Sotkamo Silver
Silver Mine

BANKABLE
FEASIBILITY STUDY
Update

May 28th 2014

Report FINAL rev 03
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ABBREVIATIONS

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<tr>
<td>ABA</td>
<td>Acid base accounting</td>
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<tr>
<td>ASL</td>
<td>Above Sea Level</td>
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<td>A-lab</td>
<td>Sample analysing laboratory</td>
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<tr>
<td>AVI</td>
<td>Regional State Administrative Agency</td>
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<tr>
<td>BAT</td>
<td>Best Available Techniques</td>
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<tr>
<td>CCR</td>
<td>Central Control Room</td>
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<tr>
<td>CPU</td>
<td>Central Processing Unit</td>
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<tr>
<td>CSS</td>
<td>Close Side Setting</td>
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<tr>
<td>DCS</td>
<td>Distributed Control System</td>
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<tr>
<td>ELY</td>
<td>Centre for Economic Development, Transport and the Environment</td>
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<tr>
<td>G-lab</td>
<td>Geological sample handling laboratory</td>
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<tr>
<td>GTK</td>
<td>Geological Survey of Finland (Geologian Tutkimuskeskus)</td>
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<tr>
<td>HP</td>
<td>High Pressure</td>
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<tr>
<td>HVAC</td>
<td>Heating, Ventilation and Air-Conditioning</td>
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<td>LP</td>
<td>Low Pressure</td>
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<tr>
<td>LPG</td>
<td>Liquefied Petroleum Gas</td>
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<td>MCC</td>
<td>Motor Control Cabinet</td>
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<td>MSDS</td>
<td>Material safety data sheets</td>
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<td>NAF</td>
<td>Non-acid forming</td>
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<td>OPC</td>
<td>Open Process Control</td>
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<tr>
<td>O/F</td>
<td>Over flow (cyclone)</td>
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<tr>
<td>O/S</td>
<td>Over size (screen, filtering)</td>
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<td>PAF</td>
<td>Potentially acid forming</td>
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<td>PPB</td>
<td>Process plant building</td>
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<td>ROM</td>
<td>Run of mine</td>
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<td>SRS</td>
<td>Safety Related System</td>
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<td>TGB</td>
<td>Tipasjärvi Greenstone Belt</td>
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<td>TMF</td>
<td>Tailings Management Facility</td>
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<td>TUKES</td>
<td>Finnish Safety and Chemicals Agency</td>
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Sotkamo Silver commissioned CTS Engtec Oy in October 2013 to prepare a Bankable Feasibility Study Update for the Silver Mine Project in Sotkamo municipality Finland. This report contains the documents prepared including production and balance calculations, process flow diagrams, mining schedule and Mineral Resource Estimate, site layouts as well as investment cost calculation, for the Silver Mine.

Mineral Recourse Estimate, pit modelling and mine planning has been carried out by Sotkamo Silver. Dr Jyrki Parkkinen, Qualified Person under “The European Federation of Geologists” (EFG) has prepared Mineral Resource Estimate. Pekka Lovén, MSc (Mining), MAusIMM (CP) has audited the Ore reserve statement as competent person.

The overall project scope has been to provide updated CAPEX/OPEX estimate with financial analysis based on updated Mineral Resource and Ore Reserve estimate in accordance with the guidelines of the JORC Code (2012), the mine evaluation and quotations of the main equipment, production and construction.

CTS Engtec Oy is an international, independent heavy industry engineering consultancy. Most consultants used in the preparation of this report have over 20 years of adequate professional experience. CTS personnel who are responsible for this report has visited the Silver Mine site.

This Bankable Feasibility Study update report was made in co-operation with Sotkamo Silver’s project team and CTS Engtec’s project team.

Kouvola, March 24th, 2014

CTS Engtec Oy

Pekka Veisto
Project Director
1 EXECUTIVE SUMMARY

1.1 Introduction

Sotkamo Silver (SoSi) is in the process of developing the Silver Mine project of Sotkamo Silver in Finland. The Silver Mine is planned to be producing silver and zinc concentrates during 8 years Life of Mine (LOM). The company has sales agreements with European smelters for both products.

Total 700 ktn of ore is planned to be quarried out from open pit mine and 2,6 Mtn from underground mine. After crushing and beneficiation on site the concentrates are transported by trucks to smelters, part of them to Gulf of Bothnia harbour and shipped to smelters.

The Silver Mine is located in Eastern Finland in the municipality of Sotkamo, approximately 475 km NE of Helsinki and 42 km from the municipality’s centre. Mine includes production decline and associated ventilation shaft down to 350 meters depth, thus ready for production. The site has a new 45 kV power line and part of mine area roads as existing infra. The local road 9005 is in reasonable condition to handle the traffic both during construction and operation phase. The mining concession area is at an elevation of approximately 220 m above sea level, and the surrounding area consists of smoothly undulating terrain, dominated by forest, bogs, and shallow lakes.

1.2 Environmental and Legal Rights

Based on legislation the Environmental Permit for the mine has been granted in April 2013. Mining concession application for the Sotkamo Silver Mine was submitted to the authorities in year 2010 and the Ministry of Employment and the Economy has granted the Mining Concession in April 2011, which covers total area of 371.44 hectares.

