PFS Study for Hautalampi Ni-, Cu-, Co-Deposit, Outokumpu, Finland

Prepared for FinnCobalt Oy.

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1 Summary

Key Features of the Hautalampi project

Measured and Indicated Resources	9.33 Mt @ Ni 0.28 %, Co 0.07 %, Cu 0.19 %
Ore Reserves	4.56 Mt @ Ni 0.30 %, Co 0.08 %, Cu 0.24 %
Diamond Drilling	
Historical (Outokumpu Oy)	103 dd holes, 11 865 meters
Finn Nickel Oy	100 holes, 11 153 meters
FinnCobalt Oy	74 holes, 8 381 meters
Mining and Processing Capacity	500 000 tpa
Capital Costs	EUR 74 840 000 (incl. 15 % contingency)
Operating Costs (€/t)	
Admin	3.10
Mining	21.55
Concentrator Plant	15.53
Logistics	0.98
Total	41.16
Concentrates:	
Ni-Co concentrate (nominal capacity)	16400 tpa (dry tonnes) Ni: 7 % (Ni recovery 82%) Co: 1.93 % (Co recovery 82%)
Cu Concentrate (nominal capacity)	3754 tpa (dry tonnes) Cu: 26.5 % (Cu recovery 86.5%)
Total metal production in concentrates	
Ni-Co Concentrate:	11 400 dry tonnes of Ni 2 900 dry tonnes of Co
Cu Concentrate	9 600 dry tonnes of Cu
Average Metal Prices in the DCF model	Ni 20 000 EUR/t Co 70 000 EUR/t Cu 9 750 EUR/t Exchange rate USD: EUR 1.00
Base Case Economic Return (contingency excluded)	NPV @ 5% 58.8 MEUR NPV @ 10 % 32.7 MEUR IRR 20.0 %

1.1 Introduction

FinnCobalt Oy commissioned AFRY Finland Oy (AFRY) to prepare a pre-feasibility study on the Hautalampi and Mökkivaara Ni-Cu-Co deposits. The study has been prepared and reported in accordance with the recommendations of the 2012 Australasian Code for Reporting of Mineral Resources and Ore Reserves (JORC 2012).

The mineral resource estimate dated October 14th, 2022 has been used as a basis for this study. This report has an effective date of March 7th, 2023.

The October -22 estimate was completed by Ville-Matti Seppä who is a Competent Person as defined by the Australasian Code for the Reporting of Mineral Resources and Ore Reserves (JORC Code) 2012 Edition.

1.2 Location

The Hautalampi property is located at Latitude 62.7151 °N, Longitude 28.9730 °E in the Outokumpu municipality, eastern Finland, about 2 km southwest of the town centre of Outokumpu, about 45 km WNW of the city of Joensuu, and about 350 km NE of Helsinki.

1.3 Ownership and History

The project is in the historic Keretti mine site, previously developed and operated by Outokumpu Oy. Suomen Nikkeli Oy (Finn Nickel Oy) acquired a 100% interest in the property in 2007. Followed by a FinnNickel bankruptcy in 2009 the mineral rights and the ground together with the Luikonlahti plant were purchased from the Finn Nickel bankruptcy estate by Vulcan Resources Pty Ltd. Hautalampi asset was sold to Vulcan Hautalampi Oy in September 2016. In May 2020 FinnCobalt Oy (formerly Vulcan Hautalampi Oy) agreed on the farm-in agreement with Eurobattery Minerals AB, where Eurobattery agrees to finance the future development of the company and subsequently earns the right to purchase all FinnCobalt Oy shares. Currently, FinnCobalt does not own the total land area covered by the mining concession.

Mining concession proceedings have been initiated in accordance with the original application. FinnCobalt has been prepared to update the mining concession area to have a better match to the planned land use and land ownership. Areas not owned by the company would be partly or totally left out.

The total area of the valid FinnCobalt mining concession 7802/1 is 283.5 hectares.

1.4 Geology and Mineralization

The geological setting of the Hautalampi mineralization is the same as that for the main Keretti Cu- rich ore, the main differences being in the mineralization of the mineralization zone within the Outokumpu stratigraphy, and the nature of the mineralization body itself.

Mineralisation mainly occurs as disseminations in bands due to metamorphosis. The mineralization zone has in sometimes a very sharp contact with the wall rocks. However, in many places, a transitional zone from one meter up to three meters occurs between the mineralization zone and wall rocks.

1.5 Project Status

The Hautalampi Project is an advanced exploration project that has seen extensive exploration throughout the years. The recent development includes core drilling for metallurgical sampling 2017-2018, which was followed by the flotation test work by GTK Mintec laboratories. Commercial grade Cu- and Ni-Co-concentrates were produced. Further Ni-Co-concentrate leaching test work aiming for battery chemicals production was done by Outotec Oyj. All test work succeeded well and confirmed that Hautalampi mineralisation is suitable for battery chemicals production.

The deposit has an Environmental Permit for underground mining in force and Mining Lease appropriation is ongoing. On June 17th 2022 the company published anew Environmental Impact Assessment for the project including underground mining and on-site ore processing and battery chemicals production plant.

1.6 Mineral resource estimates

The data that has been used for this work has been collected and compiled during the last mineral resource estimate work done by Outotec (Finland) Oy, dated 15th March 2009, and from the latest drilling campaign conducted by FinnCobalt Oy from 2020 to 2021.

The estimate has been prepared and reported in accordance with the recommendations of the 2012 Australasian Code for Reporting of Mineral Resources and Ore Reserves (JORC 2012).

The Mineral resource estimates at Hautalampi and Mökkivaara are presented below (Table 1-1, Table 1-2 and Table 1-3).

Table 1-1 Hautalampi Mineral Resources as of the September 29th, 2022 @ 0.25% NiEq cut-off

Hautalampi									
	Tonnes (t)	Ni %	Cu %	Co %	S %	Fe %	Zn %	Ni Eq %	Cu Eq %
Measured	2 808 000	0.35	0.26	0.08	2.02	3.92	0.04	0.70	1.57
Indicated	6 523 000	0.25	0.16	0.06	3.03	4.50	0.09	0.51	1.14
total	9 331 000	0.28	0.19	0.07	2.73	4.33	0.07	0.57	1.27
Contained Metals	tonnes	26 100 (Ni)	17 700 (Cu)	6 200 (Co)					

Table 1-2 Hautalampi Inferred Mineral Resources as of the September 29th, 2022 @ 0.25% NiEq cut-off

Hautalampi									
	Tonnes	Ni %	Cu %	Co %	S %	Fe %	Zn %	Ni Eq %	Cu Eq %
Inferred	216 000	0.21	0.12	0.02	1.91	3.79	0.07	0.32	0.72
Contained Metals	tonnes	450 (Ni)	260 (Cu)	40 (Co)					

Table 1-3 Mökkivaara Inferred Mineral Resources as of the September 29th, 2022 @ 0.25% NiEq cut-off

Mökkivaara									
	Tonnes	Ni %	Cu %	Co %	S %	Fe %	Zn %	Ni Eq %	Cu Eq %
Inferred	3 188 000	0.22	0.13	0.05	2.32	4.41	0.09	0.44	0.97
Contained Metals	tonnes	7 000 (Ni)	4 100 (Cu)	1 600 (Co)					

1.7 Ore Reserve estimates

Hautalampi Ore Reserve Estimate is based on the JORC (2012) compliant Mineral Resource Estimate prepared by AFRY Finland Oy (September 2022).

Table 1-4 summarises the Ore Reserve of Hautalampi deposit at 30 €/t NSR cut-off.

Table 1-4 Hautalampi Ore Reserves as of 1st March 2023 @30€/t NSR Cut-off. Mineral Resources are inclusive of these Ore Reserves.

Underground Ore Reserve		Grade			Contained Metals		
	Tonnes	Ni	Cu	Co	Ni (t)	Cu (t)	Co (t)
Proven	1 871 000	0.36 %	0.30 %	0.09 %	6 800	5 700	1 600
Probable	2 693 000	0.25 %	0.19 %	0.07 %	6 900	5 300	1 900
Total	4 564 000	0.30 %	0.24 %	0.08 %	13 700	11 000	3 500

For the mine design and reserve reporting purposes, a 30€/t NSR cut-off value was used. All excavated material needed to be above 30€/t NSR value and the average NSR value for the individual stope needs to be above 50€/t. The selected NSR cut-off is justified when compared against the Hautalampi project estimated OPEX (Table 1-5).

Table 1-5 OPEX breakdown

Costs	€/t
Administration	3.10
Mining	21.55
Concentrator	15.53
Logistics	0.98
total:	41.16

A yearly summary of the LOM plan is presented in Table 1-6. It is to be noted that the LOM plan includes also 369 000 tonnes of material from mineral resources that are in the inferred category. The inferred material comprises 7 % of the total LOM tonnes.

Table 1-6 LOM summary

	1	2	3	4	5	6	7	8	9	10	11	12	Totals
ORE t (Stopes)	78 000	445 000	352 000	332 000	331 000	379 000	464 000	458 000	464 000	470 000	417 000	83 000	4 274 000
ORE t (Drifts)	105 000	48 000	112 000	139 000	144 000	109 000	2 000	-	-	-	-	-	659 000
Total Ore	183 000	493 000	464 000	471 000	475 000	488 000	466 000	458 000	464 000	470 000	417 000	83 000	4 933 000
Feed Grades													
Ni (%)	0.24 %	0.29 %	0.32 %	0.32 %	0.34 %	0.33 %	0.26 %	0.28 %	0.25 %	0.26 %	0.20 %	0.17 %	0.28 %
Cu (%)	0.21 %	0.23 %	0.27 %	0.23 %	0.31 %	0.30 %	0.17 %	0.18 %	0.17 %	0.19 %	0.20 %	0.18 %	0.22 %
Co (%)	0.06 %	0.06 %	0.08 %	0.07 %	0.08 %	0.09 %	0.07 %	0.07 %	0.07 %	0.07 %	0.06 %	0.06 %	0.07 %
S (%)	3.36 %	2.62 %	2.72 %	2.20 %	2.38 %	2.42 %	2.35 %	2.43 %	2.18 %	2.46 %	2.75 %	3.26 %	2.49 %

1.8 Process

The process plant consists of a comminution circuit (crushing and grinding), copper flotation concentrate production, nickel/cobalt flotation concentrate production, sulphur removal flotation and tailings handling. The plant is designed for 500 ktpa nominal ore throughput. Design parameters are largely based on pilot test work conducted by GTK and experience from former Keretti operation with similar ore.

Concentrator plant availability has been assumed to be 92%, which is equal to an annual operation of 8059 h/a. 500 000 tpa feed rate to plant with 8059 h/a operation equals to 62 t/h nominal plant feed rate. The crushing circuit has been assumed to be operating 5 days per week in two shifts (16 h/d), which is equal to an annual operation of 4171 h/a. 500 000 tpa feed rate to the crushing with 4171 h/a operation equals to 120 t/h nominal feed rate to the crushing. Nominal main operational parameters are summarized in Table 1-7

Table 1-7. Nominal operational design parameters.

	Throughput (t/a)	Operation hours (h/a)	Feed rate (t/h)
Crushing Plant	500 000	4171	120
Process Plant	500 000	8059	62

Concentrate specifications have been estimated according to the pilot plant test work conducted by GTK. Concentrate specifications are based on average values for concentrates from two pilot runs (30.1.2019 and 31.1.2019). Specifications are presented in Table 1-8.

Table 1-8. Estimated concentrate specifications.

	Cu conc.	Ni/Co conc.
Cu (%)	26.50	0.76
Ni (%)	0.50	7.00
Co (%)	0.15	1.93
Cu-Rec to conc. (%)	86.5	10.5
Ni-Rec to conc. (%)	1.4	82.0
Co-Rec to conc. (%)	1.5	82.0
Fe (%)	28.06	37.19
S (%)	29.75	26.00

MgO (%)	3.48	5.39
SiO2 (%)	7.21	11.02
Au (g/t)	5.68	0.77
Ag (g/t)	42.00	5.80
Zn (%)	0.34	0.49
Pb (%)	0.001	0.001
As (%)	0.030	0.057
Bi (%)	0.011	0.01
Sb (%)	0.004	0.003
Te (%)	0.001	0.001

1.9 General infrastructure

Process water for process and backfill will be taken from the underground mine dewatering system.

Power supply will be provided through the national grid by a local supplier.

Office, warehouses and contractors' facilities will be located on-site. Some of them will be built by the local municipality and rented for FinnCobalt.

The mine access road of 1 km will be upgraded and paved to suit mine traffic.

The tailings are deposited as a slurry onto the old tailings storage facility. The pond will have a permeable dyke which will keep solids inside the pond.

1.10 Project design and schedule

The project's next study phases will take about 1.5 years and project execution will take less than two years.

Project permitting will be a time-consuming and critical task from the schedule point of view.

Preliminary schedules are presented in Table 1-9 and Table 1-10.

Table 1-9 Project development and execution schedule

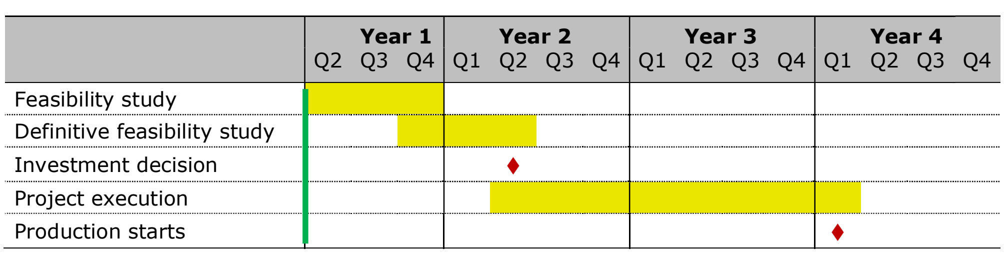


Table 1-10 Project permitting schedule

	Year 1			Year 2				Year 3				Year 4				Year 5			
	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Environmental impact assessment																			
Permitting process																			
Environmental permit																			
Possible Appeals																			

*) Green line marks the project's present status

1.11 Environment

The project area and underground mine are located in a connection with the old Keretti underground mine and the immediate vicinity of the town of Outokumpu. The old mining area is partially forested, and the area currently houses a golf course. The Jyri waste station is located west of the mining area. There are also other industrial players in the vicinity of the area, such as Outotec and GTK's pilot plant.

The mining area is located in the Sysmänjoki catchment area (catchment code 04.353) which is a part of the Vuoksi major catchment area (04). The waterbodies inside the project area are small lakes, ditches and streams. The surface waters of the mining area discharge through river Ruutunjoki (04.353_002) into the lake Sysmäjärvi (04.353.1.019_001), from which through river Sysmänjoki (04.353_004) and river Taipaleenjoki (04.351_001) further to Heposelkä lake area in Orivesi (04.311.1.001_002).

Lake Alimmainen Hautalampi cannot be considered a natural waterbody due to former mining operations, because it has been used as a final sedimentation pond.

Heavy metal concentrations are high, especially in Ruutunjoki, Lahdenjoki, Sysmänjärvi and Sysmänjoki water bodies, where the Environmental Quality Standard (EQS) has been occasionally exceeded. Part of the load is due to contaminated groundwaters. Lake Sysmänjärvi has been receiving a considerable load for several years, including load from both mining and the community. The impact of residential wastewater and agriculture can be seen in elevated nutrient levels and the mining water effect in the elevated electrical conductivity, as well as elevated sulphate and metal concentrations. Arsenic, nickel, copper and zinc have accumulated in the bottom sediment of the lake, due to the mining industry's influence. The Regional State Administrative Agency for Eastern Finland has granted a permit for the mixing zone in part of lake Sysmänjärvi. Article 4 of the WFD allows for the application of mixing zones which can be designated adjacent to discharge points. Within the mixing zones the concentration of one or more of the priority substances listed in Part A of Annex I, may exceed the relevant EQS values. But the rest of the water body should comply with EQS standards and not be affected by mixing zones (WFD, 2008/105/EC). In Sysmänjärvi, the mixing zone is set for nickel.

Watercourses near the project site are classified as poor (river Ruutunjoki) or moderate (lake Sysmänjärvi, river Lahdenjoki, Sysmänjoki, Taipaleenjoki) ecological condition according to EU:s Water Framework Directive (WFD) classification system. There are small streams, lakes and ponds on the project site that have no ecological classification due to their small size. Lake Sysmänjärvi is also a Natura 2000-area with high natural value.

The nearest classified groundwater area (to the project area) is Valkeisensärkkä is located approximately 2.2 km northwest of the mine site, and the classified groundwater area of Niilonpatama is located approximately 3.2 km west of the mine site and the classified groundwater area of Saari-Oskamo located approximately 4.4 km east from the mine site.

The groundwater in the project area has already been polluted during previous mining operations. Groundwater contains high levels of sulphate, manganese and iron. Groundwater is also acidic in many monitoring points. The use of groundwater in the area is prohibited.

An ESIA (environmental and social impact assessment) procedure for mining projects in Finland is required before the permitting. The EIA report has been submitted in 2022 and the EIA coordinating authority requested complementary EIA work on 19th September 2022. Environmental permit applications must be submitted to the relevant authority AVI (Regional Administrative Agency), as defined in the Environmental Protection Act (527/2014) and Decree (713/2014). The EIA phase must be completed before the permit application. After filing a permit application, the authority will publish the application to allow the relevant other authorities and anyone affected by the plans to comment and make proposals concerning the requirements for the permit. Permit decisions may be appealed to the Administrative Court of Vaasa and subsequently to the Supreme Administrative Court.

Mineral rights, including decisions concerning mining permits, are regulated under the Mining Act (621/2011), and Tukes, the Finnish Safety and Chemicals Agency, are the responsible authorities. A mining permit is required before an environmental and water permit can be granted. In Finland, a mining permit covers the whole operation area (not just the deposit) and requires therefore quite advanced project plans. The project area has a valid mining permit K7802.

AFRY Finland Oy has not carried out an assessment or reviewed any existing assessment of liabilities concerning the previous mining. There is historical extractive waste on the site and groundwater has already been polluted in the past. Contamination inventory and risk assessment have been carried out as part of the national "KAJAK inventory", an environmental inventory program for historical mining areas. Anyhow, this type of inventory does not define if responsibilities will be transferred to a potential new operator and to what extent. New operation on the site is unlikely without disturbing historical extractive waste. Liability assessment requires environmental inventory, definition and impact assessment concerning future operations and legal expertise.

A preliminary social and environmental risk assessment has been carried out. Its results are presented in paragraph 20.6. Environmental and social risks.

1.12 Mine closure

In Finland, mine closure requirements are largely based on European best-available technology (BAT) definitions (EC 2018). AFRY underlines the relevance of BAT 5 and post-closure risk and impact assessments – adequacy of a closure plan should be always verified with a post-closure impact assessment. Best closure planning practices are also generally iterative – repeating the same risk and impact assessments in several study phases (ICMM 2019).

In this study, closure requirement considerations are preliminary. Closure requirements are risk-based assumptions, not numerically confirmed solutions. The actual closure plan has not yet been drafted.

Assumed closure measures include the following:

- closing the mine entrance and other openings
- dismantling unnecessary infrastructure
- profiling and covering the tailings storage facility
- dismantling the basal structure of a temporary waste rock storage facility
- dismantling the structures related to the temporary high-S tailings storage
- active water management and treatment for years 1-5 post-closure
- post-closure environmental monitoring
  - regular monitoring for early years after closure and following less frequent monitoring – until year 30 post-closure
- post-closure site maintenance (small-scale cover repairs, road-keeping for monitoring etc.)
  - repairs and snow ploughing for 30 years

Details concerning closure measures and closure cost estimates are presented in the Cost estimation appendix.

1.13 Products

Mine has two major products. Ni-Co-concentrate valuable metals are nickel and cobalt. Cu-concentrate contains also some gold and silver. The estimated Ni-concentrate production capacity will be 16400 tpa (dry) and the copper concentrate production capacity 3754 tpa (dry). Concentrate contents are presented in table 1.8. Figure 1-1 Incomes distribution by metal indicates each metal's importance from an income point of view.

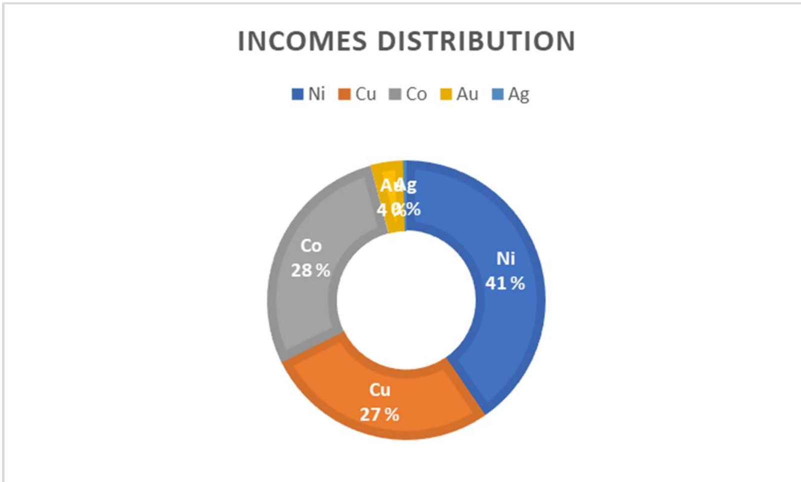


Figure 1-1 Incomes distribution by metal

The following principle for pricing has been used

Metal	Recovery	Payable	Metal price	
Nickel	82%	68%	20 000	USD/t
Copper	86.5%	96% - 65 USD/t	9 750	USD/t
Cobalt	82%	55%	70 000	USD/t
Gold	within conc.	90%	1800	USD/oz
Silver	within conc.	70%	23.05	USD/oz

1.14 Capital cost

Estimated capital costs are presented in Table 1-11 Capital cost summary.

Table 1-11 Capital cost summary

In ‘000 EUR	
Direct costs	**58 572**
Mine	5 455
Concentrator plant	24 675
Tailings and Waste Rock	11 577
Water management	2 221
Site infrastructure	11 636
Mine closure	3 009
Indirect costs	**6 505**
Base estimate	**65 078**
Contingency 15%	9 762
Total investment cost (TIC)	**74 840**

Initial CAPEX is **EUR 65 078 000** (without contingency and including direct costs)

Sustaining CAPEX is **EUR 3 886 000** (without contingency)

Contingency **EUR 9 762 000**

Main sustaining CAPEX items are tailing area costs, mining costs and mine closure. Cost estimate principles are presented in a separate appendix.

The cost estimate is based on the general price level in [Q1/2023]. However, the estimates are subject to change and due to prevailing circumstances the amounts may differ materially from those described in the report.

1.15 Operating cost

Opex summary per cost center is presented in Table 1-12 and Figure 1-2.

Table 1-12. Total Operating Cost Summary

Cost Center	Eur/a	Eur/t (Ore)	% of Total Opex
Admin	1 552 352	3.10	7.5
Mining	10 772 550	21.55	52.3
Concentrator Plant	7 763 856	15.53	37.7
Logistics	489 352	0.98	2.4
Total	20 578 109	41.16	100.0

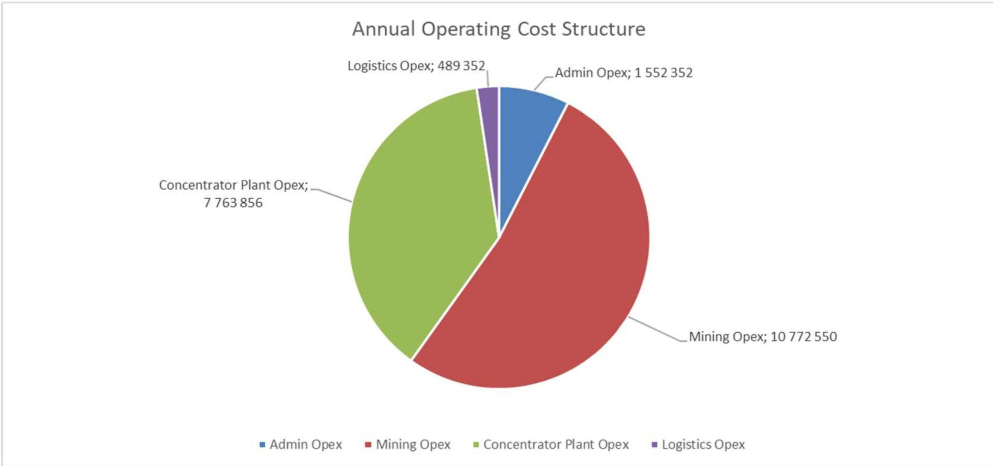


Figure 1-2. Operating Cost Summary.

1.16 Discounted cash flow

The following assumptions have been used on discounted cash flow calculations.

- USD: EUR rate is assumed to be 1:1
- Environmental guarantee is excluded from the calculation
- Post-closure costs are included in the Capex calculation
- Company taxation is excluded from the DCF calculation

Key figures of base case discounted cash flow are presented in Table 1-13 and Figure 1-3.

Table 1-13 Discounted cash flow key figures

Net present value	58.8	MEUR	(5% discount factor)
Net present value	32.7	MEUR	(10% discount factor)
IRR	20.0	%	

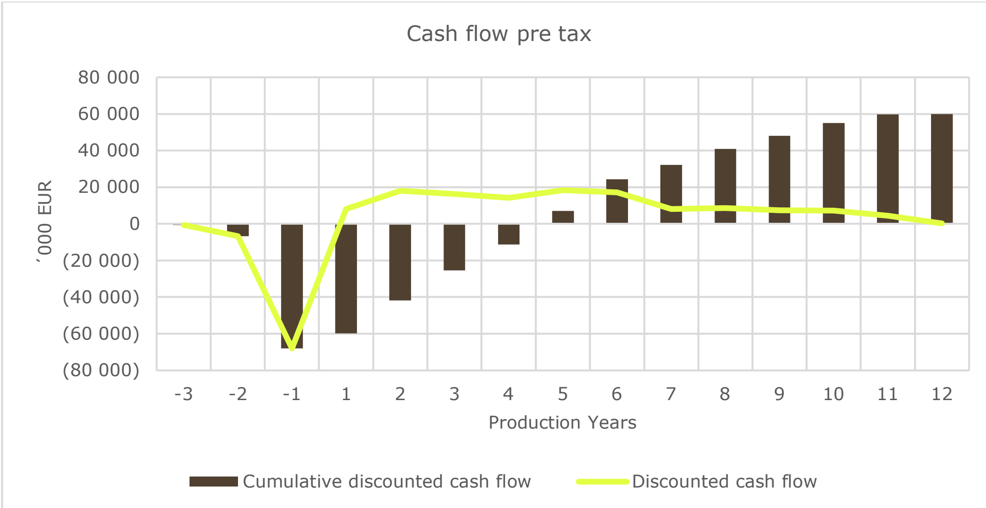


Figure 1-3 Discounted cash flow (discount factor 5 %)

Project sensitivities are presented in figures Figure 1-4 and Figure 1-5.

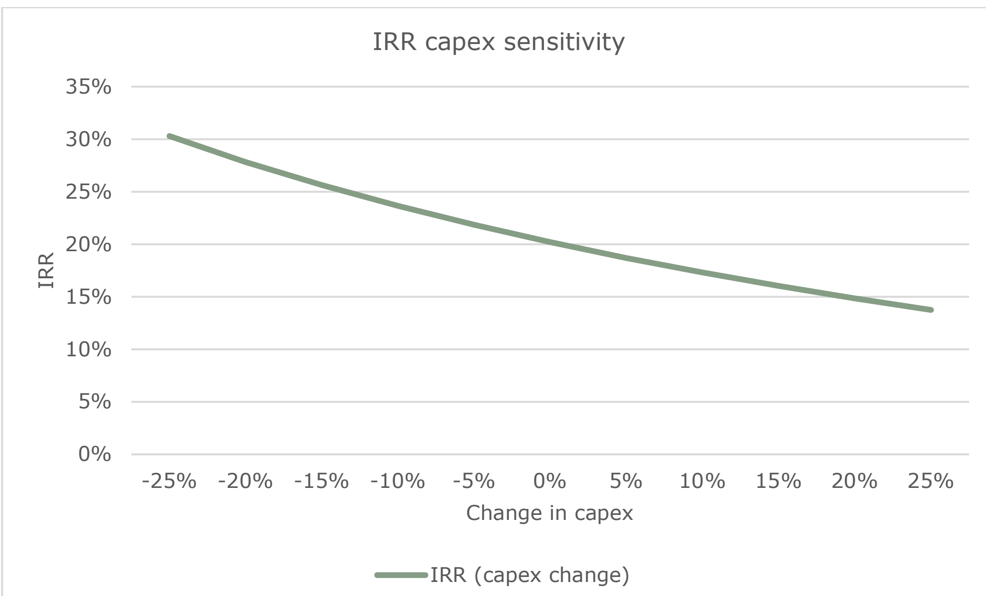


Figure 1-4 Sensitivity (IRR vs Capex)

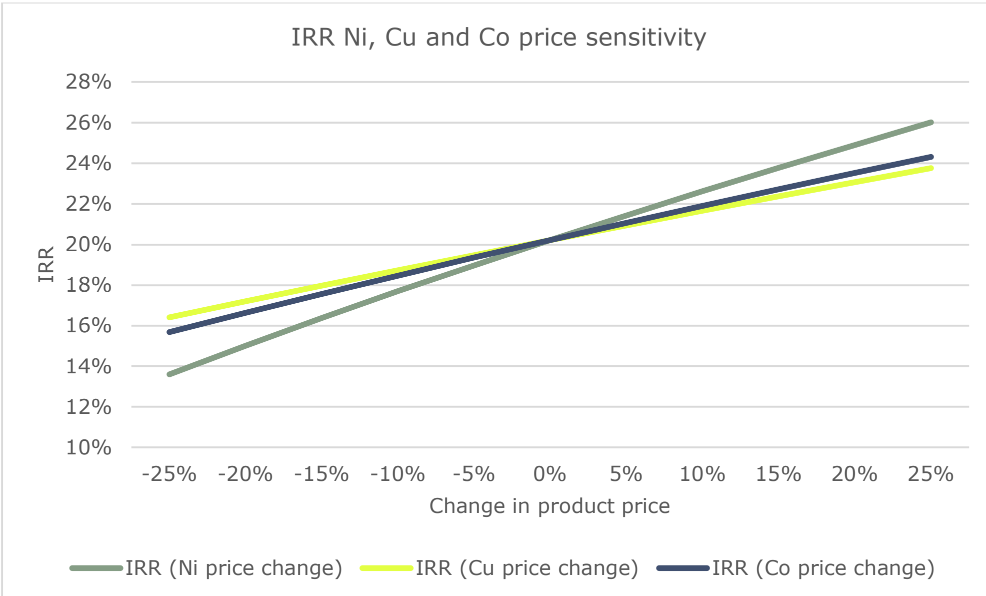


Figure 1-5 Sensitivity (IRR vs Nickel, Copper and Cobalt prices)

1.17 Conclusions

The following remarks and conclusions regarding the Hautalampi project are summarized below:

- The drilling and sampling to date support the mineral resources estimate and there is sufficient information to be used as a basis for the mineral resource estimate.
- The drilling pattern and spacing cover the known measured, indicated and inferred mineral resources. A limited amount of new drilling down-dip of the historic drilling could upgrade the indicated and inferred resources.
- The deposit geology and style of mineralization are well understood, and the property has a history of successful mining activities. However, the Mökkivaara area needs more consideration to upgrade the resource class.
- Based on the mineral resource and ore reserve estimate, the project is well suited to proceed to the next study phase.
- The process is a traditional flotation-based process. No major process-related risks are identified.
- The site is an old nickel mine site with old tailings areas. This will require special attention to tailings methods and permitting.
- The site location is close to an active city, which makes project execution much easier than in more remote locations.

Cautionary Note Regarding Forward-looking Information and Statements

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

In certain cases, forward-looking statements can be identified by the use of words such as "budget", "estimates", or variations of such words or state that certain actions, events or results "may", "would", or "occur". These forward-looking statements are based, in part, on assumptions and factors that may change, thus causing actual results or achievements to differ materially from those expressed or implied by the forward-looking statements. Such factors and assumptions include, but are not limited to, assumptions concerning base metal prices; cut-off grades; accuracy of mineral resource estimates and resource modelling; reliability of sampling and assay data; representativeness of mineralization; accuracy of metallurgical test work and timely receipt of regulatory approvals. Liability assessment concerning the historical extractive waste within the planned mining operation area is not included in the scope of this study and AFRY or CPs do not take any responsibility regarding risk or cost assessments related to the previously mentioned liabilities. AFRY or CPs do not take any responsibility regarding risks on cost estimates that are caused by fluctuations of metal prices, electricity price or other raw material prices.

Forward-looking statements involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements of FinnCobalt to be materially different from any future results, performance or achievements expressed or implied by the forward-looking statements. Such risks and other factors include, among others, fluctuation in the price of base and precious metals; expropriation risks; currency fluctuations; requirements for additional capital; government regulation of mining operations; environmental, safety and regulatory risks; unanticipated reclamation expenses; title disputes or claims; limitations on insurance coverage; changes in project parameters as plans continue to be refined; failure of plant, equipment or processes to operate as anticipated; accidents, labour disputes and other risks of the mining industry; competition inherent in the mining exploration industry; delays in obtaining governmental approvals or financing or in the completion of exploration, development or construction activities. Although FinnCobalt and the author of this Report have attempted to identify important factors that could affect FinnCobalt and may cause actual actions, events or results to differ, perhaps materially, from those described in forward-looking statements, there may be other factors that cause actions, events or results not to be as anticipated, estimated or intended.

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

2 Introduction

AFRY Finland Oy (AFRY) has been commissioned by FinnCobalt Oy (FinnCobalt) to prepare an independent Pre-feasibility study update on the Hautalampi Ni, Cu, Co deposit in compliance with the recommendations of the 2012 Australasian Code for Reporting of Mineral Resources and Ore Reserves (JORC 2012).

This report has an effective date of March 7th, 2023. This report is based on the data collected and prepared for the Technical report for the Hautalampi Co-Ni-Cu deposit in 2009 (Finn Nickel Oy) and on the FinnCobalt drilling campaign from 2020. And also to the work done for the mineral resource estimate dated June 21st 2021 and the previous PFS-study done in 2022.

The mineral resource estimate was completed by Ville-Matti Seppä who is a Competent Person as defined by the Australasian Code for the Reporting of Mineral Resources and Ore Reserves (JORC Code) 2012 Edition.

Mr Seppä visited the site on March 10th, 2021. The inspection included:

- Visiting the historic Keretti mine area.
- Visiting the drill core storage.
- Overall view of the property.
- Inspection of the number of available drill holes.
- Discussions with Markus Ekberg, CEO of FinnCobalt Oy and geologists Kalle Penttilä and Matthias Mueller of FinnCobalt.

AFRY has relied on information provided by FinnCobalt to prepare this report. AFRY has no reason to believe that this information is materially misleading, incomplete, or contains material errors. The content of this report as expressed by AFRY is based on the assumption that all the data provided by FinnCobalt is complete and correct to the best of FinnCobalt's knowledge. Liability assessment concerning the historical extractive waste within the planned mining operation area is not included in the scope of this study and AFRY or CPs do not take any responsibility regarding risk or cost assessments related to the previously mentioned liabilities.

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

3 Reliance on Other Experts

The Competent Person has relied on additional data from:

- The Exploration and Mining Registry (permitting), Finnish Safety and Chemicals Agency

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

- The Competent Person's field observations
- Data, reports, and other information are supplied by FinnCobalt and other third parties.

To the report, CPs have relied on the ownership data provided by FinnCobalt and believe that such data and information is complete and correct. CPs have not completed an extensive property title and ownership search on Hautalampi and express no legal opinion on the ownership status of the property.

4 Property Description and Location

The Hautalampi property is located at Latitude 62.7151 °N, Longitude 28.9730 °E in the Outokumpu municipality, eastern Finland, about 2 km southwest of the town centre of Outokumpu, about 45 km WNW of the city of Joensuu, and about 350 km NE of Helsinki. (Figure 4-1).

Figure 4-1 Hautalampi project location.

The project is in the historic Keretti mine site, previously developed and operated by Outokumpu Oy. The mine site infra apart from the old hoisting tower has been removed. The Hautalampi property was previously known as the Keretti (or Outokumpu) property, which included the Keretti Cu Deposit, mined between 1913 and 1989.

4.1 Mineral rights

The property is covered by FinnCoblt mining concession 7802/1. The total area of the mining concession is 283.5 hectares (Figure 4-2). The mining concession comprises from 114.95 hectares size mining area and 168.55 hectares size auxiliary area.

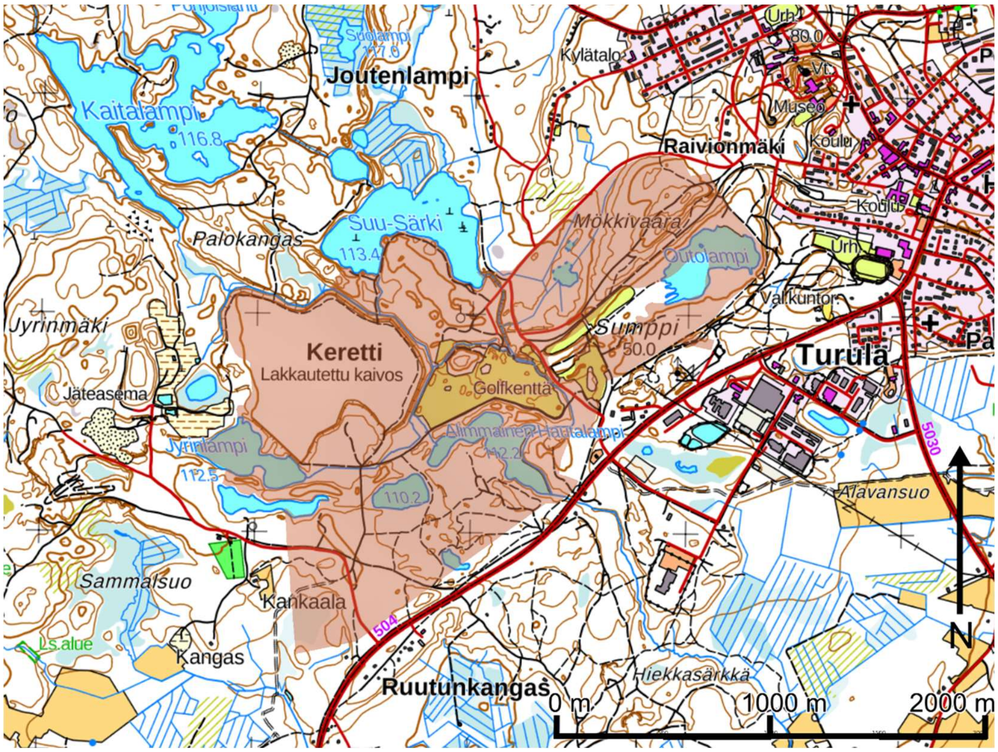


Figure 4-2 Location map of the mining concession relative to the local topography and town of Outokumpu.

Mining concession proceedings have been initiated in accordance with the original application. FinnCobalt has been prepared to update the mining concession area to have a better match to the planned land use. Also, areas not owned by the company would be partly or totally left out.

The final meeting of the mining district commission was held on October 27, 2022, and the compensation to the landowners stipulated in the commission has been paid. The new mining concession is not yet valid as determined by TUKES. Figure 4-2 illustrates the valid mining concession.

5 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

5.1 Accessibility and physiography

The Hautalampi property can be accessed all year round by a tarred road, running through the centre of the claim. The nearest town is Outokumpu, about 2 km to NE. The topography is of generally flat relief, with some low rolling hills, due to remnant outcrop or glacial features such as drumlins and moraines. Elevations range between 110 and 180 m above sea level. The area is a combination of a disused mine site (Keretti mine), forestry, farming, and an urban setting.

5.2 Climate

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

The annual precipitation is approximately 600-650 mm. The amount of precipitation increases towards summer, usually July and August are the rainiest months (Finnish Meteorological Institute, 2012).

Wintertime lasts approximately six to seven months in the Outokumpu region, and snow stays on the ground for 145 to 160 days of the year (Finnish Meteorological Institute, 2012).

5.3 Local Resources and Infrastructure

In terms of potential mining infrastructure, several high-voltage power lines cross the property. Water is readily accessible from the river/stream and nearby lake. The area is a historic mining district with disused mine buildings and tailings areas to the north of the Hautalampi claim. It is envisaged that there would be no shortage of skilled personnel in the region. Vocational school in North Karelia’s Outokumpu unit trains and educates mining professionals.

6 History

There has been no historical production from the Co-Ni-rich mineralised zone, which is the principal target of the Hautalampi property. However, mining was to start at Hautalampi and underground declines and some ore development are already in place (1200 m of decline and ca. 850 m of drifts). However, because of the rapid change in the Outokumpu Company's metal policy, the Co-production was sold and the mining of the Co-Ni ore in the Hautalampi zone was stopped in 1987. The deeper Cu-Zn ore that made up the Keretti mine was mined from 1913 until 1989 (Figure 6-1), over which time some 28.54 Mt of ore was mined, grading 3.8% Cu, 0.24% Co, 0.12% Ni, 1.1 % Zn, 8.9 ppm Ag, and 0.8 ppm Au.

The Hautalampi property was previously held by Outokumpu Mining Oy and was known variously as the Keretti Mine or Outokumpu Mine. Suomen Nikkeli Oy (Finn Nickel Oy) acquired a 100% interest in the property in 2007. Followed by a Finn Nickel bankruptcy in 2009 the mineral rights and the ground together with the Luikonlahti plant were purchased from the Finn Nickel bankruptcy estate by Vulcan Resources Pty Ltd. After Vulcan Resources withdrew from Finland the Hautalampi asset was sold to Vulcan Hautalampi Oy in September 2016. In May 2020 FinnCobalt Oy (formerly Vulcan Hautalampi Oy) agreed on the farm-in agreement with Eurobattery Minerals AB, where Eurobattery agrees to finance the future development of the company and subsequently earns the right to purchase all FinnCobalt Oy shares.

Figure 6-1 Aerial photo from the historical Keretti mining area.

6.1 Historical Reserves and Resources

The first resource estimations for Hautalampi were made by Jyrki Parkkinen in 1987 (Table 6-1) when the "nickel parallel" of the Keretti co-deposit was estimated. The following estimate for Hautalampi ("nickel parallel W") was given:

Table 6-1 Hautalampi Resource estimate (Parkkinen 1987)

Tonnes (Mt)	Ni %	Cu %	Co %	Zn %	Fe %	S %
1	0.55	0.59	0.15	0.08	5.74	3.72

Later in 1997 (Table 6-2) Jyrki Parkkinen made the following estimate:

Table 6-2 Hautalampi Resource Estimate (Parkkinen 1997)

Tonnes (Mt)	Ni %	Cu %	Co %	Au (ppm)	Zn %	Fe %	S %
1	0.47	0.35	0.16	0.15	0.1	6.3	3.9

Parkkinen (1985) calculated mineral resources for the other "nickel parallels" of the Keretti copper deposit as well (Table 6-3). Sections 33–44 were called the Raivionmäki formation and sections 44-57 Mökkivaara formation.

Table 6-3 Historical Mineral Resources of the Keretti area (Parkkinen 1985)

Sections	Area	Tonnes	Ni %	Cu %	Co %	Zn %	Fe %	S %
33-44	Raivionmäki	1 000 000	0.3	0.11	0.05	0.07	4.1	2.2
44-57	Mökkivaara	556 750	0.34	0.04	0.02	0.06	4.4	3.4
33-57	Rai + Mök	1 556 750	0.31	0.08	0.04	0.07	4.2	2.6

In addition, Parkkinen (1985) calculated the exploration potential for the "nickel parallel E" (Table 6-4), which was met at the 100 – 200 m level. It was estimated as more dispersed and of lower grade than the nickel parallel W. The calculation by Parkkinen (1985) gives the following resources for the nickel parallel E (sections 53–57 + 37–44):

Table 6-4 Mineral Resources for the "nickel parallel E" (Parkkinen 1985)

Tonnes Mt	Ni %	Cu %	Co %	Zn %	Fe %	S %
1.13	0.4	0.11	0.06	0.01	3.6	2.6

The mineral resources of the Hautalampi Deposit (profiles 89–103) are calculated In the Technical Report (Meriläinen et al. 2006) (Table 6-5), Property

Portfolio of Suomen Nikkeli Oy (Finn Nickel Ltd.) in Southern Finland, prepared 1st October 2006 for Belvedere Resources Ltd.

Table 6-5 Hautalampi Mineral Resources as of 1st October 2006 @ NSR 30€/ ore tonne cut-off.

Resource class	Tonnes Mt	Cu %	Ni %	Co %
Indicated	1.18	0.49	0.48	0.12

Resource class	Tonnes	Cu %	Ni %	Co %
Inferred	50 000	0.24	0.38	0.08

In the Technical Report, NI 43-101 Technical Report for the Hautalampi Co-Ni-Cu Deposit at Outokumpu, Eastern Finland (Meriläinen et al. 2008) (Table 6-6) reported the following mineral resource for Hautalampi:

Table 6-6 Hautalampi Mineral Resources as of 1st October 2008 @ 0.30% Ni cut-off.

Resource class	Tonnes	Ni %	Cu %	Co %	Fe %	S %
Measured	837 544	0.483	0.489	0.12	4.281	2.411
Indicated	869 250	0.431	0.306	0.105	4.098	2.366
Total M+I	1.71 Mt	0.46	0.4	0.11	4.19	2.39

The most recent Mineral Resource estimate was prepared by Outotec Oyj (Finland) Oy by Markku Meriläinen with an effective date of March 15th, 2009 (Table 6-7). The resource estimate is in compliance with the Canadian Securities National Instrument 43-101 Standards of Disclosure for Mineral Properties and Form 43-101F1.

Table 6-7 Hautalampi Mineral Resource as of 15th March 2009 @ 0.3% Ni cut-off

Resource class	Tonnes	Ni %	Cu %	Co %	Fe %	S %
Measured	1 030 000	0.47	0.47	0.13	4.71	2.65
Indicated	1 226 000	0.42	0.3	0.12	3.87	2.81
Total M+I	2 256 000	0.44	0.38	0.12	4.25	2.74
Inferred	895 000	0.4	0.3	0.1	3.6	2.9

Below (Table 6-8) is presented the Hautalampi Reserves estimated by Outotec Oyj in 2009. The ore reserves are not additional to the mineral resources in Table 6-7.

Table 6-8 Hautalampi Mineral Reserve as of 15th March 2009 @ NSR 30€/ ore tonne cut-off.

Reserve Class	Tonnes Mt	Ni %	Cu %	Co %	Fe %	S %
Proven	0.94	0.42	0.41	0.11	4.23	2.37
Probable	1.28	0.36	0.25	0.09	3.23	2.47
Total	2.22	0.38	0.32	0.1	3.66	2.43

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

6.2 Previous Estimates

Hautalampi Mineral Resources were previously published on June 21st 2021 and it was completed by AFRY Finland Oy.

After the 2021 mineral resource estimate FinnCobalt Oy completed a drilling program of 45 new diamond drill holes.

The 2021 Mineral resource estimates at Hautalampi and Mökkivaara are presented below (Table 6-9, Table 6-10 and Table 6-11).

Table 6-9Hautalampi Mineral Resources as of the June 21st, 2021 @ 0.3% NiEq cut-off

Hautalampi	Tonnes (t)	Ni %	Cu %	Co %	Ni Eq %	Cu Eq %
Measured	2 582 000	0.38	0.28	0.08	0.72	1.67
Indicated	2 701 000	0.31	0.20	0.08	0.61	1.42
total M&I	5 283 000	0.35	0.24	0.08	0.66	1.54
Contained Metals	tonnes	18289	12783	4337		

Table 6-10 Hautalampi Inferred Mineral Resources as of the June 21st, 2021 @ 0.3% Ni Eq cut-off

Hautalampi	Tonnes (t)	Ni %	Cu %	Co %	Ni Eq %	Cu Eq %
Inferred	195 000	0.26	0.14	0.05	0.45	1.04
Contained Metals	tonnes	505	267	98		

Table 6-11Mökkivaara Inferred Mineral Resources as of the June 21st, 2021 @ 0.3% Ni Eq cut-off

Mökkivaara						
	Tonnes	Ni %	Cu %	Co %	Ni Eq %	Cu Eq %
Inferred	2 186 000	0.25	0.16	0.06	0.46	1.07
Contained Metals	tonnes	5410	3509	1218		

7 Geological Setting and Mineralization

7.1 Regional Geology

The following is summarized from the Hautalampi feasibility study 2009.

The geological setting of the Hautalampi mineralisation is the same as that for the main Keretti Cu-rich ore, the main differences being in the localisation of the mineralised zone within the Outokumpu stratigraphy, and the nature of the mineralised body itself.

The Keretti deposit is located within the NE trending ca. 2 km wide horizon of black schists and serpentinite bodies that are defining the western margin of the Outokumpu structure (Figure 7-1 and Figure 7-2), and which is commonly called the "Outokumpu belt". The deposit is found in association with a long (>10 km), tubular (<1.2 x <1.5 km in cross-sections) body consisting of tightly folded serpentinite, located along its NW margin in a few metres to tens of metres layer of carbonate-skarn-quartz rocks that are enveloping and being folded with the serpentinite. Unfolded the serpentinite tube is found to consist of a ca. 150-200 m thick, possibly 5 km wide and >10 km long sheet, the thickness and width estimated for the thickest part of the tube. The carbonate-skarn-quartz enveloped, folded serpentinite tube is enclosed in the Upper Kaleva metagreywackes, with usually a few metres to a couple of tens of metres thick layers of black schist in between.

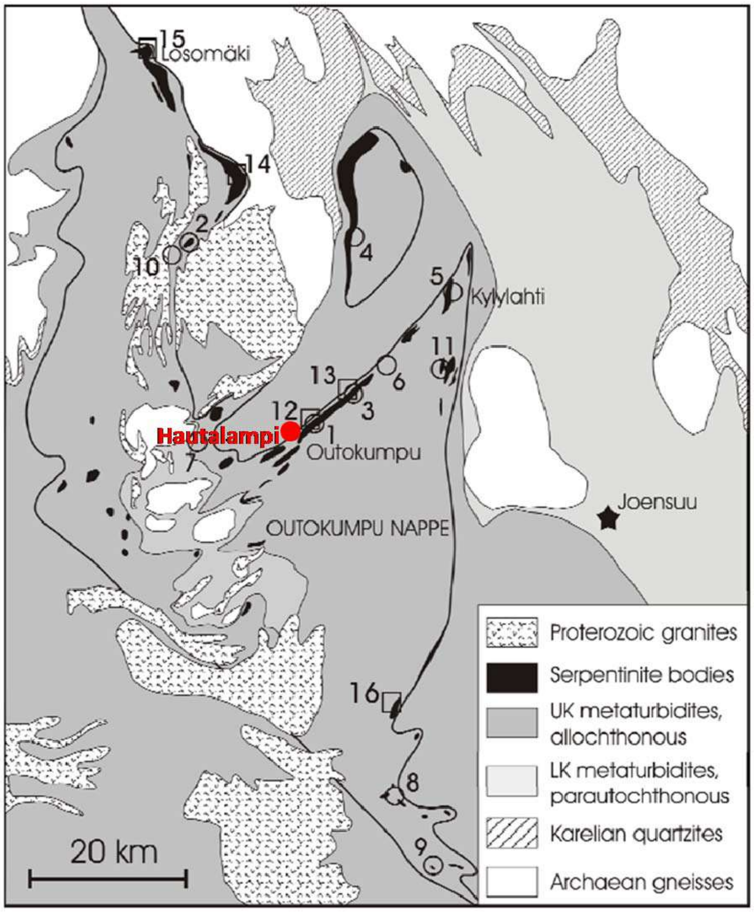


Figure 7-1 Map of the Outokumpu-type sulphide deposits in North Karelia

The Outokumpu serpentinite massif comprises very few other components but serpentinite. Pervasively chloritised and metamorphosed obvious mafic dykes occur locally, but they are nowhere abundant, comprising far less than 5 % of the total volume of the massif. The serpentinites are retrogressively serpentinitised (lizardite-chrysotile) metaperidotites, usually talc-olivine rocks in the middle part, and anthophyllite-enstatite-olivine to olivine-enstatite-carbonate rocks at the margins of the massif. The mineral assemblages of the metaperidotites and olivine-spinel thermometry indicate peak metamorphism in temperatures above 630 °C. Thermobarometry for garnet-cordierite-orthoamphibole rocks has yielded similar peak temperatures at ca. 3-4 kbar pressures.

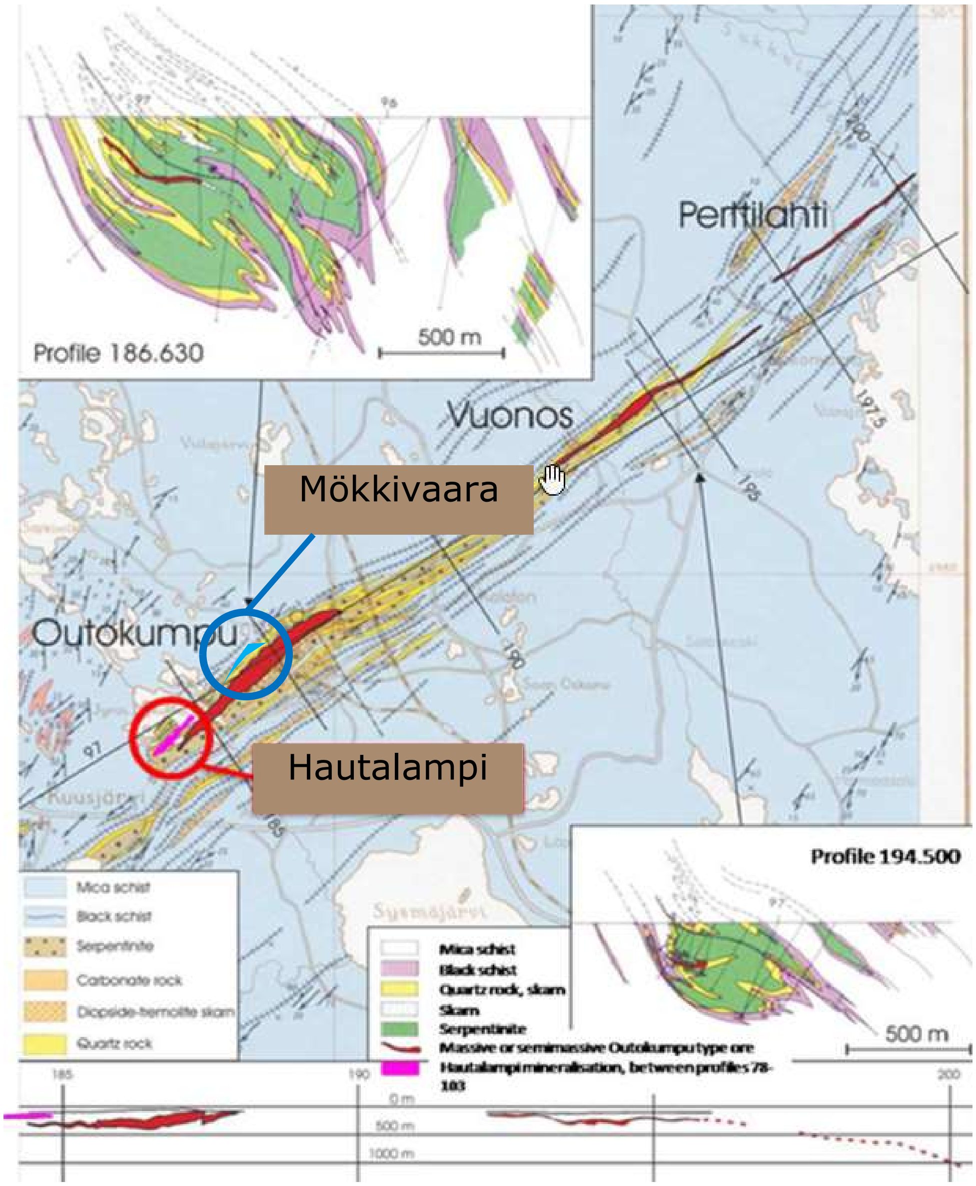


Figure 7-2 Geological map of the Outokumpu belt. Modified from GEOMEX report (2006) and Feasibility Study (2009)

The NW edge of the serpentinite tube shows several shallowly to SW plunging and 20-50° SE dipping isoclinal F1 folds. The Keretti ore plate is apparently enclosed for its entire length inside one of the F1 folds. In vertical cross-sections approximately perpendicular to the F1 axis, the ore plate is seen to broadly follow the upper limb of the host fold, defined by the contact between serpentinite and fringing carbonate-skarn-quartz rocks. In a detailed investigation of the cross-profiles it is seen that, in many of them, the ore sheet truncates the serpentinite carbonate-skarn-quartz sequence, implying that the final emplacement of the ore has to post-date the carbonate-silica alteration of the serpentinite body margins.

The Keretti and Vuonos ores are often said to be hosted by the quartz rocks or quartzites in the Outokumpu assemblage. However, this is an oversimplification. Although much of the footwall of the Keretti ore plate is generally against quartz rocks, some parts are found partly or completely enclosed in serpentinite. In addition, the hanging wall is mainly in direct contact with serpentinite and skarn–carbonate rock, and parts even in the mica and black schists. Also, the contacts of the ore with the quartz rocks, as well as the other wall rocks, are frequently very sharp and intrusive-like, suggesting an epigenetic relationship. This is further supported by the fact that ore material often brecciated the strongly shear-banded wall-rock.

7.2 Mineralization

The lower edge of the Co-Ni-Cu-mineralisation zone is typically some 150 to 200 m above and a bit to the NW of the upper edge of the main Keretti Cu-ore. Dimensions of the modelled Hautalampi mineralised zone are approximately 1000 m in length, 100-150 m in width, and 1-30 m in thickness. Some drill holes indicate that in the NW parts the mineralisation is cut by the present erosion surface. Mineralisation has a 10 - 55° dip to the SE (on average about 25-30°). The main part of the mineralisation is 70-120 m below the surface and the deepest parts of the known mineralisation are about 150 m below the surface.

Mökkivaara mineralisation is located approximately 650 meters northeast of the Hautalampi mineralisation and it has the same overall strike and dip as the Hautalampi mineralisation. More work is needed to gain confidence in the geological setting of Mökkivaara. Old interpretations suggest that the mineralisation is in synform but according to the latest drillings, the data supports an antiform structure.

The Co-Ni-Cu mineralisation, (also referred to as the Hautalampi mineralisation), consists of tightly folded metamorphic rocks. Host rocks are mainly quartz rocks with anthophyllite-tremolite skarn bands and interlayers with variable amounts of chlorite. In some places, the mineralised zone is also hosted by skarniferous dolomitic rocks. Minor diopside can occur with other skarn minerals. In places, there is also nickel-bearing black schist or black schist-bearing quartz rock in the footwall. Mineralisation mainly occurs as disseminations in bands due to metamorphosis. The mineralised zone has in places a very sharp contact with the wall rocks. However, in many places, a transitional zone from one meter up to three meters occurs between the mineralised zone and wall rocks.

Chlorite schist is locally rich in garnet and also minor cordierite is present. Garnet and cordierite occur as porphyroblasts. Phlogopite occurs in quartz rocks and it seems to be an alteration product of amphiboles. Also, cummingtonite, staurolite, and spinel are mentioned in the GEOMEX report. Chromite and its alteration products, ferrian chromite and magnetite, are present in almost all the host and wall rocks, especially in rocks that are rich in quartz and dolomite. Serpentinites contain thin magnetite bands and magnetite grains are typical.

The hanging wall rock is mainly serpentinite and quite often also quartz rock and dolomite with or without diopside-tremolite skarn bands or interlayers. Footwall rocks are quite often the same due to folding. Rock types vary a lot through a drill hole, especially between skarn-, skarniferous quartz and quartz rocks. A simplified geological cross-section through profile 92 is presented in Figure 7-3.

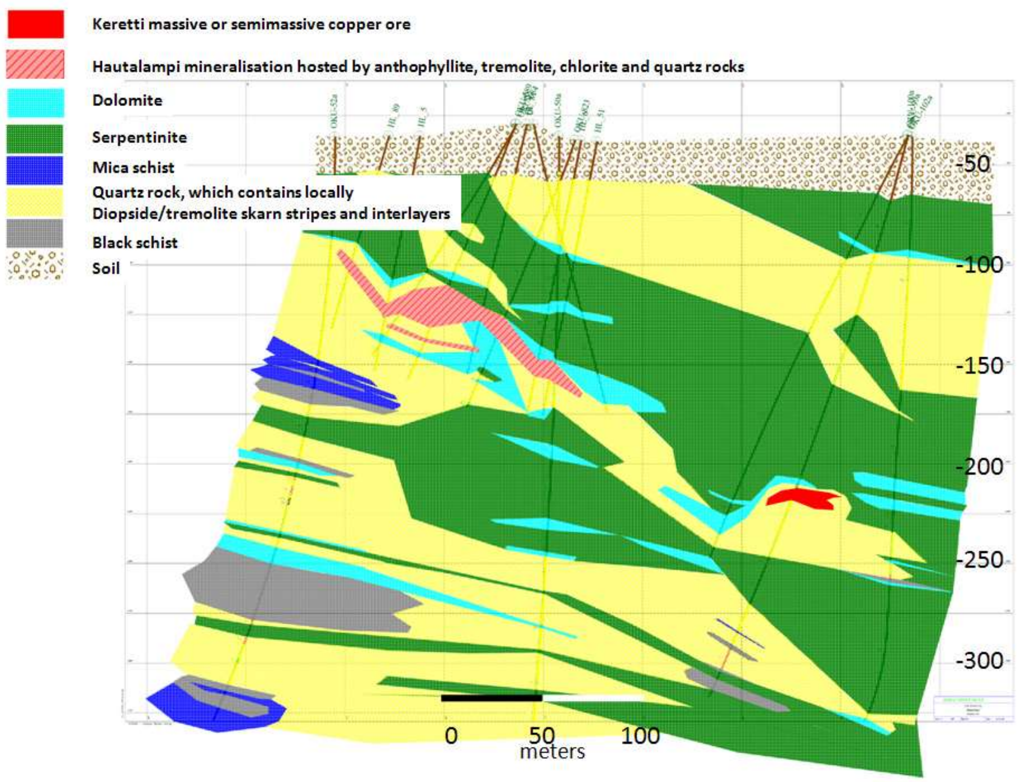


Figure 7-3 Cross-section through profile 92 of the Keretti copper (Outokumpu) and Hautalampi deposits.

