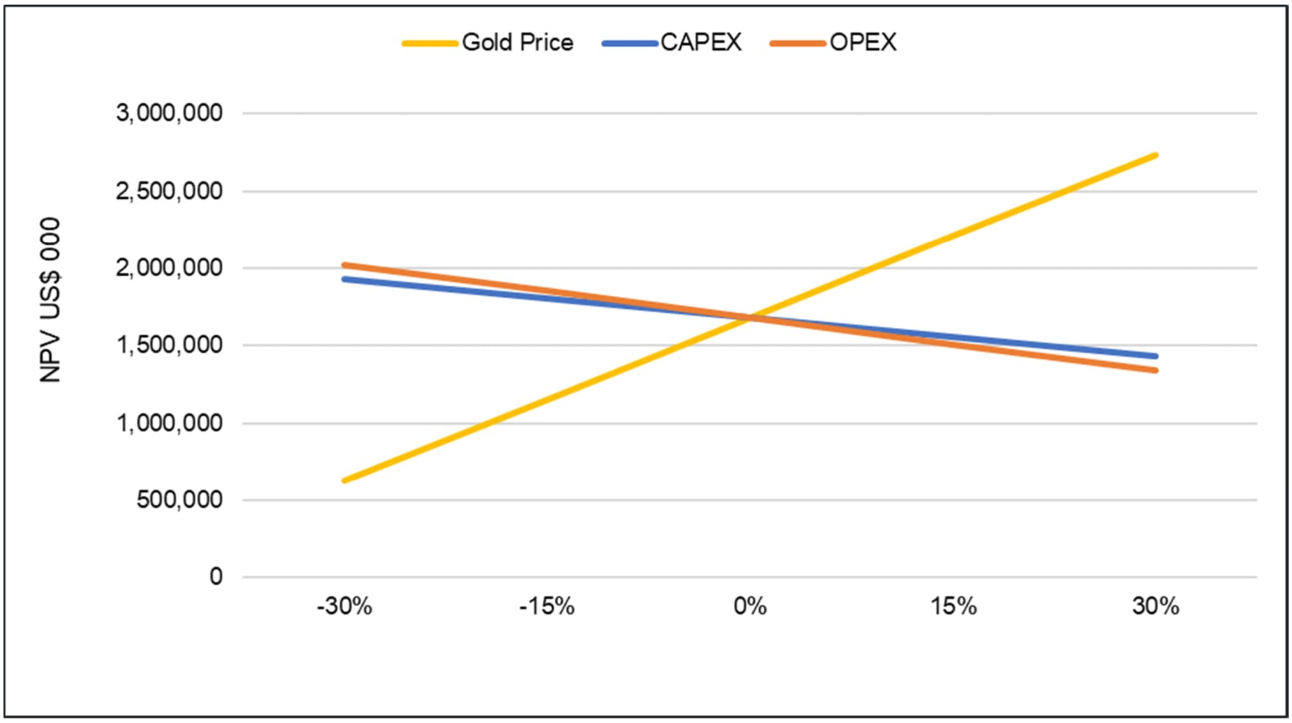


Figure 22-6 – Ikkari NPV Sensitivity Analysis

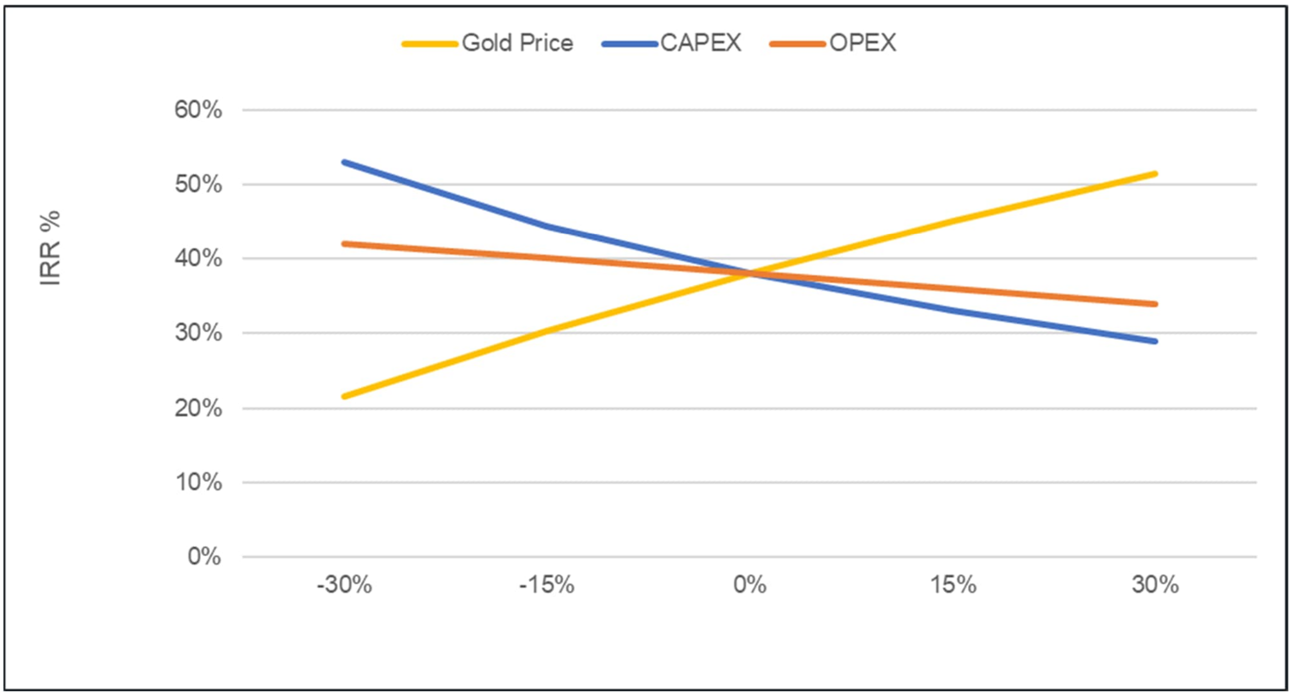


Figure 22-7 – Ikkari IRR Sensitivity Analysis

Sensitivity analysis was also performed for different gold price scenarios, with the results shown in the table below.

Table 22-5 – Financial Results for Varying Gold Prices

Gold Price [$/oz]	NPV [$ 000]	IRR [%]	Payback [Years]
1 500	617,427	21	3.7
1 700	944,489	27	3.1
2 000	1,435,034	35	2.4
2 150	**1,680,307**	**38**	**2.2**
2 650	2,497,882	49	1.7
3 000	3,070,185	55	1.4

23 ADJACENT PROPERTIES

The information in this section that relates to adjacent properties is derived from public domain information and the QP has not verified this information. As of any information related to mineral reserves or resources of adjacent properties written in this chapter WSP is not liable for the accuracy of the information, the sources are listed under the tables and in the reference list. The reader is advised to visit the companies' websites for latest press releases regarding mineral resources and reserves.

23.1 THIRD PARTY PROJECTS – INTRODUCTION

Several significant mineral discoveries have been made in the CLB, namely Suurikuusikko, better known internationally as the Kittilä gold mine and Sakatti, an orthomagmatic, polymetallic base metals deposit. Kevitsa, also a polymetallic orthomagmatic deposit discovered in 1987 entered production in 2012. Since 2015, several major mining groups have made strategic investments in the region and promising early-stage discoveries have been made at Aamurusko, Kutuvuoma and Helmi (gold) (Figure 23-1).

Table 23-1 summarises the Mineral Reserve and MREs of these deposits based on publicly available information. These estimates are not necessarily representative of the mineralisation for the Ikkari project and the QP has not verified this information.

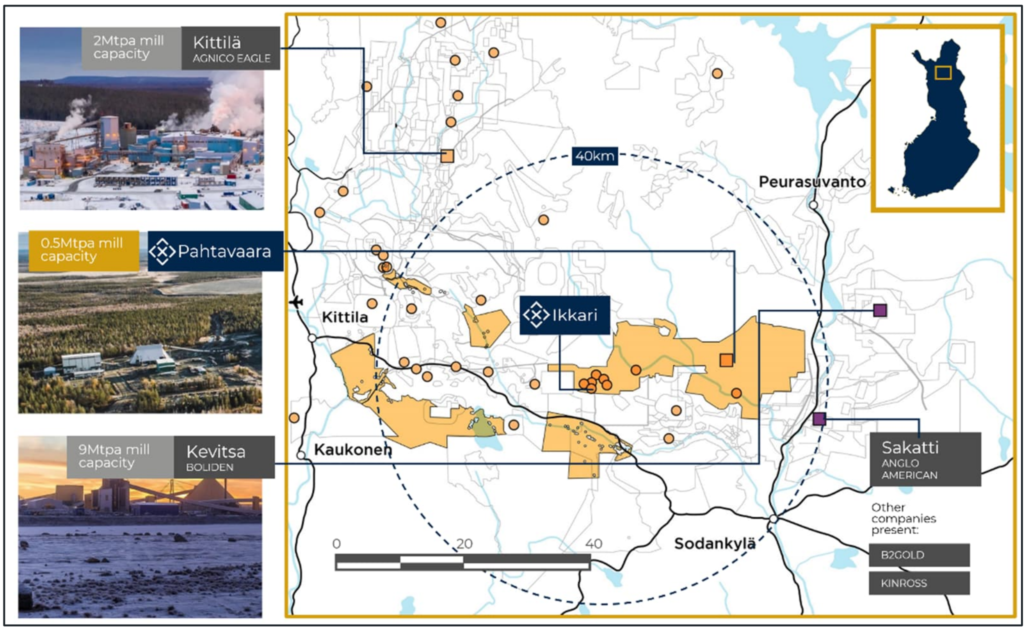


Figure 23-1 – Recent Activity in Central Lapland

Table 23-1 – Mineral Reserves and Resources of Adjacent Properties in CLB 2023

Deposit	Type	Mt	Au (g/t)	Cu (%)	Ni (%)	Co (%)	Pt (g/t)	Pd (g/t)
Reserves								
Kevitsa (Boliden)	Proven	48	0.10	0.31	0.20**	0.01***	0.20	0.13
	Probable	34	0.08	0.32	0.21**	0.01***	0.17	0.11
Kittilä (Agnico Eagle)	Proven	0.984	4.11	-	-	-	-	-
	Probable	25.943	4.14	-	-	-	-	-
Resources								
Kevitsa (Boliden) *	Measured	61	0.09	0.34	0.23**	0.01***	0.17	0.11
	Indicated	106	0.07	0.36	0.24**	0.01***	0.12	0.07
	Inferred	0.3	0.04	0.22	0.13**	0.01***	0.06	0.03
Kittilä (Agnico Eagle) *	Measured	4.299	2.91	-	-	-	-	-
	Indicated	13.632	2.93	-	-	-	-	-
	Inferred	6.565	5.06	-	-	-	-	-
Sakatti (Anglo American) *	Indicated	3.5	0.33	3.45	2.47	0.11	0.98	1.18
	Inferred	40.9	0.33	1.77	0.83	0.04	0.61	0.43

Note: * Mineral Resources are reported exclusive of Mineral Reserves (see references, Boliden 2023; Anglo American plc, 2023; Agnico Eagle 2023).

** reported as NiS

*** reported as CoS

23.2 KITTILÄ MINE (AGNICO EAGLE)

The Kittilä mine is located in the Lapland region of northern Finland, approximately 900 km north of Helsinki and 150 km north of the Arctic Circle. The Kittilä mine is the largest gold mine in Europe and annually extracts about 2.0 million tonnes of ore, yielding about 7000 kg of gold annually. With a mine life estimated through 2035, its proven and probable mineral reserves contain 3 584 Moz gold (26 926 Mt at 4.14 g/t Au) as of December 31, 2023. Ore has been mined from underground since 2010 and the mine produced 234 402 oz of gold in 2023. (Agnico Eagle Finland, 2025 and Agnico Eagle, 2025)

First gold was poured at Kittilä on January 14, 2009 and commercial production was achieved on May 1, 2009. The Kittilä orebodies were initially mined from two open pits – Suuri (beginning in 2008) and Roura (2010) – with underground operations added in October 2010. The open pits were mined out in November 2012, and since then, mining has been entirely underground in the Suuri, Roura and Rimpi areas of the Main Zone. In February 2018, the Board approved an expansion to increase throughput rates at Kittilä to 2.0 million tonnes per year from 1.6 million tonnes per year. This expansion included the construction of a 1 044-metre-deep shaft, a processing plant expansion as well as other infrastructure and service upgrades. Shaft sinking was completed in the third quarter of 2022 and the construction and commissioning of a nitrogen removal plant was completed in the fourth quarter of 2022. The installation and commissioning of the production and service hoists was completed in the third quarter of 2023.This increased mining rate will be supported by the development of the Sisar Zone and deeper portions of the Main Zone. (Agnico Eagle Finland, 2025 and Agnico Eagle, 2025)

23.3 KEVITSA MINE (BOLIDEN)

Boliden Kevitsa in Sodankylä is one of the biggest open pit mines in Finland. The main products are nickel and copper concentrates containing also platinum, palladium, gold and cobalt. The Kevitsa open pit mine in northern Finland was acquired by Boliden in June 2016. The operation, which comprises a mine and a concentrator, went into operation in 2012. The mined-out ore tonnage for 2023 was 9.405 Mt. Total mined material (ore and waste) was 36.408 Mt at 2023. Concentrates are trucked to the Gulf of Bothnia approximately 300 km south of the mine from where it is shipped to Boliden's Harjavalta smelter in southwestern Finland. (Boliden, 2025).

23.4 SAKATTI PROJECT (ANGLO AMERICAN)

The Sakatti Project is a Copper – Nickel – Platinum Group Elements (PGE) deposit that was discovered by Anglo American in 2009 and is one of the richest multi-metal deposits in Europe. The deposit is located 15 km north of Sodankylä, and the area is partly located in Viiankiaapa, a protected mire and a Natura 2000 designated area. Anglo American recommenced drilling of the project in the winter of 2016 and announced a maiden resource for the project in 2017. Anglo American commenced a PFS for the project in early 2017, which was completed in 2019. (Anglo American, 2025)

An exploration permit and a permit from the Environmental Ministry for the exploration work at Sakatti was awarded during July 2020. The Lapland Centre for Economic Development, Transport and the Environment (ELY Centre) has granted approval of the Sakatti Environmental and Social Impact Assessment (EIA) in August 2023, marking a significant milestone for the project. The Natura 2000 assessment requires an update during the next permitting stages. The project encompasses 10 valid permits covering 10 614 ha and 19 renewal applications covering 13 657 ha. (Anglo American, 2023).

23.5 AURION RESOURCES PROJECTS IN FINLAND

Aurion Resources Ltd. (Aurion) began operating in Finland in early 2014 and currently holds, or has interests in, mineral tenements that cover approximately 75 000 hectares (ha) of the Central Lapland Greenstone Belt (CLB) of the Fennoscandian Shield.

Aurion owns the Risti and Launi West Projects, which covers approximately 25 000 ha of the Central Lapland Greenstone Belt. Gold occurrences identified to date include: Aamurusko Main and Aamurusko NW where follow-up drilling of gold-bearing boulders has intersected gold bearing quartz veins in bedrock including some bonanza grade, narrow intercepts. The Kaaresselkä gold prospect discovered by GTK in the 1980's also occurs on the Risti Property as is the focus of current exploration drilling. Recently, Aurion announced an option agreement with Kinross for its Launi East Project. (Aurion Resources, 2025).

23.6 B2GOLD-AURION JOINT VENTURE

Exploration licences and exploration licence application held by B2Fingold, a joint venture held 70% by B2Gold and 30% by Aurion Resources covers approximately 293 km^2 of ground to the west of Rupert's Lapland Gold Project and adjoins the Rupert Resources project at its western boundary.

It includes several discoveries such as Helmi (highlight intercept: 2.05 g/t Au over 77.50 m) and Kutuvuoma (highlight intercept: 16.47 g/t Au over 11.0 m) which occur 1.3km and 8km to the west of Ikkari respectively.To date publicly available information indicates that B2Fingold have drilled at least 44 holes into the Helmi prospect and a further 40 holes to Kutuvuoma. (Aurion Resources, 2025).

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT EXECUTION

24.1.1. CONTEXT AND ASSUMPTIONS

A WBS Level 2 implementation schedule has been developed based on key preconstruction activities, the local climate and saturated ground conditions which combine to control the seasonal scheduling of construction activities:

- Mid-April to End June- High melt water flows and thawing ground;
- End June to Start of November - Low water flows and unfrozen ground; and
- Start of November to Mid-April – Frozen ground.

24.1.2. CRITICAL PATH AND KEY SCHEDULE DRIVERS

The WBS Level 2 implementation schedule (Appendix 2) estimates following schedule:

- Construction Phase **2.75 years; and**
- Operation Phase **20 years.**

The key schedule drivers within this schedule have been determined as follows:

24.1.3. PERMITTING

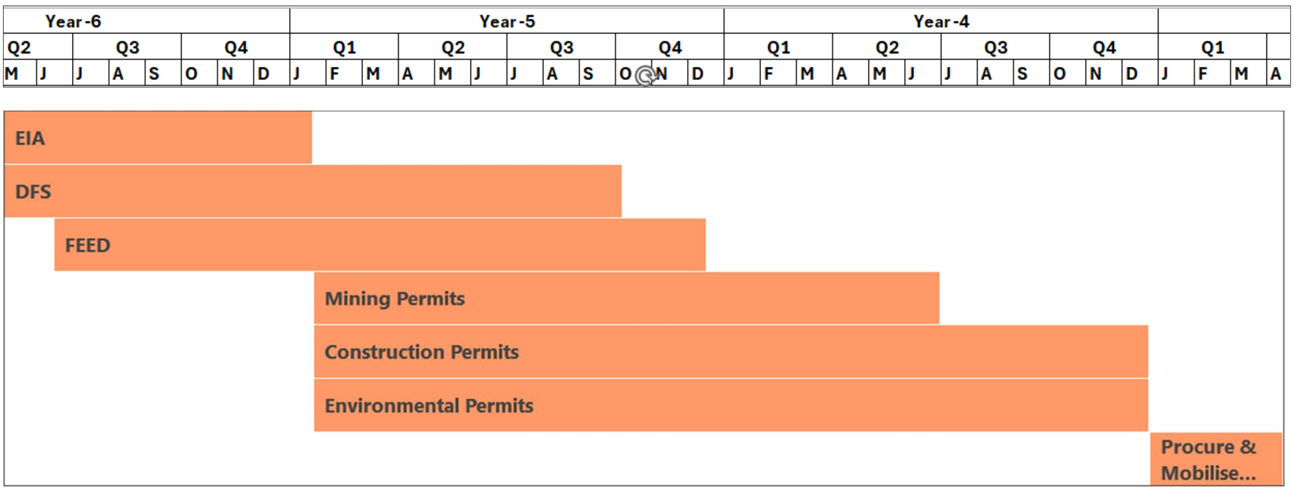


Figure 24-1 – Permitting schedule

The preconstruction critical path runs through the permitting process and cannot begin until sufficient definition of the Environment Impact Assessment has taken place (Figure 24-1). Further engagement with the permitting authorities and work package planning will be required to refine this estimate as the mineral property project develops. Early focus on EIA definition and overlapping the next study stages (Definitive Feasibility Study, AACE Class 3, FEL 3) with Front-End Engineering Design provides the opportunity to minimise the pre-construction critical path.