In February 2014 SoSi received the Mine Safety Permit from the Finish Safety and Chemicals Agency (Tukes).

The land area for the mining activities has been acquired in October 2011 and is fully owned by the company. The Building permit application has been submitted to local municipality.

1.3 Mineralogy and Mining

The Silver Mine deposit was found in 1980 and is located in the Tipasjärvi greenstone belt; southernmost part of the Tipasjärvi-Kuhmo-Suomussalmi (TKS) greenstone complex in eastern Finland. The deposit is lens shaped, dipping 65° to the southeast, plunging 60° degrees to the south-southwest, and extending to a depth of at least 550 m. The deposit has been first investigated by Kajaani Oy during 1980-1998, by Joint Venture between Outokumpu Mining and UPM-Kymmenen since 1988 and presently by SoSi since 2006.
The Mineral Resource Estimate (MRE) is based on database which has analyse results from 402 drill holes with total length of 55 052 m, a combination of 184 surface and 218 underground holes.

The Jorc code audited MRE is as follows:

Table 1  Sotkamo Silver mine mineral resource estimate at 50 g/t Ag cut off grade

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<tr>
<td>Inferred</td>
<td>1 340.0</td>
<td>76.0</td>
</tr>
</tbody>
</table>

The audited Ore Reserve Estimate is as follows:

Table 2  Sotkamo Silver Mine ore reserve (mineral reserve)

<table>
<thead>
<tr>
<th></th>
<th>Tonnage (kt)</th>
<th>Ag Grade (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Proven</td>
<td>1 800.0</td>
<td>98</td>
</tr>
<tr>
<td>Probable</td>
<td>1 536.0</td>
<td>106</td>
</tr>
<tr>
<td>Total</td>
<td>3 336.0</td>
<td>102</td>
</tr>
</tbody>
</table>

The ore reserve contains approx. 68 percentage of silver contained in the Measured and Indicated MRE. The ore reserves are mineable mineral resources and are included in total MRE.

The mineable deposit is approximately 450 m long, and extends presently approximately 500 m from top to bottom. Below the known deposit exists zone of exploration potential down to at least 1400 meters, this zone with low resistivity has been defined by geo-physical deep-penetrating Sampo-survey. This depth potential will be investigated and eventually developed for mining while production in upper parts has commenced.

The underground part of the deposit will be mined using mostly longitudinal bench and fill stoping, with rock fill, and will be mined upwards from the base of ore reserves. All operations is planned to be done by contractor.

In first two years to production of ore is limited to 350 000 tpa due Environmental Permit (EP) limitations, but will be increased to 450 000 tpa since EP is updated. The total amount of ore processed is 3 336 000 t from which 708 000 t is taken from open pit. The total amount of waste rock from open pit is 3 149 000 t and from underground 1 050 000 t during LOM.

1.4  Processing

After the ore is quarried from the pit or UG mine it is crushed by contractor and conveyed to beneficiation plant. The Silver Mine Pb-Ag-Zn ore type is amenable to pro-
cessing by grinding and froth flotation, achieving saleable grades of lead and zinc concentrates at acceptable recoveries at a grind size of approximately 80% passing 75 microns. The two stage grinding includes both rod and ball mills. The ore will be processed using silver-lead flotation consisting of rougher, scavenger and cleaning stages. The silver-lead tailings will be further treated in a flotation circuit consisting of rougher, scavenger and cleaning stages for zinc recovery. Zinc tailings will pass to a pyrite flotation section prior to final tailings disposal. The Silver-Gold-Lead and Zinc-Silver products will be thickened and filtered to suitable transportation moisture levels below 10%. Pyrite-Silver concentrate is handled similarly.

The Tailings Management Facility (TMF) will consist of the ordinary tailings pond 28 ha and three settling ponds and after this treatment the purified excess water will be discharged through wetland treatment back to nature. Process water is circulated back to plant.

In first phase the plant is constructed to handle 350 000 tpa of ore and will be upgraded to 450 000 tpa production level for the third year. The average Silver-Gold-Lead concentrate production is 1 944 tpa and Zinc-Silver concentrate production 5 032 tpa during LOM, but in first three years the Silver-Lead concentrate production is over 2 230 tpa due to higher Ag grade of the feed.

1.5 Capital and Operating Cost, Economic Evaluation

The total workforce employed by SoSi will be total 31 people. It’s estimated that outsourced mining activities and crushing plant will have 10 people multiplied by 5 shifts. The BFS contains investment cost estimate calculation (CAPEX) and operation cost estimate calculation (OPEX) with accuracy ±15%. Estimates are mainly based on inquiries. The total Initial Capital costs before start-up are 23.9 M€ as divided in table below.

<table>
<thead>
<tr>
<th>Discipline</th>
<th>Initial</th>
<th>Sustaining</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine</td>
<td>3 013</td>
<td>650</td>
</tr>
<tr>
<td>Crushing and beneficiation</td>
<td>12 744</td>
<td>6 912</td>
</tr>
<tr>
<td>Site General, incl. TMF</td>
<td>4 213</td>
<td>2 187</td>
</tr>
<tr>
<td>State support</td>
<td>-</td>
<td>1 851</td>
</tr>
<tr>
<td>Contingency and owners cost</td>
<td>3 933</td>
<td>549</td>
</tr>
<tr>
<td>TOTAL</td>
<td>23 903</td>
<td>8 447</td>
</tr>
</tbody>
</table>

| TOTAL PROJECT CAPEX               | 32 350 k€ |

The sustaining capital includes the investment costs of production increase to 450 000 tpa level in the second operation year, TFM area enlargement in the fourth operation year and mine closure costs.

