Report to:

Rupert Resources Ltd.

Preliminary Economic Assessment: Ikkari and Pahtavaara - Finland

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Report to:

RUPERT RESOURCES LTD.

PRELIMINARY ECONOMIC ASSESSMENT IKKARI AND PAHTAVAARA - FINLAND

Effective Date: 10th January 2023

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RUPERT RESOURCES

REVISION HISTORY

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A	10th January 2023	Andrew Carter William Harding Matthew Randall James Seccombe Toni Uusimaki James Vardy Anton von Wielligh Brian Wolfe	Andrew Carter	Andrew Carter	Issued

TABLE OF CONTENTS

APPENDICES

LIST OF TABLES

LIST OF FIGURES

GLOSSARY

UNITS OF MEASURE

Centimetre	cm
Cubic centimetre	cm^3
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
Dry metric tonne	dmt
Gigawatt	GW
G-force	g
Gram	g
Grams per cubic centimetre	g/cm^3
Grams per tonne	g/t
Greater than	>
Hectares	Ha
High voltage	HV
Hour	hr
Kilo ounce	koz
Kilo Troy ounces per year	kozT/a
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per tonne	kg/t
Kilograms per year	kg/a
Kilometres per hour	km/h
Kilotonnes per year	kt/a
Kilovolt	kV
Kilowatt	kW
Kilowatt hour	kWh
Less than	<
Litres per second	l/s
Megavolt ampere	MVAr
Megawatt	MW
Metre	m
Metres below ground level	m bgl
Metres per second	m/s
Micron / micron metres	µm
Milliamp	mA
Milligram	mg
Milligrams per kilogram	mg/kg
Milligrams per litre	mg/l
Millimetre	mm
Million	M
Million dollars	M$
Million dollars per year	M$/a

Million ounces .... Moz
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 metre .... m^2
Tonne .... t
Tonnes per hour .... t/h
Tonnes per year .... t/a
Troy ounce .... ozT
Weight for weight .... w/w
Wet metric tonne .... wmt

Abbreviations and Acronyms

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
Billion year old .... Ga
Bismuth .... Bi
Block caving .... BC
Bond abrasive index .... BAi
Bond Working index .... BWi
Cadmium .... Cd
Caesium .... Cs
Calcium .... Ca
Calcium Hydroxide .... $CA(OH)_2$
Canadian National Instrument 43 101 .... NI 43-101
Capital Expenditure .... CAPEX
Carbon dioxide .... CO_2
Carbon in leach CIL
Carbon in pulp .... CIP
Cemented rock fill .... CRF
Central Lapland Greenstone Belt .... CLGB
Cerium .... Ce
Certified reference material .... CRM
Chalcopyrite .... Cp

Characteristic grind .... P_{80}
Chromium .... Cr
Cobalt .... Co
Co-disposal facility .... CDF
Coefficient of variation CV
Concentrate .... conc
Conceptual hydrogeological model .... CHM
Conditional cumulative distribution function .... ccdf
Copper .... Cu
Cut-off grade .... COG
Cyanide destruct .... CN (wad)
Differential Global Positioning System .... DGPS
Digital terrain model .... DTM
Dissolved Oxygen .... DO
Dollars per dry metric tonne .... $/dmt
Dollars per kilogram .... $/kg
Dollars per tonne .... $/t
Earnings Before Interest, Taxes, Depreciation, and Amortisation .... EBITDA
East .... E
Effective grinding length .... EGL
Electromagnetic (survey) .... EM
Energy dispersive spectroscopy .... EDS
Environmental impact assessment .... EIA
Environmental and social impact assessment .... ESIA
Euro per hectare .... EUR/ha
Finnish Environment Institute .... SYKE
First phase of deformation D1
Gallium .... Ga
General and administration .... G&A
General Mine Expenses .... GME
Geological Survey of Finland .... GTL
Germanium .... Ge
Global positioning system .... GPS
Gold .... Au
Gold per tonne .... Au/t
Grinding Solutions Ltd .... GSL
Hafnium .... Hf
Heating Ventilating and Air Conditioning .... HVAC
Human machine interface .... HMI
Hydraulic radium .... HR
Hydrochloric Acid .... HCL
Indium .... In
Induced polarisation .... IP
Inductively Coupled Plasma - Atomic Emission Spectroscopy .... ICP-AE
Inductively Coupled Plasma - Mass Spectronomy .... ICP-MS
Insulated metal panel .... IMP
Internal Rate of Return .... IRR
Iron .... Fe
Joint Venture .... JV
Lanthanum .... La

Lead .... Pb
Lerch Grossman .... LG
Life of Mine .... LOM
Lithium Li
Load Haul Dump .... LHD
Local area network .... LAN
Locked cycle tests .... LCT
Long hole open stoping .... LHOS
Lower control limit LCL
Lower warning limit .... LWL
Magnesium .... Mg
Manganese .... Mn
Mean annual precipitation .... MAP
Mean paired relative difference .... MPRD
Mercury .... Hg
Mine Environment Management Ltd. .... MEM
Minable shape optimiser .... MSO
Micon International co ltd. .... Micon
Molybdenum .... mo
Multiple Indicator Kriging .... MIK
Net Present Value .... NPV
Net Smelter Return .... NSR
Nickel .... Ni
Niobium .... Nb
North .... N
NPV Scheduler .... NVPS
Open pit .... OP
Operating costs .... OPEX
Optical ground wire .... OPGW
Ordinary Kriging .... OK
Overall slope angle .... OSA
Phosphorous .... P
Plant control system .... PCS
Platinum group elements .... PGE
Potassium .... K
Pre-feasibility Study .... PFS
Preliminary economic assessment .... PEA
Price factors .... PF
Process control system .... PCS
Pyrite .... Py
Pyrrhotite .... Po
Qualified Person .... QP
Quality assurance and quality control .... QA/QC
Refinery charge .... RC
Relative level .... RL
Remote input/output .... RIO
Resource model .... RM
Revenue factor .... RF
Reverse circulation .... RC
Rhenium .... Re

Rock quality designation .... RQD
Rod mill work index .... Rwi
Rubidium .... Rb
Run of mine .... ROM
Rupert Resources Ltd. .... Rupert Resources
Scandium .... Sc
Scanning Electron Microscopy .... SEM
Second phase of deformation D2
Selected mining unit .... SMU
Selenium .... Se
Semi-autogenous grinding .... SAG
Silicon .... Si
Silver .... Ag
Sodium .... Na
Sodium cyanide .... NaCN
South .... S
Specific gravity .... SG
SRK Consulting Finland Oy .... SRK
Strontium .... Sr
Structure 1 .... S1
Sub-level caving .... SLC
Sulphur .... S
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
Treatment charge / Refining charge .... TC/RC
Tungsten .... W
Ultramafic schist .... MSCU
Underground .... UG
Unmanned aerial vehicle .... UAV
Upper control limit .... UCL
Upper warning limit .... UWL
Uranium .... U
Vanadium V
Very low frequency radar .... VLF-R
Vibrating wire piezometer .... VWP
West .... W
Wide area network .... WAN
Yttrium .... Y
Xanthate-potassium Amyl Xanthate .... PAX
Zinc .... Zn
Zirconium .... Zr

1.0 SUMMARY

1.1 INTRODUCTION

The Preliminary Economic Assessment (PEA) was commissioned by Rupert Resources Ltd. (Rupert Resources). The study team was led by Tetra Tech Ltd. (Tetra Tech), a global provider of consulting and engineering services for mining projects. Tetra Tech was supported by International Resource Solutions Pty Ltd (resource estimation), Axe Valley Mining Consultants Ltd (mining), SRK Ltd (geotechnical and hydrological studies), Grinding Solutions Ltd (metallurgy), Paterson & Cook (tailings and mining backfill) and Envineer Oy (environmental studies).

This document supports the updated Mineral Resource Estimate for the Ikkari gold deposit, the Pahtavaara gold mine and the Heinä Central gold-copper deposit. The document supports disclosures by Rupert Resources Ltd in a news release dated 28th November 2022 entitled "Rupert Resources reports Preliminary Economic Assessment for Ikkari outlining after-tax net present value (NPV) of US $1.6b". The PEA provides a base case assessment for developing the Ikkari and Pahtavaara deposits as open pit mines and subsequent underground mines upon completion of the open pit portion of the deposits. The Heinä Central Mineral Resource Estimate is not included in the PEA portion of this document. All monetary units in the Study are in US dollars unless otherwise specified. Quantity and grades are rounded to reflect that the reported values represent approximations.

1.2 RELIANCE

The major Study contributors and their respective areas of responsibility are presented in Table 1.1.

Table 1.1 PEA Contributors, Competent Person(s) and Responsibilities

Qualified Person / Consulting Firm	Overview of Responsibilities
International Resource Solutions Pty Ltd Brian Wolfe	• Geology, exploration, drilling • Sample prep, QA/QC, Data verification • Mineral Resource Estimation
Axe Valley Mining Consultants Ltd Dr Matthew Randall Anton von Wielligh	• Open pit and underground mine design • Mine scheduling and development • Mine capital and operating costs estimation • Cut-off grade estimation
SRK Ltd William Harding Michael Di Giovinazzo	• Mining hydrogeology • Mine dewatering • Mining geotechnical investigations

Grinding Solutions Ltd Mike Cook	• Mineralogy • Laboratory metallurgical characterisation studies • Metallurgical flotation and cyanidation extraction studies • Cyanide destruction test work
Paterson & Cook Steve Wilson	• Mine backfill studies
Envineer Oy Toni Uusimaki Niko Karjalainen Heli Uimarihuhta	• Baseline environmental studies • EIA investigations
Tetra Tech Andrew J. Carter James Seccombe James Vardy Justin Taylor Jay Li Nigel Goldup Armin Hayatbakhsh	• Metallurgy • Process design and engineering • Water treatment • Mine waste management • Infrastructure • Engineering capital and operating costs estimation • Financial evaluation

1.3 PROPERTY DESCRIPTION AND LOCATION

The “Rupert Lapland Project” is located on a package of exploration and mining licences controlled by Rupert Resources in Lapland, northern Finland (Figure 1.1). The project area includes the Pahtavaara Mine and associated mining licences, located near Rajala village in the municipality of Sodankylä approximately 25 kilometres (km) northwest of Sodankylä town. The Ikkari and Heinä Central deposits lie 20 km west of Pahtavaara at 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. Heinä Central is located only 1.5 km north of the Ikkari deposit within pine dominated forest at the southern margin of a further low-lying aapa-mire.

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

Figure 1.1 Location of Rupert Lapland Project

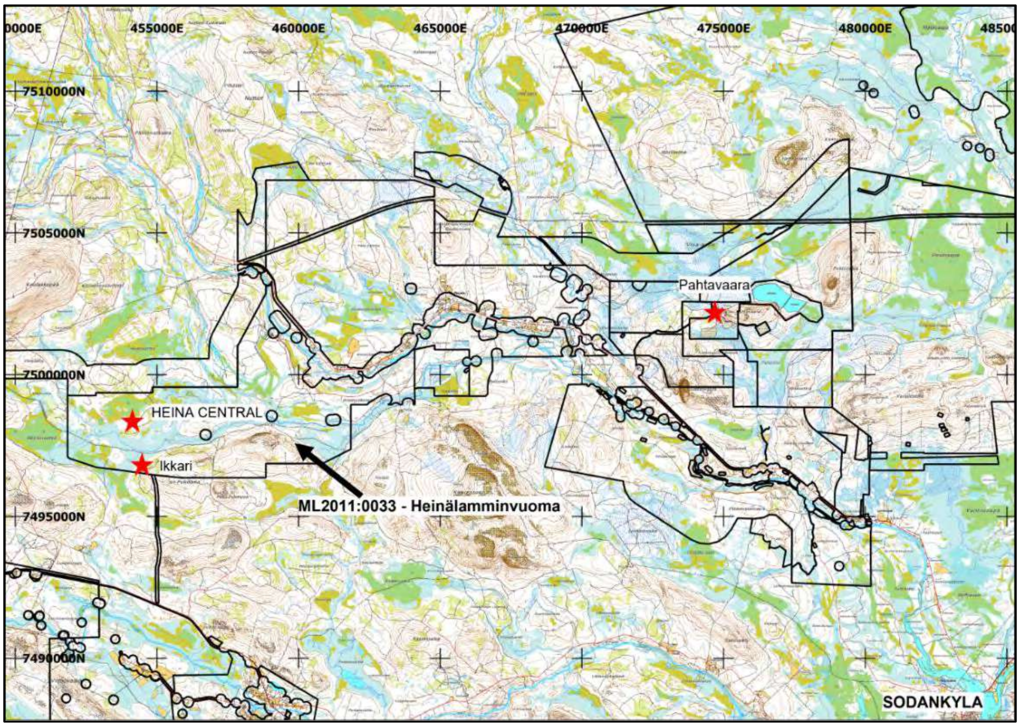
1.4 OWNERSHIP

The Rupert Lapland Project area is 100% owned by Rupert Finland Exploration Oy & Rupert Finland Oy, wholly owned subsidiaries of Rupert Resources, a company incorporated in British Columbia, whose office is at 82 Richmond Street

East, Suite 203, Toronto, Ontario, Canada, M5C 1P1. The Pahtavaara mine and resources defined in this report are contained within a four square kilometre (km²) mining licence (Pahtavaara – 3921 and Pahtavaara laajennus - KL2013:0001-01). The Rupert Lapland Project property is subject to a 1.5 percent (%) royalty on revenue, capped at US $2.0 million (M).

The Ikkari and Heinä Central resources defined in this report are contained on an 84 km² exploration licence (Heinälamminvuoma - ML2011:0033) contained within a wider contiguous land position of 380 km² (Figure 1.2), as well as additional permits in the Central Lapland Belt region (total 714 km², Table 4.2) referred to as the Rupert Lapland Project area.

Figure 1.2 Location of the Pahtavaara Gold Mine, Ikkari Deposit and Heinä Central Deposit Located Within the 380 km² Contiguous Land Package Known as the Rupert Lapland Project Area



1.5 Geological Setting and Mineralisation

1.5.1 Regional Geological Setting

The Rupert Lapland Project area is located within the Central Lapland Greenstone Belt (CLGB), part of the Fennoscandian shield, which hosts 1,700 known incidences of mineralisation in Finland, Sweden, Norway and Russia including around 80 mines. The CLGB has two gold mines of significance. Agnico Eagle's 4.1 million-ounce (Moz) Kittilä mine which produced 221,914 ounces (oz) of gold in 2021 and the historically producing Pahtavaara mine which mined an estimated 441 kilo ounces (koz) of gold in three periods of ownership between 1996 and 2014 (GTK, Mineral Deposit Report) with a remnant Inferred Resource of 4.6 million tonnes (Mt) grading 3.2 grams per tonne gold (g/t Au) (estimated by Rupert Resources in 2018). The Ikkari deposit, 1.5 km to the south-southwest (SSW) of Heinä Central, with an inferred resource of 3.95 Moz at 2.5 g/t (estimated by Rupert Resources in 2021: Wolfe, 2021) is a further example of the CLGB prolific gold endowment.

Ikkari is a new, grassroots, undercover, orogenic gold discovery in the Paleoproterozoic CLGB, 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. 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).

The Rupert Lapland exploration permits occur at a significant regional geological domain boundary zone, which trends predominantly east-west (E-W) through the westernmost extent of the Rupert exploration licences. An approximately four kilometre wide zone of 2.05 billion year old (Ga) Savukoski Group rocks, comprising fine-grained metasedimentary rocks, including phyllite, carbonaceous shale and mafic intrusive rocks, as well as komatiites, occurs between younger Kittilä Group rocks to the north (dominantly tholeiitic metabasalts) and Kumpu Group rocks (molasse-type fluviatile quartzites, subarkoses and polymictic conglomerates) to the south. This zone broadly corresponds with the ‘Sirkka Line’ structure that corresponds to the Savukoski/Kittila Group contact zone to the west of Ikkari and continues as distinctive magnetic lineament(s) to the E/S of Ikkari. Some 25 to 30 gold deposits/occurrences have been reported along this structural zone.

1.5.2 Deposit Geology

Both Ikkari and Heinä Central are grassroots discoveries, located under 5 to 40 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 host rocks of Ikkari mineralisation:

- Dark pyritic shales and siltstones (intruded by gabbro) comprise the majority of the northern fault block and bound mineralisation to the north.
- A central komatiite-dominant zone with complex intercalations of texturally diverse ‘felsic’ facies, one of the main hosts to gold mineralisation.
- A northern, banded ‘felsic’ facies, intensely albite-altered in places, that pinches out in the eastern part of the deposit which also hosts substantial quantities of gold mineralisation.
- A southern zone comprising dominantly coarse ‘felsic’ siliciclastics – massive, banded, conglomeratic and typically more quartz-rich than the northern facies and hosts only minor sporadic gold mineralisation.

Approximately 1 km to the north of Ikkari is the Heinä Central gold-copper deposit which occurs within a relatively low strain lozenge-shaped block of Savukoski Group rocks bound by shear corridors to the north and south. The geology at its most basic level, is fairly simple though the distribution of sulphides, the key control on gold mineralisation is less so

- Dark green-grey, chlorite altered, fine-grained gabbro defines the outside of a moderately northwest (NW) plunging fold closure.
- Grey to black coloured, fine-grained sediment and volcano-sedimentary sequence form the interior of the plunging fold.
- Massive, white to light grey quartz-albite- carbonate-veins which occur only within the sedimentary sequence and in turn host fracture fill and massive sulphides.

The host rocks to gold mineralisation at Pahtavaara are a sequence of amphibolites within a substantial talc-carbonate altered komatiite sequence.

1.5.3 MINERALISATION

The Ikkari deposit can be described as an orogenic, hydrothermal gold deposit. Gold is hosted by disseminated and vein-related pyrite, frequently occurring as ~1 millimetre (mm) visible gold grains. Multi-phase breccias are well developed within the mineralised zone, with early silicified cataclastic phases overprinted by late, carbonate- iron-oxide- rich, hydrothermal breccias which display a subvertical control. All breccias frequently host disseminated pyrite, and are often associated with bonanza gold grades, particularly where magnetite or haematite is prevalent. In the sedimentary lithologies, albite alteration is intense and pervasive, with pyrite-magnetite (± gold) hosted in veinlets in brittle fracture zones.

Mineralisation at the Pahtavaara Mine is structurally controlled and associated with low sulphidation, polyphase quartz-carbonate veining commonly within and on the contacts to amphibolite units. Biotite alteration as a result of potassic alteration is ubiquitous with magnetite alteration more proximal to the mineralisation.

The Heinä Central deposit is distinct from the other two more classical orogenic gold deposits, mineralisation occurs as a sulphide-rich fracture-fill and cement to a brecciated quartz-albite-carbonate vein and also as veinlets in immediately adjacent albite-altered sediments. Sulphide content ranges from a few percent up to greater than 90% in massive sulphides. Gold and copper mineralisation are however poorly correlated with the magnitude of sulphide present.

1.6 EXPLORATION

Ikkari and Heinä Central represent new grass roots discoveries that were initially identified through systematic base of till sampling beginning in early 2019. In the Ikkari area, a single anomalous base of till 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 mineralised zones with a high-grade component over an initial strike length of greater than (>) 500 m.

In the Heinä Central area, anomalous base of till samples of up to 1.0 ppm Au were followed up reconnaissance drill testing. The first drill hole into this geochemical anomaly (hole 119033) was drilled in April 2019 and identified sequences of massive to semi-massive sulphides and included several mineralised intercepts. Follow up drill holes in September 2019 identified further mineralisation with drill hole 119044 intersecting 31 m grading 1.4 g/t Au from 64 m.

Pahtavaara was discovered by the Geological Survey of Finland in 1986 when high grade gold mineralisation with visible gold was found in outcrop. Prior to the discovery, gold anomalies in till and bedrock had been detected during regional exploration.

1.7 Drilling

The vast majority of historical drilling on the Rupert Lapland Project licences focussed on the Pahtavaara Gold Mine and near mine exploration only. Rupert Resources are the only entity to have drilled in the vicinity of the Ikkari and Heinä Central deposits. At the Pahtavaara Gold Mine numerous operators have drilled extensively (Table 1.2) whilst drilling by Rupert Resources across the wider exploration permits is set out in Table 1.3.

Table 1.2 Drilling by Operator at the Pahtavaara Gold Mine

Operator	Period	DH Type	Holes	Metres	% of Total
GTK	1989	Diamond	44	4,372	0.8
Terra Mining	1992-2000	Diamond	152	14,853	2.8
		RC	84	9,976	1.9
		Sludge (UG)	116	117	0.0
		Unknown	8	300	0.0
Scan Mining	2003-2007	Diamond	815	94,563	17.9
		RC	21	1,116	0.2
		Sludge (UG)	2,268	49,902	9.4
Lapland Goldminers	2009-2014	Diamond	1,232	154,573	29.2
		RC	78	1,135	0.2
		Sludge	6,675	124,867	23.6
Rupert Resources	2016-2022	Diamond	596	71,346	13.5
		RC	33	2,224	0.4
Total			**12,122**	**529,344**	**100%**

Within the Heinälamminvuoma exploration permit area, Rupert Resources has used diamond drilling to predominantly target base of till gold anomalies. Since late 2019, following the initial target generation work drilling was undertaken at specific prospect locations at Area 1. These drilling statistics are summarised in Table 1.3.

Table 1.3 Drill Hole Summary for Drilling Undertaken by Rupert Resources on the Rupert Lapland Exploration Licence (Outside of the Pahtavaara Mine Area, up to end May 2022)

Prospect Name	DH Type	Holes	Metres	% of Total
Paskamaa	Diamond	18	1,605	1.1
Heinä South	Diamond	83	15,685	10.9
Heinä North	Diamond	12	1,857	1.3
Heinä Central*/**	Diamond	94	21,772	15.1
Island North	Diamond	18	3,348	2.3
Saitta	Diamond	21	4,196	2.9
Ikkari*/**	Diamond	192	80,925	56.1
Others	Diamond	87	14,903	10.3
Total		**525**	**144,291**	**100%**

Note: Prospect on coding in database, not all holes are necessarily targeting the same mineralisation occurrence.
* Including extensions to drill holes and wedge holes.
** Including holes completed but not assayed, and therefore not included in the resource estimation (section 14.2).

1.8 MINERAL PROCESSING

Metallurgical test work was carried out on a series of samples considered to be representative of the Pahtavaara and Ikkari ore bodies to evaluate different gold concentration methods and allow evaluation of different processing methods of gold recovery. Comminution tests were also performed to determine the Bond Working index (BWi) and Bond Abrasive index (BAi) (Table 1.4). Based on the proposed flowsheet (Figure 1.3) the overall metallurgical recoveries for gold at Ikkari and Pahtavaara are presented in Table 1.4.

Table 1.4 Projected Metallurgical Recoveries for Au and Comminution Properties Across the Ikkari and Pahtavaara Deposits

Deposit / Recovery Method	Grind Size P_{80} (µm)	BWi	BAi	Gold Recovery Percentage (%)
Ikkari via Gravity	175	-	-	34
Ikkari via floatation	175	15.5	0.35	96
Ikkari total Recovery	175	-	-	95
Pahtavaara Recovery	250	-	0.59	89

Key: µm - micron

1.9 MINERAL RESOURCE

The Mineral Resource Estimates for the Ikkari Project, Pahtavaara Project and Heinä Central Project are reported in accordance with National Instrument 43-101 and have been estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) “Estimation of Mineral Resources and Mineral Reserves best Practice Guidelines". These mineral resource estimates are classified as a combination of Indicated and Inferred Mineral Resources as defined by the CIM. Numbers displayed in the consolidated Table 1.5 are affected by rounding.

Table 1.5 Consolidated Rupert Lapland Projects Consolidated Resource Statement

Classification	Deposit	Mining Method	Cut-off Grade Au (g/t)	Tonnes	Grade Au (g/t)	Au Content (Ounces)
Indicated	Ikkari	OP	0.5	30,000,000	2.5	2,400,000
		UG	1.0	16,500,000	2.4	1,280,000
		Total		46,400,000	2.5	3,680,000
	Pahtavaara	OP	0.5	900,000	2.2	60,000
		UG	1.5	1,000,000	3.7	120,000
		Total		1,900,000	3.0	180,000
	Rupert Lapland Projects Total			**48,300,000**	**2.5**	**3,860,000**
Inferred	Ikkari	OP	0.5	3,100,000	1.5	150,000
		UG	1.0	8,700,000	2.0	550,000
		Total		11,800,000	1.9	710,000
	Pahtavaara	OP	0.5	3,700,000	1.6	190,000
		UG	1.5	2,200,000	3.1	220,000
		Total		5,900,000	2.1	410,000
	Heinä Central	OP	0.5	2,200,000	1.7	120,000
		UG	1.2	400,000	2.1	30,000
		Total		2,700,000	1.8	150,000
	Rupert Lapland Projects Total			**20,400,000**	**1.9**	**1,260,000**

Key: OP = open pit, UG = underground

These are mineral resources not mineral reserves as they do not have demonstrated economic viability. Results are presented in situ. Ounce (troy) = metric tonnes x grade / 31.103475. Calculations used metric units (meters, tonnes, g/t). Any discrepancies in the totals are due to rounding effects.

The effective date of the 2022 Mineral Resource Estimate for Ikkari is 28 November 2022. The Mineral Resource Estimate at Ikkari is calculated using the multiple indicator kriging (MIK) method and is reported both within a designed open pit and as a potential underground operation outside that. The Mineral Resource Estimate at Ikkari is reported using a cut-off grade of 0.5 g/t Au for mineralisation potentially mineable by open pit methods and 1.0 g/t Au for mineralisation potentially extractable by underground methods. The potential open pit mine and cut-off grade is calculated using a gold price at $1,650 per ounce, 5% mining dilution, 95% Au recovery. Open pit mining costs at $2.5 per tonne (/t), process costs at $11.3 /t, other costs (including co-disposal, water and closure) at $4.0 /t and general and administration (G&A), including royalties and refining at $3.2 /t. The calculated cut-off grade is rounded up to 0.5 g/t for reporting. The underground cut-off grade is calculated at underground mining cost $21.8 /t and underground mining dilution at 8% based on sub level caving.

The calculated underground cut-off grade is rounded up to 1.0 g/t as the resource is not constrained within mineable shapes.

The effective date of the 2022 Mineral Resource Estimate for Pahtavaara is 28 November 2022 and the is calculated using the MIK method. The Mineral Resource Estimate is reported both within a designed open pit and as a potential underground operation outside that. The Mineral Resource Estimate at Pahtavaara is reported using a cut-off grade of 0.5 g/t Au for mineralisation potentially mineable by open pit methods and 1.5 g/t Au for mineralisation potentially extractable by underground methods. The potential open pit mine and cut-off grades are calculated using a gold price at $1,650 per ounce, 20% mining dilution, 89% Au recovery, and a mining cost at $2.6 /t. Process cost at $10.2 /t (concentration at Pahtavaara and transport to Ikkari), other costs (including co-disposal facility [CDF] costs and closure) at $1 /t and G&A including royalties and refining at $3.1 /t. The calculated cut-off grade is rounded up to 0.5 g/t for reporting. The underground cut-off grade is calculated at an underground mining cost $49.6 /t and underground mining dilution at 10% based on long hole open stoping. The calculated underground cut-off grade is rounded up to 1.5 g/t for reporting.

The effective date of the 2022 Mineral Resource Estimate for Heinä Central is 28 November 2022 and is calculated using the ordinary kriging (OK) method. The Mineral Resource Estimate is reported both within an optimised open pit and as a potential underground operation outside that. The Mineral Resource Estimate is reported at a 0.5 g/t Au cut-off grade for mineralisation potentially mineable by open pit methods and at 1.2 g/t Au for mineralisation potentially extractable by underground methods. The potential open pit mine and cut-off grade are calculated using a gold price at $1,650 /oz, 5% mining dilution, 78% Au recovery. Open pit mining costs at $2.5 /t, process costs at $10.01 /t (concentrate production at Heinä and transport to Ikkari), other costs (including CDF and closure) at $3.20 /t and G&A including royalties and refining at $1.66 /t. The calculated open pit cut-off grade is rounded up to 0.5 g/t for reporting. The underground cut-off grade is calculated at underground mining cost $30 /t and underground mining dilution of 5%. The calculated underground cut of grade is rounded up to $1.2 g/t for reporting. The Heinä Central deposit also contains potentially recoverable copper. At the 0.5 g/t Au cut-off grade for mineralisation potentially mineable by open pit methods Heinä Central also contains 12,000 tonnes of in situ copper. At the 1.2 g/t Au cut-off grade for mineralisation potentially mineable by underground methods, Heinä Central also contains 1,800 tonnes of in situ copper. No economic value is applied to the copper content when designing the optimised open pit or calculating the potential cut-off grade at Heinä Central.

1.10 MINING

The PEA considers that Ikkari will be initially developed as an open pit with a target production rate of 3.5 million tonnes per year (Mt/a) of plant feed. As the open pit reaches the end of its life (after 11 years) the underground development will be completed so that the underground operation can continue as the open pit is depleted. The transition point between open pit and underground operations was determined by operating costs as well as the limitation of the current exploration permit boundary. Open pit mining at Ikkari is expected to utilise a conventional shovel and truck configuration (140 t medium sized haul trucks matched with 300 t hydraulic excavators). Underground mining at Ikkari was modelled assuming the sub-level caving method. The mine at Pahtavaara will be re-developed as an open pit and underground mine (employing the long hole open stoping [LHOS] method) to produce a high-grade concentrate which will then be hauled by road to the Ikkari plant for final processing.

Mineral inventories developed in preparation of the mining estimates are shown in Table 1.6 and Table 1.7.

Table 1.6 Ikkari Open Pit Mineral Inventory by Pit Stage

Stage	Waste	Mineral Inventory[2]			Strip Ratio
	(tonnes)[1]	(tonnes)[1]	(Kg Au)	(g/t Au)	
1	11,145,000	7,139,000	17,901	2.51	1.6
2	37,030,000	13,956,000	31,523	2.26	2.7
3a	27,643,000	5,525,000	4,948	0.90	5.0
3b	56,623,000	10,361,000	22,703	2.19	5.5
Total	**132,441,000**	**36,981,000**	**77,075**	**2.08**	**3.6**

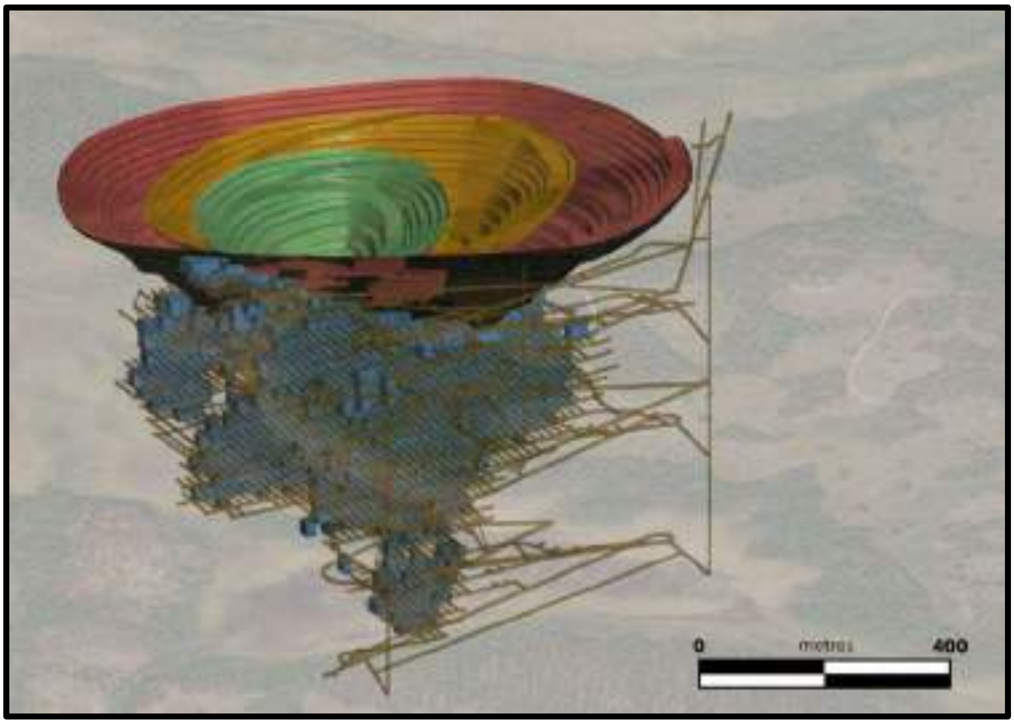
Notes:
1. Tonnages have been rounded
2. Mining recovery and dilution set to 95% and 5% respectively

Table 1.7 Pahtavaara Open Pit Mineral Inventory by Pit Stage

Stage	Waste	Mineral Inventory[2]			Strip Ratio
	(tonnes)[1]	(tonnes)[1]	(Kg Au)	(g/t Au)	
1	22,432,000	2,185,000	2,927	1.34	10.30
2	21,466,000	1,084,000	1,734	1.60	19.80
3	20,026,000	2,410,000	3,052	1.27	8.30
Total	**63,924,000**	**5,679,000**	**7,712**	**1.36**	**11.63**

Notes:
1. Tonnages have been rounded to the nearest thousand.
2. Mining recovery and dilution set to 100% and 0% respectively.

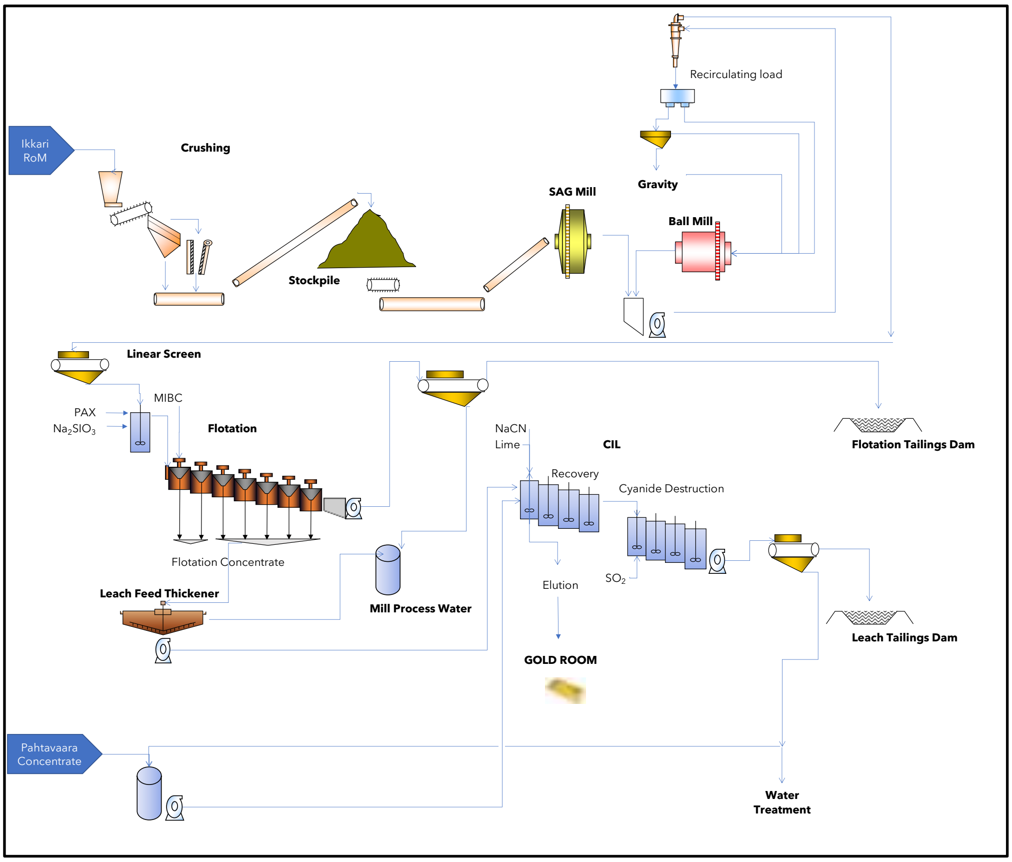
Figure 1.3 Ikkari Mine – Open Pit Stages and Underground Mining Infrastructure



1.11 RECOVERY

A new plant is envisaged to process 3.5 Mt/a of run-of-mine (ROM) ore from the Ikkari open pit and underground at an average grade of 1.82 g/t Au (including processing low grade stockpiles towards the end of life of mine). Test work shows that the gold at Ikkari is non-refractory and occurs in the native form or associated with pyrite. The process considered comprises crushing and grinding to reduce the ROM material to a characteristic grind (P80) of 175 µm, and a gravity circuit to recover the native gold. The pyrite associated gold will be recovered by flotation and fed, with the re-pulped concentrate from Pahtavaara, into the leach circuit where lime and cyanide are added in the presence of air to extract the gold. The gold will be then recovered in an adsorption, desorption, and recovery (ADR) circuit. The leach tails will be treated to remove cyanide and filtered for co-disposal with waste rock. The liquor recovered from the filtration is treated prior to re-use. See flowsheet (Figure 1.4).

Figure 1.4 Rupert Lapland Project Recovery Flowsheet



The Pahtavaara ROM ore would be processed through a 0.5 Mt/a gravity and flotation concentration facility that is envisaged to be expanded to 0.75 Mt/a after 6 years of operation. The resulting high-grade concentrate product would be fed in to the Ikkari carbon in leach (CIL) circuit for gold recovery to doré.

1.12 ACCESS AND INFRASTRUCTURE

1.12.1 ACCESS

Ikkari is well supported by existing infrastructure and is accessed by tarmac and a five km gravel road from the towns of Kittilä (50 km west) and Sodankylä (40 km east) both of which provide support services and labour to two existing mines in the area.

1.12.2 CO-DISPOSAL OF MINE WASTE AND TAILINGS

Mine waste and tailings are planned to be combined and disposed of together to increase physical and chemical stability of the waste. Initial studies suggest that the potential for acid generation could be significantly reduced by the buffering effect on acid solutions by carbonates present in the Ikkari rock and lime in the leach tails. Detailed waste material characterisation studies are underway for optimisation of the long-term storage facility design parameters for construction, operation and eventual closure.

1.12.3 HYDROGEOLOGY & WATER TREATMENT

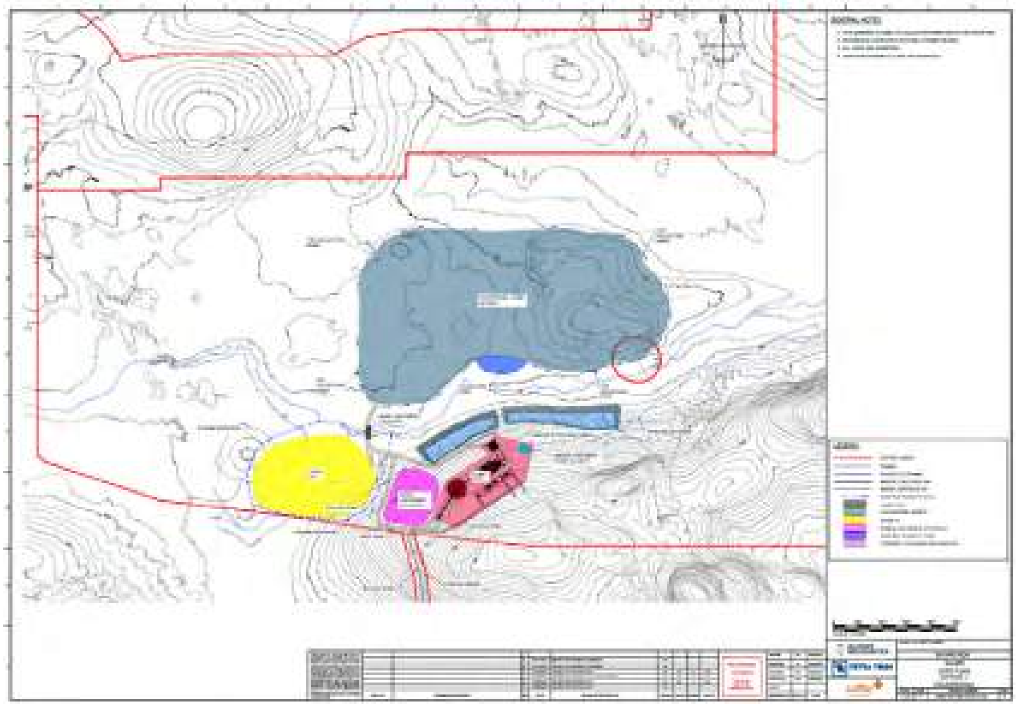
Initial hydrogeological studies have been undertaken at the Ikkari project site to formulate a management plan. To reduce contact water, surface water flows will be diverted where possible to avoid the open pit, plant site and waste facilities and a series of dewatering bores would be installed to reduce water flows into the mining operation. All water that comes in contact with the operation will be collected in lined storage dam structures and treated in a water treatment facility to remove potential contaminants before discharge via pipelines to nearby water courses. Further hydrogeological studies are being undertaken as part of data collection for future studies.

The plant design envisages water to be recirculated within the plant to minimise water consumption with water recovered from the cyanide destruction filtration to be treated in the water treatment area. Treated water is to be used for the re-pulping of the Pahtavaara concentrate and for reagent make-up. Brine produced by the water treatment plant will be added to the cyanide destruction tailings.

1.12.4 POWER

A 220 kilovolt (kV) power transformer substation is located 9 km from Ikkari that can be used as a connection point to the national grid for a 110 kV power line to the Ikkari minesite. A power surplus is envisaged in Lapland towards the end of the decade, with a significant contribution expected from renewables.

Figure 1.5 Ikkari Site Layout



1.13 ENVIRONMENTAL STUDIES

Rupert Resources has been pro-actively engaged in baseline data collection and stakeholder engagement since 2018. The base case PEA presented here is one of three potential development options that will be the subject of the environmental impact assessment program (EIA Programme) that will be presented to all relevant authorities and presented for feedback at public meetings expected to occur in early 2023. On completion of the EIA Programme work and the planned project Pre-feasibility Study (PFS), the results will be presented in the project Environmental Impact Assessment document that will form the basis for submission of an environmental permit application. Rupert Resources has begun a parallel programme of land use planning with the local and regional authorities and has also set up a stakeholder co-operation group that will have the opportunity to comment and give opinions and feedback during the EIA process.

1.14 CAPITAL AND OPERATING COSTS

Table 1.8 Project Capital Costs

Initial Capex	US $ Millions
Mining o/p pre-production	16.6
Process Plant	131.0
Civils and infrastructure	29.5
Water treatment	96.4
Tailings	20.4
First fills & spares	10.0
Owner's Costs	20.0
Closure bond	37.2
Contingency	43.5
Total Initial Capex	**404.6**

Table 1.9 Sustaining Capital Costs

Sustaining Capex	US $ Millions
Pahtavaara initial capex	41.0
Underground mining	178.8
Water treatment	34.0
Tailings & waste dump	34.9
Plant sustaining	101.0
Pahtavaara closure bond	5.0
Total	**394.7**

Table 1.10 Operating Costs

Life of Mine Operating Cost	US $ / Tonne Milled	US $ /oz
Mining	18.1	333
Water treatment	1.4	26
Concentrate freight	0.1	2
Processing	10.9	204
Tailings	1.6	28
Closure fund	0.8	15
G&A	2.4	44
Freight/Refining	0.1	3
Royalty	0.7	12
Total Cash Costs	**36.1**	**667**

1.15 ECONOMIC ANALYSIS

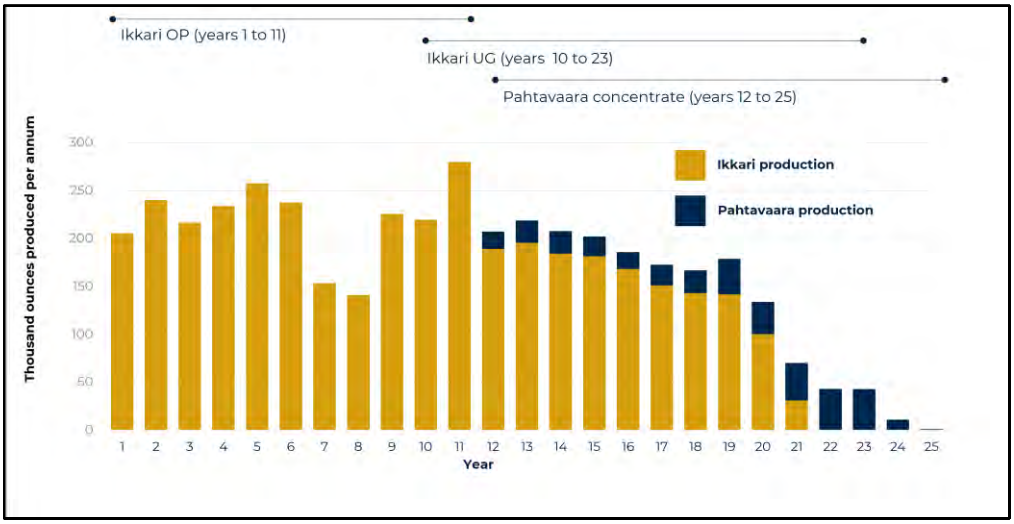
The production model considered an open-pit operation at Ikkari in the first 11 years, transitioning to Ikkari underground (years 10-23) and with a contribution from the existing Pahtavaara mine in years 12 to 24. The core 22-year Life of Mine (LOM) includes recovered gold of 4.25 Moz with average annual production of 200,000 oz. The open pit operation is expected to support average annual production of 220,000 oz in years one to 11 principally owing to higher grades.

Table 1.11 Project Production Summary

	Unit	Years 1 to 11	LOM (22 years)
Milled tonnes	Million tonnes	37.9	71.6
Mill throughput	Million tonnes per annum	3.5	3.5
Strip ratio	Waste : Ore	3.6	4.6
Average processed grade	Grams per tonne (gold)	2.1	1.9
Average metallurgical recovery	%	95.0	95.0
Average annual gold production	000 troy ounces	220.0	200.0
Recovered gold	Million troy ounces	2.4	4.2
Total Cash Cost	$ / troy ounce	501.0	667.0
Sustaining capital	$ / troy ounce	95.0	93.0
All in Sustaining Cost (AISC)	$ / troy ounce	596.0	759.0

Note: As per the World Gold guidance (Gold All in Sustaining Costs | Gold AISC | World Gold Council), the objective of the all-in sustaining costs (AISC) metric is to provide key stakeholders (i.e. management, shareholders, governments, local communities, etc.) with comparable metrics that reflect as close as possible the full cost of producing and selling an ounce of gold, and which are fully and transparently reconcilable back to amounts reported under Generally Accepted Accounting Principles (GAAP) as published by the Financial Accounting Standards Board (FASB also referred to as US GAAP) or the International Accounting Standards Board (IASB also referred to as IFRS). AISC and AIC are non-GAAP metrics subject to regulatory and disclosure requirements of the various jurisdictions applicable to the reporting company.

Figure 1.6 Project Production Profile



Initial, preproduction capital was estimated to be $404.6 M with a further $394.7 M of sustaining capital required over the LOM. On a unit basis expected all-in sustaining cost (AISC) was shown to be $759 /oz over LOM, and $596 /oz during open-pit operation. The study showed an After-tax NPV (5% discount) of $1.6 billion with unlevered internal rate of return (IRR) of 46% and payback after only two years, assuming a gold price of $1,650 per troy ounce.

Table 1.12 Project Economics

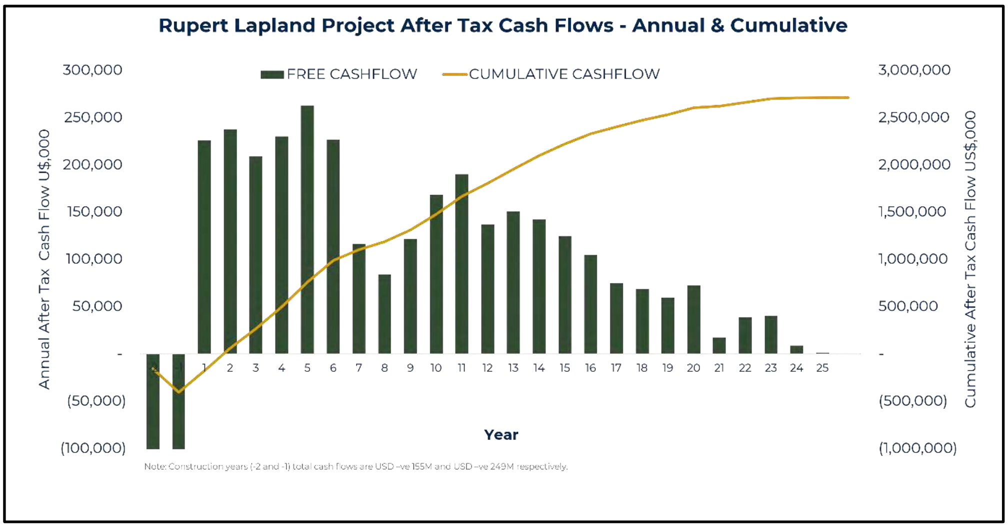
	Unit	Value
Life of mine	Years	22
Net Present Value (5% discount rate)	US $ million	1,600
Internal rate of return (unlevered)	%	46
Payback	Years	2.0
Capital expenditure (Initial)	US $ million	405
Capital expenditure (Sustaining)	US $ million	395
Revenue	US $ million	6,955
Operating cost	US $ million	2,775
Free cash (after tax)	US $ million	2,710

The gold price assumption of US $1,650 /troy ounce of gold used throughout the study was derived from mean consensus long term pricing assumptions from a population of 40 energy and metals analysts. The study assumes payability of 99.92% and a freight and refining cost of $2.50 /oz on doré product based on industry benchmarking. Gold doré produced at Ikkari is expected to have no deleterious elements and to be able to be refined in Europe.

Table 1.13 Model Inputs

Assumption	Unit	Value
Gold price	US $ / troy ounce	1650
Exchange rate	EUR / US $	1:1
Corporate tax rate	%	20

Figure 1.7 Project Model After Tax Cash Flow



1.16 CONCLUSIONS

The PEA results provide a high-level initial estimate of the potential economic value of the mineral resources discovered to date. The report also shows Ikkari to be robust technically and capable of sustaining high margin production over a mine life of more than 20 years. Eighty four percent of resource ounces at Ikkari are expected to report to an Indicated resource category based on 73,000 m of diamond drilling which defines a cohesive deposit with broad intervals of consistent high-grade gold.

1.17 RECOMMENDATIONS

The PEA has demonstrated the economic potential of the Ikkari project and justifies accelerated development toward implementation with the required next steps being further resource and exploration drilling; completion of a PFS and project EIA programme. On completion of the EIA programme and the planned PFS, the results will be presented in the project EIA document that will form the basis on which an environmental permit application is submitted. Rupert Resources has also begun a parallel programme of land use planning with the local and regional authorities and has also set up a stakeholder co-operation group that will have the opportunity to comment and give opinions and feedback during the EIA process.

2.0 INTRODUCTION

2.1 GENERAL

Rupert Resources 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, mill, and exploration permits and concessions located in the CLGB of Northern Finland. The Rupert Lapland Project is located within the CLGB, 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. The regional industrial base is currently dominated by small businesses involved in forestry, agriculture and manufacturing.

The town of Rovaniemi in Finland is located some 150 km south-southwest of Pahtavaara. Rovaniemi has a population of approximately 60,000 inhabitants and is the administrative centre of Finnish Lapland.

Open-pit operation at Ikkari is suggested for the first 11 years, transitioning to Ikkari underground (years 10-23) and Pahtavaara concentrate (years 12 to 24). The 24-year life of mine includes recovered gold of 4.25 million ounces with average annual production of 200,000 ounces. The open pit operation is expected to support average annual production of 220,000 ounces in years one to 11.

The PEA study team was led by Tetra Tech Ltd supported by International Resource Solutions Pty Ltd (resource estimation), Axe Valley Mining Consultants Ltd (mining), SRK Ltd (geotechnical and hydrological studies), Grinding Solutions Ltd (metallurgy), Paterson & Cook (tailings) and Envineer Oy (environmental studies).

This report summarises the findings of the study and is published for internal use by Rupert Resources.

2.2 SITE VISITS

A site visit of the properties was conducted in May 2022, personnel in attendance are shown in Table 2.1.

Table 2.1 Personnel on Site Visit

Name	Company	QP
Nigel Goldup	Tetra Tech	Yes
Matthew Randall	Axe Valley Consultants	Yes
James Vardy	Tetra Tech	Yes
Brian Wolfe	International Resource Solutions	Yes

2.3 QUALIFIED PERSONS

A summary of the Qualified Persons (QPs) responsible for this report is provided in Table 2.12.

Table 2.2 Summary of QPs

Report Section		Company	Qualified Person
1.0	Summary	All	Sign-off by Section
2.0	Introduction	Tetra Tech	Andrew Carter
3.0	Reliance on Other Experts	All	Andrew Carter
4.0	Property Description and Location	Tetra Tech	Andrew Carter
5.0	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Tetra Tech	Andrew Carter
6.0	History	Tetra Tech	Andrew Carter
7.0	Geological Setting and Mineralisation	International Resource Solutions	Brian Wolfe
8.0	Deposit Types	International Resource Solutions	Brian Wolfe
9.0	Exploration	International Resource Solutions	Brian Wolfe
10.0	Drilling	International Resource Solutions	Brian Wolfe
11.0	Sample Preparation, Analyses, and Security	International Resource Solutions	Brian Wolfe
12.0	Data Verification	International Resource Solutions	Brian Wolfe
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	Andrew Carter
14.0	Mineral Resource Estimates	International Resource Solutions	Brian Wolfe
15.0	Mineral Reserve Estimates	Axe Valley	Matthew Randall
16.0	Mining Methods	Axe Valley	Matthew Randall
17.0	Recovery Methods	Tetra Tech	Andrew Carter

Report Section		Company	Qualified Person
18.0	Project Infrastructure	Tetra Tech	Andrew Carter
18.1	Site Conditions	Tetra Tech	Andrew Carter
18.2	Existing Infrastructure	Tetra Tech	Andrew Carter
18.3	General Site Layout	Tetra Tech	Andrew Carter
18.4	On Site Roads	Tetra Tech	Andrew Carter
18.5	Structural Design	Tetra Tech	Andrew Carter
18.6	Site Buildings	Tetra Tech	Andrew Carter
18.7	Heat, Ventilation and Air Conditioning	Tetra Tech	Andrew Carter
18.8	Fire Detection & Protection	Tetra Tech	Andrew Carter
18.9	Electrical Distribution System	Tetra Tech	Andrew Carter
18.10	Instrumentation, Control and Communication System	Tetra Tech	Andrew Carter
18.11	Hydrological Review	SRK	William Harding
18.12	Groundwater Inflow Assessment	SRK	William Harding
18.13	Water Management Concept	SRK	William Harding
18.14	Site Wide Water Balance	Tetra Tech	Andrew Carter
18.15	Tailings Management & Storage	Paterson & Cook	Andrew Carter
18.16	Mining Contractor Facilities	Tetra Tech	Andrew Carter
18.17	Offsite Infrastructure	Tetra Tech	Andrew Carter
19.0	Market Studies and Contracts	Tetra Tech	Andrew Carter
20.0	Environmental Studies, Permitting, and Social or Community Impact	Envineer Oy	Toni Uusimaki
20.1	Political, Legal and Institutional Framework	Envineer Oy	Toni Uusimaki
20.2	Stakeholder Information and Consultation	Envineer Oy	Toni Uusimaki
20.3	Baseline Description	Envineer Oy	Toni Uusimaki
20.4	Impact Identification and Assessment	Envineer Oy	Toni Uusimaki
20.5	Environmental Risks	Envineer Oy	Toni Uusimaki

Report Section		Company	Qualified Person
20.6	Environmental and Social Management Plan	Envineer Oy	Toni Uusimaki
20.7	Closure and Rehabilitation Plan	Envineer Oy	Toni Uusimaki
20.8	Acid Rock Drainage and Metal Leaching	Envineer Oy	Toni Uusimaki
20.9	Site Water Management	Envineer Oy	Toni Uusimaki
21.0	Capital and Operating Costs	Tetra Tech	Andrew Carter
22.0	Economic Analysis	Tetra Tech	Andrew Carter
23.0	Adjacent Properties	Tetra Tech	Andrew Carter
24.0	Other Relevant Data and Information	Tetra Tech	Andrew Carter
25.0	Interpretation and Conclusions	All	Andrew Carter
26.0	Recommendations	All	Andrew Carter
27.0	References	All	

2.4 Sources of Information

All sources of information for this study are located in Section 27.0.

2.5 Units of Measurement and Currency

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

All currency is in US dollars unless otherwise noted.

3.0 RELIANCE ON OTHER EXPERTS

3.1 RELIANCE

Tetra Tech has placed reliance on technical, legal, and financial opinions regarding the development of the Property provided by external QPs and other experts who are not qualified persons. Responsibility for the various sections of the report were indicated earlier. The extent of reliance on others is indicated below.

The various agreements under which Rupert Resources holds title to the mineral resources for the project have not been reviewed by Tetra Tech and Tetra Tech offers no opinion as to the validity of the mineral title claimed. Tetra Tech has reviewed the licence information provided by International Resource Solutions and Rupert Resources.

Regarding open pit and underground mine design, development, scheduling, and cost estimation Tetra Tech has relied on the work conducted by Axe Valley Mining Consultants (AXE Valley) who were contracted independently by Rupert Resources.

In relation to metallurgical extraction test work, Tetra Tech has reviewed and considered the reports compiled by Grinding Solutions Limited (GSL) on mineralogical materials from Ikkari and Pahtavaara deposits. Tetra Tech has not carried out any independent test work of its own.

Regarding geotechnical assessments relating to pit design and site infrastructure, Tetra Tech and Axe Valley have relied on work completed by SRK Consulting (Finland) Oy (SRK) engaged separately by Rupert Resources.

Regarding hydrogeology Tetra Tech has relied on site investigations, modelling and technical reports undertaken by SRK Consulting (UK) Limited (SRK), engaged separately by Rupert Resources.

In relation to environmental aspects, Tetra Tech has not completed any independent work in the development of this study and has relied on the work completed to date by Envineer Oy, Environmental Consulting (Envineer) engaged separately by Rupert Resources.

Regarding Finnish taxation requirements Tetra Tech has relied on information provided by Rupert Resources.

The maps and tables for this report were reproduced or derived from reports written on the property and supplied to Tetra Tech by Rupert Resources. Unless otherwise stated, the photographs used in this report were taken by the authors of the various sections or Tetra Tech during the site visits.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION OF IKKARI GOLD DEPOSIT

The Ikkari Gold Deposit is located within Rupert Resources' "Rupert Lapland Project" exploration licences, which occur across an area surrounding the Rajala village in the municipality of Sodankylä approximately 30 km northwest of Sodankylä in northern Finland. The Ikkari Gold Deposit occurs in the westernmost extents of the Rupert Lapland Project, approximately 10 km north-northwest (NNW) of Jeesiö village (for coordinates see Table 4.1) and 22 km west-southwest (WSW) of the Pahtavaara Mine.

Table 4.1 Deposit Coordinates

Deposit	Reference Grid	Easting	Northing
Ikkari	EUREF	454,100	7,496,950
	YKJ	3,454,253	7,500,083
Heinä Central	EUREF	454,225	7,498,287
	YKJ	3,454,379	7,501,420
Pahtavaara	EUREF	475,137	7,501,765
	YKJ	3,475,300	7,504,900

4.2 RIGHT OF TENURE

The Rupert Lapland Project area (in which the Ikkari, Heinä Central and Pahtavaara Gold Deposits occur) is comprised of a contiguous package of mining licences, exploration licences, claims and reservations for exploration totalling an area of 380 km², and additional permits elsewhere in the Central Lapland Belt, for a grand total of 714 km² (see Table 4.2 for component parts, expiry and annual fees). The mineral resource for Ikkari and Heinä defined in this report is contained within the existing valid exploration permit Heinälamminvuoma - ML2011:0033, with an area of 84 km². Pahtavaara is included in a separate licence. The rights conveyed to the landholder are defined in the Mining Act of Finland (621/2011) and summarised as follows:

4.2.1 MINING PERMIT

A mining permit is required for the establishment of a mine and the undertaking of mining activity. The mining permit entitles the holder to exploit the mining minerals found in the mining area, the organic and inorganic surface materials, waste rock and tailings generated as by-products of mining activities as well as other materials belonging to the bedrock and soil of the mining area to the extent that their use is necessary for the purposes of mining operations in the mining *area. The mining permit also entitles its holder to perform ore prospecting within the mining area.*

4.2.2 Exploration Permit

The holder of an exploration permit has the right to explore the structures and composition of geological formations on the permit holder's own land and on land owned by another landowner within the area referred to in the permit (exploration area). The permit holder also has the right to conduct other prospecting 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 in accordance with the exploration permit.

Table 4.2 Land Components of the Rupert Lapland Project

Type	Code	Status	Name	Company	Area (km²)	Granted	Expires	Fee Eur/ha
Mining Licence	3921	Valid	Pahtavaara	Rupert Finland Oy	3.86	14 Sep 1993	N/A	100
	KL2013:0001-01	Valid	Pahtavaara laajennus	Rupert Finland Oy	0.35	12 Sep 2013	Review after 10 years	100
Sub total					*4.21*			
Exploration Permit	ML2011:0033-01	Valid	Heinälamminvuoma	Rupert Exploration Finland Oy	83.91	11 Jun 2021	10 Jun 2024	**30**
	ML2017:0079-01	Valid	Rajala	Rupert Exploration Finland Oy	2.94	27 May 2019	26 Jun 2023	20
	ML2017:0080-01	Valid	Liikavaara	Rupert Exploration Finland Oy	3.71	05 Feb 2019	07 Mar 2023	20
	ML2012:0196-01	Valid	Soretiajärvi 4 (Hirvilavanmaa)	Rupert Exploration Finland Oy	0.96	19 Jul 2018	20 Aug 2021	50
	ML2011:0008-02	Valid	Soretiajärvi 3 (Hirvilavanmaa)	Rupert Exploration Finland Oy	0.09	19 Jul 2018	20 Aug 2021	50
	ML2019:0005	Valid	Satta	Rupert Finland Oy	4.54	02 Jul 2019	02 Aug 2022	40
	ML2019:0023	Valid	Satta SE	Rupert Exploration Finland Oy	43.49	07 Nov 2019	18 Mar 2024	20
	ML2019:0024	Valid	Pahta NW	Rupert Exploration Finland Oy	37.82	07 Nov 2019	18 Mar 2024	20
	ML2020:0007	Valid	Liika	Rupert Exploration Finland Oy	0.79	01 Sep 2021	08 Oct 2025	20
	ML2020:0006	Valid	Area 51	Rupert Exploration Finland Oy	65.56	17 Feb 2020	08 Oct 2025	20
	ML2021:0003	Valid	Jeesiö	Rupert Exploration Finland Oy	58.28	28 Jul 2021	03 Sep 2025	20
	ML2013:0014	Valid	Paskamaa 1-5	Rupert Finland Oy	4.88	31 May 2021	30 May 2024	50

	ML2012:0195	Valid	Pahtarimpi 2-3	Rupert Finland Oy	1.66	31 May 2021	30 May 2024	50
	ML2013:0013	Valid	Pahtarimpi 10-11	Rupert Finland Oy	5.46	31 May 2021	30 May 2024	50
	ML2012:0080-02	Valid	Liikamaa 1-4	Rupert Finland Oy	1.97	31 May 2021	30 May 2024	50
	ML2011:0034-01	Valid	Paskahaara 1	Rupert Finland Oy	16.77	08 Mar 2022	14 Apr 2025	N/A
Sub total					*316.06*			
Exploration Permit Application	ML2013:0012-01	Application	Paskamaa 2b-3b	Rupert Finland Oy	0.09	20 Apr 2021	N/A	N/A
	ML2021:0081	Application	Rako	Rupert Exploration Finland Oy	0.46	27 Sep 2021	N/A	N/A
	ML2021:0113	Application	Sattanen West	Rupert Exploration Finland Oy	1.36	15 Oct 2021	N/A	N/A
	ML2011:0008	Application	Soretiajärvi 3	Rupert Exploration Finland Oy	0.09	20 Apr 2021	N/A	N/A
	ML2012:0196	Application	Soretiajärvi 4	Rupert Exploration Finland Oy	0.95	20 Apr 2021	N/A	N/A
	ML2022:0025	Application	Jeesio 2	Rupert Exploration Finland Oy	1.63	10 May 2022	N/A	N/A
	ML2022:0058	Application	Kuusajärvi 1	Rupert Exploration Finland Oy	43.74	22 Sep 2022	N/A	N/A
	ML2022:0071	Application	Kuusajärvi 2	Rupert Exploration Finland Oy	31.98	25 Oct 2022	N/A	N/A
	ML2022:0071	Application	Kuusajärvi 3	Rupert Exploration Finland Oy	38.19	25 Oct 2022	N/A	N/A
Sub total					*135.26*			
Reservation for Exploration Licence	VA2021:0050-01	Valid	Kallo	Rupert Exploration Finland Oy	119.35	13 Jul 2021	12 Jul 2023	N/A
	VA2021:0063	Valid	Kuusajärvi	Rupert Exploration Finland Oy	138.89	20 Dec 2021	26 Oct 2023	N/A
Sub total					*258.24*			
TOTAL					**713.77**			

Key: EUR/ha = Euros per hectare

The permit holder may build or transfer to the exploration area temporary constructions and equipment necessary for exploration activity in accordance with the exploration permit. An exploration permit does not authorise the exploitation of the deposit. It does, however, provide the holder with a privilege for the mining permit, which in turn provides the right to exploit the deposit. The prerequisites for the granting of the mining permit are to do with the size, ore content and technical characteristics of the deposit concerning its exploitability.

Exploration permits are valid for up to 15 years.

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. The reservation does not entitle the applicant to perform exploration. Instead, the reservation grants a privilege as regards the submission of an ore prospecting application.

4.3 ANNUAL FEES AND ROYALTIES

Legislation requires holders of exploration and mining permits to make annual payments to landowners on EUR/ha basis (see Table 4.3). A statutory mining royalty of 0.15% of the value of the exploited mineral / metal is also payable to the landowner. Discussions in the Finnish Parliament about the implementation of a further 0.6% state royalty are underway and this has been included in the economic evaluation of the project.

Table 4.3 Annual Royalty Payments According to Finland Mining Act 2011

Permit Type	EUR/ha
Exploration (years 1 - 4)	20
Exploration (years 5 - 7)	30
Exploration (years 8 - 10)	40
Exploration (years 11 - 15)	50
Mining	50

The Rupert Lapland Project is subject to a 1.5% net smelter return (NSR) royalty that is capped at a value of US $2 M.

4.4 ENVIRONMENTAL BONDS

Rupert has funded environmental reclamation bonds of EUR 850,000 for the Pahtavaara Gold Mine and a further EUR 106,103 in exploration related bonds covering the Rupert Lapland Project Area.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 PROPERTY ACCESS

The airport of Rovaniemi has several scheduled domestic flights daily to and from Helsinki. The distance from Rovaniemi to Sodankylä is 140 km by road and takes just under two hours to drive. To reach Ikkari, and Heinä Central 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 Pulkittama road for 7.5 km where forest tracks lead directly to the exploration site.

To reach Pahtavaara from Sodankylä, continue to follow the E75 road north towards Sattanen. At Sattanen take a left turn and follow the road for another 15 km until the road sign directs towards the mine site.

Access to all sites is possible throughout the year.

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 bogs and lakes. Hills are mostly covered by glacial moraine and sands and are forested, primarily with birch, pine, and spruce. Bedrock outcrops on the hills and along riverbanks but is limited to some two % or less of the project area. The Ikkari and Heinä Central gold deposits are located at the margins of low-lying bog terrain, cut by a small stream, rising towards boulder-dominated gentle slope in the south/southeast (SSE). The area in general is approximately 225 m above sea level. This terrain largely drains to the north and then east into the Saitta River and then into the Sattanen River and further into the catchment basin of the Kitinen River, and eventually the area drains into the Kemi River.

The Pahtavaara gold deposit is located in a region of incised undulating terrain of low relief. The terrain in general is approximately 240 m to 250 m above sea level. This terrain largely drains to towards the North-North East to Ala-Postojoki river and through that to Kitinen river.

5.3 CLIMATE

According to Köppen climate classification, northern Finland is classified as Dfc (Continental, without a dry season and a cold summer). The region has cold, wet winters, where the mean temperature of the warmest month is no lower than 10 degrees centigrade (°C) and that of the coldest month no higher than -3°C. The rainfall is, on average, moderate in all seasons.

The climate is typical of northern Fennoscandia with temperate summers and cold winters. 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 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 extreme heat 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 located 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.

5.4 LOCAL RESOURCES AND REGIONAL INFRASTRUCTURE

The town of Rovaniemi in Finland is located some 150 km south-southwest of Pahtavaara. 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 Pahtavaara mine and Rupert Lapland exploration permits, including accredited sample preparation facilities operated by ALS Minerals and Eurofins Labtium. ALS Minerals and Eurofins Labtium are internationally accredited laboratories and are

ISO compliant (ISO 9001:2008, ISO/IEC 17025:2005). The regional industrial base is currently dominated by small businesses involved in forestry, agriculture and manufacturing. There are several hotels, shops, and restaurants which accommodate a growing year-round influx of tourists into Lapland. A skilled work force is in place.

Hydroelectric power in the region is 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.

Surface infrastructure at Pahtavaara Mine Site includes a heavy vehicle workshop, administration building, two core sheds and a processing plant located 22 km east-northeast (ENE) of Ikkari. 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 is under construction that will service two temporary facilities buildings that are in place. From the end of 2022 the logistical hub for Ikkari exploration activities will move from the Pahtavaara Mine Site to a newly constructed facility 10 km south of the town of Sodankyla. Management and administration functions have already been relocated to an office in the town of Sodankyla.

Figure 5.1 Regional Infrastructure

6.0 HISTORY

The Pahtavaara deposit was discovered by the Geological Survey of Finland in 1984 when high grade gold mineralisation and visible gold were found in outcrop. Prior to the discovery, gold anomalies in till and bedrock had been detected during regional exploration. Swedish company Terra Mining bought the rights to the deposit from the Ministry of Trade in 1991 and Davy International completed a feasibility study in 1994 and production commenced from open pit mining. Two million tonnes of ore were mined from open pit between 1996 and 2000 when Terra Mining filed for bankruptcy due to low gold prices. Pahtavaara was bought and re-opened in 2003 by Scan Mining and in 2004, the company commenced underground mining. In December 2007 Scan Mining went bankrupt due to financial difficulties in the parent company in Sweden and the failed commissioning of the Blaiken mine. In April 2008 Pahtavaara was bought by Lappland Goldminers and underground mining was recommenced in December 2008. Lappland Goldminers operated until 2014 when the parent company in Sweden filed for bankruptcy and the operation was placed in care and maintenance. Rupert purchased the operation from the administrators of Lappland Goldminers in September 2016.

The Heinälamminvuoma exploration permit on which the Ikkari and Heinä Central Gold Deposits are located, was applied for in 2011 by Lapland Goldminers, who were then owners of the Pahtavaara Mine. The Heinälamminvuoma exploration permit has been part of the Rupert Lapland Project area since that time, although very little exploration was undertaken and exploration field activities confined to the easternmost parts of the licence, adjacent to the Pahtavaara Mine itself. Lappland Goldminers held the Heinälamminvuoma exploration permit, as part of its Pahtavaara Mine operations, until 2014 when the parent company in Sweden filed for bankruptcy. Rupert Resources purchased the Pahtavaara operation, and associated exploration permits, from the administrators of Lappland Goldminers in September 2016.

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

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.

However, in the Pahtavaara area, during the 1980s, the GTK conducted systematic percussion drilling to take samples from the bedrock surface below the overburden. Outcrop mapping and trenching was also performed. Due to the thickness of the till layer and the lack of exposed outcrops, most of the mapping was completed in trenches. Approximately fifty trenches were dug during the 1990s by Terra Mining. The trenches were sampled by sawing channel samples and percussion drilling. When overburden was removed from what was to become the 'C' open pit, the bedrock was considered to be too weathered to complete mapping. As a result, the only maps available prior to the development of the open pits are compiled from the maps of the trenches.

In 2006, the open pit was mapped by Warren Pratt of Specialised Geological Mapping. The overburden of the Länsi ore bodies was removed in 2006 and a detailed map was produced by Pratt in 2007. Both the open pit and Länsi area were sampled by grab samples. Since the production was moved underground in 2004, all drifts have been systematically mapped and the maps have been digitised.

6.2 Previous Geochemical Surveys

Regional and detailed till geochemistry and stratigraphy were analysed by the GTK in the 1980s. Geochemical surveys were performed in the area around Pahtavaara in the 1990s by Terra Mining. All sampling was conducted through the analysis of both till and the underlying bedrock. In 2003 Scan Mining conducted percussion drilling and took samples from both till and the underlying bedrock.

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.

6.3 Previous Geophysical Surveys

The GTK has conducted low-altitude, airborne magnetic, electromagnetic and radiometric surveys and systematic ground magnetic and slingram surveys. The Geological Survey has also conducted ground gravity, Airborne Electromagnetic (AEM), Induced polarisation (IP) and very low frequency radar (VLF-R) surveys in the area. Scan Mining analysed the ground geophysics in 2007. Since 2016, Rupert has completed 27 line km of IP geophysics and has re-flown low altitude airborne geophysics on discrete areas of the property.

6.4 DRILLING BY PREVIOUS EXPLORERS

A total of 452,788 m of drilling has been completed at Pahtavaara from 11,847 holes (Table 6.1). Review of the drill hole assay database has indicated that much of the drilling has been selectively sampled. This relates mostly to the diamond drill holes with approximately 42% of diamond core unsampled and approximately 7% of 'sludge' drill holes unsampled.

Table 6.1 Summary of Available Drill Data for Pahtavaara Gold Deposit

Company	DH Type	Holes	Metres	% of Total
Geological Survey of Finland (pre-1992)	Channel	55	309	0.01
Lappland Goldminers (2009 to 2014)	Diamond	1,232	154,573	34
	RC	78	1,135	0.1
	Sludge (UG)	6,675	124,867	28
	Channel	123	89	0.1
Scan Mining (2004 to 2008)	Diamond	815	94,563	21
	RC	21	1,116	0.1
	Sludge (UG)	2,268	49,902	11
	Channel	134	213	0.1
Terra Mining (1992 to 2000)	Diamond	152	14,853	3
	RC	84	9,976	2
	Sludge (UG)	116	117	0.1
	Unknown	8	300	0.1
Unknown	Sludge	18	668	0.1
	Channel	68	107	0.1
Total		**11,847**	**452,788**	**100**

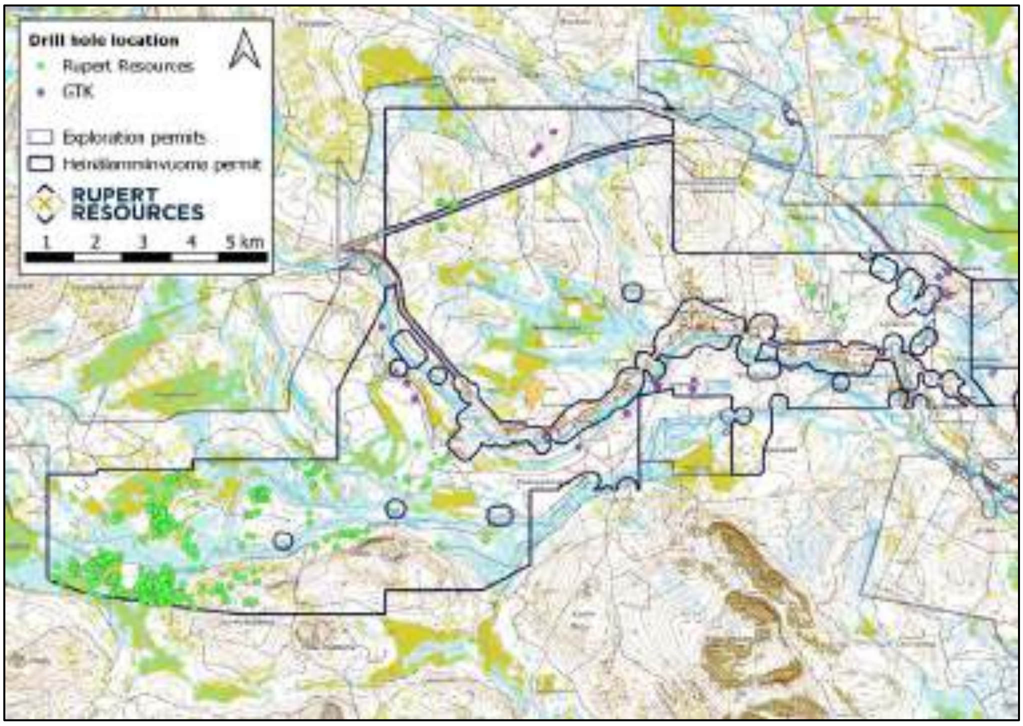
Within the Heinälamminvuoma exploration licence area, a total of 2,420 m of historic diamond drilling has been completed within the licence area, from 26 drill holes (Table 6.2). Very limited drilling has been undertaken by any previous explorers and the majority of these holes are confined to the eastern extent of the licence area (Figure 6.1,).

Table 6.2 Summary of Historic Drill Data for Heinälamminvuoma Exploration Permit Area

Company	DH Type	Holes	Metres % of Total
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.

Figure 6.1 Location of Drilling on the Heinälamminvuoma Exploration Licence



6.5 HISTORICAL RESOURCE AND RESERVE ESTIMATES

6.5.1 PAHTAVAARA

At Pahtavaara, an estimate prepared by Davy in 1994, as part of a feasibility study, which resulted in an open pit reserve of 1,051,000 t grading 3.05 g/t Au with a strip ratio of 4.5:1 and an underground reserve of 512,000 t grading 3.73 g/t Au.

The first resource reported according to NI 43-101, as recorded by the GTK in 2010, was completed by Lappland Goldminers at a 1.5 g/t Au cut off and comprised a Measured and Indicated Resource of 574,000 t grading 3.3 g/t Au and an Inferred resource of 88,000 t grading 7.14 g/t Au. Proven and Probable Reserves were stated as 678,000 t grading 2.79 g/t Au.

Lappland Goldminers published a further NI 43-101 resource and reserve in 2013 using a 0.5 g/t Au cut off. Proven and Probable Reserves were 1,397,000 t grading 1.7 g/t Au derived from a Measured and Indicated Resource of 1,274,000 t grading 2.1 g/t Au. Inferred Resources were estimated as 1,482,000 t grading 1.77 g/t Au.

Rupert Resources published a NI 43-101 resource for Pahtavaara in 2018 using a 1.5 g/t Au cut off with and Inferred Resource of 4,640,000 t grading 3.2 g/Au for 474 koz. This resource included over 50,000 m of drilling completed by Rupert from 2016 up to the end December 2017, along with drilling by the previous owners since the last resource estimate. The modelling work also estimated that 441 koz had been mined from Pahtavaara historically (consistent with production data from 1996 to 2014) indicating a yield of over 2,000 oz/vertical meter for the Pahtavaara Project.

6.5.2 IKKARI

The only previous estimate of the mineral resource at the Ikkari deposit is the inferred, maiden mineral resource released by Rupert Resources in September 2021. Based on 36,000 m of drilling the estimated resource was 49.33 Mt at 2.5 g/t for 3.95 Moz. This estimate was produced by Brain Wolfe of International Resource Solutions Pty Ltd who is a Qualified Person for the reporting of mineral resources in accordance with NI 43-101.

The September 2021 mineral resource was reported at a 0.6 g/t cut-off within a Whittle pit at a revenue factor of 0.4. Outside of this open pit, resources with reasonable prospects for eventual economic extraction were defined using an elevated 1.2 g/t cut-off. The cut-off grades were based on operating costs derived from comparable operations and first principles calculations resulting in estimated operating costs of US $25.2 and US $49.0 for the open pit and underground components respectively. Further assumptions included process recoveries of 92%, a revenue royalty of 0.15% and a gold price of US $1,430 per ounce.

Table 6.3 Ikkari Gold Deposit: Mineral Resource Report (Inferred Resource) – September 2021

Mine Type	Lower Cut-off Grade (g/t Au)	Tonnes (Mt)	Average Grade (g/t Au)	Gold Metal (Moz)	Gold Metal (Kg)
Open Pit	0.6	30.53	2.6	2.51	78,200
Underground	1.2	18.80	2.4	1.44	44,600
Total		**49.33**	**2.5**	**3.95**	**122,800**

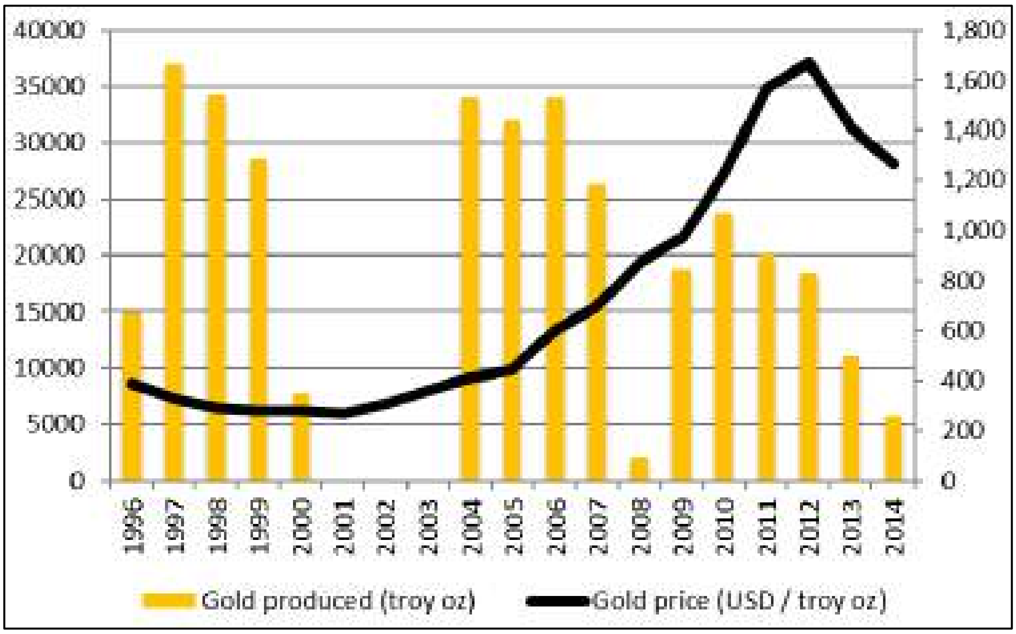
Note: Appropriate rounding has been applied.

6.6 PRODUCTION HISTORY

Reported production is solely from the Pahtavaara mine, with the Ikkari and Heinä Central deposits still being in preliminary exploration and development phases.

Pahtavaara has produced an estimated 348,996 oz of gold from 6,419,226 t ore processed over 16 years in three periods of prior ownership between 1996 and 2014 (see Figure 6.2). The highest recorded production from the open pit was 36,941 oz in 1997, primarily as a result of record throughput of 539,658 t. The highest recorded production from underground was 33,983 oz from mill throughput of 507,002 t (GTK).

Figure 6.2 Production History of Pahtavaara Mine (GTK)



7.0 GEOLOGICAL SETTING AND MINERALISATION

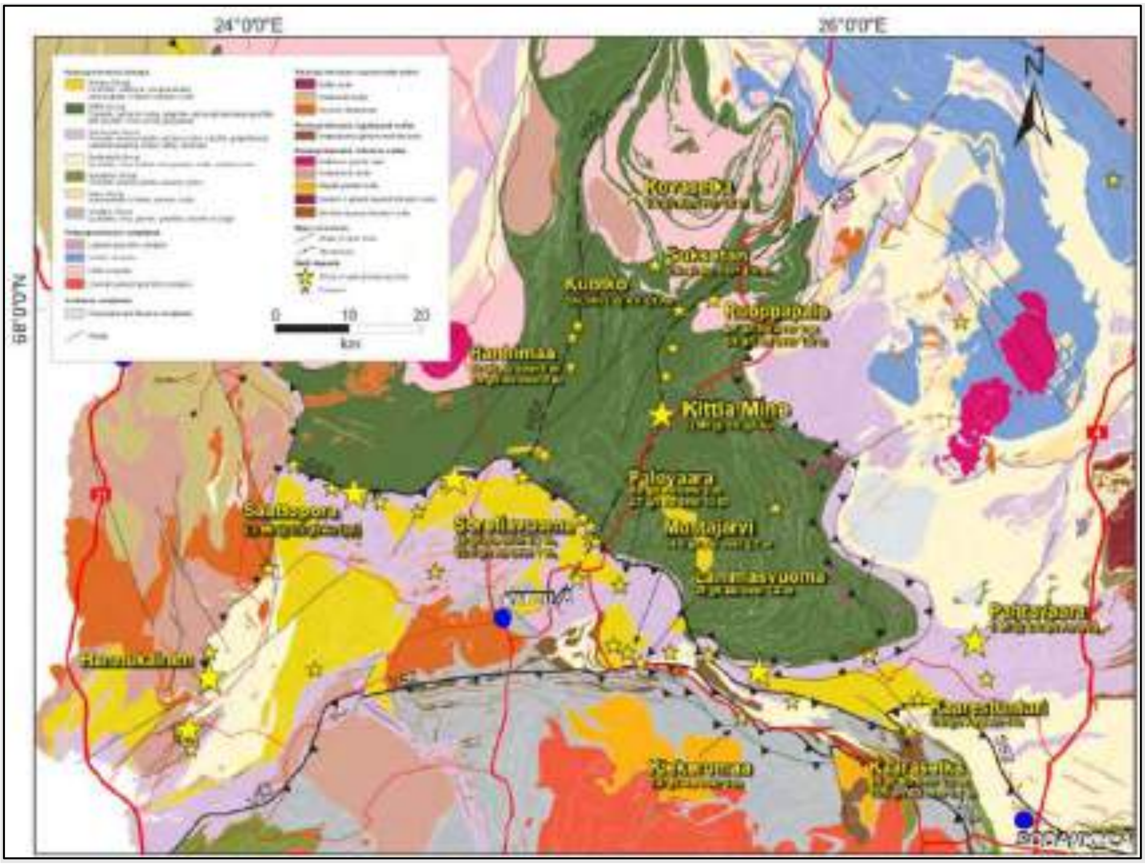
7.1 REGIONAL GEOLOGICAL SETTING

The Rupert Lapland Project area is located within the CLGB, part of the Fennoscandian shield, which hosts 1,700 known incidences of mineralisation in Finland, Sweden, Norway and Russia including around 80 mines. The CLGB has two gold mines of significance. Agnico Eagle's 4.1 Moz Kittilä mine which produced 221,914 oz of gold in 2021 and the historically producing Pahtavaara mine which mined an estimated 441 koz of gold in three periods of ownership between 1996 and 2014 (GTK, Mineral Deposit Report) with a remnant Inferred Resource of 4.6 million tonnes grading 3.2 g/t Au (estimated by Rupert Resources in 2018). The Ikkari deposit, 1.5 km to the south-southwest of Heinä Central, with an inferred resource of 3.95 Moz at 2.5 g/t (estimated by Rupert Resources in 2021: Wolfe, 2021) is a further example of the CLGB prolific gold endowment.

Copper, along with nickel and platinum group elements (PGE's) are mined from Boliden's Kevitsa mine and reported as part of the resource at Anglo American's Sakatti Project located within 16 km northeast and 7 km east of the nearest Rupert Lapland Project Area exploration permit. These two deposits are examples of magmatic sulphide deposits, hosted by an ultramafic intrusive, and are also distinct from the styles of mineralisation encountered within the Rupert Lapland Project area to date.

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).

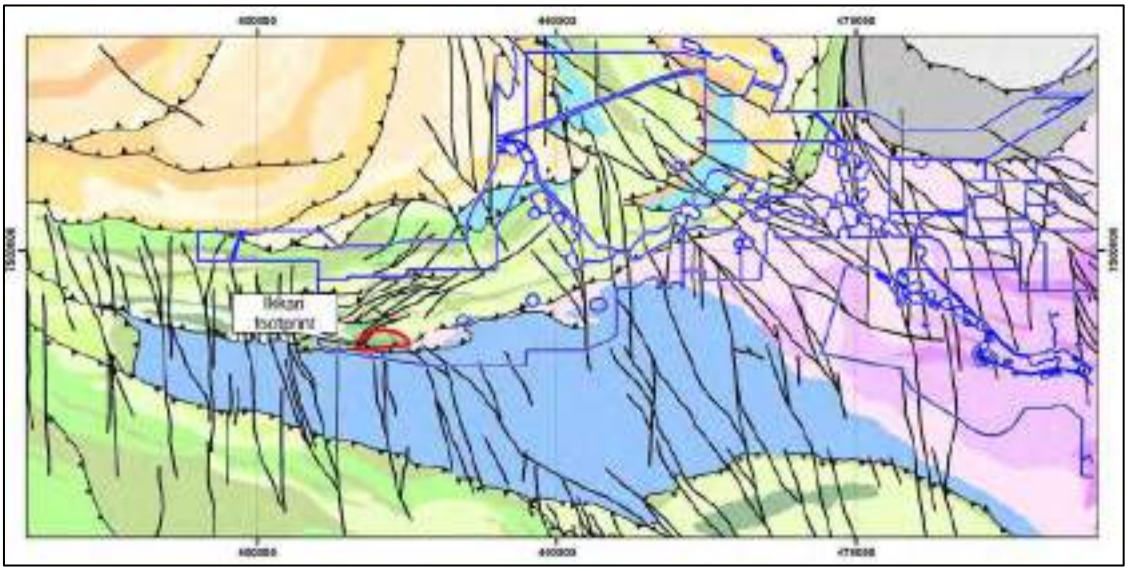
Figure 7.1 Geological Map of Central Lapland Greenstone Belt



Source: GTK Regional Geology Map

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 exploration licences (Figure 7.2). An approximately four kilometre wide zone of 2.05 Ga Savukoski Group rocks, comprising fine-grained 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 (dominantly tholeiitic metabasalts) and younger still Kumpu Group rocks (molasse-type fluviatile quartzites, subarkoses and polymictic conglomerates) to the south.

Figure 7.2 Structural Domain Map of the Ikkari-Pahtavaara District, Based on Potential Field Data



Source: Internal Rupert Resources Ltd database and interpretation, 2022, after Selley, 2019

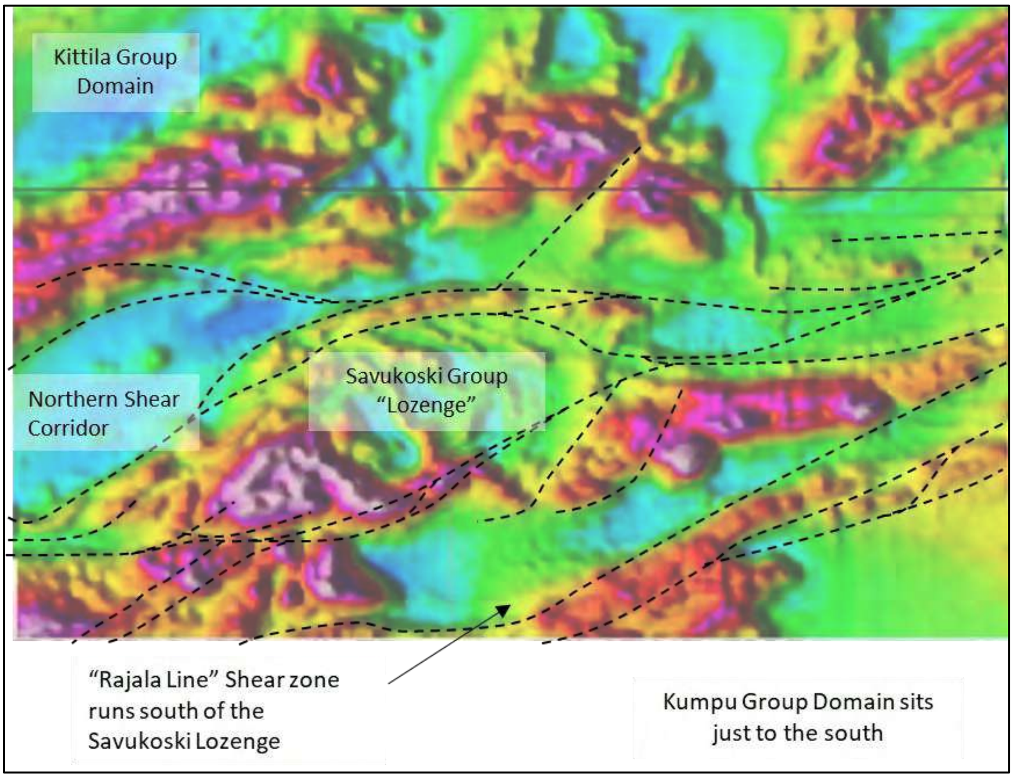
Regional drilling and mapping by Rupert Resources, indicate that the Savukoski Group ‘corridor’ across the Heinälamminvuoma permit area is primarily composed of basalts and fine-grained sediments cut by a multitude of dominantly mafic intrusions. Early, layer-parallel foliation related to the major recumbent NW-SE orientated folds is interpreted to fold the basalts and sediments together during structural thickening prior to upright folding during the second phase of deformation (D_2).

The locally termed “Rajala Line”, a 073° trending 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 sediments, nominally belonging to the Savukoski Group. The structure, possibly originating during the first phase of deformation (D_1) and active during D_2, accommodated ductile thrusting during NNW-SSE compression as the Kittilä and Savukoski Group were thrust towards the south, over the Kumpu Group sediments. The highest intensity ductile deformation seen to date within the Rupert Lapland Project occurs along this trend. A series of shears orientated at 065° intersect the Rajala Line at an angle of 10° anticlockwise to its strike with possible offset markers indicating a greater than one kilometre in-plane sinistral displacement. Despite this interpretation of major shears, there are no major offsets within the Rajala Line or other E-W trending zones. The system of shears is therefore interpreted to detach into these more E-W oriented zones. The Ikkari deposit sits at the south-eastern extent of the Rajala Line, whilst Heinä Central lies to the north of this feature, both within the Rupert Lapland property (Bonson 2021). The relationship between the Pahtavaara deposit and this structural lineation is less apparent.

The Ikkari deposit occupies a complex structural position between thrust imbricated Savukoski Group metavolcanics and metasediments, and synorogenic molasse-type siliciclastic strata of the Kumpu Group. Regionally, a southward convex thrust corridor is dissected by ENE-striking elements to the north of the deposit, whereas an array of apparently late-stage N-S to NW-striking faults propagates from the Kumpu Group 'basin' to the south (Figure 7.2). The unconformity between the Savukoski and Kumpu Groups has been structurally modified and forms the locus of hydrothermal alteration and gold mineralisation.

North of the Savukoski Group lozenge, host to the Heinä Central deposit, a further shear corridor locally parallel to Rajala Line, separates the allochthonous Kittilä Group tholeiitic metabasalts which occur over a significant expanse to the north and northwest.

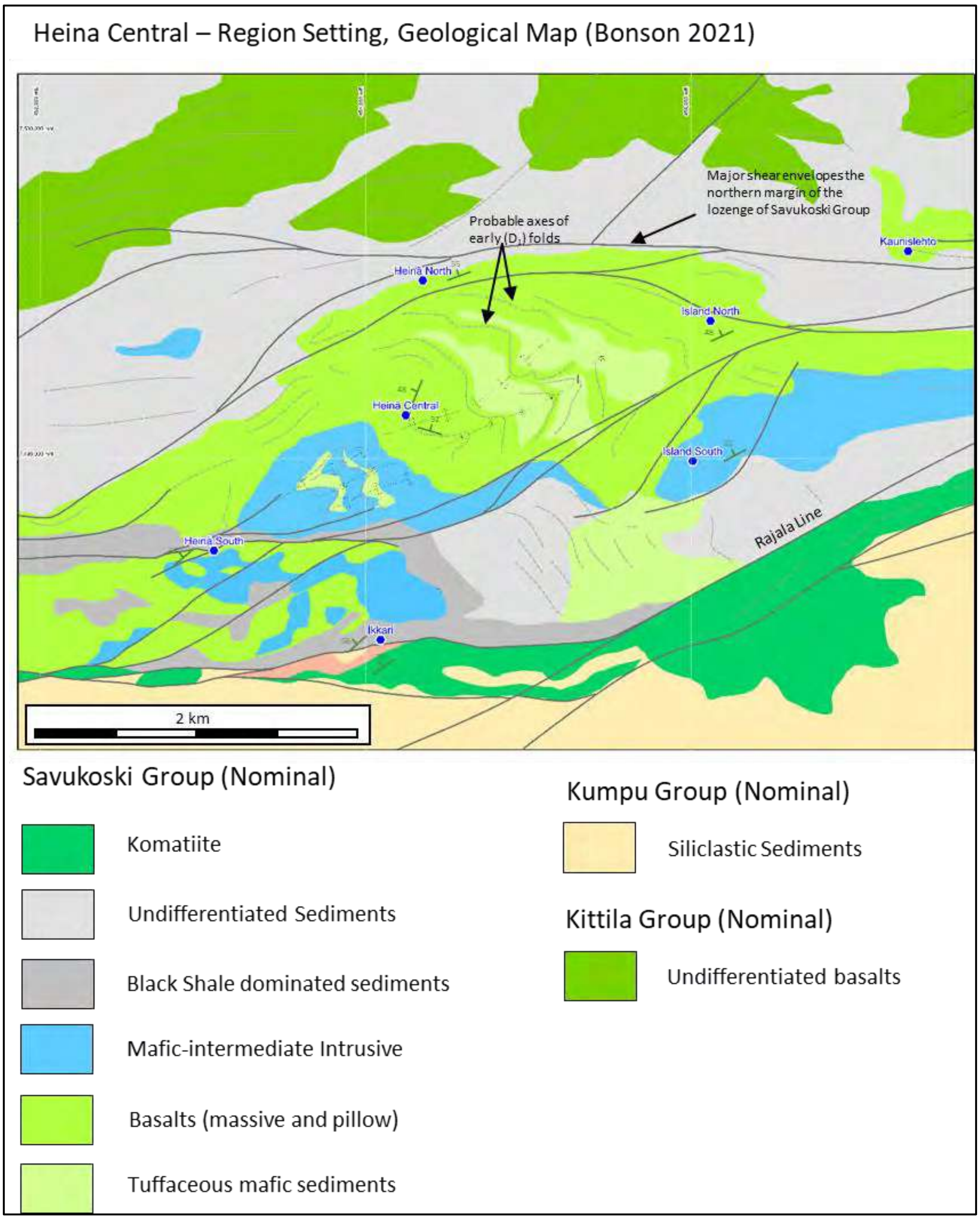
Figure 7.3 Total Magnetic Intensity (TMI), TRP Magnetic Map of the Western Portion of Rupert's Lapland Project – Major Geological Domains and Interpreted Structures



Source: After Bonson, 2021

Within the Savukoski Group lozenge NW-SE trending contacts between mafic volcanics and fine-grained sediments and volcaniclastics are subsequently crenulated by NE trending D_2 fold hinges. The nature of the mafic-sediment contacts is not well understood, these may be primary contacts representing differential basin infill or equally they may represent structural interleaving during recumbent, layer parallel to D_1 deformation. Evidence for the latter occurs at both the Ikkari deposit and within the Kumpu sediments several kilometres to the south of Heinä Central (Selley, 2019).

Figure 7.4 Geological Map of the Heinä Central Regional Setting

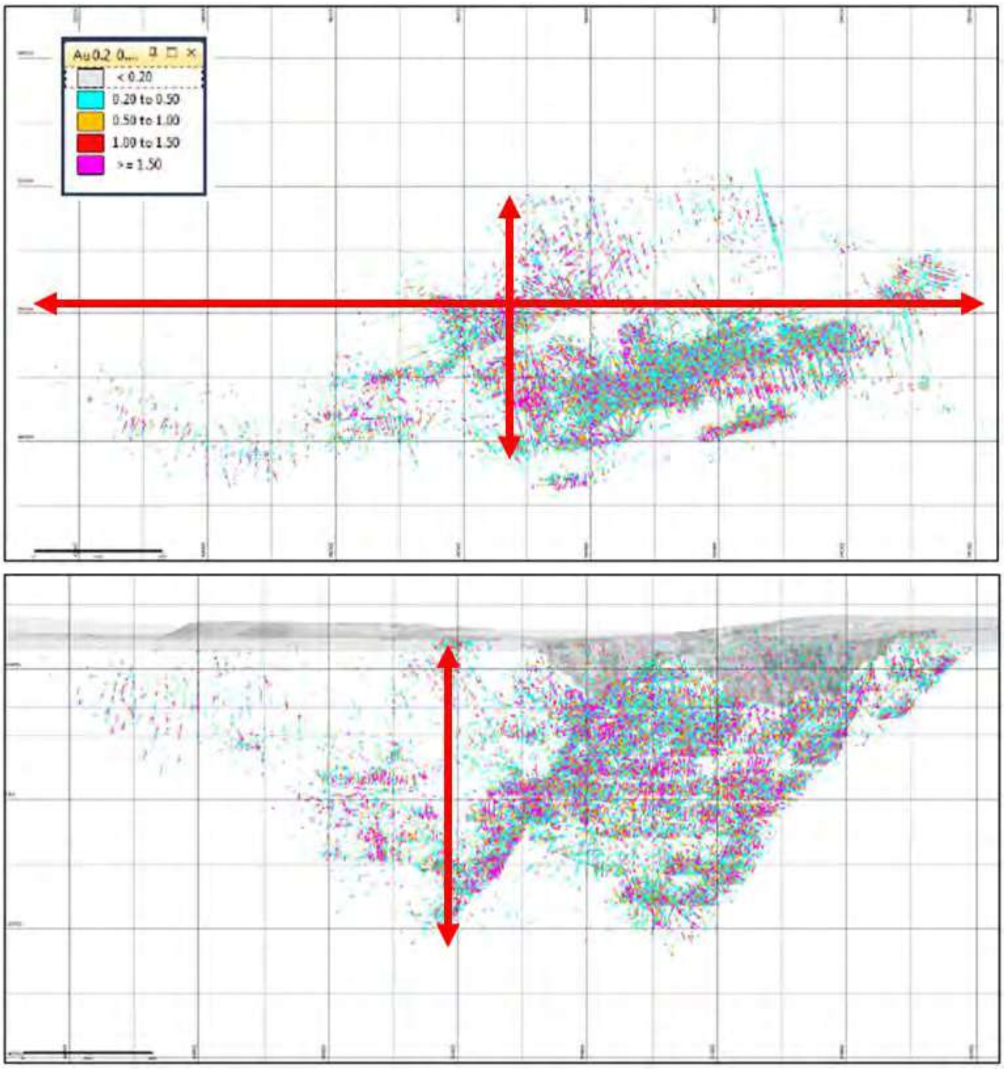


Source: Bonson, 2021

7.2 DEPOSIT GEOLOGY - PAHTAVAARA

The Pahtavaara gold deposit can be described as an orogenic, metamorphic, hydrothermal gold deposit. Geological modelling utilising over 500,000 m of drilling available for the Pahtavaara deposit has shown the deposit to lie within a mineralised envelope up to 500 m wide and up to 1.5 km long (see Figure 7.5) and that the deposit remains demonstrably open at depth and along strike. In the 1994 Feasibility Study the deposit was described as occurring in a gold-bearing alteration zone covering 100 m x 600 m, dipping 80° to 85° NNW (Davy, 1994).

Figure 7.5 Currently Defined Limits of Pahtavaara Mineralisation



Scale bar: 200 m, Grid: 100 m

Source: Internal Rupert Resources Ltd database, 2022

7.2.1 STRUCTURE AND MINERALISATION

Pahtavaara is hosted by ultramafic rocks (komatiites to high magnesian basalts). Gold mineralisation is structurally controlled and associated with low sulphidation, polyphase quartz-carbonate veining, multiple deformation phases varying from brittle to ductile and back to brittle and intense alteration, both prograde and retrograde. This has resulted in a complex vein overprinting history (Figure 7.6).

Figure 7.6 Free Gold in Drill Core (A), Polyphase Structures and Veining (B & C)

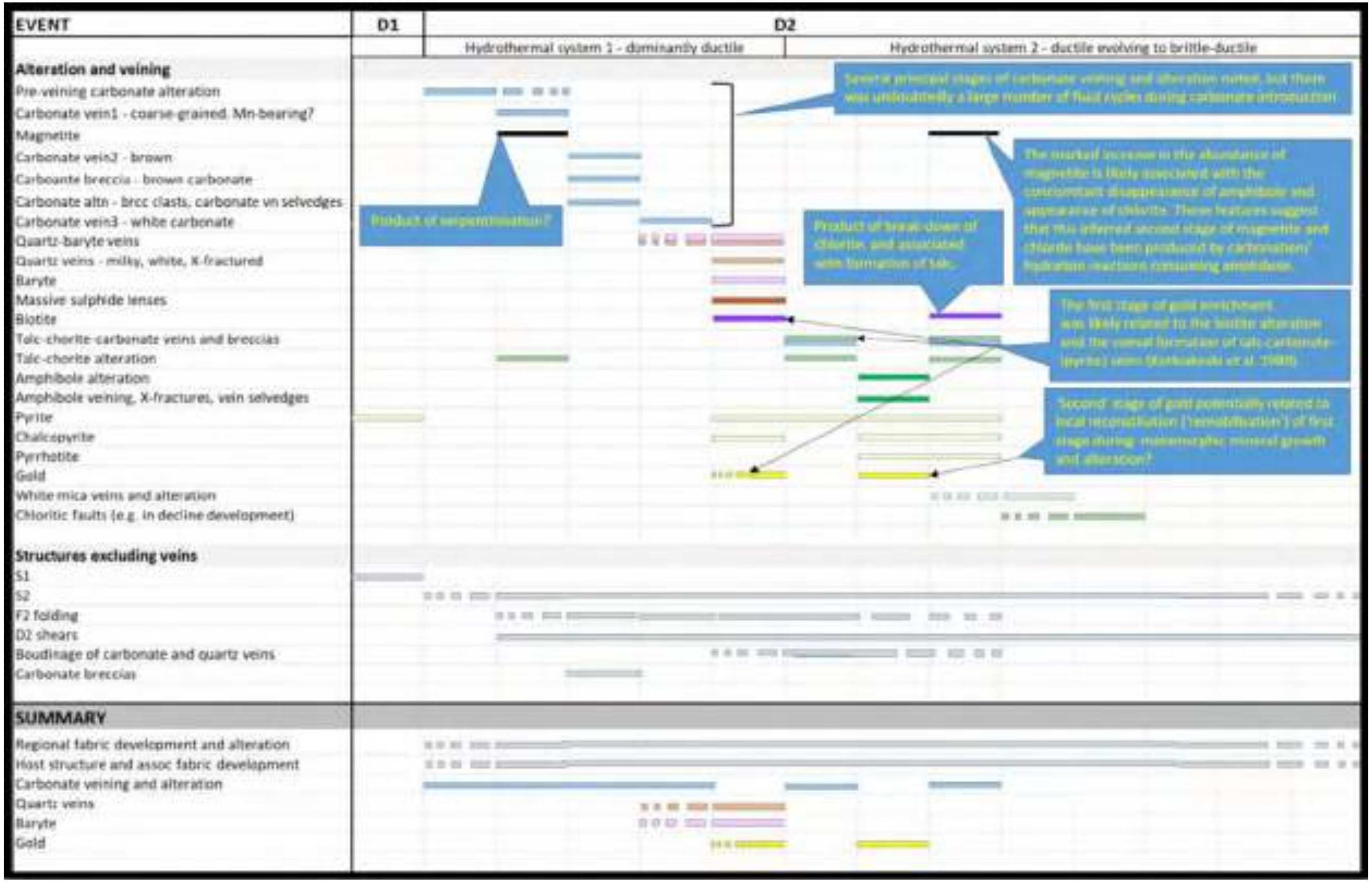
Source: Davis 2018

A complex alteration sequence has been identified, with multiple phases of overprinting, governed by changes in fluid chemistry and their reaction with host rocks of differing mineralogy. Alteration assemblages include initial implied serpentinisation, regional intense carbonate, talc-carbonate +/- chlorite +/- pyrite, amphibole-chlorite, amphibole and biotite.

Unusual aspects of the gold mineralisation at Pahtavaara include a Ni-Cu-Co geochemical signature (most probably due to the ultramafic host rocks) and minor massive sulphide lenses formed during prograde metamorphic and ductile conditions.

Two phases of gold mineralisation have been observed; early fine grained and later, more coarse-grained (Figure 7.6, Davis, 2018). Both are ‘free’ gold, as the deposit does not exhibit refractory metallurgical characteristics.

Figure 7.7 Geological History of Pahtavaara



Source: Internal Rupert Resources Ltd interpretation, 2022, after Davis, 2018

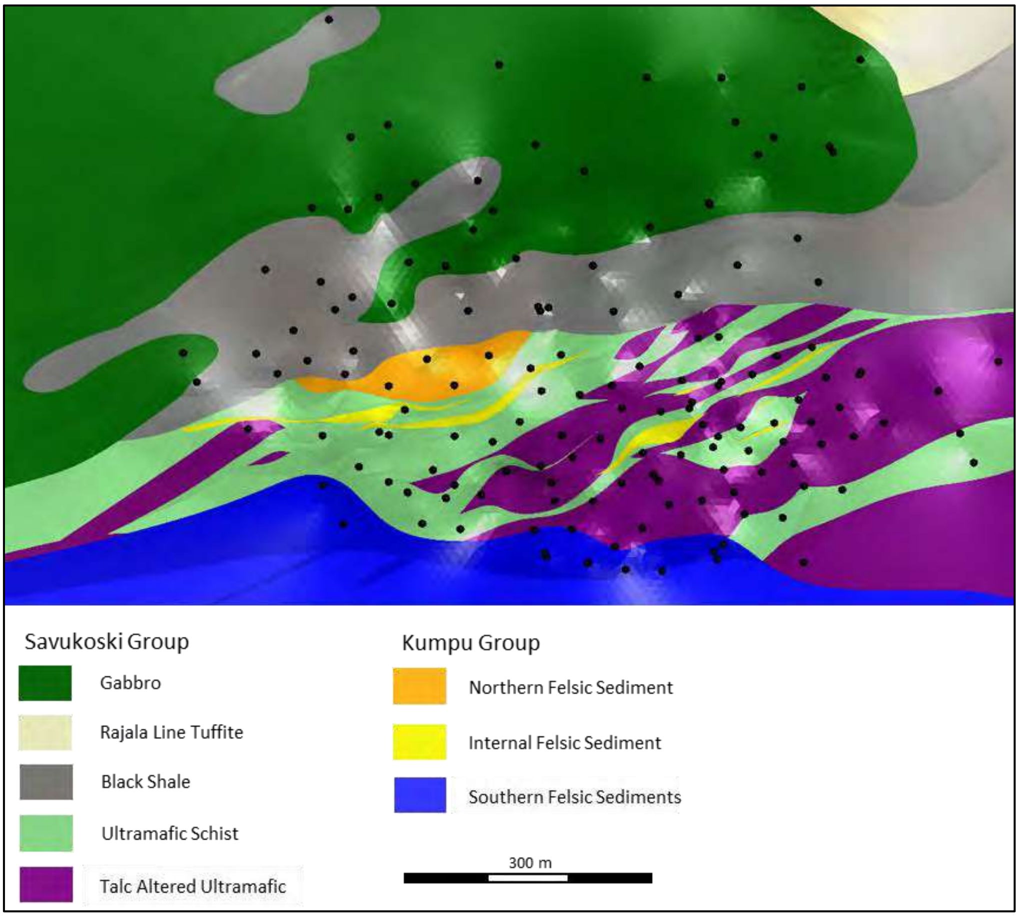
7.3 DEPOSIT GEOLOGY – IKKARI

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 host rocks of Ikkari mineralisation (Figure 7.2):

- Dark pyritic shales and siltstones (intruded by gabbro) comprise the majority of the northern fault block.
- 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.
- 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 7.8 Plan Map of Ikkari Taken From 3D Geological Model with Overburden Removed



Source: Internal Rupert Resources Ltd database, 2022

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 7.9).

Figure 7.9 Basic Relationship Between Protolith and Alteration Products at Ikkari

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

Source: Internal Rupert Resources Ltd Interpretation, 2022

7.3.1 Rock Types

In more detail, ultramafic units are dark grey to green-grey schistose extrusive rocks, which occasionally exhibit volcanoclastic textures with lapilli-like deformed clasts. Geochemically, they are komatiitic in composition (> 60% Mg) and are almost completely altered to talc-chlorite composition, but also variably contain serpentine, amphibole and biotite and characteristic narrow, wispy calcite veinlets (Figure 7.10). In places, intensely altered ultramafic rocks may appear as a more mafic lithology (magnesium replaced by iron), in proximity to the mineralised zone). The ultramafic sequence forms an over 100 m-thick continuous unit between the Ikkari mineralisation and footwall quartzites.

Figure 7.10 Example of Barren Ultramafic Rocks with Characteristic Calcite Veining

Note: Hole 120065 at 138 m *Core size HQ (63.5 mm core width).

Source: Internal Rupert Resources Ltd database, 2022

Figure 7.11 Example of Chaotically Veined, Stockwork-like Siderite ± Chlorite ± Sulphide Veins in Altered Mafic-ultramafic Rock with Strong Sericite-Silica Overprint

Note: Fine-grained disseminated pyrite hosts 5.78 g/t Au in this sample. Hole 120086 at 166 m. *Core size NQ (50.7 mm core width).

Source: Internal Rupert Resources Ltd database, 2022

The mixed ultramafic-sedimentary package (shown on plan map as ultramafic schist and internal sediment) is characterised by highly variable alteration styles, in places intense veining and foliation that frequently overprint texture, making identification of the original lithology difficult. Geochemical characterisation of these rocks also gives a 'mixed' signature and it is inferred that the sediments range from felsic (pelites) through to heterogeneous mafic-dominated composition (greywacke), implying a variable volcanogenic component. Where original textures are preserved, finely laminated dark grey to green-brown silty sediments are common, with occasional coarse grained (up to gravel-sized clasts) units. Hydrothermal alteration overprints a biotite-chlorite-mica greenschist assemblage and commonly comprises quartz-dolomite-chlorite-magnetite (± haematite), particularly in association with veining (Figure 7.12). In places, sedimentary banding is commonly defined (or enhanced) by albite flooding.

Figure 7.12 Example of Boudinage Quartz-carbonate Veins in Altered Mafic-ultramafic Rock

Note: Hole 1120071 at 240 m. *Core size HQ (63.5 mm core width).

Source: Internal Rupert Resources Ltd database, 2022

Felsic sediments are intensely and pervasively albite-altered, particularly forming a large block of albitised rock in the north western 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.

Figure 7.13 Example of Strongly Albite-altered Felsic Sedimentary Rock with Micro-veinlets Hosting Pyrite-magnetite

Note: Hole 120072 at 106 m *Core size HQ (63.5 mm core width). Assays of 4.1 g/t Au in this sample.
Source: Internal Rupert Resources Ltd database, 2022

Laminated carbonaceous shale (black schist) forms the hangingwall (northern margin) to mineralisation in most places. It contains significant amounts of syngenetic disseminated pyrite Figure 7.14), which is often banded, and although graphite content is overall low, graphitic fractures occur in places.

Figure 7.14 Example of Laminated Carbonaceous Shale Displaying Folding with Unmineralised Pyrite

Source: Internal Rupert Resources Ltd database, 2022

In the northernmost extent of the deposit is a mafic intrusive of gabbroic composition, which intrudes the carbonaceous shale, including locally, narrow dykes.

To the south of the mineralised zone, the contact with the Kumpu Group quartzites is poorly defined. The Kumpu Quartzites are coarse-grained, relatively unaltered and weakly strained. Whilst in places the contact appears to be faulted and dips irregularly towards the north drilling beyond this, drilling in very limited areas has indicated that the intercalation of komatiitic strata continues albeit in decreasing abundance to the south. Minor mineralisation is seen in quartz veinlets at faulted contacts to the quartzites and rarely within the quartzite.

It should be noted that the age and relative timing of the felsic sediments, particularly those intercalated within the ultramafics, is currently unclear. Preliminary dating of some of these units indicates that they are all of a similar Kumpu age approximately 1.88 Ga (Harju, 2021) which suggests that these younger rocks must have been complexly structurally interposed within the older komatiite units prior to mineralisation.

Breccias are common throughout the deposit and occur is 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

Structure$_2$ (S_2) foliation (Figure 7.15). This style of brecciation is interpreted to be depositional in origin.

- A relatively early cataclastic tourmaline-carbonate-silica-cemented breccia commonly contains clasts of albite-altered sediments (Figure 7.16). In places these are overprinted by the mesothermal alteration regime and are tentatively interpreted to represent D_1 structures.
- A late, carbonate-iron-oxide-rich, hydrothermal breccia that contains rounded quartz grains in a fine-grained matrix and is sometimes vuggy (Figure 7.17). With typically narrow (10 to 30 centimetres [cm] wide) cross-cutting geometries that indicate fluidised injection (Figure 7.18), these breccias frequently host disseminated pyrite, and associated gold grades. Breccias appear to have a dominant sub-vertical control and are associated with high-grade gold mineralisation and sulphide concentrations, within and particularly at margins.

Figure 7.15 Example of Chlorite Alteration and Disseminated Pyrite (seen here as rusty staining) Within Disrupted Coarse Carbonate-veined Ultramafic Rock

Note: Hole 120059 at 132 m, containing 1.57 ppm Au. Thicker, coarser-grained siderite veins frequently appear to occupy a marginal position to the mineralised zone. *Core size NQ (50.7 mm core width).

Source: Internal Rupert Resources Ltd database, 2022

Figure 7.16 Example of Polymictic, Cataclastic Breccia with Fushsitic Siltstone Clasts and Rounded Quartz Grains

Note: Hole 120059 at 276 m. *Core size NQ (50.7 mm core width).

Source: Internal Rupert Resources Ltd database, 2022

Figure 7.17 Example of Late Haematite-cemented Breccia in Albite-altered Felsic Sediments

Note: Hole 120075 177 m, containing 4.35 ppm Au. *Core size NQ (50.7 mm core width).
Source: Internal Rupert Resources Ltd database, 2022

Figure 7.18 Example of Narrow, Iron-oxide-rich Breccia

Note: Hole 120123 at 196.4 m, containing 64.9 ppm Au. *Core size NQ (50.7 mm core width).
Source: Internal Rupert Resources Ltd database, 2022

Figure 7.19 Example of Narrow, Late Calcite Veins Cross-Cutting Breccia Development (left) and Sedimentary Banding (right)

Note: Hole 120086 at 299 m. *Core size NQ (50.7 mm core width).
Source: Internal Rupert Resources Ltd database, 2022

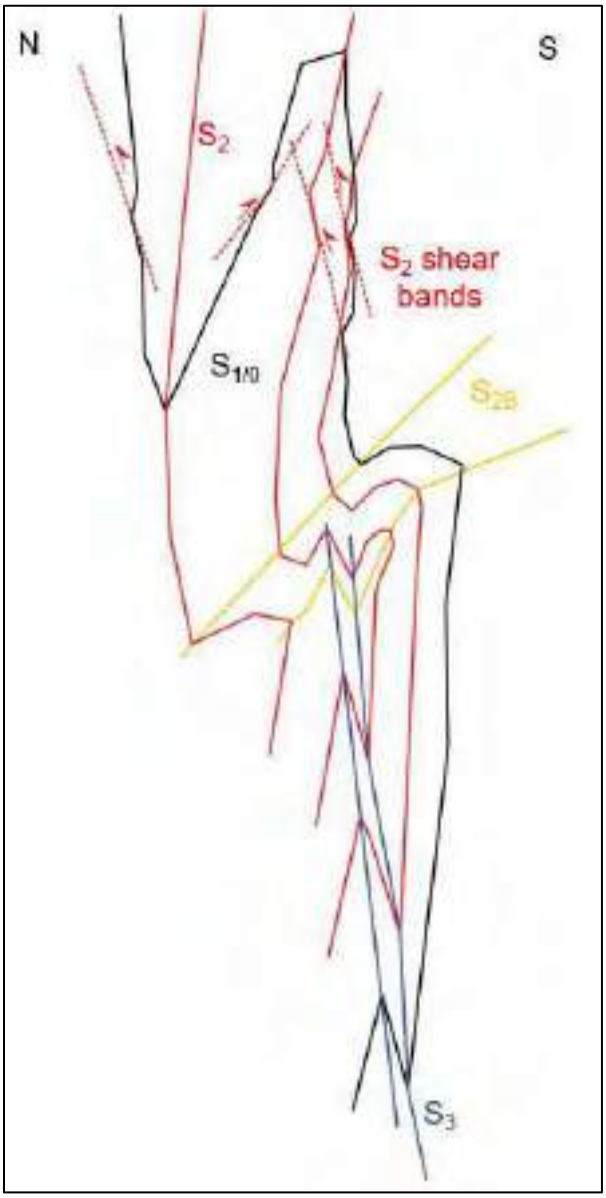
7.3.2 STRUCTURE

Occurring 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 and komatiite-dominated Savukoski groups strata. A steeply 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 mineralised corridor is less well constrained by drilling but appears to comprise an outlier of quartzitic Kumpu Group sandstones and conglomerates that is at least locally in sheared contact with komatiitic rocks to the north. In other localities Kumpu quartzites become progressively more dominant though interleaved komatiitic rocks persist in decreasing quantities. The intensity of the foliation developed also drops markedly to the south.

Initial structural studies of representative drill hole intersections from Ikkari (Selley, 2021) indicate three distinct phases of deformation that are texturally and geometrically analogous to the deformation history recorded throughout the region (Figure 7.8). 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.

A first phase of deformation (D_1) records early orogenic, large-scale recumbent folding and thrust stacking, with layer-parallel fabrics developed (e.g. Figure 7.27). Although this deformation is poorly preserved it interpreted to be responsible for the complex interleaving of shale and felsic sediments with komatiitic facies, which appears to have been a necessary 'pre-conditioning' for gold mineralisation throughout the deposit.

Figure 7.20 Schematic Representation of Three Phases of Structural Deformation at Ikkari Deposit, Showing Planar Fabric Relationships and Resulting Complex Structural Meshwork

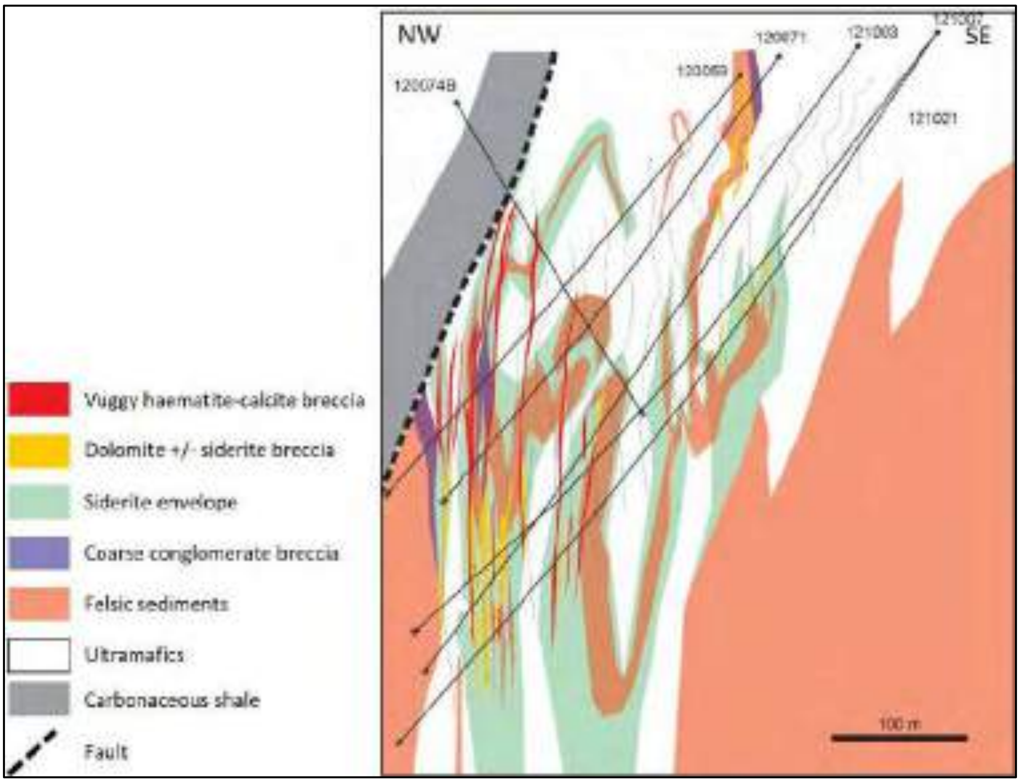


Source: Selley, 2021

A later deformational event (D_2) N-S to NW-SE compression of the thrust stack, resulted in the development of tight (meter-scale) upright folds (shallowly SW-plunging fold axes), which resulted in the complex geometries of the interleaved sediments that are now observed (Figure 7.21, Figure 7.22 and Figure 7.23). It is apparent that this geometry also contributes to the localisation of gold mineralisation, fold hinges appear preferentially mineralised and host some of the high-grade intercepts. This deformation phase included NE- to E-W striking, steeply dipping shear zones that dissect the fold geometry and provide the dominant foliation across the deposit.

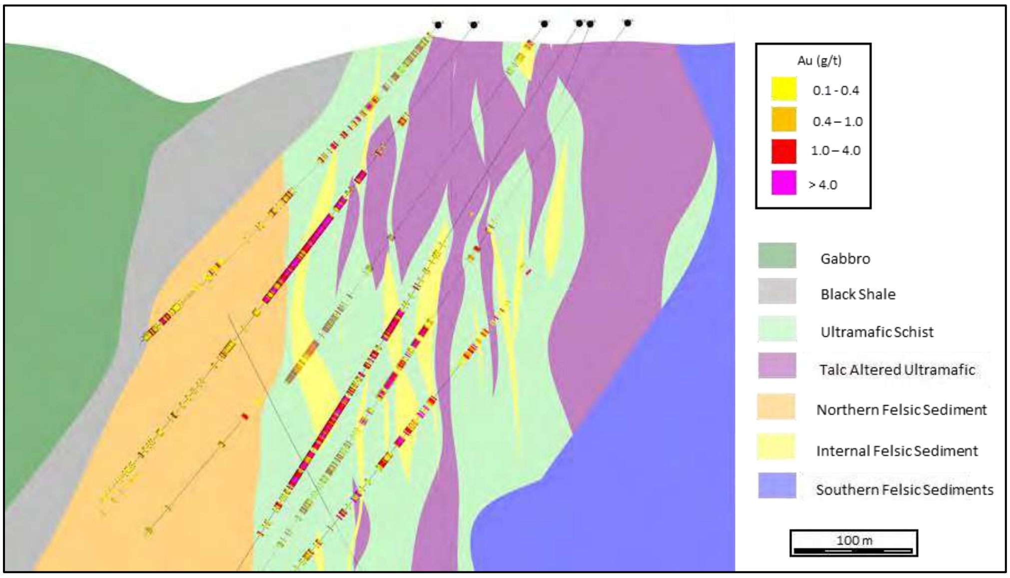
D_2 fabrics are most closely associated with gold-related alteration. An early phase of quartz-sericite-pyrite+/-dolomite veining appears to account for the many of the low and moderate grade gold intervals. The central komatiitic zone is affected by semi-ductile shears, surrounded by a sericite alteration envelope. However, pyrite-rich zones of chlorite-magnetite-siderite alteration (reflecting early iron-concentration) are also typically well mineralised.

Figure 7.21 Cross Section (Looking NE) Through Ikkari Deposit (Central Section)



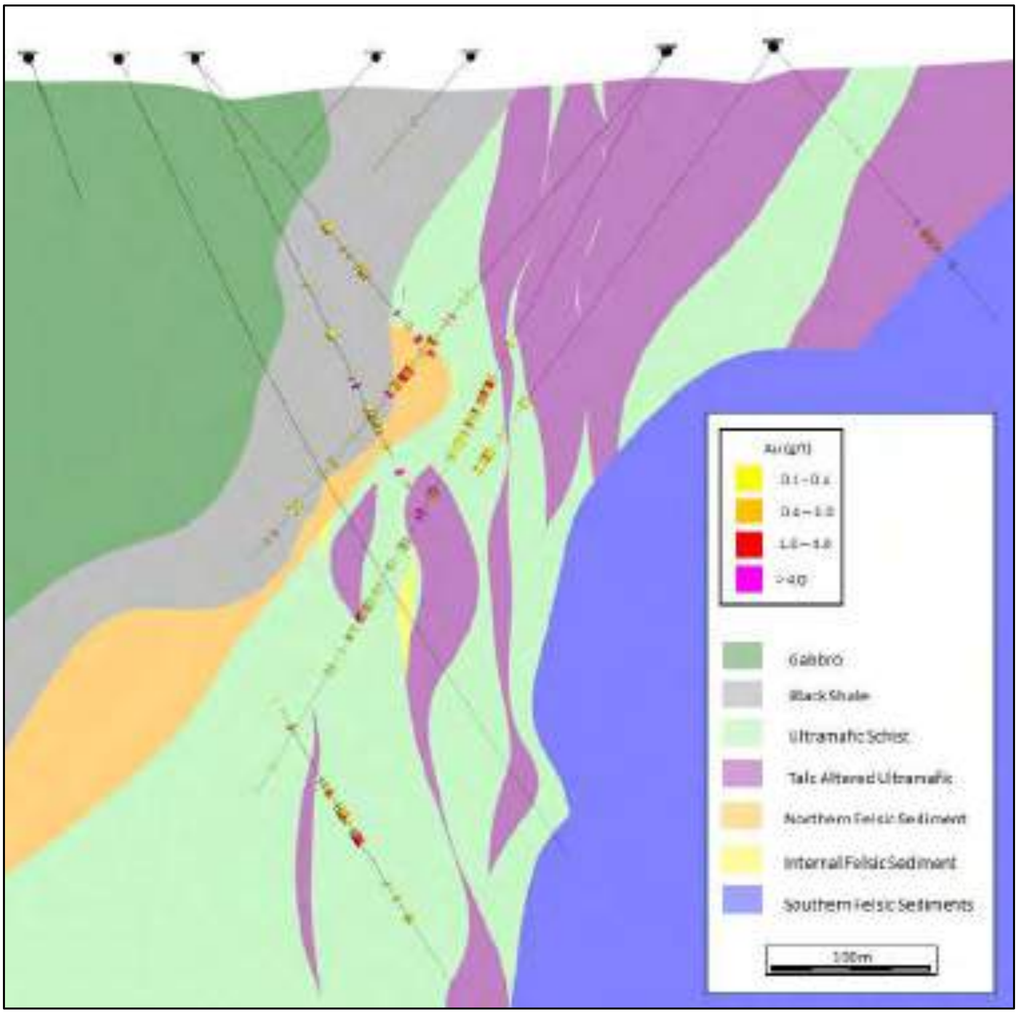
Source: Selley, 2021

Figure 7.22 Cross Section (Looking NE) Through Ikkari Deposit (Central Section)



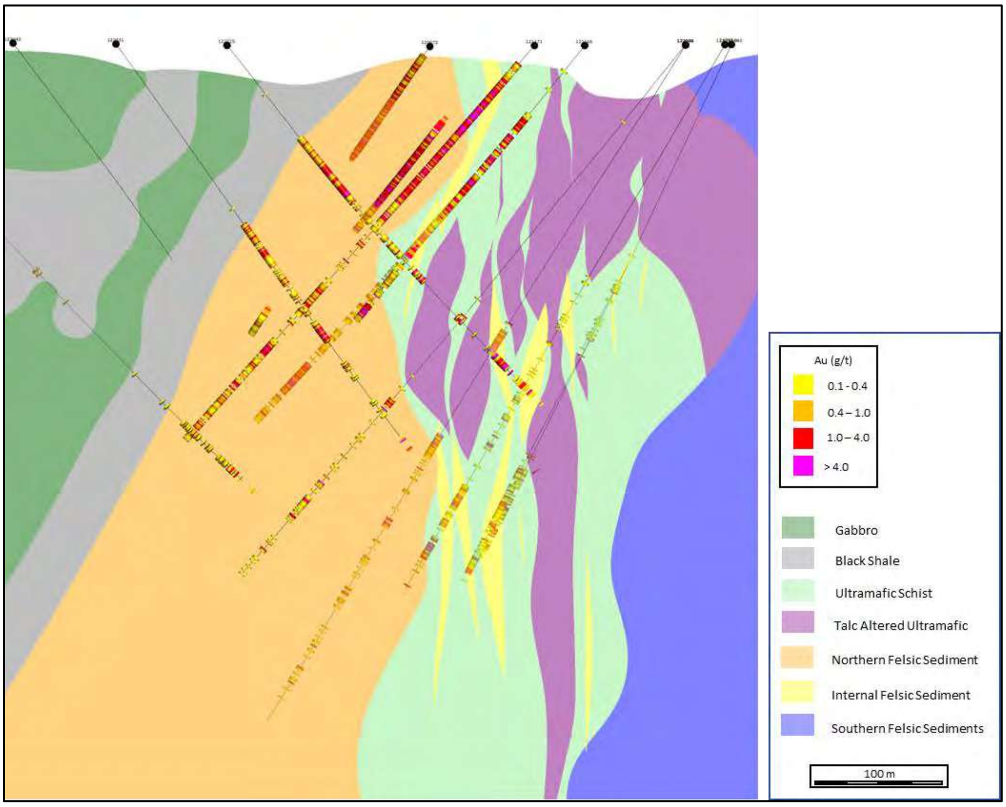
Source: Internal Rupert Resources Ltd database and interpretation, 2022

Figure 7.23 Cross Section (Looking NE) Through Ikkari Deposit (Eastern Section)



Source: Internal Rupert Resources Ltd database and interpretation, 2022

Figure 7.24 Cross Section (Looking NE) Through Ikkari Deposit (Western Section)



Source: Internal Rupert Resources Ltd database and interpretation, 2022

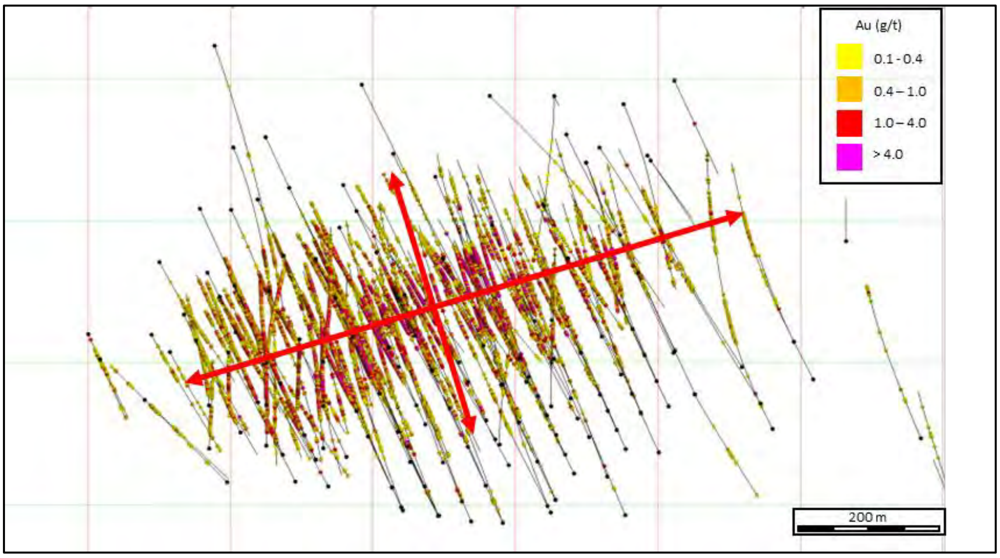
A third deformation phase (D_3) records E-W compression which reactivated subvertical shears, recording strike-slip movement and developed steep, sub-vertically plunging fold axes which contribute to the difficulty of tracing sedimentary intercalations between sections. This D_3 folding is responsible 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.

Late-stage vuggy, carbonate-rich breccias with a subvertical control are also considered to be controlled by the D_3 folding, and the close spatial association with the youngest phase of gold-rich (typically high-grade) quartz-haematite-calcite-pyrite brecciation suggests a second phase of gold-bearing fluid injection at this time. Emplacement of the N-dipping, barren, carbonaceous shale that marks the northern boundary of the tightly folded, mineralised block and cross-cuts the upright foliation in the east is also believed to be late stage and may correlate with D_3.

7.3.3 MINERALISATION

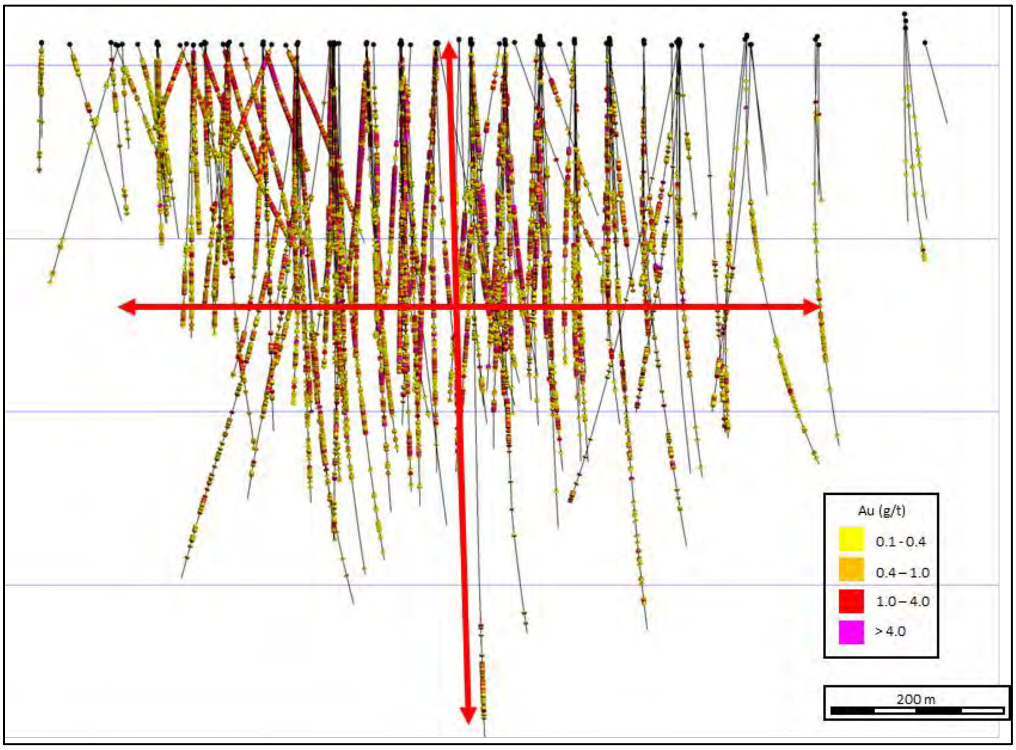
The Ikkari deposit can be described as an orogenic, hydrothermal gold deposit. Modelling of the mineralisation, using over 80,000 m of drilling available, shows the deposit to lie within a mineralised envelope of up to 800 m long, 300 m wide and 750 m deep Figure 7.25) and that the deposit remains demonstrably open at depth and along strike.

Figure 7.25 Currently Defined Limits of Ikkari Mineralisation (Plan View)



Scale bar: 200 m, Grid: 200 m
Source: Internal Rupert Resources Ltd database, 2022

Figure 7.26 Currently Defined Limits of Ikkari Mineralisation (Looking Northwest)



Scale bar: 200 m, Grid: 200 m
Source: Internal Rupert Resources Ltd database, 2022

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 controlled by faulting, folding intensity and localisation at lithological contacts. The style of mineralisation is principally controlled by the host lithology with significant controls on mineralisation localisation including:

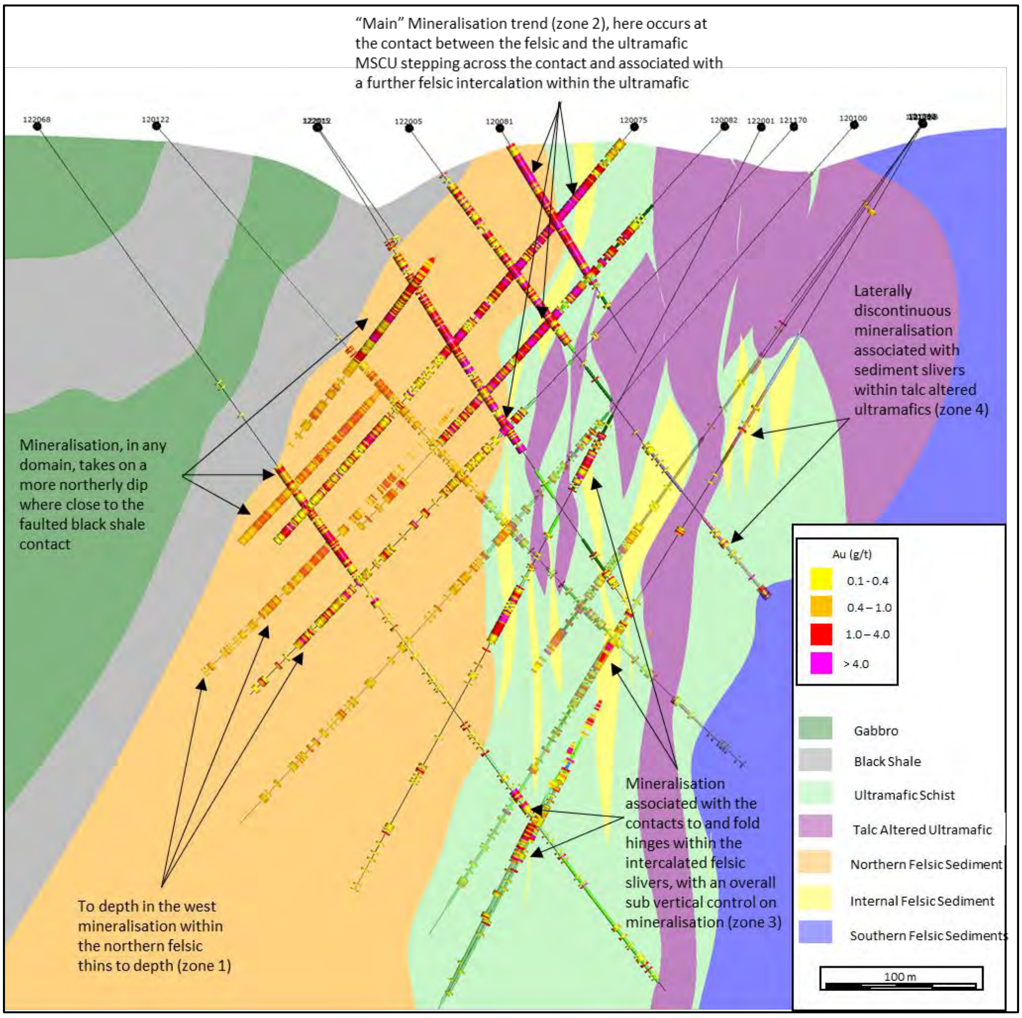
- Brittle-fracture regime in intensely albite-altered felsic sediments that controls veinlets of gold associated with fine-grained pyrite and magnetite (e.g. Figure 7.11). Given that this style of mineralisation is limited to the albite-altered sediments it is most prevalent in the north-western portion of Ikkari where the felsic sediments form a large block. It also occurs in larger felsic intercalations within the komatiite domain.
- Complex and concentrated short-wavelength (metre-scale) parasitic folding of narrow felsic sediment intercalations within intensely chlorite-sericite-altered mafic-ultramafic rocks (e.g. Figure 7.30), that appears to have focused fluid flow and pyrite deposition, particularly at fold hinges and lithological contacts. Intense, irregular carbonate-quartz veining is frequently developed in these zones and are also mineralised (e.g. Figure 7.31).
- At lithological contacts; notably within intensely sericite-pyrite-(±fuchsite)-altered sediments, at contacts with felsic sediments or mafic-ultramafic rocks; often as narrow intercalations.
- Within and at the margins of, several phases of hydrothermal and tectonic brecciation (e.g. Figure 7.14), 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 (e.g. Figure 7.31).

Ikkari is unusual among orogenic gold deposits in the width of mineralisation when compared to the strike. 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 can be attributed to multiple, stacked mineralised zones perpendicular to the strike. These stacked zones are interpreted to arise from the interleaving of diverse lithologies pre-mineralisation, 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 more weakly albite-altered felsic sediments (Figure 7.27).

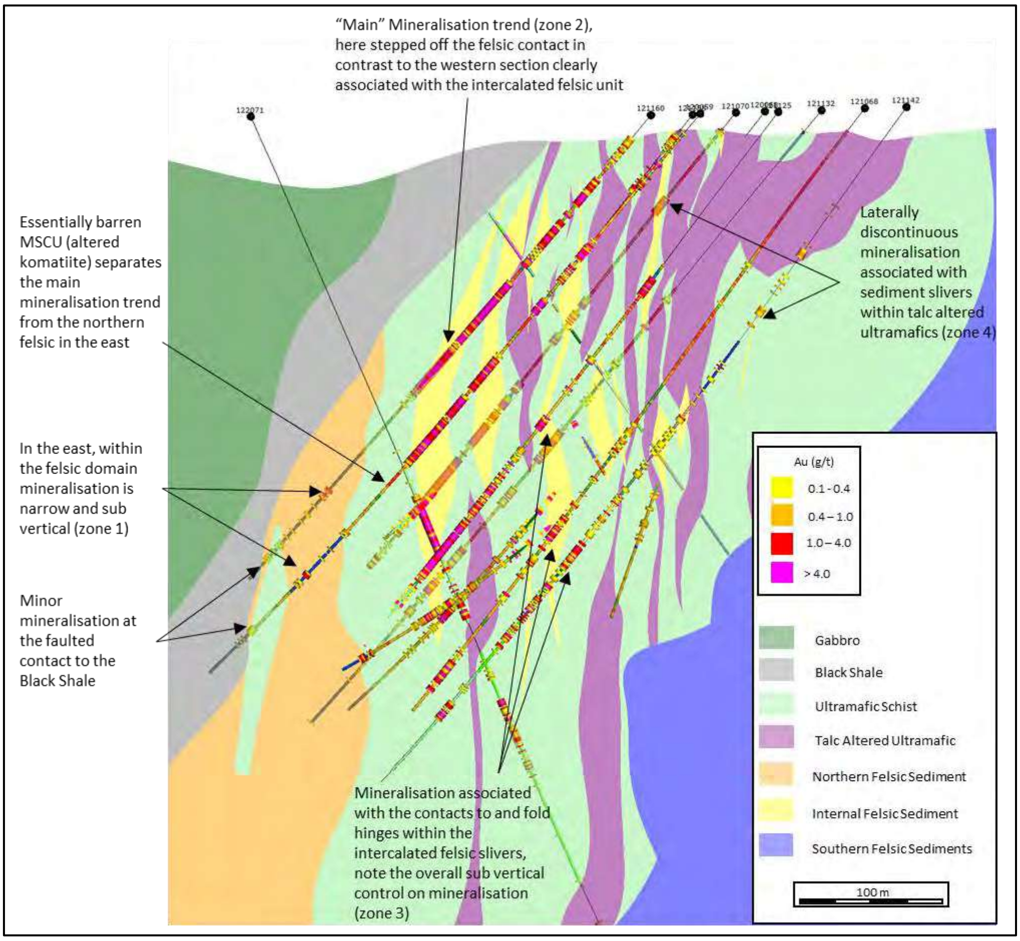
- At the contact between the main felsic block and the komatiite domain in the west, and then stepping off this contact to the east to be entirely within the komatiite domain, is the next zone of mineralisation. In the west, mineralisation occurs at both sides of the felsic-komatiite contact with the intensely albite-altered felsic sediments hosting a 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 intercalated felsic and phyllitic sediments, the contacts of these and fold hinges within (Figure 7.27)
- 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 7.29) consistent with the shale but elsewhere the dip is more vertical and the apparent plunge is approximately 30° to the east (Figure 7.28).
- Further south still are several parallel mineralisation trends within the komatiite domain 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, as well as the hinges of parasitic folds in these intercalations. Mineralisation in this portion of the deposit plunges back to the WSW at approximately 20° which is consistent with the S_1-S_2 intersection lineation and F_2 fold hinges as measured throughout the komatiite domain. (Figure 7.27)
- 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
- To the south, within the 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 7.27 and Figure 7.28).

Figure 7.27 Cross Section from the Central Western Portion of the Deposit with Different Zones and Styles and Mineralisation Highlighted



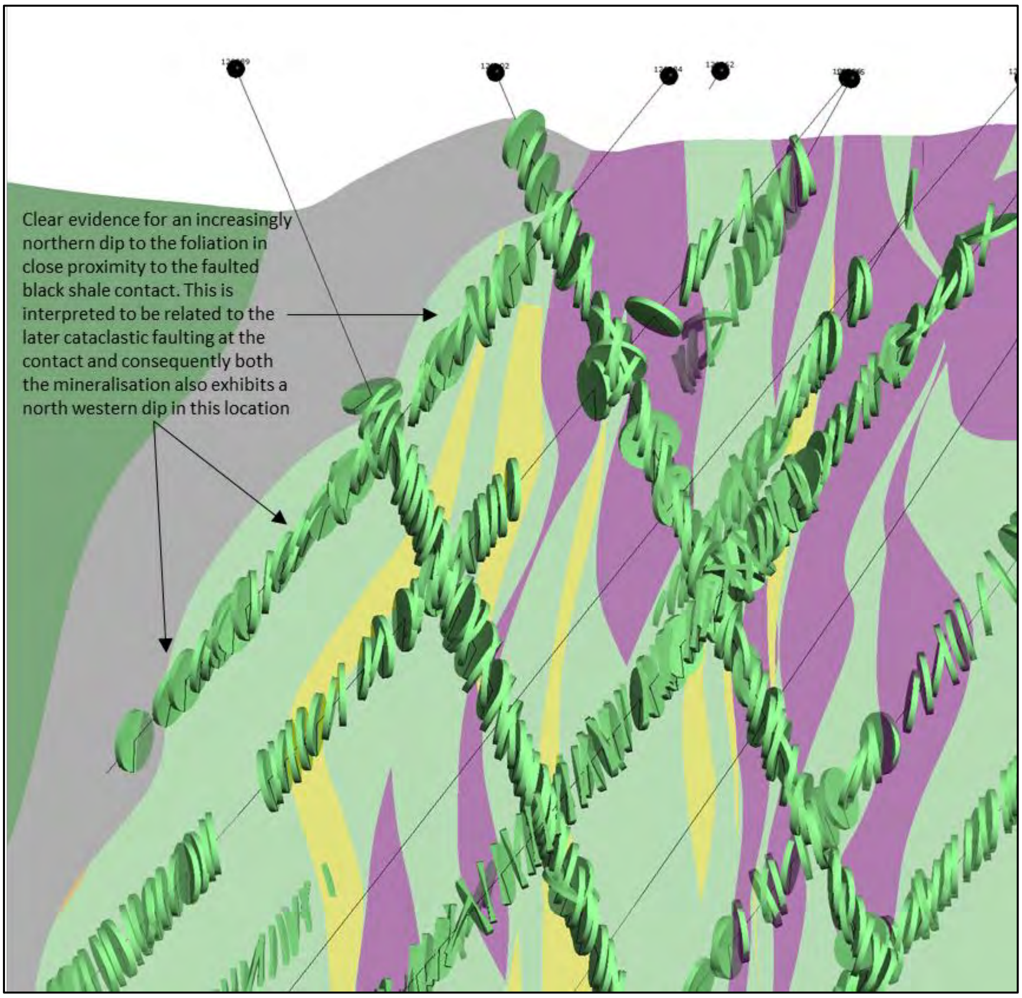
Source: Internal Rupert Resources Ltd database and interpretation, 2022

Figure 7.28 Cross Section from the Central Eastern Portion of the Deposit with Different Zones and Styles and Mineralisation Highlighted



Source: Internal Rupert Resources Ltd database and interpretation, 2022

Figure 7.29 Cross Section from the Central Eastern Portion of the Deposit Showing High Confidence Foliation Measurements as Discs on the Drill holes



Source: Internal Rupert Resources Ltd database and interpretation, 2022

Although vein arrays and stockwork zones are considered to be linked to the main gold phase, in that pyrite content increases, there is little consistent relationship between vein density, vein volume, and gold grade. There appears to be a further relationship between gold content, sulphide 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 (+ pyrite) in the matrix, very high grades (>10 g/t Au) are observed. These iron-rich fluids are interpreted to be D_3-related and may have been introduced during late-stage E-W strike slip, that also resulted in shearing textures within pre-existing quartz-siderite veins. 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 which 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 include chlorite, magnetite, syngenetic pyrite, graphite (related to carbonaceous shale) or the presence of a pre-existing reduced fluid. However, the spatial association of high-grade gold zones to apparently later D_3 structural domains suggests that a later gold-bearing fluid phase was also present.

Mineralisation remains open at depth, down-plunge in many zones and also up-plunge in several places.

Figure 7.30 Example of 'Felsic' Intercalations within the Central Komatiite Zone. Vibrant green fuchsitic wisps are locally developed at the unit boundaries, and are interpreted as selectively replaced, sheared siltstone.

Note: *Core size NQ (50.7 mm core width).
Source: Internal Rupert Resources Ltd database, 2022

Figure 7.31 Visible Gold Within Brecciated Carbonate Veining

Note: Hole 120102 at 224 m (assay 56.2 ppm Au). *Core size NQ (50.7 mm core width).
Source: Internal Rupert Resources Ltd database, 2022

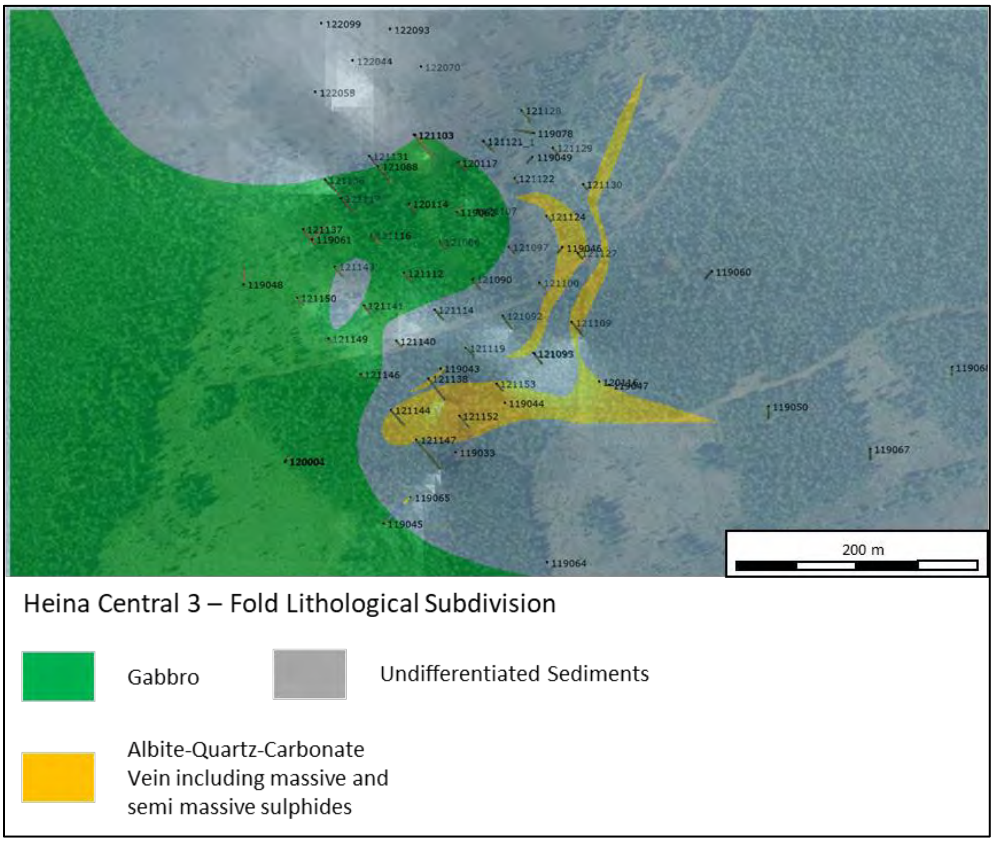
7.4 DEPOSIT GEOLOGY – HEINÄ CENTRAL

It should be noted that outcrop across most of the Heinälamminvuoma permit area and especially in the immediate vicinity of the Heinä Central deposit, is virtually non-existent. The deposit is a grassroots discovery, located under more than 10 m of transported glacial till cover and locally up to 40 m of glacial till.

Heinä Central is located within a relatively low strain lozenge-shaped block of Savukoski Group rocks bound by shear corridors to the north and south. The geology at its most basic level is fairly simple though the distribution of sulphides, the key control on gold mineralisation is less so. At the most basic level, a three-fold lithologic subdivision is constructed for the host rocks of Heinä Central deposit (Figure 7.32):

- Dark green-grey, chlorite altered, fine-grained gabbro which defines the outside of the plunging fold.
- Grey to black coloured, fine-grained sediment and volcano-sedimentary sequence composed of carbonaceous shales, volcanic tuffs, of variable mafic content, and other fine grained sediments. These form the interior of the plunging fold.
- Massive, white to light grey quartz-albite- carbonate-veins which occur only in the sedimentary sequence and in turn host fracture fill and massive sulphides. Massive and semi-massive sulphides are dominated by pyrrhotite (Po) with lesser pyrite (Py) and chalcopyrite (Cp). Where the sulphides are semi-massive in nature clasts are almost always of the quartz-carbonate-albite veins.

Figure 7.32 Basic Geological Model of the Heinä Central (Plan View) Showing the Key Lithologies Present According to the Simplified Three-fold Lithological Subdivision



Source: Internal Rupert Resources Ltd database and interpretation, 2022

In summary it is interpreted that during D_2 folding of the sequence, the pre-emplaced quartz-albite-carbonate vein brittlely deformed within the sedimentary sequence allowing for sulphides to fill fractures within the vein system. Near the hinge of the fold, where the vein was most strongly deformed and thus broken, the sulphides form a massive and semi-massive infill.

7.4.1 Rock Types

Dark green-grey, fine grained gabbro, commonly exhibiting plagioclase phenocrysts, is variably logged as a massive basalt or fine-grained gabbro. The distinctive dark green colour, a result of chlorite alteration, the characteristic irregular calcite veinlets and the magnetic nature of the unit are ubiquitous features. The lithology is very massive in nature with no foliation visible in hand specimen throughout much of the unit and no mineralisation of potential economic significance occurs within this unit.

Figure 7.33 Example of Barren Fine-grained Gabbroic Rocks with Characteristic Irregular Calcite Veining

Note: Hole 120117 62-64 m. *Core size NQ (50.7 mm core width).
Source: Internal Rupert Resources Ltd database, 2022

In contrast to fine-grained gabbro the sedimentary sequence at Heinä Central is very heterogeneous with composition varying between volcanic and sedimentary end members. At the most volcanic end of this continuum are minor, laterally discontinuous basalts, the distribution of which is very sporadic. More commonly, volcaniclastic and sedimentary dominated sequences appear intercalated with local compositional variations, both laterally and over time, interpreted to be responsible for this variation. Attempts to correlate horizons within the sedimentary sequence, both using logged lithology and lithogeochemical data has to date proven difficult on a deposit scale.

At the sedimentary end of this continuum are carbonaceous shales with a strong planar foliation, chaotically folded in places, and significant quantities of syngenetic disseminated and veined pyrite. Graphite content is overall low to moderate with graphite preferentially occurring on fracture surfaces however, locally the content can be more significant. These black shales are commonly sericite- and albite-altered in proximity to the albite veins.

Figure 7.34 Examples of the Variability within the Sedimentary / Volcano-sedimentary Sequence with, from top to bottom, Carbonaceous Shale, Mafic Tuff, Albite-altered Meta-siltstone (non-carbonaceous shale)

Note: Carbonaceous shale example - Hole 121122 at 210 m.

Note: Volcaniclastic example - Hole 121090 at 46 m (wet photo).

Note: Albite altered meta-siltstone – Hole 121117 at 187 m. *Core size NQ (50.7 mm core width) for all images.

Source: Internal Rupert Resources Ltd database, 2022

The sedimentary package hosts significant quantities of albite-quartz-carbonate veins which in turn host the sulphide mineralisation. The veins are universally brecciated with a sulphide cement, however the extent of the brecciation and thus the quantity of the sulphide fill is variable (Figure 7.35).

Figure 7.35 Example of Massive Quartz-albite-carbonate Veining within the Sedimentary Package, with Sulphide Cementation

Note: Hole 120086 at 93 m.

Note: Hole 121131 at 132 m –The contact between the albitised sediments and the quartz-albite-carbonate vein can be clearly seen in the top run of core. Both Cp and Po cements are present. *Core size NQ (50.7 mm core width) for both images.
Source: Internal Rupert Resources Ltd database, 2022

Semi-massive and massive sulphides occur within the previously described veins through the deposit, with maximum widths in the several 10's meters on the sections where the northern and southern mineralisation trends coalesce. Massive sulphides are always pyrrhotite dominated with chalcopyrite occurring preferentially towards the contacts, within the quartz-carbonate-albite veins and commonly as sulphide-cemented, brecciated veins rather than massive sulphides.

Figure 7.36 Example of Semi-massive Sulphides with Quartz-albite-carbonate Vein Material Clearly Present Between Short Sections of Massive Sulphides and as Clasts Within Sulphide Cement

Note: Hole 121095 at 60 m – See next photo for white box in more detail. *Core size NQ (50.7 mm core width).