8 Deposit types

The Hautalampi mineralised zone is the south-westernmost part of the Co-Ni-Cu-mineralisation zone, which is situated within the hanging wall roughly parallel to the Outokumpu Keretti Cu-ore body. It belongs to the "Outokumpu-type" deposits within the rock associations of the Outokumpu Formation.

The Co-Ni-Cu zone has some aspects that are distinct to the main Cu ore environment. One is the frequent occurrence variably cummingtonite, anthophyllite, cordierite (usually extensively pinitised), staurolite, garnet, phlogopite, and spinel-bearing chlorite-rich rocks/schists, hosted as thin layers (usually < 1m) or patches in skarn (diopside-tremolite)-quartz rocks forming the bulk of the Co-Ni-Cu zone. Another distinct feature is the abundance of often very coarse-grained, usually highly zincian chromite in almost all the rock types in the zone. And a third one is the relative cobalt-nickel enrichment of the included sulphide mineralisation (modified from the GEOMEX Report and references therein).

It was earlier thought that the Hautalampi zone represents a feeder zone for the main Keretti Cu-ore. According to the now widely accepted Geomex model the silicate nickel was transformed to the sulphide fraction during the obduction and adjacent carbonate-quartz alteration of the seafloor around 1.9 Ga. After that, during the areal deformation phases D1-2, the Ni-bearing sulphides were remobilised and recrystallised. It is important to note that according to both models, the nickel-enriched zone was made before the folding. Consequently, understanding the fold structures at Hautalampi is important in trying to follow the mineralised zone.

9 Drilling

9.1 Historical Drilling

The earliest drillings in the Co-Ni enriched zone nearby the Keretti Cu ore were made by Outokumpu Oy already in the 1930s connected with the drillings of the Cu ore. Later during the 1950s and 1960s drillings were focused on the Co-Ni enriched zone, including the Hautalampi area. In 1979 a drilling program for inventing the mineral resources in the Co-Ni enriched zone was commenced. Still, in 1984 a new drilling campaign was made. All together around 40 km was drilled (Figure 9-1). Some of the holes however include partly Keretti Cu-ore drilling.

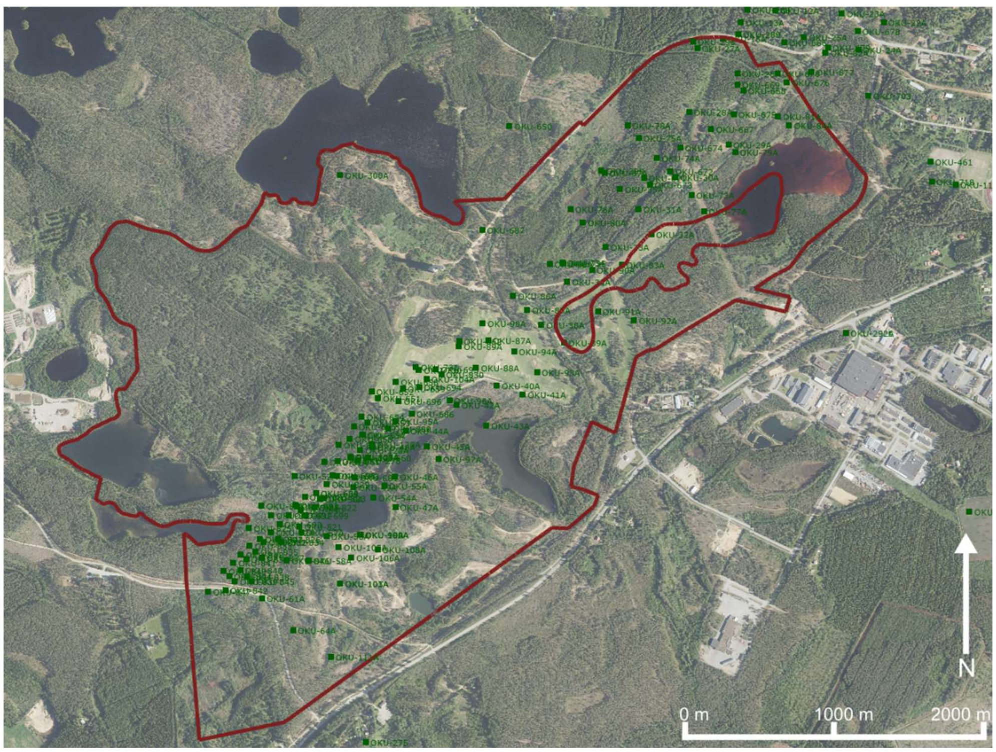


Figure 9-1 Overview map of the Outokumpu Oy drillings in relation to the applied mining concession.

9.2 Finn Nickel drilling 2007–2008

Between years Finn Nickel Oy drilled 92 drill holes (Figure 9-2) totalling 10 120.45 meters. The target of Finn Nickel's drilling program was to confirm the continuity of the mineralised zone. Diamond drilling and surveying (dip measuring) of the boreholes were contracted to Suomen Malmi Oy (SMOY). The collected drill core was 42 mm in diameter excluding three holes that were drilled for flotation tests using a 62 mm core diameter.

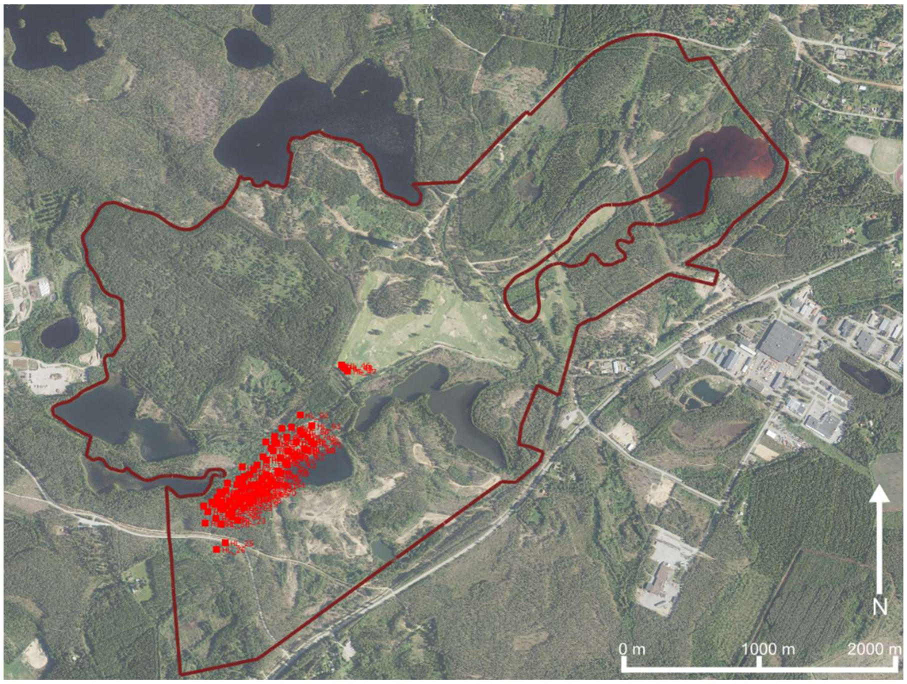


Figure 9-2 Overview map of the 2007-2008 Finn Nickel Oy drilling in relation to the applied mining concession.

9.3 FinnCobalt (Vulcan Hautalampi Oy) 2017-2018

A total of eight holes (Figure 9-3) with 993.7 meters of drill core were drilled for metallurgical testing. The collected sample diameter was 62 mm.

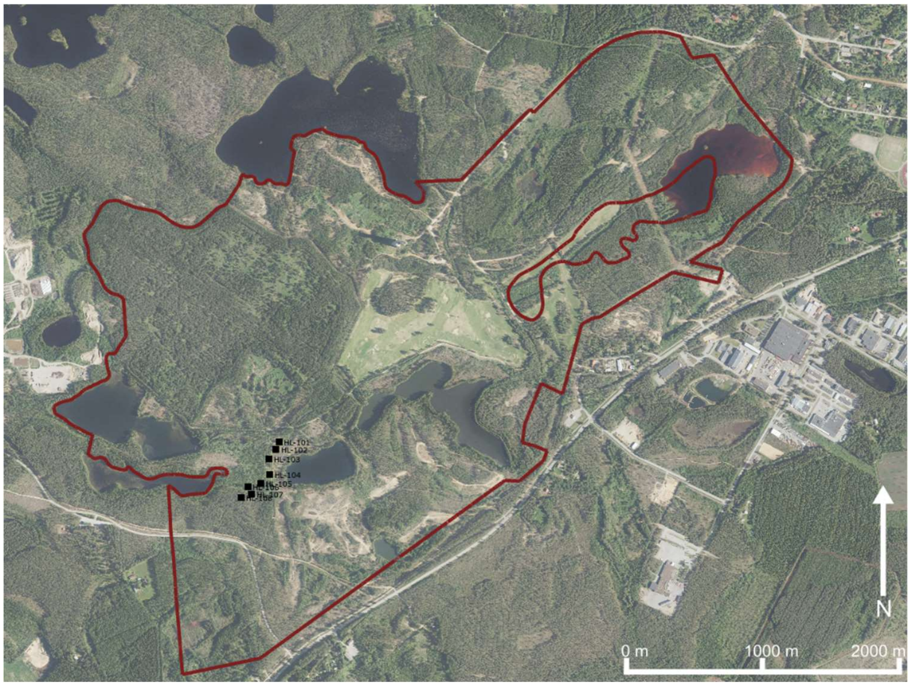


Figure 9-3 Overview map of the 2017-2018 FinnCobalt (Vulcan Resources) drilling in relation to the applied mining concession.

9.4 FinnCobalt Oy 2020 and 2021

A total of 29 (Figure 9-4) drill holes were drilled between 13.07.2020 and 26.09.2020 comprising 3768.0m of drill holes. The diamond drilling was carried out by the Finnish contractor company Northdrill Oy using WL-76 equipment (57.5 mm sample diameter). The samples were oriented. Before drilling, all drill holes were marked in the field with a DGPS, supplied by Northdrill Oy, allowing maximum collar accuracy. Once drilling was finished at a hole, the entire drill hole was surveyed for its azimuth and dip deviation, using a Devico DeviFlex instrument. The most recent drilling campaign comprises 45 drill holes and a total of 4613.2 meters. The drilling was contracted by Northdrill Oy and the used equipment size was WL-76.

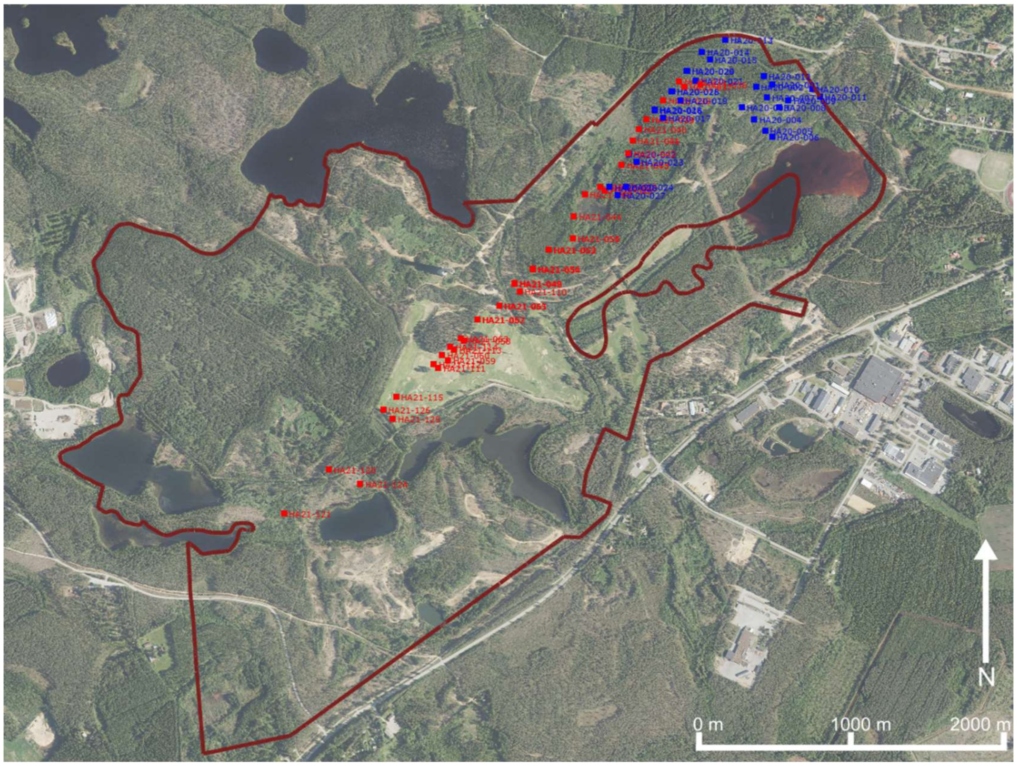


Figure 9-4 Overview map of the 2020 and 2021 FinnCobalt drilling in relation to the applied mining concession (Blue=2020, Red =2021).

10 Sample Preparation, Analyses and Security

10.1 Outokumpu Oy

Detailed information on the procedures from the Keretti Mine underground drilling, 1950 – 1986, and from Outokumpu Oy Exploration 1950–1987 program, are missing. Apart from the assaying methodology, the sample preparation and security measures are unknown. The detailed QA/QC procedures of the historical database (drilled 1961 – 1989) are not known in detail. However, this work was carried out by the large mining and exploration company Outokumpu, for their internal use, it is believed that the work was carried out to industry standards and Outokumpu exploration practices for that time and is believed to be reliable. A sampling of the drill core would have been undertaken by company-authorized professional personnel. In accordance with good and established industry practices. The Competent Person has no reason to believe that the documentation provided is misleading in any way.

10.2 Finn Nickel drilling 2007–2008

Sample preparation was made in Outokumpu town by Okun Autolähetti Oy which is now owned by ALS Chemicals Ltd. Each core was halved with a diamond saw with one half prepared for analysis. The remaining half-core was retained for verification and reference purposes. For assay samples, density measurements were done before sawing. Except for the first 25 drill holes, density was measured for all samples. The samples were dried and crushed in an Mn steel jaw crusher to a grain size of > 70 % < 6 mm. Instruments used in crushing were Retsch or Rocklabs. The sample was then pulverized to a grain size of > 85 % < 75 µm (Essa LM5). A 150 g subsample was taken for assays. Figure 10-1 illustrates the used sampling protocols and used QAQC methods.

Assaying was made in two different laboratories. The first 49 drill cores (holes HL-1 to HL-49) were analyzed in Kuopio by Labtium Oy. The Labtium laboratory has been accredited since 1994 according to the SFS-EN ISO/IEC 17025 standard to perform chemical analyses of geological samples. The quality system of the laboratory complies with the requirements of the Standards Council of Canada (CAN-P- 1579) "Guidelines for Accreditation of Mineral Analysis Testing Laboratories". The quality assurance- quality control (QAQC) program of Labtium Oy inserts into every batch of 50 samples, two standards, one blank, and three laboratory duplicates.

Base metal analyses were made using the following procedures (Labtium method code 510P): 0.15 g subsample is digested with 2.25 ml of aqua regia (3:1 mixture of concentrated hydrochloric acid and concentrated nitric acid) by heating at 90°C in an aluminium heating block for 1.5 hours and diluted to 15 ml with water. The solution is diluted with water before instrumental analysis. The following elements were analyzed in this method: Ag, As, Cd, Co, Cr, Cu, Fe, Mn, Mo, Ni, Pb, Sb, Zn and S. Instrument used was an "Inductively Coupled

Plasma Optical Emission Spectrometer Thermo Electron iCAP 6600 Duo View with Cetac ASX-520HS Autosampler".

Gold was analyzed systematically from all samples in the first 49 drill holes. After that, samples for gold analysis were selected after the results of base metal assays from the Luikonlahti laboratory. Only samples above the cut-off value and one sample outside from both sides of the ore intersection was selected. The samples were analyzed at Labtium using their method 521U. This method includes the following procedures:

5.0 g subsample is leached with 15 ml of aqua regia at room temperature for 16 hours. After dilution, the analytes are separated and pre-concentrated from the matrix by using Hg-coprecipitation and stannous chloride as reductants. Analysis of Au is carried out by Graphite Furnace Atomic Absorption Spectrometry, Perkin Elmer Analyst 600 equipped with AS-600 Autosampler, or Perkin Elmer SIMAA 6000 instruments equipped with AS-72 Autosampler.

Platinum and palladium were also analyzed from six selected drill holes. Holes were selected to cover the whole research area and especially areas of high gold content. Analyses were made in Labtium's Rovaniemi Laboratory using their high-precision classical Pb fire assay method 703P/704P. These methods include the following procedures:

12.5 g (703P) or 25 g (704P) subsample is weighted depending on nickel content. The high nickel content in the sample needs a smaller subsample. After that, the sample is smelted with the help of some flux material (among other things borax, soda, and silica). When fusion is completed, the sample is leached at 70°C with aqua regia and analyzed using ICP-AAS as in method 510P, described above.

The rest of the drill core samples (holes HL-50 to HL-93) were analyzed in Finn Nickel Oy's laboratory at Luikonlahti. The following elements were analyzed: Cu, Co, Ni, Zn, and S. The following procedures were used:

0.5 g subsample is leached at 100°C with 20 ml of aqua regia (3:1 mixture of concentrated hydrochloric acid and concentrated nitric acid) for one hour. After leaching, 50 ml of distilled water is added and then boiled followed by cooling and dilution with water in a volume of 250 ml.

Analysis of Cu, Co, Ni, and Zn is carried out by Graphite Furnace Atomic Absorption Spectrometry, Perkin Elmer 1100B instrument. For quality control, samples analyzed both in Labtium and at Luikonlahti were used. Every thirteenth sample was a control sample.

Sulphur was analyzed by S-analyzer. This instrument was used also for samples, which had Sulphur content over 5 % in Labtium's 510P assay result (ICP assay is unreliable for Sulphur contents greater than 5 %).

Sulphur assaying procedures in Labtium:

Method codes are 510P, which is described above, and 810L, which is more accurate, especially for higher Sulphur grades. Method 810L is used in check analysis for Luikonlahti assays. In method 810L the following procedure is used:

The sample is weighed in a combustion boat on an electronic balance which is interfaced with the PC. By pressing a key, the sample weight is transferred to the PC. If required, the sample weight can also be entered manually. Subsample weight is usually 100-200 mg. The ceramic boat with the sample is placed on the furnace platform. The start key is pressed, and the analysis cycle begins. The sample is pushed into the furnace at 1400 °C temperature. Infrared detectors are used to analyze the amount of sulphur. At the end of the cycle, the assay results appear on the PC screen. Instrument is ELTRA CS-500.

Sulphur assays at Luikonlahti are made using the same procedures as at Labtium and with the instrument (ELTRA CS-530). The calibration samples used are BaSO4 and NIST 1633b. For quality control, standard samples from Geostats Pty Ltd, Australia were used. At least every 10th sample was calibration or a control sample.

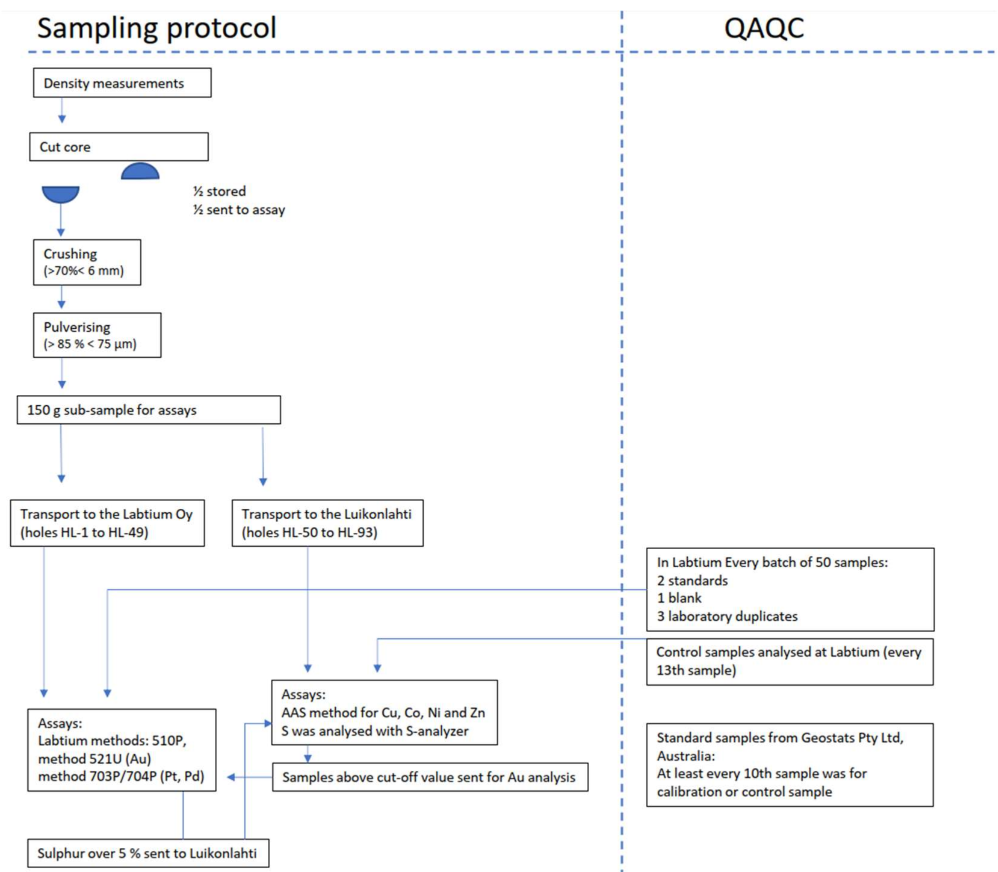


Figure 10-1 QAQC and Sampling protocol for Finn Nickel drill cores

10.3 FinnCobalt Oy 2020 to 2021

Sampling protocols and QAQC measures are well documented from the year 2020 drilling campaign. The following chapters describe the logging, sampling, and QAQC protocols that were used. The protocol is illustrated below in Figure 10-2.

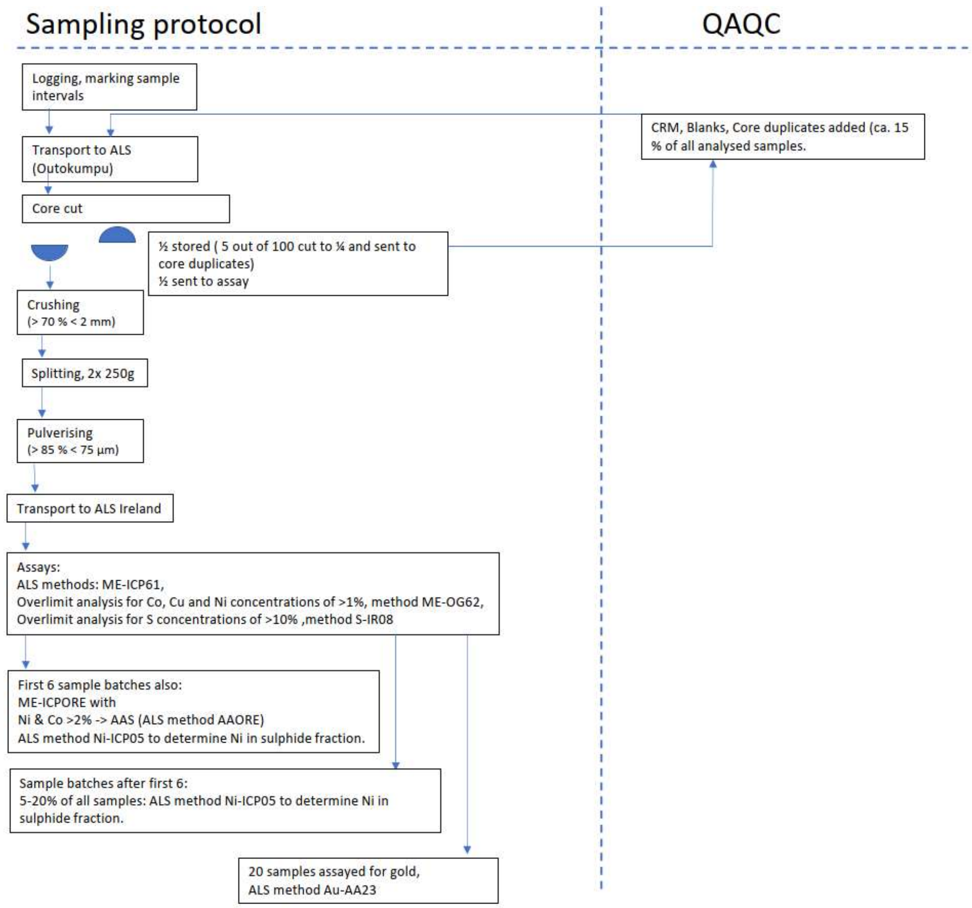


Figure 10-2 QAQC and Sampling protocol for FinnCobalt Oy's drill cores

10.3.1 Logging and sampling protocol

Logging

- Before logging, the drill core pieces were organised, the core metered, and an orientation line was drawn where possible. The orientation line is facing downwards on the core. The orientation line is drawn in blue colour and is dashed if the orientation is uncertain.
- Logging was performed on a field laptop in Microsoft Excel templates. Logging information was subsequently imported to a Microsoft Access drill hole database. Logged information included lithological intervals (from and to), rock type, colour, foliation, grain size,

texture, degree of weathering and fracturing, mineral constituents, alteration, and relative abundance and style of ore minerals.
- Logged lithologies including intervals and meter marks are marked on the core boxes.
- Magnetic Susceptibility readings (S.I.) were taken every 1m.
- Specific gravity measurements were taken before and after every lithological change and every ~3 metres in homogenous rock. The Archimedes method (submersion in water) was used.
- All logging was conducted by a trained geologist (Figure 10-3).

Figure 10-3 Geologist Kalle Penttilä logging drill core at FinnCobalts logging facility and warehouse in Outokumpu.

Sampling

- Sample intervals are based on geological contacts and/or degree of mineralisation.
- The maximum sample interval is 2.0m, and the minimum sample interval is 0.2m.
- The “barren” rock enveloping mineralised intervals is sampled for at least 4m on each side (e.g., 2x2m intervals on each side).
- Sample intervals start and end at core loss.
- All sampling was conducted by a trained geologist.
- Sample intervals and sample numbers are noted on the core boxes.
- After marking of sample number and sample intervals on the core boxes the drill core within the core boxes is photographed, dry, and wet.

- The drill core boxes were subsequently submitted to ALS for sample preparation and analysis.
- For assaying the core is sawn and half of the core is analysed. The remaining ½ core remains for archive purposes, except for samples where core duplicate QAQC is performed, here half of the ½ core is analysed and ¼ core remains for archiving.
- Control samples (CRM, blank, and core duplicates) were inserted and submitted together with normal samples and were analysed. Control samples represented ~15% of all for analysis submitted samples (5% CRM, 5% blanks, 5% core duplicates). One control sample follows five normal core samples -> 1 out of 6 samples is a control sample. The type of control sample rotates within the batch.

QAQC

As part of the FinnCobalt QAQC protocol coarse blanks, standards, and core duplicates were inserted into the regular samples at a rate of 15:100 (1 out 6 samples is a QC sample). An overview of the QC samples is represented below (Figure 10-1).

Table 10-1 Overview of the QC samples in FinnCobalts 2020 drilling campaign (total samples submitted = 1780)

Type	Material	Number of insertions	Insertion frequency
Gade CRM	Oreas 13b	57	3.20 %
Grade CRM	Oreas 680	44	2.47 %
Coarse blank	ALS wash rock	105	5.90 %
Core duplicate	Quarter drill core	92	5.17 %

Blanks

- All utilized blanks were coarse blanks.
- Each batch started with a coarse blank.
- 5 out of 100 (5%) of all samples submitted to ALS were coarse blanks.
- The blank material was put into a sturdy plastic bag together with a sample ID and was sealed. The sample ID was also written on the sample bag. The blanks were submitted to the laboratory together with the drill core.

The coarse blank rock material was sourced from *Savon Kuljetus Oy* via *ALS* Laboratories. The blanks are produced in a quarry near Joensuu and are also used by *ALS* as a "wash rock" in-between runs. *ALS* is periodically analysing the rock material every month to ensure its homogeneity and barrenness. Based on *ALS*'s analysis (Table 10-2) the rock is suitable to act as a blank for *FinnCobalt's* assay purposes.

Table 10-2 ALS analyses of the blanks. Gold is analysed with ALS code Au-ICP21 (Au 30g FA ICP-AES Finish) and all other elements with code ME-MS61 (48 elements four-acid ICP-MS)

Analysis	Au ppb	Ag ppm	As ppm	Co ppm	Cr ppm	Cu ppm	Fe %	Mg %	Ni ppm	Pb ppm	S %	Zn ppm
Apr 20	<1	0.03	0.5	3.8	7	12.8	1.47	0.4	3.6	14.7	0.03	42
May 20	1	0.02	0.3	3.5	7	12.6	1.43	0.37	3.7	13.3	0.03	37
Jun 20	<1	0.03	0.6	3.4	7	14.2	1.55	0.4	3.3	12.9	0.02	41

The regular submission of blank material was used to assess potential contamination during sample preparation and to identify possible drifts in assay results over time. Figure 10-4 shows the FinnCobalt assay results of the utilized blanks throughout the 2020 drilling campaign. The results indicate that good precision was present and no inherent drift in assay results can be observed. Only in one sample, for Ni, contamination can be interpreted (in VHO009). However, since the utilized coarse blanks are not certified, the Ni spike might also represent a higher Ni content in the source rock. Furthermore, the Ni spike only comprises an additional ~30ppm Ni, which is well within the acceptable range.

Figure 10-4 Coarse blank analysis results of the 2020 FinnCobalt drilling campaign, sorted by submitted sample batch in chronological order.

Certified Reference Material (CRM)

- 5 out of 100 samples (5%) of all samples submitted to ALS were standards (CRMs).
- The CRMs were sourced from OREAS, Australia.
- Two different CRMs were utilized – one for high-grade mineralized zones (OREAS 680) and one for low-grade mineralized zones (OREAS 13b) to match the tenor of the mineralisation.
- The standards were delivered to the laboratory in their original 10g sachet packages. Before sample submittal, the OREAS CRM codes were erased from their packaging and were replaced by FinnCobalts' sample number.

CRM Code	Cu	Ni	Co	State	Matrix	Mineralization
OREAS 13b	2327ppm	2247ppm	75ppm	primary	gabbronorite	disseminated magmatic
OREAS 680	0.904%	2.15%	334ppm	primary	gabbronorite	magmatic Ni-Cu-PGE

Two different standards were utilized to check the accuracy of the laboratory. Specific pass/fail criteria were determined from the standard deviation provided for the CRMs. The conventional approach to setting acceptance limits is to use the mean assay ± 2 standard deviations as a warning limit and ± 3 standard deviations as a failure limit, which are provided by the CRM manufacturer (Oreas). The results for Co, Cu, Ni, and S analysis of the CRMs Oreas 13b and Oreas 680 are given, respectively, (Figure 10-5 and Figure 10-6). The results show that failures only occurred in the low-grade Oreas 13b standard for Cu analysis. Here the highest Cu assay is 10.9% higher than the certified value. However, compared to the SDs of the other elements and the other standard, it appears that the Cu standard deviation for this standard is especially tight. Besides the multiple failures in the Cu assays of Oreas 13b, there are warnings in the S assays for the higher-grade standard Oreas 680. Furthermore, a positive drift is apparent in all presented elements for both standards. It is worth noting that FinnCobalts 2020 drilling campaign targeted low-grade areas at the beginning of the drilling and switched to higher-grade targets starting from batch VHO007. However, the last batch (VHO020) was from a low-grade Co and Cu area and still shows above-average assays for CRM Oreas 13b and Oreas 680, indicating that the observed drift cannot be solely attributed to generally higher ore grades in the regular drill core samples. Also, the coarse blank analyses do not show any indications of systematic contamination.

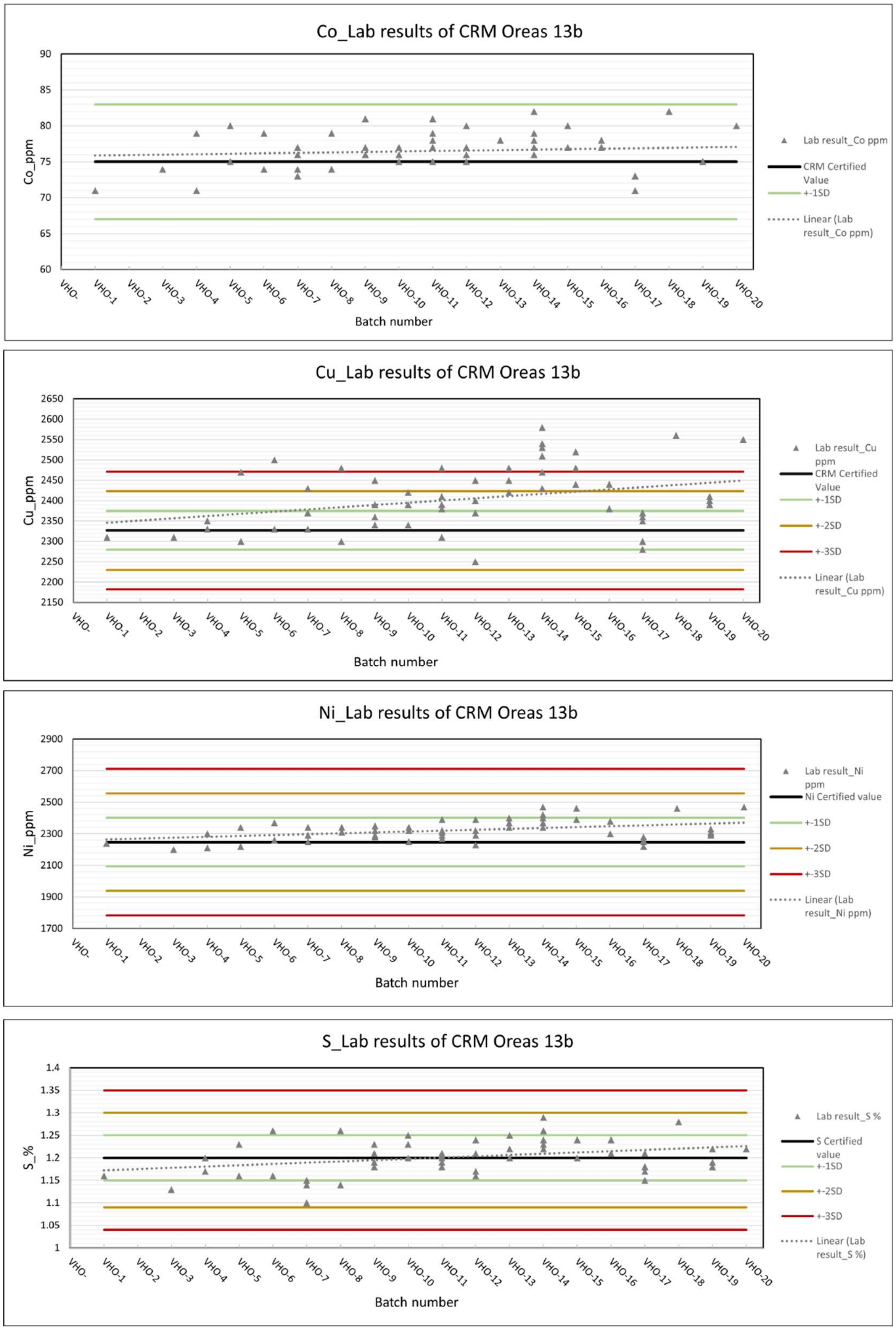
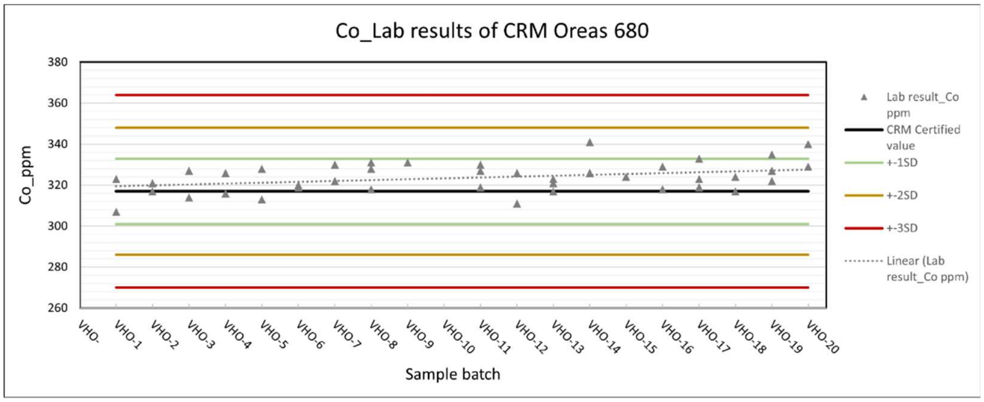
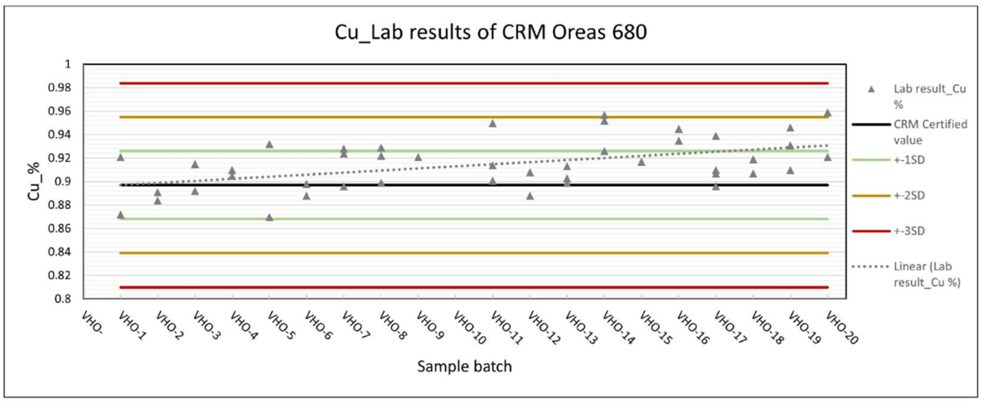
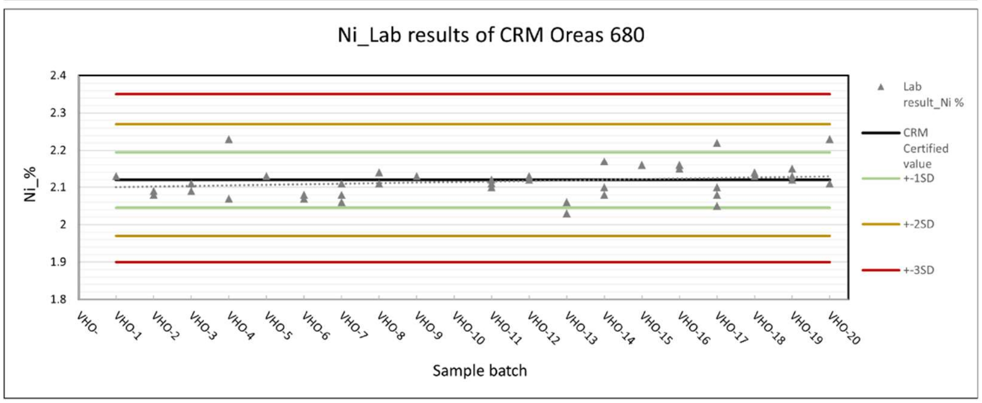


Figure 10-5 Assay results of standard Oreas 13b, in chronological order.







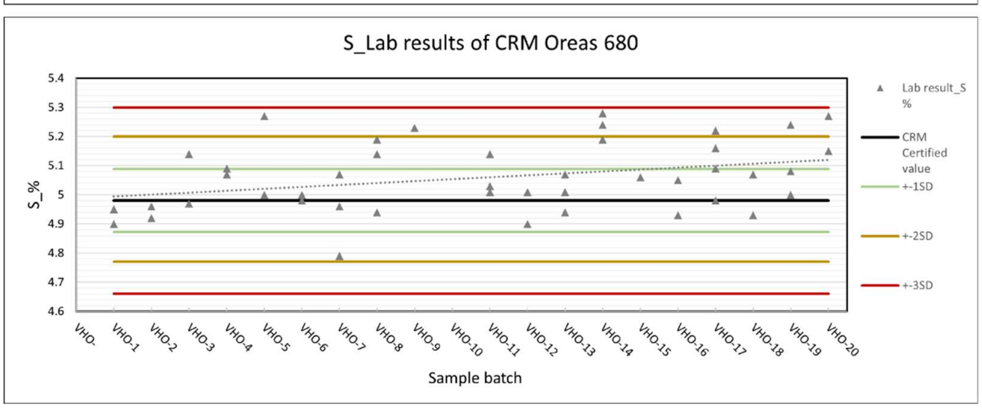


Figure 10-6 Assay results of standard Oreas 680, in chronological order.

Duplicates

- 5 out of 100 samples (5%) of all samples submitted to ALS are core duplicates.
- For core duplicates, the ½ core remaining after normal sampling is quartered and one quarter is analysed (¼ core remains).
- The original sample and the duplicate sample have the same sample interval.
- The original sample and the duplicate sample have different sample numbers.

Duplicates were inserted to assess the precision of sample taking and to assess the representativity of sampled drill core. Figure 10-7 shows a comparison of the course duplicate assays with their original counterparts. Results suggest a good correlation between the original and duplicate core assays with R2 ranging between 0.958 and 0.972.

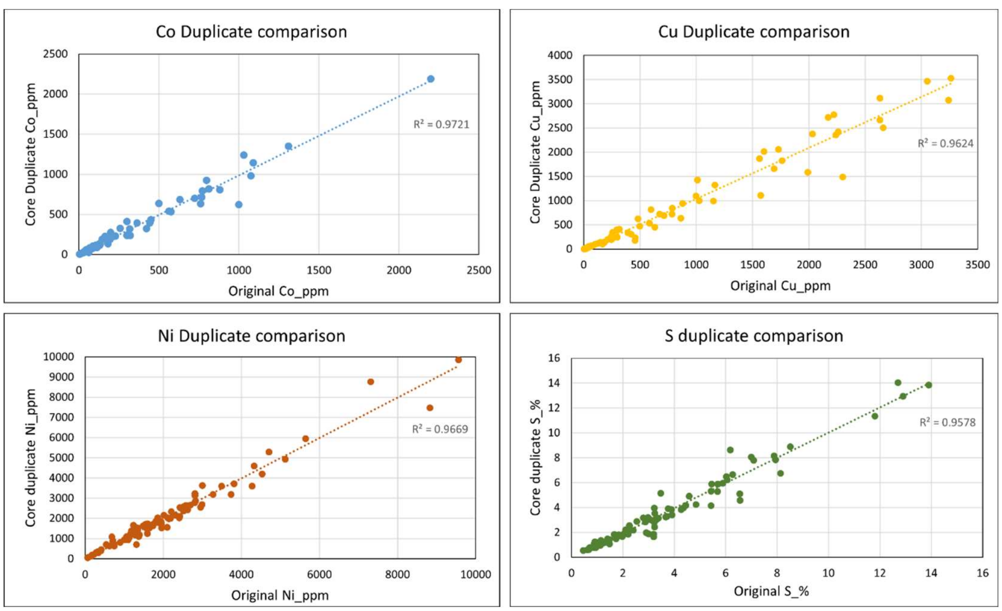


Figure 10-7 Assays of coarse duplicate samples plotted against their original counterparts.

10.3.2Laboratory and assay methods

Sample transport

The drill core samples, still located in their core boxes, were picked up with a forklift by ALS from FinnCobalts warehouse and logging facility. At all times were the QAQC samples transported together with the drill core samples. The ALS laboratory site is only ~500m away from the FinnCobalt premises. Chain of custody certificates is collected for each batch sent to the lab (company batch number, lab work order form).

Laboratory and sample preparation

FinnCobalt utilized ALS laboratories in Outokumpu where the drill core was sawn, crushed, split, and pulverized. ALS is an international, fully independent, and accredited analytical services firm whose Quality Management System framework follows the most appropriate ISO standard i.e. ISO 9001:2015 for survey/inspection activity and ISO/IEC 17025:2017 UKAS ref 4028 for laboratory analysis. At the ALS laboratory in Outokumpu, half-core samples were placed in an industry-standard sample preparation sequence (ALS code: PREP-31) comprising crushing to >70% passing 2mm, splitting, and pulverizing a 250g split portion to >85% passing 75µm. Subsequently, the pulps were shipped to Ireland to the ALS analysis facilities.

Assay methods

The sample analysis was performed at the ALS laboratory in Ireland. An overview of which assay method was utilized in each hole, including sample amounts, is given in Table 10-3.

FinnCobalt's main assay method consisted of 0.25g sample four acid digestion with ICP-AES (Inductively Coupled Plasma – Atomic Emission Spectroscopy) finish (ALS code: ME-ICP61), yielding assays for 33 elements. Overlimit analysis for Co, Cu, and Ni concentrations of >1% consisted of ALS code: ME-OG62. Overlimit analysis for S concentrations of >10% consisted of ALS code: S-IR08, which comprised Leco furnace and infrared spectroscopy. All FinnCobalts 1780 drill core samples were analysed with this method.

For the first 6 sample batches (325 samples) FinnCobalt also utilized strong oxidising digestion comprising HNO_3, $KClO_3$, and HBr with aqua regia, with an ICP-AES finish (ALS code: ME-ICPORE) yielding assays for 19 elements. The over-limit assay method for Ni and Cu contents of >2% consisted of an analytical AAS (Atomic Absorption Spectroscopy) finish (ALS code: AAORE). The ME-ICPORE assay method applies to base metal ores and is particularly suitable for massive sulphides. This additional assay method was chosen to compare the assay results to those of the ME-ICP61 method and to evaluate any grade differences. Since no grade differences were recognized, ME-ICPORE was discontinued after batch VHO006.

Additionally, a partial digestion method (ALS code: Ni-ICP05) was utilized for 448 core samples to further determine the Ni content for exclusively the sulphide fraction. Ni-ICP05 analysis was conducted for every sample for the first 6 sample batches and ~5-20% of all samples for the remaining batches.

To get a better understanding of the Au concentration in the mineralised zones, a total of 20 Au assays were taken. For gold assays, the analysis method consisted of fire assay with AAS analysis using 30g sample sizes (ALS code: Au-AA23), yielding a 0.005 ppm Au detection limit.

Table 10-3 Drilling summary of the 2020 FinnCobalt drilling campaign.

SEQUENCE	HOLE_ID	DRILL TYPE	PROSPECT	AZIMUTH	DIP	STATUS	DRILL START	DRILL END	HOLE DEPTH	COMMENT	ASSAYS RECEIVED	SAMPLES (amount)	SAMPLE BATCH	Assay method (assay amount)	Sample Nr. from	Sample Nr. To
1	HA20-001	DD	BLUE SKY 2 & 3	315	-80	Extended	13/07/2020	14/07/2020	94.80		13/08/2020	50	VHO001	ME-ICPORE (50); ME-ICP61 (50); Ni-ICP05 (50)	447001	447050
2	HA20-002	DD	BLUE SKY 2 & 3	315	-70	Target depth	15/07/2020	15/07/2020	62.60		21/08/2020	34	VHO002	ME-ICPORE (34); ME-ICP61 (34); Ni-ICP05 (34)	447051	447084
3	HA20-003	DD	BLUE SKY 2 & 3	315	-70	Target depth	15/07/2020	16/07/2020	74.45		21/08/2020	50	VHO003	ME-ICPORE (50); ME-ICP61 (50); Ni-ICP05 (50)	447085	447134
4	HA20-004	DD	BLUE SKY 2 & 3	315	-80	Target depth	16/07/2020	17/07/2020	125.75		31/08/2020	67	VHO004	ME-ICPORE (67); ME-ICP61 (67); Ni-ICP05 (67)	447135	447201
5	HA20-005	DD	BLUE SKY 2& 3	315	-80	Terminated	17/07/2020	20/07/2020	204.40	Severe problems with this hole: several drill bits broken and water kept flowing out of the hole.	31/08/2020	62	VHO005	ME-ICPORE (62); ME-ICP61 (62); Ni-ICP05 (62)	447202	447263
6	HA20-006	DD	BLUE SKY 2 & 3	315	-80	Target depth	21/07/2020	22/07/2020	119.40		02/09/2020	34	VHO006	ME-ICPORE (34); ME-ICP61 (34); Ni-ICP05 (34)	447264	447297
7	HA20-007	DD	BLUE SKY 2 & 3	315	-80	Extended	22/07/2020	23/07/2020	111.00		02/09/2020	28	VHO006	ME-ICPORE (28); ME-ICP61 (28); Ni-ICP05 (28)	447298	447325
11	HA20-008	DD	BLUE SKY 2 & 3	315	-80	Terminated	27/07/2020	30/07/2020	230.90	Hole was collapsing and had to be abandoned. Drillers did not want to drill any deeper. Old mining tunnels only 15m below end depth.	18/09/2020	29	VHO007	ME-ICP61 (29); Ni-ICP05 (2)	447326	447354
8	HA20-009	DD	BLUE SKY 2 & 3	315	-80	Target depth	23/07/2020	24/07/2020	80.80		29/09/2020	36	VHO010	ME-ICP61 (36); Ni-ICP05 (2)	447622	447657
9	HA20-010	DD	BLUE SKY 2 & 3	315	-80	Terminated	25/07/2020	26/07/2020	176.50		06/10/2020	25	VHO011	ME-ICP61 (25)	447681	447705
10	HA20-011	DD	BLUE SKY 2 & 3	315	-80	Target depth	26/07/2020	27/07/2020	110.30		18/12/2020	54	VHO020	ME-ICP61 (54)	448727	448780
12	HA20-012	DD	BLUE SKY 2 & 3	315	-80	Extended	31/07/2020	31/07/2020	41.65		29/09/2020	23	VHO010	ME-ICP61 (23); Ni-ICP05 (1)	447658	447680
13	HA20-013	DD	Blue Sky 1	315	-75	Target depth	31/07/2020	01/08/2020	86.50		06/10/2020	48	VHO011	ME-ICP61 (48)	447706	447753
14	HA20-014	DD	Blue Sky 1	315	-80	Extended	01/08/2020	02/08/2020	89.80		06/10/2020	79	VHO011	ME-ICP61 (79); Ni-ICP05 (2)	447754	447832
15	HA20-015	DD	Blue Sky 1	315	-80	Extended	02/08/2020	04/08/2020	145.20		12/10/2020	100	VHO012	ME-ICP61 (100); Ni-ICP05 (4)	447833	447932
16	HA20-016	DD	Blue Sky 1	315	-80	Extended	05/08/2020	07/08/2020	163.25		25/09/2020	83	VHO008	ME-ICP61 (83); Ni-ICP05 (3)	447448	447530
17	HA20-017	DD	Blue Sky 1	315	-80	Extended	07/08/2020	09/08/2020	180.00		18/09/2020	93	VHO007	ME-ICP61 (93); Ni-ICP05 (2)	447355	447447
18	HA20-018	DD	Blue Sky 1	315	-80	Extended	09/08/2020	10/08/2020	152.70		25/09/2020	91	VHO009	ME-ICP61 (91); Ni-ICP05 (6); Au-AA23 (3)	447531	447621
21	HA20-019	DD	Blue Sky 1	315	-80	Terminated	14/09/2020	15/09/2020	155.50		29/10/2020	71	VHO014	ME-ICP61 (71); Ni-ICP05 (3)	448031	448101
20	HA20-020	DD	Blue Sky 1	315	-80	Extended	12/09/2020	13/09/2020	149.15		29/10/2020	88	VHO014	ME-ICP61 (88); Ni-ICP05 (1)	448102	448189
19	HA20-021	DD	Blue Sky 1	315	-80	Extended	09/09/2020	12/09/2020	290.60		20/10/2020	98	VHO013	ME-ICP61 (98); Ni-ICP05 (16); Au-AA23 (7)	447933	448030
22	HA20-022	DD	Blue Sky 1	315	-80	Target depth	16/09/2020	17/09/2020	137.55		29/10/2020	70	VHO015	ME-ICP61 (70); Ni-ICP05 (2)	448190	448259
26	HA20-023	DD	Blue Sky 1	315	-80	Target depth	22/09/2020	23/09/2020	130.85		23/11/2020	50	VHO017	ME-ICP61 (50); Ni-ICP05 (9)	448329	448378
23	HA20-024	DD	Blue Sky 1	315	-80	Terminated	17/09/2020	18/09/2020	140.80		23/11/2020	111	VHO017	ME-ICP61 (111); Ni-ICP05 (25)	448379	448489
27	HA20-025	DD	Blue Sky 1	315	-45	Target depth	23/09/2020	24/09/2020	91.50		26/11/2020	61	VHO018	ME-ICP61 (61); Ni-ICP05 (2); Au-AA23 (5)	448490	448550
24	HA20-026	DD	Blue Sky 1	315	-80	Target depth	19/09/2020	20/09/2020	146.10		07/12/2020	89	VHO019	ME-ICP61 (89); Ni-ICP05 (27)	448607	448695
25	HA20-027	DD	Blue Sky 1	315	-80	Terminated	20/09/2020	21/09/2020	91.10	Hole intersected serious cavings area and had to be abandoned. The last 2m before drilling stop were massive pyrrhotite mineralization	07/12/2020	31	VHO019	ME-ICP61 (31); Ni-ICP05 (8)	448696	448726
28	HA20-028	DD	Blue Sky 1	315	-45	Target depth	24/09/2020	25/09/2020	87.50		26/11/2020	56	VHO018	ME-ICP61 (56); Ni-ICP05 (1); Au-AA23 (5)	448551	448606
29	HA20-029	DD	Blue Sky 1	315	-45	Extended	25/09/2020	26/09/2020	97.35		29/10/2020	69	VHO016	ME-ICP61 (69); Ni-ICP05 (7)	448260	448328
29									**3,768.00**			**1,780**				

10.4 Assay comparison for the Hautalampi resource and Blue-Sky area

FinnCobalt Oy made two re-assay studies regarding the Hautalampi resource area and Blue-Sky area (Mökkivaara) in 2018 and 2021. During 2018 a re-logging and re-assaying program was applied to 31 OKU drill holes, giving 416 new ICP assays. The old Outokumpu assay results were then compared against the re-assayed results. Although the sampled intercept can be the same the sampling method has varied. In some of the old cores during Outokumpu exploration, the core has not been split but pieces of the core have been sampled and average values calculated. The re-assay sampling was made by splitting the core when possible. In some cases, however, the old core left was so narrow that splitting was not made, and core pieces were taken instead. In summary, the data from the 2018 study shows that the re-assay values for cobalt, copper, nickel and sulphur are 24 to 28 % lower than the old assay values.

It is CP's opinion that because of the varying sampling techniques used in the old Outokumpu sampling, the comparison to FinnCobalt samples can be biased. The re-assays made in 2021 can be considered more reliable.

The old Outokumpu Oy (OKU) drill core assays were compared to the re-assays made in 2021 by FinnCobalt in the Hautalampi resource area and Blue-Sky area (Mökkivaara). The assay method in the re-assaying was ALS/ME-ICP61. The old Outokumpu assays have been made mainly by FAAS. The total number of samples was 131 from 20 holes and a total length of 467.54 m of drill core (Figure 10-8 and Table 10-4). Sampling for the re-assaying was made by Matthias Mueller and Kalle Penttilä in January 2021 at the GTK drill core storage in Loppi. The existing half core was halved, by the GTK staff with a diamond saw, so the re-assay sample was one-quarter of the core. Regarding QAQC measures, blanks, standards, and pulp duplicates were inserted in the sample batch. QAQC samples made up ~15% of the total submitted samples, with 1 out of 6 samples being a QAQC sample. The 15% QAQC samples were divided equally into 5% coarse blanks, 5% standards, and 5% pulp duplicates. All utilized blanks were coarse blanks, sourced from Savon Kuljetus Oy via ALS Laboratories, who use the rock as a "wash rock" in-between runs. ALS is periodically analysing this blank rock material every month to ensure its homogeneity and barrenness. The utilized standards were sources from OREAS, Australia. Two different standards were used – one for high-grade mineralized zones (OREAS 680) and one for low-grade mineralized zones (OREAS 13b) to match the tenor of the mineralisation. Both standards have certified values for the elements of interest (Co, Cu, Ni, and S).

The results of the comparison are shown in – Figure 10-14 and in Table 10-4 figures 2-7 and Table 1. A summary the re-assays conform well with the OKU assays for the most important metals - nickel, cobalt, and copper:

- For nickel, the average difference is small – re-assay is on average 2.4 % lower, but for the ore grade samples (> 0.3 %), the re-assay is on average 7.4 % lower. This is well depicted in fig. 2.
- For cobalt, the re-assay gives in average slightly higher values – the average difference is 5.0 %.
- For copper, the difference is small – re-assay is on average 1.4 % higher.

For zinc, there is no correlation at all (Figure 10-12). This may be reflected by the tendency of zinc to occur very unevenly in the rock, so the core halves are always unequal. The same applies partly also to copper. The basic reason for this is that zinc and copper are much more mobile during all the ore-forming processes than nickel and cobalt.

OKU assay data for iron and sulphur was available only for 16 samples, so the comparison is not very reliable. In the sulphur assay comparison, the OKU assays are systematically higher (Fig. 6). In the iron assay comparison the correlation is poor (Fig. 7).