The centre of Economic Environment and Employment has granted financial support to the mining project. At this moment total 1.851 M€ is still unused and taken into account in financial calculations.
A residual value of 8,058 M€ is used in financial calculations. The total accuracy of the cost calculation is approx. 15 % and the contingency of 5 % was used based on experience of similar projects.

The average operating costs in Sotkamo Silver mine was estimated to be 1 930 €/t of concentrate products as presented in table 4. Operating costs prior to start-up are considered as capital cost.

### Table 4 Operation cost breakdown (year 4)

<table>
<thead>
<tr>
<th>Cost break down</th>
<th>Total 1 000 €/y</th>
<th>€/tn of ore</th>
<th>€/tn of products</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining</td>
<td>10 210</td>
<td>22.48</td>
<td>1 117</td>
</tr>
<tr>
<td>Beneficiation and Crusher</td>
<td>5 391</td>
<td>11.87</td>
<td>590</td>
</tr>
<tr>
<td>Site general, environmental and utilities</td>
<td>2 038</td>
<td>4.49</td>
<td>223</td>
</tr>
<tr>
<td><strong>TOTAL</strong></td>
<td><strong>17 639</strong></td>
<td><strong>38.83</strong></td>
<td><strong>1 930</strong></td>
</tr>
</tbody>
</table>

Total operation costs vary between 38…39,50 €/t of ore during main production phase. Main drivers for changes are mining costs, which vary from 18,90 to 23,50 €/t and processing costs including crushing which vary between 11,50 and 13,50 €/t of ore.

In the first operation year the production cost per ore tonne are 18,91 € in mining and 13,33 €/ore tonne in processing. Site general cost per ore tonne are 5,87 €. The higher mining cost per ore tonne in year 4 are results from larger amounts of OP waste rock movement and more than doubling the drifting after first operation year.

The Sotkamo Silver mine economical analyse base case is based on silver price of 21.50 $/oz, gold price of 1 380 $/oz, zinc price of 2 651 $/tn and lead price of 2 380 $/t. Euro/USD exchange rate of 0.76 was used for product pricing.

The total incomes from products in the Base Case are presented on yearly basis in following table after smelter charge and penalties, as so called Net Smelter Return (NSR). The sum of smelter charges and penalties was calculated as total 20.55 M€ per LOM.

### Table 5 Total incomes from products 1000 € in Base Case

<table>
<thead>
<tr>
<th>Year</th>
<th>AgPb conc</th>
<th>Zn Conc</th>
<th>Total</th>
<th>€/tn of products</th>
</tr>
</thead>
<tbody>
<tr>
<td>year 1</td>
<td>24 102</td>
<td>2 765</td>
<td>26 867</td>
<td>4 663</td>
</tr>
<tr>
<td>year 2</td>
<td>23 900</td>
<td>3 613</td>
<td>27 513</td>
<td>4 027</td>
</tr>
<tr>
<td>year 3</td>
<td>27 495</td>
<td>3 958</td>
<td>31 452</td>
<td>4 128</td>
</tr>
<tr>
<td>year 4</td>
<td>25 151</td>
<td>5 322</td>
<td>30 473</td>
<td>3 334</td>
</tr>
<tr>
<td>year 5</td>
<td>18 787</td>
<td>4 356</td>
<td>23 143</td>
<td>3 154</td>
</tr>
<tr>
<td>year 6</td>
<td>18 485</td>
<td>4 387</td>
<td>22 871</td>
<td>3 112</td>
</tr>
<tr>
<td>year 7</td>
<td>17 752</td>
<td>3 992</td>
<td>21 744</td>
<td>3 210</td>
</tr>
<tr>
<td>year 8</td>
<td>11 148</td>
<td>3 085</td>
<td>14 232</td>
<td>2 8499</td>
</tr>
<tr>
<td><strong>TOTAL</strong></td>
<td>166 820</td>
<td>31 476</td>
<td>198 296</td>
<td>3 3553 avg</td>
</tr>
</tbody>
</table>

Capital and operating cost estimates were generated and inputted to an economic model along with other economic inputs including product prices and exchange rates. Production data from the mining plan were also input.
An annualized cash flow position is shown in figure 1 below.

The key financial numbers of Sotkamo Silver project base case are presented in table below based on discount rate of 8% (after tax).
The updated Bankable Feasibility Study (BFS) has demonstrated both the technical and economic viability of the Sotkamo Silver mining project.