Note: More detailed look at the breccia textures with sulphide cement with the semi-massive sulphides. *Core size NQ (50.7 mm core width).
Source: Internal Rupert Resources Ltd database, 2022

Figure 7.37 Example of Late-stage Chalcopyrite Within the Massive Sulphides

Note: Hole 121131 at 132 m. *Core size NQ (50.7 mm core width).
Source: Internal Rupert Resources Ltd database, 2022

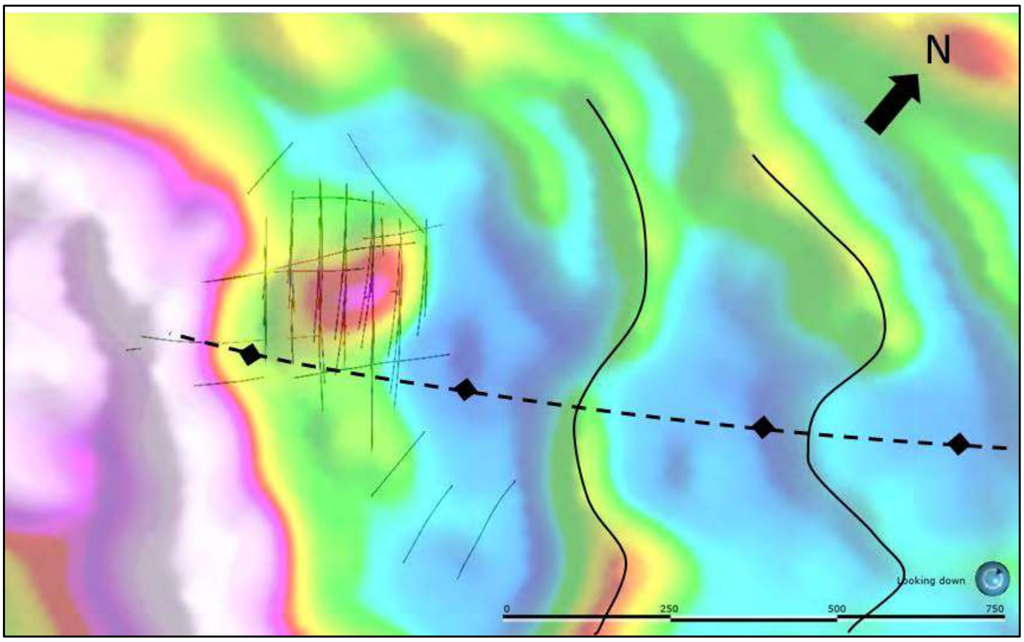
Figure 7.38 Example of Chalcopyrite Filling Narrow, Stringer Like Cracks in Brecciated Vein and Possible Veinlets Within the Quartz-albite-carbonate Brecciated Vein. Where Larger areas of Sulphide Cement Occur Between the Vein Clasts the Dominant Cement is Pyrrhotite

Note: Hole 120116 at 241 m. *Core size NQ (50.7 mm core width).
Source: Internal Rupert Resources Ltd database, 2022

7.4.2 Structure

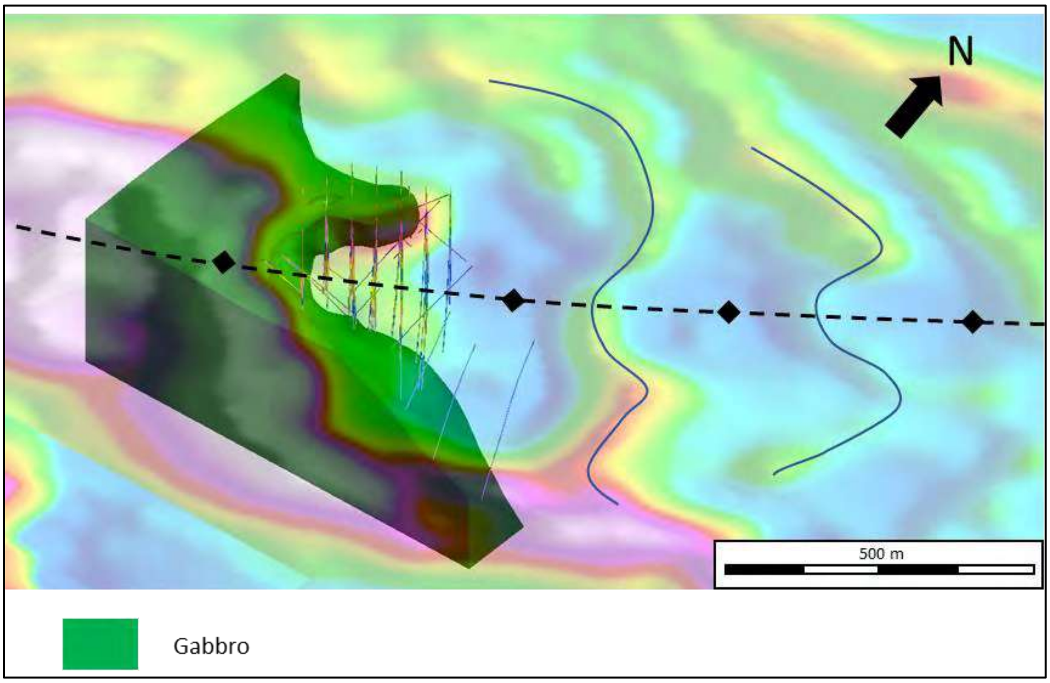
Heinä Central occurs within the relatively low strain Savukoski Lozenge, and this is reflected in the structure of the deposit. The gabbro-sediment contact defines a NW plunging synclinal fold (Figure 7.39) the that mirrors the NE-trending D_2 fold hinges discussed previously in the context of the wider Savukoski Lozenge (section 7.1) and evident in the magnetic imagery to the east of the Heinä Central deposit (Figure 7.40).

Figure 7.39 TMI, RTP Magnetic Map in Plan View Looking NW Showing the Fold Pattern Visible as Crenulations in the Magnetic Units to East of the Heinä Central Deposit. The Trending Hinge to One of these Major Crenulations Extends Through the Heinä Central Deposit



Source: Internal Rupert Resources Ltd database and interpretation, 2022

Figure 7.40 TMI, RTP Magnetic Map Looking Downdip at 40° Towards 320° with the Modelled Gabbro (from Drilling) Shown with the Interpreted Fold Hinge from the Magnetics



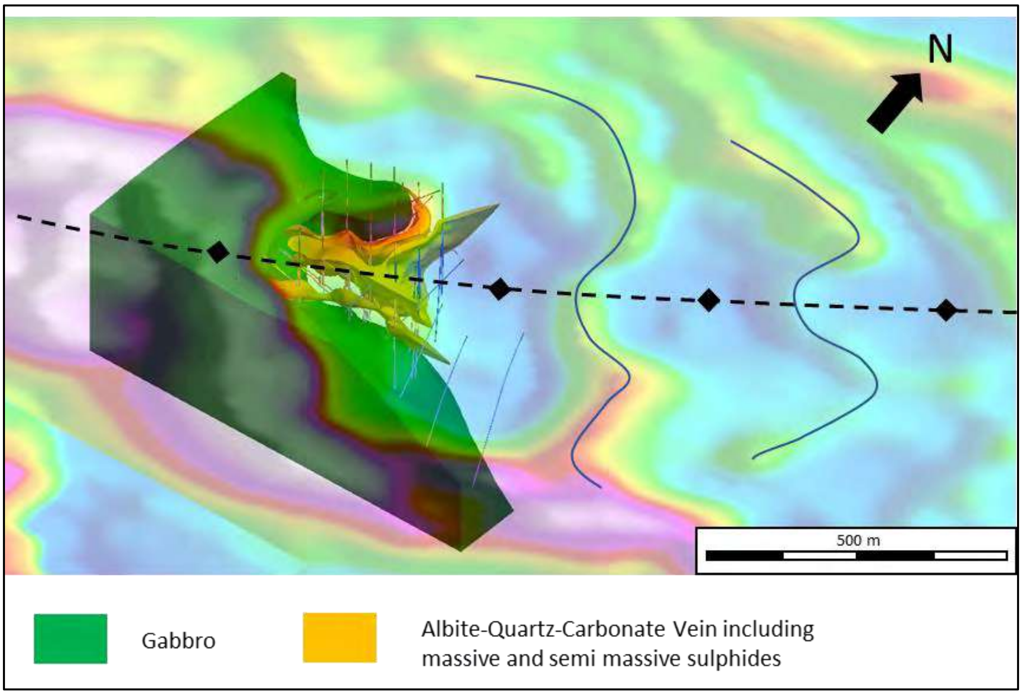
Source: Internal Rupert Resources Ltd database and interpretation, 2022

A similar fold geometry is recorded in the brecciated veins, massive and semi-massive sulphides. For simplicity, and in line with the geological interpretation that the sulphides form as an infill to brittle deformation within the vein, both the vein and sulphides are modelled together. Modelling reveals two parallel zones of veining and mineralisation with the greatest thickness and continuity occurring in the innermost zone.

The veins and sulphide form a shape consistent with the same synclinal fold, axial plane dipping approximately 40° towards 330° with the hinge plunging at a similar angle, albeit rotated slightly clockwise of this to approximately 40° towards 335°. The interlimb angle is typically around 70° but in central portion of the deposit limbs are sub-parallel whilst to the east there are suggestions that the fold structure relaxes and the interlimb angle increases (Figure 7.41).

The sulphides rarely exhibit a strong foliation which is consistent with infill and cementation of the breccia post D_2 or very late during D_2. Where massive sulphides occur on the outer edges of the quartz-albite-carbonate veins and thus in contact with the strong foliated sediments, the sulphide is seen to cross-cut to foliation (Figure 7.42).

Figure 7.41 TMI, RTP Magnetic Map Looking Downdip at 40° Towards 320° with the Modelled Gabbro and Vein +/- Sulphide Mineralisation Shown with the Interpreted Fold Hinge from the Magnetics

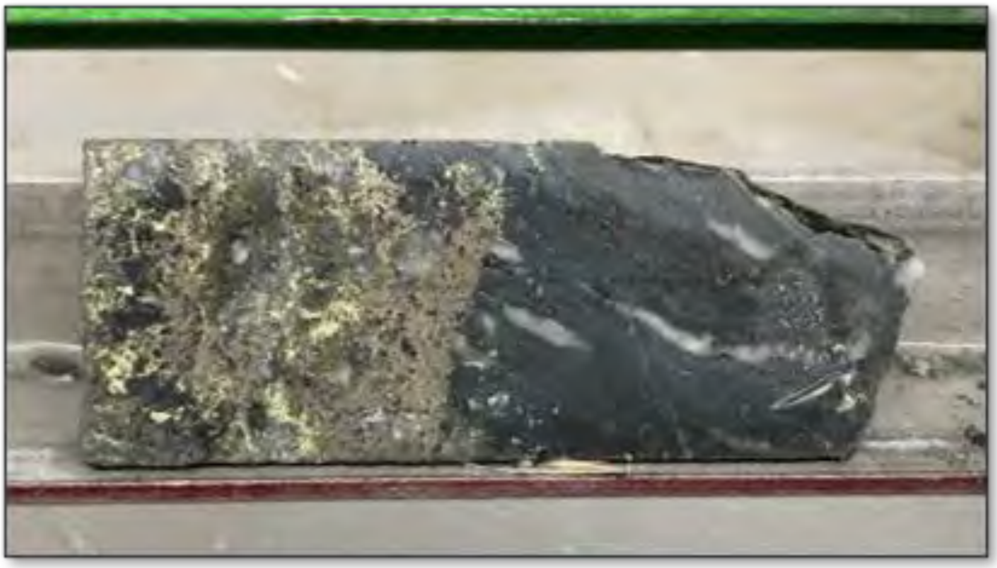


Source: Internal Rupert Resources Ltd database and interpretation, 2022

Neither the gabbro nor the mineralisation forms a simple fold shape. An additional hinge running parallel to the drilling is tentatively interpreted, as a result of NE-SW compression, creating an embayment like structure in the northern, outer gabbro and mineralisation. Although evidence for this second fold hinge orientation is limited at Heinä Central, the interpretation is consistent with observations at Ikkari Deposit only 1.5 km to the south (Selley, 2021).

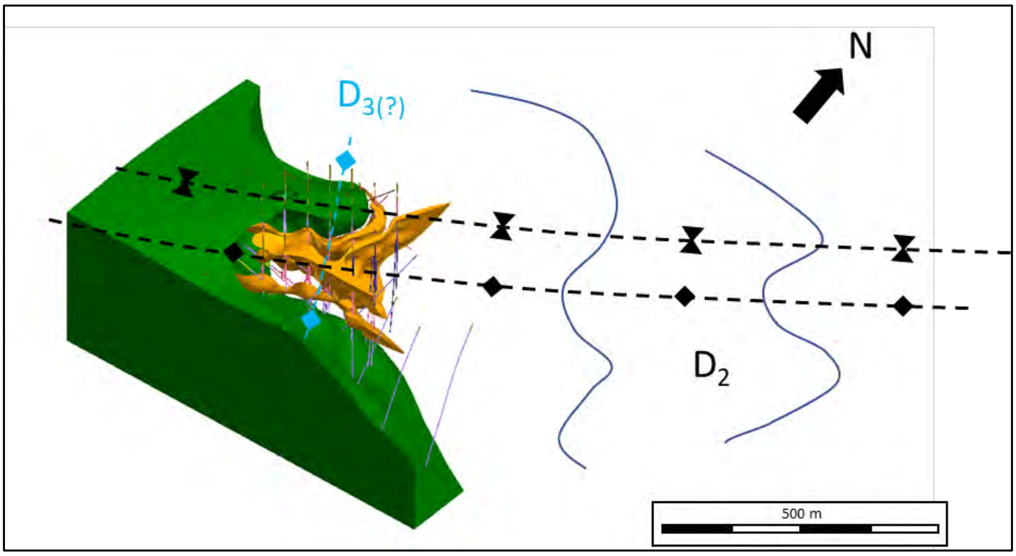
Additionally, the corresponding anticline to the host syncline, potentially occurs at the north-eastern edge of the current drilling and can be further inferred from the corresponding magnetic signatures, both immediately at Heinä Central and within corresponding magnetic units to the east. The plunge of the antiformal hinge may be rotated anticlockwise from that of the syncline plunging at a similar angle towards 310°; this is conformable with a minor, nominally D_3, axial plane trending NW-SE (Figure 7.43). Resolving the potential folded continuation of the Heinä Central deposit is one of the key objectives of current exploration at the deposit.

Figure 7.42 Massive Sulphides Cross-Cutting Plan Foliation in Tuffaceous Sediments



Note: *Core size NQ (50.7 mm core width).
Source: Internal Rupert Resources Ltd database, 2022

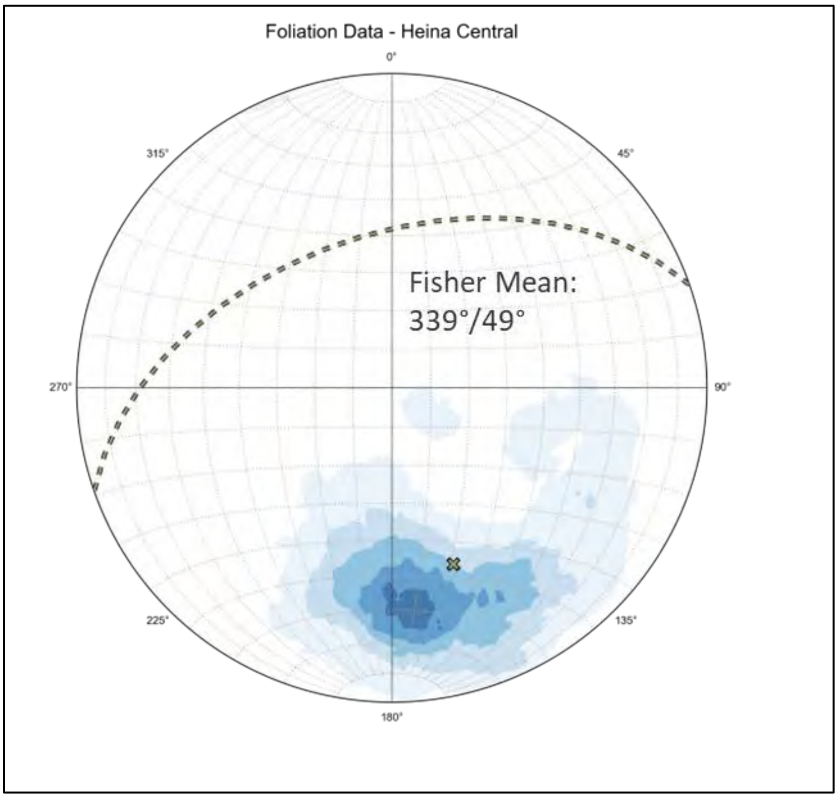
Figure 7.43 Tentatively Interpreted D_3 Rotation Around the NW-SE Trending Axial Plane



Source: Internal Rupert Resources Ltd database and interpretation, 2022

Foliation data captured by Rupert Resources from orientated drill core is consistent with this structural interpretation and the average foliation measurement 45/333 (Figure 7.44) is consistent with the axial plane of the modelled fold, whilst the range of foliation measurements is also consistent with, albeit insufficient, proof of the tentatively interpreted D_3 rotation around the NW-SE trending axial plane (Figure 7.44)

Figure 7.44 Contour Plot of Poles to the Foliation Data at Heinä Central with the Fisher Mean Plane Shown – only High Confidence Measurements within the Resource Area are PlottedHhere



Source: Internal Rupert Resources Ltd database, 2022

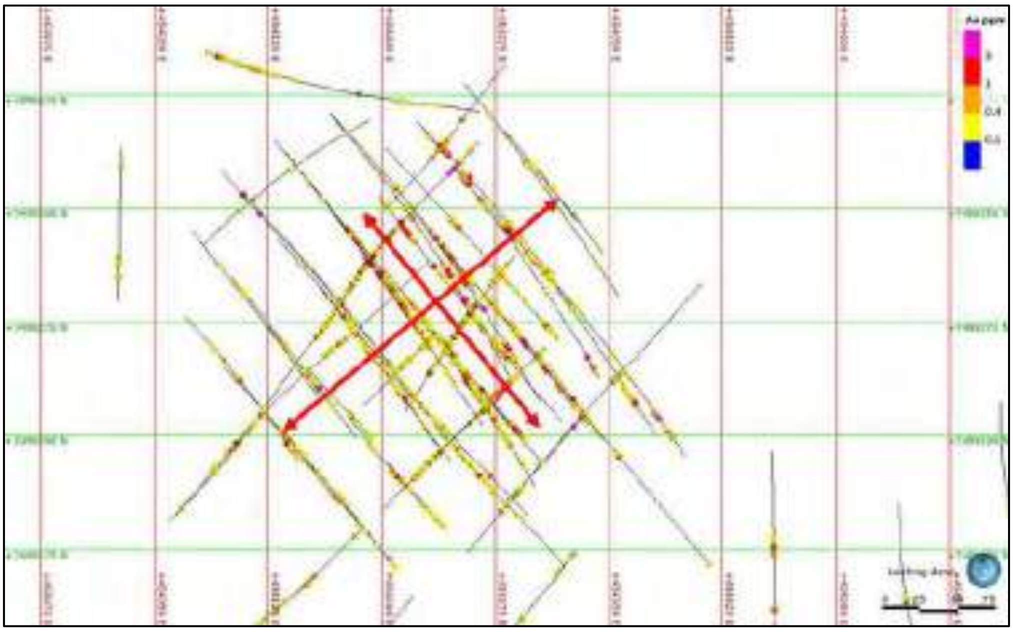
7.4.3 MINERALISATION

Gold-bearing sulphide mineralisation at Heinä Central does not correspond well to any typical universally recognised deposit type, although deposits of its type are not uncommon within greenstone belts. Whilst there are no current examples of Mineral Resource Estimates reported in the CLGB for the style of mineralisation encountered at Heinä Central, the mineralisation does share characteristics with gold-copper occurrences at Saatapoora (e.g. Korvuo 1997) and Kuutuvuoma that occur to the west of Heinä Central, along similar structural trends. Lithological, structural and geochemical relationships indicate a hydrothermal source of the gold mineralisation, possibly introduced in more than once phase of fluid event.

Modelling of the mineralisation, using the 12,403 m of drilling available, shows the deposit to have a mineralised footprint of up to 250 m (NE-SW) by 200 m

(NW-SE) extending to a maximum of 240 m below surface (Figure 7.45). The deposit remains demonstrably open at depth and along strike.

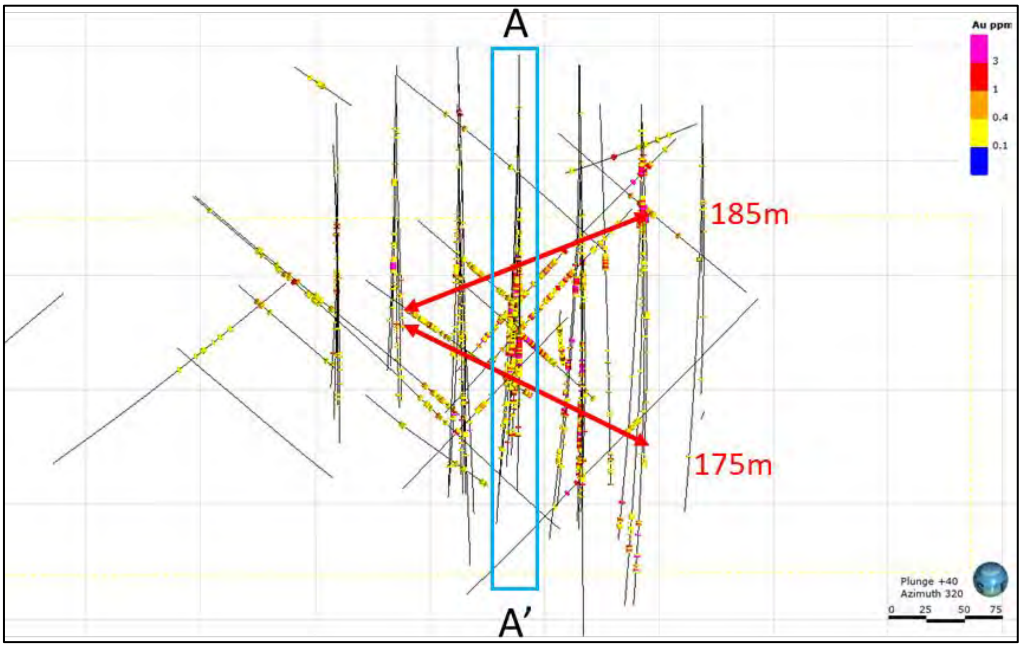
Figure 7.45 Currently Defined Limits of Heinä Central Mineralisation with Red Arrows Demonstrating the Location of the Dimensions (Plan View)



Scale bar: 75 m, Grid: 75 m
Source: Internal Rupert Resources Ltd database, 2022

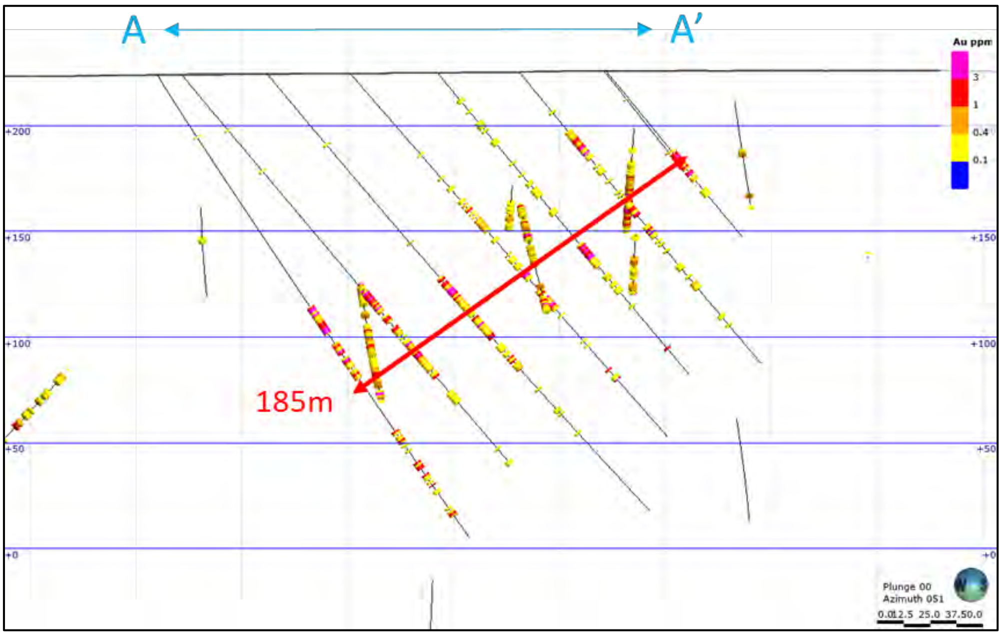
Whilst these plan view dimensions are indicative of the mineralisation footprint projected to surface they do not adequately describe the extent of the mineralisation in relation to its morphology. Mineralisation is best described as two converging trends of mineralisation which both dip at approximately 40° to the north and northwest. The converging trends can be traced for 185 m and 175 m respectively and both have been traced 185 m down-dip from the bedrock surface, which is the extent of current drilling (Figure 7.46).

Figure 7.46 Currently Defined Limits of Heinä Central Mineralisation Looking Down-plunge (40° towards 320°) with Red Arrows the Orientation of the Measurements Given



Scale bar: 75 m, Grid: 75 m
Note: The blue box A à A’ is shown in cross section below.
Source: Internal Rupert Resources Ltd database, 2022

Looking Northeast (towards 050°)



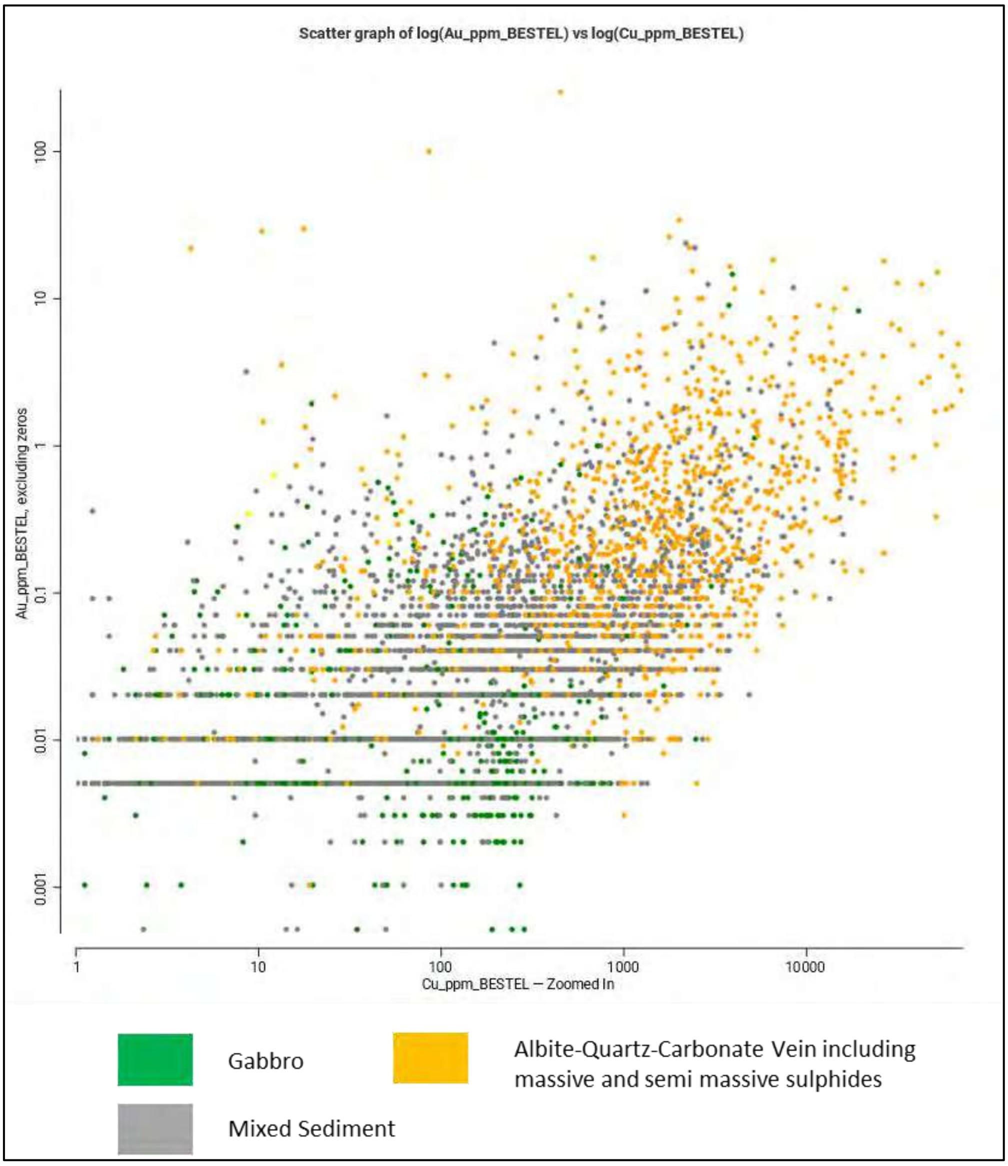
Scale bar: 50 m, Grid: 50 m
Source: Internal Rupert Resources Ltd database, 2022

Mineralisation at Heinä Central occurs as a sulphide-rich fracture-fill and cement to a brecciated quartz-albite-carbonate vein and also as veinlets in immediately adjacent albite-altered sediments. Sulphide content ranges from a few percent up to greater than 90% in massive sulphides. Gold and copper mineralisation are however poorly correlated with the sulphide mineralisation (Figure 7.49) which is dominated by pyrrhotite and to a lesser extent pyrite and chalcopyrite.

Syngenetic sulphide occurs throughout much of the sedimentary package as disseminated pyrrhotite in the foliation and is particularly prevalent in more carbonaceous portions where >5% sulphide is not uncommon. This syngenetic pyrrhotite is not related to either copper or gold mineralisation at Heinä Central though it likely contributes as a sulphide source for subsequent mineralisation phases.

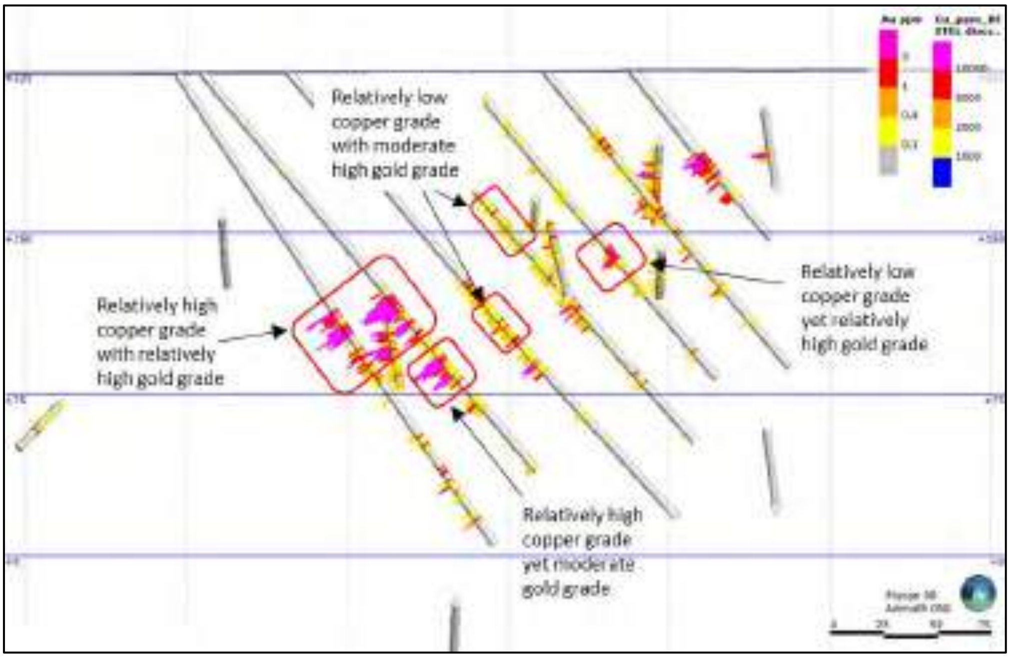
Gold is correlated with chalcopyrite mineralisation (Figure 7.47) and in hand specimen chalcopyrite content is frequently a good indicator of gold mineralisation. However, the correlation is not absolute and a minor separation between the highest grade gold and copper mineralisation (Figure 7.48) is observed. In addition, some gold occurs completely independently of the pyrrhotite-chalcopyrite sulphide assemblage and is more closely related to narrow sericite-silica alteration zones within predominantly sedimentary units, but this is a relatively minor contribution to the overall gold distribution.

Figure 7.47 Log-Log Scatter Plot of Copper v Gold Assays at Heinä Central Colours by Lithology According to the Most Basic Three-lithology Subdivision – Correlation Between the Two is Clearly Evident Though not Strong



Source: Internal Rupert Resources Ltd database, 2022

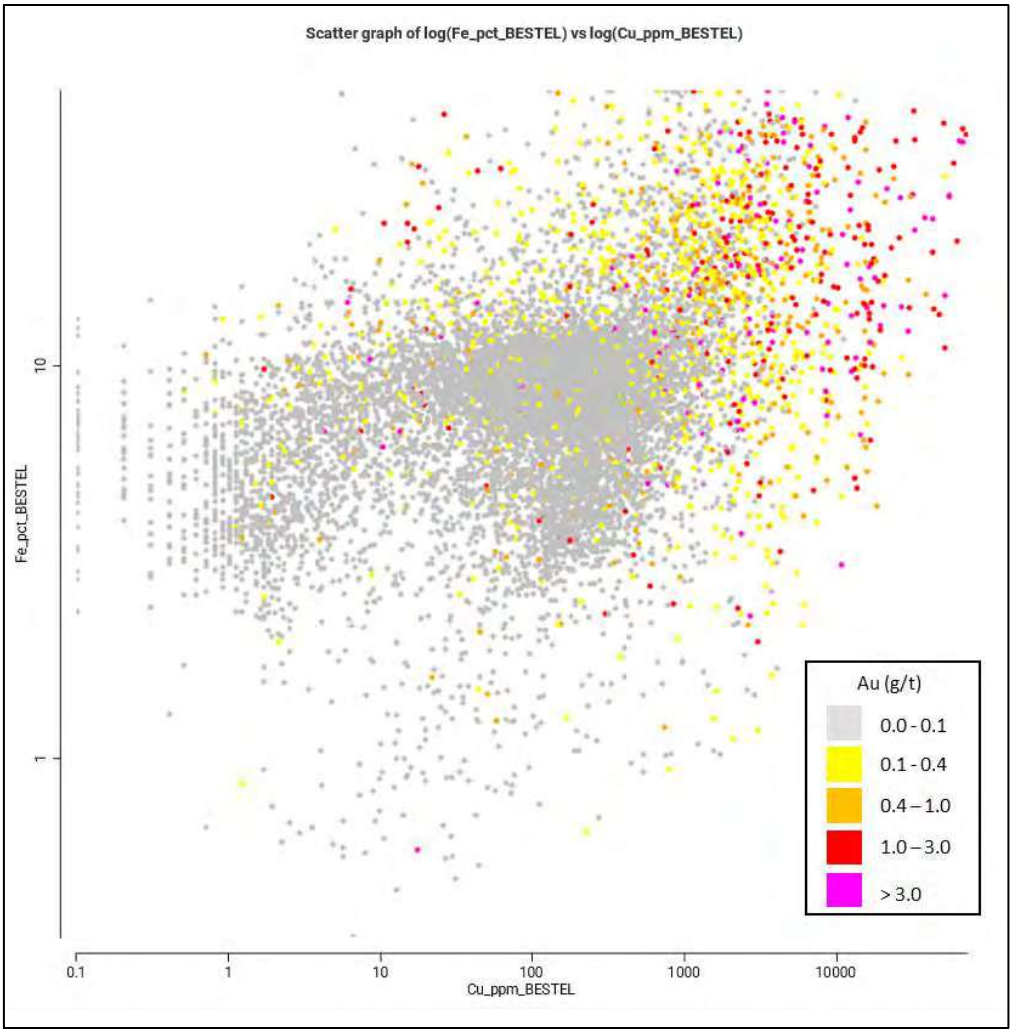
Figure 7.48 Two Cross sections within the Heinä Central Deposit Comparing the Distribution of Gold and Copper Mineralisation. Gold is Shown on the Drill Hole Cylinders whilst Copper is Plotted as a Graph to the Left-hand Side



Source: Internal Rupert Resources Ltd database, 2022

As noted previously, correlation between copper and gold mineralisation with respect to iron sulphides is poor, thus their spatial relationship is unclear. Whilst there an observable increase in copper content that corresponds to iron content above 10% (roughly implying sulphide mineralisation above that that can be attributed to syngenetic sulphide) higher copper grades, above 0.1% and higher gold grades, above 0.1 g/t, do not correlate with increasing iron content above this threshold. (Figure 7.49)

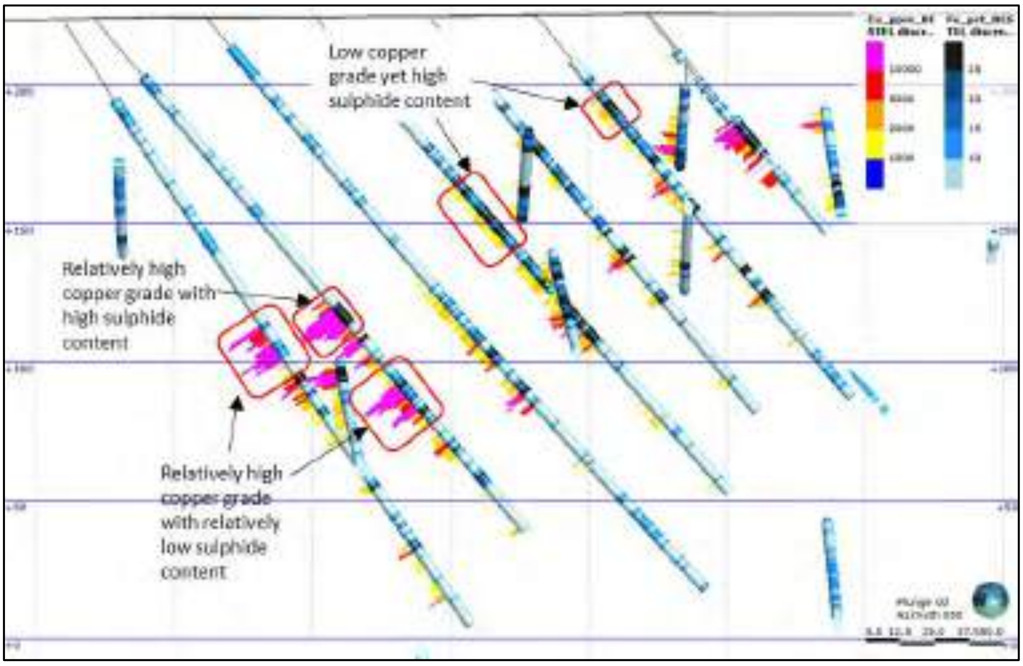
Figure 7.49 Log-Log Scatter Plot of Copper v Iron Assays at Heinä Central Coloured by Gold Grade



Note: Iron is plotted here instead of sulphur to avoid issues with the upper detection limit of sulphur.
Source: Internal Rupert Resources Ltd database, 2022

Observed in core, the copper and gold mineralisation is clearly spatially linked to the massive sulphide mineralisation, however, the precise relationship remains unclear. For example, in places, copper and gold mineralisation occurs in association with the highest concentrations of sulphide mineralisation, while elsewhere the copper and gold occur in relatively sulphide poor parts of the drillhole, although this is almost always adjacent to, or along strike of, more massive sulphide mineralisation (Figure 7.50). It is therefore inferred that the heterogeneity of copper and gold mineralisation locations is due to late-stage precipitation relative to the precipitation of pyrrhotite. Whether the copper sulphide and gold mineralisation fractionated from the same sulphide source as the pyrrhotite-dominated sulphides or whether it was introduced from a different, later source, is not currently known. The possibility of a later, cross-cutting gold-bearing fluid event is feasible.

Figure 7.50 Log-Log Scatter Plot of Copper v Iron Assays at Heinä Central Coloured by Gold Grade



Note: Iron is plotted here instead of sulphur to avoid issues with the upper detection limit of sulphur.
Source: Internal Rupert Resources Ltd database, 2022

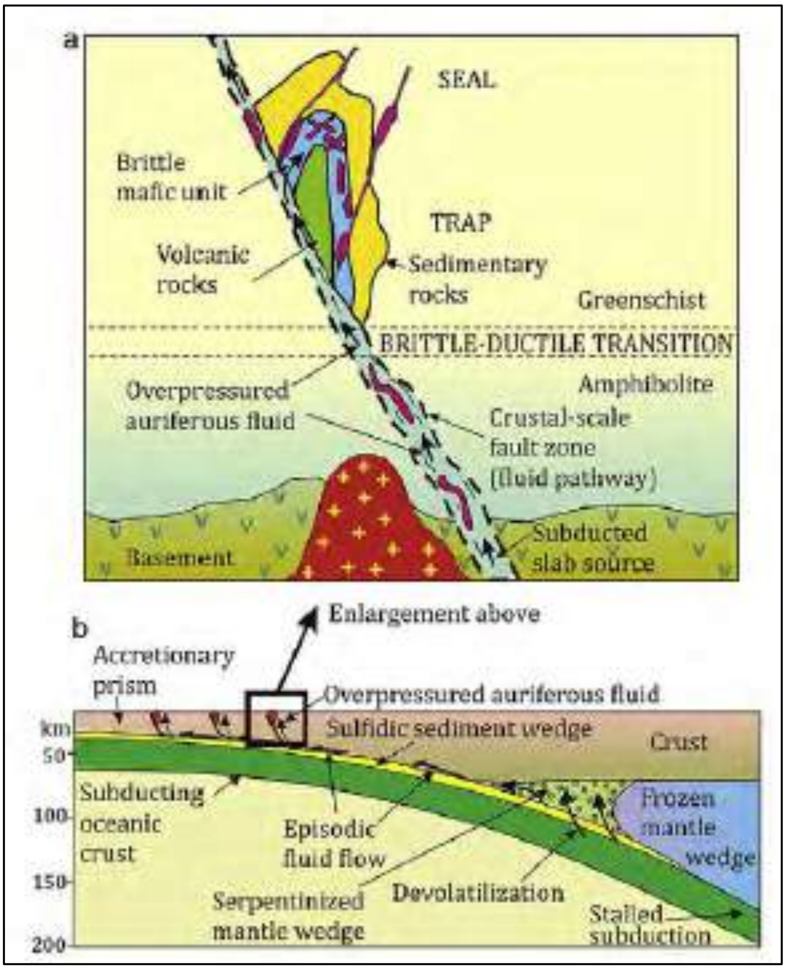
Mineralisation remains open at depth and along strike and further drilling will help resolve apparent strike variations, folds or off-sets and possible later structural controls on gold mineralisation.

8.0 DEPOSIT TYPES

All three deposits set out here are 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.
- 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 pages 733 - 734), Figure 8.1.

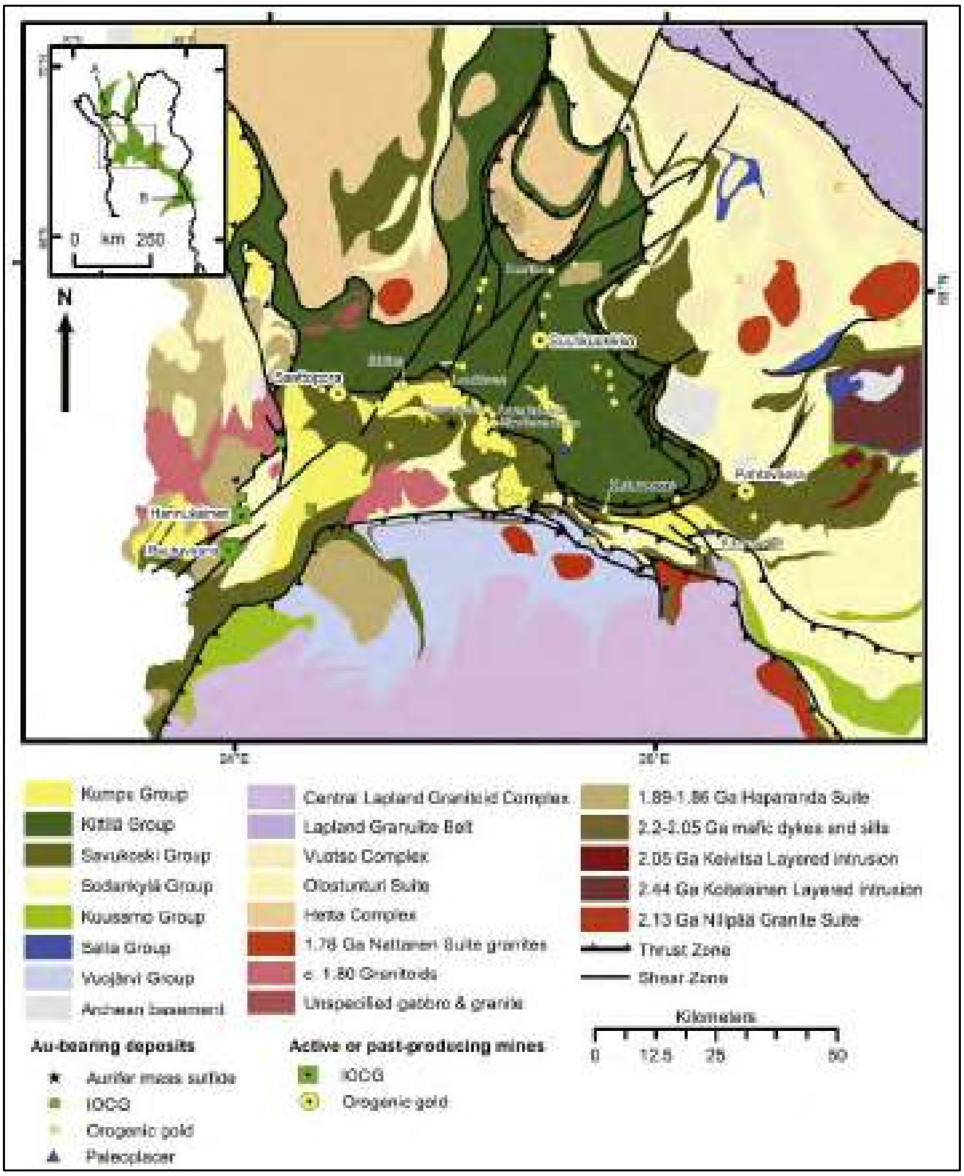
Figure 8.1 Schematic Representation of a Permissive Scenario for All Orogenic Gold Deposits



Source: Groves and Santosh, 2015

A number of orogenic gold deposits are believed to be hosted in the CLGB 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).

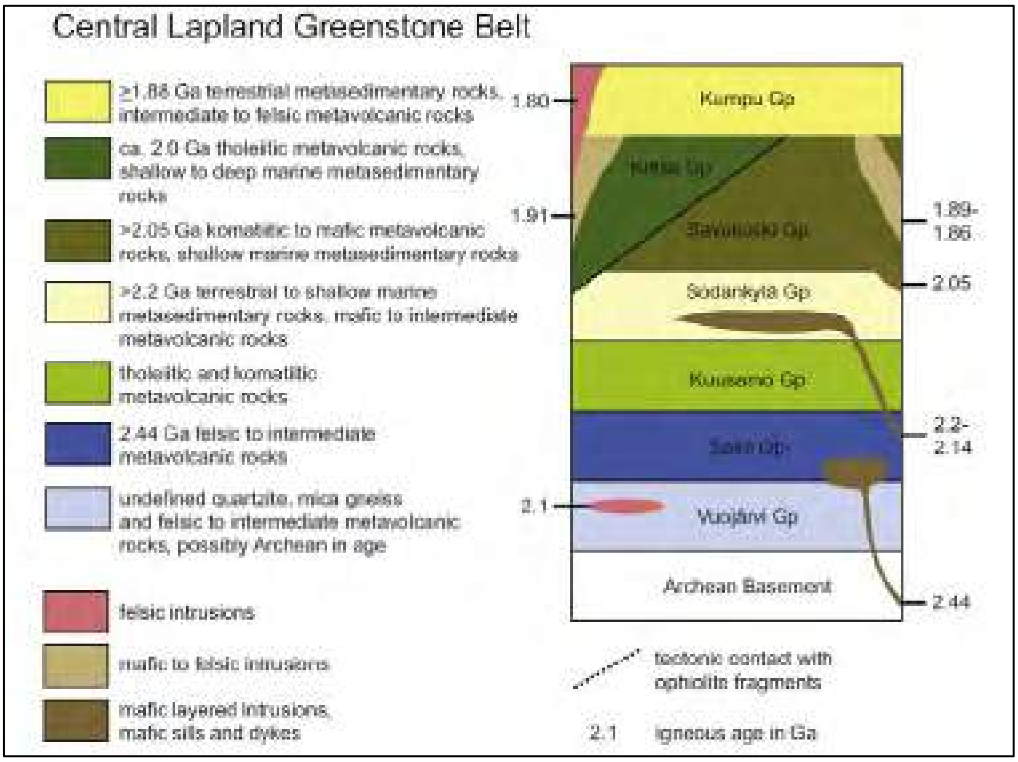
Figure 8.2 Geology and Gold Deposits of the CLGB



Source: Niiranen et al, 2015 (Modified from GTK Bedrock of Finland − DigiKP (October 15, 2013) and FINGOLD (October 15, 2013))

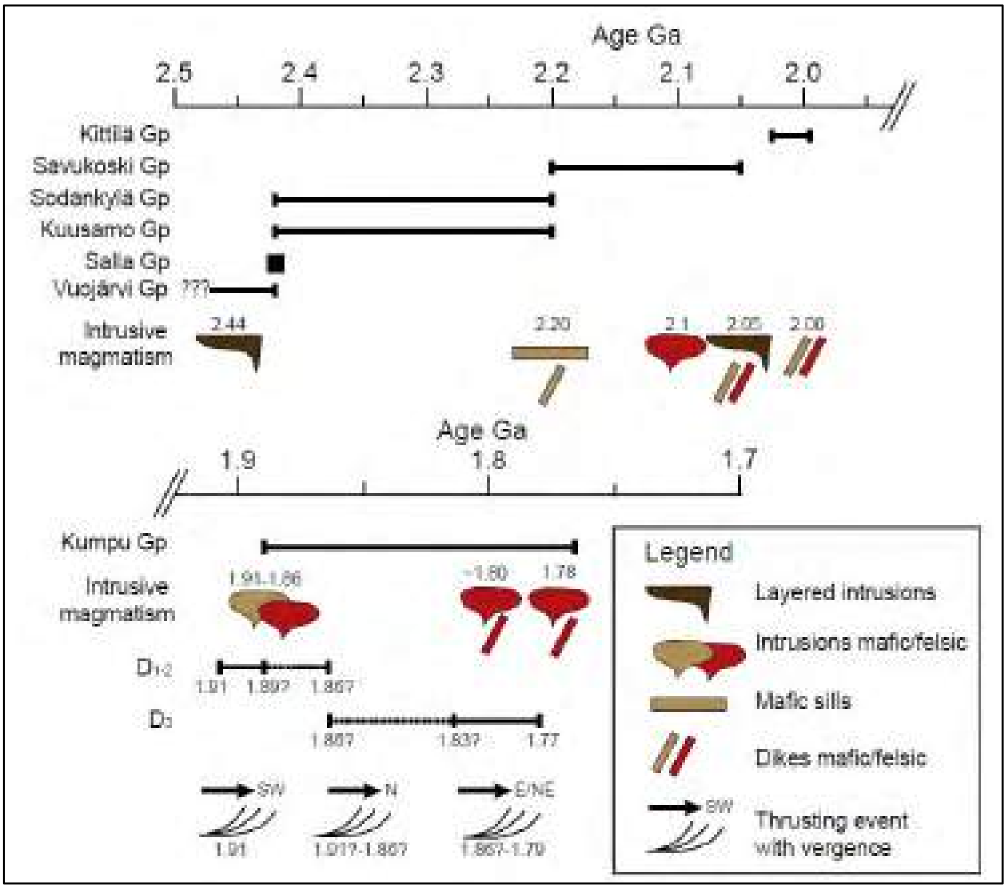
At a camp and district scale, known deposits cluster in proximity to transcrustal or other major deformation zones that are formed synchronously with the thickening of the crust during accretionary or collisional tectonic events. In most prospective districts, the deposits were formed at mid-crustal levels, as suggested by the dominant greenschist facies metamorphic assemblages of the host rocks (Nirranen et al, 2015). Within the Rupert 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 Rupert's licence, hosted within the Kittilä Group and the thrusted margin between the Kittilä and Savukoski Groups (see Figure 8.3). Timing relationships are displayed in Figure 8.4.

Figure 8.3 Stratigraphy and Main Igneous Events of the CLGB



Source: Niiranen et al, 2015 (Compiled after Hanski et al. (2001) and Bedrock of Finland − DigiKP (October 15, 2013))

Figure 8.4 A Schematic Sequence of the Lithostratigraphic Groups, Intrusive Stages and Deformation for the CLGB



Source: Wyche et al, 2015

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, tellurium, copper and tungsten. This is evident at Heinä Central and Ikkari, but much less so at Pahtavaara. In addition, spatial relationships between late-stage magnetite, K-feldspar and in places, relic plagioclase and gold mineralisation at Ikkari, particularly high-grade gold zones. It is hypothesised that a magmatic fluid from a deeper intrusive source may have been somewhat responsible for the localisation of gold mineralisation, especially high-grade gold, in favourable structural sites. Overprinting alteration events, diverse element and mineral associations at both Heinä Central and Ikkari indicate the potential for multifluid sources as a control on gold mineralisation.

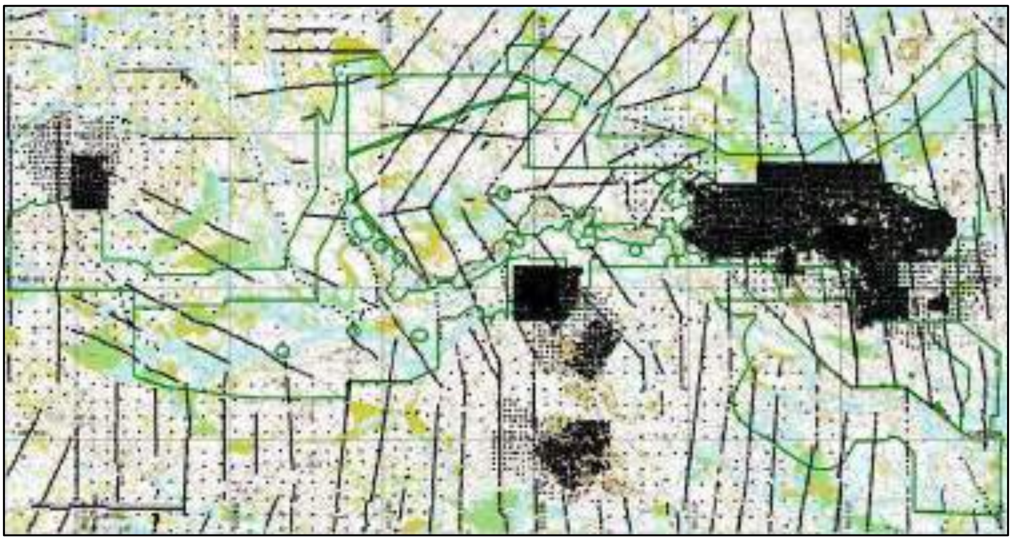
9.0 EXPLORATION

9.1 PREVIOUS EXPLORATION

In the 1970s, the GTK carried out regional geochemical mapping along lines in the Central Lapland Belt. The concentration of silicon (Si), Al, Fe, Mg, Ca, Na, K, Ti, V, Cr, Mn, Co, Ni, Cu, Zn, Pb, and Ag were analysed. The area of the Sattasvaara komatiite complex was characterised by elevated contents of Mg, Cr, Ni, and Co, and several local Cu anomalies appeared in the monotonous komatiitic environment indicating sulphide mineralisation. Additional geochemical till sampling was carried out using a grid of 50 m × 100 m in the winter of 1984 to 1985 to check the copper anomalies and gold was also analysed. A distinct gold anomaly was found in Pahtavaara and follow-up studies in 1985 including sampling of the bedrock surface by percussion drilling and excavated trenches, defined an altered zone containing visible gold between komatiitic lavas and tuffites (Pulkkinen et al,1986; Korkiakoski, 1992).

Historical till sampling comprises 426,737 samples compiled in regional programmes conducted by GTK and previous operators at Pahtavaara (see Figure 9.1). Some 38,298 samples were assayed for gold by a variety of analytical techniques and interpretation of the data is being undertaken. Very few samples were taken in the western parts of the Rupert Lapland exploration licences and the area that hosts Ikkari and Heinä Central deposits was completely unrepresented in historical sampling programmes.

Figure 9.1 Historical Base of Till Sampling



Grid: 5 km
Source: GTK & Internal Rupert Resources Ltd database, 2022

9.2 Geophysical Surveys by Previous Operators

The GTK flew airborne geophysics in the area in the 1970s and 1980s. The survey was originally flown with a low-level DC-3 system between 1973 and 1979 and was 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. These surveys included aeromagnetic surveys, electromagnetic (EM) surveys and radiometric surveys. More detailed survey methods conducted by GTK included slingram and ground magnetic surveys. In addition to these surveys, previous operators have undertaken local IP and magnetic surveys on several targets, including Lappland Goldminers' electromagnetic (VTEM) survey in 2010 on near-mine targets and SkyTEM electromagnetic and magnetic surveys in 2011.

9.3 Exploration Undertaken by Rupert Resources

9.3.1 Pahtavaara

Geophysics

At the Pahtavaara area, in 2016 an IP survey was conducted covering the mine site and the near-mine area totalling 27 line kilometres with 50 m line spacing (see Figure 9.2). During May 2018, Rupert Resources completed a programme of low-altitude magnetic surveying using remote-controlled drones, which was subsequently extended across the majority of the exploration licence package as it existed then.

Figure 9.2 2016 IP Survey

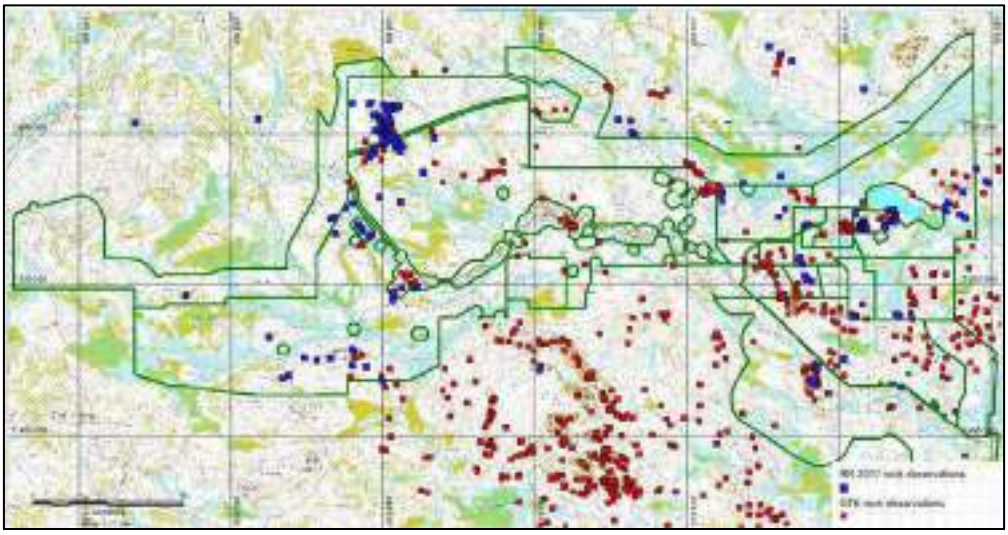
Source: Internal Rupert Resources Ltd database, 2022

GEOCHEMISTRY

Additionally, at Pahtavaara area, Rupert has completed a soil sampling programme during the 2017 field season. Soil samples include 950 ionic leach samples and 169 geochemical soil samples with multi-element assays, and 140 heavy mineral (till) samples, which were micro-panned using a Knelson concentrator and gold grains counted and classified according to grain morphology.

The bedrock mapping and boulder-hunting database contains 260 rock samples collected by Rupert Resources, as well as additional 57 field observations (Figure 9.3). There are 2,920 additional observations in the database from GTK boulder and outcrop mapping.

Figure 9.3 Boulder and Outcrop Observations at Pahtavaara



Grid: 5 km
Source: GTK & Internal Rupert Resources Ltd database, 2022

9.3.2 IKKARI AND HEINÄ CENTRAL

Focusing only on the work Rupert Resources has undertaken within the package of Rupert Lapland exploration licences, including Area 1 and the Ikkari and Heinä Central discoveries, the following exploration programmes have been completed.

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 9.4). In addition, a series of ground magnetic programmes were completed during 2020 across selected target areas in Area 1, including Ikkari and Heinä South. 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 permit area, at a 200 m spaced grid resolution (Figure 9.3), with 3,416 measurements taken.

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

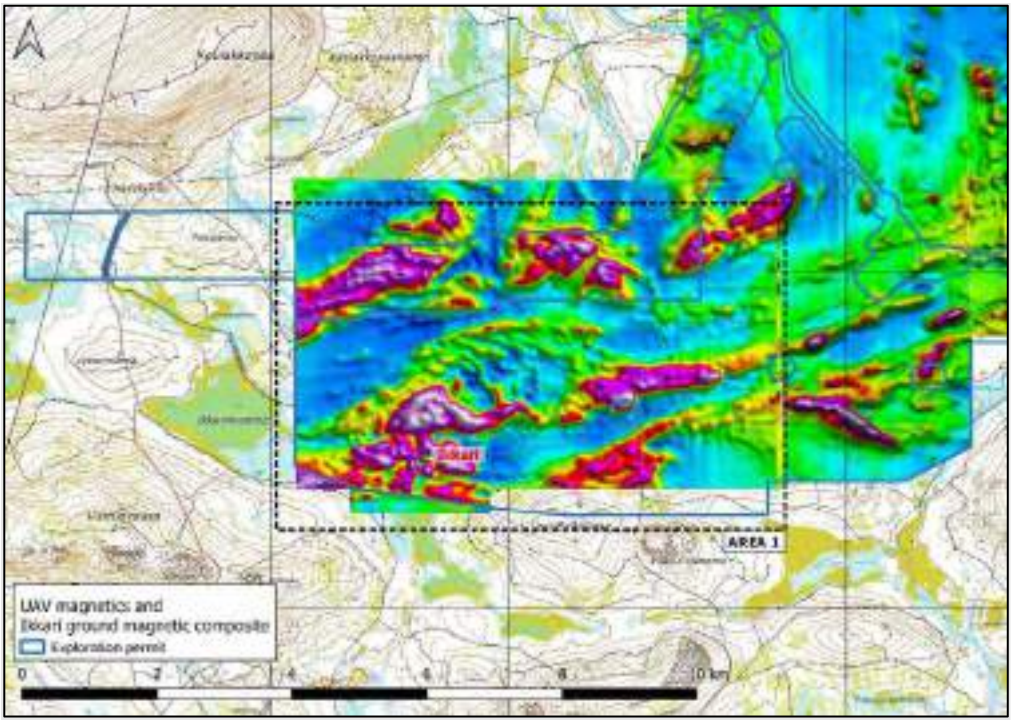
A series of ground IP pole-dipole programmes were completed across specific targets in Area 1 during 2020 (Figure 9.6), using a GDD 32cRx receiver, GDD Tx4 5 kilowatt (kW) transmitter, PbCl2 electrodes and stainless steel. Primary voltage was apparent resistivity and chargeability with 20 arithmetic 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.

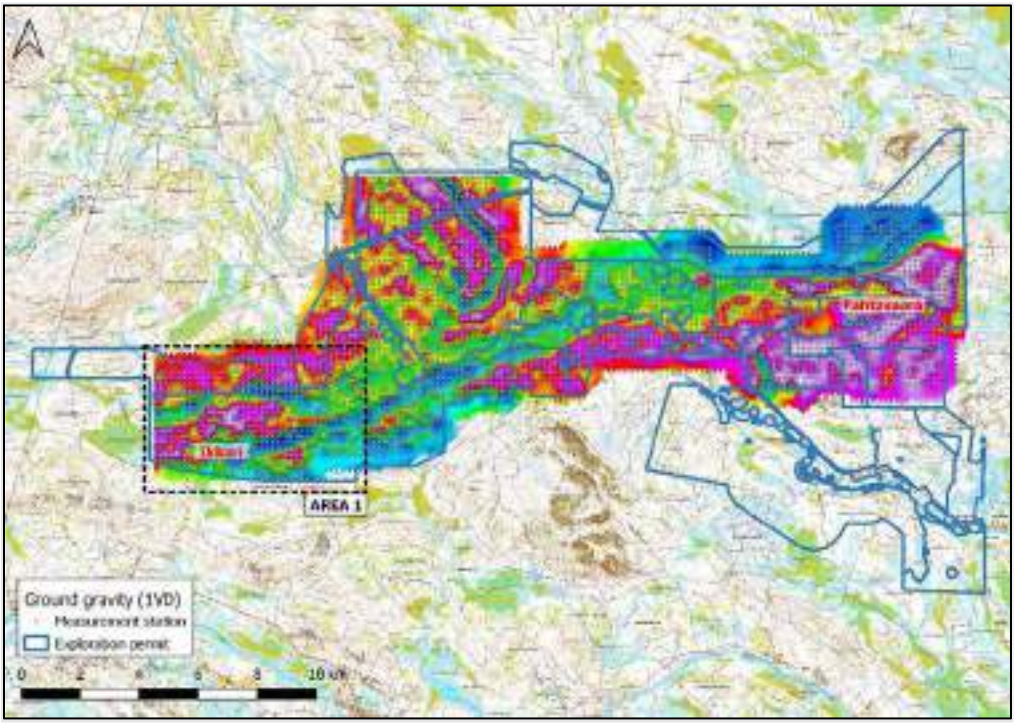
At Saitta, a 100 point programme was completed across 2 lines with electrode spacing at 25 m and 50 m.

Figure 9.4 Composite Magnetic Image of Rupert Lapland Exploration Permits, Showing Results from Drone Magnetic Survey and Ground Magnetics in Area 1



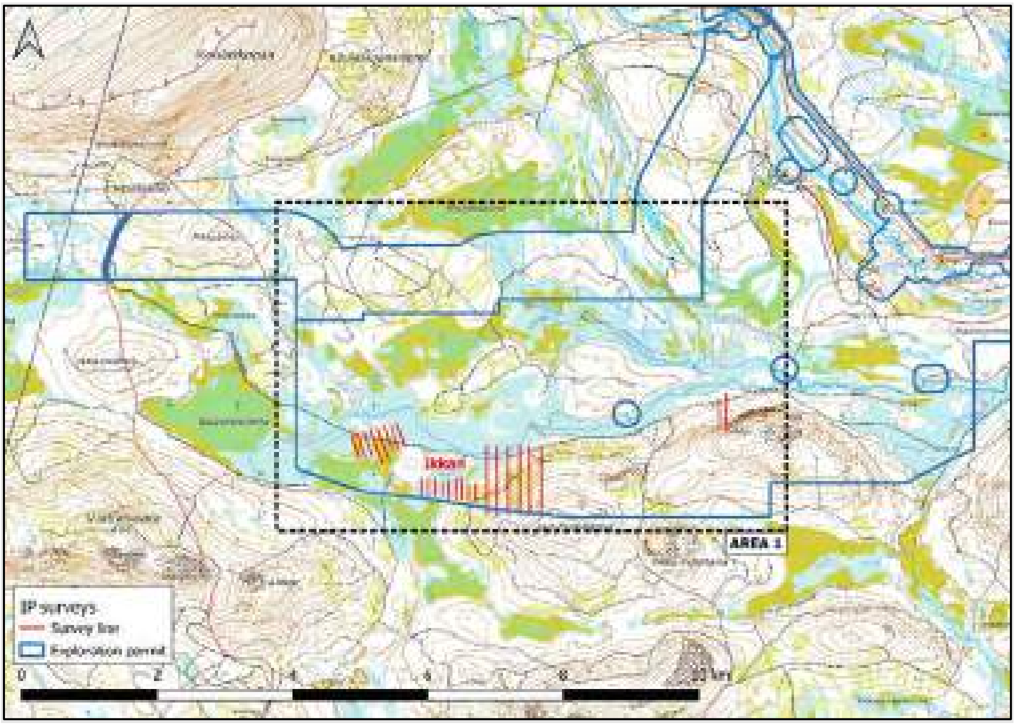
Source: Internal Rupert Resources Ltd database, 2022

Figure 9.5 Ground-gravity Programme with Points for Each Measurement Shown



Source: Internal Rupert Resources Ltd database, 2022

Figure 9.6 Location of Pole-dipole IP Lines at Target Areas Within Area 1



Source: Internal Rupert Resources Ltd database, 2022

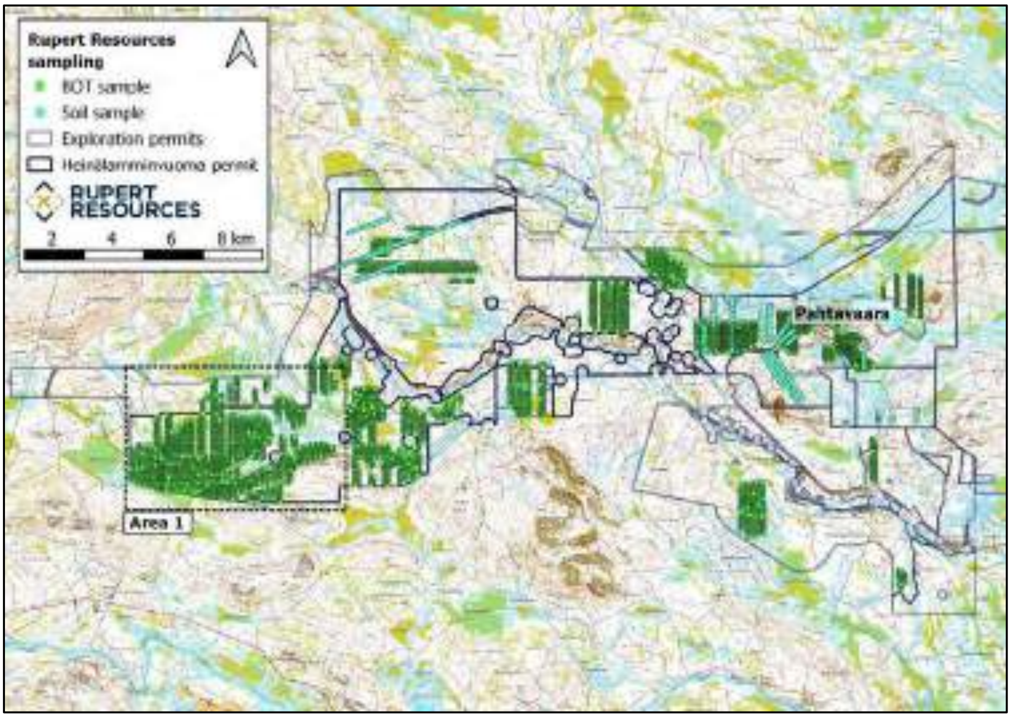
GEOCHEMISTRY

Initial work by Rupert Resources on the Rupert Lapland Project area, was focused on the area immediately surrounding the Pahtavaara mine (Figure 9.7). The bedrock mapping and boulder-hunting database of the Heinälamminvuoma permit area contains 1,365 rock observations including assayed samples collected by Rupert Resources across the project area. However, within Area 1, in which the Ikkari deposit is located, fewer rocks and boulders have been sampled by Rupert, largely due to the lack of outcrop and extensive bogs and thick till cover sequences. However, where accessible, surface geochemical sampling has been undertaken in these areas (Figure 9.8).

In early 2019, Rupert 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.

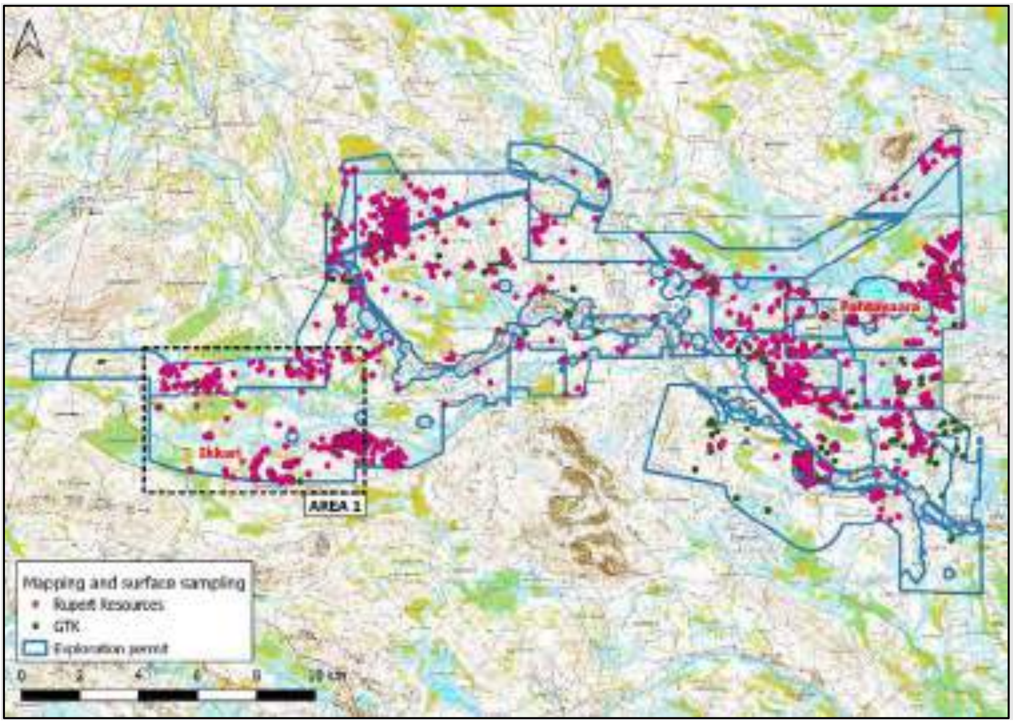
Follow up systematic drill testing of identified base of till gold anomalies was initiated with gold occurrences identified at several locations within the permit, including at Ikkari. Initial 'tram line' BoT traverses yielded a single point anomaly of 0.2 ppm Au and this was followed up with closer 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 9.7 Base of Till and Soil Sampling Locations Completed by Rupert Resources



Source: Internal Rupert Resources Ltd database, 2022

Figure 9.8 Boulder and Outcrop Observations Undertaken by Rupert Resources



Source: Internal Rupert Resources Ltd database, 2022

10.0 DRILLING

10.1 DRILLING BY PREVIOUS OPERATORS

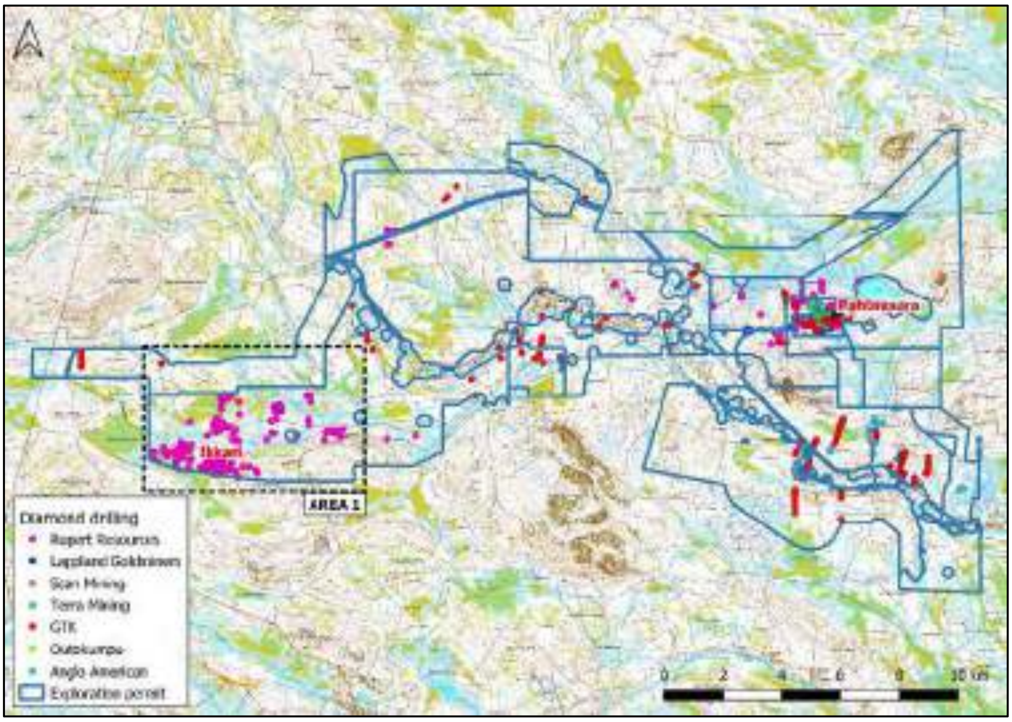
Considering initially the entire Rupert Lapland exploration licences, the vast majority of historic drilling has been carried out at the Pahtavaara Mine site, and near-mine areas with very little drilling completed elsewhere on the permits (Figure 10.1). No drilling has been undertaken by previous operators at or near the Ikkari and Heinä Central deposits.

At the Pahtavaara mine site, between 1986 and 1987, GTK drilled 114 diamond drill holes totalling 3,639 m. In 1989 GTK drilled a further 44 diamond drill holes totalling 4,372 m.

During 1992 to 2000, Terra Mining drilled 152 diamond drill holes totalling 14,853 m. Infill drilling was conducted using reverse circulation (RC) drilling and a total of 84 RC holes were drilled for 9,976 m.

Between 2003 to 2007, Scan Mining drilled 815 diamond drill holes (a total of 94,563 m) and 21 RC holes (a total of 1,116 m). Lappland Goldminers conducted an exploration programme from 2009 to 2014. Lappland Goldminers drilled 1,232 diamond drill holes (154,573 m) and over 6,600 sludge hole from underground for more than 124,000 m. The wider exploration programme consisted of diamond drilling, geophysical surveys and till sampling in areas surrounding the mine. Figure 10.1shows the location of diamond drill holes within the Pahtavaara Project licence area, subdivided by operator.

Figure 10.1 Diamond Drilling on the Rupert Lapland Licence Area



Source: Internal Rupert Resources Ltd database, 2022

10.2 PAHTAVAARA – DRILLING BY RUPERT RESOURCES

Rupert Resources began an exploration programme in June 2016 at the Pahtavaara deposit after acquiring the option on the property in March 2016. Up to end of May 2018, Rupert Resources drilled 377 diamond drill holes, totalling 52,937 m, and completed 1,010 m of channel and chip sampling underground. Drilling from 2016 to 2018 was undertaken by contractor MK Core Drilling. The core diameter for underground drilling was 40.7 mm (BQTK) and on surface 57.5 mm (WL76). The average drill hole length of the Rupert Resources drilling is 140 m. Downhole deviation surveys during 2016 were completed by the drilling contractor using a Deviflex downhole survey tool. Since mid-2017, all drilled holes were oriented using Reflex Act III core orientation tool. All collar locations were surveyed by Rupert surveyor using total station.

Further drilling by Rupert Resources, from April 2018 to July 2021, (undertaken by MK Core Drilling Oy, Arctic Drilling Company Oy Ltd and Comadev Oy), that is included in this resource update, comprises 242 diamond holes, totalling 21,618 m as well as 32 RC drill holes for 2,224 m (undertaken by Styrud Ltd). Underground drilling included in these statistics, comprises 125 holes for 7,458 m completed by MK Core Drilling Oy (2020) and Comadev Oy (2021).

10.3 IKKARI AND HEINÄ CENTRAL– DRILLING 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 summarised in Table 10.1.

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

Table 10.1 Drill Hole Summary for Drilling Undertaken by Rupert Resources on the Rupert Lapland Exploration Licence (Outside of the Pahtavaara Mine Area, up to end May 2022)

Prospect Name	DH Type	Holes	Metres	% of Total
Paskamaa (2019)	Diamond	18	1,605	1
Heinä South (2019)	Diamond	2	200	0
Heinä South (2020)		22	3,951	3
Heinä South (2021)		28	4,929	3
Heinä South (2022)		31	6,605	5
Heinä North (2019)	Diamond	10	1,612	1
Heinä North (2020)		2	245	0
Heinä Central (2019)	**Diamond**	**19**	**3,593**	**2**
Heinä Central + extn. (2020)		**10**	**2,416**	**2**
Heinä Central + extn. (2021)		**39**	**7,540**	**5**
Heinä Central (2022 to May)		**26**	**8,223**	**6**
Island North (2019)	Diamond	1	152	0
Island North (2020)		10	1,791	1
Island North (2021)		7	1,405	1
Saitta (2020)	Diamond	11	1,960	1
Saitta (2021)		2	534	0
Saitta (2022)		8	1,702	1
Ikkari (2020*)	**Diamond**	**62**	**20,320**	**14**
Ikkari (2021*)		**72**	**35,127**	**24**
Ikkari (2022 to end May)		**58**	**25,478**	**18**
Others (2019 – 2022)	Diamond	87	14,903	10
Total		**525**	**144,291**	**100%**

* Including extensions to drill holes and wedge holes.
** Including holes completed but not assayed, and therefore not included in the resource estimation (section 14.2).
Note: Any errors are 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 mineralised 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 mineralised system at Ikkari and further drill testing was prioritised, with some 62 holes for 20,320 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 mineral resource estimate in September 2021 infill drilling on 40 m sections with a 40 m spacing on section commence immediately synchronous with further step out drilling to the east, northwest and at depth. This updated resource estimate includes 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 mineral resource estimate.

At Heinä Central, an initial drill hole in April 2019, tested a base of till anomaly cluster at the margin of a magnetic anomaly. The hole returned weak gold and copper mineralisation over substantial downhole widths, hosted by sedimentary rocks and semi-massive to massive sulphides, the hole demonstrates strong sericite-silica alteration, shearing and brecciation along with substantial amounts of sulphides consisting of pyrrhotite and chalcopyrite.

Hole 120033 intersected 7 m grading 1.26 g/t Au and 0.18% Cu from 40 m.

Following these initial results, step out drilling was pursued along the magnetic anomaly boundary. These holes successfully intersected further mineralisation and indicated a potential strike length of 200 m.

Hole 119044 intersected 1.44 g/t Au and 0.5% Cu over 31 m. The hole was drilled in front of 119033, targeting near surface mineralisation of the same zone.

These holes confirmed the presence of mineralisation and drilling was continued. After this initial drilling programme it was interpreted from the drill core that the drilling orientation was parallel to the main foliation and the drilling did thus not intersect the mineralisation at a correct angle. During 2019 and beginning of 2020, 4,414 m was drilled in Heinä Central with this orientation.

This drilling confirmed the presence of a mineralised system at Heinä Central and further drill testing was planned. The target was tested in late 2020 with 806.7 m in 3 holes drilled with a preferred drilling orientation.

Drill hole 120114 intersected 19 m grading 2.52 g/t Au and 0.18% Cu from 127 m, drill hole 120116 intersected 15 m grading 1.23 g/t Au and 1.65% Cu.

With the success of this preferred drilling orientation a systematic drill programme was planned to delineate the extents and width of the mineralisation during the fall of 2021 and winter of 2022. Highlights included drill hole 121088 that intersected 14 m from 131 m grading 1.83 g/t Au and 2.92% Cu and 11 m from 166 m grading 1.67 g/t Au and 2.19% Cu and 10 m from 150 m grading 0.98 g/t Au and 0.85% Cu.

10.4 HOLE PLANNING AND SET-UP

The drilling process as undertaken by Rupert Resources and its contractors is consistent across all project areas.

Diamond Drilling at Ikkari from 2019 to 2022 was undertaken predominantly by contractor MK Core Drilling supplemented by Arctic Drilling Company (ADC), Kati and Comadev. The core diameter used was predominantly NQ2 (50.7 mm) and WL76 (57.5 mm). Diamond Drilling at Heinä Central from 2019 to 2022 and at Pahtavaara from 2016 to 2020 was undertaken predominantly by contractor MK Core Drilling. The core diameter used was predominantly NQ2 (50.7 mm) and WL76 (57.5 mm). From 2020 to 2021, underground drilling at Pahtavaara was undertaken by Comadev using NQ2 rods.

After drill holes are planned the Rupert staff surveyors get collar coordinates and also coordinates for the planned end of the hole, along with the dip and azimuth.

For drill holes collared at the ground surface, the surveyor uses Differential Global Positioning System (DGPS) to locate the collar location, orients the hole direction from the azimuth determined from the DGPS (according to direction between start and end coordinates).

The collar location is marked by 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 front in the planned drilling direction. An additional 'marker' peg is positioned in order 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).

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

10.5 Surveying and Orientations

For surface drill holes the actual collar position is picked up using DGPS total survey equipment. The drilling contractor does downhole surveys after the drill hole has been completed. Current survey tools are Reflexgyro and DeviFlex downhole survey instruments. The instruments 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 DeviSoft instrument software. The azimuth field is re-processed at all depths from the collar when the collar survey is available.

10.6 Pahtavaara - Dry Bulk Density Collection

10.6.1 Historical Bulk Density Measurements

Minor density test work was done by Scan Mining in 2005 at the Labtium laboratory in Sodankylä. The results are summarised in Table 10.2.

10.6.2 Lappland Goldminers' Bulk Density Measurements

Lappland Goldminers recognised that the densities at Pahtavaara varied between approximately 2.75 and 3.0 t per cubic metre, depending on lithology. However, no geological model detailed enough to permit the use of a variable density model had been developed by Lappland Goldminers. In the absence of this an average density of 2.9 t per cubic metre was used.

Table 10.2 Pahtavaara Gold Deposit, Density Test Work Completed by Scan Mining in 2005

Sample ID	Grade (Au/t)	Rock Type	Mass (g)	Volume (cm^3)	Density ($kg.m^3$)
05A4185	0.410	Bt-trem	7,416	2,477	2,994
05A4186	5.510	Bt-trem	6,925	2,328	2,975
05A4187	4.900	Dol vein	7,547	2,077	3,635
05A4188	0.325	tlc-bt with dol	7,538	2,165	3,482
05A4189	1.660	tlc-bt with dol	6,131	2,021	3,034
05A4213	0.300	tlc-bt+ minor qz	7,139	2,451	2,912
05A4214	0.800	tlc-bt schist	7,275	2,366	3,075
05A4215	0.940	tlc-bt schist	7,445	2,432	3,062
05A4216	0.970	qz vein rock	7,379	2,508	2,942
05A4217	0.900	qz vein rock	7,061	2,409	2,931

Key: Au/t = gold per tonne, cm^3 = cubic centimetre, g = gram, $kg.m^3$ = kilogram per cubic metre

10.6.3 Rupert Resources Bulk Density Measurements

The Pahtavaara bulk density database contains 8,466 measurements, of which 7,416 have been recorded by Rupert Resources since 2016. The most recent phase of drilling, from 2019 to 2021 comprises 5,250 additional density measurements.

Since late 2017, all diamond drill holes have been routinely measured for density. A 10 cm to 15 cm piece of core from every second meter is weighed first in air, and then in water. These values are recorded in the AcQuire database, which calculates the density using formula density = 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 barite veins.

The bulk density of the lithologies at Pahtavaara between 2.0 to 4.5 g/cm³ with an average value of 2.93 gm/cm³.

10.7 Ikkari - 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 density using formula density = 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.

The bulk density of the lithologies at Ikkari range between 2 to 4 gm/cm³ with an average value of 2.87 gm/cm³.

For this resource update the density measurements were each assigned a lithology according to their position within the geological model with statistics interrogated within each modelled geological unit (Table 10.3).

As would be anticipated the felsic lithologies 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 content commonly found within this 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 iron addition in the MSCU as a result of mesothermal alteration.

Table 10.3 Ikkari Gold Deposit – Density Statistics (gm/cm^3) by Lithology Group 2020-2021 by Rupert Resources

Lithology Series	Median	Mean	1st Quartile	3rd Quartile	Number
Ultramafic (Talc Altered)	2.88	2.89	2.86	2.92	2,164
Northern Felsic Sediments	2.74	2.75	2.71	2.77	950
Southern Felsic Sediment	2.71	2.70	2.67	2.74	260
Black shale (MSB)	2.78	2.76	2.72	2.83	364
Ultramafic Schist (MSCU)	2.93	2.93	2.87	3.00	2,523
Gabbro	2.84	2.85	2.79	2.93	324
Internal Felsic	2.83	2.85	2.72	2.97	627

Key: gm/cm3 = grams per cubic centimetre

The average of all mineralised lithologies (regardless of lithology) is 2.88 gm/cm^3, which is slightly higher than the host sediments and likely represents the overlap of mineralisation between sediments and mafic-ultramafic units.

10.8 HEINÄ CENTRAL - DRY BULK DENSITY COLLECTION

Since initiation of drilling in April 2019, 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 density using formula density = ρsubstance / ρH_2O, [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.

Using the simple three lithology classification discussed previously the statistics related to densities across these three domains across Heinä Central are shown in Table 10.4.

Considering only the average densities, all three units are very similar at 2.85 to 2.86 g/cm^3 however this masks the large variation within the brecciated vein/sulphide domain highlighted by the 0.16 variance. This variance was to be expected due to the variable but often substantial sulphide content present and the lower density of the surrounding quartz dominated vein material. Both the sedimentary unit and more especially the gabbro are of very consistent density and therefore amenable to density definition by rock type however these are not the main host of the gold-copper mineralisation. Given the variance demonstrated here in the main host it will be necessary to estimate the density as part of the resource estimation process and this is set out in section 14.8.

Table 10.4 Heinä Centrao Gold Deposit – Density Statistics (gm/cm³) by Lithology Group 2019 to 2021 by Rupert Resources

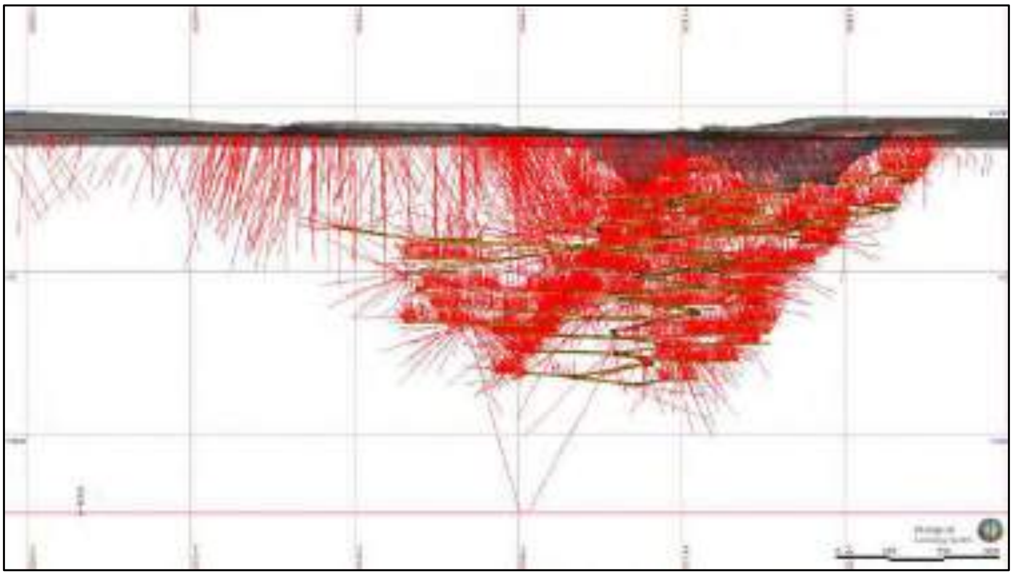
Lithology Series	Median	Mean	1st Quartile	3rd Quartile	Variance	Number
Gabbro	2.86	2.86	2.83	2.89	0.02	336
Sediments and Volcaniclastics	2.85	2.86	2.77	2.92	0.16	933
Vein and Sulphide Infilled Breccias	2.86	2.91	2.89	3.28	0.04	314

10.9 PAHTAVAARA - DRILL DATABASE

The current database contains 2,875 diamond drill holes (339,945 m), 9,137 sludge holes (186,829 m), and 16,557 m of other drilling, including RC drilling and historical sludge drilling (see Table 6.1, Figure 10.2 and Figure 10.3). Channel sampling has also been included in the drill hole database.

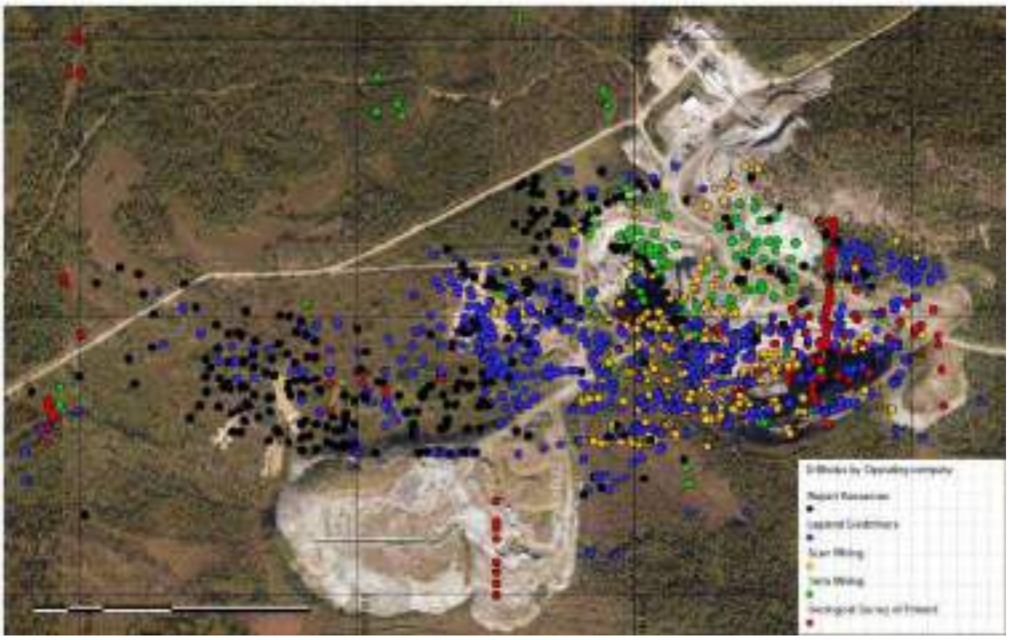
The drilling database contains 327,506 gold assays and 47,682 multi-element assays. The database contains 110,067 surveys and 8,466 density measurements.

Figure 10.2 Long Section Looking North Showing All Drilling in the Pahtavaara Deposit



Source: Internal Rupert Resources Ltd database, 2022

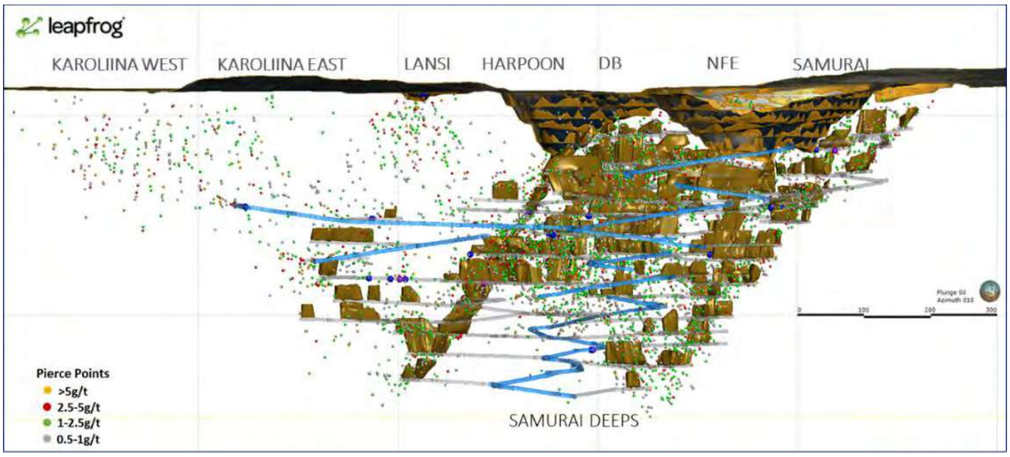
Figure 10.3 Plan View of Pahtavaara Showing Near Mine Drilling by Operator



Grid: 500 m

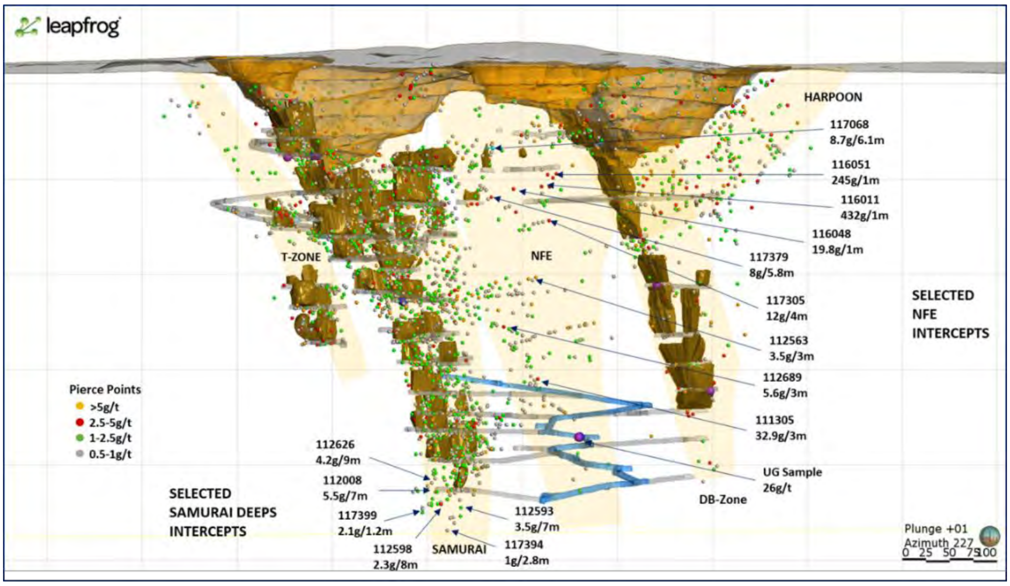
Source: Internal Rupert Resources Ltd database, 2022

Figure 10.4 Long Section of Gold Intersections at Pahtavaara (Looking North)



Source: Internal Rupert Resources Ltd database, 2022

Figure 10.5 Cross Section Looking West Showing Main Zones

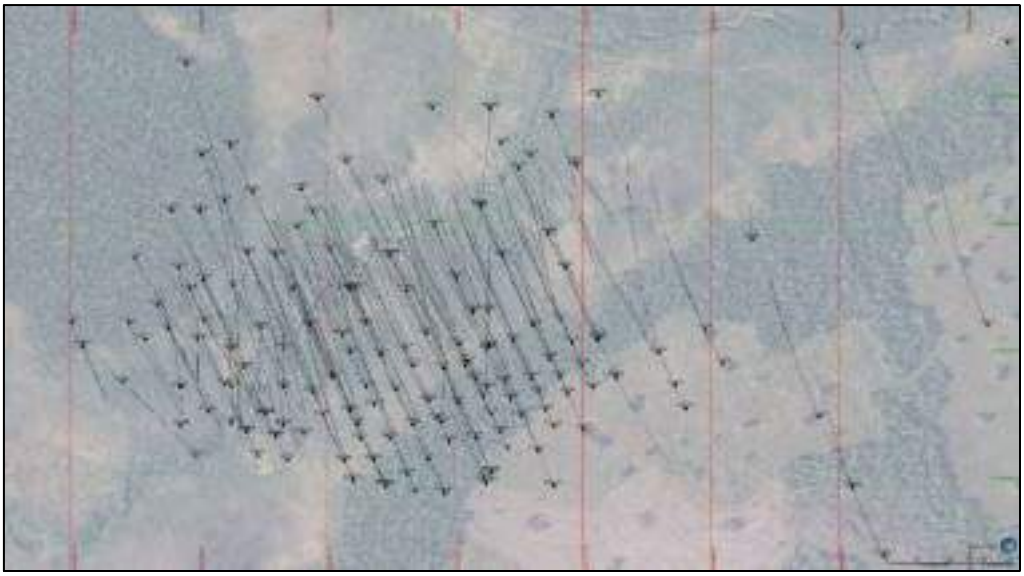


Source: Internal Rupert Resources Ltd database, 2022

10.10 IKKARI - DRILL DATABASE

The Ikkari database used in this resource evaluation contains 180 diamond drill holes (73,033) (Table 10.1 and Figure 10.1). The drilling database used in this resource calculation contains 66,946 gold assays and 48,909 multi-element assays. The database also contains 16,353 downhole survey stations and 7,161 density measurements.

Figure 10.6 Plan Map of Collar Locations and Drill Trace on Semi-transparent Aerial Photograph Draped on Topography



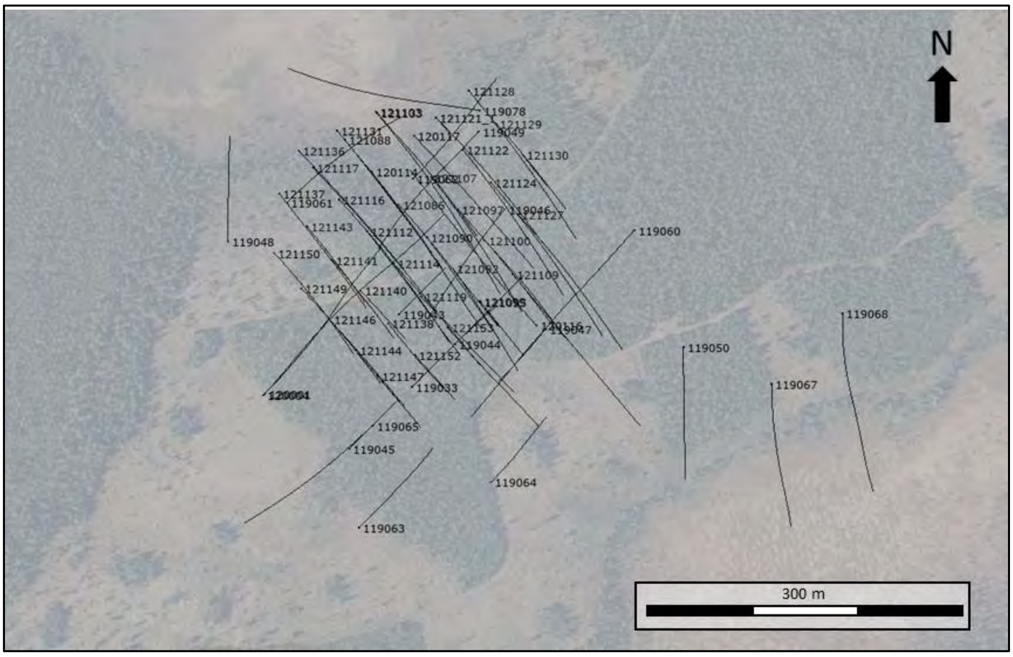
Grid and scale bar 200m

Source: Internal Rupert Resources Ltd database, 2022

10.11 HEINÄ CENTRAL - DRILL DATABASE

The Heinä Central database used in this resource evaluation contains 62 diamond drill holes (12,403 m) (Table 10.1and Figure 10.7). The drilling database used in this resource calculation contains 11,821 gold – and multi-element assays. The database also contains 4,192 downhole survey stations and 1,567 density measurements.

Figure 10.7 Plan View of Heinä Central Drill Hole Locations



Source: Internal Rupert Resources Ltd database, 2022

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLE METHOD AND APPROACH

11.1.1 PAHTAVAARA - HISTORICAL SAMPLING METHODS

Samples were typically collected for one metre intervals and the sample intervals were marked by the company geologist, based on selective sampling of visually interpreted gold-mineralised intervals. Only the areas that were believed to be mineralised were analysed. A three-metre buffer zone was used before and after the interpreted mineralised zones for additional sampling. Three quarters of the core was sent for analysis.

The underground drill chip (sludge) holes were sampled over the entire length of the drill hole with a sample of 2 kilogram (kg) to 3 kg for an average sample interval of 1.8 m being collected.

LAPPLAND GOLDMINERS SAMPLING METHODS

Samples were typically collected from one metre intervals and the samples were marked by the company geologist. Only the areas that were considered to be mineralised were sampled. A three-metre buffer zone was used before and after the interpreted mineralised areas. Three quarters of the core was sent for analysis. Blanks, standards and duplicate samples were systematically added by the geologist into the sample sequence. In exploration diamond drill holes, every 40th sample was a control sample. In production diamond drill holes every 20th sample was a control sample.

The drill chip holes were analysed over the entire length of the drill hole. A sample of 2 kg to 3 kg was analysed.

The drill core was sawed on site in the logging facilities by company personnel. Three quarters of the drill core was sent to the lab for analysis, the rest was stored in core boxes. Fire assay with a 50 g sub sample was used until June 2007. After June 2007 the core samples were analysed for gold by a 500 g subsample with the cyanide “Leachwell method” with an atomic absorption spectroscopy (AAS) finish to determine the cyanide extractable gold content.

The drill chip samples were split by an automatic splitter when drilling. Split samples (approximately 3 kg) of the drill chips were put into numbered bags and sent for analysis. Up to February 2007, fire assay with a 50 g sub sample was used. From March 2007, the drill chips were analysed for gold by a 500 g sub-sample with the cyanide “Leachwell method” and an AAS finish.

11.1.2 Historical Chain of Custody, Sample Preparation and Analyses

The drill core was delivered by the drilling contractor to the core logging facilities. The drill core was measured and logged by a company geologist. The assay sections were marked on the core boxes as well as on the core. Drill core was sawn by company personnel and put into metal boxes or plastic bags with an identical tag as on the core box. The drill chips were sampled underground by company personnel at the drill rig and the chip samples were delivered in wooden boxes to the logging facilities.

Drill chip samples and core samples were taken to the ALS preparation facility in Sodankylä by company personnel or shipped to Piteå (Sweden) by company personnel or a courier. Fire assay with a 50 g sub sample was used until February 2007. From March 2007 the drill chips were analysed for gold by a 500 g subsample using a cyanide leach method “Leachwell” method with an AAS finish. Drill core was analysed using Fire Assay with a 50 g sub sample up until June 2007. The Cyanide “Leachwell” method was used for drill core analysis from June 2007.

Lappland Goldminers’ Chain of Custody, Sample Preparation, and Analyses

All drill core, as well as chips, from percussion drilling were recovered by Goldminers’ technicians or geologists as soon as it is produced.

Drill cores were laid out on the logging tables at the core logging facility and controlled by the company geologist ensuring that the core was in right order in the boxes. Geological logging was conducted by the company geologist. Sample positions, usually of one metre length as standard, were marked on the core boxes according to specific criteria depending on the project.

Every sample interval was labelled on the core boxes by a yellow sample identification digit badge and a line was drawn on the core along which the core was cut.

The core boxes were photographed and the photos stored on the company server. Drill core was geologically logged and rock quality designation (RQD) parameters were recorded continuously along the core. Density was determined by Archimedes (immersion) method every 10th metre as a standard along the core and magnetic susceptibility measured every metre along the hole and recorded.

All core logs were printed out in paper format and stored by the company geologist in binders at the office and all recorded, geological and other data were transferred into an Access database on the company server.

The core was cut with a diamond saw by Lappland Goldminers personnel. Core was cut along the line drawn by the company personnel and three quarters of the core placed in a plastic sample bag and the other quarter placed back into the core box. Every sample bag was labelled with an identical red sample identification digit badge as the sample interval on the core box.

The Labtium laboratory in Sodankylä, Finland was used for assaying gold in the core samples. Samples were transported to the laboratory by Lappland Goldminers personnel. The preparation and assay method for core samples was as follows: The sample preparation methods were Labtium code 14, 31 and 35. Gold was assayed by PAL 1000 cyanidation leach method and values were read by Flame-AAS method (Labtium code

236A). ALS Chemex was used for assaying the underground samples considered to have importance for surface exploration.

Samples analysed for Inductively Coupled Plasma (ICP) elements were transported by courier to ALS Chemex preparation laboratory in Piteå, Sweden. Gold was assayed at ALS Chemex laboratory at Rosia Montana in Romania. Base metals and silver were analysed at ALS Chemex laboratory in Vancouver in Canada. The preparation and assay methods for core samples are as follows: The sample preparation methods are ALS Chemex code PREP-31B and SPL-33. Gold was assayed by Fire Assay and AAS analysis, ALS Chemex code Au-AA26. Base metals and silver for 35 elements were assayed by aqua regia acid digestion and ICP-AES, ALS Chemex code ME-ICP41. Each method had its lower and upper calibration range and sample results falling above the upper calibration range for elements Au, Ag, arsenic (As), Pb, Zn, Mo and Cu were re-assayed by methods with higher calibration ranges. The over limit samples were automatically re-assayed from Au-AA26 by Fire Assay with gravimetric finish, ALS Chemex code Au-GRA22, and from ME-ICP41 by aqua-regia digestion and AAS, ALS Chemex code (+)-AA46.

Blank samples, commercial standard samples and duplicate samples were inserted into the sample stream according to standard intervals set by Lappland Goldminers.

11.1.3 Chain of Custody, Sample Preparation, and Analyses

For all Rupert Resources drilling, the contractor brings the core at the end of their shift to prearranged laydown yard, from where it is collected and transported to Rupert Resources by a transport contractor.

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 organising 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. Reflex ACT III orientation tool is used by drilling contractors to get oriented core. The core is measured and metre intervals are marked on core boxes and on core.

Core logging is done by using Geobank Mobile logging software. Log sheets to be filled include lithology, structural data, magnetic susceptibility and core recovery (RQD) sheets and a sample data sheet.

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 marked on the left side of each metre line in the core box with pencil. Geobank mobile calculates RQD percentage automatically from given recovery and RQD centimetres.

The geology logging includes the geology, “geozone” code, structure and sample data including company check 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.

The Geobank Mobile 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 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 sample ID number same as 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 sample tickets are placed in a sealed bag for dispatch along with the batch of samples.

Drill core is sawn in 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. 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.

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 assay laboratory used by Rupert Resources is ALS Minerals at Sodankylä, Finland (prep lab) and gold assay laboratory ALS Geochemistry in Rosia Montana, Romania. The assay method in use has been 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, pulverising the entire sample (max 3 kg) minimum 85% to 75 microns (µm). Samples greater than 3 kg are split prior to pulverising with method SPL-22. After pulverising 250 g extra split is packed separately and returned to Rupert Resources for use in umpire lab checks. The over limit samples (>100 ppm Au) are automatically re-assayed via fire assay with gravimetric finish, code Au-GRA22. A few batches were also sent for PGM-ICP24 analyses for testing platinum group metals. Multi-elements (Ag, Al, arsenic (As), barium (Ba), Beryllium (Be), Bismuth (Bi), Ca, cadmium (Cd), cerium (Ce), Co, Cr, caesium (Cs), Cu, Fe, gallium (Ga), germanium (Ge), hafnium (Hf), indium (In), K, Lanthanum (La), Lithium (Li), Mg, Mn, molybdenum (Mo), Na, niobium (Nb), Ni, Phosphorus (P), Pb, rubidium (Rb), rhenium (Re), sulphur (S), antimony (Sb), scandium (Sc), selenium (Se), tin (Sn), strontium (Sr), tantalum (Ta), tellurium (Te), thorium (Th), Ti, thallium (Th), uranium (U), V, tungsten (W), yttrium (Y), 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 – 48 elements by HFHNO3-$HClO_4$ 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.

Some of the infill holes were assayed for gold in Eurofins Labtium Sodankylä with their Au-705P method, gold assay 50 g by fire assay with ICP-OES finish. 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, pulverising the whole sample (max 3.5 kg) in one milling (method 50, LM5). After pulverising the whole pulp is sampled to subsamples for following Fire Assay analysis. The pulp rejects are packed in plastic bags. The pulverising puck and the bowl are cleaned by pulverising 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 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 quality assurance and quality control (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 mineralised zones has averaged >99%.

11.2 ASSAY QUALITY CONTROL - PAHTAVAARA

For all drilling undertaken by Rupert Resources, analysis of internationally accredited assay standards or certified reference material ("CRM") has been carried out.

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.
- Barren samples ("blanks") submitted by both Rupert and the assay laboratory.

11.2.1 PRE-2016 DATA (PAHTAVAARA)

For information relating to drilling and sampling undertaken prior to 2016, the sections are quoted from the 2013 Micon International Co. Ltd ("Micon") independent NI 43-101 report (Micon, 2013). The relevant sections are replicated here and are identifiable as being in italics. Micon carried out a review and statistical analysis of spreadsheet data relating to the analysis of standards, duplicates and blanks completed by the main assay laboratory during 2012.

The following text summarises the approach taken by Micon (which is taken from their 2013 report):

The standards (certified reference materials) used by Pahtavaara were prepared, certified and supplied by: Ore Research & Exploration Pty Ltd (OREAS), 6-8 Gatwick Road, Bayswater North Victoria 3153, Australia.

The Pahtavaara data includes assay results for 1,262 standard, blank and duplicate samples. These data were checked and categorised. Obvious errors were corrected as appropriate.

Table 11.1 (note the table numbering has been adapted to match the current report) shows the certified assay values and expected Performance Gates (A Performance Gate is a control value specified as a Standard Deviation of the Certified Value) of the gold standards used at Pahtavaara.

Table 11.1 Pahtavaara Gold Deposit Gold Ore Reference Material Performance Gates (Standard Deviations)

Absolute Standard Deviations							
Standard	Constituent	Certified Value	1SD	2SD		3SD	
				Low	High	Low	High
OREAS 15d - (SH)	Au (ppm)	1.559	0.042	1.475	1.643	1.433	1.685
OREAS 15g - (SF)	Au (ppm)	0.527	0.023	0.481	0.573	0.458	0.596
OREAS 15h - (SK)	Au (ppm)	1.019	0.025	0.970	1.068	0.945	1.093
OREAS 19a - (SJ)	Au (ppm)	5.490	0.100	5.290	5.690	5.190	5.790
OREAS 12a - (SI)	Au (ppm)	11.790	0.240	11.310	12.270	11.070	12.510
OREAS 60b - (SD)	Au (ppm)	2.570	0.110	2.350	2.780	2.250	2.890
OREAS 61d - (SE)	Au (ppm)	4.760	0.140	4.470	5.040	4.330	5.190
OREAS 18c - (SM)	Au (ppm)	3.520	0.110	3.310	3.730	3.200	3.840
OREAS 10c - (SO)	Au (ppm)	6.60	0.16	6.27	6.92	6.61	7.08
OREAS 16a - (SN)	Au (ppm)	1.81	0.06	1.68	1.93	1.62	1.99
OREAS 62c - (SR)	Au (ppm)	8.79	0.21	8.36	9.21	8.15	9.42

Performance Gates (Absolute Values)								
Standard	Constituent	Certified Value	1SD		2SD		3SD	
			Low	High	Low	High	Low	High
OREAS 2Pd - (SA)	Au (ppm)	0.885	0.855	0.914	0.826	0.943	0.797	0.973
OREAS 6Pc - (SG)	Au (ppm)	1.520	1.460	1.590	1.390	1.660	1.320	1.720
OREAS 15Pb - (SB)	Au (ppm)	1.060	1.030	1.090	1.000	1.120	0.970	1.140
OREAS 53Pb - (SC)	Au (ppm)	0.623	0.602	0.644	0.581	0.666	0.559	0.687

The Performance Gates for each standard are as follows:

- *Value for the Upper Control Limit (UCL) and Lower Control Limit (LCL) are provided by the manufacturer;*
- *UCL and LCL are 3 standard deviations above and below the Certified Value of each standard;*
- *Values for the Upper Warning Limit (UWL) and Lower Warning Limit (LWL) are provided by the manufacturer; and*

- *UWL and LWL are 2 standard deviations above and below the Certified Value of each standard.*

None of the standard sample assay results should fall outside the UCL or LCL.

Not more than 5% (ie 1 in 20) of the assay values should fall outside the UWL and LWL.

The assay values for standard samples returned by the Main Laboratory were compared with the OREAS certified assay values by plotting the assay values against the Certified Value of the material and its Performance Gates. A summary of the analysis of the standard samples is shown in Table 11.2*.*

Table 11.2 Pahtavaara Gold Deposit Summary of Standard Assay Results

Standard Label	*Mean Grade of Standard (Au g/t)*	*Number of Data Points*	*Points Outside UCL and LCL*	*Percentage of Points Outside UCL and LCL (%)*	*Comments*
SA	*0.89*	*5*	*5*	*Not Plotted*	*-*
SB	*1.06*	*1*	*0*	*Not Plotted*	*-*
SC	*0.62*	*0*	*0*	*No Points*	*-*
SD	*2.57*	*0*	*0*	*No Points*	*-*
SE	*4.76*	*1*	*1*	*Not Plotted*	*-*
SF	*0.53*	*6*	*3*	*Not Plotted*	*-*
SG	*1.52*	*12*	*9*	*75*	*Total loss of laboratory control*
SH	*1.56*	*29*	*26*	*89*	*Total loss of laboratory control*
SI	*11.79*	*54*	*51*	*94*	*Total loss of laboratory control*
SJ	*5.49*	*72*	*63*	*87*	*Total loss of laboratory control*
SK	*1.02*	*9*	*9*	*100*	*Total loss of laboratory control*
SM	*3.52*	*35*	*25*	*71*	*Total loss of laboratory control*
SN	*1.81*	*17*	*13*	*76*	*Total loss of laboratory control*
SO	*6.60*	*27*	*25*	*92*	*Total loss of laboratory control*
SR	*8.79*	*5*	*3*	*60*	*Total loss of laboratory control*

Micon commented as follows:

If more than 2 successive points in a plot lie outside the UCL and LCL this indicates a significant loss of control by the assay laboratory. More than 4 points on one side or the other of the mean (the mean being the Certified Value for a Standard) signifies a drift in the assaying process used by the laboratory or a significant bias in the results.

The assay results have been plotted in time order. Progressive changes about the mean or cyclic variations of assay values suggest a time dependent variation and loss of control and precision in the assaying procedures of the laboratory.

The assay values for standards SA, SB, SC, SD, SE and SF were not analysed. Data points were too few to provide a meaningful analysis.

The control plots produced by Micon for the time-ordered assays of the CRM show that the assay laboratory was routinely ‘undercalling’ the expected assay grade, for all CRM investigated. Micon concluded that:

The analysis of the quality control data shows that the results provided to Pahtavaara by the assay laboratory are of poor quality. The laboratory appears to be reporting consistently lower values for the standards. This conclusion is reinforced by the analysis of the duplicate samples.

The mine should investigate the reasons for the variations in standard assay values and should carefully consider the impact of this investigation on the reliability and use of the assay results provided during this period.

It appears that the assay method used to routinely analyse samples from Pahtavaara is the ‘Leachwell’ method, which is a cyanide –extractable gold assay method with an AAS finish. The assay method is designed to be carried out on samples larger than 500 g and will only determine the cyanide-extractable component of in-situ gold, not the total gold content. It is evident that the main problem with the under-calling is that the OREAS reference standards are based on multiple fire assay/AAS analyses at a larger number of independent laboratories and is intended to quantify the total gold content of the standard in question, not the cyanide extractable component. It appears that all the standards have a proportion of gold that cannot be extracted by the Leachwell method. This point will be discussed further below.

Micon also undertook a review of some 511 blank samples. Micon commented as follows:

Of the 511 blank samples that were analysed only 4 were shown to have a value above the nominal zero detection limit of 0.010 Au g/t. This is less than 0.01% of the total number of samples. The material being used for blank samples is therefore regarded as satisfactory.

Micon reviewed the data for 75 duplicate samples. It is not specified in the Micon report what sort of duplicate sample was being reviewed. Micon noted that the largest ‘errors’ are associated with the low-grade samples (quoted as being less than 1.0 g/t). Micon concluded as follows:

The greatest errors are attributed to the lowest grade samples. This conclusion is not unexpected given the difficulty the assaying laboratories have in achieving satisfactory repeatability for the standard control samples.

Review of the figures in the Micon report suggests that the great majority of the duplicate pairs return a mean grade of between 0.1 g/t and 1.0 g/t, with the relative error between ± 10% and ±100%.

11.2.2 Post 2016 Data (Pahtavaara)

QA/QC data from sampling and analyses carried out from 2016 to the present has been compiled in Pahtavaara AcQuire 4 relational database. The relevant information has been downloaded for statistical review and analysis and includes the following datasets:

- Blanks:
  - Submitted by Rupert.
  - Internal ALS blanks.
  - Internal CRS blanks.
  - Internal Eurofins Labtium blanks.
- CRM (Standards):
  - Submitted by Rupert:
    - To ALS.
    - To CRS.
    - To Eurofins Labtium.
  - ALS internal CRM.
  - CRS internal CRM.
  - Eurofins Labtium internal CRM.
- Data Pairs:
  - Submitted to ALS, CRS and Eurofins Labtium:
    - Core duplicates (quarter core pairs).
    - Crush duplicates (duplicates taken after the jaw crush stage, crush stage duplicates not taken since July 2018).
    - Lab duplicate (duplicate samples taken after pulverised to >85% <75 µm).
    - Pulp duplicates (duplicates samples taken from within one pulp sachet).
    - 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.

Table 11.3 summarises the results of assaying blank samples. For the great majority of analyses, the blanks returned less than detection limit results.

Table 11.3 Pahtavaara Gold Deposit Blanks

Standard	Assay Method	Laboratory	Number	Expected Value	Mean	% Bias	% in Tolerance
BLK-CO01	Au-AA26-ppm	ALS	1,027	0.01	0.0053	-47.0789	100
BLK-CO01	Au-AA15-ppm	ALS	921	0.01	0.0066	-34.0934	100
BLK-CO01	Au-PAL1000-ppm	CRS	189	0.05	0.0251	-49.7354	100
BLK-CO01	Au-705P-ppm	Labtium	595	0.01	0.0102	2.0840	99.4958
BLK-ALS	Au-AA26-ppm	ALS	634	0.01	0.0053	-47.0032	100
BLK-ALS	Au-AA15-ppm	ALS	1,515	0.01	0.0058	-41.8152	100
BLK-CRS	Au-PAL1000-ppm	CRS	195	0.05	0.0250	-50.0000	100
SOKEA_Labtium	Au-705P-ppm	Labtium	353	0.01	0.0096	-4.1643	100
SOKEA_Labtium*	Au-703P-ppm	Labtium	3	0.01	0.0100	0	100

In summary all laboratories produced acceptable assaying of blank samples.

Figure 11.1 Rupert Company Blank (BLK-CO01) Performance in AA26 ALS

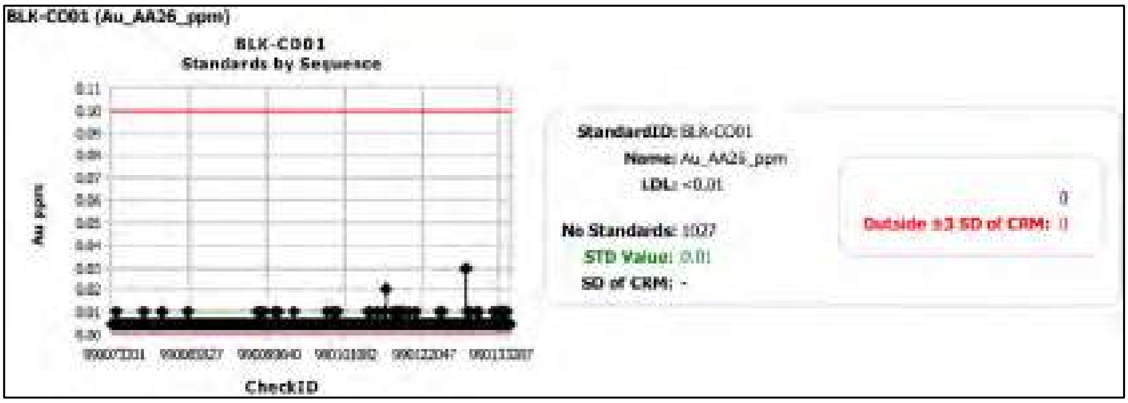


Figure 11.2 Rupert Company Blank (BLK-CO01) Performance in AA15 ALS

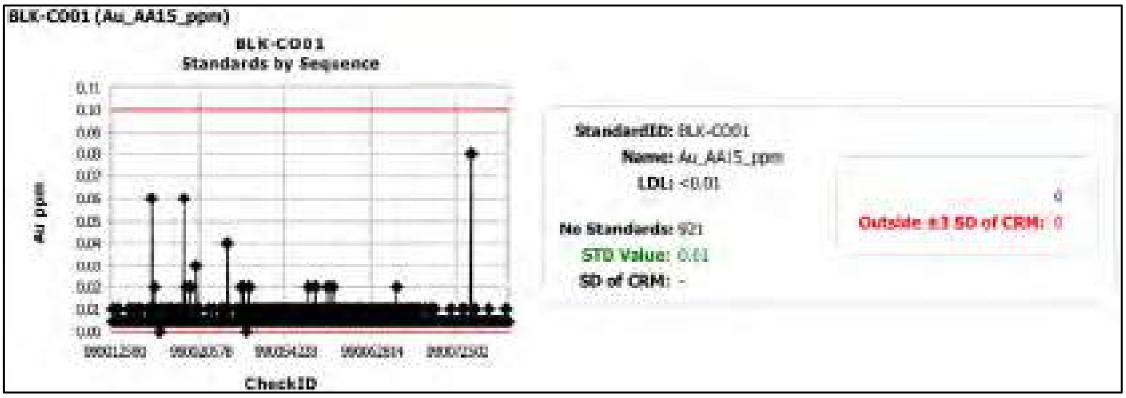


Figure 11.3 Rupert Company Blank (BLK-CO01) Performance in 705P Eurofins Labtium

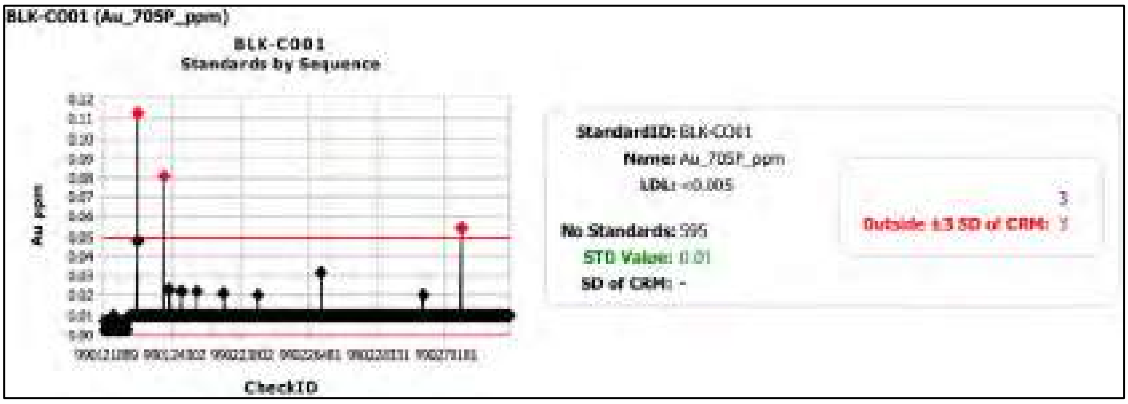
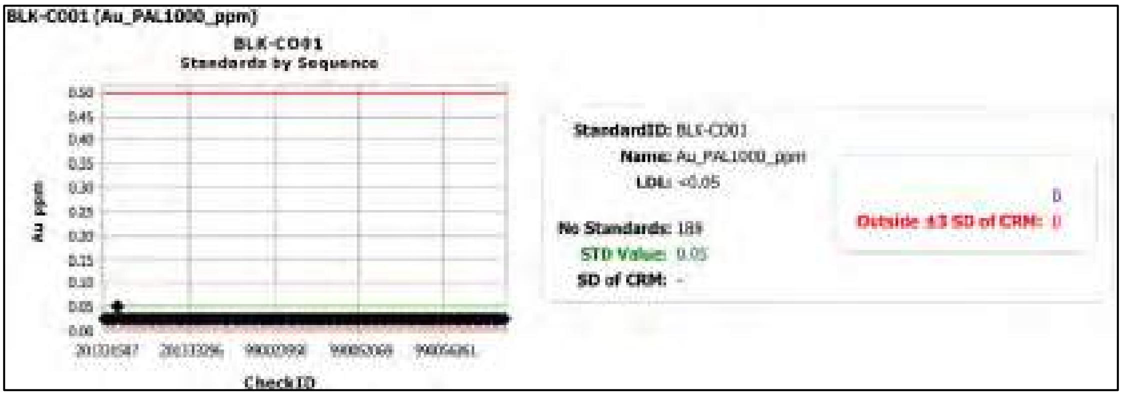


Figure 11.4 Rupert Company Blank (BLK-CO01) Performance in PAL1000 CRS



CRM Submitted by Rupert Resources

Rupert Resources routinely submitted accredited CRM. Since July 2018 QA/QC review Rupert Resources has been using gold certified reference materials produced by Geostats pty ltd. These CRM's (G912-3, G915-2, G915-6, G314-2, G315-7, G398-4, G312-4, and G917-4) have been selected to represent three different gold grades.

Table 11.4 Pahtavaara Gold Deposit Standards Submitted to ALS by Rupert Resources

Standard	Assay method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
G312-4	Au-AA26	156	5.3	5.3258	0.4862	2.3975	100
G398-4	Au-AA26	320	0.66	0.6444	-2.3674	3.0557	100
G912-3	Au-AA26	290	2.09	2.1065	0.7903	2.6338	100
G915-2	Au-AA26	27	4.98	5.0893	2.194	2.2797	100
G917-4	Au-AA26	90	5.1	5.119	0.3725	1.8288	100
OREAS-214	Au-AA26	8	2.92	3.0288	3.7243	3.5139	100
OREAS-214	Au-AA15	292	2.92	2.923	0.1044	4.4852	95.2055
OREAS-216	Au-AA26	32	6.53	6.6641	2.053	2.6055	100
OREAS-216	Au-AA15	392	6.53	6.5045	-0.3911	3.7435	98.2143
CDN-CM-17	Au-AA26	36	1.37	1.3769	0.5069	4.7156	100
CDN-CM-17	Au-AA15	19	1.37	1.2953	-5.4552	3.1557	100
CDN-CM-4	Au-AA15	114	1.18	1.0764	-8.7794	26.2365	78.9474
OREAS-10C	Au-AA15	5	6.6	5.528	-16.2424	8.7967	20
OREAS-12A	Au-AA15	3	11.79	10.5167	-10.8001	4.8095	33.333
OREAS-15D	Au-AA15	6	1.559	1.2333	-20.8895	5.1451	0
OREAS-16A	Au-AA15	7	1.81	1.3586	-24.9408	21.0152	0
OREAS-18C	Au-AA15	6	3.52	2.7733	-21.2121	6.9533	0
OREAS-19A	Au-AA15	11	5.49	4.5418	-17.2711	6.2235	0
OREAS-203	Au-AA15	5	0.871	0.864	-0.8037	34.0069	20
OREAS-204	Au-AA15	4	1.04	0.895	-13.9423	8.7741	25
OREAS-208	Au-AA15	3	9.25	7.0233	-24.0721	28.7445	66.6667
OREAS-62C	Au-AA15	3	8.79	7.3933	-15.8893	1.5677	0
OREAS-62D	Au-AA15	5	10.5	9.558	-8.9714	3.828	40

Table 11.5 Pahtavaara Gold Deposit Standard Submitted to CRS by Rupert Resources

Standard	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
CDN-CM4	2	1.180	1.14	-3.3898	2.4811	100
OREAS-10C	4	6.600	5.8850	-10.6061	1.6549	0
OREAS-12A	4	11.790	10.7700	-8.6514	1.1090	0
OREAS-15D	10	1.559	1.4750	-9.1725	9.2332	20
OREAS-16A**	4	1.810	1.5650	-33.1492	67.2120	50
OREAS-18C**	7	3.520	2.6193	-25.5885	43.8024	14.2857
OREAS-19A*	8	5.490	7.3925	34.6539	92.4218	0
OREAS-203	5	0.871	0.9480	8.8404	10.2108	80
OREAS-204	8	1.040	1.1038	6.1298	4.9609	100
OREAS-208	2	9.250	8.8800	-4.0000	3.9815	100
OREAS-214	37	2.920	2.9451	0.8608	2.3259	100
OREAS-216	44	6.530	6.4193	-1.6950	1.9216	100
OREAS-62D	4	10.500	10.2250	-2.6190	1.2306	100

Norte: *One sample returned 24.3 g/t (most likely substitution error).
** One sample returned 0 g/t (most likely substitution error).

Table 11.6 Pahtavaara Gold Deposit Standards Submitted to Euroins Labitium by Rupert Resources

Standard	Assay method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
G912-3	Au-705P	191	2.09	2.1280	1.8177	2.2560	100
G398-4	Au-705P	49	0.66	0.6508	-1.4007	4.1574	100
G917-4	Au-705P	48	5.1	5.1953	1.8689	2.0791	100
G915-2	Au-705P	117	4.98	5.1129	2.6688	2.5221	100
G915-6	Au-705P	125	0.67	0.6700	0.0048	4.6823	100

Internal CRM analysed by ALS

ALS, as part of their standard QA/QC procedures routinely analyse CRM prepared by independent suppliers. Rupert has obtained all the available internal ALS CRM analytical results and statistical analysis has been carried out on the gold data.

Table 11.7 summarises the results of the analytical performance by ALS on these internally submitted CRM. The assay method used for the different CRM is also noted in Table 11.7.

Table 11.7 Pahtavaara Gold Deposit ALS Internal Standards

Standard	Assay Method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
BLK_ALS	AA15	1515	0.01000	0.0058	-41.8152	334.0283	100
BLK_ALS	AA26	634	0.01000	0.0053	-47.0032	22.4124	100
G306-3	AA26	158	8.66000	8.7099	0.5766	2.0793	100
G312-4	AA26	192	5.30000	5.3136	0.2575	2.1716	100
G314-5	AA26	124	5.29000	5.3413	0.9696	1.9744	100
G910-3	AA15	17	4.03000	3.8988	-3.2550	3.5573	100

G912-1	AA26	5	7.29000	7.1900	-1.3717	1.2556	100
G912-5	AA26	7	0.38000	0.3643	-4.1353	2.6789	100
G913-10	AA26	113	7.09000	7.0904	0.0050	2.3473	100
GLG304-1	AA26	7	0.15391	0.1529	-0.6841	3.1922	100
GLG904-1	AA15	21	0.20408	0.2010	-1.5325	9.6884	100
GLG908-4	AA15	2	0.06588	0.0600	-8.9253	23.5702	50
OREAS-12A	AA26	2	11.79000	11.8500	0.5089	2.3869	100
OREAS-200	AA26	5	0.34000	0.3360	-1.1765	2.6620	100
OREAS-202	AA26	95	0.75200	0.7560	0.5319	2.2117	100
OREAS-204	AA15	251	1.00700	0.9579	-4.8770	3.2919	100
OREAS-217	AA26	84	0.33800	0.3326	-1.5920	2.5612	100
OREAS-250	AA15	508	0.30900	0.3024	-2.1418	5.0627	100
OREAS-252	AA26	53	0.67400	0.6730	-0.1456	2.4826	100
OREAS-253	AA15	181	1.22000	1.1680	-4.2659	3.2935	100
OREAS-256	AA15	339	7.51000	7.5439	0.4509	3.3054	100
OxA89	AA15	11	0.08360	0.0809	-3.2188	39.6488	18.1818
OxF125	AA26	2	0.80600	0.7800	-3.2258	0	0
OxJ111	AA15	12	2.16600	2.1708	0.2231	3.7499	66.67
OxP116	AA15	21	14.92000	14.7786	-0.9479	2.3937	80.95
ST14/9501	AA15	73	0.43000	0.3978	-7.4865	3.7060	100
ST-463	AA15	58	9.66000	9.3662	-3.0413	2.2795	100
OXL118	AA26	11	5.82800	5.8118	-0.2777	1.0904	100
OxT126	AA26	3	0.80600	0.7800	-3.2000	0.0000	0
LEA-16	AA26	214	0.50100	0.4985	-0.4981	2.5985	100
JK-17	AA26	113	1.99800	1.9579	-2.0082	1.9752	89.3805
JK-17	GRA22	2	1.99008	1.9850	-0.6507	1.7811	100
OXL118	AA26	10	5.82800	5.8180	-0.1716	1.0865	100
OxP116	AA15	21	14.92000	14.7786	-0.9479	2.3937	80.9524
OxQ90	GRA22	2	24.88000	25.4500	2.2910	0.2778	100
SJ63	AA15	13	2.63200	2.6515	0.7423	2.3292	100

Internal CRM analysed by CRS

CRS have also routinely undertaken internal analyses of CRM as part of their standard QA/QC procedures. Table 11.8 summarises the analytical data for the two CRM routinely used by CRS and their blank.

Table 11.8 Pahtavaara Gold Deposit CRS Internal Standards

Standard	Assay Method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
BLK-CRS	PAL1000	195	0.05	0.0250	-50.000	0	100
GR313-10	PAL1000	109	45.86	45.3367	-1.1411	2.0741	100
G915-10	PAL1000	91	48.92	47.2969	-3.3178	2.0311	100

Internal CRM analysed by Eurofins Labtium

Eurofins Labtium have also routinely undertaken internal analyses of CRM as part of their standard QA/QC procedures. Table 11.9 summarises the analytical data for the CRM Au267 and their blank SOKEA.

Table 11.9 Pahtavaara Gold Deposit Eurofins Labtium Internal Standards

Standard	Assay Method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
SOKEA_Labtium	705P	355	0.01	0.0096	-4.1408	26.0895	100
SOKEA_Labtium	703P	3	0.01	0.0100	0	0	100
Au267	705P	452	2.67	2.6877	0.6622	2.5408	98.4513

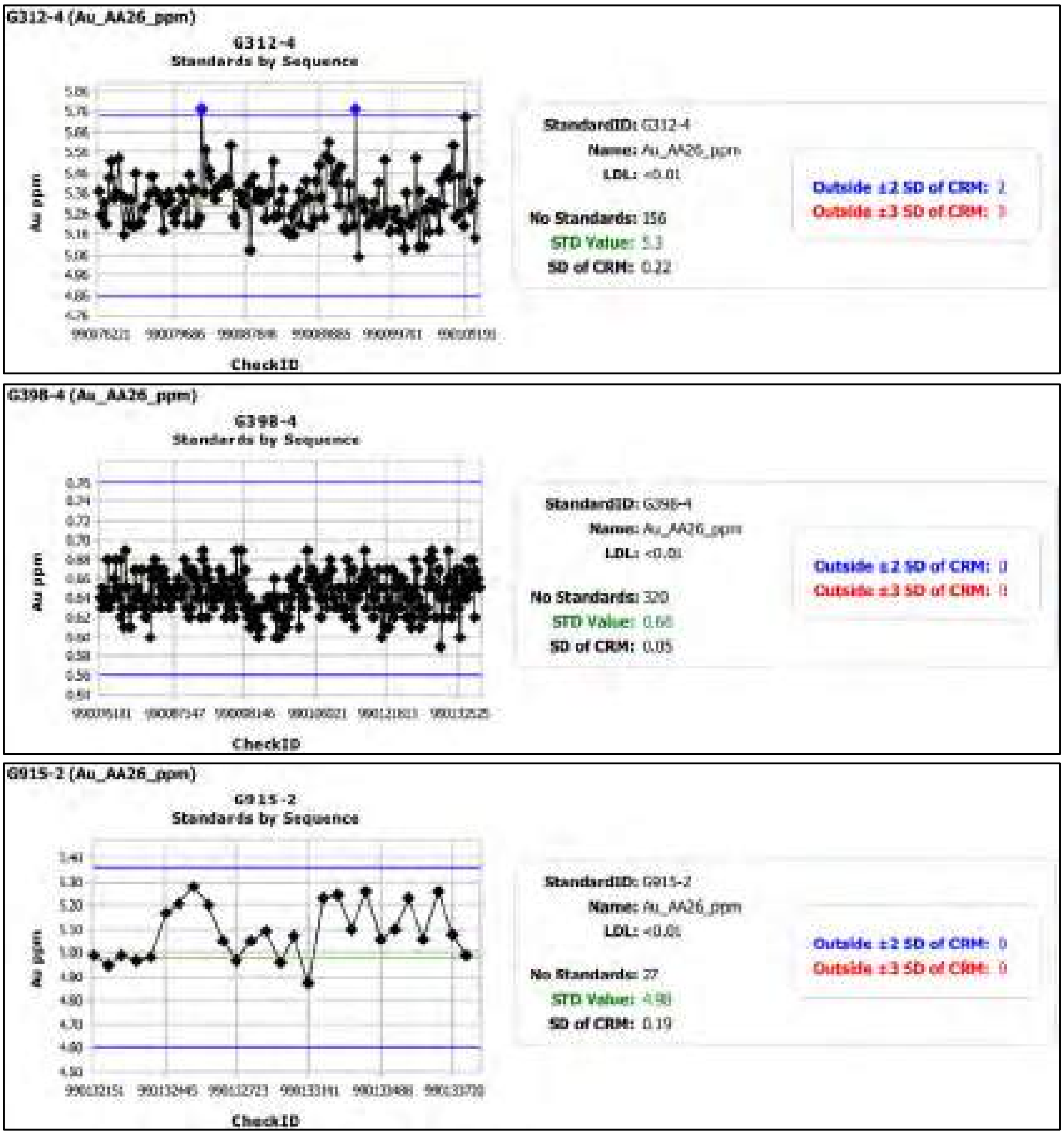
Comparison of common CRM

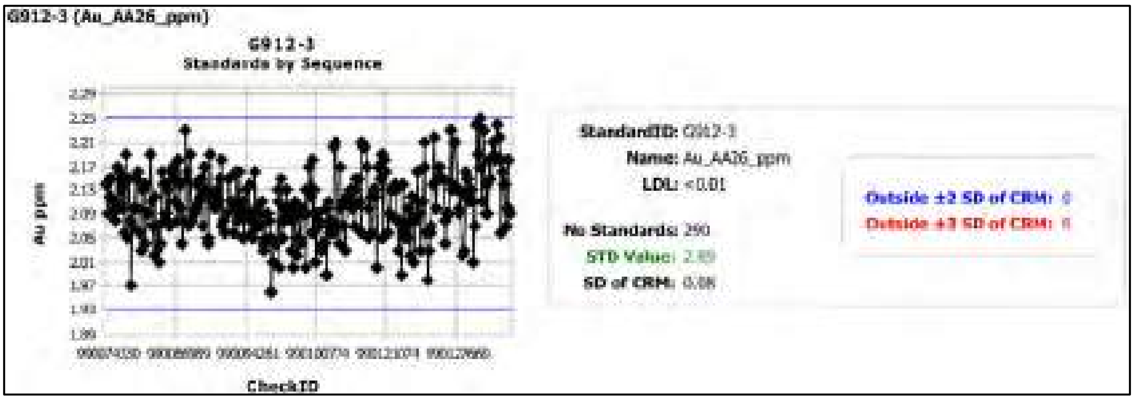
All Geostats CRM's Rupert Resources has been using since July 2018 perform very well with used fire assay methods in both laboratories, ALS and Eurofins Labtium.

Table 11.10 Pahtavaara Gold Deposit Comparison of Commonly Submitted Standards

Standard	Submitted By	Laboratory	Assay Method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
G312-4	Rupert	ALS	AA26	156	5.30	5.3258	0.4862	2.3975	100
G312-4	ALS	ALS	AA26	192	5.30	5.3136	0.2575	2.1716	100
G398-4	Rupert	ALS	AA26	320	0.66	0.6444	-2.3674	3.0557	100
G398-4	Rupert	Labtium	705P	49	0.66	0.6508	-1.4007	4.1574	100
G912-3	Rupert	ALS	AA26	290	2.09	2.1000	0.7903	2.6338	100
G912-3	Rupert	Labtium	705P	191	2.09	2.1280	1.8177	2.6396	100
G915-2	Rupert	ALS	AA26	27	4.98	5.0893	2.1940	2.2797	100
G915-2	Rupert	Labtium	705P	117	4.98	5.1129	2.6688	2.5221	100
G915-6	Rupert	Labtium	705P	125	0.67	0.6700	0.0048	4.6823	100
G917-4	Rupert	ALS	AA26	90	5.10	5.1190	0.3725	1.8288	100
G917-4	Rupert	Labtium	705P	48	5.10	5.1953	1.8689	2.0791	100

Figure 11.5 Performance of CRM's Submitted by Rupert Resources to ALS AA26





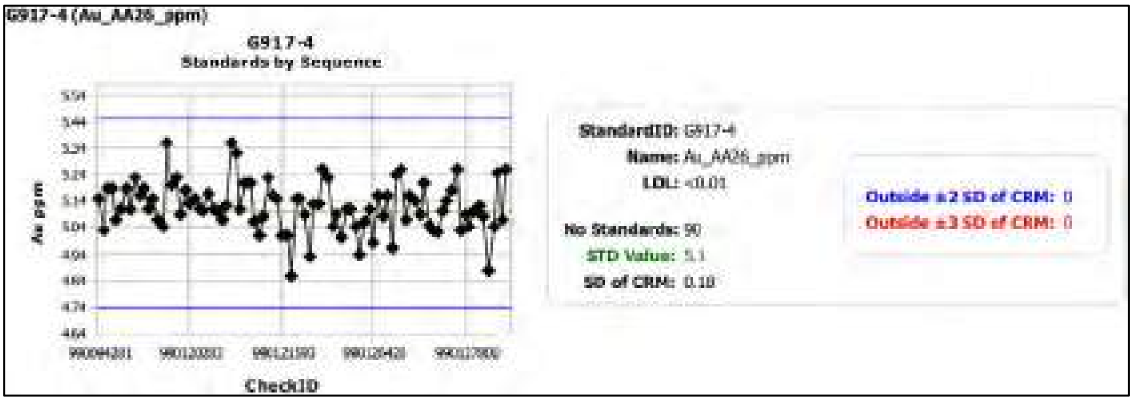
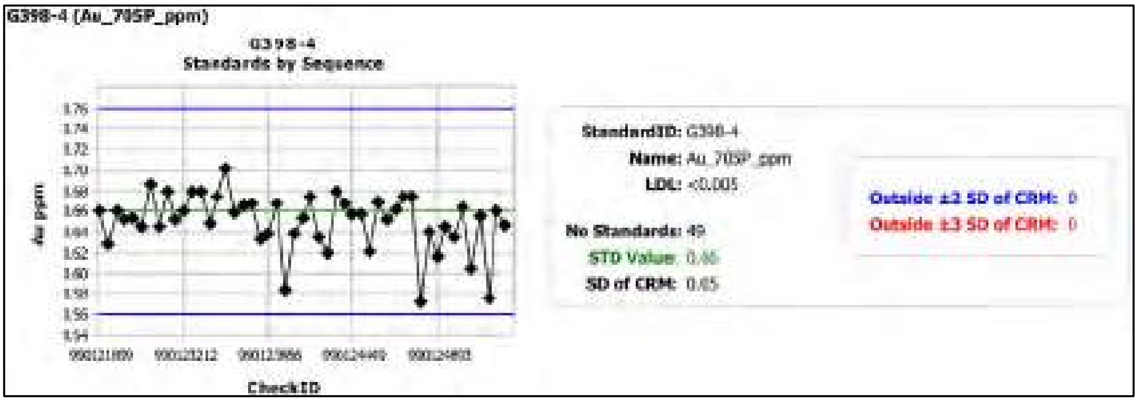
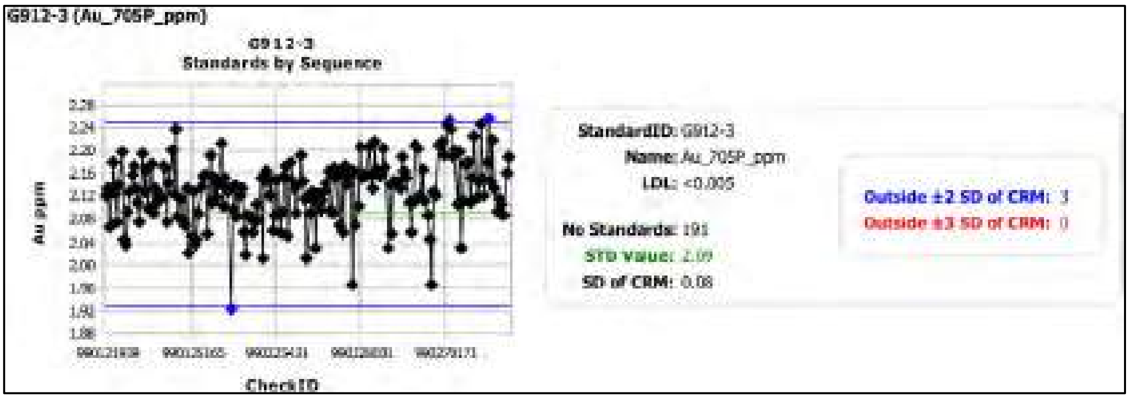
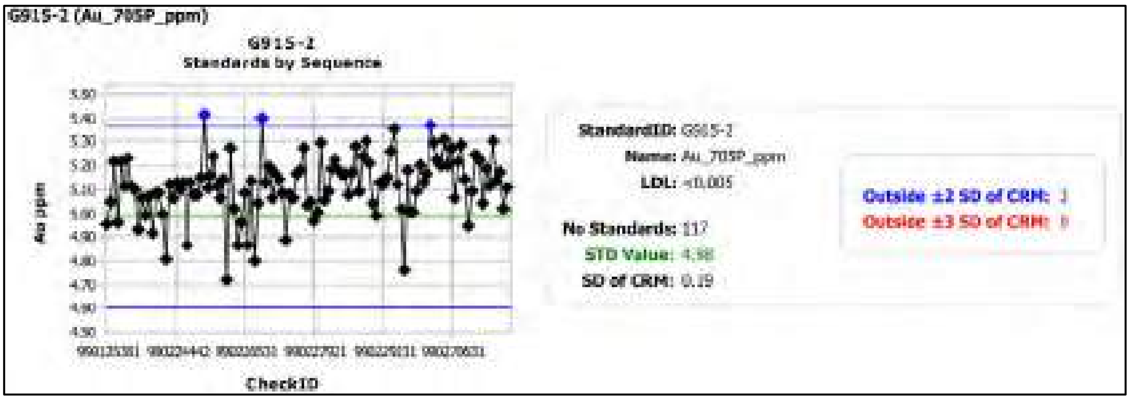
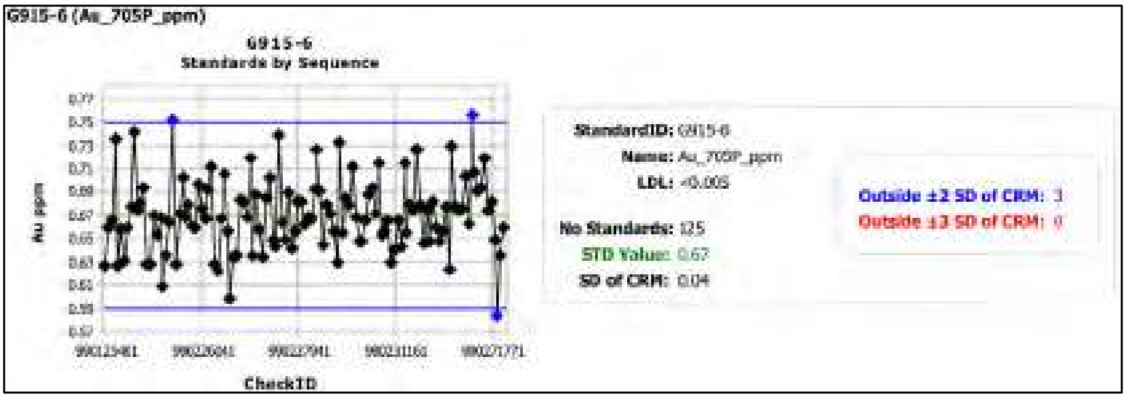


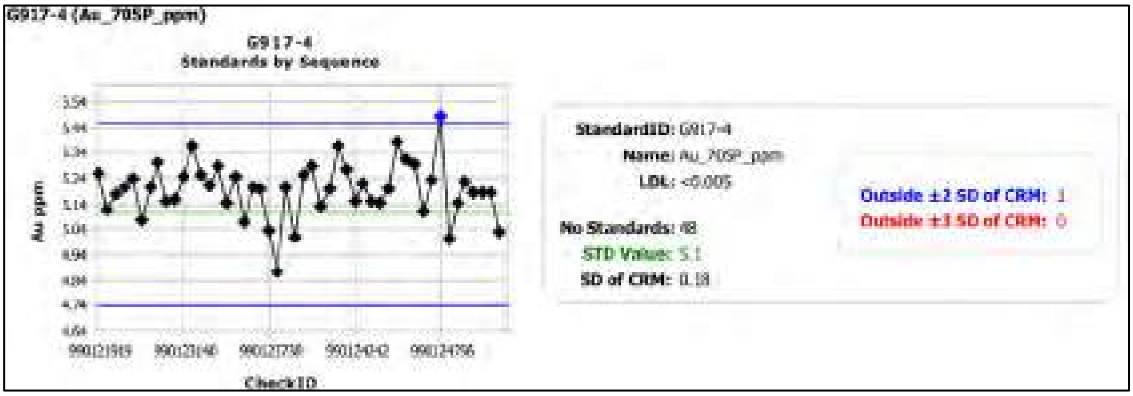
Figure 11.6 Performance of CRM's submitted by Rupert to Eurofins Labtium 705P











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 and assay method. The different types of data pairs comprise the following:

- Field duplicates (quarter core pairs).
- Lab Duplicates (two samples taken after pulverising sample material >85% <75 µm).
- Pulp duplicates (duplicates samples taken from within one pulp sachet).
- Crush duplicates (duplicates taken after the jaw crush stage. Only cyanide leach, crush stage duplicates are not part of the QC-programme since July 2018 and changing method to fire assay.).
- Umpire checks (Pulp split sent to second laboratory).

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

- Mean Paired Relative Difference (MPRD) by Mean Grade.
- Correlation Plot.
- Quantile-Quantile Plot.

Samples Submitted to ALS

Samples submitted to ALS for data pair analysis have included the following sample types:

- Field duplicates (two separate quarter core samples from the same sample interval).
- Crush duplicates (two samples taken after jaw crushing to a nominal 2 to 3 mm. No more crushing stage duplicates after July 2018).
- Lab Duplicates (two samples taken after pulverising sample material >85% <75 µm).
- Pulp duplicates (two sub-samples taken from the same pulp sachet).
- Umpire checks (pulverising stage split for samples originally assayed at CRS or Labtium).

Samples Submitted to CRS

Sample pairs submitted to CRS include the following:

- Core Duplicates.
- Crush duplicates (No more crushing stage duplicates after 2018 July).
- Lab Duplicates (LAB1 and LAB2 are two sets of sub samples from the main sample after size reduction to a notional particles size of >85% passing 75 µm).
- Umpire checks (Pulverising stage split sent to second laboratory)

Samples Submitted to Eurofins Labtium

Sample pairs submitted to Eurofins Labtium include the following:

- Core Duplicates.
- Lab Duplicates.
- Umpire checks (Pulverising stage split for samples originally assayed at ALS by fire assay).

Table 11.11 Pahtavaara Gold Deposit Data Pairs

Duplicate Type	Submitted by	Laboratory	Assay Method	Total number of pairs	Au1 Mean (g/t)	Au2 Mean (g/t)	Corr. Coeff.	Absolute Mean Paired Relative Difference (AMPRD)
Core duplicate	Rupert	ALS	AA26	1292	0.4296	0.4333	0.4512	90.8044
Crush duplicate	Rupert	ALS	AA26	20	0.1550	0.1450	1.0000	6.6741
Pulp duplicate	Rupert	ALS	AA26	998	0.4472	0.4854	0.9998	47.8493
Lab duplicate	ALS	ALS	AA26	1050	2.1361	2.1022	0.7665	55.5314
Core duplicate	Rupert	ALS	AA15	524	0.4081	0.5703	0.7738	85.1551
Crush Duplicate	Rupert	ALS	AA15	747	1.6438	1.5594	0.9937	46.7911
Pulp Duplicate	Rupert	ALS	AA15	387	0.3906	0.3383	0.9780	38.5890
Lab Duplicate	ALS	ALS	AA15	1447	1.1842	0.4997	0.8302	156.7694
Core duplicate	Rupert	Labtium	705P	509	0.6909	0.4106	0.8935	92.6319
Crush Duplicate	Rupert	Labtium	705P	-	-	-	-	-
Pulp Duplicate	Rupert	Labtium	705P	586	0.9367	1.0338	0.9356	68.7483
Lab Duplicate	Labtium	Labtium	705P	439	0.6033	0.5670	0.9681	57.5301
Core duplicate	Rupert	CRS	PAL1000	39	8.0625	2.4400	-1.0000	176.4923
Crush Duplicate	Rupert	CRS	PAL1000	160	1.0592	2.8983	-0.5653	48.3588
Pulp Duplicate	Rupert	CRS	PAL1000	-	-	-	-	-
Lab Duplicate	CRS	CRS	PAL1000	677	3.0666	3.0817	0.9969	28.5316

Table 11.12 Pahtavaara Gold Deposit umpire Assay Data Pairs

Duplicate Type	Original laboratory	Umpire Laboratory	Assay Method Original	Assay Method Check	Total number of pairs	Au1 Mean (g/t)	Au 2 Mean (g/t)	Corr. Coeff.	Mean AMPRD
Umpire	ALS	Labtium	AA26	705P	154	2.1722	2.1362	0.8645	52.0262
Umpire	ALS	CRS/MS Anal.	AA26	FAS221	334	0.4515	0.4816	0.7842	54.4268
Umpire	ALS	CRS	AA15	PAL1000	558	1.1889	1.1213	0.9411	38.6668
Umpire	Labtium	ALS	705P	AA26	143	1.5021	0.9971	0.7123	58.7376
Umpire	CRS	ALS	PAL1000	AA15	134	1.2391	1.0356	0.9387	40.9072

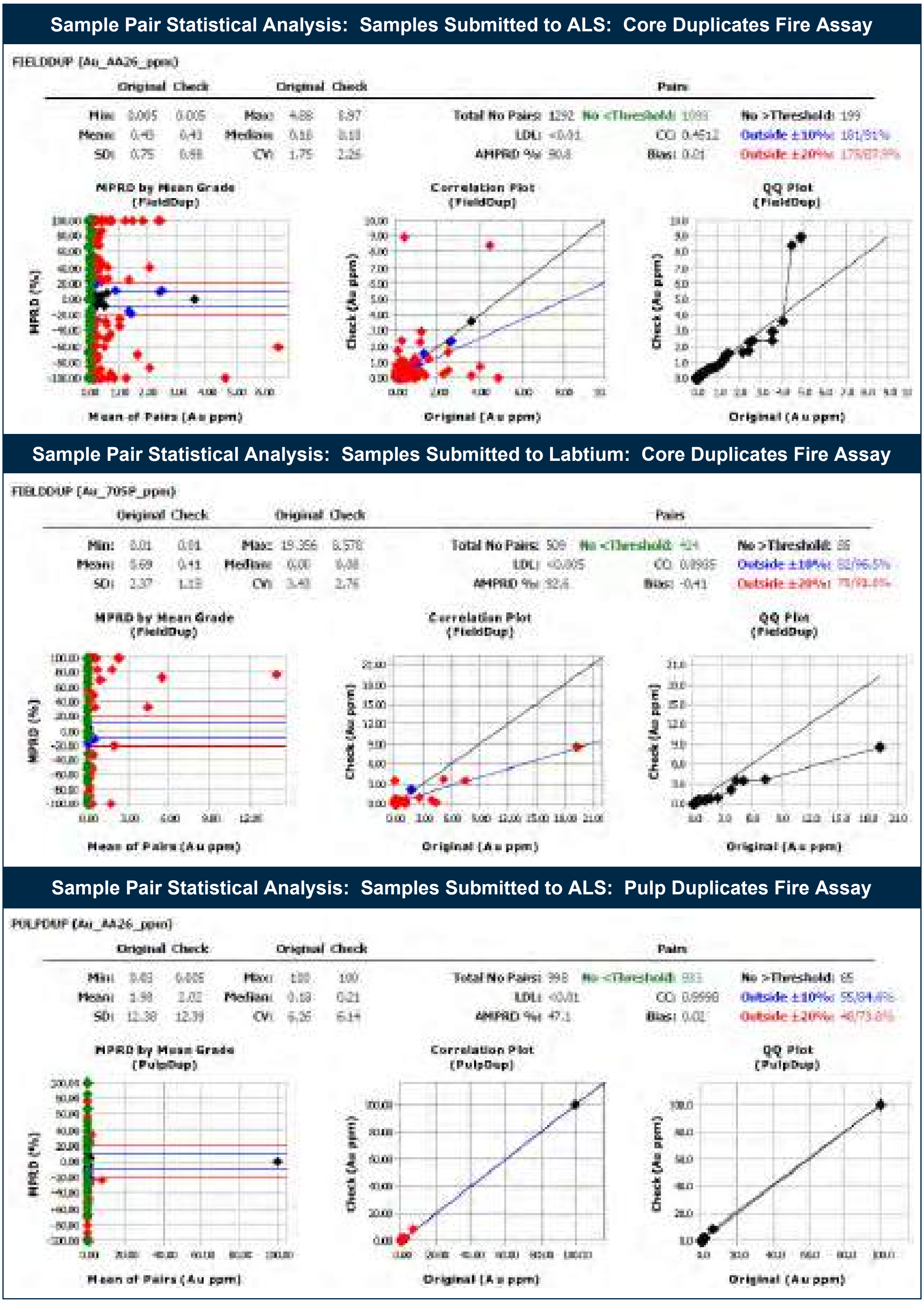
11.2.3 Discussion (Pahtavaara)

The review of the CRM submitted by Rupert Resources and the internal standards submitted by ALS and CRS have shown that the use of CRM which have an expected value based on multiple fire assay/AAS analyses with an assay technique that only determines the cyanide extractable gold content of the material has generally resulted in an apparent ‘under-call’. It is considered that this is not an issue of lab performance or sample preparation, but rather is due to the fact that the assay technique is only able to determine the cyanide-soluble portion of the CRM. Neither is it a sample size issue as the CRM material is as fine as 20 µm.

When the sample pair data is reviewed the typical pattern of a reduction in the level of variability as the particle size of the source material is reduced is noted from quarter core samples to pulp sachet samples. The levels of variability for much of the datasets are typical for gold deposits.

As such, it appears that the key areas of focus in terms improving the quality of sampling and assaying at Pahtavaara are the sample preparation flowsheet and the assay method (for example using a standard 50 gm fire assay/AAS analytical approach, with check screen fire assays for high grade samples etc).