24.1.4. PROCESS PLANT

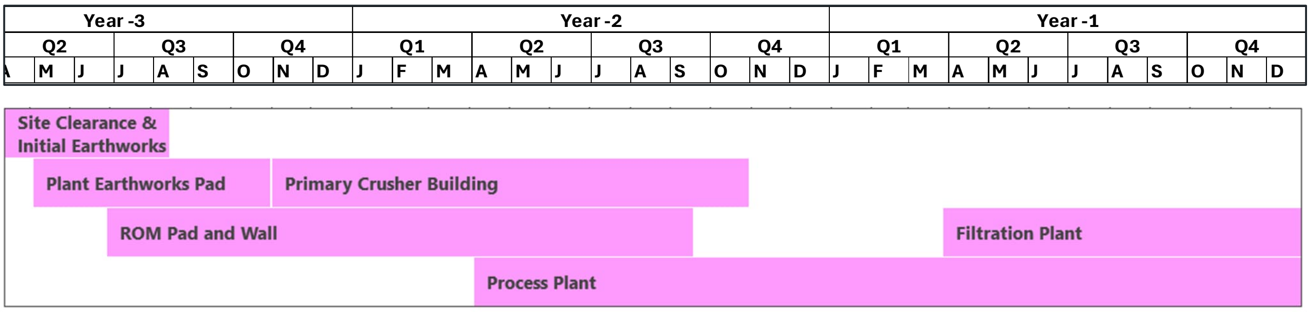


Figure 24-2 – Process plant schedule

The Construction Phase critical path runs through the Mineral Processing Plant (Figure 24-2). In future, the detailed sequencing of the ROM wall and primary crusher building are to be examined, as phased construction of each activity is assumed to enable plant construction to overlap these tasks, reducing overall construction duration.

24.1.5. RIVER DIVERSION AND PIT DEWATERING

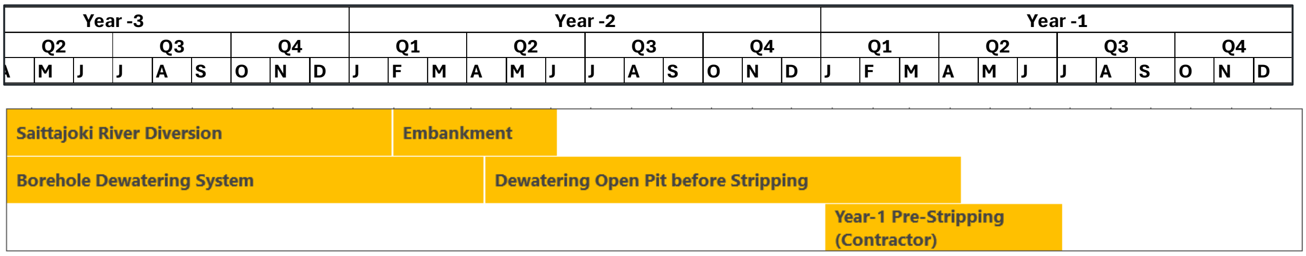


Figure 24-3 – River Diversion and Pit Dewatering schedule

Site dewatering, especially the open pit area, is a key schedule driver. The time required for open pit dewatering before stripping is not yet fully defined, potentially being 6 to 12 months, and is assumed not to begin until the River Diversion works are complete (Figure 24-3. Further technical studies will define estimates for these works and schedule.

24.1.6. MINE EQUIPMENT

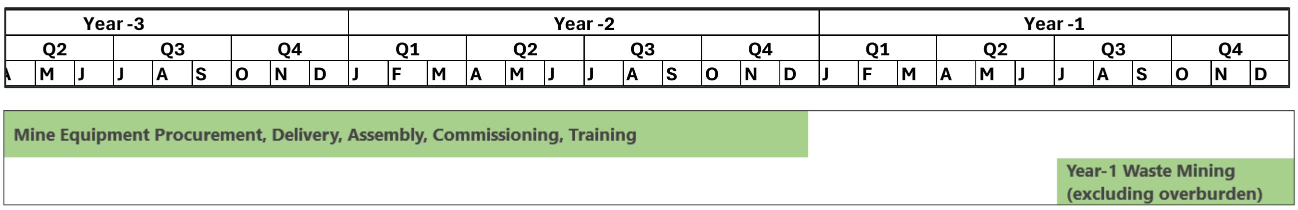


Figure 24-4 – Mine equipment schedule

Waste mining in production year -1 requires mobile mining equipment with an expected long lead time for equipment delivery (Figure 24-4). It is assumed that the project will utilise contract mining for the open pit to mitigate this risk but should be revisited by the operational contract strategy in the next stage of implementation planning.

24.1.7. WATER TREATMENT

The construction of the Water Treatment Plant and Treated Water Discharge Pipeline are relatively long lead items, both estimated to take two years and are closely linked to the Mineral Process Plant and site establishment schedules. Whilst initial scheduling works do not show these as being on the critical path, they present a risk to the overall project timeline and further analysis should consider early construction. Furthermore, the site operating permit terms and conditions agreed with the government based on an agreed EIA may also impact on the water treatment plant establishment timelines and should be re-examined at the next stage.

24.1.8. MATERIAL VOLUME (CUT AND FILL) BALANCE

Site construction involves significant material movement for flattening topography, building embankments, primary and secondary drainage, temporary works such as construction roads, potable water and sewage treatment plants, areas for cranage, material laydown and creating ponds to capture all site drainage. Without local material sources, material scheduling, logistics and value chain mapping is crucial to minimise costly imported fill. The lack of available material could delay critical tasks, like forming the Mineral Process Plant foundation pad or the Co-Disposal facility foundation. Optimising cut and fill volumes and material suitability should be addressed in the next study stage. Particular attention should be given to the Co-Disposal facility where early material requirements may be met by bringing activities that yield site won materials forward (e.g. pond construction).

24.2 CONTRACTING MODELS

The schedule assumes an EPCM strategy is adopted. RR have at the PFS stage maintained a range of options for implementation and not confirmed their strategy. The main options available can be summarised as follows:

EPC (Engineering, Procurement, and Construction) involves the contractor taking full responsibility for design, procurement, and construction. The contractor delivers the project as a turnkey solution, reducing the client’s day-to-day involvement and shifting risk to the contractor. This model is beneficial when a fixed price and timeline are preferred.

EPCM (Engineering, Procurement, and Construction Management), the contractor manages engineering, procurement, and construction services but does not directly perform construction. The client retains control over project execution and hires subcontractors for construction tasks. This model offers flexibility as the project can begin before full engineering is completed.

The procurement and contracting strategy will also influence cost and schedule control, risk control allocation and influence the management of the critical path to first gold poured. An extension of the EPCM model is integration of FEED and a nuanced use of a Project Management Contractor for control of sub-contractors and risk management.

24.3 OPTIMISATION STUDIES

During the PFS study phase, several optimisation studies have been conducted. These are listed below:

- Underground Mining: Mining method trade-off (SLC vs LHOS vs Combined), fleet power, mine access, and production rate;

- Open pit mining: Initial open pit optimisation runs with few different block model versions;
- Open pit optimisation: for MRE RPEE;
- Open pit optimisation: to support final pit design for Mineral Reserve estimation;
- Iteration No 1. MSO optimisation for UG mine design;
- Iteration No 2. MSO optimisation for UG mine design; and
- Enterprise value optimisation. Open pit and underground optimisations to study alternative material handling, stockpiling, cut-off-grade and mine scheduling strategies which have used the theory of constraints to examine maximisation of enterprise value
- Trade-of-study on processing options (flotation followed b leach, whole ore leach, leach followed by flotation);
- Gravity circuit optimization study.

25 INTERPRETATION AND CONCLUSIONS

25.1 INTERPRETATION AND CONCLUSIONS

25.1.1. MINERAL RESOURCE ESTIMATE CONCLUSIONS

It is the Qualified Persons (QP) opinion that the exploration, drilling and analytical procedures used by Rupert Resources to collect geological data are consistent with industry practises and CIM Mineral Exploration Best Practise Guidelines (November 2018) and that the data is suitable for the reporting of the MRE as summarized in this Technical Report.

The MRE for the Ikkari Project has been estimated in conformity with November 2019 CIM "Estimation of Mineral Resource and Mineral Reserves Best Practice" guidelines.

The QP has taken reasonable steps to construct the computational block model based on the interpretation of the site geology and the MRE is representative of the project data, but notes that there are risks related to the accuracy of the estimates related to the following:

- The assumptions used by the QP to prepare the data for resource estimation;
- The accuracy of the geological interpretations of lithology, structural controls and mineralisation;
- Estimation parameters used by the QP;
- Assumptions and methodologies used to estimate SG;
- Orientation of drill holes; and
- Cut-off grade and related assumptions of commodity prices, mining costs and metallurgical recovery.

For these reasons, actual results may differ materially from the reported MRE.

25.1.2. MINERAL RESERVE ESTIMATE CONCLUSIONS

Mineral Reserves were estimated in accordance with the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines. The disclosure of the Reserve Estimate uses the NI 43-101 guidelines and has excluded the use of Inferred Mineral Resources.

Reserves are derived from the proposed open pit and underground mining areas. The open pit is based on a conventional truck and shovel operation with conventional grade control processes. Underground mining implements a Long Hole Open Stoping (LHOS) mining method with a hybrid of waste development (sub surface infrastructure) rock or paste backfill manufactured from the mineral process plant tailings.

Mineral Reserves were estimated based on the Resource contained within the open pit and underground mine designs with allowances for dilution and recovery losses. The Mineral Reserve was estimated at 52 Mt at an average grade of 2.1 g/t for 3.5 million ounces.

25.1.3. ROCK MECHANICS CONCLUSIONS

Open pit stability was analysed with kinematic analyses to determine bench scale susceptibility to structural failures. Based on the stereographic projection analysis of logged joint orientation data, the lithological domains in the open pit area generally only have two relatively steeply dipping joint orientations, of which one lines up with foliation orientation. Bench failure modes are mainly toppling and planar failures (mainly South-East and East walls). Expected failure volumes are small and most of the failures are dominated by strong foliation.

Two-dimensional finite element analysis was performed to analyse large scale stability against failure. The simulation was performed using shear strength reduction method for saturated and drained slope models, separately for each open pit sector. The results demonstrate stable slopes within 45° to 55° overall slope angle, depending on the pit sector. Three-dimensional analysis is recommended in the next study phase to further optimize the pit angles. The ramp placement is recommended on the North wall due to lower rock mass quality and vicinity of property boundary on the South side of the planned open pit.

High-level rock mechanical analyses supported underground trade-off studies between Long Hole Open Stoping (LHOS) and sub-level caving (SLC). LHOS was selected as the go-forward case. Rock mechanics inputs for underground stope design have been selected to achieve a reliable and robust mine design and sequence using rock mass qualities between the 25th and 50th percentile. Maximum induced stresses on stope surfaces have been assessed with 3D numerical modelling, considering interaction between mining areas and the open pit. Two empirical design methodologies and associated criteria have been used in parallel, resulting in stable stope sizes for primary and secondary stope lines. Different lithologies and sectors within the underground orebody were considered in design recommendations. Dilution from sidewalls has been estimated based on recorded case histories at other mines.

Ground support estimates are provided for mining drives, intersections, and stope backs. They should be seen as input to economic assessments for this PFS and are not designed.

Pastefill strength requirements for primary and secondary stope lines are provided. The study assessed advantages and disadvantages of various stope sequences which have been considered with the mine design team. A general triangular retreat shape using a primary and secondary stope arrangement was selected with a mining direction away from the Southern fault zone.

25.1.4. MINING METHODS CONCLUSIONS

The Ikkari project is a combined open pit and underground gold mine with the total life of mine (LOM) at 20 years. LHOS was implemented for the underground mine design and schedule.

Overburden overlays the open pit mining area, with hard rock underneath this requiring blasting to be mined. A traditional truck and shovel configuration has been selected for open pit operations.

Drilling and blasting are planned on 10m bench heights, with double benching utilised for permanent and semi-permanent pit walls in areas where the rock mass quality is sufficient. The drilling and blasting arrangements support conventional grade control methods of working.

The open pit consists of two stages, with the ramp placed on the northern wall for both. Stage 1 is based on the Revenue Factor (RF) 0.17 optimisation shell. Stage 2 is based on RF 0.80 optimisation shell.

The Mineral Reserve was estimated at 52 Mt at an average grade of 2.1 g/t for 3.5 million ounces.

The open pit material handling schedule includes 130 Mt of waste which is mined with 17 Mt of this consisting of overburden material. A 3.5 Mtpa production rate is forecast in the open pit mining schedule.

Access to the underground mine was designed as follows:

- North ramp access with a portal located on the surface (220 m elevation) to the East of the open pit to provide access for mining equipment, personnel, services, and as part of the primary ventilation circuit. This is the initial and primary access to the underground mine; and
- South ramp access with a portal located on the 40 m elevation switchback inside the open pit. This will be developed after the open pit mining is completed.

Material from the underground mine will be hauled to the mineral process plant / RoM stockpile via trucks using these declines and site roads. The sub surface infrastructure development is designed at a 5.0 mW x 5.15 mH with an arched profile.

Ventilation is provided through four shafts: two for exhaust and two for the provision of fresh air and the two declines. Fresh air will be heated when required. Ventilation shafts vary between 2.5 to 4.0 m in diameter.

Underground stopes dimensions were designed based on the geotechnical inputs. Horizontal levels are spaced 30 m vertically apart, with stopes on the level spaced at 15 m. Stopes were generated using Datamine's MSO tool. Unplanned dilution levels are accounted for through ELOS and varies depending on location and depth in the mine. Stope recovery is set at 96% for most stopes, with stopes that are located under the open pit having a reduced recovery of 86%.

A primary-secondary stoping sequence is planned. This entails mining of the primary stopes and leaving at least one stope width between as a supporting pillar. At the completion of mining and curing of backfill in the primary stope on either side of the pillar, mining of the secondary stope may be commenced.

The underground mine development commences in Year 6 with the construction of the North ramp access. The ramp will proceed down to the -140 m elevation where the primary aim is to establish the ventilation circuit infrastructure and commence the central decline down to -320 m elevation.

Stoping commences in Year 10 in a bottom-up sequence to enable maximisation of extraction per period, with the underground mine ending in Year 20. At its peak, the underground mine will produce just over 2 Mtpa.

A 3.5 Mtpa production rate can be sustained through open pit mining. Underground mining at Ikkari by LHOS can sustain 2 Mtpa production rate, hence a reduction in processing throughput occurs in Year 10 to Year 20 where mining transitions underground.

25.1.5. PASTE BACKFILL PLANT DESIGN CONCLUSIONS

Laboratory test work was performed on tailings samples which provided the preliminary data for paste backfill recipe development and backfill material properties. This data was used to size the paste plant and underground paste distribution system. The paste plant nominal design capacity is 2.0 Mtpa.

The paste plant receives filter cake from the filtration plant and mixes it with binder to produce the various strength backfill recipes required underground. The paste plant utilises a twin shaft mixer to blend the constituent ingredients to the required slump.

The underground paste distribution system begins at a borehole situated at edge of the pit. To optimise the delivery of filter cake to the process, the paste plant was located adjacent to the filter plant rather than at the borehole. A 350 m surface pipeline is required to deliver paste to the borehole collar. The surface pipeline precludes the use of gravity distribution of paste. Piston type positive displacement pumps are required to delivery paste from the plant to the stopes underground.

The paste plant process developed, and equipment utilised is typical with many existing plants in operation.

25.1.6. MINE WATER MANAGEMENT CONCLUSIONS

To facilitate the mining operation and minimise the risk of polluting the Saittajoki River it is proposed to divert both the tributary and the main river channel around the mine site.

To minimize the amount of surface water runoff that needs treatment it is necessary to separate contact and non-contact water on the mine. Where runoff that has been in contact with ore or mine waste is considered contact, while runoff from undisturbed natural catchments is considered to be non-contact. Runoff from non-contact catchments will be diverted around the mine in open channels, while that from contact catchments will be captured and treated or used as process water. Runoff from developed areas of the mine, that are not considered contact catchments, will be passed through sediment traps before being discharged to the environment. The surface water inputs have been estimated based on the final mine development extent.

Snow will need to be cleared from areas such as the pit, ROM Pad and roads etc. to allow mine operations to progress unhindered. This will need to be stockpiled and managed appropriately.

Contact water management ponds are required to balance peak inflows during wet periods. There are two contact water ponds proposed, the Co-Disposal Runoff Collection Pond (Co-Disposal Pond) and the Raw Water Pond. A Treated Water Pond has also been provided to provide some residence time before treated water is discharged via the pipeline to River Kitinen, as well as to allow sufficient storage in the event the pipeline is shut down.

The Co-Disposal Pond will have a capacity of 440 000 m^3, whereas the Raw Water Pond will have a capacity of 600 000 m^3. The Treated Water Pond has been sized with a total volume of 290 000 m^3 to give 14 days of treated water storage if the discharge pipeline is shut down.

The optimum combination of overall raw water treatment rate and pond size resulted in a 1:200 annual exceedance probability spill risk from the Raw Water Pond. This is equivalent to a 1:11 probability of one or more spills occurring over a 20-year life of mine. Significant dilution of contaminants during such a rare spill event, due to the large proportion of hydrological water in the system, would however further mitigate such a rare event.

Peripheral external pit dewatering from a minimum of 16 dewatering wells is the recommended method of dewatering the proposed Ikkari open pit to reduce treatment costs and assist in stabilising the pit walls. The initial dewatering pumping rate will peak at 688 m^3/h (16 500 m^3/day) during the first year of dewatering. The exact location of the dewatering discharge point will need to be determined in the next phase of the project and could be either the section of the Saittajoki River channel passing through the mine site, or the upstream river diversion channel. The maximum de-watering borehole discharge rate is considered to be insignificant when compared to annual high flows in the Saittajoki River during spring snow melt. It has been assumed that the ground water discharge quality will meet environmental permit limits. In the event that this is not the case provision has been made for two 20 000 m^3 lined ponds to store at least 2 days' worth of flow. Longer term management of contaminated ground water would need additional water treatment capacity, which has not been accounted for in the PFS. Further work is required to understand whether ground water can be discharged to the environment.

The internal Open Pit water management will include bench drains, draining to a pit sump. From the sump, pumps will pump contact water to the Raw Water Pond. Depending on the final pit depth, the in-pit dewatering might consist in multiple pumping stages to reach the pit crest. Perimeter drains will be included around the Open Pit to prevent runoff to the pit from the external catchment.

From the water balance the average wet month hydrological contribution to the Open Pit is estimated to be 1 210 m^3/day. Ground water inflows to the pit will increase annually as the pit deepens and expands to a maximum of 1 200 m^3/day by Year 8 before reducing to near zero over the remaining LOM.

It is currently assumed that the pit will be dewatered to prevent flooding from a runoff event more frequent than that with a 1:20 annual exceedance probability. This restriction was imposed to limit the size of the pumps and limit the peak flows entering the Raw Water Dam to manageable levels.

Underground dewatering will be necessary, starting from the construction of the underground development during Year 8, and increase once beyond the depth of the pit, continuing until the end of LOM. Underground dewatering will peak in Year 10 at approximately 9 070 m^3/day, while the peripheral dewatering wells will be pumping approximately the same, diminishing over the remaining LOM to approximately 3 240 m^3/day.