Table 6  Financial parameters

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Units</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore Mined LOM</td>
<td>kt</td>
<td>3,350</td>
</tr>
<tr>
<td>Zinc Recovered</td>
<td>t</td>
<td>21,800</td>
</tr>
<tr>
<td>Lead Recovered</td>
<td>t</td>
<td>9,400</td>
</tr>
<tr>
<td>Gold Recovered</td>
<td>oz</td>
<td>28,250</td>
</tr>
<tr>
<td>Silver Recovered</td>
<td>oz</td>
<td>9,500</td>
</tr>
<tr>
<td>Total Tax LOM (base case)</td>
<td>€M</td>
<td>9</td>
</tr>
<tr>
<td>Total Operating Costs LOM</td>
<td>€M</td>
<td>123</td>
</tr>
<tr>
<td>Total Capital Expenditure LOM</td>
<td>€M</td>
<td>32</td>
</tr>
</tbody>
</table>

Silver Price Forecast 15 $/Oz

<table>
<thead>
<tr>
<th>Parameter</th>
<th>€M</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Total Revenue (NSR) LOM</td>
<td></td>
<td>156</td>
</tr>
<tr>
<td>Operating Free Cash Flow LOM</td>
<td></td>
<td>32</td>
</tr>
<tr>
<td>EBITDA first 4 years average/y</td>
<td></td>
<td>7</td>
</tr>
<tr>
<td>EBITDA first 4 years average</td>
<td>%</td>
<td>31</td>
</tr>
<tr>
<td>NPV 8% (before tax)</td>
<td>€M</td>
<td>1</td>
</tr>
<tr>
<td>IRR (before tax)</td>
<td>%</td>
<td>9</td>
</tr>
<tr>
<td>Payback Period</td>
<td>Years</td>
<td>8</td>
</tr>
</tbody>
</table>

Silver Price Forecast 21.5 $/Oz

<table>
<thead>
<tr>
<th>Parameter</th>
<th>€M</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Total Revenue (NSR) LOM</td>
<td></td>
<td>198</td>
</tr>
<tr>
<td>Operating Free Cash Flow LOM</td>
<td></td>
<td>75</td>
</tr>
<tr>
<td>EBITDA first 4 years average/y</td>
<td></td>
<td>14</td>
</tr>
<tr>
<td>EBITDA first 4 years average</td>
<td>%</td>
<td>46</td>
</tr>
<tr>
<td>NPV 8% (before tax)</td>
<td>€M</td>
<td>30</td>
</tr>
<tr>
<td>IRR (before tax)</td>
<td>%</td>
<td>35</td>
</tr>
<tr>
<td>Payback Period</td>
<td>Years</td>
<td>2</td>
</tr>
</tbody>
</table>

Silver Price Forecast 28 $/Oz

<table>
<thead>
<tr>
<th>Parameter</th>
<th>€M</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Total Revenue (NSR) LOM</td>
<td></td>
<td>241</td>
</tr>
<tr>
<td>Operating Free Cash Flow LOM</td>
<td></td>
<td>117</td>
</tr>
<tr>
<td>EBITDA first 4 years average/y</td>
<td></td>
<td>17</td>
</tr>
<tr>
<td>EBITDA first 4 years average</td>
<td>%</td>
<td>53</td>
</tr>
<tr>
<td>NPV 8% (before tax)</td>
<td>€M</td>
<td>59</td>
</tr>
<tr>
<td>IRR (before tax)</td>
<td>%</td>
<td>61</td>
</tr>
<tr>
<td>Payback Period</td>
<td>Years</td>
<td>1.5</td>
</tr>
</tbody>
</table>
2 INTRODUCTION

2.1 General

The Sotkamo Silver, (SoSi) commissioned CTS Engtec Oy (CTS) in October 2013 to prepare a Bankable Feasibility Study Update (BFS) for the Sotkamo Silver Mine Project (Project). SoSi is responsible for the whole BFS report covering parts of the BFS which are not included in CTS’ scope.

CTS is an international, independent heavy industry engineering consultancy. Most consultants used in the preparation of this report have over 20 years of adequate professional experience.

2.2 Terms of reference and Scope of BFS Update

The overall project scope has been to provide updated CAPEX/OPEX estimate with financial analysis based on updated Mineral Resource and Ore Reserve estimate in accordance with the guidelines of the JORC Code (2012), the mine evaluation and quotations of the main equipment, production and construction.

The scope commands the incorporation of process design, mine infrastructure and environmental studies, as well as associated operating costs for a 350 000 – 450 000 tpa operation to an accuracy of +/- 15%.

Sotkamo Silver mine will have concentration capacity of 350 000 – 450 000 t/a. The run of mine will be mined from the open pit and the underground mines. The crushing plant will have capacity of 250 t/h and the concentrator plant capacity of 45-55 t/h.

The silver mine concentration process will produce Ag-Au-Pb concentrate ca 1 550 t/a, Zn concentrate ca 3 850 t/a and S concentrate ca 9 150 t/a.

The total investment cost is 32.4 M€.
3 PROPERTY DESCRIPTION

The Silver Mine Project is located in the municipality of Sotkamo in eastern Finland, 475km from Helsinki, as shown in Figure 2.1. Finnish national grid coordinates define the deposit at approximately E 4453 500 N 7093 600, located on map sheet 4322 12.

The deposit is at an approximate elevation of 220 m above sea level.