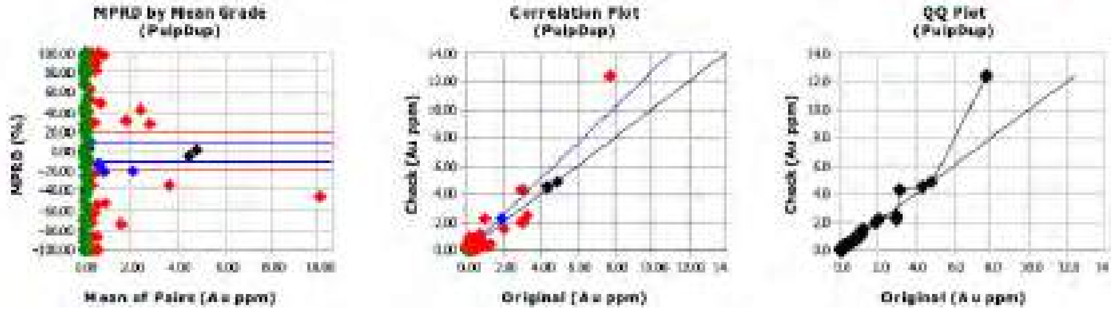
Figure 11.7 Performance of Duplicate Assays for Main Laboratories ALS and Labtium, Fire Assays AA26 and 705P



Sample Pair Statistical Analysis: Samples Submitted to Labtium: Pulp Duplicates Fire Assay

PULPDUP (Au_705P_ppm)

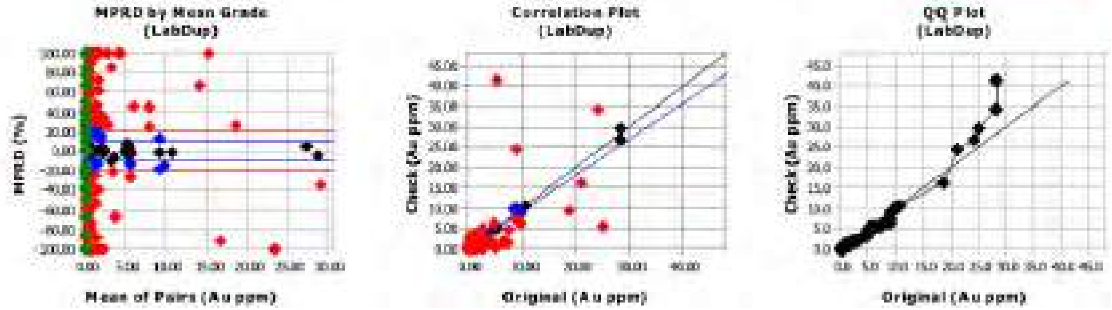
	Original	Check		Original	Check		Pairs				
Min:	0.023	0.032	Max:	7.722	12.348	Total No Pairs:	586	No <Threshold:	536	No >Threshold:	50
Mean:	0.94	1.03	Median:	0.56	0.38	LDL:	<0.005	CC:	0.9558	Outside ±10%:	47/94%
SD:	1.45	1.97	CV:	1.55	1.91	AMPRD %:	60.7	Bias:	0.1	Outside ±20%:	45/90%



Sample Pair Statistical Analysis: Samples Submitted to ALS: Lab Duplicates Fire Assay

LABDUP (Au_AA26_ppm)

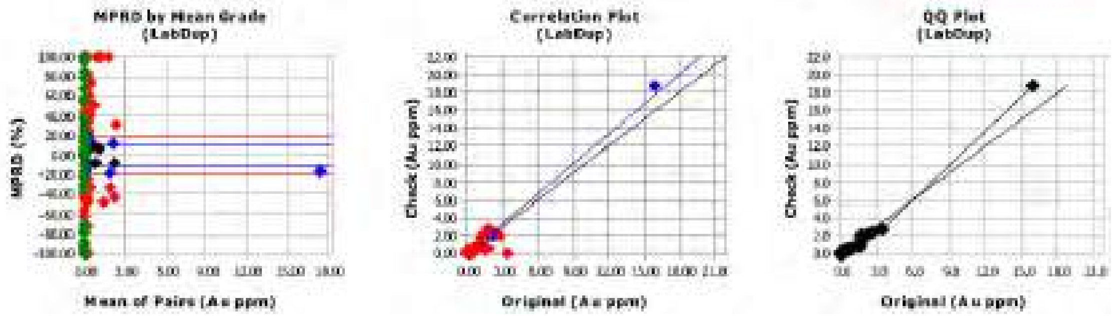
	Original	Check		Original	Check		Pairs				
Min:	0.005	0	Max:	28.2	41.3	Total No Pairs:	1050	No <Threshold:	814	No >Threshold:	236
Mean:	2.14	2.1	Median:	0.58	0.6	LDL:	<0.01	CC:	0.7665	Outside ±10%:	187/79.2%
SD:	4.26	4.92	CV:	1.99	2.34	AMPRD %:	55.5	Bias:	-0.02	Outside ±20%:	161/68.2%



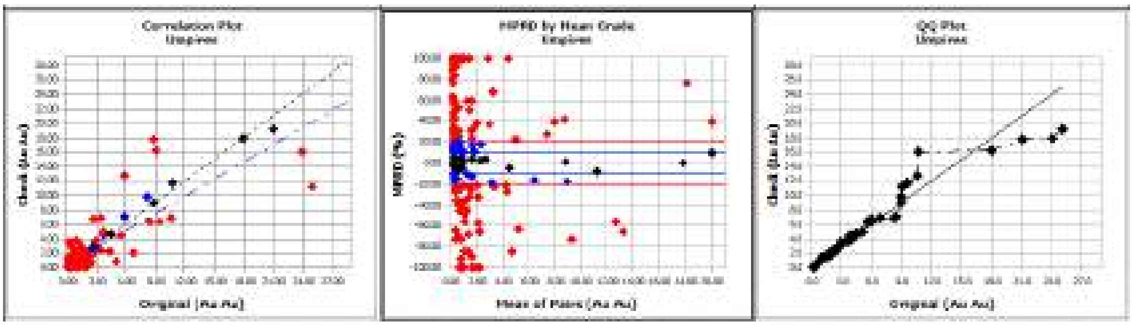
Sample Pair Statistical Analysis: Samples Submitted to Labtium: Lab Duplicates Fire Assay

LABDUP (Au_705P_ppm)

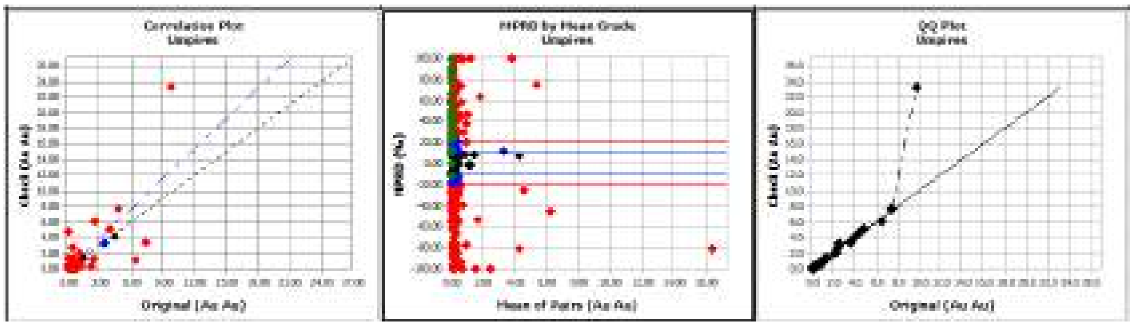
	Original	Check		Original	Check		Pairs				
Min:	0.01	0.01	Max:	15.934	18.746	Total No Pairs:	439	No <Threshold:	353	No >Threshold:	86
Mean:	0.6	0.57	Median:	0.17	0.14	LDL:	<0.005	CC:	0.9681	Outside ±10%:	73/84.9%
SD:	1.8	2.07	CV:	2.98	3.64	AMPRD %:	57.5	Bias:	-0.06	Outside ±20%:	60/69.8%



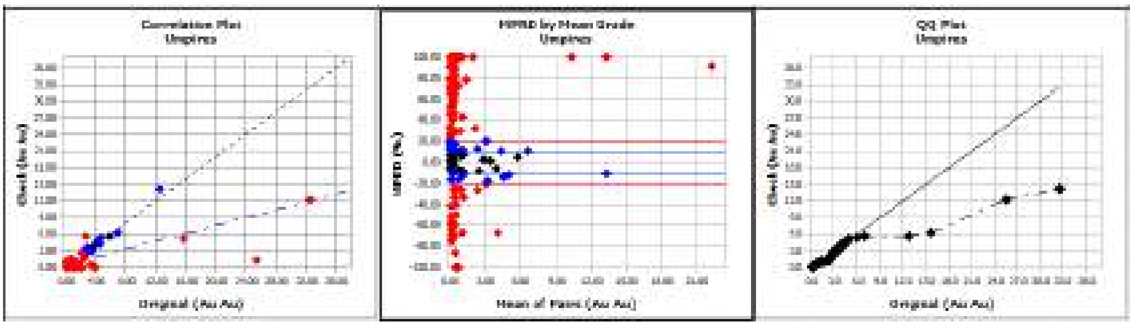
Sample Pair Statistical Analysis: Umpires: Original ALS AA26, Check Labtium 705P



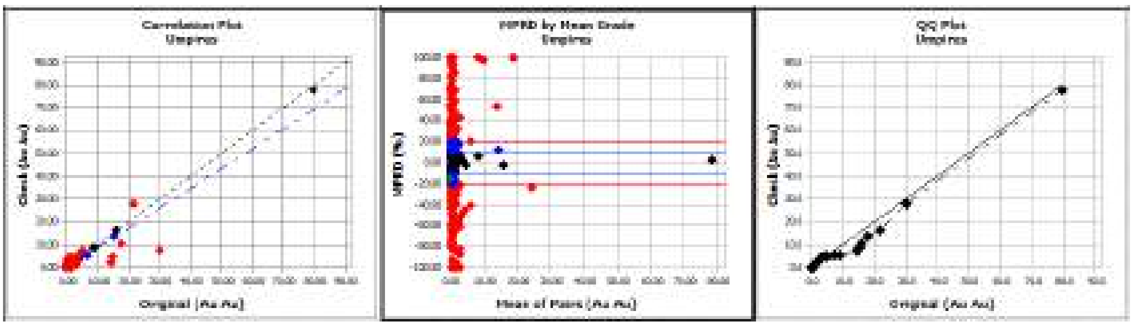
Sample Pair Statistical Analysis: Umpires: Original ALS AA26, Check MS Analytical FAS221



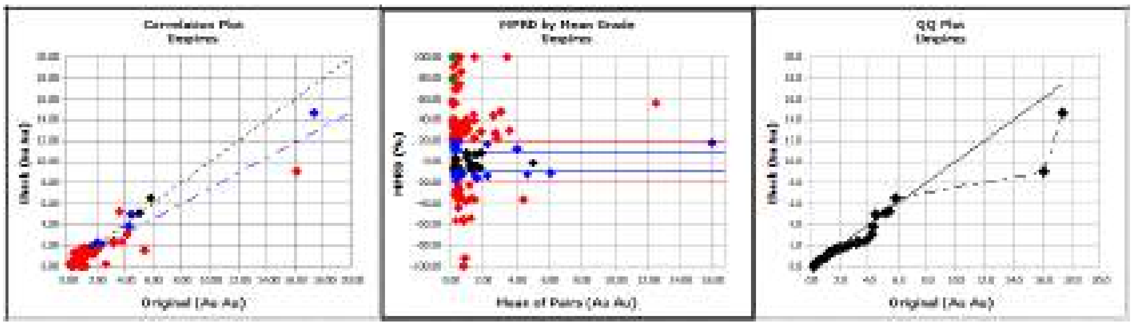
Sample Pair Statistical Analysis: Umpires : Original Labtium 705P Check ALS AA26



Sample Pair Statistical Analysis: Umpires: Original ALS AA15, Check CRS PAL1000



Sample Pair Statistical Analysis: Umpires: Original CRS PAL1000, Check ALS AA15



11.3 Assay Quality Control - Ikkari

Analysis of internationally accredited assay standards or certified reference material ("CRM") has been carried out.

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.
- Barren samples ("blanks") submitted by both Rupert Resources and the assay laboratory.

11.3.1 QA/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 and includes the following datasets:

Blanks:

- Submitted by Rupert.
- Internal ALS blanks.
- Eurofins Labtium blanks.
- MSA blanks.

CRM (Standards):

- Submitted by Rupert.
- ALS internal CRM.
- Eurofins Labtium CRM.
- MSA internal CRM.

Data Pairs:

- Core duplicates (quarter core pairs).
- Lab duplicate (duplicate samples taken after pulverised to >85% <75 µm).

- Pulp duplicates (duplicates samples taken from within one pulp sachet).
- 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 has been using and continues to use is quartz gravel provided by Sibelco Nordic/Nilsiä kvartsi.

Table 11.13 and Figure 11.8 and Figure 11.9 summarise the results of assaying blank samples. For the great majority of analyses, the blanks returned less than detection limit results.

Table 11.13 Ikkari Gold Deposit Blanks

Standard	Assay Method	Laboratory	Number	Expected Value	Mean	% Bias	% in Tolerance
BLK-CO01	Au-AA26-ppm	ALS	3568	0.010	0.0060	-40.1205	100
BLK-CO01	Au-ICP24-ppm	ALS	18	0.001	0.0016	55.5556	100
BLK-CO01	Au-705P-ppm	Labtium	373	0.020	0.0119	18.7131	100
BLK-CO01	Au-FAS121-ppm	CRS/MSA	92	0.005	0.0029	-42.5000	100
BLK-ALS	Au-AA26-ppm	ALS	2,049	0.010	0.0060	-39.8487	100
BLK-ALS	Au-GRA22-ppm	ALS	7	0.025	0.0250	0	100
BLK-ALS	Au-ICP24-ppm	ALS	49	0.001	0.0007	-28.5714	100
BLK-ALS	Au-ICP27-ppm	ALS	6	0.010	0.0050	-50.0000	100
SAUNA	Au-705P-ppm	Labtium	34	0.020	0.0109	9.1176	100
SOKEA	Au-705P-ppm	Labtium	75	0.020	0.0109	8.6667	100
BLK-MSA	Au-FAS121-ppm	CRS/MSA	55	0.010	0.0025	-75.0000	100

Figure 11.8 Rupert Company Blank (BLK-CO01) Performance

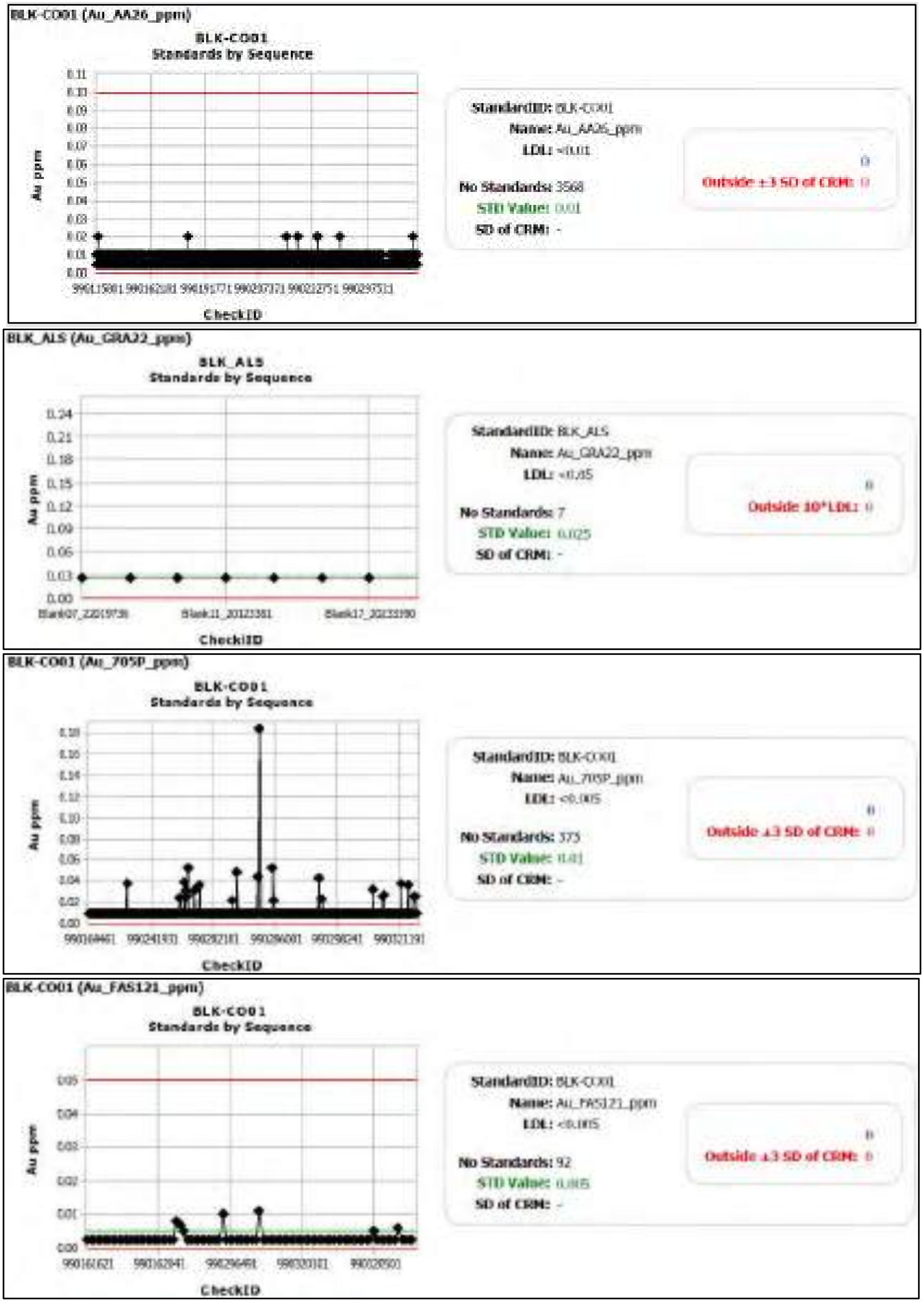


Figure 11.9 ALS Internal Blank (BLK-ALX) Performance

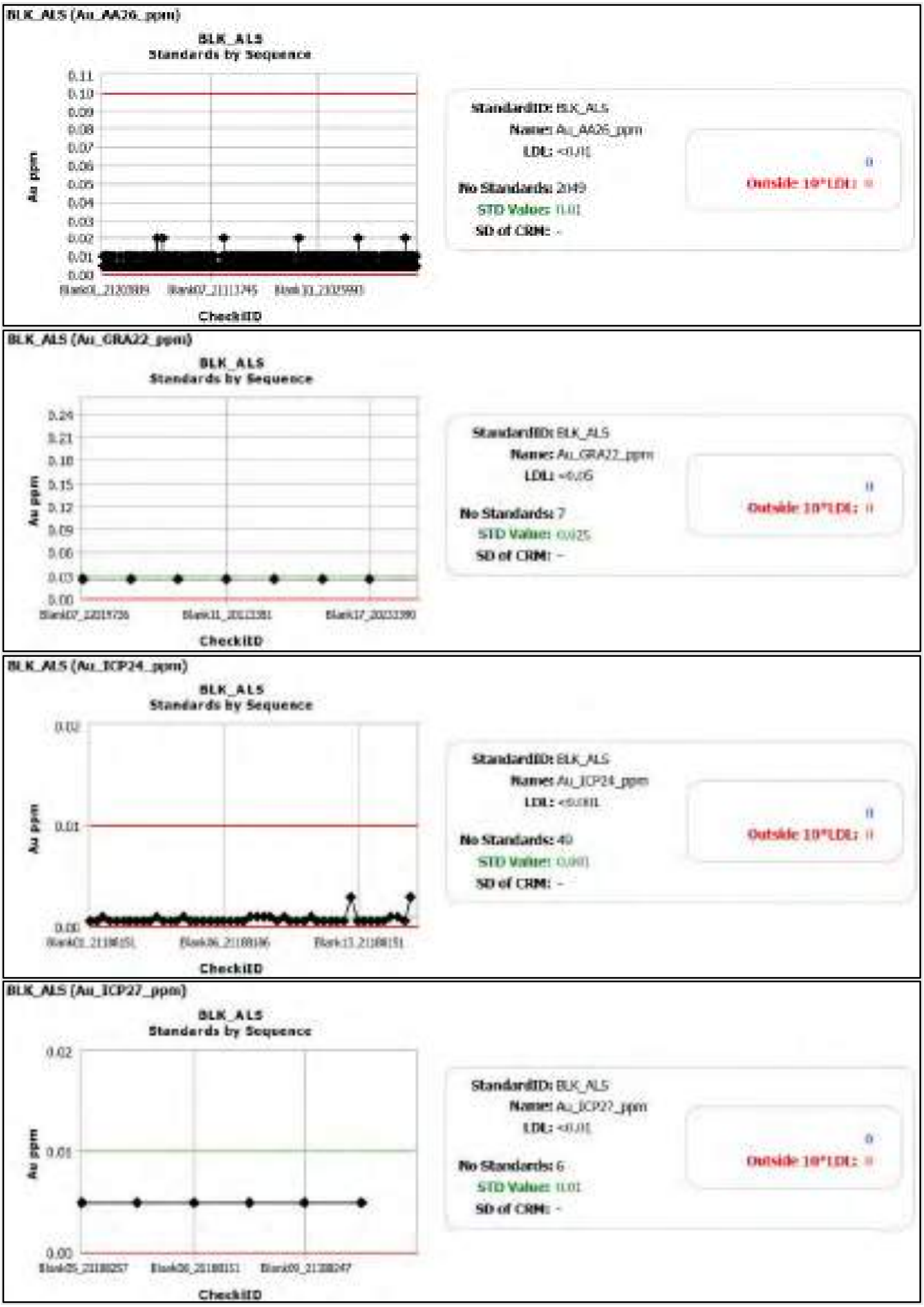


Figure 11.10 Labtium Internal Blanks (Sauna_Labtium and SOKEA_Labtium Performance

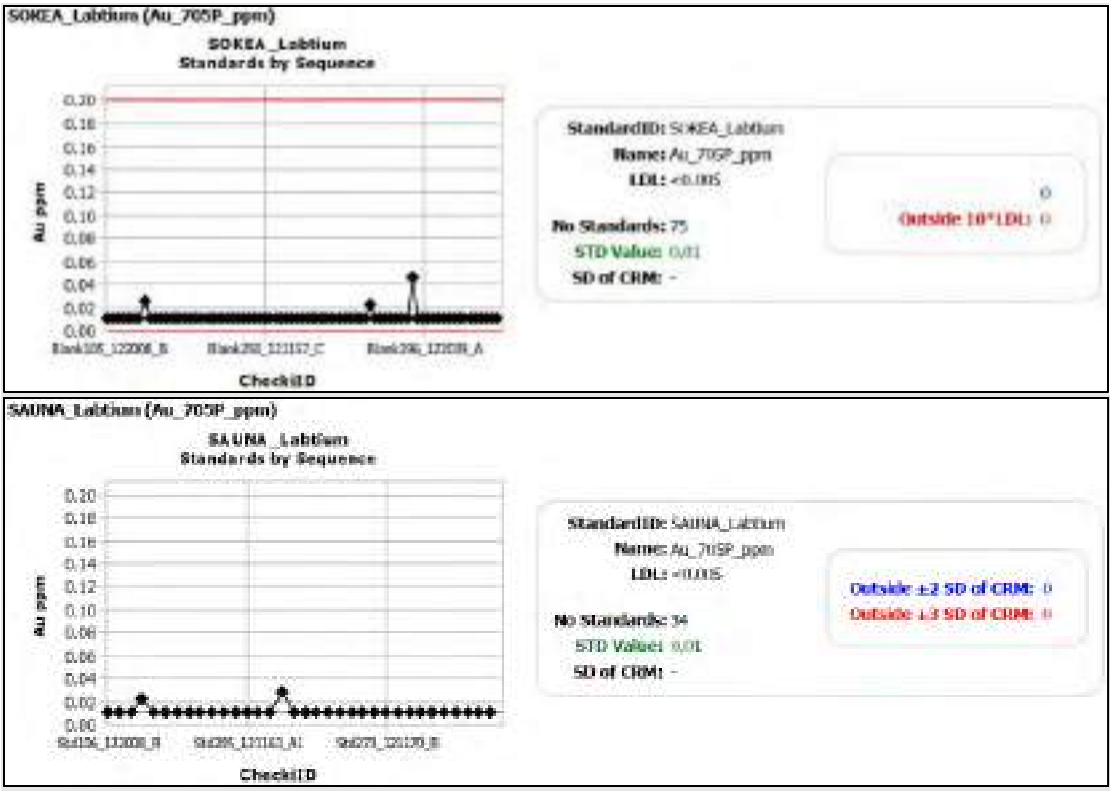


Figure 11.11 MSA Internal Blank (BLK-MSA) Performance

CRM Submitted by Rupert Resources

Rupert Resources routinely submitted accredited CRM. Rupert Resources has been using gold certified reference materials produced by Geostats Pty Ltd. These CRM's (G912-3, G915-2, G915-4, G915-6, G314-2, G315-7, G398-4, G312-4, G320-10 and G917-4) have been selected to represent four different gold grades. Rupert has also used CRM's prepared by CDN Resource Laboratories Ltd (CDN-GS-3H, CDN-GS-3K, CDN-GS-P7B and CDN-GS-P7H).

Table 11.14 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	2	5.30	5.135	-3.1132	1.7901	100
G314-2	Au-AA26	54	0.99	0.9817	-0.8418	2.5706	100
G315-7	Au-AA26	48	0.30	0.2931	-2.2917	2.6478	100
G320-10	Au-AA26	15	0.65	0.65	0	2.9074	100
G398-4	Au-AA26	15	0.660	0.6460	-2.1212	3.1399	100
G912-3	Au-AA26	983	2.090	2.0869	-0.1494	2.8752	100
G915-2	Au-AA26	1,043	4.980	5.0231	0.8664	2.1397	100
G915-4	Au-AA26	330	9.160	9.0340	-1.3752	1.7943	100
G915-6	Au-AA26	933	0.670	0.6527	-2.5835	3.9521	100
G917-4	Au-AA26	6	5.100	5.1000	0	1.4881	100
CDN-GS-3H	Au-AA26	14	3.040	3.0179	-0.7284	2.5870	100
CDN-GS-3K	Au-AA26	11	3.190	3.1464	-1.3679	2.7535	100
CDN-GS-P7B	Au-AA26	1	0.710	0.6900			100
CDN-GS-P7H	Au-AA26	26	0.799	0.8019	0.3658	2.2589	100

Table 11.15 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	10	0.65	0.6559	0.9077	3.0682	100
G912-3	Au-705P	99	2.09	2.1278	1.8075	2.204	100
G915-2	Au-705P	89	4.98	5.0967	2.3426	2.2473	100
G915-4	Au-705P	92	9.16	9.2702	1.2036	1.9135	100
G915-6	Au-705P	71	0.67	0.6713	0.1976	4.4010	100

Table 11.16 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	2.0367	-2.5489	3.7479	100
G915-2	Au-FAS121	24	4.98	4.9709	-0.1824	2.6384	100
G915-4	Au-FAS121	24	9.16	8.8343	-3.5558	2.6133	100
G915-6	Au-FAS121	21	0.67	0.6293	-6.0768	3.4451	100

Figure11.12 Rupert Resources CRM's Performance in ALS

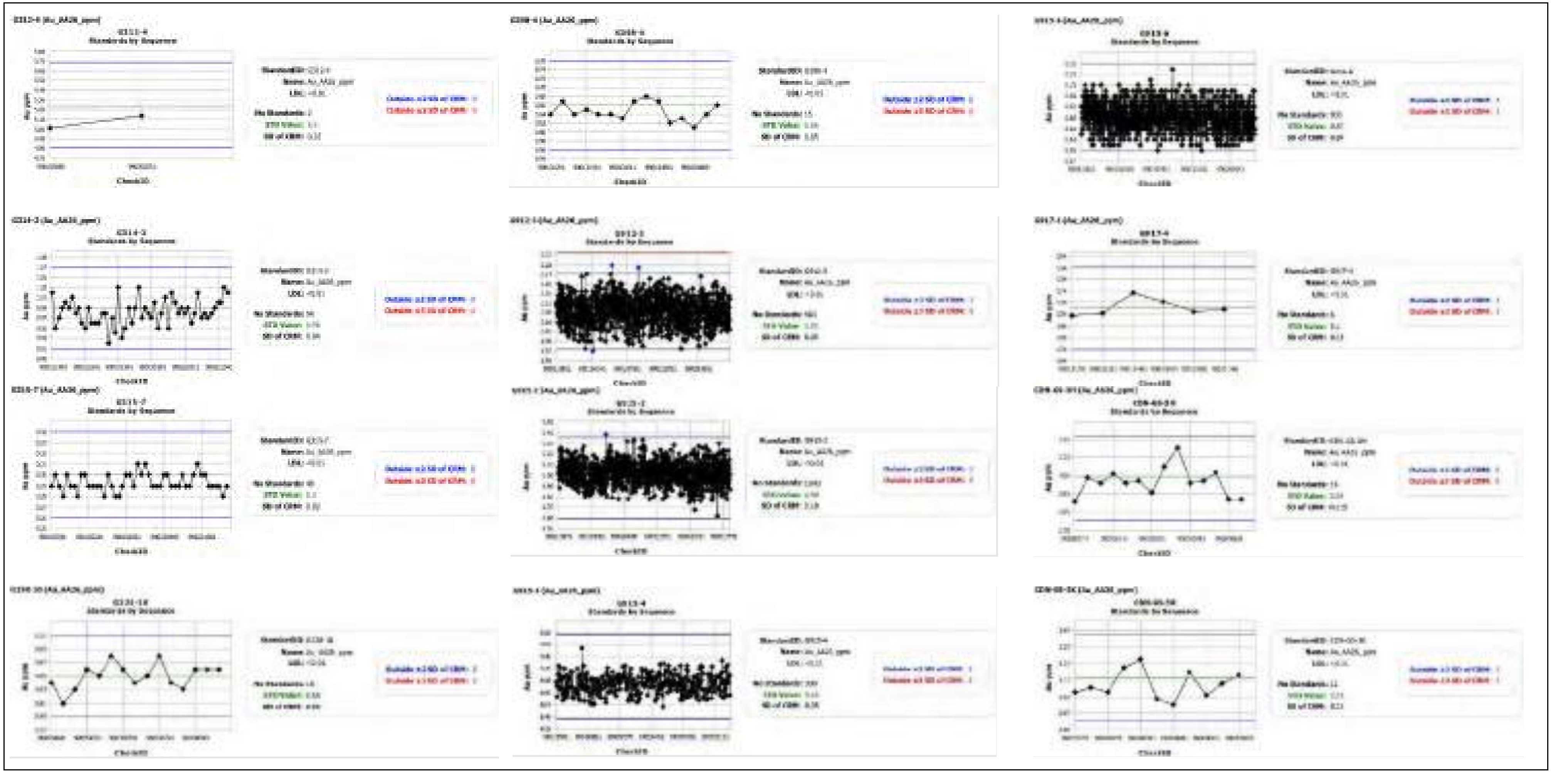


Figure11.13 Rupert Resources CRS's Performance in Eurofins Labtium

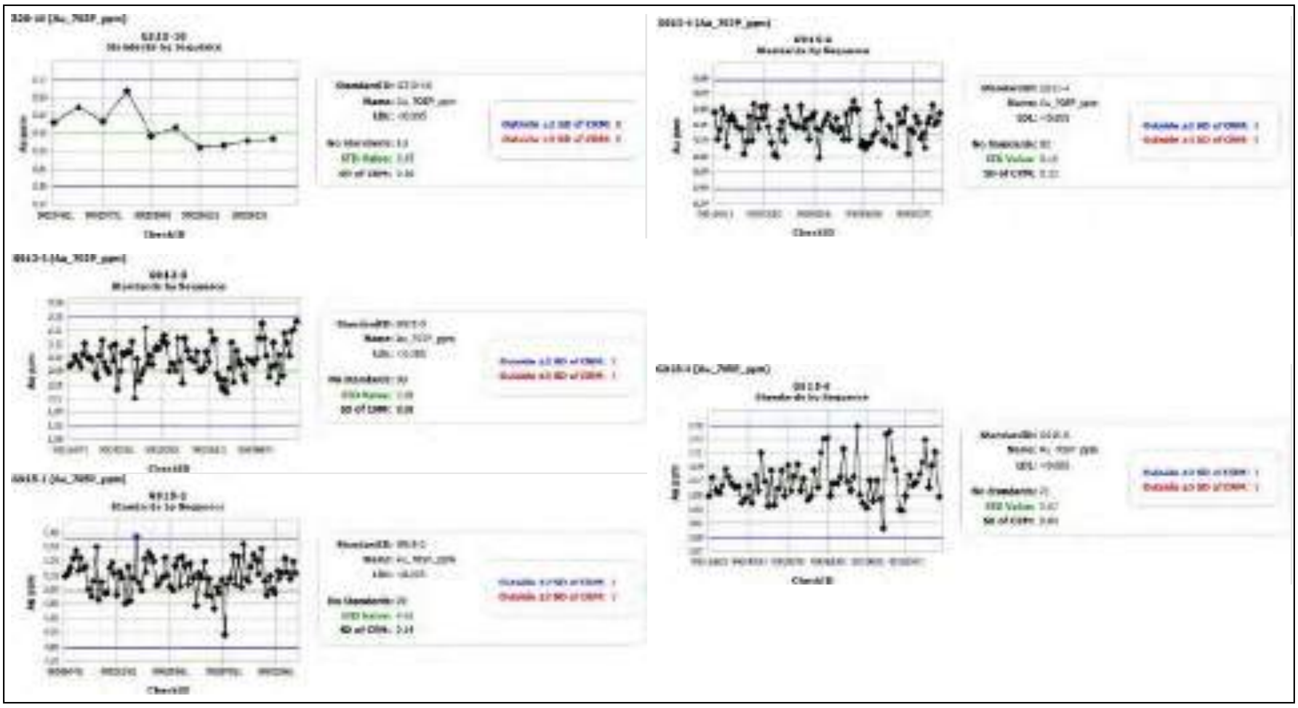
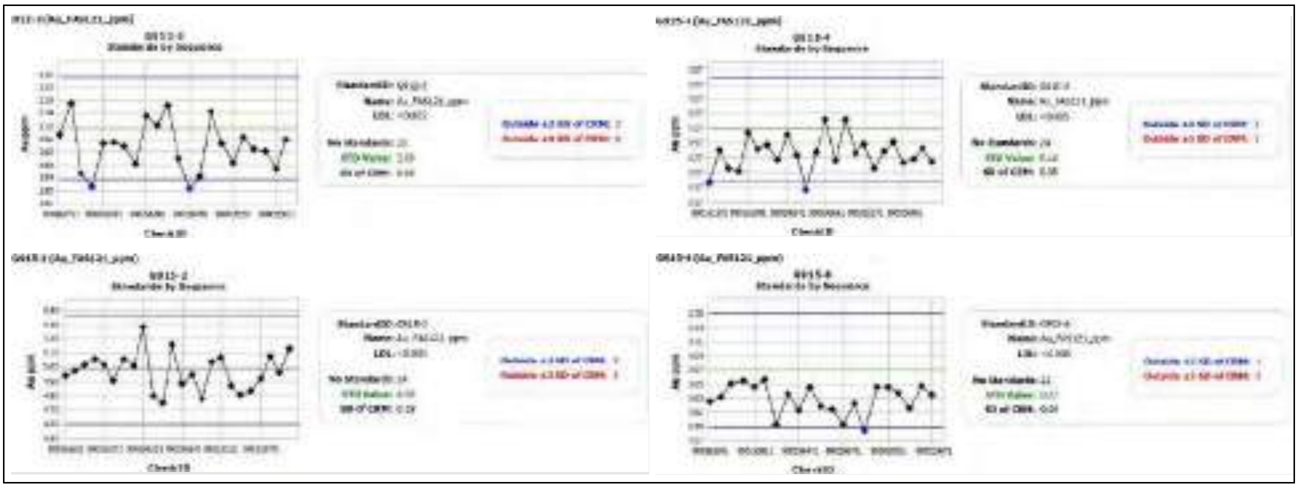


Figure11.14 Rupert Resources CRM's Performance in MSA Laboratories



Laboratories' Internal CRM

ALS, Eurofins Labtium and MSA as part of their standard QAQC procedures routinely analyse CRM prepared by independent suppliers. Rupert has obtained all the available internal CRM analytical results and statistical analysis has been carried out on the gold data.

Table 11.17 and Table 11.18 summarise the results of the analytical performance on these internally submitted CRM. The assay method used for the different CRM is also noted in these tables.

Table 11.17 Ikkari Gold Deposit ALS Internal Standards

Standard	Laboratory	Assay Method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
BLK_ALS	ALS	Au-AA26	2,049	0.0100	0.0060	-39.8487	35.2138	100
BLK_ALS	ALS	Au-GRA22	7	0.0100	0.0250	0	0	100
BLK_ALS	ALS	Au-ICP24	49	0.0100	0.0007	-28.5714	72.8583	100
BLK_ALS	ALS	Au-ICP27	6	0.0100	0.0050	-50.0000	0	100
G312-4_ALS	ALS	Au-AA26	14	5.3000	5.3207	0.3908	2.3995	100
G313-5	ALS	Au-AA26	277	7.0700	7.0380	-0.4524	2.6535	100
G398-10_ALS	ALS	Au-AA26	682	4.0700	4.0952	0.6182	2.3355	100
G913-10_ALS	ALS	Au-AA26	69	7.0900	7.1010	0.1554	2.7033	100
G914-10_ALS	ALS	Au-AA26	280	10.2600	10.1492	-1.0801	2.4975	100
JK-17_ALS	ALS	Au-AA26	32	1.9980	1.9659	-1.6047	1.5816	100
JK-17_ALS	ALS	Au-GRA22	3	1.9980	2.0200	1.1011	1.7849	100
KIP-19_ALS	ALS	Au-AA26	10	2.4300	2.500	2.8807	1.3597	100
KIP-19_ALS	ALS	Au-ICP24	15	2.4300	2.4207	-0.3841	1.0055	100
LEA-16_ALS	ALS	Au-AA26	14	0.5010	0.5057	0.9410	2.1542	100
OREAS-214_ALS	ALS	Au-AA26	9	3.0300	3.0067	-0.7701	0.9978	100
OREAS-219_ALS	ALS	Au-AA26	511	0.7600	0.7547	-0.7004	4.9605	99.8
OREAS-221_ALS	ALS	Au-AA26	11	1.0600	1.0673	0.6861	3.1661	100
OREAS-226_ALS	ALS	Au-AA26	216	5.4500	5.4555	0.1011	1.8381	100
OREAS-228b_ALS	ALS	Au-AA26	22	8.5700	8.5300	-0.4667	2.5758	100
OREAS-250_ALS	ALS	Au-AA26	172	0.3030	0.3115	2.8091	2.9054	100
OxA131_ALS	ALS	Au-AA26	149	0.0770	0.0746	-3.1640	8.4653	100
OxC129_ALS	ALS	Au-AA26	44	0.2050	0.2002	-2.3282	15.7746	97.73
OxD151_ALS	ALS	Au-AA26	211	0.4300	0.4242	-1.3557	2.7766	93.39
OxF125_ALS	ALS	Au-AA26	46	0.8060	0.8017	-0.5286	1.8650	82.61
OxF162_ALS	ALS	Au-AA26	68	0.8320	0.8154	-1.9902	1.7399	100
OxI121_ALS	ALS	Au-AA26	172	1.8340	1.8126	-1.1691	1.7744	100
OxP116_ALS	ALS	Au-AA26	263	14.9200	14.9878	0.4546	2.0799	73.00
OxP154_ALS	ALS	Au-AA26	11	15.2600	15.2364	-0.1549	1.7617	100
OxP158_ALS	ALS	Au-AA26	308	15.1500	15.2263	0.5036	2.3493	100
OxQ90_ALS	ALS	Au-GRA22	3	24.8800	25.4667	2.3580	2.1627	100
PMP-18	ALS	Au-AA26	601	0.3000	0.3023	0.7543	2.7189	99.30
SL76_ALS	ALS	Au-AA26	15	5.9600	5.9187	-0.6935	1.1296	100
TAZ-20	ALS	Au-AA26	424	0.3020	0.2994	-08653	3.1758	93.40

Table 11.18 Ikkari Gold Deposit Eurofins Labtium and MSA Laboratories Internal Standards

Standard	Laboratory	Assay Method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
SAUNA_Labtium	Labtium	Au-705P	34	0.010	0.0109	9.1176	34.3915	100
SOKEA_Labtium	Labtium	Au-705P	75	0.010	0.0109	8.6667	43.6467	100
BLK_MSA	MSA	Au-FAS121	55	0.010	0.0025	-75.0000	0	100
CDN-CM-27_MSA	MSA	Au-FAS121	2	0.636	0.6315	-0.7075	7.9501	100
CDN-CM-47_MSA	MSA	Au-FAS121	3	1.130	1.1027	-2.4189	3.6775	100
CDN-ME-1709_MSA	MSA	Au-FAS121	7	0.1789	0.1761	-1.5412	3.6415	100
OxB167_MSA	MSA	Au-FAS121	7	0.462	0.4604	-0.3401	3.1694	100
OxK160_MSA	MSA	Au-FAS121	9	3.674	3.7363	1.6966	1.8711	100
OxN155_MSA	MSA	Au-FAS121	3	7.776	7.7013	-0.9602	1.3252	100
OxQ115_MSA	MSA	Au-FAS425	4	25.220	25.3750	0.6146	0.8124	100

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 (Figure11.12 to Figure11.14 and Table 11.14 to Table 11.16).

Data Pairs

Rupert's 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 and assay method (Table 11.19). The different types of data pairs comprise the following:

- Field duplicates (quarter core pairs).
- Lab duplicates (two samples taken after pulverising sample material >85% <75 µm).
- Pulp duplicates (duplicates samples taken from within one pulp sachet).
- Umpire checks (Pulp split sent to second laboratory).

Table 11.19 Ikkari Gold Deposit Data Pairs

Duplicate Type	Submitted by	Laboratory	Assay Method	Total Number of Pairs	Au1 Mean (g/t)	Au2 Mean (g/t)	Corr. Coeff.	Mean AMPRD
Field duplicate	Rupert	ALS	Au-AA26	3,520	0.3928	0.38700	0.8809	43.5185
Field duplicate	Rupert	ALS	Au-ICP24	37	0.5689	0.4795	0.8772	51.651
Pulp duplicate	Rupert	ALS	Au-AA26	,3476	0.5505	0.5513	0.9870	26.7256
Pulp duplicate	Rupert	ALS	Au-ICP24	29	1.8811	1.8884	0.9997	52.9741

Pulp duplicate	Rupert	ALS	Au-ICP27	4	15.9425	15.4750	0.9883	8.5473
Lab duplicate	ALS	ALS	Au-AA26	3316	0.9512	0.9560	0.9758	28.3121
Lab duplicate	ALS	ALS	Au-GRA22	6	122.4167	120.3667	0.9276	5.9013
Lab duplicate	ALS	ALS	Au-ICP24	24	0.7797	0.8282	0.9648	45.7797
Lab duplicate	ALS	ALS	Au-ICP27	6	17.0583	16.5900	0.9649	6.9377
Field duplicate	Rupert	Labtium	Au-705P	378	0.5607	0.6167	0.6536	35.1796
Pulp duplicate	Rupert	Labtium	Au-705P	373	0.3825	0.3634	0.9946	16.1448
Lab duplicate	Labtium	Labtium	Au-705P	207	2.3893	2.3691	0.9844	13.1729
Field duplicate	Rupert	CRS/MSA	Au-FAS121	93	0.0619	0.0930	0.6077	59.5664
Lab duplicate	CRS/MSA	CRS/MSA	Au-FAS121	45	0.6260	0.6657	0.9935	19.9267

Table 11.20 Ikkari Gold Deposit Umpire Assay Data Pairs

Duplicate Type	Original laboratory	Umpire Laboratory	Assay Method Original	Assay Method Check	Total Number of Pairs	Au1 Mean (g/t)	Au 2 Mean (g/t)	Corr. Coeff.	Mean AMPRD
Umpire	ALS	Eurofins	Au-AA26	705P/704P	1,749	5.6439	5.9045	0.9387	17.4957
Umpire	ALS	Eurofins	AuSCR24_tot	705P/704P	455	7.394	7.5387	0.9825	11.8022
Umpire	ALS	MSA	Au-ICP24/27	Au-FAS121/425	184	9.9283	9.9790	0.5995	25.6106
Umpire	ALS	MSA	Au-AA26	Au-FAS121/425	496	5.12	4.1851	0.8298	24.1726

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

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

Samples Submitted to ALS

Samples submitted to ALS for data pair analysis have included the following sample types:

- Field duplicates (two separate quarter core samples from the same sample interval) (Figure11.15).
- Pulp duplicates (two sub-samples taken from the same pulp sachet) (Figure11.16).
- Lab Duplicates (two samples taken after pulverising sample material >85% <75 µm) (Figure11.17).

Figure11.15 Sample Pair Statistical Analysis: Samples Submitted to ALS: Field Duplicates

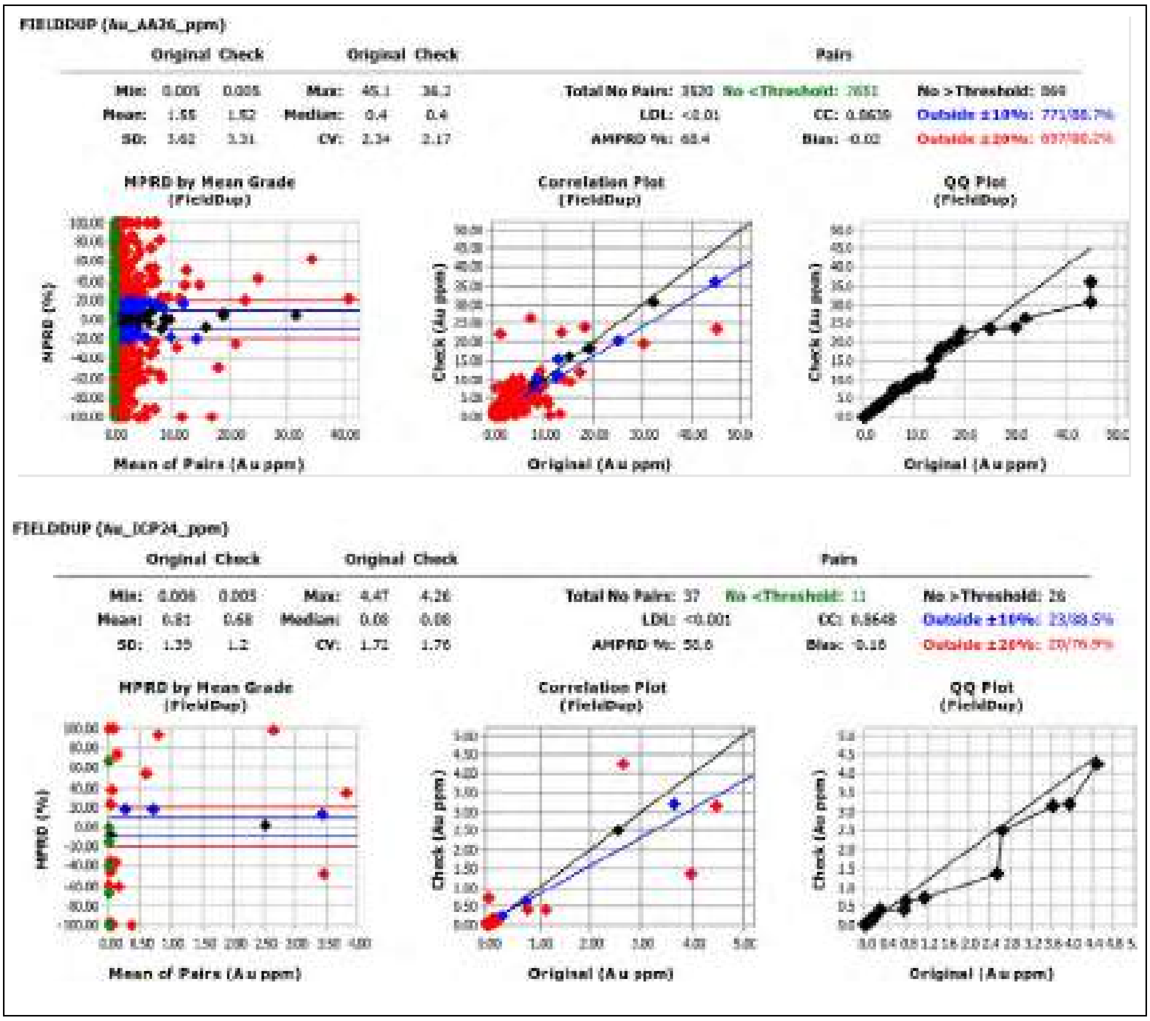
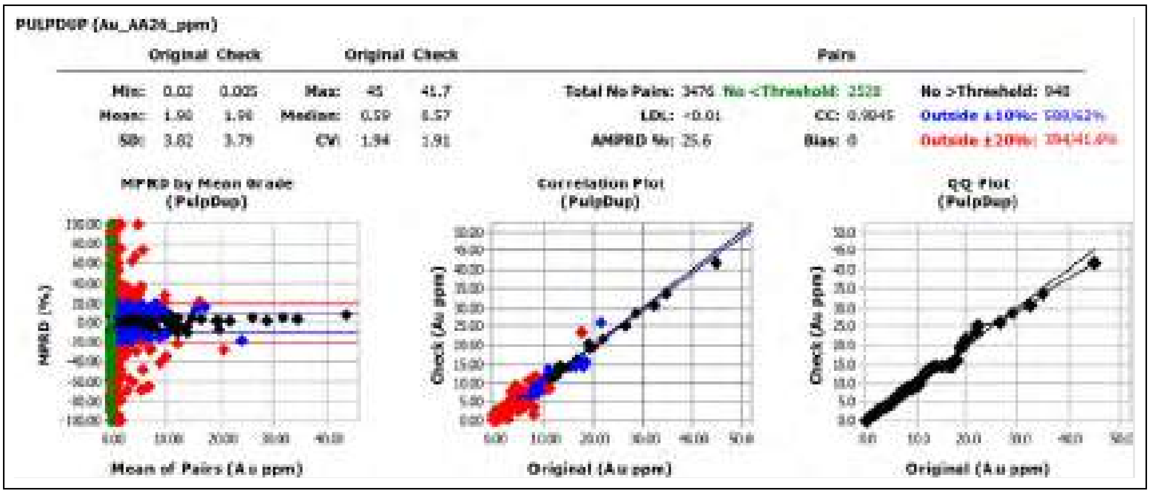


Figure11.16 Sample Pair Statistical Analysis: Samples Submitted to ALS: Pulp Duplicates



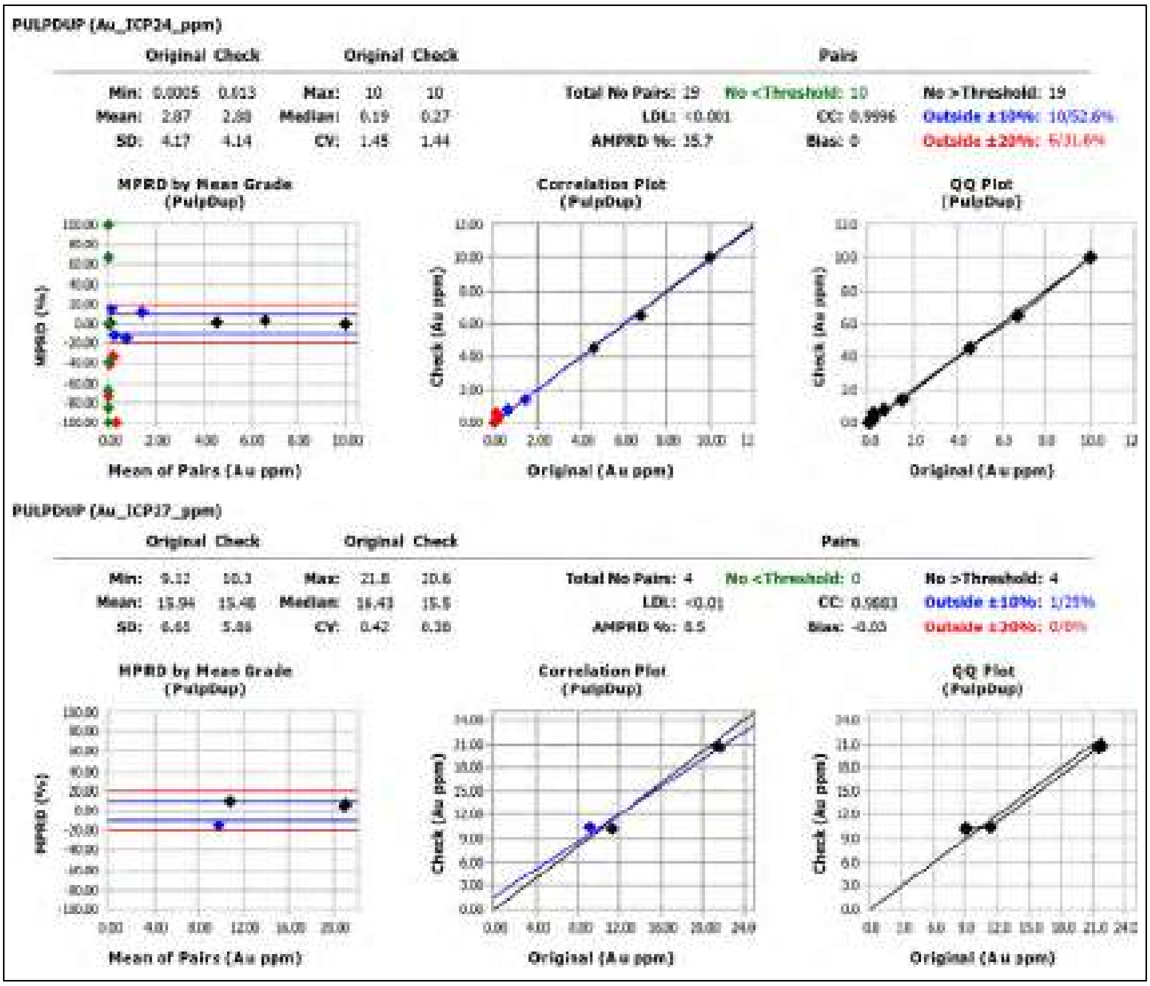
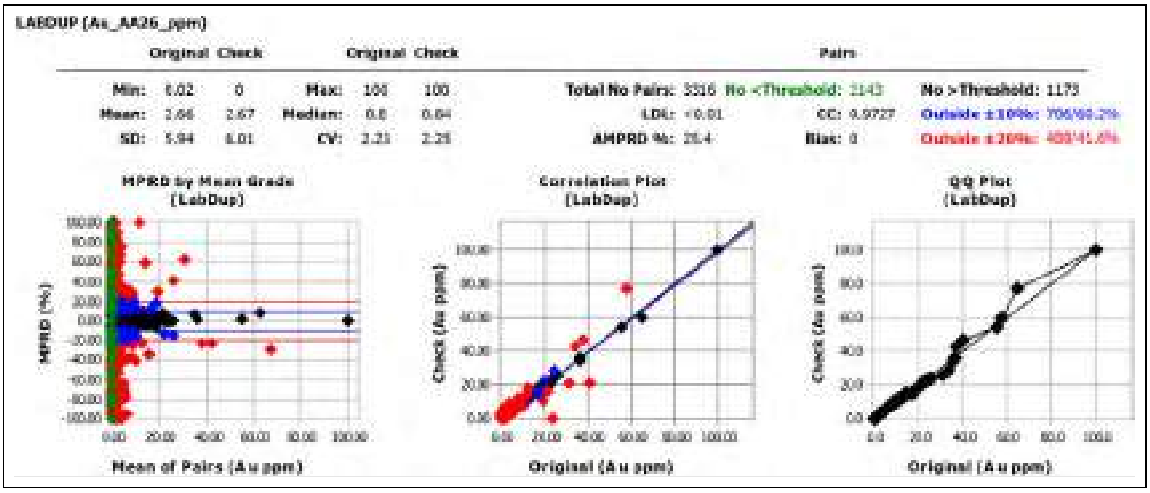
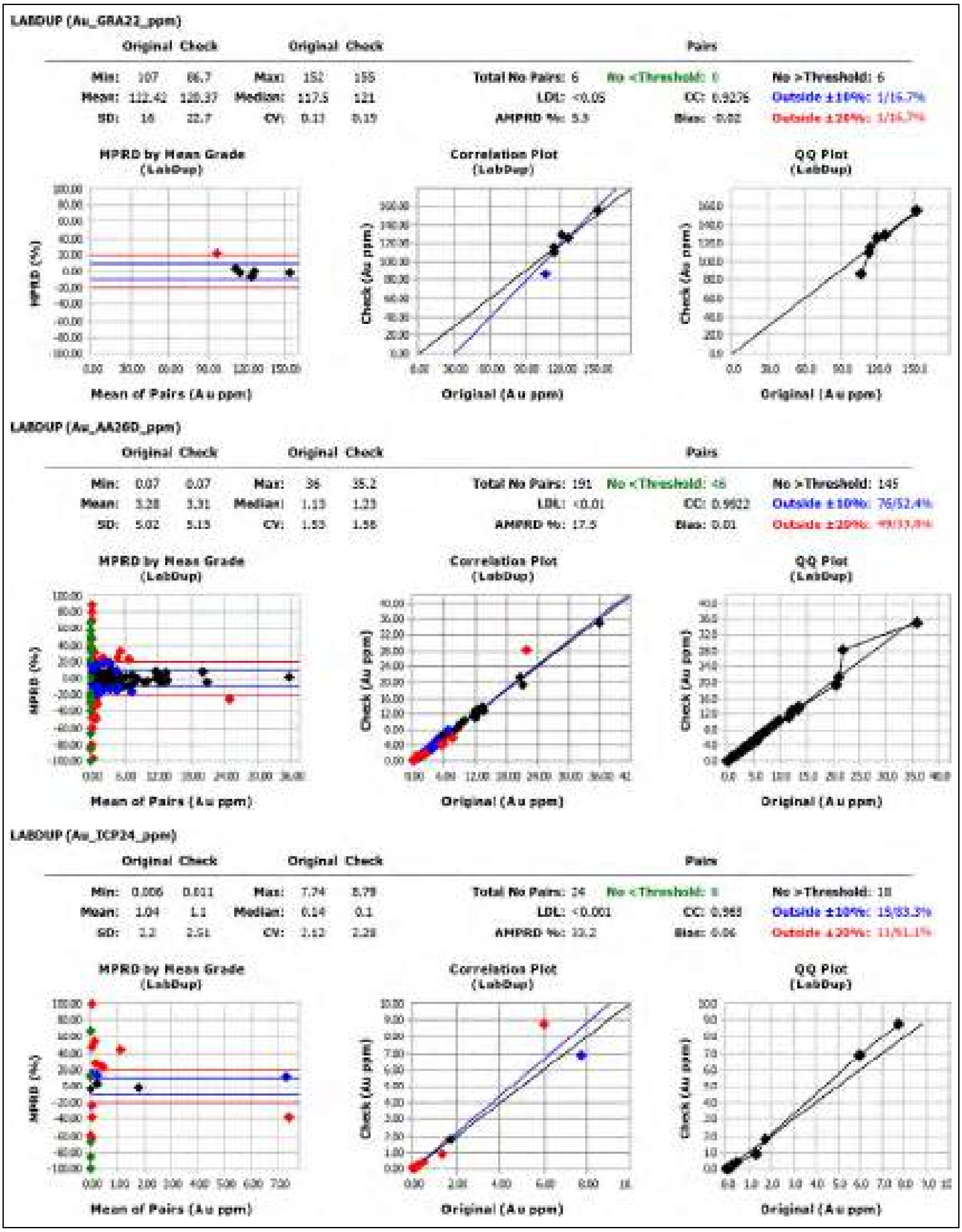
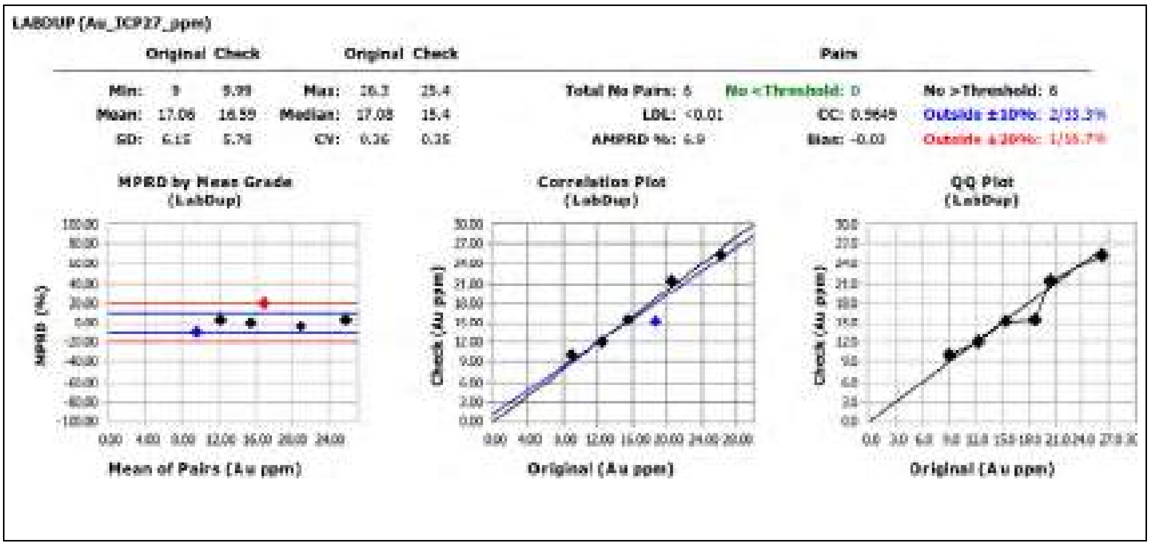


Figure11.17 Sample Pair Statistical Analysis: Samples Submitted to ALS: Laboratory Duplicates




LABDUP (Au_GRA22_ppm)
Original Check Original Check Pairs
Min: 107 86.7 Max: 152 155 Total No Pairs: 6 No <Threshold: 0 No >Threshold: 6
Mean: 122.42 120.37 Median: 117.5 121 LDL: <0.05 CC: 0.9276 Outside ±10%: 1/16.7%
SD: 16 22.7 CV: 0.13 0.19 AMPRD %: 5.5 Bias: -0.02 Outside ±20%: 1/16.7%
MPRD by Mean Grade (LabDup)
MPRD (%)
Mean of Pairs (Au ppm)
Correlation Plot (LabDup)
Check (Au ppm)
Original (Au ppm)
QQ Plot (LabDup)
Check (Au ppm)
Original (Au ppm)
LABDUP (Au_AA26D_ppm)
Original Check Original Check Pairs
Min: 0.07 0.07 Max: 36 35.2 Total No Pairs: 191 No <Threshold: 46 No >Threshold: 145
Mean: 3.28 3.31 Median: 1.15 1.23 LDL: <0.01 CC: 0.9922 Outside ±10%: 76/52.4%
SD: 5.02 5.13 CV: 1.53 1.56 AMPRD %: 17.5 Bias: 0.01
MPRD by Mean Grade (LabDup)
MPRD (%)
Mean of Pairs (Au ppm)
Correlation Plot (LabDup)
Check (Au ppm)
Original (Au ppm)
QQ Plot (LabDup)
Check (Au ppm)
Original (Au ppm)
LABDUP (Au_ICP24_ppm)
Original Check Original Check Pairs
Min: 0.006 0.011 Max: 7.74 8.79 Total No Pairs: 24 No <Threshold: 6 No >Threshold: 18
Mean: 1.04 1.1 Median: 0.14 0.1 LDL: <0.001 CC: 0.985 Outside ±10%: 15/83.3%
SD: 2.2 2.51 CV: 2.12 2.28 AMPRD %: 33.2 Bias: 0.06 Outside ±20%: 11/61.1%
MPRD by Mean Grade (LabDup)
MPRD (%)
Mean of Pairs (Au ppm)
Correlation Plot (LabDup)
Check (Au ppm)
Original (Au ppm)
QQ Plot (LabDup)
Check (Au ppm)
Original (Au ppm)

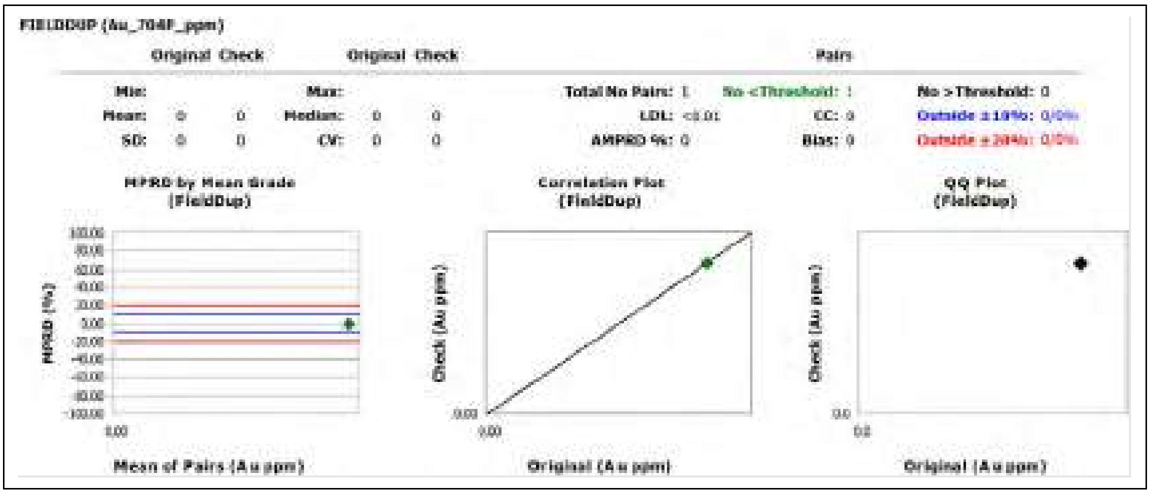


Samples Submitted to Eurofins Labtium

Samples submitted to Eurofins Labtium for data pair analysis have included the following sample types:

- Field duplicates (two separate quarter core samples from the same sample interval) (Figure11.18).
- Pulp duplicates (two sub-samples taken from the same pulp sachet) (Figure11.19).
- Lab Duplicates (two samples taken after pulverising sample material >85% <75 µm) (Figure11.20).

Figure11.18 Sample Pair Statistical Analysis: Samples Submitted to Labtium: Field Duplicates



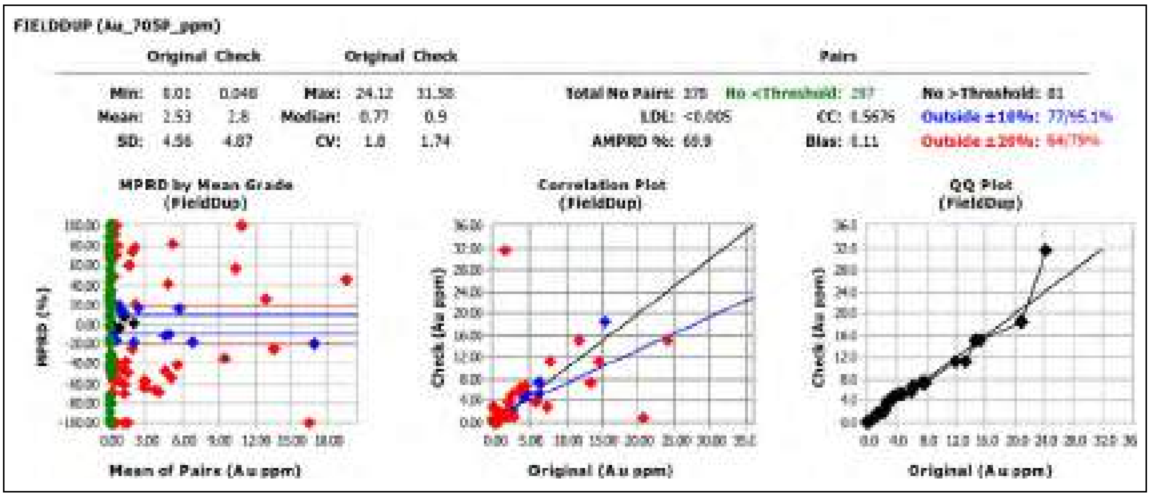


Figure11.19 Sample Pair Statistical Analysis: Samples Submitted to Labtium: Pulp Duplicates

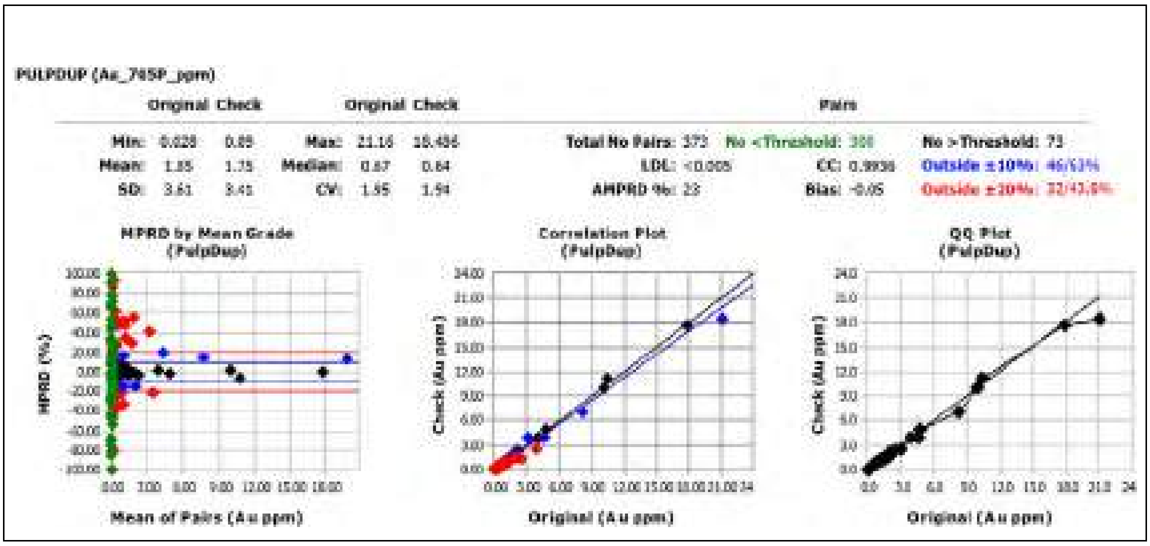
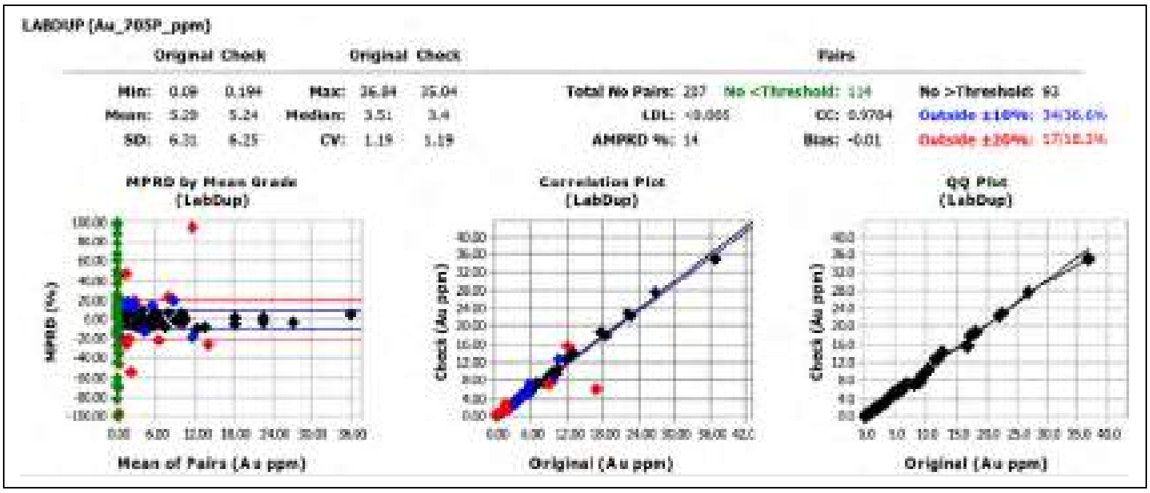


Figure11.20 Sample Pair Statistical Analysis: Samples Submitted to Labtium: Laboratory Duplicates



SAMPLES SUBMITTED TO CRS/MSA LABORATORIES

Samples submitted to CRS/MSA for data pair analysis have included the following sample types:

- Field duplicates (two separate quarter core samples from the same sample interval) (Figure11.21).
- Lab Duplicates (two samples taken after pulverising sample material >85% <75 microns) (Figure11.22).

Figure11.21 Sample Pair Statistical Analysis: Samples Submitted to MSA/CRS: Field Duplicates

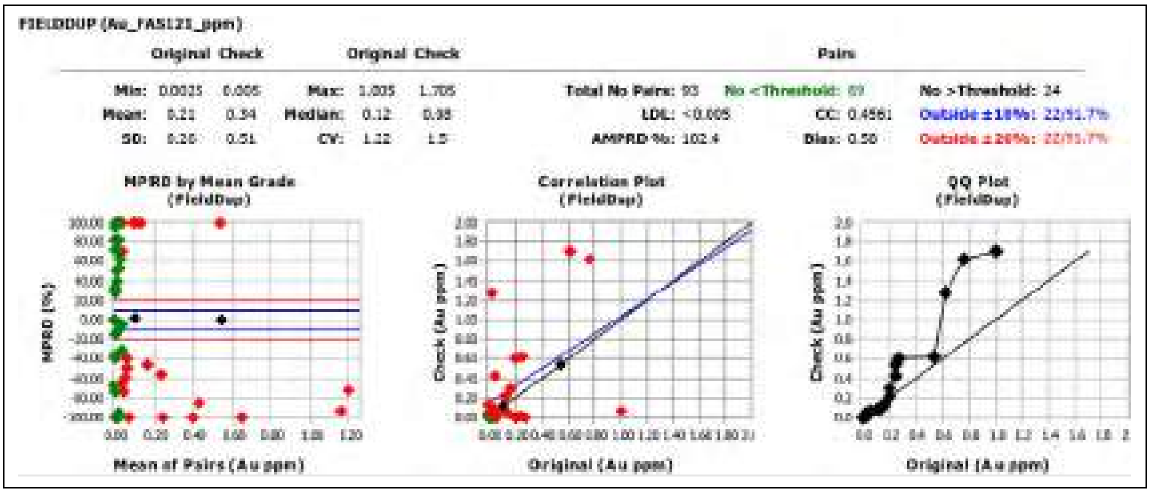
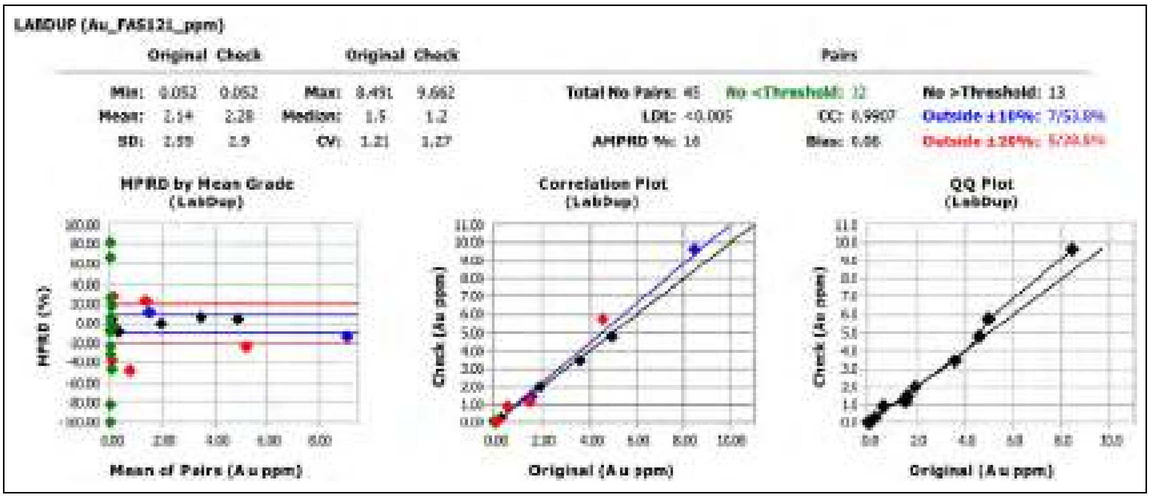


Figure11.22 Sample Pair Statistical Analysis: Samples Submitted to CRS/MSA: Laboratory Duplicates



UMPIRE CHECKS

ALS Minerals laboratory has been instructed to make 250 g extra split at pulverising stage, to be sent to second laboratory for laboratory check. Five percent of all samples have been sent to Eurofins Labtium or to MSA Labs for gold fire assay.

- Umpire checks (Pulverising stage split for samples originally assayed at ALS by fire assay AA26) (Table 11.21 and Figure 11.23 to Figure 11.28).

Table 11.21 Ikkari Gold Deposit Data Pairs

Duplicate Type	Original laboratory	Umpire Laboratory	Assay Method Original	Assay Method Check	Total Number of Pairs	Au1 Mean (g/t)	Au 2 Mean (g/t)	Corr. Coeff.	Mean AMPRD
Umpire	ALS	Eurofins	Au-AA26	705P/704P	1,749	5.6439	5.9045	0.9387	17.4957
Umpire	ALS	Eurofins	AuSCR24_tot	705P/704P	455	7.3940	7.5387	0.9825	11.8022
Umpire	ALS	MSA	Au-ICP24/27	Au-FAS121/425	184	9.9283	9.9790	0.5995	25.6106
Umpire	ALS	MSA	Au-AA26	Au-FAS121/425	496	5.1200	4.1851	0.8298	24.1726

Figure 11.23 Sample Pair Statistical Analysis: Umpires: Original ALS AA26, Check Labtium 705P/704P

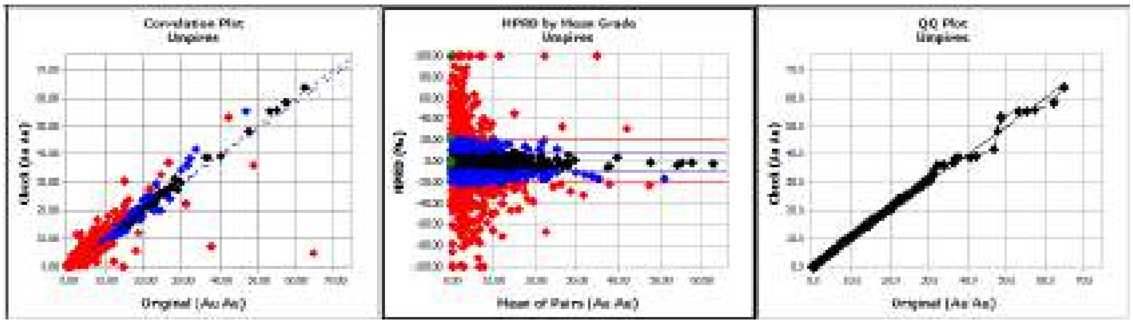


Figure 11.24 Sample Pair Statistical Analysis: Umpires: Original ALS SCR24, Check Labtium 705P/704PFigure 11.25Figure 11.26

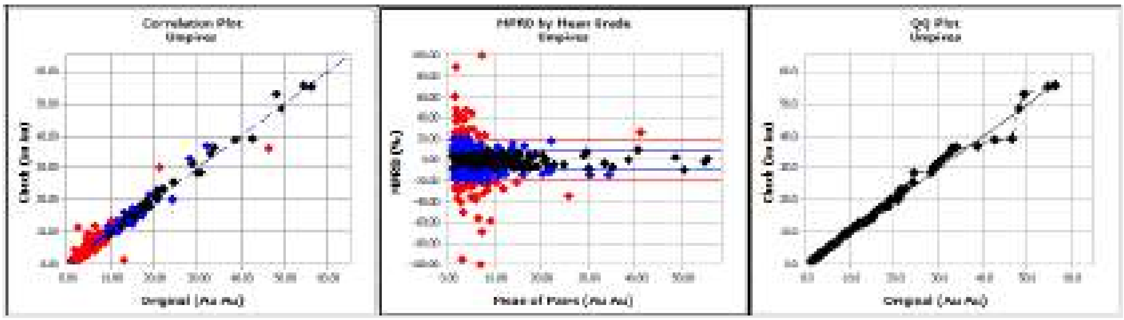


Figure 11.27 Sample Pair Statistical Analysis: Umpires: Original Als ICP24/ECP27, Check MSA FAS121/425

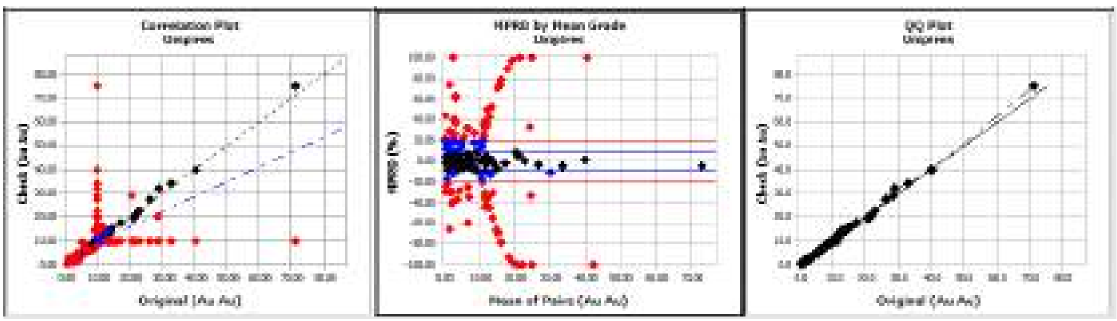
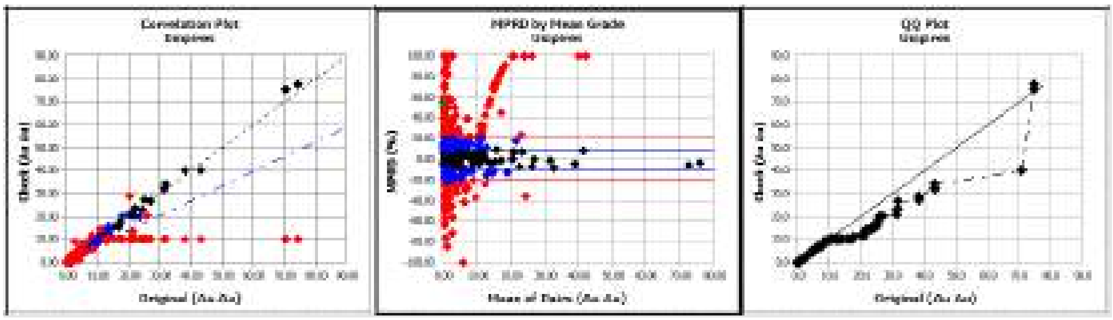


Figure 11.28 Sample Pair Statistical Analysis: Umpires: Original Als AA26, Check MSA FAS121/425



11.4 ASSAY QUALITY CONTROL – HEINÄ CENTRAL

Analysis of internationally accredited assay standards or certified reference material (“CRM”) has been carried out.

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.
- Barren samples (“blanks”) submitted by both Rupert Resources and the assay laboratory.

11.4.1 QA/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 and includes the following datasets:

Blanks:

- Submitted by Rupert Resources.
- Internal ALS blanks.

CRM (Standards):

- Submitted by Rupert Resources.
- ALS internal CRM.

Data Pairs:

- Core duplicates (quarter core pairs).
- Lab duplicate (duplicate samples taken after pulverised to >85% <75 µm).
- Pulp duplicates (duplicates samples taken from within one pulp sachet).
- 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 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.

Table 11.22 and Figure 11.29 and Figure 11.30 summarise the results of assaying blank samples. For 100% of analyses, the blanks returned less than detection limit results.

Table 11.22 Heinä Central Gold Deposit - Blanks

Standard	Assay Method	Laboratory	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
				Blanks Submitted by Rupert				
BLK-CO01	Au-AA26-ppm	ALS	804	0.0100	0.0057	-43.0970	2.1753	100
BLK-CO01	Au-ICP24-ppm	ALS	42	0.001	0.001	4.7619	68.9699	100
				Internal ALS blanks				
BLK-ALS	Au-AA26-ppm	ALS	450	0.010	0.0057	-43.2222	35.3017	100
BLK-ALS	Au-ICP24-ppm	ALS	96	0.001	0.0006	-40.1042	53.6505	100
BLK-ALS	Cu-MS61-ppm	ALS	604	0.010	0.2424	24138.4000	151.8380	95

Figure 11.29 Rupert Resources Company Blank (BLK-CO01) Performance

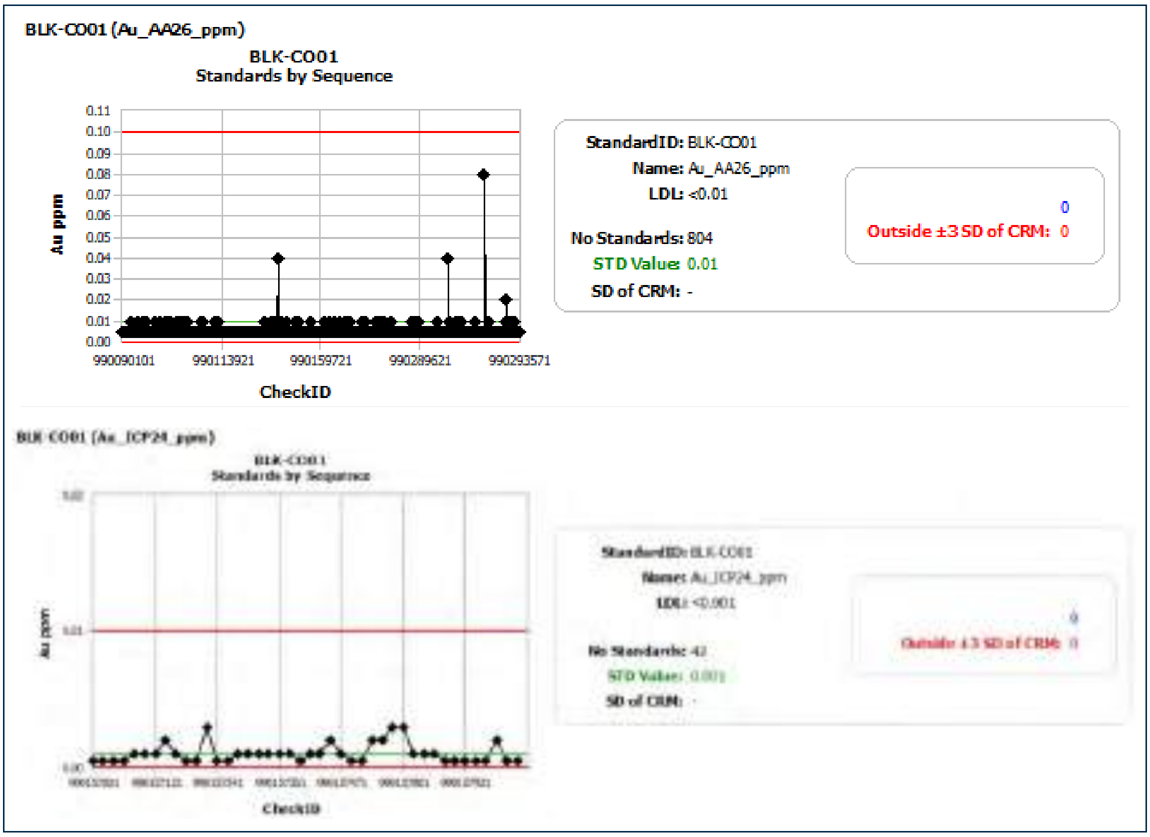
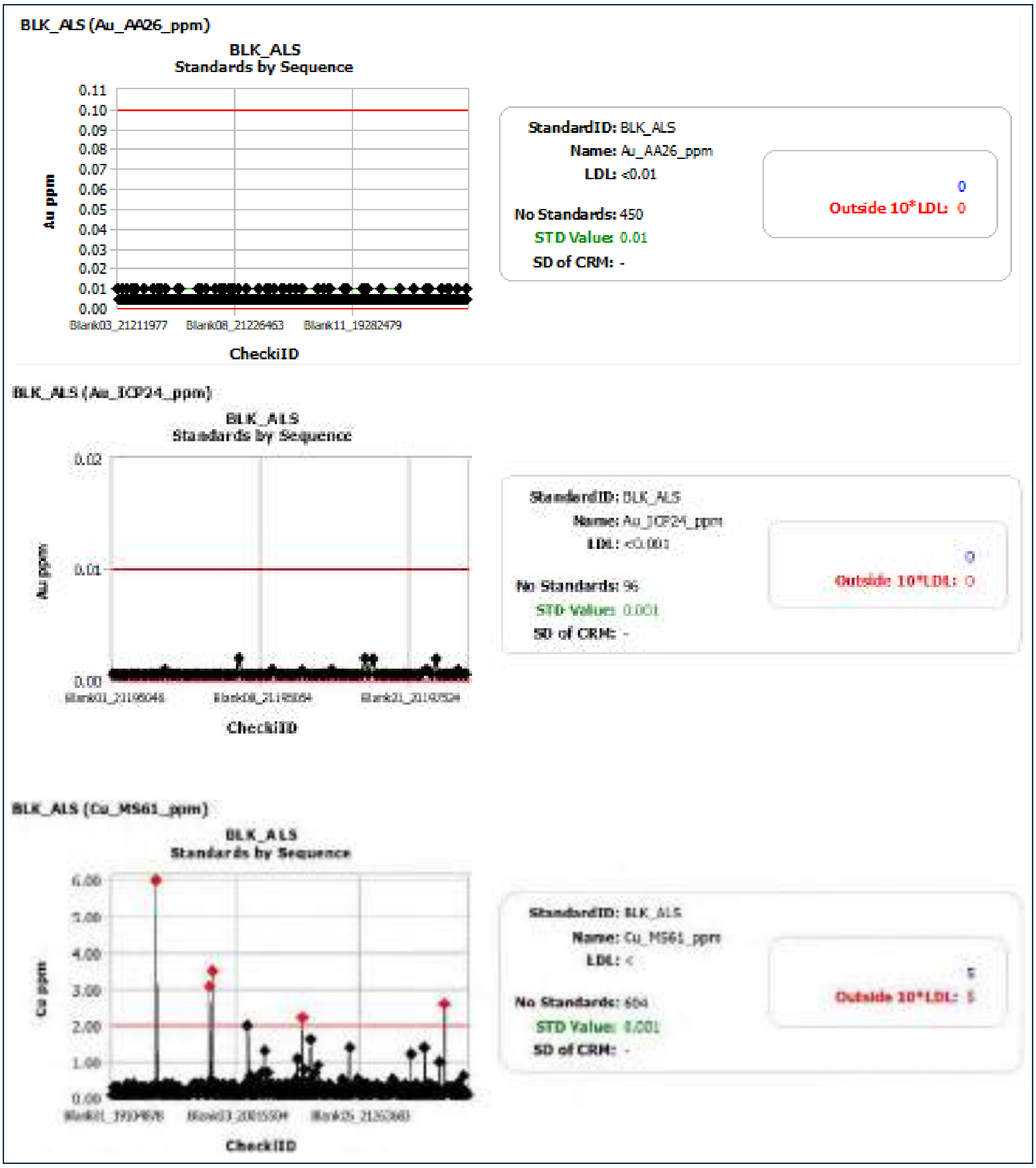


Figure 11.30 ALS Internal Blank (BLK-ALS) Performance



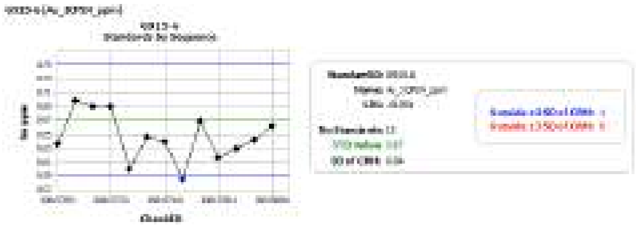
CRM Submitted by Rupert Resources

Rupert Resources routinely submitted accredited CRM. Rupert has been using gold certified reference materials produced by Geostats Pty Ltd. These CRM's (G912-3, G915-2, G915-6, G398-4, G312-4, and G917-4) have been selected to represent three different gold grades. Rupert has used also commenced use of Cu- Co- and Ni-certified standard materials, but results for these are pending.

Table 11.23 Heinä Central gold Deposit Standards Submitted to ALS by Rupert Resources

Standard	Assay method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
G312-4	Au-AA26	29	5.30	5.3614	1.1581	2.5762	100
G398-4	Au-AA26	103	0.66	0.6485	-1.3580	3.0182	100
G912-3	Au-AA26	253	2.09	2.0821	-0.3763	2.9067	100
G912-3	Au-ICP24	14	0.09	2.0939	0.1880	2.5343	100
G915-2	Au-AA26	161	4.98	4.9868	0.1372	2.2994	100
G915-2	Au-ICP24	12	4.98	5.0358	1.1212	1.8560	100
G915-6	Au-AA26	150	0.67	0.6428	-4.0597	3.6811	100
G915-6	Au-ICP24	13	0.67	0.6469	-3.4443	5.3764	100
G917-4	Au-AA26	60	5.10	5.1360	0.7059	0.2537	100

Figure 11.31 Rupert Resources CRM's Performance in ALS



INTERNAL CRM ANALYSED BY ALS

ALS, as part of their standard QAQC procedures routinely analyse CRM prepared by independent suppliers. Rupert has obtained all the available internal ALS CRM analytical results and statistical analysis has been carried out on the gold data.

Table 11.24 summarises the results of the analytical performance by ALS on these internally submitted CRM. The assay method used for the different CRM is also noted in Table 11.24.

Table 11.24 Heinä Central Gold Deposit ALS Internal Standards

Standard	Assay Method	Number	Expected Value	Mean	% Bias	% RSD	% in Tolerance
BLK_ALS	Au-AA26	450	0.010	0.5700	-43.2222	30.1788	100
BLK-ALS	Au-ICP24	96	0.010	0.0006	-40.1042	53.6505	100
G306-3_ALS	Au-AA26	2	8.660	8.7650	1.2125	2.0168	100
G312-4_ALS	Au-AA26	65	5.300	5.3378	0.7141	2.2252	100
G313-5_ALS	Au-AA26	36	7.070	7.0119	-0.8212	2.0463	100
G398-10_ALS	Au-AA26	123	4.070	4.0605	-0.2337	2.8389	100
G913-10_ALS	Au-AA26	62	7.090	7.0574	-0.4595	2.0833	100
G914-10_ALS	Au-AA26	48	10.260	10.0542	-2.0062	2.3545	100
G998-4_ALS	Au-AA26	12	4.360	4.4375	1.7775	1.4061	100
KIP-19_ALS	Au-ICP24	24	2.430	2.4288	-0.0514	1.5325	100
LEA-16_ALS	Au-AA26	67	0.501	0.5021	0.9410	2.5604	100
OREAS 214_ALS	Au-AA26	9	3.030	3.0033	-0.8801	1.3109	100
OREAS 217_ALS	Au-AA26	58	0.338	0.3303	-2.2648	2.4034	100
OREAS-219_ALS	Au-AA26	23	0.760	0.7496	-1.3730	2.1840	100
OREAS-226_ALS	Au-AA26	44	5.450	5.4468	-0.0584	1.9099	100
OREAS 250_ALS	Au-AA26	125	0.303	0.3126	3.1551	2.9832	100
OxA131_ALS	Au-AA26	4	0.077	0.0745	-2.5974	7.6980	100
OxD151_ALS	Au-AA26	72	0.430	0.4208	-2.1218	2.6076	86.11
OxF162_ALS	Au-AA26	11	0.832	0.8182	-1.6608	1.8791	100
OxI121_ALS	Au-AA26	3	1.834	1.8533	1.0542	0.3115	100
OxP116_ALS	Au-AA26	23	14.920	14.9435	0.1574	2.4718	60.87
OxP158_ALS	Au-AA26	68	15.150	14.9404	-1.3832	2.7239	100
OxQ90_ALS	Au-GRA22	3	24.880	24.5667	-1.2594	2.3851	100
PMP-18_ALS	Au-AA26	57	0.300	0.3011	0.3509	3.3090	98.25
SL76_ALS	Au-ICP24	23	5.960	5.9326	-0.4596	1.4178	100
TAZ-20_ALS	Au-AA26	116	0.302	0.2956	-2.1181	3.1321	86.21
BLK-ALS	Cu-MS61	604	0.010	0.2424	24138.4100	151.8384	95.00
GBM908-10_ALS	Cu-MS61	3	3,601.000	3,626.6700	0.7128	2.2960	100
MRGeo08_ALS	Cu-MS61	269	631.000	629.9300	-0.1703	2.2433	100
OGGeo08_ALS	Cu-MS61	284	8,540.000	8,287.2500	-2.9596	1.8983	100
OREAS 905_ALS	Cu-MS61	93	1,533.000	1,518.3300	-0.9567	2.3649	100
OREAS 920_ALS	Cu-MS61	48	112.000	111.865	-0.1209	3.1110	100
OREAS 922_ALS	Cu-MS61	153	2,122.000	2,180.000	2.7333	2.1335	100

Comparison of Common CRM

All CRM's Rupert Resources has been using since July 2018 perform very well with used fire assay methods (Figure 11.31 and Table 11.24)

Data Pairs

Rupert Resources current 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 and assay method (Table 11.25). The different types of data pairs comprise the following:

- Field duplicates (quarter core pairs).
- Lab duplicates (two samples taken after pulverising sample material >85% <75 µm).
- Pulp duplicates (duplicates samples taken from within one pulp sachet).
- Umpire checks (Pulp split sent to second laboratory).

Table 11.25 Heinä Central Gold Deposit Data Pairs

Duplicate Type	Submitted by	Laboratory	Assay Method	Total Number of Pairs	Au1 Mean (g/t)	Au2 Mean (g/t)	Corr. Coeff.	Mean AMPRD
Field duplicate	Rupert	ALS	Au-AA26	663	0.1417	0.1310	0.7860	31.506
Field duplicate	Rupert	ALS	Au-ICP24	40	0.5168	0.3258	0.7576	47.3354
Pulp duplicate	Rupert	ALS	Au-AA26	755	0.3085	0.2790	0.9931	20.2210
Pulp duplicate	Rupert	ALS	Au-ICP24	39	0.2016	0.2279	0.9260	22.8348
Lab duplicate	ALS	ALS	Au-AA26	705	0.3067	0.2940	0.9349	29.3906
Lab duplicate	ALS	ALS	Au-ICP24	30	0.3785	0.3842	0.9990	23.3046
Lab duplicate	ALS	ALS	Au-GRA22	3	179.3333	160.6667	0.9547	73.6026
Field duplicate	Rupert	ALS	Cu-MS61	6	106.3667	116.7667	0.0978	51.3488
Lab duplicate	ALS	ALS	Cu-MS61	718	494.9217	492.0517	0.9999	4.7037

Table 11.26 Heinä Central Gold Deposit Umpire Assay Data Pairs

Duplicate Type	Original laboratory	Umpire Laboratory	Assay Method Original	Assay Method Check	Total Number of Pairs	Au1 Mean (g/t)	Au 2 Mean (g/t)	Corr. Coeff.	Mean AMPRD
Umpire	ALS	CRS/MSA	Au-AA26	Au-FAS221	183	0.0825	0.0935	0.8261	41.6863

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

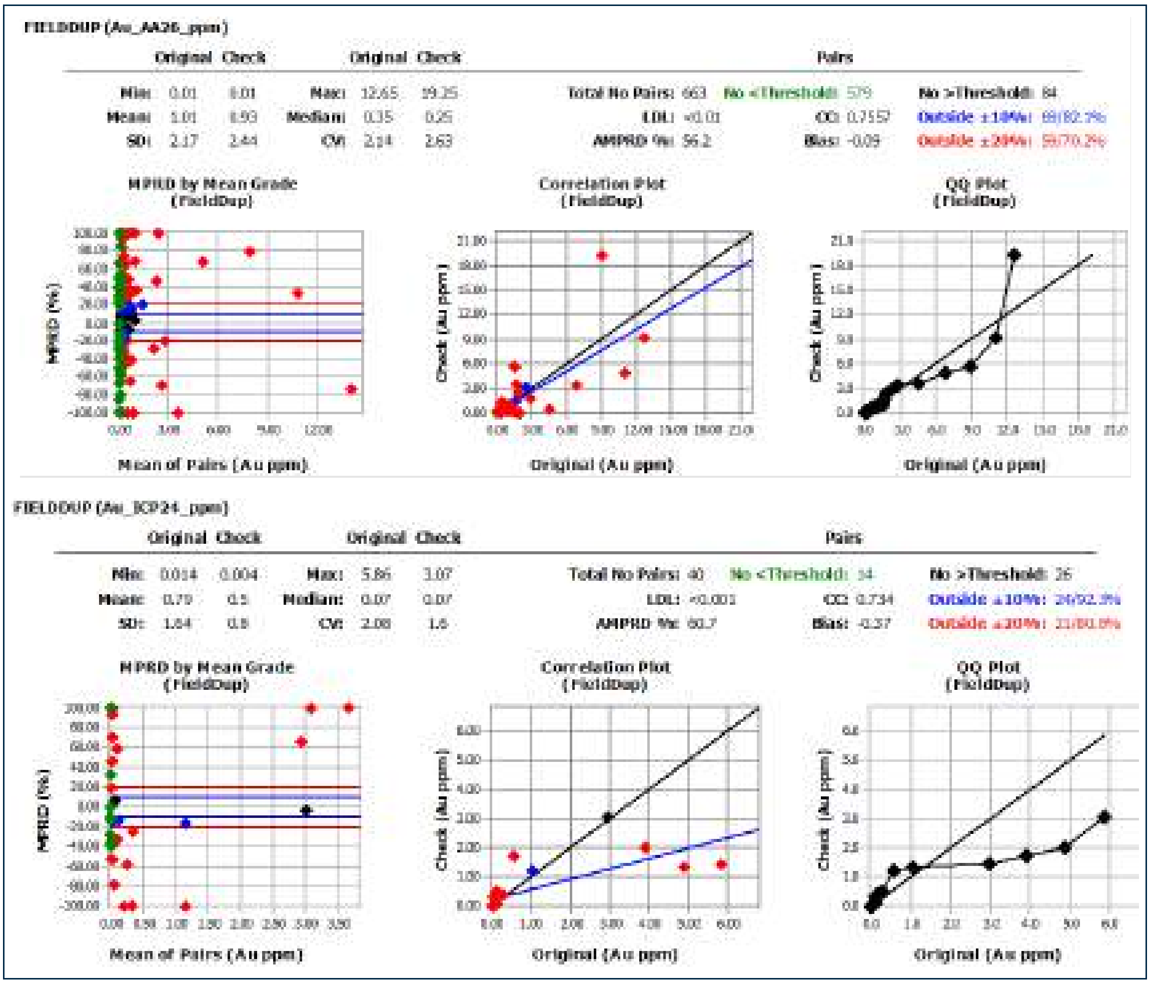
- MPRD by Mean Grade.
- Correlation Plot.
- Quantile-Quantile Plot.

Samples Submitted to ALS

Samples submitted to ALS for data pair analysis have included the following sample types:

- Field duplicates (two separate quarter core samples from the same sample interval) (Figure 11.32).
- Pulp duplicates (two sub-samples taken from the same pulp sachet) (Figure 11.33).
- Laboratory Duplicates (two samples taken after pulverising sample material >85% <75 µm) (Figure 11.34).

Figure 11.32 Sample Pair Statistical Analysis: Samples Submitted to ALS: Field Duplicates



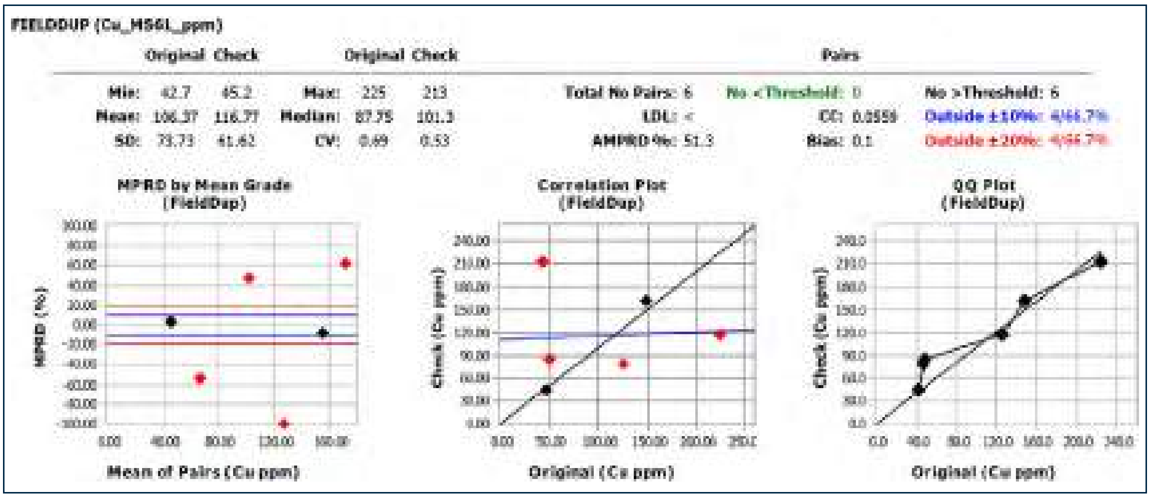


Figure 11.33 Sample Pair Statistical Analysis: Samples Submitted to ALS: Pulp Duplicates

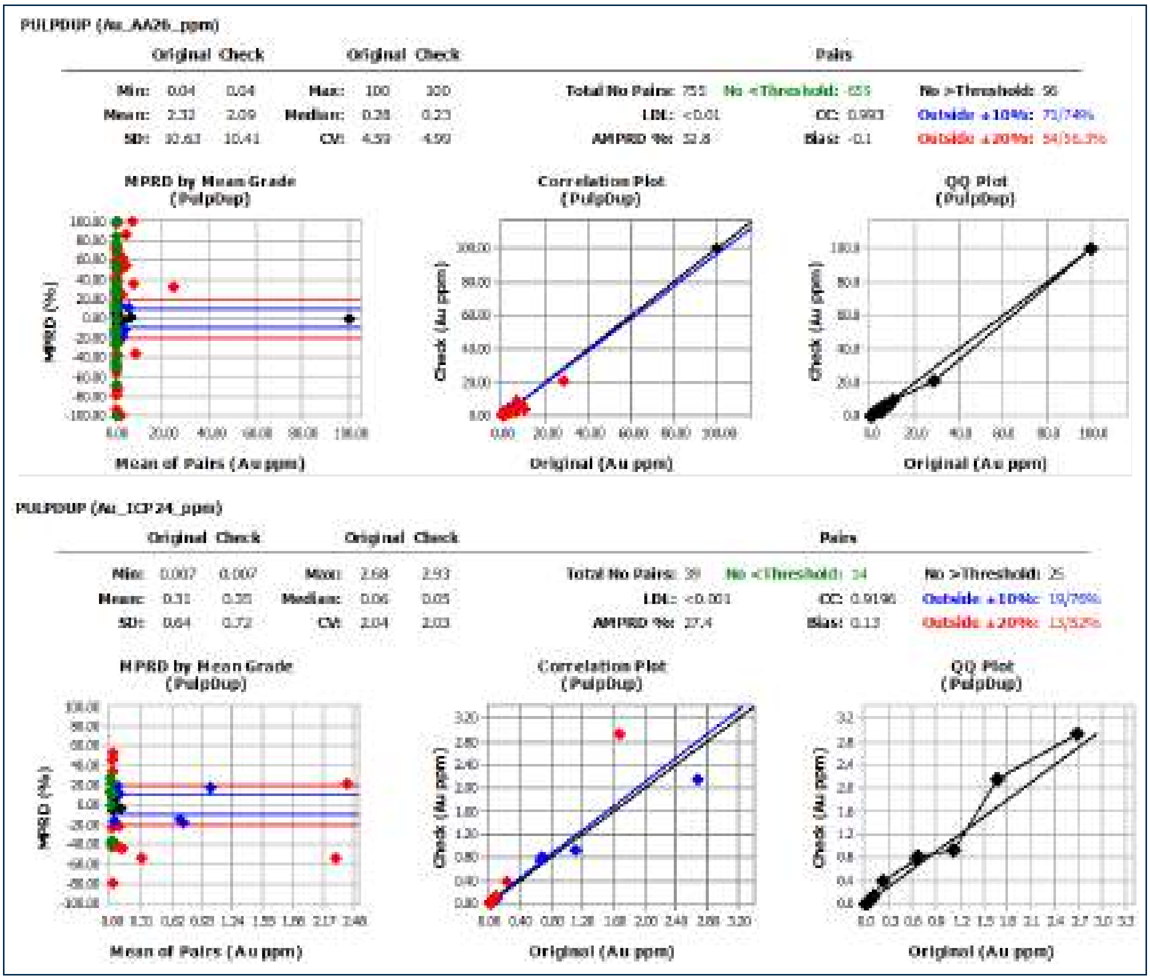
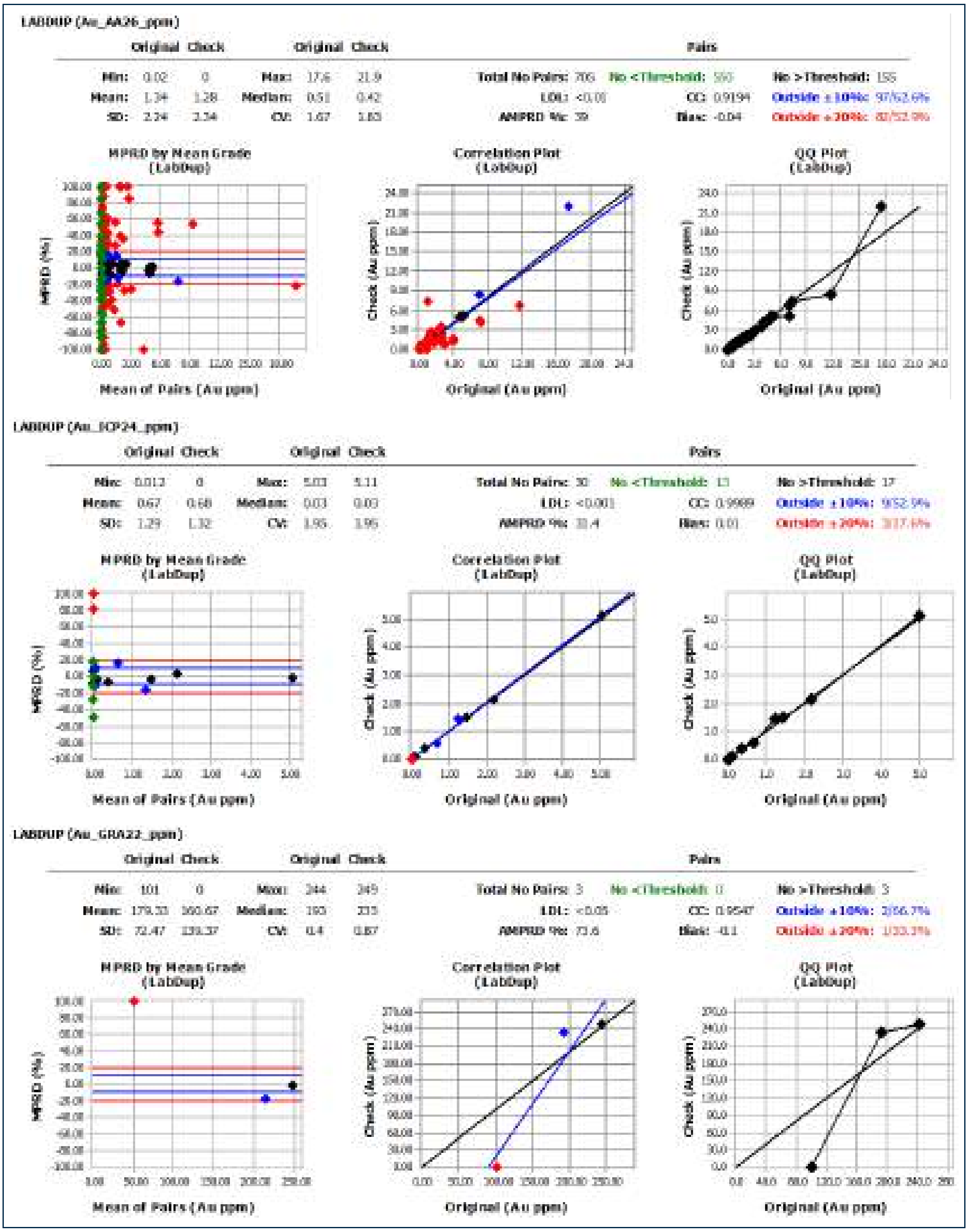
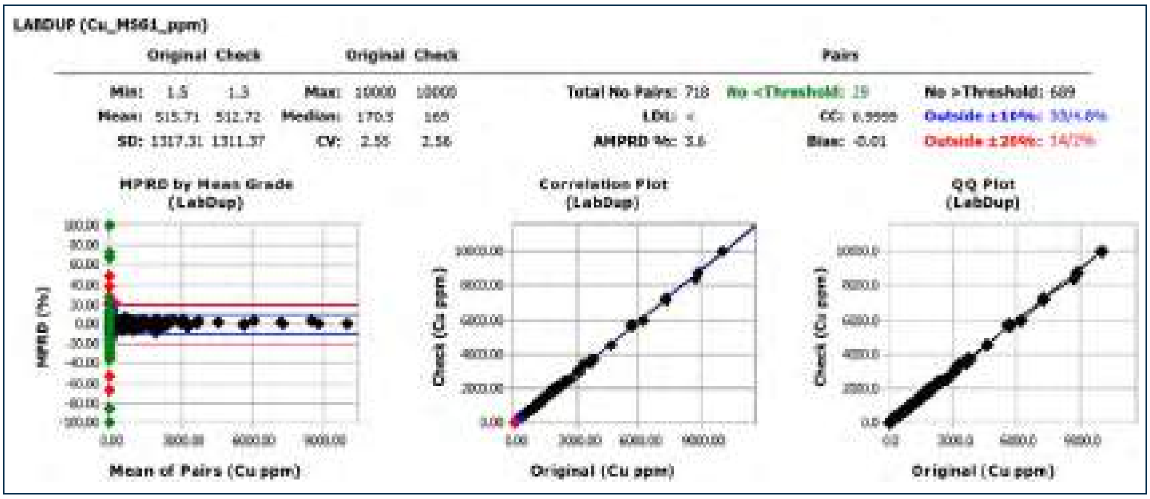


Figure 11.34 Sample Pair Statistical Analysis: Samples Submitted to ALS: Laboratory Duplicates





11.5 CONCLUSIONS

These methods of data verification are considered at or above industry standard. The results of the QA/QC data analyses discussed in the preceding sections demonstrate that the quality of the data is acceptable for use in mineral resource estimation.

All sample preparation was carried out at independent laboratory in Finland, and analyses were carried out at independent laboratories in Romania, Ireland, or Finland (apart from largely historic samples from Pahtavaara, as discussed in section 11.1.2. No aspect of laboratory sample preparation or analysis was conducted by an employee, officer, director or associate of either Rupert Resources or its predecessors.

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 good 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.

12.0 DATA VERIFICATION

12.1 INDEPENDENT QUALIFIED PERSON REVIEW AND VERIFICATION

Mr Brian Wolfe visited the Pahtavaara Gold Project in February 2018 and again in March 2022, along with the Ikkari and Heinä Central projects. Steps undertaken to verify the integrity of data used in this report include:

- Field visits to the areas outlined in this report, including Pahtavaara, Ikkari and Heinä Central sites.
- Site visit to underground while channel sampling and mapping was under way (Pahtavaara, 2018).
- Inspection of diamond drill core.
- Inspection of diamond drilling activities, sampling and logging procedures.
- Review of data collection, database management and data validation procedures.
- Review of the previous technical documentation for the Pahtavaara Gold Project.

The Qualified Person (QP) has reviewed and cross-checked sections of this Report prepared by Rupert Resources geologists.

The QP completed the updated resource estimate for the Pahtavaara, Ikkari and Heinä Central Gold Deposits. Additional data verification steps undertaken during this estimate process included the following:

- Validation of drilling, geology and assay database (including checks overlapping intervals, samples beyond hole depth and other data irregularities.
- Review of Rupert Resources' QA/QC data and charts for standards, blanks and duplicates.
- Visual and statistical analysis of resource estimate model outputs versus primary data.
- Random cross checks of assay reports against the database.

Based on this review work, the QP is of the opinion that the dataset provided by Rupert Resources is of an appropriate standard to use for resource estimation work.

12.2 QA/QC Data Analysis

The quality control data has been statistically evaluated, and summary plots have been produced for interpretation as described in the previous sections.

12.3 Conclusions

These methods of data verification are considered at or above industry standard. The results of the QA/QC data analyses discussed in the preceding sections demonstrate that the quality of the data is acceptable for use in mineral resource estimation.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 SUMMARY

Test work was carried out on material from the Pahtavaara ore body and the Ikkari ore body to evaluate different gold concentration methods and extractability to be able to evaluate different processing methods of gold recovery. Metallurgical test work has not yet been carried out on material from the Heinä Central ore body.

13.1.1 PAHTAVAARA SUMMARY OF RESULTS

Gold head grade analysis of the Pahtavaara sample has shown to have a gold content of 1.19 g/t. During testing, it was noted that there was a significant nugget effect with variance in the back-calculated head grades. The average back calculated head grade during the metallurgical testing was shown to be 2.58 g/t.

Comminution testing via Semi-autogenous Grinding (SAG) Mill Comminution (SMC), Rod Mill Work Index (RWi), and Bond Working index (BWi) testing has shown that the submitted sample is of average hardness. The Bond Ai test showed that the sample is considered abrasive with a value of 0.59.

The gravity release shows that for a grind below 250 µm gravity concentration is viable. The gravity recoverable and GRG test showed that gold recovery of 38.9% with a mass pull of 1.89% was achievable.

The flotation tests showed that a grind finer than 125 µm decreased gold recovery. The reagent optimisation tests showed that the Xanthate-Potassium Amyl Xanthate (PAX) addition could be decreased to 10 g/t. The addition of a cleaner circuit was not viable as the overall recovery decreased significantly. The bulk flotation test using these conditions achieved a gold recovery of 82.8% with a mass pull of 12.9%. It is believed this can be improved upon with a longer flotation residence time.

Both the whole ore and flotation leaching tests showed that greater than 98% extraction was achievable. The leaching of the flotation concentrate was higher with an extraction of greater than 99.5%.

Thickening test work conducted on bulk flotation tailings samples indicated that flocculant N2354 could be used to effect flocculation of the tailings at feed well densities of 15%. Settling rates were shown to be fast at around 2000 m^3/m^2.day.

The environmental test conducted showed that the sample was classified as potentially acid neutralising with the net acid generation test showing a final pH of between 11.27 and 11.43. The material was found to be acceptable for inert waste landfill.

13.1.2 Ikkari Summary of Results

Two samples were analysed for the Gold head grade. The results of the analysis were higher than the back-calculated head grade. It is believed that the cause of this is due to "nugget" effect. The back-calculated head grade for Au averaged between 3.6 to 4.2 g/t Au. The second sample's back-calculated gold grade was lower at 1.81 milligrams (mg) Au /kg.

The results of the comminution test work showed the material to be abrasive and harder than "medium" with a bond BWi of 15.5 kWh /t.

The gravity release shows that for a grind below 600 µm gravity concentration is viable. The gravity recoverable and GRG test showed that gold recovery of 65.22% with a mass pull of 2.14% was achievable.

The flotation tests with different grind sizes (P^{80}) showed that the gold recovery was insensitive up to a grind size 190 µm. The smaller the grind size resulted in a higher mass pull to achieve the same gold recovery. The reagent optimisation tests showed that the PAX addition could be decreased to 32 g/t. The addition of a cleaner circuit was not viable as the overall recovery decreased significantly. The bulk flotation test using these conditions achieved a gold recovery of greater than 97.4% with a mass pull of 7.21% or less.

The whole ore cyanidation testing on grind sizes between 106 µm and 38 µm had gold extractions greater than 94.5%, with the results showing that grind had little effect on extraction.

The leaching of flotation concentrate did not vary significantly with different grind sizes. The addition of lead nitrate increased the gold leaching kinetics but decreased the overall extraction. The introduction of an oxygen preconditioning stage did not improve the leach kinetics.

The leaching of the flotation tails showed that cyanide extractable gold in the flotation tailings samples was between 47% and 56%.

Cyanide destruction using the Inco SO_2/air method resulted in most of the cyanide being broken down with the residual cyanide being less than 3 ppm.

Initial settling rates in the order of 4,500 m^3/m^2.day were observed. Changing of the flocculant addition from 10 g/t to 35 g/t had little effect on the settling rate.

Chemical analysis of each of the waste streams for U, Th, Cd, and Mercury (Hg) below detection limits of 0.0001%. Four of the five waste rock samples tested were classified as acid neutralising. A sample of the flotation tailings was tested and found to be acid neutralising. The waste rock samples were tested for landfill compliance and found to be acceptable for inert waste landfill.

13.2 Source Documents

Tetra Tech has access to the original laboratory reports from Preliminary Economic Assessment (PEA) test work listed below.

- Research report from samples AEM 001-AEM020 Ikkari Au prospect, Kari Kojonen FT, spring 2021.
- A21518 Two (2) samples from Ikkari, Pahtavaara Project for Rupert Resources Limited, ALS Metallurgy Services, January 2021.
- 21-1882 Ikkari Deposit Gold Recovery Testing - Rupert Resources, Grinding Solutions, May 2021.
- 22-1970 Rupert Resources - Pahtavaara Deposit Gold Recovery Testing, Grinding Solutions, January 2022.
- 22-1967 Rupert Resources Phase II - Ikkari Gold Recovery Optimisation Testing1, Grinding Solutions, February 2022.
- 22-2061 Rupert Resources - Pre-aerated Cyanide Leach Testing, Grinding Solutions, May 2022.

All the test work reported in this section is taken from the information presented in these documents.

13.3 Mineralogical Test Work

Mineralogical examination of Ikkari (previously Agnico Eagle's) mineral exploration samples AEM001-AEM020 was carried out in the spring of 2021, in which the samples were examined with a polarising microscope in reflected and transmitted light. Electronically with Scanning Electron Microscopy (SEM) / Energy dispersive spectroscopy (EDS) equipment and an electron microprobe, the last two of which were used for both point analysis of mineral grains and elemental distribution analysis.

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. 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. To a lesser extent, chalcopyrite, sphalerite, and galena were found. Accessory minerals include monazite, xenotime, zircon, brannerite, and apatite. The main minerals in the ore samples are sericite, carbonate, quartz, biotite and chlorite.

Based on elemental distribution images and point analyses, Co is most abundant in pyrite, which occurs as two generations: Anhedral massive-like and subhedral to euhedral grains in veinlets and as dissemination. Various Co and Ni concentrations have also been analysed in magnetite and the gangue minerals biotite and chlorite are Ni bearing. 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).

13.4 Pahtavaara Metallurgical Test Work

The gravity release shows that for a grind below 250 µm gravity concentration is viable. The gravity recoverable and GRG test showed that gold recovery of 38.9% with a mass pull of 1.89% was achievable.

13.4.1 Head Grade

Pahtavaara Testing on the submitted sample from the Pahtavaara deposit has shown that the sample has a gold content of 1.19 g/t. During testing, it was noted that there was a significant nugget effect with variance in the back-calculated head grades. The average back calculated head grade during the metallurgical testing was shown to be 2.58 g/t.

13.4.2 Comminution

Comminution testing via SMC, RWi, and BWi testing has shown that the submitted sample is of average hardness. The Bond Ai test showed that the sample is considered abrasive with a value of 0.59.

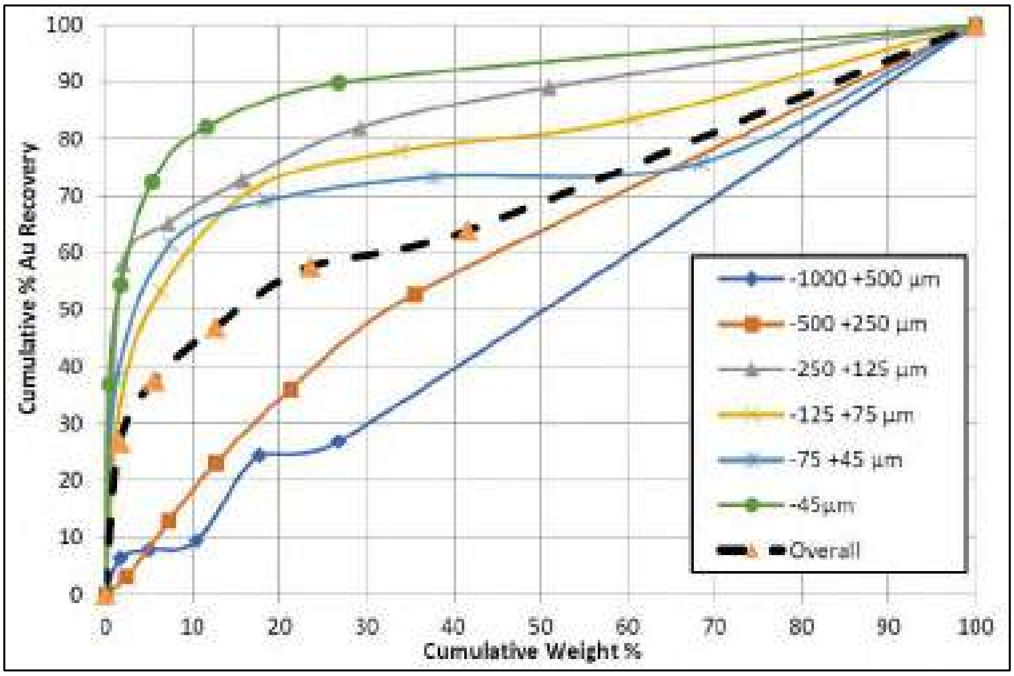
13.4.3 Gravity Recovery

The gravity release shows that for a grind below 250 µm gravity concentration is viable. The gravity recoverable and GRG test showed that gold recovery of 38.9% with a mass pull of 1.89% was achievable.

Gravity Release

Gravity release analysis, shown in Figure 13.1, performed on -1 mm ground feed material showed that grinding to a P^{80} of around 250 µm and using single G separation devices such as spirals or tables could provide a 75% gold recovery to a 15% mass product grading at around 15 g/t.

Figure 13.1 Mass Pull vs Gold Recovery Curves for Gravity Release Analysis



GRAVITY RECOVERABLE GOLD AND GRG TEST

Gravity recoverable gold testing showed the sample to possess a GRG content of 38.9% with a 1.89% mass pull. The size distribution of this gold content was shown to be moderately coarse.

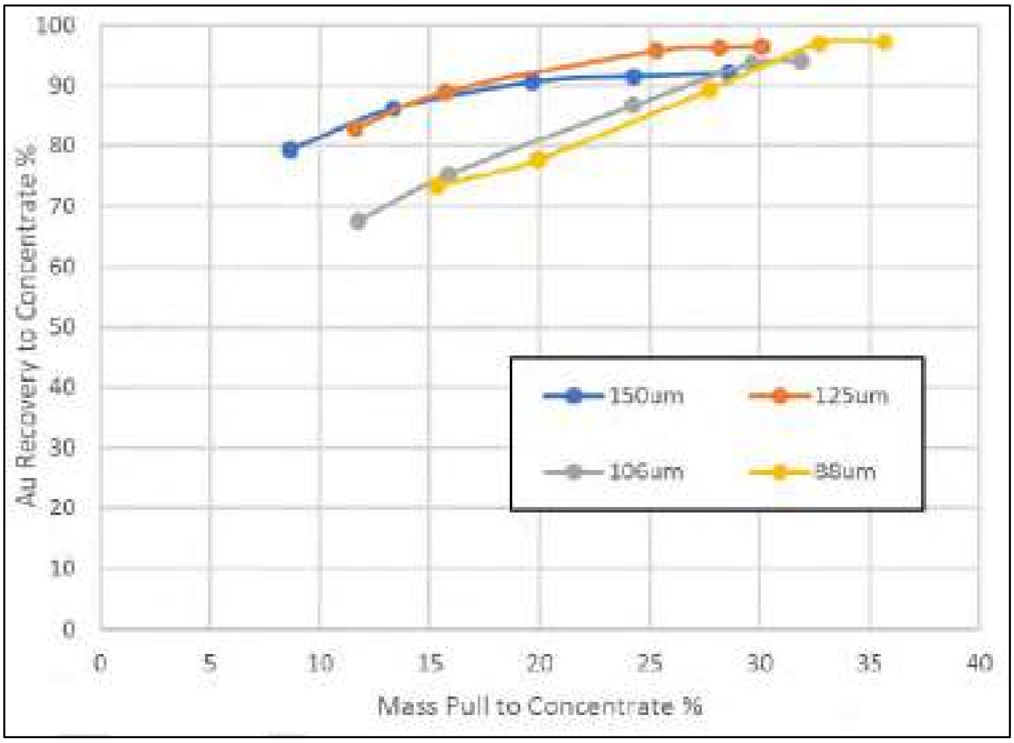
13.4.4 FLOTATION

The flotation tests showed that a grind finer than 125 µm decreased gold recovery. The reagent optimisation tests showed that the PAX addition could be decreased to 10 g/t. The addition of a cleaner circuit was not viable as the overall recovery decreased significantly. The bulk flotation test using these conditions achieved a gold recovery of 82.8% with a mass pull of 12.9%. It is believed this can be improved upon with a longer flotation residence time.

GRIND SIZE

Rougher flotation test work was carried out on different feed sizes, from a P^{80} of 88 µm to 150 µm. The results are shown in Figure 13.2. The results show that gold recovery to the flotation concentrate was high for all completed tests reaching between 92% and 97% for the tests. Mass pull to the flotation concentrates was high at between 28.6% and 35.7%, with a trend for increasing mass pulls at finer grind sizes. Gold grades to the rougher concentrate were between 11.6 g/t and 6.2 g/t Au with a trend of decreasing grade with decreasing grind size.

Figure 13.2 Kinetic Mass Pull vs Gold Recovery Plots for Mesh of Grind Flotation Tests

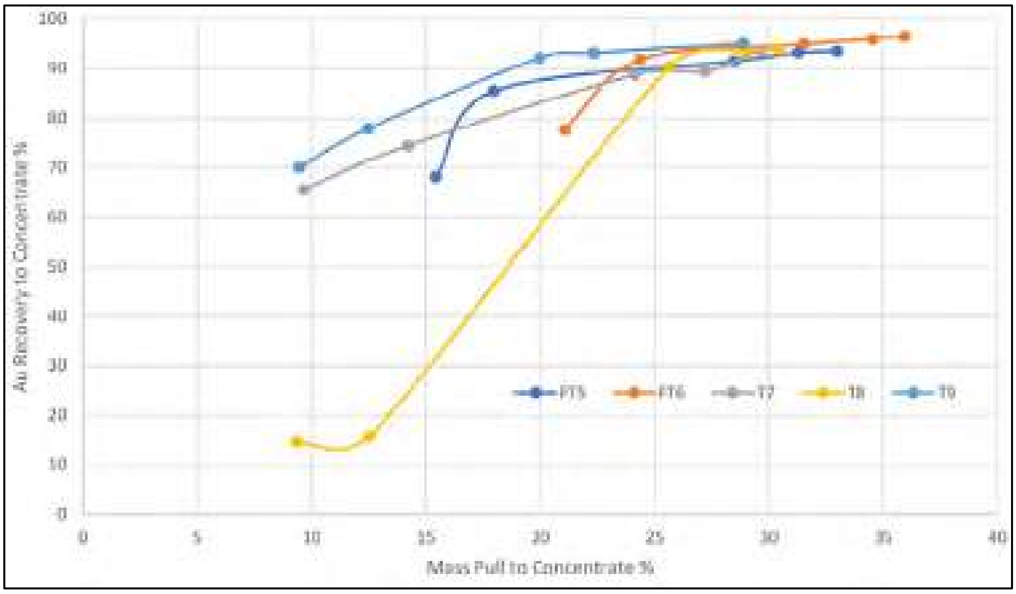


Rougher flotation testing has shown that a high gold recovery can be achieved to the flotation concentrate. Testing showed that a 96% gold recovery could be achieved with a mass pull of 30% and a gold grade of around 8 g/t.

REAGENT OPTIMISATION

Reagent optimisation was done in the form of reducing PAX addition. The result of the test is shown in Figure 13.3 where FT9 had the lowest PAX addition with 10 g/t and FT5 had the highest PAX addition with 64 g/t.

Figure 13.3 Flotation Kinetic Mass Pull vs Gold Recovery Plots for Varying Reagent Addition Rates



It was found that reducing the PAX dosage in the rougher circuit from 80 g/t to 10 g/t had minimal impact on overall gold recovery to concentrate.

CIRCUIT DEVELOPMENT

Tests were carried out evaluating the suitability of pre-flotation and cleaner stages on the material. Test CL1 was completed as a rougher and 2-stage cleaner test with no regrinding of the rougher concentrate. Both CL2 and CL3 were completed in the same manner except for including a pre-float to float naturally floating amphibole and clay species to reduce the mass pull through the final concentrate and increase the grade.

Table 13.1 Summary of Flotation Cleaner Tests

Float Test	Pre-Float			Rougher			Cleaner 1			Cleaner 2		
	Mass pull %	Au Recovery	Au Grade g/t	Mass Pull %	Au Recovery %	Au Grade g/t	Mass Pull %	Au Recovery %	Au Grade g/t	Mass Pull %	Au Recovery %	Au Grade g/t
CL1	-	-	-	21.84	91.71	13.31	9.61	80.78	18.57	5.12	49.43	27.32
CL2	11.75	8.19	1.58	19.12	85.87	9.54	9.71	80.06	16.65	0.70	45.83	146.70
CL3	11.35	1.53	0.54	16.80	90.79	12.80	5.07	61.72	28.63	0.53	44.70	199.30

The results show that gold recovery to the rougher concentrate of around 90% to 91% was achieved. The inclusion of other flotation steps was shown to decrease the gold recovery substantially.

BULK FLOTATION

Test work was done on a sample to generated flotation concentrate for leach test work. The results indicated that a gold recovery to the rougher concentrate was lower than had previously been achieved at 82.8%. The mass pull to the concentrate was also lower at just 12.9%.

Comparing this result to previous kinetic tests at similar mass pull points, the recovery is similar. It is considered that the degree of agitation achieved in the larger cell may not have been sufficient which most likely resulted in the lower mass and so lower gold recovery. A longer flotation time would likely have resolved this.

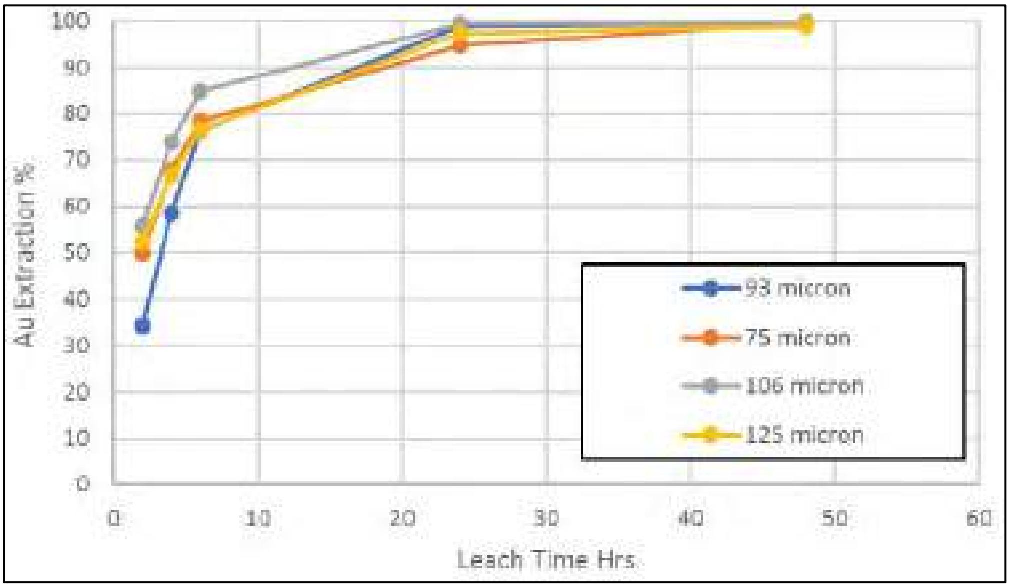
13.4.5 Leaching

Both the whole ore and flotation leaching tests showed that greater than 98% extraction was achievable. The leaching of the flotation concentrate was higher with an extraction of greater than 99.5%.

Whole Ore Leaching

Whole ore cyanide leach testing was completed on a range of grind sizes of a P^{80} between 75 µ and 125 µ. All tests, shown in Figure 13.4, showed excellent gold extractions to solutions all over 98%. Indicating that the particle sizes with a P^{80} of 125 µm and less do not substantially affect gold extraction.

Figure 13.4 Kinetic Gold Extraction Plots for Whole Ore Cyanide Leach Tests

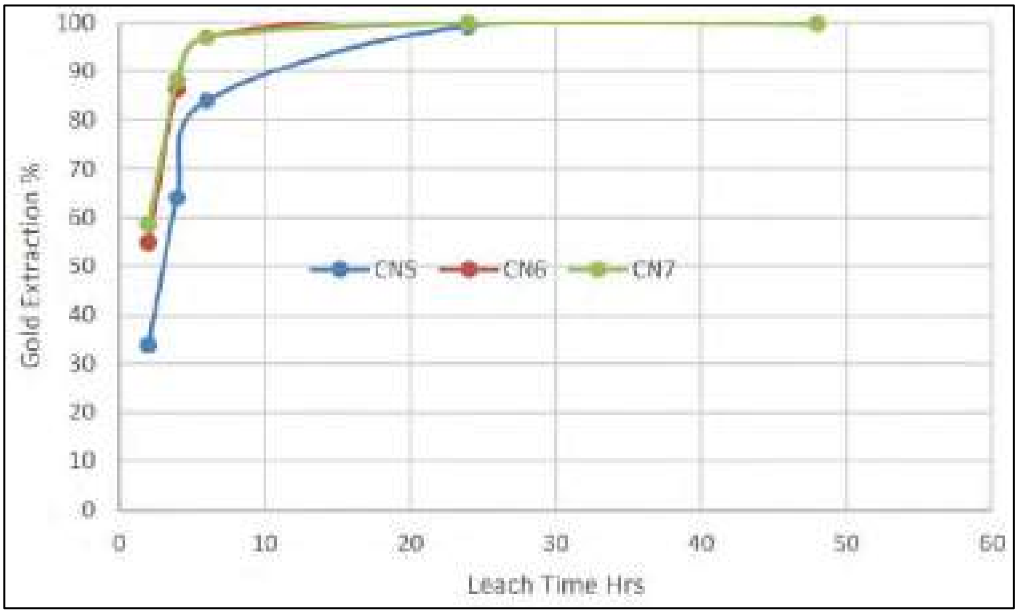


Cyanide consumptions for the tests were low at between 0.01 and 0.25 kilograms per tonne (kg/t) Sodium Cyanide (NaCN). The data shows that the extraction is nearing completion after 24 hours (hr), however, this may be sooner if there was kinetic data between the 6 hr and 24 hr sample points.

Leaching of Flotation Concentrate

Figure 13.5 shows the results of the leaching of the flotation concentrate.

Figure 13.5 Kinetic Gold Extraction Plots for Flotation Concentrate Leach Tests



Gold extraction to solution for all tests was excellent with all indicated extraction being above 99% indicating that no regrind of the flotation concentrate would be required.

Cyanide consumptions for the tests ranged between 0.12 and 0.92 kg/t NaCN. Lime consumptions for the tests ranged between 0.41 and 0.44 kg/t Calcium Hydroxide ($Ca(OH)_2$).

13.4.6 Solid Liquid Separation

Thickening test work was conducted on bulk flotation tailings samples using the flocculant N2354 manufactured by Nasaco. The results of the tests are shown in Table 13.2.

Table 13.2 Conditions and Summary Results for Thickening Tests on Flotation Tailings

Settling Test	Feed Solids %	Flocculent	Dosage g/t	Settling Rate m3/m2	Thickener Unit Area Underflow M2/MTPD	Solids % at Compression Point	Final Underflow Solids Density %
1	10	N2354	14.25	4818.53	0.630	63.0	71.6
2	10	N2354	21.37	4744.92	0.332	61.2	65.8
3	10	N2354	28.26	4523.9	0.245	53.4	60.9
4	10	N2354	35.62	5096.4	0.123	63.0	67.3
5	12.5	N2353	34.74	4553.28	0.187	60.8	65.7
6	15	N2354	34.85	4553.28	0.136	62.3	67.3
7	17.5	N2354	40.00	3589.63	0.228	52.8	61.5
8	20	N2354	39.33	4142.02	0.166	58.5	63.7

Thickening test work conducted on bulk flotation tailings samples indicated that flocculant N2354 could be used to effect flocculation of the tailings at feed well densities of 15%. Settling rates were shown to be fast at around 2000 m^3/m^2.day.

13.4.7 Environmental Testing

Acid Base Accounting

The results showed that the sample was classified as potentially acid neutralising.

Net Acid Generation Test

The net acid generation test showed a final pH of between 11.27 and 11.43.

Waste Compliance Leach Testing

Waste compliance was carried out.

A waste rock sample sent for waste compliance testing using a cumulative two stage batch test. Compliant limits are shown in Table 13.3.

Table 13.3 Landfill Waste Compliance Acceptance Criteria Limits

Solid Waste Analysis	Units	Landfill Waste Acceptance Criteria Limits		
		Inert Waste Landfill	Stable Non-reactive Hazardous Waste in Non-Hazardous Landfill	Hazardous Waste Landfill
Total Organic Carbon	%	3	5	6
Loss on Ignition	%	-	-	10
Sum of BTEX	mg/kg	6	-	-
Sum of 7 PCBs	mg/kg	1	-	-
Mineral Oil	mg/kg	500	-	-
PAH Sum of 17	mg/kg	100	-	-
pH	ph Units	-	>6	-
ANC to pH 6	mol/kg	-	-	-
ANC to pH 4	mol/kg	-	-	-
Eluate Analysis				
Arsenic	mg/l	0.5	2	25
Barium	mg/l	20	100	300
Cadmium	mg/l	0.04	1	5
Chromium	mg/l	0.5	10	70
Copper	mg/l	2	50	100
Mercury Dissolved (CVAF)	mg/l	0.01	0.2	2
Molybdenum	mg/l	0.5	10	30
Nickel	mg/l	0.4	10	40
Lead	mg/l	0.5	10	50
Antimony	mg/l	0.06	0.7	5
Selenium	mg/l	0.1	0.5	7
Zinc	mg/l	4	50	200
Chloride	mg/l	800	15,000	25,000
Fluoride	mg/l	10	150	500

Sulphate (soluble)	mg/l	1,000	20,000	50,000
Total Dissolved Solids	mg/l	4,000	60,000	100,000
Total Monohydric Phenols (W)	mg/l	1	-	-
Dissolved Organic Carbon	mg/l	500	800	1000

Key: mg/kg = milligrams per kilogram, mg/l = milligrams per litre, mol/kg = moles per kilogram

The result of the tests shows the material to be acceptable for inert waste landfill.

13.5 Ikkari Metallurgical Test Work

The Ikkari test work was carried out in two phases.

13.5.1 Head Grade

The sample used for the first batch 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 head grade for Au averaged between 3.6 g/t Au to 4.2 g/t Au. It is considered that the result showing 9.54 g/t was an outlier. The silver content of the sample was shown to be 0.4 g/t.

The sample used for the second batch of tests indicated that the metal content was 3.14 g Au/t Au and 0.5 g Ag/t. Sulphide content is shown to be 1.3% and organic carbon content is shown to be 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 the weighted average of the back-calculated gold grade was lower (1.81 mg Au/kg) than indicated in the head assay. This value has been taken as the true gold head grade.

13.5.2 Comminution

The results of the comminution test work showed the material to be abrasive and harder than “medium” with a BWi of 15.5 kWh/t.

SMC Test Work

The results of the SMC test work are shown in Table 13.4 and

Table 13.5. The test work results are compared to the SMC classification (Table 13.6), showing that the material is harder than medium and abrasive.

Table 13.4 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 13.5 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

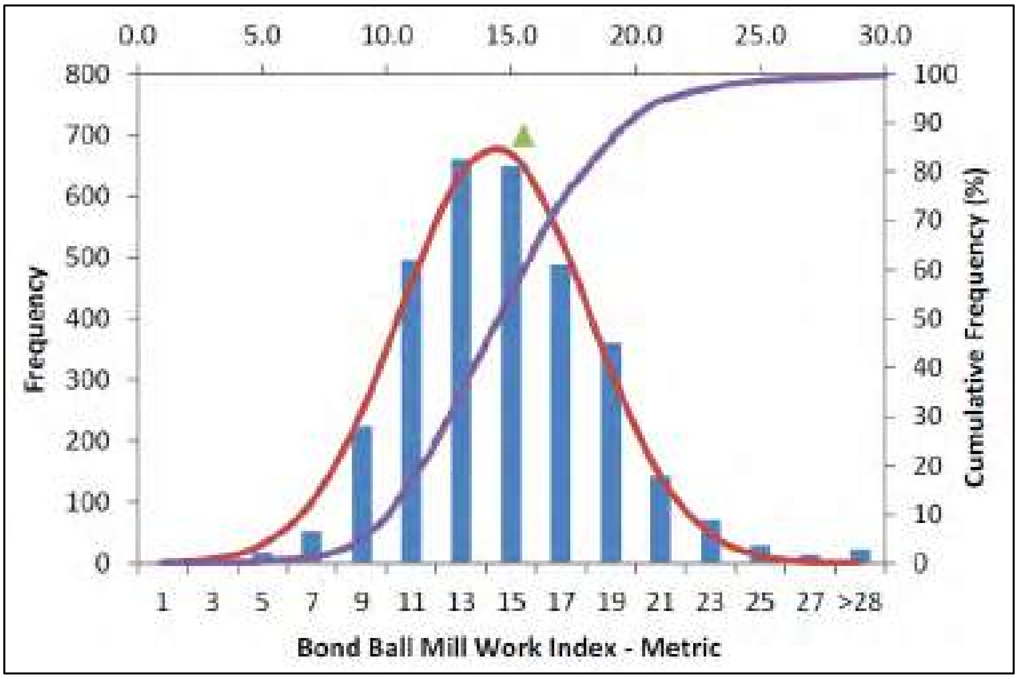
Table 13.6 Hardness Classification for the SMC Results

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

BOND ROD GRINDABILITY

Testing shown in Figure 13.6, shows that the material has a Bond Ball Index of 15.5 kWh/t (metric), indicating a medium hardness.

Figure 13.6 Comparison of Bond Ball Index Result to Data Base



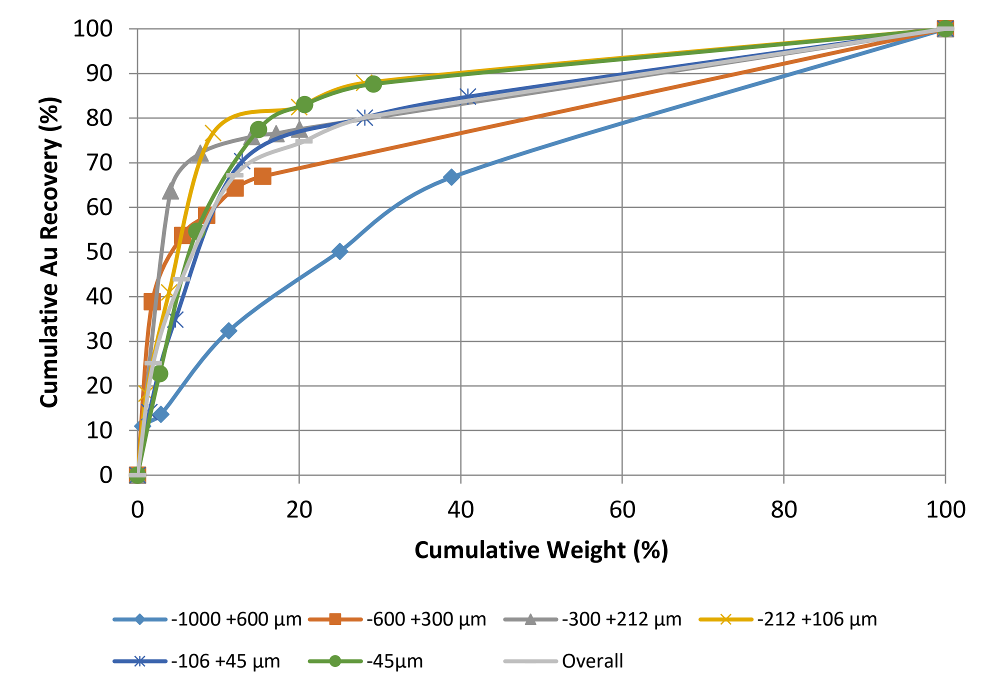
13.5.3 GRAVITY RECOVERY

The gravity release shows that for a grind below 600 µm gravity concentration is viable. The gravity recoverable and GRG test showed that gold recovery of 65.22% with a mass pull of 2.14% was achievable.

GRAVITY RELEASE ANALYSIS

Figure 13.7 shows there is a steep cumulative gold recovery for the particle size less than 600 µm. Showing that gravity recovery is a viable concentration and recovery method.

Figure 13.7 Mass Pull vs Au Recovery Curves for Gravity Release Analysis



GRAVITY RECOVERABLE GOLD AND E-GRG TEST

Gravity recoverable gold testing showed that the sample contained a GRG recovery of 65.22% achieved to a mass pull of 2.14%. The test work showed that the concentrate could be further concentrated in an additional cleaner stage.

E-GRG testing on the concentrate sample demonstrated a GRG recovery of 47% at a mass pull of 1.15% over three moderately coarse grind recovery stages. The head grade was back-calculated and found to be 1.7 g Au/t.

13.5.4 FLOTATION

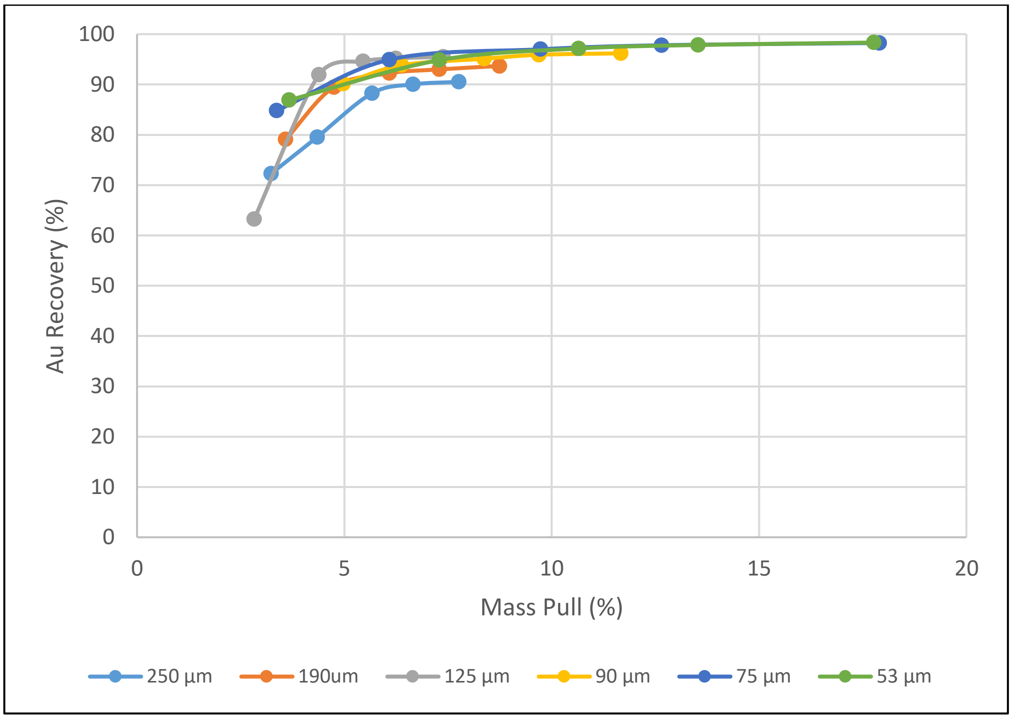
The flotation tests with different grind sizes showed that the gold recovery was insensitive to size up to 190 µm. The smaller the grind size resulted in a higher mass pull to achieve the same gold recovery. The reagent optimisation tests showed that the PAX addition could be decreased to 32 g/t. The addition of a cleaner circuit was not viable as the overall recovery decreased significantly. The bulk flotation test using these conditions achieved a gold recovery of greater than 97.4% with a mass pull of 7.21% or less.

GRIND SIZE

Tests were performed at grind sizes up to 250 µm, with results shown in Figure 13.9 and Figure 13.10. Figure 13.9 shows that the different grind sizes gave very similar recovery profiles with gold recovery decreasing at grind sizes of above 190 µm. The gold recovery was largely complete after 8 to 10 minutes of flotation. A grind of 125 µm achieved a gold recovery of 95.5% after 15 minutes of flotation with a mass pull of 7.4%.

Figure 13.10 shows that the silver recoveries were significantly lower and a trend in the results was less apparent.

Figure 13.8 Mass Pull vs Gold Recovery for Grind Sizes Between 106 µm and 190 µm



Additional test work was carried out for the grind sizes between 106 µm and 190 µm to improve on the resolution of the results. The results of the tests are shown in Figure 13.8.

Figure 13.9 Kinetic Au Recovery to Rougher Flotation Concentrate – Mesh of Grind Flotation Tests 106 µm to 190 µm

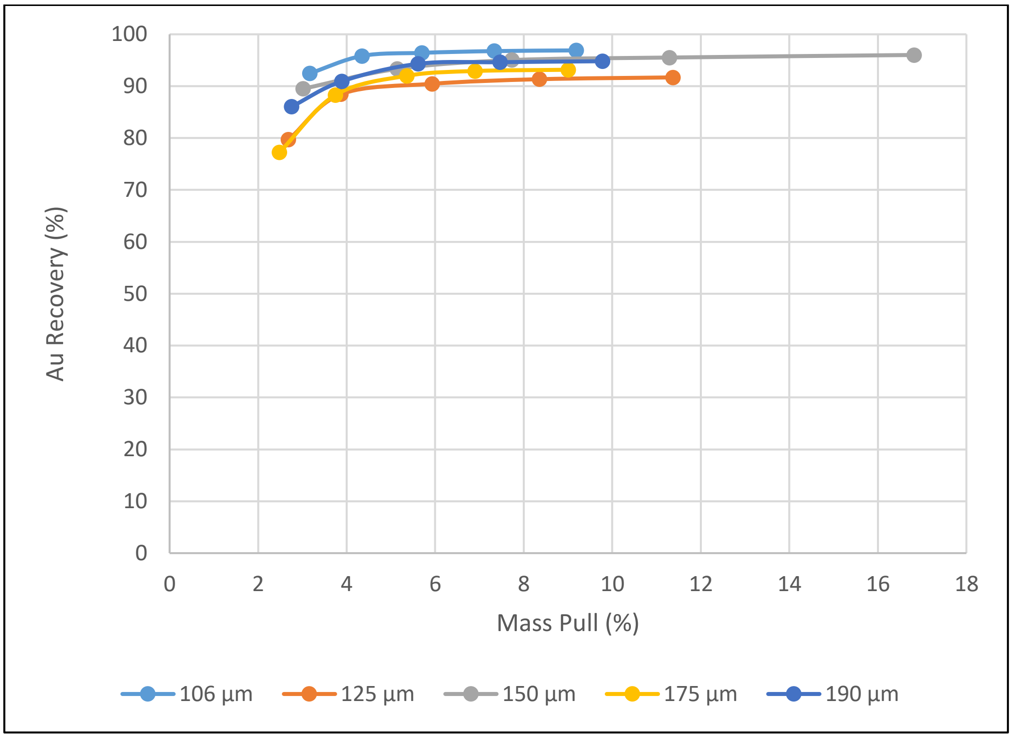
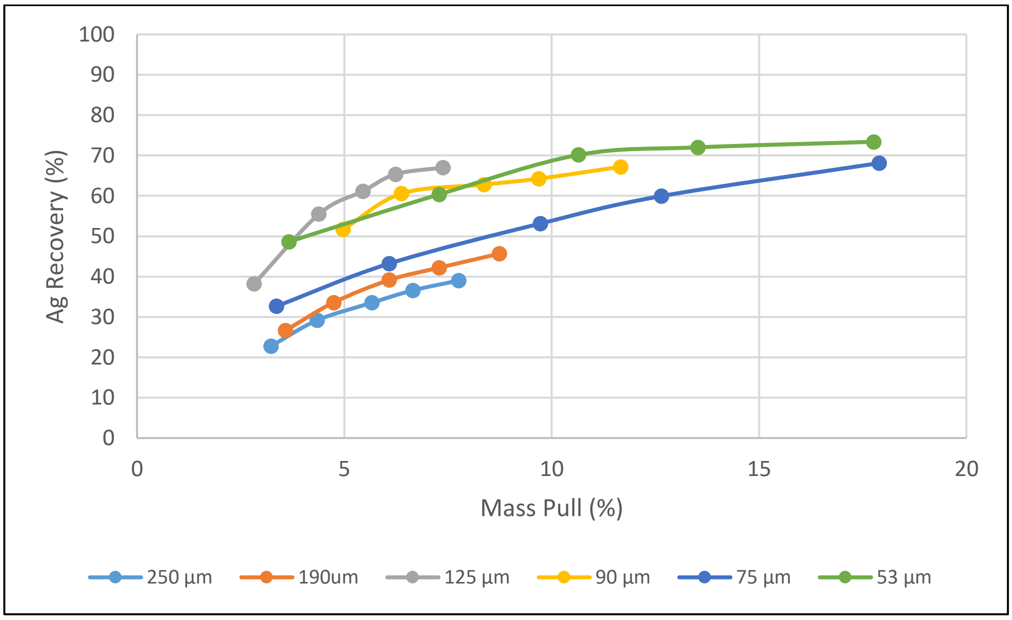


Figure 13.8 shows that at a grind of 106 µm achieved the highest recovery has a higher, above 95%, gold recovery under the same conditions.

Figure 13.10 Kinetic Au Recovery Plot for Mesh of Grind Flotation Tests



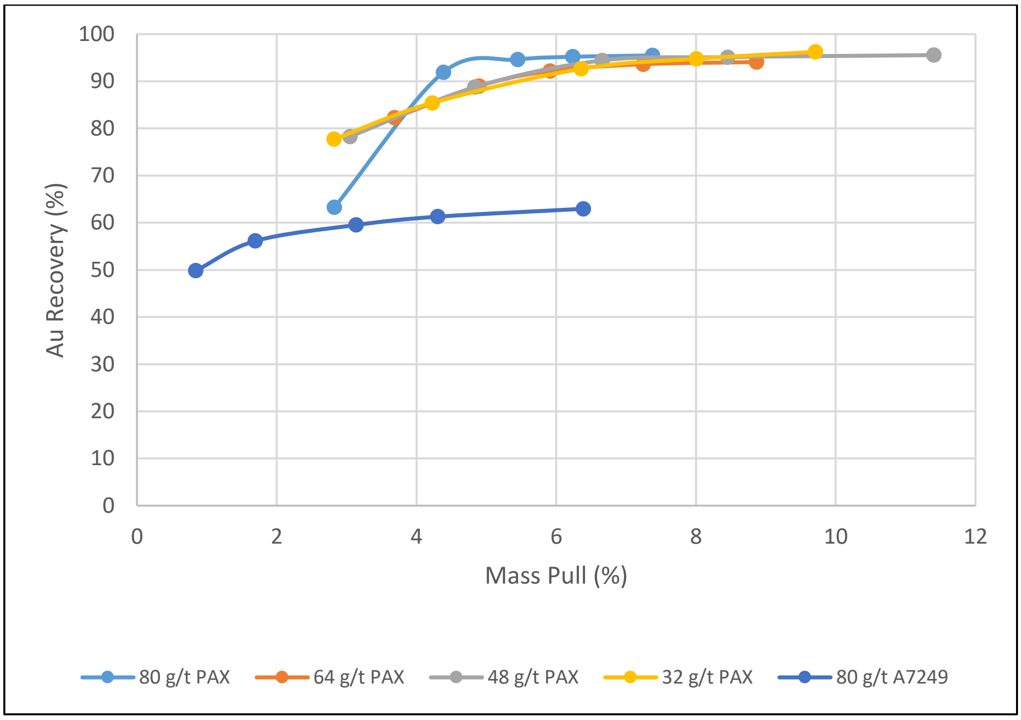
Reagent

Reagent optimisation test work was done for different reagent addition rates and using an alternative flotation collector at a grind of 125 µm. The results of the tests are shown in Figure 13.11.

Gold recovery was little impacted by reducing the PAX dosage from 80 g/t to 32 g/t with gold recoveries ranging between 94.1% and 96.2% for tests. The test conducted using Aero 7249 showed a much lower gold recovery.

The sulphur recovery was also significantly lower compared with the PAX achieved sulphur recoveries of over 90% implying that this reagent was more selective for gold despite the lower recovery.

Figure 13.11 Kinetic Recovery Plots for Reagent Scoping Flotation Tests



Further reagent optimisation was done with a wider variety of reagents and addition rates on gravity recoverable tailings that had a P^{80} of 185 µm. The results of the tests are shown in Table 13.7.