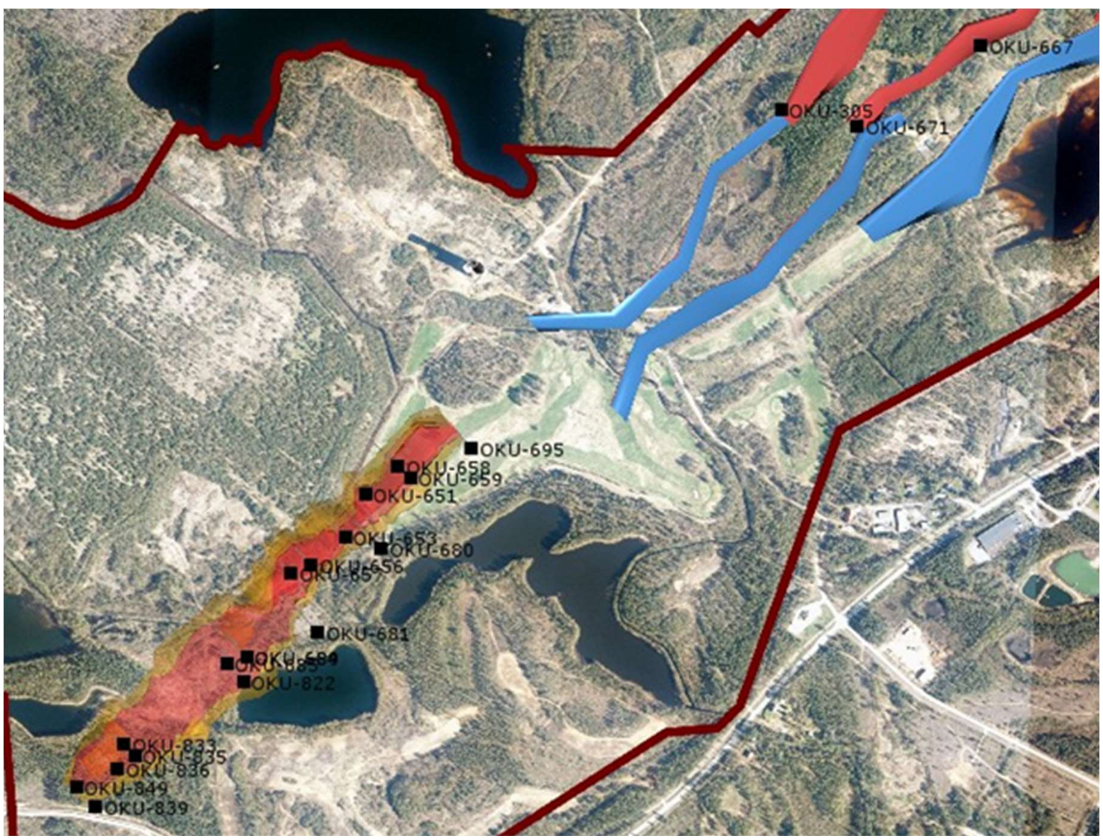
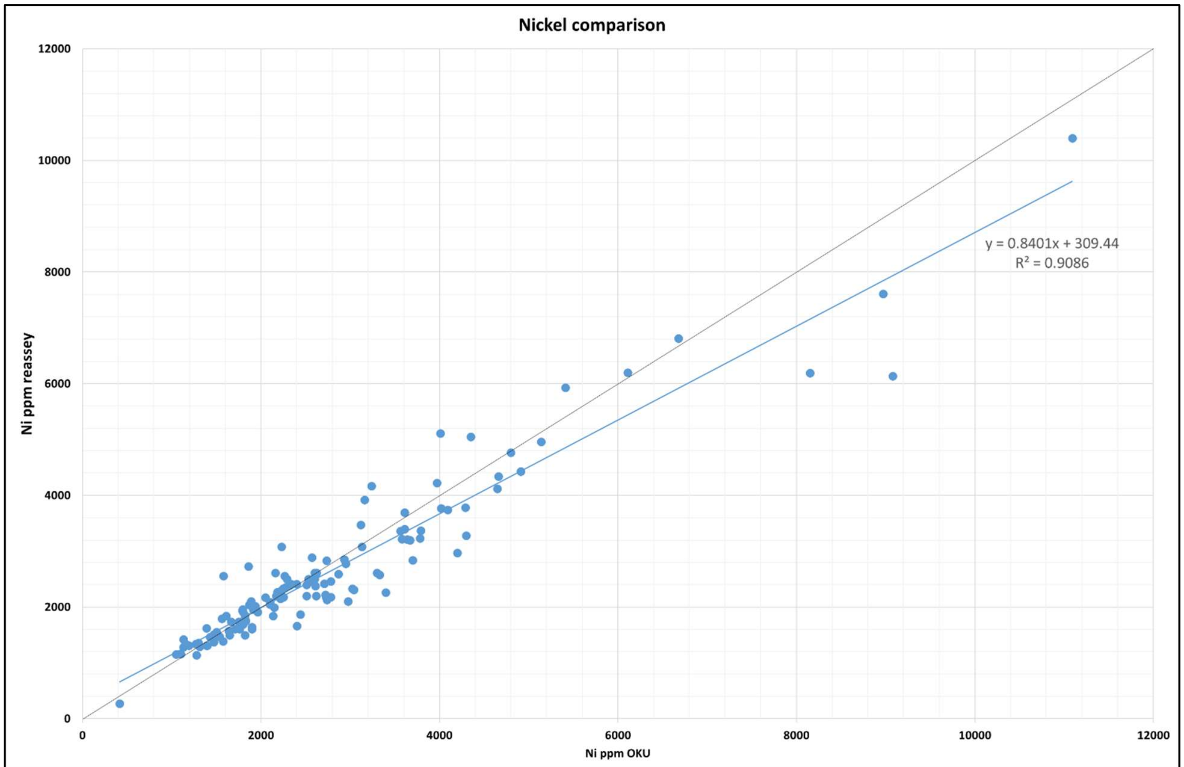
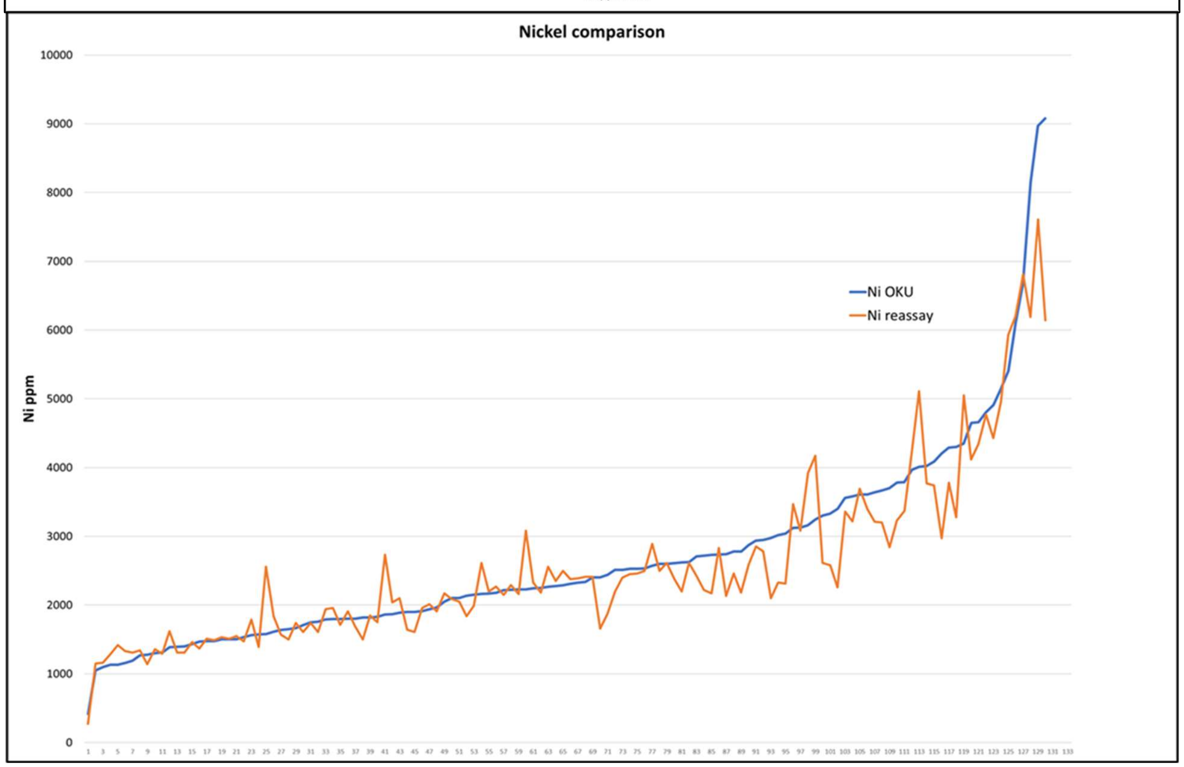
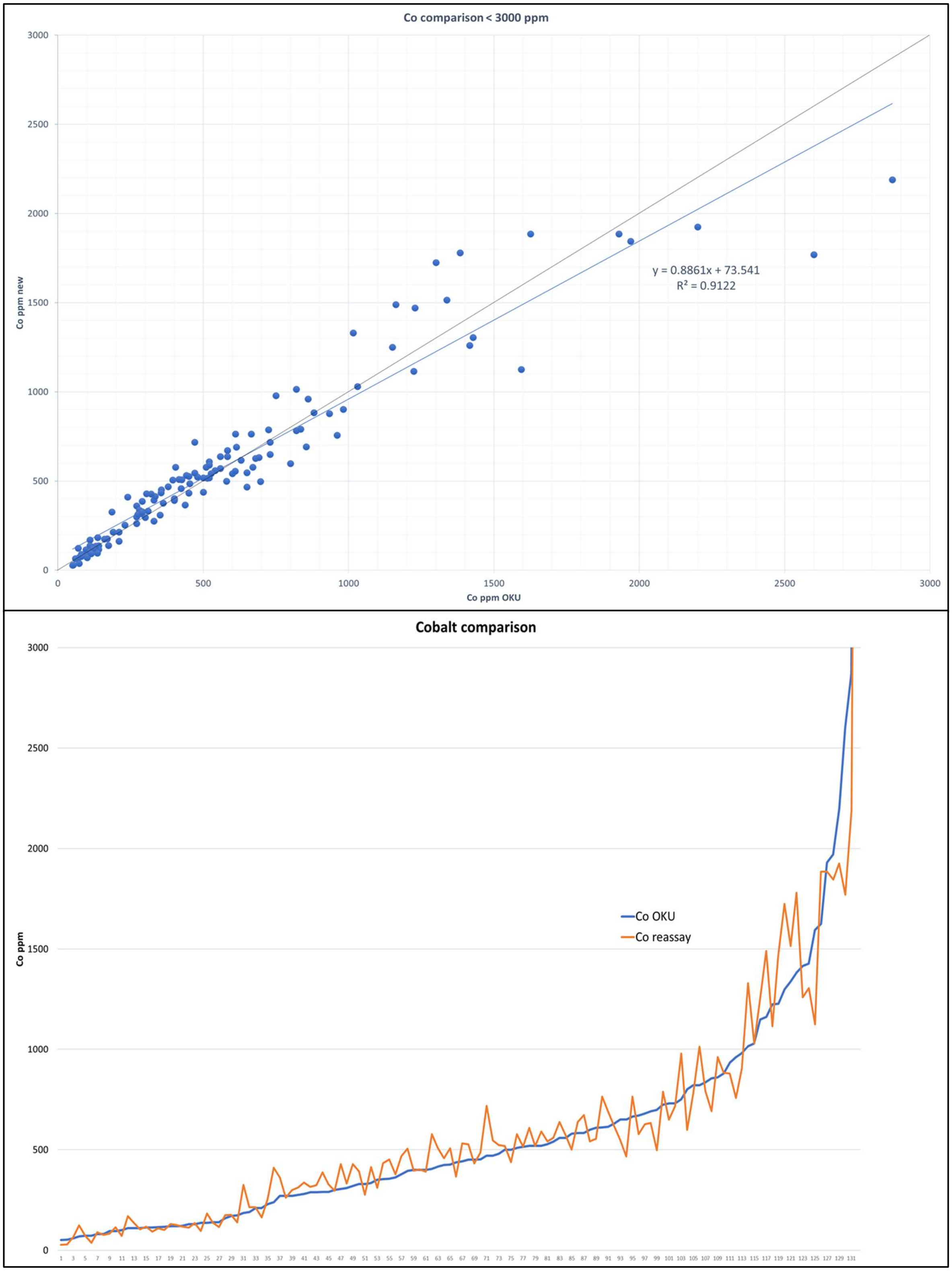


Figure 10-8 Location of the drill holes for the re-assays. Hautalampi resource area (red+yellow) and Blue-Sky areas / Mökkivaara (blue and red) are shown.





*F*igure 10-9 Nickel comparaison.



*F*igure 10-10 Cobalt comparaison.

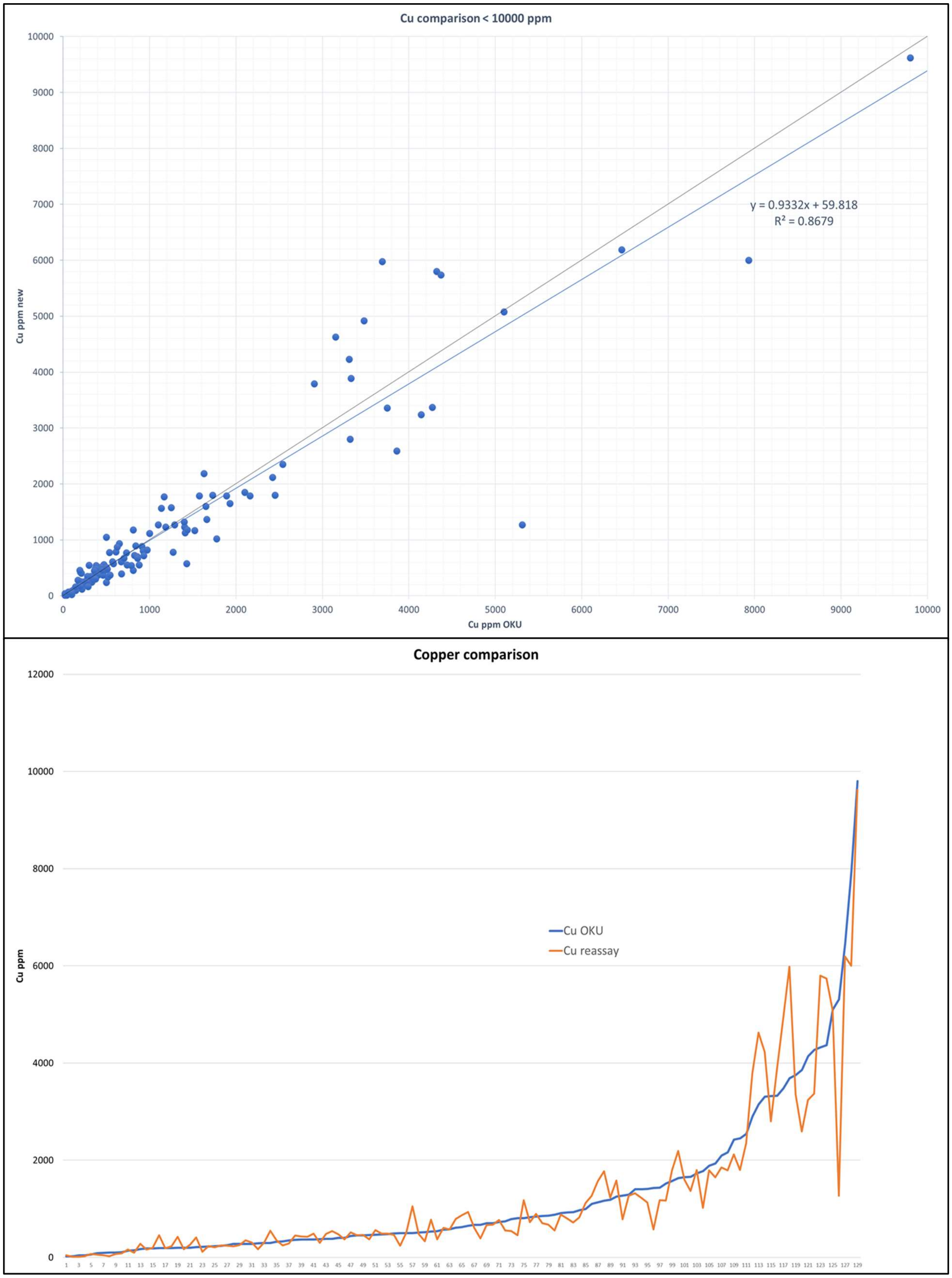


Figure 10-11 Copper comparaison.

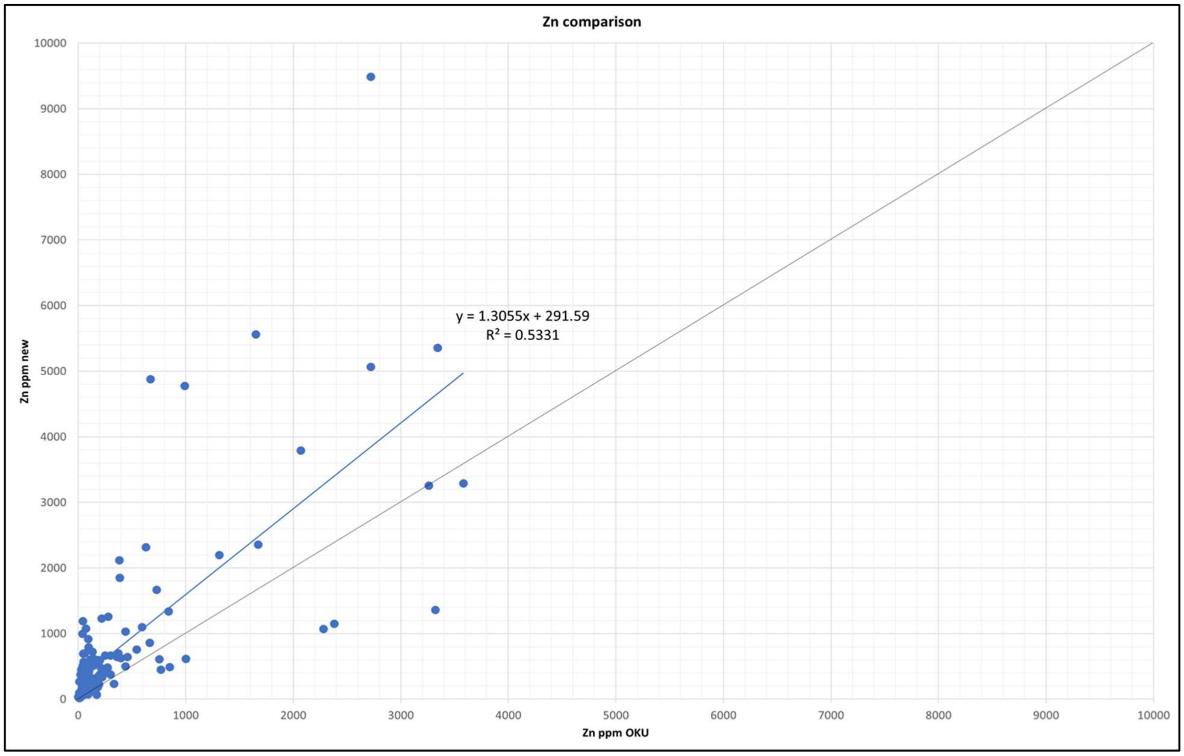


Figure 10-12 Zinc comparison.

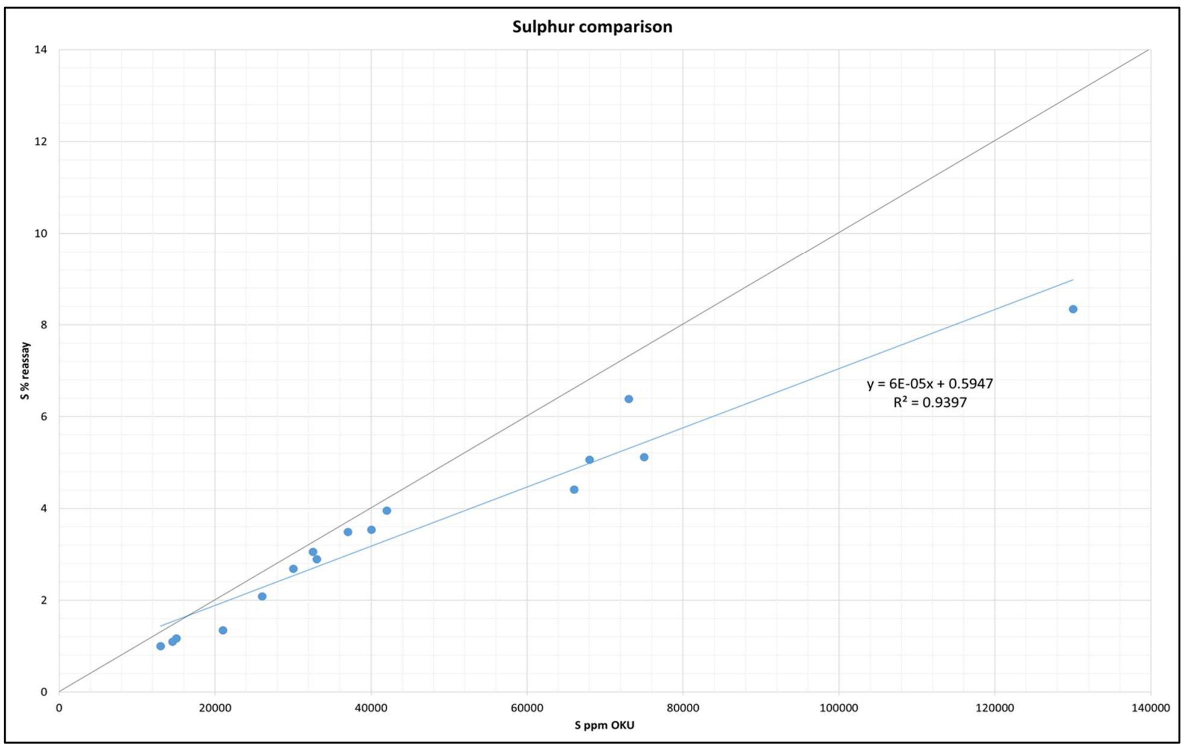


Figure 10-13 Sulphur comparison.

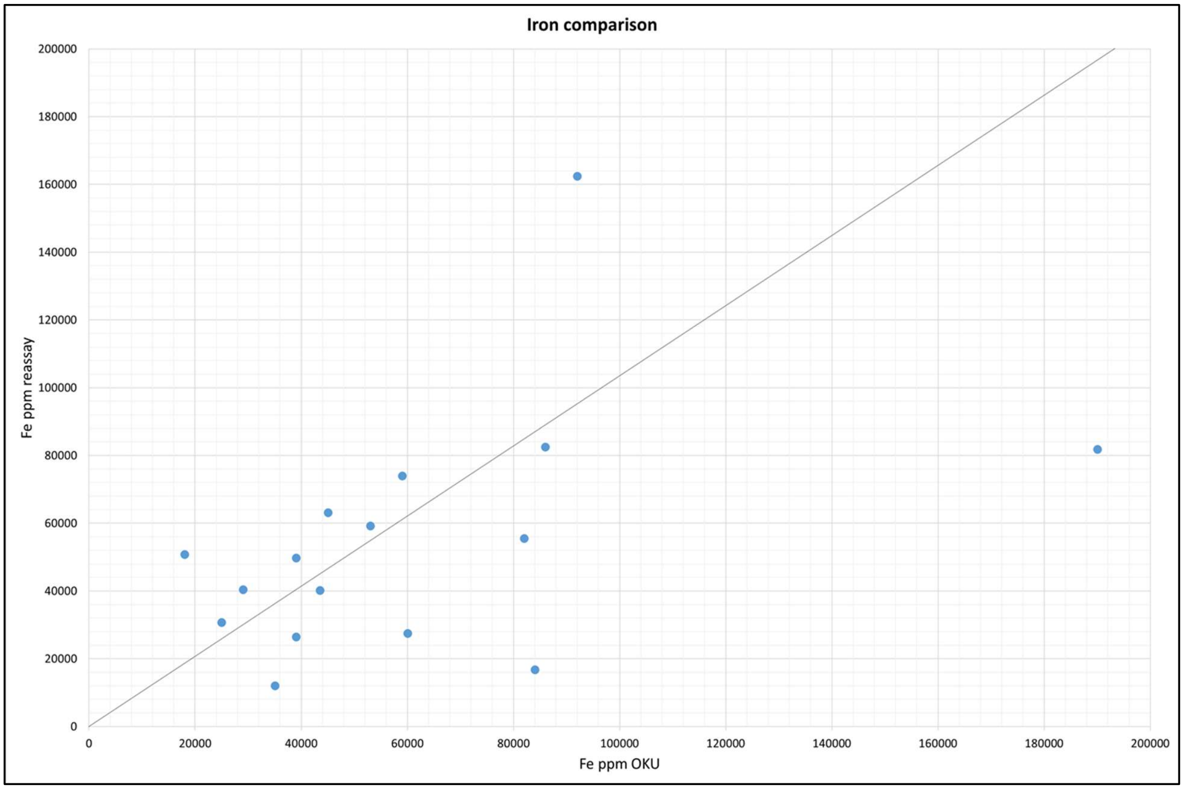


Figure 10-14 Iron comparison.

Table 10-4 Reassays and old Outokumpu Company assays (OKU) for Co, Cu and Ni.

					OKU	Reassay	Co Difference	OKU	Reassay	Cu Difference	OKU	Reassay	Ni Difference
Average of all				467.54			-5.0			-1.4			2.4
Sample ID	Hole ID	From m	To m	Length m	Co ppm	Co ppm	%	Cu ppm	Cu ppm	%	Ni ppm	Ni ppm	%
039	OKU-305	12	17	5	120	126	-5.0	100	71	29.0	1800	1910	-6.1
040	OKU-305	17	23	6	300	296	1.3	200	258	-29.0	1500	1530	-2.0
041	OKU-305	23	29	6	650	547	15.8	2100	1850	11.9	1800	1690	6.1
042	OKU-305	29	34	5	600	542	9.7	1100	1270	-15.5	2100	2050	2.4
044	OKU-305	34	39	5	580	500	13.8	1400	1320	5.7	2600	2500	3.8
045	OKU-305	39	42.2	3.2	500	518	-3.6	700	669	4.4	2600	2610	-0.4
046	OKU-305	42.2	47.5	5.3	270	262	3.0	500	508	-1.6	2400	2410	-0.4
047	OKU-305	47.5	53	5.5	100	71	29.0	100	27	73.0	1400	1310	6.4
048	OKU-305	53	59.35	6.35	130	113	13.1	300	309	-3.0	1500	1510	-0.7
002	OKU-651	19.5	24.4	4.9	450	527	-17.1	230	211	8.3	1820	1500	17.6
003	OKU-651	24.4	29.7	5.3	520	518	0.4	190	456	-140.0	2050	2170	-5.9
004	OKU-651	37.8	43.45	5.65	270	361	-33.7	470	560	-19.1	1270	1340	-5.5
005	OKU-651	43.45	47.15	3.7	190	214	-12.6	370	431	-16.5	1430	1460	-2.1
006	OKU-651	47.15	49.8	2.65	500	438	12.4	970	824	15.1	2780	2460	11.5
008	OKU-651	49.8	56	6.2	270	300	-11.1	380	486	-27.9	2510	2200	12.4
009	OKU-651	56	61.1	5.1	140	116	17.1	170	280	-64.7	2440	1870	23.4
010	OKU-651	61.1	66	4.9	110	138	-25.5	810	456	43.7	1610	1840	-14.3
011	OKU-651	83.2	87.3	4.1	130	135	-3.8	60	69	-15.0	2710	2420	10.7
026	OKU-653	58.45	62.6	4.15	70	124	-77.1	20	44	-120.0	1160	1330	-14.7
027	OKU-653	62.6	66.2	3.6	670	578	13.7	740	554	25.1	2780	2180	21.6
028	OKU-653	66.2	70.8	4.6	510	578	-13.3	300	548	-82.7	1820	1850	-1.6
029	OKU-653	70.8	72.2	1.4	540	559	-3.5	810	1180	-45.7	1760	1610	8.5
030	OKU-653	72.2	75	2.8	400	395	1.3	490	462	5.7	1790	1940	-8.4
032	OKU-653	75	79.5	4.5	400	391	2.3	1930	1650	14.5	1900	1610	15.3
033	OKU-653	79.5	83.7	4.2	290	387	-33.4	500	1050	-110.0	1500	1550	-3.3
034	OKU-653	83.7	88.3	4.6	170	177	-4.1	330	245	25.8	1050	1150	-9.5
035	OKU-653	88.3	92.9	4.6	650	466	28.3	840	896	-6.7	2740	2130	22.3
036	OKU-653	92.9	95.6	2.7	80	89	-11.3	40	15	62.5	1640	1570	4.3
038	OKU-653	95.6	100.35	4.75	60	66	-10.0	350	284	18.9	1100	1160	-5.5
513	OKU-656	60.4	65.5	5.1	520	590	-13.5	280	308	-10.0	2170	2200	-1.4
514	OKU-656	65.5	68.6	3.1	750	979	-30.5	1630	2190	-34.4	3160	3920	-24.1
515	OKU-656	68.6	72.6	4	280	337	-20.4	400	480	-20.0	1910	1960	-2.6
516	OKU-656	79.15	83.2	4.05	450	432	4.0	790	543	31.3	1830	1750	4.4
517	OKU-656	83.2	87.9	4.7	110	105	4.5	90	61	32.2	1650	1500	9.1
050	OKU-657	45.3	49.2	3.9	470	546	-16.2	140	160	-14.3	2870	2590	9.8
051	OKU-657	56.8	60.5	3.7	330	393	-19.1	380	543	-42.9	1750	1740	0.6
052	OKU-657	60.5	65.5	5	240	410	-70.8	440	516	-17.3	1860	2730	-46.8
053	OKU-657	65.5	69.1	3.6	610	555	9.0	1430	577	59.7	3040	2310	24.0
054	OKU-657	69.1	72.5	3.4	480	523	-9.0	360	449	-24.7	2510	2400	4.4
056	OKU-658	19	22.9	3.9	527	542	-2.8	703	674	4.1	3021	2330	22.9
057	OKU-658	25.8	29.5	3.7	400	402	-0.5	2422	2120	12.5	1898	1640	13.6
058	OKU-658	29.5	33.7	4.2	697	498	28.6	1777	1020	42.6	2403	1660	30.9
059	OKU-658	33.7	38	4.3	351	310	11.7	476	490	-2.9	1575	1390	11.7
060	OKU-658	38	42	4	96	115	-19.8	185	161	13.0	1193	1310	-9.8
040	OKU-658	42	46.05	4.05	334	414	-24.0	451	460	-2.0	2150	1990	7.4
041	OKU-658	46.05	50.3	4.25	583	672	-15.3	535	777	-45.2	3120	3470	-11.2
042	OKU-659	42.3	45.6	3.3	289	324	-12.1	1134	1570	-38.4	2572	2890	-12.4
044	OKU-659	45.6	52	6.4	115	110	4.3	198	424	-114.1	1476	1490	-0.9
045	OKU-659	52	57.4	5.4	73	72	1.4	20	13	35.0	1132	1420	-25.4
046	OKU-659	57.4	62.8	5.4	185	326	-76.2	239	250	-4.6	1388	1620	-16.7
047	OKU-659	62.8	68.5	5.7	611	764	-25.0	1402	1230	12.3	2327	2390	-2.7
048	OKU-659	73.7	78.15	4.45	426	507	-19.0	372	488	-31.2	1936	2010	-3.8
002	OKU-659	78.15	82	3.85	137	183	-33.6	192	229	-19.3	1561	1790	-14.7
003	OKU-659	82	86.15	4.15	724	788	-8.8	1576	1790	-13.6	2623	2610	0.5
004	OKU-659	89.35	93.4	4.05	210	163	22.4	460	373	18.9	1710	1610	5.8
005	OKU-659	104.7	110.25	5.55	230	253	-10.0	370	424	-14.6	2100	2090	0.5
006	OKU-659	110.25	114.9	4.65	210	214	-1.9	320	352	-10.0	1470	1370	6.8
008	OKU-667	40.39	42.09	1.7	121	118	2.5	199	170	14.6	3780	3230	14.6
009	OKU-667	42.09	43.44	1.35	174	139	20.1	541	374	30.9	8970	7610	15.2
010	OKU-667	43.44	46.1	2.66	136	96	29.4	92	46	50.0	3400	2260	33.5
011	OKU-667	46.1	47.72	1.62	114	93	18.4	511	482	5.7	4300	3280	23.7
026	OKU-667	47.72	50.38	2.66	73	38	47.9	373	301	19.3	2977	2100	29.5
027	OKU-671	27.7	28.9	1.2	53	29	45.3	483	491	-1.7	1276	1140	10.7
028	OKU-671	28.9	32.1	3.2	51	28	45.1	498	239	52.0	417	271	35.0

Table 10-4 *Continues*

					OKU	Reassay		OKU	Reassay		OKU	Reassay	
Sample ID	**Hole ID**	**From m**	**To m**	**Length m**	**Co ppm**	**Co ppm**	**Difference %**	**Cu ppm**	**Cu ppm**	**Difference %**	**Ni ppm**	**Ni ppm**	**Difference %**
074	OKU-680	91.2	95.35	4.15	356	451	-26.7	925	803	13.2	2181	2270	-4.1
075	OKU-680	99.7	101.45	1.75	1594	1125	29.4	3750	3360	10.4	4200	2970	29.3
076	OKU-680	101.45	105.65	4.2	730	650	11.0	826	723	12.5	2135	1840	13.8
077	OKU-680	105.65	106.85	1.2	1228	1470	-19.7	2905	3790	-30.5	4090	3740	8.6
078	OKU-680	106.85	108.6	1.75	514	516	-0.4	456	462	-1.3	1965	1910	2.8
080	OKU-680	108.6	117	8.4	140	138	1.4	275	228	17.1	1536	1470	4.3
081	OKU-680	119.5	121.75	2.25	362	377	-4.1	216	120	44.4	1472	1510	-2.6
082	OKU-680	121.75	124.75	3	854	692	19.0	1187	1230	-3.6	3330	2580	22.5
083	OKU-680	124.75	129.05	4.3	160	175	-9.4	145	98	32.4	1666	1740	-4.4
084	OKU-680	139.15	142.4	3.25	95	83	12.6	517	336	35.0	1313	1290	1.8
507	OKU-681	108.8	112.1	3.3	470	718	-52.8	650	938	-44.3	2230	3080	-38.1
508	OKU-681	112.1	114	1.9	1150	1250	-8.7	1290	1270	1.6	4800	4770	0.6
509	OKU-681	114	117.6	3.6	290	329	-13.4	1410	1130	19.9	2220	2290	-3.2
510	OKU-681	117.6	118.95	1.35	110	170	-54.5	110	86	21.8	1580	2560	-62.0
511	OKU-681	118.95	122.8	3.85	120	131	-9.2	40	23	42.5	1130	1280	-13.3
086	OKU-684	97.85	99.55	1.7	1416	1260	11.0	3320	2800	15.7	4290	3780	11.9
087	OKU-684	99.55	103	3.45	665	764	-14.9	730	771	-5.6	2736	2830	-3.4
088	OKU-684	103	105.45	2.45	981	903	8.0	1523	1170	23.2	4020	3770	6.2
089	OKU-684	105.45	109.05	3.6	442	531	-20.1	624	872	-39.7	2246	2330	-3.7
090	OKU-684	109.05	112.35	3.3	1383	1780	-28.7	1437	1180	17.9	4660	4340	6.9
092	OKU-684	112.35	118.3	5.95	354	434	-22.6	402	373	7.2	2289	2500	-9.2
093	OKU-684	118.3	119.85	1.55	1428	1305	8.6	4140	3240	21.7	4910	4430	9.8
094	OKU-684	119.85	120.9	1.05	437	366	16.2	7930	6000	24.3	1795	1710	4.7
095	OKU-684	120.9	125.35	4.45	116	101	12.9	191	186	2.6	1391	1310	5.8
116	OKU-685	85	86.5	1.5	404	578	-43.1	999	1120	-12.1	3240	4170	-28.7
117	OKU-685	86.5	91.5	5	275	313	-13.8	856	704	17.8	2531	2500	1.2
118	OKU-685	96.55	100.3	3.75	835	791	5.3	1888	1790	5.2	3790	3370	11.1
119	OKU-685	100.3	102.9	2.6	416	509	-22.4	275	255	7.3	2934	2850	2.9
120	OKU-685	108.95	115.75	6.8	288	316	-9.7	228	235	-3.1	2309	2380	-3.1
122	OKU-685	115.75	117.8	2.05	453	485	-7.1	1250	1580	-26.4	2225	2160	2.9
101	OKU-689	87.85	89.05	1.2	1162	1490	-28.2	4270	3370	21.1	4350	5050	-16.1
102	OKU-689	89.05	90.05	1	1016	1330	-30.9	3480	4920	-41.4	3970	4220	-6.3
104	OKU-689	90.05	92.95	2.9	1224	1115	8.9	9800	9620	1.8	3700	2840	23.2
105	OKU-689	92.95	96.7	3.75	614	691	-12.5	1167	1770	-51.7	2338	2410	-3.1
106	OKU-689	96.7	98	1.3	1338	1515	-13.2	4370	5740	-31.4	6110	6200	-1.5
107	OKU-689	98	100.85	2.85	691	633	8.4	571	609	-6.7	3300	2610	20.9
108	OKU-689	100.85	104.55	3.7	305	429	-40.7	210	409	-94.8	1891	2100	-11.1
110	OKU-689	104.55	106.7	2.15	1626	1885	-15.9	4320	5800	-34.3	5410	5930	-9.6
111	OKU-689	106.7	110.9	4.2	424	458	-8.0	289	169	41.5	4650	4120	11.4
112	OKU-689	110.9	116.85	5.95	395	506	-28.1	251	239	4.8	2263	2560	-13.1
113	OKU-689	116.85	118.7	1.85	934	879	5.9	3690	5980	-62.1	3670	3200	12.8
114	OKU-689	118.7	122.85	4.15	113	118	-4.4	185	194	-4.9	1301	1360	-4.5
503	OKU-695	96.4	101.65	5.25	583	638	-9.4	1729	1800	-4.1	2274	2350	-3.3
504	OKU-695	101.65	105.05	3.4	558	638	-14.3	670	611	8.8	1795	1960	-9.2
505	OKU-695	105.05	109.05	4	558	571	-2.3	673	394	41.5	2216	2150	3.0
501	OKU-822	102.2	104	1.8	820	1015	-23.8	931	718	22.9	3560	3360	5.6
502	OKU-822	114.85	119.6	4.75	379	468	-23.5	879	554	37.0	3610	3690	-2.2
096	OKU-833	66.2	69.15	2.95	860	961	-11.7	3150	4630	-47.0	3610	3400	5.8
098	OKU-833	69.15	71.05	1.9	2200	1925	12.5	3330	3890	-16.8	11090	10400	6.2
064	OKU-835	59.15	61.25	2.1	2870	2190	23.7	6460	6190	4.2	8150	6190	24.0
065	OKU-835	61.25	64.35	3.1	880	883	-0.3	2450	1800	26.5	3580	3220	10.1
066	OKU-835	66.15	67.15	1	800	599	25.1	5310	1270	76.1	2620	2200	16.0
068	OKU-835	68.35	69.85	1.5	1930	1885	2.3	15200	13550	10.9	6680	6810	-1.9
069	OKU-835	69.85	71.9	2.05	820	782	4.6	3310	4230	-27.8	2950	2780	5.8
070	OKU-835	71.9	73.25	1.35	1300	1725	-32.7	17900	18200	-1.7	4010	5110	-27.4
071	OKU-835	73.25	76.95	3.7	1030	1030	0.0	1650	1600	3.0	3130	3080	1.6
072	OKU-835	76.95	79.15	2.2	960	757	21.1	860	674	21.6	3640	3210	11.8
522	OKU-836	56	57.4	1.4	520	609	-17.1	910	882	3.1	1870	2040	-9.1
523	OKU-836	57.4	59.35	1.95	1970	1845	6.3	3860	2590	32.9	5140	4960	3.5
525	OKU-836	59.35	63.3	3.95	730	717	1.8	2540	2350	7.5	2530	2450	3.2
526	OKU-836	63.3	64.75	1.45	2600	1770	31.9	28220	19000	32.7	9080	6140	32.4
527	OKU-836	66.05	69.45	3.4	630	617	2.1	1270	782	38.4	2250	2180	3.1
519	OKU-839	71.1	73.85	2.75	680	627	7.8	1660	1370	17.5	2730	2170	20.5
520	OKU-839	73.85	78.65	4.8	320	428	-33.8	610	789	-29.3	2160	2610	-20.8
521	OKU-839	78.65	81.35	2.7	330	276	16.4	580	576	0.7	2610	2380	8.8
533	OKU-849	40.05	41.7	1.65	310	332	-7.1	5100	5080	0.4	2530	2460	2.8
534	OKU-849	46.5	48.5	2	80	77	3.8	280	351	-25.4	2720	2220	18.4

11 Data Verification

11.1 Database Validation

The resource database was validated for double assays, overlapping intervals and missing data. Drill holes used in resource estimation passed the validation. However, the database contains some errors that should be fixed in future.

11.2 Down-Hole Survey Validation

The drill hole data was validated by checking the consistency of consecutive survey results.

11.3 Assay Verification

The collar, geology, survey and assay files were provided in Microsoft Access®. All From-To data are either zero or a positive value. No intervals exceeded the total depth of its drill hole. Intervals with no assay data were listed as blank in the database.

11.4 Geologic Data Verification and Interpretation

The author has compared the lithological drill core loggings against the drill core photos taken during the drill core logging process.

11.5 QA/QC Protocol

Quality control and quality assurance work are well documented from the drilling campaign done by FinnCobalt Oy. Finn Nickel era sampling and QA/QC protocol are documented but the procedures used are not as high as current-day standards. Nevertheless, the data collected by Finn Nickel Oy is in CP’s opinion suitable to be used in this resource estimate work as is Outokumpu era data.

11.6 Conclusion

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

12 Mineral processing and metallurgical testing

12.1 GTK Laboratory Test Work 2007

Finn Nickel Oy tested Hautalampi ore in the GTK Mintec laboratory in Outokumpu, Finland in August 2007. Finn Nickel Oy delivered nine drill core samples from the Hautalampi Ni-Co-Cu deposit in June 2007. The target of the project was to produce separate smelter-grade Cu- and Ni-concentrates as well as smelter-grade bulk concentrate. The target of cobalt content in the nickel concentrate was about 4 % Co. Cu-content and the recovery of Cu-concentrate is were 24.4 % Cu and recovery of 76.1 % in selective flotation. The best results from Ni/Co flotation were 10.0 % Ni and 40.9 % recovery; 3.8 % Co and 42.4 % recovery. The content of Au in the Cu-concentrate was 2.0 g/t and recovery was 40.2 %. The content of Ag in the Cu-concentrate was 55.9 g/t and recovery was 35.8 %. The content of Ag in the bulk concentrate was 27.9 % and recovery was 62.4 %.

12.2 GTK Laboratory Test Work (2018-2019)

Hautalampi ore beneficiation was tested in the GTK Mintec laboratory in Outokumpu, Finland in a bench scale in August 2018-January 2019 to confirm findings from 2009 test work. Copper and nickel/cobalt concentrates were produced in bench-scale flotation tests. Drill cores samples were crushed and ground to P80 ~72-83 µm for the flotation tests. Copper flotation was tested with two cleaning flotation stages and nickel/cobalt flotation with three cleaning flotation stages.

In the copper flotation average copper grade of 26.2% and 78.8%, copper recovery to the concentrate was achieved. In the nickel/cobalt flotation average nickel grade to 3rd cleaner concentrate was 7.7% with 64% recovery. The average cobalt grade to 3rd cleaner concentrate was 1.8% with 62% recovery.

Preliminary acid leaching tests were done for the nickel/cobalt concentrate. Preliminary acid leaching tests for nickel-cobalt concentrate showed poor results, only 4.5 % of nickel and cobalt were leached. Tests indicated that a longer leaching time than tested 72 hours is required. The leaching kinetic seemed to depend on the grinding fineness.

12.3 GTK Pilot Test Work (2019)

Hautalampi ore was tested in continuous pilot plant operation in the GTK Mintec pilot plant in Outokumpu, Finland. The target of the pilot plant operation was to produce 10 kg of copper concentrate and 20-40 kg of nickel/cobalt concentrate. Target grades and recoveries for the concentrates were as follows:

- Copper concentrate: Cu grade 25%, Cu recovery to concentrate 85%
- Nickel/Cobalt concentrate: Ni grade 6%, Co grade 1.6%, Ni and Co recovery to concentrate 80%

The pilot flowsheet consisted of two-stage ball milling in a closed circuit, a copper flotation circuit with two cleaning stages and a nickel/cobalt flotation circuit with three cleaning stages. The pilot campaign was done in a five-day period, of which concentrates were produced during the final three days. The total amount and grades of produced concentrates were:

- Copper concentrate: 7.2 kg produced, Cu grade 27.2%
- Nickel/Cobalt concentrate: 34.3 kg produced, Ni grade 7.5%, Co grade 1.9%

In the best pilot sampling period (31.1.2019 9.30-13.30) copper recovery of 86.5% was achieved in the copper concentrate with 26.7% Cu grade. During the same period, nickel grade and recovery to the Ni/Co concentrate were 8.0% and 82.0%. Cobalt grade and recovery to the Ni/Co concentrate were 1.9% and 82.6%.

The work index in the mini-pilot grinding was defined and it was 33.3 kWh/t from particle 80 % -1240 µm to 80 % -58 µm. The net energy consumption in grinding was 34.3 kWh/t with magnetic separation, and the grade of the Ni-Co concentrate could be raised from 6.9 % Ni to 9.2 % Ni with a recovery of 89.6 %. Cobalt grade changed from 1.7 % Co to 2.3 % Co.

12.4 Sulphur Flotation Laboratory Test Work (2021)

Sulphur removal flotation was tested in the GTK Mintec laboratory in Outokumpu, Finland in 2021. The campaign aimed to test sulphur removal by flotation from the Hautalampi nickel/cobalt flotation tailings. Tailings from a separate bench flotation test for copper and nickel/cobalt concentrates were used as feed material for the sulphur removal tests.

The average sulphur feed grade in the sulphur flotation removal feed was 0.62%. The sulphur removal concentrate had a sulphur grade of 5.40%, and sulphur recovery to the concentrate was 56.1%. Sulphur flotation tailings had a remaining sulphur grade of 0.29% with 43.9% sulphur recovery.

12.5 Leaching Test Work by Outotec

Outotec carried out leaching test work with Hautalampi Ni/Co concentrate in 2019. The test work aimed to confirm concentrate leaching extractions, PLS purification efficiency and purity of the products. Laboratory scale test work was conducted for two concepts:

1. Processing and purification of nickel/cobalt concentrate to produce battery-grade Ni/Co sulphate solution
2. Processing and purification of nickel/cobalt concentrate to produce mixed hydroxide precipitate (MHP)

Ni/Co concentrate main grades in the tests were Ni 7.42 %, Co 1.24 % and Co 1.73 %.

The highest Ni and Co extractions achieved with atmospheric leaching were 88.9 % and 85 %, respectively. In pressure oxidation leaching (220 °C and 7 bars oxygen partial pressure) higher extractions were achieved, Ni 98.7 % and Co 99.7 %.

Iron was removed efficiently from PLS down to < 10 mg/l with jarosite precipitation follows by hydroxide precipitation at pH 3.5. In battery-grade sulphate production, copper and zinc were removed from PLS down to levels < 1 mg/l and 10 mg/l, respectively.

Extraction and stripping tests were done for impurity SX, cobalt SX and nickel SX. Based on the results it is expected to be possible to produce nickel and cobalt sulphate solutions for battery-grade metal sulphate productions.

In MHP precipitation 100 % Ni and Co recoveries were obtained in precipitation at pH 8, though at this pH level product contained impurities, mainly magnesia. Low-impurity MHP was achieved at lower pH of 7.5. In pH 7.5 Ni recovery was 95.6 % and Co recovery 97.4 %.

13 Mineral Resource Estimates

The Mineral Resource estimate has been prepared by Ville-Matti Seppä/ AFRY Finland Oy. Mr Seppä has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources, and Ore Reserves". The data that has been used for this work has been collected and compiled during the last mineral resource estimate work done by Outotec (Finland) Oy, dated 15th March 2009 and from the latest drilling campaigns conducted by FinnCobalt Oy in 2020 and 2021.

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

13.1 Data

The Hautalampi database was provided by FinnCobalt Oy in the form of an Acess® database containing collar locations, down-hole survey information, geologic data and assay results, density measurements, susceptibility measurements, structural measurements, and Q-value loggings. Digital copies of drill core photos, historic reports and Keretti mine maps / geological interpretations were also available. Drillhole locations from different drilling campaigns are presented in Figure 13-1.

The resource database contains data from four different drilling campaigns (Table 13-1) and assays from elements: Ag, As, Cd, Co, Cr, Cu, Fe, Mn, Mo, Ni, Pb, S, Sb, Zn and Au. The lithology file contains 61 different lithological units.

*Table 13-1 Summary of the Resource database. *Sludge drilling data was not used in resource estimation.*

				Samples	
Campaign	Holes	Length	Ni	Cu	Co
HA	74	8381.2	3 033	3 033	3 033
k	27	4 808	376	376	157
HL	100	11 153	2 736	2 734	2 736
OKU	301	62 411	6 575	6 610	6 534
Total	457	82 140	11 159	11 192	10 899
s*	102	1684	2 268	2 268	2 262

Notes:

Campaign	*Description*	*Assaying*
HA	*2020,2021 FinnCobalt Oy*	*ICP-AES*
HL	*2007–2008, Finn Nickel Oy*	*ICP/AAS*
OKU	*1950–1987, Outokumpu Oy Exploration and Keretti Mine*	*AAS/ICP*
k	*1950–1986, Outokumpu Keretti mine, underground drilling*	*AAS*
s	*1985–1987, Outokumpu, tunnelling project, sludge drilling.*	*X-Met*

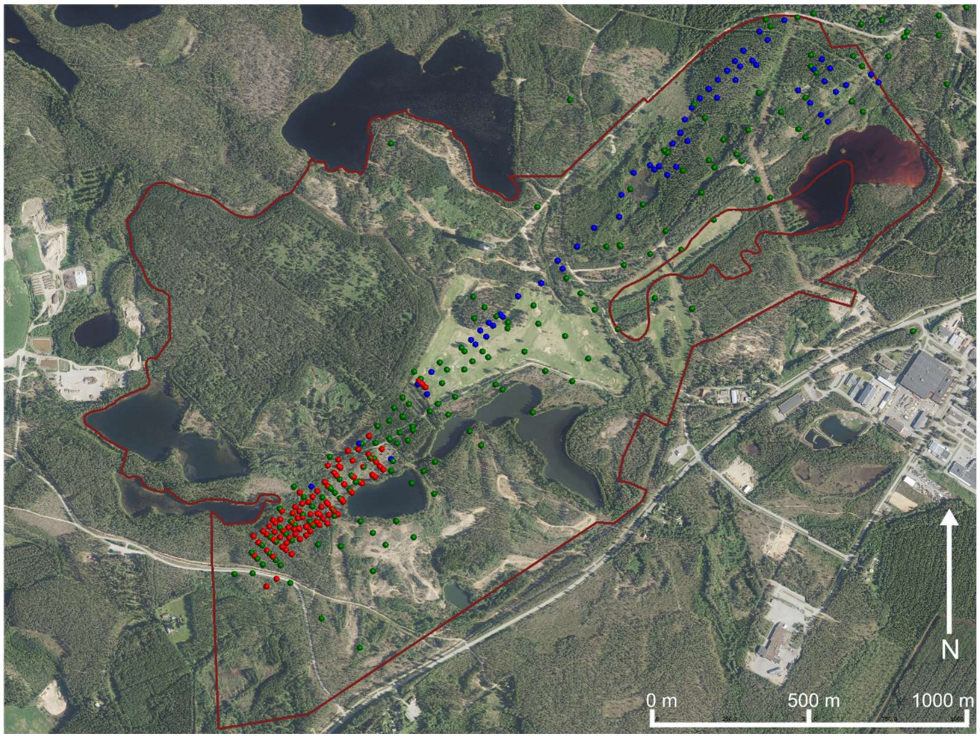


Figure 13-1 Drillholes from different drilling campaigns in relation to the mining concession.(Green=Outokumpu Oy, Red=Finn Nickel Oy, Blue = FinnCobalt Oy)

13.2 Resource modelling

The resource model for the Hautalampi project was created with Seequent LeapFrog Geo software. Numeric Ni-equivalent composites were created which were then used to create the resource solids. NiEq grade was calculated using the following prices:

- nickel US $20,000 /t
- copper US$ 9,000 /t
- cobalt US$ 60,000 /t

NiEq grade calculation = Ni% + (Co%*60000+Cu%*9000) /20000.

No metallurgical or recovery factors have been assumed at this stage of the Project. A USD/EUR exchange rate of 1.0 was used.

The modelling cut-off was selected to be 0.25 % NiEq, based on the following assumptions for underground mining and processing costs:

• Drilling and blasting	10 € / tonne
• Mucking	5 € / tonne
• Trucking	5 € / tonne
• Backfilling	5€ / tonne
• Processing costs	10 € / tonne
Total	35 € / tonne

The value of processed material with 0.25% NiEq and with a nickel price of UD $ 20,000 is 50.0 EUR which is well above the selected modelling cut-off and thus is suitable to be used in this project. The compositing length for NiEq composites was selected to be 1.0 meters. The residual end length of the composite less than 0.2 meters was discarded. The resource modelling process was aided by accepting small inclusions of material with less than 0.25 % NiEq inside the solids.

Thirteen ore lenses were modelled to the Hautalampi deposit using the intrusion tool in Leapfrog software (Figure 13-2). The interpolation was guided by inserting polylines and points and assigning individual trends to each created solid.

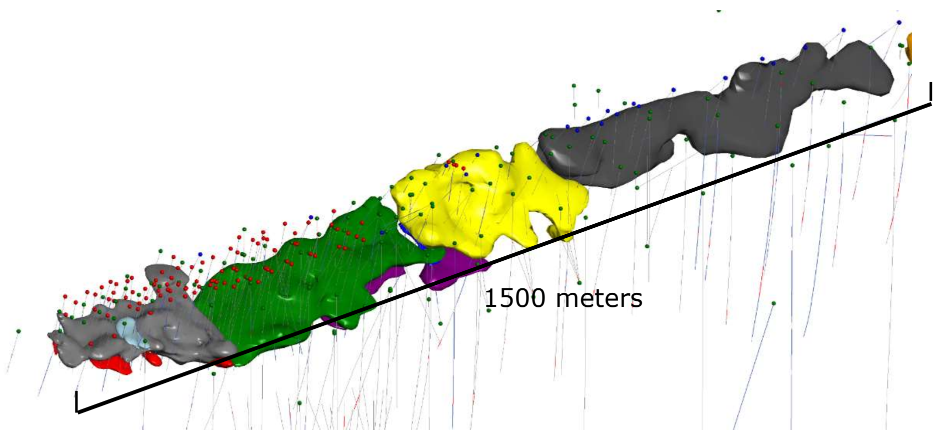


Figure 13-2 Oblique view of Hautalampi resource model. Looking towards North-East.

For the Mökkivaara area, four solids were modelled (Figure 13-3). Both Hautalampi and Mökkivaara models used drill holes drilled by Outokumpu, Finn Nickel Oy and FinnCobalt Oy, sludge holes were not used in resource modelling or grade estimation.

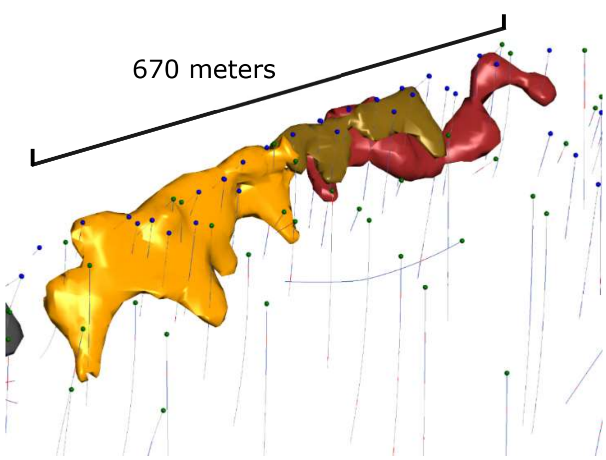


Figure 13-3 Oblique view of Mökkivaara resource model. Looking towards North-East.

Figure 13-4 shows the location of Hautalampi and Mökkivaara resource models in relation to FinnCobalt Oy's mining concession. The northernmost tip of the Mökkivaara model is outside of the mining concession and the volumes that are outside are not included in this report's estimations. The total length of the Hautalampi modelled mineralisation is 1500 meters along the strike of the deposit. The vertical extent of the Hautalampi model is 130 meters but local variations exist. Mökkivaara mineralisation length along the strike is 670 meters.

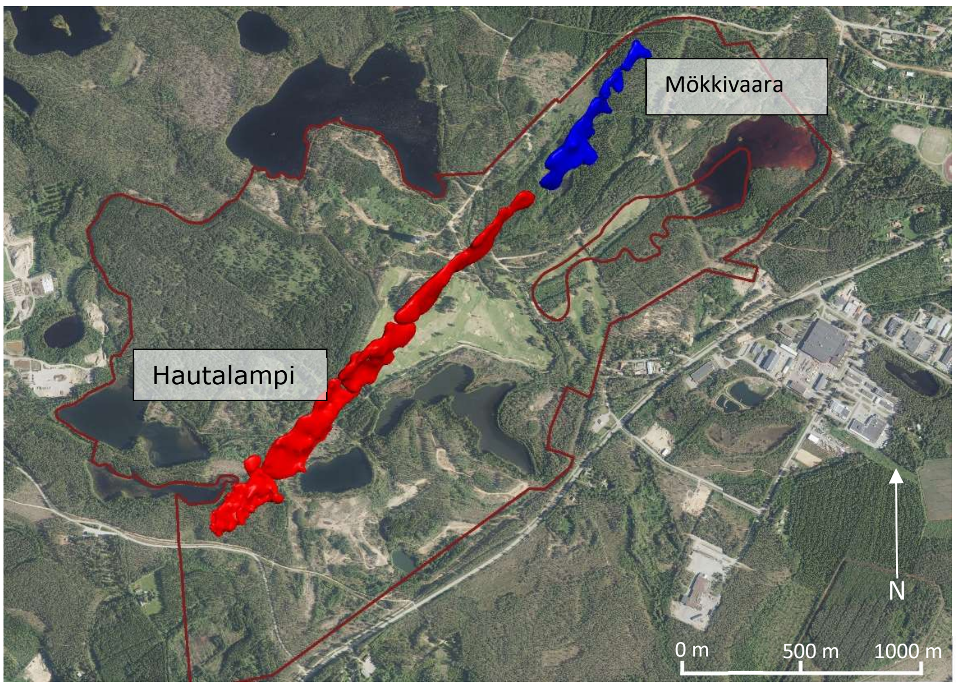


Figure 13-4 Aerial views of Hautalampi and Mökkivaara Resource models

13.3 Drill hole compositing

The resource estimate was based on resource intersections defined using the wireframes of the mineralized zones. Intersection data was used to extract samples for statistical analysis and for compositing the data for grade interpolation. Drill-hole sample composites were generated to standardize the data for further statistical evaluation which would eliminate any adverse effects related to sample length. The average assay interval in the database for samples above 0.3 % NiEq is 2.1 meters. The average assay interval length in HL and HA drilling campaigns was 1.3 meters. For the resource estimation, the drill holes were composited to be 1.5 meters in length. The selected length honours the created resource model boundaries. Basic statistics related to the Hautalampi composites used in grade estimates are presented in Table 13-2. The data set shows a fairly low Coefficient of Variation (CV) for Ni and Co. Usually, values less than 0.5 indicate a fairly well-behaved set of data, meaning low variability of the data. CV for Cu is higher than Ni or Co but it's still at an acceptable level and shows that the data can be used for predictive models. CV values greater than 2.0 or 2.5 indicates a distribution of data with significant variability, such that some predictive models may not be appropriate.

Table 13-2 Basic statistics of the Hautalampi composited data used in the grade estimations.

Variable	1.5 m composites Ore Ni (ppm)	Cu (ppm)	Co (ppm)
Number of samples	3734	3734	3732
Minimum value	0	0	0
Maximum value	27524	60310	7160
Mean	3005	2030	753
Median	2472	956	555
Geometric Mean	Not Calculated	Not Calculated	Not Calculated
Variance	3320627	10224244	438529
Standard Deviation	1822	3198	662
Coefficient of variation	0.61	1.57	0.88

Basic statistics of Mökkivaara composites are presented in (Table 13-3). CV for nickel and cobalt is at a good level but a value of 2.45 for copper can indicate a possible uncertainty in the models. In the Mökkivaara case, the high variance is caused by a small number of high Cu grades.

Table 13-3 Basic statistics of the Mökkivaara composited data used in the grade estimations.

Variable	1.5 m composites Ore Ni (ppm)	Cu (ppm)	Co (ppm)
Number of samples	690	690	690
Minimum value	213	6	24
Maximum value	7252	39100	2416
Mean	2244	1121	491
Median	2112	613	429
Geometric Mean	2119	559	401
Variance	656249	7529042	99424
Standard Deviation	810	2744	315
Coefficient of variation	0.36	2.45	0.64

13.4 Block model

The block model created for this resource estimation is made up of 5 m x 5 m x 5 m parent blocks and 2.5 m x 2.5 m x 2.5 m sub-blocks. The block model is rotated 45 degrees around the z-axis to match the general strike of the mineralized bodies. The selected block size was selected partly based on the drilling density and partly to match the geometric constraints. The summary of the block model parameters is given in Table 13-4.

Table 13-4 FinnCobalt resource block model parameters

Type	Y	X	Z	
Minimum Coordinates	6956362.5	4447194	-100	
Maximum Coordinates	6957162.5	4449994	160	
User Block Size	5	5	5	
Min. Block Size	2.5	2.5	2.5	
Rotation	-45	0	0	
Attribute Name	**Type**	**Decimals**	**Background**	**Description**
anisotropic_dist_to_nearest	Real	3	-99	
average_anisotropic_dist	Real	3	-99	
average_true_distance	Real	3	-99	
block_variance	Real	3	-99	
co	Float	3	0	resource grade (ppm)
cu	Float	3	0	resource grade (ppm)
cu_eq	Float	3	0	(Cu%+Ni%*20 000+Co%*60 000)/9 000
distance_to_dh	Real	3	-99	
fe	Float	3	0	resource grade (ppm)
id2_co	Float	2	-99	grade validation, inverse distance, Co
id2_cu	Float	2	-99	grade validation, inverse distance, Cu
id2_ni	Float	2	-99	grade validation, inverse distance, Ni
kriging_variance_co	Real	3	-99	
kriging_variance_cu	Real	3	-99	
krigink_efficiency	Real	3	-99	
krigink_variance	Real	3	-99	
lagrange_multiplier	Real	3	-99	
ni	Float	3	0	resource grade (ppm)
ni_eq	Float	3	0	Ni% + (Co%*60 000+Cu%*9 00)/20 000
nn	Float	2	0	closest sample
nn_co	Float	2	-99	grade validation, nearest neighbour, Co
nn_cu	Float	2	-99	grade validation, nearest neighbour, Cu
nn_ni	Float	2	-99	grade validation, nearest neighbour, Ni
number_of_drillholes	Integer	-	-99	

number_of_negative_weights	Integer	-	-99	
number_of_samples	Integer	-	-99	
resource_class	Integer	-	0	1=measured, 2=indicated, 3=inferred, 4= mineralised materials outside mining concession
s	Float	3	0	resource grade (ppm)
sg	Float	2		specific gravity
true_dist_to_nearest	Real	3	-99	
zn	Float	3	0	resource grade (ppm)

13.5 Geostatistical analysis and kriging parameters

For the Hautalampi deposit, mineralisation continuity for Ni, Cu, Co, S, Fe and Zn was examined by using variogram analysis. Variography was used to examine the spatial relationship between composites and identify the directions of mineralisation continuity and quantify the ranges of grade continuity. As a result, kriging parameters were obtained for resource estimation. The experimental variograms were calculated with the major axis aligned along the main mineralisation strike, the second was aligned in the plane of mineralisation at 90° to the first orientation. And the third was orientated perpendicular to the mineralisation plane, across the width of the mineralisation.

The variograms displayed reasonable structure, and the best continuity was observed to be in the plunge direction of the mineralisation. The variograms created for Ni, Cu and Co are shown in Figure 13-5, Figure 13-6 and Figure 13-7 respectively.