The Reservation area covers a total of 265km² and the Mine Concession area 371 ha. Reservation area covers whole TGB including several exploration targets.
4 DEPOSIT GEOLOGY

4.1 Geological setting

The Silver Mine deposit of Sotkamo Silver in eastern Finland is located in the Tipasjärvi Archaean greenstone belt (TGB), in the southern part of a north-south-trending about 200 km long and from 3 to 5 km wide Tipasjärvi-Kuhmo-Suomussalmi greenstone complex that is a part of the Archaean Kianta terrain. The deposit is characterized by a low content of sulphide ore minerals, 5 per cent on average, disseminated and vein-type ore textures, and the very pronounced geochemical alteration zones typical of epithermal precious metal deposits of young volcanic areas. The environment includes a few minor deposits of banded iron-formations and layers of massive pyrite, which have been targets of repeated exploration activities by mining companies. In the 2000’s the Geological Survey of Finland (GTK) performed a thorough study on geology, geochemistry and geophysics and evaluated the prospectivity of the Tipasjärvi greenstone belt (Figure 3).

4.2 Property geology

The stratigraphic sequence in the vicinity of the Silver Mine deposit area comprises the Koivumäki Formation, as the oldest stratigraphic unit and overlying volcanic Vuoriniemi, Kallio and Taivaljärvi Formations and the sedimentary Kokkoniemi Formation. The deposit itself is situated in the Taivaljärvi Formation. Some details of the formations from youngest to oldest units:

Kokkoniemi Formation – mica schists of metasedimentary origin, maximum age 2.75 Ga.

Taivaljärvi Formation – generally comprising quartz-sericite schists, interpreted as representing metamorphic equivalents of volcanic breccias, and layered tuffs and tuffites indicating shallow water or subaerial eruption. U-Pb isotopic ages of 2798±2 Ma indicate the termination of volcanic succession of the greenstone belt. The Silver Mine deposit is located in the middle of the Formation. An extensive 5-25 m kyanite-quartz rock layer is present approximately 100 m stratigraphically above the mineralised zone, interpreted as representing argillic alteration of a primary cap rock. The felsic rocks between the deposit and the kyanite-quartz rock are hydrothermally altered with plagioclase replaced by K feldspar and sericite. Alteration is not recorded stratigraphically above the kyanite-quartz rock. The upper most part of the formation comprises sulphide-graphite schists and BIFs, possibly indicating a short break in volcanic activity. Overall thickness of the formation is 500m. Deformation of the formation caused the development of an antiform structure with the deposit located in the eastern limb of the antiform. The felsic pyroclastic metavolcanic rocks of the Taivaljärvi Formation are underlain by homogeneous felsic porphyry which grades transitionally into plutonic rocks of TTG series.

Kallio Formation – ultramafic metavolcanics generally comprising theoleiitic basalts, thin interlayers of banded iron formations (BIF) and thin komatiitic lavas.

Vuoriniemi Formation – mafic to felsic metavolcanics dominated by amphibolites, which are interpreted as representing mafic lavas, tuffs and gabbroic sills.
Koivumäki Formation – felsic to intermediate metavolcanics dated to 2828±3 Ma

Figure 3  Regional geology of the TGB around the Silver Mine deposit by GTK

The Map width is approximately 18 km.
4.3 Deposit Geology

The Silver Mine deposit is located in the upper part of the Taivaljärvi Formation. The deposit is lens shaped, dipping 65° to the southeast, plunging 60° degrees to the south-southwest, and extending to a depth of at least 500m.

Four ore bodies make up the deposit with different base and precious metal ratios. The ore bodies are sub-parallel to primary stratigraphic bedding and identifies from the footwall upward as, D, C, B and A. Layers vary laterally in thickness and locally join up with neighbouring layers (Figure 4).

- Ore body A – Ag
- Ore body B – Ag-Zn-Pb-Au
- Ore body C – Ag-Zn-Au
- Ore body D – Ag-Zn-Pb-Au-Cu
4.4 The Silver Deposit

The outcrop of the ore body is 400 m long and from 5 to 110 m wide, averaging 40 m (Figure 5). The ore body dips 65° to southeast, plunges 60° to south-southwest, and drilling indicated ore extends to a depth of at least 500 m.

The mineralized section is composed of several parallel zones, and the main zones were named from the footwall upward the D, C, B, and A ores with specific compositions and metal ratios. However, the ore zones vary laterally in thickness and locally join up with the neighbouring zones. The interspaces are not totally barren but contain a weak dissemination of ore minerals, and the mineable ore consists of the whole mineralised section.

![Figure 5](image.png)

Figure 5 Surface plane and a vertical section of the Silver Mine deposit

To the west of the deposit the footwall consists of a 150m wide zone of quartz-sericite-biotite schist with bands rich in garnet, tremolite, ankerite, sulphides and accessory chlorite, epidote, cordierite and rutile.
4.5 Mineralogy of the Deposit

4.5.1 Carbonates

Carbonates occur in intersecting quartz-carbonate-sulphide veins and conformable bands of the A and B ores, in calc-silicate interlayers of the D ore and in the footwall. In the A and B ores the content of carbonates varies from 6 to 8 per cent and in the C and D ores from 1 to 2 per cent. According to microprobe analyses the carbonates are commonly manganoankerites with MnO content of 3 to 6 wt per cent. The FeO content of manganoankerite increases from the A ore toward the footwall: in the A ore FeO is 5.27 per cent, in the D ore, 6.42 per cent, and in the footwall, 7.55 per cent; the MgO content decreases correspondingly. Mangano-calcite with 3 per cent MnO is associated with ore minerals in the quartz-carbonate veins in the B ore.