Table 13.7 Summary of Test Conditions and Results of Rougher Reagent Flotation Tests

Test	Charge Mass kg	% Solids	Grind Size µm	Na2SiO3 g/t	A5688 g/t	Aero 8045 g/t	Aero MX983 g/t	PAX Dosage g/t	MIBC g/t	Mass Pull %	Au Recovery %	Au Grade g/t
FT6	1	33	185*		30	-	-	50	10	17.18	94.17	3.89
FT7	1	33	185*		-	30	-	50	10	11.64	70.12	5.52
FT8	1	33	185*		-	-	30	50	10	23.12	90.25	2.65
FT9	1	33	185*	500	-	-	-	80	10	10.61	96.44	3.13
FT10	1	33	185*		-	-	-	50	10	12.32	93.58	5.35
FT11	1	33	185*		-	-	-	32	10	18.27	94.48	5.36
FT12	1	33	185*		-	-	-	20	10	12.68	91.73	7.64
FT13	1	33	185*		-	-	-	15	10	20.00	88.82	5.99
FT14	1	33	150	500	150	-	-	-	10	20.00	88.82	6.99

* GRG Tailings

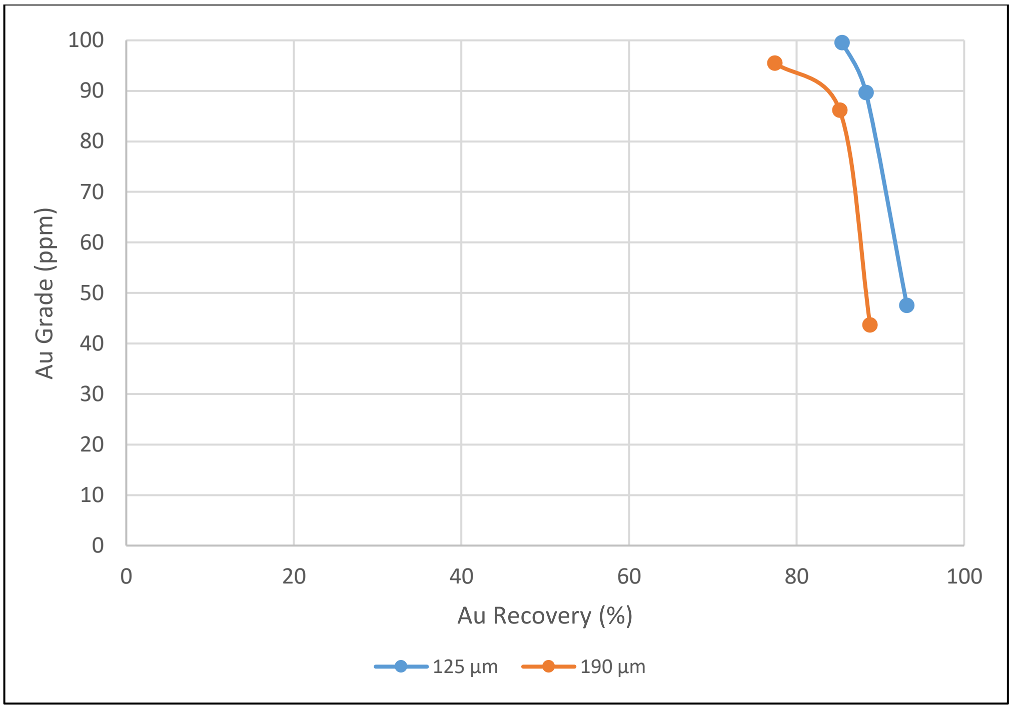
The results show that the highest gold recovery was achieved during test FT9 where 80 g/t of PAX was used. Tests FT6 and FT11 also performed well with gold recoveries over 94% to the rougher concentrate.

CLEANER FLOTATION TESTS

Two stages of rougher cleaner tests was done on the two grind sizes, 125 µm and 190 µm, to produce a gold grade recovery curve. The results of the tests are shown in Table 13.12. The 125 µm had a better recovery of gold than the 190 µm.

The results show that for the test performed using a 125 µm primary grind size the rougher recovery reached 93% Au. After a single stage of cleaning, this dropped to 88% Au recovery and then to 85% Au recovery after the second stage of cleaning. These recovery numbers represent a stage recovery between the rough and cleaner 1 stage of 94% and between cleaner 1 and 2 of 96%.

Figure 13.12 Au Grade Recovery Curves for Cleaner Tests Performed at 125 µm and 190 µm Primary Grind Sizes



BULK FLOTATION

The performance of a bulk flotation test using the selected primary grind size of 125 µm, test conditions shown in Table 13.8, saw a gold recovery of 99.70% to a rougher concentrate with a mass pull of 5.97% yielding a gold grade to the concentrate of 60.0 g/t Au.

Table 13.8 Conditions Used for Bulk Flotation Test

Float Test	Grind Size	Float Time	pH	PAX Dosage g/t	MIC
Bulk	125	15 min	Natural	80	As required

The bulk flotation test was repeated to generate more flotation concentrate sample for leach testing, conditions shown in Table 13.9.

Table 13.9 Operating Conditions for Additional Bulk Flotation Test

Charge Mass	% Solids	Grind Size	Na_2Sio_3 g/t	PAX Dosage g/t	MIBC g/t
150	33	175	500	32	10

The results of the test show that a mass pull of 6.55% was achieved to the flotation rougher concentrate and a gold recovery of 97.46%. The grade of this product was 27.40 g/t Au. This result is comparable to rougher tests performed using similar reagent regimes on fresh feed material.

The bulk flotation test was repeated to generate more flotation concentrate sample for leach testing. This was done using the same conditions shown in Table 13.9. The results indicate that 96.3% of the gold was recovered to the float concentrate to a mass pull of 7.21% producing a gold grade of 16.72 mg/kg Au.

LOCKED CYCLE FLOTATION

A locked cycle test was performed on the tailings generated from the GRG test carried out with a P^{80} of 185 µm. The results of the locked cycle test show that a gold recovery from the GRG tailings to a flotation cleaner 2 concentrate of 88.2% was achieved to a concentrate grade of 31.95 g/t Au.

This provides a combined recovery of gold to GRG concentrate and to flotation cleaner 2 concentrate of 93.75% at a grade of 50.02 g/t Au.

13.5.5 LEACHING

The whole ore cyanidation testing on grind sizes between 106 µm and 38 µm had gold extractions greater than 94.5%. The extraction results showed that grind had little effect on extraction.

The leaching of flotation concentrate did not vary significantly with different grind sizes. The addition of lead nitrate increased the gold leaching kinetics but decreased the overall extraction. The introduction of an oxygen preconditioning stage did not improve the leach kinetics. The leaching of the flotation showed that cyanide extractable gold in the flotation tailings samples was between 47% and 56%.

GRIND WHOLE ORE LEACHING

The gold and silver extraction kinetic results are shown in Figure 13.13 and Figure 13.14. The results show that there is little improvement in the gold extraction after 24 hours. It can be concluded that particle size is not a major factor in limiting gold extraction for the tested particle sizes. When comparing the extraction for the different grind sizes, there is very little improvement in the gold extraction for smaller grind sizes. Silver showed that there was an improvement in the extraction with smaller grind sizes.

The cyanide consumption (NaCN) for the tests ranged between 0.329 and 0.523 kg/t of feed showing a slight increase with the finer grind sizes. Lime consumption was shown to range between 0.312 to 0.586 kg/t of feed again showing an increase in finer grind sizes.

Figure 13.13 Kinetic Extraction Plots for Au for the Mesh of Grind Cyanidation Tests

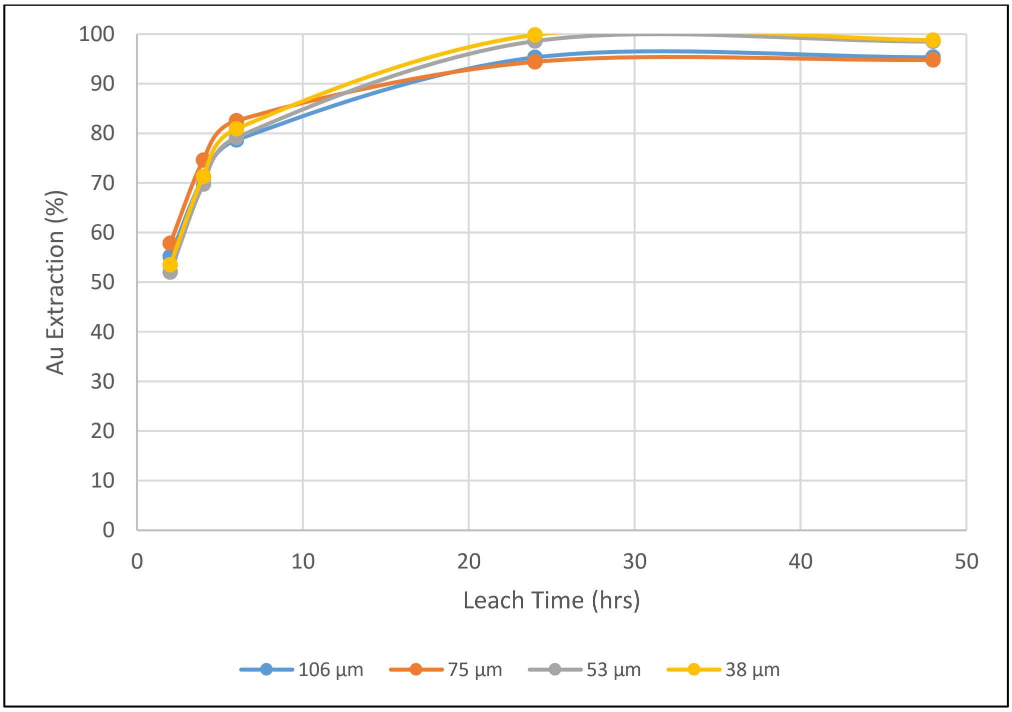
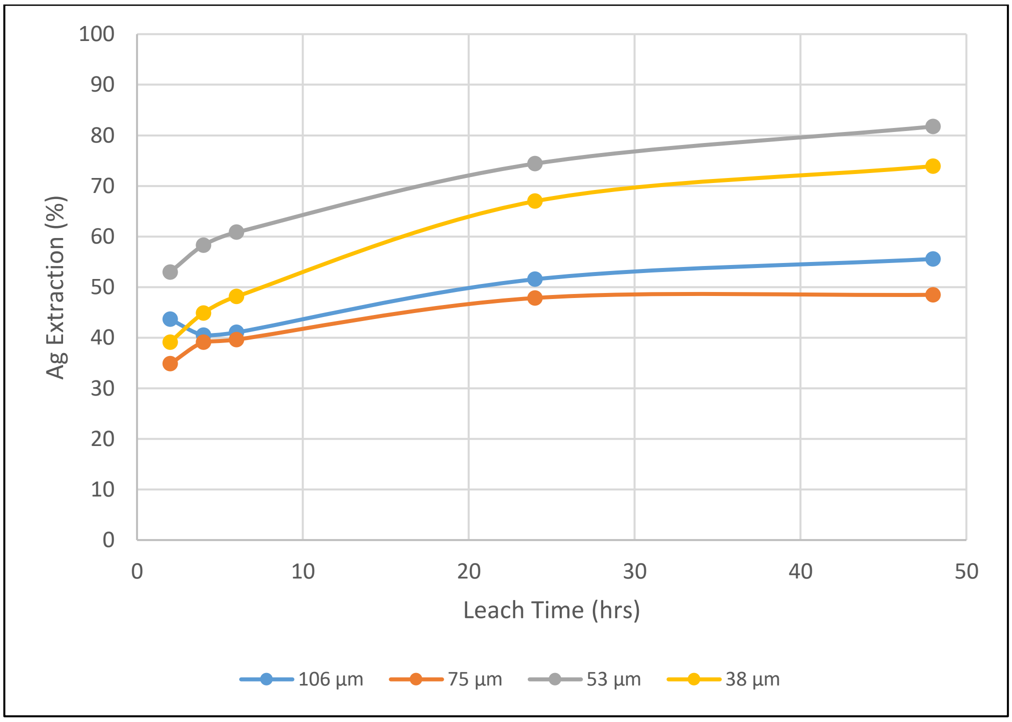


Figure 13.14 Kinetic Extraction Plots for Ag for the Mesh of Grind Cyanidation Tests



WHOLE ORE LEACHING

Whole ore cyanidation testing showed that high gold extractions were seen for all grind sizes tested. Gold recovery ranged between 94.79% to 98.79% for the tests conducted at grind sizes between 106 µm and 38 µm. A trend for higher recovery was seen for the two tests conducted at the finer grind sizes of 53 µm and 38 µm. Cyanide consumptions for the tests were between 0.329 kg/t to 0.523 kg/t NaCN. Lime ($Ca(OH)_2$) consumption ranged between 0.312 kg/t to 0.586 kg/t.

FLOTATION CONCENTRATE LEACHING

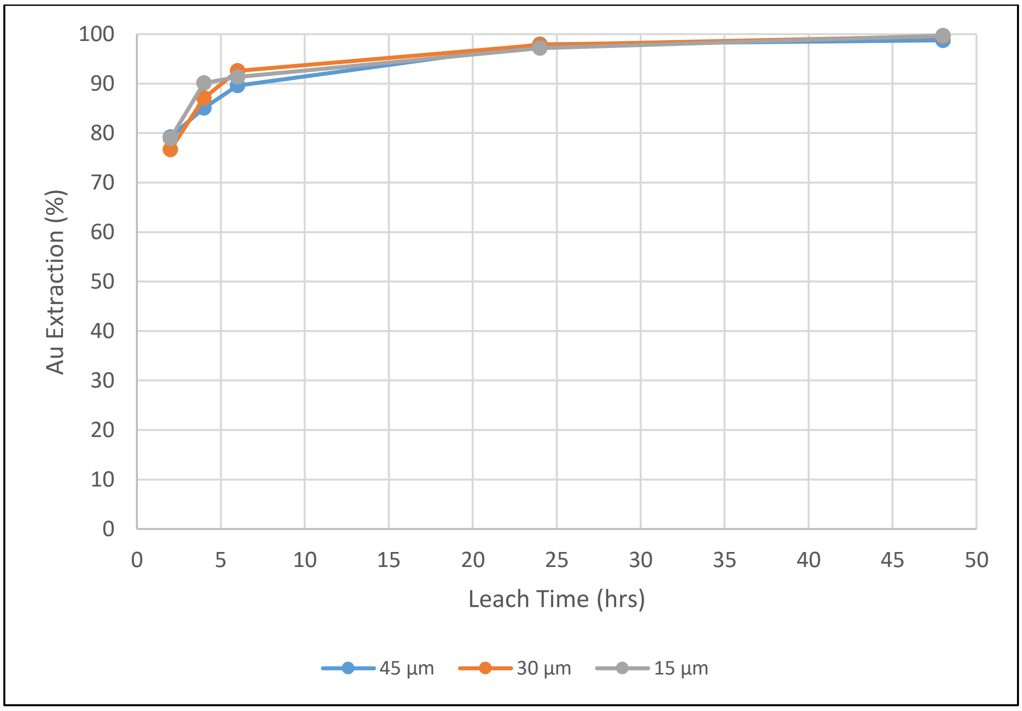
The leaching of flotation concentrate did not vary significantly with different grind sizes. The addition of lead nitrate increased the gold leaching kinetics but decreased the overall extraction. The introduction of an oxygen preconditioning stage did not improve the leach kinetics.

Effect of Grind on Gold Extraction

Flotation concentrate was subjected to further grinding and leaching. The results of the test are presented in Figure 13.15.

The results show that excellent gold extractions were seen for the tests at all grind sizes tested. The highest gold extraction was seen in the test performed at a regrind size of 15 µm where an extraction reached 99.7% and was lowest in the coarsest regrind size test where gold extraction was 98.75%.

Figure 13.15 Au Kinetic Extraction Plots for Intensive Cyanidation Tests Performed on Reground Bulk Flotation Rougher Concentrate



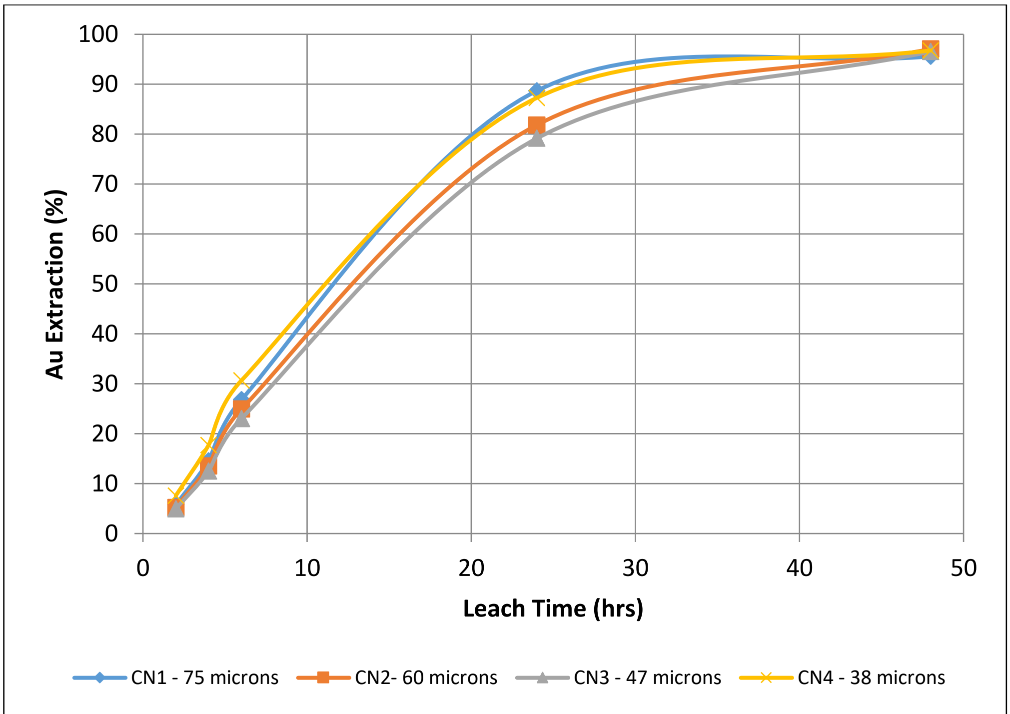
Cyanide consumption ranged between 17 kg/t and 26 kg/t with a trend for increasing consumption at the finer grind size tested which is explained through the increased surface area of the sulphide content. Lime consumption ranged between 6.00 kg/t and 40.39 kg/t with increasing consumption seen at the finer re-grind sizes.

The above test was repeated for additional grind sizes. The test conditions are shown in Table 13.10, with results showing in Figure 13.16 . The results show that gold extractions for all tests were high over 95.5% after 48 hours.

Table 13.10 Operating Conditions for Mesh of Grind Cyanidation Tests

Test	Mass g	Grind Size µm	% Solids	NaCN g/l	PH	$PbNO_3$
CN1	580	75	40	2	10.5 - 11	0
CN2	580	60	40	2	10.5 - 11	0
CN3	580	47	40	2	10.5 - 11	0
CN4	580	38	40	2	10.5 - 11	0

Figure 13.16 Kinetic Extraction Plots for Au During Mesh of Flotation Concentrate Grind Cyanidation Tests

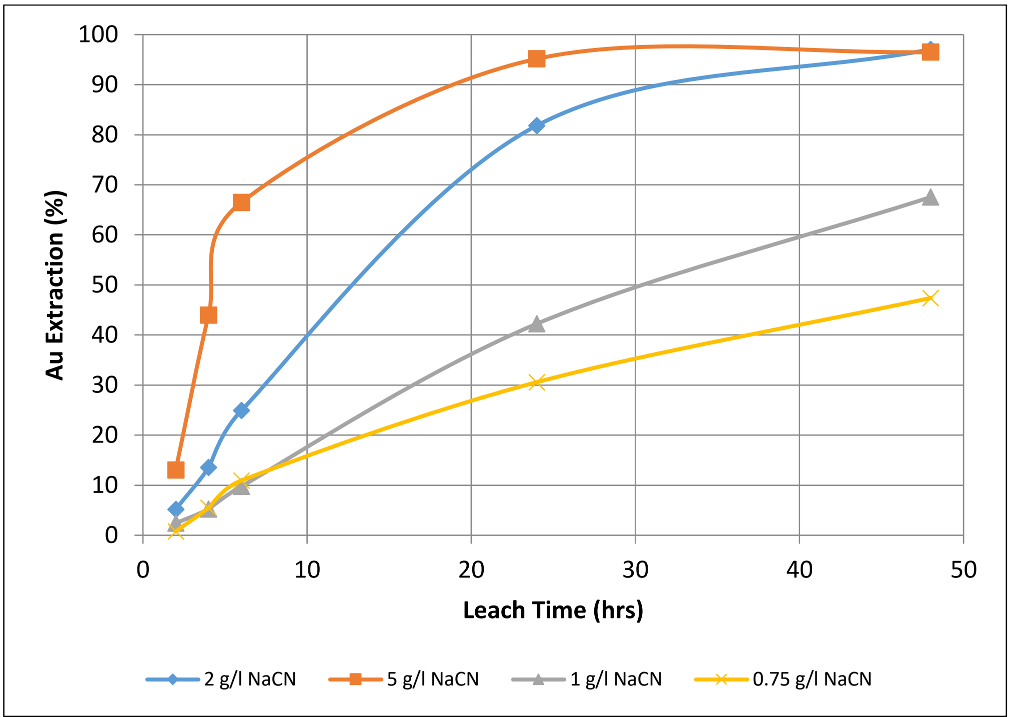


Effect of Cyanide Concentration on Gold Extraction

Table 13.11 Operating Conditions for Cyanide Dosage Cyanidation Tests

Test	Mass g	Grind Size µm	% Solids	NaCN g/l	PH	$PbNO_3$
CN5	580	60	40	5	10.5 - 11	0
CN6	580	60	40	1	10.5 - 11	0
CN7	580	60	40	0.75	10.5 - 11	0

Figure 13.17 Kinetic Extraction Plots for Au during Cyanide Dosage Cyanidation Tests



Cyanide consumptions for the tests ranged between 2.43 kg/t and 3.85 kg/t NaCN showing a trend of increasing consumption with increasing cyanide dosage as expected. Lime consumption ranged between 0.66 kg/t to 0.95 kg/t $Ca(OH)_2$ with a trend showing increasing consumption rates with increasing cyanide dosage.

Effect of Lead Nitrate Addition on Leach Kinetics

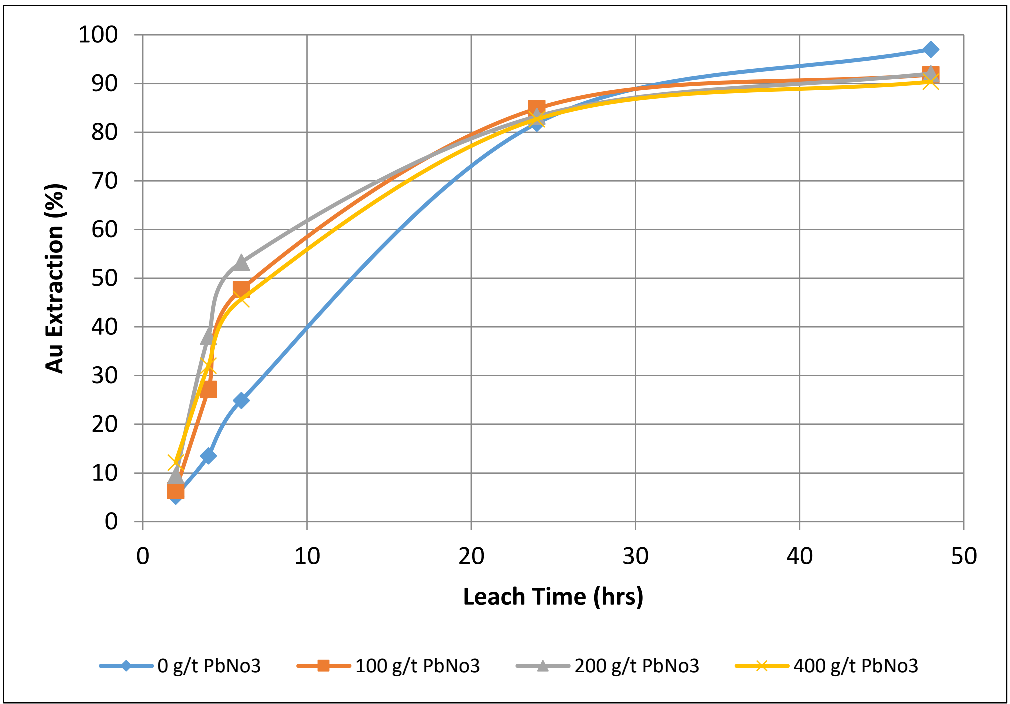
A series of cyanide leach tests were completed to investigate the addition of lead nitrate to monitor its addition on leach rate and reagent consumption. Conditions for the tests are shown in Table 13.12.

Table 13.12 Operating Conditions for Lead Nitrate Cyanidation Tests

Test	Mass g	Grind Size µm	% Solids	NaCN g/l	PH	$PbNO_3$
CN8	580	60	40	2	10.5 - 11	100
CN9	580	60	40	2	10.5 - 11	200
CN10	580	60	40	2	10.5 - 11	400

The results show, in Figure 13.18, that after 48 hrs the gold extraction was lower with the addition of lead nitrate than without. Gold extractions to solution ranged between 90% and 92% with the addition of lead nitrate compared to 97% (CN_2) without. However, the initial kinetics of gold extraction were faster with the addition of lead nitrate.

Figure 13.18 Kinetic Extraction Plots for Au during Lead Nitrate Cyanidation Tests



Cyanide consumptions ranged between 3.24 kg/t and 3.48 kg/t NaCN. No clear trend between lead nitrate addition rates and cyanide consumptions were observed. Lime consumptions ranged between 0.41 kg/t and 0.75 kg/t $Ca(OH)_2$ with a trend of decreasing lime consumption with increasing lead nitrate addition.

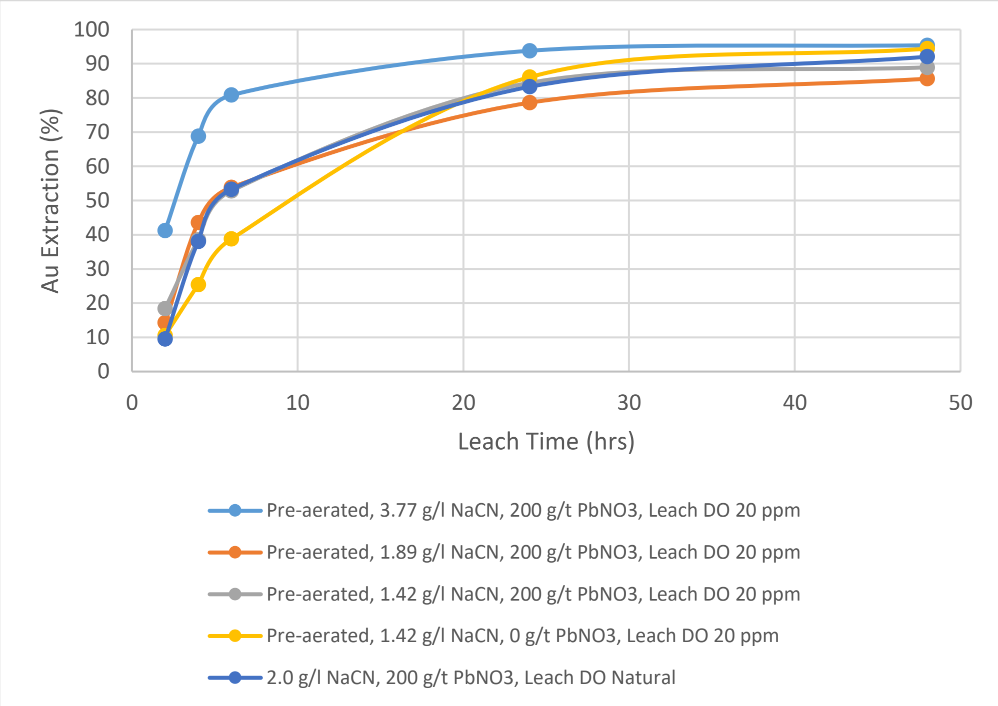
Effect of Oxygen Preconditioning

The effect of oxygen addition in a preconditioning step and during the leach on the gold extraction kinetics and reagent consumptions was tested. The conditions of the test work are shown in Table 13.13.

Table 13.13 Conditions Used During Pre-aerated Cyanide Leach Tests

Test	Pre-aeration	Pre-aeration DO2 mg/l	NaCN g/l	$PbNO_3$	Leach DO2 mg/l
1	Y	40+	3.77	200 g/t	20
2	Y	40+	1.89	200 g/t	20
3	Y	40+	1.42	200 g/t	20
4	Y	40+	1.42	0 g/t	20
22-1967 CN2	N	-	2.00	200 g/t	Natural

Figure 13.19 Kinetic Au Extraction Plots for Pre-aerated Cyanide Leach Tests



The results, shown in Figure 13.19, show that the gold extraction to solution for the tests conducted with pre-aeration and aeration during leach ranged between 85.63% to 95.41%. Cyanide consumptions during the test work ranged between 2.78 kg/t and 3.96 kg/t NaCN showing increasing consumption with increasing cyanide dosage rate.

There was no improvement in the extraction or the kinetics with a pre-aeration step or the higher dissolved oxygen level (DO).

Leaching of Flotation Tailing

The flotation tailings generated from the bulk flotation tests and locked cycle tests (LCT) was subjected to cyanide leach test. The condition for the test is shown in Table 13.14.

Table 13.14 Conditions for Cyanide Leach Testing of Flotation Tailings Samples

Sample	Mass	% Solids	NaCN g/l	Retention Time Hrs
LCT Tails	1 kg	40	1	48
Bulk Flotation Tails	1 kg	40	1	48

Table 13.15 Summary of Flotation Tailings Cyanide Leach Results

Leach Feed Sample	Au Extraction to Solution %	Au in Residue g/t	Back Calculated Head g/t Au	Cyanide Consumption kg/t NaCN
LCT Tailings	46.91	0.07	0.14	0.12
Bulk Flotation Tailings	56.31	0.06	0.14	0.12

The back calculated gold grades for the tests both came back at 0.14 mg/kg Au, significantly higher than the previously reported direct assay of 0.07 and 0.06 for the LCT and bulk flotation tailings respectively. The results show that cyanide extractable gold in the flotation tailings samples was between 47% and 56%.

The cyanide consumptions for both tests were 0.12 kg/t NaCN.

13.5.6 Cyanide Destruction

Cyanide destruction using the Inco SO_2/air method resulted in most of the cyanide being broken down with the residual cyanide being less than 3 ppm.

INCO SO_2 / Air Method

The pulp from the bulk leach was submitted for INCO SO_2 / air cyanide destruction testing. The batch test showed that a higher pH was required. Table 13.16 shows the analysis of the feed to the test with Table 13.17 showing the conditions used for the continuous cyanide destruction test.

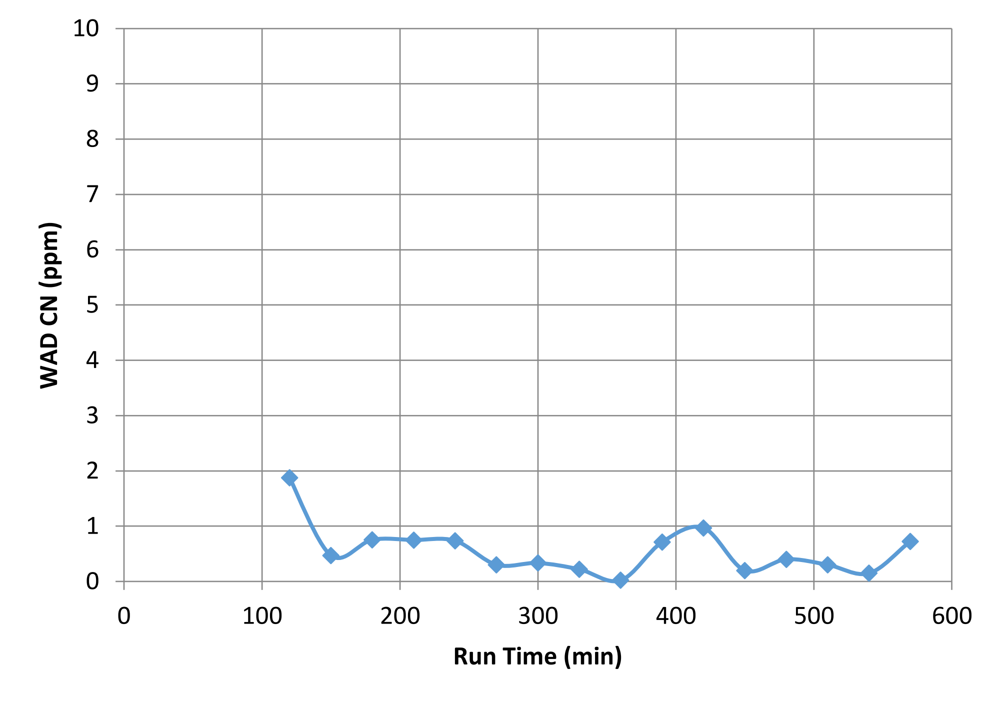
Table 13.16 Feed Analysis of the Feed to INCO Cyanide Destruction

Feed Solution Analysis	Mg/l
Copper	551.00
Zinc	0.59
Iron	6.41
Nickel	1.70
Cyanide total	1,130.00
Cyanide wad	770.00

Table 13.17 Test Operating Conditions for INCO Test

Parameter	Value
Feed Solids	30%
SO_2 g/g CN wad	6.5
$CuSO_4$	NA
pH Target	9.5
Retention Time mins	180

Figure 13.20 Plot of WAD Cyanide Levels in Cyanide Destruction Reactor Discharge



The results, shown in Figure 13.20, of the continuous test show that the WAD Cyanide concentration levels was maintained below 1ppm. An ICP scan of the final sample had a WAD cyanide concentration of 2.8 ppm.

13.5.7 Solid Liquid Separation

Flocculant Screening

Flocculant screening was carried out on the flotation tailings. The flocculants tested were:

- No flocculant.
- Nasfloc 2286.
- Nasfloc 2132.
- Nasfloc 2326.
- Nasfloc 2354.

The results showed that Nasfloc 2354 was the most effective.

Flotation Tailings

Initial solid liquid separation testing has shown that the bulk flotation tailings responded well to flocculation and settling. The results from the test work are shown in Table 13.18.

Table 13.18 Summary Results of Settling Tests Performed on Bulk Flotation Tails

Settling Test	Feed Solids %	Flocculant	Dosage g/t	Settling Rate m3/m2/d	Thickener Unit Area Underflow M2/MTPD	Solids % at Compression Point	Final Underflow Solids Density %
1	10	N2354	14.25	4818.53	0.639	63.0	71.6
2	10	N2354	21.37	4744.92	0.332	61.2	66.8
3	10	N2354	28.26	4525.9	0.245	53.4	60.8
4	10	N2354	35.62	5096.4	0.123	63.0	67.5
5	12.5	N2353	34.74	4553.28	0.187	60.8	65.7
6	15	N2354	34.85	4553.28	0.136	62.3	67.3
7	17.5	N2354	40.00	1689.63	0.228	52.6	61.5
8	20	N2354	39.33	4142.02	0.166	58.5	61.7

Initial settling rates in the order of 4500 m^3/m^2.day were observed. Changing of the flocculant addition from 10 g/t to 35 g/t had little effect on the settling rate. The thickener unit area underflows of 0.187 m^2/MTPD achieved with feed-well solids contents of 10%.

13.5.8 ENVIRONMENTAL TESTS

Chemical analysis of each of the waste streams for U, Th, Cd, and Hg below detection limits of 0.0001%. Four of the five waste rock samples tested were classified as acid neutralising. A sample of the flotation tailings was tested and found to be acid neutralising. The waste rock samples were tested for landfill compliance and found to be acceptable for inert waste landfill.

WASTE ROCK CHEMICAL ANALYSIS

Analysis of each of the waste streams for U, Th, Cd, and Hg were below detection limits of 0.0001%. The waste streams analysed were:

- Gravity tails.
- Flotation Tails.
- Leach Residue.
- Cyanide Destruction Residue.

ACID BASE ACCOUNTING TEST

There were five waste rock samples that were tested. Four of the five were classified as acid neutralising and one sample was classified as potentially acid generating.

A sample of the flotation tailings was tested and found to be acid neutralising.

NET ACID GENERATION TEST

The results from the net acid generation test on the five waste rock samples and the flotation tailings sample was showed a final pH of between 8.23 and 12.03

WASTE COMPLIANCE TEST EN12457-3

Five waste rock samples and a sample of the flotation tails was sent for waste compliance testing. Compliant limits are shown in Table 13.19.

Table 13.19 Landfill Waste Compliance Acceptance Criteria Limits

Solid Waste Analysis	Units	Landfill Waste Acceptance Criteria Limits		
		Inert Waste Landfill	Stable Non-reactive Hazardous Waste in Non-Hazardous Landfill	Hazardous Waste Landfill
Total Organic Carbon	%	3	5	6
Loss on Ignition	%	-	-	10
Sum of BTEX	mg/kg	6	-	-
Sum of 7 PCBs	mg/kg	1	-	-
Mineral Oil	mg/kg	500	-	-
PAH Sum of 17	mg/kg	100	-	-
pH		-	>6	-
ANC to pH 6	mol/kg	-	-	-
ANC to pH 4	mol/kg	-	-	-
Eluate Analysis				
Arsenic	mg/l	0.5	2	25
Barium	mg/l	20	100	300
Cadmium	mg/l	0.04	1	5
Chromium	mg/l	0.5	10	70
Copper	mg/l	2	50	100
Mercury Dissolved (CVAF)	mg/l	0.01	0.2	2
Molybdenum	mg/l	0.5	10	30
Nickel	mg/l	0.4	10	40
Lead	mg/l	0.5	10	50
Antimony	mg/l	0.06	0.7	5
Selenium	mg/l	0.1	0.5	7
Zinc	mg/l	4	50	200
Chloride	mg/l	800	15000	25000
Fluoride	mg/l	10	150	500
Sulphate (soluble)	mg/l	1000	20000	50000
Total Dissolved Solids	mg/l	4000	60000	100000
Total Monohydric Phenols (W)	mg/l	1	-	-
Dissolved Organic Carbon	mg/l	500	800	1000

Each of the results were shown to be acceptable for inert waste landfill.

Further Test work underway:

Additional bench scale comparison runs (3 of) are to be completed on bulk milled Ikkari material to provide repeatability data for the envisaged flowsheet. Processing will include GRG, flotation of GRG tails, and leaching of flotation concentrate. Furthermore, the test work is designed to safely produce quantities for environmental testing based on mass pulls defined during previous test phases and provide sufficient material for the cyanide destruction test trials. Additional mass is being processed from the Pahtavaara ore body to produce carbon in pulp (CIP) tailings sample that will be available for environmental testing should this be required in the future.

Approximately 900 kg of Ikkari material and 500 kg of Pahtavaara material are being used to generate samples for the following environmental tests:

Ikkari:

- Flotation tailings (short column).
- CIL tailings after cyanide destruction (short column).
- Flotation tailings (large column).
- CIL tailings after cyanide destruction (large column).

Pahtavaara:

- Existing tailings beach area (short column).
- Existing tailings pond area (short column).
- Tailings TBC (large column).
- Flotation tailings low sulphide (short column).
- Flotation tailings high sulphide (short and large column).

14.0 MINERAL RESOURCE ESTIMATES

14.1 PAHTAVAARA

14.1.1 INTRODUCTION

This Mineral Resource for the Pahtavaara Gold Deposit has been estimated as at the effective date of the 28th November 2022. Gold grade estimation was completed using MIK for the main mineralised domains with the secondary low-grade domains estimated by ordinary kriging (OK). MIK grade estimates have been localised to a selective mining unit (SMU) dimension using an analogous methodology to localised uniform conditioning. This estimation approach was considered appropriate based on review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralisation, and the style, geometry and tenor of mineralisation. The estimation was constrained with geological and mineralisation interpretations.

14.1.2 DATABASE VALIDATION

The resource estimation was based on the available exploration drill hole database which was compiled in-house by Rupert Resources. The database has been reviewed and validated prior to commencing the resource estimation study.

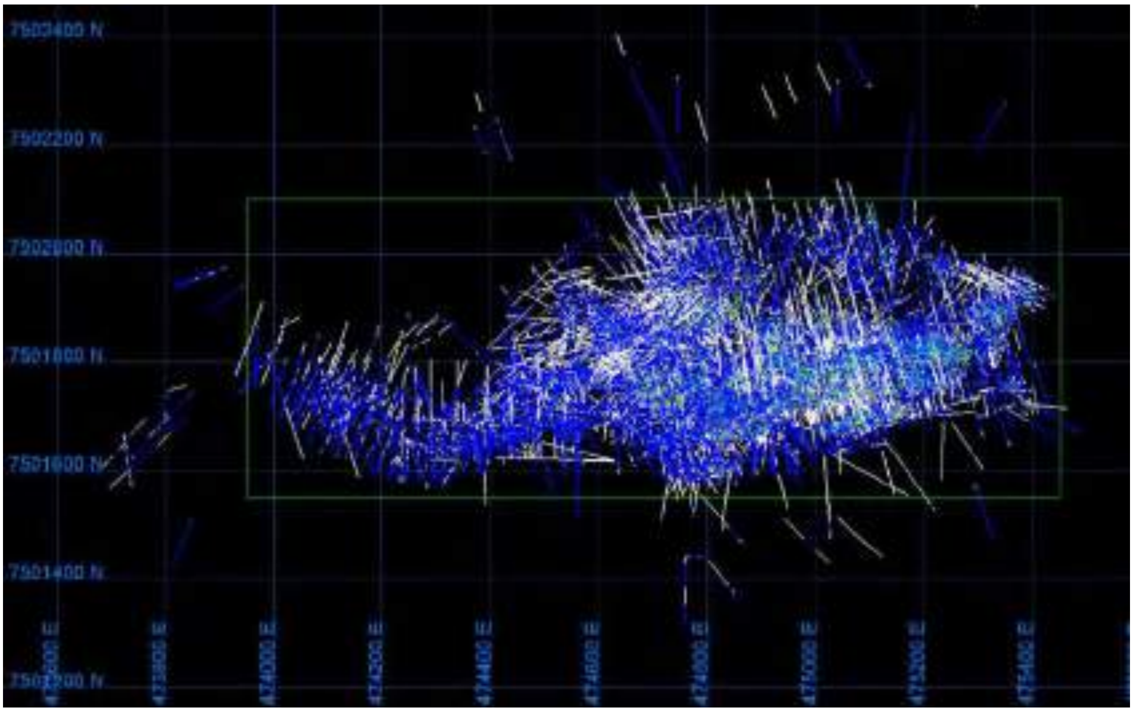
The database consists of surface and underground diamond drilling together with underground sludge sampling, some RC drilling and channel sampling. Database statistics are provided as Table 14.1 and it can be seen that the majority of the data originates from diamond drilling and sludge sampling. A plan view of all drilling is presented in Figure 14.1.

Table 14.1 Summary of the Available Drill Hole Database

Company	DH Type	Holes	Metres	% of Total
Rupert	Diamond	596	71,346	13.4
	RC	32	2,224	0.4
	Channel	590	3,568	0.7
Geological Survey of Finland	Diamond	44	4,372	0.8
Lappland Goldminers	Diamond	1,232	154,573	28.9
	RC	78	1,135	0.2
	Sludge (UG)	6,675	124,867	23.4
	Channel	123	89	0.0
Scan Mining	Diamond	815	94,563	17.7
	RC	21	1,116	0.2
	Sludge (UG)	2,268	49,902	9.3
	Channel	134	213	0.0
Terra Mining	Diamond	152	14,853	2.8
	RC	84	9,976	1.9

	Sludge (UG)	116	117	0.0
	Unknown	8	300	0.1
Unknown	Sludge	18	668	0.1
	Channel	68	107	0.0
Total		**13,054**	**533,989**	**100%**

Figure 14.1 Pahtavaara Gold Deposit, Plan View of all Drilling



Much of the historic drill hole assay database has been selectively sampled. This relates mostly to the diamond drill holes with ~42% of diamond core unsampled and ~7% of sludge drill holes unsampled. For the purposes of the current resource estimate it has been assumed that the unsampled portions of the drill core are essentially unmineralised and therefore those absent intervals in the database have been set to 0.001 ppm Au. In the case of all other unsampled data (sludge etc) the unsampled intervals have been ignored as it is less certain why the intervals remained unsampled. Therefore, all following data analysis is on the basis of the described data substitution.

The resultant database was validated, and the checks made to the database prior to use included:

- Check for overlapping intervals.
- Downhole surveys at 0 m depth.
- Consistency of depths between different data tables.
- Check gaps in the data.
- Replacing less than detection samples with half detection.
- Replacing intervals with no sample with -999.
- Replacing intervals with assays not received with -999.

14.1.3 INTERPRETATION AND MODELLING

MINERALISATION INTERPRETATION

Mineralisation at the Pahtavaara Project is hosted by amphibolitised komatiites. The principal geological control in the area is considered to be a linear structural corridor that trends between east-west and northeast-southwest, with gold mineralisation identified in both the larger structures parallel to this trend and oblique fractures and steeply plunging zones that represent the intersection of these structures or possibly fold hinges. The mineralised structural corridor identified at the Pahtavaara Project is characterised by hydrothermal alteration and mineralisation within komatiites that have been subjected to several phases of intense, pervasive alteration. The hydrothermal alteration and the gold-bearing structures and veins associated are a result of a prolonged period of ductile deformation and later brittle-ductile deformation related to a belt scale thrusting event. Mineralisation occurs over at least 1.4 km of strike length and has been interpreted to extend to more than 500 m below the surface. Mineralisation remains open at depth along the entire zone. Gold occurs mostly as free gold with a smaller proportion associated with magnetite.

Typically for many deposits of this type, the mineralisation often presents as generally somewhat discontinuous and irregularly distributed on the scale of approximately 10 m to 50 m. Figure 14.2 presents a north south sectional view 474,900 m E demonstrating variability in grade, thickness and orientation of gold mineralisation. This commonly makes the traditional approach of wireframing on a sectional and plan basis extremely difficult with multiple plausible geometrical solutions often existing.

To establish appropriate grade continuity, the mineralisation models were therefore based upon a nominal 0.3 ppm Au indicator mineralisation shell estimated using 3 m unconstrained downhole composites. This interpretation is designed to capture the broad mineralisation halo that encompasses the geological vein system and is not intended to constrain individual veins or vein clusters. As the main grade estimation technique is MIK with change of support technique, this type of mineralisation constraint is deemed appropriate.

The mineralisation grade shells were generated by grade estimation via indicator kriging at a single cut-off, 0.3 g/t Au. Grade estimation was into block models with cell dimensions of 5 m E × 5 m N × 5 m relative level (RL). Grade shell triangulations were then generated by constraining the block model at a 20% and 35% probability cut-off (Figure 14.2). The purpose of selecting two probability cut-offs is to generate a nested series of mineralisation constraints. The lower grade shell is in effect forming a lower-grade halo to the main mineralisation. The lower grade shell also serves the function of collecting higher grade data that may have not been included in the main mineralisation shell due to issues with drilling orientation and geometry. The probability cut-offs may be considered somewhat subjective and may seem arbitrary, however were selected based on extensive review of a range of probability cut-off. The selected probability shells are considered optimal to capture the observed continuity and tenor of mineralisation while excluding obvious low-grade material. Grade shells were reviewed in multiple orientations and in plan and section views prior to being accepted for grade estimation and block modelling purposes.

Mineralisation estimation domains were thus defined with further sub-division being differentiated on the basis of orientation, flexures in the shear and tenor of gold grade. A total of 14 main estimation domains (Table 14.2 and Figure 14.3 to Figure 14.5) have been defined. The main mineralisation shells generated at the 35% cut-off are designated with the prefix 35 and are numbered 3510 to 35140. The nested 20% probability cut-off shells that constitute the lower grade envelope to the main mineralisation are designated with the prefix of 20 and are numbered 2010 to 20140.

Figure 14.2 Pahtavaara Gold Deposit, Section 474, 900 m East

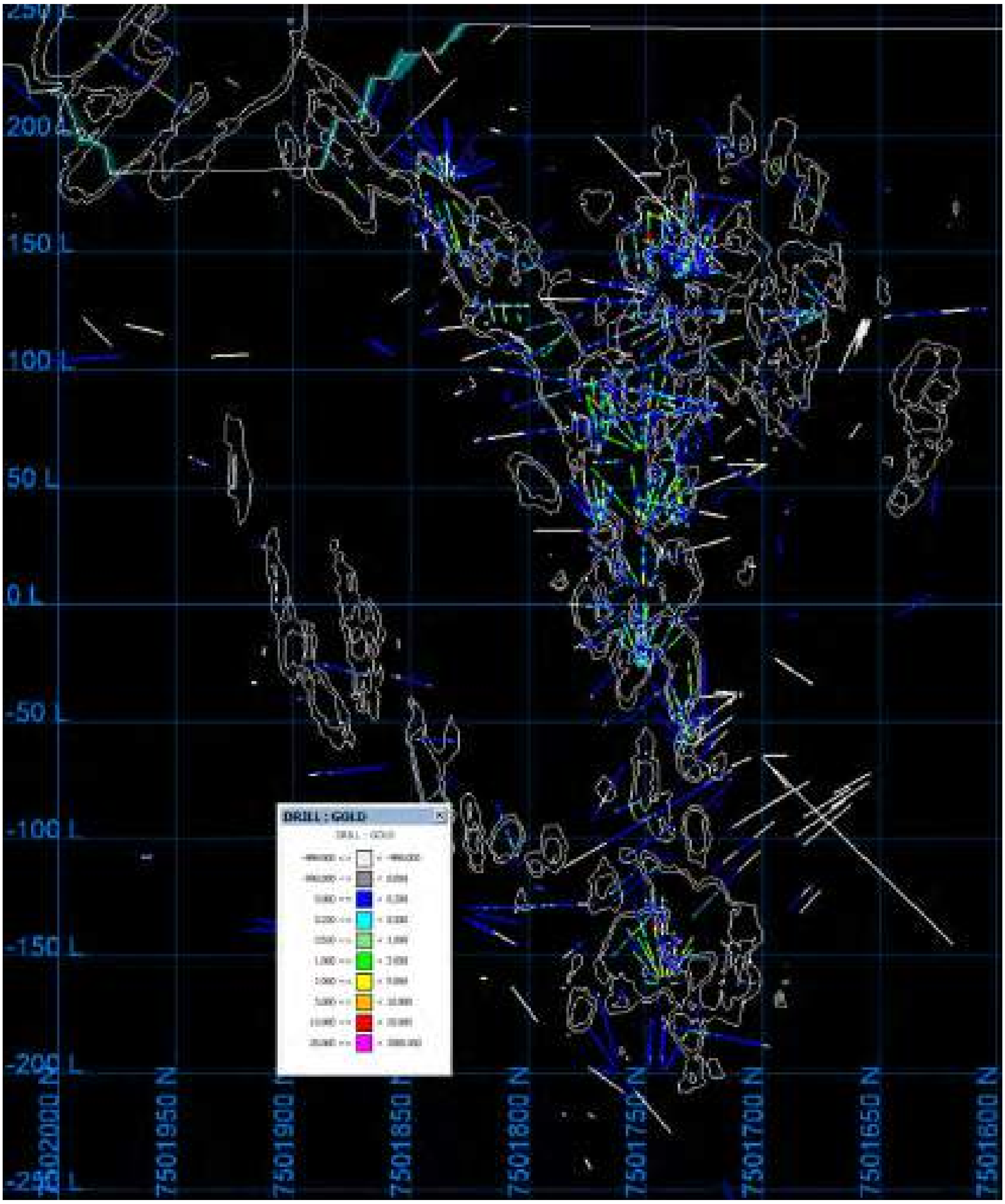


Table 14.2 Pahtavaara Gold Deposit, Estimation Domain Description

Domain	Description	Area
3510	Steeply dipping to sub-vertical to the north on central portion of deposit.	Samurai/NFE
3520	Steeply dip NNW and between 4,490 m E and 4,815 m E.	Karoliina East
3530	Steeply dip NNE and west of 4,490 m E.	Karoliina West
3540	Sub vertical on S flank of the deposit.	T-Zone
3550	Sub vertical on S flank of the deposit, north of 3540.	Samurai
3560	Sub vertical on S flank of the deposit, north of 3540.	Samurai
3570	Sub-vertical with a westerly plunge on SE side.	T-Zone
3580	Steeply dipping with a westerly plunge on E side.	NFE
3590	Steeply dipping with a westerly plunge on lower-central location.	NFE
35100	Steep westerly plunging shoot in central part.	DB/NFE
35110	Steep westerly plunging shoot in central part.	DB
35120	Crescent shape with a westerly plunge at NW side.	DB/Harpoon
35130	Southerly dipping on the NE flank of deposit.	NFE/Samurai
35140	North of 4,800 m N and west of 5,130 m E. Westerly plunging shoots.	Lansi

Figure 14.3 Pahtavaara Gold Deposit, Estimation Domains Plan View

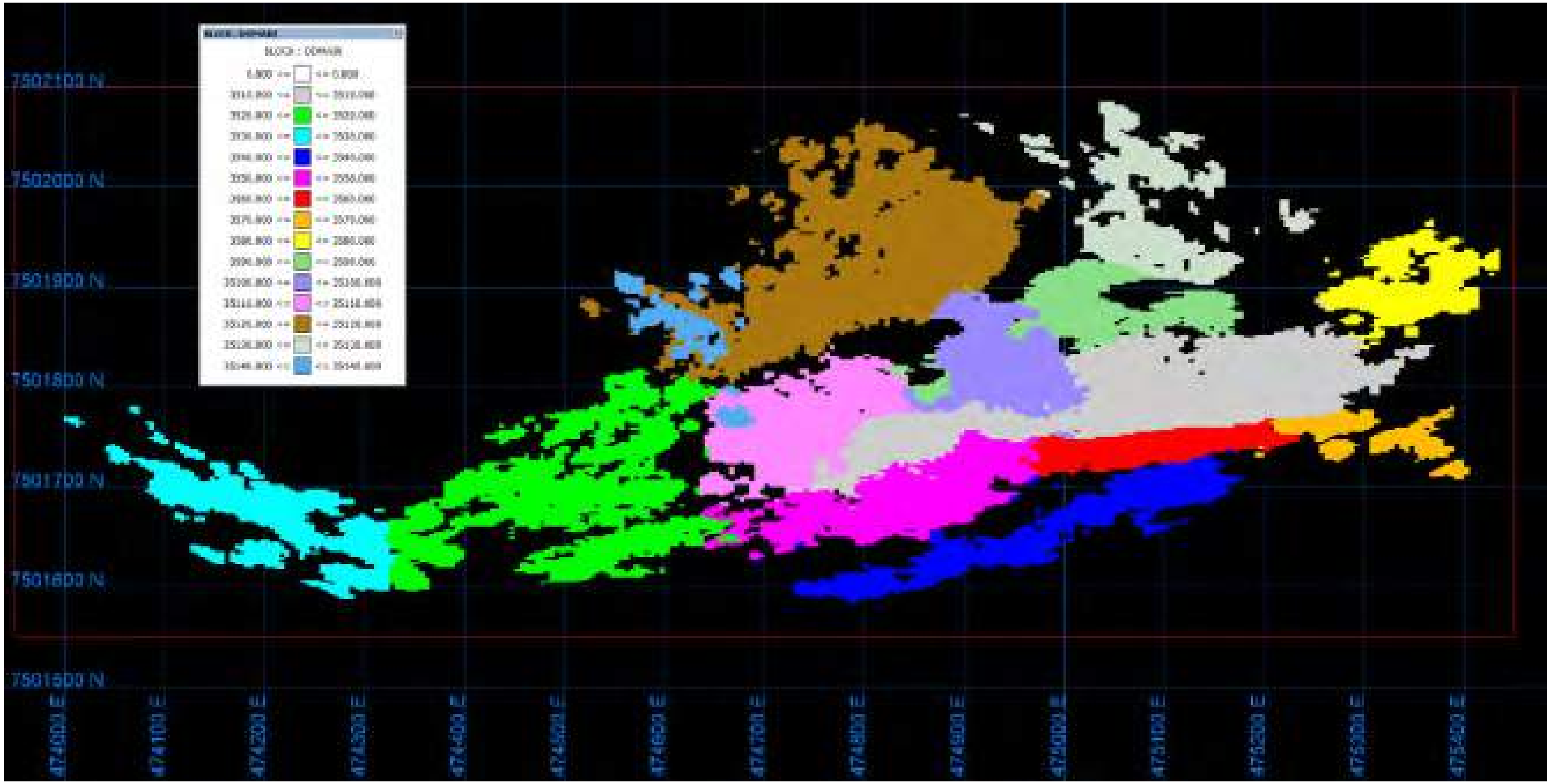


Figure 14.4 Pahtavaara Gold Deposit, Estimation Domains Isometric SW View

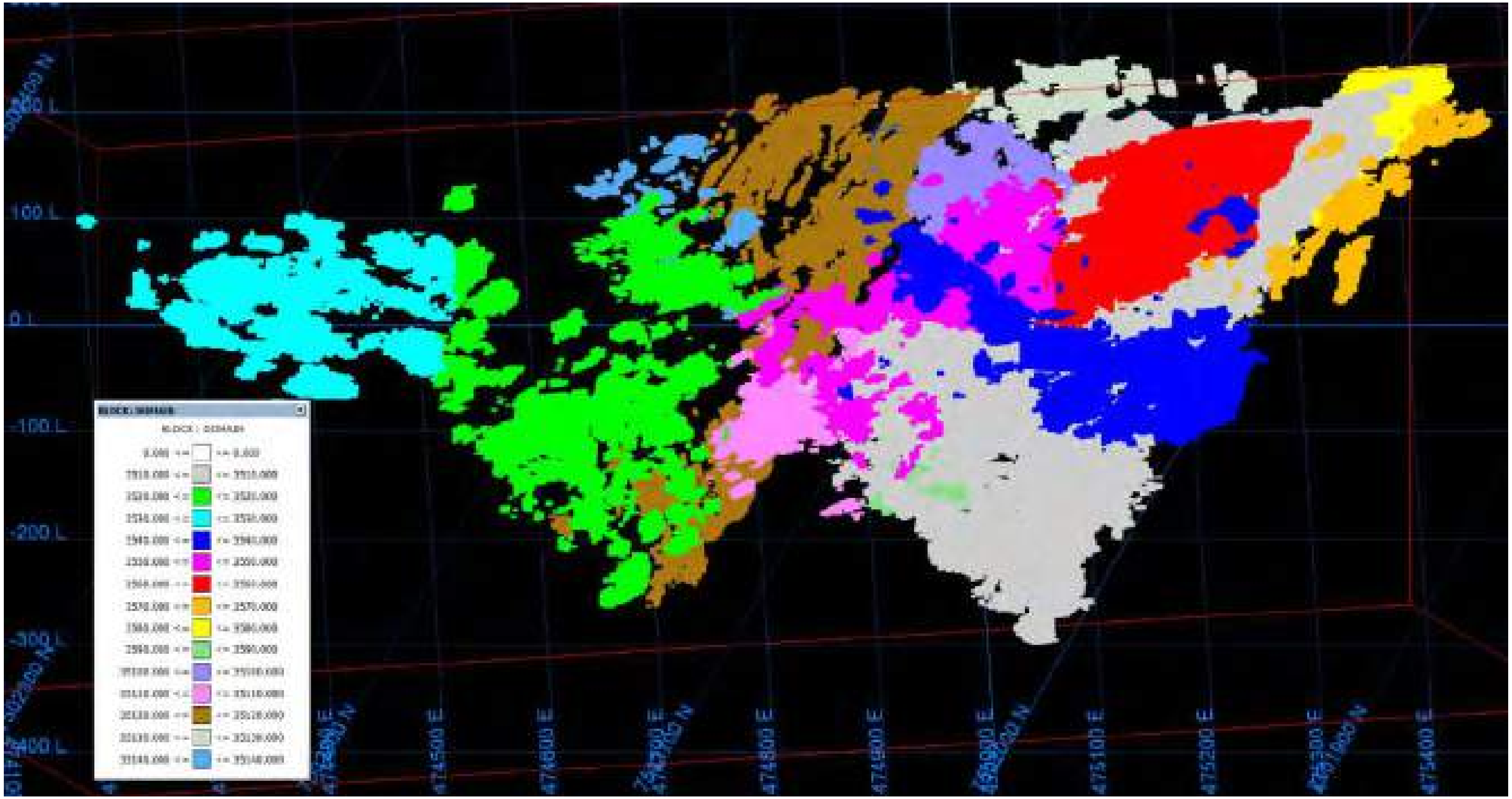
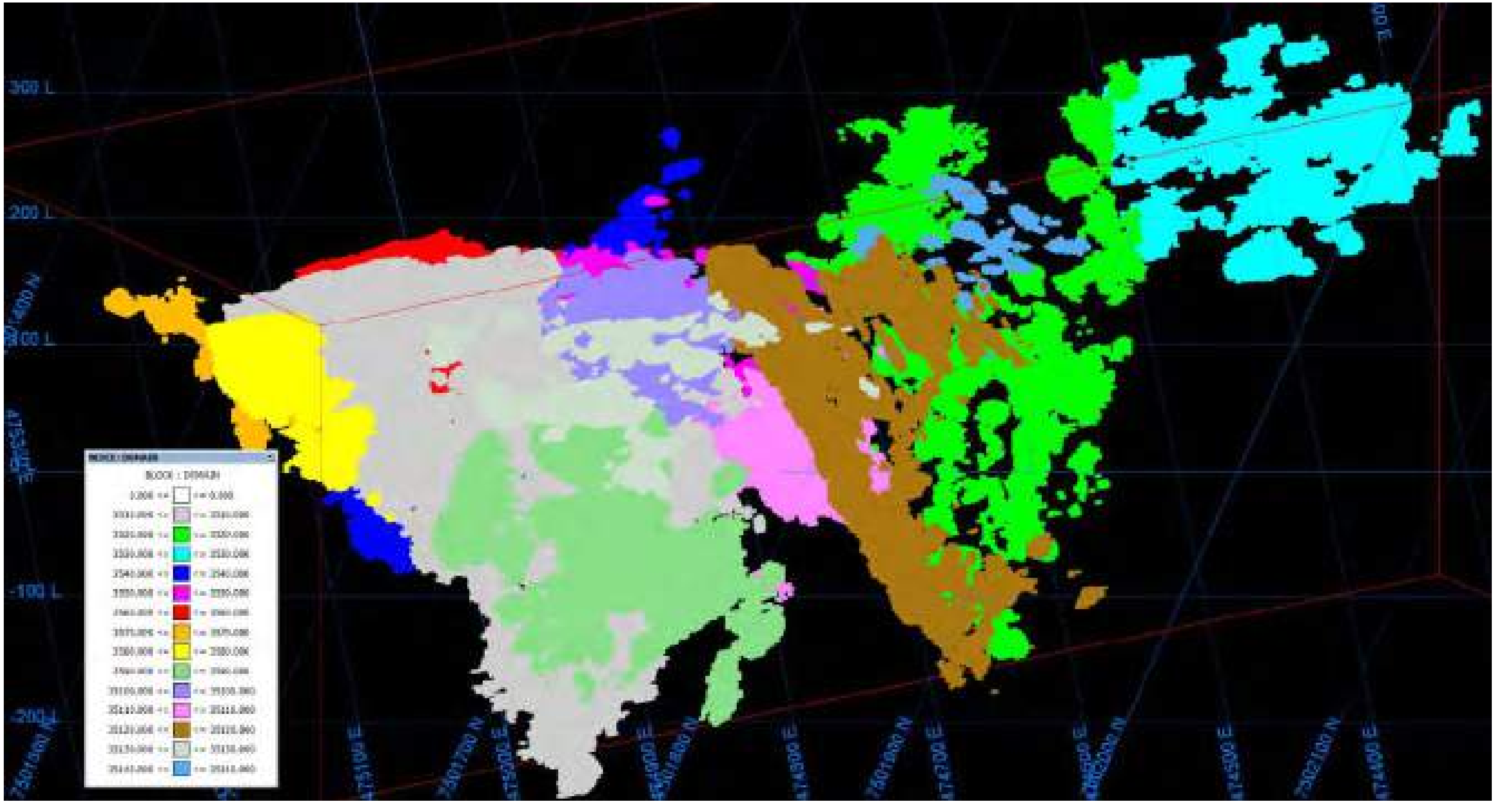


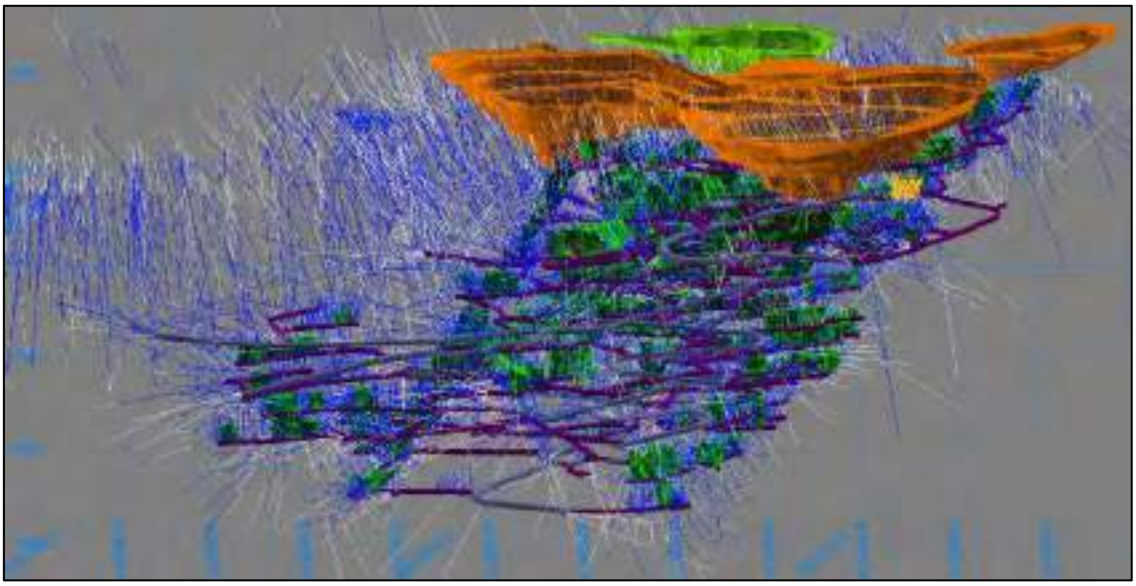
Figure 14.5 Pahtavaara Gold Deposit, Estimation Domains Isometric NE View



MINE INFRASTRUCTURE

Pahtavaara has been mined since 1996 from a series of open cuts and underground. As such, a series of extensive underground infrastructure including declines, drives and open stopes exist in conjunction with the open pits. The relationship between the open pits and underground infrastructure is presented in Figure 14.6.

Figure 14.6 Pahtavaara Gold Deposit, Open Pits and Underground Infrastructure, Isometric SW View



14.1.4 DATA FLAGGING AND COMPOSITING

Drill hole samples were flagged with the relevant indicator grade shells, topographical surfaces and both the underground and open pit wireframes described in previous sections. Coding was undertaken on the basis that if the individual sample centroid fell within the grade shell boundary it was coded as within the grade shell. Each sub-domain has been assigned a unique numerical code to allow the application of hard boundary domaining if required during grade estimation.

The drill hole database coded within each grade shell or mineralisation wireframe was then composited as a means of achieving a uniform sample support. It should be noted, however, that equalising sample length is not the only criteria for standardising sample support. Factors such as angle of intersection of the sampling to mineralisation, sample type and diameters, drilling conditions, recovery, sampling/sub-sampling practices and laboratory practices all affect the 'support' of a sample. Exploration/mining databases which contain multiple sample types and/or sources of data provide challenges in generating composite data with equalised sample support, and uniform support is frequently difficult to achieve.

After consideration of relevant factors relating to geological setting and mining, including likely mining selectivity and bench/flitch height, a regular two metre run length (downhole) composite was selected as the most appropriate composite interval to equalise the sample support at Pahtavaara Gold Deposit. Compositing was broken when the routine encountered a change in flagging (grade shell boundary) and composites with residual intervals of less than two metres were retained in the composite file.

14.1.5 Statistical Analysis

Summary Statistics

The composites flagged as described in the previous section were used for subsequent statistical, geostatistical and grade estimation investigations.

Summary descriptive statistics were generated for all domains (Table 14.3 and Table 14.4). The grade distributions are typical for gold deposits of this style and show a positive skew or near lognormal behaviour (Figure 14.7). The coefficient of variation (CV - calculated by dividing the standard deviation by the mean grade) is moderately high, consistent with the presence of high outlier grades that potentially require cutting (capping) for grade estimation.

Table 14.3 Pahtavaara Gold Deposit, Summary Statistics Low Grade Domains for Two Metre Composites of Uncut Gold Grade (g/t)

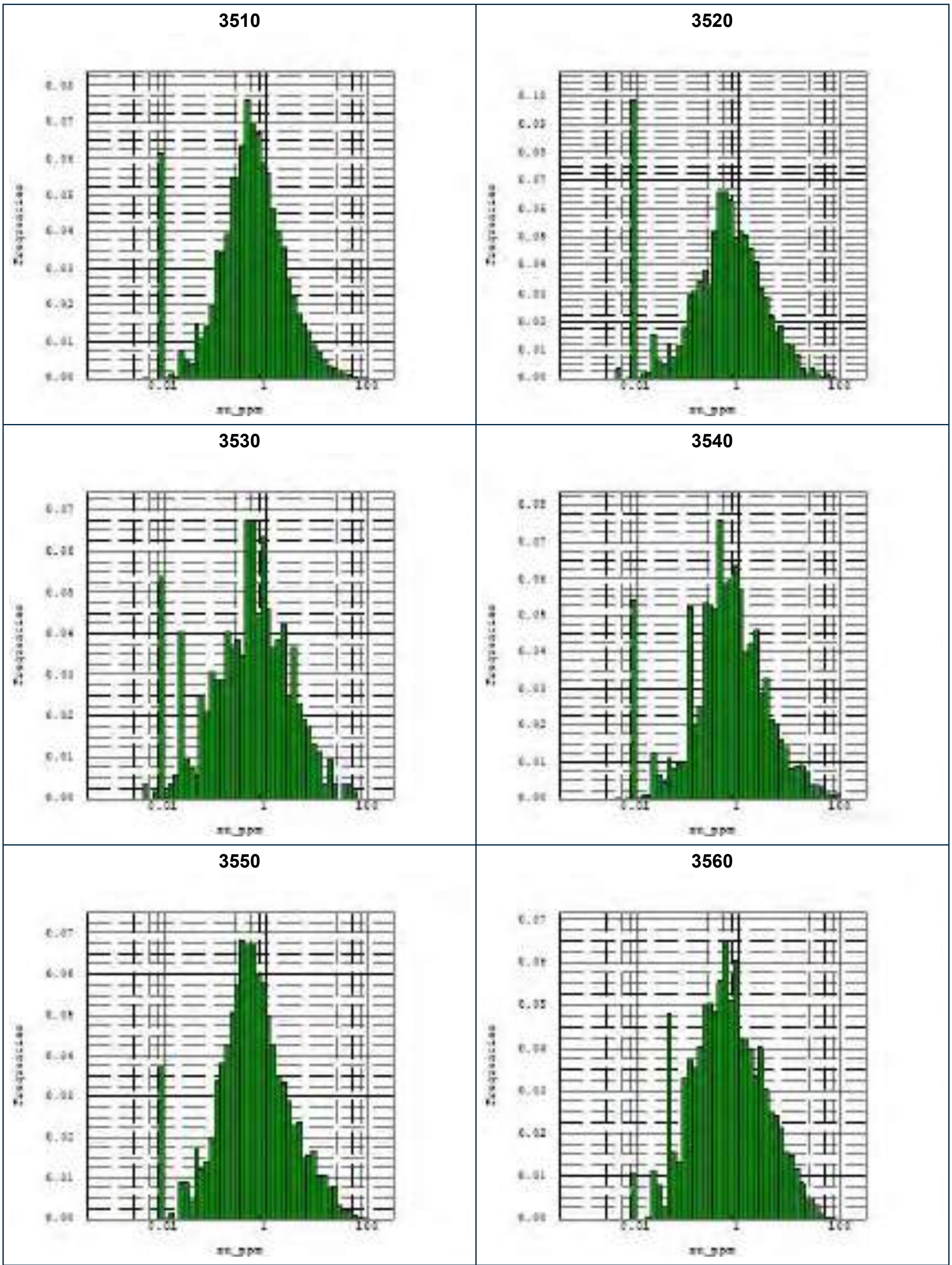
Domain	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
2010	12,054	0.005	28.610	**0.222**	0.698	0.487	3.144
2020	1,966	0.005	20.700	**0.294**	1.019	1.038	3.466
2030	486	0.005	25.484	**0.383**	1.444	2.085	3.770
2040	1,398	0.001	23.602	**0.251**	0.965	0.932	3.845
2050	1,739	0.007	15.545	**0.248**	0.784	0.614	3.161
2060	679	0.009	8.890	**0.21**	0.599	0.359	2.852
2070	443	0.005	16.800	**0.388**	1.392	1.937	3.588
2080	936	0.010	2110.000	**2.515**	68.925	4750.705	27.406
2090	727	0.005	12.900	**0.246**	0.663	0.439	2.695
20100	1,414	0.001	25.400	**0.262**	1.193	1.424	4.553
20110	1,500	0.001	10.900	**0.214**	0.556	0.309	2.598
20120	2,843	0.005	56.300	**0.274**	1.357	1.840	4.953
20130	461	0.005	216.105	**1.663**	13.935	194.186	8.379
20140	465	0.001	24.200	**0.534**	1.994	3.974	3.734

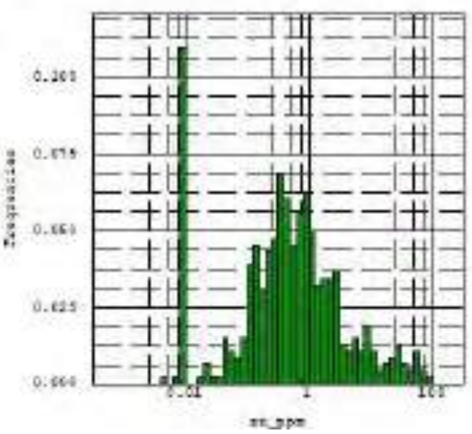
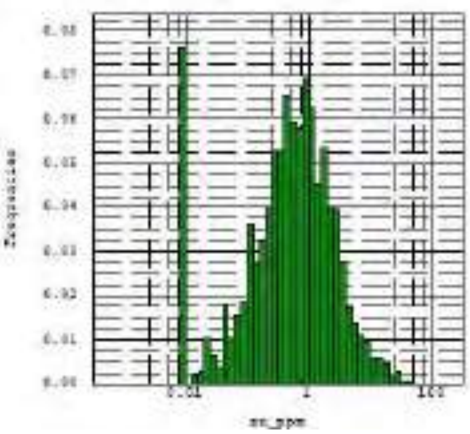
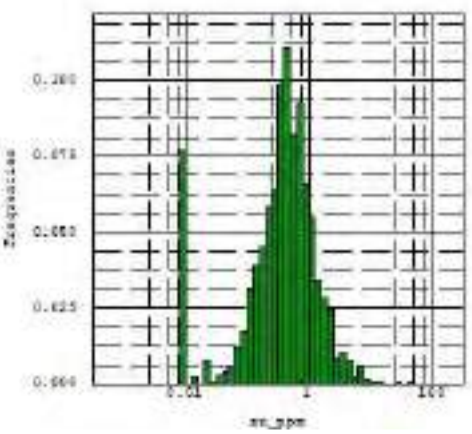
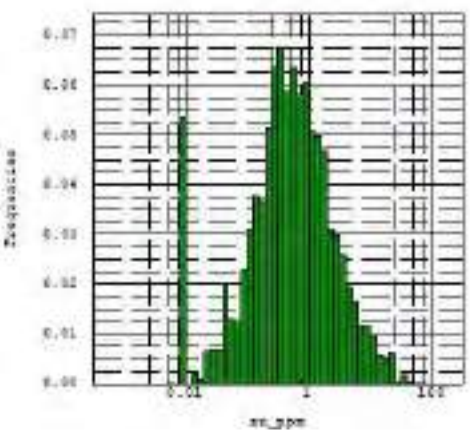
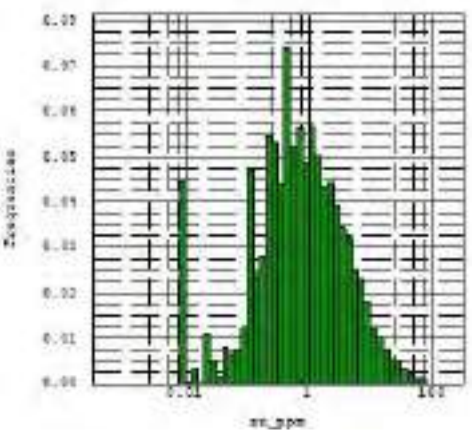
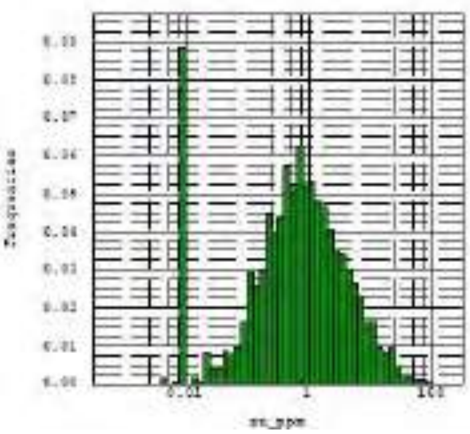
Table 14.4 Pahtavaara Gold Deposit, Summary Statistics High Grade Domains for Two Metre Composites of Uncut Gold Grade (g/t)

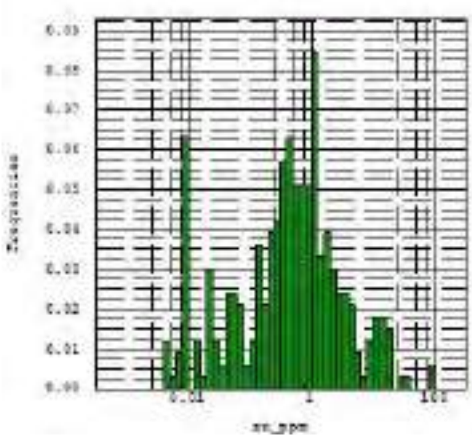
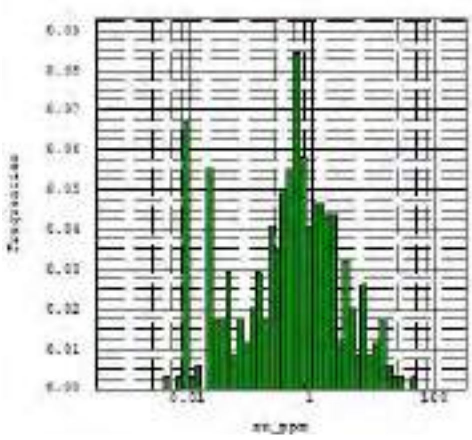
Domain	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
3510	33,087	0.004	3450.000	**1.668**	19.560	382.577	11.727
3520	2,742	0.005	138.000	**1.933**	5.777	33.378	2.989
3530	519	0.005	57.555	**1.979**	5.055	25.553	2.554
3540	3,759	0.005	226.105	**2.601**	9.309	86.653	3.579
3550	3,723	0.010	387.455	**2.293**	9.129	83.347	3.981
3560	2,121	0.010	65.664	**2.156**	4.709	22.177	2.184
3570	466	0.005	100.950	**3.570**	10.994	120.858	3.080
3580	1,548	0.010	44.510	**1.467**	3.097	9.592	2.111
3590	1,265	0.010	115.000	**0.853**	3.668	13.453	4.300
35100	3,366	0.006	2368.914	**2.446**	41.178	1695.616	16.835
35110	4,664	0.010	197.001	**2.635**	8.233	67.782	3.124
35120	5,995	0.005	817.262	**2.758**	12.842	164.914	4.656

35130	332	0.005	467.675	**4.651**	31.097	967.032	6.686
35140	344	0.005	44.130	**1.811**	4.161	17.317	2.298

Figure 14.7 Log Histograms of Uncut Gold Grade by Domain



3570	3580
	
3590	**35100**
	
35110	**35120**
	

35130	35140
	

HIGH GRADE OUTLIER ANALYSIS

MIK has been selected as the main method to estimate the gold grades for the Pahtavaara Gold deposit. However, the grade datasets for the various estimation domains are characterised by moderately high CV values, indicating that high-grade values may contribute significantly to the mean grades reported for the various datasets.

It should be noted that while gold grades are not cut or capped for the purposes of MIK estimation the use of cut grades is often employed for variography and the change of support process. As MIK estimates are essentially a series of OK estimates applied to the binary transformation of a series of indicator cut-offs, high-grade cutting will have no effect on the resultant MIK estimate unless the high-grade cut is lower than the chosen upper indicator cut-off and this scenario would be considered highly sub-optimal in the context of MIK estimation. A full description of the MIK estimation method with change of support is provided in Section 14.1.9.

The effects of the highest-grade composites on the mean grade and standard deviation of the gold dataset for each of the estimation domains have been investigated by compiling and reviewing statistical plots (histograms and probability plots). The resultant plots were reviewed together with probability plots of the sample populations and an upper cut for each dataset was chosen coinciding with a pronounced inflection or increase in the variance of the data. A list of the determined upper cuts applied and their impact on the mean grades of the datasets is provided in Table 14.5.

Table 14.5 Pahtavaara Gold Deposit, Summary Statistics High Grade Domains for Two Metre Composites of Top Cut Gold Grade (g/t)

Domain	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
3510	33,087	0.004	30.00	**1.460**	3.256	10.602	2.230
3520	2,742	0.005	30.00	**1.762**	3.754	14.091	2.131
3530	519	0.005	30.00	**1.857**	4.060	16.483	2.186
3540	3,759	0.005	55.00	**2.361**	6.053	36.641	2.564
3550	3,723	0.010	55.00	**2.115**	5.298	28.066	2.505
3560	2,121	0.010	55.00	**2.151**	4.647	21.592	2.160
3570	466	0.005	55.00	**3.302**	9.216	84.944	2.791

3580	1,548	0.010	44.51	**1.467**	3.097	9.592	2.111
3590	1,265	0.010	30.00	**0.775**	1.735	3.011	2.239
35100	3,366	0.006	30.00	**1.618**	3.512	12.336	2.171
35110	4,664	0.010	30.00	**2.315**	4.587	21.044	1.981
35120	5,995	0.005	55.00	**2.518**	5.770	33.287	2.292
35130	332	0.005	55.00	**2.479**	7.052	49.728	2.845
35140	344	0.005	44.13	**1.811**	4.161	17.317	2.298

Composite data was viewed in 3D to determine the clustering or otherwise of these highest grades observed in each domain to assess the appropriateness of the high-grade cut. Clustering of the highest grades in one or more areas may indicate that the grades do not require cutting.

Cell Declustering Analysis

Visual inspection of the available datasets for each of the estimation domains indicated some clustering of the data within higher grade regions of the deposit. Data clustering often occurs when drilling campaigns selectively target higher grade regions of the deposit, resulting in an artificially high mean grade in many cases. Declustering was therefore completed to remove any effects of preferential sampling of high-grade areas that may have occurred.

Cell declustering was completed with weights w(i) associated with composite (i) determined as w(i) = mv/n(i) where mv is the mean of all the samples within a moving window centred on composite (i) and n(i) is the number of samples within the moving window. This normalisation allows the weights not to decrease with the number of data.

Declustered composite statistics are presented in Table 14.6. As expected, the declustered mean grades tend to be less than the composite mean grades due to the data configuration issues discussed above, however in some instances the mean grade increases over a wide range of cell declustering sizes.

Table 14.6 Pahtavaara Gold Deposit, Summary Statistics Low Grade Domains for Two Metre Composites of Declustered Gold Grade (g/t)

Domain	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
2010	12,054	0.005	28.610	0.225	0.623	0.388	2.769
2020	1,966	0.005	20.700	0.261	0.958	0.918	3.670
2030	486	0.005	25.484	0.366	1.457	2.122	3.981
2040	1,398	0.001	23.602	0.218	0.843	0.711	3.867
2050	1,739	0.007	15.545	0.235	0.824	0.679	3.506
2060	679	0.009	8.890	0.207	0.625	0.391	3.019
2070	443	0.005	16.800	0.385	1.373	1.886	3.566
2080	936	0.010	2110.000	1.616	53.380	2849.440	33.032
2090	727	0.005	12.900	0.244	0.501	0.251	2.053
20100	1,414	0.001	25.400	0.302	1.447	2.095	4.791
20110	1,500	0.001	10.900	0.187	0.457	0.208	2.444
20120	2,843	0.005	56.300	0.269	1.565	2.449	5.818
20130	461	0.005	216.105	0.912	9.581	91.799	10.505
20140	465	0.001	24.200	0.511	2.141	4.583	4.190

Table 14.7 Pahtavaara Gold Deposit, Summary Statistics High Grade Domains for Two Metre Composites of Top Cut Declustered Gold Grade (g/t)

Domain	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
3510	33,087	0.004	30.00	**1.299**	2.931	8.593	2.256
3520	2,742	0.005	30.00	**1.631**	3.647	13.302	2.236
3530	519	0.005	30.00	**1.665**	3.845	14.783	2.309
3540	3,759	0.005	55.00	**1.789**	4.498	20.233	2.514
3550	3,723	0.010	55.00	**1.840**	4.928	24.287	2.678
3560	2,121	0.010	55.00	**1.994**	4.511	20.346	2.262
3570	466	0.005	55.00	**1.948**	6.353	40.355	3.261
3580	1,548	0.010	44.51	**1.421**	2.770	7.675	1.949
3590	1,265	0.010	30.00	**0.736**	1.848	3.417	2.511
35100	3,366	0.006	30.00	**1.523**	3.460	11.971	2.272
35110	4,664	0.010	30.00	**1.673**	3.967	15.736	2.371
35120	5,995	0.005	55.00	**2.113**	5.164	26.664	2.444
35130	332	0.005	55.00	**1.961**	6.348	40.294	3.237
35140	344	0.005	44.13	**1.408**	3.084	9.512	2.190

Domain Grouping

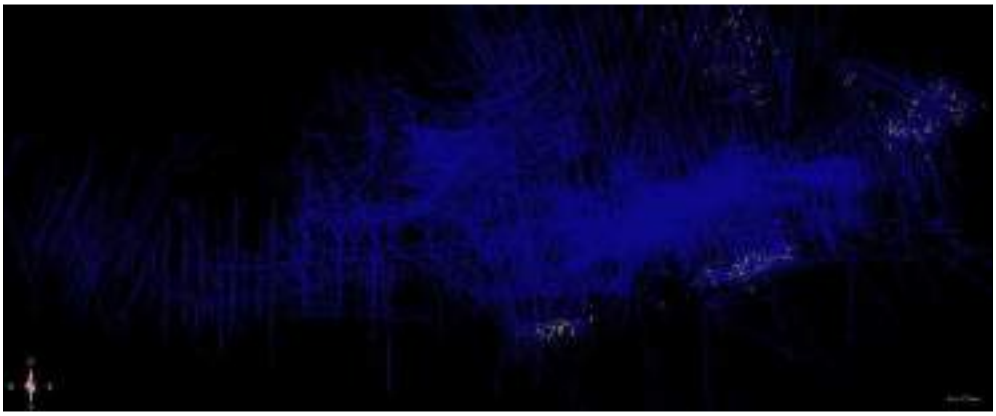
The fourteen estimation domains have been grouped for the purposes of MIK estimation. The grouping was on the basis of domain statistics, with consideration also given to location within the deposit and overall domain geometry and orientation. The grouping is outlined in Table 14.8 and Figure 14.8.

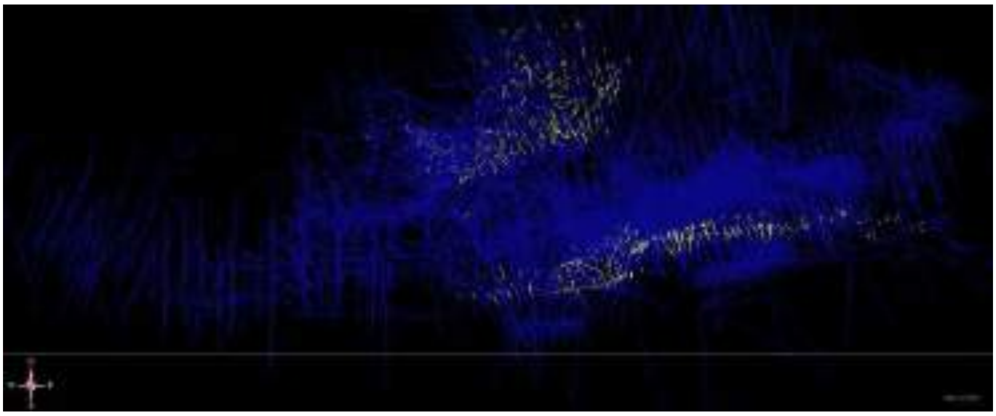
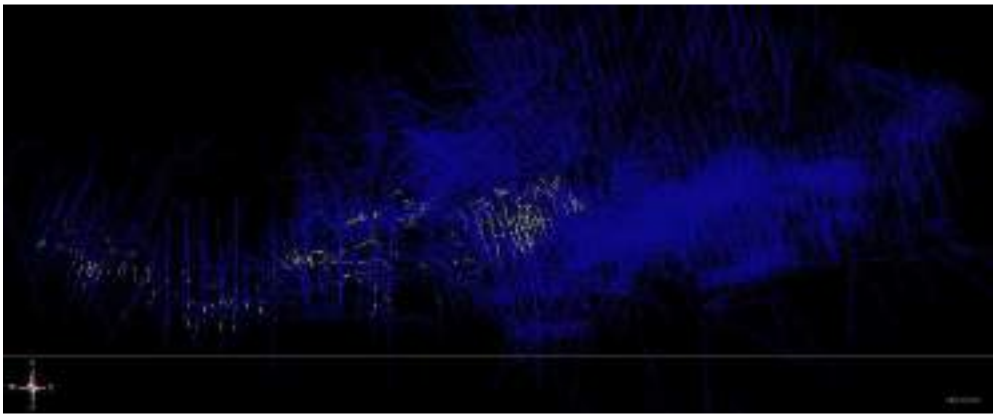
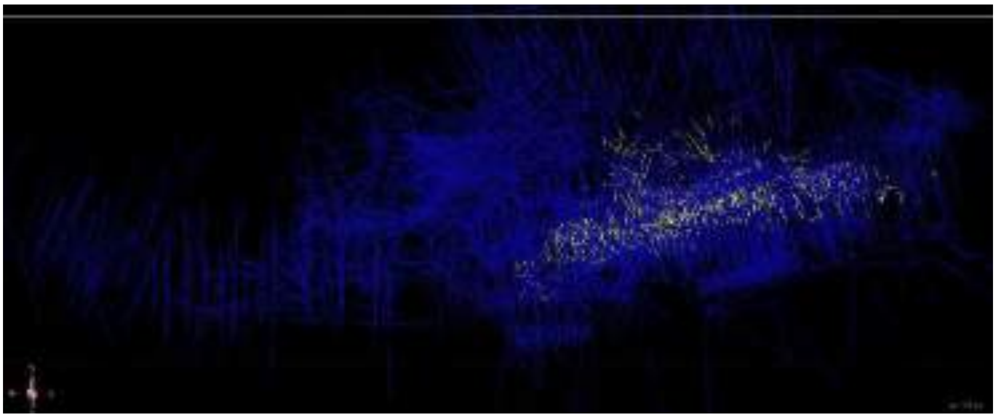
Table 14.8 Pahtavaara Gold Deposit, Domain Grouping

Domain Group	Domains
High Grade	3540, 3580,35130
Medium Grade	3550, 3560, 3570, 35120, 35140
Medium Low Grade	3520, 3530, 35110
Low Grade	3510, 3590, 35100

Figure 14.8 Domain Grouping

High Grade



Medium Grade	
Medium / Low Grade	
Low Grade	

Multiple Indicator Kriging Cut-offs and Indicator Class Statistics

Indicator Kriging cut-offs or indicator bins were selected for each domain group to be estimated by MIK. Cut-offs were based upon population distributions and metal proportions above and below the mean composite value of the proposed cut-off bins. Conditional statistics for data within each domain grouping to be estimated by MIK are listed in Table 14.9. A total of 17 cut-offs were applied to each domain group for estimation via MIK. Top cuts have not been applied for the purposes of conditional statistics calculation.

Table 14.9 Pahtavaara Gold Deposit, Indicator Class Statistics

Domain Group					
High Grade Group			Medium Grade Group		
Grade Threshold (Au g/t)	Probability Threshold	Class Mean (Au g/t)	Grade Threshold (Au g/t)	Probability Threshold	Class Mean (Au g/t)
0.15	0.273	0.0525	0.15	0.249	0.0588
0.30	0.385	0.2202	0.30	0.370	0.2167
0.50	0.505	0.3834	0.45	0.460	0.3700
0.75	0.607	0.6136	0.60	0.532	0.5169
1.00	0.676	0.8631	0.80	0.602	0.6879
1.30	0.729	1.1375	1.05	0.665	0.9138
1.85	0.785	1.5865	1.40	0.719	1.2056
2.25	0.810	2.0611	1.90	0.773	1.6332
2.90	0.841	2.5458	2.70	0.826	2.2680
3.50	0.869	3.1295	3.70	0.867	3.1487
4.80	0.903	4.0425	4.90	0.900	4.2676
5.90	0.923	5.2867	6.80	0.927	5.7740
8.70	0.939	7.6978	8.80	0.946	7.6910
12.00	0.956	10.0753	13.00	0.966	10.4953
18.00	0.975	14.6131	18.00	0.979	15.3826
24.00	0.984	21.2429	26.00	0.989	21.6117
35.00	0.991	29.8638	44.00	0.996	33.8274
Max	Max	91.3312	Max	Max	91.1683
Medium – Low Group			Low Grade Group		
Grade Threshold (Au g/t)	Probability Threshold	Class Mean (Au g/t)	Grade Threshold (Au g/t)	Probability Threshold	Class Mean (Au g/t)
0.15	0.288	0.0519	0.15	0.269	0.0617
0.30	0.401	0.2186	0.30	0.416	0.2173
0.50	0.509	0.3997	0.50	0.555	0.3933
0.70	0.596	0.5804	0.70	0.652	0.5885
0.95	0.669	0.8117	0.95	0.730	0.8108
1.25	0.726	1.0782	1.25	0.791	1.0842
1.60	0.778	1.4113	1.65	0.838	1.4387
2.15	0.816	1.8476	2.10	0.874	1.8572
2.80	0.853	2.4745	2.75	0.904	2.3864
3.60	0.881	3.1829	3.55	0.927	3.1337
4.50	0.905	3.9833	4.50	0.944	3.9795
5.70	0.921	4.9832	5.70	0.958	5.0886
7.00	0.942	6.1819	7.20	0.970	6.3949
8.50	0.954	7.7375	8.90	0.980	7.9521
11.00	0.967	9.7365	13.00	0.988	10.4556
14.00	0.979	12.4175	17.00	0.993	14.8375
23.00	0.991	17.8152	27.00	0.998	20.6258
Max	Max	37.9414	Max	Max	42.0621

Data Type Comparisons

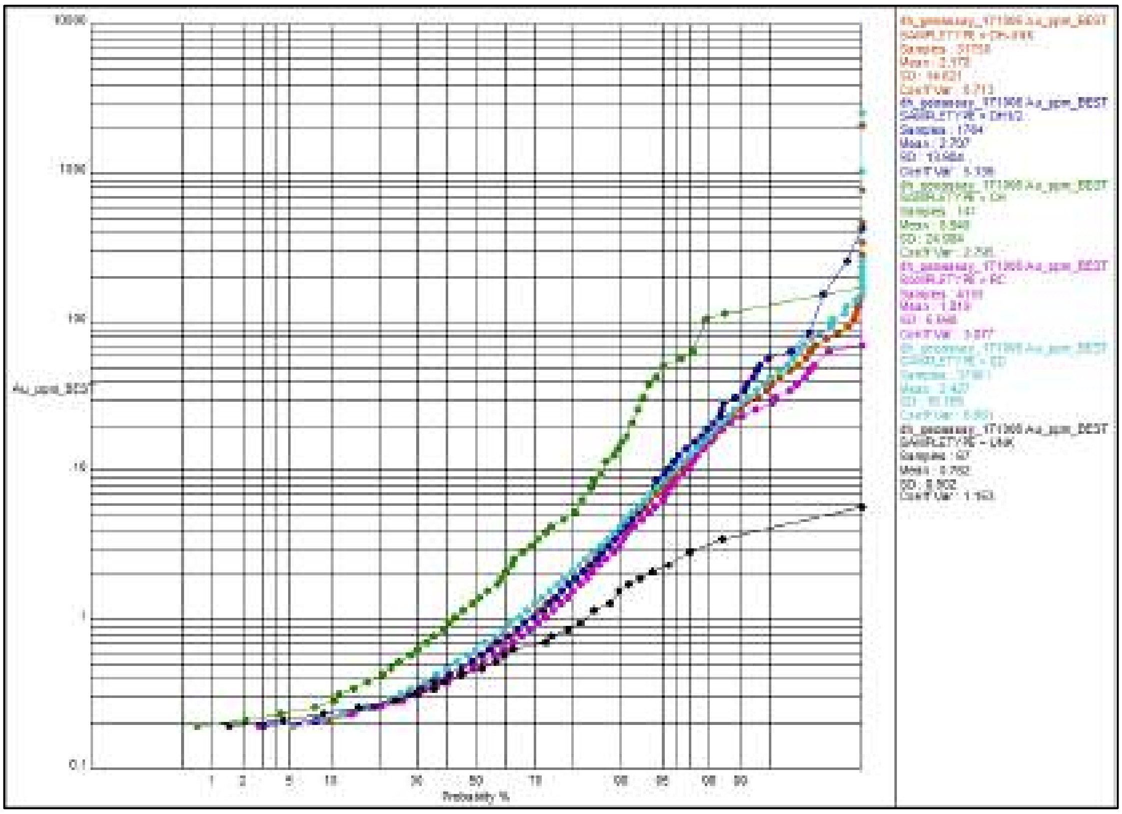
The drill hole database contains different data types (Table 14.10) and the issue of concern is that bias may exist between the data types. The combination of various data types may therefore be unsuitable for the purposes of resource estimation. The main data types are diamond drilling and sludge sampling. Raw sample type statistics are presented in Table 14.10.

Table 14.10 Pahtavaara Gold Deposit, Summary Statistics Sample Gold Grades

Sample Type	All Data			>0.2 g/t Au		
	Number	Mean	%	Number	Mean	%
Channel	321	3.95	0	141	8.94	0
Dia unknown	152,302	0.48	53	31,750	2.18	42
Dia ½ core	24,927	0.22	9	1,764	2.71	2
RC	16,606	0.50	6	4,155	1.82	5
Sludge	95,171	0.99	33	37,681	2.43	50
Unknown	201	0.30	0	67	0.78	0
Total	**289,527**	**0.63**	**100**	**75,738**	**2.31**	**100**

It is evident the main dataset is composed of diamond and sludge drilling. Above 0.2 g/t Au 50% of the data is sludge drilling. Both RC and channel sampling form relatively insignificant proportions of the total dataset. While the sludge may appear to be biased high on the basis of the total dataset, equivalency can be demonstrated on the basis of the subset of data greater than 0.2 g/t Au. A log probability plot of the different data types is presented in Figure 14.9. Virtually identical distributions can be observed for sludge (light blue), diamond (red and dark blue) and additionally RC samples (pink).

Figure 14.9 Pahtavaara Gold Deposit, Log Probability Plot Different Sampling Types



As Figure 14.9 demonstrates equivalency of global data distribution only, additional tests have been carried out to determine if different sample types co-located within discrete 3D volumes demonstrate equivalency of gold grades. These tests have been undertaken in Isatis geostatistical software. The generalised approach is as follows:

- Create a grid of blocks with dimensions of 5 m E x 5 m N x 5 m RL (125 m³) and 10 m E x 10 m N x 10 m RL (1,000 m³).
- Record statistics for each data type enclosed within each individual block to that block i.e. number, minimum, maximum, mean, etc.
- In this way the different type of samples contained within each block may be compared. Filters may be applied so that any given block enclosing too few samples of any type will be excluded from the overall comparison.

Statistics for both grid dimensions have been calculated and results compared. Only blocks where both types of samples are co-located have been considered. Results are presented in Table 14.11. Results indicate equivalency of diamond and sludge sample gold grades when both occur in close proximity. It can be concluded that both types of data can be combined for the purposes of resource estimation.

Table 14.11 Pahtavaara Gold Deposit, Summary Statistics Sample Gold Grade Spatial Correlation

Sample Type	5 m E x 5 m N x 5 m RL (125m³)			10 m E x 10 m N x 10 m RL (1,000m³)		
	Number Blocks	Average Grade	Total Samples	Number Blocks	Average Grade	Total Samples
Diamond	804	2.9	1,327	949	2.08	4,043
Sludge	804	3.1	1,769	949	2.20	8,987

14.1.6 VARIOGRAPHY

INTRODUCTION

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

Several types of variogram calculations are employed to determine the directions of the continuity of the mineralisation:

Traditional variograms are calculated from the raw assay values:

- Log-transformed variography involves a logarithmic transformation of the assay data.
- Gaussian variograms are based on the results after declustering and a transformation to a Normal distribution.
- Pairwise-relative variograms attempt to ‘normalise’ the variogram by dividing the variogram value for each pair by their squared mean value.

- Correlograms are 'standardised' by the variance calculated from the sample values that contribute to each lag.

Fan variography involves the graphical representation of spatial trends by calculating a range of variograms in a selected plane and contouring the variogram values. The result is a contour map of the grade continuity within the domain.

The variography was calculated and modelled in the geostatistical software, Isatis. The rotations are tabulated as dip and dip direction of major, semi-major and minor axes of continuity. Modelled variograms were generally shown to have moderate to good structure and were used throughout the MIK estimation and the change of support process.

Pahtavaara Variography

Grade and indicator variography was generated to enable grade estimation via MIK and change of support analysis to be completed. In addition, Gaussian variograms were also examined as part of the change of support process. Indicator thresholds for Domain groups to be estimated via MIK had variograms modelled with every third variogram typically modelled. Variograms not modelled have had their parameters interpolated based on the bounding modelled variograms.

Interpreted anisotropy directions correspond well with the modelled geology and overall geometry of the interpreted domains. All grade variography has been based on the back-transformed Gaussian variograms. A common feature of all the grade variography is the relatively short ranges, especially for the first modelled structure, and the dominance of the overall variance by the nugget and the first sill. This outcome can be expected in cases like Pahtavaara where much of the data is dominated by close spaced drilling.

Grade variography as modelled for OK grade estimation and change of support analysis is presented in Table 14.12 and indicator variography for the various MIK estimation domains in Table 14.13 to Table 14.16. Modelled grade variograms are presented in Figure 14.10 to Figure 14.13.

Table 14.12 Pahtavaara Gold Deposit, Grade Variogram Models Au g/t

Domain Group	Nugget (C0)	Rotation (dip→dip dir)			Structure 1				Structure 2			
		Major	Semi Major	Minor	Relative Sill 1 (C1)	Range (m)			Relative Sill 2 (C2)	Range (m)		
						Major	Semi Major	Minor		Major	Semi Major	Minor
High Grade	11.1	10→260	80→80	0→170	8.1	20	12	4	5.3	57	43	15
Medium Grade	11.6	50→270	0→180	40→90	11.2	10	8	2	4.1	31	20	8
Medium Low Grade	7.0	33→257	11→174	55→60	6.6	9	7	3	2.9	36	23	9
Low Grade	5.0	90→90	0→80	0→170	3.5	9	7	3	1.2	43	27	7

Note: All grade variograms derived from back transformed Gaussian Variogram

Table 14.13 Pahtavaara Gold Deposit, Domain Group High Grade Indicator Variogram Models Au g/t

Grade Variable or Indicator Threshold	Nugget (C0)	Rotation (dip→dip dir)			Structure 1				Structure 2			
		Major	Semi Major	Minor	Relative Sill 1 (C1)	Range (m)			Relative Sill 2 (C2)	Range (m)		
						Major	Semi Major	Minor		Major	Semi Major	Minor
0.15(1)	0.0483	10→260	80→80	0→170	0.0788	25	17	5	0.0830	65	55	17
0.30(1)	0.0554	10→260	80→80	0→170	0.0904	25	17	5	0.0952	65	55	17
0.50	0.0570	10→260	80→80	0→170	0.0930	25	17	5	0.0980	65	55	17
0.75(2)	0.0574	10→260	80→80	0→170	0.0882	25	17	5	0.0934	63	53	17
1.00(2)	0.0552	10→260	80→80	0→170	0.0798	25	17	5	0.0850	62	52	16
1.30	0.0520	10→260	80→80	0→170	0.0710	25	17	5	0.0760	60	50	16
1.85(3)	0.0494	10→260	80→80	0→170	0.0634	24	17	5	0.0682	58	48	16
2.25(3)	0.0461	10→260	80→80	0→170	0.0558	23	17	5	0.0602	57	47	16
2.90	0.0420	10→260	80→80	0→170	0.0480	22	17	5	0.0520	55	45	16
3.50(4)	0.0395	10→260	80→80	0→170	0.0415	22	17	5	0.0449	55	45	16
4.80(4)	0.0335	10→260	80→80	0→170	0.0324	22	17	5	0.0351	55	45	16
5.90	0.0280	10→260	80→80	0→170	0.0250	22	17	5	0.0270	55	45	16
8.70(5)	0.0224	10→260	80→80	0→170	0.0191	18	15	5	0.0205	47	40	15
12.0(5)	0.0186	10→260	80→80	0→170	0.0153	14	12	5	0.0161	38	35	13
18.0	0.0140	10→260	80→80	0→170	0.0110	10	10	5	0.0115	30	30	12
24.0(6)	0.0088	10→260	80→80	0→170	0.0069	10	10	5	0.0072	30	30	12
35.0(6)	0.0046	10→260	80→80	0→170	0.0036	10	10	5	0.0038	30	30	12

Note: 1) Assumed model based on 0.50 Au g/t variogram model.
2) Assumed model based on 0.50 Au g/t and 1.3 Au g/t variogram models.
3) Assumed model based on 1.3 Au g/t and 2.9 Au g/t variogram models.
4) Assumed model based on 2.9 Au g/t and 5.9 Au g/t variogram model.
5) Assumed model based on 5.9 Au g/t and 18 Au g/t variogram model.
6) Assumed model based on 18 Au g/t variogram model.

Table 14.14 Pahtavaara Gold Deposit, Domain Group Medium Grade Indicator Variogram Models Au g/t

Grade Variable or Indicator Threshold	Nugget (C0)	Rotation (dip→dip dir)			Structure 1				Structure 2			
		Major	Semi Major	Minor	Relative Sill 1 (C1)	Range (m)			Relative Sill 2 (C2)	Range (m)		
						Major	Semi Major	Minor		Major	Semi Major	Minor
0.15(1)	0.0388	50→270	0→180	40→90	0.1085	12	6	5	0.0457	40	25	8
0.30(1)	0.0469	50→270	0→180	40→90	0.1314	12	6	5	0.0554	40	25	8
0.45	0.0500	50→270	0→180	40→90	0.1400	12	6	5	0.0590	40	25	8
0.60(2)	0.0511	50→270	0→180	40→90	0.1396	12	6	4	0.0589	40	25	8
0.80(2)	0.0508	50→270	0→180	40→90	0.1353	12	6	4	0.0572	40	25	7
1.05	0.0500	50→270	0→180	40→90	0.1300	12	6	3	0.0550	40	25	7
1.40(3)	0.0470	50→270	0→180	40→90	0.1148	12	6	3	0.0492	38	25	7
1.90(3)	0.0439	50→270	0→180	40→90	0.1009	12	6	3	0.0438	37	25	6
2.70	0.0420	50→270	0→180	40→90	0.0910	12	6	3	0.0400	35	25	6
3.70(4)	0.0357	50→270	0→180	40→90	0.0700	10	5	3	0.0318	32	25	6
4.90(4)	0.0287	50→270	0→180	40→90	0.0511	9	5	3	0.0240	28	25	5
6.80	0.0240	50→270	0→180	40→90	0.0390	7	4	3	0.0190	25	25	5
8.80(5)	0.0191	50→270	0→180	40→90	0.0295	6	4	3	0.0135	22	21	4
13.0(5)	0.0129	50→270	0→180	40→90	0.0188	6	4	2	0.0081	18	17	4
18.0	0.0090	50→270	0→180	40→90	0.0125	5	4	2	0.0050	15	13	3
26.0(6)	0.0038	50→270	0→180	40→90	0.0053	5	4	2	0.0021	15	13	3
44.0(6)	0.0014	50→270	0→180	40→90	0.0019	5	4	2	0.0008	15	13	3

Note: 1) Assumed model based on 0.45 Au g/t variogram model.
2) Assumed model based on 0.45 Au g/t and 1.05 Au g/t variogram models.
3) Assumed model based on 1.05 Au g/t and 2.7 Au g/t variogram models.
4) Assumed model based on 2.7 Au g/t and 6.8 Au g/t variogram model.
5) Assumed model based on 6.8 Au g/t and 18 Au g/t variogram model.
6) Assumed model based on 18 Au g/t variogram model.

Table 14.15 Pahtavaara Gold Deposit Domain Group Medium/Low Grade Indicator Variogram Models Au g/t

Grade Variable or Indicator Threshold	Nugget (C0)	Rotation (dip→dip dir)			Structure 1				Structure 2			
		Major	Semi Major	Minor	Relative Sill 1 (C1)	Range (m)			Relative Sill 2 (C2)	Range (m)		
						Major	Semi Major	Minor		Major	Semi Major	Minor
0.15(1)	0.0500	33→257	33→257	33→257	0.0837	10	6	3	0.0743	35	30	6
0.30(1)	0.0582	33→257	33→257	33→257	0.0973	10	6	3	0.0864	35	30	6
0.50	0.0640	33→257	33→257	33→257	0.1070	10	6	3	0.0950	35	30	6
0.70(2)	0.0585	33→257	33→257	33→257	0.0960	10	6	3	0.0855	35	28	6
0.95(2)	0.0543	33→257	33→257	33→257	0.0875	10	6	3	0.0782	35	27	6
1.25	0.0550	33→257	33→257	33→257	0.0870	10	6	3	0.0780	35	25	6
1.60(3)	0.0437	33→257	33→257	33→257	0.0667	10	6	3	0.0586	35	23	6
2.15(3)	0.0388	33→257	33→257	33→257	0.0571	10	6	3	0.0491	35	22	6
2.80	0.0400	33→257	33→257	33→257	0.0570	10	6	3	0.0480	35	20	6
3.60(4)	0.0299	33→257	33→257	33→257	0.0397	10	6	3	0.0324	32	18	6
4.50(4)	0.0265	33→257	33→257	33→257	0.0328	10	6	3	0.0258	28	17	5
5.70	0.0250	33→257	33→257	33→257	0.0290	10	6	3	0.0220	25	15	5
7.00(5)	0.0182	33→257	33→257	33→257	0.0194	10	6	3	0.0153	25	15	5
8.50(5)	0.0158	33→257	33→257	33→257	0.0155	10	6	3	0.0127	25	15	5
11.0	0.0150	33→257	33→257	33→257	0.0135	10	6	3	0.0115	25	15	5
14.0(6)	0.0075	33→257	33→257	33→257	0.0068	10	6	3	0.0058	25	15	5
23.0(6)	0.0041	33→257	33→257	33→257	0.0037	10	6	3	0.0032	25	15	5

Note: 1) Assumed model based on 0.50 Au g/t variogram model.
2) Assumed model based on 0.50 Au g/t and 1.25 Au g/t variogram models.
3) Assumed model based on 1.25 Au g/t and 2.8 Au g/t variogram models.
4) Assumed model based on 2.8 Au g/t and 5.7 Au g/t variogram model..
5) Assumed model based on 5.7 Au g/t and 11 Au g/t variogram model.
6) Assumed model based on 11 Au g/t variogram model.

Table 14.16 Pahtavaara Gold Deposit Domain Group Low Grade Indicator Variogram Models Au g/t

Grade Variable or Indicator Threshold	Nugget (C0)	Rotation (dip→dip dir)			Structure 1				Structure 2			
		Major	Semi Major	Minor	Relative Sill 1 (C1)	Range (m)			Relative Sill 2 (C2)	Range (m)		
						Major	Semi Major	Minor		Major	Semi Major	Minor
0.15(1)	0.0520	90→90	0→80	0→170	0.1066	10	6	3	0.0494	40	30	10
0.30(1)	0.0615	90→90	0→80	0→170	0.1260	10	6	3	0.0585	40	30	10
0.50	0.0610	90→90	0→80	0→170	0.1250	10	6	3	0.0580	40	30	10
0.70(2)	0.0594	90→90	0→80	0→170	0.1106	10	6	3	0.0530	40	30	10
0.95(2)	0.0557	90→90	0→80	0→170	0.0945	10	6	3	0.0468	40	30	10
1.25	0.0490	90→90	0→80	0→170	0.0760	10	6	3	0.0390	40	30	10
1.65(3)	0.0444	90→90	0→80	0→170	0.0628	10	6	3	0.0327	37	28	9
2.10(3)	0.0380	90→90	0→80	0→170	0.0490	10	6	3	0.0260	33	27	7
2.75	0.0330	90→90	0→80	0→170	0.0390	10	6	3	0.0210	30	25	6
3.55(4)	0.0270	90→90	0→80	0→170	0.0302	10	6	3	0.0167	28	22	6
4.50(4)	0.0214	90→90	0→80	0→170	0.0226	10	6	3	0.0129	27	18	6
5.70	0.0170	90→90	0→80	0→170	0.0170	10	6	3	0.0100	25	15	6
7.20(5)	0.0121	90→90	0→80	0→170	0.0115	10	6	3	0.0064	23	15	6
8.90(5)	0.0093	90→90	0→80	0→170	0.0084	10	6	3	0.0043	22	15	5
13.0	0.0060	90→90	0→80	0→170	0.0052	10	6	3	0.0025	20	15	5
17.0(6)	0.0031	90→90	0→80	0→170	0.0027	10	6	3	0.0013	20	15	5
27.0(6)	0.0013	90→90	0→80	0→170	0.0011	10	6	3	0.0005	20	15	5

Note: 1) Assumed model based on 0.50 Au g/t variogram model.
2) Assumed model based on 0.50 Au g/t and 1.25 Au g/t variogram models.
3) Assumed model based on 1.25 Au g/t and 2.75 Au g/t variogram models.
4) Assumed model based on 2.75 Au g/t and 5.7 Au g/t variogram model.
5) Assumed model based on 5.7 Au g/t and 13 Au g/t variogram model.
6) Assumed model based on 13 Au g/t variogram model.

Figure 14.10 Pahtavaara Gold Deposit, Domain Group High Grade, Grade Variogram

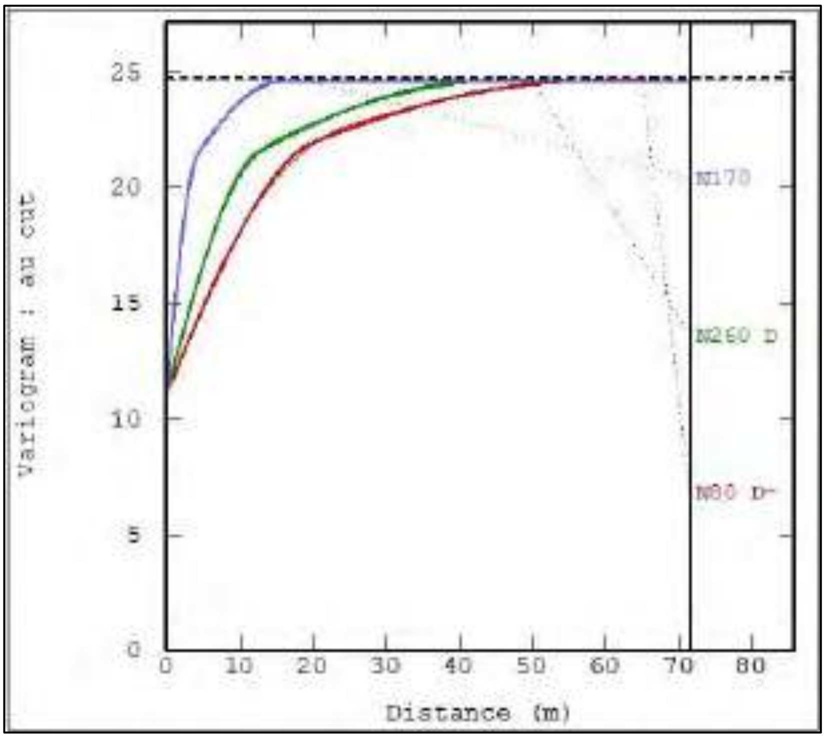


Figure 14.11 Pahtavaara Gold Deposit, Domain Group Medium Grade, Grade Variogram

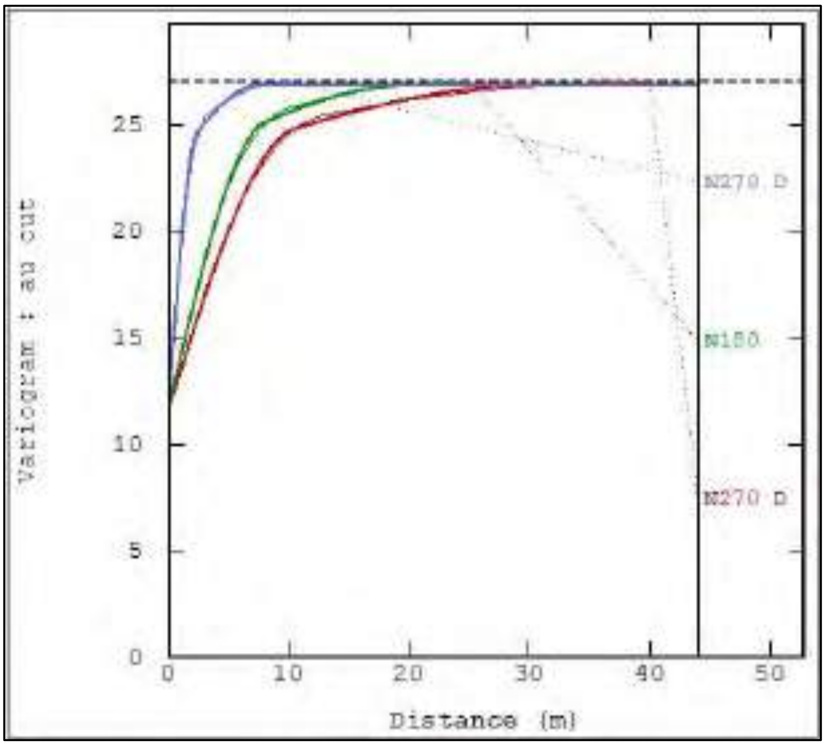


Figure 14.12 Pahtavaara Gold Deposit, Domain Group Medium/Low Grade, Grade Variogram

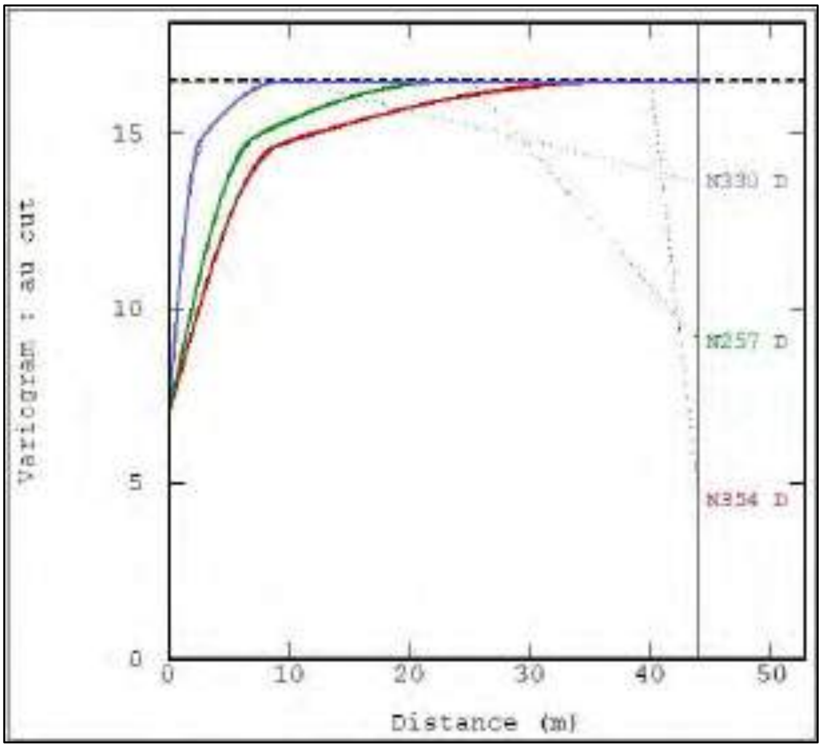
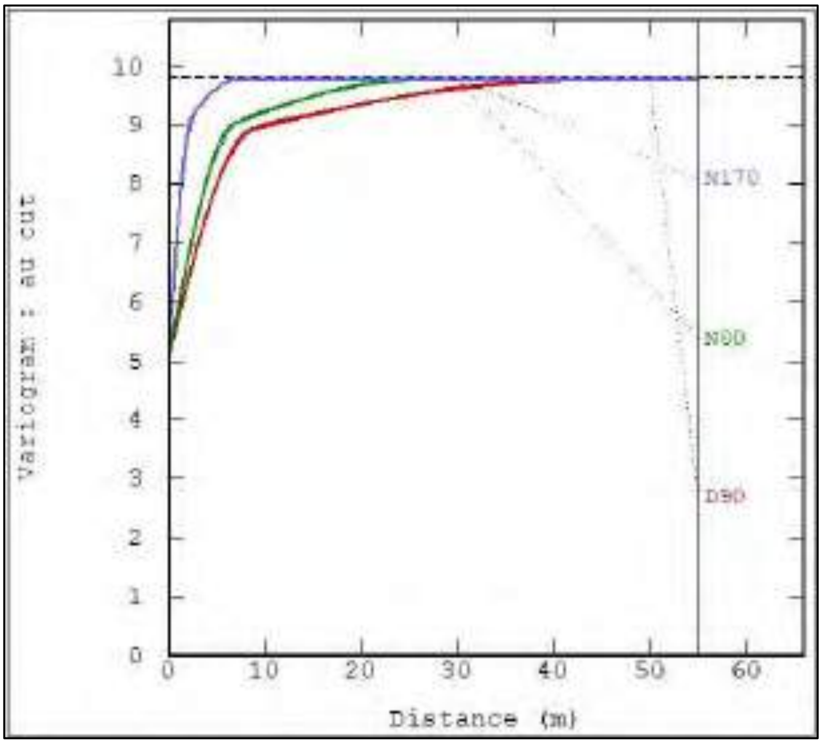


Figure 14.13 Pahtavaara Gold Deposit, Domain Group Low Grade, Grade Variogram



14.1.7 Block Modelling

A 3D block model was created in the ETRS89 (European Terrestrial Reference System) grid using Vulcan mining software. The parent block size was selected on the basis of the average drill spacing together with consideration of potential mining parameters. A parent cell size of 20 m E by 10 m N by 10 m RL which was sub-blocked down to 5 m E by 2.5 m N by 2.5 m RL (to ensure adequate volume representation). The models covered all the interpreted mineralisation zones and included suitable additional waste material to allow later mining engineering studies. Block coding was completed on the basis of the block centroid, wherein a centroid falling within any wireframe was coded with the wireframe solid attribute. The block model is unrotated.

The main block model parameters are summarised in Table 14.17. Variables were coded into the block models to enable MIK and OK estimation and subsequent MIK change of support and grade tonnage reporting. A visual review of the wireframe solids and the block model indicated correct flagging of the block model. Additionally, a check was made of coded volume versus wireframe volume which confirmed the above.

Table 14.17 Pahtavaara Gold Deposit, Block Model Parameters

	Northing (Y)	Easting (X)	RL (Z)
Min Coordinates	7,501,550	473,950	-300
Max Coordinates	7,502,100	475,450	330
Block size (m)	10.0	20.0	10.0
Sub Block size (m)	2.5	5.0	2.5
Rotation (° around axis)	0°	0°	0°

14.1.8 Bulk Density Data

A dry bulk density database has been supplied containing a total of 8,466 data. The database can be subdivided based on work carried out by Lappland Goldminers in 2009 and 2010 and subsequent work by Rupert Resources. Review of the two sets of data indicate no material difference Table 14.18.

Table 14.18 Pahtavaara Gold Deposit, Density Statistics

Company	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
Lappland Goldminers	752	2.05	4.12	2.921	0.142	0.020	0.049
Rupert	7,714	2.03	4.5	2.93	0.094	0.009	0.032

Rupert have calculated dry bulk densities on the basis of the weight in water method. Density readings have been taken on whole drill core and are distributed across all areas of the deposit. It is recognised that across the deposit, different lithologies are likely to have different densities, however a sufficiently coherent geological model does not yet exist to allow for differentiation between the lithologies present. A bulk density of 2.9 t/m^3 has therefore been applied as a tonnage factor to allow for appropriate grade tonnage reporting.

14.1.9 GRADE ESTIMATION

INTRODUCTION

MIK was applied to grade estimation at the Pahtavaara Gold Project within the defined indicator mineralisation shells. The minor domains forming a low-grade halo to the main mineralised domains were estimated via OK. Estimation was completed in the mining package Vulcan using the GSLib geostatistical software while geostatistical change of support parameters were developed in Isatis geostatistical software. MIK is considered a robust estimation methodology for grade estimates for gold deposits such as Pahtavaara where high levels of short scale variability are present. MIK grade estimation with change of support has been applied to produce 'recoverable' gold estimates targeting a SMU of 5 m E x 2.5 m N x 2.5 m RL.

THE MULTIPLE INDICATOR KRIGING METHOD

The MIK technique is implemented by completing a series of OK estimates of binary transformed data. A composite sample, which is equal to or above a nominated cut-off or threshold, is assigned a value of 1, with those below the nominated indicator threshold being assigned a value of 0. The indicator estimates, with a range between 0 and 1, represent the probability the point will exceed the indicator cut-off grade. The probability of the points exceeding a cut-off can also be considered broadly equivalent to the proportion of a nominated block that will exceed the nominated cut-off grade.

The estimation of a complete series of indicator cut-offs allows the reconstitution of the local histogram or conditional cumulative distribution function (ccdf) for the estimated point. Based on the ccdf, local or block properties, such as the block mean and proportion (tonnes) above or below a nominated cut-off grade can be investigated.

POST MIK PROCESSING - E-TYPE ESTIMATES

The E-type estimate provides an estimate for the grade of the total block or bulk-mining scenario. This is achieved by discretising the calculated ccdf for each block into a nominated number of intervals and interpolating between the given points with a selected function (e.g. the linear, power or hyperbolic model) or by applying intra-class mean grades. The sum of all these weighted interpolated points or mean grades enables an average whole block grade to be determined.

The following example shows the determination of an E-type estimate for a block containing three indicator cut-offs.

The indicator cut-offs and associated probabilities calculated are shown in Table 14.19.

Table 14.19 Pahtavaara Gold Deposit, Indicator Cut-off and Probability

Indicator	Cut-off Grade Au g/t	Indicator Probability (cumulative)
minimum grade *	0	0.00 **
indicator 1	1	0.40
indicator 2	2	0.65
indicator 3	3	0.85
maximum grade *	4	1.00 **

Note: *Cut-off grades determined by the user.
**Indicator probability is assumed at the minimum and maximum cut-off.

The whole block grade can now be determined in this block with the following parameters used for the purposes of the interpolation:

- Number of discretisation intervals: 4.
- Linear extrapolation between all points (median grade between nominated cut-offs).

The worked example is then calculated with the following steps:

- Interval 1 (0-1 g/t Au) median grade x probability/proportion attributed to the interval (0.5 g/t Au x 0.40 = 0.200).
- Interval 2 (1 – 2 g/t Au) median grade x proportion (1.5 g/t Au x 0.25 = 0.375).
- Interval 3 (2 – 3 g/t Au) median grade x proportion (2.5 g/t Au x 0.20 = 0.500).
- Interval 4 (3 – 4 g/t Au) median grade x proportion (3.5 g/t Au x 0.15 = 0.525).

Calculate total grade average all calculated intervals ((0.2 + 0.375 + 0.500 + 0.525)/1) = 1.60 g/t Au.

It is also possible from this example to calculate the proportion and grade above a nominated cut-off (e.g. 2 g/t - at sample support or complete selectivity). The following steps would be undertaken to calculate the tonnes and grade at sample selectivity using a 2 g/t cut-off:

- Interval 3 (2 – 3 g/t Au) median grade x proportion (2.5 g/t Au x 0.20 = 0.500).
- Interval 4 (3 – 4 g/t Au) median grade x proportion (3.5 g/t Au x 0.15 = 0.525).
- Calculate total grade average all calculated intervals ((0.500 + 0.525)/0.35) = 2.93 g/t Au with 0.35% of the block above the cut-off.

The effect of using a non-linear model to interpolate between cut-offs is to shift the grade weighting associated with that cut-off away from the median. The intra-class means based on the cut composite data have been used to reconstitute the ccdf and produce block statistics.

It is noted, however, that the calculation of the E-type estimate and complete selectivity often does not allow mine planning to the level of selectivity which is proposed for production. To achieve an estimate which reflects the levels of mining selectivity envisaged, a SMU correction is often applied to the calculated ccdf.

Support Correction (Selective Mining Unit Estimation)

A range of techniques are known to produce a support correction and therefore allow for selective mining unit emulation. The common features of the support correction are:

- Maintenance of the mean grade of the histogram (E-type mean).
- Adjustment of the histogram variance by a variance adjustment factor (the 'f' factor).

The variance adjustment factor, used to reduce the histogram or ccdf variance, can be calculated using the variogram model. The variance adjustment factor is often modified to account for the likely grade control approach or 'information effect'.

In simplest terms, the variance adjustment factor takes into account the known relationship derived from the dispersion variance.

Total variance = variance of samples within blocks + variance between blocks.

The variance adjustment factor is calculated as the ratio of the variance between the blocks and the variance of the samples within the blocks, with a small ratio (e.g. 0.10) indicating a large adjustment of the ccdf variance and large ratio (e.g. 0.80) representing a small shift in the ccdf.

Two simple support corrections that are available include the Affine and Indirect Lognormal correction, which are both based on the permanence of distribution. The discrete Gaussian model is often applied to global change of support studies and has been generated on the composite dataset as a comparison. The indirect lognormal correction was applied to the MIK grade estimates.

Indirect Lognormal Correction

The indirect lognormal correction can be implemented by adjusting the quantiles (indicator cut-offs) of the ccdf with the variance adjustment factor so that the adjusted ccdf represents the statistical characteristics of the block volume of interest.

This is implemented with the following formula:

$$q' = a \times q^b$$

q = quantile of distribution,

q' = quantile of the variance-reduced distribution.

where the coefficients a and b, are given by the following formula:

$$a = \sqrt{\frac{m}{f.\,CV^2+1}}\left[\sqrt{\frac{CV^2+1}{M}}\right]$$

$$b = \sqrt{\frac{\ln(f.\,CV^2+1)}{\ln(CV^2+1)}}$$

m = mean of distribution.

f = variance adjustment factor.

CV = coefficient of variation.

At the completion of the quantile adjustments, grades and tonnages (probabilities are then considered a pseudo-tonnage proportion of the blocks) at a nominated cut-off grade can be calculated using the methodology described above (E-type). The indirect lognormal correction, as applied to Pahtavaara, is the best suited of the common adjustments applied to MIK to produce selective mining estimates for positively skewed distributions.

Multiple Indicator Kriging Parameters

MIK estimates were completed using the indicator variogram models (Section 14.1.6), and a set of ancillary parameters controlling the source and selection of composite data. The sample search parameters were defined based on the variography and the data spacing, and a series of sample search tests performed in Isatis geostatistical software. A total of 17 indicator thresholds were estimated for all estimation domains (see Table 14.20).

OK estimates were completed on the minor estimation domains forming a halo to the main domains using the grade variogram models (Section 14.1.6), and a set of ancillary parameters controlling the source and selection of composite data. The sample search parameters were defined based on the variography and the data spacing, and a series of sample search tests performed in Isatis geostatistical software.

The sample search parameters for the MIK estimations are provided in Table 14.20. A combination of soft domain boundaries was used for the estimation throughout to reflect continuity between domains or otherwise. A three-pass estimation strategy (where required) was applied to each domain, applying a progressively expanded and less restrictive sample search to the successive estimation pass, and only considering blocks not previously assigned an estimate. Parent cell estimations (20 m E by 10 m N by 10 m RL) were applied throughout and discretisation was applied on the basis of 3 X by 3 Y by 2 RL for 18 discretisation points per block.

Table 14.20 Pahtavaara Gold Deposit, MIK Sample Search Criteria

Domain	Pass	Sample Search Orientation (dip/dip direction˚)			Sample Search Distance (m)			Numbers of 2 m Composites			% Blocks Estimated
		Major	Semi Major	Minor	Major	Semi Major	Minor	Min.	Max.	Max Per Drill Hole	
3510	Pass 1	0→75	80→165	10→345	40	40	20	24	72	16	98
	Pass 2	0→75	80→165	10→345	80	80	40	18	72	-	99
	Pass 3	0→75	80→165	10→345	80	80	40	12	72	-	100
3520	Pass 1	0→80	60→170	30→350	40	40	20	24	72	16	78
	Pass 2	0→80	60→170	30→350	80	80	40	18	72	-	95
	Pass 3	0→80	60→170	30→350	160	160	50	12	72	-	100
3530	Pass 1	0→110	15→200	15→20	40	40	20	24	72	16	85
	Pass 2	0→110	15→200	15→20	100	100	30	24	72	-	100
	Pass 3									-	
3540	Pass 1	0→260	65→170	25→350	40	40	20	24	72	16	83
	Pass 2	0→260	65→170	25→350	120	120	60	18	72	-	94
	Pass 3	0→260	65→170	25→350	240	240	120	12	72	-	100
3550	Pass 1	-30→260	60→260	0→350	40	40	20	24	72	16	93
	Pass 2	-30→260	60→260	0→350	80	80	40	18	72	-	98
	Pass 3	-30→260	60→260	0→350	80	80	40	12	72	-	100
3560	Pass 1	-40→265	49→250	8→349	40	40	20	24	72	16	96
	Pass 2	-40→265	49→250	8→349	80	80	40	18	72	-	99
	Pass 3	-40→265	49→250	8→349	120	120	60	12	72	-	100
3570	Pass 1	-40→260	42→218	23→330	40	40	20	24	72	16	80
	Pass 2	-40→260	42→218	23→330	120	120	60	18	72	-	99
	Pass 3	-40→260	42→218	23→330	120	120	60	12	72	-	100
3580	Pass 1	0→260	65→170	25→350	40	40	20	24	72	16	91
	Pass 2	0→260	65→170	25→350	80	80	40	18	72	-	100
	Pass 3									-	
3590	Pass 1	-75→185	0→95	15→185	40	40	20	24	72	16	79
	Pass 2	-75→185	0→95	15→185	120	120	60	18	72	-	94

	Pass 3	-75→185	0→95	15→185	160	160	80	12	72	-	100
35100	Pass 1	-50→250	19→316	34→213	40	40	20	24	72	16	93
	Pass 2	-50→250	19→316	34→213	80	80	40	18	72	-	99
	Pass 3	-50→250	19→316	34→213	120	120	60	12	72	-	100
35110	Pass 1	-30→295	17→215	54→331	40	40	20	24	72	16	91
	Pass 2	-30→295	17→215	54→331	80	80	40	18	72	-	98
	Pass 3	-30→295	17→215	54→331	160	160	80	12	72		100
35120 North	Pass 1	-60→205	0→115	30→205	40	40	20	24	72	16	84
	Pass 2	-60→205	0→115	30→205	80	80	40	18	72	-	97
	Pass 3	-60→205	0→115	30→205	120	120	60	12	72	-	100
35120 South	Pass 1	-60→295	0→205	30→295	40	40	20	24	72	16	96
	Pass 2	-60→295	0→205	30→295	80	80	40	18	72	-	100
	Pass 3									-	
35130	Pass 1	-42→159	28→97	35→209	80	80	40	24	72	16	85
	Pass 2	-42→159	28→97	35→209	160	160	80	18	72	-	100
	Pass 3									-	
35140	Pass 1	-30→250	0→160	60→250	40	40	20	24	72	16	73
	Pass 2	-30→250	0→160	60→250	120	120	60	18	72	-	99
	Pass 3	-30→250	0→160	60→250	120	120	60	12	72	-	100

The sample search parameters for the OK estimations are provided in Table 14.12. A combination of soft and hard domain boundaries was used for the estimation throughout to reflect continuity between domains or otherwise. Only one estimation pass was considered with a search neighbourhood of sufficient parameters to enable estimation of all required blocks. Estimations were on the basis of SMU block dimensions (5 m E by 2.5 m N by 2.5 m RL) and discretisation was applied on the basis of 2 X by 2 Y by 2 RL for 8 discretisation points per block.

Table 14.21 Pahtavaara Gold Deposit, OK Sample Search Criteria

Domain	Sample Search Orientation (dip/dip direction°)			Sample Search Distance (m)			Numbers of 2 m Composites		
	Major	Semi Major	Minor	Major	Semi Major	Minor	Min	Max	% Estimated
2010	0→75	80→165	10→345	80	80	40	6	8	100
2020	0→80	60→170	30→350	120	120	60	4	8	100
2030	0→110	15→200	15→20	120	120	60	4	8	100
2040	0→260	65→170	25→350	160	160	80	4	6	99.7
2050	-30→260	60→260	0→350	60	60	30	4	8	100
2060	-40→265	49→250	8→349	80	80	40	4	8	98.3
2070	-40→260	42→218	23→330	60	60	40	4	8	100
2080	0→260	65→170	25→350	60	60	30	4	6	100
2090	-75→185	0→95	15→185	160	160	80	4	6	100
20100	-50→250	19→316	34→213	80	80	40	4	8	100
20110	-30→295	17→215	54→331	120	120	60	4	8	100
20120 North	-60→205	0→115	30→205	80	80	40	6	8	99.9
20120 South	-60→295	0→205	30→295	60	60	30	6	8	99.6
20130	-42→159	28→97	35→209	80	80	40	4	6	100

20140	-30→250	0→160	60→250	80	80	40	4	8	99.7

Change of Support

Applying the modelled variography, variance adjustment factors were calculated for to emulate a 5 m E x 2.5 m N x 2.5 m RL SMU via the indirect lognormal change of support. The intra-class composite mean grades were used in calculating the whole block and SMU grades. The change of support study also included the calculation of the theoretical global change of support via the discrete Gaussian change of support model.

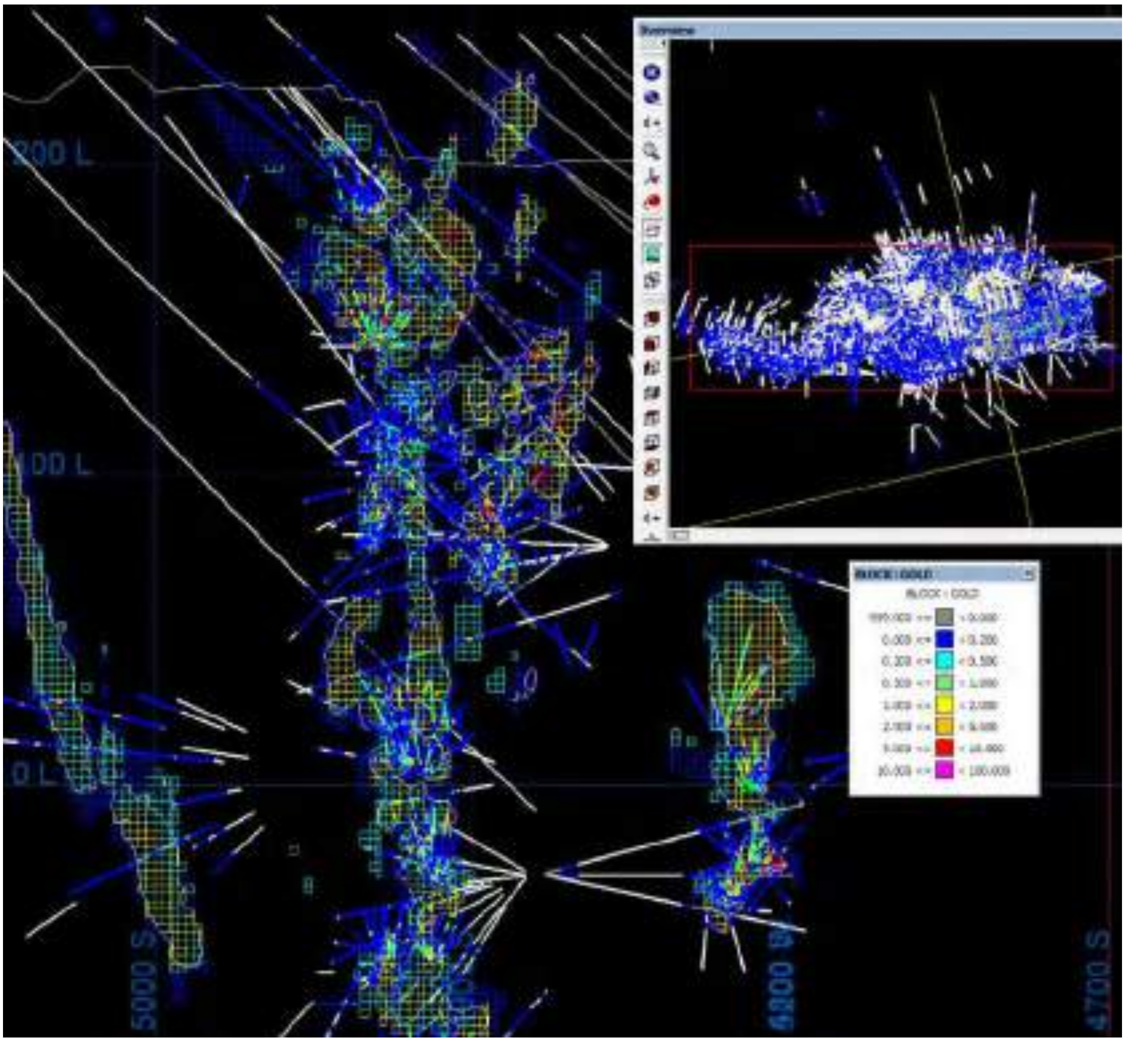
An 'information effect' factor is commonly applied to the originally derived panel-to-block variance ratios to determine the final variance adjustment ratio. The goal of incorporating information effect is to calculate results taking into account that mining takes place based on grade control information. There will still be a quantifiable error associated with this data and it is this error we want to incorporate. This is achieved in practice by running a test kriging estimation of an SMU using grade control data (the results required to incorporate this option in the change of support do not depend on the assay data so the grade control data can be hypothetical). The incorporation of the information effect is commonly found to be negligible, however can have a significant effect in some cases. In this case, the information effect factor was found to have a minor effect and has been incorporated in the calculation.

The variance adjustment ratios as applied to all mineralised domains was 0.1.

Grade Localisation

MIK grade estimates are generated in large blocks or panels (in the case of Pahtavaara, 20 m E x 10 m N x 10 m RL) and are inherently not intuitive to review. Post processing of these MIK estimates aims to simplify the presentation by producing a single SMU dimension block grade where the distribution of the grades in the panel matches that of the distribution in the SMU's. The MIK panel grades have been localised to SMU dimension blocks in Isatis software. The SMU dimension was 5 m E x 2.5 m N x 2.5 m RL. Validation of the results indicates a near identical distribution and the resultant model has been accepted. A typical section is presented below (Figure 14.14).

Figure 14.14 Typical Sectional View Displaying Localised Au Grades



Estimate Validation

All relevant statistical information was recorded to enable validation and review of the MIK estimates. The recorded information included:

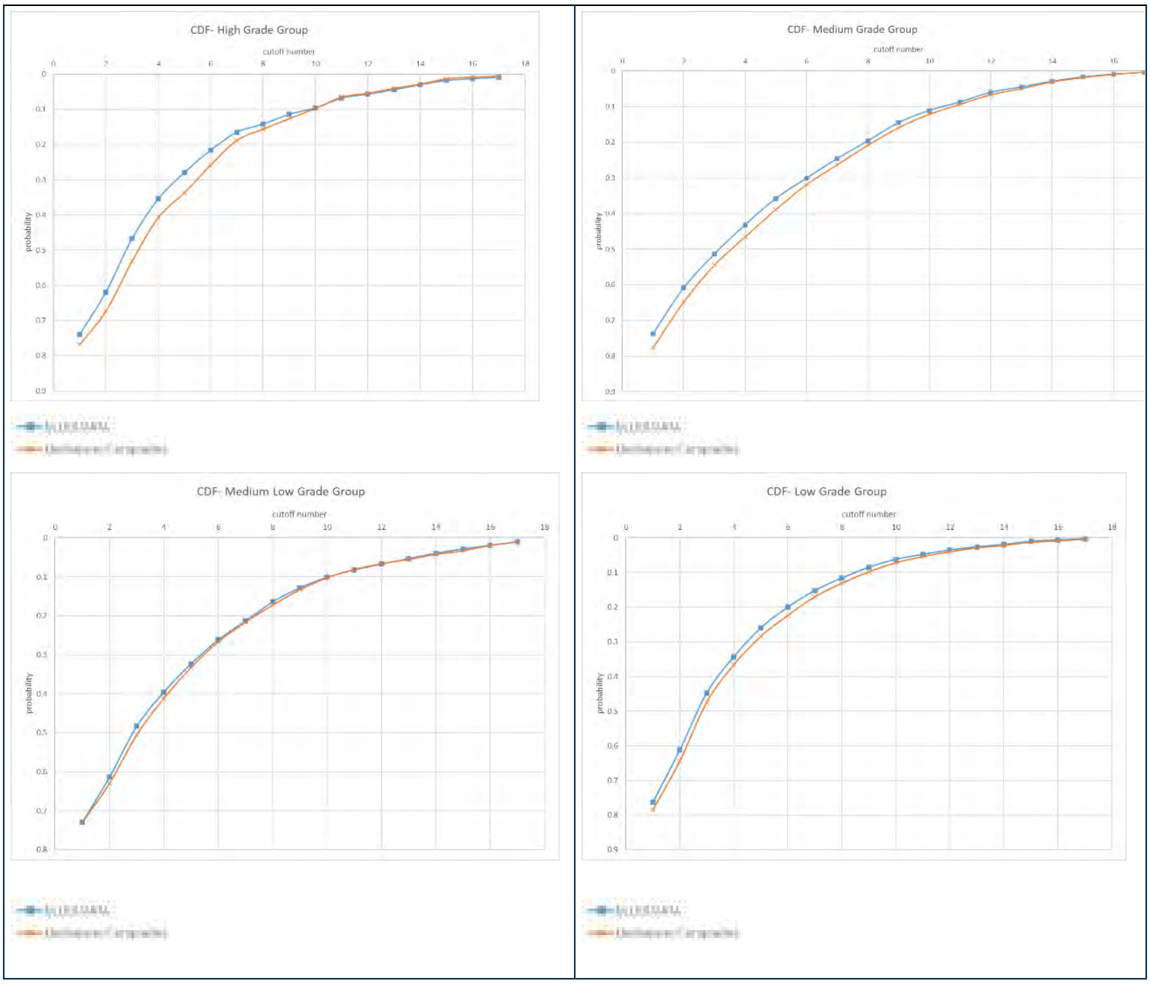
- Number of samples used per block estimate.
- Number of drill holes from which samples selected.
- Average distance to samples per block estimate and distance to nearest sample.
- Estimation flag to determine in which estimation pass a block was estimated.
- Number of drill holes from which composite data were used to complete the block estimate.

The estimates were reviewed visually and statistically prior to being accepted. The review included the following activities:

- Comparison of the E-type estimate versus the mean of the composite dataset, including weighting where appropriate to account for data clustering.
- Comparison of the reconstituted cumulative conditional distribution functions of the estimated blocks (indicator kriging) versus the input composite data (Figure 14.15).

- Visual checks of cross sections, long sections, and plans.

Figure 14.15 CCDF Validation



Alternative estimates were also completed to test the sensitivity of the reported model to the selected MIK interpolation parameters. An insignificant amount of variation in overall grade was noted in the alternate estimations.

Validation of localised block Au grades has been undertaken on a per domain grouping basis for the MIK domains and on a domain by domain basis for the OK domains. Validation was achieved by comparing the block mean grades with the relevant composite mean grades (Table 14.22).

Table 14.22 Pahtavaara Gold Deposit, Comparison of Block Grades with Composite Mean Grades, All Data Used

	Zone	All Composites, (declustered, capped)	All Composites, (non-declustered, capped)	Block Model Grades	% Diff Block Model versus Declustered Mean
OK Domains (uncapped)	2010	0.225	0.222	0.215	-4.4%
	2020	0.261	0.294	0.233	-10.7%
	2030	0.366	0.383	0.291	-20.5%
	2040	0.218	0.251	0.205	-6.0%
	2050	0.235	0.248	0.227	-3.4%
	2060	0.207	0.21	0.213	2.9%
	2070	0.385	0.388	0.338	-12.2%
	2080	1.616	2.515	0.279	-82.7%
	2090	0.244	0.246	0.196	-19.7%
	20100	0.302	0.262	0.248	-17.9%
	20110	0.187	0.214	0.209	11.8%
	20120	0.269	0.274	0.2	-25.7%
	20130	0.912	1.663	0.488	-46.5%
	20140	0.511	0.534	0.563	10.2%
MIK Domains	High	1.803	2.12	1.908	5.82%
	Medium	1.763	2.347	1.754	-0.51%
	Med low	1.707	2.094	1.699	-0.47%
	Low	1.242	1.452	1.114	-10.31%

For the MIK grade domains, a reasonable correlation can be drawn with most domains falling within the range of approximately ±10%. The low grade OK grade domains demonstrate greater variability in comparison to the input composites, however the difference is overwhelmingly negative. As these domains are intended as a dilution skin to the main mineralised MIK grade domains, the OK grade estimates are considered acceptable for this purpose.

14.1.10 Depletion for Mining Activity

Depletion to account for mining activity has been applied to the model. Depletion has been applied as at the effective date via the use of surveyed topographic surfaces, underground stopes, declines and other associated infrastructure. Depletion has been applied by block model flag to identify the mined and in-situ portions of the models.

14.1.11 Resource Classification

The resource categorisation was based on the robustness of the various data sources available, including:

- Geological knowledge and interpretation.
- Variogram models and the ranges of the first structure in multi-structure models.
- Drilling density and orientation.
- Estimation quality statistics.

The resource estimates for the Pahtavaara Gold Deposit have been classified as Indicated and Inferred Mineral Resources based on the confidence levels of the key criteria as presented in Table 14.23.

Table 14.23 Pahtavaara Gold Deposit, Confidence Levels by Key Criteria

Items	Discussion	Confidence
Drilling Techniques	Diamond/percussion sludge - Industry Standard approach.	High for diamond, Moderate/Low for sludge
Logging	Standard nomenclature has been adopted but not used in entire database.	Moderate
Drill Sample Recovery	Recoveries are not recorded in entire database but diamond core recoveries assumed acceptable. Unknown recoveries for sludge.	Moderate
Sub-sampling Techniques and Sample Preparation	Diamond and RC sampling conducted by industry standard techniques.	Moderate/High
Quality of Assay Data	Appropriate quality control procedures only available for work completed by Rupert. They were reviewed on site and considered to be of industry standard.	Moderate/High
Verification of Sampling and Assaying	Sampling and assaying procedures have been assessed and are considered of appropriate industry standards.	Moderate
Location of Sampling Points	Survey of all collars conducted with accurate survey equipment. Investigation of downhole survey indicates appropriate behaviours.	Moderate/High
Data Density and Distribution	Majority of regions defined on a notional 25 m E x 25 m N drill spacing. Grade control spaced drilling available.	Moderate/high
Audits or Reviews	Data collection assessed during site review.	N/A
Database Integrity	Data base is largely legacy with numerous campaigns and UG grade control. Industry standard approach applied by Rupert.	Moderate
Geological Interpretation	Mineralisation controls are moderately well understood. The mineralisation constraints are robust but relatively broad and therefore of moderate confidence.	Moderate
Estimation and Modelling Techniques	MIK is considered to be appropriate given the geological setting and grade distribution. Minor domains are estimated by OK.	High
Cut-off Grades	MIK is independent of cut-off grade although the mineralisation constraints were based on a notional 0.3 g/t Au lower cut-off grade. A 0.5 g/t lower cut-off grade is considered appropriate for open pit mining and a 1.5 g/ cut-off grade is considered appropriate for mineralisation that would be mined using underground methods.	Moderate/High
Mining Factors or Assumptions	A 5 m E x 2.5 m N x 2.5 m RL SMU emulated for gold. UG mining assumed. Change of support for Inferred component has higher degree of uncertainty.	Moderate
Metallurgical Factors or Assumptions	Not applied or available.	N/A
Tonnage Factors (In-situ Bulk Densities)	Sufficient data exists to enable high confidence in the applied density values.	High

14.1.12 Resource Reporting

The Mineral Resource is reported both within a designed open pit and as a potential underground operation below that.

The summary resources for the Pahtavaara Gold Project are provided in Table 14.24 along with the mineral resource estimate at a range of additional cut-off grades to demonstrate grade tonnage relationships at higher and lower cut-off grades. The preferred lower cut-off for reporting is 0.5 g/t for the open pit portion and 1.5 g/t for the underground mining portion.

Table 14.24 Pahtavaara Gold Deposit, Mineral Resource Report, Summary Grade Tonnage Report

Resource Category		Lower Cut-off Grade (g/t Au)	Tonnes	Average Grade (g/t Au)	Gold Metal (Troy ounces)	Gold Metal (Kg)
Indicated	Open Pit	0.4	1,000,000	2.0	63,000	1,900
		0.5	**900,000**	**2.2**	**62,000**	**1,900**
		0.6	800000	2.3	60,000	1,800
		0.7	700,000	2.5	59,000	1,800
	Underground	1.0	1,500,000	2.8	140,000	4,400
		1.5	**1,000,000**	**3.7**	**120,000**	**3,700**
		2.0	700,000	4.6	100,000	3,200
		2.5	500,000	5.5	90,000	2,800
Indicated Open Pit and Underground			**1,900,000**	**3.0**	**180,000**	**5,600**
Inferred	Open Pit	0.4	4,200,000	1.5	200,000	6,100
		0.5	**3,700,000**	**1.6**	**190,000**	**5,900**
		0.6	3,300,000	1.7	180,000	5,700
		0.7	3,000,000	1.8	180,000	5,500
	Underground	1.0	3,900,000	2.3	290,000	8,900
		1.5	**2,200,000**	**3.1**	**220,000**	**6,800**
		2.0	1,400,000	3.9	170,000	5,400
		2.5	900,000	4.8	140,000	4,400
Inferred Open Pit and Underground			**5,900,000**	**2.1**	**410,000**	**12,700**

Note: Appropriate rounding has been applied.

The designed pit and cut-off grade are based on the following economic parameters: Gold price $1,650 /oz, 80% mining recovery, 91% Au recovery to concentrate with subsequent 98% recovery to Doré. Estimated open pit mining costs are $2.6 /t, process costs $10.2 /t, and $1 /t other costs including CDF and closure costs. Estimated process costs at Pahtavaara include production of concentrate at Pahtavaara and transport of concentrate to Ikkari for refining. G&A including refining and royalties are $3.1 /t. The calculated cut-off grade is rounded up to 0.5 g/t for reporting.

At underground mining costs of $49.6 /t and 95% mining recovery, the calculated underground cut-off grade is rounded up to 1.5 g/t for reporting.

The effective date of this Mineral Resource is 28th November 2022. It is not anticipated that this Mineral Resource estimate will be materially affected, to any extent, by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors. It should be noted that mineral resources that are not mineral reserves do not have demonstrated economic viability

14.2 IKKARI

14.2.1 INTRODUCTION

This Mineral Resource for the Ikkari Gold Deposit has been estimated as at the effective date of the 28th November 2022. Gold grade estimation was completed using MIK for the mineralised domains. MIK grade estimates have been localised to an SMU dimension using an analogous methodology to Localised Uniform Conditioning. This estimation approach was considered appropriate based on review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralisation, and the style, geometry and tenor of mineralisation. The estimation was constrained with geological and mineralisation interpretations.

14.2.2 DATABASE VALIDATION

The resource estimation was based on the available exploration drill hole database which was compiled in-house by Rupert Resources. The database has been reviewed and validated prior to commencing the resource estimation study.

The database consists of solely of surface diamond drilling. Database statistics are provided below as Table 14.25. A plan view of all drilling is presented in Figure 14.16.

Table 14.25 Summary of Available Drill Data for Ikkari

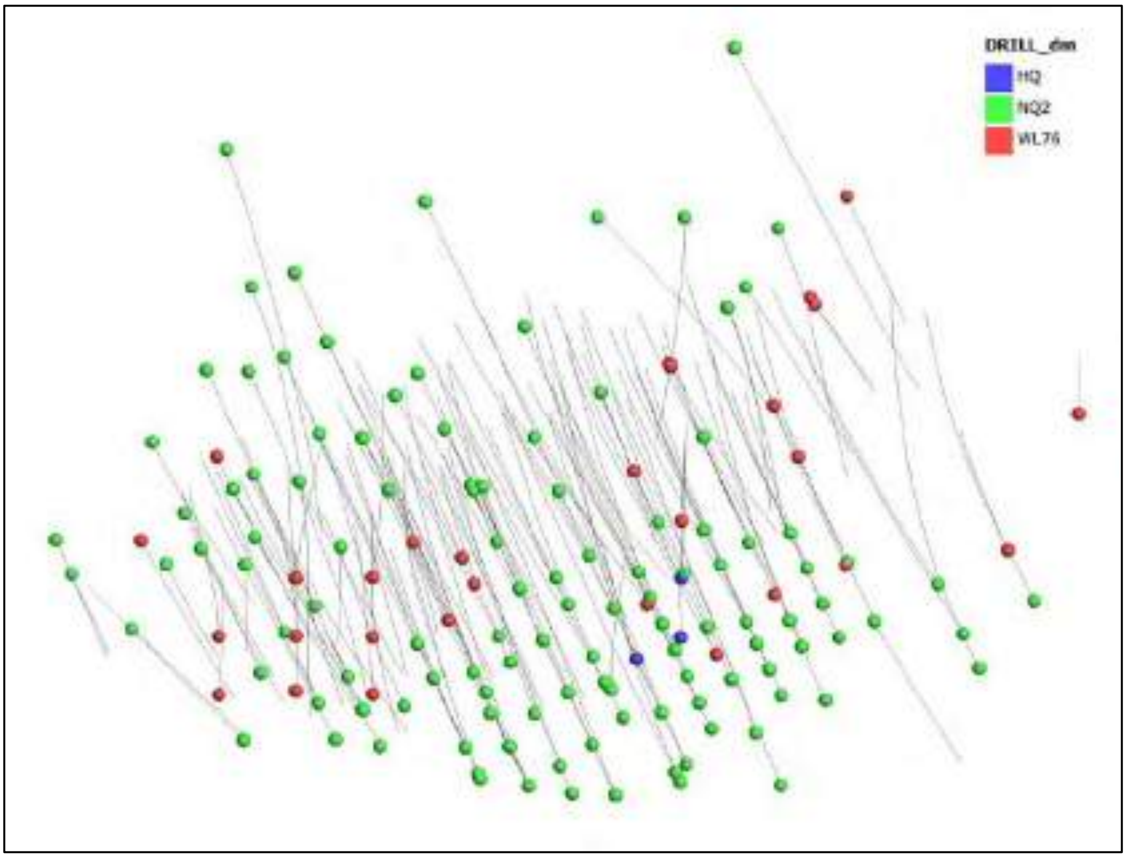
Company	DH Type	Holes	Metres	% of Total
Rupert Resources (April 2020 to May 2022)	Diamond (HQ)	4	1,098	1.3
	Diamond (NQ2)	162	72,485	88.7
	Diamond (WL76)	32	8,143	10.0
Total		**197**	**81,726**	**100%**

All diamond drilling has been sampled per meter and assayed for gold as well as 50 other multi-elements.

The resultant database has been validated, and the checks made to the database prior to use included:

- Check for overlapping intervals.
- Downhole surveys at 0 m depth.
- Consistency of depths between different data tables.
- Check gaps in the data.
- Replacing less than detection samples with half detection.
- Replacing intervals with no sample with -999.
- Replacing intervals with assays not received with -998.

Figure 14.16 Ikkari Gold Deposit, Plan View of all Drilling



14.2.3 INTERPRETATION AND MODELLING

MINERALISATION INTERPRETATION

Mineralisation at Ikkari is hosted by sedimentary intercalations within extrusive ultramafic rocks. This heterogeneous package has been folded into a series of tight (metre-scale), upright folds, which have been subsequently cut by later hydrothermal breccias (sub-vertical). Early thrusts bound the mineralised zone to the north and south.

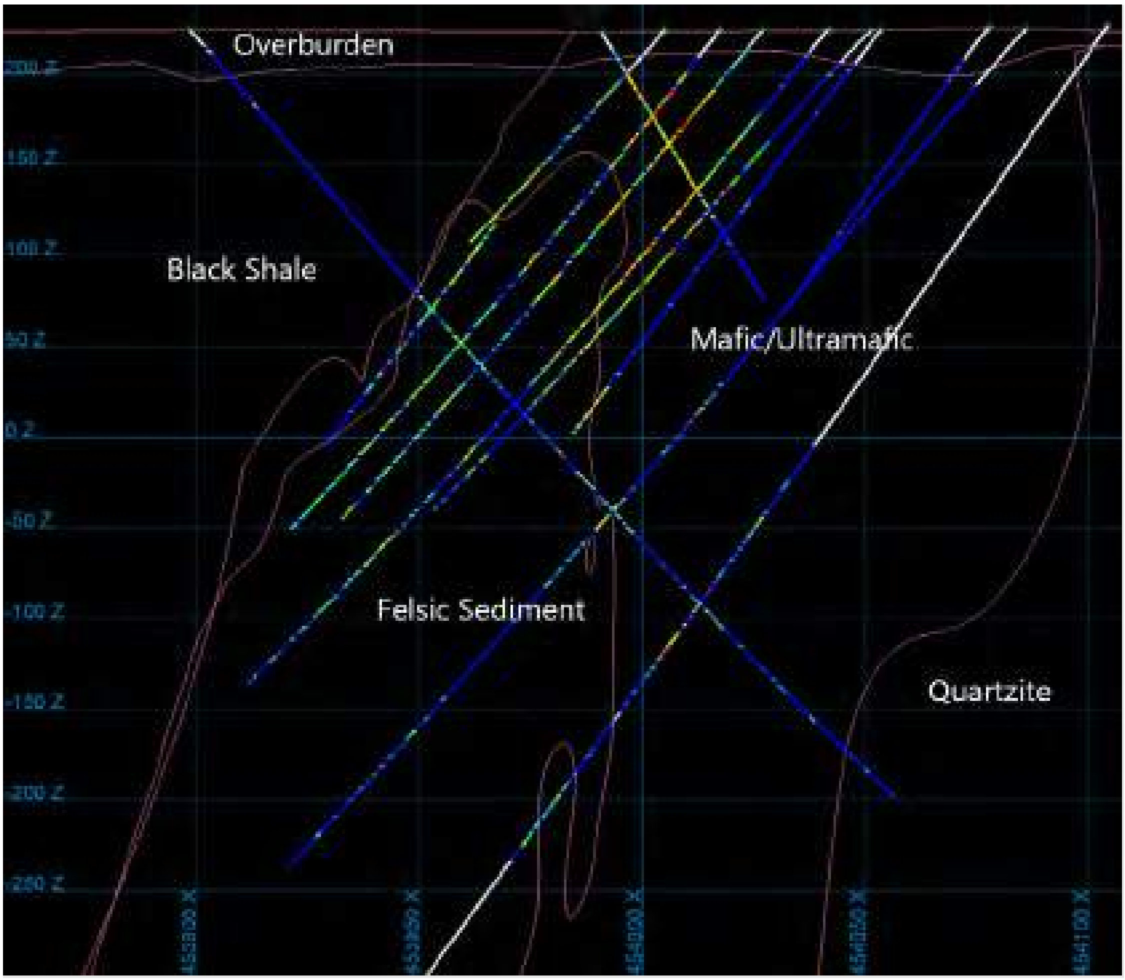
Mineralisation at Ikkari occurs in four principal styles:

- Brittle-fracture in intensely albite-altered felsic sediments that controls veinlets of gold associated with fine-grained pyrite and magnetite. This type of mineralisation is particularly prevalent in the north-western part of Ikkari where felsic sediments form a large block which pinches out eastwards.
- Complex and concentrated short-wavelength (metre-scale) parasitic folding of narrow felsic sediment intercalations within intensely chlorite-sericite-altered mafic-ultramafic rocks. Intense, irregular carbonate-quartz veining is frequently developed in these zones and are also mineralised. This type of mineralisation comprises the bulk of the high-grade, broad drill intercepts within the central part of the deposit.
- At lithological contacts; notably within intensely sericite-pyrite-(±fuchsite)-altered sediments, at contacts with felsic sediments or mafic-ultramafic rocks.
- Within and the margins of, several phases of hydrothermal and tectonic brecciation, 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.