Figure 13-5 Experimental variogram models for Ni

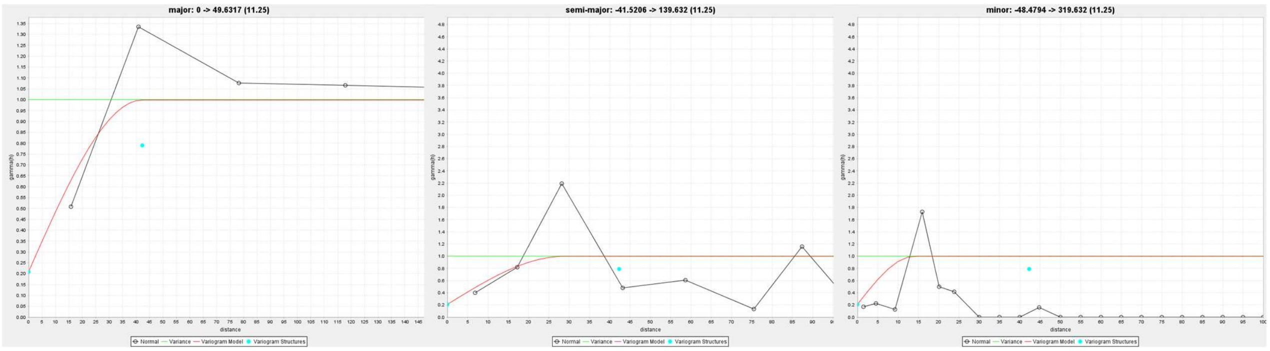


Figure 13-6 Experimental variogram models for Cu

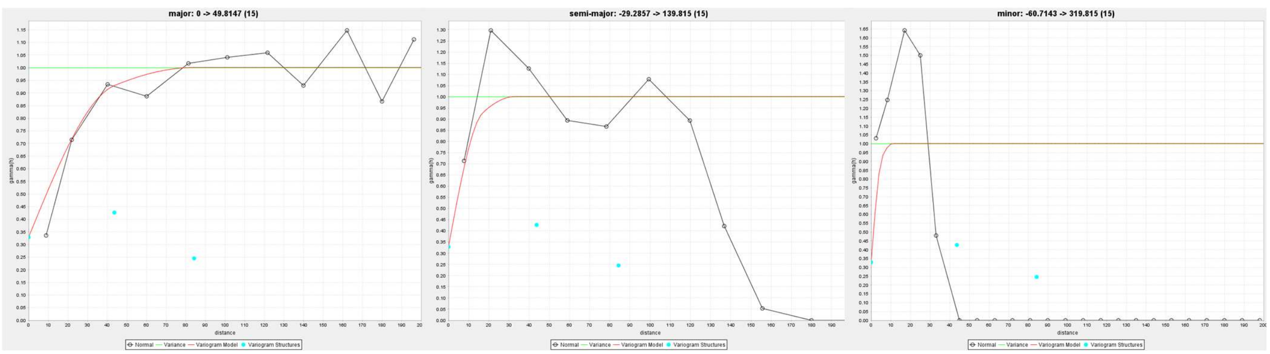


Figure 13-7 Experimental variogram models for Co

13.6 Grade interpolation

For the Hautalampi deposit, all elements (Ni, Cu, Co, S, Fe, Zn) were estimated by using Ordinary Kriging (OK) interpolation. For the Mökkivaara deposit, the elements were estimated by using an inverse distance squared (ID2) method. In Mökkivaara the major, semi-major and minor axes of the search ellipsoid were set to match the geometry of the Mökkivaara mineralisation.

Interpolation parameters for the estimation are shown below in Table 13-5. the Second pass with doubled range was used if empty blocks remained after initial interpolation.

Table 13-5 Interpolation parameters used

Hautalampi		C1	C2	Range	Azimuth	Plunge	Dip	Semi Ratio	Minor Ratio	Min samples	Max samples
Ni	Pass 1	0.638		115.0	39.02	0	-19.16	2.77	9.55	3	15
	Pass 2	0.638		180.0	39.02	0	-19.16	2.77	9.55	3	15
Cu	Pass 1	0.790		42.3	49.63	0	-41.521	1.425	3.115	3	15
Cu	Pass 2	0.790		90.0	49.63	0	-41.521	1.425	3.115	3	15
Co	Pass 1	0.427		43.7	49.82	0	-29.285	2.523	7.791	3	15
			0.25	84.4	49.82	0	-29.285	2.523	7.791	3	15
S	Pass 1	0.979		125.7	33.22	0	-38.572	3.112	9.193	3	15
Fe	Pass 1	0.881		129.8	58.78	0	-43	1	1.376	3	15
Zn	Pass 1	0.802		56.0	49.69	0	-50.19	1.288	2.89	3	15
	Pass 2	0.802		120.0	49.69	0	-50.19	1.288	2.89	3	15
Mökkivaara				Range	Azimuth	Plunge	Dip	Semi Ratio	Minor Ratio	Min samples	Max samples
All elements	Pass 1			120	30.2	0	-43.93	1.59	1.96	5	20

13.7 Bulk Density

Bulk density was estimated based on available density measurements. A total of 3113 samples were available and from those 478 were inside modelled resource solids. 1301 samples were outside of resource solids and were taken from samples below the selected modelling cut-off of 0.25 % NiEq. Statistics for the Density measurements are presented below (Table 13-6). A density of 2.82 was used for both waste and mineralised material.

Table 13-6 Statistics for bulk density data

	Number of intersections	Length	Mean	Standard deviation	Coefficient of variation	Variance
Inside resource model	478	223.54	2.82	0.132	0.047	0.018
waste	1301	321.08	2.82	0.160	0.058	0.027
All	3113	748.6	2.81	0.167	0.058	0.026

13.8 Mineral resource classification

Mineral Resource classification was considered based on drill hole spacing, continuity of mineralisation and data quality. Throughout the Hautalampi and Mökkivaara deposits, the grade continuity is good, with generally uniform Ni Cu and Co grades. Measured class material is located in the area where the drill section spacing is generally 20 meters. Indicated material has drill section spacing of 20 to 40 meters and inferred material has spacing generally greater than 50 meters. The kriging variance was also considered in the resource classification process. Generally, areas measured category material has a kriging variance of 0 to 0.4 whereas in an indicated class the variance is in the range of 0.2 to 0.6. In the measured class, the average true distance to samples is below 25 meters and in indicated class, in most cases, the average true distance is over 25 meters. Resource classes are illustrated below (Figure 13-8):

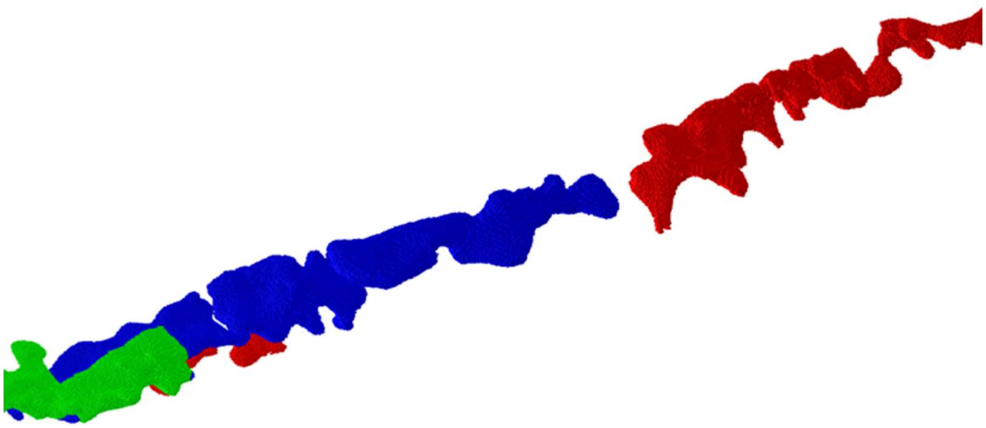


Figure 13-8 FinnCobalt Oy resources (Blue=Measured, Green= Indicated, Red =inferred resource class)

13.9 Mineral Resource Cut-off

The "reasonable prospects for eventual economic extraction" requirement mentioned in the JORC 2012 Code generally implies that quantity and grade estimates meet certain economic thresholds and that mineral resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recovery.

To ensure that the mineral resource estimate can be considered for eventual economic extraction, the following economic assumptions and operating costs have been used (Table 13-7Table):

Table 13-7 Assumed commodity prices and OPEX costs

Prices:		
Nickel price	20 000	US$ / tonne
Copper price	9 000	US$ / tonne
Cobalt price	60 000	US$ / tonne
US $/EUR exchange rate	1.00	
OPEX:		
Drilling and blasting	10	€ / tonne
Mucking	5	€ / tonne
Trucking	5	€ / tonne
Backfilling	5	€ / tonne
Processing costs	10	€ / tonne

A cut-off value for NiEq was estimated by using a NiEq value calculation (NVC). The (NVC) represents the combined metal values for nickel, copper and cobalt in the mineralized material.

The metal prices were provided by FinnCobalt Oy and they are based on the S&P Consensus price forecasts as of 4th of July 2022. The mining costs were estimated assuming contractor mining and using AFRY Finland Oy's in-house prices from similar-sized mining operations and FinnCobalt Oy's internal reference prices.

The NVC values (€/ tonne) for a mineralized material tonne were calculated using varying NiEq grades. The NVC was then compared against the operating cost (OPEX) broken down in Table 13-7 to see what the break-even value and the related cut-off grade should be. Using the assumed metal prices and operating costs the break-even cut-off grade is estimated to be 0.25 % NiEq (Figure 13-9). The selected modelling and reporting cut-off is supported by the estimated break-even cut-off. It should be noted that the NVC calculation is based on assumed economic and technical parameters presented earlier (Table 13-7).

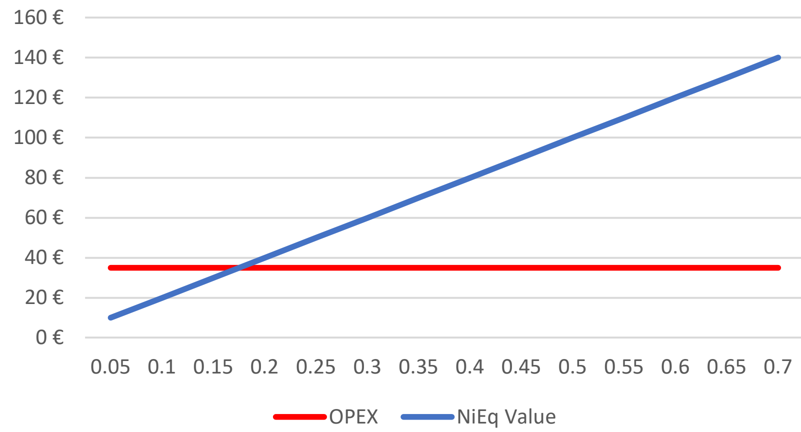


Figure 13-9 Cut-off breakeven calculation

13.10 Mineral Resources

Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore reserves (The JORC Code 2012) define a mineral resource as:

"A 'Mineral Resource' is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade (or quality), and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade (or quality), continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories.

Table 13-8 below summarizes the Hautalampi mineral resources using a NiEq cut-off grade of 0.25%. The author is not aware of any factors related to environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors which could materially affect the mineral resource estimate contained in this Report.

Table 13-8 Hautalampi Mineral Resources as of the September 29th, 2022 @ 0.25% NiEq cut-off

Hautalampi									
	Tonnes (t)	Ni %	Cu %	Co %	S %	Fe %	Zn %	Ni Eq %	Cu Eq %
Measured	2 808 000	0.35	0.26	0.08	2.02	3.92	0.04	0.70	1.57
Indicated	6 523 000	0.25	0.16	0.06	3.03	4.50	0.09	0.51	1.14
total	9 331 000	0.28	0.19	0.07	2.73	4.33	0.07	0.57	1.27
Contained Metals	tonnes	26 100 (Ni)	17 700 (Cu)	6 200 (Co)					

The Mökkivaara deposit and parts of Hautalampi are in Inferred Mineral Resource class. The FinnCobalt Oy's Inferred mineral resources are reported below (Table 13-9 and Table 13-10). There are not sufficient data to categorise these resources into Indicated resources. However, some of the inferred mineral resources could be upgraded to indicated resources with more detailed geological investigations and lithological modelling of the deposit.

Table 13-9 Hautalampi Mineral Resources as of the September 29th, 2022 @ 0.25% NiEq cut-off

Hautalampi									
		Ni	Cu	Co	S	Fe	Zn	Ni Eq	Cu Eq
	Tonnes	%	%	%	%	%	%	%	%
Inferred	216 000	0.21	0.12	0.02	1.91	3.79	0.07	0.32	0.72
Contained Metals	tonnes	450 (Ni)	260 (Cu)	40 (Co)					

Table 13-10 Mökkivaara Inferred Mineral Resources as of the June 21st, 2021 @ 0.3% Ni Eq cut-off

Mökkivaara									
		Ni	Cu	Co	S	Fe	Zn	Ni Eq	Cu Eq
	Tonnes	%	%	%	%	%	%	%	%
Inferred	3 188 000	0.22	0.13	0.05	2.32	4.41	0.09	0.44	0.97
Contained Metals	tonnes	7 000 (Ni)	4 100 (Cu)	1 600 (Co)					

13.11 Validation

Validation of the block model was performed visually against the drill hole data in cross-section views (Figure 13-10 and Figure 13-11). The block model was also validated at the domain level by comparing the mean values of the composited and estimated data (Table 13-11). These reviews did not reveal any inconsistencies between block model results and drill hole assays.

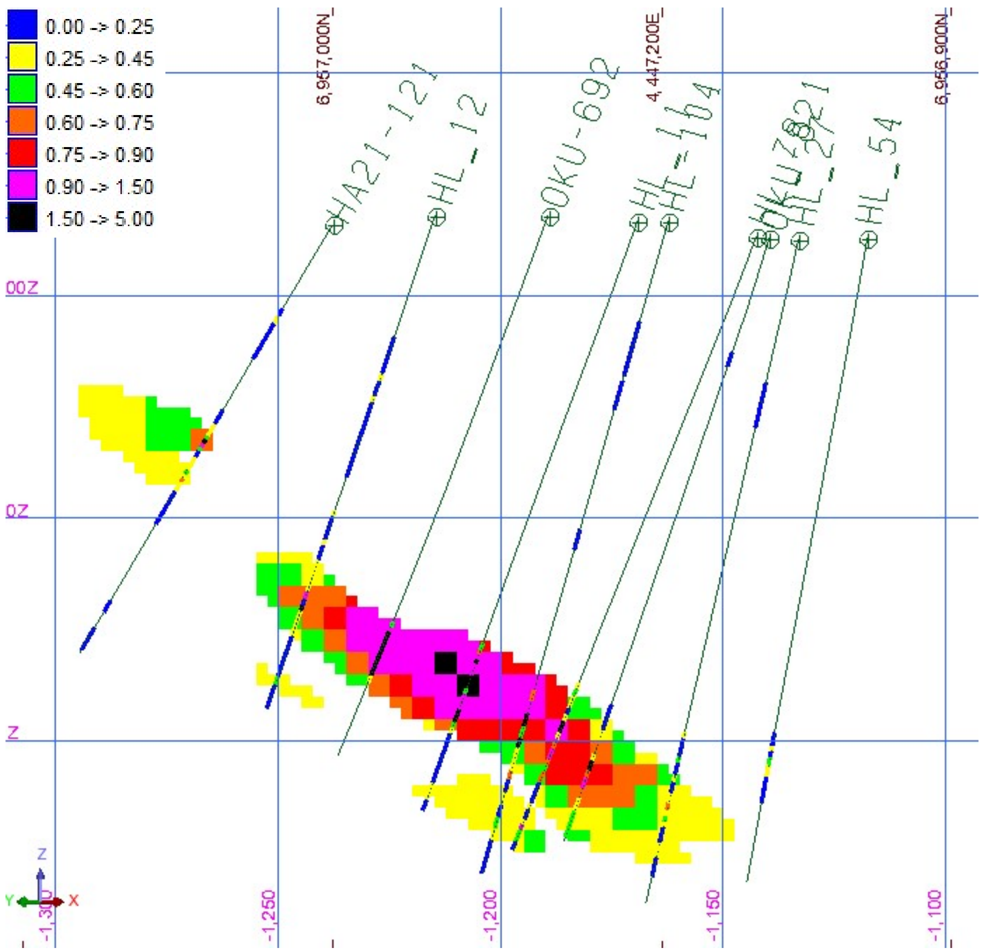


Figure 13-10 Profile 95 viewing North East displaying NiEq % in blocks and drill holes.

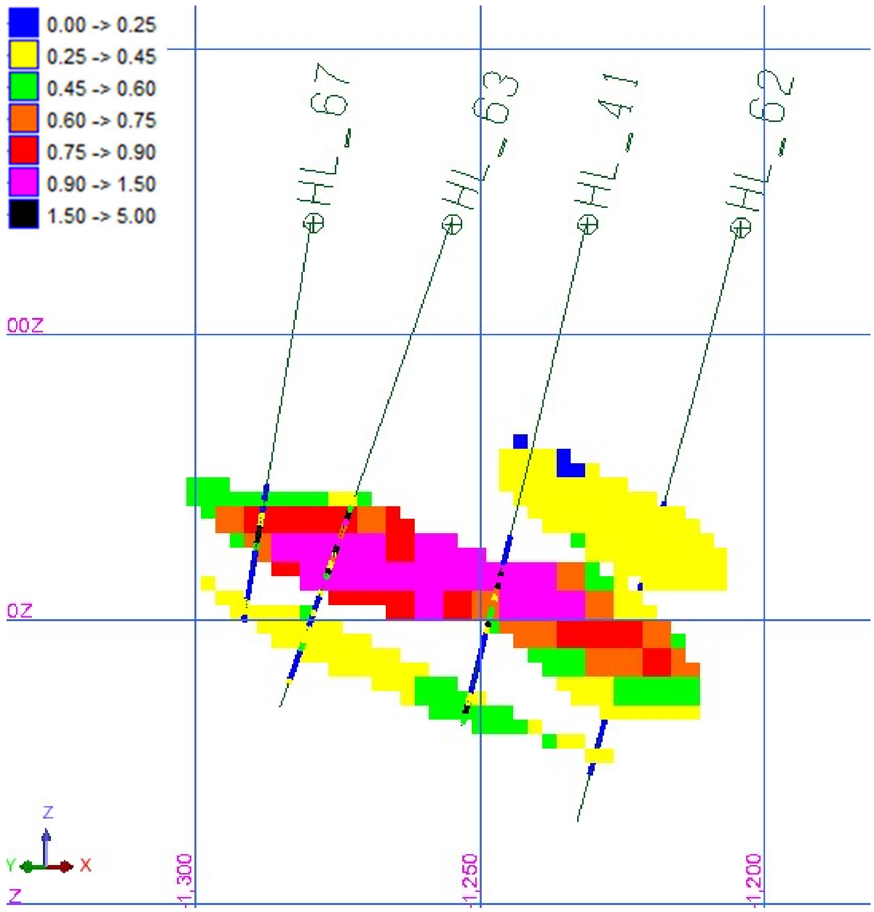


Figure 13-11 Profile 99+20m viewing North East displaying NiEq % grades in blocks and drill holes.

Table 13-11 Basic statistics of the block model and composites used to estimate the block grades

	1.5 m composites Inside resource model			Blockmodel Inside resource model		
Variable	Ni (ppm)	Cu (ppm)	Co (ppm)	Ni (ppm)	Cu (ppm)	Co (ppm)
Number of samples	3734	3734	3732	36517	36517	36517
Minimum value	0	0	0	0	0	0
Maximum value	27524	60310	7160	9875	17178	3444
Mean	3005	2030	753	2688	1737	557
Median	2472	956	555	2470	1204	528
Geometric Mean	Not Calculated	Not Calculated	Not Calculated	Not Calculated	Not Calculated	Not Calculated
Variance	3320627	10224244	438529	835277	2531405	170392
Standard Deviation	1822	3198	662	914	1591	413
Coefficient of variation	0.61	1.57	0.88	0.34	0.92	0.74

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

When comparing the volume of the geological 3D solids against the block model cells, a good congruence between the volumes can be seen. Figure 13-12 illustrates an oblique view of the Hautalampi 3D ore solid and the block model.

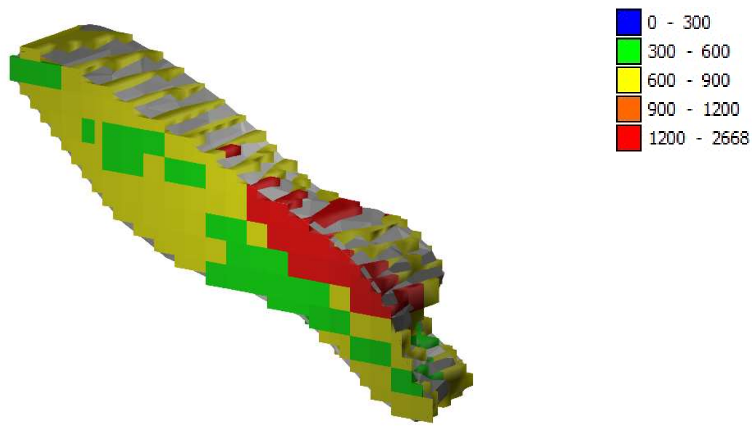


Figure 13-12 Volume comparison of 3D solid vs block model (Co grade)

The total volume difference between the 3D solid and the block model is only 0.02% (Table 13-12) and can be concluded that the volume difference is in a good range.

Table 13-12 Volumes of the 3D solid and the reported block model cells

The volume of 3D solid	**4 997 264 m^3**
The volume of reported block model cells	**4 996 172 m^3**
% difference	**0.02%**

The nearest neighbour (NN) method is a fast way to do a global validation of the resource model and it was used for the initial check-in block model validation for Mökkivaara and Hautalampi estimation. In addition to NN validation, also ID2 method was used to validate Hautalampi kriging results. Table 13-13 shows the comparison between the Ordinary kriging, inverse distance and the NN method. All methods produced identical grades for Ni, Cu and Co.

Table 13-13 Comparison between estimation methods

Resource Class	Tonnes	Ni			Cu			Co		
		Kriging	Nearest neighbour	Inverse distance	Kriging	Nearest neighbour	Inverse distance	Kriging	Nearest neighbour	Inverse distance
	Mt	%	%	%	%	%	%	%	%	%
Measured	2.81	0.35	0.35	0.35	0.26	0.26	0.27	0.08	0.08	0.08
Indicated	6.52	0.25	0.25	0.25	0.16	0.16	0.16	0.06	0.07	0.07
Total M&I	9.33	0.28	0.28	0.28	0.19	0.19	0.19	0.06	0.07	0.07
Inferred	3.40	0.22	0.22	0.22	0.13	0.14	0.13	0.05	0.05	0.05

Swath plot analysis showed a good correlation between the composited grades versus the estimated grades from the block model. Northing Swath plot analyses for Ni, Cu and Co grades are presented in Figure 13-13, Figure 13-14 and Figure 13-15. Easting Plots in Figure 13-16, Figure 13-17 and Figure 13-18. Figure 13-19, Figure 13-20 and Figure 13-21 illustrates the elevation swath plots.

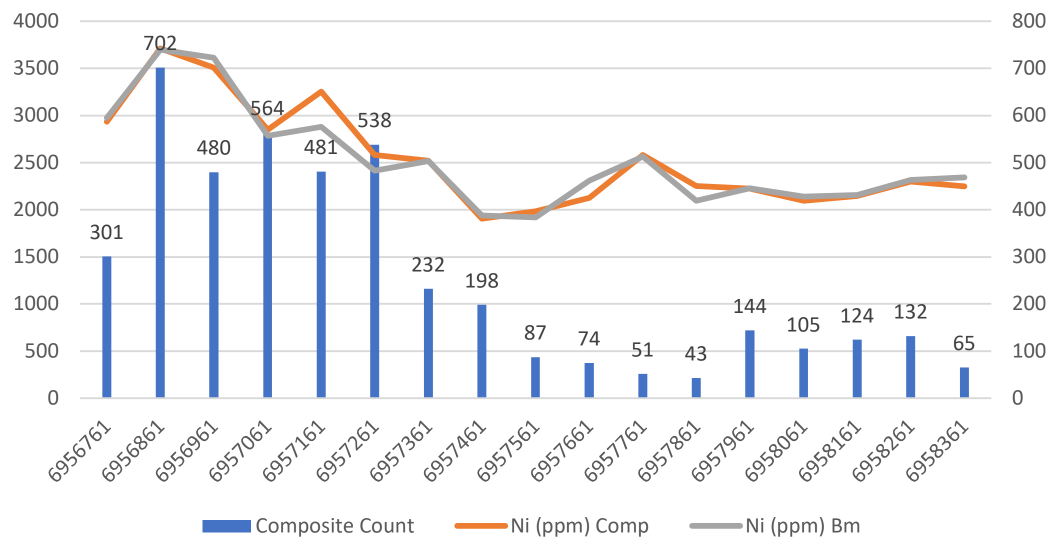


Figure 13-13 Swath Analysis: Ni, Northing

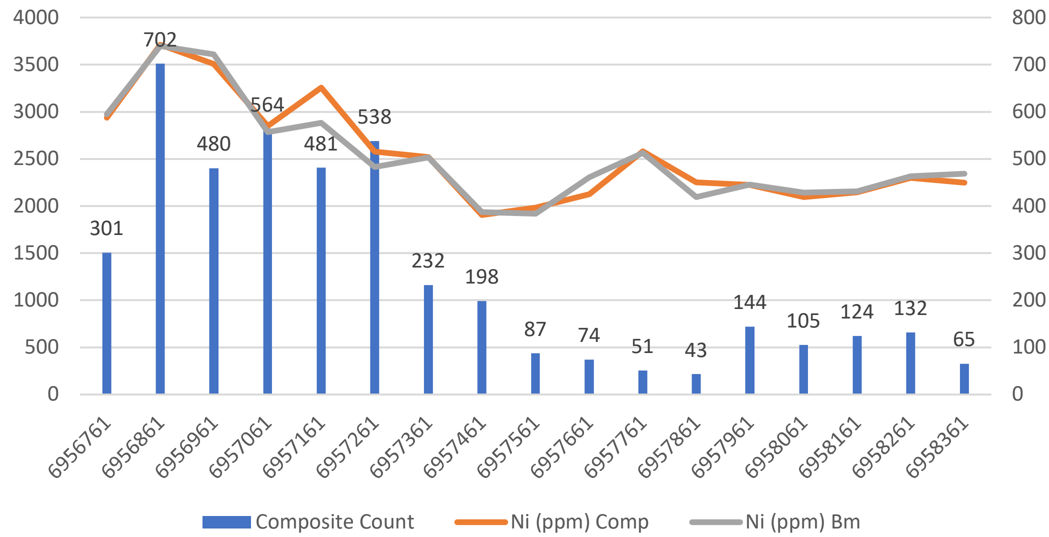


Figure 13-14 Swath Analysis: Cu, Northing

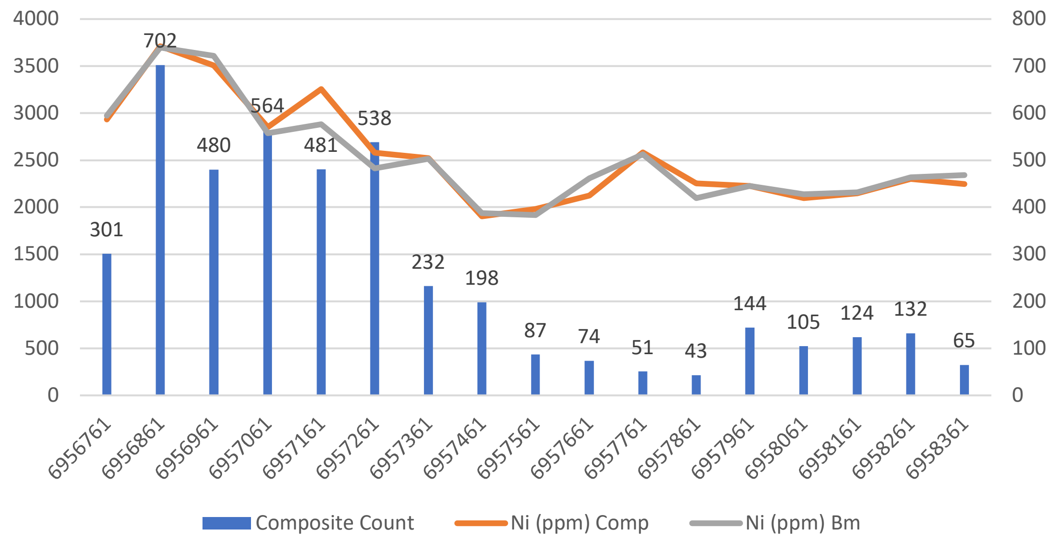


Figure 13-15 Swath Analysis: Co, Northing

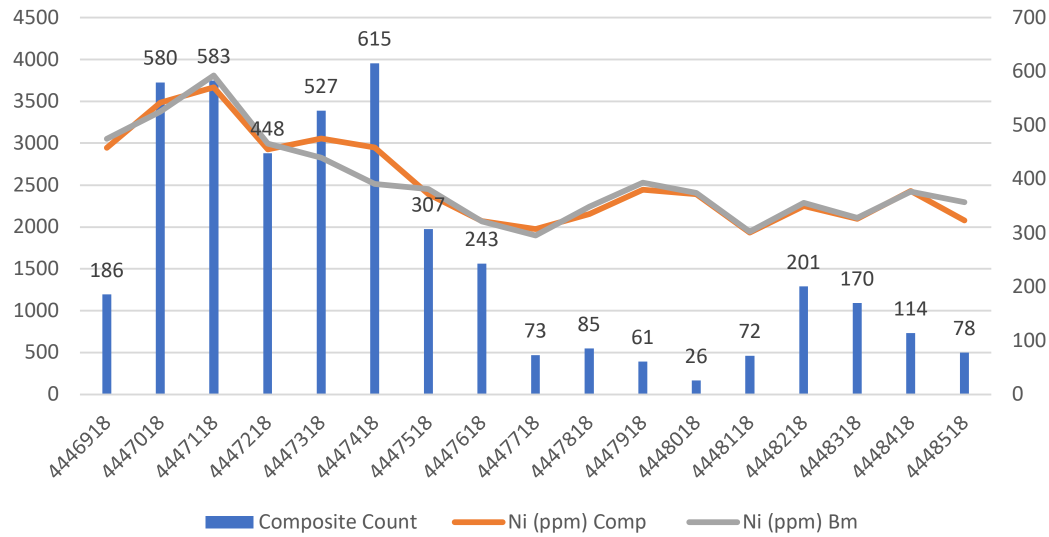


Figure 13-16 Swath Analysis: Ni, Easting

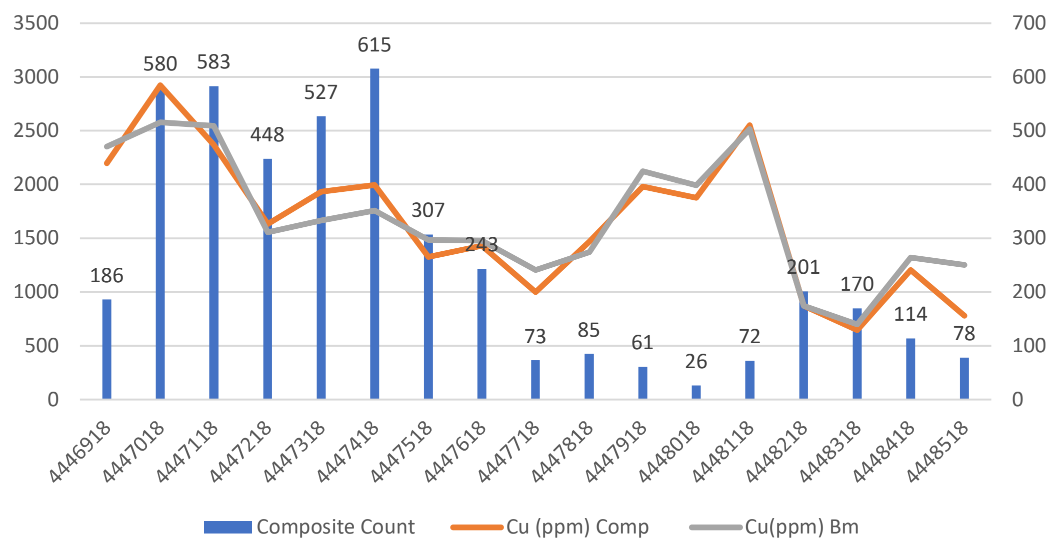


Figure 13-17 Swath Analysis: Cu, Easting

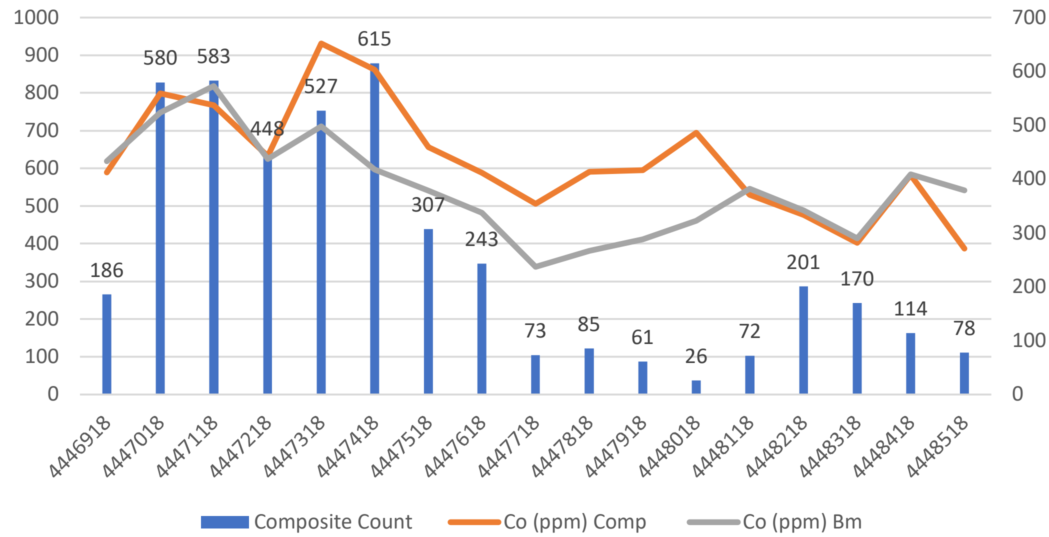


Figure 13-18 Swath Analysis: Co, Easting

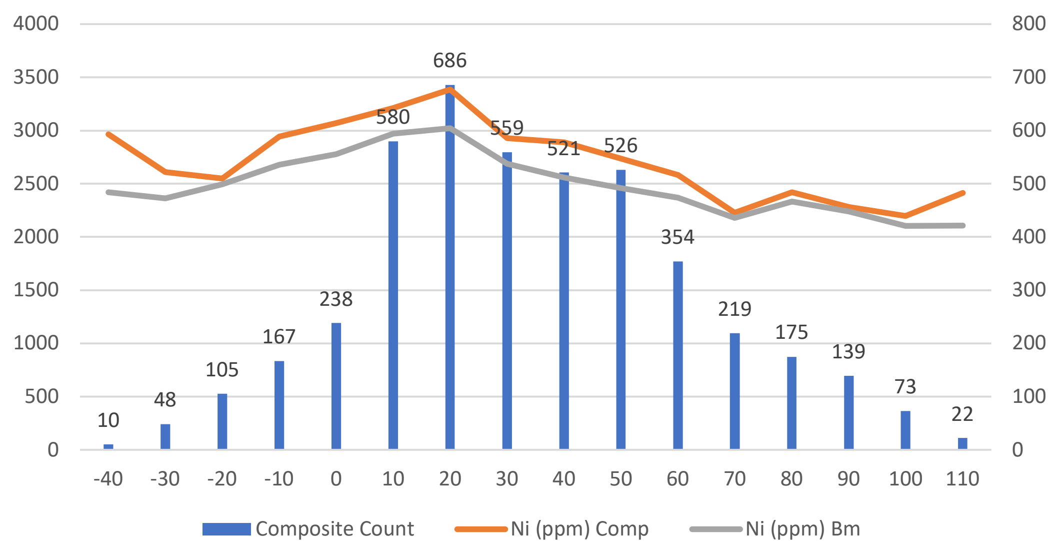


Figure 13-19 Swath Analysis: Ni, Elevation

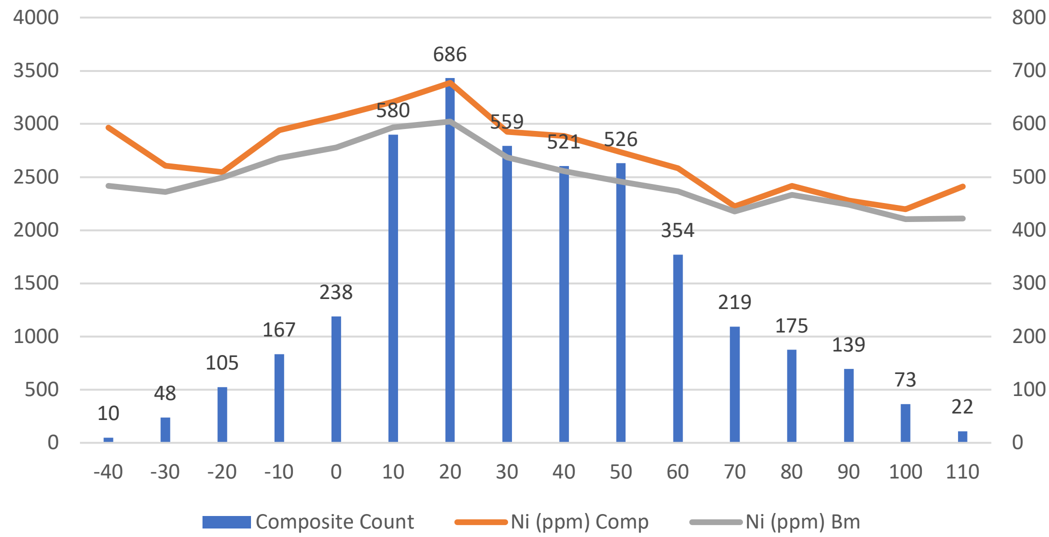


Figure 13-20 Swath Analysis: Cu, Elevation

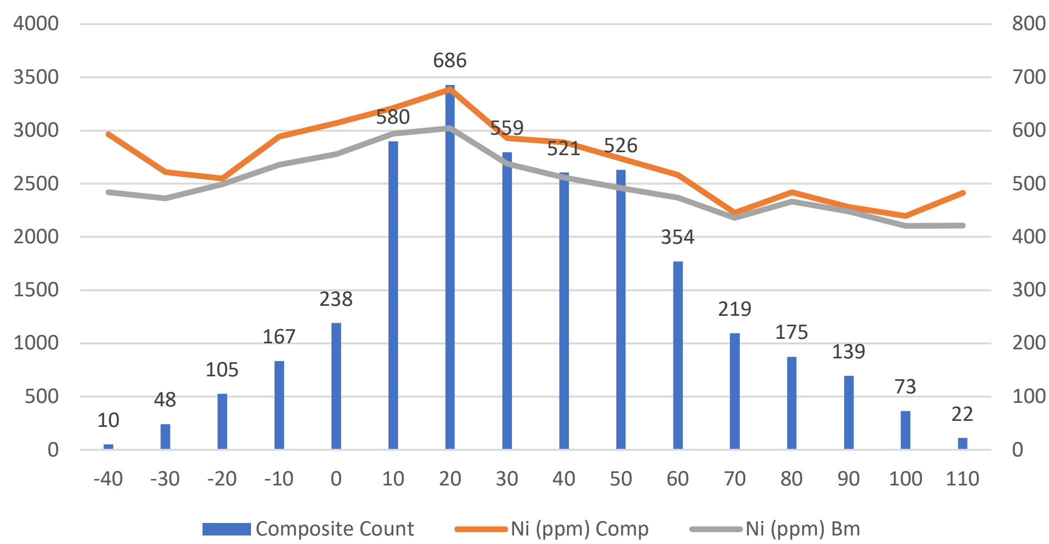


Figure 13-21 Swath Analysis: Co, Elevation

13.12 Sensitivity of Mineral Resources

The relationship between the NiEq cut-off grade and the resource tonnage is shown in Figure 13-22. The effects of selected cut-off grade on Measured and Indicated Mineral resources are shown in Table 13-14. In Table 13-15the sensitivity of Inferred Mineral resources is shown against varying cut-off grades.

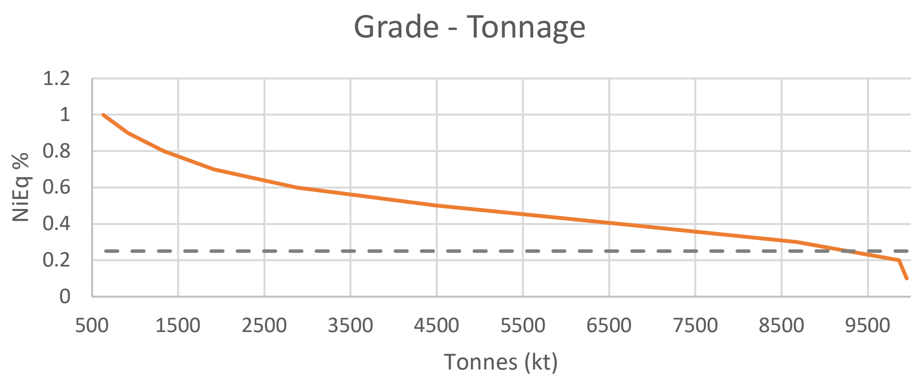


Figure 13-22 Hautalampi Grade-Tonnage curve for Measured & indicated resource class material, dashed line shows the selected cut-off.

Table 13-14 Sensitivity of Measured + Indicated Mineral Resource to varying cut-off grades

		Average grade			Contained Metal		
	Tonnes	Ni	Cu	Co	Ni	Cu	Co
Cut-off	kt	%	%	%	kt	kt	kt
0.1	9 949	0.27	0.18	0.06	27.2	18.2	6.0
0.2	9 860	0.27	0.18	0.06	27.1	18.1	5.9
0.25	**9 331**	**0.28**	**0.19**	**0.06**	**26.1**	**17.7**	**5.9**
0.3	8 673	0.29	0.20	0.07	24.8	17.1	5.9
0.4	6 634	0.31	0.23	0.08	20.4	15.1	5.2
0.5	4 490	0.34	0.27	0.09	15.3	12.3	4.1
0.6	2 876	0.38	0.33	0.11	11.0	9.5	3.1
0.7	1 911	0.42	0.39	0.12	8.0	7.5	2.3
0.8	1 331	0.45	0.45	0.13	6.0	6.0	1.7

Table 13-15 Sensitivity of Inferred Mineral Resource to varying cut-off grades

		Average grade			Contained Metal		
	Tonnes	Ni	Cu	Co	Ni	Cu	Co
Cut-off	kt	%	%	%	kt	kt	kt
0.1	3 489	0.22	0.13	0.05	7.7	4.5	1.7
0.2	3 478	0.22	0.13	0.05	7.7	4.5	1.7
0.25	3 404	0.22	0.13	0.05	7.5	4.4	1.7
0.3	3 224	0.22	0.13	0.05	7.2	4.3	1.6
0.4	1 694	0.24	0.19	0.06	4.0	3.2	1.0
0.5	515	0.26	0.37	0.08	1.3	1.9	0.4
0.6	219	0.27	0.64	0.08	0.6	1.4	0.2
0.7	92	0.25	1.15	0.09	0.2	1.1	0.1
0.8	51	0.23	1.73	0.09	0.1	0.9	0.0

14 Mineral reserve estimates

Hautalampi Ore Reserve Estimate is based on the JORC (2012) compliant Mineral Resource Estimate prepared by AFRY Finland Oy (October 2022).

Table 14-1summarises the Ore Reserve of Hautalampi deposit at 30 €/t NSR cut-off.

Table 14-1 Hautalampi Ore Reserves as of 7th March 2023 @30€/t NSR Cut-off. Mineral Resources are inclusive of these Ore Reserves.

Underground Ore Reserve		Grade			Contained Metals		
	Tonnes	Ni	Cu	Co	Ni (t)	Cu (t)	Co (t)
Proven	1 871 000	0.36 %	0.30 %	0.09 %	6 800	5 700	1 600
Probable	2 693 000	0.25 %	0.19 %	0.07 %	6 900	5 300	1 900
Total	4 564 000	0.30 %	0.24 %	0.08 %	13 700	11 000	3 500

Internal dilution is estimated from the waste proportions of the waste rock in the drill cores inside the ore solids and it is taken into account in the resource estimation.

The external dilution is estimated as a constant percentage. In Hautalampi underground mining operation the external dilution percentage is estimated to be 10 % in mining areas a and b and 15 % in c and d areas. Mining recovery for the underground ores was estimated to be 95 %. The diluting material was estimated based on 100 % waste.

The mining recovery and waste rock dilution are considered to be typical for this type of deposit and operation. Hautalampi Ore Reserves are illustrated in Figure 14-1.

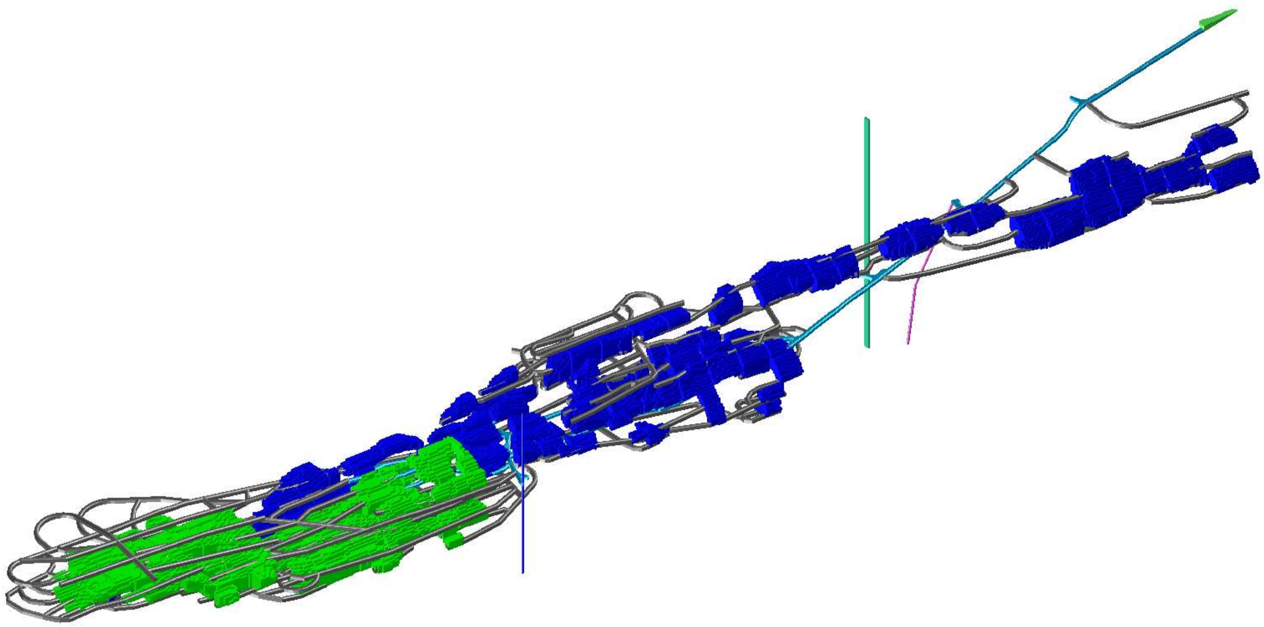


Figure 14-1 Hautalampi Ore Reserves, Green= Proven Reserve, Blue = Probable Reserve.

Net Smelter Return value (NSR) €/ tonne was calculated based on the following economic and processing assumptions (Table 14-2):

Table 14-2 NSR calculation values

	Recovery (%)	payable (%)	USD/t
Ni	82	70	20 000
Co	82	50	60 000
Cu	86.5	90	9 000
USD/EUR = 1.00			

For the mine design and reserve reporting purposes, a 30€/t NSR cut-off value was used. All excavated material needed to be above 30€/t NSR value and the average NSR value for the individual stope needs to be above 50€/t. The selected NSR cut-off is justified when compared against the Hautalampi project estimated OPEX (Table 14-3).

Table 14-3 OPEX breakdown

Costs	€/t
Administration	3.10
Mining	21.55
Concentrator	15.33
Logistics	0.98
	41.16

Additional mineral resources after ore reserve conversion are presented below in Table 14-4, Table 14-5 and Table 14-6. The remaining mineral reserves are illustrated in Figure 14-2.

Table 14-4 Hautalampi *Mineral resource as of 7th March 2023. @ 0.25 Ni Equivalent Cut-off. Mineral resources are additional to ore reserves.*

Hautalampi									
	Tonnes (t)	Ni %	Cu %	Co %	S %	Fe %	Zn %	Ni Eq %	Cu Eq %
Measured	254 000	0.26	0.07	0.04	2.03	4.09	0.01	0.41	0.92
Indicated	1 049 000	0.21	0.07	0.04	2.14	3.80	0.07	0.36	0.80
total	1 303 000	0.22	0.07	0.04	2.12	3.86	0.06	0.37	0.82
		Ni	Cu	Co					
Contained Metals	tonnes	2 900	1 000	500					

Table 14-5 Hautalampi inferred Mineral resources as of 7th March 2023. @ 0.25 Ni Equivalent Cut-off. Mineral resources are additional to ore reserves.

Hautalampi									
	Tonnes	Ni %	Cu %	Co %	S %	Fe %	Zn %	Ni Eq %	Cu Eq %
Inferred	216 000	0.21	0.12	0.02	1.91	3.79	0.07	0.32	0.72
		Ni	Cu	Co					
Contained Metals	tonnes	450	260	40					

Table 14-6 Mökkivaara inferred Mineral resources as of 7th March 2023. @ 0.25 Ni Equivalent Cut-off. Mineral resources are additional to ore reserves.

Mökkivaara									
	Tonnes	Ni %	Cu %	Co %	S %	Fe %	Zn %	Ni Eq %	Cu Eq %
Inferred	3 188 000	0.22	0.13	0.05	2.32	4.41	0.09	0.44	0.97
		Ni	Cu	Co					
Contained Metals	tonnes	7 000	4 100	1 600					

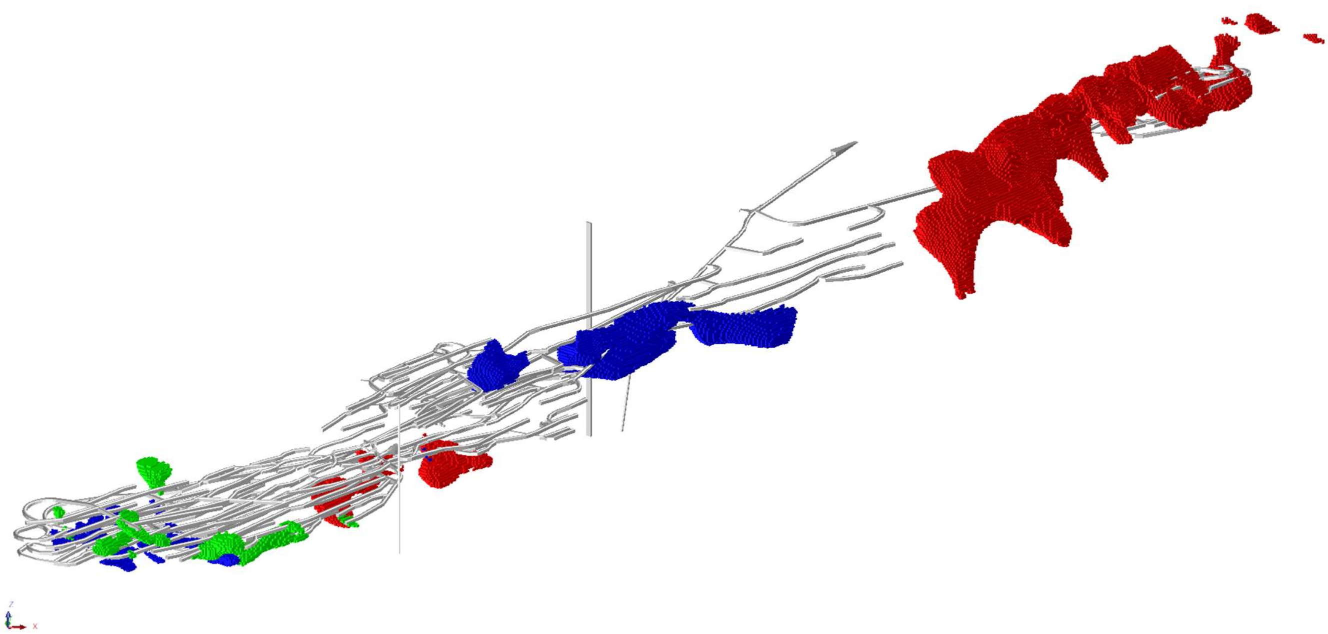


Figure 14-2 Additional mineral resources to ore reserves. Green=Measured resources, Blue= Indicated resources, Red= inferred resources.

15 Mining

15.1 Introduction

The underground mine design for Hautalampi has been performed by JK Kaivossuunnittelu Oy. The updated mine design utilizes the existing decline and drifts. The general mine layout is presented in Figure 15-1, Figure 15-2, and Figure 15-3.

The estimated production rate for the mine has been based on a sustainable mining rate. The production reaches a level of 492 000 tpa in Year 2 and roughly maintains at 470 ktpa level for the next nine years. The total life of mine is estimated to be twelve years.

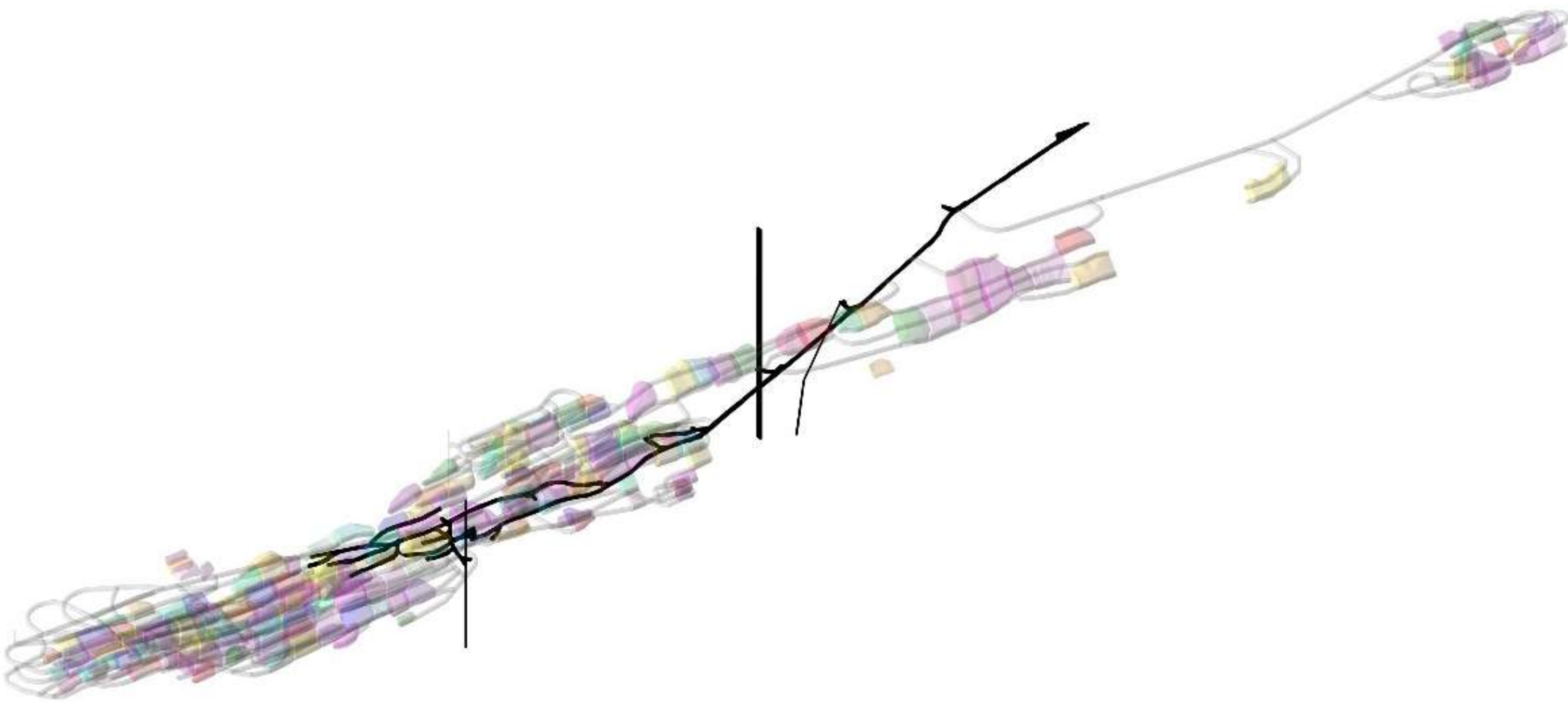


Figure 15-1. Hautalampi mine design in perspective view (design by JK Kaivossuunnittelu Oy). The existing mine decline, shafts, and drifts are presented in black colour. Mökkivaara mineralization is included and shown in the top right corner.

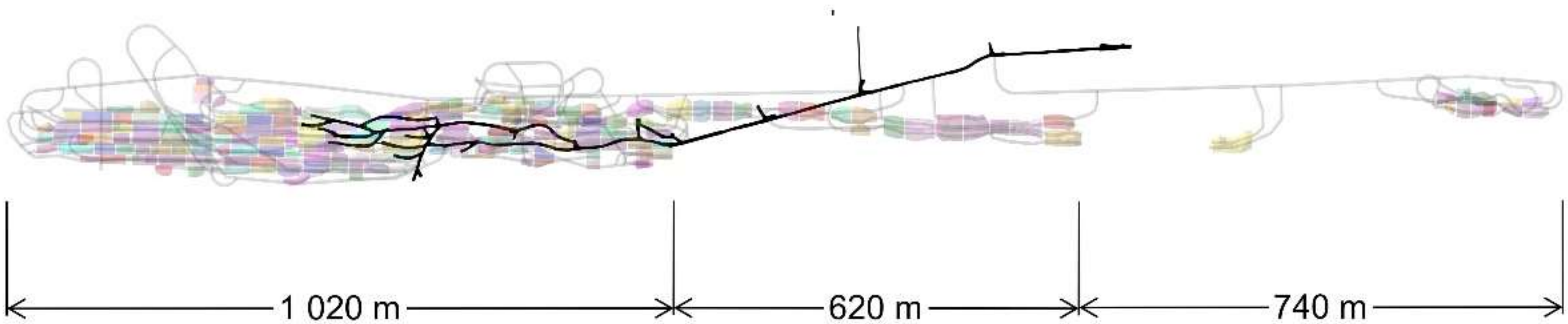


Figure 15-2. Hautalampi mine design in plan view (design by JK Kaivossuunnittelu Oy). The existing mine decline, shafts, and drifts are presented in black colour. Mökkivaara mineralization is included and shown on the right.

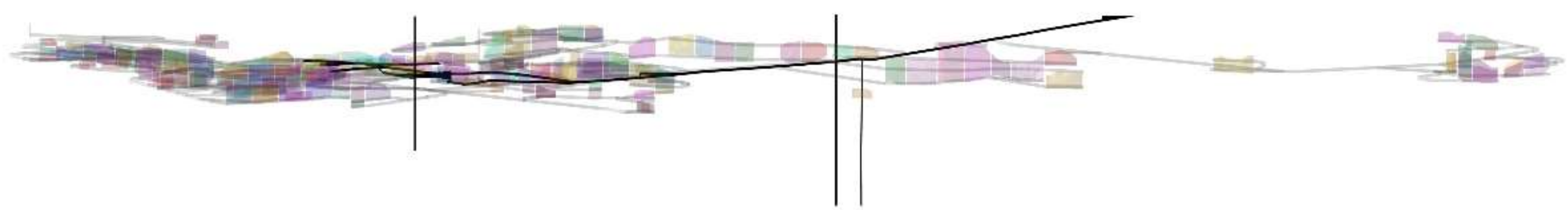


Figure 15-3. Hautalampi mine design in longitudinal view (design by JK Kaivossuunnittelu Oy). The existing mine decline, shafts, and drifts are presented in black colour. Mökkivaara mineralization is included and shown on the right.

15.2 Decline access

Accessing the mine will be through an existing decline, which is presented in Figure 15-4 and Figure 15-5. The existing decline is 5 m wide and 5 m high. Furthermore, the inclination varies from 1:6 to 1:14. All traffic (including transport) to and from the mine will go through the decline. The decline is also conveniently reached from a nearby road. However, the existing roads and the decline must be reconstructed, meaning that the surface of the roads must be removed, and a new surface constructed.

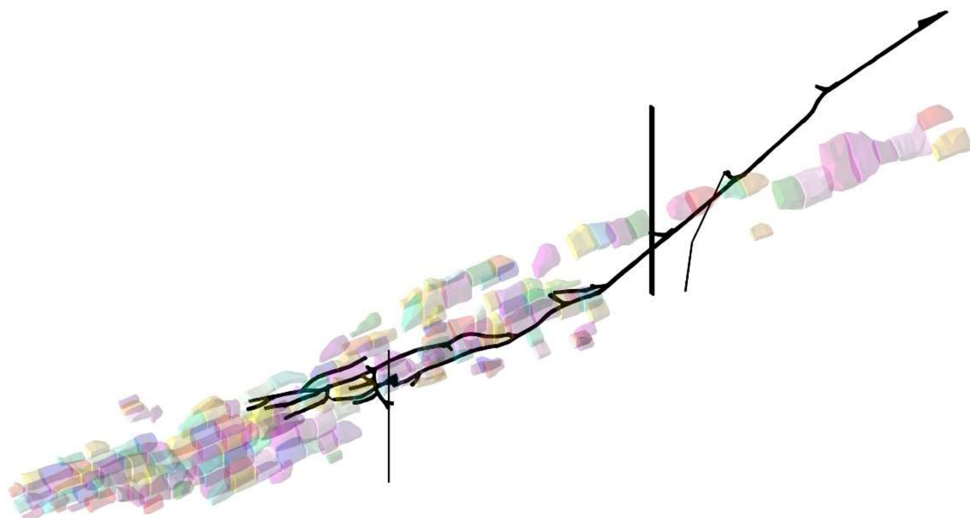


Figure 15-4. Hautalampi existing mine decline, shafts, and drifts in relation to designed stopes.