Seepage from the Co-disposal facility was estimated at an average seepage rate of approximately 2 100 m^3/day (88 m^3/h), reporting to the Co-Disposal Pond during the warmer months of the year where temperatures remain mostly above freezing.

The water treatment requirements were defined based on the site water balance, available water quality data from monitoring and other relevant studies. The definition of acceptable discharge water quality to the environment were based on a review of the current regulatory framework, current permitting practices and known future changes in the national and EU Regulations, which were modelled for the Kitinen River.

Contact mine water and process water are to be treated in two separate treatment plants; this is to treat the two different water streams through fit-for-purpose treatment processes.

Contact mine water will be treated such as its effluent water quality complies with environmental discharge water quality. The treated water will first be stored in the treated water pond before being discharge to the Kitinen River via a 37 km pipeline. The plant aims at reducing concentration of suspended solids, metals, uranium and nitrogen compounds as well as protecting the discharge pipeline against corrosion and fouling. Depending on the treatment chosen for nitrogen removal, biological or ferric sludge and/or reject from ion exchange process will be produced.

To minimise the requirement for freshwater abstraction, process water from processing plant will be treated such as it can be reclaimed and used as a freshwater source to the process. Contact water is to be added to the inlet of this second plant to meet water demand from the ore processing plant. The plant aims at reducing concentration of suspended solids, sulphate, nitrogen compounds, inorganics introduced by the ore processing process. The plant has been designed to be zero liquid discharge, producing a ferric sludge, gypsum slurry and a mixed salt precipitate.

In addition to the mine water treatment requirements, domestic potable and sewerage systems will be provided to serve welfare facilities throughout the site. The potable water will be generated from treated water from the external pit dewatering borehole and will meet Finnish drinking water quality standards. A sewage treatment plant will be provided; the effluent will be discharged via the discharge pipeline.

Biological and ferric sludges will be dewatered before being off-site alongside the mixed salt slurry. Reject from the ion exchange is to be treated through the zero liquid discharge treatment process. Gypsum slurry will be stored on-site in the Gypsum slurry pond.

25.1.7. METALLURGY AND PROCESSING CONCLUSIONS

The processing flowsheet has been established based on metallurgical testwork completed on representative samples taken from the Ikkari deposit. The Ikkari gold bearing ore is amenable to gravity recovery and leaching.

A circuit consisting of a primary jaw crusher, a SABC grinding circuit (SAG mill, pebble crusher, ball mill), gravity concentration, intensive cyanidation, carbon-in-leach, elution, electrowinning, gold refining and cyanide destruction has been designed to process 3.5 Mt/a of run-of-mine (ROM) from the open pit mine. Minor modifications to the plant are expected to be made in order to efficiently process the lower throughput from the underground mine (2.0 Mt/a) in the second half of the life-of-mine (LOM).

Tailings are planned to be used in co-disposal and will be dewatered in a filtration plant located about 350 m from the processing plant. Once the underground mine will be in operation, a portion of the tailings will also be sent to the paste plant.

Based on the PFS ROM characteristics and plant design parameters, an average gold recovery of 95.8% is expected.

The processing and filtration plants are designed using standard and well accepted technologies and equipment in the gold industry.

25.1.8. MINE WASTE CONCLUSIONS

An initial design and appraisal for the storage of mine waste streams for the project is provided. This considers the use of co-disposal for filtered tailings from the whole ore leach (WOL) process combined with waste rock from the open pit in the same facility. Outline design requirements and constructability are discussed. Details of the filling and operation and anticipated material balance are provided for the process plant throughput and mining output during the life of mine. Preliminary slope stability analysis was undertaken. These identify the significance of deep layers of tailings being placed and the requirement for rock layers to provide stability. The PFS design of the runoff collection pond associated with the facility is also provided. Recommendations for further investigations and studies at DFS level are given. Reference is also made to other similar filtered tailings and co-disposal projects.

25.1.9. SURFACE INFRASTRUCTURE CONCLUSIONS

An initial design and appraisal for the surface infrastructure at the plant site, water treatment ponds and roads South of the Saittajoki stream is provided. It considers the development of the general arrangement for the surface infrastructure and terracing (or pads) required for the process plant, filter plant, both heavy mobile (LME) and light mobile equipment (LME), maintenance, administration and other building assets. This excludes discussion on the open pit, underground mine and the co-disposal facility with runoff collection pond, which are discussed separately. Risks are also identified for inclusion in the overall project register. Recommendations for further investigations and studies at DFS level are given.

25.1.10. MINE CLOSURE CONCLUSIONS

The Ikkari project has been designed to accommodate progressive mine closure action to be initiated prior to the start of mining with a permanent geomorphically-designed river diversion, and continuing throughout the operational and active closure period. Accordingly, the anticipated active closure timeframe is only three years, with post-closure monitoring and maintenance timelines being similarly reduced where applicable.

The overarching goal of the planned closure actions is to reclaim mine-impacted land to support similar land uses to those present prior to mining, albeit in a different arrangement. The closure actions outlined in earlier sections of this document support the post-mining land use vision, which is focused on re-establishment of pre-mining land uses, mainly locally common habitat (mires and mixed forest), support for local passive recreational enjoyment of nature including snowmobile and hiking trails, and reindeer husbandry (WSP 2025).

25.1.11. CAPITAL COST ESTIMATES CONCLUSIONS

The capital costs were estimated in accordance with the AACE Cost Estimation guidelines and to a Class 4 level of accuracy. The contingency included equates to 11% of the total cost estimate.

The capital costs include pre-production expenditure and sustaining capital. The pre-production capital designates capital spent until commercial production is reached and is based on RR managing the indirect activities to the point of gold doré production. This includes capital spent in pre-production years -3, -2 and -1, as well as associated indirect and management costs until the mine ramps up to full production. Pre-production capital totals $575 million.

Sustaining capital is all capital spent after full planned production. This includes the major maintenance and replacement of worn-out or exhausted plant, property, machinery, equipment and other site assets. Capital related to the development of the underground mine are included in the sustaining capital estimate. Sustaining capital totals $571 million.

25.1.12. OPERATING COST ESTIMATES CONCLUSIONS

Total operating costs are estimated at $46.8/t ore, totalling $2 432 million over the life of the project.

The operating costs were estimated in accordance with the AACE Cost Estimation guidelines and to a Class 4 level of accuracy. There is no contingency include in the operating cost estimate.

25.1.13. ECONOMIC ANALYSIS CONCLUSIONS

The forecast Ikkari industrial enterprise generates a positive, post-tax NPV5 of $1 680 million with an IRR of 38% at a gold price of $2 150 USD and payback from the start of production of 2.2 years. The discount rate was set at 5%.

The planned Ikkari enterprise $NPV_{5\%}$ is most sensitive to gold price.

The forecast AISC (including selling costs) total is $918/oz with AIC at $1 306/oz.

25.2 RISKS AND OPPORTUNITIES

25.2.1. RISK ANALYSIS

Minimising project risk is a key element of a feasibility study (Scoping, Prefeasibility, Definitive). As such, WSP has considered project risks throughout the duration of this Prefeasibility Study and, where possible, has incorporated mitigation strategies into the concepts presented. This section of the document outlines the risk work completed to date and highlights outstanding work for possible later progression and integration as the Project develops.

WSP has recorded all risks in a register (see Appendix 3) through a process of identification, qualitative and quantitative assessment, and the development of associated mitigation actions.

A quantitative assessment to rank risks was utilised as described below in Figure 25-1.

Probability (P)

Consequence (C)	Rare	Unlikely	Possible	Likely	Almost Certain
Insignificant	Low	Low	Low	Low	Medium
Minor	Low	Low	Medium	Medium	Medium
Moderate	Low	Medium	Medium	High	High
High	Low	Medium	High	High	Very high
Major	Medium	Medium	High	Very high	Very high

Figure 25-1 – Utilised risk matrix

25.2.2. VERY HIGH/ HIGH RANKED RISKS

The Project risks ranked as Very High or High along with mitigation measures are presented in full below (Table 25-1).

Table 25-1 – Project risks

ID	Discipline	Risk Title	Cause(s)	Consequence(s)	Existing Controls	Risk Rating			Proposed Additional Controls	Residual Risk Rating		
						Likeli-hood	Severity	Rank		Likeli-hood	Severity	Rank
1-01	General	Impact of climate change on engineering designs	Climatic influences outside those assumed in the design assumptions (e.g. warmer/colder temperatures, more intense storm events, more/less precipitation, groundwater changes)	Failure of engineering controls (e.g. facility design) and operational (e.g. water) controls. Wildfires. Authorities ask to consider longer event horizons (100 year, 1000 year).	Inclusion of climate risk in design criteria. Climate change model.	Possible	Major	High	Climate change impact assessment to identify design parameters that account for anticipated/selected climate change scenario. Revisit throughout the LOM.	Unlikely	Major	Medium
1-03	Health & Safety	Fibrous minerals health hazard.	Risk of fibrous mineral release during mining and processing.	Air quality and health safety impacts.	Comminution and processing testing work underway by others.	Possible	Major	High	Mineralogical analysis of waste rock and ore. Dust sampling during trial processing. If required dust control and exposure management.	Possible	Moderat e	Medium
1-04	Health & Safety	Under-ground fire risks	Overheating, faulty equipment. Electric fault. Fuel spill and ignition.	Air quality impact from carbon monoxide, toxic gas, smoke or oxygen depletion. Risk to life.	Standard U/G fire safety procedure. Safety havens, egress routes, self-rescuers, equipment maintenance, fuel storage practice.	Possible	Major	High	Detailed ventilation and fire risk modelling in future phases to optimise safe evacuation plans and mitigation measures.	Rare	Major	Medium
1-05	Operational Readiness/ Project Implementation	Difficulty securing skilled workers for the mine.	Logistics and skill sets for workers. Risk perception of operational duties. Competing projects and limited resource pool.	Increased labour costs. Poor technical performance of under skilled staff (blast efficacy, mill recovery)	Not yet determined	Likely	Moderate	High	Labour force analysis. Implement local training programmes to fill gaps. Increased contractor involvement. xpatriate labour.	Possible	Moderat e	Medium
1-06	Operational Readiness/ Project Implementation	Delays to critical path items	Unforeseen delays in key schedule items such as Permitting, Process Plant/ River Diversion & Pit Dewatering/ Mine Equipment/ Water Treatment or Co-Disposal Facility	Delay to first production, increased cost and longer schedule impact.	Project Implementation Planning at this Study Level	Likely	High	High	Further detailed scheduling at next study stage, inclusion of contigency in schedule.	Unlikely	High	Medium
1-07	Operational Readiness/ Project Implementation	Site Material Volumes not balanced.	Excess or deficit of construction material resulting from the scheduling, cut/fill volumes or the quality of site won material compared with construction specification.	In the case of too little site won material, the import of large volume will have a higher cost and potentially schedule impact. In the case of too much site won material, additional space will need to be found on site for stockpiling and disposal.	High level material volume estimate included in Prefeasibility Study.	Likely	High	High	Detailed project implementation planning in subsequent studies and further material balance estimates based on additional ground investigations, as well as testing site materials for geotechnical properties. Identification and testing of local sources of rock. Trade off against import.	Unlikely	High	Medium

ID	Discipline	Risk Title	Cause(s)	Consequence(s)	Existing Controls	Risk Rating			Proposed Additional Controls	Residual Risk Rating		
1-08	Environment and permitting	Future regulatory changes	Changes to current regulations making permitting more challenging and delayed. There are planned authority responsibility changes that may impact start 2026.	Permitted area changes. Changes to water quality requirements/legislation. Changes to requirements to containment structures, and / or other changes in design. Potentially stricter regulations.	Permitting risks identified in Prefeasibility Study	Likely	Moderate	High	Permitting road map. Follow closely changes in legislation and practice related to environmental and mine permitting.	Likely	Moderat e	High
1-09	Environment and permitting	Permitting takes longer than expected.	Planning authority not satisfied with initial applications and requires revisions and updates leading to delays.	Project is delayed, costs increased.	Permit routes identified, experts and stakeholders engaged.	Almost Certain	Moderate	High	Plan engagement with relevant authorities and stakeholders. Production of high-quality permitting application addressing all elements required by authorities	Likely	Moderat e	High
1-12	Environment and permitting	Co-disposal facility concept/design perceived not credible for permitting due to authority/ public perception	Concept/design not credible for permitting due to authority/ public perception. Misidentification of co-disposal facility using layering as a co-mingling facility.	Permitting refused. Change of tailings concept. Delay.	Global best practice. Presented in town halls.	Possible	High	High	Engagement with public/ authorities to educate about co-disposal. Opportunity- safer tailings disposal method. Demonstrated technical underpinning in Prefeasibility Study document, explanation of co-disposal unit in Prefeasibility Study, client works closely with engineers to present concept correctly. and future engineering works.	Unlikely	Moderat e	Medium
1-19	Environment and permitting	Leväsaarenoja ground-water quality impact in restriction area and Naattuankanga s.	Insufficient understanding of regional hydrogeology and underestimation of the cone of depression.	Reduced usability (there is at least one household well) and quality in Leväsaarenoja groundwater restriction area, and Naattuankangas. Negative public/authority attention.	Ground Water modelling and field investigations	Possible	High	High	Additional groundwater modelling (underway by others) additional hydrogeological investigations to be conducted in the future	Unlikely	High	Medium
1-21	Environment and permitting	Mine dewatering lowers river flow.	Pumping natural drainage from the watershed to discharge pipe reduces natural flow.	Saittajoki runs dry or suffers drought on dry for even longer periods. Ecological changes or damage.	Existing high level modelling. Gap in knowledge identified in Prefeasibility Study.	Likely	Moderate	High	Integrated geomorphological and hydrogeological planning teams. Reduction of river intake. Further assessment and control implementations, e.g. diversion of surface water to maintain levels.	Unlikely	Moderat e	Medium
1-25	Geology	Drill hole orientations of some holes may be subparallel to mineralisation resulting in potential local grade bias.	Poor drill hole orientation with respect to orientation of mineralisation.	Reduction in quantity of metal in deposit.	None.	Likely	Moderate	High	Replace holes with new holes at better angles during further drilling programs	Unlikely	Moderat e	Medium

ID	Discipline	Risk Title	Cause(s)	Consequence(s)	Existing Controls	Risk Rating			Proposed Additional Controls	Residual Risk Rating		
1-26	Geology	Revised interpretation of mineral domains reduces metal content.	Mineral domain and lithological models were interpreted from drill hole data and may not accurately represent the geology or account for the full scale of geological variability due to the complex structurally deformed nature of the deposit.	Reduction in quantity of metal in deposit.	Considered in MRE classification. Indicated aim is +/-15% accuracy on an annual basis.	Likely	Moderate	High	Further resource conversion drilling to increase resource confidence to Measured. Constant re-evaluation and re-interpretation of data. In portion where further drilling occurs likelihood of fewer ounces reduces.	Possible	High	High
1-30	Geology	Composition of waste rock	Risk classification of waste rock incorrectly assigned due to incorrect geological interpretation and/or estimation of relevant elements	Flow of material to co-disposal not what is expected leading to incorrect placement of Non-Acid Generating and Potentially Acid Generating waste	Majority of drillholes have multielement data and substantial effort to interpret distribution of waste rock.	Likely	Moderate	High	Design co-disposal is such a way that flexibility exists to classify waste material during waste stripping. Overall design with tolerances to allow for changes in overall balance of waste material.	Unlikely	Minor	Low
1-31	Geology	Dewatering wells not performing to expectation	Structures that control hydraulic conductivity not correctly interpreted and wells drilled in wrong place or more permeability in unfractured bedrock	Lower volumes of non-contact water and Higher volumes of contact water leading to increased water treatment costs and/or impact on production.	Structures modelled from all available drillhole but these are not optimised for these structures.	Likely	Moderate	High	Drill program to firm up location of structures prior to installation of dewatering infrastructure.	Rare	Moderat e	Low
1-32	Mine Water	Water balance - excess or insufficient water	Insufficient understanding of groundwater and surface water regime, operational controls, water storage and demand.	Excess of water requiring disposal, in particular during storm events, or insufficient water for process requirements. Potential to impact springs, leading to them drying out and vulnerable species destruction.	High level study completed in Prefeasibility Study. Spring surveys completed in 2023. Thermo-images identify water at surface.	Possible	High	High	Development of a dynamic probabilistic water balance to quantify risks, based on an understanding of the mine design and hydrological information at later stage. Updating models through life of mine. Spring surveys and baseline studies.	Unlikely	High	Medium
1-33	Mine Water	Water management - insufficient inflow control	Insufficient hydrogeological and hydrological characterisation leading to large uncertainty in the hydrogeological conceptual model.	Water ingress in excess of control capacity leading to mine flooding and or uncontrolled discharges from site	High level study	Possible	Major	High	Hydrogeological and hydrological investigation, including the installation of monitoring facilities and modelling of inflows.	Unlikely	Major	Medium
1-34	Mine Water	Hydrogeological conceptual model uncertainty	Hydrogeological conceptual model is uncertain. Conflicting information and concepts being applied. Current test work is focussed on the orebody but may not be reflective of the rest of the site.	Rate of groundwater ingress to the open pit and or underground mine are underestimated or overestimated and pore pressures poorly understood.	Previous hydrogeological study by others. Ongoing study by others.	Likely	High	High	Additional investigation required and hydrogeological conceptual model to be revisited in conjunction with geological and geotechnical model. Further detailed study of river diversion including hydraulic modelling and flood risk modelling.	Unlikely	High	Medium

ID	Discipline	Risk Title	Cause(s)	Consequence(s)	Existing Controls	Risk Rating			Proposed Additional Controls	Residual Risk Rating		
1-35	Mine Water	Flooding of Underground or Open Pit mine	Groundwater and/or surface water flooding. Underground is more likely to be affected by groundwater flooding. Flooding could result from river diversion seepage at tributary head or river during diversion. Flooding is most likely during the snowmelt season	Mine is flooded, affects work schedule and could affect personnel working in the mine Rehabilitation of mine needed post flooding. Health & Safety Risks.	PFS considers at high level, ongoing study by others.	Possible	Major	High	Creation of flood embankments as river is being diverted. Additional studies of the river diversion, including hydrological modelling. Determination of acceptable surface flood risk at the bottom of the pit. Additional data in the hydrological model	Unlikely	Major	Medium
1-36	Mine Water	Surface water diversion route not suitable	Low level of knowledge regarding terrain traversed by proposed surface water diversion from a topographic, geotechnical, hydrological, hydrogeological perspective.	Route not achievable for technical reasons. Potential for backflow into pit if gradients not accurate	Further testing and design recommended by Prefeasibility Study	Possible	High	High	Detailed topographic, hydrological and geotechnical survey of proposed route. Hydraulic modelling to be completed on final permittable design.	Unlikely	High	Medium
1-37	Mine Water	Ground-water quality issues	More saline (or mineralised) than anticipated	Process efficacy, water treatment, discharge to environment. Potential mixing with shallower groundwater and changing quality.	Limited testing.	Likely	Moderate	High	More testing. Further characterisation. Contingency planning for process/treatment.	Possible	Moderat e	Medium
1-38	Mine Water	Cannot discharge untreated groundwater from boreholes to Saittajoki	Groundwater quality does not meet discharge permit.	Delays in construction, increased cost of construction.	None. Groundwater quality assumed suitable for discharge.	Possible	High	High	Complete additional testing on groundwater quality. Potential to implement solutions such as temporary water treatment plant (pre-mine development), build full water treatment plant early or advanced pipeline construction to larger river which could be permittable.	Unlikely	Moderat e	Medium
1-39	Geo-chemistry	ARD and deleterious mine drainage	Understanding of the geochemical behaviour of the mine waste (rock and tailings) and in-situ rock (pit and underground) is limited hence understanding of the potential impacts on the environment (especially water) is limited.	Potential for significant impact on water quality from mine water discharges, seepage from waste and construction materials.	Initial Consultant (MEM) geochemistry study	Likely	High	High	Ongoing geochemistry study. Integrate with hydrogeological and hydrological studies.	Unlikely	High	Medium
1-40	Water Treatment and discharge pipeline	Water Treatment Plant demand exceeds capacity and leads to spill	Higher than expected demand due to uncertainty in groundwater quality and volumes. Bench-scale data for tailing seepage delayed or change in water balance	Demand exceeds capacity. Water ponds could overtop & spill into environment. A stop work order may result from authorities, reputation of RR and designer damaged.	Balancing pond size and water treatment plant capacity estimates.	Likely	Major	Very high	Further underpinning on inflow and water quality estimates (focus next study stage on ground investigations and more detailed water balance) and technical design of mitigation (focus on buffer and plant	Likely	High	High