4.5.2 Ore minerals

The most important ore minerals are freibergite, dyscrasite, pyrargyrite, native silver, electrum, sphalerite, galena, and chalcopyrite. The following minerals have also been identified: pyrite, pyrrhotite, arsenopyrite, cubanite, covellite, gudmundite, acanthite, miiargyrite, freieslebenite, bournonite, scheelite, native Sb, and native Bi. The grain size of the common sulphides varies from 0.1 to 0.5 mm and that of the Ag-bearing minerals from 0.01 to 0.1 mm. Galena and the associated Ag minerals are more abundant in the stratigraphically uppermost ore zones, where they occur in quartz-carbonate veins and bands. The sulphide content in the ore varies between 5 and 8 per cent, and more than 50 per cent of the sulphides are composed of pyrite and pyrrhotite. Silver mineralogy in the whole deposit stays basically similar, > 95% of silver occurs in sulphides and antimonides, galena may contain in silver-rich parts ca 0,1 % silver, in the ores with highest silver contents there exists also silver and silver-gold alloy, amount of metallic silver is small and upper cut-off 700 g/t for silver is used in our resource estimates; thus the possible nugget effect is under control.

The hanging wall (to the east of the deposit) consists of quartz-sericite schists with minor cordierite and iron sulphides, and accessory sphalerite and galena.

4.6 Deposit type

Felsic volcanic breccias and coarse fragmental volcanic rocks, typical for volcanic centers, characterize the Silver Mine volcanic succession. In the Silver Mine area these “Mill rocks”, acid volcanic rocks containing ejecta fragments as evidence from violent volcanic eruption, are exposed in several outcrops. These rocks containing fragments and sulphides are often host rocks for high-value deposits of Zn-Pb-Cu-Ag and Au in Archaean Greenstone belts, thus named “mill-rocks” while one often can hear mill sound while looking at these rocks. The primary volcanic centers are high heat-flow areas, and hence probable targets for post-eruption hydrothermal activity. Although metamorphosed and deformed, the association of the ore deposit with felsic pyroclastic rocks, the stratiform setting and the compositional alteration features around the ore deposit refer to a low-sulphur epithermal deposit associated with a geothermal system.
4.7 Geophysical approach

Geophysical deep-penetrating Sampo-survey indicates that the mineralized zone continues down to 1.5 or even two kilometres (Figure 6 and Figure 7). Survey was conducted in the spring 2013 by GTK (Niskanen, 2012) and interpretation of the survey results and synthesis of geophysics and structural observations of the depth extension was prepared by Jyrki Parkkinen (2012, 2013).

Figure 6  Original interpretation of the GTK geophysical Sampo surveys by Niskanen 2011-12
Figure 7  Further 3D interpretations of the Sampo deep electromagnetic surveys

A. 1100 Ohm’s 3D surface with –600 m grid and some mineralised zones in drill holes
B. Original anomaly interpretation by Matti Niskanen 2011.
C. Adjustment of the Sampo soundings to the layered structure of the deposit by Parkkinen 2013: with the application of geostatistics, the original depth of -1 400 m obtained by Niskanen could be extended to the depth of –2 000 m.
D. The same in horizontal projection: probable intersecting ore zones (in magenta) were recognized (shear zone)

Based on geophysical survey the depth continuation of formation containing Silver deposit continues at least to a depth of 1 400 meters indicating huge future potential.

More detailed information is presented in Appendix 1 Description of Silver Deposit with implications for further exploration.
5  EXPLORATION WORK AT SILVER MINE

The deposit was initially discovered in 1980. Since the discovery of the deposit there have been a number of phases of exploration. Works carried out range from remote sensing techniques and geochemical works through to intrusive drilling investigations and development of underground drives. After discovery of the deposit Kajaani Oy, a local paper company having exploration department investigated the target by geochemical soil sampling, diamond drillings and geophysics since 1980 until 1988.

Further information is presented in Appendix 2 Exploration targets of Sotkamo Silver on the Tipasjärvi Greenstone Belt

5.1  Taivalhopea

In 1988 the prospect was transferred to a joint venture between UPM-Kymmene (Kajaani Oy was merged to UPM) and Outokumpu Mining Oy, called “Taivalhopea”. It performed in 1988 - 1991 additional inventory drillings, concentration tests, planning of mine and concentrating plant and completed a feasibility study. In order to investigate ore quality and side rock properties the JV also constructed a 2 600 meters long production decline down to the depth level of 350 meters and opened a ventilation shaft to the level of 340 meters. Due to low metal prices in 1991 the project was put into hold.

5.2  Sotkamo Silver

In 2005 Silver Resources Oy (at present Sotkamo Silver Oy) was established. The company acquired the mineral rights and all previous investigation material. The company performed additional surface and underground drillings, which presently totals to ca 53 km.

5.3  2010-2011 Sotkamo Silver exploration work

In 2010 a total of 11 surface drill holes totalling 1 750.40m was completed and followed up in 2011 by 28 underground holes (2 398.15m) and 26 surface holes totalling 2 112.80m. Surface drilling undertaken in 2010 and 2011 produced 52mm diameter core; underground drilling yielded core of 62mm diameter.