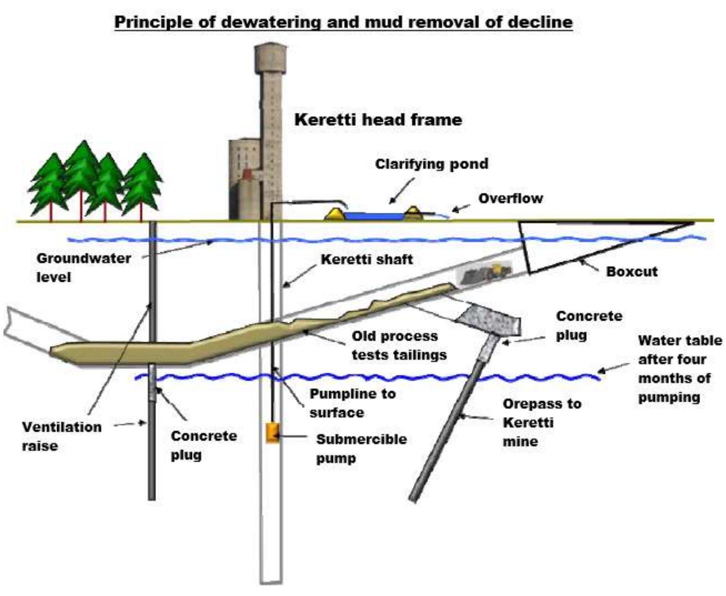


Figure 15-5. Dewatering and mud removal of the existing decline, conceptual design.

Soil from the local surroundings has been used to fill the box-cut of the access decline, and it is estimated that 10 000 m^3 of soil has been used. Before this box-cut filling, tailings from the Talvivaara black schist nickel leaching residue were dumped into the box-cut. Roughly 25 000 m^3 of tailings is estimated to have been dumped. It is also unknown to what extent the tailings have flowed and filled the decline, so this study assumes that about 1000 m of the decline will be full of tailings.

In a crosscut of the decline in Figure 15-5, about 380 m from the entrance of the decline, there is an ore pass connected to the old Keretti mine ore handling system. It is assumed that part of the tailings could have flowed into the ore pass.

In the upper part of the decline, there is a significant water inflow to the tunnel, which is planned to be grouted tightly in during the decline maintenance.

The 10 000 m^3 of soil currently placed in the box-cut will be removed by an excavator. The soil is then transported 500 m by trucks to a stockpile area close to the old Keretti mine headframe.

There is uncertainty about how leaching residue and other sludge have settled in the decline, and whether it is possible to muck it or if it must be pumped to a vessel or tank lorry and transported to the surface. It is estimated that 10 m of the decline can be cleaned during one day in two working shifts if pumping must be used.

The grouting of the water leakages in the upper part of the decline is calculated to take two weeks with two working shifts. Drilling of the grouting holes is estimated to take one day at 16 hours/day and the cement grouting to take nine days at 16 hours/day. Cement consumption is estimated to be five tons.

The washing of the mud and scaling of the decline is estimated to advance 100 m/shift. This work will require a platform and a workforce of three men.

The bulkhead which prevents water inflow from the shaft, which is connected to the Keretti mine, is planned to be a 5 m thick cement plug in the drift from the decline to the shaft. Two-meter-long holes are drilled in a 1 m grid into the roof, walls, and bottom of the drift and 4 m-long armour steel is cement grouted into these holes. After this, the cement is cast for the plug.

As summary, the rehabilitation of the existing decline and its facilities requires the following tasks:

- Lowering of the water level in the Keretti shaft
- Removal of the fill from the box-cut 10 000 m^3
- Removal of the tailings from the decline of 25 000 m^3
- Grouting the water inflows in the upper part of the decline
- Washing of the mud from filled parts of the decline

- Scaling and supporting the existing decline
- Construction of the decline road
- Construction of the bulkhead to the ore pass in section 71
- Emptying the ventilation shaft in section 88 from mud and water
- Construction of the bulkhead to the ventilation shaft to prevent water inflow from the old Keretti mine.

The cost of the above-mentioned tasks is estimated to be 820 000€. This sum is based on earlier cost estimates and overall inflation. See Table 15-1 for a more detailed cost breakdown structure for the decline rehabilitation.

Table 15-1. Cost estimates for the decline rehabilitation.

Task description	Cost estimate in 2009	Cost estimate for 2023 (price index = 6.3 %)
Removal of fill from box-cut	28 000 €	34 000 €
Removal of mud from decline	206 672 €	255 000 €
Injection of water leakage	16 048 €	20 000 €
Washing of mud-filled rooms	12 080 €	15 000 €
Scaling of existing rooms	7 248 €	9 000 €
Support of existing rooms	70 400 €	87 000 €
Road construction in decline	17 720 €	22 000 €
Building of bulkhead (section 71)	66 500 €	82 000 €
Drilling 60 m hole (section 88)	6 000 €	7 000 €
Removal of the bulkhead (section 88)	880 €	1 000 €
Removal of mud (section 88)	1 141 €	1 000 €
Building of bulkhead (section 88)	66 500 €	82 000 €
Equipment	166 375 €	205 000 €
Sum	**665 564 €**	**820 000 €**

15.3 Ground conditions and ground support

15.3.1 Tunnel support

Tunnel support requirements have been estimated using the historical experience from the development of the existing decline and experience from similar rock conditions encountered during mining in the 1980s at the now-closed Vuonos mine. It is estimated that the tunnel roof and shoulders will require basic support at all intersections. The basic support consists of a 30 mm thick layer of fibre-reinforced shotcrete and the use of fully grouted 2.4 m long rock bolts at a spacing of 1.5 m.

According to regulations, ground condition control and scaling of the tunnels must be done every third month, and if necessary, even more often. The active tunnels, close to where mining is progressing, must be inspected and scaled after every blast. This applies also to the decline and other spaces in the mine.

15.3.2 Stope support

Stope development, which is also assumed to be within competent ground conditions, will be subject to stress redistribution and blast vibration damage, especially close to the stope brows or draw points.

The backs and the walls of the ore drives (stope development drifts) will require basic support consisting of a 30 mm thick layer of fibre-reinforced shotcrete and the use of fully grouted 2.4 m long rock bolts at a spacing of 1.5 m. Additionally, six-meter-long cement grouted cable bolts are required to support the backs and the roof of the stopes. The cable bolts will be installed at a spacing of two meters.

15.3.3 Ventilation raises

Excavation of the ventilation raises can be done by raise boring or drill and blast. In total, 3 raises are planned to be utilized for Hautalampi underground mine. Figure 15-6 shows the three ventilation raises for Hautalampi underground mine. There is one existing raise which will work as an exhaust air raise, during the development phase, and additionally, two new ventilation raises are required to be excavated. These new ventilation raises will both work as exhaust air raises, and once production starts the old ventilation raise will be changed to a fresh air raise. A 30 mm layer of fibre-reinforced shotcrete together with 2.1 m long fully grouted rock bolts at a spacing of 1.5 m will be required for reinforcement. Due to the thick overburden, a concrete collar structure will be used in the construction of the upper part of the ventilation raises.

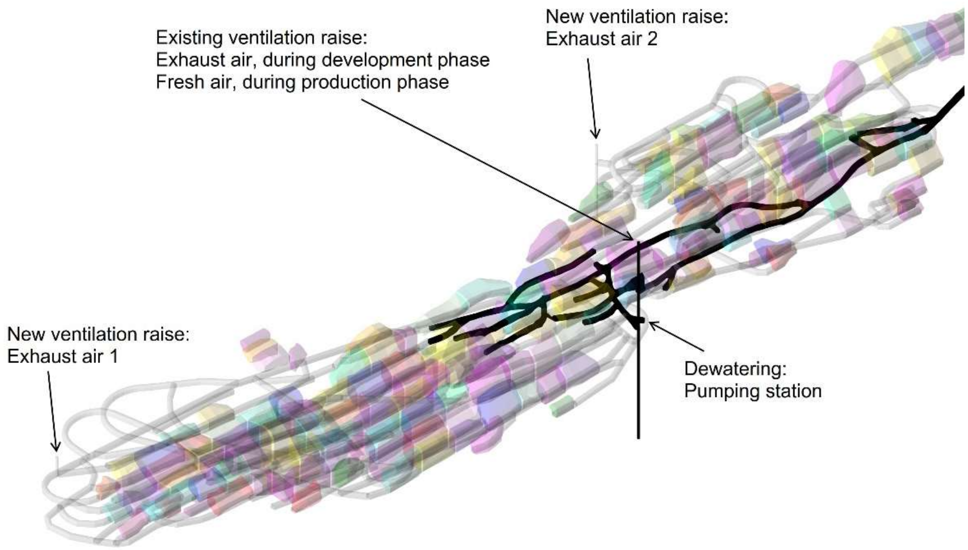


Figure 15-6. Hautalampi ventilation raises and dewatering pumping station.

15.3.4 Decline portal

After removing the fill material placed in the box-cut and portal, the support of the existing decline portal may be decided. It is assumed that some additional shotcrete and rock bolts are required for sufficient reinforcement.

15.3.5 Stope designs

The dimensions of the Hautalampi main orebody are estimated to be approximately 1000 m long, 100-150 m high, and 1-30 m wide. A plan view of the underground mine is shown in Figure 15-2. The stope designs have taken into account the rock mass conditions and the expected effects of the structures. Once stopes have been opened and their performance monitored, the span and length can be optimized to suit local conditions.

The size and the shape of the stopes vary according to the dimensions of the orebody. A cross-sectional view of the Hautalampi main stope configuration is presented in Figure 15-7. The average stope dimensions are roughly 20 m in height, 15 – 20 m in horizontal width and 20 – 40 m in length.

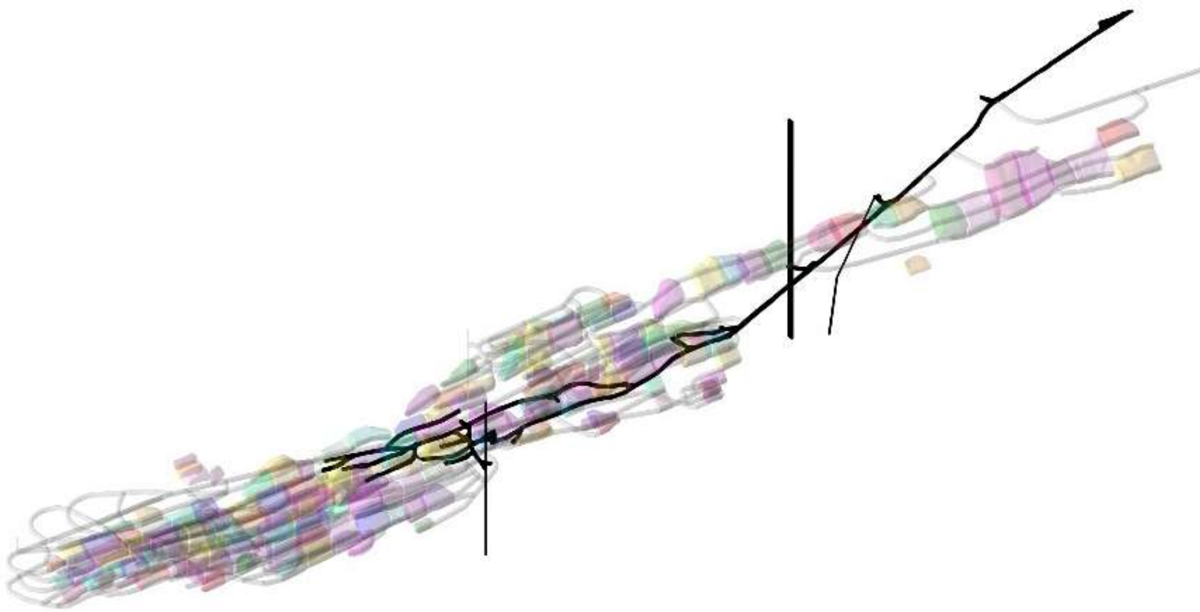


Figure 15-1

Figure 15-7. Cross-sectional view of Hautalampi main stope configuration.

15.4 Mining Methods

The geometry of the orebody and the rock mass properties are suitable for the long hole stoping (LHS) method. Consolidated hydraulic backfilling (including waste rock and mine tailings) will be used to maximize ore extraction and minimize waste rock storage on the surface. Mining will be carried out primarily as bottom-up excavation with primary and secondary stopes. Primary stopes will be mined first and secondary stopes will be excavated once the adjacent primary stopes have been backfilled with consolidated backfill. The secondary stopes will act as rock pillars before excavation and will improve stope stability. The mining sequence is presented in Figure 15-8.

Two different mining methods will be utilized in the Hautalampi underground mine. Longitudinal long hole stoping will be the primary mining method and covers most of the underground mining operation.

Up-hole benching will be the secondary mining method and used in the underground mine whenever there is no access drift above. The size and the shape of the stopes vary according to the dimensions of the orebody.

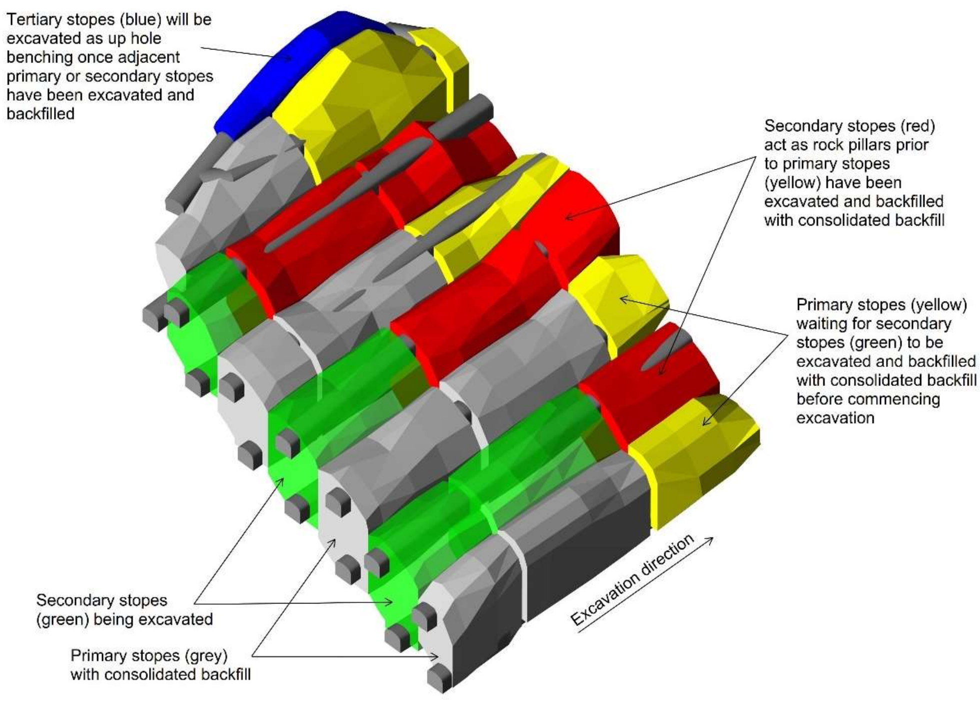


Figure 15-8. Mining sequence of longitudinal long hole stoping and up-hole benching.

15.4.1 Longitudinal long hole stoping

Longitudinal long-hole stoping is the primary mining method. Longitudinal LHS is used due to the relatively narrower orebody, which varies in width from 1 – 30 m.

Longitudinal LHS is a mechanised mining method where the strike of the stope is the same as the strike of the orebody. Ore drives are developed from a footwall drive, which is subparallel to the orebody. The ore drives advance in the orebody to the back of the stope and the mining of the ore is done in reversed order. The ore drive dimensions are 5 m x 5 m, but should be optimized during detailed mine planning to further approach dimensions of 8 m x 5 m if stope dimensions allow for the width increase.

The longitudinal LHS will be mainly mined as 40 m long stopes. Depending on the ore body shape, the stope lengths can vary between 20 – 60 m. If increased stope stability is required during production, narrow rock pillars can be left in between the stopes, perpendicular to the excavation direction. The stopes will be backfilled with waste rock and consolidated hydraulic backfill. The excavation direction of the stopes will commence from the back of the ore drive and progress reversing back towards the start of the ore drive.

The longitudinal LHS mining method requires that a proportion of the ore, from the blasted stope, is mucked using a tele-remote loader. A general conceptual design for longitudinal long-hole bench stoping is presented in Figure 15-9.

A slot raise is required to start an individual stope. The slot raise can be drilled with either a slot raise drill rig or by using the long-hole drilling equipment to create a slot raise drill pattern.

The 89 mm diameter blast holes will be drilled down from the upper ore drive. Smooth blasting will be used to limit the breakage of stope walls.

The stoping efficiency is estimated to be roughly 10 t per drilled meter.

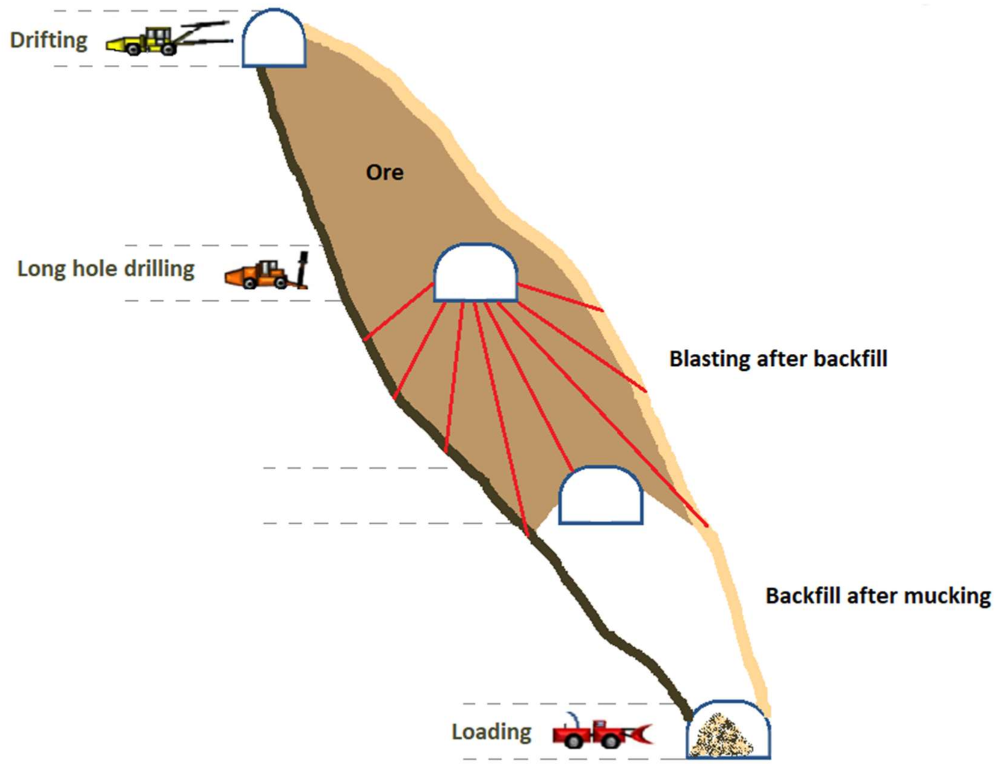


Figure 15-9. General conceptual design for longitudinal long hole bench stoping.

15.4.2Up-hole benching

Up-hole benching will be used when the ore thickness or the dip of the ore is not suitable for long-hole stoping. Up-hole benching will also be necessary when there is no drift access above the ore boundary.

The stopes are developed by driving a drift along the strike of the orebody. This drift is also used for blast hole drilling and mucking. The size of the ore drive is 5 m x 5 m.

The blast holes are drilled upwards with 89 mm diameter holes. The general design for up-hole benching is shown in Figure 15-10.

The length and size of the up-hole bench stopes vary according to the ore dimensions. The stoping efficiency will also be roughly 10 t per drilled meter.

The commencement of the stopes will be done by pre-splitting. Mucking will be done from the footwall drift with a tele-remote loader. Backfilling will be not used for the up-hole benching stopes, which are primarily excavated last in the sequence.

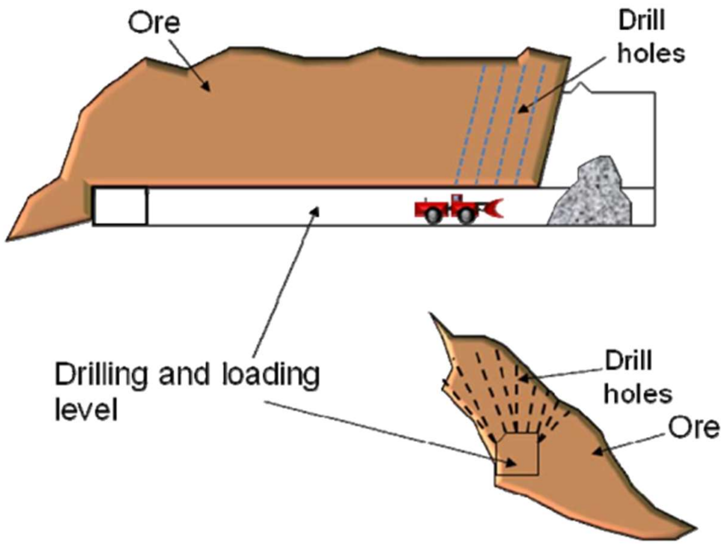


Figure 15-10. General conceptual design for up-hole benching.

15.5 Operating hours

The underground mine operating hours have been calculated for 240 production days per year (Table 15-2). The mining operations will be theoretically ongoing for 17 hours per day, five days a week and working with two crews on 8.5-hour shifts (Table 15-3). It is assumed that 4 weeks of production will be lost per year, due to vacations, bad weather and breakdowns. Additionally, one hour per shift will be lost due to mealtimes, breaks and shift change. The total actual operating hours per day is therefore 15 hours.

Table 15-2. Mining operating schedule for Hautalampi underground mine.

Mining operating schedule	Unit	Quantity
Actual operating hours	hours/day	15
Operating days	days/week	5
	days/year	240
Operating weeks	weeks/year	48
Vacation, weather, downtime, etc.	weeks/year	4
Maximum operating hours	hours/year	3 600

Table 15-3. Shift specifications for Hautalampi underground mine.

Shift specifications	Unit	Quantity
Shifts	shifts/day	2
Shift length	hours	8.5
Meal, breaks, shift change (h/shift)	hours/shift	1
Crews	pcs	2

The two crews will be rotating morning and evening shifts every second week (Table 15-4) from Monday to Friday. Weekends will be free. Weekly working hours will sum up to 42,5 hours for each crew member.

Table 15-4. Crew rotation schedule for Hautalampi underground mine.

Crew rotation			Week 1							Week 2				h/week
Crew 1	M	M	M	M	M	F	F	E	E	E	E	E	F	F	42,5
Crew 2	E	E	E	E	E	F	F	M	M	M	M	M	F	F	42,5
Morning shift	M														
Evening shift	E														
Weekend free	F														

Extra capacity is available by changing into three shift daily rotation by adding a night shift into the cycle. Furthermore, weekends are also available for extra capacity or scheduled maintenance if required.

15.6 Loading and hauling of ore and waste rock

Specifications for loading and hauling of ore and waste rock are presented below. The specifications are based on production quantities and available actual operating hours. The loader specifications (Table 15-5) are based on a 12 t loader capacity and the hauling specifications (Table 15-6) are based on a 30 t haulage truck.

Any brand name used in this report is purely for sizing and specification purposes and is not a recommendation to purchase.

The performance of the loader is summarized in Table 15-5. These are based on standard performance figures. They assume that a truck is always available for loading. These figures are used to estimate fleet numbers. The selected loader is an Epiroc Scooptram ST14. A theoretical cycle time of 75 seconds per pass has been used in the loading specifications.

Two Epiroc Scooptram ST14's will be sufficient during the life of mine, with one in operation and one as a spare.

Table 15-5. Epiroc Scooptram ST14 (12 t) loader with operation specifications.

Loader	**Epiroc Scooptram ST14**	
Measure		
Bucket capacity	6.4	m^3
Theoretical bucket payload	12.0	t
Fill factor	83 %	
Actual bucket payload	10.0	t
Overall job efficiency	90 %	
Mechanical availability	92 %	
Operator efficiency and equipment utilization	87 %	
Number of passes	3	
Minutes per truck load	3.8	min
Maximum trucks per hour	11.5	
Tonnes per hour	345	t/h
Tonnes per day	5 180	t/d
Maximum tonnes per year	1 243 109	t/a

The selected haul truck is a Scania 560 (30 t). Table 15-6 summarises the average hauling performance of the haul truck. The LOM average distance of hauling ore and waste one way is calculated as 2.02 km. Additionally, it is assumed that loaded and empty hauling speeds are averaged at 20 km/h. The actual truck payload is calculated for three passes with the loader.

Two to four Scania 560 will be sufficient during the life of mine, with one to three in operation and one as a spare. A detailed study of the haulage profiles is recommended for future studies.

Table 15-6. Scania 560 (30 t) haul truck with operation specifications.

Haul truck	**Scania 560**	
Measure		
Truck capacity	16	m^3
Actual truck payload	30	t
Truck speed	20	km/h
Haulage distance one way (LOM average)	2.02	km
Cycle		
Load	3.8	min
Haul	6.1	min
Tip	1.5	min
Return	6.1	min
Wait	0	min
Total cycle (LOM average)	17.4	min

The one-way hauling distances, shown in Table 15-7, are yearly averages from the underground loading points to the ROM pad for ore, and from underground loading points to initially estimated stopes for backfill material. For year one, waste rock from tunnel development will be hauled to the surface, adjacent to the decline portal for temporary storage. Once stopes are available for backfilling, waste rock from the temporary surface storage will be hauled back underground for stope backfilling.

Table 15-7. Haul truck yearly average haulage distances and cycle times.

Yearly production	Haulage distance one-way (km)		Cycle time (min)	
	Ore	Waste	Ore	Waste
1	2,35	1,90	19,4	16,7
2	2,55	1,70	20,6	15,5
3	2,75	1,50	21,8	14,3
4	2,95	1,30	23,0	13,1
5	2,75	1,10	21,8	11,9
6	2,55	0,90	20,6	10,7
7	2,35		19,4	
8	2,15		18,2	
9	2,05		17,6	
10	1,95		17,0	
11	1,85		16,4	
12	1,75		15,8	

The use of 30 t multi-axle on-highway trucks (Scania 560) in underground haulage, instead of large rigid mine trucks, is favorable, because of their lower cost, higher availability, and higher tramming speed in the gently sloping incline. Additionally, they are easier found from contractors and have lower maintenance costs etc.

The loading and trucking operation would be flexible with trucks being loaded immediately upon arrival at the stoping level. There would be little need for stockpiles of ore to be generated except to speed up the loading of trucks directly from a small stockpile. These small stockpiles would then be refilled by the loader while the trucks are on their way to and back from the ROM pad on the surface.

The annual ore production for the peak years is roughly 488 000 t. For 240 working days, the daily ore production is 2 033 t. This is 10 167 t per week, for a five-day work week.

15.7 Production drilling

The selected underground production drill rig is Sandvik DL 431. The primary production drill rig is suitable for the drive size, production drill hole diameter and length, and stope drill ring pattern. Sandvik DL 431 is a top hammer long-hole drill rig which will utilise drill tubes to maintain the accuracy of the 89 mm production holes.

The production drill rig specifications have been used to estimate the equipment fleet numbers. Two Sandvik DL 431 production drill rigs will be sufficient during the life of mine, with one in operation and one as a spare.

Table 15-8. Sandvik DL 431 production drill rig with operation specifications.

Production drill rig	**Sandvik DL 431**	
Measure		
Hole size	64 - 89	mm
Bench height	20.0	m
Hole length	23.0	m
Drill rate	23.5	m/h
Single-hole	59	min
Drill move	5	min
Overall job efficiency	90 %	
Mechanical availability	92 %	
Operator efficiency and equipment utilization	87 %	
Holes per hour	0.7	
Holes per day	10.2	
Drill meters per t	0.098	m/t
Drill meters per day per rig	234	m/d/rig
Drill meters per year per rig	56 161	m/a/rig
Tonnes per year per rig	573 118	t/a/rig

15.8 LOM -plan

The mine production schedule for the Hautalampi Project used the created underground mine design. The objectives of the production schedule are to:

- Achieve targeted annual production in terms of quantity and quality.
- Develop a production schedule suitable for estimating PFS mining operating cost estimates.

Scheduling parameters:

Yearly production rate: ca. 470 000 tonnes per annum

Development rate: ca.1200–1300 meters per annum

The production schedule has been developed on monthly basis. Production scheduling was completed using Mine Sched-software. A yearly summary of the LOM plan is presented in

Table 15-9. Metal tonnes in the table below represents the contained metal in the plant feed.

Table 15-9 LOM summary

	1	2	3	4	5	6	7	8	9	10	11	12	Totals
ORE (stopes)	78 316	444 942	352 283	332 439	331 422	379 444	463 980	458 442	463 728	470 289	416 512	82 626	4 274 424
ORE (drifts)	104 746	48 059	112 466	138 864	143 621	109 158	1 882	-	-	-	-	-	658 795
ORE total	183 062	493 001	464 749	471 304	475 043	488 602	465 861	458 442	463 728	470 289	416 512	82 626	4 933 219
Waste (Tonnes)	135 553	193 228	122 347	101 437	97 876	131 801	331	-	-	-	-	-	782 573
Ni (t)	440	1 411	1 482	1 491	1 605	1 621	1 194	1 301	1 180	1 232	836	144	13 936
Cu (t)	380	1 140	1 247	1 095	1 493	1 467	804	817	780	886	818	149	11 076
Co (t)	112	320	354	343	400	420	325	318	338	306	266	54	3 555
S (t)	6 143	12 898	12 626	10 355	11 296	11 805	10 953	11 130	10 112	11 561	11 456	2 692	123 027
Ni (%)	0.24 %	0.29 %	0.32 %	0.32 %	0.34 %	0.33 %	0.26 %	0.28 %	0.25 %	0.26 %	0.20 %	0.17 %	0.28 %
Cu (%)	0.21 %	0.23 %	0.27 %	0.23 %	0.31 %	0.30 %	0.17 %	0.18 %	0.17 %	0.19 %	0.20 %	0.18 %	0.22 %
Co (%)	0.06 %	0.06 %	0.08 %	0.07 %	0.08 %	0.09 %	0.07 %	0.07 %	0.07 %	0.07 %	0.06 %	0.06 %	0.07 %
S (%)	3.36 %	2.62 %	2.72 %	2.20 %	2.38 %	2.42 %	2.35 %	2.43 %	2.18 %	2.46 %	2.75 %	3.26 %	2.49 %
Development (meters)	3 650	3 660	3 650	3 650	3 650	3 660	34	-	-	-	-	-	21 954
In Ore (m)	1 591	730	1 708	2 109	2 182	1 658	29		-	-	-	-	10 007
In Waste (m)	1 524	2 661	1 687	1 490	1 358	1 627	5	-	-	-	-	-	10 351
Decline (included in waste)	535	286	172	51	171	375		-	-	-	-	-	1 589

In addition to the reported reserves, the life of mine plan includes the mining of inferred mineral reserve of Mökkivaara. The modifying factors applied are the same as in measured and indicated mineral resources. The inferred resources comprise a total of 7% of the life of the mine plan.

15.9 Supportive mining equipment

The main mining fleet consists of loaders, haul trucks and production drill rigs, as described in chapters 15.6 and 15.7. In addition to the main mining fleet, a supportive mining fleet, including tunnelling equipment for development, is required. The minimum requirements for the supportive fleet are listed below. The requirements have been adjusted according to ore and waste production rates.

Drill jumbos:
For the first six years of the life of mine, three drill jumbos will be sufficient to drill the decline, access drifts, and ore drives. Two will be required in operation and one as a spare. The selected drill jumbo is an Epiroc Boomer E2, which is a twin-boom diesel-hydraulic jumbo. With powerful motors for long tramming distances, the Epiroc Boomer E2 suits ideally for mine development in Hautalampi underground mine.

Bolting rig:
For the first six years of the life of mine, three bolting rigs will be sufficient to install rock bolts for the decline, and access drifts. Two will be required in operation and one as a spare. The selected bolting rig is an Epiroc Boltec E. Epiroc Boltec E is a fully mechanized rock bolting rig suitable for the installation of a range of bolt types from 2.4 to 6 meters in length.

Cable bolting rig:
For the first six years of the life of mine, two cable bolting rigs will be sufficient to install cable bolts for tunnel intersections, and ore drives. One will be required in operation and one as a spare. The selected cable bolting rig is an Epiroc Cabletec M. The cable bolting rig has a fully mechanized drilling system, a 500 kg dry cement capacity, and approximately 775 m of 15.2 mm diameter cable storage capacity. With a hole diameter range of 51–76 mm and 20 m mechanized drilling hole depth, Epiroc Cabletec M will also serve as a secondary production drill rig during unscheduled maintenance and breakdowns of the primary production drill rigs.

Shotcreting rig:
For the first six years of the life of mine, two shotcreting rigs will be sufficient for the reinforcement of the decline, access drifts, and ore drives. One will be required in operation and one as a spare. The selected shotcreting rig is a Normet Spraymec 8100. The Normet Spraymec 8100 is an electro-hydraulic self-propelled mobile concrete sprayer, which optimizes concrete spraying in drifts. It provides efficient spraying from one set-up in tunnels of up to 10.3 m in height and 16 m in width. The maximum vertical spraying reach is 14 m.

Scaling rig:

For the first six years of the life of mine, two scaling rigs will be sufficient for removing loose blocks from the decline, access drifts, and ore drives. One will be required in operation and one as a spare. The selected scaling rig is a Volvo EC 140 excavator with a hydraulic hammer. With its small to medium-sized dimensions, the Volvo EC 140 is well suited for the drift size of 5 – 8 m x 5 m.

Charging rig:
For the life of mine, two to three charging rigs will be sufficient. One to two will be required in operation and one as a spare. The selected charging rig is a Normet Charmec MF605. The Normet Charmec MF605 charger has a diesel-hydraulic drive, which is designed for production and tunnel face charging in underground mines and tunnels for up to 36m² cross-sections, where the max face height is 6.5 m.

Utility rig:
For the life of mine, one utility rig will be sufficient. The selected utility rig is a Normet Himec MF905. This will function as a supportive machine for various installations in the drifts.

Wheel loader:
For the life of mine, one wheel loader will be sufficient. The selected wheel loader is a CAT 930M. The wheel loader will be utilized for loading of ore at the ROM pad. During the first year of mine development, the wheel loader can be additionally used for re-handling work at the temporary waste rock storage facility.

Graders:
For the life of mine, one grader will be sufficient to ensure coverage for the mining operation. The selected grader is a CAT 14. The CAT 14 will be used for road cleaning and grading, and general drainage and ditching work. In underground mining, the condition of the haul roads is one of the key issues to a successful operation, so good grading is always important.

Personnel vehicles:
For the life of mine, three to six light vehicles will be sufficient. Two to five will be required in operation and one as a spare. The selected light vehicle is a Toyota Hilux twin cab 4x4. Personnel mobility is another key issue to a successful operation, and therefore the above-mentioned light vehicle fleet is included.

15.10 Equipment schedule

Table 15-10 summarises the yearly mining equipment requirements, including spare equipment, for the 12-year duration of the mining operation.

Table 15-10. Yearly mining equipment requirements, including spare equipment.

Function	Model type	1	2	3	4	5	6	7	8	9	10	11	12
LHDs	Epiroc Scooptram ST14	2	2	2	2	2	2	2	2	2	2	2	2
Mine trucks	Scania 560	3	4	4	4	4	4	3	3	3	3	3	2
Production drill rigs	Sandvik DL 431	2	2	2	2	2	2	2	2	2	2	2	2
Drill jumbos	Epiroc Boomer E2	3	3	3	3	3	3	-	-	-	-	-	-
Bolting rigs	Epiroc Boltec E	3	3	3	3	3	3	-	-	-	-	-	-
Cable bolting rigs	Epiroc Cabletec M	2	2	2	2	2	2	-	-	-	-	-	-
Shotcreting rigs	Normet Spraymec 8100	2	2	2	2	2	2	-	-	-	-	-	-
Scaling rigs	Volvo EC 140	2	2	2	2	2	2	-	-	-	-	-	-
Charging rigs	Normet Charmec MF605	3	3	3	3	3	3	2	2	2	2	2	2
Utility rigs	Normet Himec MF905	1	1	1	1	1	1	1	1	1	1	1	1
Wheel loaders	Caterpillar 930M	1	1	1	1	1	1	1	1	1	1	1	1
Graders	Caterpillar 14	1	1	1	1	1	1	1	1	1	1	1	1
Light vehicles	Toyota Hilux twin 4x4	5	6	6	6	6	6	3	3	3	3	3	3

Table 15-11 summarises the yearly mining equipment requirements in operation, for the 12-year duration of the mining operation. The mining equipment quantity, in operation, is used for operational cost estimates.

Table 15-11. Yearly mining equipment requirements in operation.

Function	Model type	1	2	3	4	5	6	7	8	9	10	11	12
LHDs	Epiroc Scooptram ST14	1	1	1	1	1	1	1	1	1	1	1	1
Mine trucks	Scania 560	2	3	3	3	3	3	2	2	2	2	2	1
Production drill rigs	Sandvik DL 431	1	1	1	1	1	1	1	1	1	1	1	1
Drill jumbos	Epiroc Boomer E2	2	2	2	2	2	2	-	-	-	-	-	-
Bolting rigs	Epiroc Boltec E	2	2	2	2	2	2	-	-	-	-	-	-
Cable bolting rigs	Epiroc Cabletec M	1	1	1	1	1	1	-	-	-	-	-	-
Shotcreting rigs	Normet Spraymec 8100	1	1	1	1	1	1	-	-	-	-	-	-
Scaling rigs	Volvo EC 140	1	1	1	1	1	1	-	-	-	-	-	-
Charging rigs	Normet Charmec MF605	2	2	2	2	2	2	1	1	1	1	1	1
Utility rigs	Normet Himec MF905	1	1	1	1	1	1	1	1	1	1	1	1
Wheel loaders	Caterpillar 930M	1	1	1	1	1	1	1	1	1	1	1	1
Graders	Caterpillar 14	1	1	1	1	1	1	1	1	1	1	1	1
Light vehicles	Toyota Hilux twin 4x4	4	5	5	5	5	5	2	2	2	2	2	2

15.11 Mine services

15.11.1 Ventilation air requirement

The required fresh air ventilation is calculated based on diesel emission dilution and removal. The theoretical maximum amount of simultaneously used diesel equipment in the underground mine is presented in Table 15-12. The table shows the motor effect in kW for each individual piece of equipment and the total motor effect for each equipment type, with an estimated motor utilization factor.

Table 15-12. Operational underground mining equipment with motor effects.

Function	Model type	Motor effect (kW)	Motor utilization (%)
LHDs	Epiroc Scooptram ST14	250	75%
Mine trucks	Scania 560	324	75%
Production drill rigs	Sandvik DL 431	110	50%
Drill jumbos	Epiroc Boomer E2	120	50%
Bolting rigs	Epiroc Boltec E	120	50%
Cable bolting rigs	Epiroc Cabletec M	120	50%
Shotcreting rigs	Normet Spraymec 8100	75	50%
Scaling rigs	Volvo EC 140	90	50%
Charging rigs	Normet Charmec MF605	120	50%
Utility rigs	Normet Himec MF905	120	50%
Wheel loaders	Caterpillar 930M	187	50%
Graders	Caterpillar 14	178	50%
Light vehicles	Toyota Hilux twin 4x4	80	25%

Based on the total motor effects of the diesel-powered mining equipment, theoretically simultaneously in operation (Table 15-13), the amount of fresh air required (Table 15-14) can be calculated using the widely approved dilution factor of 0,06 m^3 / s per kW of motor effect (Brake and Nixon 2008).

Table 15-13. Total motor effect (kW) of yearly mining equipment in operation.

Function	1	2	3	4	5	6	7	8	9	10	11	12
LHDs	188	188	188	188	188	188	188	188	188	188	188	188
Mine trucks	486	729	729	729	729	729	486	486	486	486	486	243
Production drill rigs	55	55	55	55	55	55	55	55	55	55	55	55
Drill jumbos	120	120	120	120	120	120	-	-	-	-	-	-
Bolting rigs	120	120	120	120	120	120	-	-	-	-	-	-
Cable bolting rigs	60	60	60	60	60	60	-	-	-	-	-	-
Shotcreting rigs	38	38	38	38	38	38	-	-	-	-	-	-
Scaling rigs	45	45	45	45	45	45	-	-	-	-	-	-
Charging rigs	120	120	120	120	120	120	60	60	60	60	60	60
Utility rigs	60	60	60	60	60	60	60	60	60	60	60	60
Wheel loaders	94	94	94	94	94	94	94	94	94	94	94	94
Graders	89	89	89	89	89	89	89	89	89	89	89	89
Light vehicles	80	100	100	100	100	100	40	40	40	40	40	40
Total (kW)	**1554**	**1817**	**1817**	**1817**	**1817**	**1817**	**1071**	**1071**	**1071**	**1071**	**1071**	**828**

Table 15-14. Total fresh air (m^3/s) required to dilute diesel emissions of yearly mining equipment in operation. The dilution factor of 0,06 m^3 / s per kW.

Function	1	2	3	4	5	6	7	8	9	10	11	12
LHDs	11	11	11	11	11	11	11	11	11	11	11	11
Mine trucks	29	44	44	44	44	44	29	29	29	29	29	15
Production drill rigs	3	3	3	3	3	3	3	3	3	3	3	3
Drill jumbos	7	7	7	7	7	7	-	-	-	-	-	-
Bolting rigs	7	7	7	7	7	7	-	-	-	-	-	-
Cable bolting rigs	4	4	4	4	4	4	-	-	-	-	-	-
Shotcreting rigs	2	2	2	2	2	2	-	-	-	-	-	-
Scaling rigs	3	3	3	3	3	3	-	-	-	-	-	-
Charging rigs	7	7	7	7	7	7	4	4	4	4	4	4
Utility rigs	4	4	4	4	4	4	4	4	4	4	4	4
Wheel loaders	6	6	6	6	6	6	6	6	6	6	6	6
Graders	5	5	5	5	5	5	5	5	5	5	5	5
Light vehicles	5	6	6	6	6	6	2	2	2	2	2	2
Total (m^3/s)	**93**	**109**	**109**	**109**	**109**	**109**	**64**	**64**	**64**	**64**	**64**	**50**

The total fresh air ventilation requirement fluctuates between 50 – 109 m^3/s during the life of mine. With the maximum total amount of 109 m^3/s of fresh air required to dilute the diesel emissions, the corresponding air velocity in the fresh air ventilation raise will be around 19.1 m/s as shown in Table 15-16. This air velocity is below the maximum recommended value of 20 m/s (SME Mining Engineering Handbook 2011). Recommended maximum air velocities for different areas are presented in Table 15-15.

Table 15-15. Recommended maximum air velocities m/s for different areas in underground mining (SME Mining Engineering Handbook 2011).

Area	Air velocity (m/s)
Working faces (Ore drives)	4
Main haulage routes (Decline)	6
Ventilation shafts	20

Table 15-16. Maximum air velocities m/s for Hautalampi underground mine cross sections, theoretically assuming all of the maximum airflow (109 m^3/s) is directed to one area.

Area	Cross section (m^2)	Air velocity for 109 m^3 / s airflow (m/s)
Working faces (Ore drives, 8 x 5 m)	37,4	2,9
Main haulage routes (Decline, 5 x 5 m)	23,4	4,7
Ventilation shafts (Fresh air, r = 1,35 m)	5,72	19,1

Even with the theoretical maximum airflow of 109 m^3/s directed into one area, all resulting air velocities are below the maximum recommendations. Furthermore, it should be noted that all equipment, simultaneously in operation, will be working at multiple different locations and therefor fresh air will be routed to multiple areas. This will decrease the amount of airflow per area and therefor also decrease the air velocity. With these air velocities, the underground mine will not likely face draft, draught and dust problems.

These yearly airflow requirements are comparable to what is commonly used in Finnish underground mines.

15.11.2 Ventilation power requirement

To account for the maximum yearly fresh air ventilation requirement, the following ventilation fan specifications are required. These ventilation fan specifications are presented in Table 15-17.

Table 15-17. Ventilation fan specifications.

Ventilation fan specifications	Maximum airflow capacity @ maximum power (m^3/s)	Maximum power capacity (kW)
Fresh air ventilation	120	500
Exhaust ventilation	70	200
Local booster ventilation	20	50

The estimated yearly ventilation fan quantities, corresponding utilized power requirements and utilized energy consumptions, and maximum power capacities are presented in Table 15-18. The yearly energy consumption is calculated for 4 560 hours per year, which is equivalent to 19 hours per day for the 240 operating days a year.

Table 15-18. Yearly ventilation fan quantities, corresponding utilized power requirements and utilized energy consumptions, and maximum power capacities.

Function	1	2	3	4	5	6	7	8	9	10	11	12
Fresh air ventilation (pcs)	1	1	1	1	1	1	1	1	1	1	1	1
Exhaust ventilation (pcs)	1	2	2	2	2	2	1	1	1	1	1	1
Local booster ventilation (pcs)	8	10	10	10	10	10	6	6	6	6	6	4
Maximum power capacity (kW)	1100	1400	1400	1400	1400	1400	1000	1000	1000	1000	1000	900
Utilized power requirement (kW)	988	1266	1266	1266	1266	1266	750	750	750	750	750	551
Utilized energy consumption (MWh)	4503	5771	5771	5771	5771	5771	3418	3418	3418	3418	3418	2513

15.11.3 Ventilation principle

The primary fresh air ventilation will be provided through ducts from the decline portal during the decline rehabilitation phase and during the ramp-up of tunnel development in the first year. The existing ventilation raise and the decline will serve as exhaust ventilation routes during the development. Once the existing shaft can be accessed, rock conditions are approved safe, and production starts, downcast fresh air ventilation will be transferred to the existing 2.7 m diameter shaft. New ventilation raises, exhaust air 1 and exhaust air 2 will be constructed in the numbered sequence during development. The main ventilation principle is presented in Figure 15-6.

The main fresh air ventilation fan will be mounted to the shaft collar and air blown down the shaft with a maximum capacity of 120 m^3/s. In an emergency, the fan direction can be reversed to exhaust air up the shaft. The fan will also incorporate a heat exchanger to warm the air during the winter. The main exhaust ventilation fans will be mounted to the shaft collar and exhaust air sucked with a maximum capacity of 70 m^3/s each. Secondary ventilation will consist of booster fans and flexible ventilation ducts in each active heading. The maximum capacity of the booster fans is 20 m^3/s each.

On the surface, a heating plant will be required to heat the freezing fresh air during winter time. Additionally, it is recommended to use silencers for the ventilation fans in order to minimize noise pollution.

15.11.4 Ventilation air heating

To avoid freezing of the fresh air raise, heating of the fresh air is needed during the winter months. Heat can be bought from an external energy company and equipment for the oil burner can be leased. According to experience from similar mine operations, the heating capacity for 100-120 m^3/s of air is roughly 3 MW (for heavy heating oil). The oil burner heats water, which is pumped through heat exchangers. Then, the air warms up by flowing through the heat exchangers and into the fresh air raise. For heat blending, an external building is required.

The possibility of using lost heat from mine site operations, for example from processing, should be further investigated.

The energy consumption for ventilation air heating is calculated for 2 280 hours at full capacity. This is half of the 4 560 hours used for ventilation per year. It is assumed that the 2 280 hours per year account for the cold winter months when heating is required. The maximum power requirement for ventilation air heating is 6 840 MWh/a during production years 2-6.

15.11.5 Exhaust raise construction

Investigations indicate that the topsoil thickness is around 27 m at the new exhaust air 1 raise and around 22 m at the new exhaust air 2 raise. Due to the thickness of the topsoil, both exhaust raises will need a concrete casted collar for the topsoil part. The soil quality and quantities should be further studied at these designed locations. Furthermore, diamond drill holes should be drilled and drill core logging done to evaluate the rock quality in detail. The length of exhaust air raise 1 is hard rock is approximately 20 m and the length of exhaust air raise 2 in hard rock is approximately 60 m. For the exhaust airflow capacity and velocity requirements, 2-meter diameter exhaust air raises are sufficient.

The earthworks consist of:

- road connection to the site on the surface and excavation of overburden to the depth of 6 metres.
- lowering the ground water table at the site by pumping from wells
- construction of a cast mould
- caisson installation (shaft collar) by digging the loose earth and by sinking the concrete casting mould step by step
- after the collar is finished, make the raises with the Alimak method

15.11.6 Pumping

The pumping requirements, based on excavation volume, have been calculated based on a 475 000 m^3 per year ground water seepage and industrial waters flowrate into the underground mine. Additionally, it is estimated that around 25 000 m^3 of water from stope backfilling needs to be pumped to the surface. The estimated yearly pumping quantities are shown in Table 15-19. This is a rough estimate and should be followed up and updated during development and production. Run-off water from the portal opening will be diverted out of the underground mine.

Table 15-19. Estimated yearly pumping quantities for Hautalampi underground mine.

Estimated yearly pumping quantities	Quantity	Unit
Ground water seepage and industrial waters	475 000	m^3/a
Water from stope backfilling	25 000	m^3/a
Total	500 000	m^3/a
	1 370	m^3/d
	57	m^3/h
	0.016	m^3/s
	16	l/s

One pumping station will be required for the underground mine (Figure 15-6). The pumping station will be equipped with two main pumps, with a 55kW maximum power capacity for each. The utilized power requirement to pump the estimated water quantities is 38 kW, with a vertical head of 150 meters, a pumping efficiency of 68% and a pipe friction loss of 10%. The additional power capacity of the pumps is justified to account for unexpected inflows to the mine. Only one pump is required for dewatering the estimated inflow whereas the second pump will be used as a backup, during maintenance, and when necessary.

Furthermore, a maximum of 10 submersible pumps will be allocated within the underground mine and will follow the progress of development and mine production. The underground mine pumping power requirements are shown in Table 15-20.

Table 15-20. Utilized pumping power requirements for Hautalampi underground mine.

Pumping	Quantity (pcs)	Maximum power capacity per unit (kW)	Utilized pumping power requirement per unit (kW)	Utilized pumping power requirement total (kW)
Main Pumps *	2	55	38	38
Submersible pumps	10	10	6.2	62
Total				100 kW

* one pump on standby

Water will be pumped up to the surface in a pipeline along the main decline during the development phase. During production, water from the pump pit can be pumped up to the surface in a pipeline which is installed into the existing fresh air ventilation raise. On the surface, water will be routed into clarification basins for further treatment.

Pumps will be operated 24 hours a day 365 days a year, which is a total of 8760 hours per year. The yearly utilized energy consumption for pumping is estimated to be a maximum of 876 MWh.

15.11.7 Mine backfill

Consolidated hydraulic backfilling (waste rock 20% and mine tailings 80% of the volume) will be used to maximize ore extraction and minimize waste rock storage on the surface.

Primary stopes will be mined first and secondary stopes will be excavated once the adjacent primary stopes have been backfilled with consolidated backfill. Once secondary stopes have been mined, they will also be backfilled with consolidated backfill. The mining sequence is presented in Figure 15-8.

Tailings streams from sulphur flotation are directed to either mine backfill or the tailings area. High- and low-sulphur streams from sulphur flotation are handled mainly separately in the tailings handling section.

Ore drives at the bottom of the stopes will be blocked with waste rock barricades. Waste rock backfill material will be hauled with trucks and dumped into the stopes. The tailings backfill material will be pumped through pipes and directed into stopes. Ore drives at the top of primary stopes will then be blocked with waste rock barricades and filled up with consolidated hydraulic backfill.

The tailing's pumpability is recommended to be further assessed in the next design phase. Waste rock haulage for backfilling is included in mining opex. Pumping, pipelines and backfill additives are included under mining infrastructure.

15.11.8 Underground power requirements

The underground mining equipment power requirements are presented in Table 15-21. The total utilized power requirement for mining equipment, ventilation and dewatering is estimated to be around 1.65 MW. A large part of the power requirement is due to ventilation requirements.

Depending on the used driving force for the mining equipment, diesel-powered or electrical, the electricity power requirements change. If the mining equipment is diesel-powered, less electricity is needed for the equipment. However, more electricity is then required for ventilation purposes to dilute the diesel emissions. Alternatively, if the mining equipment is electrically powered, less electricity is required for ventilation purposes, due to little or no diesel emissions. A trade-off study is recommended for the next design phase.

Table 15-21. Maximum power capacities and utilized power requirements for underground equipment.

Power requirements	Quantity (pcs)	Maximum power capacity total (kW)	Utilized power requirement total (kW)
Loader (primary)	1	-	-
Truck	3	-	-
Subtotal			-
Production drill rig	1	80	72
Drill jumbo	2	200	180
Bolting rig	2	120	108
Cable bolting rig	1	80	72
Shotcreting rig	1	75	68
Scaling rig	2	-	-
Charging rig compressor	2	40	36
Utility rig	1	-	-
Wheel loader	1	-	-
Grader	1	-	-
Personnel vehicle	6	-	-
Subtotal		595 kW	536 kW
Fresh air ventilation	1	500	455
Exhaust ventilation	2	400	311
Local ventilation	10	500	500
Subtotal		1400 kW	1266 kW
Main Pumps *	2	55	38
Submersible pumps	10	100	62
Subtotal		155 kW	100 kW
Total		2150 kW	1902 kW

* 1 in use at a time

15.11.9 Communication

A fixed telephone system will be installed at fixed points in the mine. A leaky feeder cable two-way telephone system will be installed to cover the decline and production areas.

The fixed telephones will be relocated as production progresses in underground stopes. The fixed telephones will also be installed in areas of significant importance, e.g. pump stations and rescue chambers.

15.12 Personnel

The minimum requirements of personnel for mine site activities and support functions are listed in Table 15-22. The labour costs are derived from benchmarking similar operations in Finland.

Table 15-22. Mine staff requirements and costs.

Mine staff requirements	**(Persons)**	Salary €/year	Increase for experienced personnel 35 %	Cost €/person	Total Cost €/year
Not included in mining OPEX:					
Mine superintendent	1	96 000	33 600	129 600	129 600
Mine Planning Engineer	1	72 000	25 200	97 200	97 200
Shift foreman / Con. supervisor	2	60 000	21 000	81 000	162 000
Maintenance crew	2	42 000	14 700	56 700	113 400
Senior geologist	1	84 000	29 400	113 400	113 400
Shift geologist	1	60 000	21 000	81 000	81 000
Senior surveyor	1	72 000	25 200	97 200	97 200
Total	**9**				**793 800**

16 Process Plant

The process plant consists of a comminution circuit (crushing and grinding), copper flotation concentrate production, nickel/cobalt flotation concentrate production, sulphur removal flotation and tailings handling. The plant is designed for 500 ktpa nominal ore throughput. Design parameters are largely based on pilot test work conducted by GTK and experience from former Keretti operation with similar ore. The process plant overview is presented in Figure 16-1.

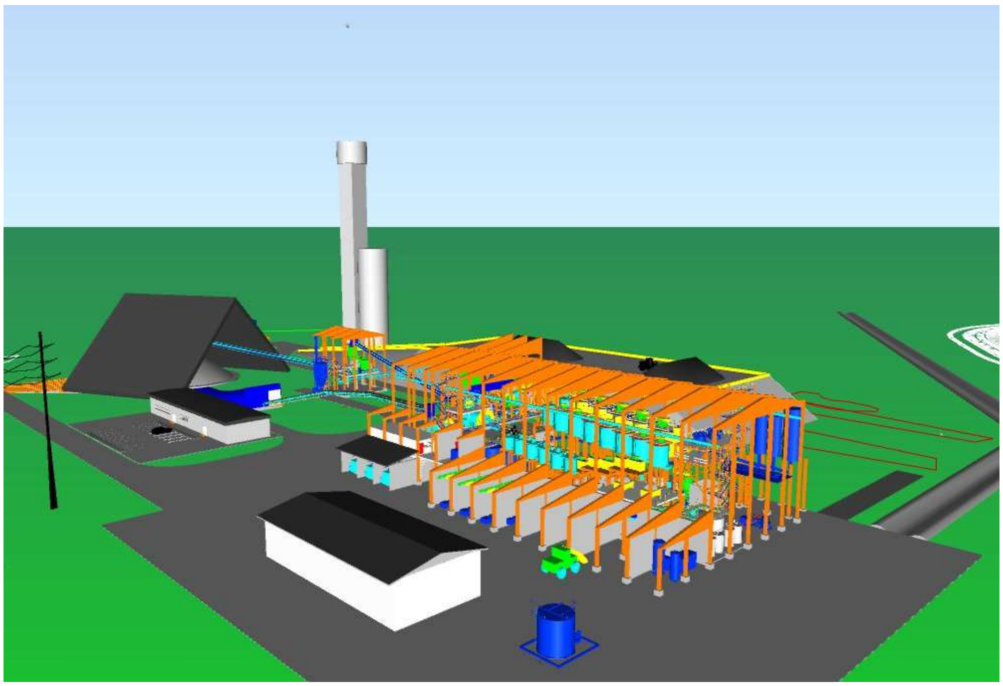


Figure 16-1 Process plant overview

16.1 Process Summary

Concentrator plant availability has been assumed to be 92%, which is equal to an annual operation of 8059 h/a. 500 000 tpa feed rate to plant with 8059 h/a operation equals to 62 t/h nominal plant feed rate. The crushing circuit has been assumed to be operating 5 days per week in two shifts (16 h/d), which is equal to an annual operation of 4171 h/a. 500 000 tpa feed rate to the crushing with 4171 h/a operation equals to 120 t/h nominal feed rate to the crushing. Nominal main operational parameters are summarized in Table 16-1.

Table 16-1. Nominal operational design parameters.

	Throughput (t/a)	Operation hours (h/a)	Feed rate (t/h)
Crushing Plant	500 000	4171	120
Process Plant	500 000	8059	62

Process areas for the concentrator plant are as follows:

- Crushing (100)
- Grinding (200)
- Copper Flotation and Dewatering (300)
- Nickel/Cobalt Flotation and Dewatering (400)
- Tailings Handling (500)
- Utilities (600)
- Sulphur Removal (700)

The flowsheet for the concentrator plant is presented in Figure 16-2.

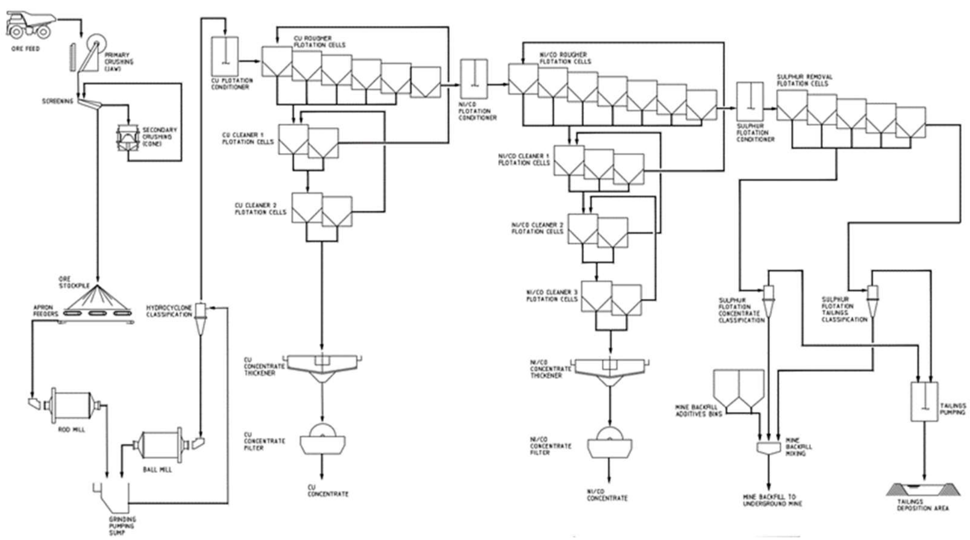


Figure 16-2. Concentrator plant flowsheet.

16.2 Mass Balance

Mass balance for the concentrator plant has been developed in the PFS project phase. The mass balance is based on 500 kt annual feed, feed grades defined in the mining plan and pilot-scale mass balance from the GTK test work. The mass balance is presented in Appendix 3 (101016697-02003).

16.3 Concentrate Specifications

Concentrate specifications for the copper and nickel/cobalt concentrates have been estimated according to the pilot plant test work conducted by GTK. Concentrate specifications are based on average values from two pilot runs (30.1.2019 and 31.1.2019). Specifications are presented in

.

Table 16-2. Estimated concentrate specifications.

	Cu conc.	Ni/Co conc.
Cu (%)	26.50	0.76
Ni (%)	0.50	7.00
Co (%)	0.15	1.93
Cu-Rec to conc. (%)	86.5	10.5
Ni-Rec to conc. (%)	1.4	82.0
Co-Rec to conc. (%)	1.5	82.0
Fe (%)	28.06	37.19
S (%)	29.75	26.00
MgO (%)	3.48	5.39
SiO2 (%)	7.21	11.02
Au (g/t)	5.68	0.77
Ag (g/t)	42.00	5.80
Zn (%)	0.34	0.49
Pb (%)	0.001	0.001
As (%)	0.030	0.057
Bi (%)	0.011	0.01
Sb (%)	0.004	0.003
Te (%)	0.001	0.001

16.4 Process Description

16.4.1 Crushing

The crushing section consists of a two-stage crushing circuit, ore stockpile and ore feed system to the grinding section. Crushing operations are placed indoors for noise and dust prevention purposes.