ID	Discipline	Risk Title	Cause(s)	Consequence(s)	Existing Controls	Risk Rating			Proposed Additional Controls	Residual Risk Rating		
			may lead to incorrect treatment plant sizing.						capacity). Allow for flexibility for retrofitting.			
1-41	Water Treatment and discharge pipeline	Water treatment plant oversized.	Lower than expected demand due to uncertainty in groundwater quality and volumes. Bench-scale data for tailing seepage delayed or change in water balance may lead to incorrect treatment plant sizing.	Capacity far exceeds demand. Conservative design (Higher CAPEX) or not being able to comply with environmental permit discharge into the Kitinen River.	Coordination with consultant preparing long tailing seepage tests in next phase. Use of best practice design, benchmark against existing mines.	Likely	High	High	Water treatment requirement to be reviewed in subsequent design stage. Allow for flexibility for retrofitting.	Possible	High	High
1-43	Water Treatment and discharge pipeline	Waste streams disposal	Production of sludge, gypsum and mixed slat slurry by the 4 treatment streams. Unable to dispose the waste streams on-site (capacity fully utilised) or off-site (third party not accepting waste streams)	Site operation to be ceased.	Disposal area provided on-site for gypsum slurry only. Assumption that sludge (from wastewater treatment and coagulation/clarification process from water treatment plant 1) is tankered off-site to the nearest wastewater treatment works. Assumption that Mixed salt slurry is to be disposed off-site in accordance with waste management regulations.	Possible	Major	High	Further engineering and underpinning work on full feasibility study for disposing waste streams	Possible	Major	High
1-45	Geotechnical	Mining-induced stress (interaction)	Only preliminary understanding of interaction between Open Pit & Underground operations, various underground mining fronts, primary & secondary stope lines, stopes and mining drives, through elastic Boundary Element Method modelling without fault zones.	Underground excavation stability impacted. Changes to underground mine design based on numerical modelling with plastic yield zones, and explicitly including fault zones.	Recommended 3D mine-scale elastic Boundary Element Modelling for next phase.	Likely	Moderate	High	3D mine-scale elastic-plastic modelling, accounting for anisotropic behaviour where appropriate, and including fault zones, to investigate whether yield and de-stressing occurs in secondary pillars, what the impact is of underground excavations on pit wall stability, crown pillar stability, and stand-off distance of infra-structure.	Unlikely	Moderat e	Medium
1-46	Geotechnical	Effect of hydrogeology for open pit stability	Coupling of rock mechanical stability analyses and hydrogeological model not completed.	Over or under estimation of stability, leading to either inadequate or overly conservative design.		Possible	High	High	Addition of hydrogeological model into slope stability modelling.	Unlikely	High	Medium
1-47	Geotechnical	Uncertainties in structural modelling and characteristics	Lack of geotechnical drilling data especially on Southern and Eastern areas to accurately define and understand thickness and location of fault zones. Length of solid core pieces	Effect on overall slope stability. Where large-scale structures (e.g., Southern Fault Zone) intersect the orebody, pillars might have to be left in place or stope sizes will have to be reduced.	Selection of a retreating sequence away from the Southern Fault Zone to reduce the amount of potential seismic energy release in close proximity to the fault plane.Reduced	Possible	High	High	Improved structural model. Definition/Estimation of the characteristics of the Southern Fault Zone, allowing inclusion in elastic-plastic model to assess interaction with pit wall and underground mining headings,	Unlikely	High	Medium

ID	Discipline	Risk Title	Cause(s)	Consequence(s)	Existing Controls	Risk Rating			Proposed Additional Controls	Residual Risk Rating		
			from fault zones typically insufficient for laboratory testing.		overall open pit slope angles in affected areas.				and more detailed assessment of the effect of mining sequencing on fault slip tendency. Perform a critical stress analysis on the Southern Fault Zone (if the shear stress across the fault exceeds its shear strength, the two opposing faces slip, i.e., the fault is critically stressed).			
1-49	Geotechnical	General under-ground operational geotechnical risks	Fall of Ground causing fatality or permanent disability Severe damage to remotely operated scoop tram due to loose from stope walls during open stope mucking. Major rehabilitation campaign of mining area due to unexpected ground behaviour. Ground support system or elements do not perform.	Fatality or permanent disability, significant delays to the mining schedule, increase in Operational Expenditure, damage to mining fleet.	Systematic approach to ground control using support categories and only allowing personnel under supported ground.	Possible	Major	High	To be developed in run up to start-up of operations, through best practice project implementation, design and construction planning: Ground Control Management Plan. Good Standard Operating Practices and reporting culture. Implementation of Design Monitoring and Safety Monitoring Devices. Just-in-time development reduces costs and delays related to preventative maintenance of ground support and rehabilitation. Stope reconciliation. Face mapping / Scanline mapping. Frequent visual inspections by qualified geotechnical engineer	Unlikely	Major	Medium
1-52	Geotechnical	Effect of foliation and rock anisotropy not captured in rock mechanical analyses	Established analysis methods expect isotropic, homogenous rock mass and might fail to capture failure mechanisms induced by strongly anisotropic rocks.	Under estimation of bench scale and/or overall stability. Design angles are not achieved in production. Stability issues causing loss of production/equipment or even personal injuries/deaths.	Usage of anisotropic material models where applicable. Choose of analysis methods. Conservative estimates of rock mass strength for heterogenous, anisotropic rock masses.	Possible	High	High	Further characterization of anisotropy and anisotropic properties of rock mass domains. and strength anisotropy. Assess variability within MSCU and assign sub-domains if required (for example on heterogeneity or foliation intensity).	Unlikely	Moderat e	Medium
1-54	Paste Backfill	Binder Cost Uncertainty	The cost of cement and other cementitious supplemental materials have increased in price significantly in the last few years. Future environmental considerations with the manufacture of cement are likely to result in the continued increase in pricing beyond inflationary values.	Increased backfill costs beyond current estimates	Not yet determined	Likely	Moderate	High	Consider adding escalation factor to the cost of binder being carried in the cost estimate. Opportunity to investigate binder replacements to mitigate cost increases.	Likely	Moderat e	High

ID	Discipline	Risk Title	Cause(s)	Consequence(s)	Existing Controls	Risk Rating			Proposed Additional Controls	Residual Risk Rating		
1-55	Paste Backfill	Surface Paste Line Blockage	The paste plant is 350 m from the borehole at the edge of the pit. This horizontal line on surface is more susceptible to blockage than underground or vertical lines	Lost backfill plant production time	Operating procedures, high pressure flush pump	Likely	Moderate	High	A standby surface pipeline was added to mitigate any issues with the operating surface pipeline.	Rare	Moderat e	Low
1-56	Paste Backfill	Power Outage Risk to Backfill System	Power outage	Lost backfill plant time, loss of equipment if cannot be cleaned.	Putting plant equipment on backup power	Likely	Moderate	High	Provide an emergency power source.	Unlikely	Minor	Low
1-59	Mine Waste	Use of Overburden in Foundation to Co-Disposal Facility	Availability/ suitability (material properties) of overburden from open pit for use as engineering fill in foundation to co-disposal facility, including clay required for the low permeability liner not yet determined.	Insufficient materials during construction leading to increased costs or delayed construction.	Identified at Prefeasibility Study	Possible	High	High	SI and best practice design recommended. Specifically by testing of the overburden's geotechnical properties to confirm suitability and further material balance underpinning through construction scheduling to confirm availability.	Unlikely	High	Medium
1-60	Mine Waste	Overall placement of co-disposal facility is located somewhere not permissible/per mittable	Proximity to local water sources, etc	Relocation of co-disposal. Refusal of permit.	Site selection work.	Possible	High	High	Engagement with public/ authorities to educate about co-disposal location and design. Consider 'Plan B' locations. Environmental Impact Assessment to consider alternatives.	Unlikely	High	Medium
1-61	Mine Waste	Suitability of Soils and Bedrock Beneath the Co-Disposal Facility	Suitability of soils and Bedrock underlying co-disposal area, to avoid differential settlement to be assessed.	Seepage of contact water into the environment and slope instability of stack, possible loss of life.	Risk identified at Prefeasibility Study.	Possible	High	High	Ground surveys and hydrogeological investigations to be continued 2024-2025 winter. Further site investigations and best practice design recommended	Unlikely	High	Medium
1-68	Infrastructure	On-site water treatment sludge storage facility	Requirements for storage of sludge from water treatment process not defined at Prefeasibility Study	Additional area may be required at a later stage	Further testing and design recommended by Prefeasibility Study	Almost Certain	Moderate	High	Sludge characteristics and quantities to be determined, then on site footprint needs sizing and technology/disposal method costed.	Possible	Moderat e	Medium
1-72	Infrastructure	Flooding water at portal	Proximity of water discharge pipeline to Underground portal	Water flowing underground	Pipeline buried underground	Possible	High	High	Provide facility for flush-out into to the raw water pond as needed and portal entrance drains.	Rare	High	Low
1-73	Process	Spills to environ-ment	Operational error, failure of control systems and instrumentation	Intervention from Authorities, possibility of halting operations and environmental/reputational damage.	More frequent checks of instrumentation/calibration/ equipment	Likely	High	High	Ensure secondary containment for reagents/CIL area meets best practice/ Finnish regulation. Ensure building has sufficient containment in the Definitive Feasibility Study stage. Show	Unlikely	High	Medium

ID	Discipline	Risk Title	Cause(s)	Consequence(s)	Existing Controls	Risk Rating			Proposed Additional Controls	Residual Risk Rating		
									containments in the 3D layout in the next phase.			
1-82	Mine Closure	Pit lake water quality unsuitable for discharge.	Pit water quality may not be suitable for discharge to the environment once the pit flooding is complete. Groundwater naturally does not meet anticipated discharge quality.	Reduced surface water entering post-closure re-connected stream. Water treatment plant will be decommissioned following Year 29 (approximately), but the pit is not expected to be flooded until year 45 or 50. At this time if water quality is unsuitable for discharge direct to the environment, solutions are predominantly undesirable (costly, extend active closure activities):	Confirmation of discharge water quality requirements / permit requirements. Conduct water quality and quantity modelling and update with renewed data (pump rates, water chemistry) regularly such that closure water management of the open pit may be planned for with greater certainty over the operational mine life. Update the Open Pit Closure Plan and water treatment plan/design as needed during operations based on new information/data recorded.	Likely	Major	Very high	1. Consider leaving the water treatment plant in place for re-start once the pit floods but prior to the discharge elevation being reached. 2. Construct a new water treatment facility if initial WTP is demolished per schedule. 3. Consider leaving dewatering wells in place for down drain/pumping and recycling through pit. 4. Re-establish dewatering wells if closed, and pump groundwater such that pit lake does not overflow, 5. investigate passive water treatment and storage options.	Likely	Major	Very high
1-83	Mine Closure	No site-wide water balance or water quality modelling to date.	No site-wide water balance or water quality modelling completed to date.	Water treatment could be required for longer than current assumption. Water treatment may be required for a greater capacity/volume than the plant was designed for. Higher costs for water treatment than estimated.	More detailed studies will progress in next stage of design.	Possible	High	High	Complete water balance and water quality modelling to understand full environmental and economic implications. Develop updated water treatment plan. Conduct trials to demonstrate proof of concept, and refine treatment plan and design. Complete hydrogeology modelling and water balance model site wide. Contingency in closure cost estimate.	Unlikely	High	Medium
1-84	Mine Closure	Potential breach of permit conditions	The water discharge permit conditions are unknown. Surface water from the site post-closure is to be discharged into the local water courses, likely to have high ecological status. Closure Plan currently assumes treatment to enable discharge into the Kitinen river.	Degree of water treatment required is higher than expected. Substantial additional expense to achieve discharge criteria, for extended period of time.	Coordination with Environmental Consultant (Envineer), mine closure and mine water team.	Likely	High	High	Assessment of water quality requirements for a discharge into the local water body plus treatment adjustment as required. Site wide water quality model and water balance model both for operations and closure timeframe. Determine treatment.	Unlikely	High	Medium
1-86	Mine Closure	Insufficient financial planning (estimation) and	Optimism bias. Poor data or lack of data. Failure to allocate sufficient funds for progressive	Insufficient funds to close mine on schedule/budget. Inability to surrender permits or draw down on any closure	Closure cost estimate has contingencies included, intended to compensate	Likely	Major	Very high	Additional studies required to reduce uncertainty of closure cost estimate.	Possible	Major	High

ID	Discipline	Risk Title	Cause(s)	Consequence(s)	Existing Controls	Risk Rating			Proposed Additional Controls	Residual Risk Rating		
		allocation of funds for closure during operation.	reclamation and/or closure trials during operations.	security "bonds". Inability to transition land to next use; reputational damage with community and regulator. Potential to impact ability to open future mines or achieve approval of mine life extensions. Long term unacceptable risks to the environment and human health. Failure of social transition. Long term liability.	for lack of information at this stage.							
1-91	Mining	Underground Achieving ramp-up	Mine production ramp-up less than planned due to schedule slippage, high demands, ground conditions, new equipment etc.	Potential reduced production and lower feed to the plant and reduction in revenue.	Prefeasibility Study Production schedule provides an understanding of critical path development and infrastructure required to meet ramp up.	Possible	Major	High	Appropriate Operational Readiness will be required to support ramp up. Further studies will allow for a better understanding of requirements and allow for fine tuning of the schedule to eliminate unnecessary development/mining. Further investigations of critical paths.	Unlikely	Major	Medium
1-92	Mining	Underground Inability to meet advance rates.	High advance rates required to meet production requirements.	Potential reduction in plant feed and reduction in revenue.	Prefeasibility Study production schedule provides a detailed understanding of required development to achieve production rates. Lateral development set at 75 m/mo. Main access at 110 m/mo. Main risk is multi-heading development.	Possible	Major	High	Similar to ramp up, further studies will allow for a better understanding of requirements and allow for fine tuning of the schedule to eliminate unnecessary development.	Unlikely	Major	Medium
1-93	Mining	Underground Inability to develop the required number of headings.	Unrealistic number of headings required to meet production requirements.	Potential reduction in plant feed and reduction in revenue.	Prefeasibility Study production schedule provides a detailed understanding of critical headings to achieve production rates.	Possible	Major	High	Similar to ramp up, further studies will allow for a better understanding of requirements and allow for fine tuning of the schedule to eliminate unnecessary development.	Unlikely	Major	Medium
1-102	Mining	Open Pit Pre-Stripping not achieved	Pre-stripping not achieved when planned due to equipment productivity levels not being met, worse ground conditions than expected or other reason.	Prolonged ramp-up, increased stripping ratio and increased costs.	Production schedule outlines specific targets required for production. Stripping can start early to enable targets are reached.	Possible	High	High	Scheduling of stripping works planned in more detail at later phases. Procurement model: contractor to provide depth of resource.	Unlikely	High	Medium
1-107	Financial & Economics	Commodity Price	Risk: Price goes down. Opportunity: Price goes up.	Changes to viability/profitability of projects	Financial modelling sensitivity analysis on gold price completed as part of statutory reporting.	Likely	Moderate	High	Additional strategies once closer to production including hedging and take-off agreements.	Possible	Moderate	Medium

The most prominent and repeated risks are areas for mitigation focus:

- **Water Management:** The extensive works required for dewatering and drainage within the context of unspoilt watercourses requires effective and robust methodologies with suitable contingent capacity for water capture, treatment and discharge. Further work should be completed to specify these systems with a particular focus on refining the understanding of groundwater inflows and quality to address uncertainties in capacity requirements and the associated capital risk of this uncertainty.
- **Permitting:** Obtaining the necessary construction, environmental and operational permits in a manner that minimises the impact on timeline will require a focus on the EIA, stakeholder engagement and 'right first time' application documents. Water Management (particularly the river diversion) and Mine Closure will be a considerable focus of these efforts due to their wide-ranging impacts. The Co-Disposal facility is likely to be unfamiliar to permitting authorities and while offering significant advantages for minimisation of environmental impact will require specialised attention in permitting application documents.
- **Project Implementation:** There are several significant schedule items with the potential to delay the construction which will need to be addressed through detailed planning (see section 24.1).
- **Closure:** Closure and post-industrial activities are linked to potentially long-term environmental impacts and financial commitments. Suitable and sufficient planning for site closure will underpin permitting, mitigate the potential for closure cost escalation and fulfil responsible industrial and mine site obligations.

25.2.3. OPPORTUNITIES

The Ikkari Project development team have completed extensive optioneering trade-offs and is presented in its optimal form based on the Prefeasibility Study constraints, however, there remain several opportunities going forwards to add value to the project:

- Modification of co-disposal geomorphology and optimising waste rock and tailings layering to reduce mine closure costs; and
- Optimised configuration of site infrastructure, layout and underground access to enable a decreasing cut-off grade strategy in operations.

26 RECOMMENDATIONS

Technical recommendations derived from the Prefeasibility Study are outlined in Table 26-1.

Table 26-1 – Technical Recommendations

Discipline	Recommendation
Geology	▪ Conduct further exploration and drilling near the deposit to fully assess the potential of mineralisation for satellite deposits; ▪ Perform test grids to determine the drill density required for the conversion of resources to the Measured Category and likely grade control pattens required during mining; ▪ Conduct further exploration and drilling to expand the footprint of the deposit; ▪ Conduct infill drilling to potentially increase confidence in the existing OP and UG resources; ▪ Switch to using a bar-coded sample tag that can be read when received at the laboratory to reduce any potential risk of transcription errors; ▪ Conduct third party density measurements that are consistent with the sample intervals in the next infill drill programme; ▪ Evaluate the potential implementation of security tags with sample shipments; ▪ Produce formal written procedures for database management; ▪ Assess the sensitivity of outlier data through various top-cuts and restriction techniques; and ▪ Replace holes oriented along-strike and down-dip with holes oriented perpendicular to mineralisation.
Mining	▪ Utilise updated geotechnical information to improve and optimise stope inputs including ELOS, exclusion of Southern Fault Zone, and stope dimension (sizing); ▪ Continue to engage with open pit and underground mining contractors to ensure there is no delay to the implementation schedule or reliance on equipment delivery when required; ▪ To expediate project development, initiate operational readiness guidelines and planning; ▪ Update the open pit/underground stope offset with updated geotechnical information; and ▪ Update all mine design and planning work with updated geological, hydrogeological, geotechnical and processing inputs.