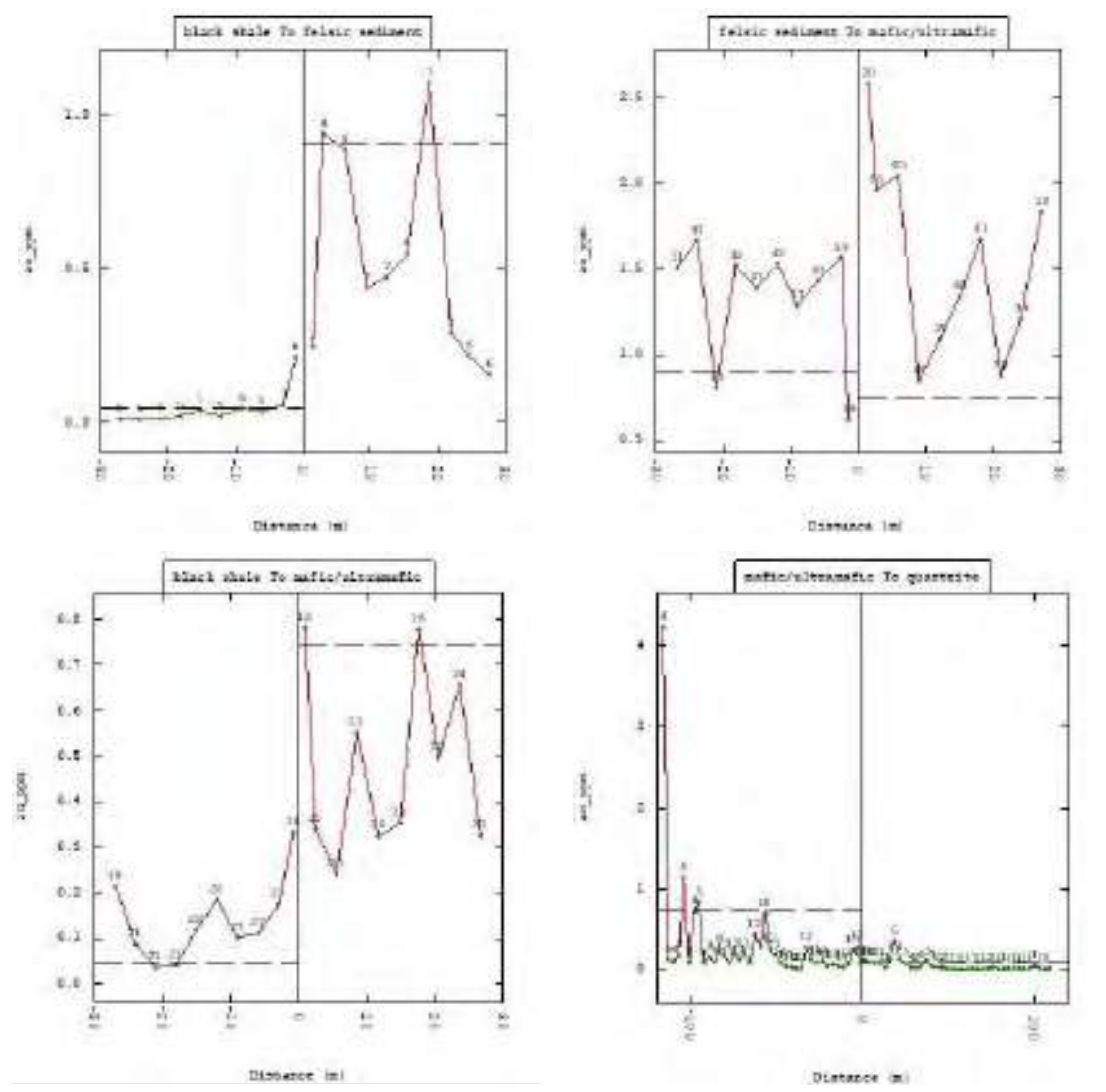
As a consequence of the structural complexity of the mineralised lithologies, the mineralisation can present as somewhat irregularly distributed on the scale of the drill sections completed to date (largely infilled to 40 m), although mineralised zones overall persist across multiple sections. Figure 14.17 presents a sectional view demonstrating variability in grade, thickness and orientation of gold mineralisation. The intricate folding and faulting of the sedimentary intercalations and the overprinting breccias makes the traditional approach of wireframing 'host lithology' on a sectional and plan basis extremely difficult with multiple plausible geometrical solutions often existing.

Figure 14.17 Cross Section Showing Main Mineralised Zones (Looking Towards 065°)



Contact analysis tests were performed to determine if changes in gold grades were related to the various lithological contacts. The tests examine the applicability of the lithological boundaries as estimation boundaries as significant changes in gold grades sustained at and across the lithological boundaries may require these to be used as such. The Contact Analysis application calculates and displays the mean value of a variable in a domain as a function of the distance of the samples to the contact with another domain. The mean value is calculated on samples at a predetermined lag distance (3 m in this case) along the drill holes. Results are graphically displayed in Figure 14.18.

Figure 14.18 Ikkari Gold Deposit, Lithological Contact Analysis



The graphical representation adequately demonstrates a marked change in grade from the black shale to the felsic sediment while the transition from the black shale to the mafic/ultramafic lithology is more gradually transitional. In the case of the felsic sediment transition to the mafic/ultramafic there is no evident change indicating that this lithological boundary may not be relevant as a control on gold grade. No grade change can be demonstrated at the boundary of the quartzite and mafic/ultramafic.

To establish appropriate grade continuity, the mineralisation models were therefore based upon a nominal 0.3 ppm Au indicator mineralisation shell estimated using 5 m unconstrained downhole composites. This interpretation is designed to capture the broad mineralisation halo that encompasses the geological system and is not intended to constrain individual veins or lithologies. As the main grade estimation technique is MIK with change of support technique, this type of mineralisation constraint is deemed appropriate.

The mineralisation grade shells were generated by grade estimation via indicator kriging at a single cut-off, 0.3 g/t Au. Indicator estimation was into block models with parent cell dimensions of 5 m E × 2.5 m N × 2.5 m RL. Two domains were adopted, one to capture the minor mineralisation in the black shales and the other to cover the mafic/ultramafic and felsic sediments. Grade shell triangulations were then generated by constraining the block model at a 25% probability cut-off (Figure 14.17).

Indicator variogram parameters are presented below in Table 14.26 and the variogram in Figure 14.19.

Table 14.26 Ikkari Gold Deposit, Indicator Variogram Parameters IK Estimate

Variable	C°	Axis Orientation (°)			Structure 1				Structure 2			
		Bearing	Plunge	Dip	C1	Major	Semi-Major	Minor	C2	Major	Semi-Major	Minor
Au 0.3ppm cut-off	0.052	065	-25	90	0.073	50	40	11	0.079	250	200	60

Figure 14.19 Ikkari Gold Deposit, Indicator Variogram 0.3 g/t Au

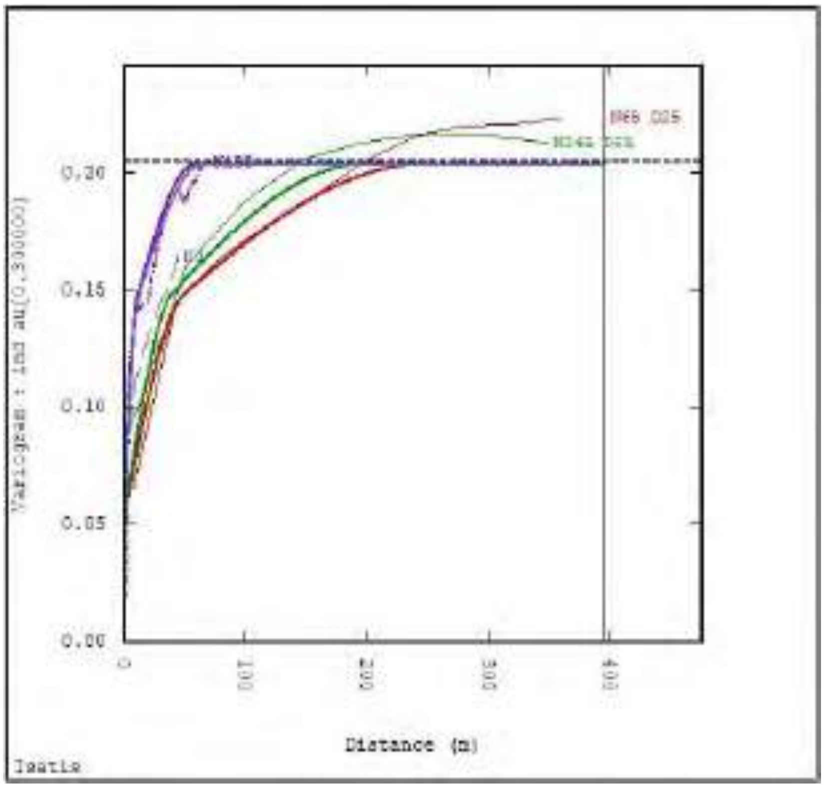


Table 14.27 details the indicator estimate sample search parameters. Estimate search axis orientations were rotated to match the overall average mineralisation geometry and the variogram for the black shale was adopted from the main mafic/ultramafic/felsic sediment domain and was rotated to match the shale orientation.

Table 14.27 Ikkari Gold Deposit, Sample Search Parameters MIK Estimate

Domain	Estimation Pass	Axis Orientation (°)			Search Distance (m)			Min No of Comp.	Max No of Comp,	Max No of Comp per DH	Discretisation
		Bearing	Plunge	Dip	X	Y	Z				
Main felsic / mafic / ultramafic	1	065	-25	90	250	200	60	12	24	4	2*2*2
Black shale	1	065	-25	55	150	100	40	12	12	4	2*2*2

The probability cut-off may be considered somewhat subjective and may seem arbitrary, however was selected based on extensive review of a range of probability cut-offs. The selected probability shell is considered optimal to capture the observed continuity and tenor of mineralisation while excluding obvious low-grade material. Grade shells were reviewed in multiple orientations and in plan and section views prior to being accepted for grade estimation and block modelling purposes.

Mineralisation estimation domains were thus defined with further sub-division being differentiated on the basis that a minor amount of mineralisation is hosted in the black shale lithology to the hanging wall side of the northern thrust that bounds one side of the main mineralised package. This mineralisation was determined to be different in orientation and tenor to the main body of mineralisation as described earlier. The black shale domain is denoted Zone 100 and the main mineralisation Zone 200. A typical section demonstrating the black shale hanging wall and the constraining grade shell outline is presented in Figure 14.20. A plan and isometric view of the mineralisation wireframe is presented in Figure 14.21 and Figure 14.22 respectively.

Figure 14.20 Ikkari Gold Deposit, Typical Cross Section (Looking East)

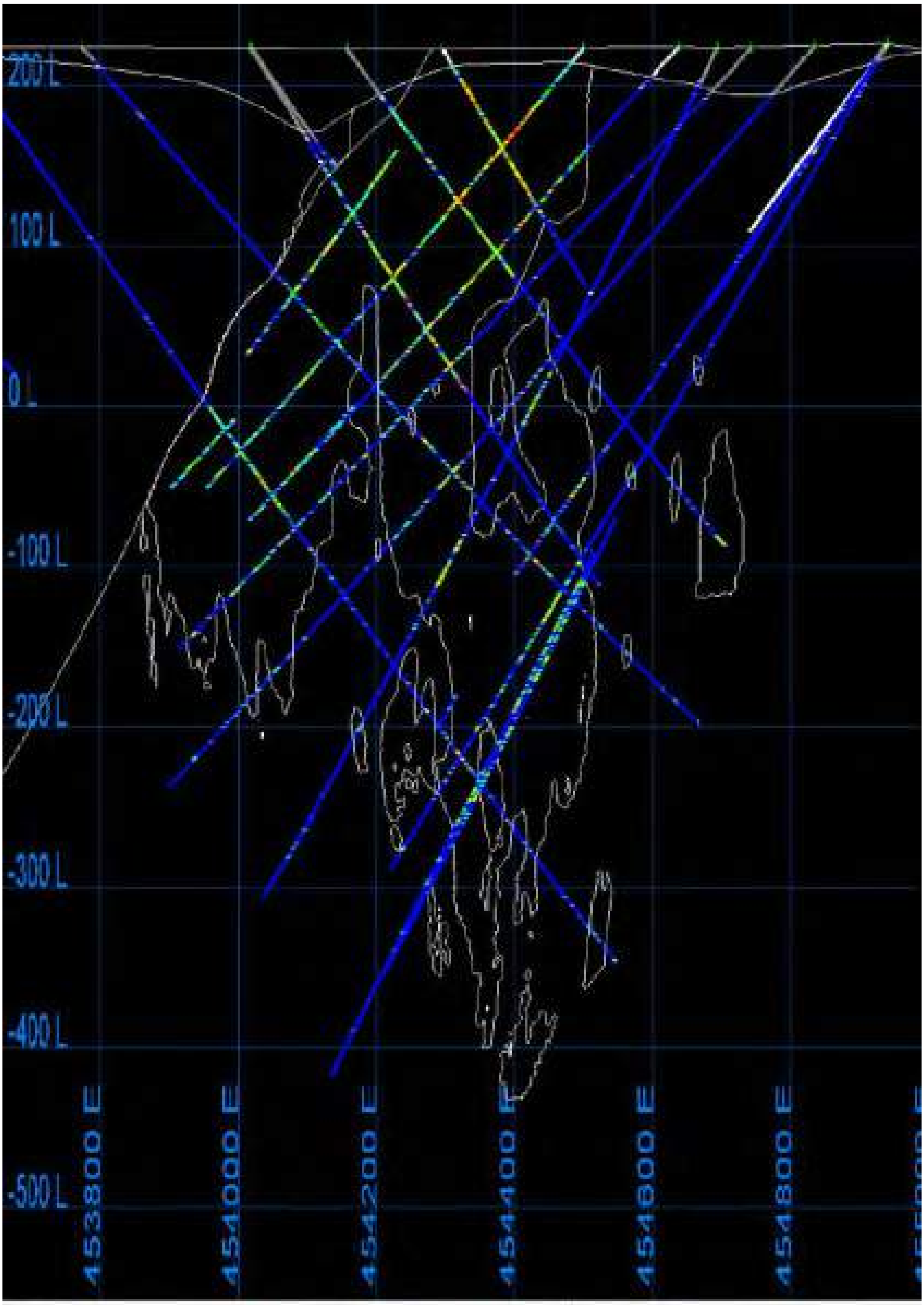


Figure 14.21 Ikkari Gold Deposit, Estimation Domains Plan View

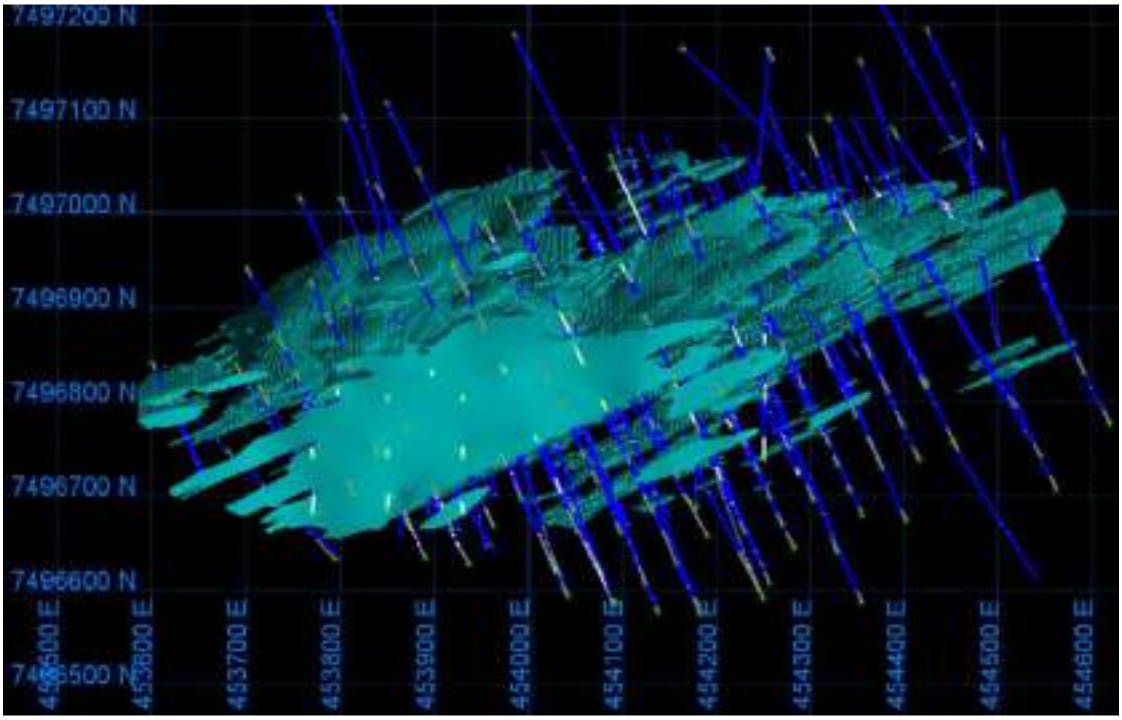
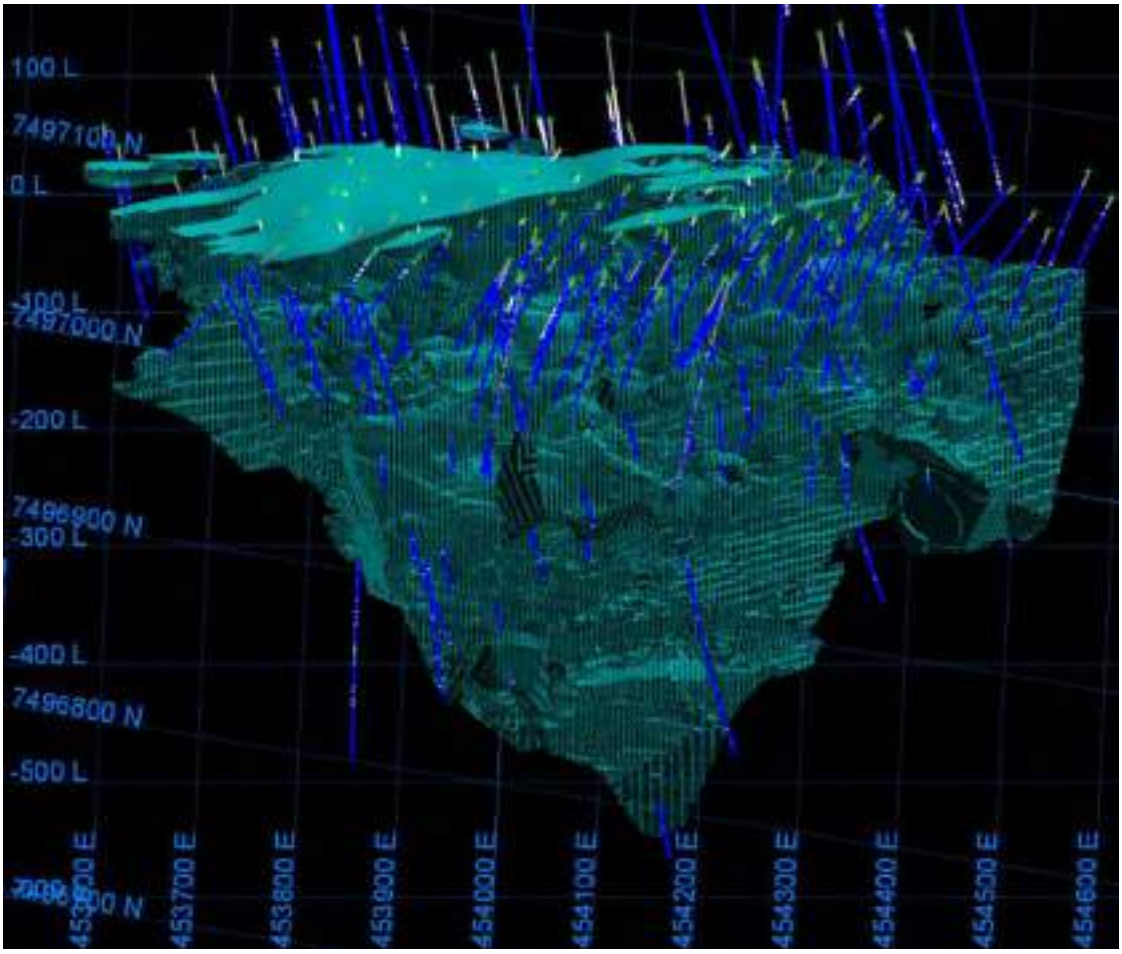


Figure 14.22 Ikkari Gold Deposit, Estimation Domains Isometric SE View



14.2.4 DATA FLAGGING AND COMPOSITING

Drill hole samples were flagged with the relevant indicator grade shell, lithological wireframes and topographical surfaces. Coding was undertaken on the basis that if the individual sample centroid fell within the grade shell boundary it was coded as within the grade shell. Each sub-domain has been assigned a unique numerical code to allow the application of hard boundary domaining if required during grade estimation.

The drill hole database coded within each grade shell or mineralisation wireframe was then composited as a means of achieving a uniform sample support. Further discussion regarding sample support equalisation is provided in Section 14.1.4.

After consideration of relevant factors relating to geological setting and mining, including likely mining selectivity and bench/flitch height, a regular 3 m run length (downhole) composite was selected as the most appropriate composite interval to equalise the sample support at the Ikkari Gold Deposit. Compositing was broken when the routine encountered a change in flagging (grade shell boundary) and composites with residual intervals of less than 3 m were retained in the composite file.

14.2.5 STATISTICAL ANALYSIS

SUMMARY STATISTICS

The composites flagged as described in the previous section were used for subsequent statistical, geostatistical and grade estimation investigations.

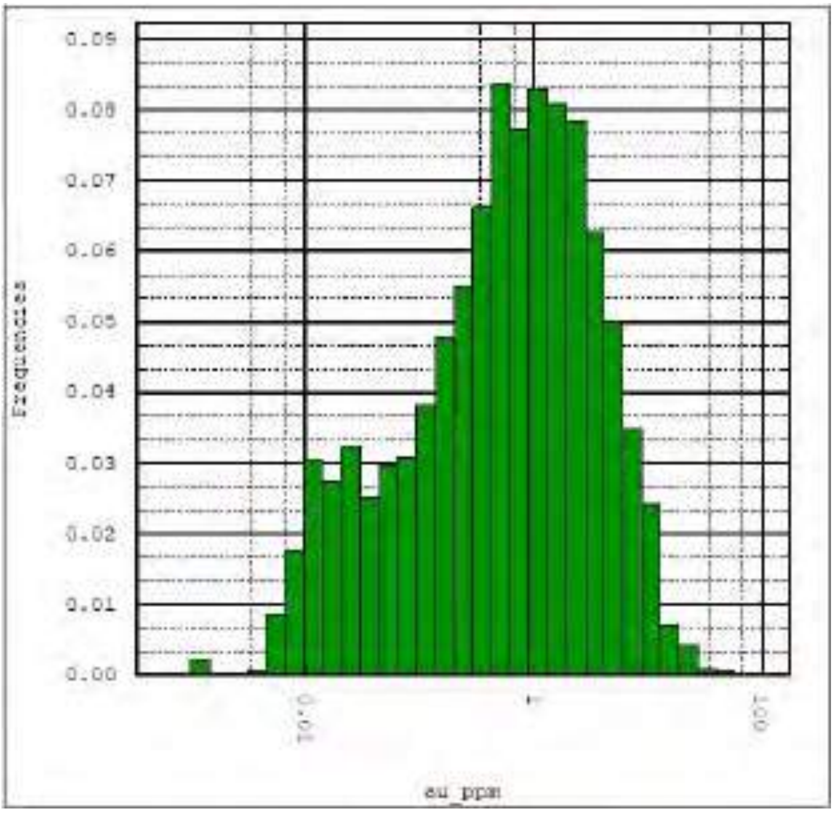
Three metre composite summary statistics for gold within the grade shell and subdivided by lithology are presented in Table 14.28. It is evident that the contribution to total metal by the black shale is minimal (<1%) and that the statistics for the felsic and mafic/ultramafic domains are more similar. The three lithological domains have therefore been combined for all further purposes of statistical and geostatistical analysis.

Table 14.28 Summary Statistics Low Grade Domains for Three Metre Composites of Uncut Gold Grade 9g/t)

Domain	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
Black shale	153	0.007	8.0630	0.648	1.041	1.084	1.606
Felsic sediment	2,088	0.005	22.930	1.410	2.032	4.127	1.440
Mafic/ultramafic	4,995	0.002	108.239	1.820	3.862	14.916	2.122

Summary descriptive statistics were generated for the combined lithological domains (Table 14.29). The grade distribution is reasonably typical for gold deposits of this style and shows a positive skew or near lognormal behaviour (Figure 14.23). The histogram indicates a bimodal distribution with a potential subordinate low grade population present. The coefficient of variation (CV - calculated by dividing the standard deviation by the mean grade) is only moderate, indicating any potential high outlier grades that do not significantly contribute to the total metal.

Figure 14.23 Log Histogram of Uncut Gold Grades



HIGH GRADE OUTLIER ANALYSIS

A high grade outlier analysis has been undertaken for the three m composite gold grades. A comparison analysis was also undertaken on the raw samples however negligible differences were observed and all further statistical analysis relates to the three m composites. The effects of the highest-grade composites on the mean grade and standard deviation of the gold dataset for each of the estimation domains have been investigated by compiling and reviewing statistical plots (histograms and probability plots). The resultant plots were reviewed together with probability plots of the sample populations and an upper cut for each dataset was chosen coinciding with a pronounced inflection or increase in the variance of the data. An upper cut was chosen at 30 g/t Au however this is considered extremely minimal as it only affects 4 composites and has resulted in a 0.8% reduction in mean grade. Further analysis of top cut variability indicates a 1.5% reduction in the mean if a top cut is performed prior to compositing. Top cut statistics are presented in Table 14.29.

Composite data was viewed in 3D to determine the clustering or otherwise of these highest grades observed in each domain to assess the appropriateness of the high-grade cut. Clustering of the highest grades in one or more areas may indicate that the grades do not require cutting.

CELL DECLUSTERING ANALYSIS

Visual inspection of the available datasets for each of the estimation domains indicated some clustering of the data within higher grade regions of the deposit. Data clustering often occurs when drilling campaigns selectively target higher grade regions of the deposit, resulting in an artificially high mean grade in many cases. Declustering was therefore completed to remove any effects of preferential sampling of high grade areas that may have occurred.

Cell declustering was completed with weights w(i) associated with composite (i) determined as w(i) = mv/n(i) where mv is the mean of all the samples within a moving window centred on composite (i) and n(i) is the number of samples within the moving window. This normalisation allows the weights not to decrease with the number of data.

Declustered composite statistics are presented in Table 14.39. As expected, the declustered mean grades are significantly less than the raw composite mean grades due to the data configuration issues discussed above.

Table 14.29 Summary Statistics Three Metre Composites Gold Grade (g/t)

Domain	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
Au ppm	7220	0.002	108.239	**1.679**	3.404	11.586	2.027
Au ppm cut	7220	0.002	30.000	**1.650**	2.946	8.68	1.785
Au ppm, declustered	7220	0.002	108.239	**1.430**	3.148	9.907	2.20
Au ppm cut, declustered	7220	0.002	30.000	**1.404**	2.736	7.486	1.949

Multiple Indicator Kriging Cut-offs and Indicator Class Statistics

Indicator Kriging cut-offs or indicator bins were selected for estimation by MIK. Cut-offs were based upon population distributions and metal proportions above and below the mean composite value of the proposed cut-off bins. Conditional statistics for data within each domain grouping to be estimated by Multiple Indicator Kriging are listed in Table 14.30. A total of 17 cut-offs were applied to each Domain Group for estimation via MIK. Top cuts have not been applied for the purposes of conditional statistics calculation.

Table 14.30 Ikkari Gold Deposit, Indicator Class Statistics

High Grade Group		
Grade Threshold (Au g/t)	Probability Threshold	Class Mean (Au g/t)
0.08	0.188	0.0299
0.20	0.300	0.1327
0.35	0.397	0.2676
0.50	0.466	0.4207
0.65	0.527	0.5711
0.86	0.592	0.7462
1.10	0.647	0.9730
1.35	0.689	1.2201
1.65	0.730	1.4876
2.00	0.767	1.8186
2.35	0.798	2.1769
2.85	0.834	2.5960
3.50	0.867	3.1658
4.30	0.896	3.8732
5.40	0.924	4.7999
7.60	0.955	6.3988

11.00	0.981	9.0733
Max	Max	17.9583

14.2.6 VARIOGRAPHY

INTRODUCTION

Discussion on variography is outlined in Section 14.1.6.

IKKARI VARIOGRAPHY

Grade and indicator variography was generated to enable grade estimation via MIK and change of support analysis to be completed. In addition, Gaussian variograms were also examined as part of the change of support process. Indicator thresholds for domain groups to be estimated via MIK had variograms modelled with every third variogram typically modelled. Variograms not modelled have had their parameters interpolated based on the bounding modelled variograms.

Interpreted anisotropy directions correspond well with the modelled geology and overall geometry of the interpreted domain. All grade variography has been based on the back-transformed Gaussian variograms. A common feature of all the grade variography is the relatively long overall ranges, especially for the second modelled structure however the dominance of the overall variance is by the nugget and the first sill. This outcome is reflective of the tightening of the drill spacing to approximately 40 m spaced sections or better.

Indicator variography for the MIK estimation domain is presented in Table 14.31. The modelled grade variogram is presented in Figure 14.24.

Figure 14.24 Ikkari Gold Deposit, Grade Variogram

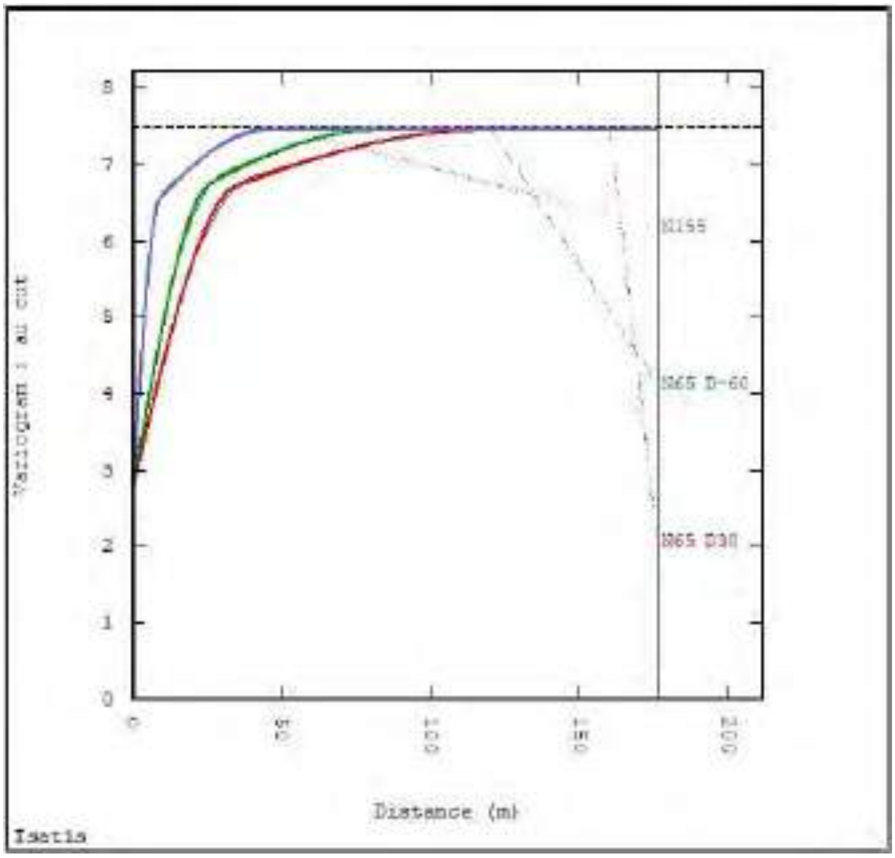


Table 14.31 Ikkari Gold Deposit, Indicator Variogram Models Au g/t

Grade Variable or Indicator Threshold	Nugget (C0)	Rotation (dip→dip dir)			Structure 1				Structure 2			
		Major	Semi Major	Minor	Relative Sill 1 (C1)	Range (m)			Relative Sill 2 (C2)	Range (m)		
						Major	Semi Major	Minor		Major	Semi Major	Minor
Grade Variogram	2.79	-30→065	60→245	0→155	3.37	34	26	9	1.29	119	82	45
0.08(1)	0.0324	-30→065	60→245	0→155	0.0906	50	40	10	0.0311	200	150	50
0.20(1)	0.0441	-30→065	60→245	0→155	0.1235	50	40	10	0.0423	200	150	50
0.35	0.0500	-30→065	60→245	0→155	**0.1400**	**50**	**40**	**10**	**0.0480**	**200**	**150**	**50**
0.50(2)	0.0554	-30→065	60→245	0→155	0.1456	47	40	10	0.0479	193	140	47
0.65(2)	0.0586	-30→065	60→245	0→155	0.1450	43	40	10	0.0456	187	130	43
0.86	0.0600	-30→065	60→245	0→155	**0.1400**	**40**	**40**	**10**	**0.0420**	**180**	**120**	**40**
1.10(3)	0.0625	-30→065	60→245	0→155	0.1272	38	38	9	0.0386	163	113	35
1.35(3)	0.0641	-30→065	60→245	0→155	0.1148	37	37	9	0.0352	147	107	30
1.65	0.0650	-30→065	60→245	0→155	**0.1030**	**35**	**35**	**8**	**0.0320**	**130**	**100**	**25**
2.00(4)	0.0608	-30→065	60→245	0→155	0.0895	35	35	8	0.0283	123	97	22
2.35(4)	0.0574	-30→065	60→245	0→155	0.0785	35	35	8	0.0253	117	93	18
2.85	0.0550	-30→065	60→245	0→155	**0.0700**	**35**	**35**	**8**	**0.0230**	**110**	**90**	**15**
3.5(5)	0.0441	-30→065	60→245	0→155	0.0533	35	35	8	0.0178	107	87	14
4.3(5)	0.0366	-30→065	60→245	0→155	0.0421	35	35	7	0.0142	103	83	13
5.4	0.0320	-30→065	60→245	0→155	**0.0350**	**35**	**35**	**7**	**0.0120**	**100**	**80**	**12**
7.6(6)	0.0173	-30→065	60→245	0→155	0.0189	35	35	7	0.0065	100	80	12
11.0(6)	0.0076	-30→065	60→245	0→155	0.0083	35	35	7	0.0028	100	80	12

Note:
1) Assumed model based on 0.35 Au g/t variogram model.
2) Assumed model based on 0.35 Au g/t and 0.86 Au g/t variogram models.
3) Assumed model based on 0.86 Au g/t and 1.65 Au g/t variogram models.
4) Assumed model based on 1.65 Au g/t and 2.85 Au g/t variogram model.
5) Assumed model based on 2.85 Au g/t and 5.4 Au g/t variogram model.
6) Assumed model based on 5.4 Au g/t variogram model.

14.2.7 Block Modelling

A 3D block model was created in the ETRS89 (European Terrestrial Reference System) grid using Vulcan mining software. The parent block size was selected on the basis of the average drill spacing together with consideration of potential mining parameters. A parent cell size of 40 m E by 20 m N by 20 m RL which was sub-blocked down to 10 m E by 5 m N by 5 m RL (to ensure adequate volume representation). The models covered all the interpreted mineralisation zones and included suitable additional waste material to allow later mining engineering studies. Block coding was completed on the basis of the block centroid, wherein a centroid falling within any wireframe was coded with the wireframe solid attribute. The block model is unrotated.

The main block model parameters are summarised below in Table 14.32. Variables were coded into the block models to enable MIK and OK estimation and subsequent MIK change of support and grade tonnage reporting. A visual review of the wireframe solids and the block model indicated correct flagging of the block model. Additionally, a check was made of coded volume versus wireframe volume which confirmed the above.

Table 14.32 Ikkari Gold Deposit, Block Model Parameters

	Northing (Y)	Easting (X)	RL (Z)
Min Coordinates	7,496,350	453,480	-600
Extent	1,080	1,400	860
Block size (m)	20	40	20
Sub Block size (m)	5	10	5
Rotation (° around axis)	0	0	90

14.2.8 Bulk Density Data

A dry bulk density database has been supplied containing a total of 7,161 data. All density measurements have been systematically taken by Rupert Resources as part of their ongoing core processing operations.

Rupert Resources has calculated dry bulk densities on the basis of the weight in water method. Density readings have been taken on whole drill core and are distributed across all areas of the deposit. Based on the geological model, statistical analysis demonstrates that across the deposit, different lithologies have different densities. Summary statistics subdivided by lithology are presented in Table 14.33. Negligible differences have been noted per lithology based on the sub-division into mineralised and unmineralised portions and therefore bulk densities have been applied to the block model based on mean grades per lithology. Additionally, overburden has been applied an arbitrary valued of 1.8 t/m^3 as no direct readings are available.

Table 14.33 Ikkari Gold Deposit, Density Statistics

Company	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
Gabbro	324	2.18	3.74	2.835	0.172	0.03	0.061
Black shale	378	2.01	3.89	2.766	0.158	0.025	0.057
Felsic sediments	359	2.08	6.82	2.903	0.243	0.059	0.084
Mafic/ultramafic	5,989	1.79	5.69	2.874	0.133	0.018	0.046
Quartzite	111	2.55	3.83	2.713	0.122	0.015	0.045

14.2.9 Grade Estimation

Introduction

MIK was applied to grade estimation at the Ikkari deposit, within the defined indicator mineralisation shell. The minor domains forming a low grade halo to the main mineralised domains were estimated via OK. Estimation was completed in the mining package Vulcan using the GSLib geostatistical software while geostatistical change of support parameters were developed in Isatis geostatistical software. MIK is considered a robust estimation methodology for grade estimates for gold deposits such as Ikkari where high levels of short scale variability are present. MIK grade estimation with change of support has been applied to produce 'recoverable' gold estimates targeting a SMU of 10 m E x 5 m N x 5 m RL.

The Multiple Indicator Kriging Method

The MIK technique has been described in in Section 14.1.9.

Post MIK Processing - E-Type Estimates

MIK post processing is described in Section 14.1.9.

Support Correction (Selective Mining Unit Estimation)

Support correction is described in Section 14.1.9.

Indirect Lognormal Correction

Indirect Lognormal Correction is described in Section 14.1.9.

Multiple Indicator Kriging Parameters

MIK estimates were completed using the indicator variogram models (Section 14.1.6), and a set of ancillary parameters controlling the source and selection of composite data. The sample search parameters were defined based on the variography and the data spacing, and a series of sample search tests performed in Isatis geostatistical software. A total of 17 indicator thresholds were estimated for all estimation domains (Table 14.30).

The sample search parameters for the MIK estimations are provided in Table 14.34. A combination of soft domain boundaries was used for the estimation throughout to reflect continuity between domains or otherwise. A three-pass estimation strategy (where required) was applied to each domain, applying a progressively expanded and less restrictive sample search to the successive estimation pass, and only considering blocks not previously assigned an estimate. Parent cell estimations (40 m E by 20 m N by 20 m RL) were applied throughout and discretisation was applied on the basis of 3 X by 3 Y by 2 RL for 18 discretisation points per block.

Table 14.34 Ikkari Gold Deposit, MIK Sample Search Criteria

Domain	Pass	Sample Search Orientation (dip/dip direction˚)			Sample Search Distance (m)			Numbers of 3m Composites			% Blocks Estimated
		Major	Semi Major	Minor	Major	Semi Major	Minor	Min.	Max.	Max Per Drill hole	
100	Pass 1	0→65	50→335	40→1555	300	240	90	24	36	6	97
	Pass 2	0→65	50→335	40→1555	900	720	270	24	36	6	100
200	Pass 1	30→65	60→245	0→155	125	100	40	24	36	6	86
	Pass 2	30→65	60→245	0→155	300	240	90	24	36	6	100

Change of Support

Applying the modelled variography, variance adjustment factors were calculated for to emulate a 10 m E x 5 m N x 5 m RL SMU via the indirect lognormal change of support. The intra-class composite mean grades were used in calculating the whole block and SMU grades. The change of support study also included the calculation of the theoretical global change of support via the discrete Gaussian change of support model.

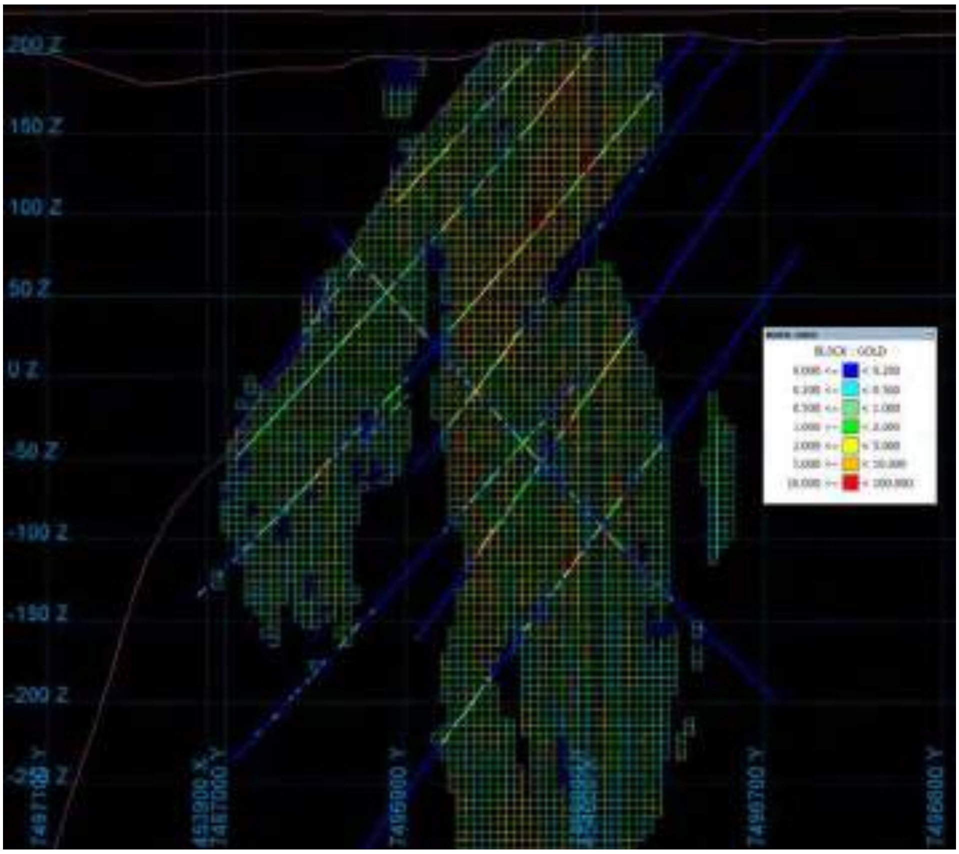
An ‘information effect’ factor is commonly applied to the originally derived panel-to-block variance ratios to determine the final variance adjustment ratio. The goal of incorporating information effect is to calculate results taking into account that mining takes place based on grade control information. There will still be a quantifiable error associated with this data and it is this error we want to incorporate. This is achieved in practice by running a test kriging estimation of an SMU using grade control data (the results required to incorporate this option in the change of support do not depend on the assay data so the grade control data can be hypothetical). The incorporation of the information effect is commonly found to be negligible, however can have a significant effect in some cases. In this case, the information effect factor was found to have a minor effect and has been incorporated in the calculation.

The variance adjustment ratios as applied to all mineralised domains was 0.15.

Grade Localisation

MIK grade estimates are generated in large blocks or panels (in the case of Ikkari, 40 m E x 20 m N x 20 m RL) and are inherently not intuitive to review. Post processing of these MIK estimates aims to simplify the presentation by producing a single SMU dimension block grade where the distribution of the grades in the panel matches that of the distribution in the SMU’s. The MIK panel grades have been localised to SMU dimension blocks in Isatis software. The SMU dimension was 10 m E x 5 m N x 5 m RL. Validation of the results indicates a near identical distribution and the resultant model has been accepted. A typical section is presented below (Figure 14.25).

Figure 14.25 Typical Sectional View Displaying Localised Au Grades



Estimate Validation

All relevant statistical information was recorded to enable validation and review of the MIK estimates. The recorded information included:

- Number of samples used per block estimate.
- Number of drill holes from which samples selected.
- Average distance to samples per block estimate and distance to nearest sample.
- Estimation flag to determine in which estimation pass a block was estimated.

Number of drill holes from which composite data were used to complete the block estimate.

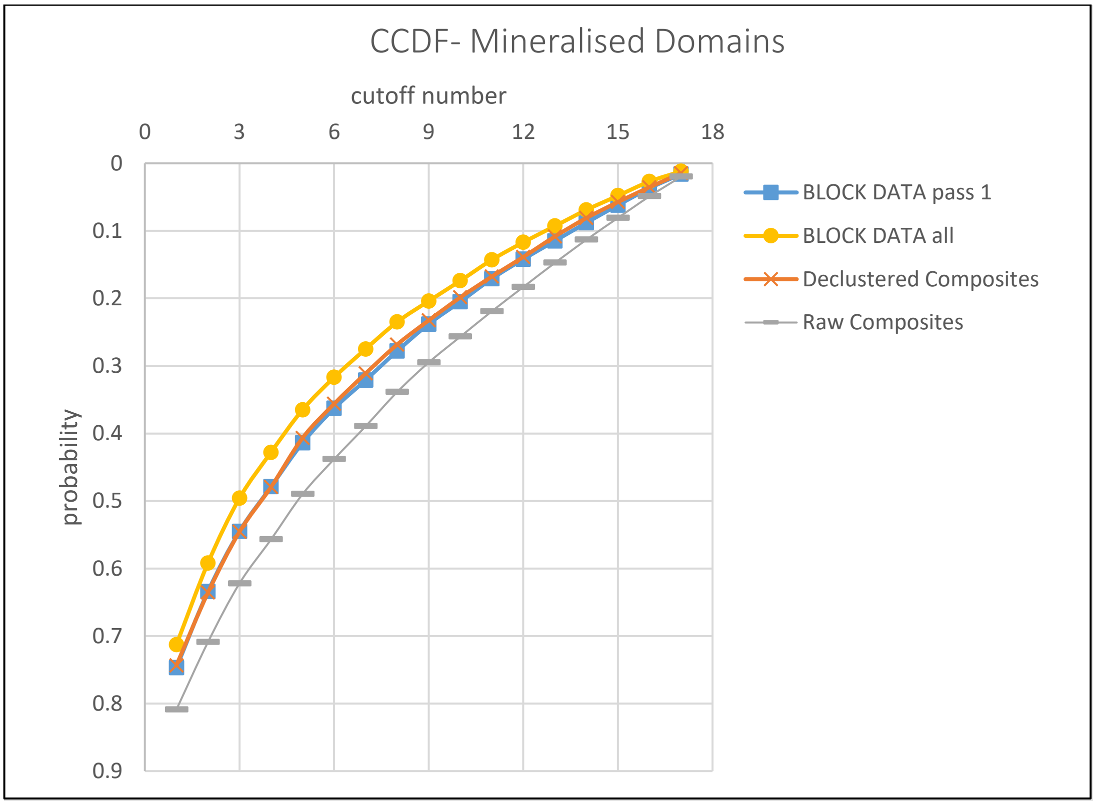
The estimates were reviewed visually and statistically prior to being accepted. The review included the following activities:

- Comparison of the E-type estimate versus the mean of the composite dataset, including weighting where appropriate to account for data clustering.

- Comparison of the reconstituted cumulative conditional distribution functions of the estimated blocks (indicator kriging) versus the input composite data (Figure 14.26).
- Production of swath plots comparing input composite grades versus block grades (Figure 14.27).
- Visual checks of cross sections, long sections, and plans.

Alternate estimates including OK estimates into varying parent cell size blocks.

Figure 14.26 CCDF Validation



Alternative MIK estimates were also completed to test the sensitivity of the reported model to the selected MIK interpolation parameters. An insignificant amount of variation in overall grade was noted in the case of the alternative estimates, with comparable mean block grades and a negligible change from the accepted MIK grade estimate.

Finally, alternative estimation methodologies of ordinary kriging and inverse distance squared were completed on a like for like basis to examine for variance between the methods. Whilst comparable whole block grades were achieved, at higher cut-off grades the reported tonnes and grades were biased high and low respectively. This is due to the ability of the change of support to emulate a practical mining selectivity dimension while the ID^2 and OK methods can only enable reporting at the parent cell dimension at which the estimate is undertaken.

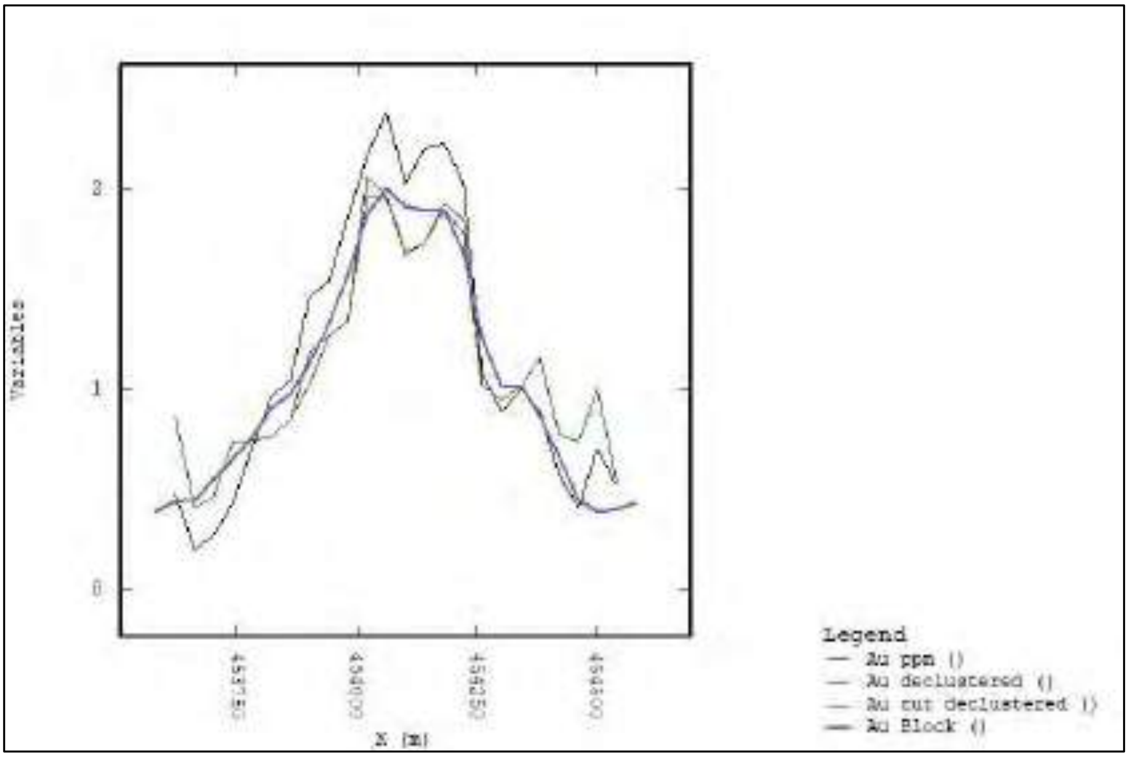
Validation of localised block Au grades has been undertaken by comparing the block mean grades with the relevant composite mean grades (Table 14.35).

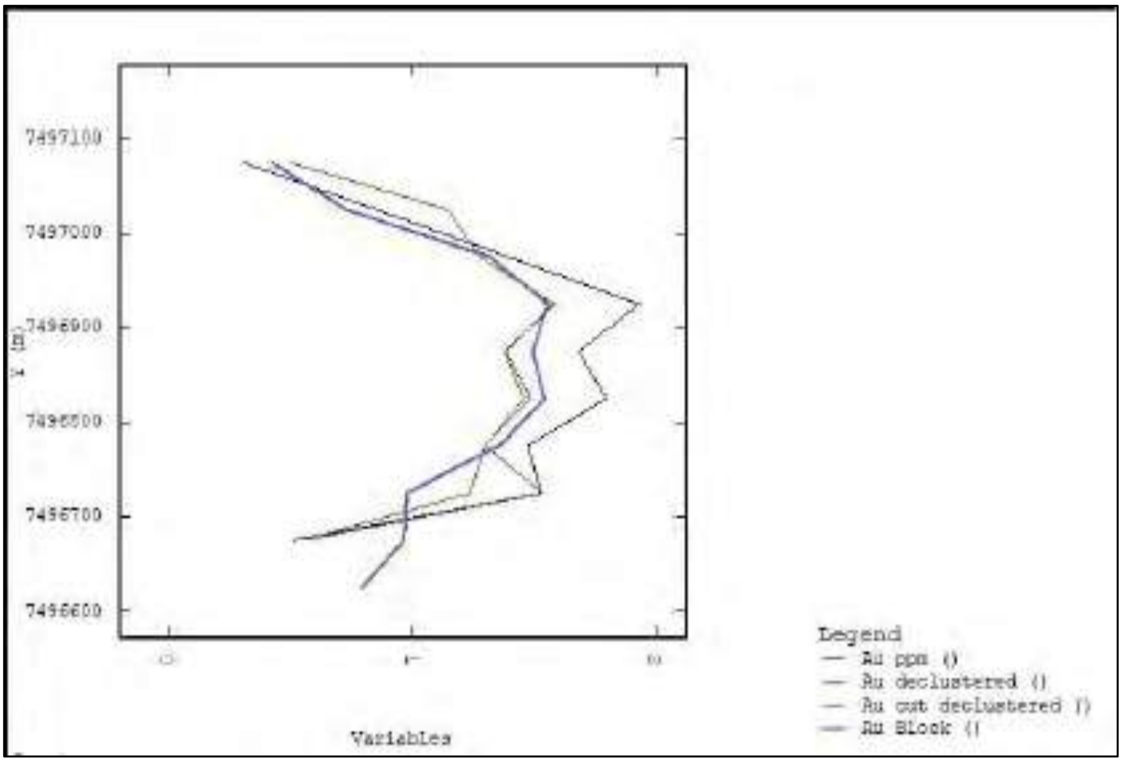
Table 14.35 Ikkari Gold Deposit, Comparison of Block Grades with Composite Mean Grades, All Data Used

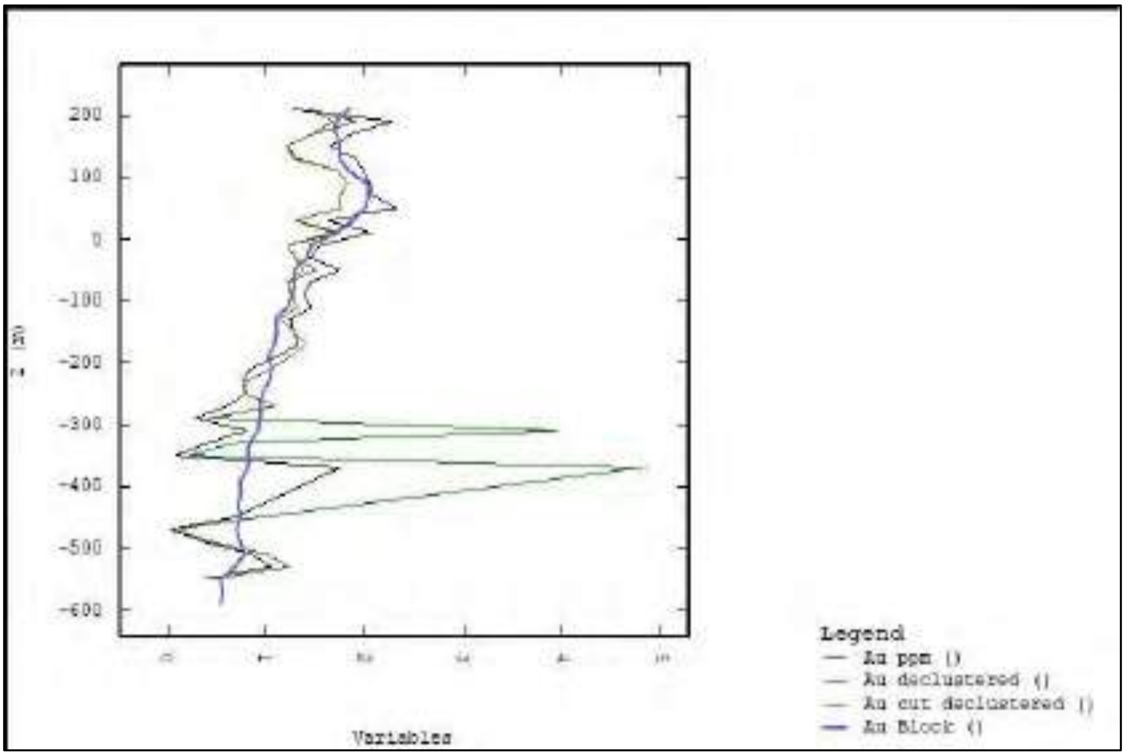
Zone	All Composites, (declustered, capped)	All Composites, (declustered, uncapped)	All Composites, (non-declustered capped)	Block Model Grades	% Diff Block Model vs Declustered Mean
100	1.404	1.430	1.650	1.432	2.0%

A good correlation may be drawn between the declustered composite mean grades and the block model mean grade.

Figure 14.27 Comparison of Swath Plot Grades for Input Composites







14.2.10 Depletion for Mining Activity

No mining activity has taken place at Ikkari therefore no depletion is applicable.

14.2.11 RESOURCE CLASSIFICATION

The resource categorisation was based on the robustness of the various data sources available, including:

- Geological knowledge and interpretation.
- Variogram models and the ranges of the first structure in multi-structure models.
- Drilling density and orientation.
- Estimation quality statistics.

The resource estimates for the Ikkari Gold Deposit have been classified as Inferred Mineral Resources based on the confidence levels of the key criteria as presented in Table 14.35.

Table 14.36 Ikkari Gold Deposit, Confidence Levels by Key Criteria

Items	Discussion	Confidence
Drilling Techniques	Diamond drilling- Industry Standard approach.	High
Logging	Standard nomenclature has been adopted.	High/Moderate
Drill Sample Recovery	Recoveries are not recorded in entire database but diamond core recoveries assumed acceptable.	High/Moderate
Sub-sampling Techniques and Sample Preparation	Diamond sampling conducted by industry standard techniques.	High
Quality of Assay Data	Appropriate quality control procedures available for work completed by Rupert. They were reviewed and considered to be of industry standard.	Moderate/High
Verification of Sampling and Assaying	Sampling and assaying procedures have been assessed and are considered of appropriate industry standards.	Moderate
Location of Sampling Points	Survey of all collars conducted with accurate survey equipment. Investigation of downhole survey indicates appropriate behaviours.	Moderate/High
Data Density and Distribution	Majority of regions defined at a minimum on a notional 80 m E x 40 m N drill spacing.	Moderate
Audits or Reviews	N/A	
Database Integrity	Industry standard approach applied by Rupert.	Moderate
Geological Interpretation	Mineralisation controls are moderately well understood. The mineralisation constraints are robust but relatively broad and therefore of moderate confidence. Controls at a local scale commonly uncertain continuity	Moderate
Estimation and Modelling Techniques	Multiple Indicator Kriging is considered to be appropriate given the geological setting and grade distribution.	High
Cut-off Grades	MIK is independent of cut-off grade although the mineralisation constraints were based on a notional 0.3 g/t Au lower cut-off grade. A 0.5 g/t lower cut-off grade is considered appropriate for reporting within a potential open pit and 1.0 g/t lower cut-off grade is considered appropriate for mineralisation that would be mined using underground methods.	Moderate/High
Mining Factors or Assumptions	A 10 m E x 5 m N x 5 m RL SMU emulated for gold. Open pit mining assumed and SMU is conditional on scale assumed.	Moderate

	Change of support for Inferred has higher degree of uncertainty due to lack of appropriate close spaced data.	
Metallurgical Factors or Assumptions	Not applied.	N/A
Tonnage Factors (In-situ Bulk Densities)	Sufficient data exists to enable high confidence in the applied density values.	High

14.2.12 Resource Reporting

The summary total Mineral Resources for the Ikkari Gold Project are provided in Table 14.37 along with the mineral resource estimate at a range of additional cut-off grades to demonstrate grade tonnage relationships at higher and lower cut-off grades. Values in tonnes and ounces are rounded to two significant figures which may cause discrepancies between rows in the table.

The Mineral Resource is reported both within a designed open pit and as a potential underground operation outside that. The preferred lower cut-off for reporting is 0.5 g/t for the portion potentially mineable by open pit methods and 1.0 g/t Au for the portion potentially extractable by underground methods.

The designed pit and 0.5 g/t cut-off grade are based on the following economic parameters: Gold price $1,650 /oz, 95% mining recovery and 95% Au recovery. Open pit mining costs at $2.5 /t, process costs at $11.3 /t and $4 /t other costs including co-disposal, water treatment and closure costs. G&A including refining and royalties at $3.2 /t. The calculated cut-off grade is rounded up to 0.5 g/t for reporting.

At underground mining costs of $21.8 /t and 92% mining recovery, based on mining by sub level caving, the calculated underground cut-off grade is rounded up to 1.0 g/t as the resource is not constrained within mineable shapes.

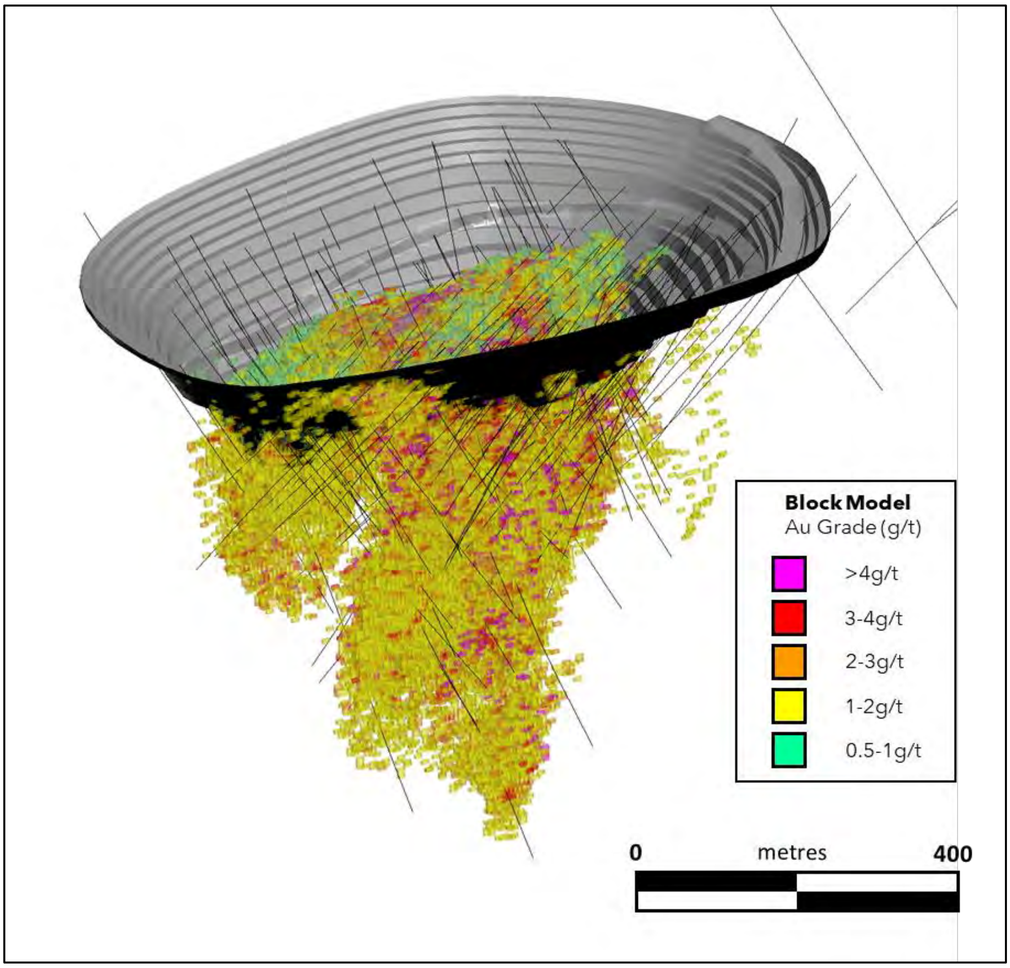
Table 14.37 Ikkari Gold Deposit, Mineral Resource Report, Summary Grade Tonnage Report

Resource Category		Lower Cut-off Grade (g/t Au)	Tonnes (t)	Average Grade (g/t Au)	Gold Metal (oz)	Gold Metal (Kg)
Indicated	Open Pit	0.3	33,300,000	2.3	2,440,000	75,900
		0.4	31,700,000	2.4	2,420,000	75,300
		0.5	**30,000,000**	**2.5**	**2,400,000**	**74,500**
		0.6	28,100,000	2.6	2,360,000	73,500
		0.7	26,400,000	2.7	2,330,000	72,400
Indicated	Underground	0.6	24,700,000	1.9	1,490,000	46,200
		0.8	19,900,000	2.2	1,380,000	42,900
		1.0	**16,500,000**	**2.4**	**1,280,000**	**39,800**
		1.2	13,900,000	2.7	1,190,000	37,000
Indicated Open Pit and Underground			**46,400,000**	**2.5**	**3,680,000**	**114,300**
Inferred	Open Pit	0.3	3,900,000	1.3	160,000	5,100

		0.4	3,500,000	1.4	160,000	5,000
		0.5	**3,100,000**	**1.5**	**150,000**	**4,800**
		0.6	2,700,000	1.7	150,000	4,600
		0.7	2,400,000	1.8	140,000	4,300
Inferred	Underground	0.6	14,900,000	1.5	710,000	22,000
		0.8	11,100,000	1.7	620,000	19,300
		1.0	**8,700,000**	**2.0**	**550,000**	**17,200**
		1.2	6,800,000	2.2	490,000	15,100
Inferred Open Pit and Underground			**11,800,000**	**1.9**	**710,000**	**22,000**

Note: Appropriate rounding has been applied.

Figure 14.28 Isometric View of Estimated Blocks Within Pit Shell looking NE with a Plunge of 20°. Blocks are Shown Above the Relevant Cut-offs, Open Pit 0.5 g/t Au and Underground Above a 1 g/t Au



14.3 HEINÄ CENTRAL

14.3.1 INTRODUCTION

This Mineral Resource for the Heinä Central Gold Deposit has been estimated as at the effective date of the 28th November 2022. Gold and copper grade estimation was completed using Ordinary Kriging (OK) for the mineralised domains. OK grade estimates have been estimated to an SMU dimension block size to maintain local grade characteristics and reduce grade smearing. This estimation approach was considered appropriate based on review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralisation, and the style, geometry and tenor of mineralisation. The estimation was constrained with geological and mineralisation interpretations.

14.3.2 Database Validation

The resource estimation was based on the available exploration drillhole database which was compiled in-house by Rupert. The database has been reviewed and validated prior to commencing the resource estimation study.

The database consists of solely of surface diamond drilling. Database statistics are provided below as Table 14.38. A plan view of all drilling is presented in Figure 14.29.

Table 14.38 Summary of Available Drill Date for Heinä Central

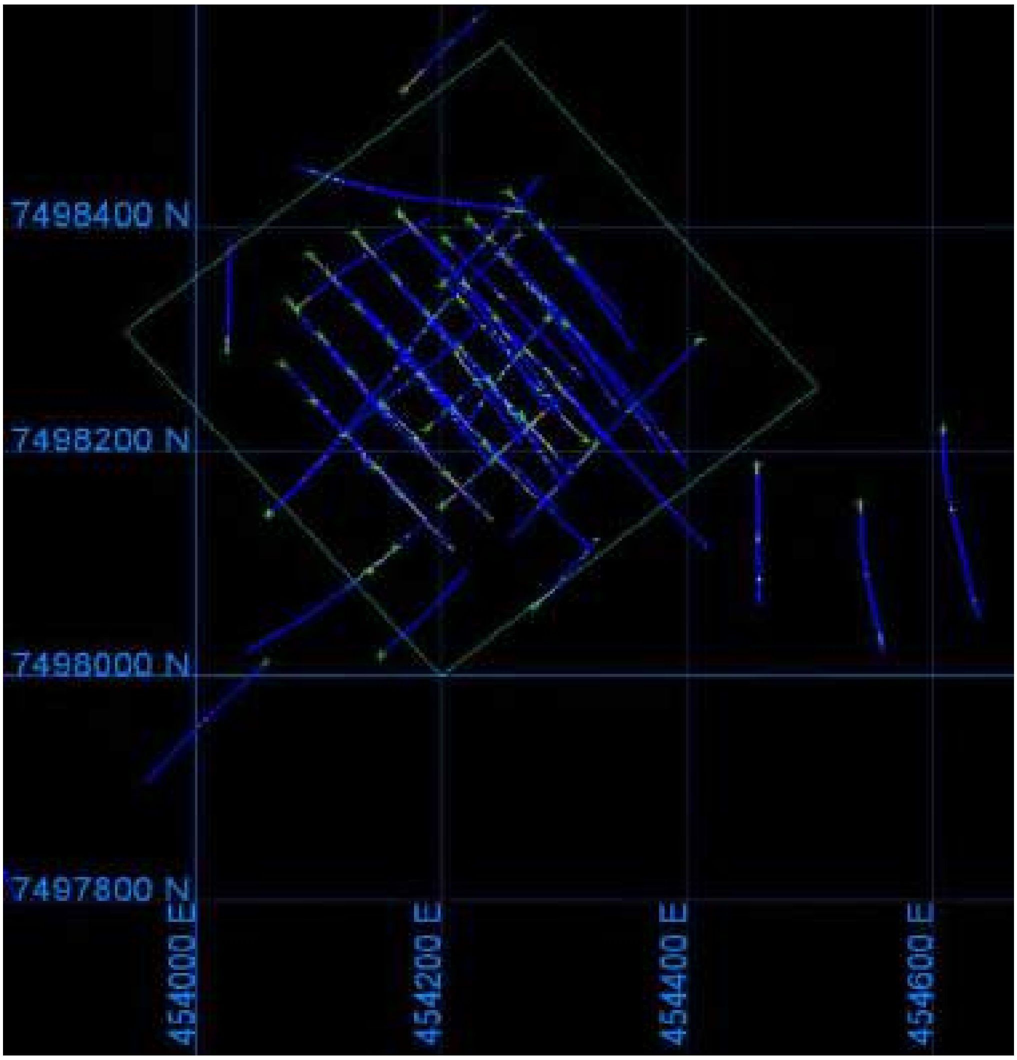
Company	DH Type	Holes	Metres	% of Total
Rupert Resources (April 2019 to April 2022)	Diamond (NQ2)	59	13,455	67
	Diamond (WL76)	33	6687	33
Total		**92**	**20,142**	**100%**

All diamond drilling has been sampled per meter and assayed for gold as well as 50 other multi-elements.

The resultant database has been validated, and the checks made to the database prior to use included:

- Check for overlapping intervals.
- Downhole surveys at 0 m depth.
- Consistency of depths between different data tables.
- Check gaps in the data.
- Replacing less than detection samples with half detection.
- Replacing intervals with no sample with -999.
- Replacing intervals with assays not received with -998.

Figure 14.29 Heinä Central Gold Deposit, Plan View of Drilling



14.3.3 Interpretation and Modelling

Mineralisation Interpretation

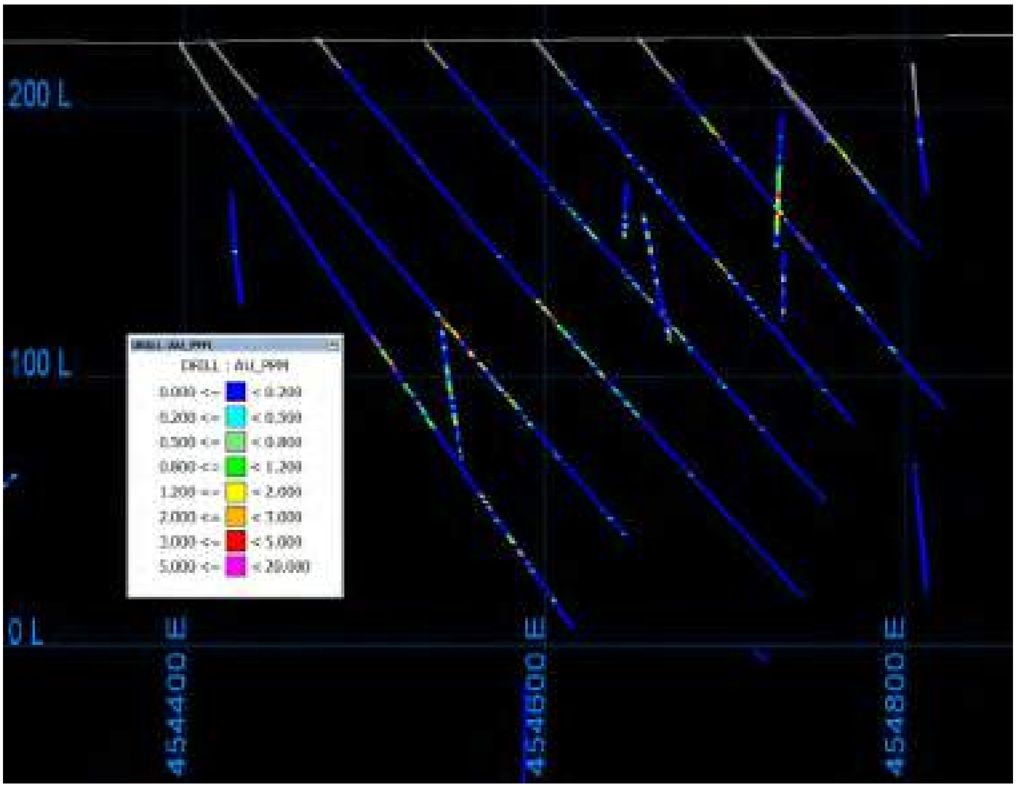
Mineralisation at Heinä Central occurs within brecciated hydrothermal, albite-quartz-carbonate dominated veins that are themselves emplaced in fine grained, partly carbonaceous sediments. In places Po, Py and Cp occur as infill to brecciated hydrothermal veins and sulphide is limited to 5 to 10% of the rock mass. Elsewhere, particularly in the hinge of the fold the hydrothermal veins are flooded by massive and semi massive sulphides of the same composition with remanent wall rock fragments and contacts composed of the same hydrothermal vein.

Chalcopyrite occurs as either infill to cracks within the pyrrhotite dominated sulphides or a late-stage veinlets cutting the pyrrhotite reflecting either the lower crystallisation temperature of Cp compared to Po or precipitation from a later fluid. Gold is positively correlated with copper but although the presence of Cp is a positive indication for gold mineralisation, at meter scale the correlation is not perfect.

As a consequence of the structural complexity of the mineralised lithologies, the mineralisation often presents as somewhat irregularly distributed on the scale of the drill sections completed to date (approximately 40 m), although mineralised zones overall persist across multiple sections.

Figure 14.30 presents a sectional view demonstrating variability in grade, thickness and orientation of gold mineralisation. The intricate folding and faulting of the sedimentary intercalations and the overprinting breccias makes the traditional approach of wireframing 'host lithology' on a sectional and plan basis difficult with multiple plausible geometrical solutions often existing.

Figure 14.30 Cross Section Showing Main Mineralised Zones



Gold is the major element of interest at Heinä Central with secondary copper. Additional complexity is introduced in the definition of the mineralised zones by the issue of copper mineralisation not being entirely coincident with the gold mineralisation. Copper mineralisation is neither as extensive as gold or as continuous. To establish appropriate grade continuity for both gold and copper, the mineralisation models were therefore based upon a nominal indicator mineralisation shell estimated using five m unconstrained downhole composites. For gold, the indicator was defined at the 0.3 g/t cut-off and for copper the cut-off was selected at 0.2% (2,000 ppm) Cu. This interpretation is designed to capture the broad mineralisation halo that encompasses the geological system and is not intended to constrain individual veins or lithologies.

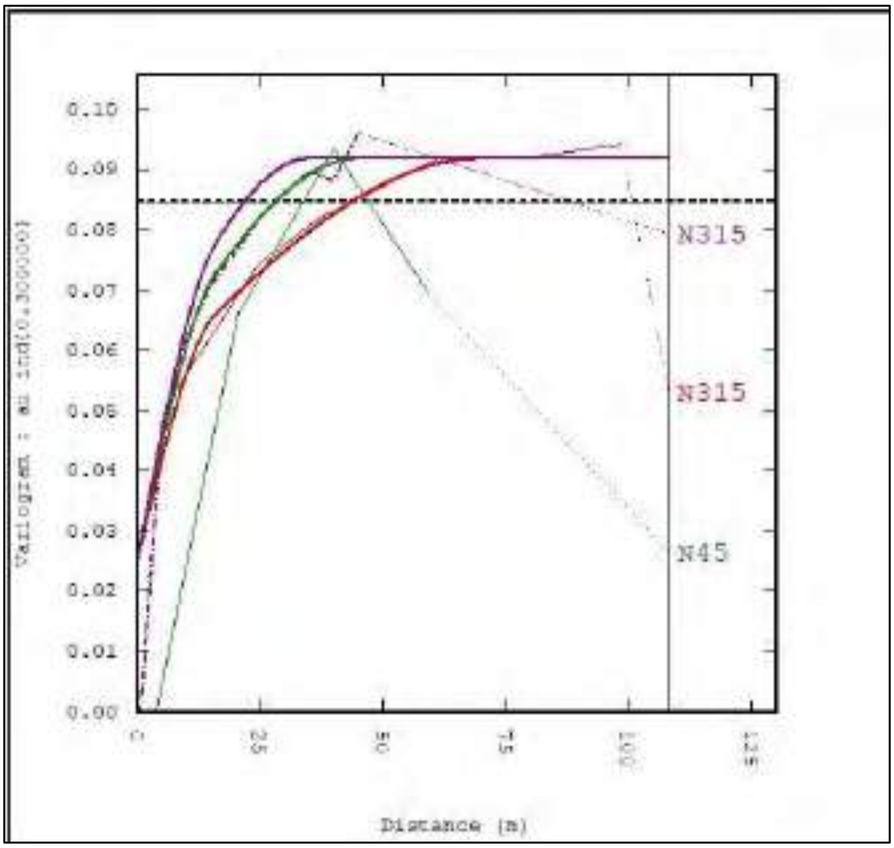
Indicator estimation was into block models with cell dimensions of 2.5 m E × 2.5 m N × 2.5 m RL. Two estimate passes were adopted, one to capture the gold mineralisation and another for the copper. Grade shell triangulations were then generated by constraining the block model at a 30% probability cut-off (Figure 14.30).

Indicator variogram parameters are presented below in Table 14.39 and the variogram in Figure 14.31. Table 14.40 details the indicator estimate sample search parameters.

Table 14.39 Heinä Central Deposit, Indicator Variogram Parameters IK Estimate

Variable	C°	Axis Orientation (°)			Structure 1				Structure 2			
		Major	Semi Major	minor	C1	Major	Semi-Major	Minor	C2	Major	Semi-Major	Minor
Au 0.3 ppm cut-off	0.025	40→315	0→225	50→135	0.028	16	16	16	0.039	70	45	35
Cu ppm 2000 cut-off	0.023	40→315	0→225	50→135	0.012	20	15	10	0.035	65	50	35

Figure 14.31 Indicator Variogram 0.3 g/t Au, 2000 ppm Cu



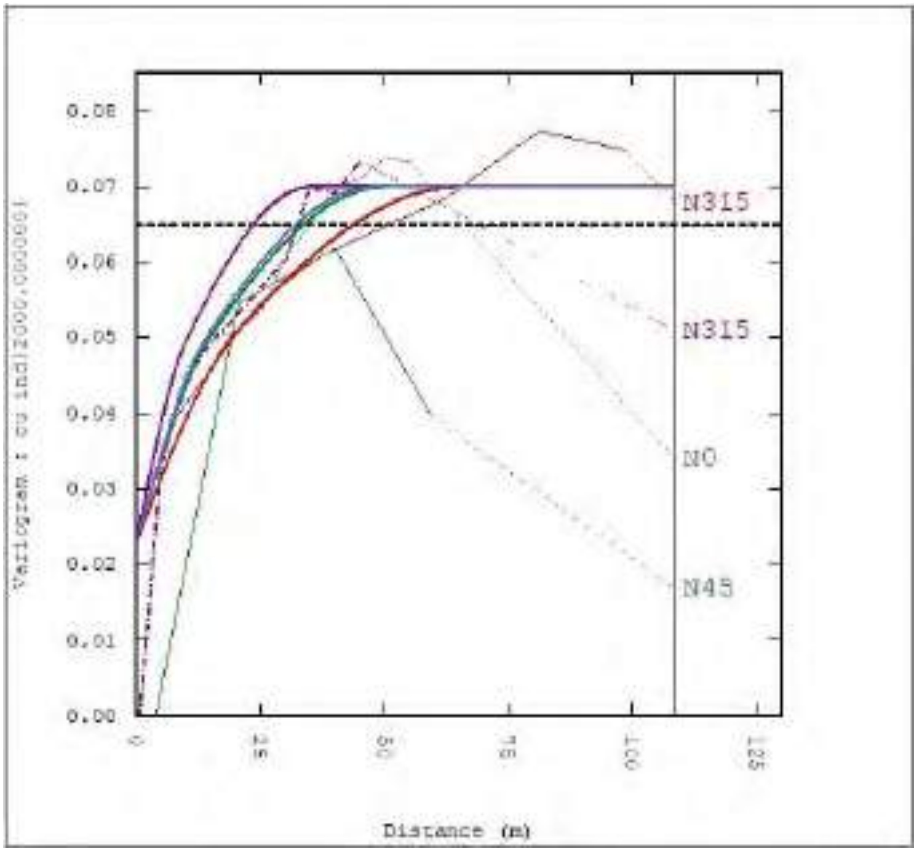


Table 14.40 Heinä Central Deposit, Sample Search Parameters IK Estimate

Domain	Estimation Pass	Axis Orientation (°)			Search Distance (m)			Min No of Comp	Max No of Comp	Max No of Comp per DH	Discret-isation
		Bearing	Plunge	Dip	X	Y	Z				
Au 0.3 ppm	1	40→315	0→225	50→135	80	60	30	8	8	5	2*2*2
Cu 2,000 ppm	1	40→315	0→225	50→135	80	60	30	8	8	5	2*2*2

The probability cut-offs may be considered somewhat subjective and may seem arbitrary, however was selected based on extensive review of a range of probability cut-offs. The selected probability shells are considered optimal to capture the observed continuity and tenor of mineralisation while excluding obvious low-grade material. Grade shells were reviewed in multiple orientations and in plan and section views prior to being accepted for grade estimation and block modelling purposes.

The resultant gold and copper grade shells are not entirely coincident with the copper shell representing approximately 60% of the volume of the gold.

A typical section demonstrating the black shale hanging wall and the constraining grade shell outline is presented in Figure 14.32. A plan and isometric view of the mineralisation wireframe is presented in Figure 14.33 and Figure 14.34 respectively.

Figure 14.32 Heinä Central Deposit, Typical Cross Section (Looking NE)

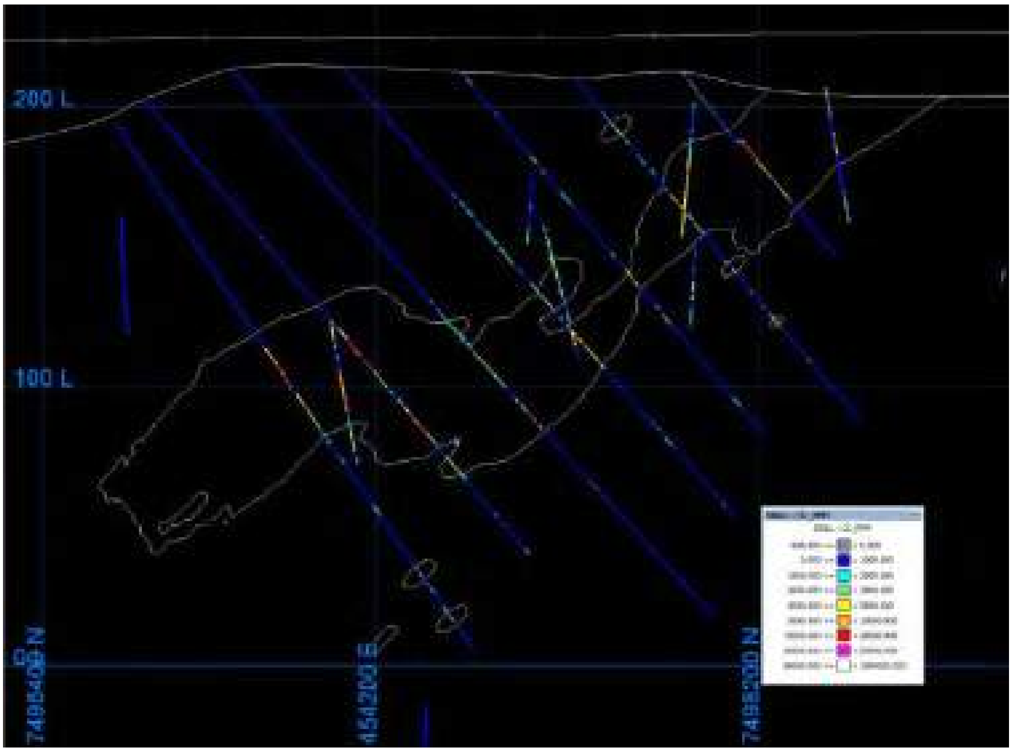


Figure 14.33 Heinä Central Deposit, Estimation Domains Plan View, Copper Domain in Red

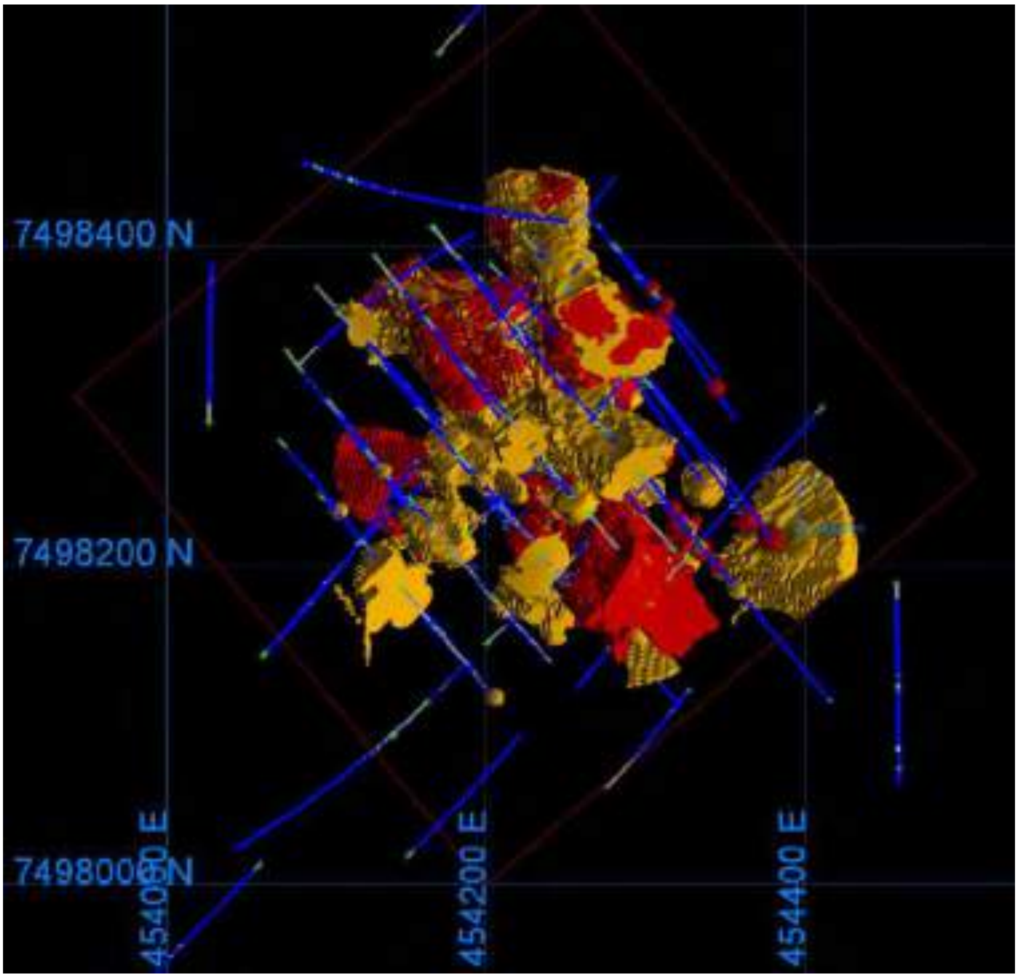
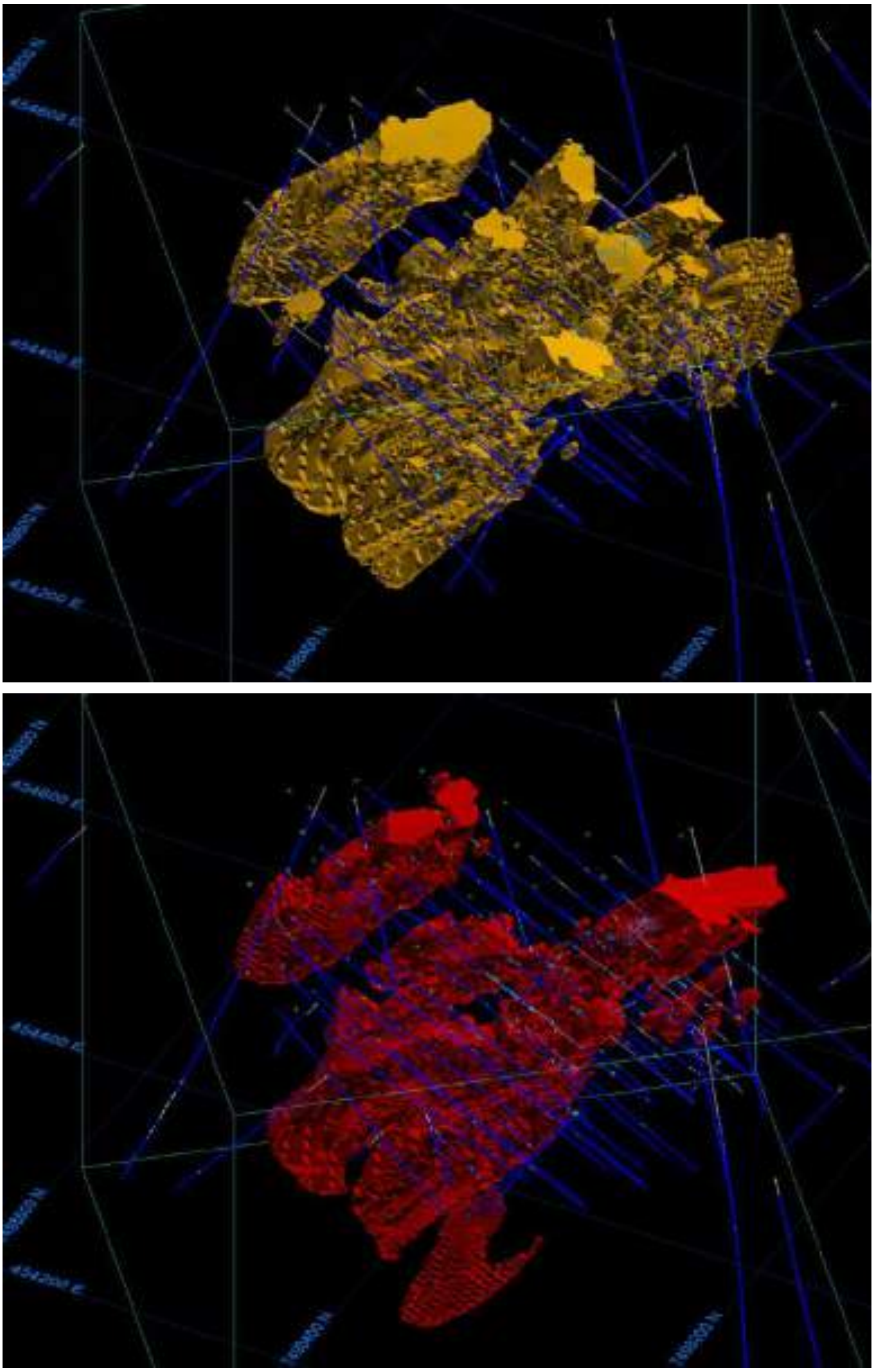


Figure 14.34 Heinä Central Deposit Estimation Domains Isometric S View Au ppm and Cu ppm Copper Domain in Red



Data Flagging and Compositing

Drill hole samples were flagged with the relevant indicator grade shells and topographical surfaces. Coding was undertaken on the basis that if the individual sample centroid fell within the grade shell boundary it was coded as within the grade shell. Each sub-domain has been assigned a unique numerical code to allow the application of hard boundary domaining if required during grade estimation.

The drill hole database coded within each grade shell or mineralisation wireframe was then composited as a means of achieving a uniform sample support. After consideration of relevant factors relating to geological setting and mining, including likely mining selectivity and bench/flitch height, a regular three m run length (downhole) composite was selected as the most appropriate composite interval to equalise the sample support at the Heinä Central Gold Deposit. Compositing was broken when the routine encountered a change in flagging (grade shell boundary) and composites with residual intervals of less than three m were retained in the composite file.

14.3.4 STATISTICAL ANALYSIS

SUMMARY STATISTICS

The composites flagged as described in the previous section were used for subsequent statistical, geostatistical and grade estimation investigations.

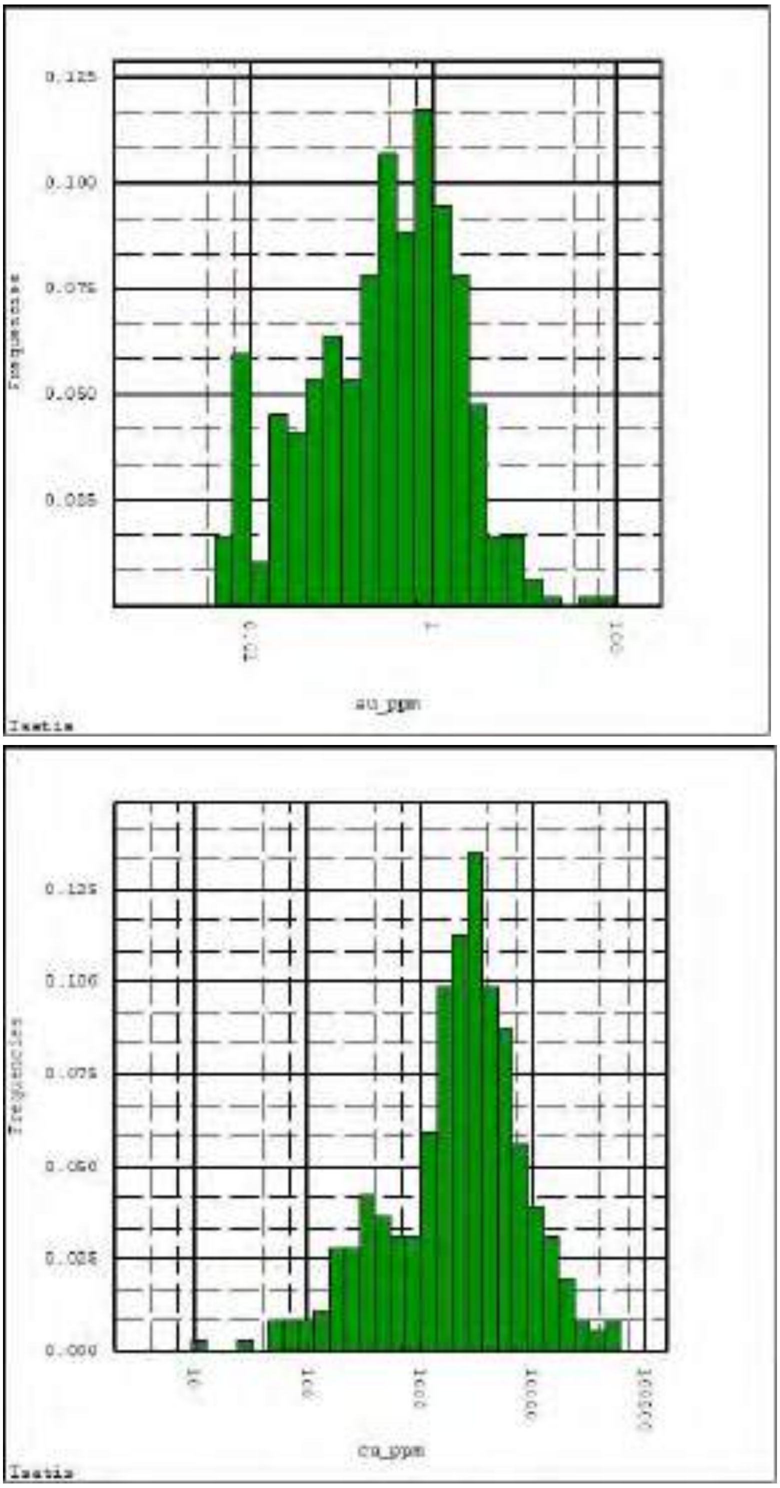
Three m composite summary statistics for gold and copper within their relevant grade shells are presented in Table 14.41.

Table 14.41 Summary Statistics Low Grade Domains for Three Metre Composites of Uncut Gold Grade (g/t)

Domain	Element	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
Gold	Au	486	0.005	86.465	1.238	4.742	22.491	3.831
Copper	Cu	355	0.001	5.613	0.450	0.684	0.468	1.519

Summary descriptive statistics were generated for the mineralised domains (Table 14.42). The grade distribution is reasonably typical for gold deposits of this style and shows a positive skew or near lognormal behaviour (Figure 14.35). The histogram indicates a potential bimodal distribution with a potential subordinate low grade population present. The coefficient of variation (CV - calculated by dividing the standard deviation by the mean grade) is high for gold, indicating any potential high outlier grades may significantly contribute to the total metal.

Figure 14.35 Log Histogram of Uncut Gold Grade



High Grade Outlier Analysis

A high grade outlier analysis has been undertaken for the three m composite gold grades. The effects of the highest-grade composites on the mean grade and standard deviation of the gold dataset for each of the estimation domains have been investigated by compiling and reviewing statistical plots (histograms and probability plots). The resultant plots were reviewed together with probability plots of the sample populations and an upper cut for each dataset was chosen coinciding with a pronounced inflection or increase in the variance of the data. An upper cut was chosen at 25 g/t. Top cut statistics are presented in Table 14.42.

Composite data was viewed in 3D to determine the clustering or otherwise of these highest grades observed in each domain to assess the appropriateness of the high-grade cut. Clustering of the highest grades in one or more areas may indicate that the grades do not require cutting.

Table 14.42 Summary Statistics, Three Metre Composites Gold Grade (g/t) with top cut

Domain	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
Au domain	486	0.001	25	1.071	2.448	5.993	2.285

CELL DECLUSTERING ANALYSIS

Visual inspection of the available datasets for each of the estimation domains indicated some clustering of the data within higher grade regions of the deposit. Data clustering often occurs when drilling campaigns selectively target higher grade regions of the deposit, resulting in an artificially high mean grade in many cases. Declustering was therefore completed to remove any effects of preferential sampling of high grade areas that may have occurred.

Cell declustering was completed with weights w(i) associated with composite (i) determined as w(i) = mv/n(i) where mv is the mean of all the samples within a moving window centred on composite (i) and n(i) is the number of samples within the moving window. This normalisation allows the weights not to decrease with the number of data.

Declustered composite statistics are presented in Table 14.52. As expected, the declustered mean grades are significantly less than the raw composite mean grades due to the data configuration issues discussed above.

Table 14.43 Summary statistics, Three Metre Composites Gold Grade (g/t) declustered

Domain	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
Gold	486	0.005	25.000	1.035	2.550	6.502	2.464
Copper	355	0.001	5.613	0.431	0.667	0.444	1.547

14.3.5 VARIOGRAPHY

INTRODUCTION

Discussion on variography is outlined in Section 14.1.6

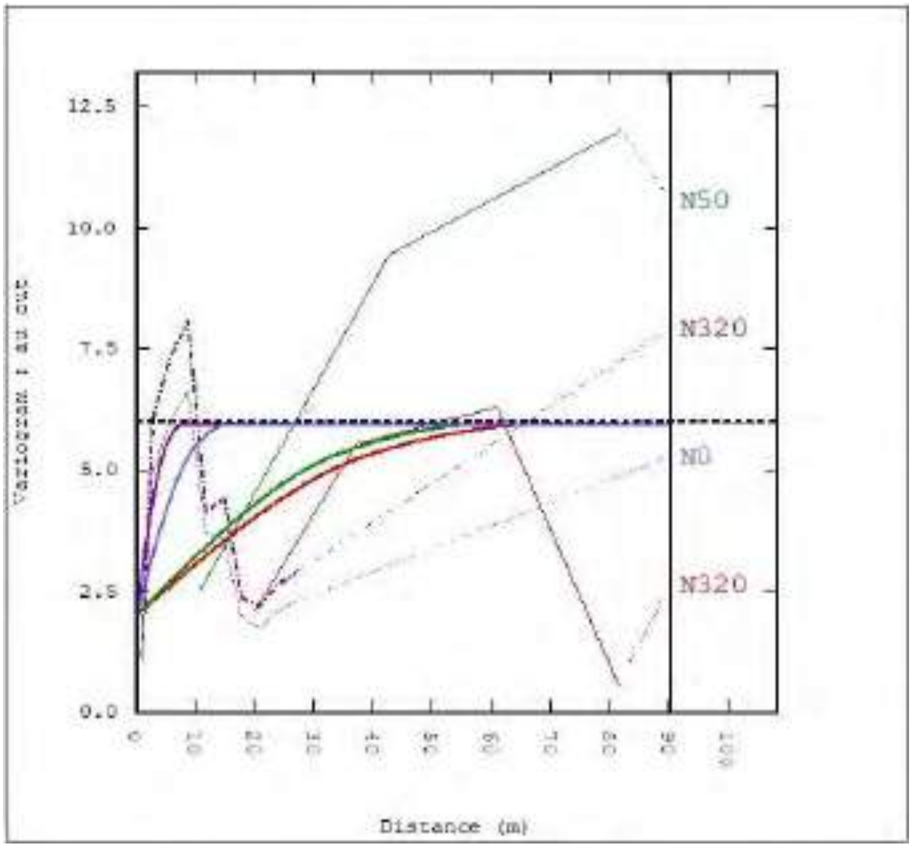
HEINÄ CENTRAL VARIOGRAPHY

Grade variography was generated to enable grade estimation via OK and change of support analysis to be completed. Interpreted anisotropy directions correspond well with the interpreted geology and overall geometry of the interpreted domains.

Indicator variography for the OK estimation domains is presented in

Table 14.44. The modelled grade variograms are presented in Figure 14.36.

Figure 14.36 Heinä Central Gold Deposit, Grade Variograms



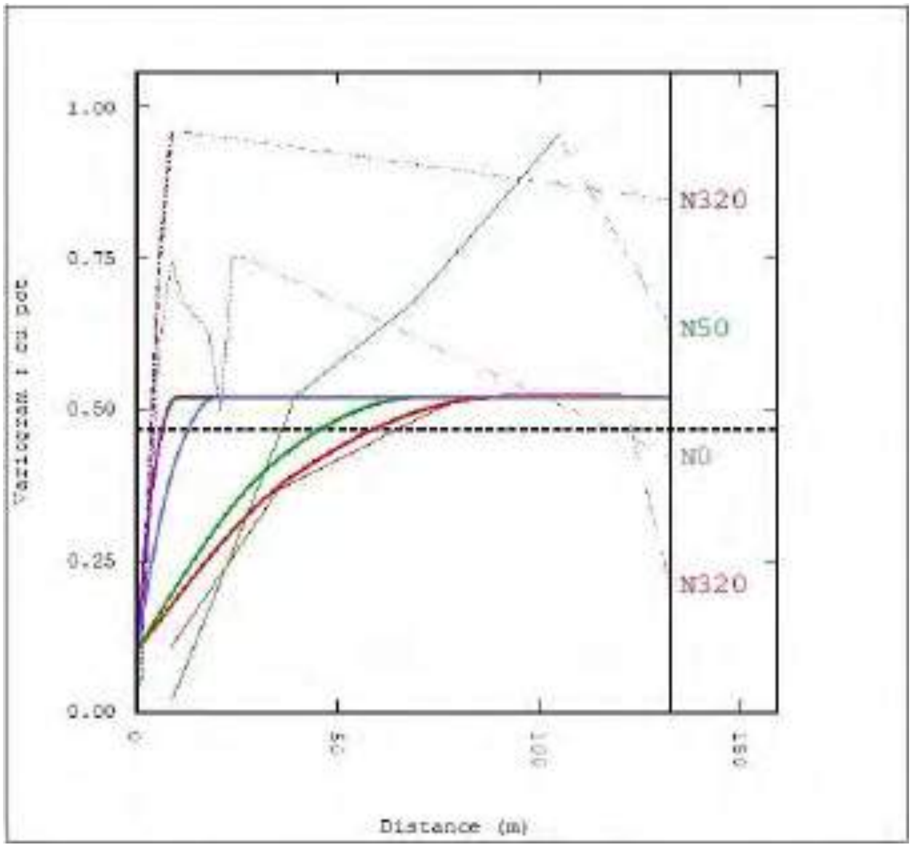


Table 14.44 Heinä Central Gold Deposit, Variogram Models Au g/t

Grade Variable	Nugget (C0)	Rotation (dip→dip dir)			Structure 1				Structure 2			
		Bearing	Plunge	Dip	Relative Sill 1 (C1)	Range (m)			Relative Sill 2 (C2)	Range (m)		
						Major	Semi Major	Minor		Major	Semi Major	Minor
Au ppm	2.0	40→320	0→230	50→140	1.35	40	35	5	2.61	70	60	8
Cu Pct	0.1	40→320	0→230	50→140	0.09	40	35	5	0.33	90	70	10

14.3.6 Block Modelling

A 3D block model was created in the ETRS89 (European Terrestrial Reference System) grid using Vulcan mining software. The parent block size was selected on the basis of the average drill spacing together with consideration of potential mining parameters. A parent cell size of 20 m E by 20 m N by 10 m RL which was sub-blocked down to 5 m E by 5 m N by 2.5 m RL (to ensure adequate volume representation). The models covered all the interpreted mineralisation zones and included suitable additional waste material to allow later mining engineering studies. Block coding was completed on the basis of the block centroid, wherein a centroid falling within any wireframe was coded with the wireframe solid attribute. The block model is rotated.

The main block model parameters are summarised below in Table 14.45. Variables were coded into the block models to enable ordinary kriging estimation and grade tonnage reporting. A visual review of the wireframe solids and the block model indicated correct flagging of the block model. Additionally, a check was made of coded volume versus wireframe volume which confirmed the above.

Table 14.45 Heinä Central Gold Deposit, Block model Parameters

	Northing (Y)	Easting (X)	RL (Z)
Min Coordinates	7,498,000	454,200	-100
Extent	400	400	350
Block size (m)	20	20	10
Sub Block size (m)	5	5	2.5
Rotation (° around axis)	0	0	50

14.3.7 Bulk Density Data

A dry bulk density database has been supplied containing a total of 1,746 data. All density measurements have been systematically taken by Rupert Resources as part of their ongoing core processing operations.

Rupert Resources have calculated dry bulk densities on the basis of the weight in water method. Density readings have been taken on whole drill core and are distributed across all areas of the deposit. Based on the mineralisation model, statistical analysis demonstrates that across the deposit, mineralised rock has a higher density than unmineralised rock. Summary statistics subdivided by mineralisation domain are presented in Table 14.46. Based on the small, but significant difference in average density between mineralised and unmineralised rock, bulk densities have been estimated to the block model based via ordinary kriging. The gold and copper domains have been amalgamated for this purpose and the waste has been estimated separately. Hard estimation boundaries were applied throughout. Additionally, overburden has been applied an arbitrary valued of 1.8 t/m^3 as no direct readings are available. An omnidirectional variogram was calculated and modelled based on the domained data. The modelled variography is presented in Figure 14.37 and Table 14.47 and the OK estimate parameters in Table 14.48.

Table 14.46 Heinä Central Gold Deposit, Density Statistics

Company	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	CV
Gold domain	186	2.46	4.57	3.089	0.38	0.144	0.123
Copper domain	125	2.36	4.65	3.157	0.417	0.174	0.132
Au/Cu Combined	223	2.36	4.65	3.097	0.390	0.152	0.126
unmineralised	1,523	2.04	4.89	2.875	0.223	0.050	0.078

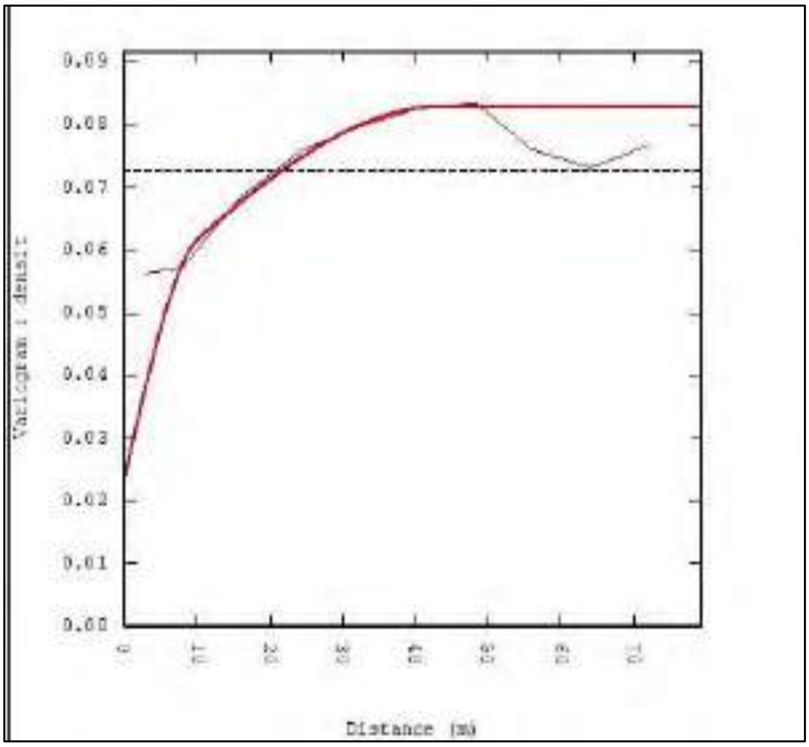
Table 14.47 Heinä Central Gold Deposit, Density Variogram model

Variable	Nugget	Sill 1	Range 1	Sill 2	Range 2
Density	0.024	0.027	10	0.032	44

Table 14.48 Heinä Central Gold Deposit, Density Estimate Parameters

Domain	Pass	Sample Search Orientation (dip/dip direction˚)			Sample Search Distance (m)			Numbers of 3 m Composites			% Blocks Estimated
		Major	Semi Major	Minor	Major	Semi Major	Minor	Min	Max	Max Per Drill hole	
Au/Cu	Pass 1	0→0	0→90	90→0	200	200	2000	6	6	3	100
Waste	Pass 1	0→0	0→90	90→0	200	200	200	6	6	3	99.3

Figure 14.37 Omnidirectional Density Variogram



14.3.8 GRADE ESTIMATION

INTRODUCTION

Ordinary Kriging (OK) was applied to grade estimation at the Heinä Central deposit, within the defined indicator mineralisation shells. Estimation was completed in the mining package Vulcan using the GSLib geostatistical software while geostatistical analysis was developed in Isatis geostatistical software. OK grade estimates have been estimated to an SMU dimension block size (5 m E x 5 m N x 5 m RL) to maintain local grade characteristics and reduce grade smearing. This estimation approach was considered appropriate based on review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralisation, and the style, geometry and tenor of mineralisation. The estimation was constrained with geological and mineralisation interpretations

ORDINARY KRIGING PARAMETERS

OK estimates were completed using the grade variogram models (Section 14.1.6), and a set of ancillary parameters controlling the source and selection of composite data. The sample search parameters were defined based on the variography and the data spacing, and a series of sample search tests performed in Isatis geostatistical software.

The sample search parameters for the OK estimations are provided in Table 14.49. Hard domain boundaries were used for the estimation throughout A kriging plan was devised whereby gold was estimated both inside and outside the gold domain and an identical strategy applied for copper in the copper domain. A two-pass estimation strategy (where required) was applied to each domain, applying a progressively expanded and less restrictive sample search to the successive estimation pass, and only considering blocks not previously assigned an estimate. Parent cell estimations (5 m E by 5 m N by 5 m RL) were applied throughout and discretisation was applied on the basis of 2 X by 2 Y by 2 RL for 8 discretisation points per block.

Table 14.49 Heinä Central Gold Deposit, OK Sample Search Criteria

Domain	Pass	Sample Search Orientation (dip/dip direction˚)			Sample Search Distance (m)			Numbers of 3m Composites			% Blocks Estimated
		Major	Semi Major	Minor	Major	Semi Major	Minor	Min.	Max.	Max Per Drill Hole	
Gold/Au	Pass 1	40→320	0→230	50→140	50	40	15	6	6	3	58
	Pass 2	40→320	0→230	50→140	300	200	90	4	6	-	100
Copper/Cu	Pass 1	40→320	0→230	50→140	50	40	15	6	6	3	62
	Pass 2	40→320	0→230	50→140	200	120	60	6	6	-	100
Waste/Au	Pass 1	40→320	0→230	50→140	50	40	15	6	6	3	45
	Pass 2	40→320	0→230	50→140	200	120	60	6	6	-	100
Waste/Cu	Pass 1	40→320	0→230	50→140	50	40	15	6	6	3	43
	Pass 2	40→320	0→230	50→140	200	120	60	6	6	-	94

ESTIMATE VALIDATION

All relevant statistical information was recorded to enable validation and review of the OK estimates. The recorded information included:

- Number of samples used per block estimate.
- Number of drill holes from which samples selected.
- Average distance to samples per block estimate and distance to nearest sample.
- Estimation flag to determine in which estimation pass a block was estimated.

Number of drill holes from which composite data were used to complete the block estimate.

The estimates were reviewed visually and statistically prior to being accepted. The review included the following activities:

- Comparison of the OK estimate versus the mean of the composite dataset, including weighting where appropriate to account for data clustering.
- Production of swath plots comparing input composite grades versus block grades (Figure 14.38).
- Visual checks of cross sections, long sections, and plans.

Alternate estimates including OK estimates into varying parent cell size blocks.

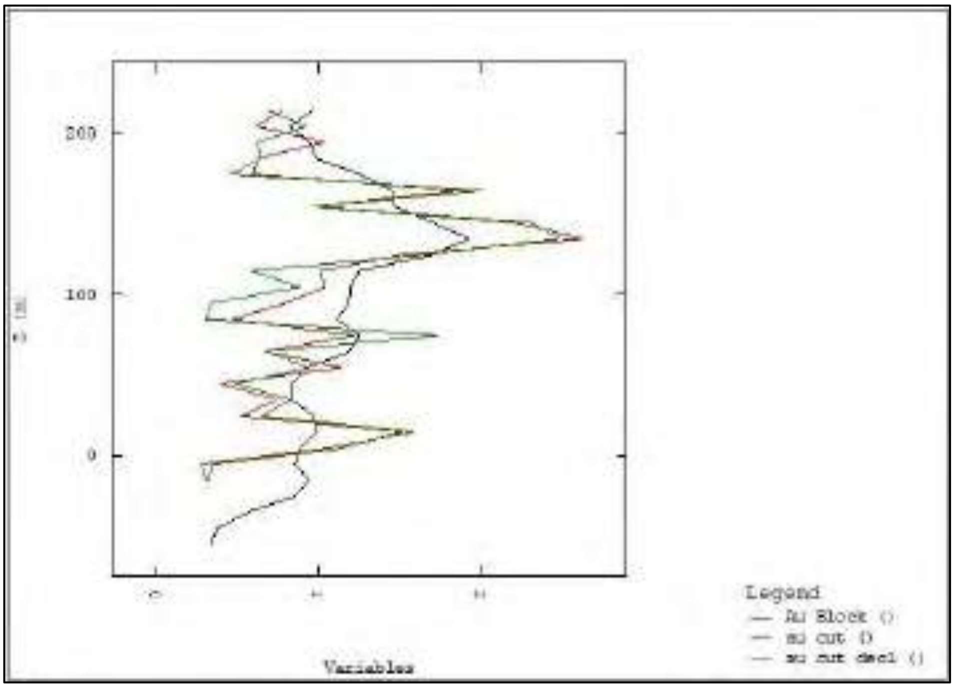
Validation of block Au and Cu grades has been undertaken by comparing the block mean grades with the relevant composite mean grades (Table 14.50).

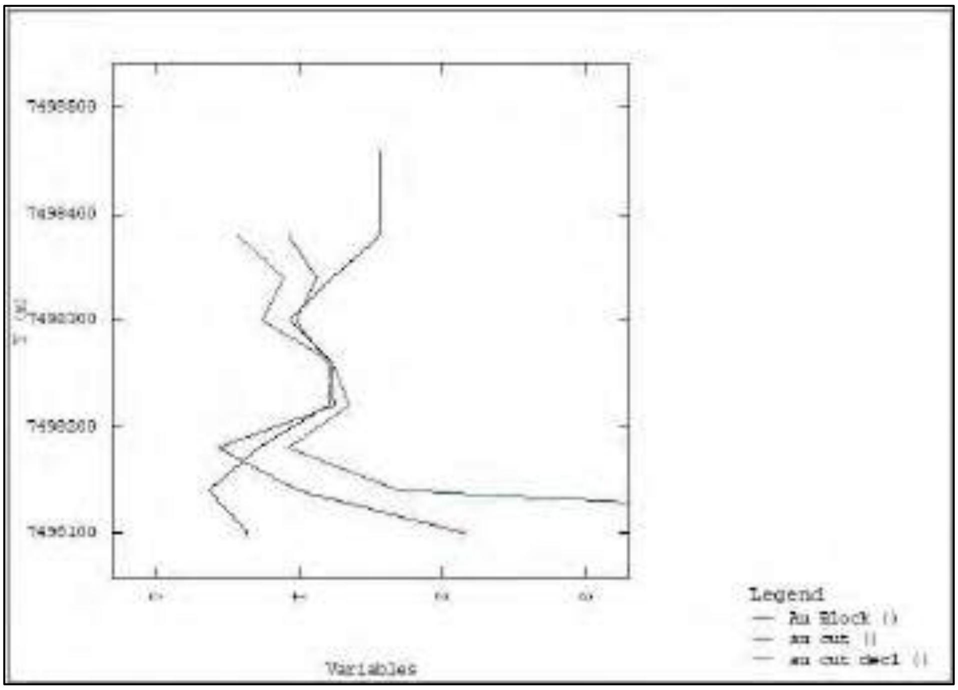
Table 14.50 Validation of Block Au and Cu Grades

Zone	All Composites, (declustered, capped)	All Composites, (uncapped)	All Composites, (non-declustered, capped)	Block Model Grades	% Diff Block Model vs declustered Mean
Au	1.035	1.238	1.071	1.148	7%
Cu	-	0.450	0.431	0.562	30%

A good correlation may be drawn between the declustered composite mean grades and the block model mean grade for gold. In the case of copper, block mean grades demonstrate an increase of 30%. It is considered likely that drillhole orientation and configuration is responsible for this anomalous increase in copper mean block grade over the composite grades.

Figure 14.38 Comparison of Swath Plot Grades for Input Composites




7498500
7498400
7498300
7498200
7498100
Variables
Legend
Au Block ()
au cut ()
au cut decl ()

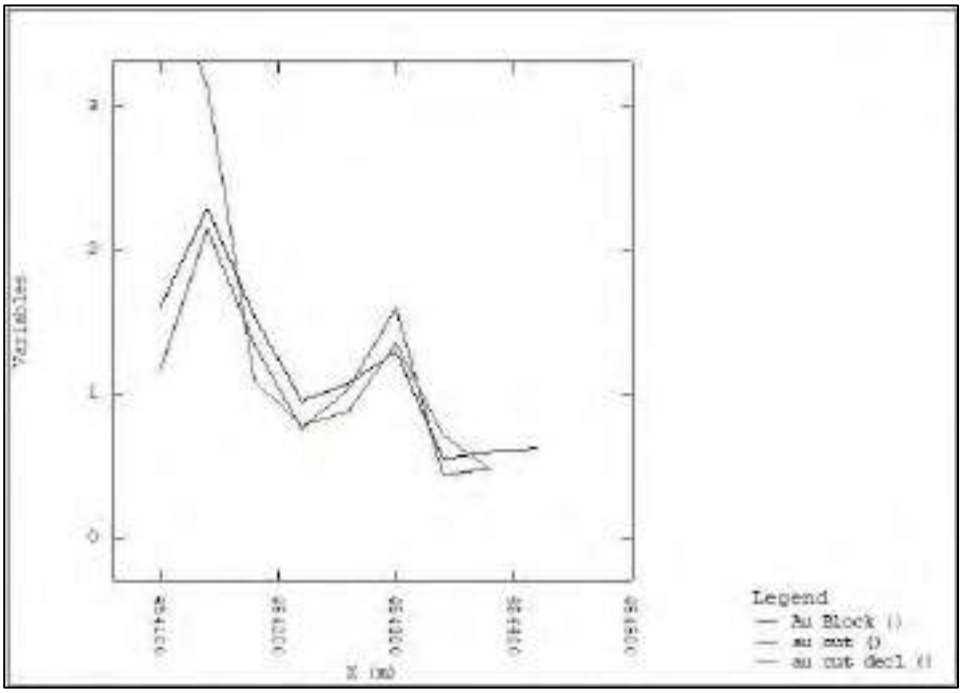
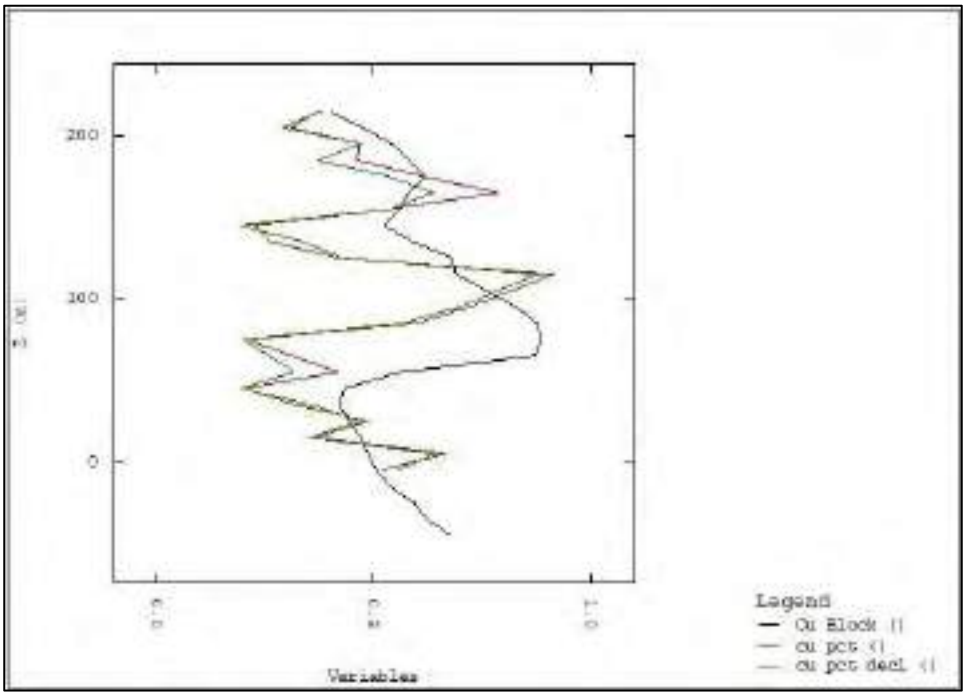
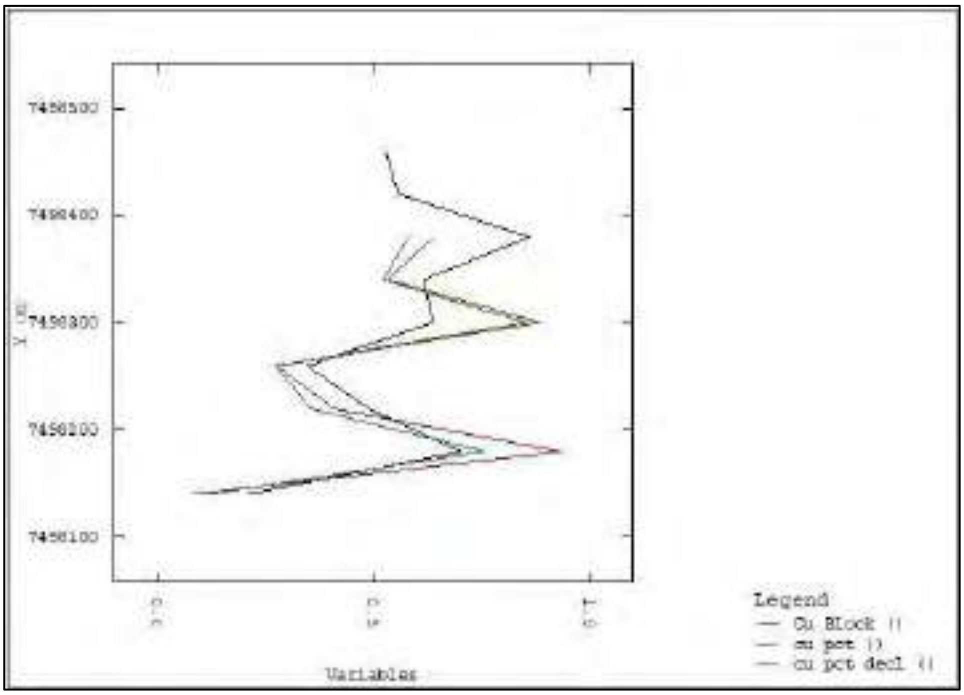
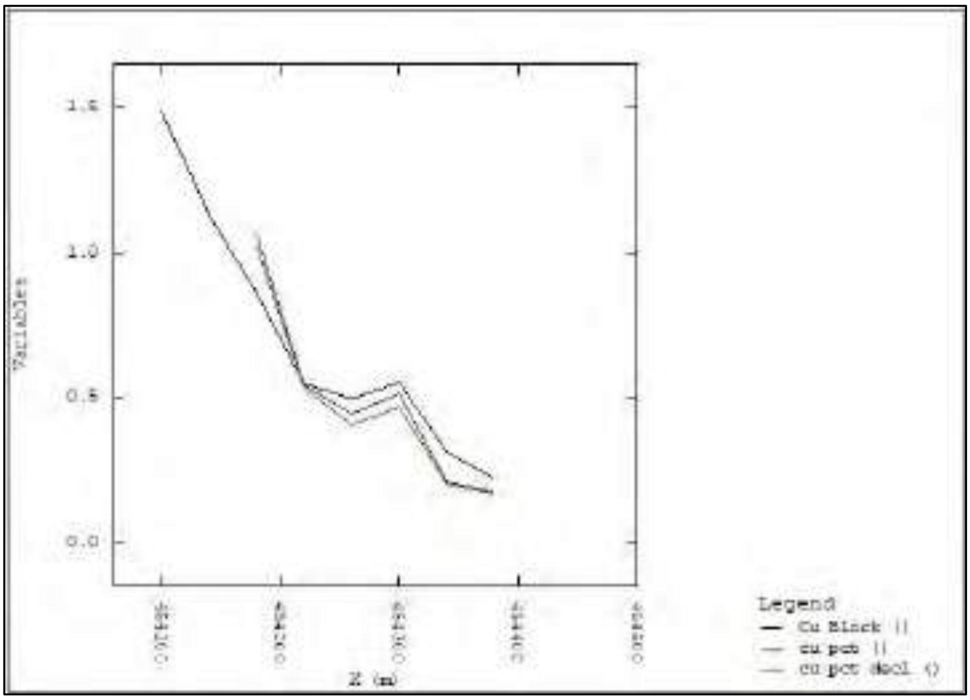

Variables
2
1
0
X (m)
Legend
Au Block ()
au cut ()
au cut decl ()

Figure 14.39 Comparison of Swath Plot Grades for Input Composites







14.3.9 Resource Classification

The resource categorisation was based on the robustness of the various data sources available, including:

- Geological knowledge and interpretation.
- Variogram models and the ranges of the first structure in multi-structure models.
- Drilling density and orientation.
- Estimation quality statistics.

The resource estimates for the Heinä Central Gold Deposit have been classified as Inferred Mineral Resources based on the confidence levels of the key criteria as presented in Table 14.51.

Table 14.51 Heinä Central Deposit, Confidence Levels by Key Criteria

Items	Discussion	Confidence
Drilling Techniques	Diamond drilling- Industry Standard approach.	High
Logging	Standard nomenclature has been adopted.	High/Moderate
Drill Sample Recovery	Recoveries are not recorded in entire database but diamond core recoveries assumed acceptable.	High/Moderate
Sub-sampling Techniques and Sample Preparation	Diamond sampling conducted by industry standard techniques.	High
Quality of Assay Data	Appropriate quality control procedures available for work completed by Rupert. They were reviewed and considered to be of industry standard.	Moderate/High

Verification of Sampling and Assaying	Sampling and assaying procedures have been assessed and are considered of appropriate industry standards.	Moderate
Location of Sampling Points	Survey of all collars conducted with accurate survey equipment. Investigation of downhole survey indicates appropriate behaviours.	Moderate/High
Data Density and Distribution	Majority of regions defined at a minimum on a notional 40 m E x 40 m N drill spacing.	Moderate
Audits or Reviews	N/A	
Database Integrity	Industry standard approach applied by Rupert.	Moderate
Geological Interpretation	Mineralisation controls are complex and not yet fully understood. The mineralisation constraints are relatively broad and therefore of moderate confidence. Controls at a local scale are commonly of uncertain continuity.	Low to Moderate
Estimation and Modelling Techniques	Ordinary Kriging is considered to be appropriate given the geological setting and grade distribution.	High
Cut-off Grades	The mineralisation constraints were based on a notional 0.3g/t Au lower cut-off grade. A 0.2% lower cut-off grade is considered appropriate for Copper. Copper is subordinate in value to gold at the grades present.	Moderate/High
Mining Factors or Assumptions	A 5 m E x 5 m N x 5 m RL SMU estimated for gold and copper. Open pit mining assumed and SMU is conditional on scale assumed.	Moderate
Metallurgical Factors or Assumptions	Not applied.	N/A
Tonnage Factors (In-situ Bulk Densities)	Sufficient data exists to enable high confidence in the applied density values.	High

14.3.10 Resource Reporting

The summary total Mineral Resource for the Heinä Central Gold Deposit is provided in Table 14.52. The Mineral Resource is reported both within an optimised open pit and as a potential underground operation below that. The preferred lower cut-off grade for reporting is 0.5 g/t Au for the portion potentially mineable by open pit methods and 1.2 g/t for the portion potentially extractable by underground methods. In view of the nature and style of the mineralisation and potential mining approach and method, these are considered appropriate cut-off grades.

Table 14.52 Heinä Central gold Deposit, Mineral Resource Report (Inferred Resource), Summary Grade Tonnage Report

	Lower Cut-off Grade (g/t Au)	Tonnes (t)	Average Grade (g/t Au)	Gold Metal (oz)	Gold Metal (Kg)	Average Grade (Cu %)	Copper Metal (tonnes)
Open Pit	0.3	2,600,000	1.5	130,000	4,000	0.46	13,000
	0.4	2,500,000	1.6	130,000	3,900	0.46	12,000
	0.5	**2,200,000**	**1.7**	**120,000**	**3,800**	**0.48**	**12,000**
	0.6	2,000,000	1.8	120,000	3,700	0.50	11,000
	0.7	1,800,000	1.9	110,000	3,600	0.53	11,000
Underground	0.8	760,000	1.6	39,000	1,200	0.32	2,400
	1.0	510,000	1.9	31,000	1,000	0.38	1,900
	1.2	**420,000**	**2.1**	**28,000**	**900**	**0.42**	**1,800**
	1.4	370,000	2.2	26,000	800	0.45	1,700
	1.6	320,000	2.3	24,000	700	0.49	1,500
Total (Open pit + Underground)		**2,650,000**	**1.8**	**150,000**	**4,700**	**0.51**	**14,000**

Note: Appropriate rounding has been applied

An isometric view of the optimised open pit with estimated blocks is presented in Figure 14.40. It should be noted that mineral resources that are not mineral reserves do not have demonstrated economic viability. Values in tonnes, ounces and kilograms are rounded to two significant figures which cause discrepancies between rows in the table.

The effective date of this Mineral Resource is 28th November 2022. It is not anticipated that this Mineral Resource estimate will be materially affected, to any extent, by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors.

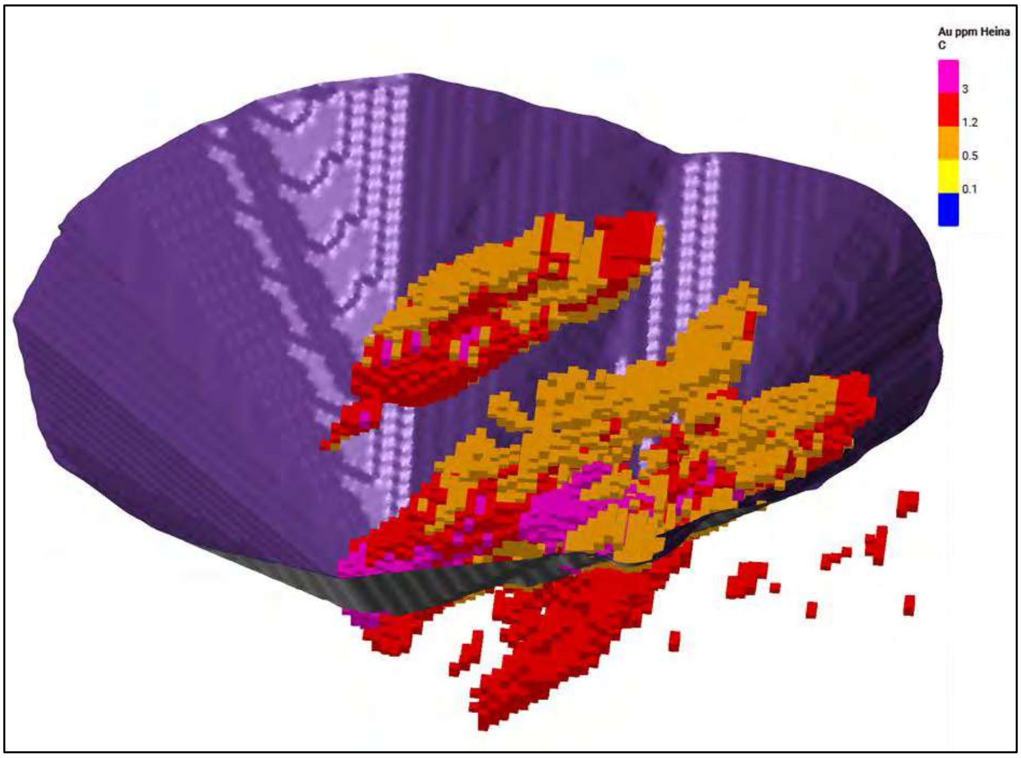
The resource model was based on costs from comparable operations notably the nearby Ikkari and Pahtavaara deposits included in this document. The preferred reporting cut-off grade for the mineral resource are 0.5 g/t Au for the open pit portion and a 1.2 g/t Au for the underground portion. The open pit resource is reported within an optimised pit with overall slope angle 45° and economic parameters as per the cut-off grade calculations set out below. Resources outside this optimised pit were considered for underground mining.

The optimised pit and cut-off grades are based on the following economic parameters: Gold price $1,650 /oz, 95% mining recovery, 80% Au recovery to concentrate with subsequent 98% recovery to Doré. Estimated open pit mining costs are $2.5 /t, process costs are $10.01 /t and $3.2 /t other costs including CDF and closure costs. Estimated process costs at Heinä Central include production of concentrate and transport of concentrate to Ikkari for refining. G&A including refining and royalties are $1.66 /t. The calculated cut-off grade has been rounded up to 0.5 g/t for reporting.

Estimated underground mining costs are $30 /t and a 95% mining recovery is assumed. The calculated underground cut-off grade has been rounded up to 1.2 g/t for reporting.

At the 0.5 g/t open pit Au cut-off grade, Heinä Central also contains a potentially recoverable resource of 12,000 t contained copper. At the 1.2 g/t underground Au cut-off grade, Heinä Central also contains a potentially recoverable resource of 1,800 t contained copper. No economic value is applied to the copper resource when designing the optimised pit or calculating the potential cut-off grades at Heinä Central. Further studies will help to better define the costs of mining at Heinä Central and recoverability of the copper resource.

Figure 14.40 Estimated Blocks Above the Respective Cut-off Grades Viewed looking NE at an 40° Plunge



15.0 MINERAL RESERVE ESTIMATES

Not applicable to this Technical Report.

16.0 MINING METHODS

16.1 INTRODUCTION

The Ikkari project is located on a package of exploration licences controlled by Rupert Resources (Rupert Lapland Project) in Lapland, northern Finland, which includes the Pahtavaara Mine project (Pahtavaara) and associated exploration licences that include the Ikkari deposit.

The Pahtavaara gold deposit is situated in a moderately dry, sparsely forested area. The landscape is reasonably flat with an elevation of approximately 240 m to 250 m above sea level. The Pahtavaara hill, located directly to the NE, has an elevation of approximately 325 m above sea level and the overburden cover is generally between 5 m to 10 m thick. In most parts of the deposit area, the ground water table is typically located a few metres below the ground surface.

Mining has previously been undertaken at Pahtavaara by open pit and underground (UG) mining methods with a total of 5.8 Mt of ore extracted over a 16-year operating history between 1996 and 2014. A total of 1.7 Mt of ore was mined from the open pit over this period with a strip ratio of 4:1.

Underground mining at Pahtavaara commenced using contractors in 2004 and continued under two periods of ownership until 2014 with 4.1 Mt mined over this period. Access was by ramp with 5 m x 5 m mine development with mining by LHOS. Ground conditions are considered excellent.

It is proposed to re-commission the process plant at Pahtavaara and to re-develop both the open pit and the underground operations to feed this plant at a rate of 500 Ktpa. This plant will then produce a high-grade concentrate (conc) that will be transported by road to the proposed Ikkari plant for final processing to Doré.

The Ikkari deposit lies 20 km west of Pahtavaara and 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 is predominantly flat with an elevation of approximately 225 m above sea level and rising slightly towards the southeast and the margins of the Iso-Pulkittama hill, which has a maximum elevation of approximately 300 m above sea level.

The overburden cover at Ikkari is comprises glacial till deposits which are generally between 5 m to 40 m in thickness and rock outcrop is very limited across most of the exploration licence area. In most parts of the deposit area, the ground water table is typically located close to the ground surface.

This is a greenfield site that will initially be mined as an open pit to a point at which the open pit is constrained by the exploration permit boundary. The mine will then be converted to an underground operation to maximise the resource recovery within the permit area.

Details of the design and scheduling of the open pit and underground operations at both Pahtavaara and Ikkari are discussed in the following sections of this report and a combined mine schedule has been prepared which sequences these operations. This combined schedule forms the basis of the evaluation of this project.

16.2 Mining Method – Pahtavaara Open Pit

16.2.1 Introduction

Pahtavaara is a brownfield site where open pit mining operations commenced in 1996, followed by intermittent periods of underground operations between 2004 and 2014, when the operation was put on care and maintenance.

It is proposed that the mine is re-developed initially as an open pit that is an expansion of the current open pit. The target initial production rate of 500 kilotonnes per year (Kt/a) of plant feed has been set based on the existing permitted production rate for the process plant.

Whilst mining the open pit the underground operations will be re-developed with access from the expanded open pit. A suitable transition point between open pit operations and underground operations has been selected on the basis of costs as well as practicality. An important consideration is the issues of mining through the existing underground voids, which will significantly add to the costs and require significant additional safety controls to manage the risks of mining near the voids.

16.2.2 Open Pit Mining

The selected mining method for the Pahtavaara open pit is a conventional shovel and truck configuration that is similar to the previous mining operations. This will include medium sized (90 t) haul trucks matched with 100 t hydraulic excavators.

The rock is relatively hard in this mine and will require drill & blast as well as pre-split on the final walls to maximise the overall slope angle.

16.2.3 Resource Model

The evaluation of Pahtavaara is based on the Resource model presented in Section 14 and summarised in Table 16.1.

Table 16.1 Pahtavaara Resource Model

Deposit	Resource Classification	Open Pit Status	Cut-off Grade (g/t)	Tonnage (t)	Grade Au (g/t)	Metal Content	
						Kg Au	Oz Au
Pahtavaara	Indicated	Open Pit	0.5	900,000	2.2	1,900	62,000
		Underground	1.5	1,000,000	3.73	3,700	120,000
		Total		***1,900,000***	***3.0***	***5,600***	***180,000***
	Inferred	Open Pit	0.5	3,700,000	1.6	5,900	190,000
		Underground	1.5	2,200,000	3.1	6,800	220,000
		Total		***5,900,000***	***2.1***	***12,700***	***410,000***
Pahtavaara Indicated and Inferred Total				**7,800,000**	**2.3**	**18,300**	**590,000**

The evaluation at a PEA level of study includes Measured, Indicated and Inferred blocks and consequently it is not possible to define an Ore Reserve and the term Mineral Inventory has been used in this report.

16.2.4 Modifying Factors

The modifying factors for Pahtavaara were investigated by regularising the block model at various parent block sizes and comparing the mineral inventory (tonnage and contained metal) of the regularised models, at various cut-off grades, with the insitu Resource Model (RM) quantities.

The cut-off grades chosen for this exercise were 0.2 and 0.5 g/t Au as these represent the expected minimum grade for sub-economic mineralised material, which may be stockpiled for the future, and the expected Mill cut-off grade for an open pit operation.

Only blocks above a Z elevation of zero were considered as this was the expected to be the maximum depth of any open pit and the dilution factors for blocks below this would not be representative of those considered for open pit mining.

An important factor in the evaluation of the modifying factors for mining recovery and mining dilution is the influence of the mined-out voids (open stopes) from the previous underground operations. These voids have been modelled as blocks with zero density and in the regularisation process this will have been considered.

There are also at least two small open pits that have been backfilled with waste and this has been accounted for by depleting the model. However, the model did not allow for a modified density for the backfill and this needs to be included in future studies.

The results from the regularisation study are shown in Table 16.2.

Table 16.2 Evaluation of Dilution Factors for Pahtavaara

Model	Cut-off Grade		Mining	Mining	Cut-off Grade		Mining	Mining
	>0.2 g/t		Recovery	Dilution	>0.5 g/t		Recovery	Dilution
SMU	(Mt)	Au g/t	(%)	(%)	(Mt)	Au g/t	(%)	(%)
RM	16.4	1.23			10.7	1.70		
5.0 x 5.0 x 2.5	18.0	1.10	99	11	11.6	1.53	97	11
5.0 x 5.0 x 5.0	**19.2**	**1.03**	**99**	**19**	**12.0**	**1.45**	**95**	**17**
10.0 x 5.0 x 5.0	21.0	0.94	98	31	12.5	1.35	92	26
10.0 x 10.0 x 5.0	23.3	0.82	96	49	12.8	1.23	87	38

It was concluded that a SMU of 5 m x 5 m x 5 m gives a reasonable approximation to the dilution factors (95% mining recovery and 17% waste dilution) that can be achieved with a small open pit mine using equipment that is suited to selective mining on 10 m benches and 5 m flitches.

Increasing the SMU size to 10 m x 5 m x 5 m or 10 m x 10 m x 5 m rapidly increases the dilution factors and results in a significant decrease in grade.

The influence of cut-off grade on dilution is also an important consideration and is related to the location of the existing voids. This means that the dilution decreases as you raise cut-off grade as the remaining higher-grade blocks are often located adjacent to the mined-out areas and a higher proportion of the dilutant will be at zero grade (i.e. a void).

16.2.5 PAHTAVAARA MINE OPTIMISATION

The open pit mine optimisation for Pahtavaara was run using Datamine's Net Present Value Scheduler (NPVS) software, which is based on the standard Lerch Grossman (LG) algorithm for pit optimisation. NPVS was run with a range of Price Factors (PF) between 2% and 100% to give a series of nested pit shells that can be used to select the optimum pit limit for further design/evaluation work.

A regularised 5 m x 5 m x 5 m model was used as the input model and the other parameters are summarised in Table 16.3.

Table 16.3 Pit Optimisation Parameters for Pahtavaara

Parameter	Units	Value	Comments
Gold Price	US $/toz	1,650	3 year trailing average
Royalty	%	0.15	Excludes initial Royalty of 1.5%
Discount Rate	%	5.0	
Pit Exit/Crusher	RL	250	
Ref Waste Mining Cost	US $/t mined	2.00	
Ref Ore Mining Cost	US $/t ore	2.50	
Incremental Mining Cost	US $/t/10 m	0.04	
Mining Recovery	%	100.0	Built into regularised model

Mining Dilution	%	0.0	Built into regularised model
Grade Control	US $/t ore	0.50	
Stockpile Rehandle	US $/t ore	0.90	
Mine G&A	US $/t ore	0.20	
G&A Other	US $/t ore	3.47	Based on historical data
Process Cost (Pahtavaara)	US $/t ore	9.10	Based on 0.5 Mt/a plant feed
Process Cost (Ikkari)	US $/dmt conc	7.20	
Concentrate Mass Pull	%	18.0	
Conc Transport Cost	US $/wmt conc	40.0	Based on 160 km round trip
TC/RC	US $/tOz	2.50	
CDF	US $/t ore	0.80	
Au Recovery (Pahtravaara)	%	91.0	
Au Recovery (Ikkari)	%	98.0	
Au Payability	%	99.92	
Overall Slope Angle	Degrees	45.0	

Key: dmt = dry metric tonne, RL = reduced level, TC/RC = Treatment charge / refining charge, wmt = wet metric tonne

Note: The discount rate for Pahtavaara was subsequently updated to 8% for Ikkari.

16.2.6 GEOTECHNICAL ANALYSIS

The geotechnical parameters for the proposed open pit at Pahtavaara are based on a previous trade-off study (SRK, 2020) that included a site visit to the historical open pit and underground mine by the SRK Geotechnical Engineer, and a desktop review of the supplied information. The observed and recommended parameters for the open pit have been summarised below.

- Overall slope angles at 120 m depth between 46 degrees (°) and 50°.
- Access via a single 20 m wide ramp.
- Berms based on existing pits and visual inspection.
- Batter angle of 65° to 75° based on existing slopes.

Based on these recommendations the pit optimisation in this study was run with a 45° Overall Slope Angle (OSA) and the pit design was constructed with a 10 m bench height, 70° batter angles, 5.4 m berms and 48° overall slope angle.

16.2.7 CUT-OFF GRADE

It is assumed that the plant at Pahtavaara will produce a high-grade concentrate that will be hauled by road to the main Ikkari plant for final processing to produce a Doré product. A NSR approach has therefore been adopted in the optimisation and an NSR value has been calculated for all mineralised blocks in the input model.

Based on the parameters shown in Table 16.3, the Mill Cut-off grade was estimated at between 0.42 and 0.45 g/t Au depending on plant throughput assumptions.

16.2.8 PIT OPTIMISATION RESULTS

Using the parameters listed in Table 16.2 the pit optimisation was run over a range of PF between 2% and 100% in steps of 2% and a graph of the cumulative Net Present Value (NPV) and Mineral Inventory was then generated between 10% and 100% (Figure 16.1). It should be noted that no pits shells were generated at a PF of less than 10% due to the relatively high initial waste stripping requirements to restart the mine and expose ore.

It was determined that greater than 98% of the maximum NPV, and 91% of the maximum Mineral Inventory, was attained at PF 0.86 and this has taken to be the selected pit limit for the purposes of open pit mine design and as the transition point between open pit and underground operations.

The cumulative grade tonnage curve and tonnage distribution by grade bin (Figure 16.2 and Figure 16.3) have been plotted for all blocks within the selected pit limit (PF 0.86) to demonstrate the variation in tonnage and grade with cut-off. At a cut-off grade of 0.40 to 0.50 g/t Au it was noted that there is:

- A near linear relationship between grade above cut-off and cut-off grade.
- A near linear relationship between tonnage above cut-off and cut-off grade.

It is not expected that the mineral inventory will be particularly sensitive to cut-off grade in the range 0.4 to 0.5 g/t Au and a high proportion (> 55%) of the inventory falls in the grade range between breakeven cut-off grade and 1.0 g/t Au, which is expected given that a significant proportion of the high-grade zones have already been mined out by the existing underground operations.

The optimum transition point between open pit mining and underground mining is typically dependent on several factors that include:

- Ore Mining Cost.
- Waste mining cost.
- Grade (allowing for dilution and mining recovery).

In this case, although the waste stripping ratio for the selected open pit is relatively high (>10:1) and the mined grade is relatively low (<1.5 g/t Au) the mining cost per recovered ounce for the selected open pit is judged to be less than a comparable LHOS operation, despite the higher grades achieved by the LHOS method.

Based on the results from NPVS, the breakeven LHOS mining cost, where the LHOS method becomes competitive, was estimated to be approximately 34 US $/t ore. This calculation considers the relative mining costs (open pit vs underground), stripping ratio for the open pit and the average mined grade from the open pit or underground operations. The analysis of the optimum transition point between open pit and underground is considered in greater detail in Section 16.3 of the report.

Figure 16.1 Pahtavaara Pit Optimisation – NPV vs Mineral Inventory

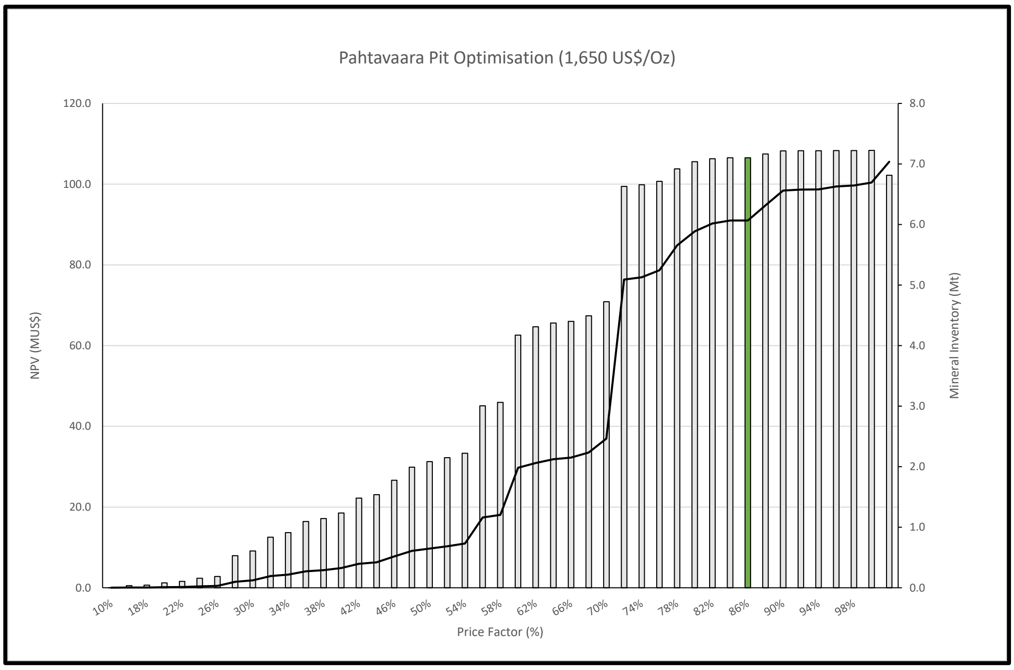


Figure 16.2 Pahtavaara Pit Optimisation – Grade Tonnage Curve

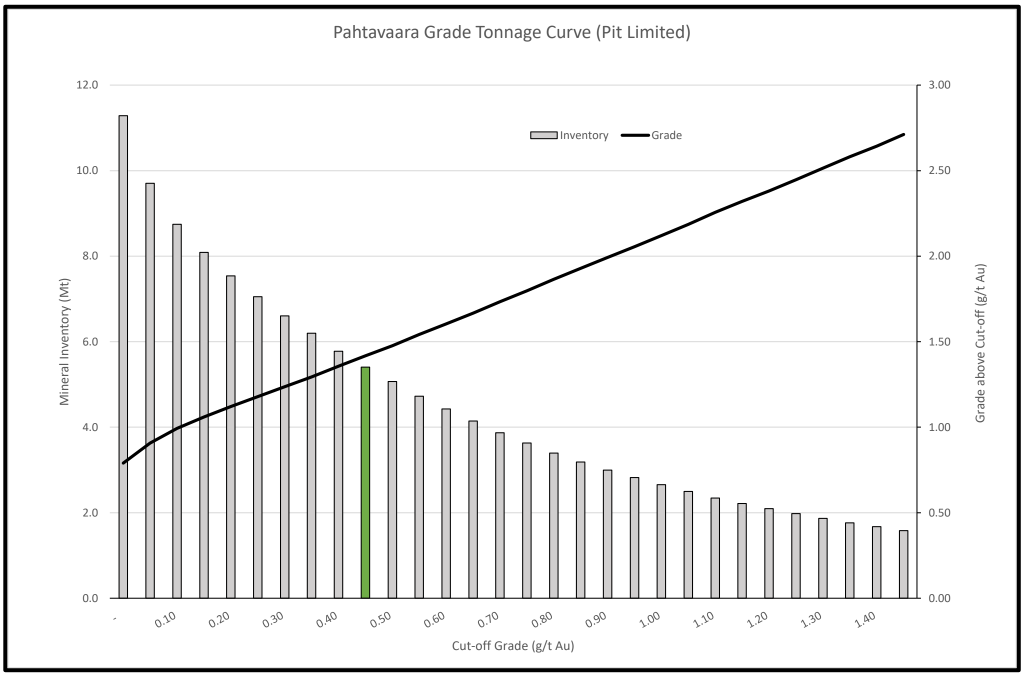
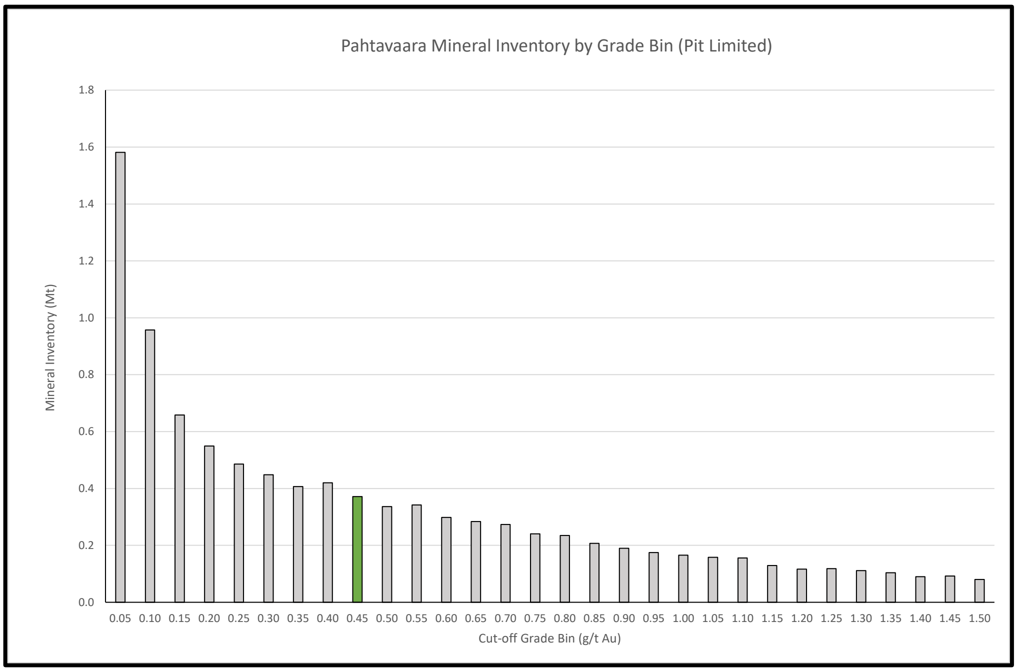


Figure 16.3 Pahtavaara Pit Optimisation – Mineral Inventory by Grade Bin



16.2.9 Optimisation Sensitivity Analysis

The sensitivity of the pit optimisation results from NPVS were tested against the assumed input parameters for OSA, mining costs and processing costs to identify the main drivers for NPV, mineral inventory and stripping ratio. The results are summarised in Table 16.4 and Table 16.5 at price factors of 1.0 and 0.86.

Table 16.4 Pit Optimisation Sensitivity Analysis – 1.0 Price Factor

	NPVS Results (Undiluted)						Variance to Base Case					
	NPV (MUS$)	Rock (Mt)	Waste (Mt)	Inventory (Mt)	Inventory (g/t Au)	Strip Ratio	NPV (%)	Rock (%)	Waste (%)	Inventory (%)	Grade (%)	Strip (%)
Base Case	108.3	75.2	68.5	6.7	1.41	10.2						
+2 degrees OSA	132.8	69.7	63.1	6.7	1.39	9.5	23%	-7%	-8%	-1%	-1%	-7%
-2 degrees OSA	112.0	79.1	72.4	6.7	1.39	10.9	3%	5%	6%	0%	-2%	6%
Mcost + 10%	108.9	72.0	65.5	6.5	1.39	10.1	1%	-4%	-4%	-3%	-1%	-2%
Mcost + 20%	96.3	68.6	62.3	6.3	1.40	9.9	-11%	-9%	-9%	-6%	-1%	-3%
Mcost - 10%	142.1	89.4	81.8	7.5	1.40	10.9	31%	19%	19%	13%	-1%	6%
Mcost - 20%	159.0	97.0	89.0	8.0	1.39	11.2	47%	29%	30%	19%	-1%	9%
Pcost + 10%	120.9	75.1	68.6	6.5	1.43	10.5	12%	0%	0%	-2%	2%	3%
Pcost + 20%	114.3	72.8	66.6	6.2	1.44	10.7	5%	-3%	-3%	-7%	2%	4%
Pcost - 10%	131.6	76.3	69.4	6.9	1.39	10.0	21%	2%	1%	4%	-1%	-2%
Pcost - 20%	134.6	76.6	69.4	7.1	1.37	9.8	24%	2%	1%	6%	-3%	-5%

Table 16.5 Pit Optimisation Sensitivity Analysis – 0.86 Price Factor

	NPVS Results (Undiluted)						Variance to Base Case					
	NPV (MUS$)	Rock (Mt)	Waste (Mt)	Inventory (Mt)	Inventory (g/t Au)	Strip Ratio	NPV (%)	Rock (%)	Waste (%)	Inventory (%)	Grade (%)	Strip (%)
Base Case	106.6	66.7	60.6	6.1	1.43	10.0						
+2 degrees OSA	131.0	60.0	54.0	6.1	1.41	8.9	23%	-10%	-11%	0%	-2%	-13%
-2 degrees OSA	112.0	67.6	61.7	5.9	1.40	10.5	5%	1%	2%	-3%	-2%	2%
Mcost + 10%	106.2	61.2	55.5	5.7	1.41	9.7	0%	-8%	-9%	-6%	-2%	-5%
Mcost + 20%	94.3	59.0	53.5	5.6	1.41	9.6	-12%	-12%	-12%	-8%	-1%	-6%
Mcost - 10%	140.6	76.5	69.7	6.8	1.42	10.3	32%	15%	15%	11%	-1%	1%
Mcost - 20%	157.1	83.4	76.3	7.1	1.42	10.7	47%	25%	26%	17%	-1%	5%
Pcost + 10%	118.7	66.7	60.8	5.9	1.45	10.2	11%	0%	0%	-2%	2%	0%
Pcost + 20%	111.8	63.8	58.1	5.6	1.46	10.3	5%	-4%	-4%	-7%	2%	1%
Pcost - 10%	130.1	70.9	64.3	6.6	1.40	9.8	22%	6%	6%	8%	-2%	-4%
Pcost - 20%	133.1	71.2	64.4	6.7	1.38	9.6	25%	7%	6%	11%	-4%	-6%

It was concluded that:

- The sensitivity results are broadly similar between the final pit limit (PF 1.0) and the selected pit limit (PF 0.86).
- The mineral inventory and waste stripping ratio are moderately sensitive to a decrease in mining cost or process cost but are less sensitive to an increase in mining or processing cost.
- The NPV is somewhat sensitive to changes in OSA, mining cost and processing with a two degree increase in OSA increasing the NPV by 23% and a 20% reduction in costs resulting in a 25% to 47% increase in NPV.

These relationships highlight the issues with low grade ore at Pahtavaara and the sensitivity to both cut-off grade and mining cost, whilst the waste stripping ratio (and NPV) are also sensitive to the slope parameters, with a two degree increase in overall slope angle resulting in a 23% increase in NPV and a 13% reduction in waste stripping ratio.

16.2.10 Open Pit Design

Using the PF 0.86 pit limit as a guide a series of three pushbacks were designed within this pit shell with the aim of accessing the highest grade first with the lowest stripping ratio. The main design parameters were as shown in Table 16.6.