The crushing area includes a run-of-mine pad (ROM pad) before the crushing circuit. ROM pad has a reserved area to store at least 3000 m³ (approximately 5000 tonnes) of ore. Ore is fed from the ROM pad to the reciprocating plate feeder, which feeds a 132 kW primary jaw crusher. The jaw crusher feed opening width size is 1200 mm. The maximum lump size in the ore has been assumed to be 500 mm. A hydraulic hammer can be used to crush oversized boulders in the ore feed.

Secondary crushing is done by a cone crusher. The jaw crusher product is first screened to remove bypass fine material from the secondary crusher. Oversize from the screen is crushed in a 315 kW secondary cone crusher. Screen undersize and secondary crusher products are fed to the ore stockpile. Transportation of material in the crushing circuit is done by four belt conveyors. Ore screening is located in a separate screening station building. Both the crushing station and screening station buildings have separate dust filtration systems.

A crushing plant has a design capacity to crush 180 t/h, which is well above the required rate of 120 t/h and gives flexibility to the operation scheme. Target P80 for the crushing circuit product (mill feed) is 13.7 mm, which is suitable as the rod mill feed material.

The ore stockpile is designed to have a sufficient buffer capacity for the plant to continue operating through the weekend without crushing plant in operation. 3500 tonnes stockpile is the minimum buffer capacity to keep the plant running for 56 hours with a 62 t/h feed rate to the plant. Ore is taken out from the stockpile with three apron feeders placed in the tunnel below the stockpile. Stockpile is covered to prevent dusting and freezing issues. Ore is fed by the apron feeders to the belt conveyor which feeds ore to the primary grinding mill.

16.4.2 Grinding

The grinding section consists of primary rod mill grinding in an open circuit and secondary ball mill grinding in a closed circuit. The target particle size (P80) to the copper flotation is assumed to be 68 µm.

Ore is fed to the primary rod mill from the ore stockpile by a belt conveyor. The primary rod mill installed power is 400 kW and the main dimensions are 2.7 x 3.6 m. Rod mill discharge is pumped to the classification hydro cyclone together with ball mill discharge. Hydrocyclone underflow is fed to the ball mill. Ball mill power is 1350 kW and the main dimensions are 3.6 x 6 m. Hydrocyclone overflow is fed to the copper flotation conditioning tank. 150 % circulation load has been assumed for the ball mill circuit. Grinding mill power requirements have been assumed based on operational values from former Keretti operations with similar type ore properties.

Grinding rod and ball consumptions per feed tonne have been assumed based on experience from former Keretti operations. Rod consumption is estimated to be 0.5 kg/t and ball consumption is 1.5 kg/t.

16.4.3 Copper Flotation and Dewatering

Copper flotation consists of rougher/scavenger flotation and two cleaning flotation stages. Dewatering of copper concentrate consists of thickening and filtration with a disc filter.

Flotation feed from the grinding circuit is first conditioned in a 40 m³ conditioning tank. pH adjustment is done with slaked lime to increase pH to ~10 for optimal copper flotation conditions. There are 6x10 m³ cells in the rougher/scavenger stage. Rougher/scavenger concentrate is fed to the first cleaning stage which consists of 2x1,5 m³ cells. The second cleaning stage is similar to the first stage, consisting of 2x1,5 m³ cells. Tailings from the first cleaning stage are returned to the rougher flotation. Concentrate from the second cleaning stage is pumped into the dewatering process. Copper flotation tailings are fed to the nickel/cobalt flotation. Retention times in the copper flotation have been estimated based on pilot operation parameters and common industry practices.

Copper concentrate is thickened in a 4 m thickener. The thickened concentrate is filtered by a disc filter to 12 % moisture content and transported to the concentrate storage by a belt conveyor.

16.4.4 Nickel/Cobalt Flotation and Dewatering

Nickel/cobalt flotation consists of rougher/scavenger flotation and three cleaning flotation stages. Dewatering of nickel/cobalt concentrate consists of thickening and filtration with a disc filter.

Flotation feed from the grinding circuit is first conditioned in a 50 m^3 conditioning tank. pH adjustment for nickel/cobalt flotation is done with slaked lime, if necessary. There are 7x30 m^3 cells in the rougher/scavenger stage. Rougher/scavenger concentrate is fed to the first cleaning stage which consists of 3x3 m^3 cells. The second cleaning stage is similar to the first stage, consisting of 2x3 m3 cells. The third cleaning stage consists of 2x1,5 m^3 cells. Tailings from the first cleaning and second cleaning stages are returned to the previous flotation stages. Concentrate from the third cleaning stage is pumped into the dewatering process. Nickel/cobalt flotation tailings are fed to the sulphur removal flotation. Retention times in the nickel/cobalt flotation have been estimated based on pilot operation parameters and common industry practices.

Nickel/cobalt concentrate is thickened in a 4,5 m thickener. The thickened concentrate is filtered by a disc filter to 12 % moisture content and transported to the concentrate storage by a belt conveyor.

16.4.5 Sulphur Flotation

Remaining sulphur in the Ni/Co flotation tailings in reduced using an additional sulphur flotation stage. The sulphur flotation stage consists of one rougher flotation stage. Cell configuration consists of 5x50 m^3 cells. The sulphur flotation concentrate is fed to the mine backfill feed hopper. In the case mine backfill is not in operation, the sulphur concentrate can be fed to the tailings thickener. Sulphur flotation tailings are fed to the mine backfill classification cyclone or directly to the tailings tank if backfilling is not in operation. Retention times in sulphur flotation have been estimated based on laboratory tests and common industry practices.

pH is dropped in the sulphur flotation to a pH level of ~8-8,5 with sulphuric acid to improve sulphur removal flotation conditions.

16.4.6 Tailings Handling and Mine Backfill

Tailings streams from the sulphur flotation are directed to either mine backfill or tailings area. High and low sulphur streams from the sulphur flotation are handled mainly separately in the tailings handling section. Tailings are pumped to the tailings area as slurry. Traditional hydraulic backfill is used as the backfill method.

The majority of the high-sulphur flotation concentrate is fed by default to the mine backfill. The remaining amount of required backfill material is taken from the low-sulphur flotation tailings. Both high-, and low-sulphur fractions are first classified in separate hydro cyclones to remove fines, which are harmful to the backfill quality. Coarse fractions are fed to the mine backfill hopper, fine fractions are fed to the tailings pump tank. Bypass options are available in the tailings handling circuit in the case mine backfill is not temporarily in use.

Cement/slag and slaked lime are used in the mine backfill as additives. E.g. blast furnace slag or fly ash can be considered to be used as an additive slag. Additives are mixed with the tailings slurry in the backfill hopper. Backfill slurry is pumped from the backfill hopper to the underground mine with a centrifugal pump. Sulphur flotation concentrate can be stored temporarily in separate high sulphide tailings temporary deposit in the tailings area in the case mine backfill is not in use.

Tailings which are not pumped to the underground mine or to the high sulphide tailings temporary deposit are pumped to the tailings deposition area. Two centrifugal pumps are used for tailings pumping, one in operation and one in stand-by mode. Estimated annual tailings tonnages and sulphur grades are presented in Table 16-3.

Table 16-3. Annual tailings tonnages and sulphur grades.

Stream	Tonnage (ktpa)	S (%)
Sulphur Flotation Feed	481	1.5
Sulphur Flotation Concentrate	31	12.9
Sulphur Flotation Tailings	451	0.7
Tailings to Mine Backfill	194	2.3
Mine Backfill (total with additives)	216	2.0
Total Tailings to Deposition Area	288	0.9

16.5 Utilities

Concentrator Plant uses both recirculation water from the tailings area and raw water. Raw water is taken in either from lake Suu-Särki or from the underground mine. Raw water has been assumed to be used as chemical make-up water, as pump gland water and as filter wash water (if required). There is a fire water network in the plant. Safety showers are installed in the plant.

The concentrator plant has process air and instrument air networks in use. The flotation process uses a separate flotation air blower.

16.6 Reagents

16.6.1 Copper and Nickel/Cobalt Flotation

Chemicals used in the copper and nickel/cobalt flotation are the following:

- Aerophine (collector). Consumption estimate based on GTK pilot test work.
- CMC (depressant). Consumption estimate based on GTK pilot test work.
- Copper Sulphate (activator). Consumption estimate based on GTK pilot test work.
- Flocculant (thickening). Consumption-based on reference operations.
- MBS (depressant). Consumption estimate based on GTK pilot test work.
- MIBC (frother) Consumption estimate based on GTK pilot test work.
- SIPX (collector). Consumption estimate based on GTK pilot test work.
- Slaked lime (pH control). Consumption estimate based on GTK bench-scale tests and reference operations.

16.6.2 Sulphur Flotation

- Sulphuric Acid 93 % (pH control). Consumption estimate-based reference operations.
- MIBC (frother). Consumption estimate based on GTK pilot test work.
- SIPX (collector) Consumption estimate based on GTK pilot test work.

16.6.3 Mine Backfill Additives

- Slag (binder). Consumption estimate based on reference operations. For example blast furnace slag and/or fly ash can be used as a binder. Regular cement can be used alternatively depending on the availability of slags.
- Slaked lime: Consumption estimate based on reference operations.

16.6.4 Chemical Consumption Summary

The chemical consumption estimate is presented in the table Table 16-4.

Table 16-4. Reagent consumption estimate.

Area	Chemical	Unit Consumption	Consumption (tpa)
Cu/NiCo Flotation	Aerophine	19 g/t ore	9
	CMC	317 g/t ore	158
	CuSO4	124 g/t ore	62
	Flocculant	25 g/t concentrates	0.5
	MBS	411 g/t ore	296
	MIBX	82 g/t ore	41
	SIPX	382 g/t ore	191
	Slaked Lime	1300 g/t ore	650
Sulphur Flotation	Sulphuric Acid (93 %)	1000 g/t ore	500
	MIBC	35 g/t ore	18

	SIPX	35 g/t ore	18
Mine Backfill Additives	Binder Slag	10 % of tailings tonnage to mine backfill	19 400
	Slaked Lime	1 % of tailings tonnage to mine backfill	1 942

17 Tailings and waste rock management

17.1 Tailings characteristics

17.1.1 Summary of Hautalampi Process Tailings Geochemistry

The initial geochemical assessment of the Hautalampi tailings has been undertaken by the Geological Survey of Finland (GTK 2021). The tailings were produced from processing trials undertaken by GTK Mintec in May-June 2021. The following is a summary of the GTK geochemical assessment.

The tested tailings sample contained 0.3%w/w total sulphur, with the predominant sulphur minerals being pyrrhotite and residual oxidised sulphide minerals from the ore, namely chalcopyrite, galena, sphalerite and pentlandite, plus some pyrite. However, the tailings are deemed to be not potentially acid-generating due to the presence of excess neutralising minerals, namely calcite and dolomite. The NAG pH was recorded as 10.5 and the AP:NP ratio was 3.1.

Potentially, under Finnish mine waste regulation VNa 190/2013 this would indicate that the mineral waste was inert, however, results of acid digestion with Aqua Regia indicate that extractable cobalt and copper concentrations exceed the PIMA threshold values, and the chromium and nickel concentrations exceed the upper PIMA guidance values. In addition, extractable molybdenum concentration exceeds slightly the SAMASE guidance value (SAMESE is an older reference, with a wider parameter range, later replaced by PIMA). Therefore, the tailings sample cannot be considered as environmentally inert rock material in accordance with the mine waste regulations (Government Decree 190/2013, Appendix 1 and further reference to Government Decree 7214/2007).

GTK report that the risk for mobilization of nickel, cobalt and possibly also chromium is high, since nickel and cobalt are bound to easily weatherable sulphide minerals, and chromium-containing silicates coexist with acid-producing pyrrhotite. Yet they go on to report that in the initial contact test, none of these elements was observed to be mobile to a significant level, and most reported results were below detection limits.

According to trail process tailings asbestos analysis (asbestos fibres > 5 µm), fibre content is 109 fibres/µg and 0.08 mass-%. The presence of asbestos must be taken into consideration in the project planning. The asbestos quantity indicates asbestos levels that may require specific health and safety measures at the site.

17.1.2 Summary of process water

Available tailings and process water samples came from a “non-locked-cycle” process trial. Therefore, actual process water quality is not yet known. Lack of process water circulation is also likely to be reflected in secondary mineral content or adsorbed substances in tailings. Process water quality impacts the tailing's pore water quality and the quality of the water separating from the tailings, together with oxidation and other long-term weathering of the tailings.

In the grinding phase, process water concentrations of sulphate, nickel, antimony and zinc were elevated (if compared to reference VNA 341/2009). In trail water balance water taken out of the process at the grinding phase was 21 % of all process water. Groundwater environmental quality standards (VNA 341/2009) was loosely used as a reference, to elucidate elevated concentrations of different substances, but there are no real limit values for process water or tailings water. There were no other “exceedances” (VNA 341/2009) in the sampled process phases, but the used tap water had very high copper concentrations (over 3-fold the groundwater environmental quality standard).

The tailings water was the largest water quantity in the trial process water balance, 42 % of all water taken out of the process. The tailings water was slightly saline and included some sulphates, heavy metals and alkaline or alkaline earth metals (for example sodium, calcium, strontium, zinc, molybdenum and nickel), but none of these was in especially large concentrations. Anyhow, tailings water alone is not representative in a process trial without water circulation. In a copper circuit, the phosphorus content of the water is noteworthy and potentially the outcome of collector chemicals. In addition to that, copper circuit water included more or less the same substances as tailings water.

In addition to the substance mobilisation risks from tailings (reported by GTK 2021), based on process water analysis, also phosphorus and sulphate loads may require attention, potentially even antimony and zinc. Phosphorus-containing promoter chemical is used, which is likely to impact the TSF water quality (but due to the process trial without water circulation, difficult to estimate from the tailings or process water).

17.2 Tailings area

Site conditions

The TSF (Tailings Storage Facility) is placed at the old Keretti tailings area (13.1 ha). The slurry disposal (spigotting) is used.

The old tailings area is approximately 45 ha in total and the new TSF will be located to the south part of it where the old tailings are supposed to consist of fine fraction. According to the few tailings samples made by FinnCobalt Oy in the summer of 2021, the old tailings are silty (88-96 % passing seave 75 μm). At the moment the depth of the old tailings, grain size distribution, water content and density are unknown.

The old tailings area has been constructed into a lake and surrounded and divided by several dams. Dam materials, dam types or raising methods of old dams are unknown. The upstream method and tailings may have been used for the dam raisings. According to the old aerial photos, the depth of tailings can be 15-20 meters in the middle of the old lake and only a few meters in shoreline areas. The tailings have been discharged as slurry from the northeast to the southwest. The coarse part of the tailings has been settled down nearby the discharge points and the fine part has settled down into the supernatant water pond. Most likely the moraine hills and isles have been used for dam raisings.

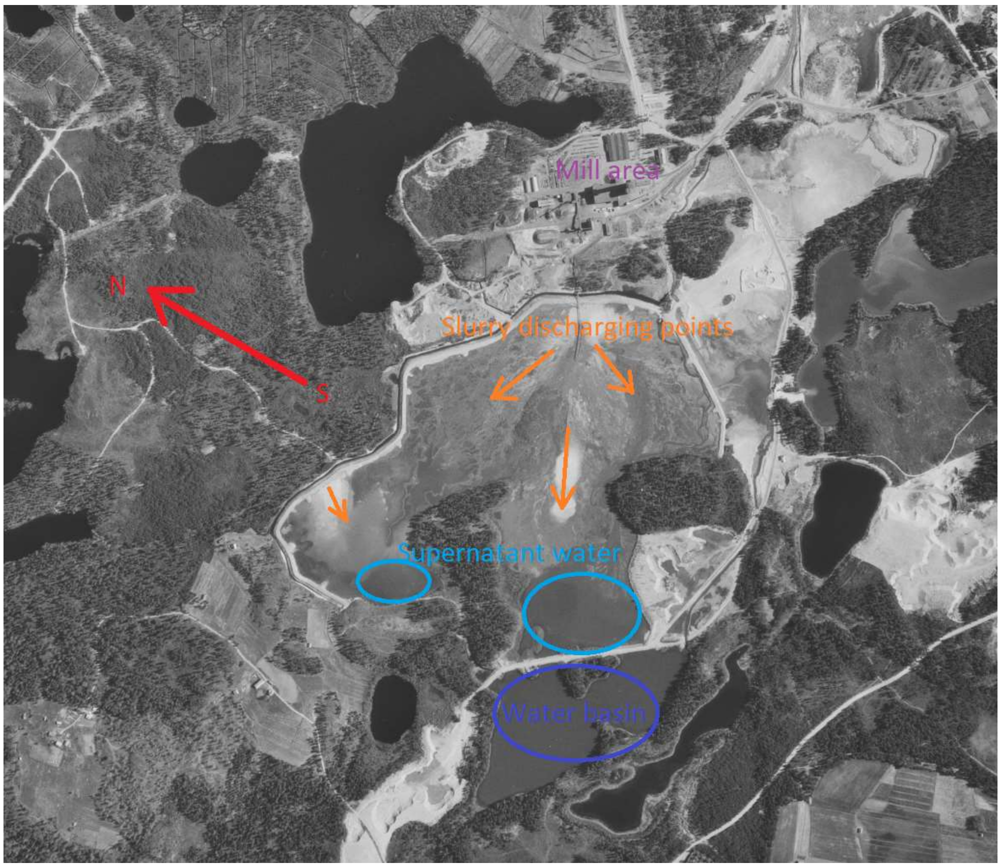


Figure 17-1 Operating in tailings area in 1971 according to the old aerial photo

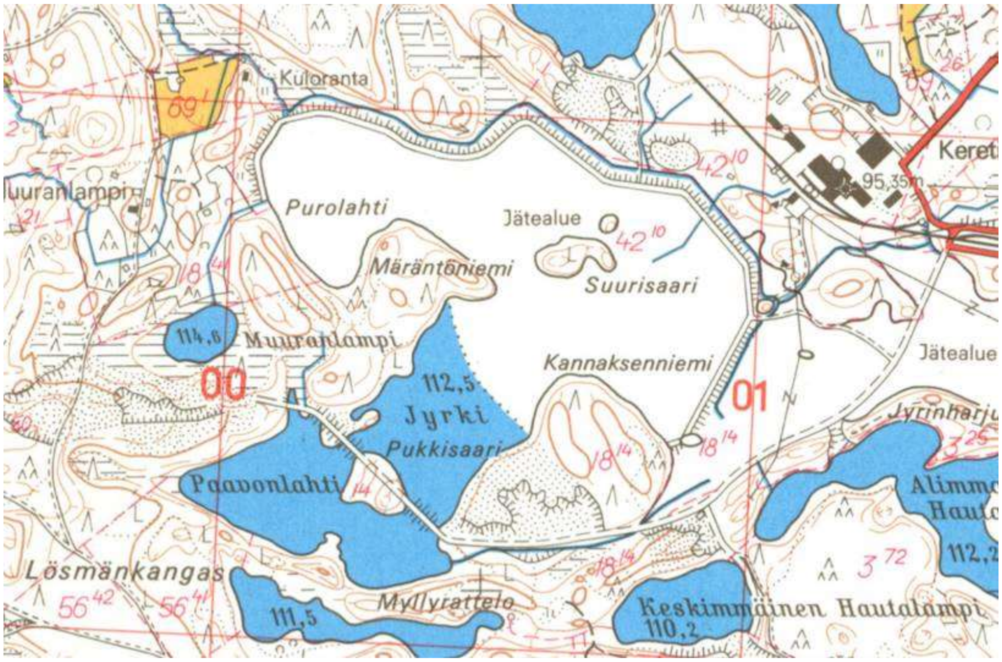


Figure 17-2 Map of area 1974

Tailings production and storage volumes

The total production of the tailings is approximately 4 683 000 tonnes. About 41 % of the produced tailings (1 920 000 tonnes) will be used as backfilling and the rest will be transported as slurry to the tailings storage facility (TSF).

The total amount of the tailings to be deposited in TSF is approximately 2.763 Mt and the required storage capacity in the tailings basin is approximately 2.033 Mm^3. The tailings in situ dry density are estimated to vary between 1.35…1.40 t/m^3 as the tailings basin can be drained also through the dams that helps the tailing to dry and compact. At the same time, dry beach areas may cause dust problems.

Tailings tonnages and volumes based on estimated in situ dry densities are presented in table 17-11-1.

Table 17-1. Estimated tailings volumes.

Year	Tailings to TSF, dry tonnages [t/a]	Annual volume [m³/a] $\gamma_{d\ in\ situ} = 1{,}40\ t/m^3$	Annual volume [m³/a] $\gamma_{d\ in\ situ} = 1{,}35\ t/m^3$
1	104 060	74 330	77 080
2	276 480	197 490	204 800
3	260 680	186 200	193 100
4	263 960	188 540	195 530
5	263 900	188 500	195 480
6	263 050	187 890	194 850
7	262 600	187 570	194 520
8	257 130	183 660	190 470
9	261 980	187 130	194 060
10	265 300	189 500	196 520
11	236 500	168 930	175 190
12	47 470	33 910	35 160
total	2 763 110	1 973 650	2 046 760

Tailings storage facility, dams and basal structure

The new tailings basins will be constructed with a sealing liner (LLPDE Linear low-density polyethylene / BGM Bituminous geomembrane) with required strengthening structures (e.g. soil net and geotextile) and protection layers. These structures must be dimensioned and soil investigations must be carried out in the next stage at the old tailings area. The uneven consolidation may cause the breakage of the liner which can be a problem. Also anchoring the liner can be difficult in large areas.

The sealing structure will be installed under the starter dams and the seepage water collected from the ditch located downstream of the dam. This requires the dam to function as a drain. The stater dam will be constructed with a thin moraine core to prevent the outflow of the fine tailings. An embankment is made of gravel or some other permeable material. If the gravel or the embankment material doesn’t fulfil the filter criteria, separate filter layers must be constructed between the core and the embankment. The starter dam height is 4-5 meters, the dam crest width is 4 meters and slopes are 1:2.5 or flatter. Later the tailings areas are planned to be raised with the tailings by using the upstream method. The tailings suitability for the dam raise is not tested (permeability, strength parameters). If the tailings are not suitable for the purpose, some other material (e.g. gravel) can be used.

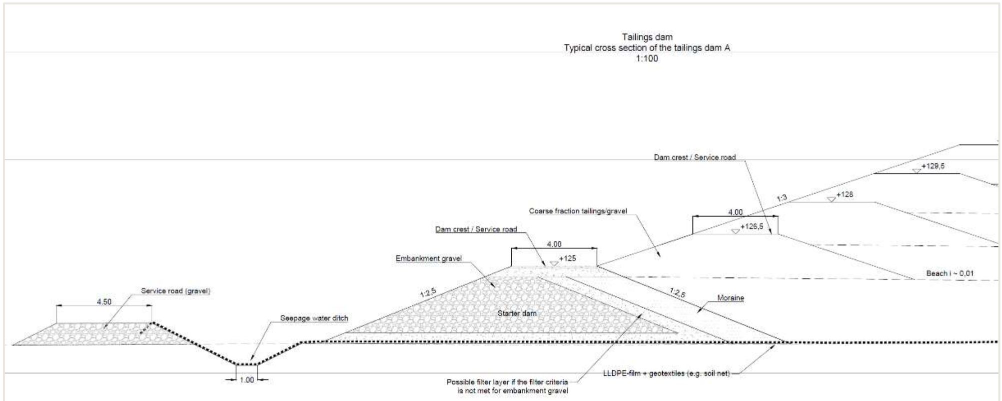


Figure 17-3 Principle of the starter dam and dam raisings (upstream).

The dam crest should have sufficient width over its entire length and a structure suitable for maintenance traffic. For class 1 and 2 earth-fill dams, a crest width of 4 m or more is considered sufficient. The crest of an embankment dam of classes 1 and 2 must be passable to traffic throughout its length. (Dam Safety Guide, Finland).

The freeboard (the difference in height between the dam crest and the highest water level) of class 1 and 2 dams should equal the frost depth that occurs at least once in ten years. At tailings dams, the HW level is defined as the highest level of free water. A moraine-core zoned embankment dam is determined with the square root of the cold content and values of coefficient k of the dam type from the formula $Z = k \times \sqrt{F}$. In Outokumpu, F_{10} is 40000 Kh and moraine core k is 0.013, this gives the frost depth of 2.6 m. (Dam Safety Guide, Finland). In this case, the dam is meant to be permeable and the phreatic surface will drop down in the embankment and the required freeboard is not valid in a conventional way.

Estimated storage capacities based on preliminary dam engineering are presented in Table 17-2.

Table 17-2. Estimated capacities (volumes) for the TSF.

Dam stage	Dam crest	Storage capacity [m^3]	Storage capacity, cumulative [m^3]	Estimated end of capacity [production years]
Starter dam	+127,0	628 600	628 600	3,5…4
1. raise	+128,5	311 900	940 500	5,5…6
2. raise	+130,0	290 000	1 230 500	6,5…7
3. raise	+131,5	282 200	1 512 700	8,5…9
4. raise	+133,0	265 900	1 778 600	9,5…10
5. raise	+134,5	254 800	2 033 400	11,5…12

High sulphide tailings (~5 t/h) will be normally used for backfilling. If the underground mine can not for some reason receive backfilling the linered (bentonite mat + HDPE liner) tailings basin for a few months production (6-8 months, 17 500 m^3) has been designed for temporary use. The basin is tentatively located to the north of the process plant area.

Tailings Geotechnical and Hydraulic Properties

According to the mini-pilot test (GTK 2018), tailings are silty material (d_{50} = 0.04 mm) and the hydraulic conductivity is estimated to be $K \sim 5 \cdot 10^{-7} \ldots 10^{-6}$ m/s. The hydraulic conductivity is highly dependent on the fine particles (d_{10}) and this information is still missing. Using the spigotting from the ring dam the fine particles will end up in the middle of the basin and form a low conductivity area and a supernatant pond from where the water must be discharged/pumped to the clarification basin. If the tailings can be used for dam rising, capillary rise above the measured position of the phreatic surface can make the tailings in this zone to be close to full saturation. This can produce unexpectedly large rises of the phreatic surface from small amounts of rainfall and this can be critical for the stability of the tailings dams.

Using the upstream method the tailings beach must form a competent foundation for the support of the next dam raising and as a general rule the discharge should contain at least 40-60 per cent sand particles.

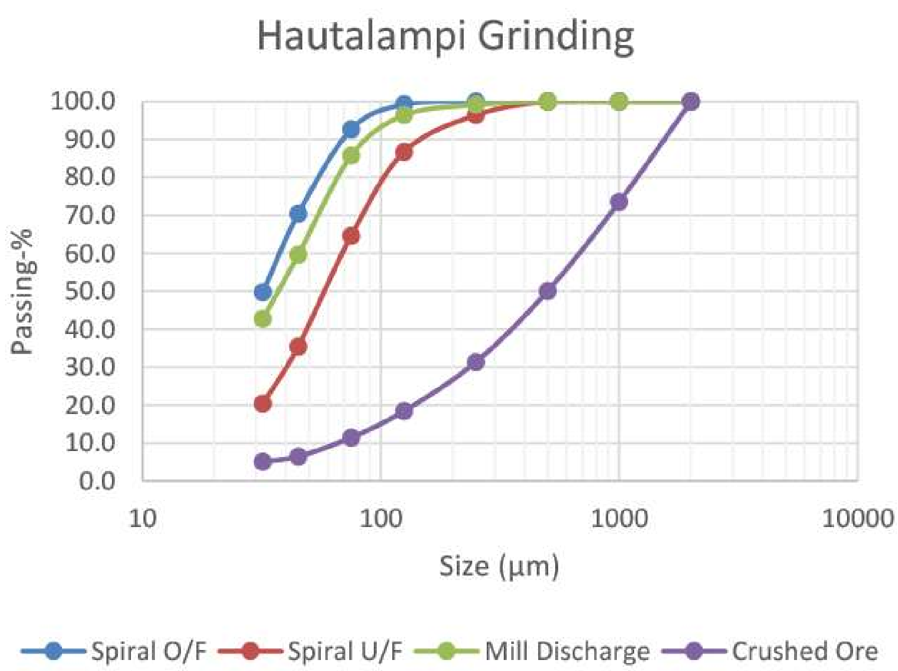


Figure 17-4 Particle size distribution curves (GTK 2018)

Specific earthquake guidelines for dam stability calculations do not exist in Finland. Since earthquakes may have a significant impact on the upstream raised dam stability, this must be considered in the next planning stage and with known material parameters of dams and basal soil.

In this case, the basins are quite small. The inclination of the tailings surface (the beach) is estimated to be 1:100 when using several spigotting points. Because of the upstream method for dam raisings, the area of the settling is getting smaller after every raise and this may cause problems for settling and decline of the beach slope. If the tailing's solid content in the slurry is about 20 %, this will cause problems for beach sloping and sediment settling. Even if the tailings will dry nearby the perimeter dams the centre of the basin might be wet and the tailings unsettled. Fine tailings may cause a poorly permeable bottom and supernatant water must be pumped to separate the clarification basin.

17.3 Waste rock characteristics

17.3.1 Waste rock quality assumptions

Actual waste rock assessments have not been done yet. Most of the waste rock is needed as backfill material, but waste rock quality and mine wall quality will have an impact on the mine water. Therefore information is needed for water quality assessments.

According to Meriläinen et al. (2008, 2009), The Hautalampi mineralised zone is a southwest part of the Co-Ni-Cu-mineralisation zone, parallel to the Outokumpu Keretti ore body. The Co-Ni-Cu zone differs from the main Cu ore environment for example due to the frequent variation between cummingtonite, anthophyllite, cordierite, staurolite, garnet, phlogopite and spinel-bearing chlorite-rich rocks/schists, hosted as thin layers or patches in skarn (diopside-tremolite)-quartz rocks forming the bulk of the Co-Ni-Cu zone. Also, the relative cobalt-nickel enrichment is distinctive. The mineralised zone has in sometimes a very sharp contact with the wall rocks. In some places, a transitional zone from one meter up to three meters occurs between the mineralised zone and wall rocks. According to Geological Survey (GTK 2021), key sulphide minerals in the mineralisation are pyrrhotite, pyrite, chalcopyrite, zincblende and pentlandite.

Waste rocks and mine wall rocks can be assumed to be variable. Both sulphide mineral and carbonate mineral presence can be expected. Considering the known metals present (nickel, cobalt), neutralisation potential is unlikely to prevent metal leaching to a significant extent; the assumed harmful elements present are well soluble in circum-neutral waters.

17.4 Waste rock area

Waste rock will be normally used for backfilling and only temporary storage for first-year waste rock will be built near the tunnel portal. The total area is about 1.5 ha and the height will be 5-6 meters with slopes of 1:1.5 to1:2. Because the waste rock comes from the tunnel and might be oxidized and weathered the waste rock basal structure will be made of a one-meter layer of fine moraine or with HDPE liner and protective layers. Seepage waters will be collected and discharged into the raw water pond.

18 Water and Wastewater Management

18.1 Site Water Management

18.1.1 Introduction

The subsequent sections present the general site water management plan, site water balance, and technical water management and treatment solutions at the pre-engineering/PFS level for the Hautalampi PFS Project.

18.1.2 Water Management Principles

The target has been to minimise the CAPEX costs by optimising the water management solution. The general site water management principles and actions are:

• Contaminated surface water is not discharged to the environment without purification.

• Freshwater abstraction from natural sources is minimised. The water of the Recirculation pond is mainly used as raw water after sand filtration.

• Any contaminated water is treated in an active treatment plant before the discharge.

Uncontaminated water streams can be diverted towards surrounding watercourses.

Water Stream	Design Principles and Justifications
Freshwater	Suu-Särki lake serves as the freshwater abstraction source. The average designed water intake flow rate is 6.9 m³/h and a maximum of 30 m³/h in case all raw water is taken from the lake. Freshwater is treated for iron and humus removal. Most of the raw water can be taken from the Recirculation Basin which is treated with sand filtration.
Mine Dewatering	The mine dewatering includes ground- and surface water seeping into the mine by pumping the water into the Tailings Basin. The advantage is that solids of the dewatering water settle into the Tailings Basin. The drawback is the mixing of water streams having probably different water qualities.
Water Released from Tailings	Tailings having 18% solids are pumped into the Tailings Basin. Solids concentration is estimated to increase up to 70% in stacking. Tailings are pumped into the tailings basins in turns.
Processing Plant Area Runoff and Site Water Streams	The Process Area corresponds to the processing plant, maintenance, industrial areas, and ore storage. Runoff from the processing plant area cannot be discharged

	without purification due to possible contamination. Runoff is directed via Equalizing Basin into the Recirculation Basin.
Tailings Area Water Management	Drainage waters of the tailings area, water released from tailings, and mine dewatering water are collected from the Tailings Basin into the Recirculation Basin. The Concentration Plant intakes water from the Recirculation Basin. The excess water is led to the environment via active water treatment.
Waste Rock Storage	The water is collected into a small equalizing basin from where it is pumped to the equalizing basin of the Processing Plant Area and from there to the Recirculation Basin.
Effluent Treatment and Water Discharge	The effluent treatment consists of an effluent precipitation process (hydroxide precipitation). The precipitate is settled in a settling basin before discharge to the environment. After active effluent treatment, the water is discharged to the Alimmainen Hautalampi Lake.
High-S tailings basin	The basin is used for the temporary storage of high sulphide tailings storage. The water of the basin is emptied when needed into the Recirculation Water Basin via Equalizing Basin.

The conceptual water cycle graph is presented in Figure 18-1 and Appendix 101016697-02021.

The general flow diagram is presented in Appendix 101016697-02024.

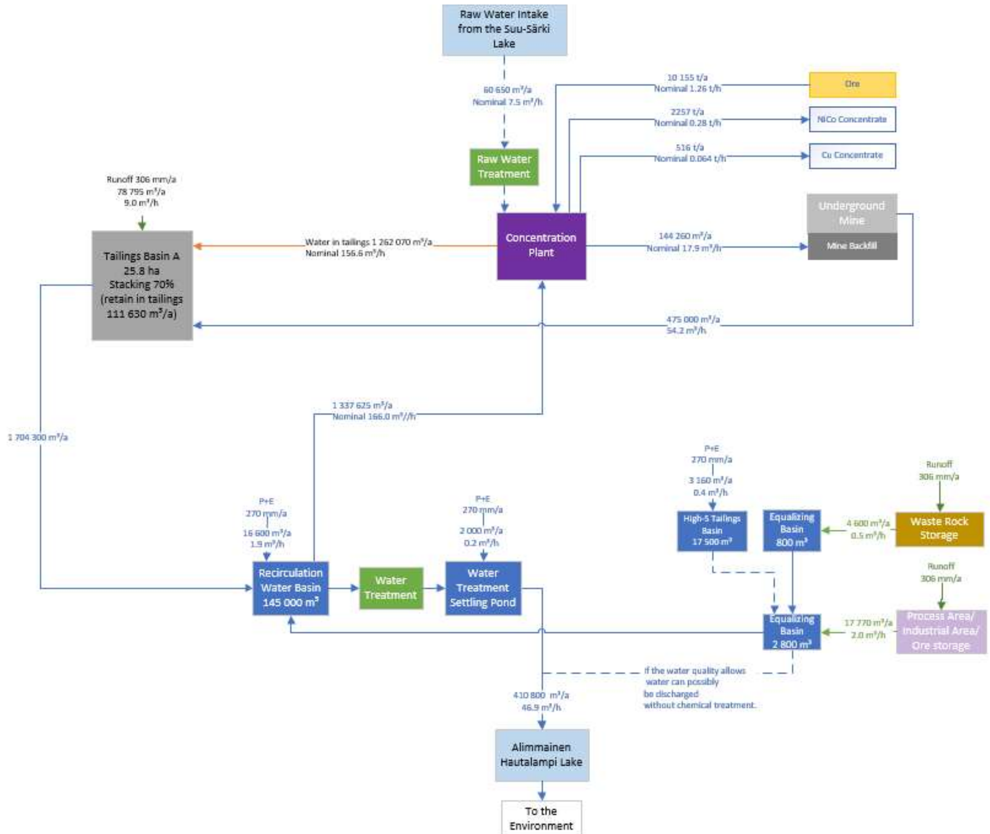


Figure 18-1. Conceptual Water Cycle Graph.

18.1.3 Site-Wide Water Balance Modelling

Introduction

As part of this PFS, the site-integrated water balance modelling has been conducted for the operational phase. The water balance is generated as a deterministic model in Microsoft Excel. Modelling is performed on monthly basis. The water balance model is Appendix 101016697-02022 Water Balance Model.

Modelling Scenarios

The water balance model has been calculated for an average hydrological scenario and a wet scenario. A 100-year return period was used to calculate the wet scenario. Dimensioning of the water systems (pumps and piping) is based on a wet May which occurs once in 100 years when runoff is 84.6 l/s*km². The wet May scenario has been generated by deriving the runoff percentages from running a frequency analysis to the May runoff values with a Gumbel distribution.

The wet scenario has been generated by multiplying the monthly precipitation and runoff of each month by the ratio of a wet and average year.

Table 18-1. Precipitation, evaporation from open water surfaces, runoff and net precipitation in a hydrologically average year.

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Precipitation (mm) Outokumpu Viuruniemi FMI 1980-2020	45.6	35.6	35.7	33.1	43.7	68.4	73.7	78.6	62.0	59.7	54.2	55.3
Evaporation from the open water surface (mm), Liperi, Lapinlinna, 1988-1995					79.4	98.5	98.4	61.4	28.1	9.8		
Runoff (l/s*km2), Kuokkalanoja Outokumpu 052	2.8	2.3	3.2	29.7	31.0	7.4	4.3	4.4	6.1	9.6	9.9	5.6
Net precipitation (Precipitation - evaporation) (mm)	45.6	35.6	35.7	33.1	-35.8	-30.0	-24.7	17.2	33.9	49.9	54.2	55.3

Table 18-2. Precipitation, evaporation from open water surfaces, runoff and net precipitation in 1/100 repeating wet year.

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Precipitation (mm) Outokumpu Viuruniemi FMI 1980-2020	65.4	51.0	51.2	47.5	62.6	98.2	105.8	112.7	88.9	85.6	77.8	79.4
Evaporation from the open water surface (mm), Liperi, Lapinlinna, 1988-1995, SYKE					79.4	98.5	98.4	61.4	28.1	9.8		
Runoff (l/s*km2), Kuokkalanoja Outokumpu 052, SYKE	4.0	3.3	4.5	42.5	44.5	10.7	6.1	6.4	8.8	13.8	14.2	8.0
Net precipitation (Precipitation - evaporation) (mm)	65.4	51.0	51.2	47.5	-16.8	-0.3	7.3	51.4	60.8	75.8	77.8	79.4

Model Input Data

Hydrological Data

The data sources used for the site hydrological assessment are:

- FMI – Finnish Meteorological Institute, open data
- SYKE - Finnish Environment Institute (Suomen ympäristökeskus), HERTTA database

Catchment Areas

The catchment areas or drainage/runoff areas that have been used in the calculations for precipitation and surface runoff are presented in Table 18-3.

Table 18-3. Catchment Areas

Area	Area (ha)
Tailings Basin A	25.8
Process Area and Ore Storage	5.8
Waste Rock Storage	1.5
Recirculation Pond	6.2
Water Treatment Settling Pond	0.8
High-S Tailings Basin	1.2
In total	**41.1**

Process Constraints and Requirements

Ore feed includes 1.26 t/h of water. The NiCo and Cu concentrates contain 0.28 and 0.064 t/h of water, respectively.

The water balance modelling estimates that the continuous raw water intake from the Suu-Särki lake would be 6.9 m³/h which is 25% of the total raw water needed. The main part of raw water 22.6 m³/h is delivered from the Recirculation Basin. The water is filtered with sand filtration to remove solid particles. In practice, raw water is needed e.g. in the making of chemical solutions and its consumption may vary. Raw water intake from the Suu-Särki Lake is minimized.

The nominal amount of water in the tailings pumped to the tailings area is 156.6 m³/h. Filtered tailings have a solid content of approximately 18%. The tailings will release water and in stacking solid concentration is estimated to increase up to around 70%. The concentration will be reviewed in the further design phases. The tailings basins are filled in turns.

Some tailings will be backfilled in the underground mine which is notified in the water balance. Backfill is fed underground at a nominal water flow rate of 17.9 m³/h.

Mine Dewatering Rates

The estimated mine dewatering rate, based on mining volume, is 475 000 m³/a (54.2 m³/h). Groundwater is assumed to be pumped continuously all year round. The mine dewatering flow rate does not take into account water released from the backfilled material. Mine dewatering water is pumped into the same tailings basin where tailings are pumped.

Modelling Results

The annual excess water flow rate from the mine site is presented in Table 18-4 for average and wet years. Water can be discharged at a maximum of 110 m³/h for example in wet May or when needed. The discharge rate depends on the regulation of the recirculation pond.

Table 18-4. Annual Excess Water Volume and Maximum Discharge Rate.

	Average Year	1/100 Wet Year
Excess Water Volume	408 800 m³/a	473 500 m³/a
Average Discharge Rate	47 m³/h	55 m³/h maximum 110 m³/h

The monthly discharge flow rate to the environment in an average year, wet year and wet May are presented in Figure 18-2. In winter, water discharge from the mine site consists mainly of mine dewatering water discharge and water released from tailings. The amount of water intake to the concentration plant from the recirculation basin is estimated to be higher compared to water separated from the tailings. Mine dewatering water pumped into the Recirculation Basin via tailings pond compensates for the water need. The Recirculation Basin is regulated to ensure water delivery to the Concentration plant, to regulate excess water discharge to the environment e.g. during snow melting and to reduce water treatment unit capacity. The highest discharge to the environment occurs in April and May due to snow melting.

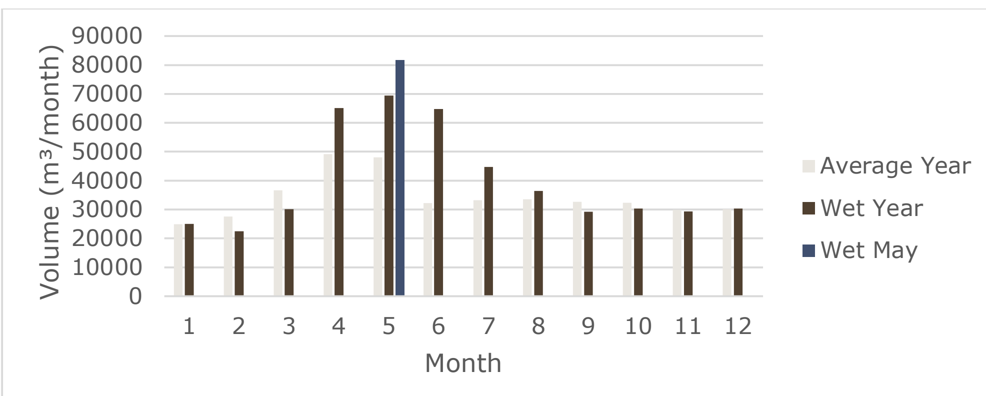


Figure 18-2. Excess Water Volume in Average Year and Wet Year and in Wet May.

18.1.3.1 Risks

It is highlighted that the water balance estimate involves the following uncertainties, which should be solved during the following design phases:

- The water balance does not take LOM into account. The water balance has been calculated for a scenario where all the areas are constructed and operating.
- The water balance takes mine backfill into account. Backfilling is not made continuously which alters the water balance temporarily.
- Water is released from the mine backfill which is not taken into account in the estimation of the mine dewatering water amount.
- The solid concentration of tailings which develops during stacking affects on water flow rate into the Recirculation Basin. The 70% stacking concentration is a rough estimate and needs to be revised in later design phases.
- Having one common water circulation instead of separating process water and mine-dewatering water can make water treatment more challenging and deteriorate mine impacted excess water quality.
- The excess water discharge into the Alimmainen Hautalampi Lake is unsure due to the ecological classification of the receiving Ruutunjoki River and the Sysmäjärvi Lake. If the Alimmainen Hautalampi Lake cannot receive the excess water a discharge pipeline may be required.

18.1.4 Loading Balance

Source term assessments have not been carried out yet and therefore load balance cannot be modelled either. The main uncertainty at this project stage is the underground mine water quality.

Based on the known lithology (Meriläinen et al. 2008, 2009), mineralogy, recent tailings assessments (GTK 2021) and process water analysis, the following assumptions can be made:

- Sulphide oxidation products will be present in the site waters.
- Key contaminants are likely to be sulphate, copper, cobalt and nickel. As potential process water impact, antimony and zinc are not yet closed out as contaminants.
- A relatively high neutralisation capacity is likely to be reflected in the waters.
  - This won´t limit significantly the solubility of nickel or cobalt, but it may limit the solubility of copper. Also, the presence of iron in the waters will impact water quality (before treatment), as it may help other metals to co-precipitate.
- The salinity of the mine water is not known.
  - The presence of sulphate is probable.
  - The increased presence of for example chlorine is possible but not certain.
- Nutrient load is also probable.

- There will be nitrogen in the mine waste, as a result of the explosives used. The nitrogen (from explosives) is also present in the waste rock, which is used as a backfill.
- Process water analysis refers to phosphorus load. This is probably related to collector chemicals.

18.1.5 Xanthates

In the concentration process xanthates, methyl isobutyl carbinol (MIBC) and sodium isopropyl xanthate (SIPX) is used as a foaming agent and as a collector, respectively. Xanthates come into the tailings pond with tailings and from there into the recirculation pond. Xanthates are biodegradable and start to degrade in the ponds. For example, the half-life of SIPX is from 7 to 14 days at pH 5.5 and +15 ºC. However, degradation slows down in cold temperatures.

To avoid xanthate emissions to the environment sufficient retention time in the tailings pond and recirculation pond is needed. Retention time in the recirculation pond is estimated to be over 20 days in a hydrologically average year while retention time in the tailings pond is not studied. Xanthate degradation needs to be studied more closely in further design phases to ensure sufficient retention time also in cold temperatures.

18.1.6 Fresh Water Abstraction

Freshwater for the concentration process is abstracted from the Suu-Särki Lake. Lake water is used mainly in the preparation of chemical solutions or in other applications where good water quality is required. Otherwise, lake water consumption is minimized. Another raw water source is the water of the Recirculation Basin which is used if the water quality is sufficient.

18.1.7 Water Ponds and Storages

The Recirculation Water Basin collects drainage water from the tailings area, mine dewatering water pumped in the tailings area, and water released from tailings. In addition, water from the high sulphide storage basin, waste rock storage, and the concentration plant and ore storage areas are led into the basin. The pond has an impermeable liner to reduce water infiltration.

Chemically precipitated water is settled in the Water Treatment Settling Basin which is separated from the Recirculation Water Basin with a dam and discharged gravitationally into the Alimmainen Hautalampi Lake.

Table 18-5. Water Management Basins and Volumes.

Basin	Volume (m³)
Recirculation Basin	145 000
Water Treatment Settling Basin	9 000
Tailings Basin A Pumping Basin	2 400
Waste Rock Storage Equalizing Basin	800
Concentration Plant Equalizing Basin	2 800

18.1.8 Water Pumping Stations

The pump design flows of the pumping stations are based on the water balance model. The flow rate considers a once in 100 years occurring Wet May to ensure sufficient pumping capacity. In locations where water has not been retained, the pumps are dimensioned according to short-term heavy rain (Suomen kuntaliitto, 2012). The lifting height is based on the geodetic head and pipe network head losses. The applied pump type is the centrifugal submersible pump. Pumping stations are configured as 1+1 (one main pump + one spare pump).

The applied pumping station types for water management are pontoon pumping stations and package pumping stations made of plastic or glass fibre. Table 18-6 summarizes the dimensioning of water management pumping stations. Pumping station locations are presented in Appendix 101016697-02023 Water Management General Layout.

Table 18-6. Summary of water management pumping stations.

Pumping Station no.	Pumping direction	Design flow (m³/h)	Total lifting height (m)	Motor Power (kW)	Pump Configuration
PS01	Catch Basin A for seepage water → Recirculation water basin	488	7	13	1+1
PS02	Tailings basin A → Recirculation water basin	292	4	5.5	1+1
PS05	Recirculation water basin → Concentration Plant	226	9	7.5	1+1

	Recirculation water basin → Water Treatment	112	9	4	1+1
PS06	Suu-Särki Lake → Water Treatment	30	14	3	1+1
PS07	Equalizing basin (Waste Rock Area) → Equalizing basin (Process Area)	53	19	4	1+1
PS08	Equalizing basin (Process Area) → Recirculation water basin	194	13	11	1+1
PS09	High Sulphide tailings basin → Equalizing basin (Process Area)	20	17	7.5	1+1
Sanitary sewage PS	To municipal wastewater treatment plant	6	19	2.4	1+1

18.1.9 Water Transportation Pipelines

The pressurized water management pipelines are planned and dimensioned as plastic high-density polyethylene (HDPE) pipes, utilizing pressure class PN10. Gravitational pipelines are planned and dimensioned as polyvinyl chloride (PVC) pipes, utilizing strength class SN8.

The Tailings pipeline from the concentration plant to the hole for high-S tailings underground backfilling is dimensioned as plastic high-density polyethylene (HDPE) pipes, utilizing pressure class PN16.

The SFS-EN 1451-1 standard and the Confederation of Finnish Construction Industries "InfraRYL" guidelines dictate the pipe material and installation. Pipelines will be installed underground below the frost line at approximately 2.7 m depth measured from the top of the pipe to the existing ground surface. The installation depth can vary depending on the subsurface or the use of insulating material. The standard flow velocity for pressure pipes is approximately 1 m/s.

The water management pipelines are summarized in Table 18-7. The pipeline routes are presented in Appendix 101016697-02023 Water Management General Layout.

Table 18-7. Summary of water management pumping stations.

Pipeline	Flowing direction	Pipe dimension	Pipe length (m)	Pipeline type
PL01	Underground Mine → Tailings basin A	200-10 PEH	1 680	Pressure pipeline
PL03	Catch basin A for seepage water → Recirculation water basin	450-10 PEH	689	Pressure pipeline
PL04	Tailings basin A → Recirculation water basin	355-10 PEH	282	Pressure pipeline
PL07	Recirculation water basin → Concentration Plant	315-10 PEH	870	Pressure pipeline
PL07b	Recirculation water basin → Water Treatment	225-10 PEH	100	Pressure pipeline
PL08	Water Treatment Settling Pond → Recharge (Alimmainen Hautalampi Lake)	250 PVC SN8	340	Gravitational pipeline
PL09	Suu-Särki Lake → Raw Water Treatment	110-10 PEH	255	Pressure pipeline
PL10	Equalizing basin (Waste Rock Area) → Equalizing basin (Process Area)	140-10 PEH	485	Pressure pipeline
PL11	Equalizing basin (Process Area) → Recirculation water basin	280-10 PEH	920	Pressure pipeline
PL11b	Option: Equalizing Basin (Process Area) → Discharge (Alimmainen Hautalampi Lake)	315 PVC SN8	230	Gravitational pipeline
PL12	High Sulphide tailings basin → Equalizing basin (Process Area)	90-10 PEH	580	Pressure pipeline
PL13 (tailings)	Concentration plant → Hole for high-S tailings underground backfilling	160-16 PEH	1020	Pressure pipeline

PL14 (tailings)	Concentration Plant → High Sulphide tailings basin	160-16 PEH	215	Pressure pipeline
PL15 (tailings)	Concentration Plant → Tailings basin A	225-16 PEH	1055	Pressure pipeline

18.1.10 Water Treatment

18.1.10.1 Hydroxide Precipitation

Treatment of mine impacted excess water relies on active water treatment methods. Active water treatment is preferred to avoid possible deterioration of water quality and ecological condition of water bodies. In addition, water quality can be controlled easier compared to passive treatment methods.

Hydroxide precipitation at pH 9-11, where metals have the lowest solubility in water, is recommended. The water treatment unit is dimensioned for a flow rate of 110 m^3/h which discharge rate may be needed e.g. in 1/100 repeating wet May. It is estimated that water can be regulated in the Recirculation Basin which reduces dimensioning of the treatment unit.

The recommended precipitation chemical is calcium hydroxide ($Ca(OH)_2$) whose dosage is estimated to be in the range of 0.1-2.0 kg/m^3. The dosage depends on the water quality. Lime is added to the water in four mixing reactors. Retention time is estimated to be 40 min or more in total. The reactors can be taken into use according to the water flow rate.

After the active water treatment process water pH might be elevated. pH may be adjusted before discharge with acid to achieve lower pH values.

The formed precipitate is separated in the water treatment settling basin. Particles are settled in the basins for at least 72 h to achieve efficient solid separation. In that time, a clay particle of #0.0015 mm settles 1 m in depth. Water leaves the ponds as an overflow to minimize solids' release into the environment.

Water treatment sludge is settled in the Water Treatment Settling Basin. If needed, the sludge settled in the basin may be emptied into the Tailings Basin.

The excess water treatment is located in the proximity of the Recirculation Water Basin and the Water Treatment Settling Pond. The raw water treatment unit is at the concentration plant area.

The excess water treatment flow diagram is presented in Appendix 101016697-02025.

18.1.10.2 Raw Water Treatment

In the treatment of the Suu-Särki lake water, the first raw water treatment step is the removal of large particles with a screen. Iron or aluminium-based coagulant can be added to water and pH is adjusted with NaOH. The formed precipitate is removed with sand filters.

The sand filters are flushed to avoid fouling. Diluting cleaning water amount can be considerable and its discharge can be challenging. To reduce its amount, a lamella clarifier, enhanced with polymer feed, can be applied. Concentrated sludge coming from the lamella is approximately 0.2 m^3/h while raw water production is 32 m^3/h. The sludge can be transported e.g. to the Tailings Basin.

18.1.10.3 Risks

It is highlighted that the suggested mine impacted excess water treatment process involves the following uncertainties, which should be solved during the following design phases:

- Water stream qualities are unknown and for example, the water treatment process has been selected based on insufficient water quality data. Therefore, the water treatment process selection needs to be reviewed in later design phases when there is more source-term data available.
- Mine dewatering water quality differs from the tailings pond water quality. The tailings pond water may contain more sodium (Na) ions compared to dewatering water due to the consumption of Na-containing chemicals in the concentration process. An increase in Na concentration may increase sulphate solubility and may increase environmental loading. Consumption of lime may increase in the hydroxide precipitation process. Having one common water circulation needs to be reviewed in later design phases.

A process flow diagram of the excess water treatment is presented in a separate drawing. A list of process equipment is presented in the Items List.

18.1.11 Water Discharge

Treated discharge waters are led via a pipeline to a ditch which leads to the Alimmainen Hautalampi Lake. The Alimmainen Hautalampi Lake discharges via the Ruutunjoki River to the Sysmäjärvi Lake. The discharge point may be reviewed in later design phases.

Ecological conditions of the surrounding watercourses may limit discharge from the mine site to the nearest watercourses. If water cannot be discharged to the Suu-Särki lake, a discharge pipeline may be needed.

18.1.12 Site Service Utilities

18.1.12.1 Potable Water

Potable water is supplied from the municipal network. The potable water network is situated in the proximity of the mine site (Figure 18-3). The connection to the municipal water network requires the construction of an underground pipeline made of high-density polyethylene (e.g. 75-10 HDPE) at a frost-free level.

The potable water consumption is estimated at approximately 70 m³/d of which analysers consume the main part. Potable water is consumed also at social premises. More detailed plans for potable water supply will be prepared in the DFS phase.

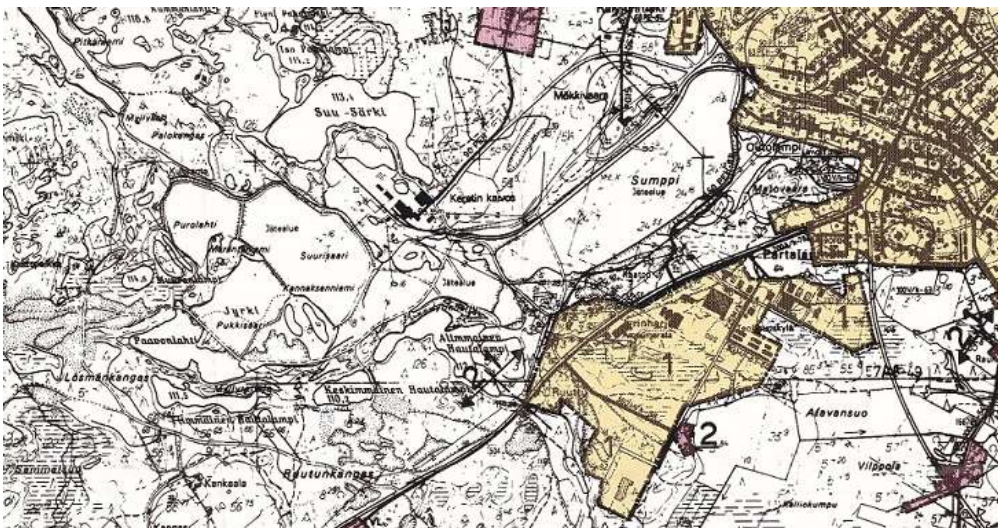


Figure 18-3. Potable Water Network Area in Yellow. (Source: https://www.outokummunkaupunki.fi/documents/291232/983109/Vesijohtoverkoston+toiminta-alue%2C%20pienennetty/39d16e87-126b-4066-96af-05bc6eb31e88?t=1486017713301)

18.1.12.2 Sanitary Sewerage

Sanitary water is supplied to municipal sewerage which is located in the proximity of the mine site (Figure 18-4). Approximately 8 m³ per day of sanitary water is expected to be generated during the operational phase, which includes 20% of leakage water.

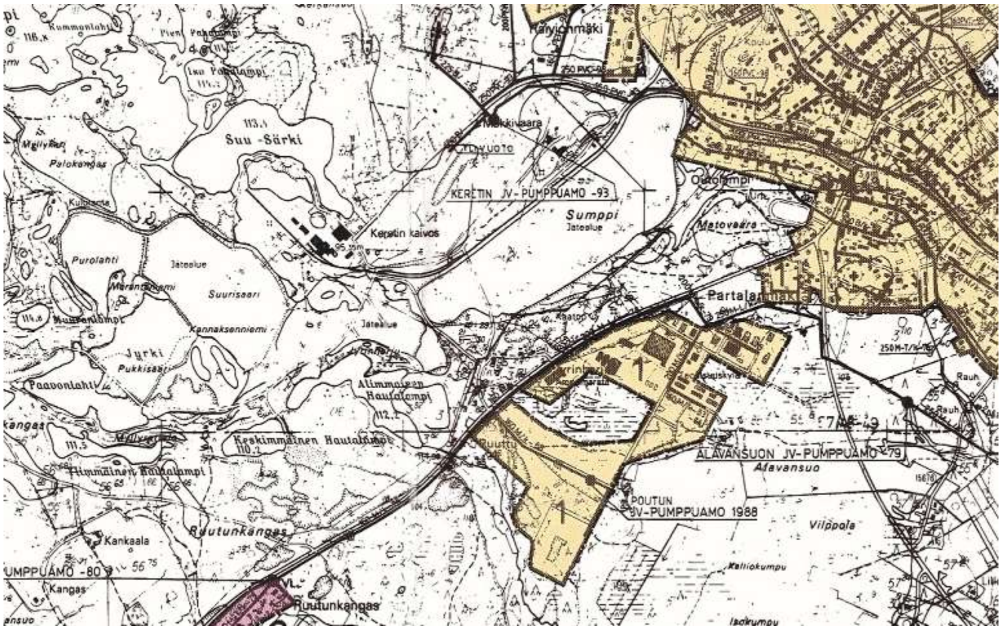


Figure 18-4. Sewerage Area in Yellow. (Source: https://www.outokummunkaupunki.fi/documents/291232/983109/J%C3%A4tevesiverkoston+toiminta-alueet%2C%20pienennetty/9e13b566-0d2a-4f48-a95c-3a3ab0c8cf88?t=1486018279620)

19 Project infrastructure

19.1 Layout

The general site layout and processing area layout has been developed for this Study to show preliminary locations and to serve as a basis for Capex. In this Pre-feasibility study phase, the purpose is to provide an idea of sizing different functions and how components could be placed and need to be reviewed and possibly updated based on supplemented data in upcoming study phases. The general site layout presents the main components covering the whole site (Figure 19-1).

A more detailed drawing showing the general site layout can be found in Appendix 101016404-10001 Site Layout.

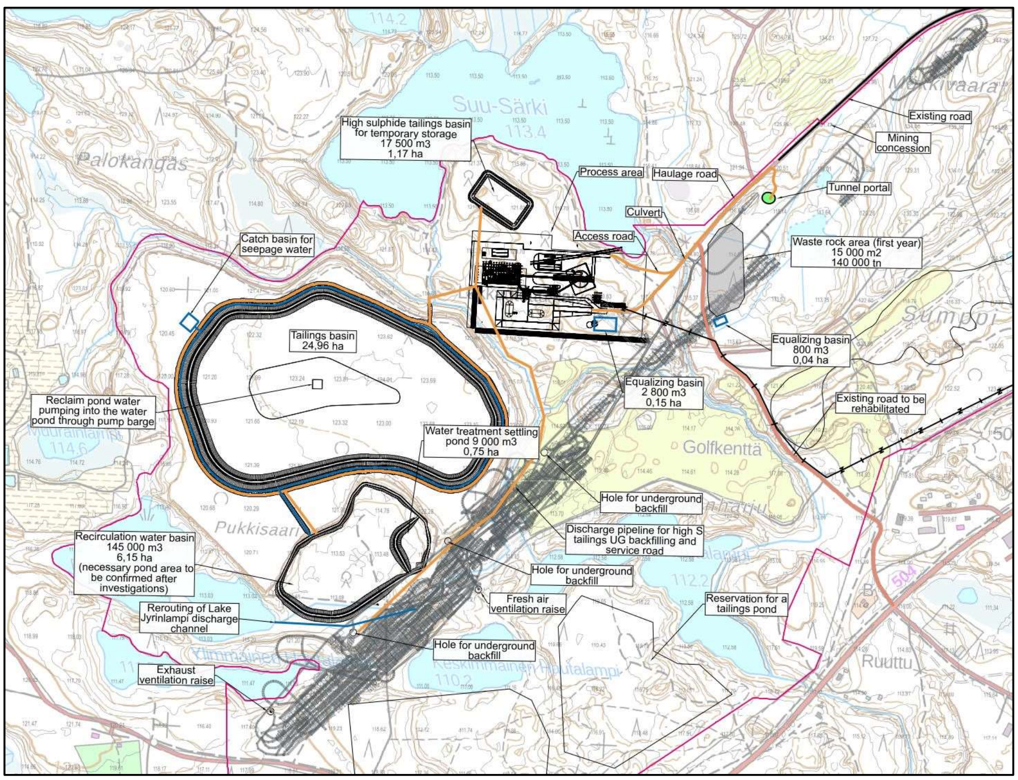


Figure 19-1. General site layout.