Discipline	Recommendation
Mine Finance and Economics	▪ With more detailed study and cost definitions and estimates, open pit and underground optimisations should be re-evaluated; and ▪ Further dialogue around the quantity and delivery of the closure bond should be undertaken to increase understanding for financial and cost modelling.
Paste Backfill	▪ Repeat paste backfill testing on a representative tailings sample to determine paste material characteristics and confirm binder requirements for paste backfill recipes; and ▪ Provide an emergency power source for the paste plant in the event of a power outage to the facility. The emergency power will allow the paste plant to clean out equipment and piping of cemented material.
Processing and Filtration	▪ Perform comminution circuit design simulations to select optimal configuration and equipment sizing; ▪ Investigate the possibility of increasing the grind size while ensuring desired recovery; ▪ Complete additional testing to confirm preg-robbing characteristics of the ore; ▪ Perform detailed trade-off study on CIL and CIP to determine best avenue in terms of gold recovery and economics; ▪ Confirm ideal CIL residence time through further leach kinetics tests; ▪ Optimize reagents dosages in CIL and cyanide destruction; ▪ Perform additional filtration tests with equipment suppliers to confirm sizing; and ▪ Study the requirement for a temporary or emergency filtered tailings storage area to provide additional storage time in case of adverse meteorological events or issues with mobile equipment.
Mine Water management	To further the understanding of the surface and groundwater systems in the Ikkari Project Area to the appropriate level to inform the Definitive Feasibility Study and support the continued development of mine water management options, further detailed studies are recommended. These studies would also be required to inform the ongoing Environmental Impact Assessment (EIA) and Permitting processes being undertaken by others. Further recommendations are as follows: ▪ Update the Conceptual and Numerical Groundwater Models to incorporate updated groundwater level and baseflow measurements, when these become available, as well as the results from the drilling and hydraulic tests to be undertaken on site in 2024/2025;

Discipline	Recommendation
	▪ Consider drilling and testing of additional boreholes outside the Ikkari Fault Intersection Zone (IFIZ) to improve knowledge of potential connection between the regional hydrogeological zone and the IFIZ; ▪ Characterize the groundwater quality across the mine site through sampling, analysing and mapping the groundwater following the drilling of investigative and production dewatering wells or any further monitoring boreholes. Continue to monitor water quality to detect quality variations over time; ▪ Run updated groundwater model scenarios to assess groundwater drawdown extents and the impacts on springs and baseflow on the likely affected surface water stream sections. This should also include an assessment of the risk of recirculation of ponded water in stream diversion sections back to the Open Pit; ▪ Include a conceptualisation of the closure phase of the mine Open Pit and underground workings with respect to groundwater level recovery post-mining and the potential implications of the likely formation of a pit lake after the pumping in and around the mine ceases. A trade off study is required between maintaining the diversion of surface water runoff around the pit after closure and directing to the pit to reduce the time take to for it to fill. ▪ The river diversion design will need to be developed further to maximise the extent of natural channel. This will be informed by a detailed geomorphological survey of the existing Saittajoki River and the Heinalamminoja Stream, topographic survey information and hydraulic modelling. The flood risk to the mine from both surface water and seepage into the pit from the adjacent diverted watercourse will need to be assessed; ▪ Develop a site-wide stochastic water balance model using GoldSim (computerised software tool) to provide the basis for sizing and interrogating the site surface water management infrastructure over the intended Life of Mine (LOM). The GoldSim model can provide the basis for appraising anticipated inflows (including runoff, direct rainfall, and groundwater ingress) into the mine, the consumption of water by the mining process, the release of water from the operation into the natural environment and the residual volumes of water that are to be managed safely on site to ensure operational integrity. It will also provide a means to compare and consider other mine water infrastructure variations and design options; ▪ Undertake a site-specific climate change study to assess the impact on mine water management at the mine; ▪ The methodology for the treatment of contact water is to be confirmed. This requires an assessment of the monthly variation of the contact water quality as well as defining best route for nitrogen compounds removal. Focus will be on chemical and power consumption as well as understanding spare heat available for the operation of a biological process; ▪ If feasible, we would recommend the use of bench-scale test and pilot plant to optimise the water treatment plant design envelop;

Discipline	Recommendation
	▪ The composition of the mixed salt slurry has been characterised and is deemed non-hazardous. It is proposed to assess options to make the slurry inert or as a product; ▪ All water treatment plants and discharge pipeline to be advanced in design to refine costs and layouts, reflecting other disciplines further studies. In particular, the design of the water treatment plant and pipeline should account for: • Uncertainty of inflows by allowing for flexibility and retrofitting; • Further buffer ponds capacity analysis – Water treatment plant capacity versus pond capacity; and • Mine flooding risk. ▪ Design and operational water treatment and conveyance risk identified through HAZOP/HAZID reviews.
Mining Waste and Tailings	▪ Further geotechnical testing required for subsequent feasibility study and detailed designs to validate assumptions made at PFS level. This includes both in-situ testing and laboratory testing. Geotechnical parameters will be required for the co-disposal facility ground conditions, tailings, waste rock and open pit overburden; ▪ Further geochemical testing; ▪ More detailed slope stability with more accurate estimate of layers of filtered tailings and waste rock during life of mine; ▪ Preparing a specification for the placement of waste and filtered tailings which is supported by compaction trials; ▪ Dimensioning of the outer wedge of waste rock; ▪ Re-shaping of top section with a flatter side slope topography during the later stages of LOM when no waste is available from underground; Modelling of seepage through facility including risk of piping; ▪ Assessment of settlement; ▪ Review of low permeability liner system; ▪ Assessment of availability of non-acid generating waste for setting aside for use in co-disposal layers during later years of LOM; and ▪ Identification of closure requirements; ▪ Further review of material balance and confirmation of availability with respect to design and construction schedule; and ▪ Consider use of temporary stockpiles for “benign” waste rock and till..

Discipline	Recommendation
Mine Closure	▪ Develop site-wide water quality and water balance models inclusive of climate change projections to inform/refine: (1) timelines and approaches for water treatment, (2) production timeline and quantities of sludge (gypsum slurry), and (3) re-assess site-wide closure water management approaches; ▪ Investigate passive water treatment options and mitigation measures, if needed based on results of site-wide water quality and water balance modelling, to address discharge of flooded pit water to the environment; ▪ Develop a nature-based, geomorphic river diversion design aligned with post-closure land use goals, and inclusive of proposed closure success criteria for discussion with regulators, and a monitoring plan for implementation during operations. The river diversion is proposed to be constructed prior to mining initiates and as such this will require action sooner than other items; ▪ Develop a more detailed geomorphic design for the co-disposal facility in consideration of the cover material quantities available and a graphic staging plan for progressive reclamation implementation; ▪ Advance the cover design and develop a cover trial development plan for the co-disposal and sludge (gypsum slurry) ponds such that this could be implemented in year one or two of operations; ▪ Advance the design of surface infrastructure and buildings such that decommissioning and demolition plans can be refined; and ▪ Further works (design/quantity) to characterise closure cost and estimate.
Surface Infrastructure	▪ Develop a more detailed definition of building footprints and layout of assets required; ▪ More accurately determine the sludge characteristics and quantities requiring the footprint sizes of the on-site disposal facilities; ▪ More accurately size the raw water pond and treated water pond; ▪ Implement site investigation campaign and design of foundations for building and assets including roads; ▪ Identify local sources of rock and moraine (or boulder clay) from within the mine and construction workings and consider when these will be available or use; ▪ Consider the option for an additional borrow area for balancing of materials; ▪ Determine water treatment sludge characteristics and quantities produced for sizing of footprint with onsite disposal method and also technology; and ▪ Power contingency planning, potentially using 20kV power line.

Discipline	Recommendation
Rock Mechanics	▪ Orientation line interval and confidence logging should be incorporated; ▪ Logging of core losses as separate geotechnical intervals; ▪ Measurement of all joint orientations and the corresponding joint surface properties, i.e., surface roughness and infill; ▪ Axial PLT testing alongside diametrical testing to estimate strength anisotropy; ▪ Future lab testing results should note an accurate measurement of the angle between the foliation plane and the loading direction for each specimen; ▪ It is recommended to perform more laboratory strength tests to address variance due to the nature of heterogenic rocks; ▪ Some lithology domains do not have enough shear samples; ▪ Drilling and geotechnical logging of drill holes oriented towards East and West is recommended to reduce orientational bias in the data; ▪ It is recommended to perform geotechnical drilling in the East sector to better understand rock quality, as current data indicate low rock quality; ▪ Obtaining televiewer data for new geotechnical holes, where all potentially open joints are logged; ▪ Better understanding of fault zones is required for more detailed analysis of pit stability; ▪ Inclusion of joint spacing data in FS stage open pit kinematic analysis; ▪ 3D analysis of pit stability recommended in next study phase, including ground water pore pressure modelling; ▪ It is recommended to perform more detailed analysis of the overburden stability and slope parameters in the DFS stage; ▪ Just-in-time development to reduce costs related to preventative maintenance of ground support and rehabilitation; ▪ Stable stope wall dimensions near the Southern Fault Zone should be optimised in the Feasibility Study; ▪ Where large-scale structures intersect the orebody, pillars might have to be left in place or stope sizes will have to be significantly reduced; ▪ Perform a critical stress analysis on the Southern Fault Zone; ▪ Better definition of the characteristics of the Southern Fault Zone;

Discipline	Recommendation
	▪ Assess variability within MSCU and assign sub-domains if required (for example on heterogeneity or foliation intensity); ▪ It is recommended that the FS includes a mine-scale elasto-plastic model, accounting for anisotropic behaviour where appropriate, and including fault zones, to investigate whether yield and de-stressing occurs in secondary pillars, what the impact is of underground excavations on pit wall stability, crown pillar stability, and stand-off distance of infrastructure; ▪ The FS should investigate susceptibility of different rock masses to kinematic failure in drifts and stopes; ▪ Confirmation of in-situ stress understanding once underground development has started by means of observations and, if warranted, over coring stress measurements; and ▪ It is recommended to keep a retreating sequence away from the Southern Fault Zone, where possible, to reduce the amount of potential seismic energy release in close proximity to the fault plane.
Environmental, Social and Permitting	▪ Rupert Resources has carried out several environmental studies for on site. To meet the current and future legislative, regulatory, and permitting requirements it is recommended to execute more detailed site investigations; ▪ Complete the ongoing EIA according to environmental impact assessment procedure act (252/2017); ▪ It is recommended that Rupert Resources proactively develop best environmental protocols and practises and openly engage all stakeholders; and ▪ Rupert Resources has carried out stakeholder work over the years. To maintain strong foundation and trust set with local people and communities it is important for Rupert Resources to continue dialog with different stakeholder groups and be proactive with communication.
Project Implementation Planning and Operational Readiness	▪ Complete labour force analysis to determine potential skills gaps, plan appropriate training scheme to produce suitable skills locally or utilisation of contractor and expatriates if required; ▪ Detailed project implementation scheduling and critical path analysis should be performed to mitigate scheduling risks; and ▪ A detailed cut and fill material balance analysis should be completed alongside additional ground investigations and sample testing to characterise the site material balance. The identification and testing of local bulk material sources should be completed and traded off against import.

Discipline	Recommendation
General	▪ Climate change impact assessment should be completed to identify design parameters that account for the anticipated and selected climate change scenario; ▪ Contingency planning should be made for the potential of adverse weather to affect operations; ▪ Mineralogical health and safety analysis of waste rock and ore for chemical, physical and radiological characteristics; ▪ Complete detailed ventilation and fire risk propagation modelling, optimized safe evacuation plans and mitigation measures should be made for underground mine; and ▪ Conduct a full HAZID, HAZOP (construction), HAZOP (operations), HAZOP (site closure) to establish critical controls for the prevention of incident and loss and enable the appropriate level of defences to be established.

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Appendix 1

QUALIFIED PERSON CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON BRIAN THOMAS

I, Brian Thomas, state that:

(a) I am a Principal Geologist at:
WSP Canada Inc.
33 Mackenzie Street, Suite 100
Sudbury, Ontario, P3C 4Y1

(b) This certificate applies to the technical report titled Ikkari Pre-Feasibility Study NI43-101 Technical Report; Rupert Resources Ltd.; with an effective date of: February 14, 2025 (the “Technical Report”).

(c) I am a “qualified person” for the purposes of National Instrument 43-101 (“NI 43-101”). My qualifications as a qualified person are as follows. I am a graduate of Laurentian University with a B.Sc. in Geology from 1994, I am a member in good standing of the Association of Professional Geoscientists of Ontario (#1366). My relevant experience after graduation, for the purpose of the Technical Report, includes over 30 years of experience in mine geology and mineral resource evaluation of mineral projects nationally and internationally in a variety of commodities including 9 years of direct working experience in gold mining operations located in northern Ontario and 13 years of consulting experience with a strong focus on gold related projects and 8 years working in the base metals mining industry.

(d) My most recent personal inspection of the property described in the Technical Report occurred between July 10 to 13, 2023 for a duration of 3 days.

(e) I am responsible for Item(s): 1.1-1.8, 1.20.1, 5-12, 14, 23, 25.1, and part of 26 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) My prior involvement with the property that is the subject of the Technical Report includes the 2023 Mineral Resource Estimate and Technical Report titled National Instrument 43-101 Technical Report; Rupert Resources Ltd. Updated Mineral Resource Estimate for the Ikkari Project - Finland; with an effective date of: December 12, 2023.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Sudbury, Ontario this 14th of February 2025.

Signed by Brian Thomas

Brian Thomas; P.Geo.

CERTIFICATE OF QUALIFIED PERSON TIMOTHY DAFFERN

I, Timothy Daffern, state that:

(a) I am a Technical Director (Mining) at:
WSP UK and Ireland Limited.
WSP House, 70 Chancery Lane,
London, WC2A, England, UK.

(b) This certificate applies to the technical report titled Ikkari Pre-Feasibility Study NI43-101 Technical Report; Rupert Resources Ltd.; with an effective date of: February 14, 2025 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of the University of New South Wales School of Mines, Sydney, NSW, Australia and The Cambourne School of Mines, Cornwall, England, UK with a combined Bachelor's Degree in Mining Engineering from 1990. I am a graduate of the Open University Business School with a Master's in Business Administration from 2000. I am a member in good standing of the Australasian Institute of Mining and Metallurgy (Grade Fellow – over 35 years), Institute of Materials, Minerals and Mining (Grade Fellow – over 35 years), a Chartered British Engineer registered with the Engineering Council (C.Eng.), Member of the Society of Mining Engineers (USA, over 35 years), Member of the Canadian Institute of Mining and Metallurgy (Canada, over 35 years).

(d) My relevant experience after graduation, for the purpose of the Technical Report, includes over 35 years of experience in mine design, mineable reserve determination, mine operations, mine management, mine closure, corporate management, opportunity assessment as a director of a merchant bank, and co-author of more than 80 due diligence and technical reports including FEL1, FEL2 (PFS) , FEL 3 of mineral properties internationally written as NI 43 101, SK 1300, CPR and JORC (2012) Table 1 disclosure formats; in a variety of commodities including more than 10 years of direct working experience in gold based mineral properties.

(e) My most recent personal inspection of the property described in the Technical Report occurred between February 5th and 6th, 2025.

(f) I am responsible for the Mineral Reserves and supervision and oversight of all sections of this Technical Report.

(g) I am independent of the issuer as described in section 1.5 of NI 43-101.

(h) I have had no prior involvement in the Ikkari, Finland mineral property.

(i) I have read the terms and conditions of the NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and

(j) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Truro, Cornwall, England this day 14th of February 2025.

Signed by Timothy Daffern

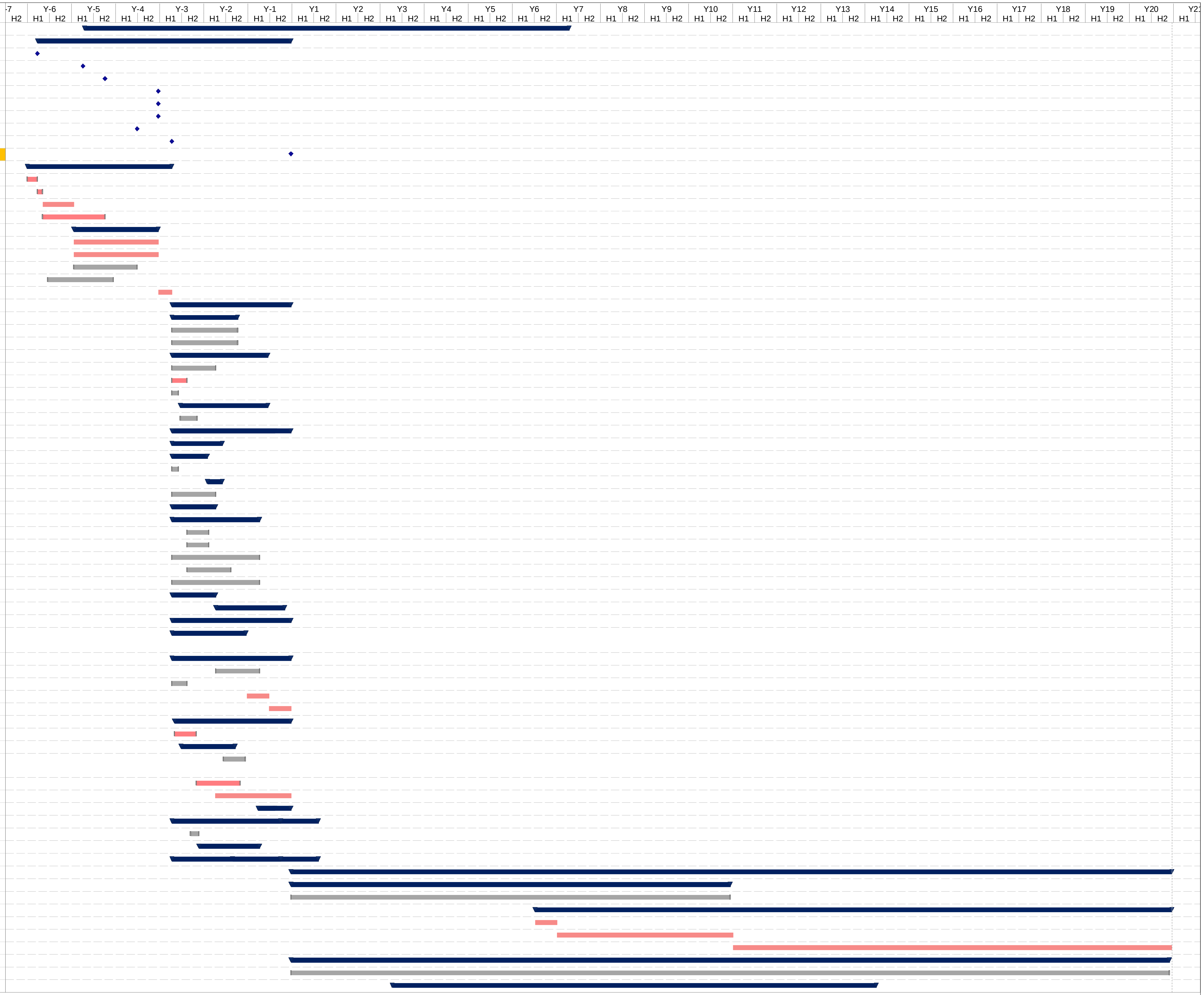
Timothy Daffern. B. Eng. (Mining). C. Eng., QMR