Table 16.6 Pushback Design Parameters - Pahtavaara

Parameter	Units	Value	Comments
Bench Height	m	10.0	
Batter Angle	°	70.0	Based on existing faces
Berm Width	m	5.4	
Overall Slope Angle	°	48.0	
Ramp Width (dual lane)	m	20.0	Based on existing ramps
Ramp Width (single lane)	m	8.0	

Ramp Gradient	%	10.0	
Minimum cut-back width	m	35.0	Based on appropriate mining equipment

The layout of the ramp system considers the current location of the primary crusher and plant, the exits to the waste dumps (East and West) and the planned expansions to the waste dumps. The conceptual layout is shown in Figure 16.4.

The general layout of the three pushbacks (pit stages) are shown in Figure 16.5, Figure 16.6 and Figure 16.7. It should be noted that a 15 m wide geotechnical berm has been introduced to provide additional temporary access between stages and to control the overall slope angle.

It should also be noted that the small satellite pit to the west of the main pit (and north-west of the existing west dump) has not been included in the final pit design as this material only becomes economic at a PF of > 0.86. However, it may need to be included if the underground access is extended in this area as a means of developing the underground operations at Pahtavaara. This is because the current underground portals are within the existing open pit and access to them will be lost whilst developing the Stage 1 pit.

Figure 16.4 General Layout Concepts for Pahtavaara

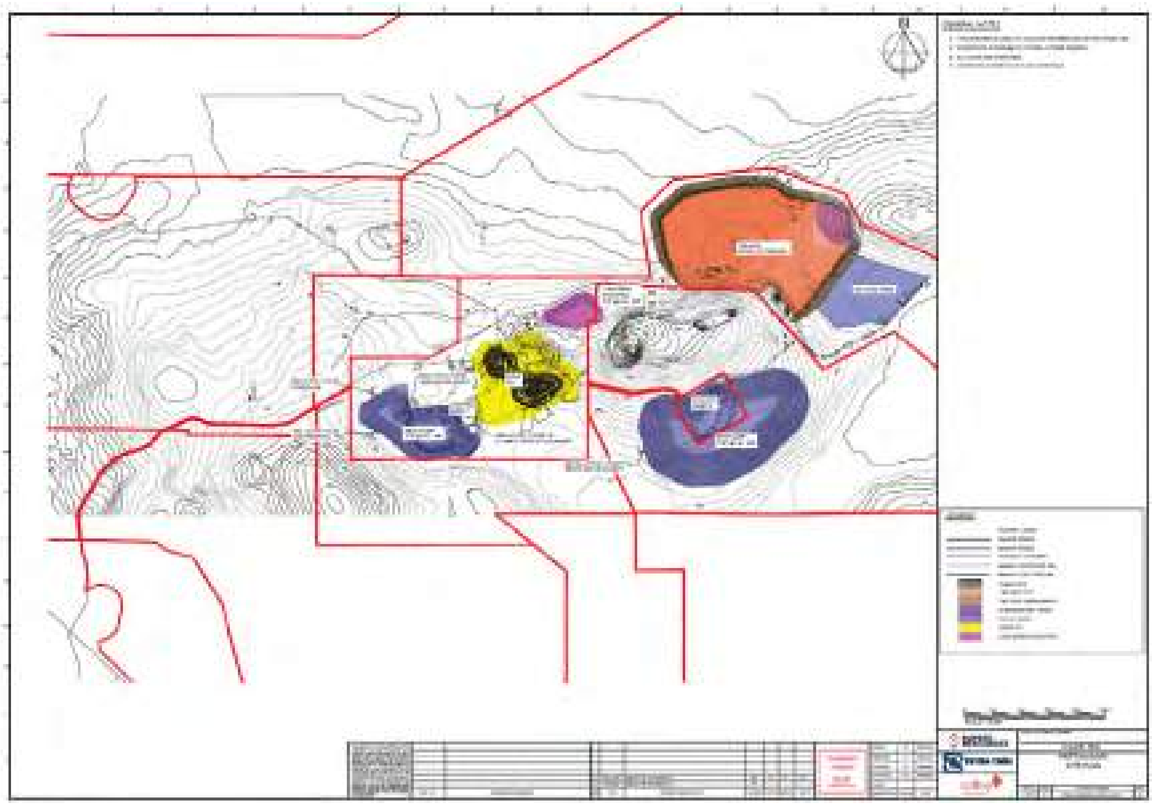


Figure 16.5 Pahtavaara Stage 1 Pit

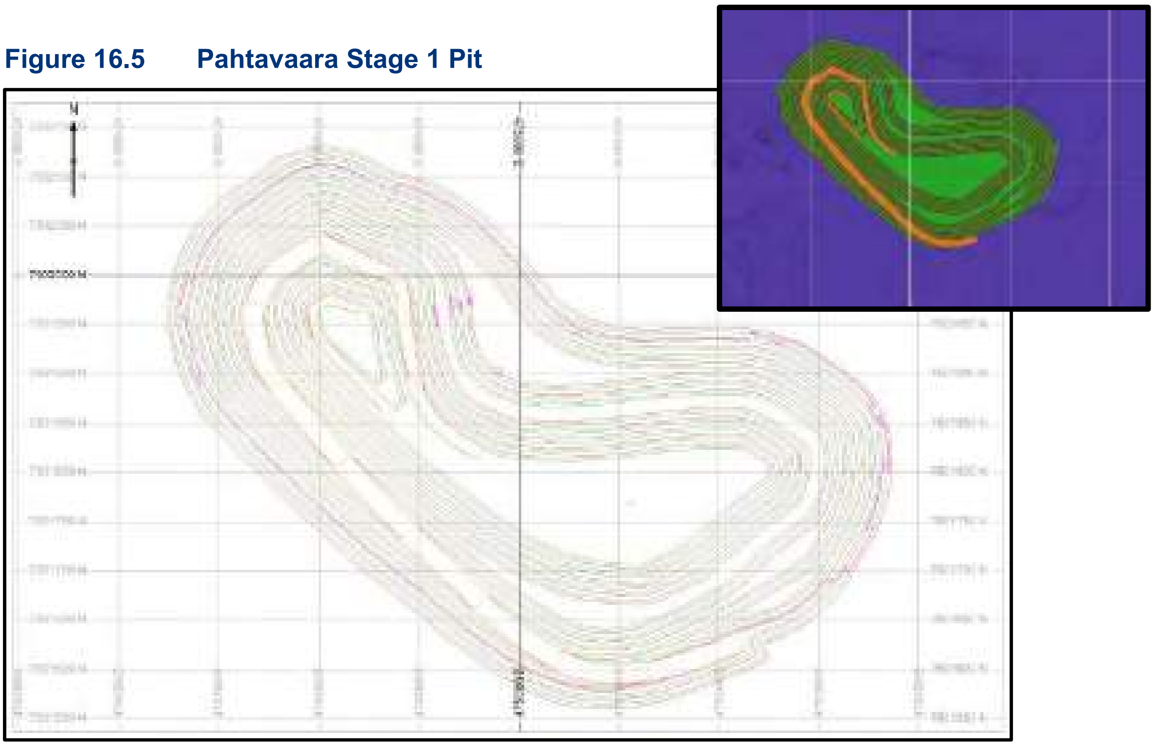


Figure 16.6 Pahtavaara Stage 2 Pit

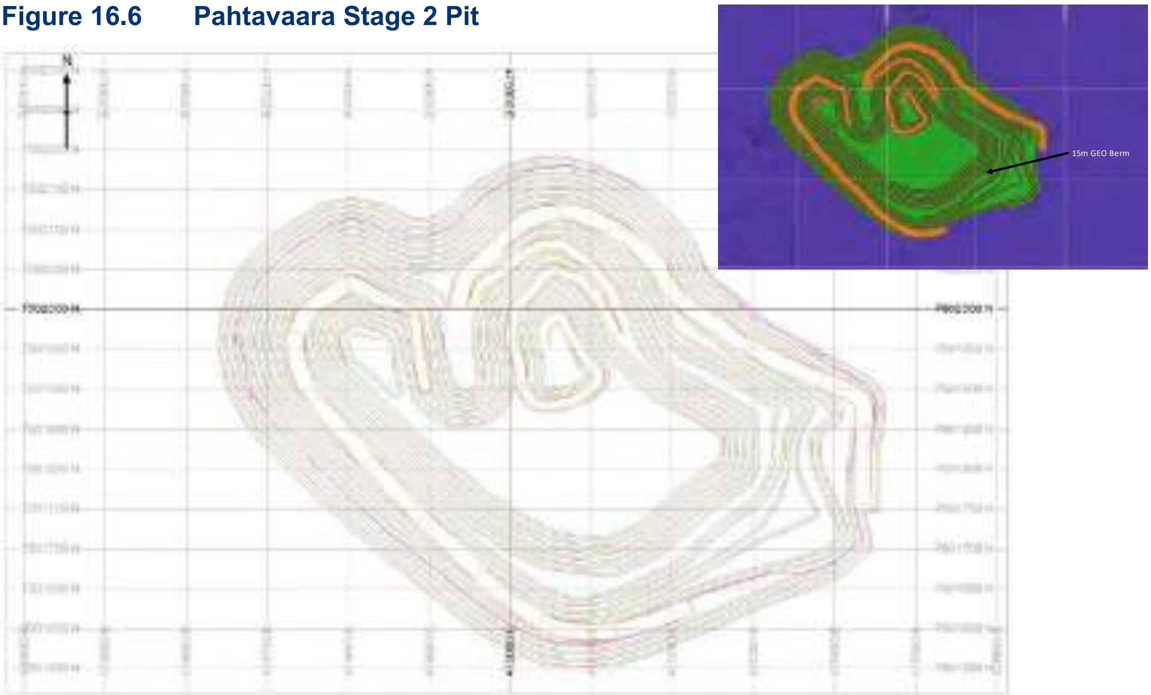
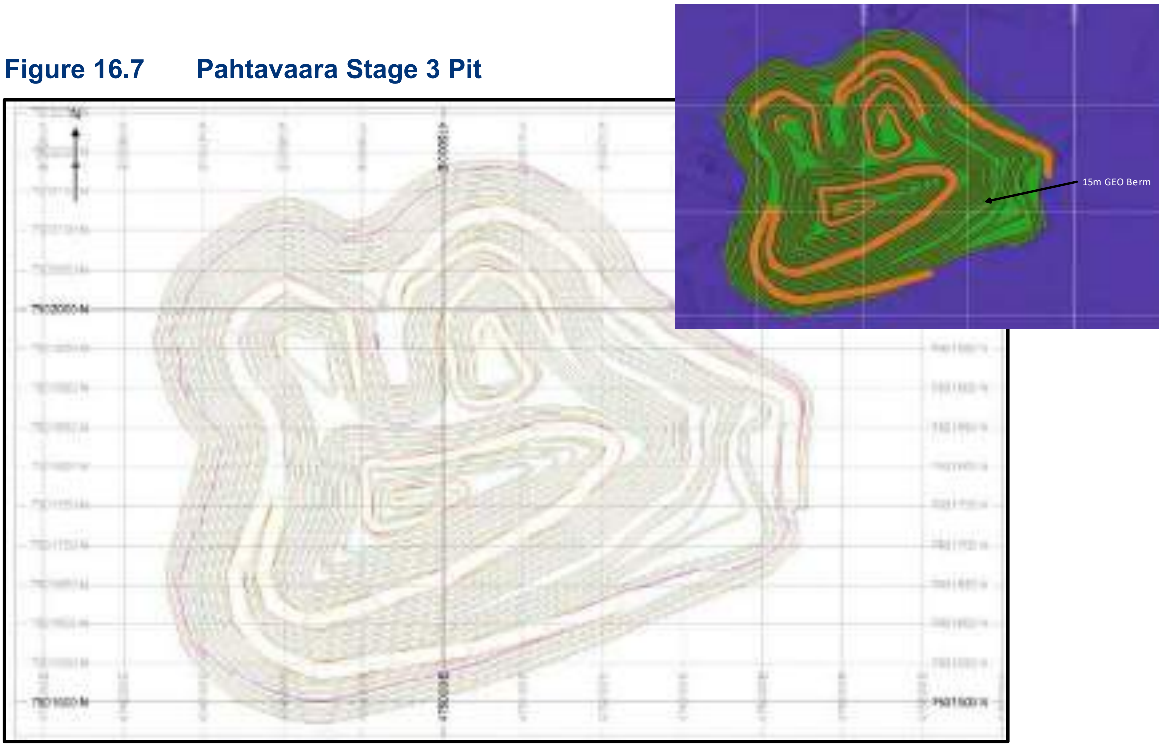


Figure 16.7 Pahtavaara Stage 3 Pit



The pit stages were evaluated using the 5 m x 5 m x 5 m regularised block model at a 0.43 g/t Au cut-off grade and the results are summarised in Table 16.7.

Table 16.7 Pahtavaara Mineral Inventory by Pit Stage

Stage	Waste	Mineral Inventory[2]			Strip Ratio
	(tonnes)[1]	(tonnes)[1]	(Kg Au)	(g/t Au)	
1	22,432,000	2,185,000	2,927	1.34	10.30
2	21,466,000	1,084,000	1,734	1.60	19.80
3	20,026,000	2,410,000	3,052	1.27	8.30
Total	**63,924,000**	**5,679,000**	**7,712**	**1.36**	**11.63**

Notes:
1. Tonnages have been rounded to the nearest thousand.
2. Mining recovery and dilution set to 100% and 0% respectively.

16.2.11 Mine Schedule

The mining schedule for Pahtavaara has been developed as part of a combined schedule for Pahtavaara and Ikkari. This assumes that a high-grade concentrate will be produced at Pahtavaara, which will hauled to the Ikkari plant for final processing.

During the PEA a decision was made to reduce the planned plant production rate at Pahtavaara from 1.0 Mt/a to 0.5 Mt/a during the initial start-up with an expansion to 0.75 Mt/a after 5 to 6 years. This decision was mainly driven by the current permitting limit of 0.5 Mt/a, but also the capital requirements to refit the plant to produce at a higher rate. It is not expected that this decision will have a significant impact on the mine design as the ramp width selected is suitable for a range of haul truck sizes.

A high grading strategy has been adopted for Pahtavaara whereby lower grade material is stockpiled for reclamation at the end of mine life for the underground operations. Besides increasing the feed grade to the plant, this strategy has the advantage that the overall mine life for Ikkari and Pahtavaara can be brought into line so that some of the open pit mining equipment from Ikkari could be transferred to Pahtavaara as the Ikkari open pit reaches the end of its life.

The low-grade stockpile also acts as a buffer during the period of ramping up production from the underground which mitigates the risk of a shortfall in plant feed due to unforeseen issues with starting up the underground.

A disadvantage of creating such a large low-grade stockpile is that there is a significant period of time at the end of the combined schedule when the only feed source to Ikkari is from reclaim and processing of the Pahtavaara stockpile. This significantly limits the production capacity of the Ikkari plant and may not prove to be feasible unless an alternative source of feed can be found to supplement the concentrate feed from Pahtavaara. Another alternative may be to find a buyer for the concentrate from Pahtavaara and to shut down the Ikkari plant at an appropriate time.

Details of the mine schedule for Pahtavaara can be found in Section 16.7.

16.2.12 Equipment Selection

It is expected that the open pit at Pahtavaara will be run as a contract miner operation and that the final choice of equipment will depend on the selected contractor. The equipment list shown below is intended as a guide to the requirements.

Table 16.8 Equipment Requirements at Pahtavaara

Open Pit Units	Description of Unit	Quarter4	Quarter8	Quarter12	Quarter16	Quarter20	Quarter24	Quarter28
Drill	*Sandvik Leopard DI550/Epiroc D60*	3	2	2	1	2	3	2
Blast Truck	*Volvo FM440 Anfo/Emulsion truck*	1	1	1	1	1	1	1
Excavator	*Cat 6018*	3	2	2	2	3	3	2
Truck	*Cat 777G*	8	7	5	4	8	10	7
FEL	*VolvoL350*	0	0	1	1	1	1	1
Large TD	*Cat D9*	3	2	2	2	3	3	2
RT	*Cat 844*	1	1	1	1	1	1	1
Small Grader	*Cat 14G*	1	1	1	1	1	1	1
Large Grader	*Cat 24M*	1	1	1	1	1	1	1
Small Water	*Volvo A45G*	2	2	2	2	2	2	2
U Excavator	*VOLVO EC350E*	1	1	1	1	1	1	1
Small FEL	*Volvo L120*	1	1	1	1	1	1	1
Service Truck	*various*	2	2	2	2	2	2	2
Pick-ups/LV	*Toyota*	9	9	9	9	9	9	9

A detailed Techno Economic Model (TEM) has been developed for Pahtavaara. The TEM was developed from first principles to estimate the capital and operating costs, showing the equipment and labour requirements over time.

The input schedule for the TEM is based on the Base Case scenario of a 3.5 Mt/a plant capacity at Ikkari and a 0.5 Mt/a plant capacity at Pahtavaara. Schedules and TEMs were also developed at 2.5 and 3.0 Mt/a as part of this study.

It should be noted that the design ramp width of 20 m at Pahtavaara is not sufficient to allow the use of a larger truck (e.g. Cat 785) and would need to be re-designed to 25 m. This would then allow compatibility between the fleet sizes for Ikkari and Pahtavaara.

16.3 Mining Methods – Ikkari Open Pit

16.3.1 Introduction

Ikkari is a greenfield site where it is proposed that the mine is developed initially as an open pit with a target production rate of 3.5 Mt/a of plant feed. As the open pit reaches the end of its life (approximately 10 years) the underground development will be completed so that the underground operation can continue for a further 13 years. The total mine life is therefore approximately 23 years with a proposed start date for pre-stripping in Q1 2027.

A suitable transition point between open pit operations and underground operations has been selected based on operating costs, as well as the limitation of the current exploration permit boundary, that impacts both surface and underground operations.

16.3.2 Open Pit Mining

The selected mining method for the Ikkari open pit is a conventional shovel and truck configuration. This will include 140 t medium sized haul trucks (e.g. Cat 785) matched with 300 t hydraulic excavators (e.g. Cat 6030).

Apart from a 40 m to 50 m covering of overburden and weathered material, the rock is relatively hard in this mine and will require drill & blast as well as pre-split on the final walls to maximise the overall slope angle.

16.3.3 Resource Model

The evaluation of Ikkari is based on the Resource model presented in Section 14 and summarised in Table 16.9.

Table 16.9 Ikkari Resource Model

Deposit	Resource Classification	Open Pit Status	Cut-off Grade (g/t)	Tonnage (t)	Grade Au (g/t)	Metal Content	
						Kg Au	Oz Au
Ikkari	Indicated	Open Pit	0.5	30,000,000	2.5	74,500	2,400,000
		Underground	1.0	16,500,000	2.4	39,800	1,280,000
		Total		***46,40,000***	***2.5***	***114,300***	***3,680,000***
	Inferred	Open Pit	0.5	3,100,000	1.5	4,800	150,000
		Underground	1.0	8,700,000	2.0	17,200	550,000
		Total		***11,800,000***	***1.9***	***39,300***	***1,260,000***
Ikkari Indicated and Inferred Total				**58,200,000**	**2.3**	**136,300**	**4,390,000**

The evaluation at this level of study includes Indicated and Inferred blocks and consequently it is not possible to define an Ore Reserve and the term Mineral Inventory has been used in this report.

16.3.4 Modifying Factors

Considering that the minimum sub-cell size in the Ikkari Resource model is relatively coarse at 10 m x 5 m x 5 m, the modifying factors for Ikkari have been estimated at 5% ore loss and 5% waste dilution. No attempt has been made to model the spatial distribution of the dilution factors.

16.3.5 Ikkari Mine Optimisation

The open pit mine optimisation for Ikkari was run using Datamine's NPVS software, which is based on the standard LG algorithm for pit optimisation. NPVS was run with a range of PF between 2% and 100% to give a series of nested pit shells that can be used to select the optimum pit limit for further design/evaluation work. The Input parameters are summarised in Table 16.10.

Table 16.10 Pit Optimisation Parameters for Ikkari

Parameter	Units	Value	Comments
Gold Price	US $/toz	1,650	3 year trailing average
Royalty	%	0.15	
Discount Rate	%	8.0	
Pit Exit/Crusher	RL	220	
Ref Waste Mining Cost	US $/t mined	1.85	
Ref Ore Mining Cost	US $/t ore	2.35	
Incremental Mining Cost	US $/t/10 m	0.04	
Mining Recovery	%	95.0	
Mining Dilution	%	5.0	
Grade Control	US $/t ore	0.50	
Stockpile Rehandle	US $/t ore	0.90	
Mine G&A	US $/t ore	2.80	
Process Cost	US $/dmt conc	9.16	
TC/RC	US $toz	2.50	
CDF	US $/t ore	0.80	
Rehabilitation	US $/t ore	0.80	
Au Recovery	%	95.0	
Au Payability	%	99.92	
Overall Slope Angle	Degrees	Varies	Defined by sector

The open pit limit was also constrained by the current exploration permit boundary. This proved to be a particularly important consideration in the pit optimisation and the choice of offset distance from this permit boundary is critical.

For this exercise an offset of 50 m was assumed in the pit optimisation in order to allow sufficient space for a security fence, safety berm and access road around the pit perimeter.

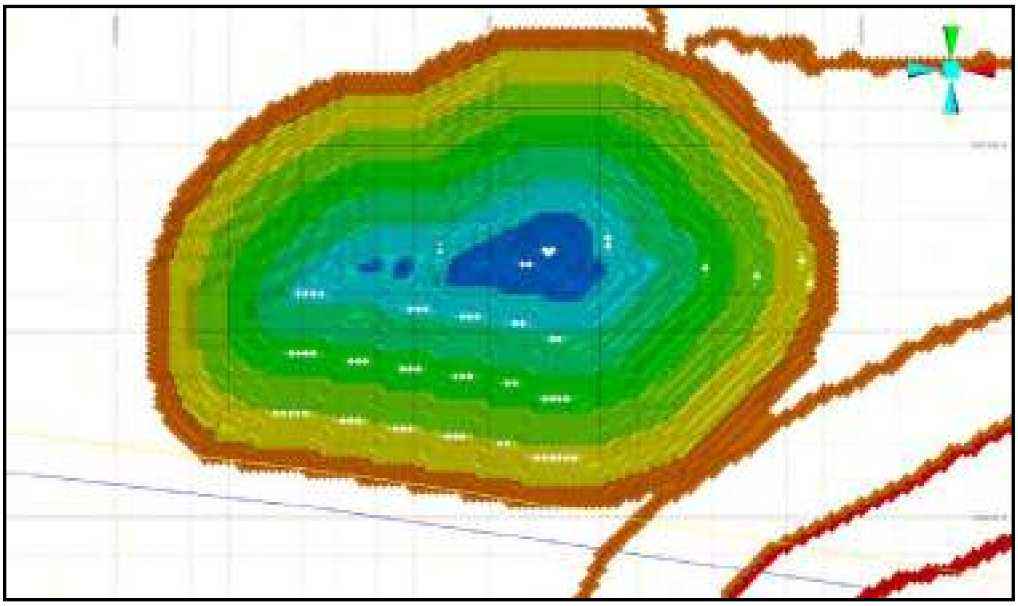
16.3.6 GEOTECHNICAL ANALYSIS

The geotechnical parameters for the proposed open pit at Ikkari are based on the geotechnical analysis of the proposed open pit to a maximum slope height of 280 m. The recommended parameters for the open pit are summarised below for a standard 10 m bench with double benching to 20 m.

Table 16.11 Recommended Overall Slope Angles - Ikkari

Domain	Sector Angle (°)		Ramp Width (m)	Geotech Berm (m)	Overall Angle (°)
	From	To			
NW	260	335	25.0	25.0	49.8
NE	335	95	25.0	25.0	49.8
SE & S	95	222	25.0	25.0	42.3
SW	222	260	25.0	25.0	43.2

Figure 16.8 Plan view of the Exploration Permit Boundary Constraint (Purple) and 50 m Offset (Yellow) in Relation to the Optimised Pit



Guidance was also given by SRK for:

- Berm width (10 m to 12 m) by sector and depth.
- Batter angle (65° to 80°) by sector and depth.
- Stack height (50 m to 100 m) by sector and depth.
- Inter Ramp Angle by sector and depth.

Based on these recommendations the pit optimisation in this study was run with the recommended OSA and the pit design was constructed with the detailed specifications for batter angle, catch berm width, maximum stack height, ramp width and Geotech berm width.

16.3.7 CUT-OFF GRADE

It is assumed that the plant at Ikkari will be fed from the Ikkari open pit plus a high-grade concentrate that will be hauled to Ikkari from Pahtavaara for final processing to produce a Doré product. A NSR approach has been adopted in the optimisation and an NSR value has been calculated for all mineralised blocks in the Ikkari input model.

Based on the parameters shown in Table 16.10, the Mill Cut-off grade was estimated at 0.33 g/t Au.

16.3.8 PIT OPTIMISATION RESULTS

Using the parameters listed in Table 16.10 the pit optimisation was run over a range of PF between 2% and 100% in steps of 2% and a graph of the cumulative NPV and Mineral Inventory was then generated between 10% and 100% (Figure 16.9).

It was determined that greater than 99% of the maximum NPV, and 95% of the maximum Mineral Inventory (ie PF 1.0 pit shell), was attained at PF 0.86 and this has taken to be the selected pit limit for the purposes of open pit mine design and as the transition point between open pit and underground operations.

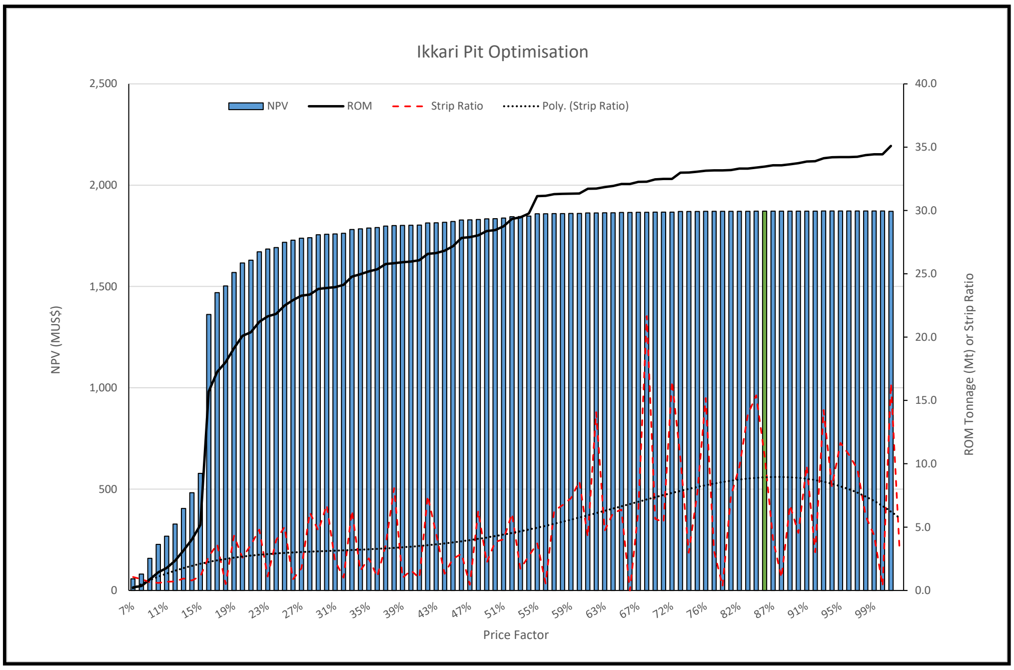
The optimum transition point between open pit mining and underground mining is typically dependent on a number of factors that include;

- Ore Mining Cost.
- Waste mining cost.
- Grade (allowing for dilution and mining recovery).

At Ikkari the main constraint on the open pit is the mining permit boundary and this defines the transition point between open pit and underground operations rather than economics. It can be seen from Figure 16.9 that the averaged strip ratio for Ikkari peaks at approximately 8:1. Consequently the maximum total ore mining cost (including waste) for the open pit will not exceed 20 US $/t, which is less than the estimated total ore mining cost of 21.6 US $/t ore for the underground Sub-level Caving (SLC) method at 3.5 Mt/a.

If the constraint on the open pit limit can be lifted the size and depth of the open pit will increase and the transition point to underground mining will change. This represents upside potential for the project in terms of NPV and resource recovery.

Figure 16.9 Pahtavaara Pit Optimisation – Cumulative NPV vs Cumulative Mineral Inventory



16.3.9 Pit Optimisation Sensitivity Analysis

The sensitivity of the pit optimisation results from NPVS were tested against the assumed input parameters for OSA, Mining Costs, Processing costs and offset from the mining permit boundary to identify the main drivers for NPV, Mineral Inventory and stripping ratio. The results are summarised in Table 16.12 and Table 16.13 at a Price Factors of 1.0 for a pit limit with and without the constraint of minimum pit bottom width.

Table 16.12 Pit Optimisation Sensitivity Analysis – 1.0 Price Factor

	NPVS Results (Undiluted)						Variance to Base Case					
	NPV (MUS$)	Rock (Mt)	Waste (Mt)	Inventory (Mt)	Inventory (g/t Au)	Strip Ratio	NPV (%)	Rock (%)	Waste (%)	Inventory (%)	Grade (%)	Strip (%)
Base Case	1,871	147.5	113.1	34.4	2.23	3.3						
+2 degrees OSA	1,971	168.9	129.5	39.4	2.15	3.3	5%	14%	14%	14%	-4%	0%
+4 degrees OSA	2,028	170.7	129.0	41.7	2.12	3.1	8%	16%	14%	21%	-5%	-6%
-2 degrees OSA	1,787	135.0	103.3	31.6	2.26	3.3	-4%	-9%	-9%	-8%	2%	-1%
-4 degrees OSA	1,666	126.0	97.5	28.6	2.27	3.4	-11%	-15%	-14%	-17%	2%	4%
Mcost + 10%	1,860	139.8	105.6	34.3	2.23	3.1	-1%	-5%	-7%	0%	0%	-6%
Mcost + 20%	1,836	138.6	104.5	34.1	2.24	3.1	-2%	-6%	-8%	-1%	1%	-7%
Mcost - 10%	1,886	152.2	117.0	35.2	2.20	3.3	1%	3%	3%	2%	-1%	1%
Mcost - 20%	1,911	157.7	122.0	35.7	2.18	3.4	2%	7%	8%	4%	-2%	4%
Pcost + 10%	1,858	143.0	108.8	34.1	2.25	3.2	-1%	-3%	-4%	-1%	1%	-3%
Pcost + 20%	1,840	139.3	105.9	33.4	2.28	3.2	-2%	-6%	-6%	-3%	2%	-3%
Pcost - 10%	1,887	149.0	113.6	35.4	2.19	3.2	1%	1%	0%	3%	-2%	-2%
Pcost - 20%	1,899	154.4	118.3	36.1	2.16	3.3	2%	5%	5%	5%	-3%	0%
60m Offset	1,822	135.3	103.1	32.2	2.26	3.2	-3%	-8%	-9%	-6%	1%	-3%
70m Offset	1,762	127.2	97.1	30.2	2.28	3.2	-6%	-14%	-14%	-12%	2%	-2%
80m Offset	1,692	120.5	92.2	28.3	2.29	3.3	-10%	-18%	-18%	-18%	3%	-1%

Table 16.13 Pit Optimisation Sensitivity Analysis – 1.0 Price Factor & 25m Pit Bottom

	NPVS Results (Undiluted)						Variance to Base Case					
	NPV (MUS$)	Rock (Mt)	Waste (Mt)	Inventory (Mt)	Inventory (g/t Au)	Strip Ratio	NPV (%)	Rock (%)	Waste (%)	Inventory (%)	Grade (%)	Strip (%)
Base Case	1,870	159.1	124.0	35.1	2.21	3.5						
+2 degrees OSA	1,970	175.4	135.6	39.8	2.13	3.4	5%	10%	9%	13%	-3%	4%
+4 degrees OSA	2,028	171.5	129.7	41.8	2.12	3.1	8%	8%	5%	19%	-4%	-6%
-2 degrees OSA	1,786	141.4	109.3	32.1	2.24	3.4	-4%	-11%	-12%	-8%	2%	4%
-4 degrees OSA	1,665	129.4	100.7	28.7	2.27	3.5	-11%	-19%	-19%	-18%	3%	7%
Mcost + 10%	1,859	147.6	112.9	34.7	2.22	3.3	-1%	-7%	-9%	-1%	1%	-1%
Mcost + 20%	1,836	139.5	105.4	34.1	2.24	3.1	-2%	-12%	-15%	-3%	2%	-6%
Mcost - 10%	1,885	159.1	123.5	35.6	2.19	3.5	1%	0%	0%	2%	-1%	5%
Mcost - 20%	1,910	164.1	128.1	36.0	2.17	3.6	2%	3%	3%	3%	-1%	8%
Pcost + 10%	1,856	152.9	118.2	34.8	2.22	3.4	-1%	-4%	-5%	-1%	1%	3%
Pcost + 20%	1,839	148.5	114.5	34.0	2.26	3.4	-2%	-7%	-8%	-3%	3%	3%
Pcost - 10%	1,885	156.6	120.9	35.7	2.18	3.4	1%	-2%	-3%	2%	-1%	3%
Pcost - 20%	1,899	158.7	122.4	36.3	2.15	3.4	2%	0%	-1%	4%	-3%	2%
60m Offset	1,821	143.0	110.3	32.7	2.24	3.4	-3%	-10%	-11%	-7%	2%	3%
70m Offset	1,761	133.2	102.7	30.5	2.27	3.4	-6%	-16%	-17%	-13%	3%	3%
80m Offset	1,691	124.7	96.2	28.5	2.28	3.4	-10%	-22%	-22%	-19%	3%	3%

Note: Although the sensitivity was run at 2.5 Mt/a and hence the absolute NPV values will be slightly lower than if they were run at 3.5 Mt/a. The variances between cases will not change significantly.

It was concluded that:

- The pit limit is primarily controlled by the mining permit boundary and consequently changes to price and costs have limited impact.
- The offset from the permit boundary, or the assumed overall slope angles, have a more significant impact (>10%) on the NPV.
- Imposing a 25 m minimum pit bottom width on the optimisation marginally increases the mineral inventory by 0.7 Mt but increases the waste striping by almost 7 Mt. This is in line with the comments over the increasing stripping ratio as you approach the pit limit.

These relationships highlight that Ikkari is not particularly sensitivity to cut-off grade (i.e. changes in price and costs) but is moderately sensitive to the slope parameters and offset distance from the permit boundary.

16.3.10 Open Pit Design

Using the selected pit limit as a guide a series of three pushbacks (stages) were designed within this pit shell with the aim of accessing the highest grade first with the lowest stripping ratio. The main design parameters are shown in Table 16.14

Table 16.14 Stage Design Parameters – Ikkari

Parameter	Units	Value	Comments
Bench Height	m	10.0	Doubled up to 20 m where possible
Batter Angle	°	65 – 80	Varies by sector and depth
Berm Width	m	10 – 12	Varies by sector and depth
Overall Slope Angle	°	43 - 50	Varies by sector and depth
Ramp Width (dual lane)	m	25.0	Sufficient for Cat 785

Ramp Width (single lane)	m	15.0	
Ramp Gradient	%	10.0	
Minimum cut-back width	m	35.0	Based on appropriate mining equipment

The layout of the ramp system considers the proposed location of the primary crusher and plant and proposed location of the waste dumps. The conceptual layout is shown in Figure 16.10.

The general layout of the three pushbacks (pit stages) are shown in Figure 16.11, Figure 16.12 and Figure 16.13. The third pushback has been split into two (Figure 16.14) to smooth out the waste stripping requirements and the grade profile.

The access decline to the underground operations is planned to be the east of the pit exit. This means that the development of the underground is not constrained by the open pit operations, and they can be run in parallel if required.

Figure 16.10 General Layout Concepts for Ikkari

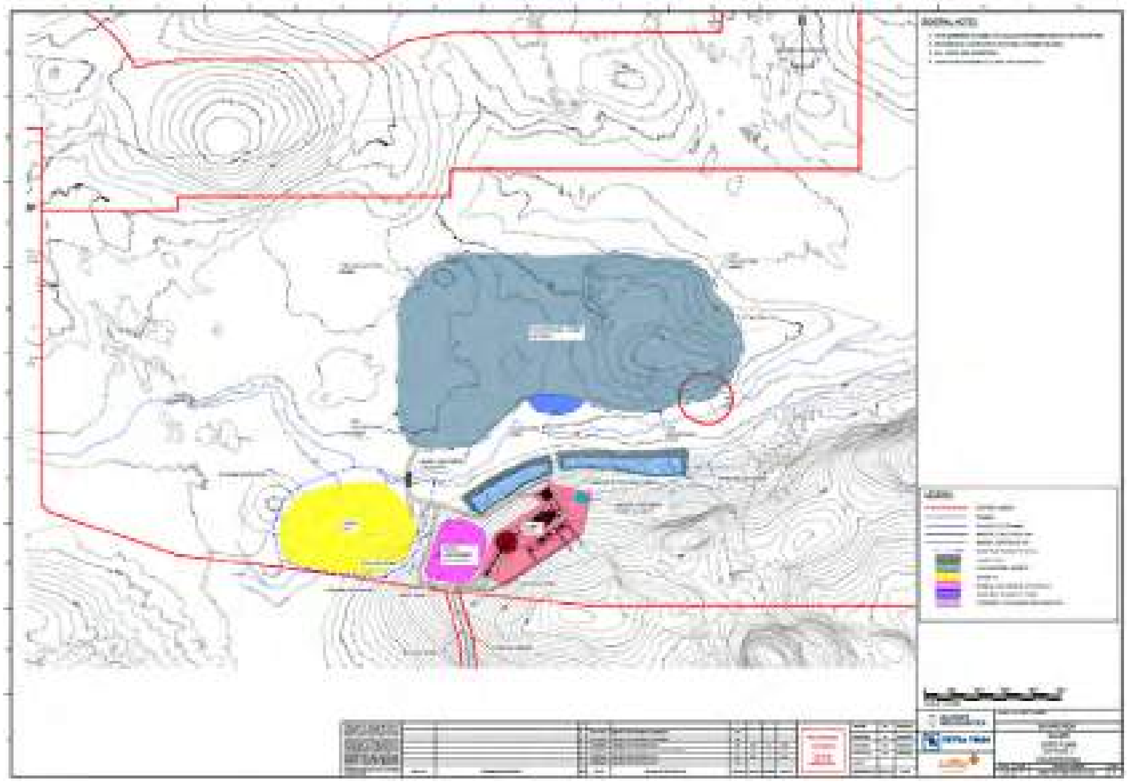


Figure 16.11 Ikkari Stage 1 Pit

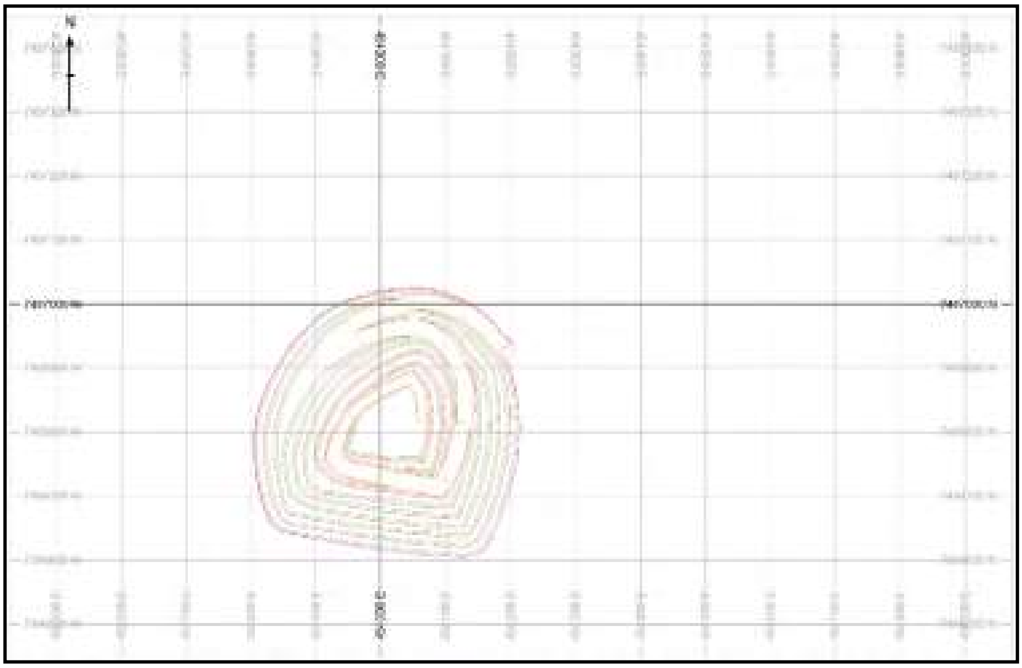


Figure 16.12 Ikkari Stage 2 Pit

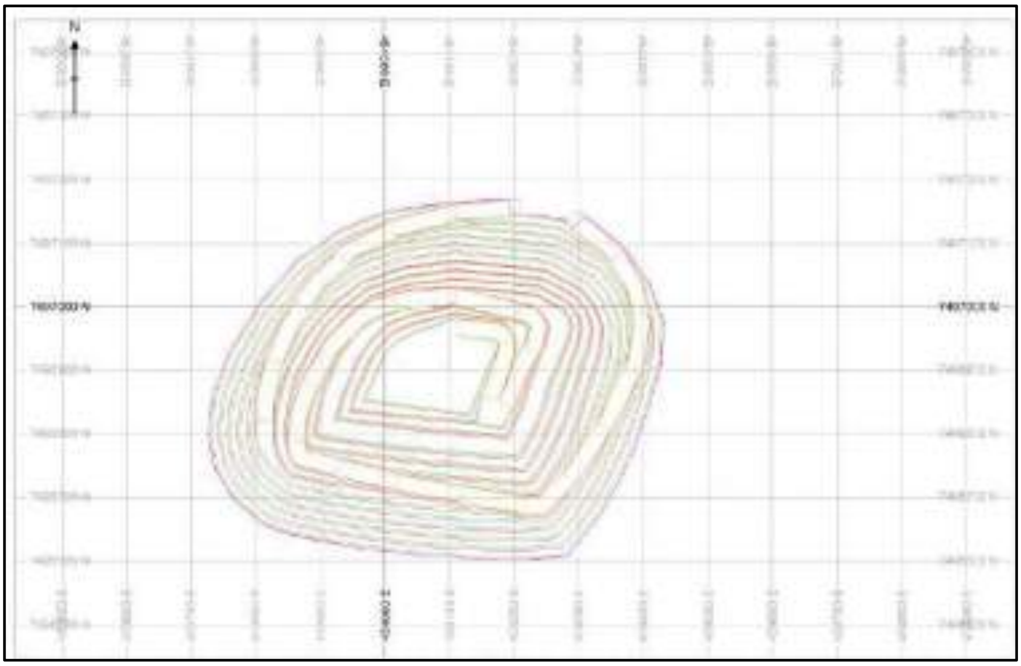


Figure 16.13 Ikkari Stage 3a and 3b Pit

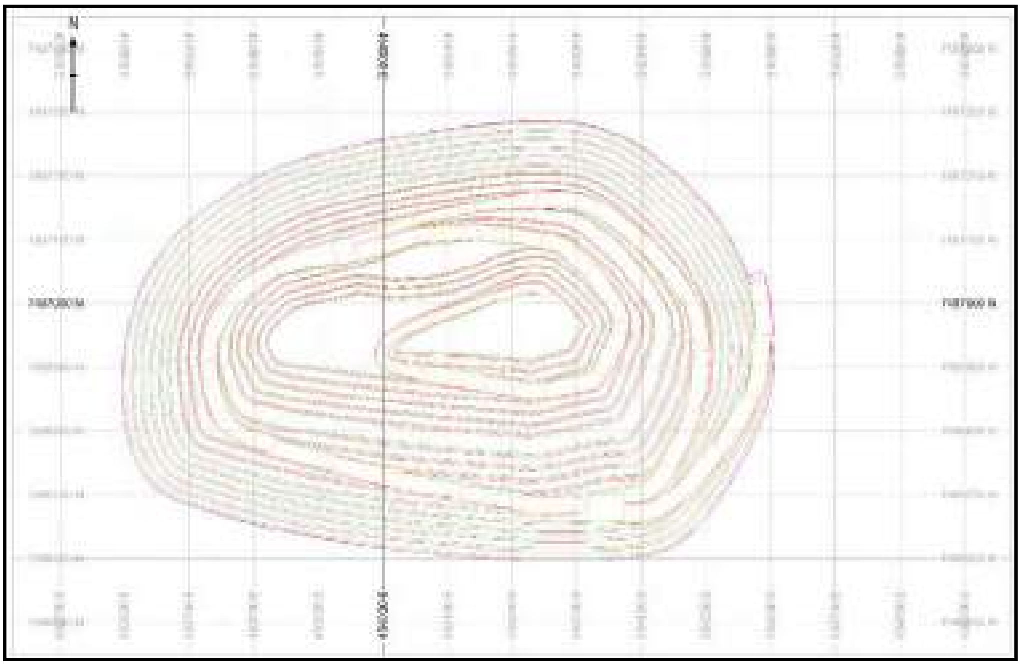
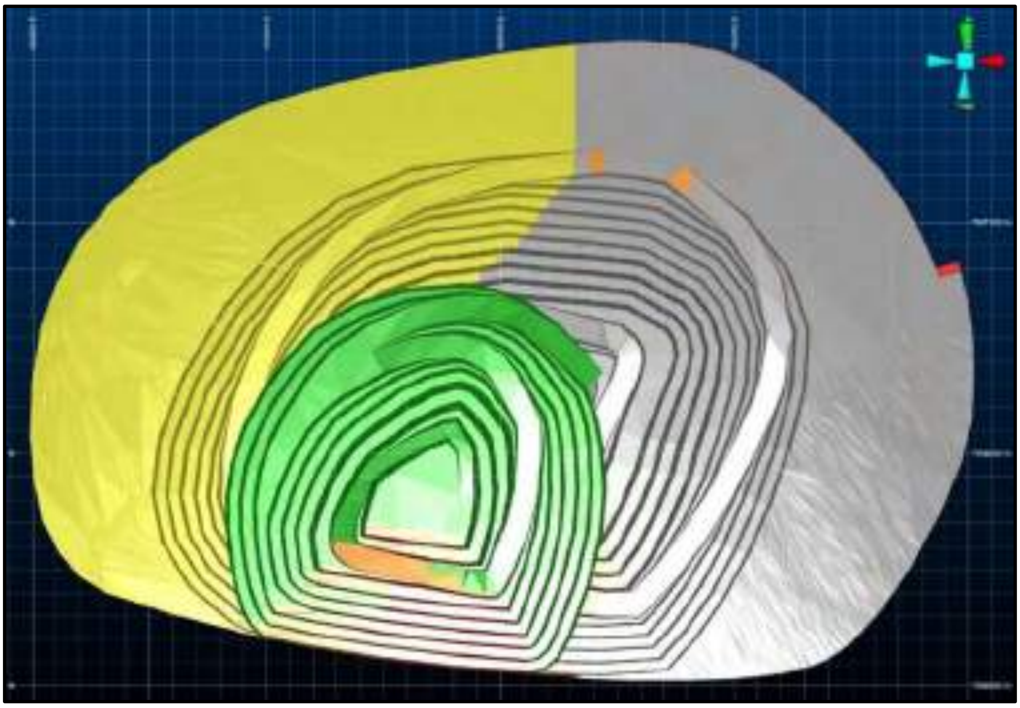


Figure 16.14 Ikkari Pit Stages 1 to 3 with subdivision of Stage 3 into 3a (yellow) and 3b (grey)



The pit stages were evaluated against the Resource block model at a 0.33 g/t Au cut-off grade. The results are summarised in Table 16.15.

Table 16.15 Ikkari Mineral Inventory by Pit Stage

Stage	Waste	Mineral Inventory[2]			Strip Ratio
	(tonnes)[1]	(tonnes)[1]	(Kg Au)	(g/t Au)	
1	11,145,000	7,139,000	17,901	2.51	1.6
2	37,030,000	13,956,000	31,523	2.26	2.7
3a	27,643,000	5,525,000	4,948	0.90	5.0
3b	56,623,000	10,361,000	22,703	2.19	5.5
Total	**132,441,000**	**36,981,000**	**77,075**	**2.08**	**3.6**

Notes:
1. Tonnages have been rounded
2. Mining recovery and dilution set to 95% and 5% respectively

16.3.11 MINE SCHEDULE

The mining schedule for Ikkari has been developed as part of a combined schedule for Pahtavaara and Ikkari. This assumes that a high-grade concentrate will be produced at Pahtavaara, which will hauled to the Ikkari plant for final processing.

During the PEA a decision was made to increase the planned plant production rate at Ikkari from 2.5 Mt/a to 3.5 Mt/a. This decision was mainly driven by the choice of SLC as the preferred underground mining method. Previous studies with the LHOS method had been limited to 3.0 Mt/a as a practical constraint on mine productivity.

The increase in production to 3.5 Mt/a has also meant that the haul truck size can be increased to 140 tonne units; A trade-off between the smaller 90 tonne trucks and the larger 140 tonne trucks demonstrated the economic advantage in capital and operating costs of increasing the size of both the trucks and the excavators.

A high grading strategy has been adopted for Ikkari whereby lower grade material is stockpiled for reclamation at the end of mine life for the underground operations. Besides increasing the feed grade to the plant, this strategy has the advantage that the low-grade stockpile acts as a buffer during the period of ramping up production from the underground, which mitigates the risk of a shortfall in plant feed due to unforeseen issues with starting up the underground.

Details of the mine schedule for Ikkari can be found in Section 16.7.

16.3.12 EQUIPMENT SELECTION

It is expected that the open pit at Ikkari will be run as a contract miner operation and that the final choice of equipment will depend on the selected contractor. The equipment list shown in Table 16.16 is intended as a guide to the requirements.

Table 16.16 Equipment Requirements at Ikkari

Open Pit Units	Description of Unit	Quarter4	Quarter8	Quarter12	Quarter16	Quarter20	Quarter24	Quarter28	Quarter40
Dril	*Sandvik DI550/Epiroc D60*	2	4	4	4	4	4	6	2
Blast Truck	*Volvo FM440 truck*	1	1	1	1	1	1	1	1
Excavator	*Cat 6030*	1	2	3	3	2	2	4	1
Truck	*Cat 785D*	2	6	7	8	8	6	12	4
Large TD	*Cat D9*	2	2	3	3	2	2	4	2
RT	*Cat 844*	1	1	1	1	1	1	1	1
Small Grader	*Cat 14G*	1	1	1	1	1	1	1	1
Large Grader	*Cat 24M*	2	2	2	2	2	2	2	2
Small Water	*Volvo A45G*	2	2	2	2	2	2	2	2
U Excavator	*VOLVO EC350E*	2	2	2	2	2	2	2	2
Small FEL	*Volvo L120*	2	2	2	2	2	2	2	2
Service Truck	*various*	6	6	6	6	6	6	6	6
Pick-ups/LV	*Toyota*	14	14	14	14	14	14	14	14

A detailed TEM has been developed for Ikkari. The TEM was developed from first principles to estimate the capital and operating costs, showing the equipment and labour requirements over time.

The input schedule for the TEM is based on the Base Case scenario of a 3.5 Mt/a plant capacity at Ikkari and a 0.5 Mt/a plant capacity at Pahtavaara. Schedules and TEMs were also developed at 2.5 Mt/a and 3.0 Mt/a as part of this study.

It should be noted that the design ramp width at Ikkari is sufficient to allow the use of a smaller truck 90 t (e.g. Cat 777) or the larger 140 t truck (e.g. Cat 785).

16.4 Mining Method - Open Pit to Underground Selection

16.4.1 Pahtavaara

Pahtavaara was historically mined via underground LHOS methods. This study identified the ability to mine and exploit Pahtavaara via open pit means then progressing underground.

There are several practical challenges that can be foreseen with this approach, yet the optimisation analyses pointed to this being the best possible opportunity for Pahtavaara to potentially yield positive economic results.

Typical challenges with an historic underground mine being mined via open pit truck and excavator and then progressing with an in-pit portal and continuing with underground mining at some advanced period or stage of the mine life are:

- Mining into historic and current underground voids poses a challenge not only from a heavy machinery point of view, but also explosive control and rock fragmentation.
- When mining into underground voids within the open pit and then later re-establishing underground operations (particularly in a reasonably wet mining environment), there could be water ingress- and underground excavation stability challenges.
- Access development and ore recovery could also be affected with this approach and therefore mine productivity and ultimately mining unit costs might be hard to optimise.

Finding the ultimate open pit to underground cutover point (best possible transition) is generally not straight forward. A basic open pit optimisation yielding a series of nested pits on revenue factor increments are a good industry approach, however, selecting where the open pit should be stopped and where the underground operations might commence could be somewhat subjective or operating and capital cost affected. Delaying underground capital might be a company objective but counter intuitive to operating margins.

The open pit optimisation at Pahtavaara considering 1% revenue increments whilst ultimate testing where the operating cost of an underground mine and underground operating margin might be better than that of the open pit was the method used for Pahtavaara. The methodology ultimately adopted to find that best open pit to underground transition point were to:

- Obtain the lowest Revenue Factor (RF) Pit shell where underground would not have any stopes formed – RF 60% meaning 60% of the gold price or of the anticipated metal recovery where underground mining would not work. The first set of RFs where underground mining could be considered is from a +- 70% RF. The RF 0.70 pit was therefore the first pit shell compared to a possible underground alternative for Pahtavaara.
- To fully understand the impacts of the open pit performance (economics) and compare those to the underground economics, two economic comparisons would be developed. The first is simply an Earnings Before Interest, Taxes, Depreciation, and Amortisation per tonne (EBITDA/t) ore (for stopes inside the respective RF open pit shells) versus the open pit material in these pit shells, the second would be to use the open pit NPV value as a discounted cashflow value and divide it by the ore tonnes (for the open pit) and to develop a conceptual underground schedule with high-level assumptions (simply in Excel) to develop a discounted cashflow basis for an underground option or stopes within these same RF open pit shells.
- The other key economic check (if one should consider continuing with an open pit or perhaps consider going underground) is the incremental unit cost component per ore tonne (from a low RF to a high RF pit shell). If the incremental open pit cost exceeds the “marginal” underground cost (stoping only cost), then there would be an economic foundation argument to not consider a further open pit pushback yet to pursue the remaining resource via underground mining methods.
- Other Risks should also be considered when trying to decide where the best open pit to underground cut-over point might be, for example:
  - Surface impoundment restrictions.
  - Environmental limitations.
  - Water/hydrology.
  - Geotechnical and stability parameters.
  - Productivity.
  - Skills and availability of skilled personnel for the respective mining methods and approach.
  - Execution risk and difficulty.

Tetra Tech considered the RF 0.70 to RF 0.90 pit shells to test when or if Pahtavaara should be mined from underground (considering a basic economic evaluation review of two competing exploitation strategies).

Tetra Tech recommends that, provided there are no serious issues, constraints, or limitations with an open pit exploitation strategy at Pahtavaara, that open pit mining should be considered at least up to an RF 0.90 pit shell. The potential for an underground option therefore should be considered from the potential stopes forming inside the RF 0.70, RF 0.80 and RF 0.90 pit shells.

Tetra Tech would not select the RF1.0 pit shell for Pahtavaara (based on the latest optimisation results) and therefore limited the open pit versus underground checks to the RF 0.90 pit shell.

Figure 16.15, Figure 16.16, Figure 16.17 and Figure 16.18 depict the material (ore) where an open pit could compete with the underground option at Pahtavaara.

Figure 16.15 Stopes from Minable Shape Optimiser (MSO) shown inside and below the RF70 Pit Shell for Pahtavaara

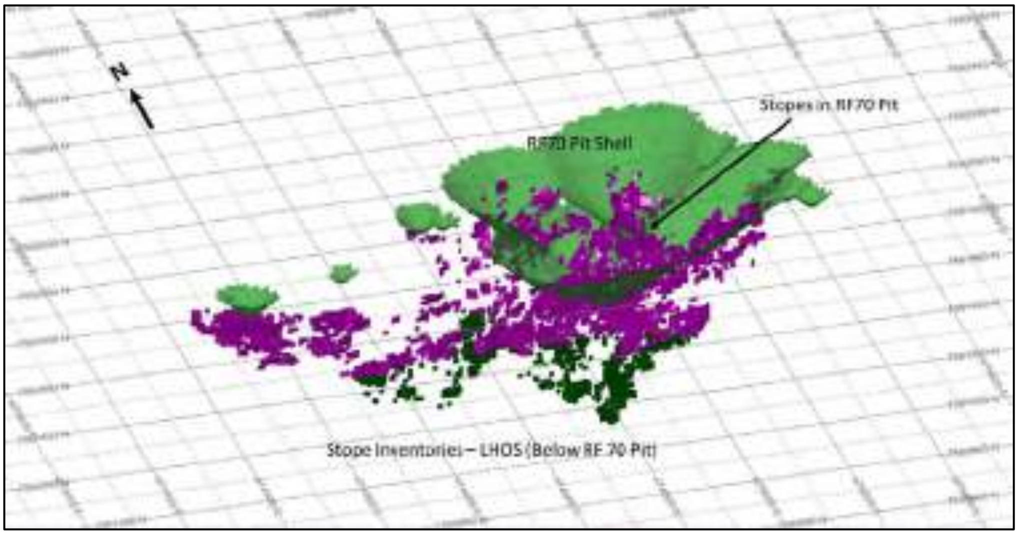


Figure 16.16 Stopes from MSP shown inside and below the RF80 Pit Shell for Pahtavaara

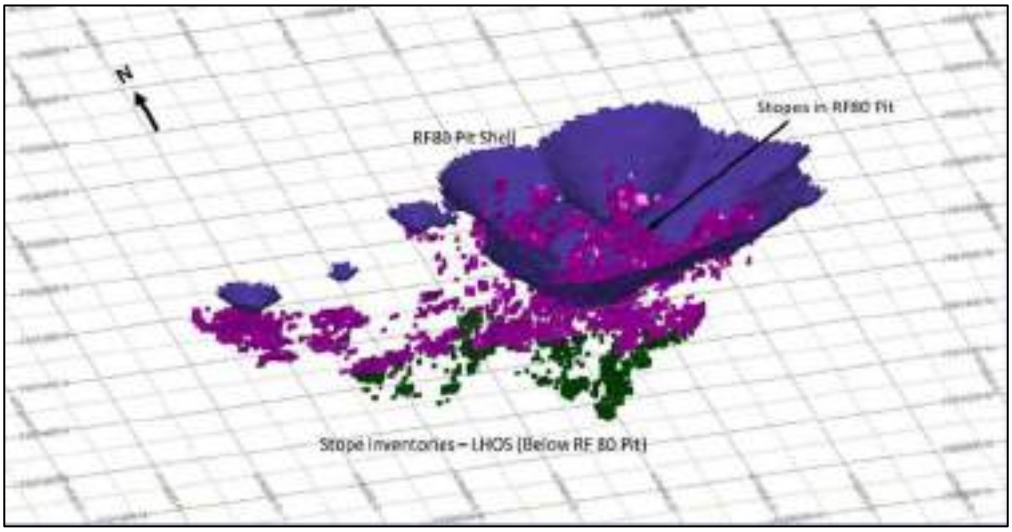
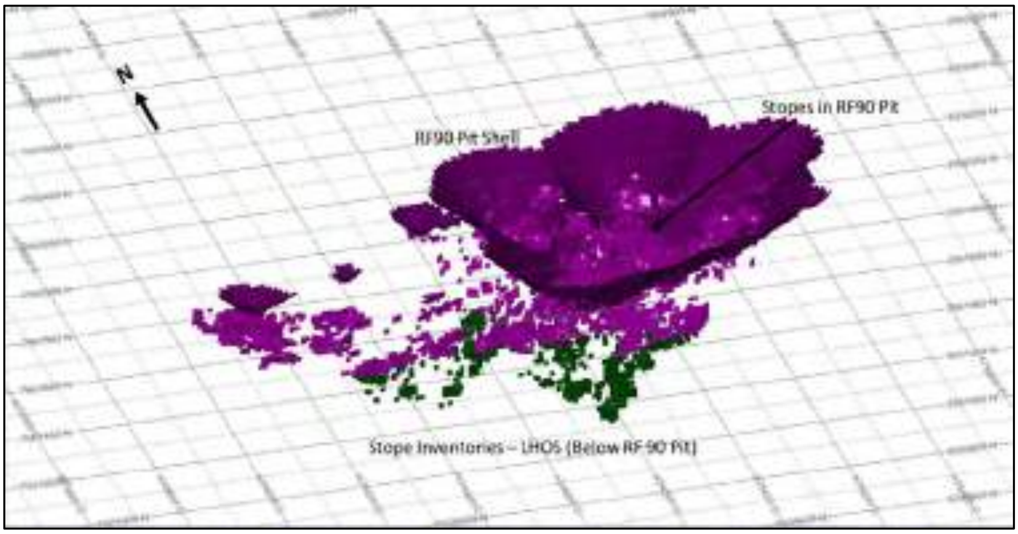


Figure 16.17 Stopes from MSO Shown Inside and Below the RF90 Pit Shell for Pahtavaara



From obtaining the incremental and cumulative NPV values within the open pit optimisation results and then developing indicative mine schedules for the stope inventories within each of these pit shells, the underground economic results were compared to those from the pit shell results (for each of these RF steps).

At an RF of 0.70 Pahtavaara yielded no MSO stope shapes. Therefore, it is safe to assume that from RF 0.70 to RF 0.90 the pit shell delta volumes, tonnages and costs of the open pit might compete with an underground alternative. Table 16.17 depicts the typical volume/tonnage and cost changes and how these compare with an underground option.

Table 16.17 Volume/Tonnage Cost Changes Comparison

Delta - RF 0.70 to RF 0.90	Units	Value
Delta Stope Ore (RF 0.70 to RF 0.90)	tonnes	361,902
Delta Open Pit Ore	tonnes	1,648,469
Delta Open Pit Waste (RF 0.70 to RF0.90)	tonnes	21,404,084
Delta SR (Open Pit)	Ratio	12.98
Delta Open pit cost	$/t ore	25.97
Underground mining Cost	$/t ore	+45

Pahtavaara will be a low productivity/low ore tonnage underground operation, therefore it is conceivable that this could be a far more expensive underground operating cost model. Provided Pahtavaara has sufficient impoundment facilities for the open pit waste rock, it is recommended that an underground option should only follow an open pit of at least an RF 0.90 size or even bigger.

16.4.2 IKKARI

Ikkari is somewhat simpler when considering an open pit and underground operation. The lowest underground mining cost option would be a sub-level cave operation targeting 3.5 Mt/a. The underground mining cost for such an operation would be approximately US $25/t ore to US $30/t ore. The Ikkari open pit mining cost for an RF 0.80 pit would still be lower than these costs, however, the surface mineral exploration /boundary inhibits the size of the potential Ikkari open pit to approximately the RF 0.70 pit.

Ikkari was therefore designed to the maximum allowable open pit with the remaining resource (economic resource) targeted by underground methods

16.5 MINING METHOD - PAHTAVAARA UNDERGROUND

16.5.1 UNDERGROUND MINING

The selected underground mining method for Pahtavaara is LHOS. Options for LHOS with and without backfilling newly created stope voids were considered. Ultimately no backfilling will be considered due to the discontinuous stope geometries and low value and productivity nature of an underground operation at Pahtavaara.

RESOURCE MODEL

The evaluation of Pahtavaara is based on the Resource model presented in Section 14 and summarised in Table 16.1.

The evaluation at a PEA level of study includes Measured, Indicated, and Inferred blocks and consequently it is not possible to define an Ore Reserve and the term Mineral Inventory has been used in this report.

Modifying Factors - Underground

The modifying factors for Pahtavaara were investigated by running various stope optimisations considering different minimum stope widths. Also studying the historic stope voids and surveys it became apparent that the strike lengths and stope widths were key to maintain a safe Hydraulic radius and therefore manage stope dilution. The following figure depicts the typical LHOS modification factors to generate a diluted ore tonnes and grade calculation and simulation for LHOS mining methods.

Figure 16.18 LHOS Modifying Factors – Stope Dilution

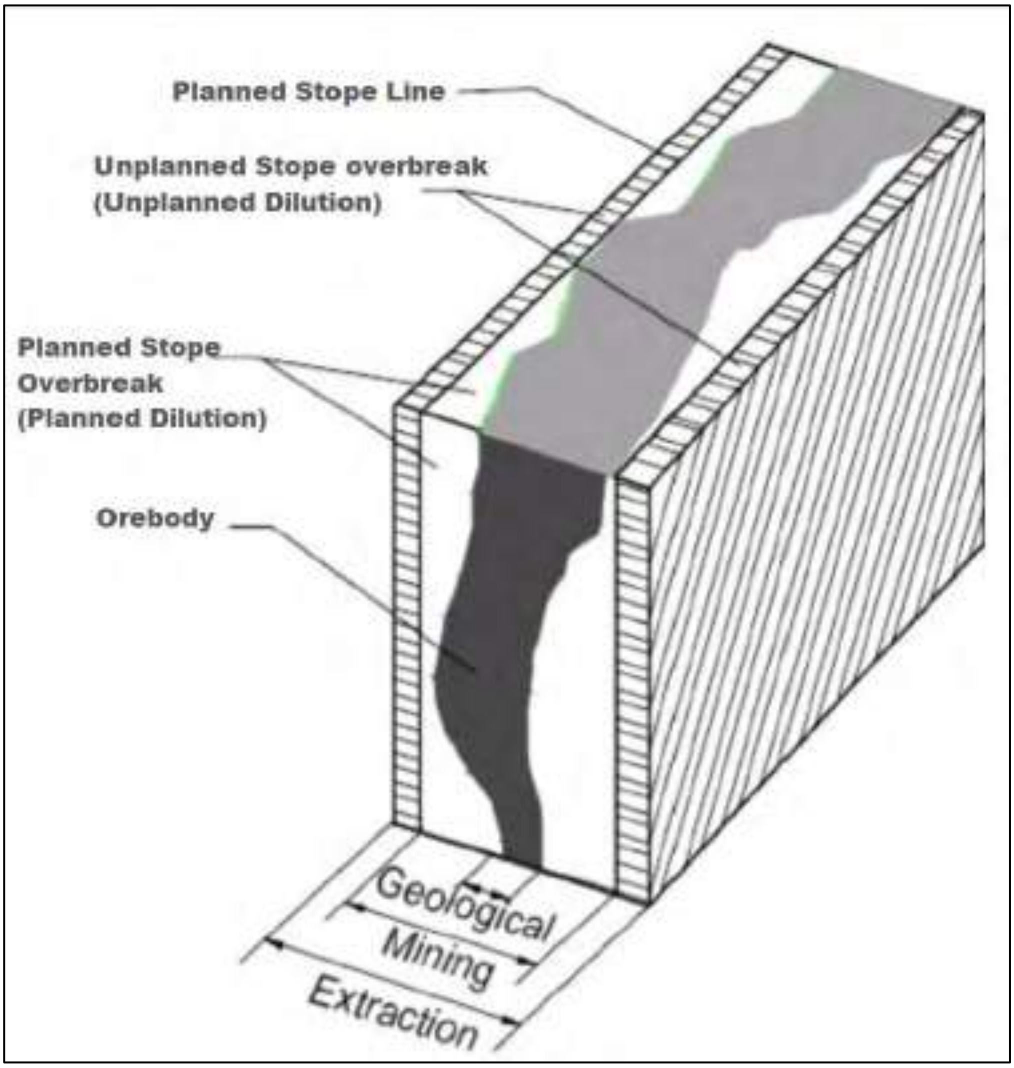


Table 16.18 LHOS Modifying Factors - Stope Dilution & Ore loss

Parameter	Unit	Value
Minimum Stope width	m	3
Hangingwall Stope overbreak Dilution	m	1
Footwall Stope overbreak Dilution	m	1
Overall minimum Stope ***Extraction*** Width	m	5
Typical Average Stope Dilution (Planned + Unplanned)	%	+-28
Average Au grade of Dilution	g/t	0
Stoping Ore loss – applied in scheduling	%	10

The stope dilution is specific to each stope (planned + unplanned dilution allowance within MSO). The range of stope dilution is 9% minimum to 40% maximum (based on the stope statistics obtained from MSO). The overall Pahtavaara stope dilution averaged close to 28%. That is waste rock width (dilution assumed at 0g/t Au) divided by the total extraction width.

Pahtavaara Stope Optimisation

The underground stope optimisations for Pahtavaara considered the parameters shown in Table 16.19 and Table 16.20.

Table 16.19 Stope Optimisation Parameters for Pahtavaara (Economics)

Parameter	Units	Value	Comments
Gold Price	US $/ozT	1,650	3-year trailing average
Royalty	%	0.15	Excludes initial Royalty of 1.5%
Discount Rate[1]	%	5.0	
Pit Exit/Crusher	RL	250	
Stope Mining Cost (Marginal Cut-off)	US $/t mined	20	
Total Mining Cost (Estimate)	US $/t ore	65	
Mining Recovery	%	90	Ore loss assumed in evaluation
Mining Dilution	%	8% - 40%	Stope overbreak of 2 m total
Ore loss (Stope pillars and impractical stope bogging)	%	10%	Crush pillars in stope, brows and loader bogging challenges in-stope
Grade Control	US $/t ore	0.50	
Stockpile Rehandle	US $/t ore	0.90	
Mine G&A	US $/t ore	0.20	
G&A Other	US $/t ore	3.47	Based on historical data
Process Cost (Pahtavaara)	US $/t ore	9.10	Based on 0.5 Mt/a plant feed
Process Cost (Ikkari)	US $/dmt conc	7.20	
Concentrate Mass Pull	%	18.0	
Conc Transport Cost	US $/wmt conc	40.0	Based on 160 km round trip
TC/RC	US $/toz	2.50	

CDF	US $/t ore	0.80	
Au Recovery (Pahtavaara)	%	91.0	
Au Recovery (Ikkari)	%	98.0	
Au Payability	%	99.92	

Key: G&A = general and administration, ozT = troy ounce, TC/RC = Treatment charge / refining charge

Note: The discount rate for Pahtavaara was subsequently updated to 8% for Ikkari.

Table 16.20 Stope Optimisation Geometry Parameters for Pahtavaara

Description	MSO Stope Criteria
Minimum Stope Extraction width	5 m (true width)
Target ore zone for drilling	3 m (true width)
Planned dilution (total) – included in minimum stope width	2 m
Strike length (per stope or cut)	15 m
Vertical Level Spacing	20 m
Maximum stope width	50 m
Minimum dip (stoping area)	45°
Planned + Unplanned dilution	2 m
Final Stope width (evaluation)-Planned + unplanned overbreak - (true width)	5 m
Minimum intact rock pillar between footwall and Hangingwall lodes	10 m

Geotechnical Analysis

The geotechnical parameters for the proposed underground mining at Pahtavaara are based on a previous trade-off study (SRK, 2020) that included a site visit and the historic underground mining operations at Pahtavaara.

One key underground stope design criterion to consider is Hydraulic Radius (HR). HR is the stope void front/panel area divided by the perimeter and this resultant number is an indication of the stability of that potential stope void.

Pahtavaara considered level spacings of 20 m (which creates a 25 m stope void when mined out with the oredrive). The stope strikes were limited to 15 m. The mine design targeted Hydraulic Radii of no more than 5.5 (ideally at or below 5).

The ore loss factor of 10% applied in scheduling assumes in-stope crush pillars to maintain an acceptable HR for Pahtavaara. The indications seem that Pahtavaara could have an HR of more than 5 and up to 7 (transition stability zones) at the modelled depths. Tetra Tech still applied stope cabling together with crush pillars (meaning allowing for supported stope spans).

Cut-off Grade

It is assumed that the plant at Pahtavaara will produce a high-grade concentrate that will be hauled by road to the main Ikkari plant for final processing to produce a Doré product. A NSR approach has therefore been adopted in the stope optimisation and an NSR value has been calculated for all mineralised blocks in the input model.

Based on the parameters shown in Table 16.20, the Breakeven and Marginal Cut-off grades estimation are shown in Table 16.21.

Table 16.21 Pahtavaara Mill Cut-off Grade

Cut-Off Grade Type	Price (US $/oz)	Process Rate (Mt/a)	NSR Cut-off ($/t ore)	Au Cut-off (g/t)
Pahtavaara-Break-even	1,650	0.5	60	1.51
Pahtavaara-Marginal	1,650	0.5	35	0.97
90% Break-Even stopes and 10% Marginal stopes included	1,650	0.5	58	1.45

The final Stope cut-off used in MSO was NSR $58 /t ore (assuming mostly break-even cut-off NSR values and approximately 10% of stopes that can be added at marginal cost/Cut-off).

Stope Optimisation Results

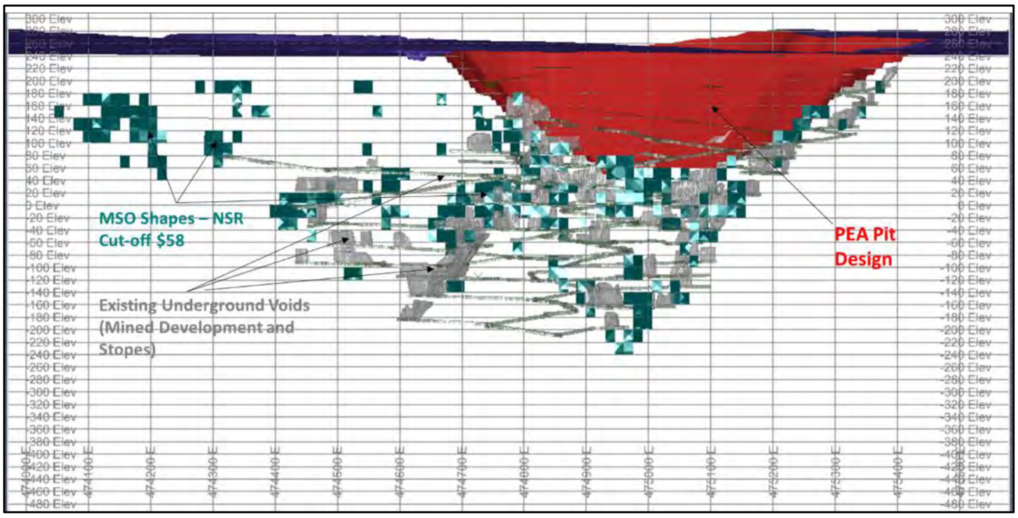
Using Datamine Studio MSO software, and attempting to avoid existing mining voids at Pahtavaara, the Pahtavaara Stope Optimisation Results are shown in Table 16.22.

Table 16.22 Pahtavaara Stope Optimisation Results

Description	Number of stopes	Stope volume	Stope tonnes	NSR US $/t	Au ppm
Stopes - Diluted-Total	347 Stope shapes	859,852	2,490,508	94.12	2.12

The MSO results had some stope shapes that formed in impractical areas and some isolated stope shapes ("orphans") were also subsequently removed during the mine design process.

Figure 16.19 Pahtavaara Stope Optimisation Shapes



Underground Design

Figure 16.20 Pahtavaara Underground Mine Design

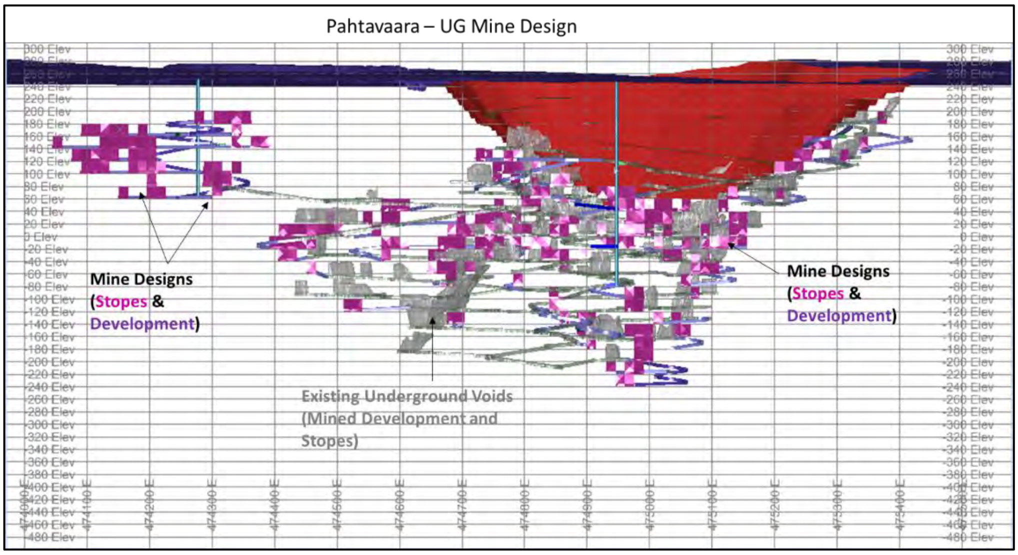


Table 16.23 Pahtavaara Underground Mine Design

Drill-Design Details	Unit	Value
Blast hole size	mm	89
Vertical Level Spacing	m	20
Drill Length	m	28
Charge Length	m	25.2
Burden (Ring) (Between 2-hole rings)	m	1.5
Toe Spacing	m	0.65
Primers per hole	no	2
Explosive (wet conditions)		Emulsion
Explosive SG	kg/m^3	940
Stope Width – Target (minimum – target mineralisation zones)	m	3
Rock SG – Ave	t/m^3	2.8
Powder Factor -Blast Design	Kg explosives / of rock m^3	1.9
Rock Broken per hole per linear meter of drillhole	m^3	2.25
Tonnage Factor per Drill m (Rounded up)	t rock /m of drill hole	6.3
Assumed re-drill (for time/cost calculations)		10%

Key: kg/m^3 = kilogram per cubic metre, SG = Specific gravity

The minimum drive dimensions were also impacted by the typical ventilation ducting sizes used at and the proposed mining equipment. The minimum duct size to be considered is 1.2 m diameter. The largest underground mining equipment that should be considered for Pahtavaara is 50 t trucks and 17 t Load Haul Dump trucks (LHD), but it is advised to consider 40 t trucks and 14 t LHD's as that would yield the same equipment unit requirements for the target production. The main declines were sized at 5 m by 5 m with some overbreak allowed, however if larger equipment is to be considered, the declines may need to be sized at 5.5 m wide by 5.8 m high (to fit 60 t trucks). Pahtavaara has a low mining rate and cost saving is essential, therefore the development drives are kept as small as possible. The stope heights of 20 m/level spacing and allowing for accurate Longhole Drilling requires the oredrives to be 5 m wide by 5 m high. There might be options to reduce these to minimise ore dilution in ore development, however, productivity and possibly selective bogging could be employed rather than reducing oredrive sizes.

Figure 16.21 and Figure 16.22 depict the equipment dimensions within the stated development drives.

Figure 16.21 50 t Haul Truck in the Typical Development Dimensions Considered

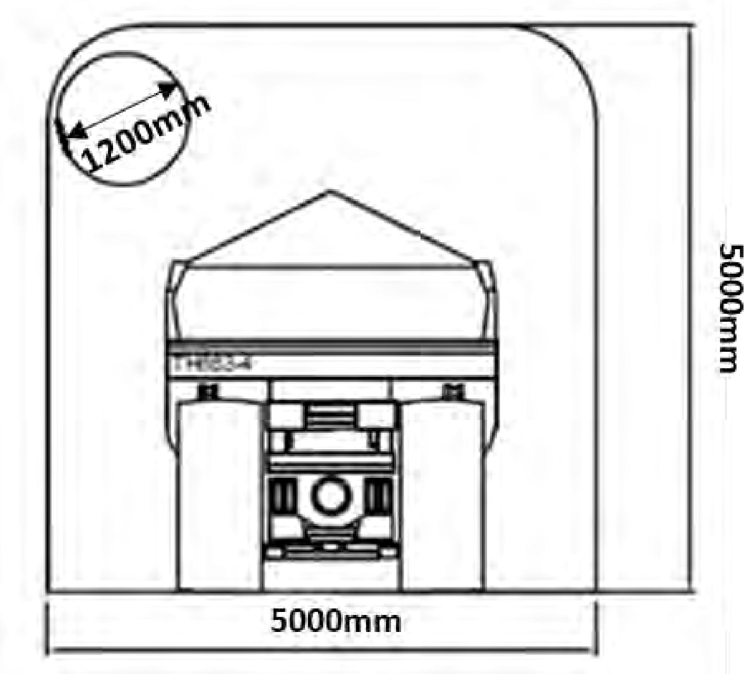


Figure 16.22 17 t LHD in a Typical 5 m by 5 m Development End

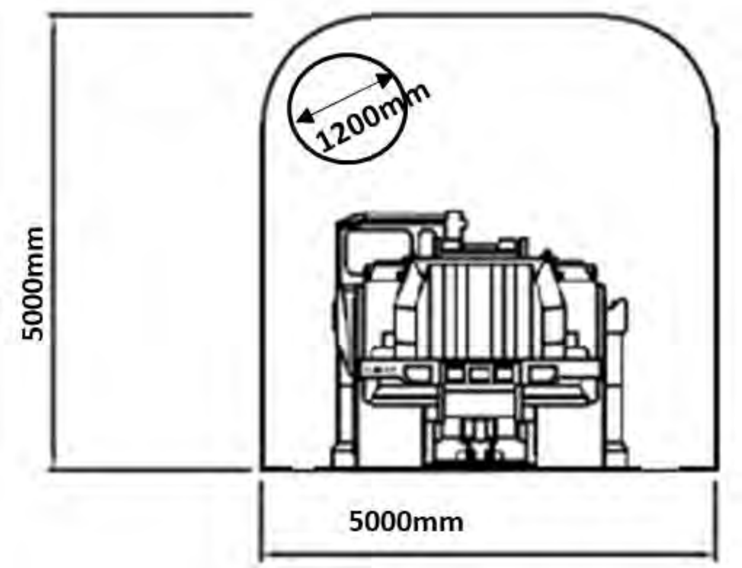


Table 16.24 Pahtavaara Underground Design & Mineral Inventory

Production Physicals		
Description	**Unit**	**Totals**
Decline length	m	1,607
Footwall drives	m	6,977
Access drives m	m	14,404
UG Remuck/Stockpile m	m	144
Sump m	m	73
Vent drive m	m	1,439
Ventilation Raise Bore	m	645
Stope ROM Tonnes	Tonnes	1,883,263
Stope ROM Au	Au grams	4,016,672.03
Stope Au g/t	Au g/t	2.13
Oredrive ROM Tonnes	Tonnes	425,946
Oredrive ROM m	m	6,199
Oredrive ROM Au	Au grams	718,636
Oredrive Au Grade	Au g/t	1.69
Total Ore (ROM)	Tonnes	2,309,209
Total Au kg (ROM)	Au kg	4,735
Ave ROM Au Grade	Au g/t	2.05

Notes:
1. Tonnages have been rounded.

2. Mining recovery and dilution set to 90% - built into the ROM extraction and metal calculations within Datamine's Studio UG and Enhanced Production Scheduler (EPS).

3. Dilution was built into the oredrive through tunnel sizes and in stopes via minimum stope widths. An overbreak assumption within the stopes of 1 m on either side of the Hangingwall and Footwall was incorporated into the stope shape development.

Mine Schedule

The mining schedule for Pahtavaara has been developed as part of a combined schedule for Pahtavaara and Ikkari. This assumes that a high-grade concentrate will be produced at Pahtavaara, which will be hauled to the Ikkari plant for final processing.

A high grading strategy has been adopted for Pahtavaara open pit whereby lower grade material is stockpiled for reclamation at the end of mine life for the underground operations. Besides increasing the feed grade to the plant, this strategy has the advantage that the overall mine life for Ikkari and Pahtavaara can be brought into line so that the open pit mining equipment from Ikkari can be transferred to Pahtavaara as the Ikkari open pit reaches the end of its life.

The low-grade stockpile also acts as a buffer during the period of ramping up production from the underground which mitigates the risk of a shortfall in plant feed due to unforeseen issues with starting up the underground.

Table 16.25 Equipment Operating Hours

Parameter	Unit	Value
Shifts per day	No	2
Hours per shift	Hr	12
Non-Working Hours per shift	Hr	2
Effective utilisation - shift utilisation	minutes per hour	50
Days per month	Days	30
Hours per month (Effective working hours – two shifts)	Hr	500
Hours per year-mining activities/equipment hours	Hr	6000

Equipment Selection

The underground mining modelled for Pahtavaara would be contractor mining. The contractor would therefore likely propose their specific mining and auxiliary equipment best suited for the project and available in their fleets. The equipment was costed as fleet lease costs within the mining cost calculations.

16.6 Mining Methods – Ikkari Underground

16.6.1 Introduction

Ikkari is a greenfield site where it is proposed that the mine is developed initially as an open pit with a target production rate of 3.5 Mt/a of plant feed. As the open pit reaches the end of its life (approximately 12 years) the underground development will be completed so that the underground operation can continue for a further 12 years. The total mine life is therefore approximately 24 years with a proposed start date for pre-stripping in Q1 2027.

A suitable transition point between open pit operations and underground operations has been selected based on operating costs, as well as the limitation of the current exploration permit boundary, that impacts both surface and underground operations.

The Ikkari underground deposit provides an excellent option for bulk underground mining and exploitation strategies. LHOS and SLC are good underground potential mining options and were subsequently studied through design, scheduling, and cost estimations. Block Cave (BC) simulations and optimisations were also concluded (at highly indicative or concept evaluation levels) to determine if block caving could potentially be an option for Ikkari.

The key aspects of BC mining are that it allows for much higher ore production rates and is mostly suited to large or very large and generally lower grade orebodies where mining and operating costs need to be minimised and ore production rates maximised. When processing rates of more than 3 Mt/a are considered, the three well practiced underground mining methods and exploitation strategies studied are LHOS (bulk), SLC and BC.

16.6.2 Underground Mining

Two underground mining methods were studied in more detail (for the Ikkari underground exploitation strategies). BC was tested (as a third mining option) during the pre-PEA studies by developing bulk stope shapes at typically minimum BC geometries expected to cave (rock broken down via gravity and caving).

Figure 16.23, Figure 16.24 and Figure 16.25 depict the different bulk underground mining methods.

Figure 16.23 Long Hole Open Stoping

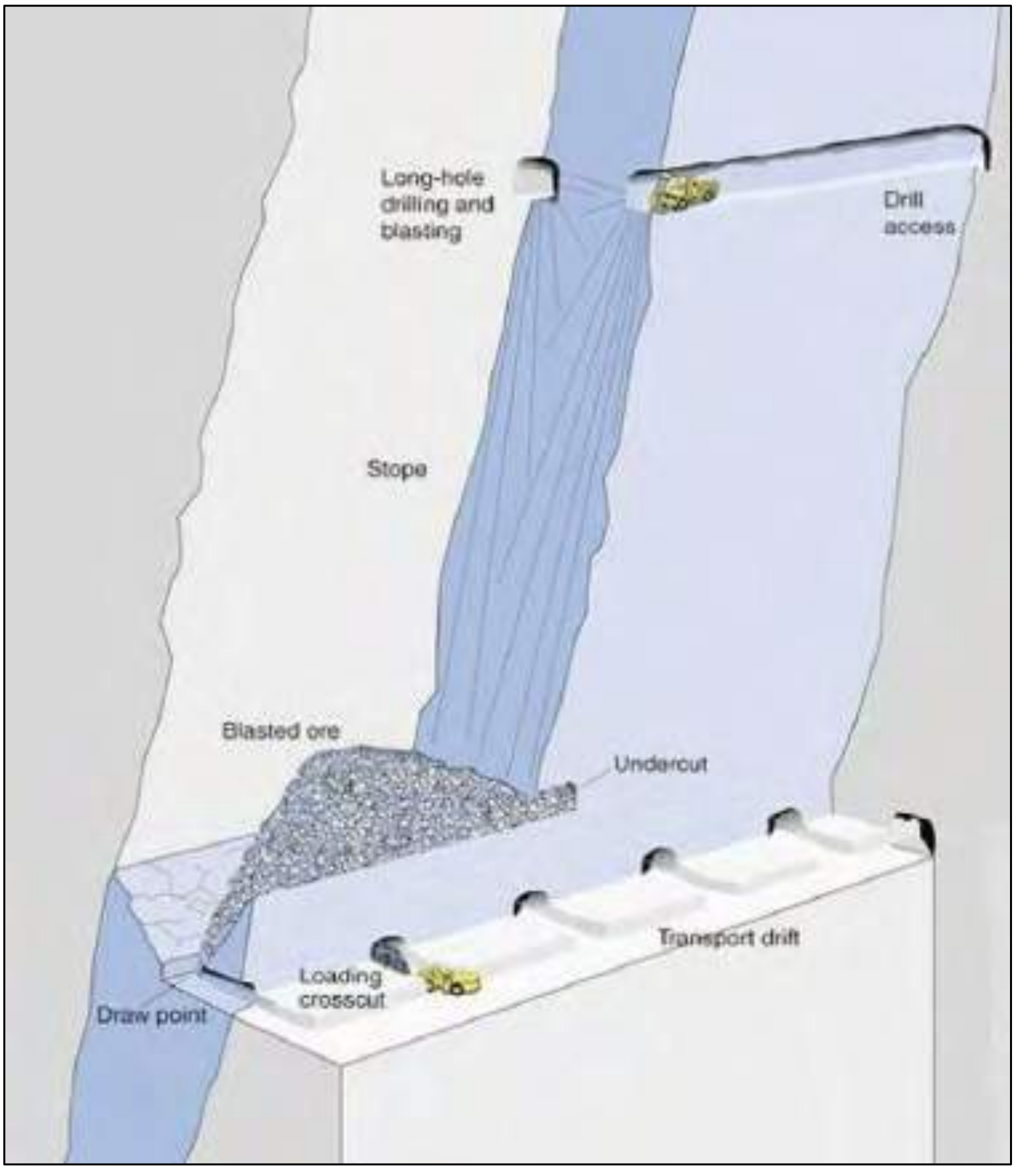


Figure 16.24 Sub-level Caving

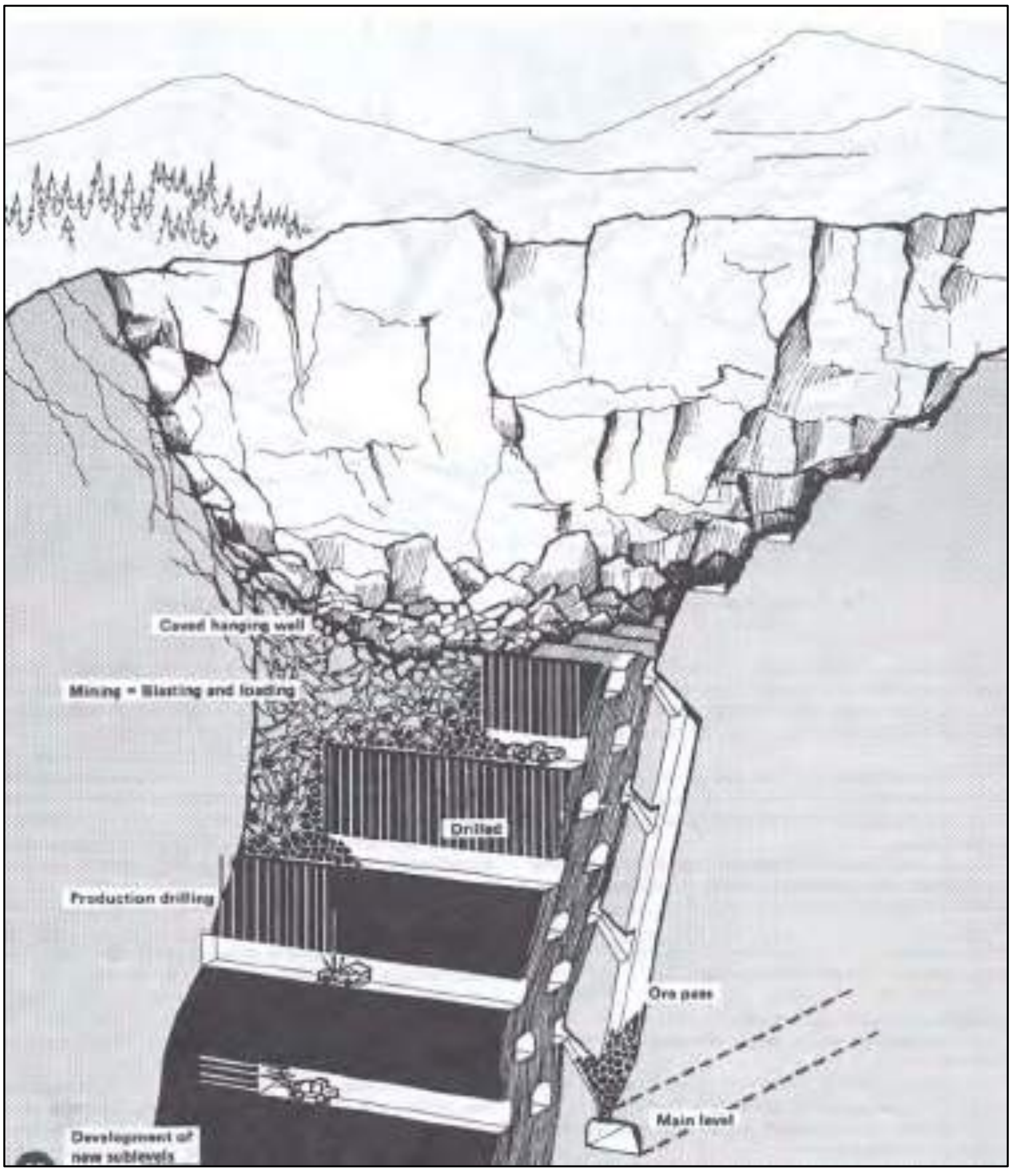
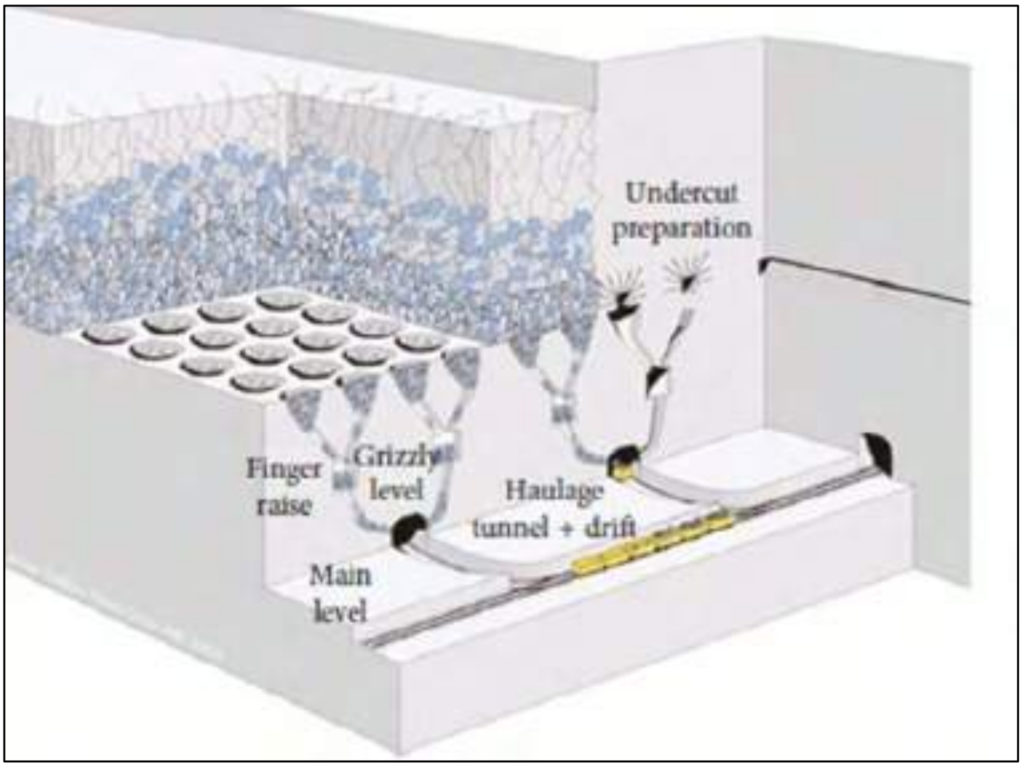


Figure 16.25 Block Caving



Ikkari has the potential to model a slightly more selective underground extraction method (SLC) compared to a complete bulk BC option. The benefit is a much better ROM or draw grade through sub-level cave controls and draw controls.

The SLC method is less selective when compared to LHOS, but the overall mine operating cost compared to ROM grades are key for more detailed analyses. This PEA study developed underground optimisation, full mine designs and schedules for both the LHOS and SLC methods.

RESOURCE MODEL

The evaluation of Ikkari is based on the Resource model presented in Section 14 and summarised in Table 16.26.

Table 16.26 Ikkari Resource Model

Deposit	Resource Classification	Open Pit Status	Cut-off Grade (g/t)	Tonnage (t)	Grade Au (g/t)	Metal Content	
						Kg Au	Oz Au
Ikkari	Indicated	Open Pit	0.5	30,000,000	2.5	74,500	2,400,000
		Underground	1.0	16,500,000	2.4	39,800	1,280,000
		Total		***46,40,000***	***2.5***	***114,300***	***3,680,000***
	Inferred	Open Pit	0.5	3,100,000	1.5	4,800	150,000
		Underground	1.0	8,700,000	2.0	17,200	550,000
		Total		***11,800,000***	***1.9***	***39,300***	***1,260,000***
Ikkari Indicated and Inferred Total				**58,200,000**	**2.3**	**136,300**	**4,390,000**

The evaluation at this level of study includes Indicated and Inferred blocks and consequently it is not possible to define an Ore Reserve and the term Mineral Inventory has been used in this report.

MODIFYING FACTORS - UNDERGROUND

The modifying factors for Ikkari were investigated by running various stope optimisations considering different minimum stope widths and considering each mining method (LHOS and SLC). Naturally the LHOS method would consider stoping widths as narrow as 4 m wide whereas the SLC method cannot consider any draw widths less than 20 m.

Table 16.27 LHOS Modifying Factors – Stope Dilution and Ore Loss

Parameter	Unit	LHOS	SLC
Minimum Stope width – target zone width	m	3	20
Hangingwall Stope overbreak Dilution	m	0.5	1.5
Footwall Stope overbreak Dilution	m	1	1.5
Overall minimum Stope Extraction Width	m	4	23
Overall average Stope Extraction Width	m	10	40
Typical Average Stope Dilution (Planned + Unplanned)	%	+-15	+8
Average Au grade of Dilution	g/t	0	0
Stoping Ore loss – applied in scheduling	%	10	9

The stope dilution is specific to each stope (planned + unplanned dilution allowance within MSO. The range of stope dilution is 6% minimum to 15% for maximum for SLC (based on the stope statistics obtained from MSO) and 10% minimum to 25% maximum for LHOS. The overall Ikkari stope dilution averaged close to 15% for LHOS and 8% for the SLC stope shapes. That is waste rock width (dilution assumed at 0g/t Au) divided by the total extraction width.

IKKARI STOPE OPTIMISATION PARAMETERS

The underground stope optimisations for Ikkari considered the parameters shown in Table 16.28.

Table 16.28 Stope Optimisation Parameters for Ikkari (Economics)

Parameter	Units	Value-LHOS	Value-SLC
Gold Price	US $/ozT	1,650	1,650
Royalty	%	0.15	0.15
Discount Rate[1]	%	8	8
Pit Exit/Crusher	RL	250	250
Stope Mining Cost (Marginal Cut-off)	US $/t mined	22	10
Total Mining Cost (Estimate)	US $/t ore	35	26
Mining Recovery	%	90	90
Mining Dilution	%	8% - 40%	8% - 40%

Ore loss (Stope pillars and impractical stope bogging)	%	10	Crush pillars in stope, brows and loader bogging challenges in-stope
Grade Control	US $/t ore	0.50	0.50
Stockpile Rehandle	US $/t ore	0.90	0.90
Mine G&A	US $/t ore	2.80	2.80
Process Cost	US $/dmt conc	9.16	9.16
TC/RC	US $/ozT	2.50	2.50
CDF	US $/t ore	0.80	0.80
Rehabilitation	US $/t ore	0.80	0.80
Au Recovery	%	95.0	95.0
Au Payability	%	99.92	99.92

Table 16.29 Stope Optimisation Geometry Parameters for Ikkari

Description	MSO Stope Criteria - LHOS	MSO Stope Criteria - SLC
Minimum Stope Extraction width	4.0 m (true width)	23 m
Target ore zone for drilling	3 m (true width)	20 m
Planned dilution (total) – included in minimum stope width	1 m	3 m
Strike length (per stope or cut)	15 m	15
Vertical Level Spacing	40 m	25
Maximum stope width	50 m	n/a
Minimum dip (stoping area)	50°	75°
Planned + Unplanned dilution	1 m	3 m
Final Stope width (evaluation)-Planned + unplanned overbreak - (min width)	4 m	23 m
Final Stope width (overall average)	10 m	40 m
Minimum intact rock pillar between footwall and Hangingwall lodes	10 m	n/a

GEOTECHNICAL ANALYSIS

The geotechnical parameters for the proposed underground mining at Ikkari are based on a previous trade-off study developed by SRK and more recent (SRK July 2022) Stope stability evaluations informing the stope criteria for Ikkari.

Ikkari considered level spacings of 40 m for the LHOS method and 25 m for the SLC. The key difference with either mining method is that the stope criteria for LHOS needs a stable unsupported Hydraulic Radius whereas SLC requires stopes to break up or cave with induction (use of blasting) together with the continuous larger rock and mined void generated as the sub-levels migrates downward with the SLC method.

Cut-off Grade

An NSR approach has therefore been adopted in the stope optimisation and an NSR value has been calculated for all mineralised blocks in the input model.

Based on the parameters shown in Table 16.29, Breakeven and Marginal Cut-off grades were estimated as shown in Table 16.30.

Table 16.30 Estimated Mill Cut-off Grade

UG Resource	Price (US $/oz)	Process Rate (Mt/a)	NSR Cut-off ($/t ore)	Au Cut-off (g/t)
Ikkari-Break even cut-off (LHOS)	1,650	3.5	50.16	1.01
Ikkari-Break even cut-off (SLC)	1,650	3.5	41.16	0.83

The final Stope cut-off used in MSO (prior to calculating the detailed mining costs) was NSR $48 /t ore (assuming mostly break-even cut-off NSR stopes for the LHOS) and NSR $40 /t ore for the SLC option.

The logic behind the different MSO (Stope optimisation) cut-off and the calculated break-even cut-off are:

- Cut-off calculation for the stope optimisation generally takes place at the onset of the mining study and mine design phase. Therefore, some initial cost calculations coupled with benchmarking mining and processing costs are used to develop economic mining and stope shapes.
- An underground stope optimisation process usually relies on most of the areas to be considering break-even economic cut-off conditions, although, if existing underground voids and mining areas exist, it is conceivable to use a higher ratio of break-even mining cut-off criteria and marginal stope cut-off criteria to define mining shapes and stoping areas for design.
- When considering a less selective/bulk underground mining option there is no/limited opportunities to apply marginal cut-off criteria as the entire block of ore will be broken and drawn, and the access development will be concentrated around a defined sub-level cave or BC. Therefore, it is generally advised to consider either Footprint Finder or PCBC software to develop the economic BC shapes or PCSLC to define a sub-level economic shape for the mine planning purposes.
- MSO is also deemed reasonably capable to define sub-level cave shapes (SLC) as it works on width, level and shape criteria that can be defined to simulate typical sub-level cave draw shapes.
- Sub-level caving is also, to some degree, able to selectively load ore, based on some sampling process. This is challenging to get right and often oversimplified in studies.
- For a SLC option it is therefore advised to run full break-even cut-off criteria and ensure the shapes are bulk and diluted within the optimiser to provide a more realistic bulk shape for an SLC mine design.

STOPE OPTIMISATION RESULTS

Table 16.31 summarises the MSO optimisation results for Ikkari (considering the two potential mining methods).

Table 16.31 MSO Optimisation Results for Ikkari

MSO Results	Number of Stopes	Stope Tonnes - MSO	Average Tonnes/Stope	Stope Tonnes – Ore loss	Stope Au metal kg	Stope Au metal kg (Ore loss applied)	Stope Au g/t
LHOS	1,218	27,282,261	22,399	24,554,035	48,398	43,558	1.77
SLC	786	35,113,052	44,673	31,952,877	55,168	50,203	1.57

These are raw MSO results for the different mining methods. There were early MSO optimisation runs developed for a BC option at Ikkari. These were developed prior to the PEA study and results were not favourable viz grade and geometry.

Figure 16.26 Ikkari LHOS Underground Mine Design

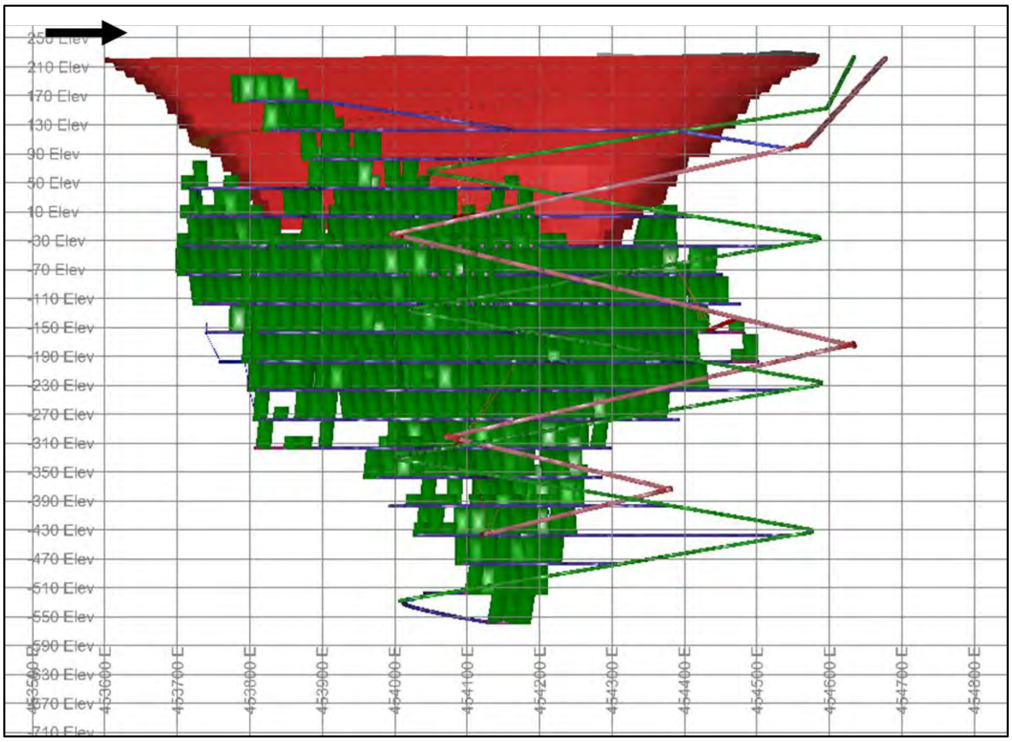


Figure 16.27 Ikkari LHOS Underground Mine Design (2)

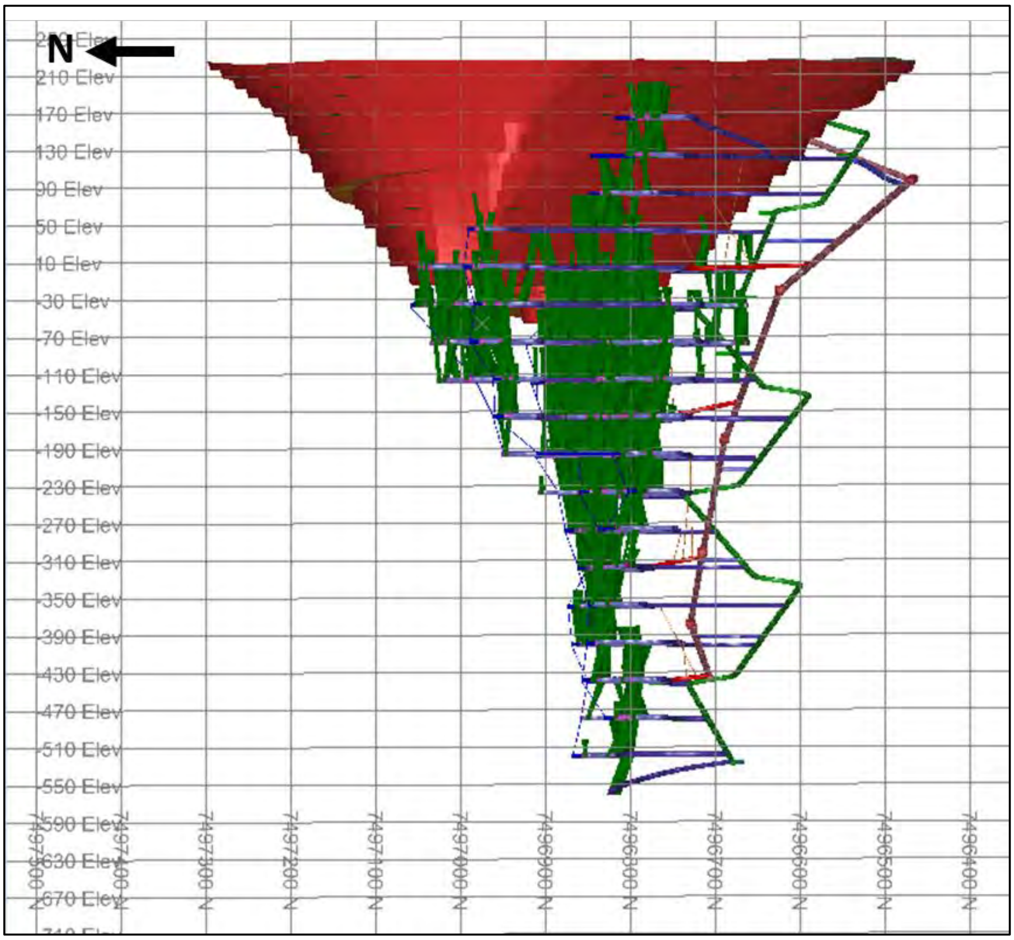


Figure 16.28 Ikkari SLC Underground Mine Design

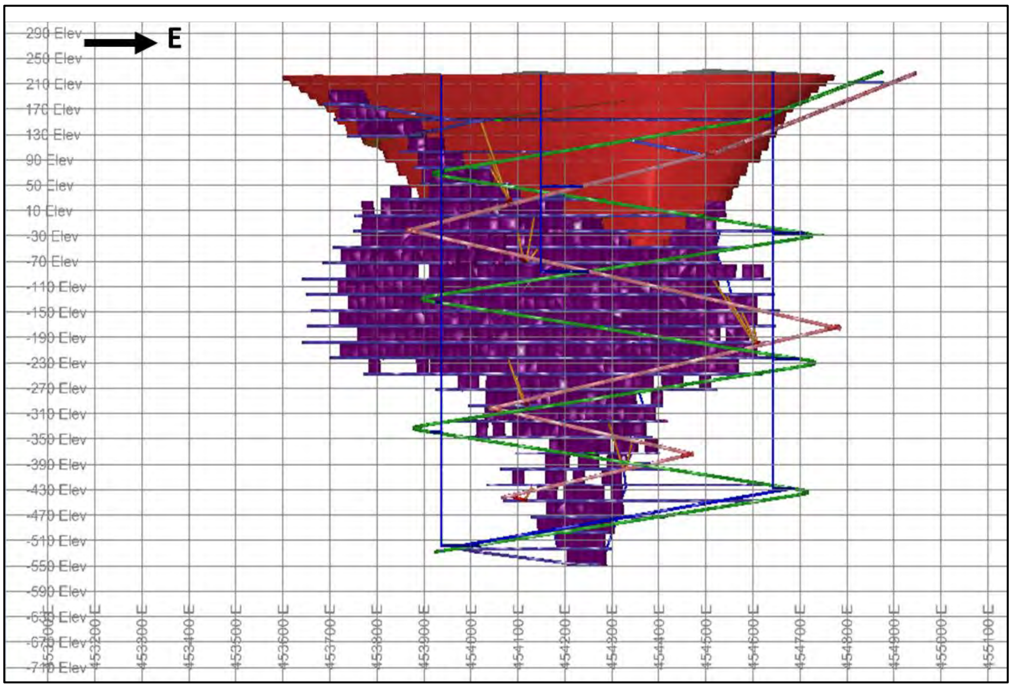
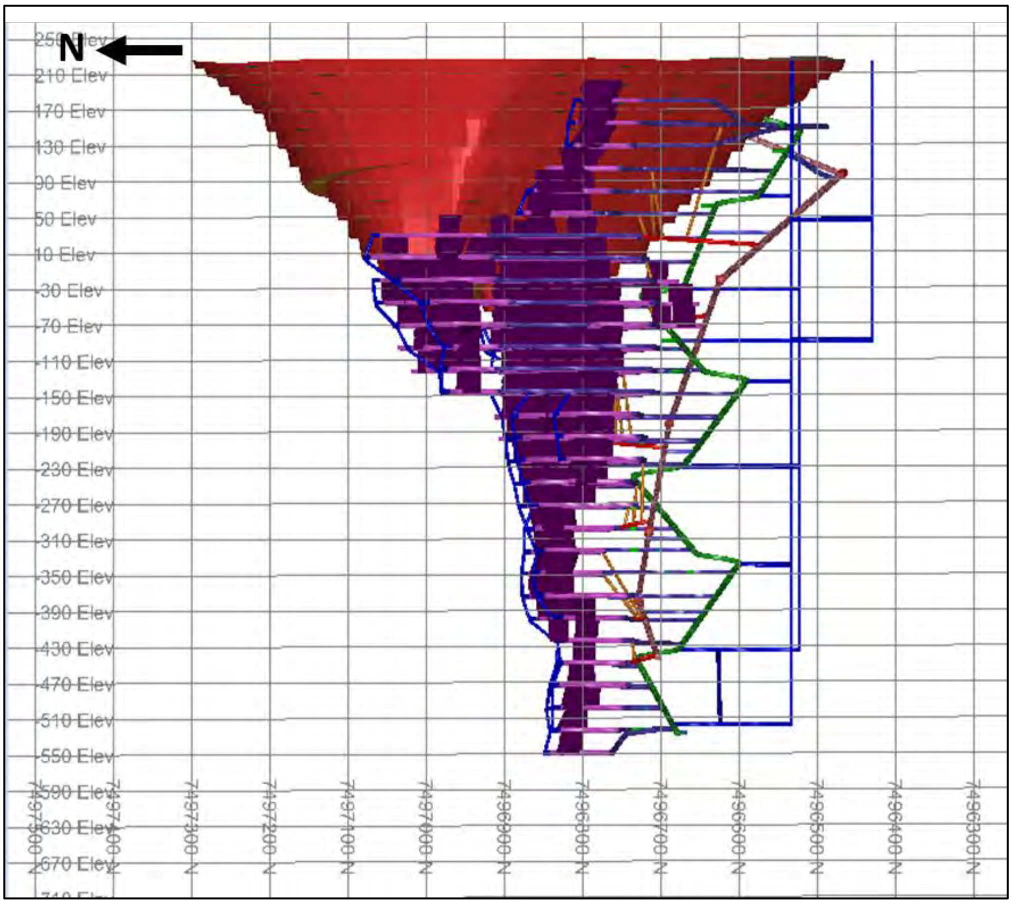


Figure 16.29 Ikkari SLC underground Mine Design (2)



The drilling and criteria for both options are shown in Table 16.32.

Table 16.32 Drill Design Details for Ikkari

Drill-Design Details	Unit	LHOS	SLC
Blast hole size	mm	89	102
Vertical Level Spacing	m	40	25
Drill Length	m	28	20
Charge Length	m	25.2	17
Burden (Ring)	m	1.5	2
Toe Spacing	m	1	1
Primers per hole	no	2	
Explosive (wet conditions)		Emulsion	Emulsion
Explosive SG	kg/m^3	940	940
Stope Width – Target (minimum – target mineralisation zones)	m	4	23
Rock SG - Ave	t/m^3	2.8	2.8
Powder Factor -Blast Design	Kg explosives / of rock m^3	1.9	1.5
Assumed re-drill (for time/cost calculations)		10%	10%

The minimum drive dimensions were also impacted by the typical ventilation ducting sizes used at and the proposed mining equipment. The minimum duct size to be considered is 1.2 m diameter, but mostly 1.4 m diameter ducting is advised as a large volume of air would be required for Ikkari. The largest underground mining equipment that should be considered for Ikkari is 60 t trucks and 17 t LHD’s, but it is advised to consider 50 t trucks and 17 t LHD’s. The main declines were sized at 5.5 m by 5.5 m but there is also a conveyor decline for Ikkari (both options) as 3.5 Mt/a generally require a consideration of alternate rock transport (conveyor, Railveyor or vertical hoisting if at greater depths).

Ikkari’s scheduling and rock transportation options included either trucking of ore or conveying of ore. Both options were modelled and costed and the key benefit with conveyance systems at these higher rock tonnage targets are flexibility, operating cost and reduction of diesel vehicles which significantly reduces ventilation requirements.

The following pictures depicts the equipment dimensions within the stated development drives:

Figure 16.30 50 t Haul Truck in the Typical Development Dimensions Considered

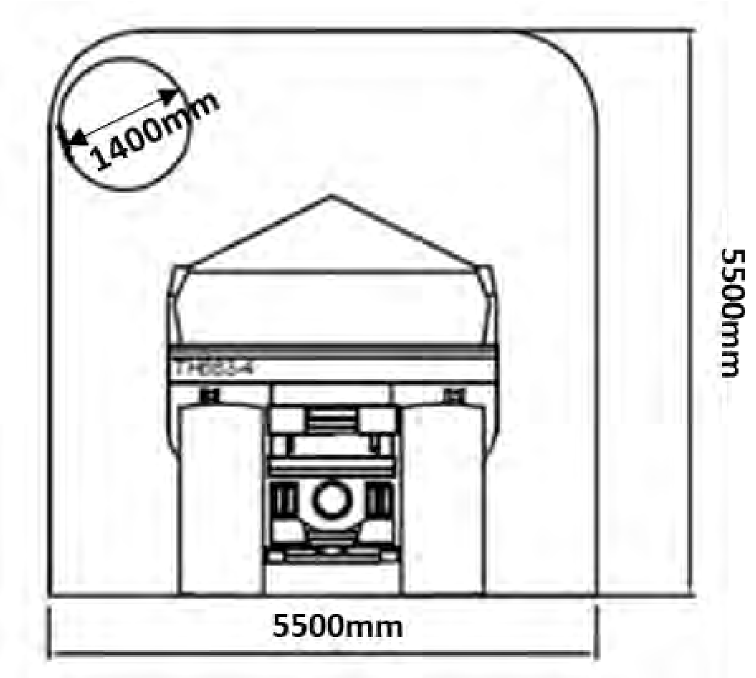


Figure 16.31 Conveyor Decline Layout (the Conveyor can be suspended for more space or installed on the ground)

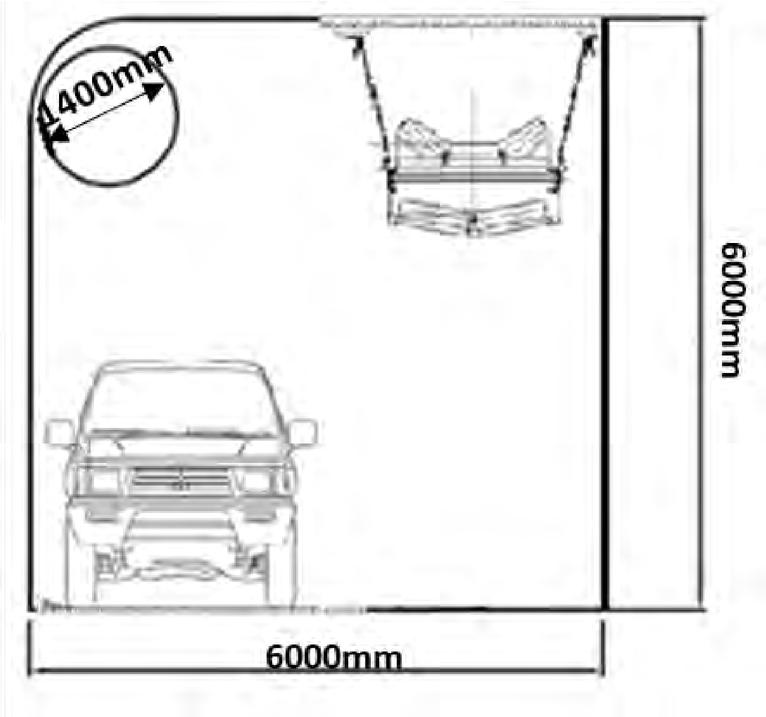


Figure 16.32 17 t LHD in a Typical 5 m by 5 m Ore

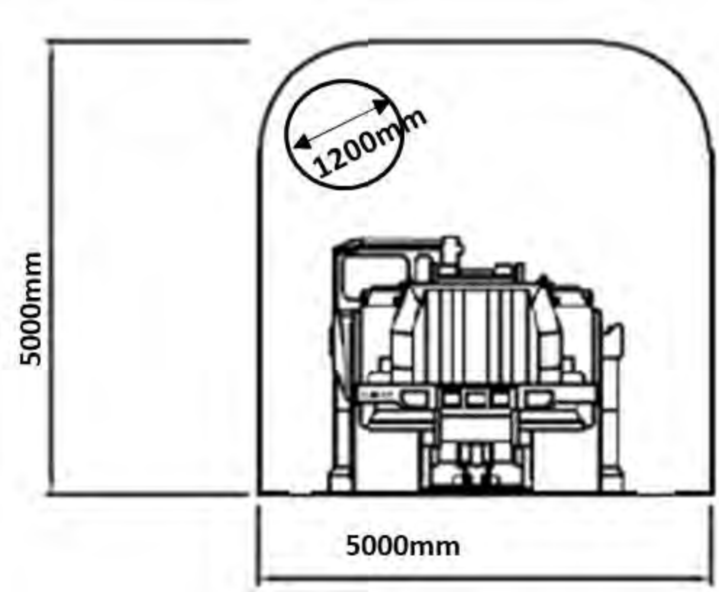


Table 16.33 Ikkari's Underground Design & Mineral Inventory

Production Physicals			
Description	**Unit**	**LHOS**	**SLC**
Dec m	m	8,548	8,029
FWD m	m	13,761	20,432
Access drives m	m	17,482	34,954
UG Remuck/Stockpile m	m	410	588
Sump m	m	180	283
Vent drive m	m	1,158	2,423
Vent RB m	m	4,453	4,938
Stope ROM Tonnes	Tonnes	25,066,123	30,515,813
Stope ROM Au	Au grams	44,575,672	47,310,165
Stope Au g/t	Au g/t	1.78	1.55
Oredrive ROM Tonnes	Tonnes	1,316,176	3,010,424
Oredrive ROM m	m	18,769	43,000
Oredrive ROM Au	Au grams	1,800,621	3,842,157
Oredrive Au Grade	Au g/t	1.37	1.28
Total Ore (ROM)	Tonnes	26,382,299	33,526,237
Total Au kg (ROM)	Au kg	46,376	51,152
Ave ROM Au Grade	Au g/t	1.76	1.53

Notes:

1. Tonnages have been rounded.
2. Mining recovery and dilution set to 90% - built into the ROM extraction and metal calculations within Studio UG/EPS.
3. Dilution was built into the ore drive through tunnel sizes and in stopes via minimum stope widths. An overbreak assumption within the stopes of 1 m on either side of the Hangingwall and Footwall was incorporated into the stope shape development.

MINE SCHEDULE

The mining schedule for Ikkari has been developed as part of a combined schedule for Pahtavaara and Ikkari. This assumes that a high-grade concentrate will be produced at Pahtavaara, which will hauled to the Ikkari plant for final processing.

EQUIPMENT SELECTION

The underground mining modelled for Ikkari is also proposed to be contractor mining. The contractor would therefore likely propose their specific mining and auxiliary equipment best suited for the project and available in their fleets. The equipment was costed as fleet lease costs within the mining cost calculations.

Typical mining equipment sizes recommended for Ikkari 's underground operations would be at least 50 t underground trucks with 17 t LHDs or even 60 t trucks with 17 t LHDs.

16.7 COMBINED SCHEDULE

16.7.1 INTRODUCTION

A combined schedule was created for the project that considers the open pit and underground operations at Ikkari and Pahtavaara. The mineral inventory for each of these operations is based on the design work described in Sections 16.2 and 16.3.

Initially schedules were developed for several potential scenarios based on a range of plant throughput rate for Ikkari (2.5, 3.0 and 3.5 Mt/a) and the underground mining methods (LHOS or SLC) for Ikkari. In all cases it was assumed that the mine plan for Pahtavaara would be limited to a plant throughput of 0.5 Mt/a and that the concentrate product from Pahtavaara will be transported by road to the Ikkari plant for final processing to Doré.

The Scenarios considered are shown in Table 16.34.

Table 16.34 Scheduling Scenarios

Scenario	Ikkari		Pahtavaara	
	Plant Rate (Mt/a)	UG Mining Method	Plant Rate (Mt/a)	UG Mining Method
1a	2.5	LHOS	0.5	LHOS
1b	2.5	SLC	0.5	LHOS
2a	3.0	LHOS	0.5	LHOS
2b	3.0	SLC	0.5	LHOS
3	3.5	SLC	0.5	LHOS

The schedules for these scenarios all start with open pit mining at Ikkari with a transition to underground mining once the open pit approaches the maximum depth and production drops off. The start of underground mining is not constrained by the open pit as the twin decline access is developed from a point outside of the open pit perimeter. The underground mine development can therefore start early so that ramping up of production from the underground coincides with ramping down of production from the open pit.

In all scenarios a high grading strategy is applied at Ikkari so as the increase revenue in the early periods and provide a buffer stock of lower grade material to mitigate the risk of a shortfall in production from either the open pit or the underground operations. This low-grade material is mainly reclaimed at the end of mine life.

The start of open pit mining at Pahtavaara is delayed as long as possible due to the lower grades and higher waste stripping ratio at Pahtavaara. However, given the constraint on plant rate for Pahtavaara the start of mining at Pahtavaara cannot be delayed much beyond the transition from open pit to underground mining at Ikkari without ending up with a long tail of concentrate production from Pahtavaara after the Ikkari underground has been completed. As discussed previously this would mean running the Ikkari plant at 180 Ktpa (limited by the concentrate production) and greatly underutilising this resource increasing the operating costs.

The other limitation on the scenarios was deemed to be the underground mining method at Ikkari with the LHOS method constrained to a maximum of 3.0 Mt/a. The SLC method on the other hand can easily be scaled up to a maximum of around 3.8 Mt/a.

It was concluded from a preliminary evaluation of the various scenarios that Scenario 3 provided was best suited to the project objectives of maximising NPV and Internal Rate of Return (IRR). This scenario was therefore developed as the Base Case for the purposes of the PEA.

16.7.2 Combined Schedule – 3.5 Mt/a SLC Case

The combined schedule was developed in Excel in quarterly periods using the mineral inventory for the various operations. In the case of the open pits the pit was divided into several stages (phases) and the mineral inventory for each stage was reported by material type for each bench, whereas the schedule for the underground operation was developed in Datamine's EPS. The underground schedules were then imported into the Excel combined schedule for integration with the open pits.

In order to allow high grading, the mineral inventory was subdivided into a number of material types based on a series of grade bins. The grade intervals were chosen to give a spread of tonnes by interval that would be suited to a high grading strategy where between 20% and 40% of the ore grade material can be stockpiled as required.

Table 16.35 Definition of Material Types (Open Pits)

Material Type	Code	Au g/t >=	Au g/t <
Waste	Wst	0.0	COG[1]
Low Grade	LG	COG[1]	0.50
Medium Grade	MG	0.70	0.90
High Grade	HG	0.90	1.00
Very High Grade	VHG	1.00	99.00

Notes:
1. Economic Cut-off Grade (COG) is varied for each mine.

For the purposes of scheduling it is assumed that all material passes through a stockpile and that the highest grade material is reclaimed first, unless a constraint is applied to a particular material type (LG, MG, HG or VHG) that limits the reclaim rate. For example, this control is used where it is required to smooth the plant feed grade.

Besides ensuring that each pit stage follows the preceding pit stage (i.e. phase lag) the mining rate for each stage of the open pit is also controlled by the maximum vertical annual bench sinking rate of 90 m (9 benches). This is reported by pit stage and schedule period.

The proposed mining sequence is summarised in Table 16.36 in terms of the start end dates for each mining operation.

Table 16.36 Start and End Dates for the 3.5 Mt/a SLC Scenario

Operation	Inventory (Mt)	Feed Rate (Mt/a)	Start Date	End Date	Mine Life[1,2,3] (Years)
Ikkari Open Pit	36.98	3.5	2027	2038	12
Ikkari Underground	33.53	3.5	2036	2048	12
Pahtavaara Open Pit	5.68	0.5	2038	2047	9
Pahtavaara Underground	2.31	0.5	2043	2048	6

Notes:
1. Mine life shown has been rounded to years.
2. LG Stockpile reclaim at Pahtavaara continues until 2052.
3. LG Stockpile reclaim at Ikkari continues until 2048.

A summary of the mined tonnes and grades for the combined schedule are presented in Figure 16.33 and Figure 16.34 and Table 16.37. It can be seen that the mined grade for Ikkari varies considerably over the life of the open pit. This is symptomatic of the grade variation with depth with lower grade material in the upper benches of Stage 2 and 3 and the changes in stripping ratio as you progress from Stage 1 (very low stripping ratio) to Stage 3 (higher stripping ratio).

Figure 16.33 Scheduled Tonnes and Grades for the Ikkari and Pahtavaara Open Pits (3.5 Mt/a SLC Case)

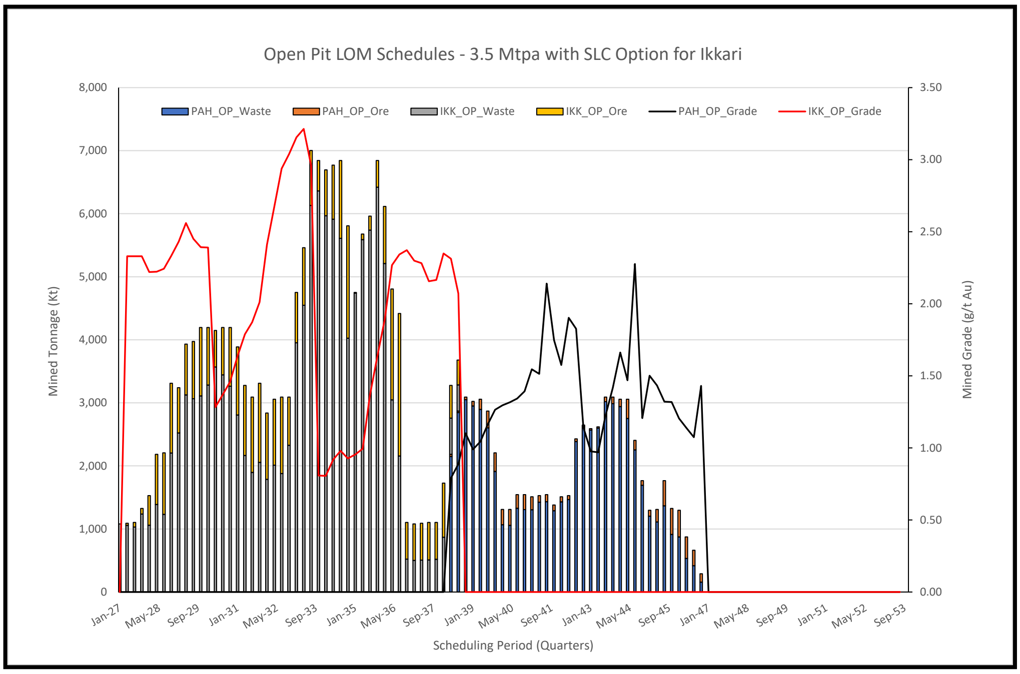
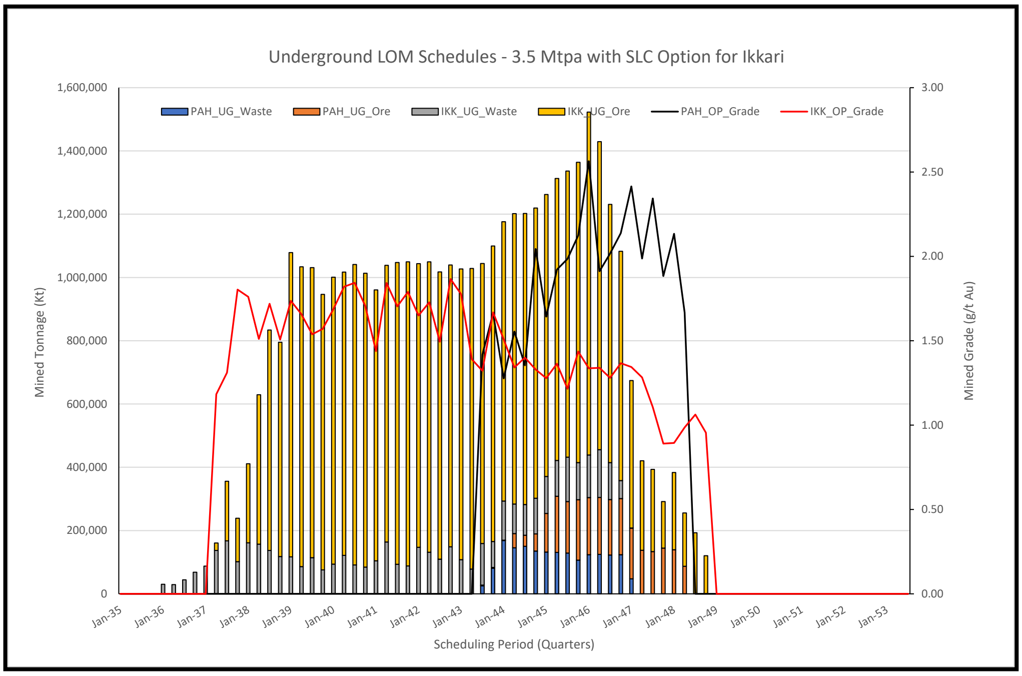


Figure 16.34 Scheduled Tonnes and Grades for the Ikkari and Pahtavaara Underground Operations (3.5 Mt/a SLC Case)



It should be noted that the tonnes and grades reported in Table 16.37 for the Ikkari Plant are inclusive of the concentrate feed from Pahtavaara. The feed also takes into account the buffering of low grade material on the stockpile and the reclamation at the end of mine life.

To try and smooth out the variation in grade the mine design for Ikkari was modified to split Stage 3 into two parts (3a and 3b). This has helped to some extent but there is still a significant dip in the mined grade in 2030 and 2034.

The stockpiling policy (high grading) at both Ikkari and Pahtavaara helps to further smooth out the variation in grade over time as shown in Figure 16.35 and Figure 16.36 but there is still a significant dip in grade in Period 2034 and 2035. It is difficult to see how to resolve this without sacrificing the high grading in the first six to seven years at Ikkari as introducing feed from the underground will not help the situation as the average grade from the Ikkari SLC underground is only 1.53 g/t Au. Similarly, the average grade from Pahtavaara open pit is only 1.36 g/t Au and the only other high grade available is the Pahtavaara underground which cannot be accessed unless the underground is developed before the open pit as the current underground access is from within the old open pit. Given that the cost per tonne for the open pit is less than the underground bringing forward the underground would significantly increase operating costs and advance the capital cost of developing the underground. Therefore, this option is rejected.

Table 16.37 Combined Life of Mine Schedule (3.5 Mt/a SLC)

	Units	Total	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052
Mining Summary																												
Total Rock	**(Kt)**	**281,394**	**4,601**	**9,233**	**15,341**	**16,735**	**13,567**	**12,079**	**24,061**	**26,119**	**23,237**	**16,612**	**5,224**	**14,444**	**15,250**	**9,785**	**10,063**	**12,269**	**15,483**	**14,351**	**9,933**	**7,913**	**2,308**	**1,935**	**624**	**226**	**0**	**0**
Total Waste	**(Kt)**	**202,899**	**4,402**	**5,884**	**11,826**	**13,560**	**8,927**	**8,007**	**20,992**	**21,516**	**22,492**	**11,110**	**2,533**	**10,481**	**10,760**	**5,151**	**5,900**	**8,443**	**11,575**	**10,068**	**5,191**	**3,040**	**499**	**494**	**48**	**0**	**0**	**0**
Total Ore	**(Kt)**	**78,495**	**198**	**3,349**	**3,515**	**3,176**	**4,640**	**4,072**	**3,069**	**4,603**	**745**	**5,502**	**2,691**	**3,963**	**4,490**	**4,634**	**4,162**	**3,826**	**3,908**	**4,284**	**4,742**	**4,873**	**1,809**	**1,440**	**576**	**226**	**0**	**0**
Open Pit Ore																												
Ikkari	**(Kt)**	**36,981**	**198**	**3,349**	**3,515**	**3,176**	**4,640**	**4,072**	**3,069**	**4,603**	**745**	**5,502**	**2,342**	**1,770**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
	(Kt Au)	**77,075**	**462**	**7,583**	**8,626**	**5,315**	**8,513**	**11,206**	**8,430**	**4,237**	**1,094**	**12,369**	**5,210**	**4,029**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
	(g/t)	**2,084.17**	**2.33**	**2.26**	**2.45**	**1.67**	**1.83**	**2.75**	**2.75**	**0.92**	**1.47**	**2.25**	**2.22**	**2.28**	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pahtavaara	**(Kt)**	**5,679**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**97**	**792**	**953**	**516**	**214**	**220**	**645**	**1,101**	**1,140**	**0**	**0**	**0**	**0**	**0**	**0**
	(Kt Au)	**7,712**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**92**	**919**	**1,273**	**880**	**357**	**278**	**1,072**	**1,492**	**1,349**	**0**	**0**	**0**	**0**	**0**	**0**
	(g/t)	**1,358.07**	-	-	-	-	-	-	-	-	-	-	-	**0.95**	**1.16**	**1.34**	**1.71**	**1.67**	**1.26**	**1.66**	**1.36**	**1.18**	-	-	-	-	-	-
Open Pit Waste																												
Ikkari	(Kt)	**132,441**	**4,402**	**5,884**	**11,826**	**13,560**	**8,927**	**8,007**	**20,992**	**21,516**	**22,492**	**10,938**	**2,038**	**1,859**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Pahtavaara	(Kt)	**63,924**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**8,048**	**10,368**	**4,759**	**5,451**	**7,906**	**11,175**	**9,641**	**4,596**	**1,981**	**0**	**0**	**0**	**0**	**0**	**0**
Underground Ore																												
Ikkari	**(Kt)**	**33,526**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**349**	**2,096**	**3,697**	**3,681**	**3,646**	**3,612**	**3,688**	**3,638**	**3,636**	**3,598**	**1,156**	**727**	**0**	**0**	**0**	**0**
	(Kt Au)	**51,152**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**522**	**3,372**	**6,020**	**6,496**	**6,188**	**6,083**	**5,681**	**5,072**	**4,816**	**4,788**	**1,408**	**705**	**0**	**0**	**0**	**0**
	(g/t)	**1,525.74**	-	-	-	-	-	-	-	-	-	-	**1.50**	**1.61**	**1.63**	**1.76**	**1.70**	**1.68**	**1.54**	**1.39**	**1.32**	**1.33**	**1.22**	**0.97**	-	-	-	-
Pahtavaara	**(Kt)**	**2,309**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**6**	**135**	**653**	**713**	**576**	**226**	**0**	**0**
	(Kt Au)	**4,735**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**9**	**229**	**1,269**	**1,540**	**1,246**	**442**	**0**	**0**
	(g/t)	**2,050.62**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**1.54**	**1.70**	**1.94**	**2.16**	**2.16**	**1.95**	-	-
Underground Waste																												
Ikkari	(Kt)	**4,786**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**172**	**494**	**574**	**392**	**392**	**450**	**538**	**400**	**427**	**488**	**460**	**0**	**0**	**0**	**0**	**0**	**0**
Pahtavaara	(Kt)	**1,747**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**106**	**600**	**499**	**494**	**48**	**0**	**0**	**0**
Plant Feed																												
Ikkari	**(Kt)**	**71,945**	**0**	**2,855**	**3,500**	**3,500**	**3,500**	**3,510**	**3,500**	**3,500**	**3,500**	**3,510**	**3,500**	**3,500**	**3,573**	**3,600**	**3,590**	**3,590**	**3,590**	**3,600**	**3,590**	**3,635**	**3,635**	**1,249**	**135**	**135**	**135**	**14**
	(Kt Au)	139,554	**0**	**6,765**	**7,900**	**7,125**	**7,701**	**8,480**	**7,811**	**5,046**	**4,639**	**7,428**	**7,226**	**9,209**	**6,799**	**7,176**	**6,809**	**6,632**	**6,101**	**5,655**	**5,460**	**5,845**	**4,371**	**2,265**	**1,372**	**1,363**	**345**	**32**
	(g/t Au)	**1.94**	-	**2.37**	**2.26**	**2.04**	**2.20**	**2.42**	**2.23**	**1.44**	**1.33**	**2.12**	**2.06**	**2.63**	**1.90**	**1.99**	**1.90**	**1.85**	**1.70**	**1.57**	**1.52**	**1.61**	**1.20**	**1.81**	**10.16**	**10.10**	**2.56**	**2.35**
Pahtavaara	**(Kt)**	**7,988**	0	0	0	0	0	0	0	0	0	0	0	0	407	501	500	500	500	501	500	750	750	752	750	750	750	76
	(Kt Au)	12,448	0	0	0	0	0	0	0	0	0	0	0	0	630	814	820	722	624	754	828	1,293	1,178	1,363	1,508	1,498	379	35
	(g/t Au)	**1.56**	-	-	-	-	-	-	-	-	-	-	-	-	**1.55**	**1.62**	**1.64**	**1.44**	**1.25**	**1.50**	**1.66**	**1.72**	**1.57**	**1.81**	**2.01**	**2.00**	**0.51**	**0.46**

Figure 16.35 Scheduled Tonnes and Grades for the Ikkari Plant Excluding Pahtavaara Concentrate - (3.5 Mt/a SLC Case)

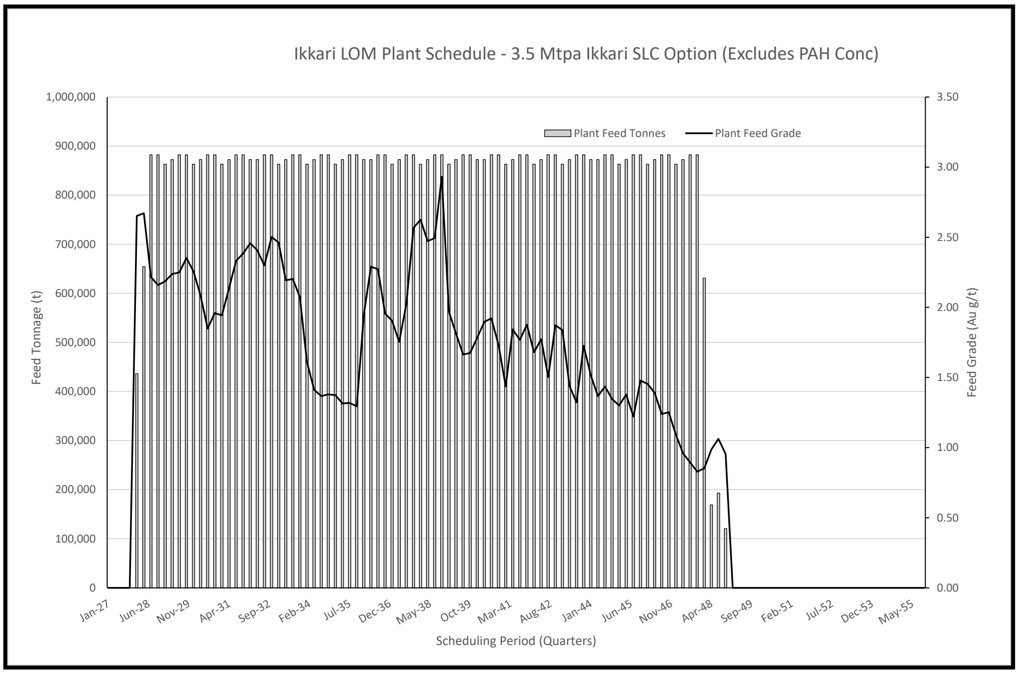
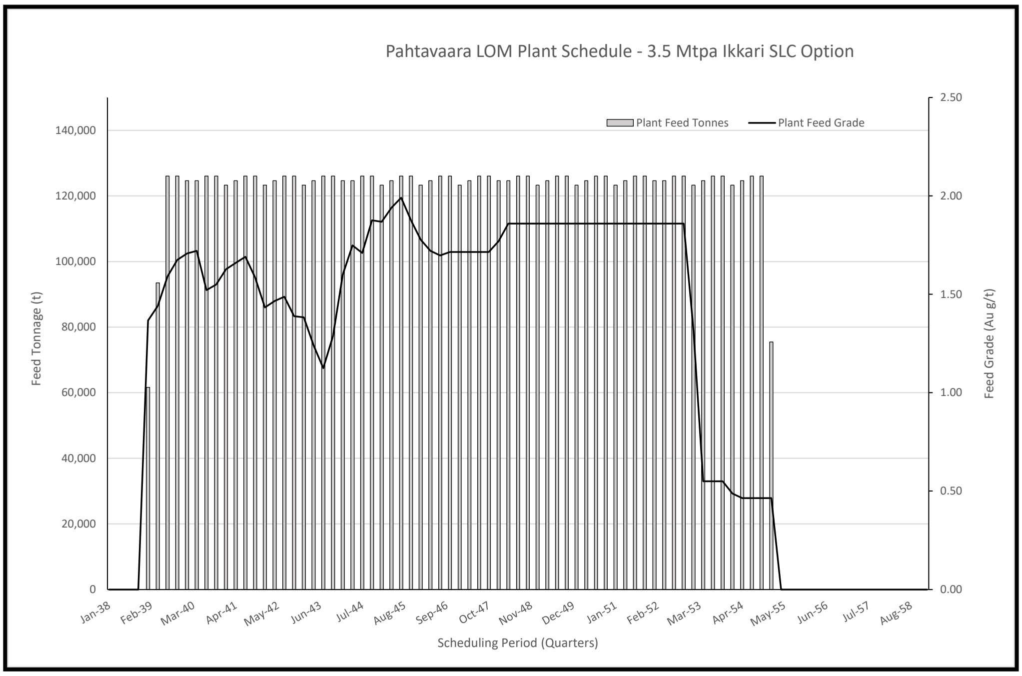


Figure 16.36 Scheduled Tonnes and Grades for the Pahtavaara Plant (3.5 Mt/a SLC Case)

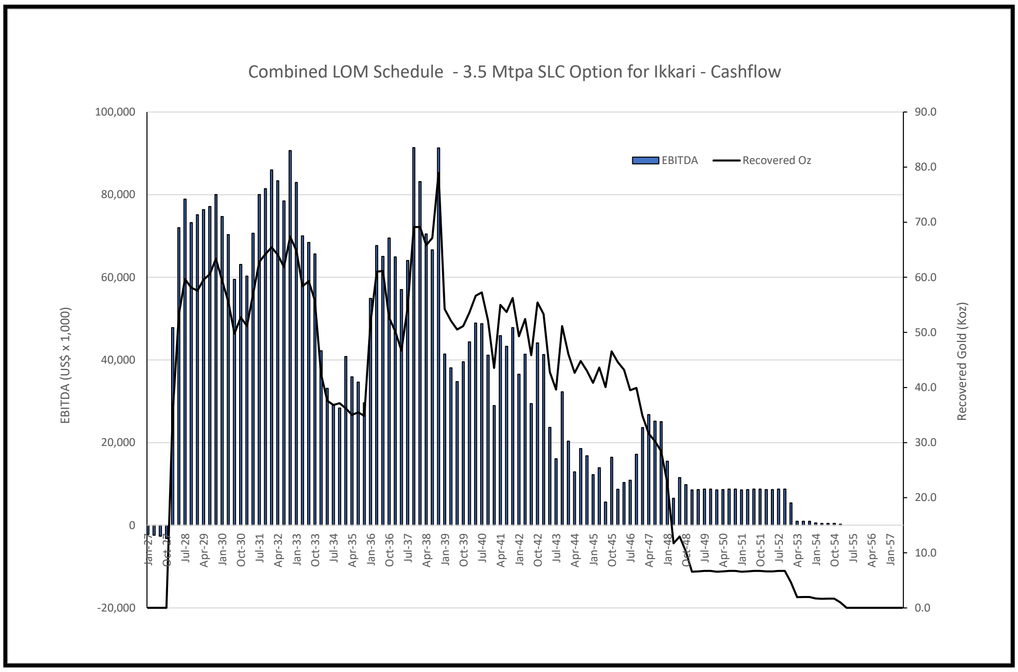


A plot of the cashflow (EBITDA) and recovered gold (Figure 16.37) demonstrates the main features of this schedule, namely;

- Pre-strip to develop Ikkari has been minimised by staging the pit.
- High cashflow maintained for at least six years by high grading Ikkari.
- Recovered ounces exceeds 230 koz/year over the first 6 years.

- Cashflow dips between Jan 2034 and July 2035 and then recovers.
- Average gold production over the first 10 year exceeds 214 koz/year.
- Gold production starts to decline from 160 koz/year by Jan 2039 to 90 koz/year by Jan 2048.
- Mine operations at both Ikkari and Pahtavaara have ceased by 2048.
- Gold production from 2048 to Jan 2055 is primarily from stockpile reclaim.

Figure 16.37 Scheduled Cashflow (EBITDA) and Recovered Gold Ounces (3.5 Mt/a SLC Case)



16.8 Backfill

16.8.1 Introduction

Tetra Tech has reviewed three backfill options for the Ikkari and Pahtavaara project; cemented paste fill, cemented rock fill (CRF) and rock fill systems. The review considered material availability and balances, location, material characteristics, mining requirements and backfill demand as part of the assessment. The requirement for backfill is currently driven by long-term regional stability and waste storage.

The Ikkari and Pahtavaara project is understood to be in the early development and optimisation stage and pending approval, considers a main carbon-in-leach (CIL) processing facility at Ikkari and a concentration plant at Pahtavaara, with subsequent processing at Ikkari.

First-pass optimisation of Ikkari has planned an open pit followed by underground mining. Ore will be processed at the main CIL facility at a rate of 3.5 Mt/a, equating to 23 years life of mine (LOM). Pahtavaara is based on open pit and underground mining (sequence to be determined), producing a concentrate for processing at the Ikkari CIL facility. The processing rate at Pahtavaara is 0.5 Mt Mt/a initially rising to 0.75 Mt/a after 5 years, equivalent to 10 years LOM.

Tetra Tech initially completed a qualitative review and quantitative trade-off study to identify the preferred backfill application for Pahtavaara and Ikkari. Further to this a PEA was completed to better resolve the likely capital and operating costs for Ikkari as backfill at Pahtavaara was subsequently discounted.

16.8.2 Qualitative Assessment

A qualitative review demonstrated paste fill as having the fastest filling cycle, lowest operating cost and uses mine tailings as its primary material but has a high capital cost for the plant and reticulation. CRF was highlighted as being highly flexible, giving a stabilised rock fill for lower capital cost but at a high operating cost driven by high binder and mobile fleet requirements. The review showed that using rock fill in isolation was not viable due the likely requirement of backfill exposure at both sites. Therefore, paste fill and CRF were compared as the primary backfill options with rock fill as a secondary consideration.

16.8.3 Trade-Off Evaluation

The backfill trade-off evaluation showed that the LOM of Ikkari and Pahtavaara are both sufficiently long for the capital cost of a paste fill system to be offset over time by operating cost savings. The shorter LOM at Pahtavaara results in a smaller cost benefit for paste fill over CRF, than that estimated at Ikkari.

Paste fill was selected for Ikkari in the LHOS scenario due to the significant cost savings and additional benefits of fast fill cycles and stable backfill exposure. Paste fill would also integrate into the planned infrastructure (dry-stack co-disposal facility within 3 to 4 km) at Ikkari and potential for bulk mining methods.

The evaluation found CRF to be unsuitable at Ikkari, not only from the costing comparison but operationally as the high mobile fleet requirement would be a significant challenge to the operation.

At Pahtavaara, the method selection is highly sensitive to operating cost and backfill demand / requirement. Although the 10-year LOM is sufficient to show payback on the higher capital cost of paste backfill, operationally, the dispersed mining locations and alternate tailings material sources (concentrate, or existing CDF tailings) may increase the paste fill operating costs significantly.

The lower mining / backfill rate and dispersed mining locations exhibited at Pahtavaara are generally suited to the flexible nature of CRF. The CRF method requires mobile fleet for placement (and mixing), but as the mining rates are lower, this requirement is expected to be accommodated by the planned production fleet. A further opportunity in pursuing a CRF fill, is it will allow loose rock fill to be placed in areas which do not require stabilised backfill, reducing costs.

Based on the current study level, although paste fill is indicated as lower overall cost, CRF is considered to be more suitable at Pahtavaara. The backfill selection can be confirmed when definitive backfill volumes, locations and a schedule is defined.

17.0 RECOVERY METHODS

17.1 TRADE-OFF

A high-level trade-off was completed to assess the probable best processing route for the gold-bearing ores at Ikkari and Pahtavaara.

The criteria for the assessment comprised the following:

- Financial performance. This was assessed using the NPV of the earnings of the process options. Concentrates may be cheaper to produce than smelter products but incur higher TC/RC costs. The difference in net financial return is not intuitive and requires assessment.
- Environmental impact. It is assumed that the plant built will be designed, constructed and operated to comply with local and international environmental standards. This criterion considered the residual environmental risk posed by the reagents, processes, and operations.
- Sustainability and Environmental and Social Governance (ESG). The criterion assesses the extent to which each processing option allows Rupert Resources to meet their sustainability targets.
- Capital cost. It is typical on gold mining projects that the overall project NPV is not sensitive to initial capital expenditures, and this lack of sensitivity has been demonstrated for this project, as shown in Section 22. The criterion was applied to account for the practical limits to the amount of capital available to build the facility.
- Operational flexibility. A facility that can deal with changes in ore from Ikkari, Pahtavaara, satellites and possibly toll-treat third-party ore would be preferred.

If possible, the selected process should be the simplest, most well-understood and most widespread option in commercial use. Process plants that are well understood will be simpler to design, to finance, to build, to commission and to operate. These characteristics make for a venture that will continue to provide value to the community for the longest possible time. See Table 17.4 for comparative notes.

The major environmental factors that were evaluated were the amount of water and the chemicals used for processing the gold.

At the time that the Trade-Off was completed, cost information for Pahtavaara was used for inferring costs for Ikkari using a scaling method and knowledge from other gold project costs. The final assessment of the Project economics utilised quotes for reagents obtained by Rupert Resources. The decisions flowing from the Trade-Off are not materially influenced by the pricing updates.

17.1.1 GSL Test Work

The values used for the Trade-Off represent the test work data from February 2022, which was available at the time of the Trade-Off, and not the process performance data used in the later modelling (September 2022). The decisions flowing from the Trade-Off are not materially influenced by the updates.

The GSL test work results showed that the ore was a free-milling sulphide ore that had good gravity, flotation and cyanide leaching recoveries. The ore had very little organic material and very low levels of uranium, thorium, cadmium, and mercury. The Pahtavaara ore body is very similar in its response to the various treatment processes. Headline numbers used in the trade-off are shown in Table 17.1.

Table 17.1 Headline GSL Test Work Results

Parameter	Unit	Ikkari	Pahtavaara
Sample head grade	g/t Au	1.6 to 4.2	2.5 to 3.5
GRG content	%	47	75
Rougher grind P80	micron	125 to 190	125
Rougher recovery	%	95	96
Flotation mass pull	%	10	30
Cyanide leach recovery	%	96 +	99 +
Cyanide consumption	kg/t	0.3 to 0.5	0.1 to 0.25
Thickening rate (tails)	$m^3/m^2.d$	4,500	2,000
Cyanide destruct CN(wad)	mg/L	2.8	

Key: $m^3/m^2.d$ = cubic metres per square metre per day, % = percent

17.1.2 Process Options

Recovery processes considered included gravity recovery, flotation, and cyanide leaching.

The gravity recovery of gold achieved in the laboratory was between 20% and 40%. Although very good, implemented on its own this extraction method is not financially attractive, nor is such a poor extraction an effective use of Lapland's natural resources. The environmental and social impacts of the mining operation are not justifiable against the low return and associated higher cut-off grades which would not sustain a long-term positive outcome for the community.

The primary choice of processing method was therefore between the production of a concentrate or doré product, using a flotation or cyanide leaching respectively. Gravity extraction was applied as a supplementary process to extract coarse gold from the mill circuit to improve recovery and reduce the size of downstream equipment.

The silver to gold ratio is not suitable to consider Merrill–Crowe, which would also be unnecessarily complicated.

Flotation Process

The flotation process assessed comprised a high intensity gravity separation circuit to remove coarse gold from the mill cyclone underflow. Gravity tailings and cyclone overflow would be floated to produce a concentrate which would be combined with the gravity concentrate. The gravity circuit would not only reduce the risk of coarse gold loss but would also allow a lower mass pull to produce a higher-grade flotation concentrate. The reduction in concentrate production would lead to significant savings in transport and refining charges. The gravity concentrate would be blended with the flotation concentrate.

Cyanide Leach Process

The cyanide leach process assessed comprised a high intensity gravity separation circuit to remove coarse gold from the mill cyclone underflow. Gravity tailings and cyclone overflow would be leached in cyanide followed by carbon-in-pulp (CIP) adsorption and subsequent elution. The gravity recovered gold will be subject to intensive leach. The pregnant solution streams from intensive leach and elution will be combined and electrowon to produce a final doré product.

17.1.3 Operational Scenarios

Owing to exceptional performance for both flotation and cyanide leach at both Ikkari and Pahtavaara, the choice of concentrate or doré product was not a function of metallurgical extraction alone. The difference lay in transport, treatment, and refinery charges. Three scenarios were identified.

- Scenario 1 (the base case):
  - Pahtavaara uses the flotation process described above to produce a concentrate which is transported by truck to Ikkari where it is blended into the Ikkari leach circuit to produce doré.
  - Ikkari uses the leach process described above to produce doré.
- Scenario 2:
  - Pahtavaara uses the flotation process to produce a concentrate which is toll refined.
  - Ikkari uses the leach process to produce doré.
- Scenario 3:
  - Both Pahtavaara and Ikkari use the flotation process to produce concentrates which are sent for toll refining.

At the time of the Trade-Off, initial mine optimisation had indicated optimal production rates for Pahtavaara and Ikkari of 1 Mt/a and 2.5 Mt/a respectively. Scenario 1 required a leach capacity to include the concentrate produced at Pahtavaara.

For the purpose of comparison, the 60 km northern road route from Pahtavaara to Ikkari was used. The shorter southern route runs through Sodankylä, an established residential and commercial centre, and regular movement of concentrate haul trucks will likely cause a significant impact.

Smelter terms are shown in Table 17.2.

Table 17.2 Treatment, Refining and Transport Cost Assumptions

Commercial Terms	Units	Low Grade	High Grade	Doré
Treatment Charge (TC)	$/dmt	225	225	$2.50 /ozT
Au – Payable	%	97.5	97.5	99.9
Refinery Charge (RC) – Au	$/kg payable	1,220	1,220	1,220
Freight	$/dmt	0.027	0.027	-
Insurance	$/dmt	0.007	0.007	-
Assay	$/dmt	0.018	0.018	-
Representative	$/dmt	0.040	0.040	-
Weight Loss	$/dmt	0.003	0.003	-
Concentrate transport and realisation	$/dmt	0.094	0.094	$2.00/ozT

Key: $/kg = dollars per kilogram

Table 17.3 summarises the outcome of the financial assessment of the two processes. The comparison of the financial performance of the three scenarios is based on a differential NPV of EBITDA.

The unit operating cost for each process was based on historical numbers for similar flow sheets, corrected for capacity, complexity and base date. The fixed component of the cost (independent of throughput rate) reflected as a decreasing unit cost with plant capacity. Variable cost components will conversely result in fixed unit cost components. It is acknowledged that the implied linearisation of the cost relationships was a simplification.

For the Trade-Off, it was estimated that at a production rate of 2.5 Mt/a the flotation and leaching options would have operating unit costs of $7.60 /t and $9.60 /t respectively.

These estimates are based on the operating costs (OPEX) expectations at the time of the Trade-Off, rather than the fully updated values used in the final economic evaluation. The updates do not materially affect the decisions flowing from the Trade-Off.

For the purpose of the high-level analysis, a Pahtavaara head grade of 1.6 g/t and an Ikkari head grade of 2.0 g/t were assumed. These were average numbers based on the initial mining inventory produced by Axe Valley.

A gold price of $1,650 /oz was assumed throughout.

Table 17.3 Summary of Scenario Financial Comparison

Item	Unit	Scenario 1 (Base case)	Scenario 2	Scenario 3
Pahtavaara				
Production	Mt/a	1.00	1.00	1.00
CAPEX	M$	25.00	25.00	25.00
OPEX	$/t ore	7.60	7.60	7.60

Process recovery	-	91%	91%	91%
Payable Au	-	100.0%	97.5%	97.5%
	kg/a	1,441	1,405	1,405
	kozT/a	46.33	45.17	45.17
Concentrate production	kt/a	180	180	180
Concentrate trucking cost	M$/a	5.40	-	-
Transport and realisation	M$/a	-	0.02	0.02
Treatment charge		-	40.50	40.50
Refinery charge		-	1.67	1.67
Ikkari				
Production	Mt/a	2.50	2.50	2.50
CAPEX	M$	177	163	46
OPEX	$/t ore	9.60	9.60	6.60
Recovery	-	95%	95%	93%
Payable Au	-	99.9%	99.9%	97.5%
	kg/a	4,850	4,689	4,464
Concentrate production	kt/a	-	-	450
Doré production	kg/a	6,058	4,689	-
	ozT/a	194,774	150,763	-
Transport and realisation	M$/a	0.3895	0.3015	0.0423
Treatment charge	M$/a	0.4869	0.3769	101
Refinery charge	M$/a	7.21	5.58	5.31
Combined				
Payable gold	kg/a	6,284	6,250	5,868
	koz	197.09	195.93	188.68
Transport cost	M$/a	5.79	0.32	0.06
TRC	M$/a	7.70	48.13	148.73
Revenue	M$/a	325.20	323.29	311.32
Net back	M$/a	317.50	275.16	162.59
Delta NPV$_{8\%}$	**M$**	**Base**	**$ -265**	**$ -1,040**

Key: CAPEX = capital expenditure, $/t = dollars per tonne, kg/a = kilograms per year, kozT/a = kilo Troy ounces per year, M$ = million dollars, M$/a = million dollars per year, TRC = treatment and refining costs

17.1.4 Analysis

Scenario 1 offered a higher return on investment. In order to consider all the key stakeholders, it was necessary also to consider:

- Financial performance
- Environmental impact
- Sustainability and Environmental and Social Governance (ESG).
- Capital cost
- Operational flexibility.