19.2 Access and Roads

Access road (ACR) to the processing area is from the south along an existing road called Keretintie which branches off from road Kuusjärventie (road nr. 504). The condition of Keretintie has not been investigated in this study phase. Based on aerial photos, the road seems to be gravel-coated and according to laser scanning data from the National Land Survey of Finland (NLS), the current width of the road is approximately 7–8 m, which is rather sufficient. The distance to the processing area from Kuusjärventie is 1.3 km.

Vertical alignment of the road has a couple of insufficient sections where there are problems with either longitudinal slope (the steepest slope ≈ 8%) or vertical curve is too small (minimum R ≈ 50 m). Those spots are, however, relatively easy to be fixed by embanking.

ACR should be following requirements stipulated for passenger vehicles and heavy traffic under the following circumstances:

- low traffic flow (< 2 500 vehicles/day)
- design speed 60 km/h

The designed width of the access road is 8.0 m with drivable width of at least 7.0 m. The road superstructure enhancement consists of local materials (sand, gravel) and crushed aggregate as the base course. Depending on groundwater level and drainage conditions, the thickness of the frost-resistant fillings superstructure should be at least 1.15 m thick so the expected frost heave is 50 to 100 mm. Open ditches are designed to be excavated on both sides of the road, and surface runoff from uphill areas is conveyed past the road via culverts. A preliminary typical cross-section of ACR is presented in Figure 19-2 below.

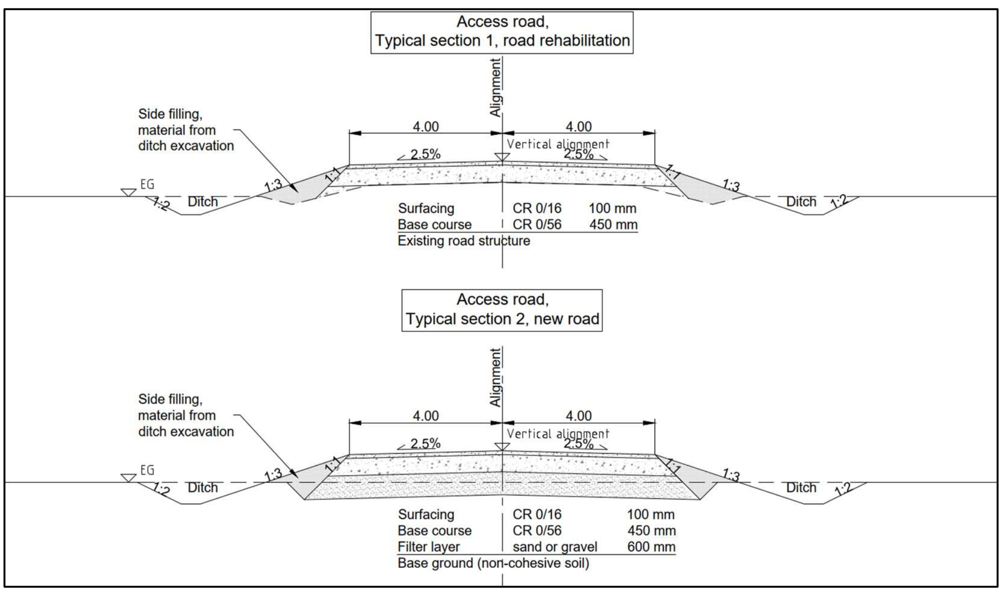


Figure 19-2. Preliminary typical cross-section of ACR

The filter layer can be constructed using local sand or gravel which is likely to be suitable for the purpose. In the case of the existing road section, the existing structure is considered to fit the purpose of a filter layer. Subsoil bearing capacity and frost susceptibility have been preliminarily estimated based on the estimation that subsoil is sandy till, sand or gravel. According to soil maps by the Geological Survey of Finland (GTK), the base ground consists of sand and/or gravel almost throughout the area. There are, however, no specific investigations at this phase to support the information. In peat or soft soil areas, settlements are prevented by mass exchange so that compressible layers are removed and the filter layer is extended to the firm subsoil. To ensure adequate drainage conditions for the road, the bottom of the roadside ditches shall be reached at least 0.25 m below the road superstructure.

In addition to ACR, the Site includes around 550 m of haulage roads (i.e. dumper roads). Haulage roads are constructed mainly between ROM pad – the mine portal. The structure and width of the haulage road are similar to ACR. The surface of all roads is designed to be crushed rock.

The typical design for the access road applies to service roads as well. However, the base course of service roads may be constructed thinner due to lighter traffic. The roads with crushed rock surfacing can be levelled during LOM with thin layers of crushed rock.

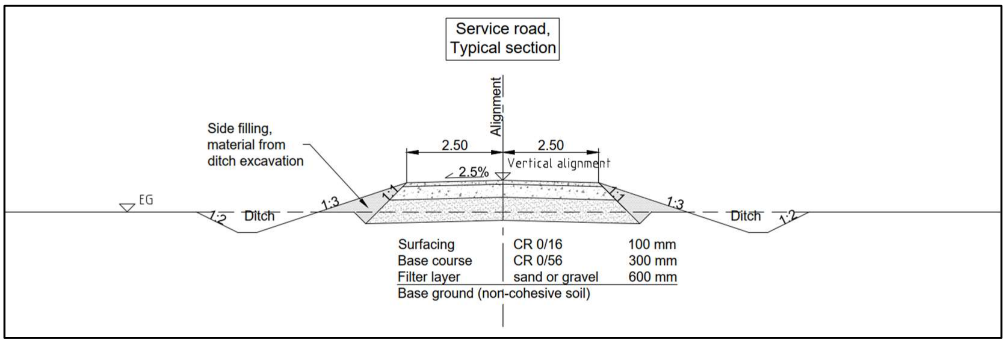


Figure 19-3. Preliminary typical cross-section of SR

The layer thicknesses of roads are subject to revision after gaining more information about the site's geotechnical conditions.

19.3 Processing Plant Area

19.3.1 Location and conditions

The processing area is preliminarily designed to be located between the existing tailings pond and lake Suu-Särki. The elevations of the processing area vary between +114.7–+124.8 in height system N2000.

According to the soil map by GTK, the base ground consists of filling and the surrounding area is mainly sand or gravel (Figure 19-4). The processing area is fully located in an area, where fillings have been made at some stage during old mining operations.

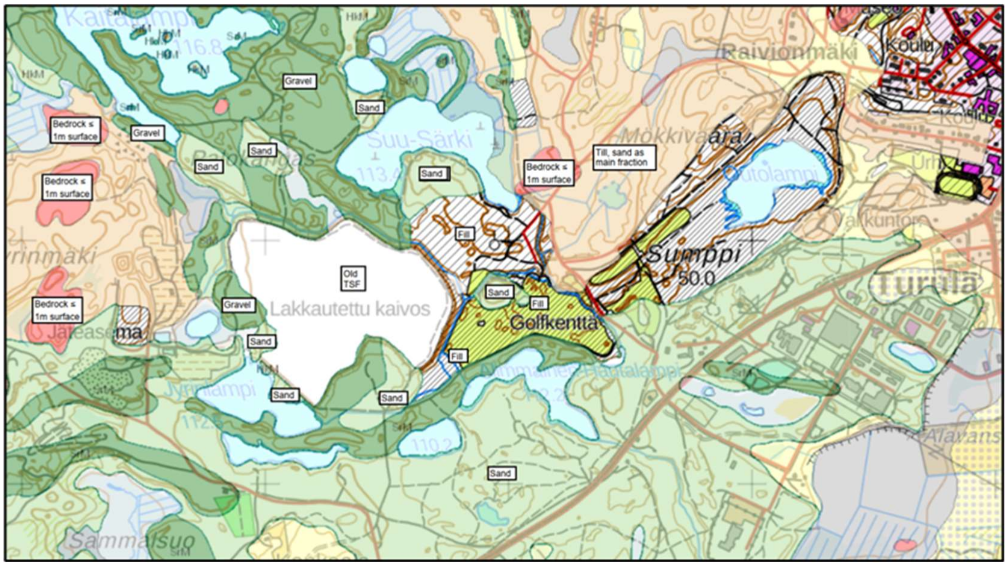


Figure 19-4. GTK soil map.

In addition, there are some investigations done at the processing plant area by GTK in 2008. The investigation consists of geotechnical and geochemical tests as follows:

- Exploratory drillings and soil sampling, 5 points
- Standpipe for groundwater table, 1 point
- Particle size distribution (PSD), 3 samples
- Geochemical analyses (ICP-AES), 12 samples

Based on the results, soil layers are generally as follows:

- Fillings generated during old mining operations, thickness generally 1.0–2.6 m
    - mainly gravelly sand, mine waste and/or soil containing organic material (peat or muck)
    - approx. 4 meters thick tailings layer on an area of the southernmost investigation point
- natural silty sand – sand layers
- till with sand as the main fraction
- very dense and/or rocky till and deeper bedrock

The exploratory drillings have terminated to boulders or bedrock at depths of 5.2–19.5 m. Based on investigations, very dense soil or bedrock is as highest approx. in the middle of the filling area (and designed processing area). The elevation of the bedrock surface has not been verified by drillings.

The aim is to keep the final surface of the whole processing area as flat as possible, i.e. floor levels of main buildings should be the same. The difference in building floor levels generally results in difficulties in construction and especially in possible later expendability.

The buildings are designed to be underdrained and additionally, an open ditch should be excavated on the sides of the whole area to cut off surface runoff and seepage.

19.3.2 Building Foundations and Earthworks

Building foundations must be designed to withstand the local ground frost and reach well below frost penetration depth. A 50-year cold content recurrence interval is used as a dimensioning value, which is the state-mandated design basis for buildings in Finland. The non-frost susceptible foundation depth of cold structures is on average 2.7 m in the Outokumpu area when excluding the protective effect of snow cover and the base ground is coarse soil (e.g. sandy till, gravel or sand). Foundation depth can be lower than 2.7 m if the bearing capacity is sufficient but that would result in the use of separate insulation materials. Decreasing the foundation depth by using insulation materials (such as XPS) is not considered to be feasible taking into account later expandability, i.e. later excavations would have to be made as supported in the case of shallow foundations.

Based on current knowledge, the buildings can be founded on ground-supported footings onto base ground or bedrock by using at least 0.5 m thick fill of compacted crushed rock layer below foundations. Filter cloth (class N3) will be installed between the ground layer and the crushed rock layer. The floor level and yard area levelling are more thoroughly designed in the upcoming design phases. There is a possibility to reduce the cuttings by raising the floor level especially if the floor levels (and thus foundation levels) of the buildings are raised individually when moving downhill to uphill however, the expandability needs to be taken into account. Piling is considered unnecessary as the base ground is assumed to be non-cohesive soil thus consolidation settlements are relatively small and those will mainly occur during the construction of fillings. Soil conditions need to be verified in upcoming design phases as GTK’s investigations show relatively high fines content in some soil layers and no such investigation data is available, which could give information about drilling resistance. However, it is likely, that foundations can, in any case, be made as ground-supported and compressive layers are replaced with a mass exchange.

Buildings shall be underdrained using pipes and wells which are discharged to the surrounding environment.

19.3.3Yard Areas

Only the parking area next to the office is designed as asphalt paved, other areas shall have crushed rock surfacing. Footprints of the yard areas are as follows:

- asphalt paved area 720 m²
- crushed rock surfacing 22 760 m²

Subsoil-bearing capacity and frost conditions constitute the main design constraints. Sand or gravel and crushed aggregate are selected as materials for the structures. The pavement structure shall be at least 1.15 m thick, thereof the filter layer thickness being 0.60 m. The design of the crushed rock surface area and the paved area includes the same structural thickness, allowing for future paving of the crushed rock surface area. The structure thickness should be dimensioned conservatively due to the unknown quality of the subsoil. The expected frost heave of the structure thickness mentioned above is less than 100 mm.

If a supporting layer of crushed rock or blasted rock is used instead of a filter layer of sand or gravel, the total thickness of the structure is at least 1.30 meter as insulation properties of coarser material is inferior to finer material.

The processing plant area, as well as the whole mine area, has been considered not to require any fencing.

19.3.4 Plant Area Surface Water Management

The processing plant area drainage system consists of ditches and a storm water drainage system (i.e. storm water piping and wells). As a majority of the yard area is not paved, an underground drainage system is recommended. Underdrainage is constructed at least for buildings. Yard underground drainage is considered unessential but shall be verified in upcoming design phases based on e.g. field investigations.

Storm water wells will be placed onto depressions on a levelled surface. Water from the storm water network will be discharged into perimeter ditches which surround the entire area. From the perimeter ditches, the water will be collected into the stormwater settling pond.

Generally, the storm water management system is designed to withstand the rainfall intensity for an event with a two (2) year return period and a 10-minute time of concentration (typically 150 l/s/ha).

In addition, the site area storm water management network including collection points and depressions in asphalt-paved areas shall be designed for operation in a fire emergency by providing sufficient capacity to collect fire water spilling from the buildings and serve as water storage. The storm water system will be designed to enable a shut-down with manual valves placed in downstream pipes to prevent fire water from leaking into perimeter ditches.

Oil separation systems are designed for places with fuel or oil handling, e.g. fuel stations.

19.3.5 ROM Pad

The ROM pad is located in the southern part of the process area. The area is considered to be an outdoor storage field with structures for environmental protection. The structure consists of crushed rock with various grain sizes and includes an HDPE liner at the bottom as a sealing layer.

At the outer edges, the liner is either raised upwards inside embankments or a ditch is formed and the liner is anchored into the embankment outside the ditch. Runoff is collected and diverted to the ROM pad buffer basin.

The designed total area of the ROM pad is 14 200 m^2

19.4 Site Surface Water Management

19.4.1 Ditches

Surface water management is planned to be executed with clean water cut-off ditches which divert surface runoff to lower ground. The ditches are placed on the uphill side of each major layout component in case the catchment area, sloping inwards, can be reduced. The ditches have bottom width of at least 0.5 m and a longitudinal gradient of 0.1 %.

At this project phase, the ditches are not presented on a map separately.

19.4.2 Settling Basins

Clarification basins settle solid particles in the water before discharging to the environment or leading the water to successive ponds or process units. Clarification is a common passive water treatment method used in the mining industry. In addition to settling basins and buffer basins mentioned in the preceding chapters, the surface runoff and seepage inside waste deposition areas need to be collected and clarified. The water collection and settling can be done either inside the waste deposition areas or outside in separate basins. As the deposition quantities and deposition arrangements are somewhat uncertain at this stage, separate settling basins are reserved on the downstream side of deposition areas.

20 Environmental

20.1 Environmental and Social Setting

20.1.1 Introduction

The project area and underground mine are located in a connection with the old Keretti underground mine and the immediate vicinity of the town of Outokumpu. The old mining area is partially forested, and the area currently houses a golf course. The Jyri waste station is located west of the mining area. There are also other industrial players in the vicinity of the area, such as Outotec and GTK's pilot plant.

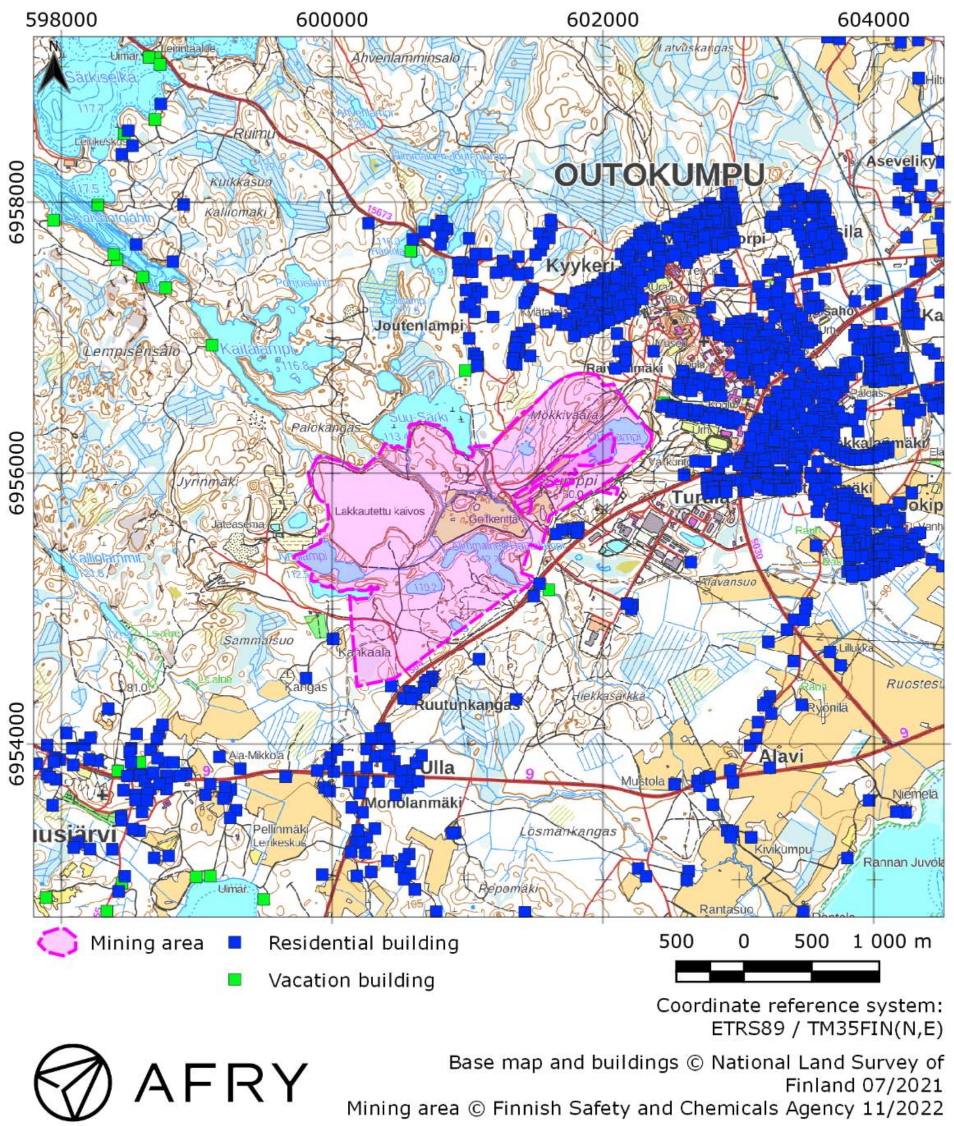


Figure 20-1. Residential and Vacation Buildings.

20.1.2 Catchment Area

The mining area is located on the Sysmänjoki catchment area (catchment code 04.353) which is a part of the Vuoksi major catchment area (04). The waterbodies inside the project area are small lakes, ditches and streams. The surface waters of the mining area discharge through river Ruutunjoki (04.353_002) into the lake Sysmäjärvi (04.353.1.019_001), from where through river Sysmänjoki (04.353_004) and river Taipaleenjoki (04.351_001) further to Heposelkä lake area in Orivesi (04.311.1.001_002).

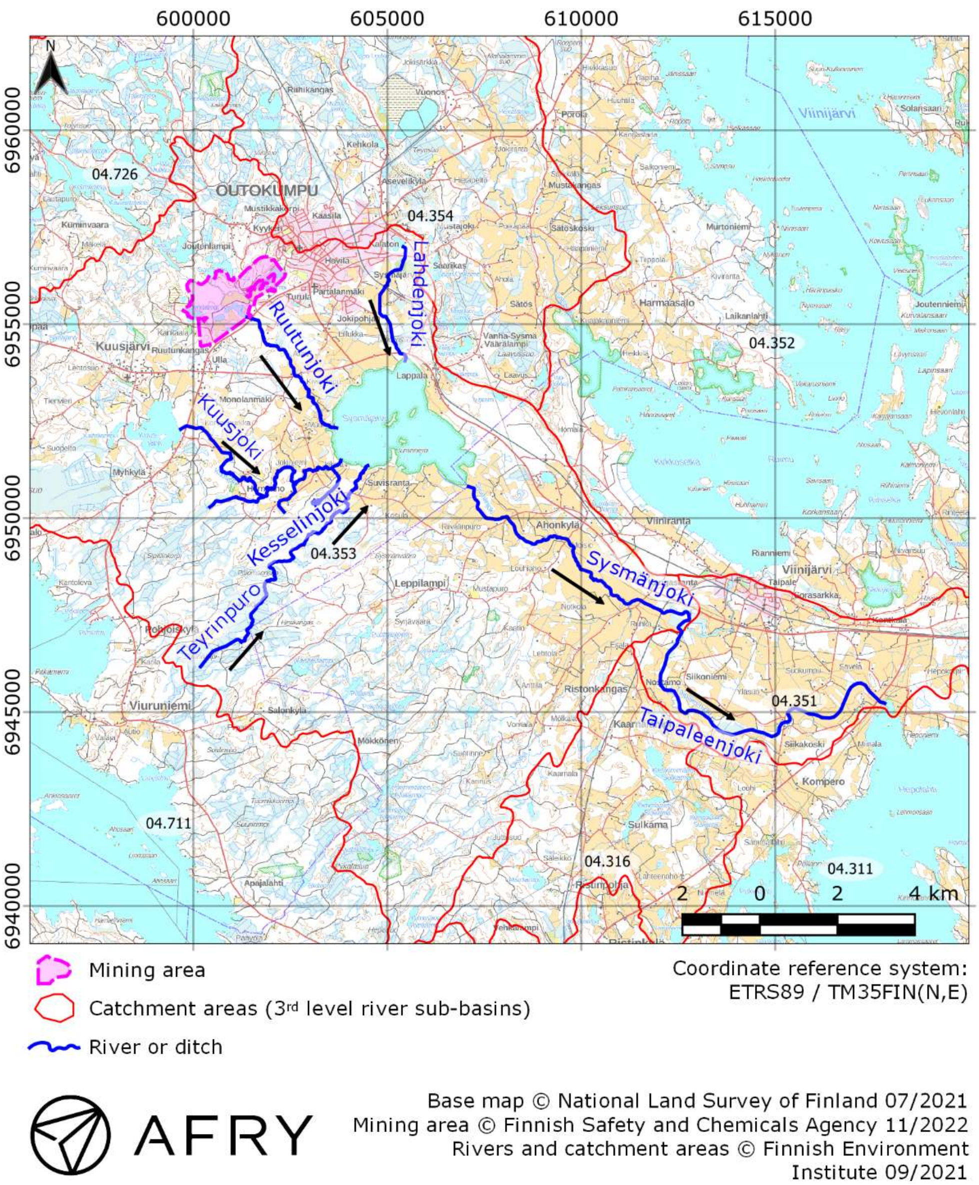


Figure 20-2 Catchment areas and closest rivers.

20.1.3 Surface Water Status and Recreational Values

All streams and ponds near the Project area are naturally humic due to peatlands in the catchment areas. The waters have a naturally brown color and elevated sulphate and metal concentrations due to former mining operations in the area. Lakes and ponds in the Project area are mainly small and shallow. Lake Alimmainen Hautalampi cannot be considered a natural waterbody due to former mining operations, because it has been used as a final sedimentation pond.

Heavy metal concentrations are high, especially in Ruutunjoki, Lahdenjoki, Sysmänjärvi and Sysmänjoki waterbodies, where the limit concentration set for nickel is occasionally exceeded (Environmental Quality Standards, EQS). Part of the load is due to contaminated groundwaters. Lake Sysmänjärvi has been receiving a considerable load for several years, including load from both mining and the community. The impact of residential wastewater and agriculture can be seen in elevated nutrient levels and the mining water effect in the elevated electrical conductivity, as well as elevated sulphate and metal concentrations. Arsenic, nickel, copper and zinc have accumulated in the bottom sediment of the lake, due to the mining industry influence. The Regional State Administrative Agency for Eastern Finland has granted a permit for the mixing zone in part of lake Sysmänjärvi. Article 4 of the WFD allows for the application of mixing zones which can be designated adjacent to discharge points. Within the mixing zones the concentration of one or more of the priority substances listed in Part A of Annex I, may exceed the relevant EQS values. But the rest of the water body should comply with EQS standards and not be affected by mixing zones (WFD, 2008/105/EC). In Sysmänjärvi, the mixing zone is set for nickel.

Some of the rivers and lakes near the Project area are classified ecologically according to the Degree of Water Resources Management (1040/2006)(Figure 20-3). Based on the EU Water Framework Directive, all classified surface waters should provide a good or even high ecological status by the end of the year 2027. After the decision of the European Court of Justice on the Weser Case (C-461/13), EU member states are required to refuse environmental permits for any project that may cause deterioration of the status of a water body or jeopardise the attainment of its status objectives. According to the decision, this covers not only the overall water quality status but also individual quality elements (Table 20-1). The overall ecological status classification for a water body is determined, according to the "one out, all out" principle, by the element with the worst status out of all the biological and supporting quality elements.

Watercourses near the project site are classified as poor (river Ruutunjoki) or moderate (lake Sysmänjärvi, river Lahdenjoki, Sysmänjoki, Taipaleenjoki) ecological condition according to EU:s Water Framework Directive (WFD) classification system. There are small streams, lakes and ponds on the project site that have no ecological classification due to their small size. Lake

Sysmänjärvi is also a Natura 2000-area with high natural value. The lake is considered as valuable bird area and it is also listed as an internationally value wetland area (RAMSAR site).

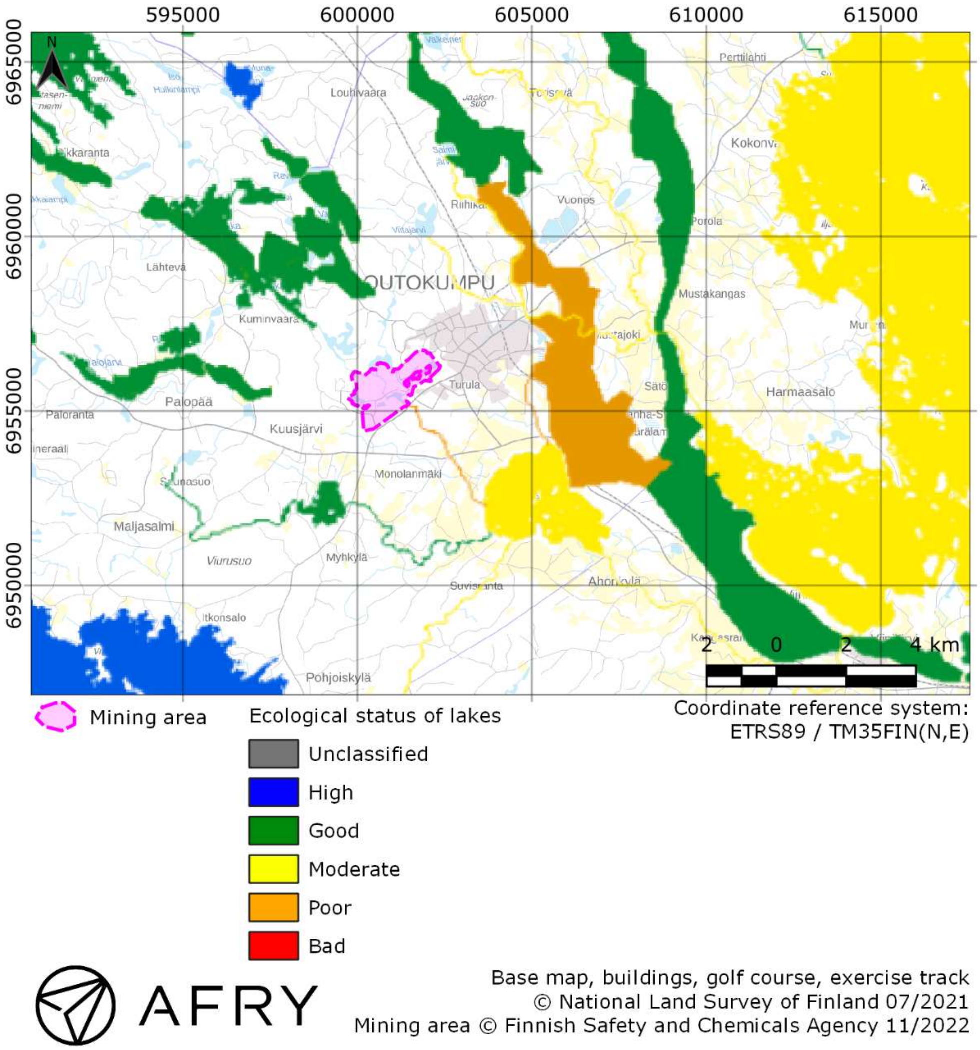


Figure 20-3 Ecological classification of lakes and rivers.

Table 20-1 The status of a water body is determined based on several quality elements.

Overall status of a waterbody	Ecological status	Biological conditions Phytoplankton Aquatic plants and diatoms Fish Benthic fauna
		Physical-chemical conditions
		Hydrological and morphological conditions
	Chemical status	Harmful and dangerous substances

In addition, planning procedures for river basin management are part of the Act on Water Resources Management (1299/2004) and stipulate the required actions to reach a good/high status for surface waters. An environmental permit is only granted when the impacts on the water are compliant with the river basin management plan.

According to available information, the project area and its vicinity waterbodies are not in recreational use (swimming, fishing etc.) due to former mining operations and impacts excluding Lake Sysmäjärvi, where recreational fishing (especially pike) is quite common. Lake Sysmäjärvi is also a popular birdwatching area. River Taipaleenjoki is used for recreational fishing (trout, rainbow trout).

20.1.4 Soil (quaternary geology)

Topography varies between 110 – 150 m above sea level and has a flat relief. Overburden landforms consist of glacial features such as drumlins and moraines. Topography slopes towards the SW and NE from the area. The northwest side of Kaitalammi is characterized by hilly terrain, with outcropping bedrock, and sandy soil types.

The quaternary map indicates outcropping areas towards Outokumpu town, and till cover in the area east of Lake Suu-Särki. South of Suu-Särki are found sandy and gravelly till. Geological Survey of Finland GTK (Saarelainen 2008) has studied the area between the past tailings area, golf course and Suu-Särki with five percussion boreholes, in which one observation well was installed.

Quaternary geology consists of a sandy formation belonging to an esker. The core of esker is located in the west part of the concession. Low-lying areas indicate peat thicknesses of 2.2 – 2.6 m deposited on the mire when in its natural state. Soil thickness varies from 5 – 20 m.

Soil thicknesses vary, but soil cover above the bedrock surface can be several tens of meters thick. Intensive exploratory drilling can be used to indicate the soil thickness at the deposit. Elsewhere information is more limited.

Original natural layering is overlain with landfill and tailings storage layers, with varying thicknesses (1 – 4 m at the area between the golf course and Lake Suu-Särki). Earlier glacial kettle holes and ponds have been filled with tailings.

A 2.5 km long area traversing between Outokumpu town in the NW and Jyrinlampi in SW, is modified with a 0.5 km wide landfill area, consisting of soil waste and mining waste. NW of this is found till soil type and to SE and S sandy area.

Grain size analysis shows the prevalent soil types consisting of silty or fine sands to gravel. Below the sands, a till layer is found above the bedrock surface.

Soil chemical analysis indicated elevated element contents (Co, Cu, Ni, Zn), reflecting the character of past mining activities.

20.1.5 Bedrock, hydrogeology and hydrology

The surface is modified from the natural stage and surface waters are diverted due to construction and road connections. Landuse is a combination of a disused mine site, forestry, farming and urban setting.

Surface water areas contain Kaitalampi (116.8 m a.s.l) and Suu-Särki (113.4 m a s l). lakes NW of the concession upstream of the area. Originally Kaitalampi drained to Lake Jyrki (112.5 m a.s.l), which had further drainage to Alimmainen Hautalampi (112.2 m). Lake Jyrki was mostly filled with tailings, and the remaining Paavonlahti bay forms a pond Jyrinlampi. Currently, surface waters from Kaitalampi are diverted from the N and NE sides of the tailings area. Suu-Särki discharges also to the Alimmainen Hautalampi.

Alimmainen Hautalampi discharges via Ruutunjoki river to Sysmäjärvi Lake (85.6 m).

Smaller lakes, Ylimmäinen Hautalampi (111.5 m), Keskimmäinen Hautalampi (110.2 m) and Outolampi reside within the concession area. Outolampi is located on a Sumppi landfill.

Suu-Särki serves as a freshwater abstraction source, with a designed maximum intake of 53 m3/h. Mine dewatering water can be used as a raw water source.

In two GTK percussion boreholes, the groundwater table was 1.5 – 2.6 m below ground level. The groundwater table after the installation of the observation well was 6 m below the ground surface. Grain size measurements were as a basis on the estimate of horizontal soil hydraulic conductivity (1E-5…1E-3 m/s in sand, 1E-7…1E-5 m/s in silty to fine sands). Soils are layered with alternating fine and more coarse interlayers.

Regional area lithology consists of Paleoproterozoic metasedimentary biotite paraschist (metagreywacke).

The deposit is well characterized, with hundreds of bedrock drilling (tens of kilometers) along the strike of the deposit. The drill hole collar data can be used in describing the soil thickness. Apart from exploratory information, core logging contains geotechnical data which is available for assessment of fracturing and hydrogeological character.

The deposit is located in NE trending ca. 2 km wide horizon of black schist and serpentinite, defining the western margin of the Outokumpu belt. The regional structure is a tightly folded serpentinite, > 10 km long tubular body, with thickness and elevation of 1.2 – 1.5 km at its thickest part. Serpentinite is enveloped with carbonate-skarn-quartz.

The main faulting can be expected to be trending NE parallel to the lithological setting. A regional thrust fault is interpreted on the footwall side of the formation.

The deposit has a moderate to gentle dip towards the SE. Ore material has often brecciated the shear-banded wall rock. Intersecting NW-SE fault directions are apparent in topographic trends, and visible in airborne geophysical data sets.

Some declines are already present, with 1200 m of decline and 850 m of drift. The Co-Ni-rich mineralised zone has not been under production.

The main deposit is 70 – 120 m below the ground surface, and the deepest known parts are 150 m below the surface.

In the upper part of the decline, there is a significant water inflow into the tunnel, which is planned to be injected tight during decline maintenance. Rehabilitation of the existing mine would include lowering the groundwater table in the Keretti shaft.

Stopes will be designed according to the shape of the orebody. The main body is 1000 m long, 100-150 m high and 1 – 30 m wide. Stope design takes into account rock mass conditions.

The estimated mine dewatering rate is 475 000 m^3/a (54.2 m^3/h) continuously all year round (chapter 15.11.6). The estimate is provided by FinnCobalt, and it is based on the tonnage to be excavated (the mining volume). Mine dewatering water is pumped to Tailings Basin A. Mine dewatering water is the main water source for the concentration plant.

The bedrock fracturing and bedrock hydraulic conductivity are not known. With the designed decline, shaft, drift and stope geometries it would be possible to make a revised estimate of the seepage rate into the mine.

Mine dewatering will cause a lowered hydraulic head near the tunnels, which in turn will cause a slow movement of groundwater in bedrock towards the excavated space. Due to the underground design and thick overburden, it is unlikely that any significant surface water drawdown would occur. Seepage groundwater has the potential to carry leached elements from past tailings storage towards the underground mine, which may affect the water quality (with respect to discharge, need for treatment, and usage as process raw water).

20.1.6 Groundwater quality and classified groundwater areas

There are no classified groundwater areas immediacy of the project site. The classified groundwater area of Valkeisensärkkä is located approximately 2.2 km northwest of the mine site, the classified groundwater area of Niilonpatama is located approximately 3.2 km west of the mine site and the classified groundwater area of Saari-Oskamo is located approximately 4.4 km east from the mine site.

The groundwater in the project area has already been polluted during previous mining operations. Groundwater contains high levels of sulphate, manganese and iron. Groundwater is also acidic in many monitoring points. The use of groundwater in the area is prohibited.

20.1.7 Protected Areas

There are no Natura 2000 sites or nature protection areas near the mining area (Figure 20-4). The nearest areas are two small private land conservation areas, both just over a kilometer to the west of the mining area (YSA238855 Lemmenkosken luonnonsuojelualue, Suomi100; YSA207124 Vilhonpuron luonnonsuojelualue).

The nearest Natura site is Sysmäjärvi (FI0700001, SPA), located about three kilometers southeast. The next closest Natura site (FI0700083 Iso-Juurikan - Leveävaaran alue, SAC) is about 5.5 kilometers southeast.

The Sysmäjärvi Natura site is protected as several private and state-owned conservation areas (YSA201164, YSA200710, LTA202187, LTA202186, YSA201165, ESA301784). Sysmäjärvi also belongs to the Waterfowl Habitats Conservation Programme (LVO070167) and the List of Wetlands of International Importance (the RAMSAR sites). In addition, Sysmäjärvi belongs to Important Bird and Biodiversity areas (IBA, 52 Outokummun lintuvedet), Finnish Important Bird Areas (FINIBA, 570376 Sysmäjärvi-Viinijärvi) and Provincially Important Bird Areas (MAALI, 570421 Sysmäjärvi ja Alavin pellot). Near Lake Sysmäjärvi, about 2.1 kilometers southeast of the mining area is also FINIBA and MAALI area (570381 Viinijärven - Outokummun pellot). (BirdLife Suomi ry 2021).

There is another large IBA area in the vicinity of the mining area. The nearest sub-area of IBA and FINIBA area (53, 570364 Outokummun-Kaavin seudun oligotrofiset järvet) is located about 200 meters northwest of the mining area. Within a five-kilometer radius, there is yet one more MAALI-area 4.2 kilometers northeast (570420 Vuonos). (BirdLife Suomi ry 2021).

There are no areas in the supplementary proposal for the protection of mires near the mining area.

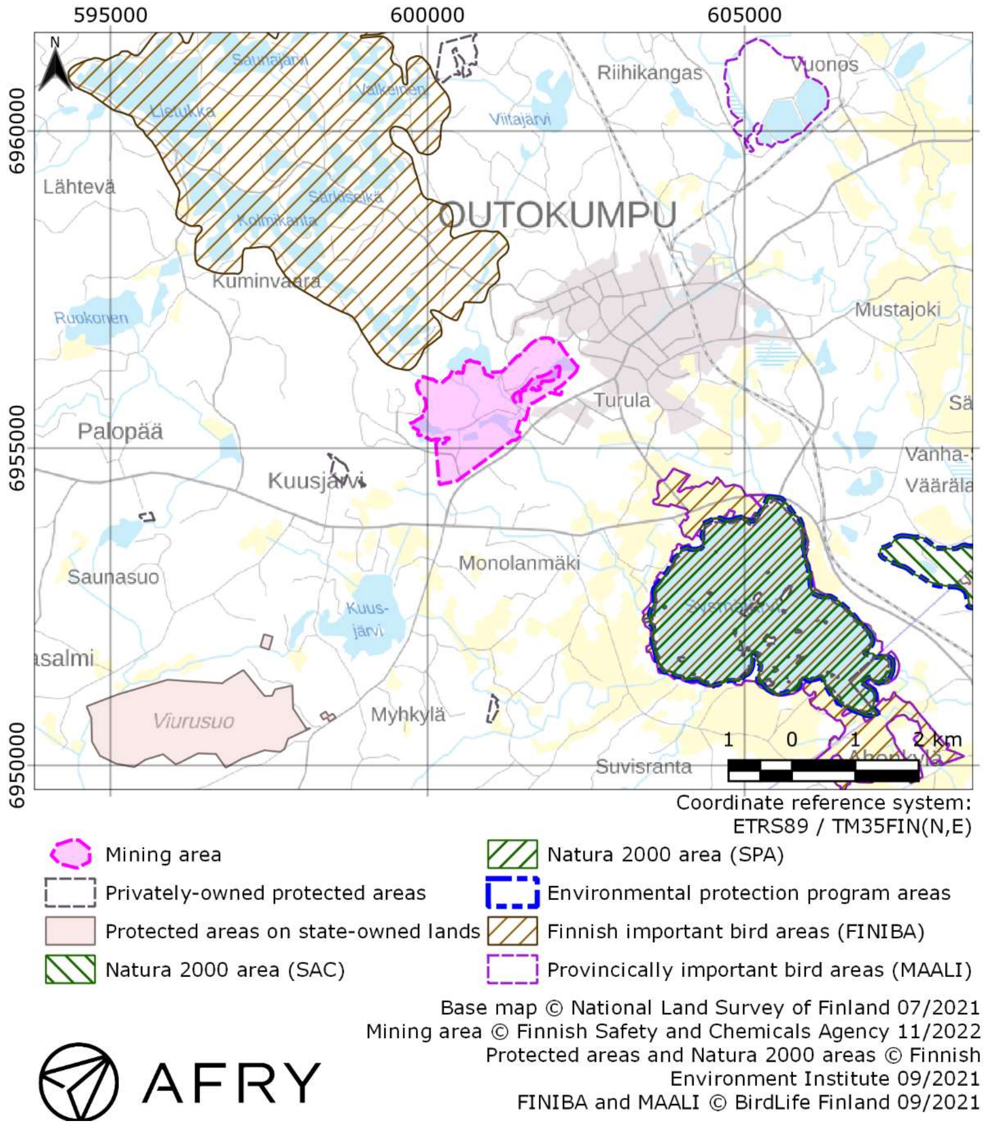


Figure 20-4 Natura 2000 sites and nature protection areas.

20.1.8 Flora and Fauna

The vegetation and avifauna of the Hautalampi area has been mapped in 2006 by Lapin Vesitutkimus Oy. In connection with the ongoing EIA assessment, the following mappings are performed in the impact area: vegetation and habitats, avifauna, moor frogs and the habitats of Ruutunjoki. The results of these mappings are not yet available. In addition, the North Karelia ELY Center has carried out bird studies in the area in 2020.

The project site is situated in the southern boreal forest vegetation zone. In grounds of maps and aerial photographs, there are forests, lawns and ponds in the area. According to the mappings conducted in 2006, the mining area has originally been dry forest land. The nature of the area is however entirely altered by previous human activity. During the 2006 survey, a significant part of the survey area was previously a sand extraction area. The old sandpit area had become forested and swampy. In the middle of the area, the historical sandpits were affected by groundwater. As a result of the groundwater impact, the vegetation was rich and exceptional (Lapin Vesitutkimus Oy 2006, according to Envineer Oy 2020).

According to 2006 surveys, no valuable habitats were found in the area. There were no protected habitat types according to the Nature Conservation Act (4:29) §. The Finnish Forest Centre (Suomen metsäkeskus) has not marked any sites protected according to the Forest Act (3:10) § in the mining area, the closest sites are located over 400 meters outside the area (Suomen metsäkeskus 2021). Forest Act sites were not observed in 2006 surveys either (Envineer Oy 2020).

According to the Water Act (2:11) § it is prohibited to endanger the natural state of a naturally occurring spring, small rivulet or a pond smaller than one hectare without a separate exception permit. In addition, brooks are protected according to the Water Act (3:2) §. Based on the map material there are no springs, small rivulets or brooks in the area. However, there is one small pond located in the southern part of the area.

In the database of endangered species, there are no observations of endangered plant species occurrences in the mining area (Suomen Lajitietokeskus 2021, situation 22.9.2021). Endangered plant species were not observed in 2006 mappings either (Envineer Oy 2020).

In the 2006 bird survey, 15 breeding bird species were recorded in the Hautalampi area. The species observed were quite common for dry pine forests and scrub areas. The most noteworthy species were the waders nesting in the marshy sandpit area in the middle of the area. The bird species observed included some noteworthy species from the point of view of nature protection. No species listed in EU legislation in accordance with Annex I of the Birds Directive were found (Envineer Oy 2020). The data on the avifauna of the area will be updated in the EIA assessment. However, it is noteworthy that important bird areas exist in the vicinity of the mining area.

The Siberian flying squirrel, moor frog, bats, otter, large predators and several insects are listed in Annex IV (a) of the Habitats Directive. No resting and breeding sites of these strictly protected species were found in the study area in 2006 mappings (Envineer Oy 2020). There are no known wolf territories in the area (Heikkinen et al. 2021).

20.1.8.1.1 Aquatic fauna

Aquatic fauna (benthic macroinvertebrates) and fisheries monitoring have been conducted in the Project area and its vicinity due to the mandatory environmental monitoring of areas mining operators (FinnCobalt, Elementis Minerals) and the city of Outokumpu. Monitoring is focused on lake Sysmänjärvi, river Sysmänjoki and Taipaleenjoki. There are no aquatic fauna data from lake Suu-Särki, river Ruutunjoki or small ponds from the project area.

Common fish species in Sysmänjärvi are pike, pikeperch, perch and cyprinid fish (roach, bream). Fish population studies conducted with Nordic-gillnets indicate eutrophicated lake which is also a result of benthic macroinvertebrates study. According to the studies, aquatic fauna is not affected by elevated metal concentrations. Metal concentrations (Ni, As, Zn) in fish muscle tissue are not elevated. River Sysmänjoki fish population includes perch, pike and roach. In the river Taipaleenjoki (Figure 20-2) probably a partly natural breeding population of brown trout has been identified. Trout has been assessed as an endangered (EN) species in Finland. In river Taipaleenjoki fish population includes also stone loach, bullhead, roach, rainbow trout and perch. Siikakoski rapids in river Taipaleenjoki are assigned as a special fishing area.

20.1.9 Landscape and Cultural Environment

There are no nationally valuable landscape areas in the vicinity of the mining area. The nearest ancient relic is located about 400 m south of the Hautalampi mining area (Figure 20-5). The old mining area of Outokumpu and the closed mine tower of Keretti are nationally significant protected cultural heritage sites. The Keretti tower is protected by the Building Protection Act.

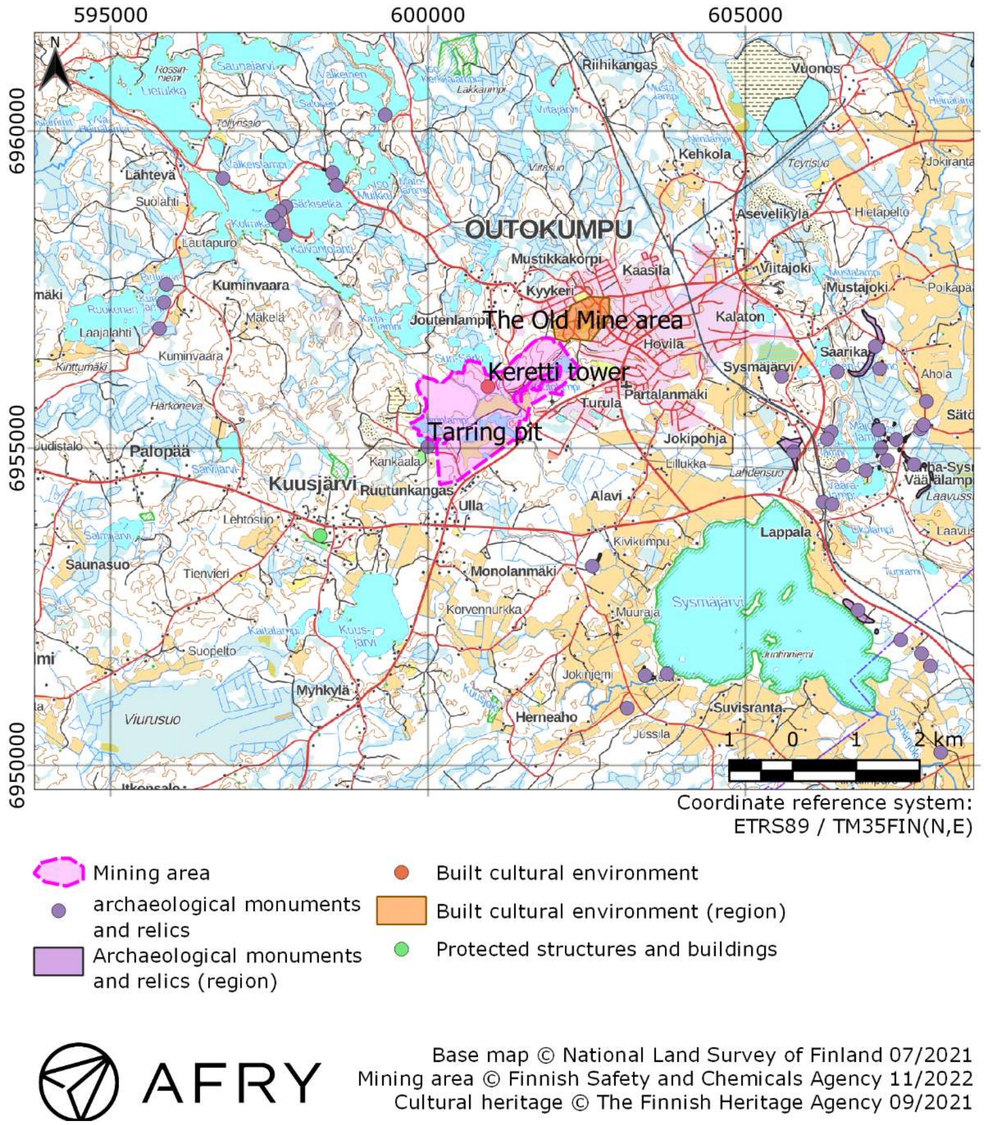


Figure 20-5. Ancient relics, built cultural environment and protected cultural heritage.

20.2 Legislation and Regulation

20.2.1 Finnish Permitting Process and Relevant Authorities'

An ESIA (environmental and social impact assessment) procedure for mining projects in Finland is required before the permitting. The ESIA procedure is based on national legislation, the EIA Act (252/2017) and the EIA Decree (277/2017). The procedure includes the ESIA program stage, where the project alternatives are described, and the actual ESIA, where the project alternatives are assessed. The purpose of the ESIA procedure is to assess the environmental and social impact but also to share information and add interaction with different stakeholders. An ESIA does not lead to a permit decision, but a finished and approved ESIA is needed for several processes like land use planning, mining permit and environmental permit. The coordination authority for the ESIA procedure is ELY (Centre for Economic Development, Transport, and the Environment). EIA-program has been submitted to North Karelia ELY Center in December 2020. According to current information, the EIA report will be submitted in 2022.

In Finland, environmental permits are required for all activities involving the risk of pollution of air and water or contamination of soil. Environmental permit applications must be submitted to the relevant authority AVI (Regional Administrative Agency), as defined in the Environmental Protection Act (527/2014) and Decree (713/2014). One important condition for a permit is the use of the Best Available Techniques (BAT). Most of the Best Available Techniques reference documents are directed to the IE-directive (Industrial Emissions Directive7) plants. According to directive annex 1, the mining industry is not included in the IE directive. For the Hautalampi project, the following document must be taken into account in future planning: BATs Reference Document for the Management of Waste from Extractive Industries (2018).

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

Water permits are required when the planned operation or activity may alter the position of the groundwater table or the groundwater quality. Also, changes related to the water flow or shoreline are subject to authorisation. Permits according to the Water Act (587/2011) and the Water Decree (1560/2011) are applied with the environmental permit application.

Finn Nickel Oy received the environmental permit in 2009 (Dnro ISY/2008/Y/185) for the excavation of the Hautalampi mine, and a water permit for pumping the groundwater from the underground mine. The permit came into force in 2011 after appeals. The permit also includes closure measures and aftercare, water treatment and monitoring of the old Keretti mine. The old permit must be updated in its entirety: The permit allows excavation of ore for 250 000 t/a, and the excavation amount investigates in this study is 400 000 t/a. The existing permit doesn´t include a concentration plant.

Mineral rights, including decisions concerning mining permits, are regulated under the Mining Act (621/2011), and Tukes, the Finnish Safety and Chemicals Agency, are the responsible authorities. A mining permit is required before an environmental and water permit can be granted. In Finland, a mining permit covers the whole operation area (not just the deposit) and requires therefore quite advanced project plans. The project area has a valid mining permit K7802.

Building permits will be required according to the Building and Land-Use Act (132/1999). Additional deviation permits according to the Nature Protection Act (1096/1996) or Natura 2000 assessments may be required in case of the presence of sensitive species or habitats. Dam safety is regulated by the Dam Safety Act (2009/494); the building permit for a dam requires a statement from the dam safety authority KAIELY (Kainuu Centre for Economic Development, Transport, and the Environment), risk assessment and a dam safety monitoring plan.

Tukes also handles applications for the utilization and storage of industrial chemicals according to the Industrial Chemical Decree (59/1999), and it is the authority for chemical registrations, labelling, and packing according to the European REACH-decree (Registration, Evaluation, Authorisation and Restriction of Chemicals) and CLP-decree (Classification, Labelling and Packaging of substances and mixtures). The use and storage of explosives, lifting equipment, and electrical work require permits from Tukes.

Destruction and deterioration of breeding sites and resting places of species listed in Annex IV (a) of the EU Habitats Directive (92/43/EEC) is prohibited under Section 49 of the Nature Conservation Act. The Centre for Economic Development, Transport and the Environment (ELY-keskus) can grant an exemption for animal species listed in Annex IV (a) and plant species listed in Annex IV only under strictly defined conditions described in Article 16(1) of the Directive. In addition, there are species under a strict protection order by degree and plant species placed under a protection order by decree, that need a derogation from the protection provisions as well. Further studies have been carried out, but results were not available at the time of this report. From Annex IV (a) species, for example, bats and pond insects, can not yet be excluded (within the impact area of the project).

20.2.2 Liability

AFRY Finland Oy has not carried out an assessment or reviewed any existing assessment of liabilities concerning the previous mining. There is historical extractive waste on the site and groundwater has already been polluted in the past. Contamination inventory and risk assessment have been carried out as part of the national "KAJAK inventory", an environmental inventory program for historical mining areas. Anyhow, this type of inventory does not define if responsibilities will be transferred to a potential new operator and to what extent. New operation on the site is unlikely without disturbing historical extractive waste. Liability assessment requires environmental inventory, definition and impact assessment concerning future operations and legal expertise.

20.2.3 Land Use Planning

Due to the new Mining Act and Environmental Protection Act, the importance of land use planning procedures has increased considerably recently. For example, a mining permit cannot be granted in case the land use plan is inadequately defined regarding the area.

Land use and building are regulated by the Land Use and Building Act (132/1999) and Decree (895/1999). The land use planning system consists of the national land use guidelines and three planning levels: the regional land use plan, the local master plan and the local detailed plan (Figure 20-6). The principle of the land use planning system is moving down the hierarchy towards more specific plans.

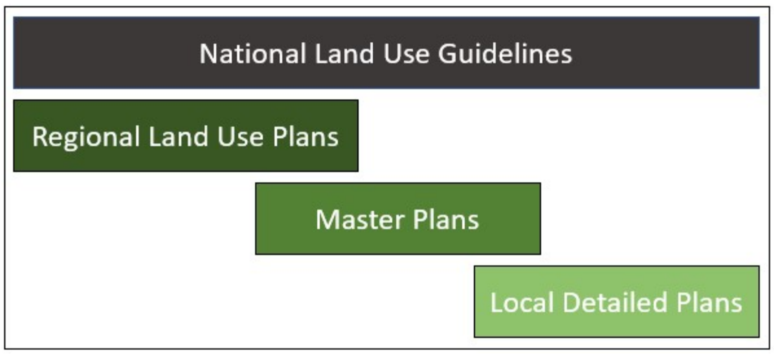


Figure 20-6. Land Use Plan Hierarchy as defined in Land Use and Building Act (132/1999) and Decree (895/1999).

Regional Land Use Plan

Finland is divided into 19 regions, each covered by a regional land use plan. The Hautalampi project area is located in North Karelia (Figure 20-7). The Council of North Karelia Region has approved the regional land use plan in 2020.

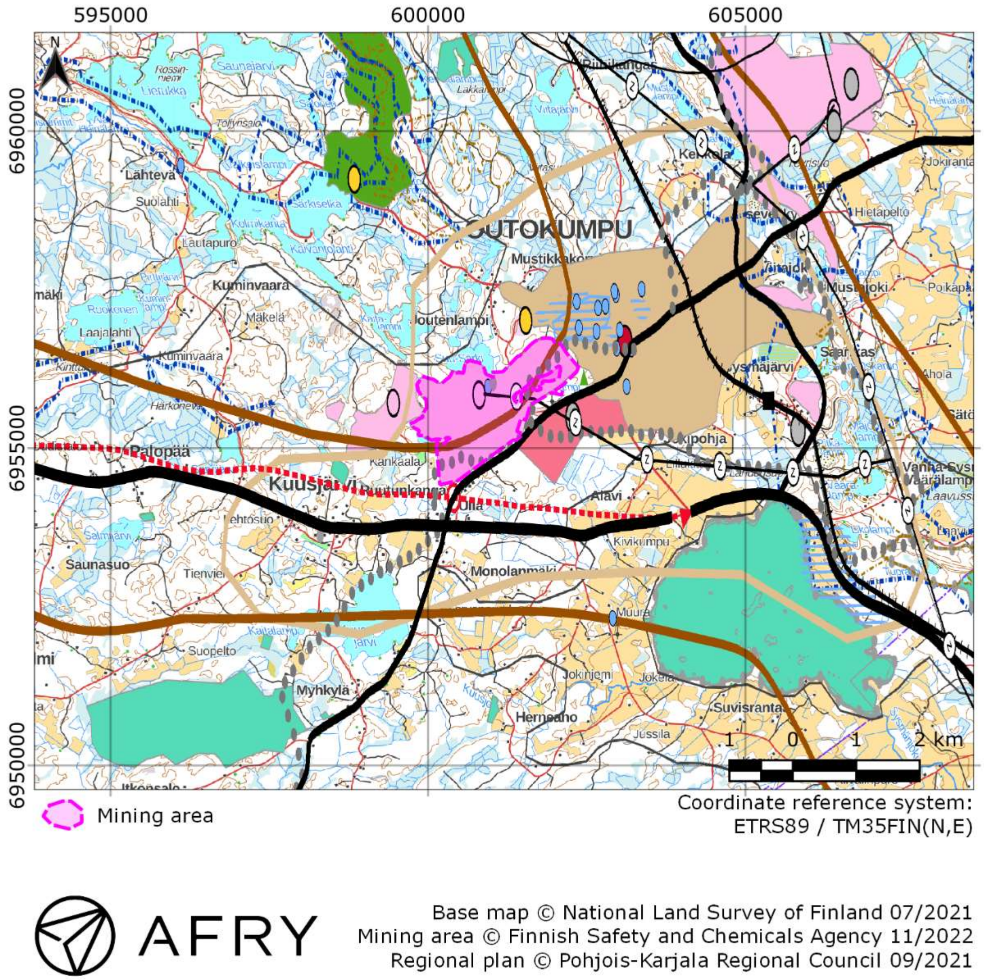


Figure 20-7 Regional Land Use Plan.

Master Plan

The purpose of a master plan is to provide general guidance regarding the urban structure and land use of a municipality and to combine and integrate all the functions.

The area has a legally valid master plan for the Joensuu region, where the area is allocated to mining.

Local Detailed Plan

A local detailed plan is the most detailed level of all land use plans. For example, it defines the location, size and purpose of buildings. The need for this land use plan shall be defined with respective authorities. On the project site, there is no local detailed plan. The need for a local detailed plan must be agreed upon with the municipality.

20.3 Environmental and Social Management Approach

Currently, Project has no actual Environmental and Social Management System. Currently, E&S professional capacity is partly outsourced. The environmental and social governance policies and E&S Management System should be developed and implemented as the project proceeds towards more advanced study phases.

20.4 Stakeholder Issues and Stakeholder Engagement

Stakeholder engagement is the process of involving people or parties who may be affected by the mining operations. Stakeholder engagement and participation are essential parts of the permitting procedure. Comprehensive management of the stakeholder issues and active communication are fundamental in promoting the acceptance of the project among the local residents and other interests and affected parties/groups.

It should be emphasised that the project potentially represents a major change to the stakeholders, particularly to the residents living near the mine. The stakeholders located in the vicinity of the mine are aware that the project will have a large impact on their properties. Currently and globally, stakeholder issues form one of the biggest risks for the mining industry.

Stakeholders impacted by the mine need to be regularly informed of the project and stakeholder concerns must be systematically documented and taken into consideration in the planning process. Currently, there have been various information releases, but there is no systematic Stakeholder Engagement Plan. It is recommended to create a detailed stakeholder engagement and a participation plan and establish a suitable grievance procedure.

20.5 Mine Closure

In Finland, mine closure requirements are largely based on European best-available technology (BAT) definitions (EC 2018). AFRY underlines the relevance of BAT 5 and post-closure risk and impact assessments – adequacy of a closure plan should be always verified with a post-closure impact assessment. Best closure planning practices are also generally iterative – repeating the same risk and impact assessments in several study phases (ICMM 2019). ICMM also underlines the social perspectives of mine closure (stakeholder expectations).

In this study, closure requirement considerations are preliminary. In other words, closure requirements are risk-based assumptions, not numerically assessed or confirmed solutions. The actual closure plan has not yet been drafted. The sensitivity and status of the receptor watercourses may set additional requirements for the closure measures. Stakeholder expectations for closure must also be known.