5.4  Sotkamo Silver, Drilling Campaign 2013

5.4.1  Field work and assaying

Twenty five sludge drill holes and one RC hole were run from the mine decline of the Sotkamo Silver Mine in 2013 to complement diamond drilling programs. First nine of sludge holes, numbered TS1—TS28, were drilled by Tolarock Oy with Tamrock Solo longhole-drill and with a 70 mm hole diameter. The rest of holes were drilled by Taipojärvi Oy with Sandvik DL 431-C longhole-drill and with a 64 mm hole diameter. The RC hole was drilled by Sotkamon Porakaivo Oy with a Casagrande C6 drill rig and with a 90 mm hole diameter. Three holes from the campaign plan remained undrilled, namely TS25-TS27.
Sampling was undertaken by Sotkamo Silver personnel; drill sludge was led into a bucket where sample was taken after manual homogenisation.

Drilling and sampling were supervised by Mine Planner Pauli Kokkonen who has 35 years’ experience in steering underground mining operations.

Sludge holes were directed according to northing vertical sections (N90E or N90W) while the RC hole was directed obliquely towards N60E. Engineer Jari Pätsi, Mitta Oy, surveyed the drilling sites and azimuthal and plunge directions of all drill holes.

Sludge sampling was carried in 1.503 m long portions and RC sampling in 3 m portions. Samples were analysed by Kemian Tutkimuspalvelut Oy (Anttila & Mäkelä, 2013). Silver, lead, zinc, sulphur and antimony were assayed using ICP-OES analysis. Gold was assayed by acid digestion followed by liquid-liquid extraction and AAS. Details of sample preparation, analytical methods, assaying results, and quality control are described in the Appendix 3 Summary of the methods and QAQC.

Four old ore samples, analysed previously by Labtium, were used as reference samples during the analytical process. The reference samples were treated and analysed among the regular test samples. Each reference sample were analysed 2-4 times during the project. Comparison of the results with those of Labtium show good correspondence. Sulphur and antimony assays by Labtium were missing.

Information about the within-batch precision was obtained by running duplicate analyses in each sample batch. In each batch 1-3 samples were analysed in duplicate. Control charts show differences from 5 % to 9 % on an average which is acceptable. Only silver charts show a few several peaks probably resulting from the nugget phenomenon causing some unpredictability in results.

As shown in Section figures Campaign results were fairly good. They are in agreement with the existing models for mining with minor adjustments and they make it possible to upgrade the volume and classification of the mineral resource. Sotkamo mine decline and the Drilling Campaign 2013 drill holes are presented in Figure 8.
Figure 8  The Silver Mine decline and the Drilling Campaign 2013 drill holes

Red intersections display mineralization with Ag-content > 50 g/t, best values are given in Table 7.

Ag-content (g/t) in Figure 8
- 0 – 10 blue
- 10 – 30 green
- 30 – 50 yellow
- > 50 red
5.4.2 Quality assurance

According to the hole survey drill hole actual azimuthal orientations differed from plan figures on an average less than +/- 5 degrees, which is acceptable. But the orientation of TS1 was 77° and the orientation of TS28 was 113.8° which means an about 13 degree difference. These holes did not reach their targets but, of course, they added a little value when ensuring already existing results.

As described above, the quality control arrangements of Kemian Tutkimuspalvelut Oy were acceptable.

5.4.3 Description of results

Campaign results are given in three formats: in Best Intersections Table below, in sectional figures on following pages, and in assay reports by Kemian Tutkimuspalvelut (CRS) in the Appendix 3.