Scenario 1 offers better financial returns and is used worldwide to treat a variety of ore bodies while limiting the environmental impact.

Pahtavaara should remain a concentrator. Cyanidation can be managed at the Ikkari operating facility which will be subjected to a greenfield project evaluation, including environmental and social impact.

The perceived downside of a cyanide leach process is mitigated by a wide body of knowledge on how to operate these plants safely. Cyanide destruction in effluent streams can be effected through a number of established technologies. The Grinding Solutions test work indicates that cyanide removal to the required standard is achievable.

Should the permitting for cyanide be declined, it is Tetra Tech's recommendation that a process that utilises gravity recovery within the mill circuit, and flotation of the gravity tails be implemented. This circuit will provide a high recovery of gold and allow the shipping of a concentrate to a toll refinery, much the same as Pahtavaara. To show the potential impact of this outcome, Table 17.4 compares the preferred cyanidation circuit for Ikkari against a flotation circuit.

Note that the table utilises the values for CAPEX, OPEX and resulting net back that was available at the time of the Trade-Off.

Table 17.4 Comparison of Flotation and CIL Options for Ikkari

Criteria	Float Concentrate	Leach Doré
Financial performance	Net back \$145 M at 2.5 Mt/a. High extraction (93%), low grade concentrate (even at 3% mass pull), high TRC (18% mass pull) (\$107 M/a). Can be reduced with lower mass pull. Standard flowsheet used extensively, but not as much in gold. Achieving target grade and recovery may take 3 to 6 months. Variation in mineralogy may affect performance. CAPEX estimate (1.0 Mt/a) = \$25 M.	Net back \$240 M at 2.5 Mt/a. High extraction (95%). TRC small fraction of payable revenue (\$8.1 M/a). Standard flowsheet used extensively worldwide on a variety of ore type. Ramp-up to full production quick. CAPEX estimate (2.5 Mt/a) = \$163 M - \$177 M.
Environmental impact	Flotation reagents are toxic. Concentrate handling and transport will have an environmental and social impact as multiple trucks per hour will use local roads that run through and past towns.	Sodium cyanide used in the leach is toxic in an acidic environment. Its use is highly regulated and its handling requires responsible management. The likelihood of an incident is very low, the impact is severe. Tests have shown that residual cyanide can be destroyed using conventional, proven methods before discharging to a tailings storage facility. Shipment of doré has a comparatively low impact owing to the small volume produced per week.

Sustainability and ESG	Plant has a low carbon footprint compared to pyro and hydrometallurgical facilities. Not all ore types will float well and future satellites may need different processes.	A robust, simple, well understood process will improve the success rate of the project and optimise the value for all stakeholders from the natural resource. The potential to treat material from Pahtavaara improves the remaining return from that existing operation.
Capital cost	A concentrator plant is mostly a modular, vendor-supplied facility with a compact footprint.	A cyanide leach plant will have more unit processes than a concentrator and cost more to engineer and construct.
Operational flexibility	Performance sensitive to ore types: oxides float poorly and can inhibit sulphide flotation performance.	Treatment of ores from multiple deposits is possible.

17.1.5 Post Trade-Off Process Optimisation

Following the Trade-Off study, further work on optimisation of the gold recovery process indicated that the best value and most practical implementation of processing would result from a hybrid recovery plant at Ikkari.

The hybrid concept utilises a high-intensity gravity circuit within the milling facility, combined with flotation of gravity tails and cyanidation of float concentrate.

The gravity system will recover coarse gold which is then subject to intensive leaching.

The milling circuit cyclone overflow is transferred to a flotation plant where the gold is recovered to a much lower volume of material, allowing for a reduction in the size of equipment in the leach train, as well as a reduction in the volumes reporting to the cyanide destruct plant.

The leach liquor reports to a CIP plant, complete with Adsorption, Desorption, Recovery (ADR) facility which generates a concentrated gold-loaded liquor. This liquor is combined with the gold-loaded leachate from the intensive leach system on the gravity circuit and fed to electrowinning for final doré production.

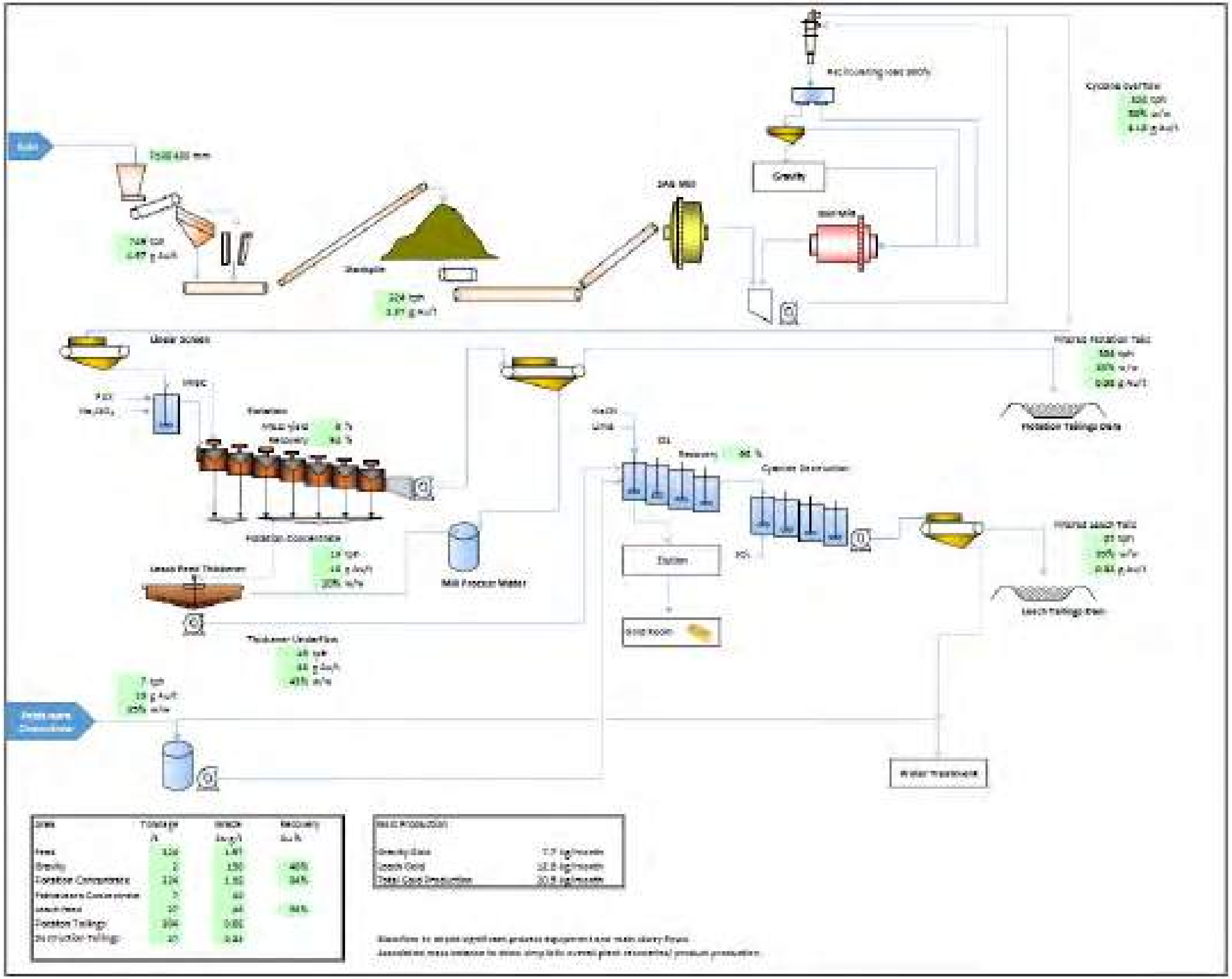
In addition, further optimisation of the mining operations has indicated that a production rate of 3,500,000 t/a is required for the Ikkari facility.

17.2 Ikkari Process Description

BFD – 03627AA-2000-BFD-Z-001

The process plant will treat 3.5 Mt/a of ROM ore from the Ikkari open pit and underground at an average grade of 1.82 g Au/t. The gold in the ore is found to be in the native form or associated with pyrite. The process comprises crushing and grinding to reduce the ROM material to a characteristic grind (P^{80}) of 175 µm, and a gravity circuit to remove the native gold. The pyrite associated gold will be recovered by flotation and fed, with the re-pulped concentrate from Pahtavaara, into the leach circuit where lime and cyanide are added in the presence of air to extract the gold. The gold will be then recovered in an adsorption, desorption, and recovery circuit. The pulp tails will be treated to remove cyanide and filtered for disposal onto the tailings dam. The liquor recovered from the filtration is treated prior to reuse. A simplified schematic of the block flow diagram for the proposed process for gold production is shown in Figure 17.1.

Figure 17.1 Schematic of the Ikkari Gold Treatment Process



17.2.1 Ikkari Basis of Design

The test work results represented in Section 13 Mineral Processing and mining optimisation were used to develop the basis of the design, shown in Table 17.5.

Table 17.5 Basis of the Process Design

Item Description	Units	Value/Comment
Operating Time		
Days per Annum	days/a	365
Statutory Holidays	days/a	0
Operating Days Available	days/a	365
Operating Hours per Day – crusher	h/day	24
Operating Hours per Day - mill	h/day	24
Operating Hours per annum – crusher	h/a	7,720
Operating Hours per annum - mill	h/a	7,720
Throughput (Dry Basis)		
Life of Ikkari Plant	years	24
Pahtavaara Concentrate Plant		
Plant feed rate – year 1 to 7	t/a	500,000
Plant feed rate – year 8 to 14	t/a	750,000
Life of Mine	years	13
Average feed grade	g Au/ t	1.56
Ikkari Processing Facility		
Plant feed rate	t/a	3,500,000
Life of Mine	years	21
Average feed grade	g Au/ t	1.82
Overall Plant Recovery		**94.6**
Pahtavaara Concentrate		
Leach Recovery	%	98
Ikkari		
Gravity Recovery	%	34
Flotation Recovery	%	96
Leach Recovery	%	95

Key: tonnes per year = t/a, w/w = weight for weight

Several unit processes are used for achieving the overall recovery. For further details on the process design refer to Appendix A Process Design Criteria. Each of these processes is described further in the sections that follow. The block flow diagram in Appendix B should be reviewed in conjunction with each of the areas described.

17.3 Pahtavaara Process Description

The process plant will be established using the current infrastructure with updated equipment to treat 0.5 Mt/a of ROM ore for the first 7 years. The process plant will then be upgraded to increase the throughput to 0.75 Mt/a of ROM ore.

The ore will be sourced from the Pahtavaara open pit and the from the underground mine at an average grade of 1.56 g Au/t. The gold in the ore is found to be in the native form or associated with pyrite. The process will comprise of crushing and grinding to reduce the ROM material to a characteristic grind (P^{80}) of 125 µm, and a gravity circuit to remove the native gold. The pyrite associated gold will be recovered by flotation. The concentrate from the gravity circuit and the concentrate from the flotation circuit will be filtered. The filtered product will be transported to the Ikkari processing plant for further treatment.

17.3.1 Pahtavaara Basis of Design

The test work results represented in Section 13 Mineral Processing and mining optimisation were used to develop the basis of the design, shown in Table 17.6.

Table 17.6 Pahtavaara Basis of the Process Design

Item Description	Units	Value/Comment
Operating Time		
Days per Annum	days/a	365
Statutory Holidays	days/a	0
Operating Days Available	days/a	365
Operating Hours per Day – crusher	h/day	24
Operating Hours per Day - mill	h/day	24
Operating Hours per annum – crusher	h/a	7,720
Operating Hours per annum - mill	h/a	7,720
Plant Throughput (Dry Basis)		
Plant operating life (Jan 2039 - Jan 2052)	years	13
Jan 2039 - Dec 2045	t/a ROM	500,000
Jan 2046 - Jan 2052	t/a ROM	750,000
Average feed grade	g Au/ t	1.56
Overall Gold Recovery to Concentrate		**91%**
Gravity Recovery	%	31%
Flotation Recovery	%	87%

Gravity concentration and flotation unit processes are used for achieving an overall recovery of 91%.

17.4 Comminution

The comminution circuit will consist of a single-stage jaw crusher, SAG mill, and closed-circuit ball mill. Cyclone will be used for targeting a P^{80} of 175 µm for size separation to the overflow. The overflow from the cyclone flows into the flotation circuit. The underflow from the cyclone is split with some of the material going to the gravity circuit and the rest returning to the ball mill.

17.4.1 Gravity Gold Recovery

A gravity gold recovery circuit will be placed within the mill closed circuit, on the cyclone underflow to treat a portion of the cyclone underflow. The cyclone underflow will be screened to ensure oversized material is not fed into the gravity concentrator. The screen oversized material and the tails from the gravity concentration will flow by gravity into the mill feed.

The concentrate from the gravity concentrator will be leached in an intensive leach reactor. The pregnant solution will be pumped to the gold room for electrowinning. The washed tails from the intensive leach reactor will be pumped into the ball mill feed.

17.5 Flotation

The cyclone overflow from the comminution circuit will be fed into the flotation conditioning tank where flotation reagents, depressant (Sodium silicate) and collector (Potassium amyl xanthate), will be added. The slurry will then be pumped into the flotation circuit where the frother (Methyl isobutyl carbinol) is added.

The gold is recovered in the flotation concentrate and is pumped to the flotation concentrate thickener for dewatering. The flotation tails are pumped to the flotation tails filtration for dewatering in the tailings area.

17.6 Leaching Circuit

17.6.1 Flotation Concentrate Solid/Liquid Separation

To preserve the water balance and to keep the solution/water containing cyanide separate from process water that does not, a solid/liquid separation is likely to be required to remove water from the flotation concentrate. This will also ensure that the flotation concentrate pulp will be transferred to the leach at the correct pulp density.

17.6.2 Pahtavaara Concentrate Re-pulping

The concentrate produced by Pahtavaara is transported from Pahtavaara to the Ikkari process plant. The material is repulped to the required pulp density in a batch process. The pulp will then be transferred to a storage tank for addition to the leach feed.

17.6.3 Leaching

The flotation concentrate pulp and Pahtavaara concentrate pulp are fed into the leach feed tank where lime is added to adjust the pH to 10.5. The pulp is then transferred into the first leach tank where cyanide and air are added to the train. The leach train has a 48-hour residence time, with the gravity transfer of pulp between each of the tanks.

17.7 ADSORPTION/DESORPTION/RECOVERY CIRCUIT

17.7.1 ADSORPTION - CARBON IN PULP

The discharge from the leach train flows by gravity into the carbon in pulp (CIP). The carbon moves counter-current to the slurry with the dissolved gold adsorbing onto the carbon. Loaded carbon is transferred from the first CIP tank to the loaded carbon tank for desorption of the gold and regeneration of the carbon. Regenerated carbon is added back to the last CIP tank in the train.

17.7.2 DESORPTION

The loaded carbon is transferred from the loaded carbon tank into the acid wash column. Where the loaded carbon is washed with a hydrochloric acid solution to remove lime that has adsorbed onto the carbon.

The carbon is then transferred to an elution column. The gold is stripped off the carbon, under high pressure and temperature, with sodium hydroxide and cyanide solution. The Eluate is then transferred to electrowinning for gold recovery in the Gold Room. The barren carbon is then transferred to the regeneration kiln for regeneration.

17.7.3 CARBON REGENERATION

The barren carbon is added to the regeneration kiln where it is heated to 750°C and regenerated. The regenerated carbon is quenched and transferred back into the last CIP tank in the train.

17.8 GOLD ROOM

The gold room handles the electrowinning of the pregnant solutions from the Adsorption/Desorption/Recovery circuit (ADR) and the gravity circuits. A gold sludge is produced from the electrowinning. This is separated from the solution and calcined. The calcined material is then smelted to produce gold doré.

17.9 CYANIDE DESTRUCTION

The slurry discharged from the Carbon in Pulp train is pumped to the cyanide destruction tanks. Sulphur dioxide is added for the destruction of the cyanide, with the pH being maintained at eight using lime. The slurry is retained in the cyanide destruction for 180 minutes before being transferred to cyanide destruction filtration in the tailings area.

17.10 Tailings Storage

17.10.1 Flotation tailings

Tailings from the flotation circuit will be filtered. The filter cake will be transported to the flotation tailings storage facility for dry stacking. The filtrate from the filtration will be pumped to the Mill process water tank for redistribution.

17.10.2 Cyanide destruction Tailings

Tailings from the cyanide destruction circuit will be filtered. The filter cake will be transported to the lined cyanide destruction tailings storage facility for dry stacking. The filtrate from the filtration will be pumped to the water treatment area for treatment.

17.11 Water

Water is recirculated within the plant to minimise water consumption and to ensure that water of a similar quality is kept together.

17.11.1 Mill Process Water

Mill process water tank is topped up with water from the raw water source. Water is recovered from the flotation tails filtration and flotation concentrate thickening into the mill process water tank. The water from the mill process water tank is used in the comminution area.

17.11.2 Water Treatment

Water recovered from the cyanide destruction filtration is treated in the water treatment area. The treated water is used for the repulping of the Pahtavaara concentrate and for reagent makeup. The brine produced by the water treatment plant will be added to the cyanide destruction tailings

18.0 PROJECT INFRASTRUCTURE

This section describes on-site and off-site infrastructure at both the Ikkari and Pahtavaara project sites The planned on-site infrastructure includes the road network, processing plant, mine support facilities, water treatment, power supply and distribution, waste and tailings storage facilities and sewage treatment facilities. Off-site infrastructure includes the access roads, high voltage (HV) power lines to the sites and water intake and discharge pipelines.

The Rupert Lapland Project is made up of a planned new facility close to the Ikkari Deposit and the existing Pahtavaara Gold Mine that has existing infrastructure in place that will be upgraded prior to the start of production in years 9 and 10 of the project.

Surface infrastructure at Pahtavaara includes a mobile equipment workshop, administration building, core sheds, and a mineral processing plant using gravity recovery. For the most part, the facilities have been well maintained since the operation was last in production in 2014 and are in good condition. The site has access, power, water, and communications systems in place. The office and mobile workshops and fueling facilities are in place and have been used by the company workforce. The mine change house is small for a re-start scenario and would need to be expanded with proper shift change lineup area for supervisors and safety personnel. A first aid room is in place, but a mine rescue room would need to be added to restart the mine. The process plant is permitted for 500 kt/y and is in good condition with the mills being rotated at regular intervals.

The subsequent sections focus only on the Ikkari development site. A detailed assessment of the infrastructure, hydrology, waste management and tailings storage facility upgrade requirements for the Pahtavaara Gold Mine would be undertaken in subsequent engineering studies.

18.1 SITE CONDITIONS

The nearest weather station is based in Sodankyla, some 40 km from Ikkari. The Köppen Climate Classification subtype for this climate is "DFC" (Continental, without a dry season and a cold summer).

18.1.1 DESIGN TEMPERATURES

The average temperature for the year in Sodankyla is 1°C. The warmest month, on average, is July with an average temperature of 14°C. The coolest month on average is February, with an average temperature of -13°C.

Table 18.1 Monthly Temperature (°C)

	Month											
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Average	-13	-13	-9	-3	4	10	14	11	5	-1	-6	-10
Minimum	-18	-19	-16	-8	0	5	8	6	1	-4	-10	-15
Maximum	**-8**	**-8**	**-3**	**2**	**9**	**16**	**20**	**17**	**10**	**2**	**-2**	**-6**

Source: www.weatherbase.com

18.1.2 Precipitation

The average amount of precipitation for the year in Sodankyla is 470 mm. The month with the most precipitation on average is July and August with 60 mm of precipitation.

Table 18.2 Monthly Precipitation (mm)

	Month											
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Average	20	20	20	20	30	50	60	60	30	40	30	20
Annual	470											

Source: www.weatherbase.com

18.1.3 Humidity

The recorded monthly average for the region ranges between 67% in June to 91% November.

Table 18.3 Average Relative Humidity (%)

	Month											
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Average	87	86	78	73	68	67	71	80	84	90	91	89

Source: www.weatherbase.com

18.1.4 Wind

The wind is most often from the South by Southwest, ranging between southerlies and westerlies for the entire year. The annual average wind speed is 11 kilometres per hour (km/h) (3.1 metres per second [m/s]), with the highest speed recorded in March and the lowest recorded in January.

Table 18.4 Average Monthly Wind Speed (km/h)

	Month											
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Direction	SSW	SSW	WSW	WNW	WSW	W	S	SSW	SSW	S	SSW	S
Average	9	9	11	11	13	11	11	11	11	9	9	9
Maximum	19	20	28	26	26	26	24	24	22	20	20	19

Source: www.weatherbase.com

18.2 EXISTING INFRASTRUCTURE

Pahtavaara process plant equipment was decommissioned and mothballed in 2016. The equipment was decommissioned and stored professionally, although some of the equipment will need to be overhauled before it can be put back into service.

The primary crusher and conveyor system structure will need work to get back into working order. The mill has been well maintained and could be put back into service without a major overhaul. It should be noted that the control panels are outdated and would need to be upgraded as some point in the future

Increasing the capacity of the current processing plant would require additional equipment and new infrastructure to be able to distribute the ground material by gravity. There is no space for adding additional equipment to increase the plant throughput. A smaller feed size to the mill could increase the throughput capacity but would need to be evaluated to understand the limitations.

Pahtavaara tailings dam has grass seed growing on the dam, this has been undertaken to limit the amount of dust created by the wind. The water flows through three dams before flowing into a natural swamp or peat bog. The discharge from the swamp flows into the river. This creates the opportunity for the water to be recycled back into the process plant to limit the amount of raw water pumped from the river. The tailings dam could potentially be lifted to increase capacity. This will need to be evaluated further.

18.3 GENERAL SITE LAYOUT

The Ikkari plant site location was selected to provide a relatively direct flow of material from mine to tailings storage. The plant facilities were arranged to minimise civil work and locate major equipment in areas with favourable geotechnical characteristics while maximising gravity-assisted material flow where possible.

Major design objectives influencing the site location and arrangement were as follows:

- Major equipment should be located on sound and competent ground, away from unstable features. Subsurface conditions must be competent enough to support the heavy static and dynamic loads from the crushing and milling equipment.
- The processing facility location should be sufficiently large to accommodate the infrastructure in a contiguous area (allowing connected terraces to minimise earthworks) and minimise the distance between the facilities.
- The facilities should be as compact as possible to minimise capital and operating costs.

- The coarse ore stockpile should be located away from or upwind of dust-sensitive facilities, such as offices, plant facilities and the electrical substation.
- The facilities should be located to prevent interference with ultimate pit limits, anticipated waste dumps and potential low-grade ore stockpiles. Arrangement of the facilities will promote segregation between mining and concentrator traffic during both construction and operation for safety and congestion considerations.
- The facilities must be a minimum of 500 m from the mine pit limits (blast radius).

The location of the ROM Pad and primary crushing station will take advantage of the natural terrain and does not conflict with waste dump and ore stockpile extents.

The processing facility will be situated on a gentle, natural slope bounded. The sloped terrain allows gravity flow of the major process streams through the in-line plant and provides good ground conditions for major equipment foundations. Terraces will be used to minimise civil excavation and structural fill.

The administration complex will be located on the highest terrace, separated by grade from the processing facility. The water ponds and tailings storage and reclaim are located at the lowest platform. This allows the large, multiple pipelines from water ponds to enter and exit the plant site without interfering with process piping.

Traffic logistics were a major consideration in configuring the plant facilities. Delivery and mine traffic will be separated with Ikkari haul vehicles arriving from the west, Pahtavaara haul vehicles arriving from the east and delivery vehicles arriving from the south.

18.4 ON-SITE ROADS

Light vehicle roads will connect all facilities for maintenance and to support operations. Heavy-vehicle roads will connect the pit to the primary crushing facilities, waste dumps, ore stockpiles, tailings storage and reclaim facility, and provide access for heavy-vehicle services, including the fuel dispensing station and truck shop. Light-vehicle and heavy-vehicle roads will be separate for safety reasons.

18.4.1 ON-SITE LIGHT-VEHICLE ROADS

The main on-site, light-vehicle road connects the administration area to the gatehouse, plant site, CDF and the mine. The road platform will be made up of two 3.5 m lanes (one in each direction), two usable shoulders of 0.5 m each, and room for a cut-side ditch. As a result, the total platform width will be 9.6 m. All light-vehicle roads will be topped with 0.2 m thick granular surface course.

18.4.2 ON-SITE HEAVY-VEHICLE ROADS

Haul roads within the mine will be designed according to international mine design standards. The main haul road connects processing facility with the pit and tailings facility. The road platform will be made up of two 9.0 m lanes (one in each direction), two usable shoulders of 1.0 m each, with room for a berm and cut-side ditch. As a result, the total platform width will be 24 m. All haul roads will be topped with 0.4 m thick granular surface course.

18.5 STRUCTURAL DESIGN

The Ikkari project site is located in a Low Seismic Zone with 0.0 to 0.1 g expected peak ground acceleration. This is a very low seismic zone, equivalent to International Building Code IBC-2018 Seismic Design Category A.

High prevailing winds of 13 km/h (26 km/h maximum) and low winter temperatures (-19°C low and -13 C average) prompted the decision to enclose buildings throughout the site. These enclosures provide protection from blowing dust and snow and allow the ambient temperature of working and wet areas to remain above freezing. This in turn will help maintain the productivity of personnel who are already subjected to the rigors of working in an arctic environment.

The 50-year return period wind three-second wind gust (68 km/h) governed the design of some structural components (roof purlins, wall cladding girts, etc.). The 150-t crane in the grinding building is the primary structural design consideration. Snow load is minimal and does not dictate structural design.

18.5.1 FROST SUSCEPTIBILITY

There is a high frost hazard potential at the Ikkari plant site. The majority of the foundations will be on bedrock and because water is readily available for the formation of ice lenses in the frost-susceptible fill. It is recommended that foundation drains are installed to drain excess water away from the foundations.

18.6 SITE BUILDINGS

18.6.1 PROCESS PLANT

The process plant will be located on the Ikkari property. Process plant buildings are summarised in Table 18.5. All process plant buildings, and ancillary buildings, conveyors and other equipment discussed in this section, will be equipped with fire suppression systems.

Table 18.5 List of Process Buildings

Building Description	Building Construction	L (m)	W (m)	H (m)	Area (m^2)
Crushing Building	Fabric	42	10	20	420
Covered Stockpile	Fabric	-	55Ø	22	2,375
Mill Building	Pre-Engineered Portal Frame	57	36	27	2,050
Flotation Building	Pre-Engineered Portal Frame	150	45	16	5,030
Leach Building	Pre-Engineered Portal Frame	102	98	19	3,700
Reagent Store	Fabric	36	21	16	750
Pahtavaara Feed Handling Building	Fabric	17	11	16	190

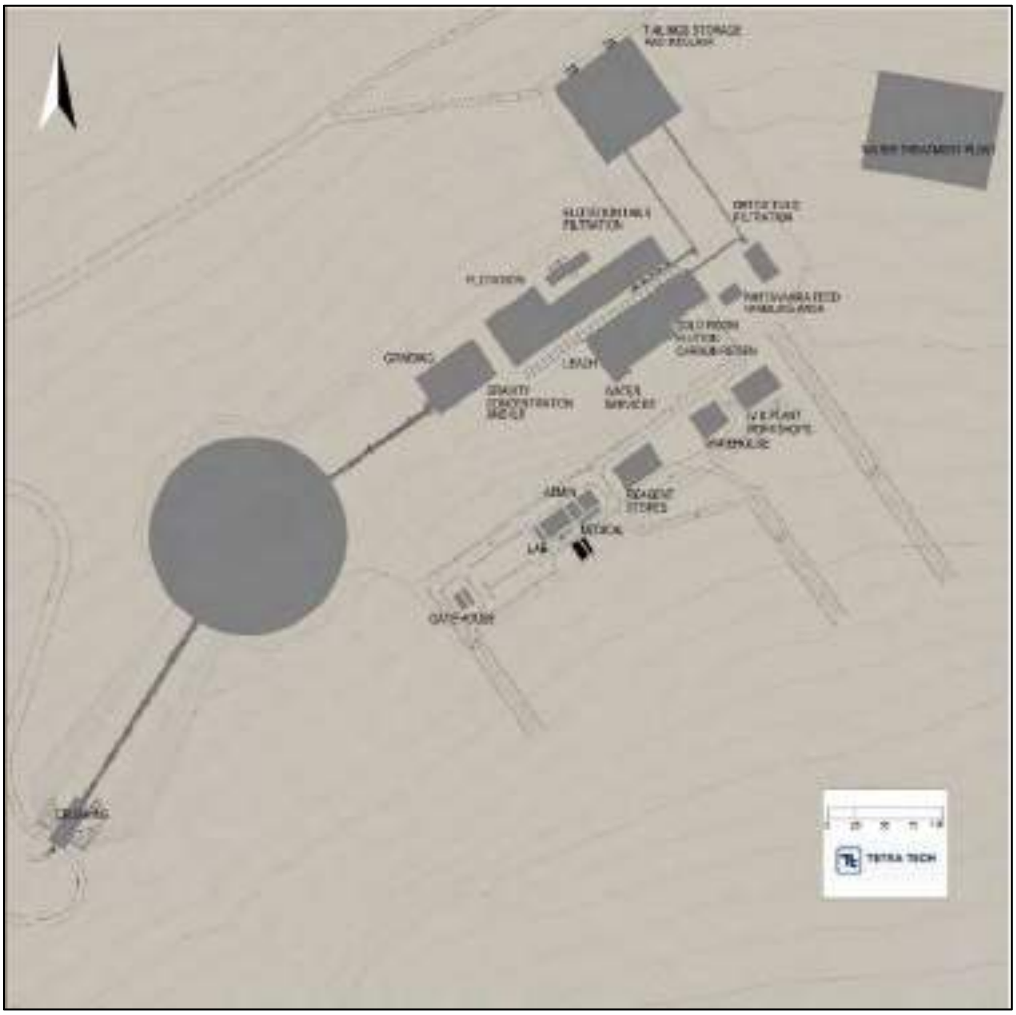
Key: Ø = diameter

The crushing building, and the stockpile cover, will be of fabric design and equipped with dust collection systems. The crushing building will house the ROM hopper equipped with a static grizzly, primary jaw crusher, chutes and additional platework. The rock breaker will also be within the building. In addition, access platforms and reinforced concrete will be utilised for the pad to support the primary jaw crusher. Secondary screening and crushing will also be completed prior to conveyance to the mill feed stockpile. Conveyors are used in the nominal operation to move the crushed material through the crushing circuit and do not rely on regular use of mobile equipment. A fabric building cover and concrete reclaim tunnel will be used for the mill feed stockpile.

The mill building, which includes the gravity concentrator and intensive leach reactor, will be constructed from pre-engineered portal frame, supported on reinforced concrete footings complete with concrete slabs and pedestals. To account for winter conditions, buildings will be built with insulated metal panel (IMP) roof and wall cladding. Area cranes will be available for equipment servicing in the mill plant. The mill building includes a ground floor and one elevated concrete floor. The various equipment will be accessed by purpose-built mezzanine platforms for maintenance, service and sampling. The mill building will contain the ball and SAG mills, cyclones, trash screen, as well as dedicated areas for the gravity circuit equipment and Intensive Leach Reactor.

The flotation building, which includes the flotation cells and the flotations tails filters, will be constructed from pre-engineered portal frame, supported on reinforced concrete footings complete with concrete slabs and pedestals. To account for winter conditions, buildings will be built with IMP roof and wall cladding. Area cranes will be available for equipment servicing in the flotation plant.

Figure 18.1 Proposed Plant Layout



The leach building, which includes the leach cells, acid wash column, the elution column, regeneration equipment, gold room, Detox filters, reagent mixing area and water services, will be constructed from pre-engineered portal frame, supported on reinforced concrete footings complete with concrete slabs and pedestals. To account for winter conditions, buildings will be built with IMP roof and wall cladding. Area cranes will be available for equipment servicing in the leach plant.

The reagent building will be of fabric design and will contain all stored reagents. The Pahtavaara feed handling building will also be of fabric design.

18.6.2 Pahtavaara Concentrator

Please see section 18.2 Existing Infrastructure.

18.6.3 Administrative Complex

The administrative complex will be northeast of the process plant, as shown in Figure 18.1. The facilities in this complex include the following:

- Administration building.

- Lunchroom/change room.
- Emergency response centre.
- Maintenance shop and plant warehouse.
- Reagent store.
- Light vehicle and plant workshop.
- Laboratory.

The main administration building will be comprised of three modular sections (lunchroom and office sections) and one stick-built section (emergency response centre). The three sections will be interconnected by a covered walkway. The building will have a disability access ramp.

The administration office will be a modular building 13 m by 50 m long by 5.5 m (eave height) with a total area of 1300 m^2. It will be comprised of 25 offices, 50 workstations, 5 meeting / training rooms, 1 centralised mine control room, 1 centralised process control room, and various service rooms. The building will be used by both the mine and plant operations groups. The building will accommodate approximately 40 mine technical and supervisory personnel per shift. Two separate main entrances will be provided for the two groups of personnel and washrooms and connecting corridors will be shared.

The mine and plant control rooms are adjacent to each other at the centre of the building and are equipped with an elevated service floor.

18.6.4 Gatehouse/Security Building

The modular gatehouse/security building will contain a security control/checking room, washrooms, a guard/personal protective equipment (PPE) room, lunchroom, and orientation room. Disability access for visitors will be included in the design as required. The building will be equipped with plumbing, electrical, lighting, data/communications, fire, Heating Ventilating and Air Conditioning (HVAC) and security systems, as necessary for the function of the building.

18.6.5 Fuel Storage

Fuel will be stored on site to supply arctic grade diesel fuel for heating and to power diesel-powered mobile equipment. The tanks will be located at least 100 m from main buildings and the prevailing wind directions will be considered.

At least one month of diesel storage will be required to give the mine the security of supply throughout the winter months. A further assessment will need to be carried out to determine the location and supply logistics for the fuel source and to determine the final site fuel requirements.

Diesel dispensing and accounting equipment will also be provided. Sealed containment equivalent to 110% is required to contain any spillage.

18.6.6 Heating Plant

A central heating plant will be located near to the offices and process plant building to provide efficient heating during the winter periods.

18.6.7 Laboratory

An assay laboratory will be near to the process plant and will contain a simple preparation area, a chemical laboratory for standard mineralisation analysis, and an environmental analysis facility.

Additional laboratory facilities will also be provided to support the mining operation and grade control activities.

18.7 Heating, Ventilation and Air Conditioning

The site ambient outdoor design temperatures used for heating and cooling calculations are -19°C dry bulb temperature in the winter (February) and +20°C dry bulb temperature in the summer (July). The ambient air is dry and often dusty. The average annual temperature is -1°C.

Heating systems will maintain a minimum air temperature of 5°C in process buildings that are infrequently occupied. A minimum room temperature of 20°C and maximum of 24°C will be maintained in the administration buildings, control rooms, electrical rooms, laboratories, and all other human-occupied spaces.

A maximum room temperature of 30°C will be maintained in the electrical rooms. All air-conditioned spaces will be maintained within a relative humidity of 25% to 65%.

Ventilation for occupied, non-process buildings (administration, offices, warehouses, etc.) will be based on ANSI/ASHRAE Standard 62.1. Where possible, the site will make use of all excess heat generated by the processing operation alongside geothermal based heat exchangers for heating the floors of all "warm" buildings.

Heat recovery fans in high buildings will be used for transferring warm air from the roof underside to lower areas through distribution ductwork. In the event of a fire, the supply air fan serving the affected area will shut down automatically. Ducts penetrating areas of fire separation will have fire dampers.

18.8 Fire Detection and Protection

Fire protection facilities will incorporate both passive and active systems. Passive systems are features that, by nature of design, resist heat damage, facilitate safe evacuation of people, and aid fire suppression operations. Active systems involve the use of systems and equipment specifically intended to extinguish or control fires, protect people or surrounding property from fire, and warn of a fire emergency. Examples of both types of systems are listed in Table 18.6.

Table 18.6 Passive Versus Active Fire Protection Systems

Passive	Active
Spatial separation	Fire detection (heat/smoke)
Drainage	Fire water systems, hoses, hydrants, sprinklers, monitors hoses
Fire separation	CO^2 gas suppression
Materials of construction	Fire alarms
Grounding	

Key: CO^2 = carbon dioxide

General design features are as follows:

- Smoke detectors and CO^2 hand-held fire extinguishers will be installed in all electrical rooms, Variable Frequency Drive rooms, and control rooms. Fire protection for "mission critical" electrical rooms will utilise clean agent (gaseous) fire suppressant room flooding.
- Electrical rooms will have two-hour fire separations.
- Hand-held, all-purpose standard ABC fire extinguishers will be provided in all buildings for local emergency firefighting.
- Smoke and heat detectors will be installed in all occupied areas not equipped with sprinklers.
- Duct smoke detectors will be installed in all air-handling units. Once a duct smoke detector is activated, the associated air-handling unit will shut down.

Firewater will be available at facilities and buildings by wet standpipes, sprinklers, and yard hydrants connected to the firewater loop, so that all areas of the facility are within reach of a hose stream. Monitors mounted on hydrants will allow water to be directed to specific hazards, such as the transformers. The firewater loop will be designed so that water can be provided from both directions.

One fire vehicle will be available for mobile firefighting.

18.9 ELECTRICAL DISTRIBUTION SYSTEM

The current estimated electrical power requirements for the processing facility are in the region of 15.65 megawatt (MW) (Table 1.7).

Table 18.7 Estimated Electrical Power Requirements

Area	Required Power (kW)
Crushing	188
Grinding	11,975
Flotation	285
Leaching	1,255
Process Ancillaries	250
Water Treatment	1,500
Administration & Maintenance	150

The site is located 3 km from 220 kV power lines and 5 km from an existing transformer station, which it is assumed has the capacity to supply the Project.

The total operating mine site load has been estimated to be between 18 to 25 MW, including any electrified mining equipment. The electrical design shall be designed to meet this need.

18.10 Instrumentation, Control and Communication Systems

Instrumentation and controls will incorporate conventional 4-20 milliamps (mA) analog with highway addressable remote transducer (HART) control and 24 Direct current volt (VDC) discrete control signaling. Field devices will be connected to field remote input/output I/O (RI/O) panels, which will then connect via industrial Ethernet over single-mode, fibre-optic cable to process control system (PCS) controller panels located in the electrical room.

The controller panels will contain redundant power supplies and controllers and will connect to redundant control system network core switches and process controller server equipment located in a central control room and adjacent control system server room.

The control system cable network will consist of optical ground wire (OPGW) run on overhead powerlines to off-site locations and conventional fibre cabling distributed throughout the concentrator process areas using armoured cable and cable tray. Both modes will be part of the plant-wide integrated fibre backbone network.

Internet communications fibre will be included in a 24-strand OPGW cable to be run with the incoming site power line from the main road to the south of the project.

Industrial Ethernet will be used for control system interfaces with motor starters and variable frequency drives. The central control room will contain three operator Human machine interface (HMI) control stations and two engineering workstations. Two remote control cabs provided at each primary crusher will contain a single operator workstation in each. Vendor-supplied Programmable Logic Controller (PLC) control systems will connect to the Plant Control Systems (PCS) via industrial Ethernet fibre cable.

The PCS will be based on a distributed control system platform. This plant-wide system will include redundant controller panels, remote I/O panels, HMIs, peripherals, networks and complete logic and control screen(s) graphic development.

Plant Local area network (LAN) communications racks, including business and process Ethernet network equipment, will be installed in identified electrical rooms and process and office buildings. Fibre distribution panels will be integrated into these racks to provide interconnection of the network switches and dedicated interconnection of various process, business and fire detection systems. Voice and data systems will be integrated using VLAN separation.

Various types of systems will support different operations and business needs, as follows:

- Voice over IP telephone services and computer networking within buildings.
- Handheld radios for remote operations within the plant area.
- LAN.
- Wide area network (WAN) connection to locations outside the plant (Internet service).

18.11 Hydrological Review

18.11.1 Introduction

Hydrological reviews are often loosely framed along the lines of the classical hydraulic cycle, describing firstly the local climate, then the surface water features, run-off and recharge, the groundwater environment and culminating in discharges to streams and rivers at the distal end of the catchment. The layout of this section is modelled on that format with a review of existing climatic, surface and groundwater data relevant to the project. This is used to develop a preliminary conceptual hydrogeological model (CHM) and together with physical parameters obtained from field testing, to build and calibrate a numerical groundwater model. The model, which is described later in Section 18.11.5, is used to predict inflows to the future open pit and underground operations.

18.11.2 Climate

The main source of climate data used for the hydrological study is the VEMALA model, which is in the form of daily temperature, precipitation and evaporation for the period 1960 to 2022. The model has been developed for the Finnish Environment Institute (SYKE) to simulate and forecast watershed systems.

A more complete summary of the climate has already been provided in Section 5 and Section 18.1 of the PEA report; however, it is important to emphasise that the large variations in temperature and light conditions over the course of the year have a considerable bearing on the dynamics of the local surface and groundwater regimes. The sub-zero temperatures during the arctic winter mean that virtually all precipitation is locked up as snow resulting in negligible run-off and recharge. The rapidly lengthening days and increasing temperatures in April and May initiate a quick thaw and a brief but intense period of high run-off as the accumulated precipitation of the previous winter is released to the catchment over a period of three to four weeks. The mean annual precipitation (MAP) recorded at the Ikkari site is 479 mm of which some 30% to 35% is released during the thaw.

18.11.3 Surface Water

The project site is characterised by gentle relief, with the floor of the Saittajoki valley occupied by a mix of streams, springs, ponds and aapa mire, and the adjacent ground to the SE rising to form a shallow ridge, the apex of which is some 60 m above the valley floor. The valley floor around the deposit is dominated by wetland and by the Saittajoki stream flowing through its centre.

The Saittajoki is the main watercourse and lies within the larger Kemijoki catchment, with surface waters draining eastwards towards the Sattanen and Kitinen Rivers. It is fed by a mixture of surface run-off and groundwater springs. The hydrograph from stream gauges installed by Rupert Resources in October 2021 show that flows peak just below 3 Cumecs during the spring thaw in May and then fall rapidly to stabilise around 0.2 to 0.3 Cumecs during the summer and autumn months. Over the winter period, flow steadily diminishes as recharge stops and the store of water from springs dries up until flows are negligible (<0.01 Cumecs).

18.11.4 Geohydrology

Physical Hydraulic Properties

Peat and Glacial Sediments (Superficial Cover)

Most of the project area is covered by a veneer of glacial sediment and peat, with this cover being most developed in the topographic depressions and pinching-out against higher ground where there is occasional bedrock exposure. The sediment cover on the valley floor ranges between 10 m in the upper parts of the catchment and 30 m towards the distal end, in the east.

Recent coring has shown the peat to be relatively thinly developed across the Saittajoki catchment with a median thickness of 1.4 m and a maximum of 4.3 m. There has been no hydrogeological testing of the peat, but using the relationship between thickness, humification levels and hydraulic conductivity developed by Jennings and Johnston (2013), the hydraulic conductivity of the peat at Ikkari might be expected to range between 10^{-6} and 10^{-7} m/s over most of the catchment, but could be as low as 10^{-8} m/s in areas of thickest development.

The glacial sediment is dominated by finer grained silts and sands in what has been described as a diamicton, with less common outcrops of esker moraine containing gravels and coarse sands. Permeameter testing and particle size distribution (PSD) analysis of the sediment reveal that the geomean hydraulic conductivity is about 10^{-6} m/s, which is typical of a silt, and the finer, clay-rich fractions common near the base of the sediment have an approximate hydraulic conductivity of 2×10^{-8} m/s and the coarser fractions present in moraine have an approximate hydraulic conductivity of 2×10^{-5} m/s.

Bedrock

The local bedrock consists of greenschist-facies metamorphosed mafic to ultramafic volcanic sediments, forming part of the CLGB. Based on the results of 68 successful packer tests and cross-hole pumping tests in 3 exploration holes, the geomean hydraulic conductivity of the bedrock is 6×10^{-7} m/s, although the range about this central value is considerable (1×10^{-9} to 5×10^{-5} m/s), which very likely reflects the heterogeneity of the fractured crystalline rock mass. The results show no strong correlation with rock type but are more closely related to the depth of testing and the incidence of faulting and brecciation. The hydraulic conductivity of the rock mass decreases with increasing depth, although the

Ikkari deposit is unusual in intersecting several features at 400 m+ depth with hydraulic conductivity as high as 10^{-5} m/s. The normal distribution of Log10 hydraulic conductivity values shows a noticeable skew right of the mean in favour of higher hydraulic conductivity between 3×10^{-7} m/s and 1×10^{-5} m/s.

Brittle style faulting, more commonly associated with groundwater flow is evident in such structures as the Rajala Line that strikes E-W through the deposit. Core logging also indicates potentially more pervasive fracturing along lithological contacts where the partings have likely arisen because of significant differences in material competence and ductility during folding.

Bedrock storativity values from cross-hole pumping tests range between 5×10^{-5} to 3×10^{-4} and average 10^{-5}. Storage in the rock mass may be enhanced locally by dissolution features present in dolomitic vein deposits, although the connectivity of these features is poorly understood.

Finally, clay has been intersected towards the base of several shallow monitoring wells targeted at the superficial sediments in the centre of the Saittajoki valley. Whether of glacial origin or the product of intense weathering of sheared bedrock, this clay will have low hydraulic conductivity properties and, where it exists serve to confine the groundwater in the underlying rock mass.

PIEZOMETRY

Peat and Glacial Sediments

The average depth to groundwater in the glacial sediments is approximately 1.5 metres below ground level (m bgl) across the study area. The phreatic surface displays fluctuations during the spring thaw, between the end of April and the end of May, with amplitudes ranging from 0.2 to 2.6 m. These are typical of the end of the winter period in Nordic regions, with the spike in level indicating the spring melt, followed by a gentle recession persisting through to the autumn.

The groundwater surface follows the topography, with flow from more elevated ground towards the water courses into which discharge occurs either as groundwater baseflow from the banks and bed of the streams, or from springs. Overall, the direction of flow is ENE towards the lower end of the Saittajoki catchment.

Bedrock

The groundwater levels associated with bedrock in the valley bottom are predominantly artesian (16 sensor observations out of 20) and higher than the groundwater in the overlying superficial deposits. The artesian elevations are on average 2.3 m above ground level although there are two outliers where the piezometric head is respectively 17 m and 27 m above ground level. The strong upward pressure gradients evident in the Vibrating Wire Piezometer (VWP) data suggest that the local faults are important conduits for groundwater flow from high ground, in this instance, probably the ridge to the SE of the deposit. The groundwater in the bedrock is mostly confined under the superficial cover on the valley floor, but artesian conditions manifest as flowing wells when exploration holes punch through the mantle of glacial sediments and weathered clay into the bedrock. A survey by Rupert Resources has so far revealed that 51 such holes are artesian.

GROUNDWATER RECHARGE

The spatial distribution of groundwater recharge, as is evident from the VEMALA model, is linked to superficial deposit properties, local ground elevation and slope. Recharge is typically greater on the ridges due to higher rock exposure and increased precipitation. Seasonal changes in recharge also vary due to the lock-up of precipitation as snow in winter and its rapid release during snow melt the following spring.

Baseflow separation analysis of stream hydrographs from the project catchment indicate recharge is approximately 13% of MAP (see Section 18.11.2). This is in line with other studies undertaken by SRK in the region that show it typically ranges between 12% and 16% of MAP.

GROUNDWATER CHEMISTRY

Samples of three types of water were collected for this study, that is from deep groundwater sourced predominantly from artesian wells, shallow groundwater from sediment wells and spring water. The pH of the deep groundwater (mean: pH 8) is generally higher than in the sediment wells and springs (mean: pH 7.2), with the latter influenced by rainfall, which is slightly more acidic. The deep groundwater wells also have higher Electrical Conductivity and sulphate, which may reflect greater water:rock and ore rock interaction. All three water sources have a similar magnesium-bicarbonate composition.

18.11.5 CONCEPTUAL HYDROGEOLOGICAL MODEL

INTRODUCTION

Groundwater and surface water are the primary pathways between potential sources and receptors, and therefore it is important to develop a CHM of the local regime, including the interaction between groundwater and surface water. This model considers the ‘ambient’ pre-mining condition and then assesses what changes are a likely consequence of the mining, both during operation and after closure.

SUMMARY OF HYDROSTRATIGRAPHIC UNITS AND AMBIENT PRE-MINING CONDITION

The hydrological review has revealed that there are several hydrogeologically-distinct formations at Ikkari, referred to as hydrostratigraphic units. The properties of these units are important in governing the way in which the local surface and groundwater regime behaves. They are broadly defined on lithological and structural grounds as the peat (Unit 1), the glacial sediment (Unit 2), the weathered and fractured bedrock (Unit 3) and the fresh bedrock (Unit 4).

From experience, the peat hydrostratigraphic unit tends to have physical properties and behaviour that mean it is at least partially isolated from the underlying glacial sediments. This is particularly so in thicker sequences of peat where the high levels of humification towards the base of the unit, also referred to as the catotelm, mean that it has a very low hydraulic conductivity. Year-round ponding of water in aapa mires and local groundwater perching above the peat are symptomatic of this condition.

The glacial sediments (Unit 2) that underlie the mires and outcrop at the surface beyond their fringes are spatially heterogeneous with till and moraine, although the till (commonly also referred to as diamicton) is the predominant material. As a unit these deposits will tend to drain quite poorly except where there are occasional stringers of moraine (esker) with coarse sediment that facilitate more rapid water movement.

The bedrock beneath the sediments comprises both weathered and fresh bedrock hydrostratigraphic units. The former (Unit 3) is present within the shear zone and along the Rajala Line. The top of the shear zone where it passes beneath the valley is heavily weathered and mantled in clay that serves to confine the groundwater in the bedrock, to an extent isolating the surface water environment from the underlying rock mass and the ore body. The groundwater regime is largely controlled by faulting and brecciation and the potential for fluid movement locally can be quite high, as evidenced by several of the artesian wells along the valley floor. This and the high (above ground level) piezometric pressures would suggest that there are several hydraulically well connected and conductive faults present in the rock mass that link the ore body to recharge zones on the higher ground to the SE with only limited attenuation.

The fresh, in-tact bedrock belonging to Unit 4 represents formations below the footwall, to the south, and above the hanging wall, to the north of the shear zone that hosts the Ikkari deposit. This rock, which in modelling terms is the dominant unit across the wider model domain, has historically been less impacted by tectonism and therefore has a much lower bulk hydraulic conductivity and more restricted groundwater movement.

Operational, Closure and Post-closure Conditions

Inflows to the mines are expected to be most elevated in the early stages of mine life during open pit development and in the initial excavation of the underground operations. Thereafter the rate should steadily decline as the storage of the surrounding rock mass is gradually depleted.

When mining the ore body, the pit, stopes, internal declines and headings may cross more open brittle faults. The permeability of some of these structures could be quite high with good hydraulic connectivity into the wider network of faults, meaning that flows could be both high and persistent. Dewatering of the mine will also create a halo of lower piezometric pressure around the excavated void with the more pronounced drawdown propagated along faults that intersect the workings.

The water management design adopted for the operation should be one that aims to prevent, as far as is possible, any substantial inflows to the mine, since it is imperative that drawdown produced by the operation does not project in any significant way to the surface. Some of the concepts for mitigating inflow are described in Section 18.13.

After closure and once the pumps have been switched-off, the water table in the mine is expected to rebound. The decommissioning process will likely entail the backfilling of remaining voids in the ore body and the installation of structures that limit decant at surface.

18.12 GROUNDWATER INFLOW ASSESSMENT

18.12.1 INTRODUCTION

Effective management of surface and groundwater in the context of a mine at the Ikkari site is essential to facilitate access to the workings, to control and, where possible prevent ingress of water to limit pumping and treatment costs and associated impacts on the surface environment caused by derogation of water features that fall within the orbit of the mine and the cone of depression created by the dewatering operation.

The objectives of the water management section of the PEA study are to understand the likely effects of the local surface and groundwater regime on the Ikkari operation, to predict inflows to the operation and use the results to help develop concept-level water management infrastructure suitable for controlling such effects.

This section briefly describes the development of a numerical groundwater model from construction and calibration through to predictive simulations with the mine infrastructure incorporated into the model mesh. The final part describes the infrastructure and practices best suited for managing water at the site using the model results and other knowledge about the local setting derived from the data review.

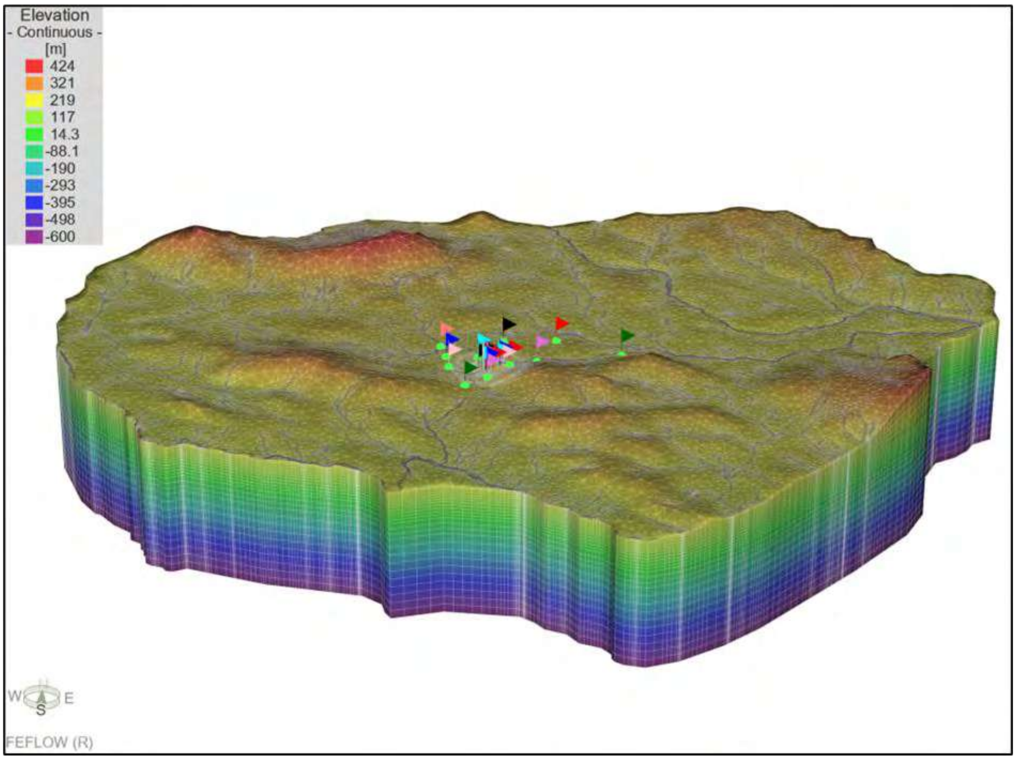
18.12.2 GROUNDWATER MODEL SELECTION, CONSTRUCTION AND CALIBRATION

SRK has conducted numerical groundwater flow modelling using the finite-element software package FEFLOW 7.5 (DHI-WHASY GmbH., 2020) to simulate water movement in both the horizontal and vertical planes in porous media assuming variably saturated conditions. Fluid flow was simulated as both steady state (long-term equilibrium) as well as transient (varying input conditions, such as seasonal recharge and mine plan development).

MODEL DESIGN

The groundwater model covers an area of 364 km^2 and extends from topographic surface to at least 50 m below the final underground mine depth (*c*.800 m bgl) using 24 layers. The discretisation of the model layers uses element dimensions of approximately 30 m near the proposed mine, 50 m along streams and up to a maximum dimension of about 200 m regionally (see Figure 18.2, with the final pit in the area surrounded flags).

Figure 18.2 Model Domain and Mesh in 3D



All model boundaries are a minimum distance of 5 km from the project site to prevent numerical boundary effects impacting on the representation of groundwater at the future mine. The bottom of the model at -600 m above sea level is assumed to be a 'no flow' boundary. Boundary conditions at the horizontal limits of the model are also 'no flow' as they either follow groundwater divides (such as ridges), or streams that are already represented by seepage nodes. The predominant flow in the model is conceptualised as being from groundwater to surface water (gaining streams).

Long term average groundwater recharge values were varied across the model according to soil type based on the VEMALA Model data, which is also comparable to equivalent soil distributions listed in the GTK database. The calibrated groundwater model annual average recharge as a per cent of MAP is c.14%. The transient models multiplied this long-term average recharge by a time-series of monthly factors. The monthly variation recharge factor was calculated from the daily VEMALA Model data.

Each of the hydrostratigraphic units (as described in Section 18.11.5) in the model was assigned a hydraulic conductivity, specific yield (Sy) and specific storage (Ss) using the results of the field studies described in Section 18.11.4. These were later adjusted during model calibration.

Open Pit and Underground Mine Representation

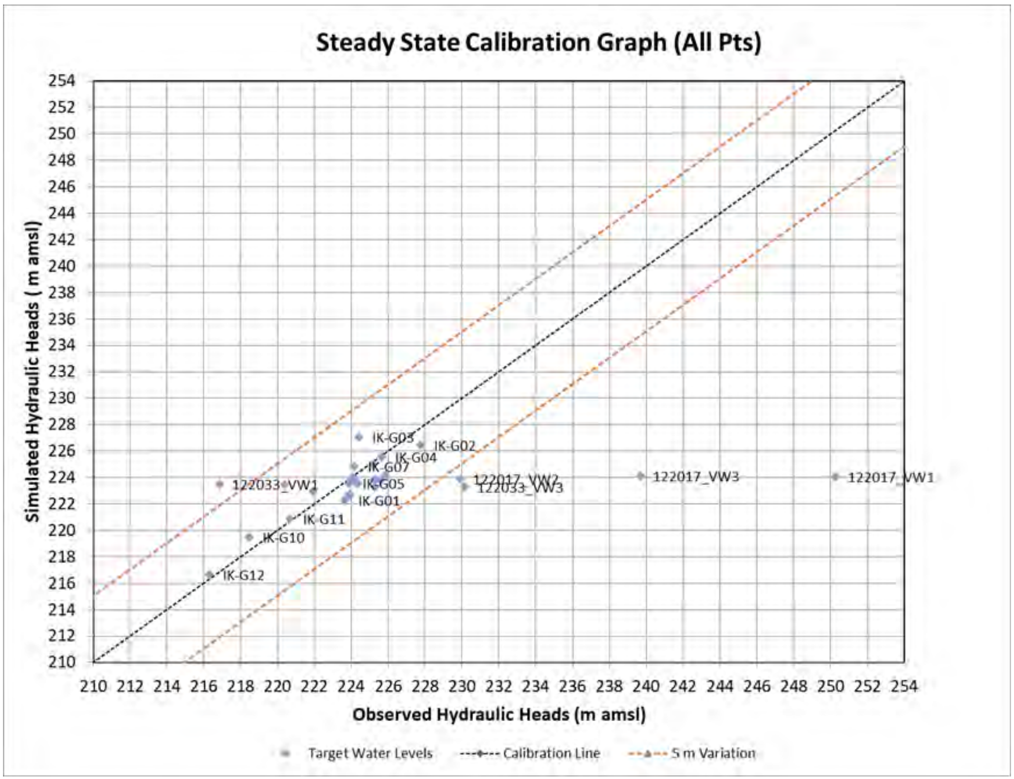
The life of mine design for a 3.5 Mt/a schedule has been incorporated into the transient model. The mine schedule of timings and the mine depth were each imported to separate FEFLOW user data nodal distributions. A Python script was developed to alter boundary conditions and the hydraulic conductivity of elements, for example assigned for grouting, at the start of each time step.

MODEL CALIBRATION

Model calibration involves an iterative process to estimate parameters describing hydrogeological properties and boundary conditions so that the model's results closely match historical observations. Model calibration focused on average water levels, seasonal variation in water levels and estimated baseflow to streams.

The imbalance (mass balance error) for the groundwater model is << 0.5%, which is within the standard acceptability criteria for numerical groundwater models. The model was also calibrated against regional groundwater contours and flow directions, as well as local spot point target locations of water levels measured in standpipes and VWP pressure heads at varying depths. The normalised root mean square error (RMS) produced by the calibration was 10% and the correlation coefficient 0.8. These are within the standard acceptability criteria for numerical groundwater models. The calibration of spot point targets (observed versus modelled) is shown in Figure 18.3.

Figure 18.3 Steady State Calibration Graph (all points)



Note Figure 18.3 includes the high head outliers in VWPs 122017 and 122033, which are first mentioned in Section 18.11.4. SRK undertook a model sensitivity study to replicate these very high heads in sensors located below the valley floor and established that this could only be done by introducing a high hydraulic conductivity structure between the area of the deposit and the ridge to the SE.

MODEL RESULTS

Various model scenarios with different combinations of engineered mitigation have been considered for this study with the object of minimising inflows and limiting the need for water clean-up. Of these, the preferred cost-benefit solution based on the model output, both in terms of the operation and the hydrological impact study, is to capture water before it enters the workings, thereby lowering the volume of water requiring treatment. This necessitates the inclusion of interception ditches to prevent surface water ingress and dewatering wells around the pit, targeted at higher flowing geological structures. These wells begin pumping during the latter stages of the ramp-up to pit development and continue for the life of operation.

Note that whilst the study has benefited from some initial site characterisation of groundwater, due to the structural complexity of this setting there remains some uncertainty about how the groundwater regime behaves. Hence, this is reflected in the uncertainty ranges that are used in the following paragraph to define flow rates. These ranges have been constrained by the results from the various hydrogeological tests (mainly packer and pumping tests) undertaken at Ikkari.

The inflow schedule produced by the model for the open pit phase of the operation (Years 1 to 9) shows that total average surface and groundwater flow ranges between 193 and 322 litres per second (l/s) with a median value of 257 l/s. Of this, some 7% reports to the pit with the remainder captured by the ex-pit well dewatering and by surface drain interception. From Year 9 onwards when the operation switches to underground working, the overall average flow to the mine remains about the same, but the proportion attributable to ex-pit dewatering declines to around 74% with the remainder seeping into the pit and the underground workings. This change is partly because the store of water in the surrounding rock mass is steadily used up causing the well production rate to naturally decline, but also because the wells are primarily targeted at limiting inflows to the open pit. Their efficiency and cost-effectiveness tend to reduce with increasing depth, in line with lower hydraulic conductivity and storage, so it is frequently more advantageous to switch from this technology to targeted grouting in the underground workings.

18.13 WATER MANAGEMENT CONCEPTS

This section describes the various approaches that are recommended for managing water in respect of the open pit and underground mines with the objective of minimising impact both on the operating capacity of the mine and on the local environment using cost-effective and pragmatic solutions.

18.13.1 OPEN PIT

Part of the water management requirements of the mine will be to ensure that the pit is kept dry enough so that normal operations are interrupted as little as possible and that the amount of contact water requiring treatment is minimised:

- The open pit will straddle a valley that contains a stream and extensive wetlands for which some form of lined diversion structure will be required to prevent large volumes of run-off finding its way into the pit.

- The periphery of the excavation will also need the construction of berms and ditches to control run-off from the upper reaches of the catchment. The ditches and associated settlement ponds will need to be lined to prevent surcharging of the rock mass behind the pit wall, increasing the risk of slope failure.
- There will need to be an appropriately sized in-pit dewatering system capable of collecting groundwater seepages and rainfall run-off and removing it from the pit; this is accomplished through gravity drainage to in pit sumps where the water is raised using pumps and riser pipes to settling ponds positioned beyond the pit perimeter.
- Consideration should be given to targeting the larger, more conductive faults with peripheral dewatering wells. This should have the added benefit of reducing the volume of 'contact' water requiring potentially expensive treatment.

18.13.2 Underground Mine and Decline Ramps

Mine Design and Grouting

The currently preferred mining method (caving) presents additional risks so far as groundwater ingress to the workings is concerned, so careful consideration needs to be given to supporting the ground to limit the development of tensional structures and ground settlement above the mine workings. However, where geological structures and more conductive lithologies cannot be avoided, then containment and exclusion of groundwater will be achieved through grouting.

For the purposes of planning all major faults should be targets for pre-grouting. In instances where high flows are considered a risk, then bitumen can be used as an alternative to grout. Where ground conditions are poor and permeability high, this might warrant a minimum stand-off distance of some 50 m between the workings and the structure concerned (Health & Safety Executive, 1993).

Pre-grouting should be implemented where probe holes drilled ahead of the excavation intercept high flow zones, most likely associated with geological structures. Cover grouting should be implemented using a split-spacing approach, only resorting to secondary and tertiary grout holes if the primary hole is insufficient to hold groundwater flows in check. Injection pressures should be at least 50% above what is anticipated in the formation around the excavation, as determined from monitoring wells installed along the path of the decline and around the mine void.

Underground Pumping Arrangements

The underground mine dewatering system will likely comprise a pumping station at or near the base of the mine (-800 m), and a second intermediate level station at -500 m. The intermediate level station will (a) prevent mixing of better-quality water from the upper levels with more mineralised water at depth and hence, limit treatment costs and (b) reduce operational costs by reducing the volume of water raised from the bottom level sump. Both stations will be equipped with duty and standby pumps together with a solids settlement system. These sumps will have a volumetric capacity for at least 24 hours of average inflow without pumps.

Water underground will be conveyed in pipes via the decline ramp to the surface and report to the mine dewatering pond.

EXPLORATION BOREHOLES

Some 70 km of exploration holes have been drilled since the discovery of the Ikkari deposit, few of which we understand have been back-filled with grout. Some of the holes are likely to intercept the future mine workings and therefore present a significant risk to the future mining operation and to the wetlands on the surface. It is important that these holes are identified and, where possible sealed with grout in advance of mining. In addition, it is recommended that a stope inrush methodology is put in place during mining to facilitate the identification and management of this risk.

18.14 SITE WIDE WATER BALANCE

18.14.1 OVERVIEW

An Ikkari site-wide water balance model was developed to support the Study. The model was used to assess process water requirements throughout the mine life, as well as to simulate the major mine facility water supply and demand. The results indicate that the co-disposal facility (CDF) will operate in a deficit during all phases of operations and under the full range of variable climatic conditions, including prolonged wet and dry cycles. Water losses will be mainly due to the physical entrainment of water within the tailing solids in the CDF. Smaller amounts will be lost to evaporation. Make-up water to supplement process water requirements during operations will be sourced from groundwater wells.

18.14.2 WATER SUPPLY AND DISTRIBUTION WATER

RAW WATER

Currently, there is limited hydrology and hydrogeology data; however, it is anticipated that raw water will be drawn from boreholes around the pit and pumped to the local river system. These boreholes can also be used to lower the water table around the pit to reduce water inflow.

Raw water will be stored in a covered raw water tank adjacent to the process plant, which will be sized in accordance with process plant requirements and water sources, to ensure security of supply.

POTABLE WATER

In the absence of water quality data, Tetra Tech proposed using a 20 cubic metres per hour (m^3/h) reverse osmosis water treatment system with 100% standby capacity to provide all site potable water requirements reliably. The system will need to be designed to take into account the sub-zero temperatures.

PROCESS WATER

During steady state operation, a large proportion of the process water requirements will be met by the recycled water from the filtration of the tailings. Process water will be collected in a raw water tank adjacent to the process plant and then pumped to the various process use areas.

Sewage Treatment

A sewage treatment plant will be located adjacent to the plant area. The facilities will consist of a packaged wastewater treatment unit and will remain on site to provide long-term sewage treatment requirements during operations. The process is defined as extended aeration activated sludge biological treatment. The sludge removed will be disposed of in the tailings management facility (TMF). Consideration must be given to the extreme climatic conditions during treatment selection.

Wastewater

Any water that is discharged to the environment will be treated to meet the minimum local water quality requirements.

18.14.3 Surface Water management

All mine contact water, which includes runoff from the plant site, CDF, waste and ore rock storage facilities, and mine dewatering flows will be collected, stored and managed within the project area. Seepage collected in collection ponds located around the CDF will be recovered for reuse in processing. Untreated contact water will not be discharged from site.

Diversion ditches will be installed around the plant site, waste storage facilities, open pit, and the CDF to convey clean or non-contact freshwater around these disturbed areas, where it is physically practical. Water that accumulates at project infrastructure will be collected and pumped to the water collection ponds for reuse in processing. No water will be discharged to the environment that would have an adverse environmental impact.

18.15 Tailings Management & Storage

18.15.1 Ikkari Co-disposal

Overview

The Ikkari Project (Ikkari) is situated in an area of relatively flat topography. The ground elevation of ranges from +220 m (above sea level) in low-lying area north of the proposed mine site to +295 m at the south of the permit boundary. The proposed open pit is located south of Saittajoki River in low-lying area and the proposed plant site is located on higher terrain to the east of the open pit. (Figure 18.4)

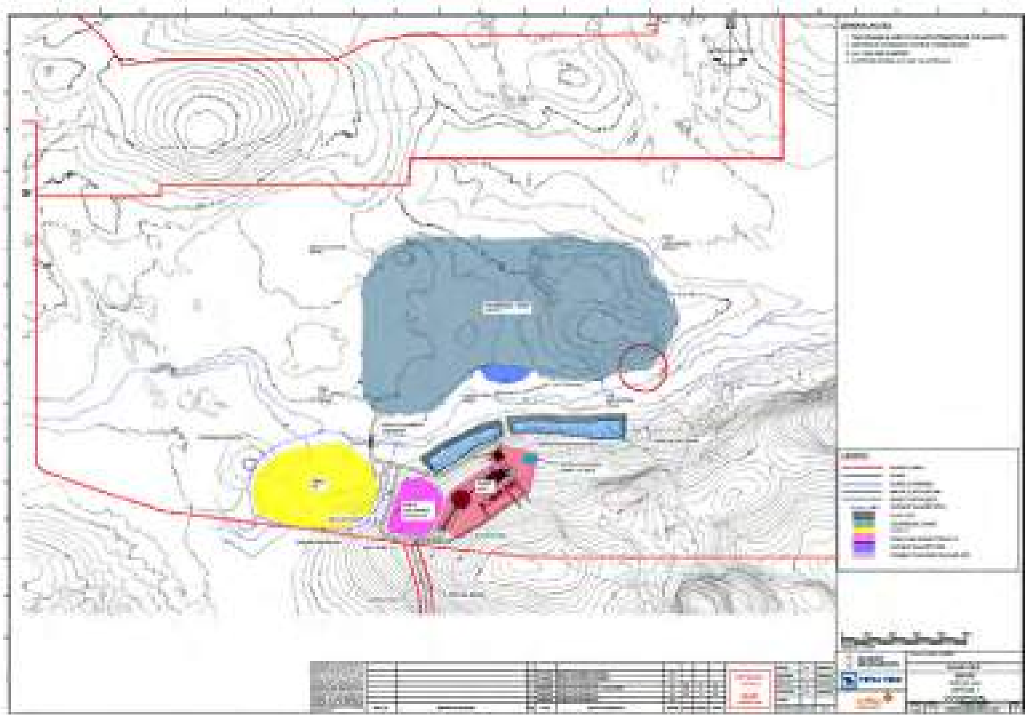
One location is proposed for the CDF of mine waste rock and filtered tailings, named North Co-disposal Facility. Tailings and mine waste rock are handled from the processing plant and open pit separately. The filtered tailings will be encapsulated inside the waste rock dump material.

Site Selection

The CDF site is proposed near the open pit and the processing plant in consideration of the transportation distance for the mine waste and filtered tailings. The footprints are constrained by the following considerations (Figure 18.4):

- Minimise marshland disturbance to preserve the local habitat and reduce operational challenges, i.e., water management and dewatering complexity.
- Maintain minimum 100 m offset from Saittajoki River in consideration of flood events.
- Maintain 50 m offset from the Rupert Resources land boundary in consideration of water management ditch or stream diversion channel.
- North Facility will maintain minimum of 150 m offset from Heinä Central mineralisation zone and future drilling programme which could generate approximately 200 koz of gold with an undefined copper component.

Figure 18.4 Mine Waste Layout of Ikkari



The conceptual waste rock dump and filtered tailings CDF was developed with consideration of available information and the project development assumptions are summarised in Table 18.8 and Table 18.9.

Table 18.8 Waste Balance Summary

Facility	Units	Total
Open pit ore	Million t	36.6
Underground ore	Million t	33.4
Open pit waste	Million t	130.0
Underground waste	Million t	4.5
Filtered tailings dry density	t/m^3	1.7
Loose rock swell factor	%	130
Waste rock generation	Loose Million m^3	61.5
Filtered tailings generation	Million m^3	41.5
Underground backfill	Million m^3	Not Available*
Co-disposal required	Million m^3	103.0

***Note:** The underground operation will be using sub-level caving and backfill is not available.

Table 18.9 Co-disposal Dumps Design Summary

Facility	Unit	North Co-disposal Facility
Maximum elevation	m	286
Design capacity	Million m^3	52
Co-disposal overall side slope	H:V	3:1
In-situ rock density	t/m^3	2.9
Placed rock density	t/m^3	2.2
Structure footprint	Hectare	119
Marshland disturbance	Hectare	59

Water Management

During the tailings filtering process, water will be recovered and recycled in the ore processing facility. Collection ponds for the waste structures are required to collect surface runoff from the tailings and local catchment. For the co-disposal structure, two water management systems will be required:

- Natural surface runoff water and groundwater from the surrounding catchment area that has not contacted the tailings is required to be intercepted and diverted. Thus, diversion ditches are required to divert surface runoff around the co-disposal areas.
- An interceptor and collection system for contact surface water, impacted groundwater, and seepage from the co-disposed tailings and waste rock. This system usually consists of perimeter ditch and collection sumps. Water collected in the ponds and sumps is used in the process plant.

Reclamation & Closure

The co-disposal placement schedule offers full control of the construction sequence and desired geometry of the structure which allows progressive reclamation, and it is amendable to dry closure landform development. A closure cap will be placed over the exposed filtered tailings surface with a layer of rock cover, moraine/till, topsoil, and vegetation. Furthermore, progressive reclamation and closure cap will further reduce erosion caused by surface water runoff and ARD/ML challenges.

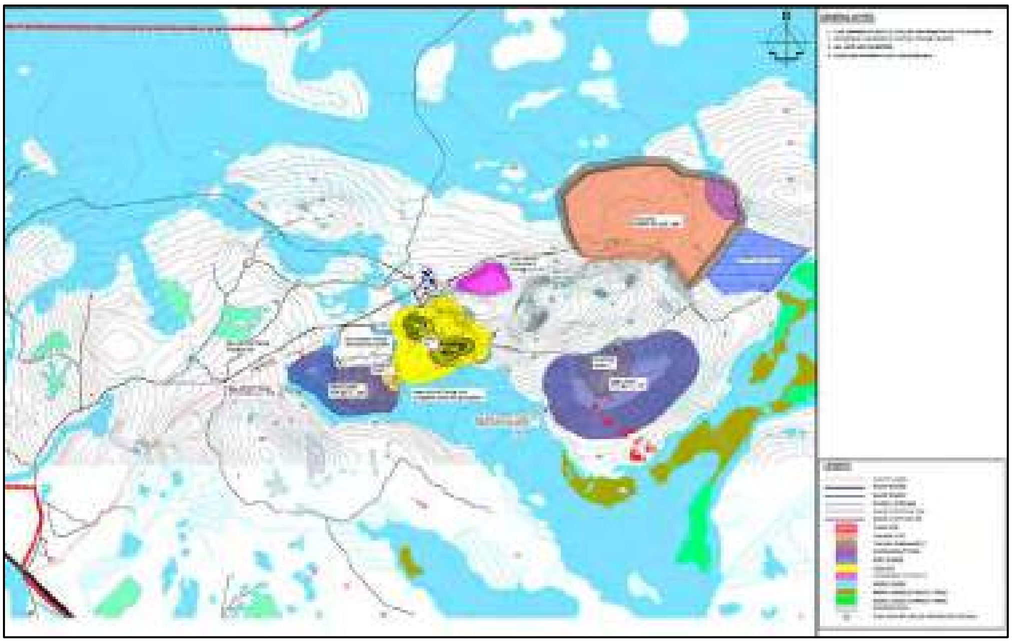
18.15.2 PAHTAVAARA

OVERVIEW

The Pahtavaara project's topography is relatively flat, ranging from 250 m to 300 m. Marshland can be found at the south of the existing open pit and waste rock dumps. Pahtavaara has operated in three periods of prior ownership between 1996 and 2014. Two existing mine waste rock dumps and associated haul road networks were established west and east of the open pit respectively. West dump is approximately 700 m from the centroid of the existing pit. East dump is approximately 1,500 m east of the pit.

The existing slurry tailings storage facility is located approximately 1,800 m northeast of the open pit. The current tailings facility is constrained by three embankments. Western, northern and central embankments contain slurry deposited tailings. A supernatant pond is formed against the central embankment and the pond level is managed by decanting the excess water into the downstream settling pond contained by the eastern embankment.(Figure 18.5).

Figure 18.5 Mine Waste Management Layout of Pahtavaara



CDF DESIGN

Due to the existing processing plant, it is recommended to continue the slurry tailings operation at the existing tailings footprint. The life of mine plan indicates 8.0 Mt of ore production from both open pit and underground; therefore, additional dam lifts will be constructed to contain approximately 5.7 Mm^3 of slurry tailings over the mine life. Table 18.10 shows slurry tailings generation and volumetric assumptions.

The west embankment and central embankment will be raised by the downstream construction method, using quarry rock sourced locally. The embankment will be placed and compacted on a woven geotextile to reinforce the foundation during construction. A layer of non-woven geotextile will be installed on the upstream side of the embankment to facilitate the filtration of slurry tailings. Furthermore, a liner and graded filter will also be installed within the embankment to further relieve hydraulic gradients and prevent seepage erosions within the structure.

Slurry tailings will be discharged at the existing discharge location, and the final elevation is estimated at elevation 249 m. As such, tailings water is expected to be collected against the embankment and continuous drainage effort may be required to direct the water toward the central embankment and decant system, and a supernatant pond is collected at the northeast quadrant of the CDF.

Tetra Tech conducted volumetric analysis for the new tailings surface and embankment as shown in Table 18.510.

Table 18.10 CDF Embankment and Slurry Beach Design Criteria

Criteria	Unit	Total
Tailings generation	(Million m^3)	5.7
Tailings quantity modeled	(Million m^3)	6.2
Existing discharge elevation	masl	240
New discharge elevation	masl	249
Existing embankment crest elevation	masl	238
New embankment crest elevation	masl	246
New embankment fill	(Million m^3)	1.4

18.16 Mining Contractor Facilities

The mining contractor will establish their own administration and workshop facilities.

18.16.1 Mine Maintenance Shop And Warehouse

The mine maintenance/warehouse complex will provide service facilities for the mining operation, along with storage of spare parts and consumables, including:

- Heavy duty repair bays.
- Weld bay.
- Light vehicle repair bays.
- Maintenance workshops.
- Machine wash/tire change bay.
- Warehouse.
- Offices.

18.16.2 Explosives Storage Area

Mine explosives will be stored in a purpose built secure complex and will conform to local regulations.

The entire complex will be surrounded by a fence that is two metres high and topped with barbed wire. To comply with European regulations the site will require constant security and will be accessed via a road. There will be guard post near the entrance of the facility.

The facility will be lit by two projector lights atop towers within the boundaries of the explosives storage area.

18.17 Offsite Infrastructure

18.17.1 Access Road

The A75 Rovaniemi to Sodankylä is 130 km by road and takes just 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 Pulkittama road for 7.5 km where forest tracks lead directly to the exploration site.

Access to the site is possible throughout the year.

18.17.2 High Voltage Power Supply

The site is located 5 km from 220 kV power lines and 9 km from an existing transformer station. A high-voltage (HV) transmission line will be constructed from the transformer station to the plant main substation for distribution to the plant facilities.

Power to the concentrator plant and other facility loads will be distributed on 11 kV rated power cables run along four 11 kV overhead power lines.

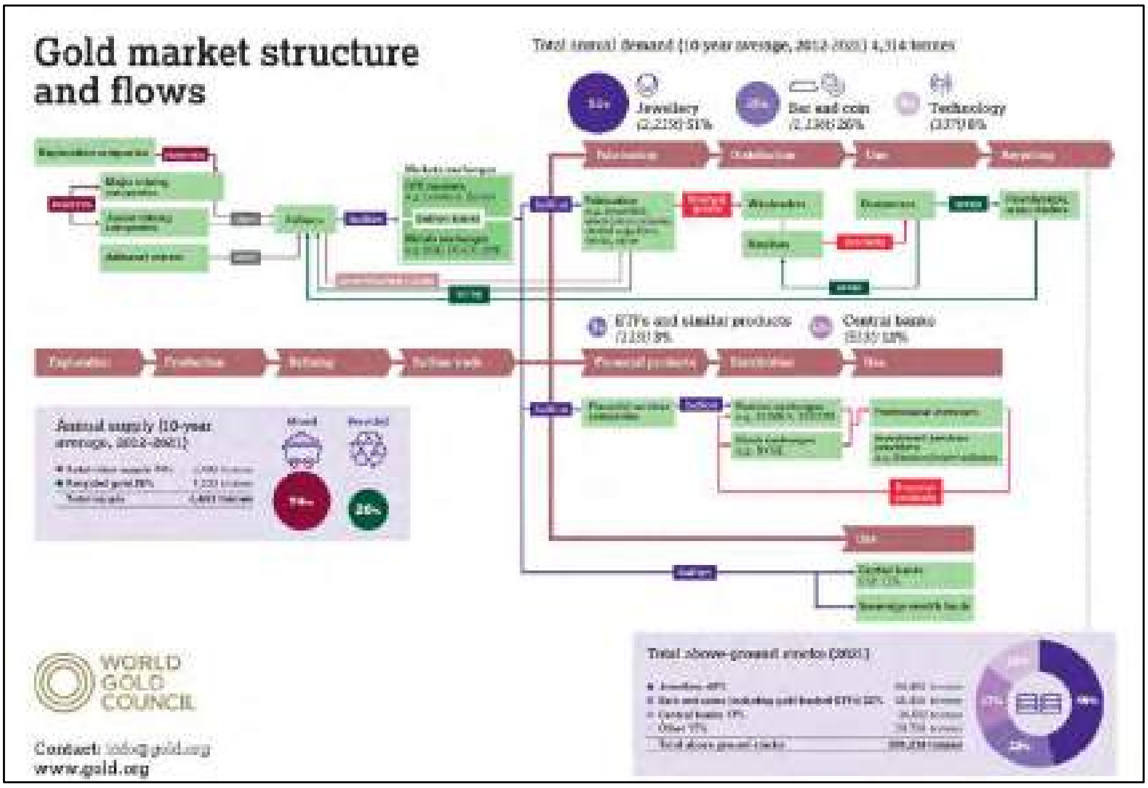
Two 40 megavolt ampere (MVAr) static VAR compensators will be installed in the plant main substation to mitigate voltage surges generated due to switching, lightning strikes and power system faults.

19.0 MARKET STUDIES AND CONTRACTS

19.1 PRICING

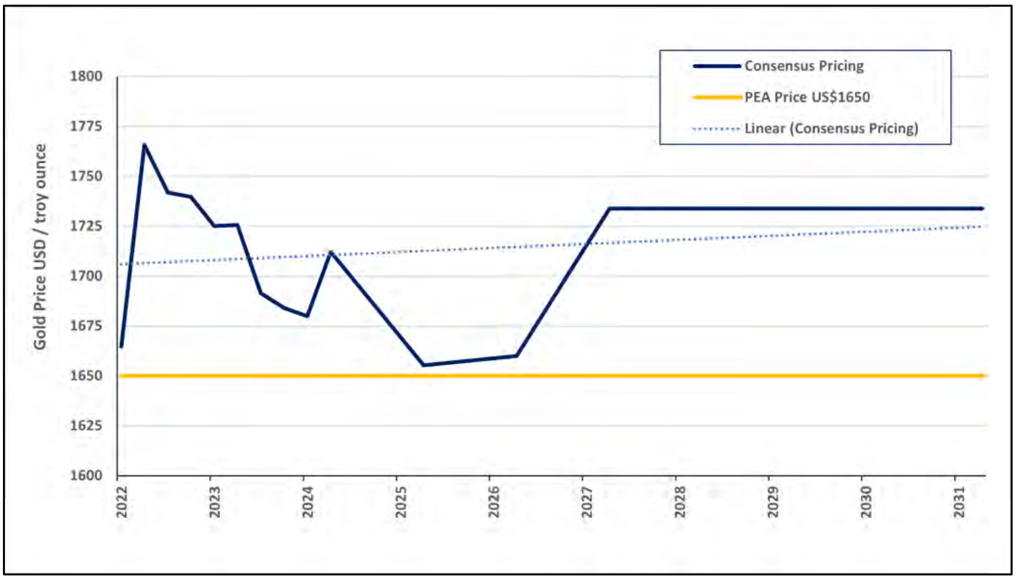
Gold is a precious metal bought by people across the world for different reasons, often influenced by socio-cultural factors, market conditions, and macro-economic drivers in their country (World Gold Council, see Figure 19.1). Daily pricing is available from Over The Counter Markets (OTC) and at Metals Exchanges.

Figure 19.1 Gold Market Structure and Flows



The gold price assumption of US $1,650 /troy ounce of gold used throughout the study was derived from mean consensus long term pricing assumptions from a population of 40 energy and metals analysts, see Figure 19.2.

Figure 19.2 Gold Pricing Forecast



Source: Consensus Economics

19.2 PAYABILITY, TREATMENT AND REFINING ASSUMPTIONS

The study assumes payability of 99.92% and a freight and refining cost of $2.50 /oz on doré product based on industry benchmarking. Gold doré produced at Ikkari is expected have no deleterious elements and to be able to be refined in Europe.

19.3 EXISTING CONTRACTS OR AGREEMENTS

The Project has no contractual or offtake sales agreements in place.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL STUDIES DONE AND RELEVANT ENVIRONMENTAL ISSUES

There are no designated protected areas within the Rupert Lapland Project area or within the Pahtavaara or Ikkari deposit impact zones. Kaaresvuoma nature conservation area (ESA302828, SSO120578) is the closest Nature conservation area to Ikkari and is situated more than 8 km east of the Ikkari deposit. Tollovuoma-Silmäsvuoma-Nunarvuoma (SAC/SPA, FI1300608, 9 673 ha) is the second closest one and located more than 10 km west of the Ikkari deposit. Ilmakkiaapa mire reservation area is the closest one to Pahtavaara site and it is located 12 km north-east from the Pahtavaara pit area.

The Ikkari project area contains forests, mires, swamps and springs, small streams, and headwaters of small rivers. Environmental baseline data collection at Ikkari began 2019 with water sampling of main streams and rivers. Currently 18 surface water locations are monitored in Ikkari and 10 surface water locations in Pahtavaara. Wide range of water analyses (total of 63 parameters) is analysed 6 times per year from all surface water samples. Groundwater monitoring has started in March 2022 with shallow sediment well installations and is done six times per year from 11 different wells at Ikkari. At the Pahtavaara mine are, six shallow sediment wells were installed in March 2022 and are monitored six times per year. Total of 65 different parameters are analysed from all groundwater samples. Also, continuous environmental monitoring stations have been installed to measure river flows, water cloudiness and pore water pressure in Ikkari area.

In addition, the following nature and environmental studies have been conducted and reported:

- Desktop review of nature values and habitats of exploration area (Eurofins, 20 February 2018).
- Pahtavaara mine waste management plan and mine waste characterisation (Eurofins, 3 July 2019).
- Moor frog –study of Ikkari & Heinälamminvuoma 2019 (Eurofins, 9th October 2019).
- Breeding bird line transect censuses in Heinälamminvuoma (Ikkari), 2019 (Eurofins, 18 September 2019).
- Pahtavaara tailings pond vegetation cover monitoring 2020 (Ramboll, 2020).

- Nature survey of exploration areas, 2019 (Ramboll, 30 January 2020).
- Desktop study of freshwater pearl mussels (Eurofins, 28 March 2021).
- Pahtavaara tailings pond vegetation cover monitoring 2021 (Ramboll, 2021).
- Status of Ikkari water systems (Envineer, 12 November 2021).
- Phase 1 Review of data to support the hydrogeological study of the Ikkari gold and satellite deposits, Northern Finland (SRK Consulting, December 2021).
- Ikkari area nature survey 2021; birds, moor frogs, bats, directive inspects, otter (Envineer, 14 January 2022).
- Preliminary review of route alternatives for Ikkari discharge water (Envineer, 10 May 2022).
- Climate change model for Ikkari gold mine (Envineer, 31 May 2022).
- Phase 2 Hydrogeological field study report for the Ikkari Au and satellite deposits, Northern Finland (SRK Consulting, 13 June 2022).
- Pahtavaara tailings pond vegetation cover monitoring 2022 (Ramboll, 2 November 2022).
- Pahtavaara waste rock area vegetation survey 2022 (Ramboll, 2 November 2022).
- Ikkari geochemistry data review, waste characterisation and gap analysis (Mine Environment Management Ltd, 2 November 2022).
- Pahtavaara geochemistry data review, waste characterisation and gap analysis (Mine Environment Management Ltd, 2 November 2022).

Carried out environmental studies that have yet to be reported:

- Ikkari area nature survey 2022; mammalian snow tracks, birds, moor frogs, bats, directive inspects, otter (Envineer).
- Pahtavaara area nature survey 2022; mammalian snow tracks, birds, moor frogs, bats, directive inspects, otter (Envineer).
- Ikkari area flora & fauna survey 2022 (Envineer).
- Pahtavaara area flora & fauna survey 2022 (Envineer).
- Ikkari area Endangered moss specie survey 2022 (Envineer).
- Ikkari area freshwater pearl mussel survey 2022 (Eurofins).
- Aquatic biological surveys in Ikkari and Pahtavaara; fish, benthos, diatoms, aquatic and coastal habitats and aquatic vegetation 2022 (Envineer).
- Archaeology survey for Ikkari and Pahtavaara area 2022 (Mikroliitti).
- Ikkari peat layer and peat quality studies 2022 (Geolite & Afry).
- Ikkari mine waste sampling and static testing results 2022 (MEM).
- Pahtavaara mine waste sampling and static testing results 2022 (MEM).

Nature baseline studies have also been undertaken at the Pahtavaara mine area by Ahma Ympäristö Ltd in 2005, 2006, 2013. There are no protected areas inside the mining area or in the area affected by the mine. Further baseline studies were completed by Eurofins Ltd for the wider Pahtavaara exploration permits in March 2018, covering an area that includes Area 1 and the Ikkari deposit.

20.2 WASTE MANAGEMENT

Ikkari and Pahtavaara mines waste characterisation programme has started in July 2022 with Mine Environment Management Ltd (MEM) and will be ongoing until the end of 2023. Ongoing programmes are based on a staged approach, where available information is reviewed to develop and understanding of the deposit before targeted selection of samples for detailed geochemical testing is carried out, and the results of this testing are used to develop a waste characterisation framework for Ikkari and Pahtavaara. The phases of the current programme can be summarised as:

- Project commencement: Data review and sample selection.
- Phase 1: Static testing, interpretation and reporting.
- Phase 2: Kinetic testing.

At this point in time data review and sample selection are completed and phase 1 with static testing programme is ongoing for both Ikkari and Pahtavaara samples (Mine Environment Management Ltd, 2.11.2022).

The work undertaken to date by MEM indicates Ikkari and Pahtavaara both present potential acid mine drainage (AMD) risks with respect to long term storage of waste materials due to the presence of sulphide sulphur and elevated metal contents consistent with the style of mineralisation in the regional area, and comparison with published information on AMD risks at other operating sites in the area (e.g. Kevitsa and Kittilä) (Mine Environment Management Ltd, 2.11.2022).

A statistical review of the elemental geochemistry of the Ikkari and Pahtavaara deposits has been undertaken which reveals very varied character across the range of recognised lithologies and derivative simplified Geogroups. Variation also occurs within each sub-group at a lithology scale thus with the exception of the black schist, all waste rocks need basic AMD characterisation to determine AMD risk levels and develop a fit for purpose classification system (Mine Environment Management Ltd, 2.11.2022).

The purpose is to collect the leachate from mining waste areas, treat it accordingly and recycle where possible, and direct it along the discharge pipeline to the receiving water body. Samples are collected from all mine waste fractions and leachates separately during the operation phase. The mine waste areas are located in the same catchment area as the Ikkari pit area, which facilitates water management after the closure phase. Closure planning is started in the environmental impact assessment (EIA) phase and becomes more specific as the investigations progress.

20.2.1 IKKARI

The geology of the Ikkari deposit is complex with many different lithotypes noted. In addition, there is polyphase structuration and several episodes of mineralisation. Nonetheless, from an AMD perspective the widespread presence of sulphide mineralisation and elevated metals indicates waste rock AMD is a tangible risk at the site that will require addressing. The MSB black shale lithology appears to carry predictable elevated sulphide which is seen as an indicator of AMD risk that can potentially be managed through logical classification schemes. Sulphides related to later economic mineralisation are more widely distributed and therefore will require inclusion in more detailed waste planning which will consider the overall net acid generating potential and metal leaching characteristics. In general metal concentrations of number of species (e.g. Ni, Co) are elevated with respect to crustal abundance indices and published assessment criteria however the relevance of absolute metal concentrations to metal leaching risks is a major uncertainty at this point (Mine Environment Management Ltd, 2.11.2022).

In this deposit the presence of significant carbonates is indicated by data assessed, which may mean that acid generating risks may not be high even though sulphide content is elevated. However, it is noted that the nature, type and distribution of carbonates is uncertain based on assessment of existing data. Certain metal species are noted to be mobile even in circumneutral conditions (Ni, Co, Mn) and as such the presence of excess carbonates (from an acid base accounting perspective) does not preclude potential for metal leaching risk (Mine Environment Management Ltd, 2.11.2022).

The gap analysis of the Rupert Resources database indicates a large amount of useful assay data has been collected from exploration drilling which can be repurposed for AMD assessment (Mine Environment Management Ltd, 2.11.2022). AMD targeted laboratory studies has been started in order to better characterise the geochemical properties of the waste rocks at the site and determine their long-term acid and metalliferous drainage potential.

The review of Carbon Capture and Storage potential indicates Ikkari in particular has excellent potential due to the presence of large volumes of ultramafic rocks. Further studies will be undertaken to determine whether a CO^2 capture project can potentially add value to the project by offsetting waste management costs.

20.2.2 PAHTAVAARA

The geology of the Pahtavaara deposit which has already been partially mined using existing open pit mining is simpler than the complex structuration of Ikkari with a greater emphasis on Amphibolite alteration and less evidence of sedimentary protoliths. However, the abundance of altered mafic and ultramafic rocks high in Mg poses the risk of fibrous asbestiform mineralisation. The sulphide mineralisation at Pahtavaara is widespread and heterogeneous with locally high levels of sulphide (pyrite + pyrrhotite). Based on assessment of sulphide distributions further detailed geochemical testing is required to determine the waste rock sulphide mineralogy, the deportment of sulphide (disseminated, vein or nugget), the elemental geochemistry and longer term leaching characteristics with respect to long term storage in waste storage facilities. In general, the sulphide content in Pahtavaara is lower than Ikkari, however the presence of elevated metals (that are mobile under circumneutral conditions) indicates that further assessment of the linkage between sulphide content and metal content and metal leaching risk is required to better determine AMD risk (Mine Environment Management Ltd, 2.11.2022).

In this deposit, the presence of significant carbonates is indicated by data assessed, which may mean that acid generating risks may not be high even though sulphide content is elevated. However, it is noted that the nature, type and distribution of carbonates is uncertain based on assessment of existing data. Certain metal species are noted to be mobile even in circumneutral conditions (Ni, Co, Mn) and as such the presence of excess carbonates (from an acid base accounting perspective) does not preclude potential for metal leaching risk (Mine Environment Management Ltd, 2.11.2022).

The gap analysis of the Rupert Resources database indicates a large amount of useful assay data has been collected from exploration drilling which can be repurposed for AMD assessment (Mine Environment Management Ltd, 2.11.2022). AMD targeted laboratory studies have been started in order to better characterise the geochemical properties of the waste rocks at the site and determine their long-term acid and metalliferous drainage potential.

20.2.3 EXISTING TAILINGS AREA

In future production, the tailings area (68 ha) may need a new operation plan, possibly new sectioning, piping, spigots and a dam raise or new dams. As built drawings for the current dams have not been reviewed and monitoring data for the existing dams has not been evaluated. The current environmental permit allows dams to be raised up to +248 m (N60). The current dam level is 232 m.

The surface of the Pahtavaara tailings basin was seeded in the summer of 2019. The seeding was part of an ongoing study to find out whether the Pahtavaara tailings pond can be closed by planting the surface of the tailings directly without a moraine cover. The tailings vegetation is studied every year to get information on how the species and their coverage develop in the basin. The studies will continue until the end of 2024, after which the company will submit the final results of the test to the authorities and decide the final closure landform. The coverage of the tailings pond vegetation has developed from 15% coverage in 2020 up to 99% coverage noticed in 2022. The vegetation has effectively prevented dusting from the area and decreased solids emissions to pond discharge water by 30 %.

20.2.4 EXISTING WASTE ROCK AREAS

There exist three waste rock areas in Pahtavaara from past production history. Two of them have not been yet closed according to the valid Pahtavaara environmental permit.

20.3 POST-CLOSURE MANAGEMENT

Closure planning of Ikkari mine is started in the EIA phase and becomes more specific as the investigations progress. The Pahtavaara mine closure plan has been updated and submitted to the permitting authority in 2020. The Permit is expected to be granted at the beginning of 2023.

20.4 SITE MONITORING

Environmental baseline data collection at Ikkari began in 2017 with water sampling of main streams and rivers. Currently 14 surface water locations are monitored from Ikkari and 18 surface water locations from Pahtavaara. Also during 2022 a total of 11 shallow groundwater monitoring wells were installed to Ikkari and 6 to Pahtavaara. Surface water and groundwater sampling is done six times per year. Four continuous environmental monitoring stations are installed to Ikkari to measure river flow.

Samples are collected and are sent for analysis by Eurofins Ltd. The results of the analyses are delivered regularly to the supervising authority (the Centre for Economic Development, Transport and Environment of Lapland).

20.5 PERMIT REQUIREMENTS, STATUS OF PERMIT APPLICATIONS AND BOND REQUIREMENTS

20.6 APPLICABLE CODES

20.6.1 MINING CODE

Mining and exploration projects in Finland are subject to the Finland Mining Act (621/2011). 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 use of natural resources.*

A further objective of the Act is to ensure the municipalities' opportunities to influence decision-making, and the opportunities of individuals to influence decision-making involving them and their living environment. 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.

20.6.2 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, inter alia, the provisions of the Nature Conservation Act (1096/1996), the Environmental Protection Act (527/2014), the Act on the Protection of Wilderness Reserves (62/1991), the Land Use and Building Act (132/1999), the Water Act (587/2011), the Reindeer Husbandry Act (848/1990), the Radiation Act (592/1991), the Nuclear Energy Act (990/1987), the Antiquities Act (295/1963), the Off-Road Traffic Act (1710/1995) and the Dam Safety Act (494/2009)"

20.6.3 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:

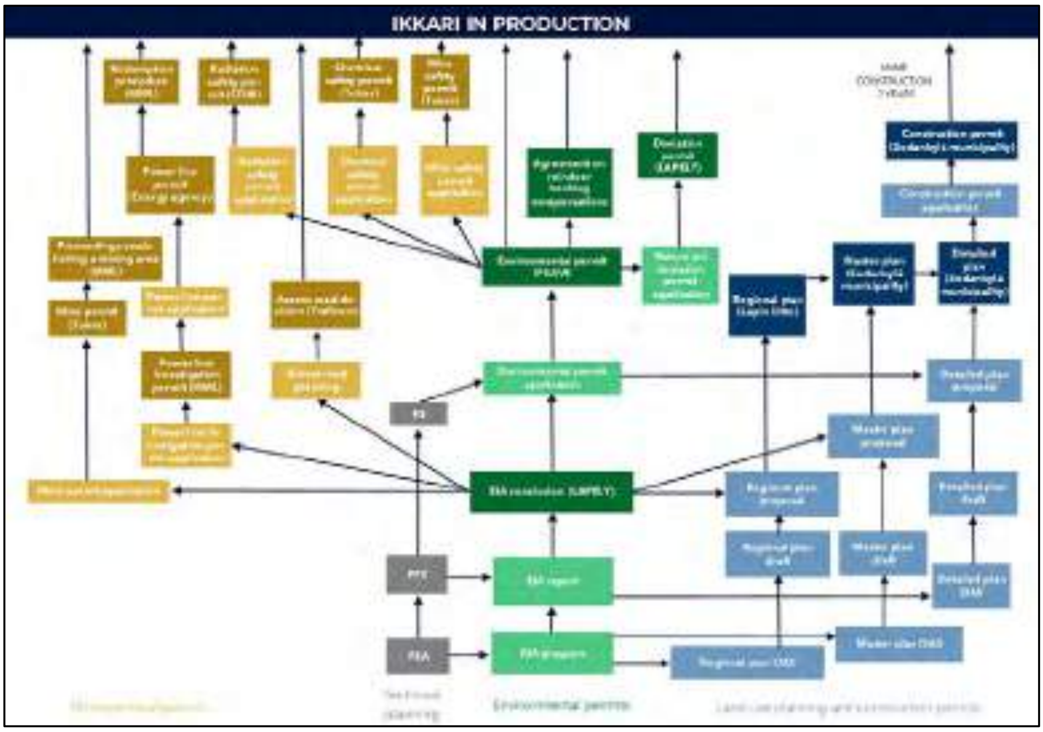
1) *the times and methods of exploration surveys and the equipment and constructions related to exploration;*
2) *measures to diminish harm caused to reindeer herding in a special reindeer herding area;*
3) *wording to ensure that activity under the permit will not endanger the status of the Sami as an indigenous people in the Sami Homeland, or the rights of the Skolts in accordance with the Skolt Act in the Skolt area;*
4) *obligation to report about exploration activities and results;*
5) *post-mining 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) *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 16; AND*
11) *other specifications that are necessary in view of public and private interests and pertaining to the implementation of the conditions of the permit.*

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 there;*
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 a special reindeer herding area;*
5) *ensuring that activity under the permit will not endanger the status of the Sami as an indigenous people in the Sami Homeland, 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; and*
9) *other specifications that are necessary in view of public and private interests and pertaining to the implementation of the conditions of the permit.*

Figure 20.1 Permitting Pathway in Finland – Company Sourced



20.6.4 Environmental Protection Policies and Strategies

Rupert Resources has a corporate social 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.

20.6.5 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.

20.6.6 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.

20.7 Social and Community Related Requirements

North Finland is the traditional area of the indigenous Sámi people. There are no Sámi people, areas or interests in the vicinity of the Rupert Lapland exploration licenses.

Reindeer herding is a common source of livelihood in Lapland. The Rupert Resources exploration permits fall within the Sattasniemi Reindeer Herding Area. Rupert has regular interaction with Sattasniemi reindeer herders and annual meetings (in January - February) to discuss matters concerning the interaction between exploration and reindeer herding and to coordinate each other's activities at the area.

At Pahtavaara, the nearest reindeer farm is located three km from the mine area, and animals are pasturing near and even inside the mine area.

At Ikkari, the nearest reindeer farm, and closest inhabited house, is located some 3.5 km to the southwest. Since reindeers are grazing freely, animals are pasturing across the whole exploration area. Sattasniemi reindeer herders have one of their main separation facilities some 3.5 km south from the Pahtavaara Mine, adjacent to the mine access road. Reindeers are collected and herded to these stations every autumn.

Rupert Resources has organised 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. A Local stakeholder survey will be done 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. Rupert Resources has established stakeholder steering committee for Ikkari EIA process, where authorities and local stakeholders can give their feedback and comments to ongoing EIA process.

20.8 MINE CLOSURE

Relating to Pahtavaara Mine only, under the current mine closure plan, the mill building will be retained whilst other buildings can be removed. Underground mine devices (transformers, electric centres, cables etc.) will be removed. Access to the underground mine will be closed.

All mine waste areas must be covered with 30 cm layer of moraine and vegetation layer and slopes shaped to assure safety.

Environmental and mining bond of EUR 850,000 is in place to ensure that the closure plan is implemented.

Rupert Resources is reviewing the closure plan as part of its evaluation of the production potential at Pahtavaara. This will define the amount of a new environmental bond.

Closure planning of Ikkari mine is initiated in the EIA phase and becomes more specific as the investigations progress.

21.0 CAPITAL AND OPERATING COSTS

21.1 SUMMARY

The preliminary economics of the Ikkari & Pahtavaara Project can be evaluated using the capital and operational cost estimates presented in this PEA. The calculations are based on underground and open pit mining operation, a processing plant, infrastructure, a tailings co-disposal facility, and the owner's expenses and provisions.

All capital and operational cost estimates are presented in US dollars, with no escalation or exchange rate variations factored in.

21.2 BASIS OF ESTIMATE

21.2.1 CLASS OF ESTIMATE

The estimate is a Class 4 Estimate prepared in accordance with the Association for the Advancement of Cost Engineering International Cost Estimate Classification System. The target accuracy of the estimate is, lower bounds -30% to -15% and upper bounds +20% to +50%, which is appropriate for the level of data available and suitable to inform the decision to proceed to the next feasibility phase.

21.2.2 BASE DATE

This estimate was prepared with a base date of Q1 2022.

21.2.3 APPROACH

The equipment and modular building supply cost estimates are based on the following:

- Mining schedule.
- Conceptual engineering design.
- Budget price enquiry for specialist equipment.
- CostMine InfoMine 2021 (InfoMine) database for processing equipment.
- Tetra Tech's in-house database or quotes from similar recent projects.
- Topographical information considered.
- Engineering design at a preliminary economic assessment level.

Non-equipment costs such as civil and structural, mechanical installation, electrical supply and installation, and logistics were calculated as a factor of the mechanical equipment supply costs.

21.3 Capital Cost Estimate

21.3.1 Summary

The total estimated capital cost of the Project is $799.3 million (-30% +50%), of which $404.6 million is initial (pre-production) capital costs, as set out in Table 21.1.

Table 21.1 Capital Cost Estimate Summary

Initial Capex	US $ millions
Mining o/p pre-production	16.6
Process plant	131.0
Civils and infrastructure	29.5
Water treatment	96.4
Tailings	20.4
First fills & spares	10.0
Owner's costs	20.0
Closure bond	37.2
Contingency	43.5
Total Initial capex	**404.6**
Sustaining Capex	**US $ millions**
Pahtavaara initial capex	41.0
Underground mining	178.8
Water treatment	34.0
Tailings & waste dump	34.9
Plant sustaining	101.0
Pahtavaara closure bond	5.0
Total Sustaining Capex	**394.7**

21.3.2 Direct Cost

Mining Capital Costs

The first-year mine costs were derived by Axe Valley as part of their techno-economic model for the project. It should be noted that contract mining is assumed, and they are also responsible for construction of the mobile fleet service facilities.

The mining capital presented here is the expected first year of invoiced costs from the mining contractor along with costs ascribed to the management of mining activities by the owner.

Table 21.2 summarises the Mine Area Capital Cost estimates for the PEA Project. It is the QP's opinion that these estimates are reasonable for the location and planned mine development and can be used for a PEA.

Table 21.2 Mining Capital Cost Summary

Description	WBS	Initial Capital Cost ($ M)	Sustaining Capital Cost ($ M)	Total ($ M)
Ikkari Open Pit	1100	16.6		16.6
Underground Mining	1200	0.0	178.8	178.8
Total		**16.6**	**178.8**	**195.4**

Process Plant Capital Costs

Conceptual process block flow diagram and process design criteria were used to generate the requirements for process equipment. Budget estimates from ongoing and completed projects comparable to the Project were used to determine the costs for mechanical equipment and building supplies, which were then scaled for size. The breakdown of costs for the process plant is shown in Table 21.3.

Table 21.3 Process Plant Capital Cost Summary

Description	Initial Capital Cost ($ M)
Equipment	45
Piping & valves	20
Instrumentation	7
Electrical	5
Process buildings	4
Construction	20
Engineering & fees	30
Total	**131**

The plant capital cost estimate is based on priced items on a major mechanical equipment list using InfoMine and in the case of specialist equipment, a price enquiry from recognised mining equipment suppliers. The equipment list items were sized and selected based on the high-level mass balance that was completed along with the process flow sheets.

On and Off-Site Infrastructure Capital Cost

The infrastructure capital costs include all powerlines, roads, concrete foundations and earthworks. In the off-site infrastructure, it is estimated that the High Voltage transmission line of 9 km includes a substation.

Table 21.4 provides a breakdown of the infrastructure costs.

Table 21.4 On and Off-Site Site Infrastructure Capital Cost Summary

Description	Initial Capital Cost ($ M)
Concrete (ancillary structures)	2.0
Earthworks (ancillary structures)	9.5
Powerlines (off-site)	5
Roads	13.0
Total	**29.5**

Tailings & Water Treatment

The capital cost of the CDF comprising the earthworks and underdrain systems, as well as the hydraulic structures and perimeter facilities has been estimated separately as $20.4 M.

The breakdown of the costs for the tailings and water management for the project is shown in Table 21.5.

Table 21.5 Water Treatment Plant Capital Cost Summary

Description	Initial Capital Cost ($ M)
Mine Impacted Water Treatment Plant	34.0
Dewatering bores	8.7
Discharge pipelines	13.8
Mine Process Water Treatment Plant	40.0
Total	**96.5**

21.3.3 Indirect Costs

Indirect costs are estimated as a percentage of certain direct costs, including the mining fleet, fuel storage, explosives magazine, onsite and off-site infrastructure, and the CDF. Project Delivery includes EPCM, environmental services, permitting and commissioning costs.

Project Delivery

Indirect costs are required during the project delivery to enable and support construction activities. The project indirect costs have been based on Tetra Tech's historical project costs of a similar nature. The indirect costs are estimated at 8% of total direct cost excluding mining area and are included together with owners costs.

Owner's Cost

The owner's costs are estimated at $20 M total direct cost excluding mining. Owner's costs have been benchmarked against comparable recent projects.

Closure Cost

Closure costs have been benchmarked against recent projects in similar jurisdictions. The project envisages an up-front closure bond of $37 M including applicable taxes as well as continuous closure costs of $0.80 /t moved. The bond is not considered returnable at the end of life in the financial analysis.

Provisions (Contingency)

Contingency is used to adjust for variations between estimated and actual costs for materials and equipment. The contingency amount fluctuates according to the contract terms and the client's demands. The estimate for capital costs must have a provision to offset the risk from these uncertainties because there were uncertainties when the estimate was created. A total contingency of $43.4 M has been included within the project's initial capital expenditures.

The contingency estimate does not allow for the following:

- Abnormal weather conditions.
- Changes to market conditions affecting the cost of labour or materials.
- Changes of scope within the general production and operating parameters.
- Effects of industrial disputations.
- Financial modelling.
- Technical engineering refinement.
- Estimate inaccuracy.

21.3.4 Sustaining Capital

Capital investment required to maintain production capacity and the costs implicated in preserving the current assets' production capability and implement the current production plan is incorporated in this category. The LOM project sustaining capital is $394.7 M as detailed in Table 21.1.

Underground Mining

The sustaining cost of the capital invested is included in the sustaining costs of mining, together with underground development. The LOM underground sustaining capital is $178.8 M.

Processing

The sustaining cost of the processing facility (plant sustaining) will be used to maintain the processing plant and associated infrastructure. The LOM processing facility sustaining cost is $101.0 M.

Pahtavaara Initial Capex

The investment in initiating production at Pahtavaara in 2038 is included in the sustaining capital estimates at a cost of $41.0 M, this includes costs for plant refurbishment and upgrades along with the initial pre-strip of the open pit.

TAILINGS & WATER TREATMENT

The sustaining cost for the tailings and water treatment caters for the planned expansion of the facility during the LOM. The LOM tailings and water treatment sustaining cost is $68.9 M.

PAHTAVAARA CLOSURE BOND

The cost of the closure bond for Pahtavaara is estimated at $5.0 M.

21.4 OPERATING COSTS

The estimated average total cash cost is $36.1 /t processed as shown in Table 21.6.

Table 21.6 Averaged LOM Operating Cost Estimate

Life of Mine Operating Cost	US $ / Tonne Milled	US $ /oz
Mining	18.1	333
Water treatment	1.4	26
Concentrate freight	0.1	2
Processing	10.9	204
Tailings	1.6	28
Closure fund	0.8	15
G&A	2.4	44
Freight/Refining	0.1	3
Royalty	0.7	12
Total Cash Costs	**36.1**	**667**

21.4.1 BASIS OF THE ESTIMATE

The process operating costs have been estimated based on test work, and the Tetra Tech database and applying a scaling factor or guidance from Rupert Resources on current costs. Tetra Tech's operating cost estimate is based on:

- A predominantly Finnish workforce. Rates are based on guidance from the client on current employee costs with a scaling factor for the different job grades.
- A 12-hour shift roster has been assumed with 3 shifts required to cover days off and holidays.
- Diesel fuel on site costs of $1.20 per litre and the cost of petrol on site of $1.53 per litre.
- Power cost of $0.06 per kilowatt hour (/kWh) based on guidance from Rupert.

21.4.2 Mining

Table 21.7 Mining Operating Costs

Description	Ikkari $/t	Pahtavaara $/t
Open Pit mining	2.51	2.60
Underground Mining	21.77	49.59
Average LOM O/P and U/G	16.20	34.30
Combined Operation LOM	$18.06 /t	

The average mining operating cost is estimated at $ 34.30 /t for Pahtavaara, $16.20 /t for Ikkari and $18 /t on a combined basis over the life of mine.

21.4.3 Processing

The total plant operating cost is estimated at $8.67 /t for Pahtavaara and $11.30 /t for Ikkari. Pahtavaara concentrates will be leached in the Ikkari plant. Treatment costs are estimated as $10.9 /t on a combined basis. Details of the breakdown of the costs are shown in Table 21.8.

Table 21.8 Process Plant Operating Costs

Description	Ikkari $ /t	Pahtavaara $ /t
Energy	2.75	3.27
Labour	1.90	3.65
Maintenance	1.29	0.50
Raw materials	5.36	1.25
Operational Costs	**11.30**	**8.67**

The process operating cost includes the following estimated components:

- Energy (Power and fuel).
- Labour (operations and maintenance).
- Raw materials (reagents and consumables).
- Maintenance spares.

Table 21.9 is a breakdown of the estimated energy costs. The process plant energy cost for Ikkari totals $2.75 /t and Pahtavaara totals $3.27/t. The fuel and diesel consumption are based on expected equipment with estimated consumption per vehicle. The Power costs are based on the equipment sizing using the limited or estimated values for equipment sizing. Only large (>100 kW) equipment sizes were estimated. Power costs have been calculated using the estimate of installed power requirements. It is anticipated that the electrical load will change with improved values for equipment sizing and an increase in the design detail.

Table 21.9 Process Energy Costs for Ikkari and Pahtavaara

Description	Unit Cost ($ /l or $ /kW)	Ikkari		Pahtavaara	
		Usage (l or GW)	Cost ($ /t)	Usage (l or GW)	Cost ($ /t)
Diesel	1.20	2,985,700	1.02	854,100	2.05
Petrol	1.53	149,285	0.07	42,705	0.13
Electricity	0.06	96.63	1.66	9.08	1.09
Total			**2.75**		**3.27**

Note: 1GW = Gigawatt = 1,000,000 kW

Labour has been estimated based on guidance from Rupert on what current pay grades are and scaled using a typical structure, from a similar project. The total labour cost was calculated to include burden costs. The process plant labour cost for Ikkari totals $1.90/t and Pahtavaara totals $3.65/t.

Reagent consumptions have been estimated based on the test work results available and from the experience of similar projects. Test work is ongoing to improve upon the values used. Reagent costs are detailed in Table 21.10.

Table 21.10 Process Materials Costs

Description	Unit Cost ($ /t)	Pahtavaara		Ikkari	
		Usage (t)	Cost ($ /t)	Usage (t)	Cost ($ /t)
Burnt lime (CaO)	230	-	0.00	900	0.06
Raw water	0	50,000	0.00	350,000	0.00
Cyanide (NaCN)	3,500	-	0.04	900	0.90
Potassium Amyl Xanthate (PAX)	3,500	5	0.88	112	0.11
Sodium Silicate (Na_2SiO_3)	1,750	250	0.02	1,750	0.88
Methyl isobutyl carbinol (MIBC)	2,200	5	0.00	35	0.02
Hydrochloric Acid (HCl) (32%)	195	-	0.00	900	0.05
Caustic/Sodium Hydroxide (NaOH)	815	-	0.00	97	0.02
Sodium metabisulphide (SMBS)	1,800	-	0.32	2,568	1.32
Flocculant	3,227	50	0.00	350	0.32
Grinding media	1,686	-	0.00	3,500	1.69
Total	-		**1.26**	-	**5.37**

Note: Reagent usage for Ikkari is inclusive of the treatment of the Pahtavaara concentrate. This has been estimated at $2.32 /t of concentrate treated at Ikkari.

The cost of transporting concentrate from Pahtavaara to Ikkari has been estimated at $8.00 /t of concentrate.

Maintenance and spares have been estimated to cost $1.29/t for Ikkari and $0.50 /t for Pahtavaara.

21.4.4 G&A Operating Cost

G&A operating costs, based on non-refractory gold projects across multiple operations and locations, have been estimated to be $2.35 /t processed.

22.0 ECONOMIC ANALYSIS

Tetra Tech has prepared this Preliminary Economic Analysis on behalf of Rupert Resources for the gold mining operations at Ikkari and Pahtavaara.

22.1 PRINCIPLE ASSUMPTIONS

The economic analysis is based on assumptions related to metallurgical recoveries and plant performance (discussed in Section 13) for the processing methods (discussed in Section 17).

Mine planning assumptions are discussed in Section 16.

Metal pricing at US $1,650 per ozT (discussed in Section 19) has been held constant through the life of the project.

Toll Refining Charges (TRC) have been estimated as US $2.50 /ozT.

Corporate taxation rate at 20%, US $ to EUR foreign exchange rate 1:1.

Royalties: State royalty 0.60% of revenues and Landowner royalty 0.15% of revenues.

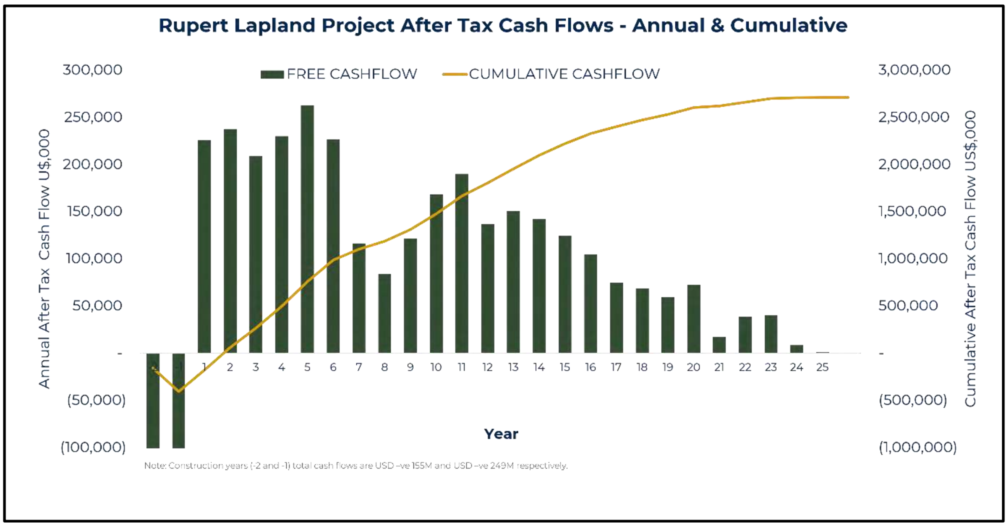
Capital and operating expenditures (discussed in Section 21) were used as the foundation for the economic assessment.

22.2 ECONOMIC HIGHLIGHTS

- 5% discount after-tax $NPV_{5\%}$ of US $1.6 billion.
- Unlevered IRR of 46% using a gold price of US $1,650 per ozT.
- After-tax payback period of two years of production.
- LOM gold production of 4.25 million ozT:
  - Ikkari gold production of 3.90 million ozT.
  - Pahtavaara gold production of 0.35 million ozT.
- LOM (Years 1-22) average gold production of 191 kozT per annum:
  - Year 1 to 11 average production of 220 kozT per annum.
  - Year 12 to 22 average production of 163 kozT per annum.
- LOM net free cash of US $2.71 billion.

Figure 22.1 shows the LOM net free cash and cumulative net free cash.

Figure 22.1 LOM Net Free Cash



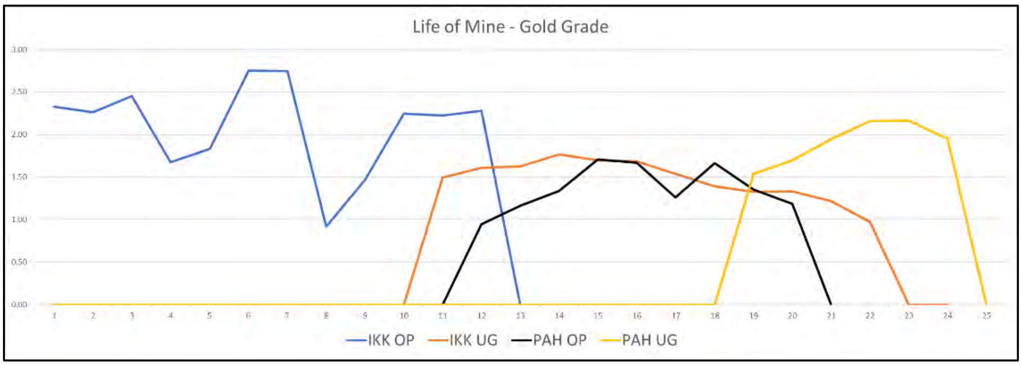
22.3 22.3 Mining Physicals

The economic analysis is based on the mining and process production schedules discussed in Section 16.

Payable metal is 99.92% of gold production.

LOM gold grades are shown in Figure 22.2.

Figure 22.2 Life of Mine Gold Grades



Key: IKK = Ikkari mine, OP = open pit, PAH = Pahtavaara mine, UG = underground

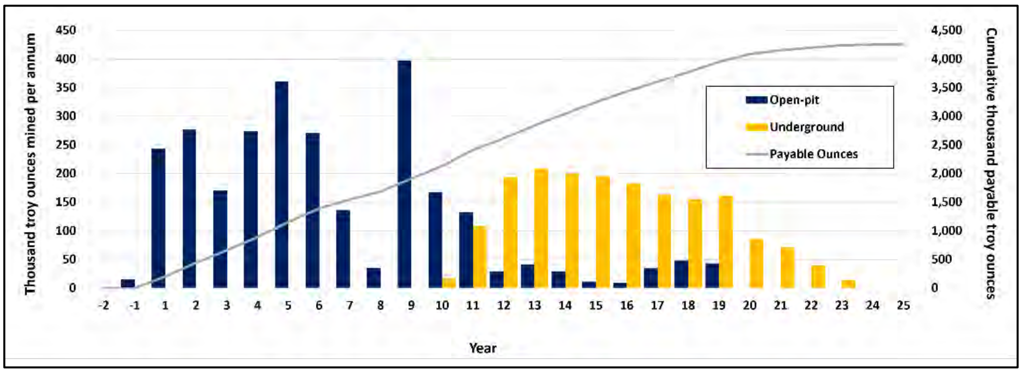
The process gold recoveries used to calculate the payable gold are based on the following overall values:

IKKARI	94.6%
PAHTAVAARA	89.2%

The above process recoveries exclude the 99.92% payable gold factor.

The mined gold and the transitions between ore bodies are shown in Figure 22.3 alongside the cumulative payable gold for the project.

Figure 22.3 Life of Mine Gold – Mined and Cumulative Payable



The total mined waste and ore are presented in Table 22.1.

Table 22.1 Life of Mine – Physicals – Ikkari and Pahtavaara Combined

Ikkari & Pahtavaara			2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051
Project Year			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
		Total/Ave.																										
PHYSICALS																												
Open-pit mining																												
Ore	kt	42,660	0	198	3,349	3,515	3,176	4,640	4,072	3,069	4,603	745	5,502	2,342	1,867	792	953	516	214	220	645	1,101	1,140	0	0	0	0	0
Au grade	g/t	1.99	0.00	2.33	2.26	2.45	1.67	1.83	2.75	2.75	0.92	1.47	2.25	2.22	2.21	1.16	1.34	1.71	1.67	1.26	1.66	1.36	1.18	0.00	0.00	0.00	0.00	0.00
Contained Au	Koz	2,726	0	15	244	277	171	274	360	271	136	35	398	167	132	30	41	28	11	9	34	48	43	0	0	0	0	0
Waste	kt	196,366	0	4,402	5,884	11,826	13,560	8,927	8,007	20,992	21,516	22,492	10,938	2,038	9,907	10,368	4,759	5,451	7,906	11,175	9,641	4,596	1,981	0	0	0	0	0
Total Material	kt	239,026	0	4,601	9,233	15,341	16,735	13,567	12,079	24,061	26,119	23,237	16,440	4,380	11,774	11,160	5,712	5,967	8,119	11,395	10,287	5,698	3,121	0	0	0	0	0
Strip Ratio	t:t	4.6	0.0	22.2	1.8	3.4	4.3	1.9	2.0	6.8	4.7	30.2	2.0	0.9	5.3	13.1	5.0	10.6	37.0	50.8	14.9	4.2	1.7	0.0	0.0	0.0	0.0	0.0
Underground mining																												
Ore	kt	35,835	0	0	0	0	0	0	0	0	0	0	0	349	2,096	3,697	3,681	3,646	3,612	3,688	3,638	3,641	3,733	1,809	1,440	576	226	0
Au grade	g/t	1.56	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.50	1.61	1.63	1.76	1.70	1.68	1.54	1.39	1.32	1.34	1.48	1.56	2.16	1.95	0.00
Contained Au	Koz	1,797	0	0	0	0	0	0	0	0	0	0	0	17	108	194	209	199	196	183	163	155	161	86	72	40	14	0
Waste	kt	6,533	0	0	0	0	0	0	0	0	0	0	172	494	574	392	392	450	538	400	427	594	1,059	499	494	48	0	0
Total Material	kt	42,368	0	0	0	0	0	0	0	0	0	0	172	844	2,670	4,090	4,073	4,096	4,150	4,088	4,065	4,235	4,793	2,308	1,935	624	226	0
Processing																												
PAH Feed	kt	7,988	0	0	0	0	0	0	0	0	0	0	0	0	0	407	501	500	500	500	501	500	750	750	752	750	750	750
Grade	g/t	1.56	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.55	1.62	1.64	1.44	1.25	1.50	1.66	1.72	1.57	1.81	2.01	2.00	0.51
Contained Au	koz	400	0	0	0	0	0	0	0	0	0	0	0	0	0	20	26	26	23	20	24	27	42	38	44	48	48	12
Concentrate Produced dmt	kt	1,046	0	0	0	0	0	0	0	0	0	0	0	0	0	53	66	66	66	66	66	66	98	98	99	98	98	98
Concentrate wmt	kt	1,151	0	0	0	0	0	0	0	0	0	0	0	0	0	59	72	72	72	72	72	72	108	108	108	108	108	108
Grade	g/t	10.82	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	10.76	11.28	11.40	10.03	8.67	10.45	11.50	11.97	10.91	12.59	13.96	13.88	3.51
Contained Au	koz	364	0	0	0	0	0	0	0	0	0	0	0	0	0	18	24	24	21	18	22	24	38	34	40	44	44	11
IKK Feed	kt	70,507	0	0	2,855	3,500	3,500	3,500	3,510	3,500	3,500	3,500	3,510	3,500	3,500	3,500	3,510	3,500	3,500	3,500	3,510	3,500	3,500	3,500	1,114	0	0	0
Grade	g/t	1.82	0.00	0.00	2.37	2.26	2.04	2.20	2.42	2.23	1.44	1.33	2.12	2.06	2.63	1.78	1.83	1.73	1.71	1.58	1.42	1.34	1.33	0.94	0.92	0.00	0.00	0.00
Contained Au	koz	4,123	0	0	217	254	229	248	273	251	162	149	239	232	296	200	207	195	192	178	160	151	150	106	33	0	0	0
Total IKK Plant Feed	kt	71,554	0	0	2,855	3,500	3,500	3,500	3,510	3,500	3,500	3,500	3,510	3,500	3,500	3,553	3,575	3,566	3,566	3,566	3,575	3,566	3,598	3,598	1,212	98	98	98
Grade	g/t	1.95	0.00	0.00	2.37	2.26	2.04	2.20	2.42	2.23	1.44	1.33	2.12	2.06	2.63	1.91	2.01	1.91	1.86	1.71	1.58	1.53	1.62	1.21	1.87	13.96	13.88	3.51
Contained Au	koz	4,487	0	0	217	254	229	248	273	251	162	149	239	232	296	219	231	219	213	196	182	176	188	141	73	44	44	11
Gold Produced	koz	4,257	0	0	206	240	217	234	258	238	153	141	226	220	280	207	219	208	202	186	173	167	179	134	70	43	43	11
Payable Au	**koz**	**4,253**	**0**	**0**	**206**	**240**	**217**	**234**	**258**	**237**	**153**	**141**	**226**	**220**	**280**	**207**	**219**	**208**	**202**	**186**	**173**	**167**	**179**	**134**	**70**	**43**	**43**	**11**

The life of mine physicals for Ikkari only are presented in Table 22.2.

Table 22.2 Life of Mine – Physicals – Ikkari only

Ikkari			2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048
Project Year			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
		Total/Ave.																							
PHYSICALS																									
Open-pit mining																									
Ore	kt	36,981	0	198	3,349	3,515	3,176	4,640	4,072	3,069	4,603	745	5,502	2,342	1,770	0	0	0	0	0	0	0	0	0	0
Au grade	g/t	2.08	0.00	2.33	2.26	2.45	1.67	1.83	2.75	2.75	0.92	1.47	2.25	2.22	2.28	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Contained Au	Koz	2,478	0	15	244	277	171	274	360	271	136	35	398	167	130	0	0	0	0	0	0	0	0	0	0
Waste	kt	132,441	0	4,402	5,884	11,826	13,560	8,927	8,007	20,992	21,516	22,492	10,938	2,038	1,859	0	0	0	0	0	0	0	0	0	0
Total Material	kt	169,422	0	4,601	9,233	15,341	16,735	13,567	12,079	24,061	26,119	23,237	16,440	4,380	3,629	0	0	0	0	0	0	0	0	0	0
Strip Ratio	t:t	3.6	0.0	22.2	1.8	3.4	4.3	1.9	2.0	6.8	4.7	30.2	2.0	0.9	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Underground mining																									
Ore	kt	33,526	0	0	0	0	0	0	0	0	0	0	0	349	2,096	3,697	3,681	3,646	3,612	3,688	3,638	3,636	3,598	1,156	727
Au grade	g/t	1.53	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.50	1.61	1.63	1.76	1.70	1.68	1.54	1.39	1.32	1.33	1.22	0.97
Contained Au	Koz	1,645	0	0	0	0	0	0	0	0	0	0	0	17	108	194	209	199	196	183	163	155	154	45	23
Waste	kt	4,786	0	0	0	0	0	0	0	0	0	0	172	494	574	392	392	450	538	400	427	488	460	0	0
Total Material	kt	38,312	0	0	0	0	0	0	0	0	0	0	172	844	2,670	4,090	4,073	4,096	4,150	4,088	4,065	4,124	4,058	1,156	727
Processing																									
Feed	kt	70,507	0	0	2,855	3,500	3,500	3,500	3,510	3,500	3,500	3,500	3,510	3,500	3,500	3,500	3,510	3,500	3,500	3,500	3,510	3,500	3,500	3,500	1,114
Grade	g/t	1.82	0.00	0.00	2.37	2.26	2.04	2.20	2.42	2.23	1.44	1.33	2.12	2.06	2.63	1.78	1.83	1.73	1.71	1.58	1.42	1.34	1.33	0.94	0.92
Contained Au	koz	4,123	0	0	217	254	229	248	273	251	162	149	239	232	296	200	207	195	192	178	160	151	150	106	33
Gold Produced	koz	3,900	0	0	206	240	217	234	258	238	153	141	226	220	280	189	196	184	182	168	151	143	142	100	31
Payable Au	**koz**	**3,897**	**0**	**0**	**206**	**240**	**217**	**234**	**258**	**237**	**153**	**141**	**226**	**220**	**280**	**189**	**196**	**184**	**182**	**168**	**151**	**143**	**142**	**100**	**31**

The life of mine physicals for Pahtavaara only are presented in Table 22.3.

Table 22.3 Life of Mine – Physicals – Pahtavaara only

Pahtavaara			2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052
Project Year			-13	-12	-11	-10	-9	-8	-7	-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
		Total/Ave.																											
PHYSICALS																													
Open-pit mining																													
Ore	kt	5,679	0	0	0	0	0	0	0	0	0	0	0	0	97	792	953	516	214	220	645	1,101	1,140	0	0	0	0	0	0
Au grade	g/t	1.36	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.95	1.16	1.34	1.71	1.67	1.26	1.66	1.36	1.18	0.00	0.00	0.00	0.00	0.00	0.00
Contained Au	Koz	248	0	0	0	0	0	0	0	0	0	0	0	0	3	30	41	28	11	9	34	48	43	0	0	0	0	0	0
Waste	kt	63,924	0	0	0	0	0	0	0	0	0	0	0	0	8,048	10,368	4,759	5,451	7,906	11,175	9,641	4,596	1,981	0	0	0	0	0	0
Total Material	kt	69,603	0	0	0	0	0	0	0	0	0	0	0	0	8,146	11,160	5,712	5,967	8,119	11,395	10,287	5,698	3,121	0	0	0	0	0	0
Strip Ratio	t:t	11.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	82.6	13.1	5.0	10.6	37.0	50.8	14.9	4.2	1.7	0.0	0.0	0.0	0.0	0.0	0.0
Underground mining																													
Ore	kt	2,309	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	6	135	653	713	576	226	0	0
Au grade	g/t	2.05	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.54	1.70	1.94	2.16	2.16	1.95	0.00	0.00
Contained Au	Koz	152	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	7	41	50	40	14	0	0
Waste	kt	1,747	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	106	600	499	494	48	0	0	0
Total Material	kt	4,056	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	111	735	1,152	1,208	624	226	0	0
Processing																													
Feed	kt	7,988	0	0	0	0	0	0	0	0	0	0	0	0	0	407	501	500	500	500	501	500	750	750	752	750	750	750	76
Grade	g/t	1.56	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.55	1.62	1.64	1.44	1.25	1.50	1.66	1.72	1.57	1.81	2.01	2.00	0.51	0.46
Contained Au	koz	400	0	0	0	0	0	0	0	0	0	0	0	0	0	20	26	26	23	20	24	27	42	38	44	48	48	12	1
Concentrate dmt	kt	1,046	0	0	0	0	0	0	0	0	0	0	0	0	0	53	66	66	66	66	66	66	98	98	99	98	98	98	10
Concentrate wmt	kt	1,151	0	0	0	0	0	0	0	0	0	0	0	0	0	59	72	72	72	72	72	72	108	108	108	108	108	108	11
Grade	g/t	10.82	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	10.76	11.28	11.40	10.03	8.67	10.45	11.50	11.97	10.91	12.59	13.96	13.88	3.51	3.22
Contained Au	koz	364	0	0	0	0	0	0	0	0	0	0	0	0	0	18	24	24	21	18	22	24	38	34	40	44	44	11	1
Gold Produced	koz	357	0	0	0	0	0	0	0	0	0	0	0	0	0	18	23	24	21	18	22	24	37	34	39	43	43	11	1
Payable Au	**koz**	**357**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**18**	**23**	**23**	**21**	**18**	**22**	**24**	**37**	**34**	**39**	**43**	**43**	**11**	**1**

22.4 FINANCIAL MODEL

Tetra Tech and the author of this section are not tax experts and have relied upon feedback from Rupert Resources for the finalisation of taxation assumptions.

The financial model incorporates both working capital and sustaining capital, as well as depreciation, royalties to both the government and landowners, plus mining fees paid to the landowners which are based on the amount of disturbed area.

Table 22.4 presents the basis for the above calculations.

Table 22.4 Financial Model Rates and Charges

GENERAL EXPENDITURE				
Sustaining capital		**4%**	**of Cash Costs less G&A and Closure OPEX**	
Working capital		**5%**	**of revenue**	
DEPRECIATION				
Rate for mobile plant		**20%**		*Rupert Resources*
Rate for fixed processing plant		**10%**		*Rupert Resources*
Rate for buildings, structures		**7%**		*Rupert Resources*
Buildings, structures		**9.7%**	**of CAPEX**	*Lang factor - buildings and structures - proportion of total CAPEX*
Fixed processing plant		**90.3%**	**of CAPEX**	*remainder of total CAPEX*
FEES				
Land owner royalty		**0.15%**	**of Revenue**	
State royalty		**0.60%**	**of Revenue**	
IKK mine fees	**€**	**50.00**	**/Ha**	*parity between US$ and Euro*
IKK OP Mine area		**1,500**	**Ha**	
IKK UG Mine area		**0**	**Ha**	
PAH mine fees	**€**	**100.00**	**/Ha**	*parity between US$ and Euro*
PAH OP Mine area initial		**420**	**Ha**	*nominal holding area - pre-production*
PAH OP Mine area final		**800**	**Ha**	*active mining chargeable area*
PAH UG Mine area		**0**	**Ha**	
CORPORATION TAX				
Tax		**20%**	**of operating profit**	
Opening losses	**$**	**150,000**	**thousand US$**	

Working capital has been estimated by comparing revenues for successive years and applying the given rate to the difference between the years. This results, as expected, in a flow of cash into and out of the working capital account as revenues fluctuate from year to year.

The estimated opening losses are carried forward to the first year of nominal profit and discounted from that year.

Figure 22.7 shows the LOM figures for gross revenue, cash costs, depreciation, corporate tax, and total royalties and mining fees. Also included are the values of EBITDA and Net Free Cash.

Table 22.5 Life of Mine – Operating Cashflows – Ikkari and Pahtavaara Combined

Ikkari & Pahtavaara			2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052
Project Year			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
		Total/Ave.																											
REVENUE																													
Gold Price	US$/oz	1,650	-	-	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650	1,650
Gross Revenue	US$ 000	7,018,224	-	-	339,201	396,136	357,261	386,173	425,220	391,666	253,046	232,603	372,491	362,351	461,784	341,964	361,161	342,769	333,729	306,950	284,792	275,136	295,199	221,094	115,806	71,267	70,823	17,932	1,670
Land Owner Royalty	US$ 000	(10,527)	-	-	(509)	(594)	(536)	(579)	(638)	(587)	(380)	(349)	(559)	(544)	(693)	(513)	(542)	(514)	(501)	(460)	(427)	(413)	(443)	(332)	(174)	(107)	(106)	(27)	(3)
State Royalty	US$ 000	(42,109)	-	-	(2,035)	(2,377)	(2,144)	(2,317)	(2,551)	(2,350)	(1,518)	(1,396)	(2,235)	(2,174)	(2,771)	(2,052)	(2,167)	(2,057)	(2,002)	(1,842)	(1,709)	(1,651)	(1,771)	(1,327)	(695)	(428)	(425)	(108)	(10)
Freight and Refining Charges	US$ 000	(10,642)	-	-	(514)	(601)	(542)	(586)	(645)	(594)	(384)	(353)	(565)	(549)	(700)	(519)	(548)	(520)	(506)	(465)	(432)	(417)	(448)	(335)	(176)	(108)	(107)	(27)	(3)
Net Revenue	**US$ 000**	**6,954,945**	**-**	**-**	**336,143**	**392,565**	**354,039**	**382,691**	**421,386**	**388,135**	**250,765**	**230,506**	**369,132**	**359,084**	**457,620**	**338,881**	**357,905**	**339,678**	**330,720**	**304,182**	**282,224**	**272,656**	**292,537**	**219,100**	**114,762**	**70,624**	**70,185**	**17,771**	**1,655**
OPERATING COSTS																													
Mining o/p	US$ 000	(573,361)	-	-	(24,910)	(35,415)	(38,259)	(36,877)	(34,396)	(49,955)	(56,517)	(49,147)	(47,464)	(22,649)	(18,060)	(23,446)	(16,426)	(15,933)	(18,950)	(25,810)	(25,277)	(18,977)	(14,893)	-	-	-	-	-	-
Mining u/g	US$ 000	(844,417)	-	-	-	-	-	-	-	-	-	-	-	(32,940)	(62,517)	(75,282)	(73,284)	(73,781)	(75,962)	(73,820)	(73,874)	(74,939)	(87,243)	(60,072)	(57,584)	(16,653)	(6,466)	-	-
Water Pumping & Treatment	US$ 000	(111,635)	-	-	(3,993)	(4,715)	(4,715)	(4,715)	(4,720)	(4,715)	(4,715)	(4,715)	(5,714)	(5,709)	(5,709)	(5,709)	(5,714)	(5,709)	(5,709)	(5,709)	(5,714)	(5,709)	(5,709)	(5,709)	(4,349)	(542)	(542)	(542)	(136)
PAH Concentrate Trucking	US$ 000	(9,209)	-	-	-	-	-	-	-	-	-	-	-	-	-	(469)	(578)	(576)	(576)	(576)	(578)	(576)	(865)	(865)	(867)	(865)	(865)	(865)	(88)
Processing	US$ 000	(868,821)	-	-	(32,250)	(39,534)	(39,534)	(39,534)	(39,642)	(39,534)	(39,534)	(39,534)	(39,642)	(39,534)	(39,534)	(43,187)	(44,141)	(44,020)	(44,020)	(44,020)	(44,141)	(44,020)	(46,263)	(46,263)	(19,328)	(6,729)	(6,729)	(7,404)	(752)
Tailings	US$ 000	(118,236)	-	-	(4,711)	(5,775)	(5,775)	(5,775)	(5,791)	(5,775)	(5,775)	(5,775)	(5,791)	(5,775)	(5,775)	(5,872)	(5,910)	(5,894)	(5,894)	(5,894)	(5,910)	(5,894)	(5,953)	(5,953)	(2,017)	(178)	(178)	(178)	(18)
Closure Fund	US$ 000	(62,796)	-	-	(2,284)	(2,800)	(2,800)	(2,800)	(2,808)	(2,800)	(2,800)	(2,800)	(2,808)	(2,800)	(2,800)	(3,126)	(3,209)	(3,200)	(3,200)	(3,200)	(3,209)	(3,200)	(3,400)	(3,400)	(1,493)	(600)	(600)	(600)	(61)
G&A Incl. Mining Fees	US$ 000	(186,181)	-	-	(6,785)	(8,300)	(8,300)	(8,300)	(8,323)	(8,300)	(8,300)	(8,300)	(8,323)	(8,300)	(8,300)	(9,262)	(9,506)	(9,480)	(9,480)	(9,480)	(9,506)	(9,480)	(10,068)	(10,030)	(4,427)	(1,805)	(1,805)	(1,805)	(221)
Total Opex	**US$ 000**	**(2,774,657)**	**-**	**-**	**(74,933)**	**(96,539)**	**(99,382)**	**(98,001)**	**(95,679)**	**(111,078)**	**(117,640)**	**(110,270)**	**(109,741)**	**(117,707)**	**(142,695)**	**(166,353)**	**(158,768)**	**(158,593)**	**(163,792)**	**(168,510)**	**(168,208)**	**(162,794)**	**(174,393)**	**(132,292)**	**(90,064)**	**(27,372)**	**(17,185)**	**(11,394)**	**(1,275)**
OPERATING CASHFLOW	**US$ 000**	**4,180,288**	**-**	**-**	**261,210**	**296,026**	**254,657**	**284,690**	**325,707**	**277,057**	**133,124**	**120,236**	**259,391**	**241,377**	**314,925**	**172,528**	**199,137**	**181,085**	**166,928**	**135,673**	**114,016**	**109,861**	**118,144**	**86,808**	**24,698**	**43,252**	**53,000**	**6,377**	**380**

Table 22.6 Life of Mine – Capital Expenditures and Pre-Tax Cashflows – Ikkari and Pahtavaara Combined

Ikkari & Pahtavaara			2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052
Project Year			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
		Total/Ave.																											
INITIAL CAPEX (IKK)																													
Mining o/p pre-production	US$ 000	(16,637)	-	(16,637)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Process Plant	US$ 000	(131,000)	(52,400)	(78,600)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Civils and Infrastructure	US$ 000	(29,494)	(11,798)	(17,696)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Water Treatment	US$ 000	(96,430)	(38,572)	(57,858)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings	US$ 000	(20,358)	(8,686)	(11,672)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
First Fills & Spares	US$ 000	(10,000)	-	(10,000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Owner's Costs	US$ 000	(20,000)	(8,000)	(12,000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Closure Bond	US$ 000	(37,200)	(18,600)	(18,600)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	US$ 000	(43,449)	(17,380)	(26,070)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Initial Capex	**US$ 000**	**(404,569)**	**(155,435)**	**(249,133)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
SUSTAINING CAPEX - INCL PAHT INITIAL																													
Pahtavaara Initial Capex	US$ 000	(41,001)	-	-	-	-	-	-	-	-	-	-	-	(3,086)	(37,915)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pahtavaara Expansion	US$ 000	(10,000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(10,000)	-	-	-	-	-	-	-	-
Underground Mining	US$ 000	(168,743)	-	-	-	-	-	-	-	-	-	-	(66,147)	(24,928)	(11,984)	(2,228)	(2,092)	(2,169)	(4,356)	(3,839)	(1,107)	(19,366)	(27,003)	(300)	(2,149)	-	(1,073)	-	-
Water Treatment	US$ 000	(34,000)	-	-	-	-	-	-	-	-	-	(17,000)	(17,000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings & Waste Dump	US$ 000	(34,945)	-	-	(3,060)	-	-	-	-	-	-	-	-	-	-	(3,086)	(4,068)	-	(4,073)	-	(5,085)	-	(6,110)	-	(6,102)	-	(3,361)	-	-
General Sustaining	US$ 000	(101,027)	-	-	(2,635)	(3,418)	(3,531)	(3,476)	(3,382)	(3,999)	(4,262)	(3,967)	(3,944)	(4,264)	(5,264)	(6,159)	(5,842)	(5,837)	(6,044)	(6,233)	(6,220)	(6,005)	(6,437)	(4,754)	(3,366)	(999)	(591)	(360)	(40)
PAT Closure Bond	US$ 000	(5,000)	-	-	-	-	-	-	-	-	-	-	-	-	(5,000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Sustaining Capex	**US$ 000**	**(394,716)**	**-**	**-**	**(5,695)**	**(3,418)**	**(3,531)**	**(3,476)**	**(3,382)**	**(3,999)**	**(4,262)**	**(20,967)**	**(87,092)**	**(32,279)**	**(60,163)**	**(11,473)**	**(12,002)**	**(8,005)**	**(14,474)**	**(10,072)**	**(22,412)**	**(25,371)**	**(39,550)**	**(5,055)**	**(11,617)**	**(999)**	**(5,026)**	**(360)**	**(40)**
TOTAL CAPEX	**US$ 000**	**(799,285)**	**(155,435)**	**(249,133)**	**(5,695)**	**(3,418)**	**(3,531)**	**(3,476)**	**(3,382)**	**(3,999)**	**(4,262)**	**(20,967)**	**(87,092)**	**(32,279)**	**(60,163)**	**(11,473)**	**(12,002)**	**(8,005)**	**(14,474)**	**(10,072)**	**(22,412)**	**(25,371)**	**(39,550)**	**(5,055)**	**(11,617)**	**(999)**	**(5,026)**	**(360)**	**(40)**
PRE-TAX CASHFLOW	**US$ 000**	**3,381,003**	**(155,435)**	**(249,133)**	**255,516**	**292,608**	**251,126**	**281,214**	**322,325**	**273,057**	**128,863**	**99,269**	**172,299**	**209,099**	**254,762**	**161,055**	**187,135**	**173,080**	**152,455**	**125,601**	**91,604**	**84,491**	**78,594**	**81,754**	**13,081**	**42,253**	**47,974**	**6,017**	**340**

Table 22.7 Life of Mine – Taxation, Working Capital and Free Cashflows – Ikkari and Pahtavaara Combined

Ikkari & Pahtavaara			2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Project Year			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
		Total/Ave.														
CORPORATE TAX																
Depreciation	US$ 000	(728,841)	-	(13,684)	(35,073)	(35,643)	(35,984)	(36,337)	(36,685)	(37,023)	(37,423)	(37,849)	(39,946)	(34,972)	(16,810)	(21,757)
Operating Profit	US$ 000	3,451,448	-	(13,684)	226,137	260,383	218,673	248,353	289,022	240,033	95,701	82,386	219,445	206,406	298,115	150,771
Tax allowance o/b	US$ 000		150,000	150,000	163,684	-	-	-	-	-	-	-	-	-	-	-
Tax allowance addition	US$ 000		-	13,684	-	-	-	-	-	-	-	-	-	-	-	-
Tax allowance used	US$ 000		-	-	(163,684)	-	-	-	-	-	-	-	-	-	-	-
Tax allowance c/b	US$ 000		150,000	163,684	-	-	-	-	-	-	-	-	-	-	-	-
Taxable income	US$ 000	3,354,336	-	-	62,454	260,383	218,673	248,353	289,022	240,033	95,701	82,386	219,445	206,406	298,115	150,771
Tax Payable	US$ 000	**(670,867)**	**-**	**-**	**(12,491)**	**(52,077)**	**(43,735)**	**(49,671)**	**(57,804)**	**(48,007)**	**(19,140)**	**(16,477)**	**(43,889)**	**(41,281)**	**(59,623)**	**(30,154)**
Working Capital																
Debtors - o/b	US$ 000		-	-	-	16,807	19,628	17,702	19,135	21,069	19,407	12,538	11,525	18,457	17,954	22,881
c/b	US$ 000		-	-	16,807	19,628	17,702	19,135	21,069	19,407	12,538	11,525	18,457	17,954	22,881	16,944
net change	**US$ 000**		**-**	**-**	**(16,807)**	**(2,821)**	**1,926**	**(1,433)**	**(1,935)**	**1,663**	**6,869**	**1,013**	**(6,931)**	**502**	**(4,927)**	**5,937**
FREE CASHFLOW	**US$ 000**	**2,710,136**	**(155,435)**	**(249,133)**	**226,218**	**237,711**	**209,318**	**230,111**	**262,586**	**226,713**	**116,591**	**83,805**	**121,479**	**168,320**	**190,213**	**136,838**
Cum. Cashflow			(155,435)	(404,569)	(178,351)	59,360	268,677	498,788	761,374	988,087	1,104,679	1,188,483	1,309,962	1,478,282	1,668,495	1,805,332
Payback	**Years**	**1.8**				1.8										
Discount Rate	**%**	**5.00%**														
NPV	**USDk**	**1,600,058**														
IRR	**%**	**45.9%**														

Ikkari & Pahtavaara	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053
Project Year	13	14	15	16	17	18	19	20	21	22	23	24	25	26
CORPORATE TAX														
Depreciation	(22,562)	(23,409)	(23,862)	(24,972)	(25,579)	(27,394)	(27,834)	(23,080)	(20,358)	(16,003)	(14,956)	(14,258)	(13,493)	(12,050)
Operating Profit	176,575	157,675	143,066	110,701	88,437	82,467	90,310	63,728	4,340	27,249	38,044	(7,881)	(13,113)	(12,050)
Tax allowance o/b	-	-	-	-	-	-	-	-	-	-	-	-	7,881	20,994
Tax allowance addition	-	-	-	-	-	-	-	-	-	-	-	7,881	13,113	12,050
Tax allowance used	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tax allowance c/b	-	-	-	-	-	-	-	-	-	-	-	7,881	20,994	33,044
Taxable income	176,575	157,675	143,066	110,701	88,437	82,467	90,310	63,728	4,340	27,249	38,044	-	-	-
Tax Payable	**(35,315)**	**(31,535)**	**(28,613)**	**(22,140)**	**(17,687)**	**(16,493)**	**(18,062)**	**(12,746)**	**(868)**	**(5,450)**	**(7,609)**	**-**	**-**	**-**
Working Capital														
Debtors - o/b	16,944	17,895	16,984	16,536	15,209	14,111	13,633	14,627	10,955	5,738	3,531	3,509	889	83
c/b	17,895	16,984	16,536	15,209	14,111	13,633	14,627	10,955	5,738	3,531	3,509	889	83	-
net change	**(951)**	**911**	**448**	**1,327**	**1,098**	**478**	**(994)**	**3,672**	**5,217**	**2,207**	**22**	**2,621**	**806**	**83**
FREE CASHFLOW	**150,869**	**142,456**	**124,289**	**104,787**	**75,015**	**68,476**	**59,538**	**72,680**	**17,430**	**39,010**	**40,387**	**8,638**	**1,146**	**83**
Cum. Cashflow	1,956,201	2,098,657	2,222,946	2,327,734	2,402,748	2,471,224	2,530,762	2,603,441	2,620,872	2,659,882	2,700,269	2,708,907	2,710,053	2,710,136

22.5 Sensitivity Analysis

The inputs to the financial model were varied in order to produce a sensitivity analysis.

The gold price, overall capital expenditure (CAPEX) and overall OPEX values were varied within the range of +20% to -20% from the base values. The results of the sensitivity analysis are presented in Table 22.8 to Table 22.10.

Table 22.8 Sensitivity Analysis – Gold Price (US $/oz)

Disc Rate	1,300	1,475	1,650	1,825	2,000
0%	1,527	2,119	**2,710**	3,302	3,893
5%	897	1,249	**1,600**	1,952	2,303
8%	664	934	**1,204**	1,474	1,744
10%	546	776	**1,007**	1,237	1,467
IRR	33%	40%	**46%**	52%	57%

Table 22.9 Sensitivity Analysis – Initial Capital Costs

Disc Rate	-20%	-10%	0%	10%	20%
0.0%	2,841	2,776	**2,710**	2,645	2,579
5.0%	1,703	1,651	**1,600**	1,549	1,497
8.0%	1,298	1,251	**1,204**	1,158	1,111
10.0%	1,095	1,051	**1,007**	962	918
IRR	56%	51%	**46%**	42%	39%

Table 22.10 Sensitivity Analysis – Operating Costs

Disc Rate	-20%	-10%	0%	10%	20%
0.0%	3,155	2,932	**2,710**	2,488	2,266
5.0%	1,843	1,721	**1,600**	1,479	1,357
8.0%	1,382	1,293	**1,204**	1,115	1,027
10.0%	1,154	1,080	**1,007**	933	859
IRR	49%	47%	**46%**	44%	43%

23.0 ADJACENT PROPERTIES

23.1 INTRODUCTION

Ikkari is located on the Rupert Lapland exploration project area surrounding the Pahtavaara Mine (Rupert Resources), which was the first mine to be developed in the CLGB belt in 1996. Since then, a number of significant mineral discoveries have been made, namely Suurikuusikko (gold), Kevitsa and Sakatti (both polymetallic base metals deposits). Since 2015, a number of 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 various deposits.

Figure 23.1 Recent Activity in Central Lapland

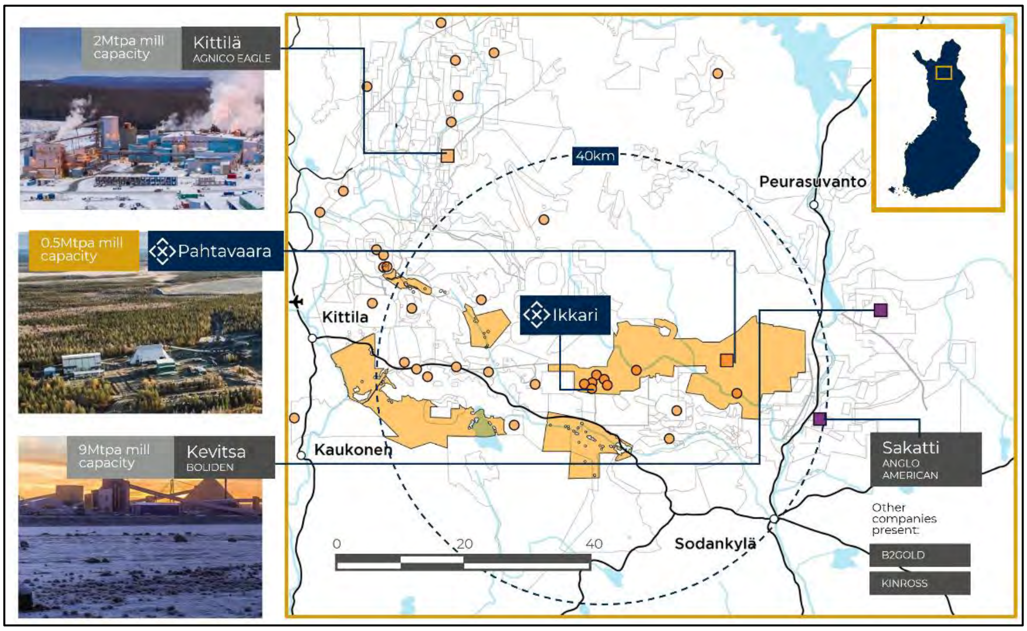


Table 23.1 Mineral Reserves and Resources in CLGB (December 2021)

Deposit	Type	Mt	Au (g/t)	Cu (%)	Ni (%)	Co (%)	Pt (%)	Pd (%)
Reserves								
Kevitsa (Boliden)	Proven	72	0.09	0.31	0.19	0.01	0.18	0.11
	Probable	51	0.10	0.33	0.27	0.01	0.23	0.15
Kittilä (Agnico Eagle)	Proven	1	3.85					
	Probable	27	4.26					
Resources								
Kevitsa (Boliden) *	Measured	50	0.08	0.33	0.21	0.01	0.17	0.11
	Indicated	88	0.07	0.36	0.23	0.01	0.11	0.07
	Inferred	2	0.03	0.19	0.11	0.01	0.03	0.02
Kittilä (Agnico Eagle) *	Measured	4	2.59					
	Indicated	19	2.6					
	Inferred	7	4.89					
Sakatti (Anglo American) *	Indicated	3.5	0.33	3.45	2.47	0.11	0.98	1.18
	Inferred	41	0.33	1.77	0.83	0.04	0.61	0.43

* Mineral Resources are reported exclusive of Mineral Reserves (see references, Boliden 2021; Anglo American plc, 2021; Agnico Eagle 2021).

23.2 SUURIKUUSIKKO/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 1.6 million tonnes of ore, yielding about 7,000 kg of gold. With a mine life estimated through 2034, its proven and probable mineral reserves contain 4.1 Moz gold (30.4 Mt at 4.16 g/t Au) as of December 31, 2020. Ore has been mined from underground since 2010 and the mine produced 221,914 oz of gold in 2021.

The Kittilä property covers 215 km^2, stretching 25 km along the Suurikuusikko Trend, a major gold-bearing shear zone. The mine area includes a group of six gold deposits along a 4.5 km segment of the trend. The largest of the deposits are the Suuri, Roura and Rimpi zones that contain most of the current reserves and resources at Kittilä. The other deposits are the undeveloped Sisar Zone, which is sub-parallel to the Main Zone, as well the Etela and Ketola zones. As part of a major expansion project at Kittilä, the commissioning of the expanded mill with its 25% increase in capacity was completed ahead of schedule in 2020 and the ramp-up towards the design capacity of 2.0 Mt per year is ongoing. The sinking of a 1,044-metre deep shaft as part of the expansion project experienced delays due to COVID-19 travel restrictions, and commissioning is now expected to be completed during the first quarter of 2023. The expansion project is expected to increase the efficiency of the mine and decrease or maintain current operating costs while providing access to the deeper mining horizons. This increased mining rate will be supported by the development of the Sisar Zone and deeper portions of the Main Zone. (Source; Agnico Eagle website).

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 concentrate and in addition to this, significant amounts of 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. In 2021, around 9,469 kt of ore were processed into metal concentrates. (Source; Boliden website).

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. A total of 166 km of drilling had been completed at the project at the end of 2020. An exploration permit and a permit from the Environmental Ministry for the exploration work at Sakatti was awarded during July 2020 enabling a three-year drilling programme, which commenced in November 2020. Environmental and social impact assessment (ESIA) was completed in December 2020 and environmental permitting commenced in January 2021. (Source; Anglo American Ore Reserves and Mineral Resources Report 2020).

23.5 AAMURUSKO PROJECT (AURION RESOURCES)

In February 2017 Aurion Resources reported the discovery of new, bonanza grade gold mineralisation on its 100% owned Risti Project in Northern Finland. The property is also known as the Aamurusko Project. The initial discovery was a 1150 m long by 700 m wide area of gold mineralisation with an apparent NE-SW trend that was discovered in late 2016. Here, 133 rock grab samples collected from predominantly large and angular sub-cropping quartz-tourmaline blocks assayed from nil to 1563.5 g/t Au, including 36 samples which assayed greater than 31 g/t Au (1 ounce per tonne). The average grade of all 133 samples was 74.3 g/t Au. Many of these samples contained abundant coarse visible gold. Aurion commenced drilling of Aamurusko in late 2017 and has subsequently identified additional mineralised zones along trend.

Aamurusko Main consists of gold-bearing quartz veins occurring near the sheared contact between sedimentary rocks and a gabbro intrusion, located on the south side of a steep, prominent ridge glacially up-ice from high-grade boulders. Drilling Highlights from Aamurusko Main include:

- 789.06 g/t Au over 2.90 m (including 3510.00 g/t Au over 0.65 m) from 116.10 m (Drill hole AM18042).
- 42.28 g/t Au over 4.00 m from 40.00 m (Drill hole AM19082).

Aamurusko Northwest is approximately 600 m northwest of Aamurusko Main target. This target consists of a 10 to 30 m wide zone of gold-bearing quartz veins within altered and mineralised clastic sedimentary rocks. Drilling has delineated Aamurusko NW to 150 m vertical depth and the target is open to extension. Drilling Highlights from Aamurusko NW include:

- 13.31 g/t Au over 19.54 m (including 22.58 g/t Au over 8.18 m) from 77.64 m (Drill hole AM19095).
- 3.51 g/t Au over 31.12 m from 55.88 m (Drill hole AM19094).