Appendix 2

IMPLEMENTATION SCHEDULE

PUBLIC

ID	Task Name	Duration
1	**Seasonal Constraints**	**2870 days**
35	**MILESTONES**	**1500 days**
36	PFS Complete	0 days
37	FEED Stage 1 Award	0 days
38	DFS Complete	0 days
39	Capital Investment Decision	0 days
40	Regulator Submission: Construction Permits	0 days
41	Regulator Submission: Enviromental Permits	0 days
42	Regulator Submission: Mining Permits	0 days
43	Construction Starts	0 days
44	First Gold Pour	0 days
45	**PRECONSTRUCTION PHASE**	**855 days**
46	PFS	60 days
47	Rupert Resource Capital Discussions	30 days
48	EIA	37 wks
49	DFS	74 wks
50	Permitting	**500 days**
51	Construction Permits	100 wks
52	Environmental Permits	100 wks
53	Mining Permits	75 wks
54	FEED	78 wks
55	Procure & Mobilise Contractor	16 wks
56	**CONSTRUCTION PHASE**	**705 days**
57	OFF-SITE INFRASTRUCTURE	**390 days**
58	Access Road: Ends at Southern Mine Site Boundary	78 wks
59	Electrical Power Supply Lines to Site	78 wks
60	ON-SITE INFRASTRUCTURE	**570 days**
61	Establishment	52 wks
62	Site Clearance & Initial Earthworks	4.5 mons
63	Construction Run Off Drainage Capture & Water Treatmer	2 mons
64	Electrical Power	**520 days**
68	Temporary Building Construction	20 wks
69	ROADS AND LAYDOWN AREA	**705 days**
76	SURFACE WATER MANAGEMENT	**300 days**
77	Saittajoki River Diversion	**210 days**
82	Sediment control Dam	2 mons
83	Embankment	**90 days**
86	Borehole Dewatering System	52 wks
87	FENCING AND ACCESS CONTROL	**260 days**
90	WATER TREATMENT	**520 days**
91	Raw Water Pond	26 wks
92	Raw Water Intake Station (Water Supply)	26 wks
93	Water Treatment Plant	104 wks
94	Treated Water Pond	52 wks
95	Treated Water Discharge Pipeline	104 wks
96	Sludge Pond 1 (East)	**260 days**
98	SURFACE BUILDINGS & STRUCTURES	**410 days**
110	MINE DEVELOPMENT	**705 days**
111	Mine Equipment Procurement, Delivery, Assembly, Commissioning, Training	**440 days**
115	OPEN PIT	**705 days**
116	Dewatering Open Pit before Stripping	52 wks
117	Circumferential road, slope stability monitoring stations.	4.5 mons
118	Year-1 Pre-Stripping (Contractor)	26 wks
119	Year-1 Waste Mining (excluding overburden)	26 wks
120	PROCESS PLANT AND ROM PAD	**690 days**
121	Plant Earthworks Pad	26 wks
122	ROM Pad and Wall	**320 days**
125	Backfill behind ROM wall and installation of ore stockpile liner	26 wks
126	Primary Crusher Building	52 wks
127	Process Plant	90 wks
128	Filtration Plant	**195 days**
132	**CO-DISPOSAL FACILITY**	**645 days**
133	Peat prestripping	2.5 mons
134	**Run-off collection pond**	**360 days**
138	**Co-Disposal Construction**	**645 days**
169	**MINING PHASE**	**5215 days**
170	**OPEN PIT**	**2600 days**
171	O/P Mining Activities Ongoing	520 wks
172	**UNDERGROUND**	**3770 days**
173	Contractor Mobilisation	26 wks
174	Construction Underground Systems	208 wks
175	U/G Mining Activities Ongoing	520 wks
176	**PLANT**	**5200 days**
177	Plant Operation	1040 wks
178	**Sludge Ponds**	**2865 days**



Project: Ikkari Implementation

Task	Summary	Inactive Milestone	Duration-only	Start-only	External Milestone	Critical Split					
Split	Project Summary	Inactive Summary	Manual Summary Rollup	Finish-only	Deadline	Progress					
Milestone	Inactive Task	Manual Task	Manual Summary	External Tasks	Critical	Manual Progress					

Appendix 3

RISK REGISTER

Client	Rupert Resources
Project	Pre-Feasibility Study
Title	Risk Register

ID	Discipline	Risk Title	Cause(s)	Consequence(s)	Existing Controls	Risk Rating			Proposed Additional Controls	Residual Risk Rating			Other Opinions/Notes
						Likelihood	Severity	Ranking		Likelihood	Severity	Ranking	
1-01	General	Impact of climate change on engineering designs	Climatic influences outside those assumed in the design assumptions (e.g. warmer/colder temperatures, more intense storm events, more/less precipitation, groundwater changes)	Failure of engineering controls (e.g. facility design) and operational (e.g. water) controls. Wildfires. Authorities ask to consider longer event horizons (100 year, 1000 year).	Inclusion of climate risk in design criteria. Climate change model.	Possible	Major	High	Climate change impact assessment to identify design parameters that account for anticipated/selected climate change scenario. Revisit throughout the LOM.	Unlikely	Major	Medium	
1-02	General	Cold Weather Risks	Excessive snow or cold due to local weather leads to loss of access or shut downs.	Impact to production.	Best practice for cold weather operations.	Unlikely	Minor	Low	Contingency planning for adverse weather.	Rare	Minor	Low	Refer to linked risk 1-103 RR experience from other Finnish mines is that this is not likely to occur.
1-03	Health & Safety	Fibrous minerals health hazard.	Risk of fibrous mineral release during mining and processing.	Air quality and health safety impacts.	Comminution and processing testing work underway by others.	Possible	Major	High	Mineralogical analysis of waste rock and ore. Dust sampling during trial processing. If required dust control and exposure management.	Possible	Moderate	Medium	
1-04	Health & Safety	Underground fire risks	Overheating, faulty equipment. Electric fault. Fuel spill and ignition.	Air quality impact from carbon monoxide, toxic gas, smoke or oxygen depletion. Risk to life.	Standard U/G fire safety procedure. Safety havens, egress routes, self- rescuers, equipment maintenance, fuel storage practice.	Possible	Major	High	Detailed ventilation and fire risk modelling in future phases to optimise safe evacuation plans and mitigation measures.	Rare	Major	Medium	
1-05	Operational Readiness/ Project Implementation	Difficulty securing skilled workers for the mine.	Logistics and skill sets for workers. Risk perception of operational duties. Competing projects and limited resource pool.	Increased labour costs. Poor technical performance of under skilled staff (blast efficacy, mill recovery)	Not yet determined	Likely	Moderate	High	Labour force analysis. Implement local training programmes to fill gaps. Increased contractor involvement. xpatriate labour.	Possible	Moderate	Medium	
1-06	Operational Readiness/ Project Implementation	Delays to critical path items	Unforeseen delays in key schedule items such as Permitting, Process Plant/ River Diversion & Pit Dewatering/ Mine Equipment/ Water Treatment or Co-Disposal Facility	Delay to first production, increased cost and longer schedule impact.	Project Implementation Planning at this Study Level	Likely	High	High	Further detailed scheduling at next study stage, inclusion of contigency in schedule.	Unlikely	High	Medium	
1-07	Operational Readiness/ Project Implementation	Site Material Volumes not balanced.	Excess or defecit of construction material resulting from the scheduling, cut/fill volumes or the quality of site won material compared with construction specification.	In the case of too little site won material, the import of large volume will have a higher cost and potentially schedule impact. In the case of too much site won material, additional space will need to be found on site for stockpiling and disposal.	High level material volume estimate included in Prefeasibility Study.	Likely	High	High	Detailed project implementation planning in subsequent studies and further material balance estimates based on additional ground investigations, as well as testing site materials for geotechnical properties. Identification and testing of local sources of rock. Trade off against import.	Unlikely	High	Medium	Refer to linked risk 1-64,1-87
1-08	Environment and permitting	Future regulatory changes	Changes to current regulations making permitting more challenging and delayed. There are planned authority responsibility changes that may impact start 2026.	Permitted area changes. Changes to water quality requirements/legislation. Changes to requirements to containment structures, and / or other changes in design. Potentially stricter regulations.	Permitting risks identified in Prefeasibility Study	Likely	Moderate	High	Permitting road map. Follow closely changes in legislation and practice related to environmental and mine permitting.	Likely	Moderate	High	Refer to linked risk 1-105
1-09	Environment and permitting	Permitting takes longer than expected.	Planning authority not satisfied with initial applications and requires revisions and updates leading to delays.	Project is delayed, costs increased.	Permit routes identified, experts and stakeholders engaged.	Almost Certain	Moderate	High	Plan engagement with relevant authorities and stakeholders. Production of high quality permitting application addressing all elements required by authorities	Likely	Moderate	High	
1-10	Environment and permitting	Insufficient climate change consideration in permitting applications.	Permitting authority requires more evidence of climate change considerations than anticipated.	Permitting delayed.	Risks identified at Prefeasibility Study.	Possible	Moderate	Medium	Further engagement with authorities to understand permitting requirements.	Possible	Moderate	Medium	
1-11	Environment and permitting	River diversion not permittable upon application	Authority denies river diversion permit	No permit. No mine.	Concept outlined at Prefeasibility Study.	Unlikely	Major	Medium	Further detail developed as part of Environmental Impact Assessment. Application for permit made.	Rare	Major	Medium	Refer to linked risk 1-36
1-12	Environment and permitting	Co-disposal facility concept/design perceived not credible for permitting due to authority/ public perception	Concept/design not credible for permitting due to authority/ public perception. Misidentification of co-disposal facility using layering as a co-mingling facility.	Permitting refused. Change of tailings concept. Delay.	Global best practice. Presented in town halls.	Possible	High	High	Engagement with public/ authorities to educate about co-disposal. Opportunity- safer tailings disposal method. Demonstrated technical underpinning in Prefeasibility Study document, explanation of co-disposal unit in Prefeasibility Study, client works closely with engineers to present concept correctly, and future engineering works.	Unlikely	Moderate	Medium	
1-13	Environment and permitting	Ecological compensation more costly/extensive than expected.	Uncertainty of regulator requirements	Additional costs.	Benchmarking existing projects.	Possible	Minor	Medium	Liaising and agreeing with relevant authorities.	Unlikely	Minor	Low	
1-14	Environment and permitting	Risks to water quality and quantity.	Waste on surface could risk contamination of water and reduce the quality.	Water may no longer meet requirements set by the authority and/or difficulty getting permit for proposed water bodies.	Modelling in Environmental Impact Assessment phase.	Possible	Moderate	Medium	Geochemistry testing. Site investigations and characterisation. Best practice surface water control on waste facility.	Unlikely	Moderate	Medium	
1-15	Environment and permitting	Flora and Fauna /sensitive habitats.	Protected/sensitive species or habitat types in impacted area.	Changes in plans, further mitigation requirements, delay, need for agreed permitting exceptions.	Flora and Fauna surveys. Impact assessment in Environmental Impact Assessment	Unlikely	Moderate	Medium	Further detailed Flora and Fauna studies if needed based on Environmental Impact Assessment; informal compensation measures; exception/derogation permit from nature authority	Rare	Moderate	Low	
1-16	Environment and permitting	Water flux from discharge.	The speed of water into the river may change due to changes in the water flux from discharge	Alteration of fish migration patterns	Modelling in Environmental Impact Assessment phase	Possible	Moderate	Medium	Further detailed water flux impact studies.	Unlikely	Moderate	Medium	
1-17	Environment and permitting	Archaeology.	Archaeological remains in project area	Changes in plans (infrastructure location etc) and/or delay in schedule	Archaeology survey as part of Environmental Impact Assessment	Unlikely	Minor	Low	Expected not required. If based on Environmental Impact Assessment it is needed, a full Archaeology inventory can be carried out.	Rare	Minor	Low	
1-18	Environment and permitting	Multiple external projects impact on the same river, Kitinen, and wider watershed of Kemijoki.	Overlapping permits for, and impacts from, different projects in same area.	Difficulty obtaining the required permit for this project, very strict permit conditions	Modelling in Environmental Impact Assessment phase/ considering ongoing and planned projects.	Possible	Moderate	Medium	Minimise impacts on surface waters, present robust controls, communicate to increase awareness and understanding among both authorities and the public.	Unlikely	Moderate	Medium	
1-19	Environment and permitting	Leväsaarenoja groundwater quality impact in restriction area and Naattuankangas.	Insufficient understanding of regional hydrogeology and underestimation of the cone of depression.	Reduced usability (there is at least one household well) and quality in Leväsaarenoja groundwater restriction area, and Naattuankangas. Negative public/authority attention.	Ground Water modelling and field investigations	Possible	High	High	Additional groundwater modelling (underway by others) additional hydrogeological investigations to be conducted in the future	Unlikely	High	Medium	
1-20	Environment and permitting	Infiltration of Kitinen river to Halssikangas Ground Water Restriction area.	Water level in Kitinen increased by combination of high flow seasons (snow melt), when flow through hydro dams is restricted or load from discharged water at Ikkari and other mines	Reduced quality in Halssikangas Ground Water restriction area. Kitinen is regulated river and regulating might also have impacts on Halssinkangas Ground Water area.	Interpretation of existing data. Environmental Impact Assessment with Environmental Evaluation Modelling System to account for other projects/actions and notes impacts.	Unlikely	Moderate	Medium	Additional monitoring data of Ground Water levels and Ground Water quality from Halssinkangas Ground Water restriction area would be useful before operation starts.	Rare	Moderate	Low	Existing data shows the Ground Water levels in Halssinkangas area are already so high that effects likely limited to only western parts of restriction area.
1-21	Environment and permitting	Mine dewatering lowers river flow.	Pumping natural drainage from the watershed to discharge pipe reduces natural flow.	Saittajoki runs dry or suffers drought on dry for even longer periods. Ecological changes or damage.	Existing high level modelling. Gap in knowledge identified in Prefeasibility Study.	Likely	Moderate	High	Integrated geomorphological and hydrogeological planning teams. Reduction of river intake. Further assessment and control implementations, e.g. diversion of surface water to maintain levels.	Unlikely	Moderate	Medium	
1-22	Environment and permitting	Unexpected nuisance impacts.	Unexpected factors, exceptional circumstances or errors during operation result in dust generation, vibration, noise or other nuisance impacts that bother residents and/or wildlife around the site.	Additional controls and remediation required. Reputation damage. Deterioration of natural environment	Design, Environmental Impact Assessment	Unlikely	Moderate	Medium	Monitoring nuisance. Outreach to locals.	Rare	Moderate	Low	
1-23	Environment and permitting	Opposition to mining causes complications in permitting	General distrust against mining (and/or this project in particular), resulting in increased scrutiny by permitting authorities and multiple appeals	Permitting is delayed, some permit conditions may be stricter	Design, Environmental Impact Assessment	Likely	Minor	Medium	Open communications with stakeholders to alleviate fears and concerns as far as is possible.	Possible	Minor	Medium	
1-24	Environment and permitting	New protected species arise during mine life	Species migrate to the mine area or immediate vicinity. Species such as eagle that are not easily fenced out.	New controls may be required to accommodate these species e.g. constraints in nesting season. Design may need to be changed.	Environmental Impact Assessment but cannot be predicted well.	Unlikely	Moderate	Medium	Nature monitoring throughout project life cycle.	Unlikely	Minor	Low	
1-25	Geology	Drill hole orientations of some holes may be subparallel to mineralization resulting in potential local grade bias.	Poor drill hole orientation with respect to orientation of mineralization.	Reduction in quantity of metal in deposit.	None.	Likely	Moderate	High	Replace holes with new holes at better angles during further drilling programs	Unlikely	Moderate	Medium	
1-26	Geology	Revised interpretation of mineral domains reduces metal content.	Mineral domain and lithological models were interpreted from drill hole data and may not accurately represent the geology or account for the full scale of geological variability due to the complex structurally deformed nature of the deposit.	Reduction in quantity of metal in deposit.	Considered in MRE classification. Indicated aim is +/-15% accuracy on an annual basis.	Likely	Moderate	High	Further resource conversion drilling to increase resource confidence to Measured. Constant re-evaluation and re-interpretation of data. In portion where further drilling occurs likelihood of fewer ounces reduces.	Possible	High	High	
1-27	Geology	High-grade outlier samples result in an over-estimation of grade and metal content.	The sample database contains some high-grade outlier values which can have a material impact on the resource	Reduction in quantity of metal in deposit	Considered in MRE. The QP has taken steps to reduce the impact of this data but there remains some uncertainty.	Possible	Moderate	Medium	See technical report. Further drilling, testing and interpretation to reduce likelihood though further domaining and/or revised capping. Ikkari is not particularly suspectable to this risk due non-extreme data skew.	Unlikely	Minor	Low	
1-28	Geology	Grade-tonnage relationship inaccurately modelled.	Lack of close spaced data and/or inaccurate modelling of spatially variance	Possibility of changes tonnage-grade relationship (either direction) leading to shorter mine life or lower grade	Theoretical relationship honoured as closely as possible for each domain	Possible	Moderate	Medium	Further drilling providing closer spaced data will better inform relationship. A small sample of the deposit drilled to grade control spacing would provide best control.	Unlikely	Moderate	Medium	
1-29	Geology	Grade continuity shorter than anticipated.	Short range continuity of grade a possible characteristic of any deposit, especially gold deposits.	Poor reconciliation between mining plan and mill/stockpiles leading to lower recoveries.	10m spacing grade control drilling included in Prefeasibility Study. Selected WOL more resilient to unexpected grade fluctuations than Floatation option.	Possible	Minor	Medium	Further drilling providing closer spaced data will better inform grade control spacing. In the end grade control drilling might need to be closer spaced to provide accurate grade to mill/stockpile data.	Unlikely	Minor	Low	
1-30	Geology	Composition of waste rock	Risk classification of waste rock incorrectly assigned due to incorrect geological interpretation and/or estimation of relevant elements	Flow of material to co-disposal not what is expected leading to incorrect placement of Non-Acid Generating and Potentially Acid Generating waste	Majority of drillholes have multielement data and substantial effort to interpret distribution of waste rock.	Likely	Moderate	High	Design co-disposal is such a way that flexibility exists to classify waste material during waste stripping. Overall design with tolerances to allow for changes in overall balance of waste material.	Unlikely	Minor	Low	
1-31	Geology	De-watering wells not performing to expectation	Structures that control hydraulic conductivity not correctly interpreted and wells drilled in wrong place or more permeability in unfractured bedrock	Lower volumes of non-contact water and Higher volumes of contact water leading to increased water treatment costs and/or impact on production.	Structures modelled from all available drillhole but these are not optimised for these structures.	Likely	Moderate	High	Drill program to firm up location of structures prior to installation of dewatering infrastructure.	Rare	Moderate	Low	
1-32	Mine Water	Water balance - excess or insufficient water	Insufficient understanding of groundwater and surface water regime, operational controls, water storage and demand.	Excess of water requiring disposal, in particular during storm events, or insufficient water for process requirements. Potential to impact springs, leading to them drying out and vulnerable species destruction.	High level study completed in Prefeasibility Study. Spring surveys completed in 2023. Thermo-images identify water at surface.	Possible	High	High	Development of a dynamic probabilistic water balance to quantify risks, based on an understanding of the mine design and hydrological information at later stage. Updating models through life of mine. Spring surveys and baseline studies.	Unlikely	High	Medium	