Assumed closure measures include the following:

- closing the mine entrance and other openings
- dismantling unnecessary infrastructure
    - no such dismantling is included in the cost estimate that cannot be funded with machinery sales and metal recycling
- profiling and covering the tailings storage facility
    - assumed cover requirement is double layer till cover, including 0.3 m sealing layer, minimum 0.8 cm protective layer (according to the regional frost penetration depth) and vegetation
- dismantling the basal structure of a temporary waste rock storage facility
- dismantling the structures related to the temporary high-S tailings storage
    - includes removal and recycling of liner and filter material, removal of dam core and crest, profiling and vegetation
- active water management and treatment for years 1-5 post-closure
- post-closure environmental monitoring
    - regular monitoring for early years after closure and following less frequent monitoring – until year 30 post-closure
    - repetitive monitoring includes site discharge, surface water and groundwater
    - one post-closure monitoring round is reserved for the flowing categories: river sediments, macroinvertebrates, diatoms, water moss, fishing enquiry, samples fishing, fish metal concentration, terrestrial moss and humus sampling, mushroom, berries, ants, needles, bird, moor frogs and fallout
- post-closure site maintenance (small-scale cover repairs, road-keeping for monitoring etc.)
    - minimalistic repairs and snow ploughing for 30 years

Details concerning closure measures and closure cost estimate are presented in the Cost estimation appendix. Aa a part of actual closure planning and definition of financial guarantees, the European Union guideline (2021) is recommended to be used as support.

Potential intervening blockings underground (to limit water flow between compartments) are not included in the cost estimate currently. Site landscaping to provide a basis for effective biodiversity development is not included in the cost estimate either. No closure measures are included for historical extractive waste – only the status of old tailings below the new tailings storage facility will benefit from the covering of the new facility.

20.6 Environmental and Social Risks

Nature and social risks related to mining projects in Finland include typically conflicts with the protection of species or habitats, watercourse impact issues, dust and air quality concerns, traffic and lasting physical impacts, and other noise and vibration issues. Also, land ownership impacts property value and recreational land use, and land access for various purposes can become conflict issues. Project risks may materialize as a significant impact on management costs, difficult technical solutions, or permitting problems.

The following preliminary project risk, as defined in Table 20-2, categories are considered especially important. The risk categories are mainly governed by Environmental Protection Act (527/2014) and Decree (713/2014), Nature Protection Act (1096/1996) and Extractive Waste Decree (190/2013). Risk prioritisation may change as the site-specific information increases.

Table 20-2. Environmental & Social Project Risks.

Risk category	**Risk description**	**Consequence type**
Terrestrial and aquatic nature	Results of the most recent field investigations were not available at the time of writing this report. The presence of Annex IV (a) species, for example, bats and pond insects, cannot yet be excluded (within the impact area of the project).	Destruction and deterioration of breeding sites and resting places of species listed in Annex IV (a) of the EU Habitats Directive (92/43/EEC) is prohibited under Section 49 of the Nature Conservation Act. The Centre for Economic Development, Transport and the Environment (ELY-keskus) can grant an exemption. Project risk type is permitting risk or schedule.
Mine Dewatering	Mine dewatering water is led to the tailings pond, according to the current plans. Generally, it is not recommended to mix water from different sources. These waters may require different treatments.	Water treatment may become more difficult and more expensive. The quality of treated water may be compromised which leads to increased environmental load. Mine water often requires oil separation, this is difficult to arrange if mine water is pumped to TSF. From this perspective changes in water management may still be needed. The project risk type is primarily cost risk.

Water quality	Discharge water quality is known only at a conceptual level (qualitative water quality definition). Quantitative/numerical water quality estimations for waste facilities and underground mine, load calculations and surface water impact modellings are still to be done. Uncertainties in water quality mean uncertainties in water management and water treatment requirements. Both metal contamination risk and eutrofication risk are included in the water quality risks.	Other water treatment methods may be required than currently estimated, (for example nitrogen, phosphorus or sulphate removal). If a load is unbearable for the nearest receptor, a discharge pipeline may be required to a larger watershed. Potential project risk types are permitting risk, risk of delay and cost risks.
Xanthates	In the concentration process xanthates, methyl isobutyl carbinol (MIBC) and sodium isopropyl xanthate (SIPX) are used.	Xanthate degradation needs to be studied more closely in further design phases to ensure sufficient retention time.
Water bodies	Based on the EU Water Framework Directive, all classified surface waters should provide a good ecological status by the end of the year 2027. Watercourses near the project site are classified as poor or moderate. Ruutunjoki river is classified as poor and Sysmäjärvi Lake as moderate. It has not been studied whether the water bodies can withstand the additional load. If they cannot, especially effective water treatment may be required. Sysmäjärvi Lake is also Natura 2000 area and a RAMSAR site. Operational and/or post-closure impacts must be small enough not to risk the natural values (primarily bird species), which are the reason for the site protection status.	The project will not get an environmental permit, if the ecological status of classified watercourses below the project area is predicted to deteriorate, or if achieving a good ecological status is jeopardized. Potential project risks are permitting risk or water treatment cost risk.

Asbestos	According to the trial process tailings asbestos analysis (asbestos fibres > 5 µm), asbestos fibre content is relatively high 109 fibres/µg and 0.08 mass-%.	The asbestos quantity indicates asbestos levels that may require specific health and safety measures at the site. Environmental concerns about extractive waste storages may also become an issue, primarily airborne asbestos risks. Especially effective dust management may be needed. Potential project risks are permitting risk and cost risk.
Closure	During operational time discharge water can be actively treated but after closure, the waters are not treated. Even if the tailings and waste rock areas are properly closed, seepage waters affect the water bodies.	The project will not get an environmental permit, if the ecological status of classified watercourses below the project area is predicted to deteriorate, or if achieving a good ecological status is jeopardized. This also applies to post-closure impacts. Tailings area closure may require a low net percolation cover system to limit water infiltration into the tailings – to limit seepage from the tailings. The current cover suggestion is a store and release cover with a sealing layer, with relatively good percolation prevention, but it is not the most effective solution.

		Potential project risk types are permitting risk and cost risk.
Stakeholder issues	Stakeholders' approval and social impacts are risks that should be considered in all mining projects. This one is located near a community, that already suffers from the impacts of the previous mining operation.	Environmental and/or safety concerns may lead to appeals. Potential project risk types are permitting risk or project delay.
Liabilities	Project impacts on the historical extractive waste or site contamination have not yet been studied in detail. New mining operations are likely to disturb old wastes to some extent. Currently, it is uncertain which responsibilities will be transferred to the new operator.	Potentially a cost risk.

21 Capital cost

The investment cost estimate is prepared with ±30% accuracy for initial and sustaining capital costs. Cost estimate involves all disciplines required for the project implementation. The investment cost estimate is presented in document 101020700-M0001. This chapter only summarizes the initial and sustaining capital cost estimates.

Table 21-1 Capital cost estimation

In '000 EUR	
Direct costs	**58 572**
Mine	5 455
Concentrator plant	24 675
Tailings and Waste Rock	11 577
Water management	2 221
Site infrastructure	11 636
Mine closure	3 009
Indirect costs	**6 505**
Base estimate	**65 078**
Contingency 15%	9 762
Total investment cost (TIC)	**74 840**

Direct costs cover the equipment and bulk material for the construction while indirect costs cover the project EPCM delivery, temporary site services and facilities, vendor services and operational readiness costs.

Office and maintenance workshop buildings will be built by local municipal.

Project direct costs are split into 8 cost breakdown items while indirect costs are considered as an additional CBS item. Project base costs (direct + indirect costs) are split between CBS items as follows:

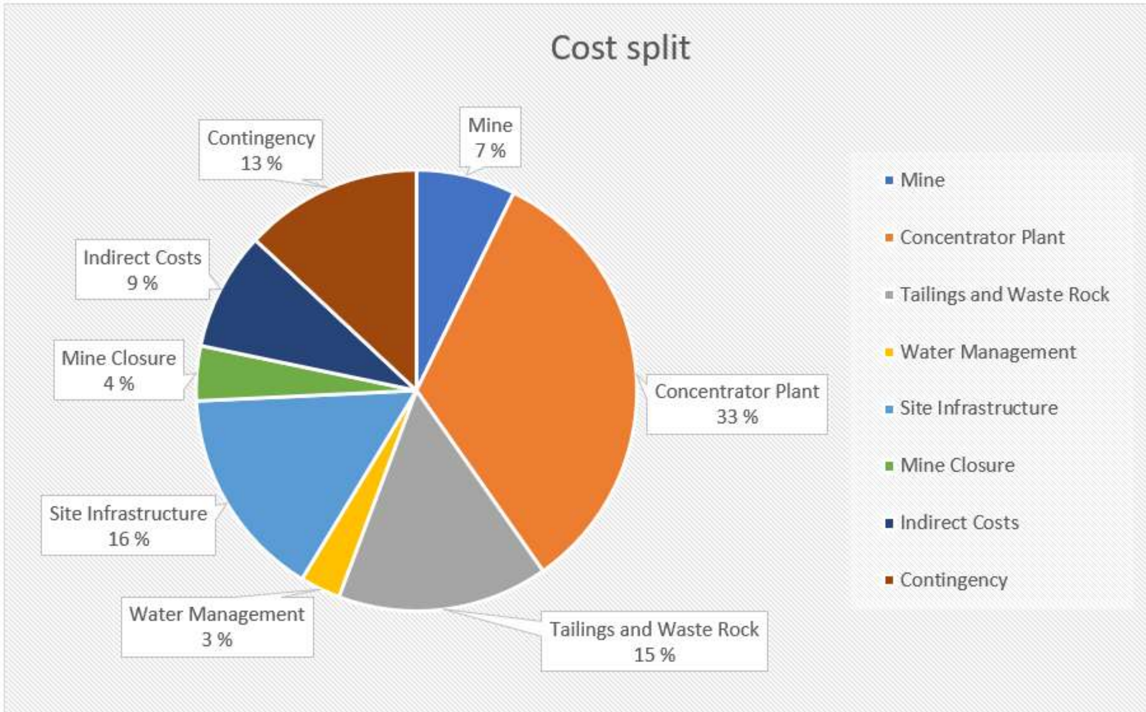


Figure 21-1 Project cost split between CBS items

Mine closure costs will be realized when mine is closed. Environmental permitting is excluded from cost calculation. Cost estimate principles inform how costs are calculated.

22 Operation costs

22.1 General

The operation cost estimate has been structured to the following four cost centers:

- Administration
- Mining
- Concentrator Plant
- Logistics

The operation cost estimate is calculated as an annual average. Annual variations during the LOM haven't been taken into account.

The detailed operating cost estimate is presented in document 101016697-02019 (Operating Cost Estimate).

22.2 Basis of estimate

- The electricity price used in the estimate is 50 Eur/MWh. Price is based on the long-term average in the Nordic energy market
- Chemical costs and other consumables (e.g. grinding media and liners) costs are based on AFRY reference projects
- Logistics costs are based on distance tables and AFRY reference project unit costs
- Labour costs have been estimated according to AFRY reference projects
    - All labour costs include salaries and social costs factor (35 %).
- Other costs are based on AFRY reference projects

The detailed basis of the estimate for Opex is presented in document 101016697-02028 (Basis of Opex Estimate).

22.3 Operating cost summary

The opex summary per cost center is presented in Table 22-1 and Figure 22-1.

Table 22-1. Total Operating Cost Summary

Cost Center	Eur/a	Eur/t (Ore)	% of Total Opex
Admin	1 552 352	3.10	7.5
Mining	10 772 550	21.55	52.3
Concentrator Plant	7 763 856	15.53	37.7
Logistics	489 352	0.98	2.4
Total	20 578 109	41.16	100.0

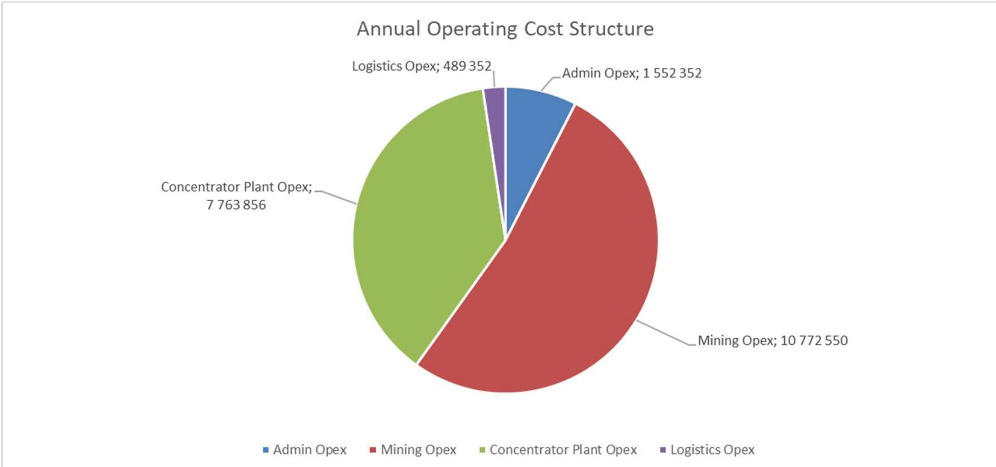


Figure 22-1. Total Operating Cost Summary, Eur/a.

22.4 Administration opex

Administration opex costs include

- General costs
    - Including e.g. environmental monitoring, dust management and IT costs
    - Office/workshop building rents
    - Mining act guarantee insurance, estimated annual 6 % cost of total closure costs including VAT
- Labour costs
    - In-house administration costs, including plant management, quality/environment/safety personnel, purchasing and warehouse labour.

The administration opex summary is presented in Table 22-2.

Table 22-2. Admin Opex Summary.

	Eur/a	Eur/t (Ore)	% of Admin Opex
General Costs	782 852	1.57	50.4
Labour Costs	769 500	1.54	49.9
Total Admin Opex	1 552 352	3.1	100.0

22.5 Mining opex

Mining operating costs include:

- Annual mining costs
    - Based on the use of contractors in mining operation
    - Includes underground mine development, ground support, LH stoping and hoisting
- Labour costs
    - Includes in-house mining labour (contractor salaries are included in the annual mining costs)
- Mining electricity costs
    - Including ventilation, pumping and mining equipment (e.g. drilling)
- Other mining costs
    - Other mining costs include mine air heating (oil heating)

The mining opex summary is presented in Table 22-3 and Figure 22-2.

Table 22-3. Mining Opex Summary.

	Eur/a	**Eur/t (Ore)**	**% of Mining Opex**
Annual mining costs	9 135 000	18.27	83.1
Labour Costs	793 800	1.59	8.3
Mining Electricity Costs	525 105	1.05	5.2
Other Costs (Mining Heating)	318 645	0.64	3.3
Total Mining Opex	9 526 237	23.82	100.0

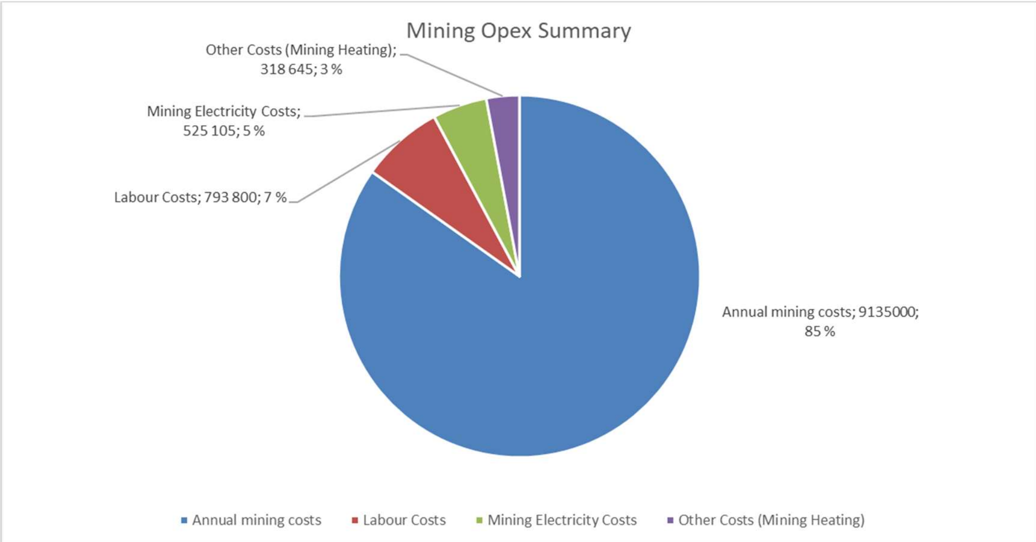


Figure 22-2. Mining Opex Summary, Eur/a.

22.6 Concentrator plant opex

Concentrator plant operating costs include:

- Consumables including
    - Chemicals
    - Mine backfill make-up additions
    - Grinding media & liners
- Electricity costs
    - Electricity cost is based on installed power, annual operation time and operation load factor
- Maintenance material costs
    - Annual maintenance materials are estimated as 2 % of equipment cost
- Laboratory & analyses costs
    - Laboratory analyses have been assumed to be outsourced to the local laboratory service
    - Elemental and particle size analyser (Courier and PSI) operation and maintenance costs
- Water handling & treatment costs
    - Water handling & treatment costs include electricity used in the water treatment, water treatment chemicals and maintenance related to water treatment
- Labour costs
    - Labour costs include plant management and engineers, process operators and maintenance engineers/crew

The concentrator plant opex summary is presented in Table 22-4 and Figure 22-3.

Table 22-4. Concentrator Plant Operating Cost Summary.

	Eur/a	Eur/t (Ore)	% of Conc. Opex
Consumables Costs	4 463 424	8.93	57.5
Electricity Costs	1 098 828	2.20	14.2
Maintenance Costs	227 736	0.46	2.9
Laboratory & Analyses Costs	300 000	0.60	3.9
Water Handling & Treatment Costs	53 868	0.11	1.0
Labour Costs	1 620 000	3.24	20.9
Total Conc. Opex	7 763 856	15.53	100.0

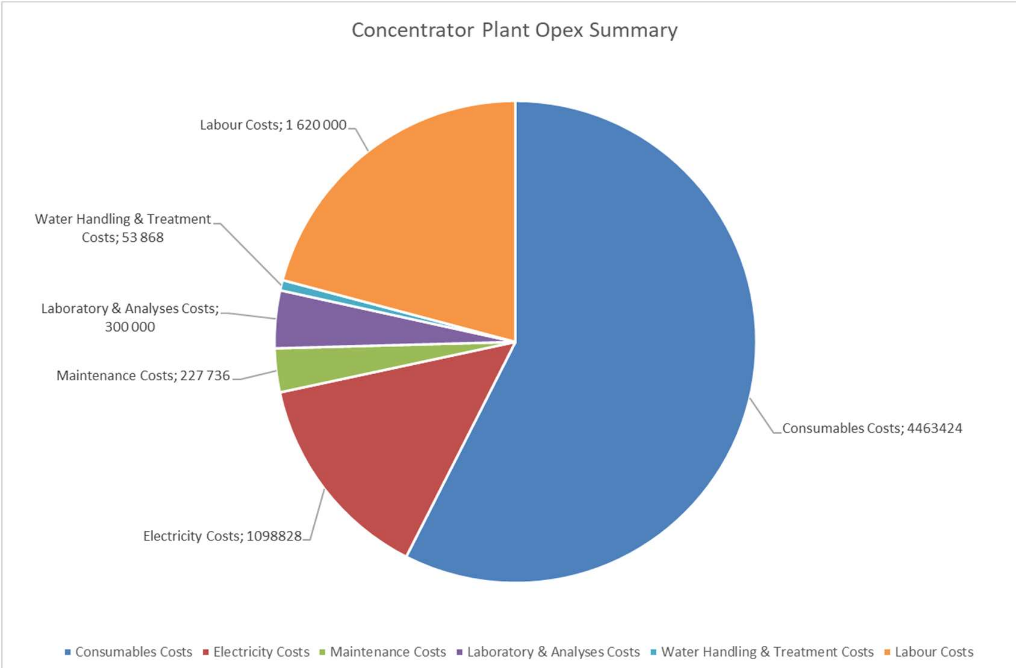


Figure 22-3. Concentrator Plant Opex Summary, Eur/a.

22.7 Logistics opex

The logistics cost estimate is based on the assumption of concentrate transportation to production sites in Harjavalta (copper concentrate) and Kokkola (nickel/cobalt concentrate), both located in Finland. The distance from the Hautalampi site to Harjavalta is approximately 500 km and to Kokkola approximately 380 km.

Chemical cost transportation costs are included in chemical unit costs in the concentrator opex estimate.

The logistics opex summary is presented in Table 22-5.

Table 22-5. Logistics Opex Summary.

	Eur/a	Eur/t (Ore)	% of Logistics Opex
Copper conc. transport	113 405	0.23	23.2
Ni/Co conc. transport	375 948	0.75	76.8
Total Logistics Opex	489 352	0.98	100.0

23 Discounted cash flow

The following assumptions have been used on discounted cash flow calculations.

- USD:EUR rate is assumed to be 1:1
- Environmental guarantee is excluded from the calculation
- Post-closure costs are included in the Capex calculation
- Company taxation is excluded from the DCF calculation
- FinnCobalt Oy provided Ni, Cu and Co price forecasts for the calculation

Key figures of base case discounted cash flow are presented in Table 23-1, Table 23-2 and Figure 23-1.

Table 23-1 The following principle for pricing has been used

Metal	Recovery	Payable	Metal price	
Nickel	82%	68%	20 000	USD/t
Copper	86.5%	96% - 65 USD/t	9 750	USD/t
Cobalt	82%	55%	70 000	USD/t
Gold	within conc.	90%	1800	USD/oz
Silver	within conc.	70%	23.05	USD/oz

Table 23-2 Discounted cash flow key figures

Net present value	58.8	MEUR	(5% discount factor)
Net present value	32.7	MEUR	(10% discount factor)
IRR	20.0	%	

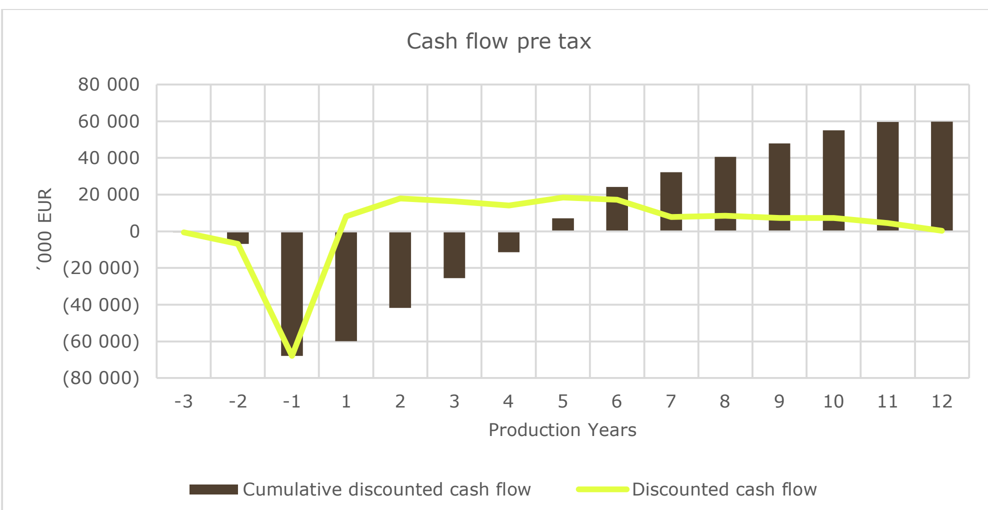


Figure 23-1 Discounted cash flow (discount factor 5 %)

Project sensitivities are presented in figures Figure 23-2and Figure 23-3.

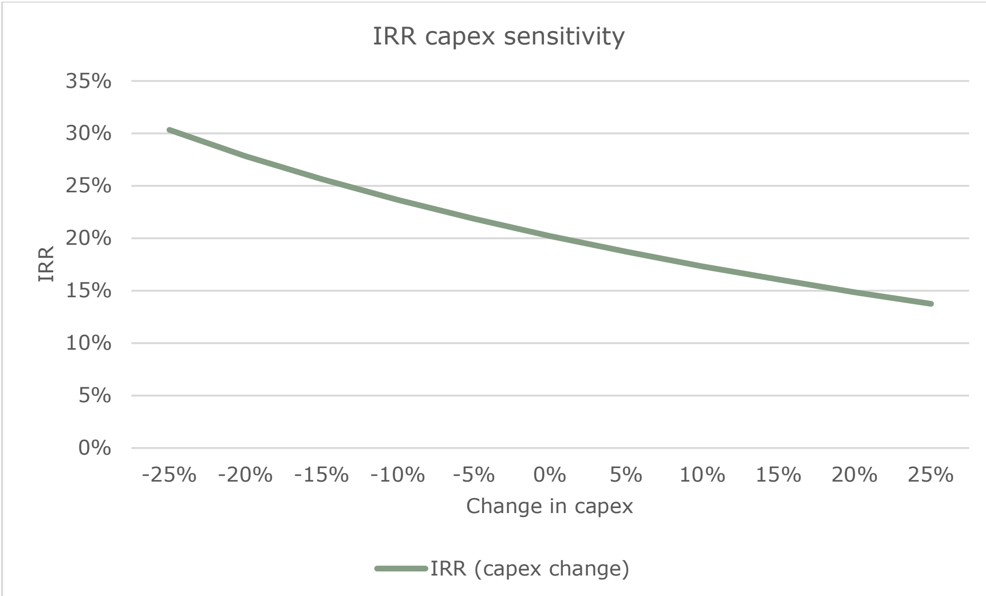


Figure 23-2 Sensitivity (IRR vs Capex)

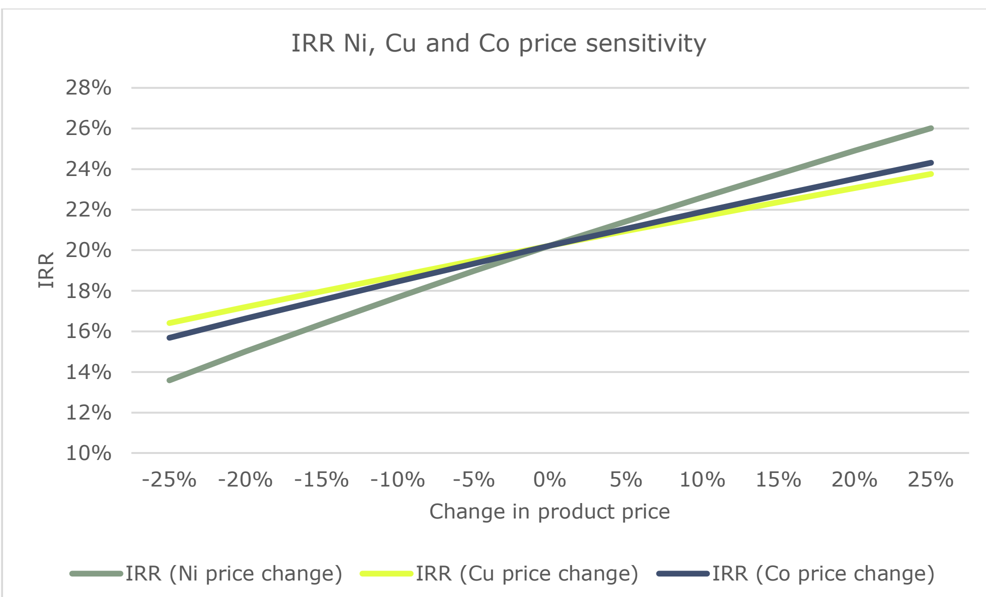


Figure 23-3 Sensitivity (IRR vs Nickel, Copper and Cobalt prices)

24 Project risk assessment

24.1 Capex / Opex risks

The study has been carried out as an update to the earlier pre-feasibility study, prepared in 2022. For the new study, the prices have escalated to the 1st quarter 2023 cost level with suitable index corrections. However, the recent cost fluctuation has been strong and this might continue in the same way in the short and also in longer term.

Electricity and consumables costs in operating cost estimates are based on long-term averages in the Nordic market. Volatility in the market leading to potentially higher prices should be taken into consideration as an Opex risk.

The volatility of costs at the time of reporting forms a risk that has been identified and the issue needs to be monitored.

24.2 Process risks

The process is a traditional flotation-based process. No major process-related risks are identified. Further testwork could be beneficial for the development of mine backfill concept and sulphur flotation optimization.

24.3 Environmental risks

The planned production site has old mine tailings facilities and that will cause some environmental risks. They are described separately in environmental paragraph 20.

24.4 Project execution risks

The site has good connections and it is close to significant cities. Also, other major infrastructures are available. No major project execution risks can be identified.

25 Project Execution

25.1 Project execution method

The Hautalampi project is rather small compared to many other mining projects. Therefore it is economical to keep the project organization lean and effective. The planned execution method is EPCM with a large owner's organization role.

Procurement packages are rather small to get the best possible price and quality for packages. No large EPC packages are considered.

25.2 Project organization

Project management tasks (PM, scheduling, purchasing, cost control) are planned to conduct with the company's personnel, which will run the plant when production starts. Therefore half of the project management costs are marked as operational expenses.

Engineering work will be outsourced to an engineering company as well as construction management.

25.3 Project schedule

The project has two different schedules to follow. The first is the project development and execution schedule and the second permitting schedule.

Table 25-1 Project development and execution schedule

	Year 1			**Year 2**				**Year 3**				**Year 4**			
	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Feasibility study															
Definitive feasibility study															
Investment decision					♦										
Project execution															
Production starts												♦			

Table 25-2 Project permitting schedule

	Year 1			**Year 2**				**Year 3**				**Year 4**				**Year 5**			
	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Environmental impact assessment																			
Permitting process																			
Environmental permit																			
Possible Appeals																			

*) Green line marks the project's present status

The schedules above indicate that permitting process is on a critical path. There is time for feasibility and definitive feasibility studies.

26 Other relevant data and information

The site is within an old nickel mine site. The old mine was opened in 1954 and closed in 1989. Old tailings areas do not fulfil today's requirements.

Liability assessment concerning the historical extractive waste within the planned mining operation area is not included in the scope of this study and AFRY or CPs do not take any responsibility regarding risk or cost assessments related to the previously mentioned liabilities.

The cost estimate is based on the general price level in [Q1/2023]. However, the estimates are subject to change and due to prevailing circumstances the amounts may differ materially from those described in the report.

27 Interpretation and Conclusions

The following remarks and conclusions regarding the Hautalampi project are summarized below:

- The drilling and sampling to date support the mineral resources estimate and there is sufficient information to be used as a basis for the mineral resource estimate.
- The drilling pattern and spacing cover the known measured, indicated and inferred mineral resources. A limited amount of new drilling down-dip of the historic drilling could upgrade the indicated and inferred resources.
- The deposit geology and style of mineralization are well understood, and the property has a history of successful mining activities. However, the Mökkivaara area needs more consideration to upgrade the resource class.
- Based on the mineral resource and ore reserve estimate, the project is well suited to proceed to the next study phase.
- The process is a typical flotation process based on well-known technologies
- The site is an old nickel mine site with old tailings areas. This will require special attention to tailings methods and permitting.
- The site location is close to an active city, which makes project execution easier compared to more remote locations.

28 Recommendations

28.1 Mining

AFRY recommends refining the mine plan for optimal mining scenarios. Furthermore, more studies are recommended in Mökkivaara as the area can provide upside potential to the Hautalampi project. In the next study phase, AFRY recommends that a trade-off study between owner-operated mining fleet versus contractor-operated mining operation is considered. Ventilation simulations and 3D rock mechanical simulations are also recommended to check the life of mine scale mine stability and to validate the ventilation principle planned for the Hautalampi mine. A detailed study of the haulage profiles is recommended in future studies.

28.2 Process

Sulphur Removal Test Work

Sulphur removal test work feed material (Ni/Co flotation tailings) in the GTK bench tests had a lower sulphur grade (0.62 % S) compared to the PFS mass balance (1.17 % S). PFS mass balance was developed based on the mining plan feed grades and GTK pilot test work results. Further test work would verify grade and recovery results and provide input for the process design. This could have the potential to reduce sulphur flotation equipment sizing.

Sulphur removal flotation efficiency is dependent on the pH level used in the flotation. Flotation tests in varying pH levels would provide more information e.g. estimation of required amounts of sulphuric acid used in the pH control.

Sulphur Removal with Magnetic Separation

Most of the sulphur in the Ni/Co flotation tailings is in the form of pyrrhotite (FeS). Magnetic separation of pyrrhotite with low-intensity magnetic separation (LIMS) could be possible. Replacement of flotation cells with LIMS as a process stage for sulphur removal could be investigated on the laboratory scale.

Mine Backfill Concept and Characteristics

The mine backfill concept has been developed based on reference operations in Finnish underground mines. The concept is based on traditional hydraulic fill operation. Separation of fine solids (which could harm backfill operation) and increase in slurry density have been assumed to be done by hydrocyclone. Mine backfill characteristics of the material haven’t been studied in this project phase. A characteristics study is recommended for the next phases of the project. GTK pilot test work report indicates that sulphur grade is higher in the fine fractions of the Ni/Co circuit tailings, which could affect the sulphur split to backfill and deposited tailings.

Continuous Leaching Test Work

Leaching of the Ni/Co concentrate to produce Ni/Co sulphate or hydroxide products have been tested at the laboratory scale. Continuous testing at the pilot scale would confirm and optimize process parameters for the possible hydrometallurgical plant design.

28.3 Tailings and waste rock management

Given that there are oxidisable sulphur species present in the tailings, and that there are environmental elements of concern associated with these minerals, it is strongly recommended that the tailings sample is subject to further geochemical characterisation through kinetic testing utilising either humidity cells or columns. These will enable an understanding of the mobilisation of these sensitive elements with respect to time and due to the degradation of sulphide minerals.

In addition, an understanding of how any potential leachate released from the new tailings will interact with the existing tailing material and ultimately what impact this will have on both surface and underground water features. This will then enable an assessment of potential mitigation controls and aid in the decision of a closure solution, particularly as the waste is not deemed an environmentally inert material.

It is recommended to implement a dam monitoring system from the beginning of the construction phase and to continue monitoring during the entire dam life span to improve dam safety and reduce the probability of environmental accidents. The most important variables that must be considered in the future are phreatic surface, pore pressure, settlement, and displacement monitoring. If the monitoring system is automated, it will be possible to retrieve real-time data from monitoring instruments and to be alerted in case of sudden changes in the dam or in the foundation. Automated monitoring is only an addition to safe dam management, it cannot replace visual and regular dam inspections.

Ground investigations (groundwater levelling, drillings, sample taking) must be done to the old tailings area and Ruutunkangas to clarify the ground conditions for the dams and the base of the liners. The ground conditions also have an impact on the basal structures, used materials and products, pricing and construction costs.

Tailings rheological, geotechnical and geochemical testing to determine properties that impact the design, stability and drainage of the impoundment (in-place and relative density, permeability, plasticity, compressibility, consolidation, shear strength and stress parameters) of the new tailings and the old tailings.

Detailed seepage and stability analysis considering updated geometry and area layout, potentially using some local construction material derived from foundation investigations and tailings material characterization.

The dusting problems, and the importance of investigating and applying alternative dust prevention methods, increase as the material gets dryer (beach areas and possible dam raisings made of tailings).

Waste rock characterization should be initiated, based on tunnel plans /mine planning. Waste rock and mine wall characterisation are recommended to be carried out as one combined task. Considering the known mineral presence in the Outokumpu area and tailings findings (Työterveyslaitos 2021), also fibrous minerals assessment is recommended.

28.4 Infrastructure

It is recommended to conduct heavy falling weight deflectometer tests (such as KUAB) together with some soil sampling for the existing road sections. The information will serve as a basis for road structure dimensioning in upcoming phases.

Before the next study phase, field investigations need to be conducted to increase the accuracy of design solutions for buildings foundations.

Some or all of the following investigation types are recommended:

- Static-dynamic penetration tests
- Swedish weight soundings
- Percussion drillings
- Groundwater pipes for sampling and especially for water table measurements
- Disturbed soil samplings
- Ground-penetrating radar

28.5 Environment

A mitigation plan to reduce recognised Environmental and Social Risks is recommended to be made. Furthermore, it is recommended to create a detailed stakeholder engagement and participation plan and establish a suitable grievance procedure.

29 References

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30 Appendices

Appendix 1 101016697-02001 Process Design Criteria_updated

Appendix 2 101016697-02002 Block diagram_updated

Appendix 3 101016697-02003 Mass Balance_updated

Appendix 4 101016697-02004 Finncobalt equipment list_updated

Appendix 5 101016697-02005-02016 Process flow sheets_updated

Appendix 6 101016697- Process plant pictures

Appendix 7 101016697-02020 Finncobalt Process Description_updated

Appendix 8 101016697-02021 Conceptual Water Cycle Graph_updated

Appendix 9 101016697-02022 Water Balance Model_updated

Appendix 10 101016697-02023 Water Management General Layout_updated

Appendix 11 101016697-02024 Water management General Flow Diagram_updated

Appendix 12 101016697-02025 Water Treatment Process Flow Diagram_no_update

Appendix 13 101016697-10001 General Site Layout

Appendix 14 Office and workshop layout

Appendix 15 Area Layout Drawing

Appendix 16 Principles of cost estimate

Appendix 17 101020700-M0001 Cost estimate

Appendix 18 101016697-02028 Basis of Opex Estimate

Appendix 19 101016697-02019 Operational costs

Appendix 20 101016687-M0003 Discounted cash flow

JORC Code, 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as downhole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay '). In other cases, more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	• All holes in the estimate were diamond drill holes. • Diamond drill core was sampled based on sample intervals determined by trained geologists. • The drill core was half-cut. Older Outokumpu-era drill holes were also sent for analysis by interval selection. • Sampling has been carried out under Outokumpu Oy, Finn Nickel Oy and FinnCobalt Oy geologists for the respective drilling campaigns. Sampling protocols and quality assurance/quality control (QAQC) procedures as per company standards relative to the company that made the drilling. • Selected samples were crushed and pulverised and sent to analysis.
Drilling techniques	• *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether the core is oriented and if so, by what method, etc).*	• The following diamond drill hole campaigns have been made: *Campaign — Company — Sample size* *HA — 2020, FinnCobalt Oy — 57.5 mm* *HL — 2007–2008, Finn Nickel Oy — 42 mm + 62mm* *OKU — 1950–1987, Outokumpu Oy Exploration and Keretti Mine — 22 mm* *k — 1950–1986, Outokumpu Keretti mine, underground drilling — 22 mm*

Criteria	JORC Code explanation	Commentary
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	• The core recovery was physically measured and recorded by the drillers for every core run. Any core loss was recorded on the drill core report by drillers. • Core recovery was double-checked and measured for all drill holes during geological logging procedure. • No additional measures were taken to maximise the core recovery. • The core recovery was generally very good. A sampling bias has not been determined.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* • *The total length and percentage of the relevant intersections logged.*	• All drill core was geologically logged determining lithology and mineralogy. Rock quality designation (RQD), Q´values and core recovery were measured for all drill cores by the Finn Nickel and FinnCobalt geologists. • All Finn Nickel and FinnCobalt drill cores are photographed in wet and dry states after logging was completed and sample intervals had been marked on the core boxes. • A total of 3113 samples were measured for density.
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise the representativity of samples.* • *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* • *Whether sample sizes are appropriate to the grain size of the material being sampled.*	• Core was cut in half using a diamond saw with generally 1-3 m half core samples submitted for analysis. Outokumpu-era samples were assayed in the company laboratory. • Finn Nickel samples were crushed with more than 70% passing the <6 mm, then reduced in a splitter to 150 g. The 150 g sample is pulverised with more than 85% passing <75 microns. • FinnCobalt samples were crushed with more than 70% passing the <2 mm, then reduced in a splitter to 250 g. The 250 g sample is pulverised with more than 85% passing <75 microns. • Finn Nickel and FinnCobalt samples were prepared by ALS as per industry best practice. Outokumpu's (large exploration and mining company at the time) work was carried out for their internal use, it is believed that the

Criteria	JORC Code explanation	Commentary
		work was carried out to industry standards and Outokumpu exploration practices for that time.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* • *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* • *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	• The Finn Nickel samples were analysed using Labtium Oy methods: 510P,521U, 703P/704P. Also, Finn Nickel's laboratory was used utilizing the AAS method and S-analyser. • FinnCobalt samples were analysed at ALS Laboratory in Ireland using methods: ME-ICP61, S-IR08, ME-ICPORE, AAORE, Ni-ICP05 and Au-AA23. QAQC: • Finn Nickel samples: every batch of 50 samples contained 2 standards, 1 blank and 3 laboratory duplicates. • FinnCobalt: CRM, Blanks and core duplicates added to sample batches (ca. 15% of analysed samples were QAQC samples).
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.* • *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* • *Discuss any adjustment to assay data.*	• FinnCobalt has carried out two reassay campaigns for the old Outokumpu samples. All information has been internally audited by various consulting groups. • Twinning of holes was not carried out. • All present-day field data is captured electronically and subsequently validated as it is imported into the centralised Access database. • Electronic copies of logs, surveys and sampling data are stored in the cloud data server. • No adjustments have been made to any assay data in this report.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.* • *Specification of the grid system used.* • *Quality and adequacy of topographic control.*	• FinnCobalt drill locations were measured with a differential GPS, and drill hole surveys were made with the Devico Deviflex instrument.

Criteria	JORC Code explanation	Commentary
		• Finn Nickel drill locations were measured with a differential GPS but no 3d surveying of the holes was made. • The grid system used is Finnish KKJ Grid Zone 4. • The topographic data used for the drill sections has been gridded from elevation data acquired from the National Land Survey of Finland.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.* • *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* • *Whether sample compositing has been applied.*	• Drill data is at a sufficient spacing to define Measured, Indicated and Inferred Mineral resources. • Compositing to 1.5 m has been applied before resource estimation.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* • *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	• Drill holes have been drilled perpendicular to the interpreted strike of the mineralisation and lithology. • No sampling bias as a consequence of orientation-based sampling has been identified.
Sample security	• *The measures taken to ensure sample security.*	• The sample "chain of custody" is managed by FinnCobalt Oy's geological personnel. • Drill cores are stored in a locked facility in Outokumpu and GTK's Loppi core archive.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• Internal company auditing and a review by AFRY Finland Oy during the resource work in May-June 2021 found that FinnCobalt Oy's data collection and QA/QC procedures were conducted to industry standards.

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	• The property is covered by FinnCoblt mining concession 7802/1. The total area of the mining concession is 283.5 hectares. • The mining concession is valid.
Exploration done by other parties	• *Acknowledgement and appraisal of exploration by other parties.*	• The earliest drillings in the Co-Ni enriched zone nearby the Keretti Cu ore were made by Outokumpu Oy already in the 1930s. Outokumpu continued the exploration between 1950 to 1987. FinnNickel drilled the property between 2007–2008.
Geology	• *Deposit type, geological setting and style of mineralisation.*	• Hautalampi is a hanging-wall Co-Ni-Cu mineralised body 150 to 200 metres vertically above the historic Keretti mine. VMS type deposit. The deposit is located in the quartz rock-skarn zone between serpentinite and mica schist, above the Keretti copper ore. The rock is banded and occasionally also slaty. Banding is attributed to the variation in grain size, in the abundance of Ca-Mg minerals, in dust-like sulphides and microcrystalline graphite and chromite.
Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* ○ *easting and northing of the drill hole collar* ○ *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* ○ *dip and azimuth of the hole* ○ *downhole length and interception depth* ○ *hole length.* • *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does*	• Not relevant. Exploration results are not being reported.

Criteria	JORC Code explanation	Commentary
	not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high-grade results and longer lengths of low-grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	• Not relevant. Exploration results are not being reported.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the downhole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	• Not relevant. Exploration results are not being reported.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	• A relevant plan showing the drilling is included within this report.
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practised to avoid misleading reporting of Exploration Results.*	• Not relevant. Exploration results are not being reported.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density,*	• Not relevant. Exploration results are not being reported.

Criteria	JORC Code explanation	Commentary
	groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.	
Further work	• *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* • *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	• Further geological modelling is recommended to increase confidence in inferred mineral resources.

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures used.*	• Drilling data is electronically stored in an Access database, that is managed by FinnCobalt Oy. • Validation of the data import includes checks for overlapping intervals, missing survey data, missing assay data, missing lithological data, and missing collars.

Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.*
	• *If no site visits have been undertaken indicate why this is the case.*
		• Mr Seppä visited the site on March 10th, 2021. The inspection included:
		  o Visiting the historic Keretti mine area.
		  o Visiting the drill core storage.
		  o Overall view of the property.
		  o Inspection of available drill holes.
		• Discussions with Markus Ekberg, CEO of FinnCobalt Oy and geologists Kalle Penttilä and Matthias Mueller of FinnCobalt.

Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	• The modelling of mineralised solids was done by modelling solids with a NiEq % cut-off of 0.25%. The mineralised areas are identified and there is a clear orientation of grade continuity present. The resource solids • Drillhole intercept logging and sample analysis results have formed the basis of the geological and mineralisation interpretations. • The extents of the modelled mineralisation zones are constrained by available drill data. Alternative interpretations are not expected to have a significant influence on the global Mineral Resource estimate. • The continuity of the geology and mineralisation can be identified and traced between drill holes by visual and assay characteristics. The geology and mineral

Criteria	JORC Code explanation	Commentary
		distribution of the system appear to be reasonably consistent. Confidence in the grade and geological continuity is reflected in the Mineral Resource classification.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	• The lower edge of the Co-Ni-Cu-mineralisation zone is typically some 150 to 200 m above and a bit to the NW of the upper edge of the main Keretti Cu-ore. • Hautalampi mineralised zone is approximately 1500 m in length, 100-150 m in width and 1-30 m in thickness. Some drill holes indicate that in the NW parts the mineralisation is cut by the present erosion surface. Mineralisation has a 10 - 55° dip to the SE (on average about 25-30°) • Mökkivaara mineralisation is located approximately 650 meters northeast of the Hautalampi mineralisation and it has the same overall strike and dip as the Hautalampi mineralisation.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer-assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.*	• The Ordinary Kriging method ("OK") algorithm for grade interpolation was used for the Hautalampi Mineral Resource using experimental variogram models created for the elements Ni, Cu, Co, S, Fe and Zn. Mökkivaara Mineral Resource estimation was carried out using the Inverse Distance Squared method ("ID2") algorithm using a search ellipsoid oriented to the average strike, plunge and dip of the mineralized zone. • Surpac software was used for the estimation. • The estimate is based on a block size of 5 m (X)by 5 m (Y) by 5m (Z), with sub-blocks of 2.5m by 2.5m by 2.5m. The block model is rotated -45 degrees around Z-axis to match the general strike of the mineralization. A bulk density value of 2.82t/m³ was assigned to all materials. • No grade cuts were applied to the estimate. • Selective mining units were not modelled in the Mineral Resource model.

Criteria	JORC Code explanation	Commentary
	• *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	• For validation, a quantitative spatial comparison of block grades to assay grades was carried out using swath plots. • Global comparisons of drill hole composites and block model grades with different modelling methods (nearest neighbour and inverse distance) were also carried out. • The estimation was constrained by interpreted resource solids.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• Tonnages have been estimated on a dry in situ basis.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• The cut-off value for NiEq was estimated by using a NiEq value calculation. • NiEq was then compared against the assumed operating cost (OPEX) to see the break-even cut-off. • 0.25% NiEq was selected to be an appropriate modelling and reporting cut-off.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	• It has been assumed that these deposits are amenable to underground mining methods and are economic to exploit to the extent currently modelled. No assumptions regarding minimum mining widths and dilution have been made.
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported*	• Bench and pilot test work on Hautalampi ore has been carried out at the Geological Survey of Finland (GTK) Mintec plant. ○ Laboratory flotation and mini-pilot tests were conducted in 2007. Cu grade and recovery of Cu concentrate were 24.4% and 76.1%. Ni/Co concentrate Ni grade and recovery were 10.0%

Criteria	JORC Code explanation	Commentary
	with an explanation of the basis of the metallurgical assumptions made.	and 40.9%. Ni/Co concentrate Co grade and recovery were 3.8% and 42.4%. Feed grades in the test were 0.76% Cu, 0.31% Ni and 0.11% Co. ○ Laboratory and mini-pilot test work was conducted again in 2009. In laboratory test work Cu grade and recovery of Cu concentrate were at best 31.2% and 76.4%. In the same test, Ni/Co concentrate Ni grade and recovery were 14.2% and 54.1%. Ni/Co concentrate Co grade and recovery were 4.1% and 56.%. Feed grades in the test were 0.676% Cu, 0.311% Ni and 0.09% Co. Recoveries in the mini-pilot were generally at a lower level, but ore had been oxidized which was found as the likely reason for lower recoveries. Based on the results GTK estimated a 25% Cu grade with an 85% Cu recovery for full-scale production. For Ni/Co concentrate Ni grade and recovery in full scale were estimated to be 6.0% and 80%. ○ Bench-scale test work was done in early 2019 where the average grades in the ore were 0.37% of copper, 0.47% of nickel and 0.14% of cobalt. The test was done as an open circuit test. The average copper grade in copper concentrate was 26.2% with 78.8% Recovery. Average Ni and Co grades and recoveries in Ni/Co concentrate were 7.7% Ni grade / 64 % Ni recovery and 1.8 % Co grade / 62 % Co recovery. ○ Continuously operated flotation pilot had a copper recovery of 86.5% nickel recovery of 82.0% and cobalt recovery of 82.6% as its highest. The copper concentrate grade was 26.7% Cu. Ni and Co grades in Ni/Co concentrates were 8.0% and 1.9%. The feed rate was between 30–35 kg/h. Test was done in late Autumn 2019. The average grades in the ore were 0.362% of copper, 0.426% of nickel and 0.112% of cobalt.

Criteria	JORC Code explanation	Commentary
		○ Historically similar kinds of processing studies have been conducted in the 1980s by Outokumpu and VTT. Leaching Test Work: • The performance of FinnCobalt nickel-cobalt concentrate leach extractions, PLS purification efficiency and purity of the final products were tested in 2019 by Outotec. Test work was conducted for two process concepts: 1) production of nickel and cobalt sulphate solution and 2) production of mixed hydroxide precipitate (MHP). Lab-scale batch test work was used in all tests. ○ In atmospheric leaching 89% nickel and 85% cobalt extraction were achieved by using 72 h leaching time, 70 g/L sulfuric acid concentration a particle size d80 < 20 µm of concentrate. ○ With pressure oxidation at 220 degrees and O2 partial pressure of 7 bars, 98.7% nickel and 99.7% cobalt extraction were achieved. ○ Iron and other impurities were removed efficiently from the PLS with jarosite precipitation, sulphide precipitation and SX process. ○ Test results indicate that it was possible to produce pure battery-grade nickel and cobalt sulphate solutions or MHP products. Ni/Co sulphate was produced to the sulphate solution. In MHP precipitation 95.6 % Ni recovery and 97.4 % Co recovery from solution were achieved at pH 7.5.
Environmental factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a*	• A clear permitting process exists in Finland. • The deposit has an Environmental Permit for underground mining in force and Mining Lease appropriation is ongoing. • In Autumn 2020 the company decided to commence a new Environmental Impact Assessment for the project

Criteria	JORC Code explanation	Commentary
	greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	including underground mining and on-site ore processing and battery chemicals production plant. The EIA was completed on May 17th 2022.
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* • *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* • *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	• Bulk density was estimated based on available density measurements. • A total of 3113 samples were available and from those 478 were inside modelled resource solids. 1301 samples were outside of resource solids and were taken from samples below the selected modelling cut-off of 0.25 % NiEq. • The average density was calculated for waste rock and mineralised material. As a result, a density of 2.82 was used for both waste and mineralised material.
Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.* • *Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).* • *Whether the result appropriately reflects the Competent Person's view of the deposit.*	• The Mineral Resource was classified in accordance with the Australasian Code for the Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC, 2012). • Classification of the Mineral Resource has accounted for the level of geological understanding of the deposit, quantity, quality and reliability of sampling data, assumptions of continuity and drill hole spacing. • The Mineral Resource is classified as an Indicated Mineral Resource for those volumes wherein the Competent Person's opinion there is adequately detailed and reliable, geological, and sampling evidence, which are sufficient to assume geological and mineralisation continuity. • The Mineral Resource is classified as an Inferred Mineral Resource where the model volumes are, in the Competent Person's opinion, considered to have more limited geological and sampling evidence, which are sufficient to

Criteria	JORC Code explanation	Commentary
		imply but not verify geological and mineralisation continuity. • The volumes located outside the mining concession were not classified. • The Mineral Resource estimate appropriately reflects the view of the Competent Person.
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• Internal audits and peer reviews were completed by AFRY Finland Oy which verified and considered the technical inputs, methodology, parameters and results of the estimate. • No external audits have been undertaken.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	• The estimate utilised good estimation practices, good quality drilling, sampling and assay data. The extent and dimensions of the mineralisation are sufficiently defined by the detailed drilling. The deposit is considered to have been estimated with a good level of accuracy. • The relative accuracy of the Mineral Resource estimate is reflected in the reporting of the Mineral Resource as per the guidelines of the JORC Code (2012 Edition). • The Mineral Resource statement relates to global estimates of in situ tonnes and grade. • No mining has taken place at this deposit to allow reconciliation with production data.

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Section 4 Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• *Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve.* • *Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.*	• The Mineral Resources for the Finn Cobalt Oy Hautalampi deposit were reported in June 2021. The resource statement is signed by Mr Ville-Matti Seppä who is a European geologist with sufficient relevant experience to qualify as a Competent Person. • The Mineral Resources are inclusive of these Ore Reserves.

Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.*
	• *If no site visits have been undertaken indicate why this is the case.*
		• Mr Seppä visited the site on March 10th, 2021. The inspection included:
		  o Visiting the historic Keretti mine area.
		  o Visiting the drill core storage.
		  o Overall view of the property.
		  o Inspection of available drill holes.
		  o Discussions with Markus Ekberg, CEO of FinnCobalt Oy and geologists Kalle Penttilä and Matthias Mueller of FinnCobalt.

Study status	• *The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.* • *The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.*	• This Ore Reserve estimate is based on a pre-feasibility study made by AFRY Finland Oy in 2022 and updated in March 2023. • The Mineral Resources have been converted to Ore Reserves utilizing underground mine study and geotechnical study. Only Mineral Resources within the LOM plan have been included in the Ore Reserves. Standard modifying factors as stated below were used.
Cut-off parameters	• *The basis of the cut-off grade(s) or quality parameters applied.*	• For the mine design and reserve reporting purposes, a 30€/t NSR cut-off value was used. All excavated material needed to be above 30€/t NSR value and the average NSR value for the individual stope needs to be above 50€/t. •
Mining factors or	• *The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of*	• Two different stoping methods will be utilized in the Hautalampi underground mine. o Longitudinal long hole stoping will be the primary

Criteria	JORC Code explanation	Commentary
assumptions	*appropriate factors by optimisation or by preliminary or detailed design).* • *The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.* • *The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling.* • *The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate).* • *The mining dilution factors used.* • *The mining recovery factors used.* • *Any minimum mining widths used.* • *The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.* • *The infrastructure requirements of the selected mining methods.*	mining method. ○ Up-hole bench stoping will be used in the underground mine whenever there is no access drift above the stope. • The selected mining methods are common in Finland and the local contractors can execute the mine plan. • 90% mining recovery has been used to convert the Underground Mineral Resources into Ore Reserves. • An external dilution factor of 15 % is used. • Reserves have only been derived from Measured and Indicated Resource categories. • Inferred Mineral Resources were not considered in the conversion.
Metallurgical factors or assumptions	• *The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.* • *Whether the metallurgical process is well-tested technology or novel in nature.* • *The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.* • *Any assumptions or allowances made for deleterious elements.* • *The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.* • *For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?*	• The process concept is based on well-known unit operations used widely in the mining industry. • Unit processes are used widely in the processing of Ni-Cu-Co ores. • The process has been tested on a bench scale and in continuous pilot-scale operation at GTK Mintec. • No deleterious elements

Criteria	JORC Code explanation	Commentary
Environmental	• *The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.*	• The project is in the early phases of the EIA procedure, which in Finland takes place before environmental permitting. The EIA programme (which defines the scope of impact assessment work) has been published. • Status of the project's environmental assessments (including waste management considerations), identification of environmental project risks and recommendations have been reported in the pre-feasibility study made by AFRY Finland Oy in 2022.
Infrastructure	• *The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided, or accessed.*	• The Hautalampi project is located in an old mining area with good access to power and water supply. Skilled labor is available to be hired or used as contractors.
Costs	• *The derivation of, or assumptions made, regarding projected capital costs in the study.* • *The methodology used to estimate operating costs.* • *Allowances made for the content of deleterious elements.* • *The source of exchange rates used in the study.* • *Derivation of transportation charges.* • *The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.* • *The allowances made for royalties payable, both Government and private.*	• Operating costs have been estimated using in-house cost data from AFRY. Operating costs are mainly based on unit consumptions derived from test work with Hautalampi ore or from reference operations with similar processing. • Transportation costs are based on distance tables and in-house unit cost data from AFRY.
Revenue factors	• *The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.* • *The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.*	• Head grade and dilution were estimated yearly basis on the LOM plan. • Metal prices are based on market assessment.
Market assessment	• *The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.* • *A customer and competitor analysis along with the identification of likely market windows for the product.* • *Price and volume forecasts and the basis for these forecasts.* • *For industrial minerals the customer specification, testing*	• The Market assessment that is used as a basis for Reserve estimation is based on a World Bank Report, April 2021, Commodity Markets Outlook and S&P Global predictions Copper: • S&P Global (Feb 2021) predicts a stable 7 500 USD/t by 2023 and 2025.

Criteria	JORC Code explanation	Commentary
	and acceptance requirements prior to a supply contract.	• S&P Global consensus target price (Feb 2021) suggests 7 200 USD/t in 2023 and 7 650 USD/t in 2025. • A poll conducted by major investment banks, economists, brokers etc. have a consensus of an average Cu price of ~8 130 USD/t by Q4 2022 (https://www.mining.com/copper-price-flies-high-but-further-out-forecasts-are-grim/). • The WorldBanks' commodity market outlook (April 2021) predicts a 38% price increase in 2021 and a following 12% price drop in 2022. • The current Cu price is at a 10-year-high at ~10 000 USD/t. • FinnCobalt Oy selected 9 750 USD/t price for DFC calculations. Cobalt: • S&P Global (Feb 2021) predicts ~43 000 USD/t in 2023 and ~52 000 USD/t in 2025. • S&P Global consensus target price (Feb 2021) suggests 46 300 USD/t in 2023 and 47 300 USD/t in 2025. • The current Co price is ~47 000 USD/t. • FinnCobalt Oy selected 70 000 USD/t price for DFC calculations. Nickel: • The nickel price outlook seems to be the most controversial one, with forecasts ranging between 14 000 to 21 500 USD/t. • S&P Global (Feb 2021) predicts 20 000 USD/t in 2023 and 21 500 USD/t in 2025. • S&P Global consensus average target price (Feb 2021) suggests 16 400 USD/t in 2023 and 16 750 USD/t in 2025. • CRU Group (March 2021) expects Ni prices to decrease over the coming years, stabilizing at ~14 000 USD/t (https://www.kitco.com/news/2021-03-17/Why-geopolitics-of-this-green-battery-metal-should-be-on-your-radar-CRU.html#:~:text=According%20to%20CRU%2C%20nickel%20prices,right%20now%2C%22%20he%20said) • The WorldBanks' commodity market outlook (April 2021)

Criteria	JORC Code explanation	Commentary
		predicts a 20% Ni price increase in 2021, followed by a price decline in 2022. • Ni price has recently touched 20 000 USD/t (end of Feb 2021) and is currently trading at 17 500 USD/t. • FinnCobalt Oy selected a 20 000 USD/t price for DCF calculations.
Economic	• *The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.* • *NPV ranges and sensitivity to variations in the significant assumptions and inputs.*	• The economics of the project have been evaluated with an Excel-based real-basis financial mode. • The average Metal Prices that were used in the DCF model: ○ Ni 20 000 EUR/t ○ Co 70 000 EUR/t ○ Cu 9 750 EUR/t • An exchange rate of 1.0 USD to Euro is used. • In general, a conservative approach has been taken in the evaluation of the Project. • The metal prices are based on FinnCobalt data.
Social	• *The status of agreements with key stakeholders and matters leading to social licence to operate.*	• The status of the project's socioeconomic assessments, identification of social project risks and recommendations have been reported in the pre-feasibility study made by AFRY Finland Oy in 2022. In Finland, EIA and environmental permitting include certain procedures for the collection of stakeholder opinions and comments. Formal (authority-driven) and additional stakeholder consultation are also discussed in the PFS report.
Other	• *To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves:* • *Any identified material naturally occurring risks.* • *The status of material legal agreements and marketing arrangements.* • *The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility*	• AFRY has no doubts regarding the necessary Government approvals and the timeframes. • Collapses in the Mökkivaara area may affect the expansion of Hautalampi and Mökkivaara resources.

Criteria	JORC Code explanation	Commentary
	study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	
Classification	• *The basis for the classification of the Ore Reserves into varying confidence categories.* • *Whether the result appropriately reflects the Competent Person's view of the deposit.* • *The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).*	• Ore Reserves have been classified into Proven and Probable categories based on the Mineral Resource categories. • No conversion from Indicated Mineral Resource into proven ore reserve was made. No conversion from the measured mineral resource into probable Ore Reserve was made. • The Ore Reserves of the deposits appropriately reflect the Competent Person's view of the deposit.
Audits or reviews	• *The results of any audits or reviews of Ore Reserve estimates.*	• No external audits have been carried out.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage.* • *It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	• It is the Competent Person's view that the quality and accuracy of the used modifying factors are in a good level. • No mining has taken place at this deposit to allow reconciliation with production data. • The relative accuracy of the Ore Reserve estimate is reflected in the reporting of the Ore Reserves as per the guidelines of the JORC Code (2012 Edition).