In late 2020, Aurion announced the discovery of multiple new, gold-bearing zones at Launi East Project (8 km southeast of their Risti project), which they report is within a 5.5 km long corridor parallel to the Sirkka Shear Zone. Many high grade boulder samples have been collected, and initial drill results indicate narrow vein-hosted mineralisation along a trend up to 1 km long. The best drill results include 63.90 g/t Au over 0.37 m, 5.50 g/t Au over 0.40 m and 3.05 g/t Au over 5.30 m.

23.6 Outapää Project (Aurion Resources and Kinross)

In February 2018, Aurion Resources reported that it had signed a non-binding letter of intent Kinross Gold Corporation giving Kinross the right to earn up to 70% of the Outa Project which comprises approximately 15,000 ha to the west of Pahtavaara.

23.7 Kutuvuoma Project (B2 / Aurion Resources)

Kutuvuoma adjoins Rupert's Lapland Project area on its westmost boundary and a small gold occurrence operated as a satellite pit for the Pahtavaara mill in the late 1990s.

In August 2015 Aurion entered into a joint venture (JV) agreement whereby B2Gold could earn 75% of the Kutuvuoma Project by spending CA $15 million and completing a feasibility study for the Project. In August 2019, B2Gold exercised its option to acquire a 51% interest in the Finland JV covering approximately 29,000 ha, which include the Kutuvuoma, Ahvenjärvi and Sinermä projects. Since inception of the agreement, dated January 13, 2016, B2Gold completed over CA $5 million in exploration expenditures, paid Aurion CA $50,000 in cash and issued 550,000 B2Gold shares over a four-year period to complete the requirements of the first option. B2Gold is currently earning an additional 19% interest by spending a further CA $10 million over two years, and, if exercised, an additional 5% interest by completing a feasibility study, for a total of 75%.

In December 2016 Aurion reported the results of the maiden drill programme and further drilling by B2Gold has been focused on identifying extensions to the Kutuvuoma resource and attempting to trace the mineralised trend along strike. They interpret over 5 km long high-grade gold mineralised target with limited shallow drilling with initial drill results returning up to 11.4 g/t Au over 13.3 m.

Most recently, drilling in late 2020 by B2Gold has followed up on base of till anomalies associated with interpreted western strike extensions of the mineralised zones in Area 1 (Rupert Resources). B2Gold's Helmi target is located along strike and in between Rupert Resources' Ikkari discovery (1.3 km to east) and the Kutuvuoma prospect (3.5 to 5 km to the west), within the metavolcanic and metasedimentary rocks of the Savukoski group near the contact with the sedimentary rocks of the Kumpu group. An initial, widely spaced, five-hole (1,259.1 m) diamond drilling programme tested selected geochemical (gold in base of till) and geophysical targets over an area extending 1,300 m in strike length. All drill holes intersected zones with elevated gold (>0.1 g/t Au) with mineralised zones encountered in multiple lithologies including ultramafic and mafic volcanic rocks, siltstones, graphitic sediments and in contacts between volcanic rocks and felsic/porphyritic dykes. Gold mineralisation was intersected in all holes (from anomalous to 28.90 g/t Au) in the maiden drill programme on the Kutuvuoma East target.

Most recently, infill and extension drilling by B2Gold at Helmi has yielded significant new intercepts that indicate potential for an economic deposit, including:

- 2.05 g/t Au over 77.50 m, including 4.18 g/t Au over 24.55 m (hole IKK22018).
- 1.42 g/t Au over 15.90 m, including 2.13 g/t Au over 6.35 m (hole IKK22019).

The information in this section that relates to adjacent properties is derived from public domain information and the QP has not been able to verify this information.

24.0 OTHER RELEVANT DATA AND INFORMATION

Rupert Resources has been developing a sustainability strategy with Grain Sustainability, a leading consultant in this field who provide organisations with sustainability assessment, planning and implementation tools to create a positive impact on people and the planet, while increasing stakeholder engagement and providing competitive advantage. The key aim for the Company is to evaluate, alongside technical studies, the broader impacts of the future development of the Rupert Lapland Project. The sustainability strategy work is informed by and aligned to a number of recognised sustainability standards including the Global Reporting Initiative (GRI), the Future-Fit Business Benchmark, and the Task Force on Climate-related Financial Disclosures (TCFD). The Company aligns itself to the globally recognised Paris Agreement and Finland's 2035 net zero target. The company is formally committed to three further frameworks which steer our sustainability strategy and action: International Council on Mining and Metals (ICMM), The Finnish Network for Sustainable Mining (FNSM) and the United Nations Global Compact (UNGC). The company has issued an annual Sustainability report for the year 2021 that includes estimates of the emissions from operations at the Project. Further technical studies will include evaluations of the potential emissions profiles of the development options for inclusion in the EIA programme and annual sustainability reports.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 CONCLUSIONS

The PEA results provide a high-level initial estimate of the potential economic value of the mineral resources discovered to date. The report also shows Ikkari to be robust technically and capable of sustaining high margin production over a mine life of more than 20 years. Eighty four percent of resource ounces at Ikkari are expected to report to an Indicated resource category based on 73,000 m of diamond drilling which defines a cohesive deposit with broad intervals of consistent high-grade gold.

Rupert plans 72,800 m of drilling over the 2022/23 drill season, ahead of the likely drill cut-off for its upcoming PFS. Seventy percent of the drilling is allocated for Ikkari and nearby satellites and 30% to regional programmes. The near-term aim is to identify potential mineralisation that should be considered in the next level of engineering study and any future development scenario planning.

Sampling and data verification methods are considered above industry standard. Rupert 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 good industry practice. Sampling has not identified any issues which materially affect the accuracy, reliability or representativeness of the resource estimates.

The production model considered an open-pit operation at Ikkari in the first 11 years, transitioning to Ikkari underground (years 10-23) and with a contribution from the existing Pahtavaara mine in years 12 to 24. The core 22-year LOM includes recovered gold of 4.25 Moz with average annual production of 200,000 oz. The open pit operation is expected to support average annual production of 220,000 oz in years one to 11 principally owing to higher grades.

Metallurgically the project is considered “free milling,” or non-refractory and ore can be processed using a conventional flow sheet with technology that is readily available and employed at similar operations worldwide. An important characteristic of the Ikkari ore is its ability to liberate most of the gold at a P_{80} of 175 µm with a mass pull of less than 10% which has positive implications for both capital and operating costs.

Inflows to the mines are expected to be mostly in the early stages of mine life during open pit development and in the initial excavation of the underground operations. Thereafter the rate should steadily decline as the storage of the surrounding rock mass is gradually depleted. The current study has focussed on initial site characterisation of groundwater, future studies will consider dewatering requirements, depressurisation of upper pit slopes, flood protection and additional investigations relating to superficial groundwater quality. Pit closure is not anticipated to present any difficulty however, co-disposal requirements and requirements for post closure seepage management will be fully assessed during the next phases of development.

Rupert Resources has initiated a sustainability programme with Grain Sustainability, a specialist consultancy. The key aim for the Company is to evaluate, alongside technical studies, the broader impacts of the future development of the Rupert Lapland Project. The work is informed by and aligned with a number of recognised sustainability standards including the Global Reporting Initiative (GRI), the Future-Fit Business Benchmark, and the Task Force on Climate-related Financial Disclosures (TCFD). Alignment will also be sought with the globally recognised Paris Agreement and Finland’s 2035 net zero target. The Company has also committed to three further frameworks which guide sustainability strategy and action: International Council on Mining and Metals (ICMM), The Finnish Network for Sustainable Mining (FNSM) and the United Nations Global Compact (UNGC). The Company has issued its first annual sustainability report for the year 2021 to act as a base for future work and includes estimates of the emissions from current operations at the Project. Further technical studies will include evaluations of the potential emissions profiles of the development options for inclusion in the EIA programme and annual sustainability reports.

Initial, preproduction capital was estimated to be \$404.6 M with a further \$394.7 M of sustaining capital required over the LOM. On a unit basis expected all-in sustaining cost (AISC) was shown to be \$759 /oz over LOM, and \$596 /oz during open-pit operation. The study showed an After-tax NPV (5% discount) of \$1.6 billion with unlevered IRR of 46% and payback after only two years, assuming a gold price of \$1,650 per troy ounce.

26.0 RECOMMENDATIONS

The PEA has demonstrated the economic potential of the Ikkari project and justifies accelerated development toward implementation. As the project moves into the PFS the following additional studies and works are recommended.

26.1 GEOLOGY

The Ikkari gold deposit was a new discovery by Rupert Resources in early 2020. The first Inferred Mineral Resource was based on an initial 36,000 m of drilling at the predominantly Indicated Mineral Resource reported here included an addition 37,000 m of drilling for 73,000 m total. Much of the additional drilling has focused on infilling the initial resource to achieve the Indicated category, as well as incremental extensions to the resource at its margins (east and west, as well as at depth). The footprint of the mineralisation still remains open at depth in the east and west, as well as vertically below the main mineralised zone. Further drilling is expected to extend the known mineralisation.

The following is recommended:

- While the recent infill drilling at Ikkari has improved the resource category from Inferred to Indicated for ~85% of the deposit extent (94% of the open pit resource is now Indicated), selected further infill drilling is required to move the remaining 15% of the deposit to the Indicated category. On the basis of the current resource, a further 20,000 m of drilling would be required to complete the infill to 40 m x 30 m spacing.
- Continue to drill further holes to step out and down from the existing coverage, given that recent drilling has continued to demonstrated depth extent to 750 m vertical depth in the central part of the deposit. Despite the mineralised intercept at this depth, adjacent sections throughout the deposit have not yet been drill tested to a similar vertical extent. Drill testing to <750 m vertical depth across the strike extent of the deposit is estimated to require approximately 30,000 m.
- Exploration around the margins of the deposit, particularly in the east where recent drilling has indicated ongoing continuity of mineralisation at depth (<500 m vertical), is considered likely to add incrementally to the resource, with approximately 20,000 m of drilling allocated to this.
- During initial stages of this PEA both LHOS and SLC were assessed as methods of underground extraction with SLC forming the base case presented. Further detailed geological modelling, based on closer spaced data, to assess the grade continuity and selectivity possible in underground mining will better inform the decision making on underground extraction techniques during subsequent studies.
- Despite initial studies included in this PEA document, additional and ongoing data collection to more fully understand the probability of

economic extraction is warranted. This should include increasingly more detailed geotechnical, geochemical and hydrogeological data extending beyond the immediate footprint of the deposit, to not only inform optimisation of pit shells and underground mining capability, but also assess the potential environmental impacts of mining activities. Geology should be used to inform and assist any Environmental impact studies and contribute (geochemical) data towards waste and tailing characterisation programmes, for example.

- Regional exploration: Given the demonstrated potential for significant mineralising systems in the vicinity of Ikkari, as well as throughout the belt, exploration is recommended to continue within the Rupert Lapland Project area, using similar techniques. It is also recommended that geological and structural studies of the Ikkari deposit be advanced, and further used to target similar structural settings on a regional scale, which may require the application for and acquisition of additional exploration permits.

An indicative budget for the recommended work outlined is summarised in Table 26.1.

Table 26.1 Budget - Recommended Work at Ikkari Deposit

Work Programme	Estimated Drill Metres	Cost (US $)
Ikkari infill drilling	20,000	3,412,500
Ikkari extension/exploration drilling	30,000	5,175,000
Ikkari hydrogeological, geotechnical and EIA support		750,000
Ikkari geological and structural studies		75,000
Regional exploration – new permits and geophysics		375,000
Regional exploration – base of till sampling		600,000
Regional exploration – target drilling	15,000	2,250,000
Total		**12,637,500**

26.2 MINING

The mining estimates in the current PEA for both open pit and underground operations and the attendant techno economic models are well developed. Tetra Tech recommends that further studies consider:

- Development of a mining reserve in relation to upgrading of resources into Measured and Indicated categories.
- Confirmation of the optimal mining rate and LOM.
- Detailed investigation into mine geotechnical criteria in relation to their impact on mine design requirements.
- Further optimisation of underground mining methods in particular detailed trade-offs in relation to LHOS and SLC.
- Cut-off grade optimisation.

- Back-fill, CRF and engineered fill requirements and testing.
- Mining equipment selection and costs.
- Optimisation of mine production schedule and stockpiling requirements.

26.3 METALLURGY

As the project moves into the PFS phase Tetra Tech recommend implementation of a comprehensive metallurgical development plan, which should include the following:

- A metallurgical sampling programme for the purposes of variability testing of identified lithological and mineralogical domains, as well as bulk composite testing. Consideration should be given to incorporation of metallurgical core sample requirements into the geological drilling programme.
- Quantitative mineralogical analysis (QEMSCAN) of variability samples, as well as polished and thin section evaluations.
- Variability comminution test work including:
  - Abrasion Index tests.
  - JK Drop-Weight tests.
  - SAG Mill Comminution tests.
  - Bond Ball Work Index and Crushability Index tests.
- Comprehensive gravity variability testing inclusive of GRG and EGRG testing.
- Variability flotation test work including:
  - Batch reagent and flotation conditions optimisation tests.
  - Variability batch flotation tests.
  - Bulk composite locked cycle testing.
- Whole ore and GRG tails variability cyanidation testing.
- Bulk concentrates cyanidation testing.
- Comprehensive cyanidation detoxification tests of bulk composite leach residues including INCO – SO_2/Air, Hydrogen Peroxide, Caros Acid, Biological de-toxification and ferri-cyanide.

Tetra Tech anticipates the estimated cost of this test work programme to be between US $600,000 and US $800,000.

Tetra Tech also recommends re-evaluating the process flowsheet and undertaking additional options trade-offs depending on the results.

26.4 HYDROLOGY

In development of a subsequent PFS it is recommended the following requirements be considered:

- Meteorology:

Installation of a site automatic weather station to allow data collection to inform future stages of the study and for operations, as well as for the design of any flood control structures.

- Surface water:

A flood study should be completed for design of the flood protection structures. The study should consider changes to natural flood lines owing to the installation of infrastructure (heavy equipment crossings and at the co-disposal dump). The study should also consider the potential for shallow groundwater to compromise the stability of the upper pit wall during flood events. Future work should also consider the seasonal variation in baseflow and how this may provide better insight into localised groundwater recharge.

- Geology (exploration drilling):

As exploration and geotechnical drilling continues at the site, it is important that the collection of hydrogeological data is also maximised. Templates for drillers logs should include comments regarding groundwater conditions, such as noticeable inflows or drilling fluid being lost to the formation. The installation of standpipe piezometers and VWP in exploration drill holes should be continued.

- Water quality:

For the next stage of the study, a comprehensive baseline dataset should be obtained, paying particular attention to the natural variability of water quality in the glacial deposits and peat. The programme should also assess variations in water quality at different times of the year (spring thaw, summer flows, winter baseflows). Comparison against Finnish water quality standards should also be undertaken to determine whether it may be feasible to discharge water directly to the environment, if required.

- Hydrogeology:

The next phase of study should include prolonged pumping trails to assess overall groundwater connectivity within the site area. Borehole data should be checked to confirm measured high heads and high vertical gradients. Further drilling and cross hole testing should be undertaken to confirm the connectivity of the mineralised zone to the wider groundwater flow system.

A programme to plug and seal exploration drill holes should also be implemented. The potential for open drill holes to allow a hydraulic connection between the weathered bedrock zone and the future underground workings should also be evaluated.

Budgetary requirements for this work are currently being assessed.

26.5 INFRASTRUCTURE

In addition to the foregoing, Tetra Tech makes the following recommendations for additional investigations at the next stage of the study:

- The following site investigations should be completed:
  - Mine geotechnical assessment.
  - Infrastructure geotechnical assessment for the site infrastructure, building, Co-disposal and WRD areas.
  - Site-seismic classification.
  - Mineralised material and waste geochemistry testing, as well as ARD test work.
  - Road condition survey.
- A TMF and Co-disposal trade-off study should be completed.
- Discussions should be held with the power authority to confirm the capacity of the high-voltage line crossing the licence area and the potential for this line to support the Project.
- The policy in regarding employee housing and transport requirements should be confirmed.
- Detailed digital terrain maps (DTMs) of the area in electronic format should be updated and their validity confirmed with local and national authorities. The DTM should have a scale of at least 1 m contours initially for a PFS level study.

Tetra Tech anticipates that many of the site investigations can be co-ordinated with the drill exploration programme, which can be used to collect geotechnical and hydrogeology data. The drilling programme can also be used to collect rock samples for the metallurgical test work and the ARD test work. The estimated additional cost of the additional ARD studies is approximately of US $250,000.

26.6 ENVIRONMENTAL STUDIES

Rupert Resources has been pro-actively engaged in baseline environmental studies and data collection and stakeholder engagement since 2018, and a comprehensive environmental and permitting programme has already been implemented, which is designed to support subsequent pre-feasibility and feasibility studies as well as licensing and permitting requirements. This work is already well advanced.

The PEA base case is only one of three potential development options that are currently being considered and that will be the subject of the EIA Programme that will be presented to all relevant authorities and at public meetings expected to occur in early 2023. The scope or works underway is inclusive of:

Completed studies:

- Desktop review of nature values and habitats of exploration area (Eurofins, 20 February 2018).
- Pahtavaara mine waste management plan and mine waste characterisation (Eurofins, 3 July 2019).
- Moor frog – study of Ikkari & Heinälamminvuoma 2019 (Eurofins, 9th October 2019).
- Breeding bird line transect censuses in Heinälamminvuoma (Ikkari), 2019 (Eurofins, 18 September 2019).
- Pahtavaara tailings pond vegetation cover monitoring 2020 (Ramboll, 2020).
- Nature survey of exploration areas, 2019 (Ramboll, 30 January 2020).
- Desktop study of freshwater pearl mussels (Eurofins, 28 March 2021).
- Pahtavaara tailings pond vegetation cover monitoring 2021 (Ramboll, 2021).
- Status of Ikkari water systems (Envineer, 12 November 2021).
- Phase 1 Review of data to support the hydrogeological study of the Ikkari gold and satellite deposits, Northern Finland (SRK Consulting, December 2021).
- Ikkari area nature survey 2021; birds, moor frogs, bats, directive inspects, otter (Envineer, 14 January 2022).
- Preliminary review of route alternatives for Ikkari discharge water (Envineer, 10 May 2022).
- Climate change model for Ikkari gold mine (Envineer, 31 May 2022).
- Phase 2 Hydrogeological field study report for the Ikkari Au and satellite deposits, Northern Finland (SRK Consulting, 13 June 2022).
- Pahtavaara tailings pond vegetation cover monitoring 2022 (Ramboll, 2 November 2022).
- Pahtavaara waste rock area vegetation survey 2022 (Ramboll, 2 November 2022).
- Ikkari geochemistry data review, waste characterisation and gap analysis (Mine Environment Management Ltd, 2 November 2022).
- Pahtavaara geochemistry data review, waste characterisation and gap analysis (Mine Environment Management Ltd, 2 November 2022).

Studies that have yet to be reported:

- Ikkari area nature survey 2022; mammalian snow tracks, birds, moor frogs, bats, directive inspects, otter (Envineer).
- Pahtavaara area nature survey 2022; mammalian snow tracks, birds, moor frogs, bats, directive inspects, otter (Envineer).

- Ikkari area flora & fauna survey 2022 (Envineer).
- Pahtavaara area flora & fauna survey 2022 (Envineer).
- Ikkari area Endangered moss specie survey 2022 (Envineer).
- Ikkari area freshwater pearl mussel survey 2022 (Eurofins).
- Aquatic biological surveys in Ikkari and Pahtavaara; fish, benthos, diatoms, aquatic and coastal habitats and aquatic vegetation 2022 (Envineer).
- Archaeology survey for Ikkari and Pahtavaara area 2022 (Mikroliitti).
- Ikkari peat layer and peat quality studies 2022 (Geolite & Afry).
- Ikkari mine waste sampling and static testing results 2022 (MEM).
- Pahtavaara mine waste sampling and static testing results 2022 (MEM).

On completion of the EIA programme of work and the planned PFS, the results will be presented in the project EIA document that will form the basis on which an environmental permit application is submitted. Rupert Resources has also begun a parallel programme of land use planning with the local and regional authorities and has also set up a stakeholder co-operation group that will have the opportunity to comment and give opinions and feedback during the EIA process.

27.0 REFERENCES

Agnico Eagle. 2021. Annual Information Form for the year ended December 31, 2021.

Anglo American plc. 2021. https://finland.angloamerican.com/en/about-sakatti.

Anglo American plc. 2021. Ore Reserves and Mineral Resources Report 2021.

Aurion Resources. 2015 to 2022. Various press releases. https://www.aurionresources.com/news/2020/.

Boliden. 2021. Boliden Summary Report Resources and Reserves, 2021, Kevitsa Mine, Prepared by Loraine Berthet.

Bonson, C. 2022. Geological Review and Map Interpretation, Area 1 – Lapland Gold Project, unpublished internal company report.

Cook, M. 2021. Ikkari Deposit – Gold Recovery Testing (Job number 21-1882), unpublished internal company report.

Davis, B. 2018. Pahtavaara gold mine, Finland – Comments on Deposit-scale Faults, unpublished internal company report.

Eilu, P. & Niiranen, T. 2013. Gold deposits in northern Finland. Excursion guidebook FIN1, SGA Biennial Meeting. Geological Survey of Sweden, Uppsala, pp. 12.

Eilu, P., Pankka, H., Keinänen, V., Kortelainen, V., Niiranen, T. & Pulkkinen, E. 2007. Characteristics of gold mineralisation in the greenstone belts of northern Finland. Geological Survey of Finland, Special Paper 44, 57−106.

Grinding Solutions Ltd. 2022. Ikkari Deposit – Gold Recovery Testing.

Grinding Solutions Ltd. 2022. Phase II – Ikkari Gold Recovery optimisation.

Grinding Solutions Ltd. 2022. Pahtavaara Gold Recovery Testing.

Grinding Solutions Ltd. 2022. Pre-Aeration Cyanidation Testing on Generated Flotation Concentrate.

Grinding Solutions Ltd. 2022. Bulk Preparation and Cyanide Destruction Trials.

Groves, D.I., Goldfarb, R.J., Gebre-Mariam, M. 1998. Orogenic gold deposits: A proposed classification in the context of their crustal distribution and relationship to other gold deposit types. Ore Geology Reviews 13, 1–28.

Groves, D. & Santosh, M. 2015. The giant Jiaodong gold province: The key to a unified model for orogenic gold deposits? Geoscience Frontiers, v. 7.

GTK. 2021. Mineral Deposit Report. Pahtavaara production history and historical resources. http://tupa.gtk.fi/karttasovellus/mdae/raportti/376_Pahtavaara.pdf.

Harju, S. 2022. U-Pb zircon dating of metasedimentary rocks within the Ikkari gold deposit, Central Lapland belt, northern Finland. Unpublished Master's thesis, University of Oulu.

Korkiakoski, E.A. 1992. Geology and geochemistry of the metakomatiite-hosted Pahtavaara gold deposit in Sodankylä, Northern Finland, with emphasis on hydrothermal alteration. Geological Survey of Finland, Bulletin 360, 96.

Niiranen, T., Lahti, I. & Nykänen, V. 2015. The Orogenic Gold Potential of the Central Lapland Greenstone Belt, Northern Fennoscandian Shield, Chapter 10.2 in Mineral Deposits of Finland.

Pulkkinen, E., Ollila, J., Manner, R., Koljonen, T. 1986. Geochemical exploration for gold in the Sattasvaara komatiite complex, Finnish Lapland. In: Prospecting in Areas of Glaciated Terrain. Institution of Mining and Metallurgy, London, pp. 129–137.

Selley, D. 2019. Regional Geology, unpublished internal company report.

Selley, D. 2020. Structural, stratigraphic, and mineralisation features of the Heinä Central prospect: A drillcore photograph- and geochemistry-based desktop study, unpublished internal company report.

Selley, D. 2021. Lithologic, structural and alteration model for the Ikkari Au deposit, unpublished internal company report.

SRK 2021. Ikkari Trade-off Results.

SRK 2020. Pahtavaara Trade-off Report.

Srivastava, M.R. and Isaacs, E. 1989. An Introduction to Applied Geostatistics. Oxford University Press, New York.

Wolfe, B. 2021. NI 43-101 Technical Report: Ikkari Project, Finland.

Wyche, N. L., Eilu, P., Koppström, K., Kortelainen, V. J., Niiranen, T., Välimaa, J. 2015. Chapter 5.2 - The Suurikuusikko Gold Deposit (Kittilä Mine), Northern Finland pp 411-433 In Mineral Deposits of Finland (Eds Maier, D. G., Lahtinen, R. & O'Brien, H.).

Yardley, B. W. D. & Graham J. T. 2002. Origins of salinity in metamorphic fluids. Geofluids, 2., 249-256.

APPENDIX A

Ikkari Mining Cut-off Grades

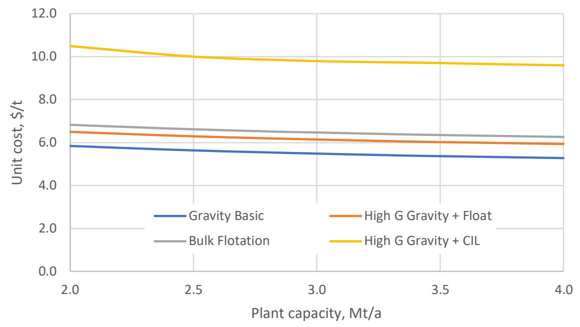
04.02.2022, Initial cost estimates and product types for first pass mine options analysis

		Inputs		
Au price	$/tr oz	1,650		
Mass	g/tr oz	31.1035		
Fx	$/Euro	1.13		
	$/g	53.0		
	Annual		*Au rem (tr oz)*	*Au lost (tr oz)*
Plant feed (ore)	Mt	3.50		
Head grade	g/t	0.37		
Au in feed	kg	1,280	1,280	
	tr oz	41,154	41,154	
	M$	67.90		
Product type	-	Gravity Basic		
Recovery	-	65.0%		
Mass pull (dry)	-	0.80000%		
Concentrate grade	-	LG		
Concentrate production	t (dry)	28,000.00		
Au recovered (in conc)	kg	832		
	tr oz	26,750	26,750	14,404
Concentrate grade	g/t conc	29.7		
Value recovered in conc	M$	44.14		
Unit value of concentrate	$/t	1,576		
Treatment Charge (TC)	$/dmt	225.00		
	M$	6.30	22,932	3,818
Au - Payable	%	97.50%	22,359	573
	kg	811		
	M$	43		
Refinery Charge (RC) - Au	$/kg p'able	1,220.0		
	M$	0.99	21,759	600
Conc transport and realisation	$/dmt	0.0938		
	M$	0.003	21,757	2
TRC	M$	7.3	21,757	19,397
NSR	M$	35.74		
			41,154	OK
OPEX	$/t ore	5.37		
	M$	18.80		
NSR - OPEX	M$	16.94		
NSR - OPEX	$/t	13.84		
NSR Including OPEX	$/t	19.21		
Total Ore Cost	$/t	10.21		
Index	Row	6		
Index	Column	2		

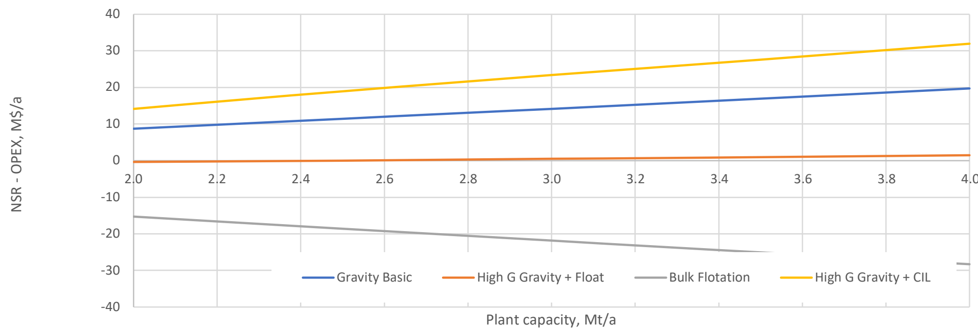
COG Calculations

Change Grade to set to zero		-

	NSRT1	NSRT2	NSRT3	NSRT4		
Column1	**Gravity Basic**	**High G Gravity + Float**	**Bulk Flotation**	**High G Gravity + CIL**	**Column2**	**Column3**
Flowsheet	Cones, spirals, tables	Falcon + float	No gravity	CIL Gravity tails		
Recovery	65.0%	95.0%	93.0%	95.0%		
Mass pull (dry)	0.8%	5%	8%	0.00025%		
Concentrate grade	LG	LG	LG	Dore		
2.0	5.85	6.50	6.83	10.50		4.65
2.5	5.63	6.29	6.62	10.00		4.37
3.0	5.49	6.14	6.47	9.80		4.31
3.5	5.37	6.03	6.35	9.70		4.33
4.0	5.28	5.94	6.26	9.60		4.32
	90%	91%	92%	91%		



NSR - OPEX (M$/a)



Commercial Terms	**Units**	**LG**	**HG**	**Dore**	**JdT comments**
Treatment Charge (TC)	USD/dmt	225	225	80,377	$2.5/oz for doré
Au - Payable	%	97.5%	97.5%	99.9%	
Refinery Charge (RC) - Au	USD/kg payable	1,220	1,220	1,220	
Freight	USD/dmt	0.027	0.027	-	
Insurance	USD/dmt	0.007	0.007	-	
Assay	USD/dmt	0.018	0.018	-	
Representative	USD/dmt	0.040	0.040	-	
Weight Loss	USD/dmt	0.003	0.003	-	
Conc transport and realisation	USD/dmt	0.094	0.094	80,377	$2/oz for doré

	LG	HG	Dore
Euro/dmt	0.024	0.024	0.024
Euro/dmt	0.006	0.006	0.006
Euro/dmt	0.016	0.016	0.016
Euro/dmt	0.035	0.035	0.035
Euro/dmt	0.003	0.003	0.003
Euro/dmt	0.083	0.083	0.083

	Process Route	**1**	**2**	**3**	**4**
	Process Code	NSRT1	NSRT2	NSRT3	NSRT4
G&A Etc	**Column1**	**Gravity Basic**	**High G Gravity + Float**	**Bulk Flotation**	**High G Gravity + CIL**
4.98	2.0	10.83	11.48	11.81	15.48
4.94	2.5	10.57	11.23	11.56	14.94
4.90	3.0	10.39	11.04	11.37	14.70
4.84	3.5	10.21	10.87	11.19	14.54
4.83	4.0	10.11	10.77	11.09	14.43

UG G&A

2.0	2.8	2.8	2.8	2.8
3.0	2.8	2.8	2.8	2.8
4.0	2.8	2.8	2.8	2.8

OP Cut-off Grade (g/t Au)

2.0	0.38	0.47	0.63	0.32
2.5	0.38	0.47	0.63	0.31
3.0	0.37	0.46	0.63	0.31
3.5	0.37	0.46	0.62	0.30
4.0	0.36	0.46	0.62	0.30

APPENDIX B

Pahtavaara Mining Cut-off Grades

04.02.2022, Initial cost estimates and product types for first pass mine options analysis

95%
30%

oz per year	
2.5	2.5
90%	90%
500,000.00	500,000.00
36,168.98	

		Inputs		
Au price	$/tr oz	1,650		
Mass	g/tr oz	31.104		
Fx	$/Euro	1.13		
	Annual		*Au rem (tr oz)*	*Au lost (tr oz)*
Plant feed (ore)	Mt	0.50		
Head grade	g/t	1.50		
Au in feed	kg	750		
	tr oz	24,113	24,113	
	M$	39.79		
Product type	-	High G Gravity + CIL		
Recovery	-	98.0%		
Mass pull (dry)	-	0.00015%		
Concentrate grade	-	Dore		
Concentrate production	t (dry)	0.75		
Au recovered (in conc)	kg	735		
	tr oz	23,630	23,630	482
Concentrate grade	g/t conc	980,000.0		
Value recovered in conc	M$	38.99		
Unit value of concentrate	$/t	51,986,883		
Treatment Charge (TC)	$/dmt	80,376		
	M$	0.06	23,594	37
Au - Payable	%	99.9%	23,570	24
	kg	734		
	M$	39		
Refinery Charge (RC) - Au	$/kg p'able	1,220		
	M$	0.90	23,027	543
Conc transport and realisation	$/dmt	80,375.514		
	M$	0.060	22,991	37
TRC	M$	1.0	22,991	1,122
NSR	M$	37.93		
			24,113	OK
OPEX	$/t ore	15.00		
	M$	7.50		
NSR - Process OPEX	M$	30.43		
NSR - ProcessOPEX	$/t	60.9		
NSR Including Process OPEX	$/t	75.87		
OPEX m	*-*	*(4.80)*		
OPEX c	*$/t*	*17.40*		
	Process Route	4		
Total Ore Cost	$/t	21.10		

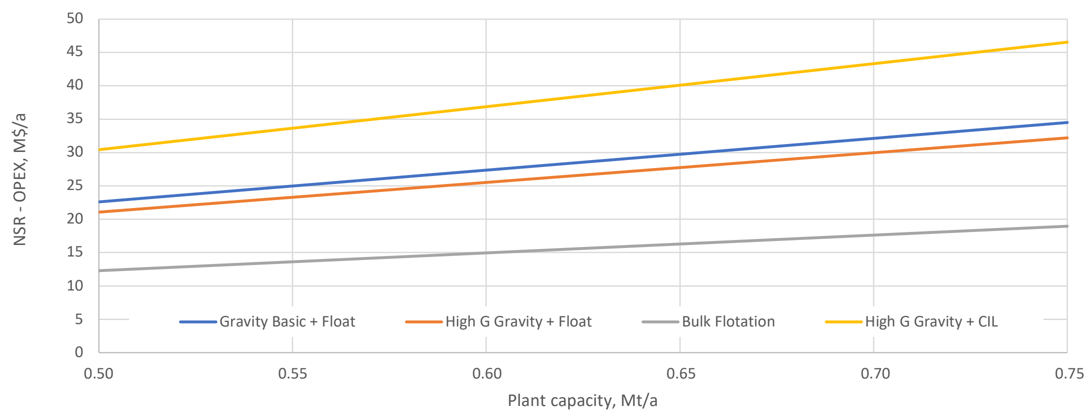
COG Calculations

Change Head Grade to set to zero	27.38

	NSRT1	NSRT2	NSRT3	NSRT4
Column1	**Gravity Basic + Float**	**High G Gravity + Float**	**Bulk Flotation**	**High G Gravity + CIL**
Flowsheet	Cones, spirals, tables, Float	Falcon + float	No gravity	CIL Gravity tails
Recovery	90.0%	80.0%	83.0%	98.0%
Mass pull (dry)	6.0%	4%	13%	0.00015%
Concentrate grade	LG	LG	LG	Dore
0.50	9.50	9.50	9.10	15.00
0.75	8.74	8.74	8.37	13.80
c	11.02	11.02	10.56	17.40
m	-3.04	-3.04	-2.91	-4.80

	kg	grade	
gravity	652	1000	652000
float	2553.4	27	68941.8
combined	3205.4	224.9147688	720941.8
	6%		

NSR - OPEX (M$/a)



Commercial Terms	Units	LG	HG	Dore	JdT comments
Treatment Charge (TC)	USD/dmt	225	225	80,376	$2.5/oz for doré
Au - Payable	%	97.5%	97.5%	99.9%	
Refinery Charge (RC) - Au	USD/kg payable	1,220	1,220	1,220	
Freight	USD/dmt	0.027	0.027	-	
Insurance	USD/dmt	0.007	0.007	-	
Assay	USD/dmt	0.018	0.018	-	
Representative	USD/dmt	0.040	0.040	-	
Weight Loss	USD/dmt	0.003	0.003	-	
Conc transport and realisa	USD/dmt	0.094	0.094	80,376	$2/oz for doré

		LG	HG	Dore
	Euro/dmt	0.024	0.024	0.024
	Euro/dmt	0.006	0.006	0.006
	Euro/dmt	0.016	0.016	0.016
	Euro/dmt	0.035	0.035	0.035
	Euro/dmt	0.003	0.003	0.003
	Euro/dmt	0.083	0.083	0.083

	NSRT1 Opex	NSRT2 Opex	NSRT3 Opex	NSRT4 Opex
Plant Rate	**Gravity Basic + Float**	**High G Gravity + Float**	**Bulk Flotation**	**High G Gravity + CIL**
0.50				15.00

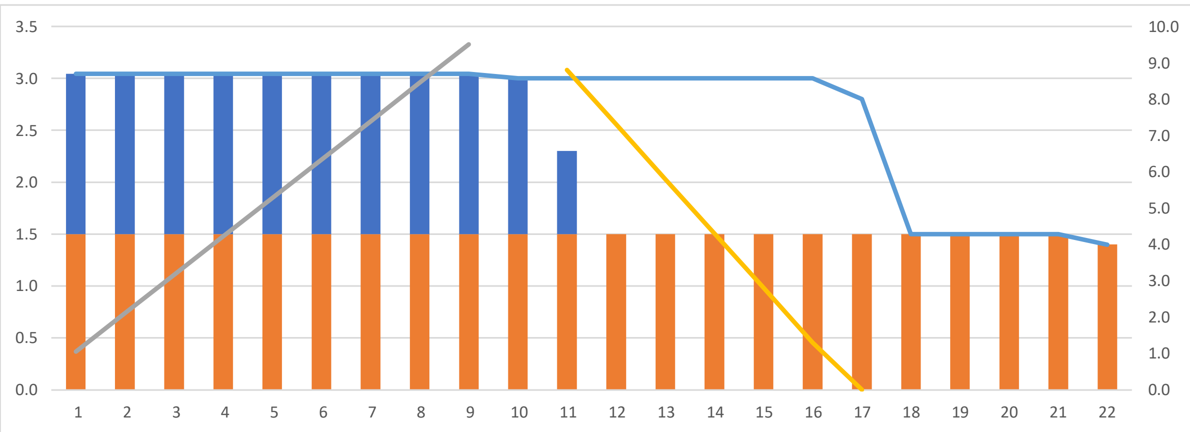
Process Route	**1**	**2**	**3**	**4**
Add Ore Cost OP	6.1	6.1	6.1	6.1
Add Ore Cost UG	3.9	3.9	3.9	3.9

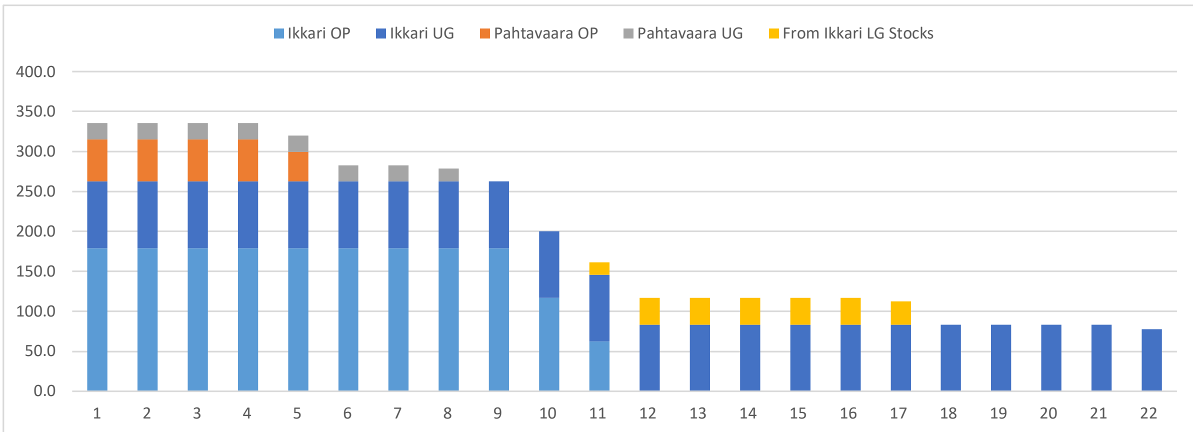
	Gravity Basic + Float	**High G Gravity + Float**	**Bulk Flotation**	**High G Gravity + CIL**
NPVS Total Ore Cost OP	0.00	0.00	0.00	21.10
NPVS Total Ore Cost OP	15.60	15.60	15.20	21.10
OP Cut-off Grade (g/t Au)	**0.64**	**0.61**	**1.06**	**0.42**

APPENDIX C

MINING SCHEDULE

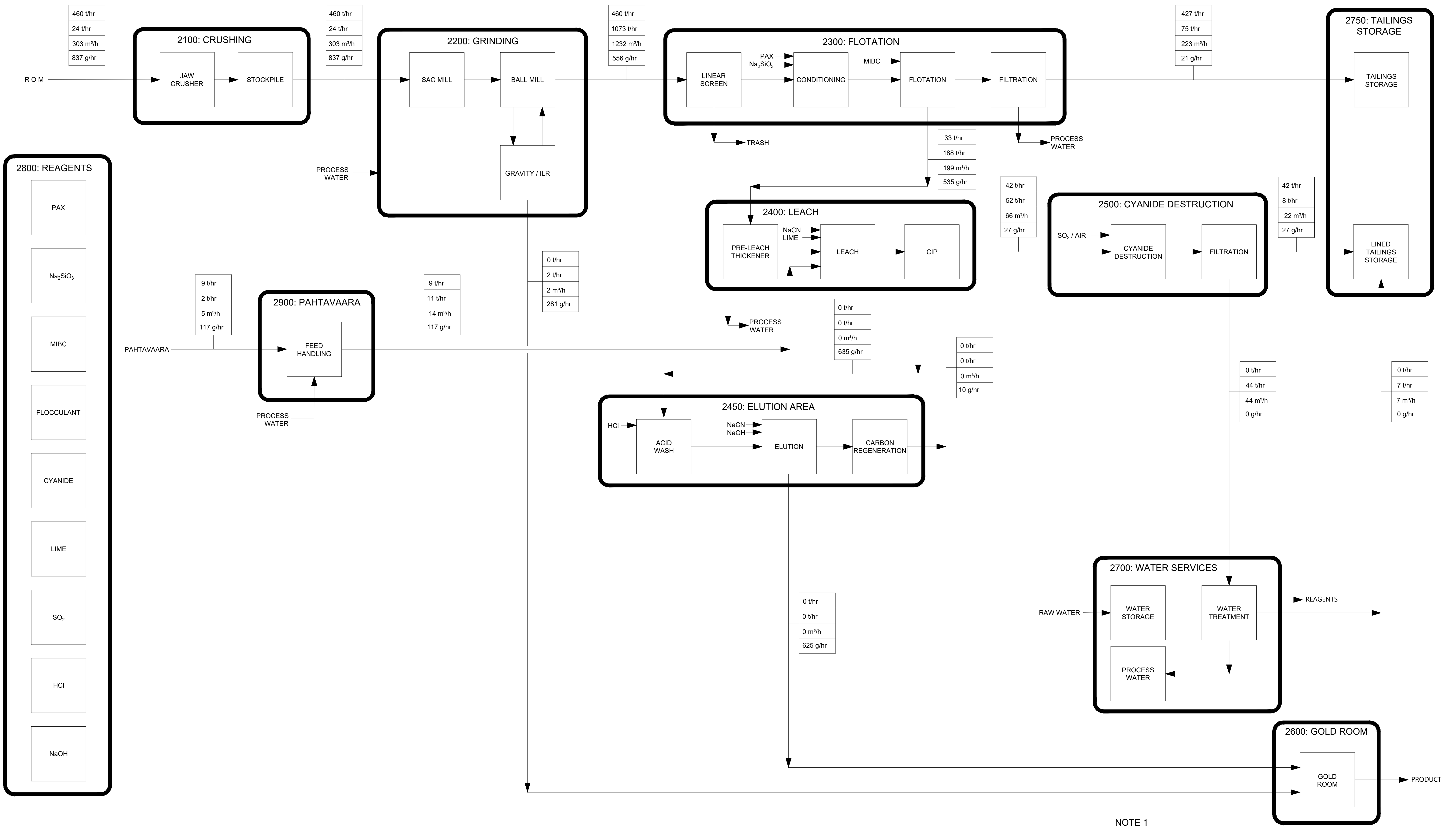
					Period																					
	Process	**Inventory (Mt)**	**Metal (Kg Au)**	**Grade (g/t Au)**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	22
Ikkari OP	High G + CIP	**25.7**	**62,339**	**2.43**																						
Rate	Total Mined				2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	1.5	0.8											
	Koz Contained				202.8	202.8	202.8	202.8	202.8	202.8	202.8	202.8	202.8	117.0	62.4	0.0	0.0									
	Remaining			25.696	23.1	20.5	17.9	15.3	12.7	10.1	7.5	4.9	2.3	0.8	0.0	0.0	0.0									
	To Plant	Mt			1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	0.8	0.0	0.0									
		Oz			179.0	179.0	179.0	179.0	179.0	179.0	179.0	179.0	179.0	117.0	62.4	0.0	0.0									
					Would you mine OP and UG together? Would it not take longer to develop UG?																					
Ikkari UG	High G + CIP	32.9		1.73																						
Rate					1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.4
	Koz Contained				83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	83.4	77.9
	Remaining			32.9	31.4	29.9	28.4	26.9	25.4	23.9	22.4	20.9	19.4	17.9	16.4	14.9	13.4	11.9	10.4	8.9	7.4	5.9	4.4	2.9	1.4	0.0
Pahtavaara OP	Float + High G + CI	4.7		1.64																						
Rate					1.0	1.0	1.0	1.0	0.7																	
	Koz Contained				52.7	52.7	52.7	52.7	36.9	0.0	0.0	0.0														
	Remaining			4.725	3.7	2.7	1.7	0.7																		
Pahtavaara UG	Float + High G + CI	3.9		1.26																						
Rate					0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.4														
	Koz Contained				20.3	20.3	20.3	20.3	20.3	20.3	20.3	16.2														
	Remaining			3.9	3.4	2.9	2.4	1.9	1.4	0.9	0.4	0.0														
To Ikkari LG Stocks	High G + CIP	9.5	6,653	0.7	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1													
	Koz Contained		213.9		23.8	23.8	23.8	23.8	23.8	23.8	23.8	23.8	23.8													
	Total				1.1	2.1	3.2	4.2	5.3	6.3	7.4	8.4	9.5													
From Ikkari LG Stocks	High G + CIP	9.5	6,653	0.7											0.7	1.5	1.5	1.5	1.5	1.5	1.3					
	Koz Contained		213.8												15.8	33.8	33.8	33.8	33.8	33.8	29.3					
	Remaining														8.8	7.3	5.8	4.3	2.8	1.3	0.0					
Ikkari Plant Feed	Mt				3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	2.8	1.5	1.5	1.5	1.5	1.4
	Oz Contained		Added Paht =>		335.4	335.4	335.4	335.4	319.6	282.7	282.7	278.7	262.5	200.4	161.6	117.2	117.2	117.2	117.2	117.2	112.7	83.4	83.4	83.4	83.4	77.9
92%	Oz Recovered				308.6	308.6	308.6	308.6	294.1	260.1	260.1	256.4	241.5	184.4	148.7	107.8	107.8	107.8	107.8	107.8	103.7	76.8	76.8	76.8	76.8	
Ikkari Inventory		35.2	68,992	1.96																						
High Grading Factor		0.03																								





APPENDIX D

BLOCK FLOW DIAGRAM



NOTE 1

LEGEND:

SOLIDS	t/hr
LIQUID	t/hr
VOLUME	m³/h
GOLD	g/hr

NOTES:

1. THE SUMMARY INFORMATION OF SOLIDS AND LIQUID MASS AND VOLUME QUOTED IN THIS BLOCK FLOW DIAGRAM ARE PROVIDED FOR THE SIMPLICITY OF UNDERSTANDING AND NOT REPRESENT THE PROCESS MASS BALANCE.

CHECK PRINT - DRAFTING

DISCIPLINE	CHECKER	DATE	INITIATOR ....
NAME (PRINT)			INITIATION DATE ....
SIGNATURE			THIS REVIEW DRAWING WILL BE AVAILABLE FOR A PERIOD OF 3 DAYS AFTER THE INITIATION DATE, AFTER WHICH TIME NON SIGNATORIES' TACIT APPROVAL WILL BE ASSUMED.

DWG. NO.	REFERENCE DRAWINGS	REV	DATE	REVISION DESCRIPTION	PREPARED	CHK'D	DESIGN	APPROVED
-	-	A	22JUN2022	ISSUED FOR COMMENT	TJ	-	JV	-

PRELIMINARY DRAWING

NOT TO BE USED FOR CONSTRUCTION

DESCRIPTION	INITIALS	DATE
DRAWN	TJ	22JUN2022
CHECKED	JV	22JUN2022
DESIGNED	-	-
APPROVED	-	-
CLIENT	-	-

RUPERT RESOURCES

TETRA TECH

coffey A TETRA TECH COMPANY

CLIENT DOCUMENT NUMBER

IKKARI GOLD PROJECT
PRELIMINARY ECONOMIC ANALYSIS
BLOCK FLOW DIAGRAM

SCALE	SIZE	DRAWING NUMBER	REV
NTS	A3	03627AA-0000-BFD-Z-001	Ab

APPENDIX E

Process Design Criteria

DESIGN CONSULTANT		CLIENT
TETRA TECH		RUPERT RESOURCES

Ikkari & Pahtavaara PEA

Process Design Criteria

DOCUMENT NO: 03627AA-0000-DSC-Z-001

APPROVALS				
	Name	Position	Signed	Date
Originator	James Vardy	Senior Process Engineer	JV	14/10/2022
Checked				
Approved				

Rev	Date	Description	By	Chk	Appd
A	14/10/2022	Issued for Preliminary Economic Assesment	JV		

Coffey Geotechnics Ltd, A Tetra Tech Company
Registered Address: 1 Northfield Road, Reading, RG1 8AH, UK
Registered in England: 06328315

1. Introduction

The process design criteria forms the basis for the design of the processing facilities and required site services. Together with the Process Flow Diagrams, this data is the basis for definition of the mass balance, process description and equipment specifications.

The design criteria have been based on data from various sources and all data is referenced to the sources. The test work carried out has been done using the Life of Mine composite sample that was selected to represent the ore body over the life of the mine. It is of particular importance to note areas in which assumptions have been made that require verification.

The design criteria and the associated mass balances are used to derive capital cost estimates and schedules for operating requirements such as power, reagents and consumables, etc. Any recovery or similar data presented herein are used for the purpose of this study only and are not statements of predicted plant performance.

2. References

The process design criteria are derived from a variety of sources. All data is referenced to a source to ensure that the basis of the information is fully understood. The referencing of data in this document is as follows:

DOCUMENT NO: 03627AA-0000-DSC-Z-001	
Source of Information	**Reference number**
Client advice/correspondence	1
Agreement with client	2
Test work data	3
Recommendation	4
Experience/database	5
Calculated data	6
Assumed or estimated data	7
Axe Valley Mining Schedule	8

Design Basis

Area / Item Number	Item Description	Units	Value/Comment	Comment	Ref	Rev
2000	**Operating Time**					
	Days per Annum	days/a	365		5	A
	Statutory Holidays	days/a	0		7	A
	Operating Days Available	days/a	365		6	A
JMENT NO: 03	Operating Hours per Day – crusher	h/day	24		4	A
	Operating Hours per Day - mill	h/day	24		4	A
	Operating Hours per annum – crusher	h/a	7,720		6	A
	Operating Hours per annum - mill	h/a	7,720		6	A
	Throughput (Dry Basis)					
	Life of Ikkari Plant	t/a ROM & concentrate	24	Jan 2028 - Jan 2052	8	A
	Pahtavaara					
	Plant Throughput	t/a ROM	500,000 750,000	Up to Dec 2045 Jan 2046 increases	1	A
	Life of Mine	years	13	Jan 2039 - Jan 2052	1	A
	Feed grade	g Au/ t	1.56		8	A
	Ikkari					
	Plant Throughput	t/a ROM	3,500,000		8	A
	Life of Mine	years	21	Jan 2028 - Oct 2048	1	A
	Feed grade	g Au/ t	1.82	Max 2.93	8	A
	Overall Plant Recovery					
	Pahtavaara		**91%**			
	Gravity Recovery	%	31%		3	A
	Gravity Mass Pull	%	0.2%		3,5	A
	Flotation Recovery	%	87%		3	A
	Flotation Mass Pull	%	10%			
	Leach Recovery	%	99.6%		3	A
	Ikkari		**94%**			
	Gravity Recovery	%	34%		3,5	A
	Flotation Recovery	%	96%		3	A
	Leach Recovery	%	95%		3	A

Site DATA:
Insert Site specific info like elevation, Wind direction (wind rose), rainfall, dry bulb temp, etc

Site Conditions

Criteria	Units	Value	Source / Responsibility
Site location description		N 67° 34,896' E 25° 54,991'	Client
Altitude	m ASL	224.50 mASL	Client
Barometric pressure	kPa	100.98 kPa	Client
Temperatures			
Average Summer	°C	14	Client
Average Winter	°C	-11.1	Client
Relative humidity	%	81%	Client
Wet bulb	°C		Client
Dry bulb	°C		
Rainfall per annum	mm	516	Client
Prevailing wind direction		191.8 deg, SSW	Client
Wind speed Max	m/s	19.7	Client
Average	m/s	2.7	Client
Min	m/s	0	Client
Design	m/s	23	

Design Criteria

Area / Item Number	Item Description	Units	Value/Comment	Comment	Ref	Rev
2100	**Crushing**					
	Ore Moisture	%	5		7	A
	Solids Density	t/m^3	2.9		3	A
	Bulk SG	t/m^3	1.6		7	A
DOCUMENT NO:	Crushing Work Index	kWh/t	12		7	A
	Head grade					
	Average	g Au/t	1.82		8	A
	Max (design) of life of Mine	g Au/t	2.93		8	A
	ROM Stockpile					
	Maximum Single Lump Size	mm	600		7	A
	Feed Method	type	Direct Tip		4	A
	Primary Grizzly Undersize material	% w/w	20%		7	A
	Crusher					
	Feed Rate	t/h	460		6	A
	Number of Crushing Stages	N^o.	1		7	A
	Crusher Type	type	Jaw		4	A
	Closed side setting	mm	150			
	Feed – Coarse					
	F_{100}	mm	600		7	A
	F_{80}	mm	450		7	A
	Product					
	P_{100}	mm	190		7	A
	P_{80}	mm	150		7	A
	P_{50}					
	Stockpile					
	Maximum Single Lump Size	mm	190		6, 7	A
	Live Capacity	h	24		7	A
		m^3	6,806		6	A
		t	10,890		6	A
	Feed Method	type	Apron		4	A
	Capacity	t/h	35		4	A
2200	**Grinding**					
	Milling Circuit		**SAB**	**SAG and Ball Mill**		
	Circuit New Feed Rate	t/hr	460		6	A
	Bond Work Index	kWh/t	15.5		3	A
	Rod Work Index	kWh/t	11.8		3	A
	Abrasion Index		0.35		3	A
	SAG Mill					
	Mill Type	type	**SAG**		5	A
	Dimensions					
	Diameter	ft	30.0		6	A
	Effective Grinding Length (EGL)	ft	15		6	A
	Installed Power	kW	5500		6	A
	Mill Discharge Density	% w/w	70%		7	A
	Ball Mill					
	Mill Type	type	**Ball**		5	A
	Dimensions					
	Diameter	ft	21.0		6	A
	Effective Grinding Length (EGL)	ft	29		6	A
	Installed Power	kW	5500		6	A
	Circulating Load	%	150 - 300		5	A
	Design Circulating Load	%	300%		7	A
	Trommel Aperture	mm	8		7	A
	Mill Discharge Density	% w/w	70%		7	A
	Mill Discharge Hopper Capacity	s	120		7	A
	Cyclone cluster					
	Inlet Pressure	kPa	120		7	A
	Cyclone O/F Classification	P_{80} (µm)	175		3	A
	Cyclone O/F Weight % Solids	w/w %	25%		7	A
	Cyclone U/F Weight % Solids	w/w %	65%		7	A
	Gravity Circuit					
	Gravity Concentration					
	Cyclone Underflow Split to Gravity Circuit	%	33%		7	A
	Gravity Concentrator	Type	Knelson/Falcon		7	A
	Mass Pull					
	Lab Mass pull	%	1.6%		3	A
	Scale up factor		10		7	A
	Design Mass pull	%	0.16%		6	A
	Gold recovery					
	Lab Recovery	%	42%		3	A
	Scale up factor	%	80%		7	A
	Design Recovery	%	34%		6	A
	Intensitive Leach					
	Feed Solids Concentration	%	30%		7	A
	Extraction	%	100%		7	A

2300	**Flotation**					
	Conditioning					
	Reagent Addition					
	PAX (Potassium amyl xanthate) - Collector	g/t	20			
	MIBC (Methyl isobutyl carbinol) - Frother	g/t	5			
	Na_2SiO_3 (Sodium Silicate) - Depressant	g/t	500			
	Reagent Contact time					
	PAX (Potassium amyl xanthate) - Collector	min	2		3	A
	Na_2SiO_3 (Sodium Silicate) - Depressant	min	4.5		3	A
	MIBC (Methyl isobutyl carbinol) - Frother	min	1		3	A
	Rougher					
	Residence Time					
	Lab Rougher	min	4.5		3	A
	Scale up factor		3		5	A
	Design Residence Time		13.5		7	A
	Scavenger					
	Reagent Addition					
	PAX (Potassium amyl xanthate) - Collector	g/t	12		3	A
	MIBC (Methyl isobutyl carbinol) - Frother	g/t	5		3	A
	Residence Time					
	Lab Savenger	min	4.5		3	A
	Scale up factor		3		5	A
	Scavenger	min	13.5		6	A
	Mass Pull	%	7.21%		3	A
	Recovery	%	96%		3	A
	Flotation Concentrate % Solids	% w/w	15%		3	A
	Flotation Tails Filtration					
	Filtration	Type	Belt		7	A
	Flux	$t/m^2.h$	1.00		7	A
	Number of Filters	#	7	6 duty & 1 standby	5,6	A
	Moisture Content	%	15%			
2400	**Leaching**					
	Flotation Concentrate Thickener					
	Thickener Type	type	High rate		7	A
	Number of Thickeners	N^o.	1		7	A
	Feedwell Density	w/w % solids	15%		5	A
	Area Settling Rate	$t/m^2/day$	4.00		7	A
	Thickener Diameter	m	18		6	A
	Underflow Density	w/w % solids	45%		5	A
	Leach					
	Extraction					
	Pahtavaara concentrate	%	99.6%		3	A
	Ikkary Flotation concentrate	%	95.0%		3	A
	Overall Extraction	%	95.7%		6	A
	Reagent Addition					
	NaCN	g/l Soln	2		3	A
	$Ca(OH)_2$	kg/t	0.95		3	A
	Retention time	hrs	48		3	A
2450	**Elution / ADR**					
	Carbon Loading	g Au/ t Carbon	2000		7	A
	Barrent Carbon Loading	g Au/ t Carbon	30		7	A
	Carbon Moisture	% w/w	5%		7	A
	Carbon Elution Mass	t	7.62		6	A
	Design Elution Mass	t	14.00	Based on higher head grade	6	A
	Number of Elutions per day		1		7	A
	Elution Reagents					
	HCL	%	3%		5, 7	A
	NaOH	%	1%		5, 7	A
	NaCN	%	1%		5, 7	A
	Carbon Density	t/m^3	2.10		5, 7	A
	Carbon Bed Bulk Density	t/m^3	0.45		5, 7	A
	Carbon bed volume	m^3	31.11		6	A
	Elution time	hrs	20		5, 7	A
	Carbon Voidage		0.5		5, 7	A
2500	**Cyanide Destruction**					
	CIP Tails WAD concentration	ppm	700		3	A
	Reagent Addition					
	SO_2 Addition	g SO_2/g CN(WAD)	6.5		3	A
	Ca(OH)2	kg/t	2.9		5	A
	Cyanide Destruction Tails Filtration					
	Filtration	Type	Belt		7	A
	Flux	$t/m^2.h$	1.00		7	A
	Number of Filters	#	4	3 duty & 1 standby	5,6	A
	Moisture Content	%	15%		7	A

APPENDIX F

PROJECT DESIGN CRITERIA

Report to:

Rupert Resources Ltd

Ikkari Preliminary Economic Analysis

Project Design Criteria

Document No. 03627AA-0000-DSC-X-001

Report to:

RUPERT RESOURCES LTD

IKKARI PRELIMINARY ECONOMIC ANALYSIS

PROJECT DESIGN CRITERIA

EFFECTIVE DATE: 1 OCTOBER,2022

Prepared by	James Seccombe	Date	
Reviewed by	James Vardy	Date	
Authorized by	Andrew Carter	Date	

JdT/lt

TETRA TECH

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REVISION HISTORY

REV. NO	ISSUE DATE	PREPARED BY AND DATE	REVIEWED BY AND DATE	APPROVED BY AND DATE	DESCRIPTION OF REVISION
A		JHPS	JV	AJC	Initial Draft

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

GLOSSARY

ABBREVIATIONS AND ACRONYMS

Tetra Tech Mining and Minerals .... Tetra Tech
Feasibility Study .... FS
issued for construction .... IFC
Direct on Line .... DOL
Variable Speed Drive .... VSD
Polychlorinated biphenyl's .... PCB
Chloro-fluoro-hydrocarbons .... CFC
International Fire Code .... IFC
Safe Working Load .... SWL
Standard Operating Procedure .... SOP
Life of mine .... LOM
Material Test Certificate .... MWE
Non-destructive Testing .... NDT
Pressure Equipment Directive .... PED
Conveyor Equipment Manufacturer's Association .... CEMA
World Health Organisation .... WHO
Totally enclosed fan cooled .... TEFC
Ingress Protection .... IP
Plant control system .... PCS
Uninterrupted power supply .... UPS
Nominal diameter .... DN

1.0 INTRODUCTION

Rupert Resources Ltd (RRL) is currently developing the Ikkari Deposit and the Pahtavarra mine, located in the Northern Finland.

The Ikkari deposit lies at the eastern extreme of the Sirkka Line, a tectonic structure that traverses northern Finland, along which some 25 to 30 gold deposits exist. The gold deposit is situated in a fairly dry, sparsely forested area. The landscape is reasonably flat with an elevation of approximately 240 to 250 metres (m) above sea level.

The Pahtavaara hill, located directly to the northeast, has an elevation of approximately 325 m above sea level. The overburden cover is generally between 5 to 10 m thick. In most parts of the deposit area, the ground water table is typically located a few metres below the ground surface.

Tetra Tech Mining and Minerals (Tetra Tech) has been contracted to complete a Preliminary Economic Analysis (PEA) for the Project. The PEA will define the elements of the project from design, construction, start-up, and operations through final closure and reclamation with a capital and operating cost accuracy to AACE® International Class 4.

This document contains the design criteria that forms the basis of the engineering design of the project as it relates to Tetra Tech's scope of work. This is a live document that will be updated continuously throughout the PEA. Formal revisions will be published from time to time at key design stages or at the discretion of the Project Engineer and Project Manager.

2.0 PROJECT DESCRIPTION

The Ikkari deposit occurs within rocks that have been regionally mapped as 2.05-2.15 Ga old Savukoski group greenschist-metamorphosed mafic-ultramafic volcanic rocks, part of the Central Lapland Greenstone Belt (CLGB). Gold mineralisation is largely confined to the structurally modified unconformity between Savukoski and Kumpu groups strata. The two units are complexly interleaved, the result of early low angle thrusting and folding and subsequent upright folding and shearing.

Pahtavaara gold mine is hosted by the predominantly pyroclastic, voluminous ultramafic volcanics of the Sattasvaara komatiite complex, part of the 2.05Ga Savukoski group (Mutanen, 1997). Least altered komatiites consist of a talc-chlorite assemblage resulting from regional greenschist facies metamorphism. Extrusive ultramafics, including pillow lavas and hyaloclastites are intercalated with minor sedimentary lenses, comprising predominantly carbonaceous shales and coarse, crystalline actinolite-tremolite assemblages believed to be the alteration product of a more pyroxenitic volcanic phase.

Figure 2.1 Project Location Map

3.0 SITE DESCRIPTION

3.1 Location

Ikkari is located near Rajala village in the municipality of Sodankylä approximately 40km west of Sodankylä town in Northern Finland.

3.2 Access to Site

The airport of Rovaniemi has several scheduled domestic flights daily to and from Helsinki. The distance from Rovaniemi to Sodankylä is 130km by road and takes just under two hours to drive. To reach Ikkari from Sodankylä, turn towards Kittilä onto main road 80. Continue to follow road 80 towards Kittilä, 4.5km after Jeesiö village turn right to Pulkittama. Continue to follow Pulkittama road for 7.5km where forest tracks lead directly to the exploration site.

Access to the site is possible throughout the year.

3.3 Climatic Conditions

The nearest wather station is based in Sodankyla, sum 30 kms from Site. The Köppen Climate Classification subtype for this climate is "DFC" (Continental, without a dry season and a cold summer).

3.3.1 Design Temperatures

The average temperature for the year in Sodankyla is 1°C. The warmest month, on average, is July with an average temperature of 14°C. The coolest month on average is February, with an average temperature of -13°C.

Table 3.1 Monthly Temperature (°C)

	Month											
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Average	-13	-13	-9	-3	4	10	14	11	5	-1	-6	-10
Minimum	-18	-19	-16	-8	0	5	8	6	1	-4	-10	-15
Maximum	-8	-8	-3	2	9	16	20	17	10	2	-2	-6

Source: www.weatherbase.com

3.3.2 Precipitation

The average amount of precipitation for the year in Sodankyla is 470 mm. The month with the most precipitation on average is July and August with 60 mm of precipitation.

Table 3.2 Monthly Precipitation (mm)

	Month											
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Average	20	20	20	20	30	50	60	60	30	40	30	20
Annual	470											

Source: www.weatherbase.com

3.3.3 Humidity

The recorded monthly average for the region ranges between 67% in June to 91% November.

Table 3.3 Average Relative Humidity (%)

	Month											
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Average	87	86	78	73	68	67	71	80	84	90	91	89

Source: www.weatherbase.com

3.3.4 Wind

The wind is most often from the South by Southwest for the entire year. The annual average wind speed is 11 kph (3.1 m/s), with the highest speed recorded in March and the lowest recorded in January.

Table 3.4 Average Monthly Speed (kph)

	Month											
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Direction	SSW	SSW	WSW	WNW	WSW	W	S	SSW	SSW	S	SSW	S
Average	9	9	11	11	13	11	11	11	11	9	9	9
Maximum	19	20	28	26	26	26	24	24	22	20	20	19

Source: www.weatherbase.com

4.0 AREA BREAKDOWN STRUCTURE

Table 4.1 Area Breakdown Structure

Area Number	Area Description
0000	Project Management
1000	Mining
2000	Processing
2100	Crushing
2200	Grinding
2250	Gravity Separation
2300	Flotation
2400	Leaching
2450	Elution
2500	Cyanide Destruction
2600	Gold Room
2700	Water Services
2750	Tailings Handling
2800	Reagents
2900	Pahtavaara Feed
2950	Plant Utilities
3000	Infrastructure and Services
4000	Tailings General
5000	Blank
6000	Blank
7000	Blank
8000	Offsite Infrastructure
9000	Construction

5.0 CODES AND STANDARDS

The latest editions of codes and standards of the organisations shown in Table 8.1 are applicable.

Table 5.1 Codes and Standards

Institution	Description
ABMA	American Bearing Manufacturer's Association
ACI	American Concrete Institute
AGMA	American Gear Manufacturer's Association
AISC	American Institute of Steel Construction
ANSI	American National Standards Institute
API	American Petroleum Institute
ASCE	American Society of Civil Engineers
ASME	American Society of Mechanical Engineers
ASTM	American Society for Testing and Materials
AWWA	American Water Works Association
BS EN	British Standards Institute (BSI)
CE	European Compliance Services
CEMA	Conveyor Equipment Manufacturer's Association
CEN	European Committee of Standardisation
CENELEC	European Committee of Electrotechnical Standards
FEBMA	Federation of European Bearing Manufactures Association
FEM	European Federation of Material Handling
GHS	Globally Harmonised System (Hazardous Labelling)
HI	Hydraulic Institute
ISO	International Organisation for Standardisation
IEC	International Electrotechnical Commission
IHEOH	Industrial Ventilation: A Manual of Recommended Practice published by Industrial Hygiene, Environmental, Occupational Health
IEEE	Institute of Electrical & Electronics Engineers
ICEA	Insulated Cable Engineers Association
IBC	International Building Code
IFC	International Fire Code
ISA	International Society of Automation
MSS	Manufactures Standardisation Society
NFSC	Natural Fire Safety Concept
PED	Pressure Equipment Directive
PFI	Pipe Fabrication Institute

Institution	Description
	Table continues…
PPI	Plastics Pipe Institute
SAE	Society of Automotive Engineers
SMACNA	Sheet Metal & Air Conditioning Contractors' National Association
SNT	Society of Non-Destructive Testing
SSPC	Steel Structure Painting Council
TEMA	Tubular Exchanger Manufacturers' Association
TEPPFA	The European Plastic Pipes and Fittings Association
UL/FM	Underwriters Laboratories / Factory Mutual

Unless specifically otherwise noted all equipment, devices and systems shall be designed, manufactured and tested in accordance with the latest applicable codes and standards as listed below.

Table 5.2 Applicable Codes and Standards

Institution	Description
CE	European Compliance Services
CEN	European Committee of Standardisation
CENELEC	European Committee of Electrotechnical Standards
IBC	International Building Code
IFC	International Fire Code
IEC	International Electrotechnical Commission
ISA	International Society of Automation
ISO	International Organisation for Standardisation

Note: Suppliers shall specify which of these standards their equipment shall be in compliance with. Failure to so stipulate will result in application of the first listed applicable codes and standards. Suppliers shall also nominate what certification marks their equipment will bear.

Referenced publications within this specification will be the latest revision, unless otherwise specified and applicable parts of the referenced publications will become a part of this specification as if fully included.

All requirements as stipulated by local, state or federal governments shall have jurisdiction.

5.1 HAZARDOUS MATERIALS AND ENVIRONMENTAL CONSIDERATIONS

The following materials shall not be used:

- Asbestos and compounds thereof.
- Polychlorinated biphenyl's (PCBs) and compounds thereof.

The following materials shall not be used without approval:

- Chloro-fluoro-hydrocarbons (CFCs) and compounds thereof.

The Supplier’s equipment, materials and products (including, but not limited to, products of a toxic, hazardous, flammable or corrosive nature) shall be identified by the required labelling at the place of origin or manufacture for transportation, handling and storage of such equipment, materials and products in accordance with all applicable legislation at that place.

The Supplier shall comply with all applicable laws, orders and regulations concerning the control and abatement of land, water and air pollution.

The Supplier's on-site activities shall be performed by methods that will prevent entrance or accidental spillage of solid, liquid or gaseous matter, contaminants, debris, and other objectionable pollutants and wastes on unprotected ground, into streams, water courses, lakes, underground water sources and the atmosphere. Such pollutants and wastes shall include but are not limited to refuse, garbage, cement, concrete, sewage effluent, industrial waste, radioactive substances, oil and other petroleum products, aggregate processing tailings, substances capable of producing toxic or otherwise objectionable leachate, mineral dust and thermal pollution. Sanitary wastes shall be disposed of at an approved landfill site or by other approved means.

5.2 ENVIRONMENTAL MANAGEMENT

Table 5.3 Environmental Considerations

Environmental aspect	Criteria
Stormwater Management	Diversion of all stormwater via ditches and berms
Sediment and Erosion Control	Minimal erosion via the control of surface water runoff
Surface Flows	Diversion of surface water via dikes, ditches and berms
Terrestrial Flora & Fauna Control	Minimal disruption to flora & fauna and a scheme to replace and regenerate after life of mine
Dust Emissions and Lime Spill Management	Control of dust via water spraying management and dust suppression measures
Acid & Reagent Control	Control via spill kits, bunds and containment areas
Hydrocarbon Emissions	
Scrubber Emissions	

5.3 UNITS

The project will be designed in the S.I. unit (metric) system. It is requested that Suppliers use these units in all specifications and drawings, whenever possible.

6.0 OPERATING PARAMETERS

6.1 OPERATING CONDITION

10,480 t/d for 334 dpa = circa 3.5 mtpa

Tonnages and volumes for equipment sizing have been derived by applying the availability and operating regime to the daily tonnage rate above.

6.2 AVAILABILITY

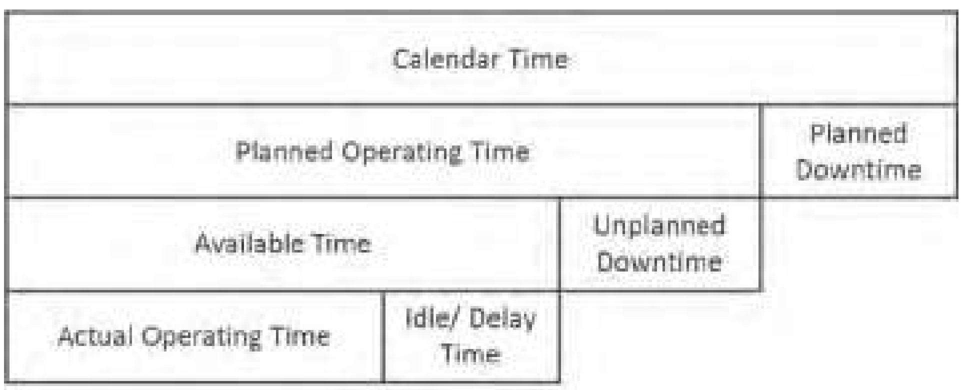
6.2.1 DEFINITION

Availability is defined as the probability that a system will be able to operate at design capacity and quality during the planned operating time. This excludes planned and unplanned downtime. Distinction is drawn between system availability and component availability as the former is the result of a specific configuration of components, each with its own availability.

A system capacity and quality target are set in narrow bands. Typically, a system is deemed available if it achieves 95% or more of its design capacity and the quality is within 90% of the specified parameter.

Availability is calculated by convention in accordance with Figure 5.1.

Figure 6.1 System Time Allocation



- Calendar time: Total number of hours in a calendar year (defined as 8,760 hours).
- Planned downtime: Including statutory holidays, planned maintenance and down shifts.
- Unplanned downtime: Where not mechanically (electrically) operable when it should be.
- Idle and delay time: Equipment operable but not producing e.g. no feed or short-term interruptions to production.

- Availability (%) = Available Time / Planned Operating Time.
- Utilisation (%) = Actual Operating Time / Available Time.

Four main systems are indicated for the Project:

- Comminution (Grinding, Crushing & Gravity)
- Flotation & Leach
- ADR & Gold Room
- Reagents, Water Treatment, Tailings & Utilities

6.2.2 COMMINUTION PLANT

The plant will operate 24 hours per day on a 2-shift system, 7 days per week and will not observe any national or local holidays. The plant will have monthly maintenance shutdowns, 10 inspection shutdowns for 12 hours, a minor maintenance shutdown for 7 days, and a major maintenance shutdown for 14 days.

Crushing plant design availability will be at least 88%.

Table 6.1 Comminution Plant Design Availability

Time Component	Frequency	Duration (Hours)	Annual Hours
Calendar Time			**8,760**
National Holidays	*0*	*24*	*0*
Major shutdown (2 Week)	*1*	*336*	*336*
Minor shutdown (1 week)	*1*	*168*	*168*
Inspection shutdown (1 day)	*10*	*12*	*120*
Planned Downtime			624
Planned Operation Time			**8,136**
Unplanned Downtime (5%)			416
Available Time			**7,720**
Idle Time (0%)			0
Operating Time			**7,720**

6.2.3 FLOTATION & LEACH PLANT AVAILABILITY

The plant will operate 24 hours per day on a 2-shift system, 7 days per week and will not observe any national or local holidays. The plant will have monthly maintenance shutdowns, 10 inspection shutdowns for 12 hours, a minor maintenance shutdown for 7 days, and a major maintenance shutdown for 14 days.

Flotation & Leach plant design availability will be 88%.

Table 6.2 Flotation & Leach Plant Design Availability

Time Component	Frequency	Duration (Hours)	Annual Hours
Calendar Time			**8,760**
National Holidays	*0*	*24*	*0*
Major shutdown (2 Week)	*1*	*336*	*336*

Minor shutdown (1 week)	*1*	*168*	*168*
Inspection shutdown (1 day)	*10*	*12*	*120*
Planned Downtime			624
Planned Operation Time			**8,136**
Unplanned Downtime (5%)			416
Available Time			**7,720**
Idle Time (0%)			0
Operating Time			**7,720**

6.2.4 ADR & Gold Room Plant Availability

The plant will operate 24 hours per day on a 2-shift system, 7 days per week and will not observe any national or local holidays. The plant will have monthly maintenance shutdowns, 10 inspection shutdowns for 12 hours, a minor maintenance shutdown for 7 days, and a major maintenance shutdown for 14 days.

The ADR & Gold Room plant design availability will be 88%.

Table 6.3 Crystallisation Plant Design Availability

Time Component	Frequency	Duration (Hours)	Annual Hours
Calendar Time			**8,760**
National Holidays	*0*	*24*	*0*
Major shutdown (2 Week)	*1*	*336*	*336*
Minor shutdown (1 week)	*1*	*168*	*168*
Inspection shutdown (1 day)	*10*	*12*	*120*
Planned Downtime			624
Planned Operation Time			**8,136**
Unplanned Downtime (5%)			416
Available Time			**7,720**
Idle Time (0%)			0
Operating Time			**7,720**

6.2.5 Reagent, Water Treatment, Tailings and Utilities Plant Availability

The plant will operate 24 hours per day on a 2-shift system, 7 days per week and will not observe any national or local holidays. The plant will have monthly maintenance shutdowns, 10 inspection shutdowns for 12 hours, a minor maintenance shutdown for 7 days, and a major maintenance shutdown for 14 days.

The reagent and utilities plant design availability will be 88%.

Table 6.4 Reagent, Water Treatment, Tailings & Utilities Plant Design Availability

Time Component	Frequency	Duration (Hours)	Annual Hours
Calendar Time			**8,760**
National Holidays	*0*	*24*	*0*
Major shutdown (2 Week)	*1*	*336*	*336*
Minor shutdown (1 week)	*1*	*168*	*168*
Inspection shutdown (1 day)	*10*	*12*	*120*

Planned Downtime			624
Planned Operation Time			**8,136**
Unplanned Downtime (5%)			416
Available Time			**7,720**
Idle Time (0%)			0
Operating Time			**7,720**

6.3 OPERATIONAL CAPACITY

Annual throughput and operating hours will be as shown in Table 6.5.

Table 6.5 Operational Parameters

Plant Area	Comminution Plant	Flotation & Leach Plant	ADR & Gold Room Plant	Reagent & Utilities Plant
Annual available operating hours	7,720	7,720	7,720	7,720
Type of operation	Continuous 24 h/d	Continuous 24 h/d	Continuous 24 h/d	Continuous 24 h/d
Nominal Capacity	453 t/h	453 t/h	14 t/h	Various
Annual Throughput	3.5 mtpa	3.5 mtpa	108 ktpa	Various

7.0 PROCESS PARAMETERS

Table 7.1 Process Parameters

Description	Unit	Value	Comment
Plant Operating Schedule			
Shifts / Day - Crushing	Shift	2	Tetra Tech assumption
Shifts / Day - Grinding	Shift	2	Tetra Tech assumption
Shifts / Day - Purification	Shift	2	Tetra Tech assumption
Shifts / Day – Solvent Extraction	Shift	2	Tetra Tech assumption
Shifts / Day – Reagents & Utilities	Shift	2	Tetra Tech assumption
Hours / Shift	h	12	Calculation
Hours / Day	h	24	-
Days / Year	d	365	-
Operational Days / Year	d	334	31 days planned shutdown
Throughput/Processing Rate			
Annual Processing Rate, Overall	t/a (dry)	3,500,000	Client Instruction
Daily Processing Rate - Mill	t/d	12,960	Calculation
Hourly Processing Rate - Mill	t/h	453	Calculation
Ore Characteristics			
Head Grade	g Au/t	1.84	Client
Overall Recovery	%	95	Client
Ore SG	-	2.9	Client
Ore Bulk Density	t/m^3	1.6	Tetra Tech assumption
Ore Moisture Content	%	5	Client
Bond ball Mill Index	kW/t	14.8	Tetra Tech assumption

8.0 STORAGE & RESIDENCE REQUIREMENTS

8.1 STORAGE

Storage Parameters for On-Site primary commodities are shown in Table 8.1.

Table 8.1 Storage Parameters for On-site Primary Commodities

Commodity	Chemical Symbol	Consumption	Storage	Storage Capacity
Lime	CaO	0.116 t/h	1 Month	85 t
Cyanide	NaCN	0.116 t/h	14 Days	40 t
Xanthate-Potassium Amyl Xanthate	PAX	0.014 t/h	1 Month	10 t
Sodium Silicate	Na_2SiO_3	0.226 t/h	14 Days	75 t
Methyl isobutyl carbinol	MIBC	0.004 t/h	1 Month	3 t
Hydrochloric Acid (32%)	HCl	0.116 m³/h	14 Days	40 m³
Caustic/Sodium Hydroxide	NaOH	0.013 t/h	1 Month	8 t
Sodium Meta-Bisulphite	SMBS	0.168 t/h	14 Days	60 t
Grinding media	-	0.635 t/h	14 Days	225 t
Fuel – Operational	-	400 l/h	7 Days	70 m³

8.2 RESIDENCE

Residence time parameters are listed in Table 8.2.

Table 8.2 Residence Requirements

Commodity or Equipment	Residence time or volume
Fire water tank	150 m³
Raw water tank	200 m³
ROM stockpile	50,000 t
Crushed Ore Stockpile	24 hours (live)

9.0 LAYOUT

9.1 SITE LAYOUT

There are 4 distinct area of the site:

- Pit
- Process Plant
- Infrastructure
- Tailings

Each of these areas have unique and distinct criteria in regard to layout.

- Pit – The mine location is restricted by the ore body extent, depth and push back areas. Lighting should be directed as to minimised light pollution. Drilling, blasting and other high noise level activities should only be performed during the day, and the strength and direction of the wind should be considered before performing any activities. Dust suppression and minimisation measures shall be employed at all times.

- Process Plant – The process plant should be located on relatively flat ground within a short haul distance of the pit. Reagents will be stored in a separately bunded area on the downwind side of the main process plant. Acids and bases shall be stored separately. Lighting should be directed as to minimised light pollution. Dust suppression and minimisation measures shall be employed at all times.

- Infrastructure – The mine site infrastructure shall be location close to the main access road and the borehole. The main site electrical HV substation shall to be located in this area. Lighting should be directed as to minimised light pollution. Dust suppression and minimisation measures shall be employed at all times.

- Tailings –The tailing management facility shall be located down stream of all other mine areas. Lighting should be directed as to minimised light pollution. Dust suppression and minimisation measures shall be employed at all times.

9.2 PLANT LAYOUT

Equipment shall be arranged in accordance with the current and approved process flowsheets.

The design criteria for equipment layout are as follows:

- Gravity and natural properties of material flow shall be utilised to the maximum extent possible, to reduce energy inputs.
- Arrangements shall provide a smooth process flow and allow for merging with other process flow streams.

- All material transfer points shall be designed to minimise spillage.
- Adequate accessibility and clearance around equipment shall be provided for installation, operation and maintenance.
- Suitable Safe Working Load (SWL) rated cranes, monorails and hoists shall be provided for operation and maintenance purposes and at all equipment that may require replacement.
- Wherever possible, ladders and cat ladders shall be avoided, and only after discussion with the Owner and/or the Owner's Representative shall they be used in a design.
- Optimal use of the structures and available space within the structures shall be implemented.
- Floors shall be suitably sloped and drains / sumps shall be provided and positioned at the lowest point to collect spillage and wash-down water. Each area's slope will be determined based on possible spills particle size and will be shown on the layout drawings. The slope range will be between 2% to 10%.
- Fire protection system shall be provided conforming local codes and regulations.
- All buildings will be accessible by hard stand roads of a minimum width of 8 metres, laid out in a one way system around the plant.

10.0 PERSONNEL SAFETY AND OPERATION

Personnel safety and protection shall be prime factors in the mechanical design and layout of equipment. The following issues shall be handled in complete accordance with all applicable codes and regulations:

- Safety devices for handling of bulk material.
- Storage of hazardous material.
- Dust control of hazardous airborne material.
- Radiation hazards.
- Building ventilation.
- Ventilation of confined spaces and self-contained air supplies.

All mechanical moving parts shall be guarded. The design of the guards shall allow their removal without having to remove other items of equipment.

All openings, sumps, vessels, bins, hoppers, elevated platforms or pits that constitute a hazard shall be adequately fenced or otherwise guarded. Equipment shall be provided with appropriate access areas where required for operation, maintenance or cleaning.

10.1 HEATING, VENTILATION AND AIR CONDITIONING

Electrical type rooms and control rooms shall be maintained at a positive pressure by mechanically supplying filtered air into the space.

Sufficient make-up air shall be introduced into the buildings to exceed the exhaust requirements of the various rooms. This make-up air shall be filtered and heated or cooled where required.

10.2 SITE VEHICLES

Qty	Type	Consumption		Hour	Consumption	
1	D11 Dozer	180	l/h	14	2520	l/d
2	2 x 994K FEL	120	l/h	14	3360	l/d
2	Forklift	60	l/h	8	960	l/d
4	Maintenance Truck	20	l/h	14	1120	l/d
10	Light Vehicle	5	l/h	14	700	l/d
	Average Total	**361**	**l/h**	**Total**	**8660**	**l/d**

11.0 MATERIALS

Equipment materials selected shall be compatible with the process fluid temperature, pressure, chemical composition and the operating environment.

Materials containing asbestos shall not be used.

All materials shall be new and of appropriate quality. No welding, filling or plugging of defective parts shall be permitted without prior approval of the authorised Engineer.

12.0 EQUIPMENT LOADING

All equipment shall be designed to withstand the following, or a combination the following:

- Dead and live loads.
- Wind loads.
- Earthquake loads.
- Vibration loads.
- Pressure induced loads.
- Loads applied by machine action (e.g. torque).
- Acceleration or deceleration (inertia) loadings (e.g. braking forces).
- Impact loads.
- Loads induced by expansion and contraction of materials of construction.
- Loads produced by material spillage or abnormal operation (e.g. conveyor spillage onto adjacent walkways, blocked chutes).
- Loads produced during the course of plant maintenance (e.g. resting of equipment on adjacent platforms, leak testing of vessels).

Supplier shall provide relevant information on any inertial and dynamic loads caused by improper balance in the equipment and shall specify installation requirements for vibration control or isolation. Vibration isolation pads shall be provided if required.

Design service factors for all drive components, couplings, gear reducers and other major equipment shall be listed by the Supplier in the Equipment Datasheets.

13.0 NAMEPLATES AND TAGS

Each item of equipment shall have the manufacturer's standard nameplate, showing at least the following:

- Equipment name.
- Name and address of manufacturer.
- Model and serial number.
- Date of manufacture.
- All pertinent technical data.
- Equipment design data and capacity.
- Design code.

A tag showing the equipment number for each item shall also be provided.

14.0 MAINTENANCE, ACCESS AND LIFTING PROVISIONS

All access ways and platforms shall be designed in accordance with the requirements of Structural Design Criteria. Except for restricted areas, there shall be dual access - ingress and egress - to all areas. Operator access shall be by stairs.

Where tools are required for normal / routine maintenance work, stairs shall be provided. For areas where there is restricted access, ladder access is acceptable.

The lifting / handling of other external equipment shall be by 14 tonne pick and carry crane, or equivalent, (based on site). For large lifts, where the overhead travelling cranes, hoists and yard cranes are unsuitable, such as for mill motors and major crusher components, a 100 tonne mobile crane will be used.

For removal / handling of Ball mill liners, a wheel mounted, 7 axis mill reline machine shall be used. Suitable access and hard standing points for mobile cranes shall be allowed for in the plant layout. Additional (manual) overhead monorail hoists shall be provided for maintenance / removal of equipment where electric hoists are not provided, and mobile crane is found to be impractical.

15.0 NOISE LIMITS AND VIBRATION CONTROL

Maximum permissible sound power level from equipment, including auxiliaries, when operating under normal operating conditions, shall be no more than 85 dB(A), measured one metre from the source of noise, in accordance with ISO 7574 or other recognised international code for noise measurement.

For some items of equipment, for example such as a primary crusher, this noise level limit may prove impossible and impractical to achieve. In areas where such equipment is located clear and concise signs advising the use of hearing protection shall be adopted.

Vibration levels for rotating machinery should be in accordance with ISO 1940.

16.0 FIRE PROTECTION

As the fire protection / detection specification is dependent on negotiations between the client and the insurance underwriter, which at the time of writing has not been documented, the following shall apply until instructed otherwise.

16.1 FIRE DETECTION

The fire detection measures shown in Table 17.1 shall apply.

Table 16.1 Fire Detection Requirements

Area	Fire Detection Method
Crushing, process, reagents, water treatment and services areas	Nil
Tailings storage facility	Nil
Buildings	Smoke detectors
MCC's	Very Early Smoke Detection Apparatus (VESDA)
Transformers	Break-glass bulb, where sprinklers used

16.2 FIRE PROTECTION

The water for the fire water system will be drawn for the lowest practical location on the raw water tank. The raw water nozzles shall be located above the fire water nozzles by the distance needed to provide the required independent fire water tank capacity.

Design of the fire protection system shall be in accordance with NFPA 122, Standard for Fire Protection and Control in Metal / Nonmetal Mining and Metal Mineral Processing Facilities.

Fire pumps, comprising (electric) jockey and (electric) booster pumps and emergency diesel pump shall be in accordance with NFPA 20, Standard for Installation of Stationary Pumps for Fire Protection. Fire protection systems to be used on the project include those shown in Table 17.2.

Table 16.2 Fire Protection Requirements

Area	Fire Protection Method
Crushing, process, reagents, water treatment and services areas	Hydrants and hose reels at ground level. Hose reels above ground level. Hand extinguishers.
Tailings storage facility	Hand extinguishers.
Buildings	Hose reels. Hand extinguishers.
MCC's	Point smoke detection, and VESDA in MCC room. Highly visible and audible alarm to central control room and guard house. Hand extinguishers.
Transformers	Sprinkler system, if required: TBC.

17.0 DESIGN FOR EQUIPMENT TRANSPORT LIMITS

All equipment and materials will be delivered and will be transported by vehicle / truck to site.

The size of plant and equipment (shop assemblies) that can be delivered to site is dictated by road transport limitations and site.

The Engineering Specification Packaging and Transport covers the load weights and dimensions that can be handled.

17.1 SHIPPING

Shipping configuration drawings for the equipment including detailing supports, lifting attachment, weights, dimensions and restrictions shall be submitted for review by the Owner and / or the Owner's Representative. This review shall be approved before packing and shipping. Sensitive equipment and material (e.g. to temperature, humidity and impacts) shall be properly preserved for shipping and storage.

Where possible, equipment shall be mounted on structural steel skids, fabricated in accordance with all applicable structural codes. Each skid shall be provided with the following:

- Proper lifting lugs for safe lifting, with lifting capacity clearly marked on skid.
- Proper drainage holes on skid base members.
- Fully welded structural skid members – stitch welding is not acceptable.
- Completely painted skid assembly, as per painting specification.
- Skid elements and final assembly free of sharp edges and corners.
- Easy access to equipment for maintenance.
- Complete structural skid drawings, showing all dimensions, details, lifting lugs location and capacity, foundation loading and drain holes. Structural drawings shall be reviewed and approved by the Owner and / or the Owner's Representative.

18.0 SURFACE PROTECTION

Surface protection of plant and equipment shall be in accordance with the project specification for Surface Protection of Structural Steelwork and Platework, with the following clarifications.

Table 18.1 Surface Protection Requirements

Item	Mechanical Requirements
Fabricated (carbon steel) steelwork, platework, and piping (excluding Supplier's lube piping and equipment)	*Un-insulated surfaces ≤ 100 °C: TBC.*
Un-insulated surfaces > 100 °C:	*Coating system per paint manufacturer's recommended high temperature paint system.*
Insulated surfaces	*Temporary coating system only.*
Supplier's carbon steel lubrication piping and equipment	*Supplier's standard preparation and paint system [See Note, below].*
Carbon steel tanks and hoppers (internal protection, uninsulated)	TBC.
Proprietary equipment (gearboxes, valves, instrumentation, etc)	*Supplier's standard paint system [See Note, below].*
Electric motors	*Motor supplier's standard paint system [See Note, below].*
Stainless steel	*No surface protection system required.*
Exposed machined surfaces	*Approved corrosion inhibitor compound and protected against damage.*
Topcoat paint colours	*Plant and equipment: Oatmeal.*
Electric motors	*Blue-Grey or Munsell 7.5BG4/2.*
Water tanks (internal)	*White.*
Other tanks and hoppers (internal)	*Black.*
Guards and handrails	*Golden Yellow.*
Fire protection equipment cabinets and piping	*Signal Red.*

Note: Where Supplier's paint system is used for proprietary equipment item or for packaged plant, it shall be suitable for the application and shall be approved by the responsible engineer prior to placement of order / application of surface protection.

19.0 POWER SUPPLY

The electrical system shall be designed using the voltage levels, frequency and earthing as listed below.

Table 19.1 Power Supply

Application	Criteria
Medium Voltage Level	11 kV, 3 Phase, 50 Hz, high resistance grounded
Low Voltage Level	400 V, 3+N, 50 Hz high resistance grounded
Lighting & Small Power	220 V, 1+N, 50 Hz
Low Voltage Motor Contactors	220 V, 1+N, 50 Hz
Equipment Heaters	220 V, 1+N, 50 Hz
MCC Control Circuits	24 VDC
Plant Control System Hardware	24 VDC
Electrical Field Controls	24 VDC
Instrumentation	24 VDC

19.1 ELECTRICAL EARTHING AND LIGHTNING PROTECTION

Electrical Earthing and Lightning protection for the entire site will be subject to a detailed design study.

All electrical equipment and devices installed on the package / skid shall be provided with facilities / connections for the termination of earth conductors.

Two M10 earth bosses (as a minimum) shall be installed at diagonally opposite corners of each package. Such earth bosses will be utilised for earthing the package. All tanks, vessels and structures not welded to the package shall be bonded with the items in the package via earth bosses or an earth bar. The earth cable shall be 35 mm^2 (minimum) cross sectional.

Cable armours of all armoured cables shall be bonded to earth via the gland and gland-plate at both ends of the cable.

The metallic enclosures of all electrical and instrument equipment / devices and all metallic piping / equipment shall be bonded to the skid base either directly via an earth boss or earth bar. Such earth wires shall be of 10 mm^2 (minimum) cross sectional area.

Cable ladders shall be electrically bonded by the installation of 35 mm^2 single core insulated earth cable installed at each end of run.

Each cable ladder and/or tray run shall be earthed by connection to the structural steel skid by the installation of a 10 mm^2 single core insulated earth cable installed at each extremity of the cable ladder.

20.0 EQUIPMENT & MATERIALS SELECTION CRITERIA

Manufacturers and fabricators shall be given the latitude to use their experience to employ the best design, installation practice and/or procedure, except where the latter would contravene Standard Operating Procedure or other standards of the mine or these criteria.

The equipment shall be robust and fit for heavy-duty applications found in a mining environment.

All equipment shall be conservatively rated and sized to withstand capacity changes due to process upsets and variations.

All equipment shall be designed to meet site conditions, such as altitude, ambient temperatures, seismic, wind, rain, humidity and any corrosive surrounding atmosphere.

The equipment shall meet or exceed the project specified production requirements for the expected life of mine operation.

Wherever possible, standard “off the shelf” equipment and components shall be used.

All materials used in the construction or assembly of equipment shall be new and free of any defects. Material Test Certificate and non-destructive testing shall be requested where applicable.

The equipment shall meet or exceed the current environmental standards of the jurisdiction in which it is installed, as well as any environmental restriction(s) that could or should be anticipated.

Where required for operation, maintenance or cleaning, equipment shall be provided with appropriate access. This access may include stairs and/or ladders, walkways and platforms complete with handrails, knee rails and kickboards that comply with the current applicable health and safety regulations.

All equipment shall be designed and/or selected in accordance with the process and site condition requirements. Other factors to be considered for equipment selection shall include, but not be limited to the following:

- Maximisation of personnel health, safety and protection.
- Ease of installation, operation, inspection, cleaning, maintenance, equipment removal and repairs.
- Minimisation of vibration and excessive noise.
- Minimisation of operating and maintenance costs.
- Minimisation of capital costs.

- Minimisation of thermal expansion stresses.
- Maximisation of standardised components.
- Availability of spare parts.
- Demonstrated successful operational history of equipment and components in comparable installations.

All equipment heavier than 34 kg shall be provided with lifting lugs or another convenient lifting arrangement.

All equipment shall have a transport weight of less than 10 tonnes per axle or be able to be broken down into subcomponents weighing less than 10 tonnes per axle, in order to meet the transportation requirements to site.

Drawings approved by the Owner and the Owner's Representative shall take precedence over other design information.

The equipment manufacturer and model shall be that which is specified in the mine standards unless agreed upon by Owner and/or the Owner's Representative.

Spare parts for equipment shall be readily available or procured with equipment at time of purchase. A list of required spares and their lead times shall be provided to the Owner and/or the Owner's Representative.

21.0 CONTROL SYSTEM

The control system shall be capable of full automated integrated process control, PID (Proportional. Integral and Derivative) loops, and sequence logic control.

Table 21.1 Control Systems

System	Specification	
Programming Software	WINCC	
SCADA Software	Siemens PCS 7	
Historian Software	OSIsoft PI Historian	
Central Site Control Point	Central control room in process plant	
Local / Remote SCADA	Employ Local and Remote Control and Monitoring	
Level of Operators	Semi-skilled	
Distributed Controllers	Yes, I/O Extension: Siemens Simatic ET200M	
Programmable logic controller (PLC)	Preferred PLC: Siemens S7-300 & 400 (Consider new S7-1500)	
Communications Protocol	Ethernet for PLC to SCADA; Profibus for Variable Speed Drive	
Communication Links	Optic Fibre Ground Wire some overhead lines (Preferred)	
If Radio	Frequency Band	D-Link radio (2.4 GHz)

The control system shall have the capability of communicating via various industrial protocols including but not limited to DeviceNet, Foundation Field Bus, Profibus DP, Ethernet IP and Modbus TCP/IP.

The control system shall have, as a minimum, redundancies on: Controllers, operator consoles, power supplies, and communications modules. Critical I/O and I/O for duty / standby shall not be installed on the same card / controller.

An Uninterruptible Power Supply (UPS) shall be installed to power the control system and field instruments. The UPS should have a minimum capacity of 30 minutes. The UPS system will be sized so that the connected load is only 60% of the nameplate full load rating of the UPS. The inverter units will be redundant so that in the event that a failure of one of the units, the additional load will be handled by the balance of the inverter units.

The control system shall operate from a global database to allow ease of engineering and operational data access modification and troubleshooting.

The control system shall have built-in diagnostics to allow system troubleshooting down to device I/O level. The system diagnostics shall be accessible to offsite technical personnel via a secure remote connection.

All interlock systems shall be designed to be “fail-safe”. On device failure, loss of power or loss of instrument air, the outputs that control process streams shall fail to a pre-defined safe state, e.g. output contacts fail open, solenoid valves fail de-energised, and control valves fail closed, motors fail stopped.

The control system shall include a continuous historian to collect process data. Access to the data historian shall be via the control system operator stations.

Control system equipment located in electrical rooms shall be housed in IP52 enclosures. Field mounted control system equipment shall be housed in IP66 enclosures. Control system equipment installed outdoors shall be housed in weather and dust proof enclosures.

All trip and Emergency Shut Down signals shall be hard wired unless otherwise specified in the project documentation.

All enclosures shall be suitable for the environmental conditions specified in the project documentation. Local Control Panel enclosures mounted in locations exposed to direct sunlight shall be provided with a hood above the enclosure.

Where panels are mounted outdoors, the Supplier shall ensure Liquid crystal display or Light emitting diode display devices are either legible in sunlight or effectively shaded.

Digital signals from the package to external control systems shall be fail-safe volt-free contact closures. Digital signals from external control systems to the package will be fail-safe volt-free contact closures.

22.0 QUALITY ASSURANCE

Suppliers, vendors, manufacturers, etc. are required to have an established Quality Assurance Programme of Quality Plan that complies with the quality objectives of the Owner's Representative.

The quality obligations of suppliers, vendors, manufacturers, etc. are defined in the project supplier quality assurance specifications and inspection data sheets of the Owner's Representative.

Suppliers, vendors, manufacturers etc. shall be responsible for all quality control, inspection and testing. Where applicable, the equipment shall be shop assembled and tested before shipment to the extent required to ensure satisfactory assembly, installation and operation at site. If disassembly is required for shipment, all sub-assemblies and components shall be match-marked for re-assembly onsite. The extent of assembly and testing shall be fully described and documented and approved by the Owner's Representative.

APPENDIX G

Mechanical Equipment List

DESIGN CONSULTANT		CLIENT
TETRA TECH	LIST	RUPERT RESOURCES

Ikkari & Pahtavarra PEA

DOCUMENT NO: 03627AA-0000-LST-M-001

MECHANICAL EQUIPMENT LIST

APPROVALS				
	Name	Position	Signed	Date
Originator	James Seccombe	Project Engineer		
Checked				
Approved				

Rev	Date	Description	By	Chk	Appd
A		Preliminary Issue	JS		

Ikkari & Pahtavarra PEA WBS Project No 03627AA

No	Area
0000	**General**
0100	Project Management
0200	Project Administration
0300	Client Meetings
0400	Risk Management
0500	Gap Analysis
0600	Site Visit
0700	Reporting
1000	**Mining**
1100	Open pit
1200	Underground Mine
1300	Waste Rock Dump
1400	Ore Stockpile
1500	Heavy Vehicle Workshop
1600	Haul Road
1700	Explosive Storage
2000	**Processing**
2100	Crushing
2200	Grinding
2300	Flotation
2400	Leaching
2450	Elution
2500	Cyanide Destruction
2600	Gold Room
2700	Water Treatment
2750	Tailings Handling
2800	Reagents
2900	Pahtavarra Feed Handling
2950	Plant Utilities
3000	**Onsite Infrastructure**
3100	Water Management
3200	Roads
3300	Buildings
3500	Waste Management
3600	Electrical
3700	Fuel Farm
3800	Laboratory
3900	Mobile Plant
4000	**Tailings & Water Treatment**
4100	Tailings Storage Facility
4500	Mine Impaction Water Treatment
4600	Mine Process Water Treatment
5000	**Offsite Infrastructure**
8100	Power
8200	Buildings
8300	Road
6000	**Project Delivery**
6100	EPCM
6200	Construction
6300	Temperory
7000	**Owners Costs**
8000	**Reclaimation and Salvage**
9000	**Provisions**

Discipline	Codes
Architectural	A
Civil	C
Electrical	E
Geotechnical	G
Instrumentation & Control	I
Mechanical	M
Environmental	N
Piping	P
Structural	S
General	X
Process	Z

Documents	Codes
Bill of Material	BOM
Block Flow Diagram	BFD
Calculation	CAL
Data Sheet	DAT
Design Criteria	DSC
Drawing	DRG
List	LST
Material Take Off	MTO
Memorandum	MEM
Minutes of Meeting	MOM
Model	MOD
Non Conformance Report	NCR
Piping & Instrumentation Di	PID
Plan	PLN
Process Flow Diagram	PFD
Report	RPT
Request for Information	RFI
Request for Quotation	RFQ
Schedule	SCH
Scope of Work	SOW
Sketch	SKT
Specification	SPC
Technical Bid Evaluation	TER
Transmittal	TXL
Vendor Data Request	VDR
Work Package	WPK

DESIGN CONSULTANT	Mechanical Equipment List	CLIENT
TETRA TECH		RUPERT RESOURCES

Project Title	Ikkari & Pahtavarra PEA	Document Number	03627AA-2000-LST-M-001
Project No	03627AA	Revision	A

AREA	EQUIPMENT DESIGNATOR	SEQUENTIAL NO	EQUIPMENT NO	VENDOR PACKAGE NO	DESCRIPTION	QUANTITY	OPERATIONAL UNIT CAPACITY T / h	OPERATIONAL UNIT CAPACITY m3 / h	CHARACTERISTIC (DIMENSIONAL, SPECIFICS ETC)	INSTALLED POWER (kW)	POWER FACTOR	OPERATION	DIVERSITY	LOAD (kW)	UNIT WEIGHT (t)	Procurement Package	PFD	P&ID	SUPPLIER	NOTES
										16281				13262						
2100	BN	001	2100-BN-001		ROM Bin	1	-	-		-	-	-	-	-						
2100	FE	001	2100-FE-001		Apron Feeder	1	460.00	302.50	1.09m x 6m	22	0.85	Duty	100%	18.7						
2100	GR	001	2100-GR-001		Grizzly	1	460.00	302.50	1.1m x 6.1m	55	0.85	Duty	100%	46.8						
2100	CR	001	2100-CR-001		Jaw Crusher	1	460.00	302.50	110 kW jaw crusher, with 1200mm open side setting and 150mm close side setting	110	0.85	Duty	100%	93.5						
2100	CV	001	2100-CV-001		Crushed Ore Conveyor	1	460.00	302.50	800 mm Wide Belt, 250 m Long	90	0.85	Duty	100%	76.5						
2200	FE	001	2200-FE-001		Reclaim Feeder No1	1	460.00	302.50	1.09m x 6m	55	0.85	Duty	100%	46.8						
2200	FE	002	2200-FE-002		Reclaim Feeder No2	1	460.00	302.50	1.09m x 6m	55	0.85	Standby	10%	4.7						
2200	CV	001	2200-CV-001		Mill Feed Conveyor	1	460.00	302.50	800 mm Wide Belt, 120 m Long	55	0.85	Duty	100%	46.8						
2200	ML	001	2200-ML-001		Sag Mill	1	1539.70	1237.70	30 ft diameter with EGL of 15ft, 5.5MW single pinion drive motor	5500	0.85	Duty	100%	4675.0						
2200	SC	001	2200-SC-001		Sag Mill Screen	1	1539.70	1237.70	Single deck, 1.83m x 2.74m	22	0.85	Duty	100%	18.7						
2200	PP	001	2200-PP-001		Cyclone Feed Pump No1	1	1539.70	1237.70	16/14	300	0.85	Duty	100%	255.0						
2200	PP	101	2200-PP-101		Cyclone Feed Pump No2	1	1539.70	1237.70	16/14	300	0.85	Standby	10%	25.5						
2200	CY	001	2200-CY-001		Cyclone Cluster	1	1539.70	1237.70	38.1 cm urethane lined x 6	-	-	-	-	-						
2200	ML	002	2200-ML-002		Ball Mill	1	1539.70	1237.70	29 ft EGL with 21ft diameter, 5.5MW single pinion drive motor	5500	0.85	Duty	100%	4675.0						
2250	GC	001	2250-GC-001		Gravity Concentrator No1	1	2.50	2.00	Gravity Concentrator, feed up to 250 t/h, 56KW motor	55	0.85	Duty	100%	46.8						
2250	GC	002	2250-GC-002		Gravity Concentrator No2	1	2.50	2.00	Gravity Concentrator, feed up to 250 t/h, 56KW motor	55	0.85	Duty	100%	46.8						
2250	RE	001	2250-RE-001		Intensive Leach Reactor	1	1.70	1.70	12t intensive leach reactor	-	-	-	-	-						
2300	SC	001	2300-SC-001		Pre Leach Screen	1	1533.00	1232.00	Single deck, 1.83m x 2.74m	22	0.85	Duty	100%	18.7						
2300	TK	001	2300-TK-001		Flotation Conditioning Tank	1	-	-	10 m diameter, 15m height, 15 kW agitator installed	15	0.85	Duty	100%	12.8						
2300	TK	002	2300-TK-002		Flotation Tank No1	1	-	-	10 m diameter, 15m height, 15 kW agitator installed	15	0.85	Duty	100%	12.8						
2300	TK	003	2300-TK-003		Flotation Tank No2	1	-	-	10 m diameter, 15m height, 15 kW agitator installed	15	0.85	Duty	100%	12.8						
2300	TK	004	2300-TK-004		Flotation Tank No3	1	-	-	10 m diameter, 15m height, 15 kW agitator installed	15	0.85	Duty	100%	12.8						
2300	TK	005	2300-TK-005		Flotation Tank No4	1	-	-	10 m diameter, 15m height, 15 kW agitator installed	15	0.85	Duty	100%	12.8						
2300	TK	006	2300-TK-006		Flotation Tank No5	1	-	-	10 m diameter, 15m height, 15 kW agitator installed	15	0.85	Duty	100%	12.8						
2300	TK	007	2300-TK-007		Flotation Tank No6	1	-	-	10 m diameter, 15m height, 15 kW agitator installed	15	0.85	Duty	100%	12.8						
2300	TK	008	2300-TK-008		Flotation Tank No7	1	-	-	10 m diameter, 15m height, 15 kW agitator installed	15	0.85	Duty	100%	12.8						
2300	FL	001	2300-FL-001		Flotation Tails Filter No1	1	218.70	172.20	Belt filter, width 4m, length 20m, no wash	37.5	0.85	Batch	66%	21.0						
2300	FL	002	2300-FL-002		Flotation Tails Filter No2	1	218.70	172.20	Belt filter, width 4m, length 20m, no wash	37.5	0.85	Batch	66%	21.0						
2300	FL	003	2300-FL-003		Flotation Tails Filter No3	1	218.70	172.20	Belt filter, width 4m, length 20m, no wash	37.5	0.85	Batch	66%	21.0						
2300	FL	004	2300-FL-004		Flotation Tails Filter No4	1	218.70	172.20	Belt filter, width 4m, length 20m, no wash	37.5	0.85	Batch	66%	21.0						
2300	FL	005	2300-FL-005		Flotation Tails Filter No5	1	218.70	172.20	Belt filter, width 4m, length 20m, no wash	37.5	0.85	Batch	66%	21.0						
2300	FL	006	2300-FL-006		Flotation Tails Filter No6	1	218.70	172.20	Belt filter, width 4m, length 20m, no wash	37.5	0.85	Batch	66%	21.0						
2300	FL	007	2300-FL-007		Flotation Tails Filter No7	1	218.70	172.20	Belt filter, width 4m, length 20m, no wash	37.5	0.85	Batch	66%	21.0						
2400	TH	001	2400-TH-001		Pre-Leach Thickener	1	221.00	199.00	18 m diameter, high density thickener, 7.5 kW motor installed	7.5	0.85	Duty	100%	6.4						
2400	TK	001	2400-TK-001		Leach Tank	1	-	-	6 m diameter, 9m height, 5 kW agitator installed	5	0.85	Duty	100%	4.3						
2400	TK	002	2400-TK-002		CIL Tank No1	1	-	-	6 m diameter, 9m height, 5 kW agitator installed	5	0.85	Duty	100%	4.3						
2400	TK	003	2400-TK-003		CIL Tank No2	1	-	-	6 m diameter, 9m height, 5 kW agitator installed	5	0.85	Duty	100%	4.3						
2400	TK	004	2400-TK-004		CIL Tank No3	1	-	-	6 m diameter, 9m height, 5 kW agitator installed	5	0.85	Duty	100%	4.3						
2450	CO	001	2450-CO-001		Acid Wash Column	1	-	-	to be able to treat 8 tons of carbon per day	-	-	-	-	-						
2450	CO	002	2450-CO-002		Elution Column	1	-	-	to be able to treat 8 tons of carbon per day	-	-	-	-	-						
2450	KN	001	2450-KN-001		Carbon Regeneration Kiln	1	-	-	1.5m x 13.7m	37.5	0.85	Duty	100%	31.9						
2500	TK	001	2500-TK-001		Cyanide Destruction Tank No1	1	-	-	5 m diameter, 7.5m height, 3 kW agitator installed	5	0.85	Duty	100%	4.3						
2500	TK	002	2500-TK-002		Cyanide Destruction Tank No2	1	-	-	5 m diameter, 7.5m height, 3 kW agitator installed	5	0.85	Duty	200%	8.5						
2500	TK	003	2500-TK-003		Cyanide Destruction Tank No3	1	-	-	5 m diameter, 7.5m height, 3 kW agitator installed	5	0.85	Duty	300%	12.8						
2500	TK	004	2500-TK-004		Cyanide Destruction Tank No4	1	-	-	5 m diameter, 7.5m height, 3 kW agitator installed	5	0.85	Duty	400%	17.0						

AREA	EQUIPMENT DESIGNATOR	SEQUENTIAL NO	EQUIPMENT NO	VENDOR PACKAGE NO	DESCRIPTION	QUANTITY	OPERATIONAL UNIT CAPACITY		CHARACTERISTIC (DIMENSIONAL, SPECIFICS ETC)	INSTALLED POWER (kW)	POWER FACTOR	OPERATION	DIVERSITY	LOAD (kW)	UNIT WEIGHT (t)	Procurement Package	PFD	P&ID	SUPPLIER	NOTES
							T / h	m3 / h												
2500	FL	001	2500-FL-001		Detox Tails Filter No1	1	31.30	22.00	Belt filter, width 1m, length 12m, no wash	5.5	0.85	Batch	66%	3.1						
2500	FL	002	2500-FL-002		Detox Tails Filter No2	1	31.30	22.00	Belt filter, width 1m, length 12m, no wash	5.5	0.85	Batch	66%	3.1						
2500	FL	003	2500-FL-003		Detox Tails Filter No3	1	31.30	22.00	Belt filter, width 1m, length 12m, no wash	5.5	0.85	Batch	66%	3.1						
2500	FL	004	2500-FL-004		Detox Tails Filter No4	1	31.30	22.00	Belt filter, width 1m, length 12m, no wash	5.5	0.85	Batch	66%	3.1						
2600	EL	001	2600-EL-001		Electrowinning Cell No1	1	-	-	1.27 m^3 Capacity	110	0.85	Duty	100%	93.5						
2600	EL	002	2600-EL-002		Electrowinning Cell No2	1	-	-	1.27 m^3 Capacity	110	0.85	Batch	66%	61.7						
2600	EL	003	2600-EL-003		Electrowinning Cell No3	1	-	-	1.27 m^3 Capacity	110	0.85	Batch	66%	61.7						
2600	KN	001	2600-KN-001		Kiln	1	-	-		55	0.85	Batch	66%	30.9						
2600	MR	001	2600-MR-001		Mercury Retort	1	-	-	0.37 m^3 capacity	75	0.85	Duty	100%	63.8						
2600	FR	001	2600-FR-001		Arc Furance	1	-	-		185	0.85	Batch	66%	103.8						
2700	XM	001	2700-XM-001		Water Treatment Plant	1	-	-		2500	0.85	Duty	100%	2125.0						
2750	PP	001	2750-PP-001		Tailings Discharge Pump No1	1	-	-	8/6	150	0.85	Duty	100%	127.5						
2750	PP	101	2750-PP-101		Tailings Discharge Pump No2	1	-	-	8/6	150	0.85	Standby	10%	12.8						
2800	XM	001	2800-XM-001		Reagents Handling Plant	1	-	-		75	0.85	Duty	100%	63.8						
2900	XM	001	2900-XM-001		Pahtavarra Feed Handling	1	20.00	14.00		37.5	0.85	Duty	100%	31.9						
2950	SR	001	2950-SR-001		Wet Gas Scrubber	1	-	-	500 m^3/h	37.5	0.85	Duty	100%	31.9						
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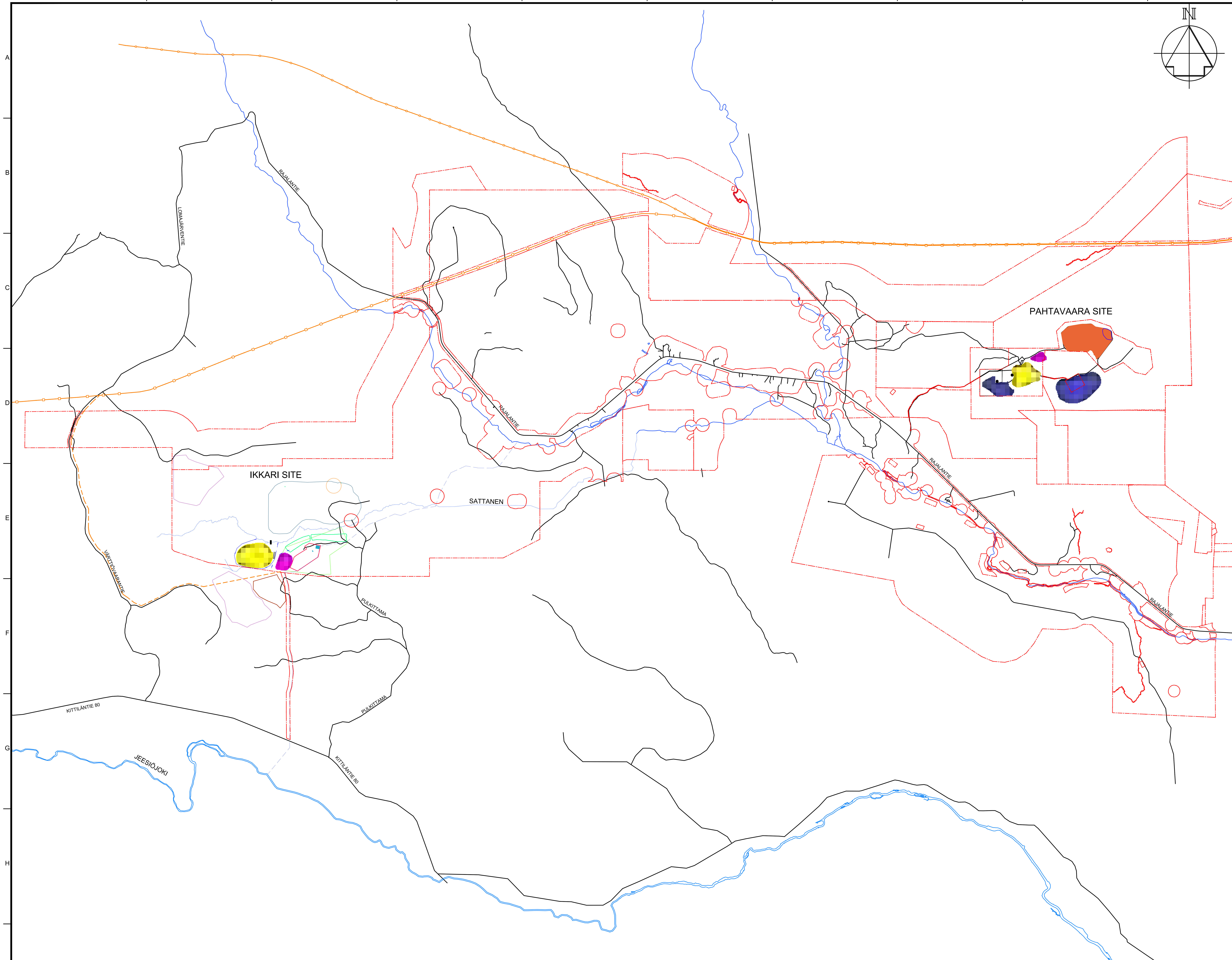
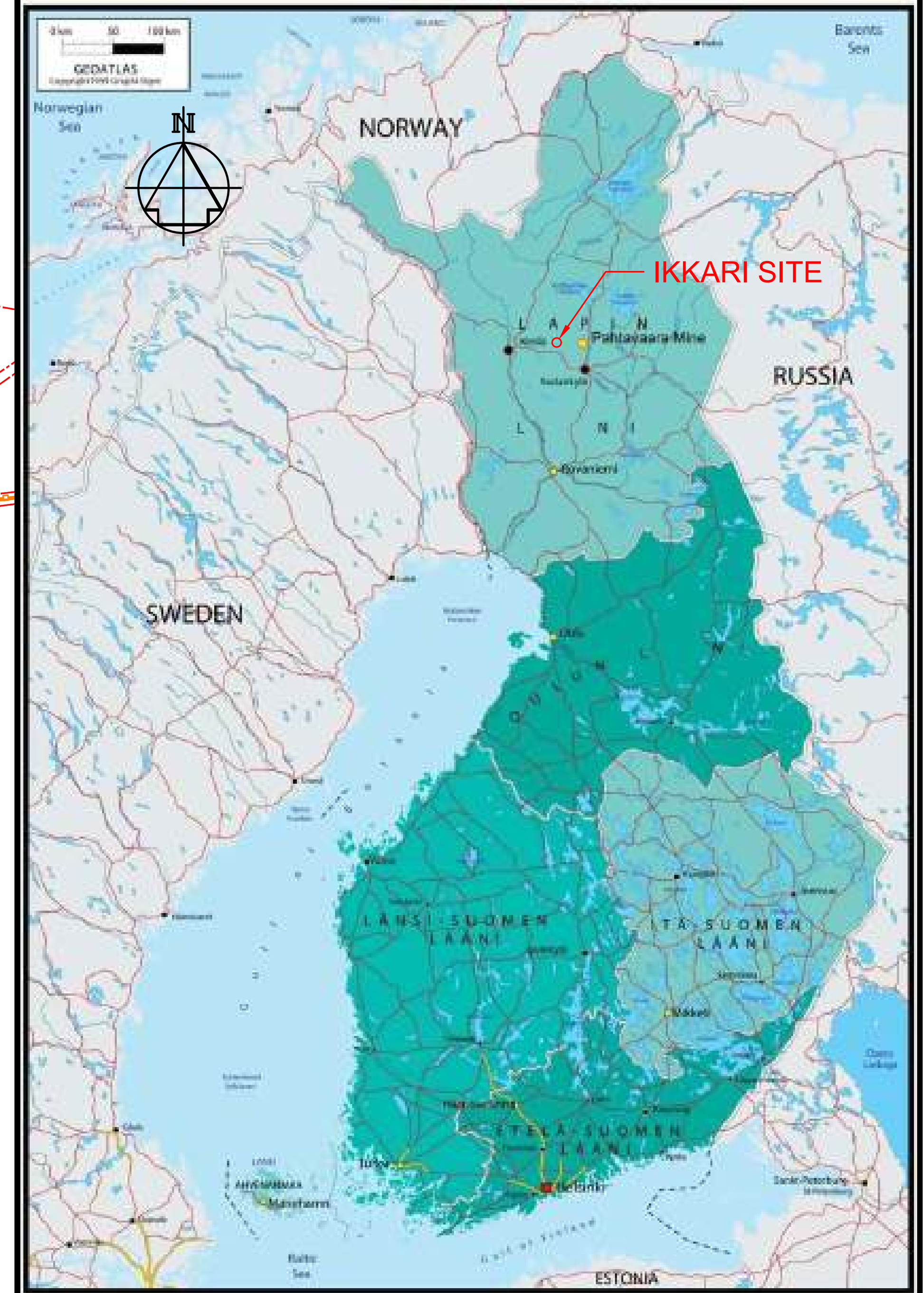
APPENDIX H

SITE PLANS



GENERAL NOTES

1. THIS DRAWING IS USED TO COLLATE INFORMATION ON THE IKKARI PEA.
2. REFERENCE COORDINATE SYSTEM - ETRS89.TM-35/Fin



MAP

NTS

LEGEND:

- RUPERT LANDS
- MINOR ROADS
- IKKARI TO PAHTAVAARA ACCESS +-62km
- RIVERS & STREAMS
- ELECTRIC OVERHEAD POWERLINES
- PROPOSED POWER CONNECTION ROUTE
- IKKARI PLANT SITE
- PAHTAVAARA TAILINGS SURFACE
- OPEN PIT
- IKKARI CO-DISPOSAL DUMPS
- PAHTAVAARA ROCK DUMPS
- ROM & LOW GRADE STOCKPILES
- IKKARI SURFACE RUN-OFF POND
- IKKARI POSSIBLE THICKENED TAILINGS

0 1.0 2.0 3.0 4.0 5.0km

SCALE 1:50 000

DWG. NO.	REFERENCE DRAWINGS

REV	DATE	REVISION DESCRIPTION	DRAWN	CHCK'D	DESN'D	APPV'D
D	22/11/2022	ISSUED FOR REVIEW & COMMENT	DO	NG	JL	AJC
C	27/10/2022	ISSUED FOR REVIEW & COMMENT	DO	NG	JL	AJC
B	17/08/2022	ISSUED FOR INFORMATION	DO	NG	JL	AJC
A	16/03/2022	ISSUED FOR INFORMATION	DO	-	-	-

PRELIMINARY DRAWING

NOT TO BE USED FOR CONSTRUCTION

DESCRIPTION	INITIALS	DATE
DRAWN	DO	16/03/2022
DESIGNED	JL	16/03/2022
CHECKED	NG	16/08/2022
APPROVED	AJC	16/08/2022
CLIENT	-	-

RUPERT RESOURCES

TETRA TECH

coffey

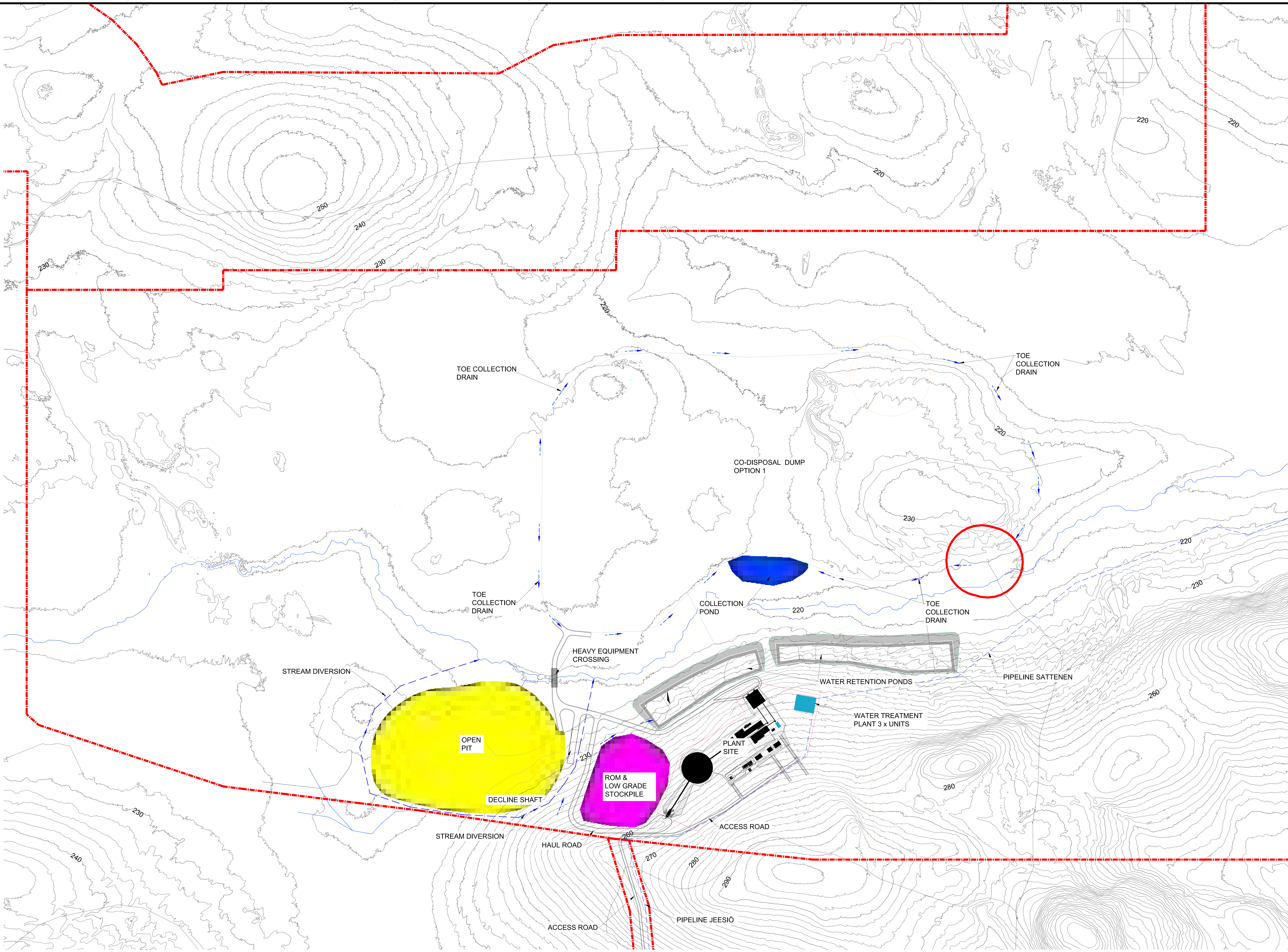
A TETRA TECH COMPANY

CLIENT DOCUMENT NUMBER

IKKARI PEA

OVERALL SITE PLAN IKKARI & PAHTAVAARA SITE LAYOUTS

SCALE	SIZE	DRAWING NUMBER	REV.
1:50000	A1	03627AA-0000-DRG-X-001	D



GENERAL NOTES

1. THIS DRAWING IS USED TO COLLATE INFORMATION ON THE IKKARI PEA.
2. REFERENCE COORDINATE SYSTEM - ETRS89.TM-35/Fin
3. ALL UNITS ARE IN METRES.
4. CONTOURS SHOWN AT 2 AND 10m INTERVALS

LEGEND:

- RUPERT LANDS
- ROADS
- RIVERS & STREAMS
- MAJOR CONTOURS 25m
- MINOR CONTOURS 5m
- SURFACE RUN-OFF DITCH
- PLANT SITE
- CO-DISPOSAL DUMPS
- OPEN PIT
- ROM & LOW GRADE STOCKPILE
- SURFACE RUN-OFF POND
- POSSIBLE THICKENED TAILINGS SITE

0 2km 4km 6km 8km 10km

SCALE 1:10 000

DWG. NO.	REFERENCE DRAWINGS

REV	DATE	REVISION DESCRIPTION	DRAWN	CHCK'D	DESN'D	APPV'D
F	22/11/2022	ISSUED FOR REVIEW & COMMENT	DO	-	-	-
E	27/10/2022	ISSUED FOR REVIEW & COMMENT	DO	-	-	-
D	13/10/2022	ISSUED FOR INFORMATION	DO	NG	JL	AJC
C	07/09/2022	ISSUED FOR INFORMATION - PLANT ADDED	DO	NG	JL	AJC
B	17/08/2022	ISSUED FOR INFORMATION	DO	NG	JL	AJC
A	16/03/2022	ISSUED FOR INFORMATION	DO	-	-	-

PRELIMINARY DRAWING

NOT TO BE USED FOR CONSTRUCTION

DESCRIPTION	INITIALS	DATE
DRAWN	DO	16/03/2022
DESIGNED	JL	16/03/2022
CHECKED	NG	16/08/2022
APPROVED	AJC	16/08/2022
CLIENT	-	-

RUPERT RESOURCES

TETRA TECH

coffey A TETRA TECH COMPANY

CLIENT DOCUMENT NUMBER

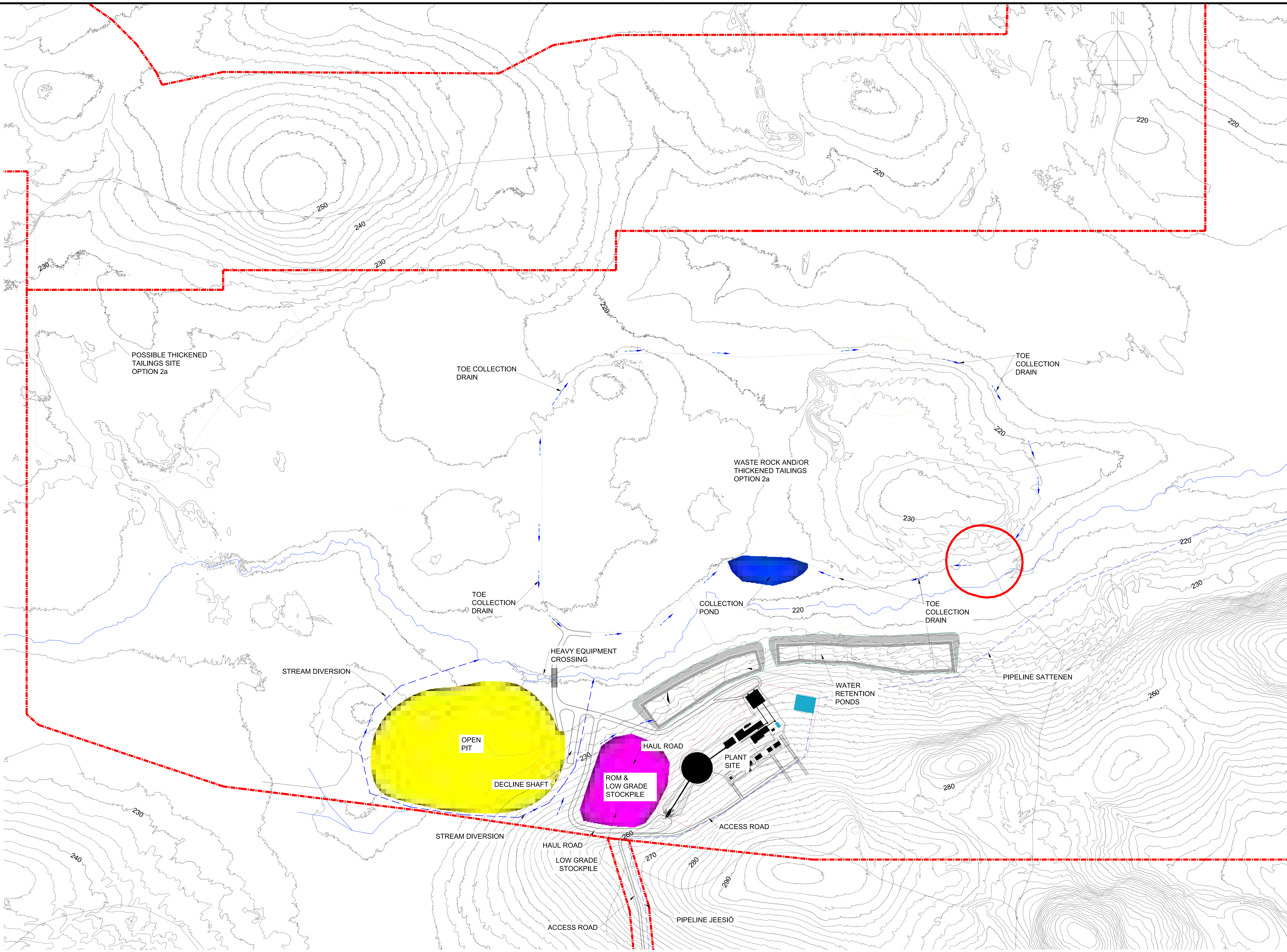
IKKARI PEA

IKKARI
SITE PLAN
OPTION 1
CO-DISPOSAL

SCALE	SIZE	DRAWING NUMBER	REV.
1:10000	A1	03627AA-0000-DRG-X-002	F

GENERAL NOTES

1. THIS DRAWING IS USED TO COLLATE INFORMATION ON THE IKKARI PEA.
2. REFERENCE COORDINATE SYSTEM - ETRS89.TM-35/Fin
3. ALL UNITS ARE IN METRES.
4. CONTOURS SHOWN AT 2 AND 10m INTERVALS



LEGEND:

- RUPERT LANDS
- ROADS
- RIVERS & STREAMS
- MAJOR CONTOURS 25m
- MINOR CONTOURS 5m
- SURFACE RUN-OFF DITCH
- PLANT SITE
- CO-DISPOSAL DUMPS
- OPEN PIT
- ROM & LOW GRADE STOCKPILE
- SURFACE RUN-OFF POND
- POSSIBLE THICKENED TAILINGS SITE

0 2km 4km 6km 8km 10km

SCALE 1:10 000

DWG. NO.	REFERENCE DRAWINGS

REV	DATE	REVISION DESCRIPTION	DRAWN	CHCK'D	DESN'D	APPV'D
B	22/11/2022	ISSUED FOR REVIEW & COMMENT	DO	-	-	-
A	27/10/2022	ISSUED FOR REVIEW & COMMENT	DO	-	-	-

PRELIMINARY DRAWING

NOT TO BE USED FOR CONSTRUCTION

DESCRIPTION	INITIALS	DATE
DRAWN	DO	16/03/2022
DESIGNED	JL	-
CHECKED	NG	-
APPROVED	AJC	-
CLIENT	-	-

RUPERT RESOURCES

TETRA TECH

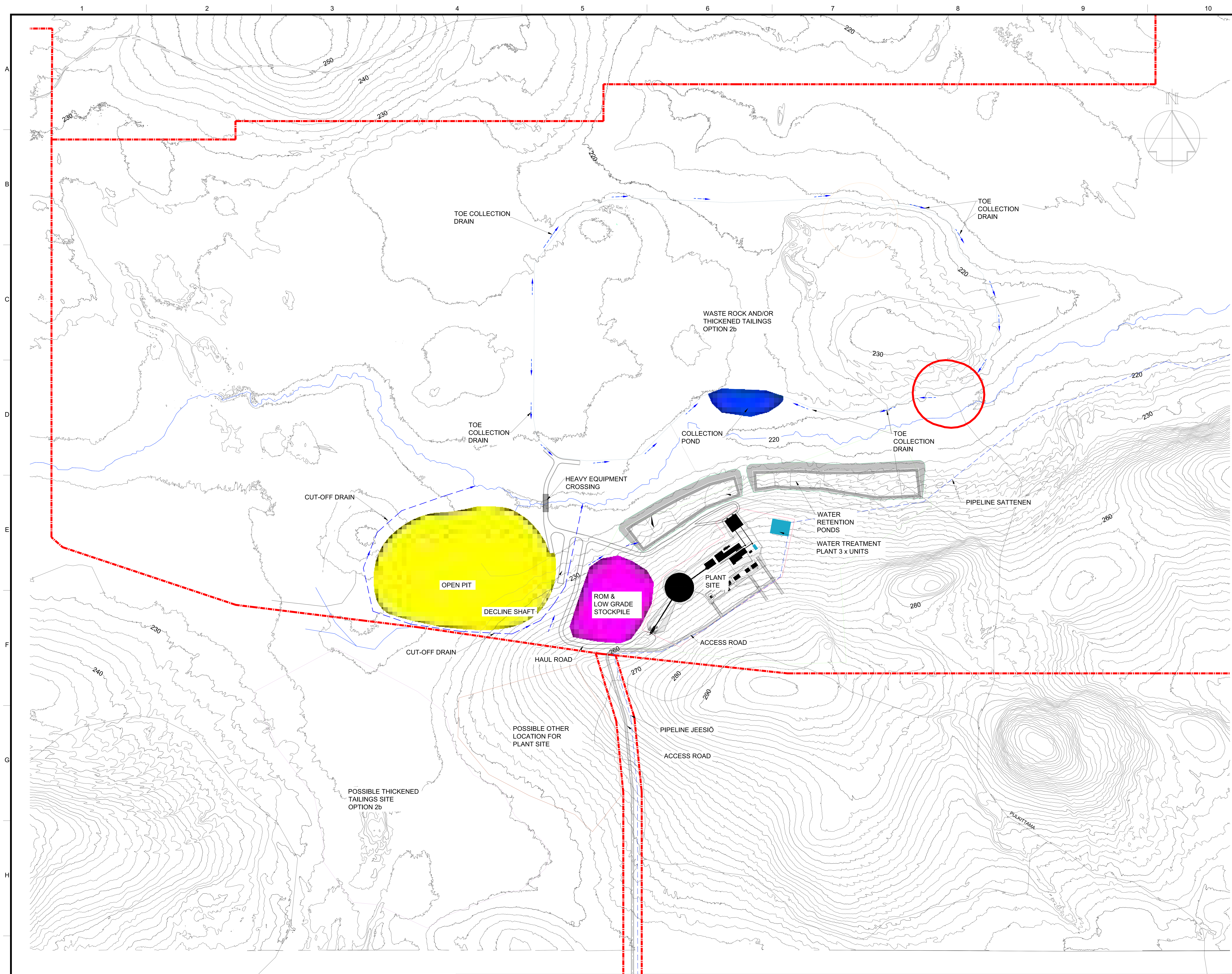
coffey A TETRA TECH COMPANY

CLIENT DOCUMENT NUMBER

IKKARI PEA

IKKARI
SITE PLAN
OPTION 2a
CO-DISPOSAL

SCALE	SIZE	DRAWING NUMBER	REV.
1:10000	A1	03627AA-0000-DRG-X-002	B



GENERAL NOTES

1. THIS DRAWING IS USED TO COLLATE INFORMATION ON THE IKKARI PEA.
2. REFERENCE COORDINATE SYSTEM - ETRS89.TM-35/Fin
3. ALL UNITS ARE IN METRES.
4. CONTOURS SHOWN AT 2 AND 10m INTERVALS

LEGEND:

- RUPERT LANDS
- ROADS
- RIVERS & STREAMS
- MAJOR CONTOURS 25m
- MINOR CONTOURS 5m
- SURFACE RUN-OFF DITCH
- PLANT SITE
- CO-DISPOSAL DUMPS
- OPEN PIT
- ROM & LOW GRADE STOCKPILE
- SURFACE RUN-OFF POND
- POSSIBLE THICKENED TAILINGS SITE

0 2km 4km 6km 8km 10km

SCALE 1:10 000

DWG. NO.	REFERENCE DRAWINGS

REV	DATE	REVISION DESCRIPTION	DRAWN	CHCK'D	DESN'D	APPV'D
B	24/11/2022	ISSUED FOR REVIEW & COMMENT	DO	-	-	-
A	27/10/2022	ISSUED FOR REVIEW & COMMENT	DO	-	-	-

PRELIMINARY DRAWING

NOT TO BE USED FOR CONSTRUCTION

DESCRIPTION	INITIALS	DATE
DRAWN	DO	16/03/2022
DESIGNED	JL	-
CHECKED	NG	-
APPROVED	AJC	-
CLIENT	-	-

RUPERT RESOURCES

TETRA TECH

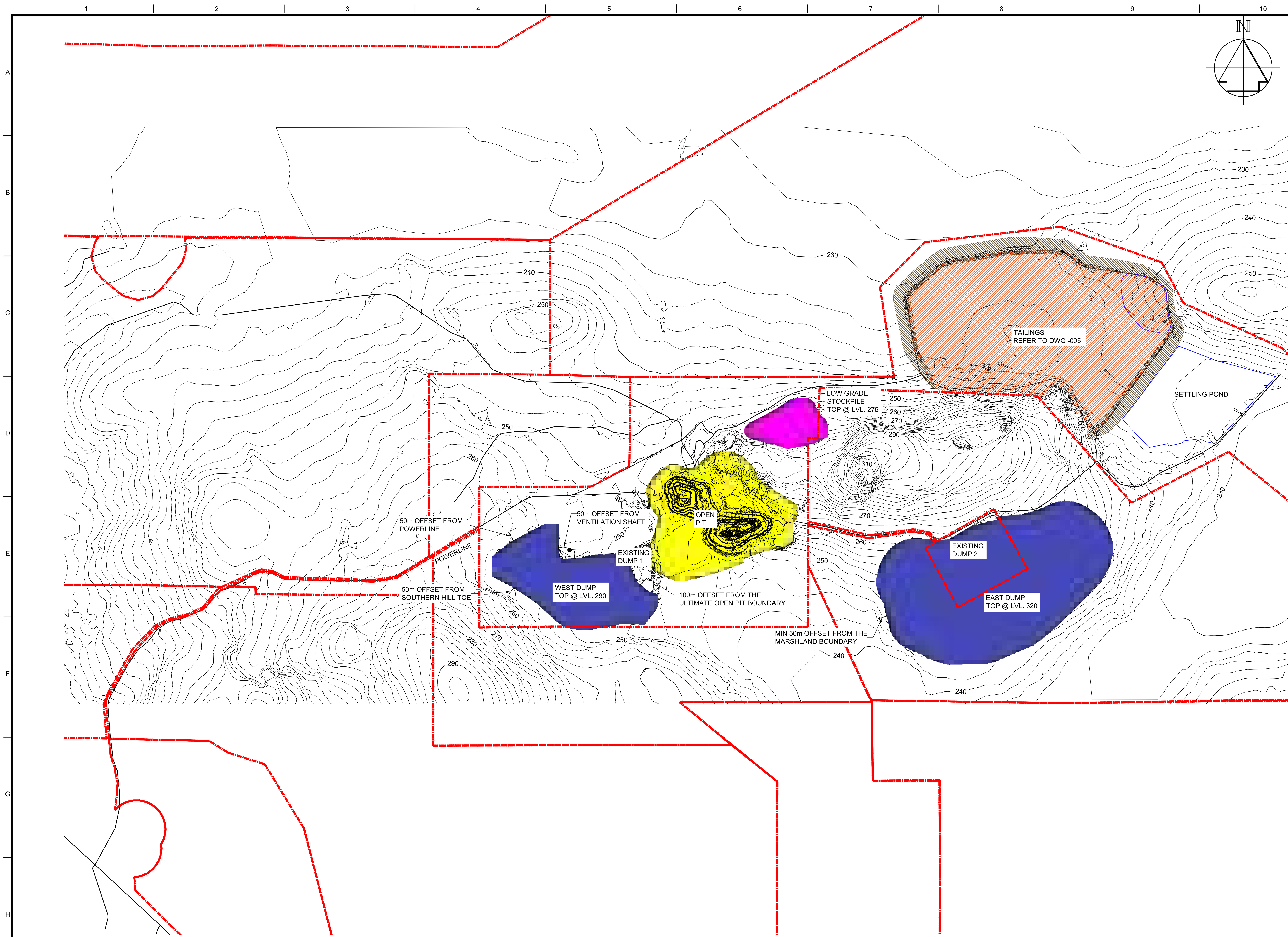
coffey A TETRA TECH COMPANY

CLIENT DOCUMENT NUMBER

IKKARI PEA

IKKARI
SITE PLAN
OPTION 2b
WASTE ROCK & THICKENED TAILINGS

SCALE	SIZE	DRAWING NUMBER	REV.
1:10000	A1	03627AA-0000-DRG-X-002	B



GENERAL NOTES

1. THIS DRAWING IS USED TO COLLATE INFORMATION ON THE IKKARI PEA.
2. REFERENCE COORDINATE SYSTEM - ETRS89.TM-35/Fin
3. ALL UNITS ARE IN METRES.
4. CONTOURS SHOWN AT 2m & 10m INTERVALS

LEGEND:

- RUPERT LANDS
- MAJOR ROADS
- MINOR ROADS
- RIVERS & STREAMS
- MAJOR CONTOURS 25m
- MINOR CONTOURS 5m
- PLANT SITE
- TAILINGS SITE
- TAILINGS EMBANKMENT
- SUPERNATANT POND
- ROCK DUMPS
- OPEN PIT
- LOW GRADE STOCKPILE

0 2km 4km 6km 8km 10km
SCALE 1:10 000

DWG. NO.	REFERENCE DRAWINGS

REV	DATE	REVISION DESCRIPTION	DRAWN	CHCK'D	DESN'D	APPV'D
B	22/11/2022	ISSUED FOR INFORMATION	DO	NG	JL	AJC
A	16/03/2022	ISSUED FOR INFORMATION	DO	-	-	-

PRELIMINARY DRAWING
NOT TO BE USED FOR CONSTRUCTION

DESCRIPTION	INITIALS	DATE
DRAWN	DO	16/03/2022
DESIGNED	JL	16/03/2022
CHECKED	NG	16/08/2022
APPROVED	AJC	16/08/2022
CLIENT	-	-

RUPERT RESOURCES
TETRA TECH
coffey
A TETRA TECH COMPANY

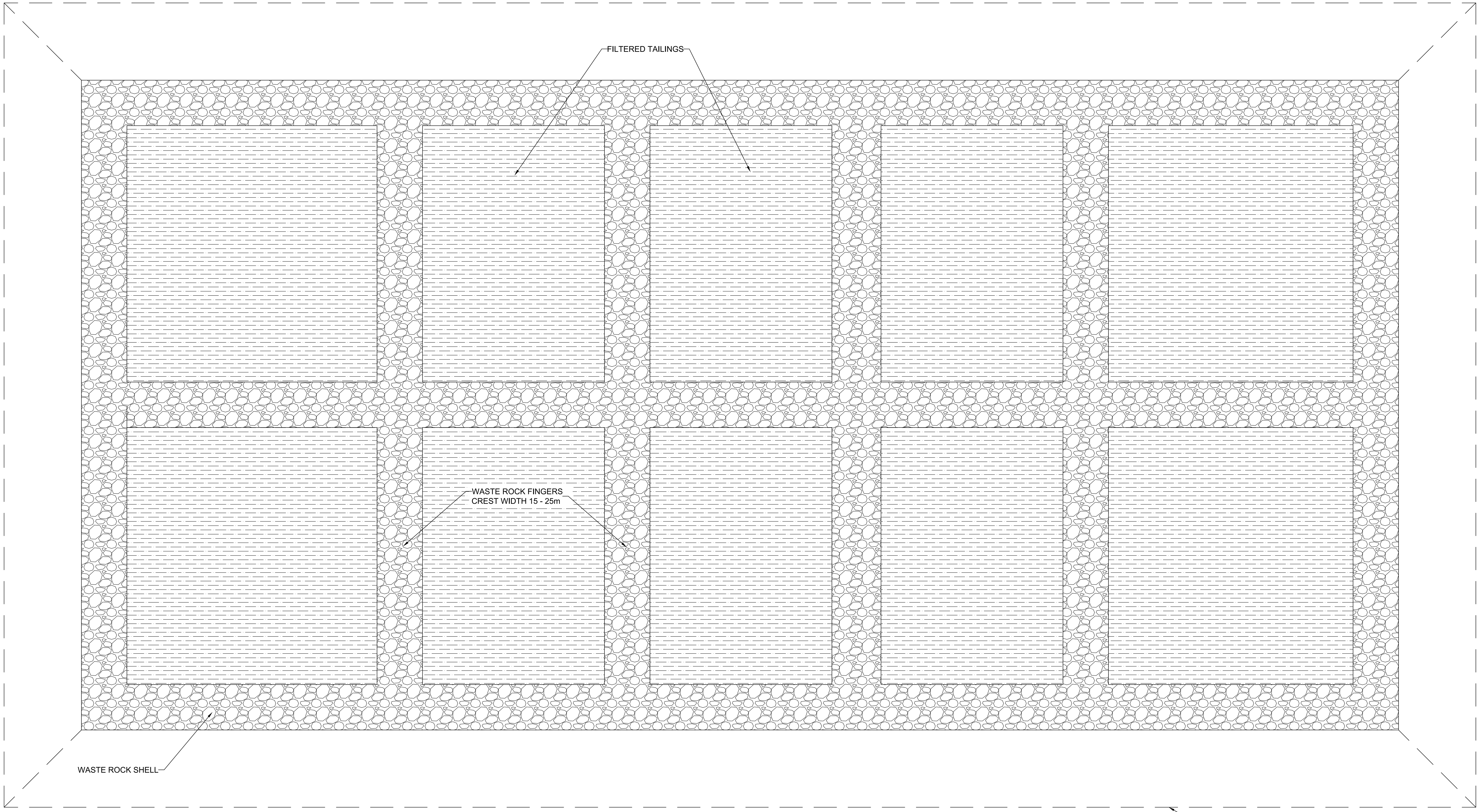
CLIENT DOCUMENT NUMBER

IKKARI PEA
PARTHAVAARI
SITE PLAN

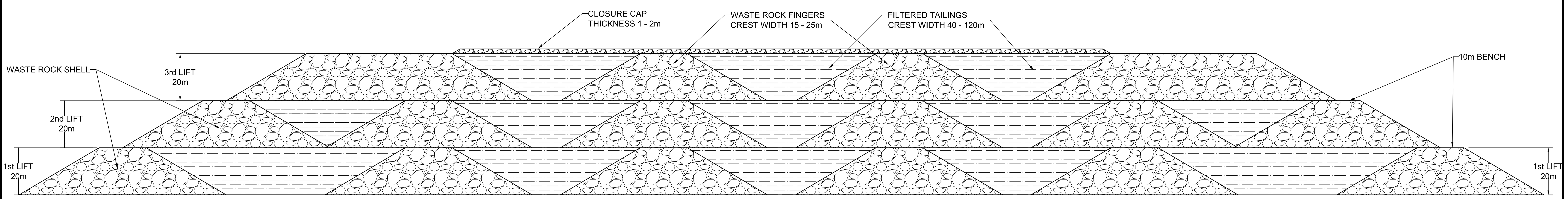
SCALE	SIZE	DRAWING NUMBER	REV.
1:10000	A1	03627AA-0000-DRG-X-003	B

GENERAL NOTES

1. THIS DRAWING IS USED TO COLLATE INFORMATION ON THE IKKARI PEA.
2. ALL UNITS ARE IN METRES.



CO-DISPOSAL FINGER CONSTRUCTION
1st LIFT PLAN
(SKETCH)



CO-DISPOSAL FINGER CONSTRUCTION
TYPICAL CROSS SECTION THROUGH 3 LIFTS
(SKETCH)

DWG. NO.	REFERENCE DRAWINGS

REV	DATE	REVISION DESCRIPTION	DRAWN	CHCK'D	DESN'D	APPV'D
A	17/08/2022	ISSUED FOR INFORMATION	DO	NG	JL	AJC

PRELIMINARY DRAWING

NOT TO BE USED FOR CONSTRUCTION

DESCRIPTION	INITIALS	DATE
DRAWN	DO	08/08/2022
DESIGNED	JL	08/08/2022
CHECKED	NG	16/08/2022
APPROVED	AJC	16/08/2022
CLIENT	-	-

RUPERT RESOURCES

TETRA TECH

coffey
A TETRA TECH COMPANY

CLIENT DOCUMENT NUMBER

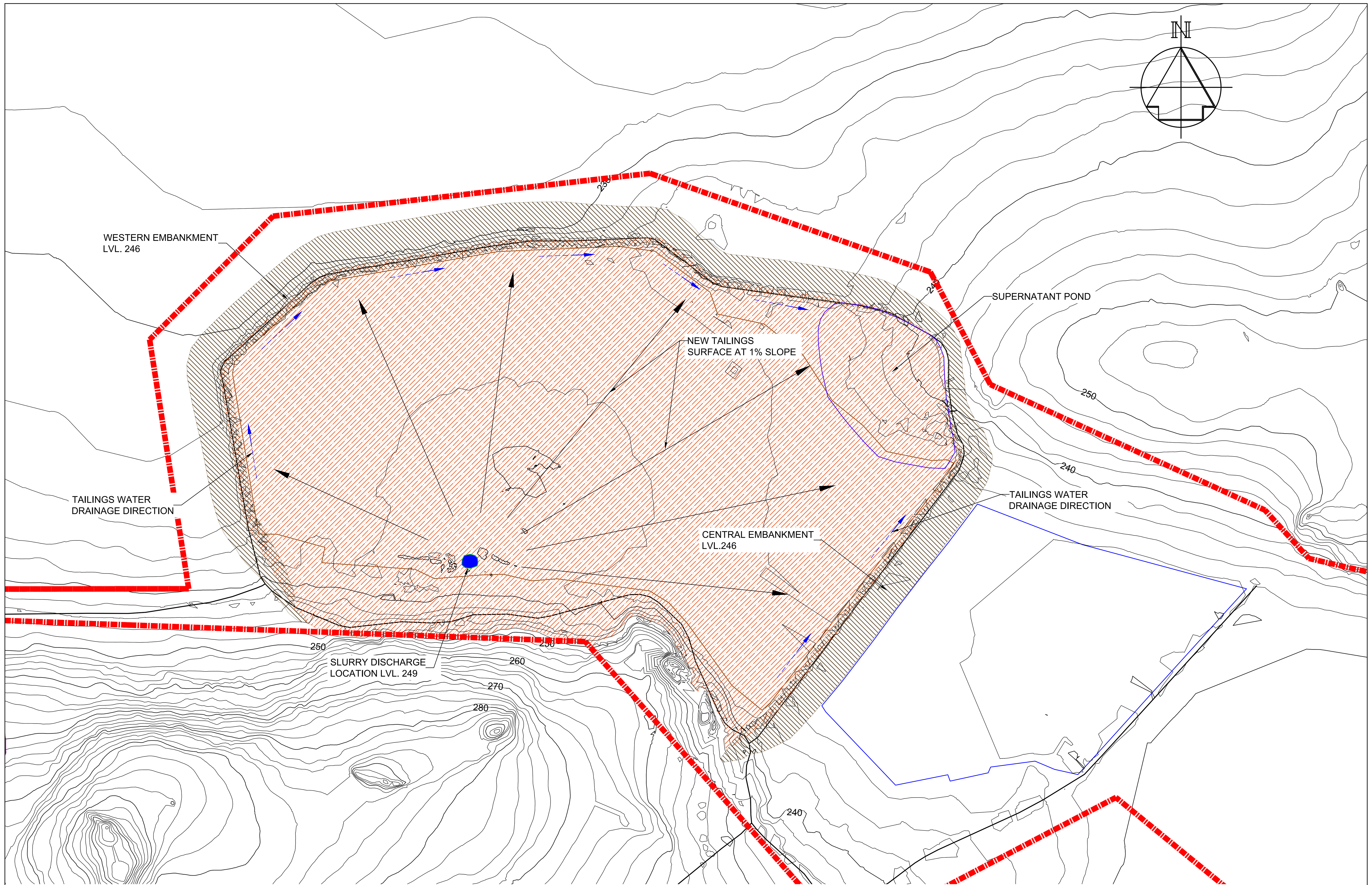
IKKARI PEA

IKKARI
TAILINGS FACILITY
CO-DISPOSAL
FINGER CONSTRUCTION SKETCH

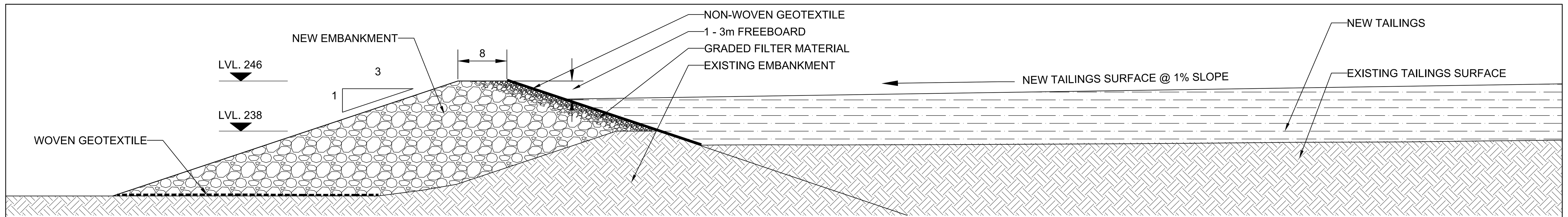
SCALE	SIZE	DRAWING NUMBER	REV.
1:1000	A1	03627AA-0000-DRG-X-004	A

GENERAL NOTES

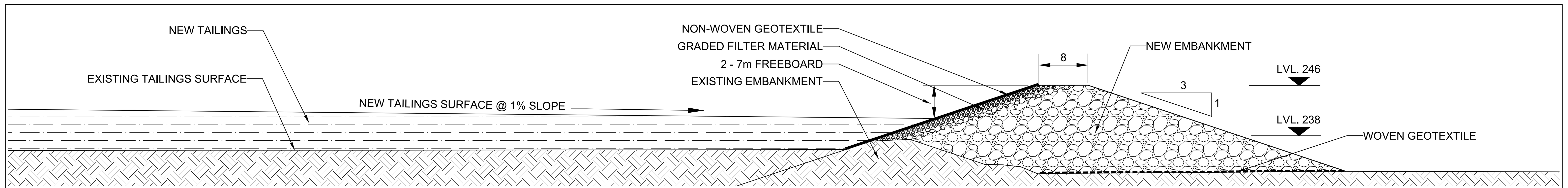
1. THIS DRAWING IS USED TO COLLATE INFORMATION ON THE IKKARI PEA.
2. REFERENCE COORDINATE SYSTEM - ETRS89.TM-35/Fin
3. ALL UNITS ARE IN METRES.
4. CONTOURS SHOWN AT 2m & 10m INTERVALS



TAILINGS DAM
SCALE 1:5000



TYPICAL SECTION THROUGH WESTERN EMBANKMENT
SCALE 1:500



TYPICAL SECTION THROUGH CENTRAL EMBANKMENT
SCALE 1:500

LEGEND:

- RUPERT LANDS
- MAJOR ROADS
- MINOR ROADS
- RIVERS & STREAMS
- MAJOR CONTOURS 25m
- MINOR CONTOURS 5m
- PLANT SITE
- TAILINGS SURFACE
- TAILINGS EMBANKMENT
- SUPERNATANT POND
- SETTLEMENT POND

0 50 100 150 200 250 300 350 400 450 500m
SCALE 1:5000 (FOR LAYOUT)

0 5 10 15 20 25 30 35 40 45 50m
SCALE 1:500 (FOR SECTIONS)

DWG. NO.	REFERENCE DRAWINGS	REV	DATE	REVISION DESCRIPTION	DRAWN	CHCK'D	DESN'D	APPV'D
		A	17/08/2022	ISSUED FOR INFORMATION	DO	NG	JL	AJC

PRELIMINARY DRAWING
NOT TO BE USED FOR CONSTRUCTION

DESCRIPTION	INITIALS	DATE
DRAWN	DO	08/08/2022
DESIGNED	JL	08/08/2022
CHECKED	NG	16/08/2022
APPROVED	AJC	16/08/2022
CLIENT	-	-

RUPERT RESOURCES
TETRA TECH
coffey
A TETRA TECH COMPANY

CLIENT DOCUMENT NUMBER

IKKARI PEA
PAHTAVAARA
TAILINGS FACILITY
PLAN AND SECTIONS

SCALE	SIZE	DRAWING NUMBER	REV.
AS SHOWN	A1	03627AA-0000-DRG-X-005	A

GENERAL NOTES

1. THIS DRAWING IS USED TO COLLATE INFORMATION ON THE IKKARI PEA.
2. REFERENCE COORDINATE SYSTEM - ETRS89.TM-35/Fin

DWG. NO.	REFERENCE DRAWINGS

REV	DATE	REVISION DESCRIPTION	DRAWN	CHCK'D	DESN'D	APPV'D
B	22/11/2022	ISSUED FOR REVIEW & COMMENT	DO	-	-	-
A	27/10/2022	ISSUED FOR REVIEW & COMMENT	DO	-	-	-

PRELIMINARY DRAWING
NOT TO BE USED FOR CONSTRUCTION

DESCRIPTION	INITIALS	DATE
DRAWN	DO	26/10/2022
DESIGNED	JL	-
CHECKED	NG	-
APPROVED	AJC	-
CLIENT	-	-

RUPERT RESOURCES
TETRA TECH
coffey
A TETRA TECH COMPANY

CLIENT DOCUMENT NUMBER

IKKARI PEA

OVERALL SITE PLAN
IKKARI & PAHTAVAARA
SERVICES LAYOUT

SCALE	SIZE	DRAWING NUMBER	REV.
1:50000	A1	03627AA-0000-DRG-X-006	B