Client Rupert Resources
Project Pre-Feasibility Study
Title Risk Register

ID	Discipline	Risk Title	Cause(s)	Consequence(s)	Existing Controls	Risk Rating			Proposed Additional Controls	Residual Risk Rating			Other Opinions/Notes
						Likelihood	Severity	Ranking		Likelihood	Severity	Ranking	
1-33	Mine Water	Water management - insufficient inflow control	Insufficient hydrogeological and hydrological characterisation leading to large uncertainty in the hydrogeological conceptual model.	Water ingress in excess of control capacity leading to mine flooding and or uncontrolled discharges from site	High level study	Possible	Major	High	Hydrogeological and hydrological investigation, including the installation of monitoring facilities and modelling of inflows.	Unlikely	Major	Medium	
1-34	Mine Water	Hydrogeological conceptual model uncertainty	Hydrogeological conceptual model is uncertain. Conflicting information and concepts being applied. Current test work is focussed on the orebody but may not be reflective of the rest of the site.	Rate of groundwater ingress to the open pit and or underground mine are underestimated or over estimated and pore pressures poorly understood.	Previous hydrogeological study by others. Ongoing study by others.	Likely	High	High	Additional investigation required and hydrogeological conceptual model to be revisited in conjunction with geological and geotechnical model. Further detailed study of river diversion including hydraulic modelling and flood risk modelling.	Unlikely	High	Medium	
1-35	Mine Water	Flooding of Underground or Open Pit mine	Groundwater and/or surface water flooding. Underground is more likely to be affected by groundwater flooding. Flooding could result from river diversion seepage at tributary head or river during diversion. Flooding is most likely during the snowmelt season	Mine is flooded, affects work schedule and could affect personnel working in the mine Rehabilitation of mine needed post flooding. Health & Safety Risks.	PFS considers at high level, ongoing study by others.	Possible	Major	High	Creation of flood embankments as river is being diverted. Additional studies of the river diversion, including hydrological modelling. Determination of acceptable surface flood risk at the bottom of the pit. Additional data in the hydrological model	Unlikely	Major	Medium	
1-36	Mine Water	Surface water diversion route not suitable	Low level of knowledge regarding terrain traversed by proposed surface water diversion from a topographic, geotechnical, hydrological, hydrogeological perspective.	Route not achievable for technical reasons. Potential for backflow into pit if gradients not accurate	Further testing and design recommended by Prefeasibility Study	Possible	High	High	Detailed topographic, hydrological and geotechnical survey of proposed route. Hydraulic modelling to be completed on final permittable design.	Unlikely	High	Medium	Refer to linked risk 1-11
1-37	Mine Water	Groundwater quality issues	More saline (or mineralised) than anticipated	Process efficacy, water treatment, discharge to environment. Potential mixing with shallower groundwater and changing quality.	Limited testing.	Likely	Moderate	High	More testing. Further characterisation. Contingency planning for process/treatment.	Possible	Moderate	Medium	Refer to linked permit risks 1-14,1-16,1-18,1-19,1-20,1-21
1-38	Mine Water	Cannot discharge untreated groundwater from boreholes to Saittajoki	Groundwater quality does not meet discharge permit.	Delays in construction, increased cost of construction.	None. Groundwater quality assumed suitable for discharge.	Possible	High	High	Complete additional testing on groundwater quality. Potential to implement solutions such as temporary water treatment plant (pre-mine development), build full water treatment plant early or advanced pipeline construction to larger river which could be permittable.	Unlikely	Moderate	Medium	
1-39	Geochemistry	ARD and deleterious mine drainage	Understanding of the geochemical behaviour of the mine waste (rock and tailings) and in-situ rock (pit and underground) is limited hence understanding of the potential impacts on the environment (especially water) is limited.	Potential for significant impact on water quality from mine water discharges, seepage from waste and construction materials.	Initial Consultant (MEM) geochemistry study	Likely	High	High	Ongoing geochemistry study. Integrate with hydrogeological and hydrological studies.	Unlikely	High	Medium	Refer to linked permit risks 1-14,1-16,1-18,1-19,1-20,1-21
1-40	Water Treatment and discharge pipeline	Water Treatment Plant demand exceeds capacity and leads to spill	Higher than expected demand due to uncertainty in groundwater quality and volumes. Bench-scale data for tailing seepage delayed or change in water balance may lead to incorrect treatment plant sizing.	Demand exceeds capacity. Water ponds could overtop & spill into environment. A stop work order may result from authorities, reputation of RR and designer damaged.	Balancing pond size and water treatment plant capacity estimates.	Likely	Major	Very high	Further underpinning on inflow and water quality estimates (focus next study stage on ground investigations and more detailed water balance) and technical design of mitigation (focus on buffer and plant capacity). Allow for flexibility for retrofitting.	Likely	High	High	
1-41	Water Treatment and discharge pipeline	Water treatment plant oversized.	Lower than expected demand due to uncertainty in groundwater quality and volumes. Bench-scale data for tailing seepage delayed or change in water balance may lead to incorrect treatment plant sizing.	Capacity far exceeds demand. Conservative design (Higher CAPEX) or not being able to comply with environmental permit discharge into the Kitinen River.	Coordination with consultant preparing long tailing seepage tests in next phase. Use of best practice design, benchmark against existing mines.	Likely	High	High	Water treatment requirement to be reviewed in subsequent design stage. Allow for flexibility for retrofitting.	Possible	High	High	
1-42	Water Treatment and discharge pipeline	Asset or water quality failure	Change in influent water quality (higher concentration /lower concentration) Power cut Treatment failure	Unable to treat water Water quality out of specification	Provision of storage - raw water pond, treated water pond. Monitoring provided from the inlet to the outlet of the treatment plant Diversion to treated water pond if water quality is out of specification	Possible	Moderate	Medium	Further data on water quality. HAZOP/HAZID reviews at later stage. Pilot water treatment plant.	Unlikely	Moderate	Medium	
1-43	Water Treatment and discharge pipeline	Waste streams disposal	Production of sludge, gypsum and mixed slat slurry by the 4 treatment streams. Unable to dispose the waste streams on-site (capacity fully utilised) or off-site (third party not accepting waste streams)	Site operation to be ceased.	Disposal area provided on-site for gypsum slurry only. Assumption that sludge (from wastewater treatment and coagulation/clarification process from water treatment plant 1) is be tankered off-site to the nearest wastewater treatment works. Assumption that Mixed salt slurry is to be disposed off-site in accordance with waste management regulations.	Possible	Major	High	Further engineering and underpinning work on full feasibility study for disposing waste streams	Possible	Major	High	
1-44	Water Treatment and discharge pipeline	Discharge pipeline failure/unavailable	Pipeline burst. Pipeline maintenance.	Unable to discharge water into pipeline. Pollution event.	Treated water storage for a minimum of 7 days Pipeline monitoring to identify any loss/burst.	Possible	Moderate	Medium	Detailed potential failure mode analysis at future stages e.g. HAZOP/HAZID.	Unlikely	Moderate	Medium	
1-45	Geotechnical	Mining-induced stress (interaction)	Only preliminary understanding of interaction between Open Pit & Underground operations, various underground mining fronts, primary & secondary stope lines, stopes and mining drives, through elastic Boundary Element Method modelling without fault zones.	Underground excavation stability impacted. Changes to underground mine design based on numerical modelling with plastic yield zones, and explicitly including fault zones.	Recommended 3D mine-scale elastic Boundary Element Modelling for next phase.	Likely	Moderate	High	3D mine-scale elastic-plastic modelling, accounting for anisotropic behaviour where appropriate, and including fault zones, to investigate whether yield and de-stressing occurs in secondary pillars, what the impact is of underground excavations on pit wall stability, crown pillar stability, and stand-off distance of infra-structure.	Unlikely	Moderate	Medium	Refer to linked risk 108
1-46	Geotechnical	Effect of hydrogeology for open pit stability	Coupling of rock mechanical stability analyses and hydrogeological model not completed.	Over or under estimation of stability, leading to either inadequate or overly conservative design.		Possible	High	High	Addition of hydrogeological model into slope stability modelling.	Unlikely	High	Medium	
1-47	Geotechnical	Uncertainties in structural modelling and characteristics	Lack of geotechnical drilling data especially on Southern and Eastern areas to accurately define and understand thickness and location of fault zones. Length of solid core pieces from fault zones typically insufficient for laboratory testing.	Effect on overall slope stability. Where large-scale structures (e.g., Southern Fault Zone) intersect the orebody, pillars might have to be left in place or stope sizes will have to be reduced.	Selection of a retreating sequence away from the Southern Fault Zone to reduce the amount of potential seismic energy release in close proximity to the fault plane. Reduced overall open pit slope angles in affected areas.	Possible	High	High	Improved structural model. Definition/Estimation of the characteristics of the Southern Fault Zone, allowing inclusion in elastic-plastic model to assess interaction with pit wall and underground mining headings, and more detailed assessment of the effect of mining sequencing on fault slip tendency. Perform a critical stress analysis on the Southern Fault Zone (if the shear stress across the fault exceeds its shear strength, the two opposing faces slip, i.e., the fault is critically stressed)	Unlikely	High	Medium	Refer to linked risk 1-88
1-48	Geotechnical	Uncertainty in rock mass properties (i.e. rock mass jointing) and rock mass quality	Limited amount and spatial distribution of geotechnical logging and laboratory testing, especially in heterogeneous domains.	Limited confidence in ability to accurately identify and characterize geotechnical domains. Over or under estimation of stability, leading to either inadequate or overly conservative design. This will result in higher Capital Expenditure due to design alterations or reduced productivity from increased dilution.	Identification of potential heterogeneous domains and current variability of laboratory data through CoV (Coefficient of Variation)	Possible	Moderate	Medium	Design targeted geotechnical data collection campaign as part of the next phase of drilling with best practice sample collection for laboratory testing, focussing on areas with limited coverage, critical mine design aspects.	Unlikely	Moderate	Medium	
1-49	Geotechnical	General underground operational geotechnical risks	Fall of Ground causing fatality or permanent disability Severe damage to remotely operated scoop tram due to loose from stope walls during open stope mucking. Major rehabilitation campaign of mining area due to unexpected ground behaviour. Ground support system or elements do not perform.	Fatality or permanent disability, significant delays to the mining schedule, increase in Operational Expenditure, damage to mining fleet.	Systematic approach to ground control using support categories and only allowing personnel under supported ground.	Possible	Major	High	To be developed in run up to start-up of operations, through best practice project implementation, design and construction planning: Ground Control Management Plan. Good Standard Operating Practices and reporting culture. Implementation of Design Monitoring and Safety Monitoring Devices. Just-in-time development reduces costs and delays related to preventative maintenance of ground support and rehabilitation. Stope reconciliation. Face mapping / Scanline mapping. Frequent visual inspections by qualified geotechnical engineer	Unlikely	Major	Medium	
1-50	Geotechnical	In-situ stress	The orientation of the in-situ stress field is estimated, but no stress measurements have been conducted on site. Cause if estimate differs from actual.	Different stress orientation or magnitude will cause different mining-induced stresses on stopes and mining drives.	Benchmarks from World Stress Map and nearby Nordic mines, Structural geology review, sensitivity analysis on stress orientation	Possible	Moderate	Medium	Confirmation of in-situ stress understanding once underground development has started by means of observations and, if warranted, over coring stress measurements.	Rare	Moderate	Low	
1-51	Geotechnical	Inaccurate numerical modelling of large scale Open Pit stability	Prefeasibility Study stage rock mechanics incorporated only 2D numerical modelling of the Open Pit stability to determine overall angles.	Over or under estimation of stability, leading to either inadequate or overly conservative design.	PFS used stability analysis methodology can be argued to be on conservative side.	Possible	Moderate	Medium	3D analysis to be performed in subsequent study phases.	Unlikely	Moderate	Medium	
1-52	Geotechnical	Effect of foliation and rock anisotropy not captured in rock mechanical analyses	Established analysis methods expect isotropic, homogenous rock mass and might fail to capture failure mechanisms induced by strongly anisotropic rocks.	Under estimation of bench scale and/or overall stability. Design angles are not achieved in production. Stability issues causing loss of production/equipment or even personal injuries/deaths.	Usage of anisotropic material models where applicable. Choose of analysis methods. Conservative estimates of rock mass strength for heterogenous, anisotropic rock masses.	Possible	High	High	Further characterization of anisotropy and anisotropic properties of rock mass domains. and strength anisotropy. Assess variability within MSCU and assign sub-domains if required (for example on heterogeneity or foliation intensity).	Unlikely	Moderate	Medium	
1-53	Paste Backfill	Backfill Recipe Uncertainty	The backfill testing performed for this study was done on the leach tails portion of the sample (Option 1:Flotation - flotation concentrate to leach process). The sample tested is not representative of the current mill process and may respond to binder differently. The current mill process is whole ore leach (no flotation prior).	The binder content required to achieve the design backfill strength may be different than the values carried in this study	None	Likely	Minor	Medium	Repeat backfill testing when a representative sample becomes available	Unlikely	Minor	Low	
1-54	Paste Backfill	Binder Cost Uncertainty	The cost of cement and other cementitious supplemental materials have increased in price significantly in the last few years. Future environmental considerations with the manufacture of cement are likely to result in the continued increase in pricing beyond inflationary values.	Increased backfill costs beyond current estimates	Not yet determined	Likely	Moderate	High	Consider adding escalation factor to the cost of binder being carried in the cost estimate. Opportunity to investigate binder replacements to mitigate cost increases.	Likely	Moderate	High	
1-55	Paste Backfill	Surface Paste Line Blockage	The paste plant is 350 m from the borehole at the edge of the pit. This horizontal line on surface is more susceptible to blockage than underground or vertical lines	Lost backfill plant production time	Operating procedures, high pressure flush pump	Likely	Moderate	High	A standby surface pipeline was added to mitigate any issues with the operating surface pipeline.	Rare	Moderate	Low	
1-56	Paste Backfill	Power Outage Risk to Backfill System	Power outage	Lost backfill plant time, loss of equipment if cannot be cleaned.	Putting plant equipment on backup power	Likely	Moderate	High	Provide an emergency power source.	Unlikely	Minor	Low	Refer to linked risk 1-70
1-57	Paste Backfill	Excessive Paste Pipeline Wear	Tailings properties or high operating pressures.	Increase backfill operating costs.	Sustaining capital includes some replacement piping	Possible	Minor	Medium	Different piping materials. i.e. induction hardened.	Unlikely	Minor	Low	
1-58	Paste Backfill	Paste Recipe Underperformance	Variations in backfill plant feed material (e.g. different ore body, grind)	Possible sterilization of a stope	Backfill testing program	Rare	Moderate	Low	Ensure adequate paste testing program is in place and requirement are followed by operation	Rare	Moderate	Low	

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1-59	Mine Waste	Use of Overburden in Foundation to Co-Disposal Facility	Availability/ suitability (material properties) of overburden from open pit for use as engineering fill in foundation to co-disposal facility, including clay required for the low permeability liner not yet determined.	Insufficient materials during construction leading to increased costs or delayed construction.	Identified at Prefeasibility Study	Possible	High	High	SI and best practice design recommended. Specifically by testing of the overburden's geotechnical properties to confirm suitability and further material balance underpinning through construction scheduling to confirm availability.	Unlikely	High	Medium	Liner Seepage and GW mass transport modelling recommended following SI to map potential GW during operations, closure and long term
1-60	Mine Waste	Overall placement of co-disposal facility is located somewhere not permissible/permittable	Proximity to local water sources, etc	Relocation of co-disposal. Refusal of permit.	Site selection work.	Possible	High	High	Engagement with public/ authorities to educate about co-disposal location and design. Consider 'Plan B' locations. Environmental Impact Assessment to consider alternatives.	Unlikely	High	Medium	
1-61	Mine Waste	Suitability of Soils and Bedrock Beneath the Co-Disposal Facility	Suitability of soils and Bedrock underlying co-disposal area, to avoid differential settlement to be assessed.	Seepage of contact water into the environment and slope instability of stack, possible loss of life.	Risk identified at Prefeasibility Study.	Possible	High	High	Ground surveys and hydrogeological investigations to be continued 2024-2025 winter. Further site investigations and best practice design recommended	Unlikely	High	Medium	Liner Seepage and GW mass transport modelling recommended following SI to map potential GW during operations, closure and long term
1-62	Mine Waste	Co-disposal facility: Undefined Tailings and Waste Parameters	Tailings and waste parameters are undefined.	Re-evaluation of stability and safety measures	Identified at Prefeasibility Study	Possible	Moderate	Medium	Lab testing and best practice design recommended	Unlikely	Moderate	Medium	
1-63	Mine Waste	Balance of engineering soil volumes.	Undefined quantities of cut and fill for engineering soils. Too much or too little.	Redesign of facilities to accommodate balance. Deficit; Borrow pit potentially required. Excess; elevation of co-disposal facility raised.	Requirement for storage facility identified at Prefeasibility Study	Possible	Moderate	Medium	Allow flexibility to designs to accommodate variances. Review design and volume balance at engineering stage gates. Consider using temporary stockpiles of "benign" waste rock and till for construction and closure balance	Unlikely	Moderate	Medium	Refer to linked risk 1-07
1-64	Mine Waste	Lower early strip ratio results in thicker tailings layers at Co-disposal facility base.	Change to mine plan results in lower strip ratio during early life of mine.	Deeper layers of tailings at lower elevations of the co-disposal facility. Re-evaluation of stability and safety measures. Higher cost from additional drainage layers and engineering. Deviation from agreed permitting, potential re-permit or permit refusal.	Consider minimum strip ratio and layer depth	Unlikely	High	Medium	Potential to mine waste in advance. Consider stockpiles of waste rock outside of planned area. Further definition of mine planning to reduce potential for change. Consider flexibility in the permitting application for this risk.	Rare	High	Low	
1-65	Mine Waste	Cannot achieve density of co-disposal facility planned.	System of working not established. Not achieving compaction of tailings, especially during winter months. Assumptions on rock variance, recompaction efficiency and tailings density found to be optimistically high.	Sub-optimal strength/density from tailings in co-disposal facility. Re-evaluation of stability and safety measures. Potential instability of facility. Greater volume of material, larger co-disposal facility.	Compaction trials recommended in Prefeasibility Study. Prefeasibility Study has contingency capacity in concept.	Possible	Moderate	Medium	Complete compaction trials. Establish working method. Construction Quality Assurance. Density testing at later study stages. Incorporation of specific unit densities in later study stages.	Unlikely	Moderate	Medium	
1-66	Mine Waste	Sterilisation of Heina Central and Eastern outside limits of open pit by Co-Disposal Facility	Co-disposal facility preventing exploitation of Heina Central or Eastern extension of open pit.	Resources sterilised or waste and tailings will need to be relocated.	These deposits not included in resource estimate or mine plan for Prefeasibility Study.	Possible	Minor	Medium	None	Possible	Minor	Medium	
1-67	Infrastructure	Limited Lands Made Available	Limited lands made available for surface infrastructure	More costly surface infrastructure	Site layout in Prefeasibility Study	Possible	Moderate	Medium	More detailed definition of building footprints and layout required at later stages.	Unlikely	Moderate	Medium	
1-68	Infrastructure	On-site water treatment sludge storage facility	Requirements for storage of sludge from water treatment process not defined at Prefeasibility Study	Additional area may be required at a later stage	Further testing and design recommended by Prefeasibility Study	Almost Certain	Moderate	High	Sludge characteristics and quantities to be determined, then on site footprint needs sizing and technology/disposal method costed.	Possible	Moderate	Medium	
1-69	Infrastructure	Undefined Foundation Conditions	Insufficient SI data for design of foundations for surface structures	Failure of foundations of surface structures and/or delays during construction	Identified at Prefeasibility Study	Possible	Moderate	Medium	SI and best practice design recommended	Unlikely	Moderate	Medium	
1-70	Infrastructure	Power Supply Risks	Arctic conditions and high winds or some other issue with national power grid leading to a loss of power	Impact to production from equipment or plant downtime.	Best practice for cold weather operations.	Unlikely	Moderate	Medium	Contingency planning for loss of mains power. The 20kV line used for temporary site power is independent of the national grid and could be used a backup.	Rare	Minor	Low	RR experience from other Finnish mines is that this is not likely to occur. Generator not thought to be required.
1-71	Infrastructure	Site Access Road Ground Condition Uncertainty	The depth to rock under the site access road is unknown	Potential cost and schedule overruns during construction due to deeper/shallower rock than planned	Ground radar (low accuracy)	Likely	Minor	Medium	Further ground investigations (geophysics/shallow boreholes) to determine rock profile	Unlikely	Minor	Low	
1-72	Infrastructure	Flooding water at portal	Proximity of water discharge pipeline to Underground portal	Water flowing underground	Pipeline buried underground	Possible	High	High	Provide facility for flush-out into to the raw water pond as needed and portal entrance drains.	Rare	High	Low	
1-73	Process	Spills to environment	Operational error, failure of control systems and instrumentation	Intervention from Authorities, possibility of halting operations and environmental/reputational damage.	More frequent checks of instrumentation/calibration/equipment	Likely	High	High	Ensure secondary containment for reagents/CIL area meets best practice/ Finnish regulation. Ensure building has sufficient containment in the Definitive Feasibility Study stage. Show containments in the 3D layout in the next phase.	Unlikely	High	Medium	
1-74	Process	Sample grade higher than planned feed in plant design	Operating Run Of Mine grade not matching planned ROM grade.	Impact on plant recovery/performance	Variability testing started in Prefeasibility Study stage. Blending stockpile included in Prefeasibility Study.	Possible	Moderate	Medium	Mining to examine blending further, complete variability testing during Definitive Feasibility Study (to make changes to design if necessary)	Unlikely	Moderate	Medium	
1-75	Process	Insufficient filtration capacity for plant design	Insufficient test work data supporting filtration	Not meeting production targets, not meeting final tails moisture targets (too wet)	Spare filter press in design if additional capacity required.	Possible	Moderate	Medium	Do thorough filtration test work during Definitive Feasibility Study. Ensure spare remains just as a spare.	Unlikely	Moderate	Medium	
1-76	Process	Plant design not meeting production target	More preg-robbing material than anticipated	Lower Au recovery	Design consists of CIL circuit	Possible	Moderate	Medium	Investigate preg robbing with additional test work in Definitive Feasibility Study	Rare	Moderate	Low	
1-77	Process	Plant design: Reagents OPEX can be higher/(lower) than anticipated	Reagent consumptions test work results not always usable, depending on ore coming to the mill.	Higher costs with potential impact to profit	Preliminary test work has been performed during Prefeasibility Study	Possible	Moderate	Medium	Complete reagent dosage test work on variable samples. Re-do optimization test work for Definitive Feasibility Study. Regular testing recommended during operations.	Unlikely	Moderate	Medium	
1-78	Process	Plant design: Filtered tailings storage capacity is not sufficient	Issues with tailings deposition equipment availability or major meteorological event.	Run out of room in tails shed, operations will need to stop	Tailings shed is designed for 18h production capacity	Possible	Moderate	Medium	Review building temporary storage or review capacity during Definitive Feasibility Study stage.	Rare	Moderate	Low	
1-79	Process	Failure of filter plant/ no emergency tailings disposal facility	Maintenance issues in filter plant	Stop in production	Spare filter press in design. There are 3 filter presses in the design (2 operating, 1 standby). Plant can also operate at reduced tonnage with just 1 filter press available.	Unlikely	Moderate	Medium	None	Unlikely	Moderate	Medium	
1-80	Process	Recycled water quality not suitable	Accumulation of elements in water not taken out by treatment.	Impact on recovery, reagent consumptions and OPEX	Water treatment concept at Prefeasibility Study	Possible	Moderate	Medium	Further testing. Modelling.	Unlikely	Moderate	Medium	RR instructed Pilot plant not possible due to project constraints.
####	Process	Low temperature impact on processing	Frozen pipes. Mill feed material temperature (compacted frozen chunks of feed).	Shut downs, reduced availability of plant.	Prefeasibility Study design, tanks inside, heating, insulation etc.	Possible	Moderate	Medium	Further detailed design/specification. Heat tracing/Insulated pipes in the P&ID. Ore stockpile in covered dome, heated reclaim tunnel. Consider in the HAZID/HAZOP.	Rare	Moderate	Low	
1-82	Mine Closure	Pit lake water quality unsuitable for discharge.	Pit water quality may not be suitable for discharge to the environment once the pit flooding is complete. Groundwater naturally does not meet anticipated discharge quality.	Reduced surface water entering post-closure re-connected stream. Water treatment plant will be decommissioned following Year 29 (approximately), but the pit is not expected to be flooded until year 45 or 50. At this time if water quality is unsuitable for discharge direct to the environment, solutions are predominantly undesirable (costly, extend active closure activities):	Confirmation of discharge water quality requirements / permit requirements. Conduct water quality and quantity modelling and update with renewed data (pump rates, water chemistry) regularly such that closure water management of the open pit may be planned for with greater certainty over the operational mine life. Update the Open Pit Closure Plan and water treatment plan/design as needed during operations based on new information/data recorded.	Likely	Major	Very high	1. Consider leaving the water treatment plant in place for re-start once the pit floods but prior to the discharge elevation being reached. 2. Construct a new water treatment facility if initial WTP is demolished per schedule. 3. Consider leaving dewatering wells in place for down drain/pumping and recycling through pit. 4. Re-establish dewatering wells if closed, and pump groundwater such that pit lake does not overflow, 5. investigate passive water treatment and storage options.	Likely	Major	Very high	Refer to linked permit risks 1-14,1-16,1-18,1-19,1-20,1-21
1-83	Mine Closure	No site-wide water balance or water quality modelling to date.	No site-wide water balance or water quality modelling completed to date.	Water treatment could be required for longer than current assumption. Water treatment may be required for a greater capacity/volume than the plant was designed for. Higher costs for water treatment than estimated.	More detailed studies will progress in next stage of design.	Possible	High	High	Complete water balance and water quality modelling to understand full environmental and economic implications. Develop updated water treatment plan. Conduct trials to demonstrate proof of concept, and refine treatment plan and design. Complete hydrogeology modelling and water balance model site wide. Contingency in closure cost estimate.	Unlikely	High	Medium	Refer to linked permit risks 1-32
1-84	Mine Closure	Potential breach of permit conditions	The water discharge permit conditions are unknown. Surface water from the site post-closure is to be discharged into the local water courses, likely to have high ecological status. Closure Plan currently assumes treatment to enable discharge into the Kitinen river.	Degree of water treatment required is higher than expected. Substantial additional expense to achieve discharge criteria, for extended period of time.	Coordination with Environmental Consultant (Envineer), mine closure and mine water team.	Likely	High	High	Assessment of water quality requirements for a discharge into the local water body plus treatment adjustment as required. Site wide water quality model and water balance model both for operations and closure timeframe. Determine treatment.	Unlikely	High	Medium	Refer to linked permit risks 1-14,1-16,1-18,1-19,1-20,1-21
1-85	Mine Closure	Co-disposal facility- Inability to achieve post-mining land use	Slope gradients of co-disposal facility are too steep for reindeer to comfortably traverse. Costs of a larger co-disposal footprint are prioritized over sustainable post-closure land use.	Post-mining land use not achieved. Reputational damage with regulator and reindeer herding community; potential implication for future approvals and social license.	Use of slope characteristics for reindeer herding lands (north facing slopes only, steepest gradients) in design.	Possible	Moderate	Medium	Increase co-disposal footprint in subsequent design stage and reduce slope gradients. Find and review reindeer behaviour/data at Definitive Feasibility Study stage.	Unlikely	Moderate	Medium	Refer to linked risk 1-12
1-86	Mine Closure	Insufficient financial planning (estimation) and allocation of funds for closure during operation.	Optimism bias. Poor data or lack of data. Failure to allocate sufficient funds for progressive reclamation and/or closure trials during operations.	Insufficient funds to close mine on schedule/budget. Inability to surrender permits or draw down on any closure security "bonds". Inability to transition land to next use; reputational damage with community and regulator. Potential to impact ability to open future mines or achieve approval of mine life extensions. Long term unacceptable risks to the environment and human health. Failure of social transition. Long term liability.	Closure cost estimate has contingencies included, intended to compensate for lack of information at this stage.	Likely	Major	Very high	Additional studies required to reduce uncertainty of closure cost estimate.	Possible	Major	High	
1-87	Mine Closure	Material Availability Risk	Quality or quantity of till and/or waste rock available on site is inadequate for co-disposal closure uses.	Need to re-design cover, or source material from an off-site location.	Estimates from geological and geochemical block model properties.	Possible	Moderate	Medium	Refine quantity estimates on a continual basis moving forward using further testing and underpinning works to understand site won material properties.	Possible	Moderate	Medium	Refer to linked risk 1-07

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1-88	Mining	Underground development impacted by southern fault structures	Uncertainty in geotechnical properties and support requirements for southern fault structures.	Slower than planned rates of advance through these zones, production plans thrown off schedule.	Geotechnical based design	Possible	Moderate	Medium	Further geotechnical data collection and analysis	Unlikely	Moderate	Medium	Refer to linked risk 1-47
1-89	Mining	Underground stope design with erratic ore boundary.	Assumed boundary to orebody is more erratic than assessed at this stage.	Higher ore loss and dilution at ore boundary stopes	Geological model basis of design	Possible	Moderate	Medium	Assess any further drilling crossing the geological model boundaries to improve understanding of ore boundary.	Unlikely	Moderate	Medium	
1-90	Mining	Underground Dilution, Ore Loss & Stope Recovery underestimated	Modelled estimates during study underestimate compared to actual mining	Reduction in recovered gold resulting in reduction in revenue. Increased processing costs due to lower ROM grade.	Current dilution estimates quite conservative. Allowance for re-drilling in cost estimates. Provisions for development overbreak, paste/waste overbreak also made.	Possible	Moderate	Medium	Detailed reconciliation program once mining to ensure dilution, ore loss and stope recovery are understood.	Unlikely	Moderate	Medium	
1-91	Mining	Underground Achieving ramp-up	Mine production ramp-up less than planned due to schedule slippage, high demands, ground conditions, new equipment etc.	Potential reduced production and lower feed to the plant and reduction in revenue.	Prefeasibility Study Production schedule provides an understanding of critical path development and infrastructure required to meet ramp up.	Possible	Major	High	Appropriate Operational Readiness will be required to support ramp up. Further studies will allow for a better understanding of requirements and allow for fine tuning of the schedule to eliminate unnecessary development/mining. Further investigations of critical paths.	Unlikely	Major	Medium	
1-92	Mining	Underground Inability to meet advance rates.	High advance rates required to meet production requirements.	Potential reduction in plant feed and reduction in revenue.	Prefeasibility Study production schedule provides a detailed understanding of required development to achieve production rates. Lateral development set at 75 m/mo. Main access at 110 m/mo. Main risk is multi-heading development.	Possible	Major	High	Similar to ramp up, further studies will allow for a better understanding of requirements and allow for fine tuning of the schedule to eliminate unnecessary development.	Unlikely	Major	Medium	
1-93	Mining	Underground Inability to develop the required number of headings.	Unrealistic number of headings required to meet production requirements.	Potential reduction in plant feed and reduction in revenue.	Prefeasibility Study production schedule provides a detailed understanding of critical headings to achieve production rates.	Possible	Major	High	Similar to ramp up, further studies will allow for a better understanding of requirements and allow for fine tuning of the schedule to eliminate unnecessary development.	Unlikely	Major	Medium	
1-94	Mining	Underground Stoping Sequence	Stoping sequence not suitable for mine.	Potential reduction in plant feed and reduction in revenue.	A primary/secondary sequence has been implemented with appropriate pillar sizes. Sequence is not overly demanding.	Unlikely	High	Medium	Implementation of mine planning/ operation controls.	Rare	High	Low	
1-95	Mining	Open Pit/ Underground Mismatched equipment	Equipment chosen is mismatched to the production or development requirements.	Reduction in productivity leading to increased mining costs.	Underground Equipment has been sized according to heading dimensions and production requirements. Open Pit specified by manufacturer.	Unlikely	High	Medium	Reassessment at next study stage. Opportunity for Open Pit- improved equipment selection for both main load/haul and ancillary equipment. Potential for lower cost runs with different equipment than that specified by the manufacturer.	Rare	High	Low	
1-96	Mining	Underground Ventilation heat load	Heating requirements for mine different than estimated.	Risk to worker safety as mine temperature is not meeting regulations. Additional development may be required including drifts and shafts.	Ventsim modelling performed to Prefeasibility Study standard.	Unlikely	Moderate	Medium	Further progression through studies will refine ventilation requirements.	Rare	Moderate	Low	
1-97	Mining	Underground Lack of ventilation	Infrastructure in place not enough to supply/exhaust air to and from the mine.	Additional development may be required including drifts and shafts.	Ventsim modelling performed to Prefeasibility Study standard.	Unlikely	Moderate	Medium	Further progression through studies will refine ventilation requirements.	Rare	Moderate	Low	
1-98	Mining	Open Pit/Underground Second Access from Pit Not Possible	Issues with switchback currently used for second decline access to underground mine, such as slope stability in area, traffic/separation safety issues make too large to accommodate here.	Inability to mine in the prescribed sequence could result in delays/reduction to plant feed. Relocation of second means of egress, potentially a hoist egress in a raise. Reduction in productivity leading to increased cycle times and mining costs.	Schedule not critically dependent on second access. Conservative truck speeds used in truck hour estimation	Possible	Moderate	Medium	Further progression through studies will assess open pit and underground access interaction in further detail.	Unlikely	Moderate	Medium	
1-99	Mining	Open Pit/Underground Geotechnical Interaction	Stresses higher than expected. Open pit mined slower or underground mined quicker than expected.	Loss of recoverable ore below open pit and in underground mine. Possible hazards in open pit.	10m offset from open pit installed. No mining to be performed within 50m of open pit until open pit is concluded.	Possible	Moderate	Medium	Further studies will allow for iterative analysis of open pit and underground interactions. More data will provide further understandings of rock mechanic properties.	Unlikely	Moderate	Medium	Refer to linked risk 1-45
1-100	Mining	Open Pit accessed by single ramp, single failure point.	Any stability, traffic, weather, breakdown, blockage, or other issues and failures occurring on the ramp.	Loss of access to open pit until ramp is remediated or recovered. Reduction in plant feed and revenue. Increased mining costs.	Pit design based on Prefeasibility Study rock mechanics work. Face angle and bench/berm configuration implemented to reduce geotechnical risk.	Unlikely	Major	Medium	Radar/movement monitoring of the slope. Add sidewall rock reinforcement. Further ground investigations including detailed geotechnical and hydrogeological characterisation of access ramp area.	Rare	Major	Medium	
1-101	Mining	Open Pit Stage1/Stage2 Interaction	Fulfilling the mine plan requires the mining of Stage 1 and Stage 2 together. Issues could arise with interactions/operations.	Reduction in revenue due to reduced production. Increased costs due to higher cycle times and truck hours.	Minimum mining widths	Possible	Moderate	Medium	Detailed planning and implementation works.	Unlikely	Moderate	Medium	
1-102	Mining	Open Pit Pre-Stripping not achieved	Pre-stripping not achieved when planned due to equipment productivity levels not being met, worse ground conditions than expected or other reason.	Prolonged ramp-up, increased stripping ratio and increased costs.	Production schedule outlines specific targets required for production. Stripping can start early to enable targets are reached.	Possible	High	High	Scheduling of stripping works planned in more detail at later phases. Procurement model: contractor to provide depth of resource.	Unlikely	High	Medium	
1-103	Mining	Open Pit Weather Delays/Issues	Ice, Snowmelt or adverse weather conditions in the open pit mining.	Reduction in productivity leading to increased cycle times and mining costs.	Productivity estimates have availability and cycle time disruptions incorporated.	Unlikely	Moderate	Medium	Contingency planning/ seasonal activities planning. Equipment selection for conditions.	Rare	Moderate	Low	Refer to linked risk 1-02
1-104	Mining	Open Pit Poor blasting quality	Poor blasting quality from inexperience/learning curve. Overcharging for economic/ productivity reasons.	Catch bench failures / higher dilution/ target slope angles not reached/ Fly rock	None	Possible	Minor	Medium	Use of experienced contractor. Consider seasonal variation of starting blasting when sequencing blasting. Further detailed engineering works. Supervision and quality controls.	Unlikely	Minor	Low	
1-105	Mining	Mandated green equipment.	Legislative changes.	Changing from diesel to green diesel, other synthetic fuels or even to electric.	Options Study considered these options.	Unlikely	Moderate	Medium	Revision of equipment to match any changes in requirements	Rare	Moderate	Low	Refer to linked risk 1-08
1-106	Financial & Economics	Inflation	High inflation (the cash flow model of this study does not take inflation into account)	Decrease in purchasing power; higher costs	Cost estimates have been performed to an AACE Class 4 estimate with estimates within an accuracy range between 30 to 50%. This range captures the risk of inflation.	Possible	Moderate	Medium	Financial model could have price escalation incorporated to assess financial impacts in more detail.	Possible	Minor	Medium	
1-107	Financial & Economics	Commodity Price	Risk: Price goes down. Opportunity: Price goes up.	Changes to viability/profitability of projects	Financial modelling sensitivity analysis on gold price completed as part of statutory reporting.	Likely	Moderate	High	Additional strategies once closer to production including hedging and take-off agreements.	Possible	Moderate	Medium	
1-108	Financial & Economics	Change in tax rate & royalty	Increase in rates	Reduction in NPV, IRR.	None	Possible	Minor	Medium	None	Possible	Minor	Medium	
1-109	Financial & Economics	Depreciation uncertainty	Depreciation will be calculated on a straight-line basis on project level	Depreciation will be a high level estimate	None	Possible	Minor	Medium	None	Possible	Minor	Medium	
1-110	Financial & Economics	Exchange rate fluctuations	Change in exchange rate (The cash flow model of this study has been calculated in US dollars.)	Impact on costs/revenue	Financial model uses FX as a key assumption and input. This allows for easy analysis of changes in FX and greater understanding of effects.	Possible	Minor	Medium	Further FX controls should be adopted closer to construction and with more defined study definition.	Possible	Minor	Medium	
1-111	Financial & Economics	Uncertainty around closure bond value (Environmental Act bond, and to some degree Mining Act bond)	Final footprint remains uncertain. Cost/square meter is assumed - could fluctuate as much as $5/m2 -$10/m2 for environmental bond.	Higher bond expense than anticipated; impact to project viability/payback timeframe.	Check biases in formulation of bonding estimates.	Possible	Moderate	Medium	Meet with regulator to refine unit cost and understand their considerations in determining unit rates.	Rare	Minor	Low	

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