Table 7 Sotkamo Silver Drilling Campaign 2013 Best Intersections

<table>
<thead>
<tr>
<th>Hole_Id</th>
<th>From</th>
<th>To</th>
<th>Length m</th>
<th>Ag</th>
<th>Au</th>
<th>Zn</th>
<th>Pb</th>
<th>S</th>
<th>Sb g/t</th>
</tr>
</thead>
<tbody>
<tr>
<td>TRC3</td>
<td>60.0</td>
<td>69.0</td>
<td>9.0</td>
<td>25.0</td>
<td>2.00</td>
<td>3.4</td>
<td>0.7</td>
<td>3.9</td>
<td>478.7</td>
</tr>
<tr>
<td>TRC3</td>
<td>69.0</td>
<td>99.0</td>
<td>30.0</td>
<td>107.8</td>
<td>0.16</td>
<td>0.7</td>
<td>0.1</td>
<td>2.1</td>
<td>33.2</td>
</tr>
<tr>
<td>TS12</td>
<td>7.3</td>
<td>23.8</td>
<td>16.5</td>
<td>96.7</td>
<td>0.30</td>
<td>0.7</td>
<td>0.3</td>
<td>1.8</td>
<td>53.3</td>
</tr>
<tr>
<td>TS13</td>
<td>14.6</td>
<td>25.6</td>
<td>11.0</td>
<td>161.2</td>
<td>1.02</td>
<td>1.7</td>
<td>0.7</td>
<td>2.4</td>
<td>66.3</td>
</tr>
<tr>
<td>TS15</td>
<td>34.8</td>
<td>51.2</td>
<td>16.5</td>
<td>97.8</td>
<td>2.16</td>
<td>0.4</td>
<td>0.6</td>
<td>0.8</td>
<td>150.3</td>
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<td>TS2</td>
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<td>26.0</td>
<td>4.6</td>
<td>497.3</td>
<td>1.23</td>
<td>4.0</td>
<td>1.8</td>
<td>0.7</td>
<td>457.0</td>
</tr>
<tr>
<td>TS3</td>
<td>45.9</td>
<td>55.1</td>
<td>15.3</td>
<td>164.3</td>
<td>1.14</td>
<td>1.8</td>
<td>0.7</td>
<td>2.0</td>
<td>206.5</td>
</tr>
<tr>
<td>TS4</td>
<td>39.8</td>
<td>55.1</td>
<td>15.3</td>
<td>268.7</td>
<td>0.26</td>
<td>0.3</td>
<td>0.3</td>
<td>2.5</td>
<td>104.8</td>
</tr>
<tr>
<td>TS5</td>
<td>42.8</td>
<td>55.1</td>
<td>12.2</td>
<td>273.7</td>
<td>0.37</td>
<td>0.2</td>
<td>0.2</td>
<td>2.2</td>
<td>67.9</td>
</tr>
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<td>TS6</td>
<td>47.4</td>
<td>55.1</td>
<td>7.7</td>
<td>266.9</td>
<td>0.42</td>
<td>0.5</td>
<td>0.4</td>
<td>2.6</td>
<td>63.8</td>
</tr>
<tr>
<td>TS8</td>
<td>30.6</td>
<td>35.2</td>
<td>4.6</td>
<td>308.3</td>
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<td>0.9</td>
<td>0.4</td>
<td>2.7</td>
<td>101.0</td>
</tr>
</tbody>
</table>
6 MINERAL RESOURCE ESTIMATES

6.1 Overview


For more detailed information of the previous Mineral Resource Estimates is presented in the chapter 13 of the Taivaljärvi (Silver Mine) Mine Bankable Feasibility Study 2012 by Wardell-Armstrong.

6.2 Sample database

The database, Taival2013.mdb in Microsoft Access format, contains drill hole collar surveys, downhole surveys, assays, rock quality designation (RQD) and lithology details.

Total drilling so far is 55 052 metres consisting of 402 drill holes, a combination of 184 surface and 218 underground holes. Of the latter 189 are diamond drill holes, 28 sludge holes and one RC-hole. 14 718 samples have been assayed for silver, gold, copper, lead, and zinc; 13 817 have been assayed for manganese, 3 682 for sulphur. A smaller number of samples have been assayed for arsenic, cadmium, cobalt, chromium, iron, molybdenum, nickel and antimony.

6.3 Geological interpretation

The mineralisation at Silver Mine comprises mineralised veins and lower grade mineralised host rock as described by Papunen et al. (1989). The hanging wall contact is quite sharp while the footwall contact is a lot more diffuse with the mineralisation gradually zoning out. In order to wireframe the deposit, modelling was undertaken based on the distribution of silver and using the nominal cut-off grade of 50 g/t Ag.

A block diagram of the Silver Mine deposit looking west is presented in Figure 9; the mineralised body is marked with red, blue indicates Ag values over 50 g/ton; local coordinate system.
Figure 9  A block diagram of the Silver Mine deposit
6.3.1 Structural analytics

Based on the detailed tectonic observations obtained from the mine decline and geochemistry of the ore elements Parkkinen (2010, 2012) interpreted the structure of the Silver Mine sequence as a pack of flexural slip z-folds, originated from one single layer. Recent results of diamond and percussive drilling campaigns do not support this interpretation. Instead, they seem to support the original idea of four layers modified with the possible existence of altogether six layers. Deformation would have been limited to intense small scale folding and pinch and swell structures inside the deposit. Probably simultaneously to the folding a generation of quartz veins, in places mineralised, was formed. This phase was followed by brittle deformation with another generation of barren quartz veins. No remarkable faults have been recognized, except indirectly via aerial geophysics, but the Silver Mine pack of layers may be bordered by sub-vertical faults, especially at the SE side of the pack.

According to a structural analysis, focused to the geometry of bedding, foliation and fold axis, and based on observations by geologists in the 2.5 km long mine decline in 1990-91, the structural elements give a uniform and simple geometry illustrated in a stereographic projection (Figure 12). It seems that the latest foliation phase (local S2), while also overlapping with bedding, forms an average a 10° angle clockwise to bedding. This might indicate that folding within the deposit is sinistral.

From the ore geological point of view, the most important features are the pervasive lineation that controls the deposit similarly to several gold deposits in Finland (Pampalo, Orivesi, Pahtavaara), and also planar controls as in the Kittilä Gold Deposit. The main geometrical controls are illustrated in Figure 12, Figure 13 and Figure 14.

6.3.2 Solid (wireframe) modelling

To construct orebody solids, three approaches were applied. First, a global 3D lithology model, bound to drill hole samples was generated. For the model, category geostatistics for several rock types were run and a category block model was generated (Figure 10).

A block diagram of the Silver Mine deposit looking east is presented in Figure 11.

Second, an initial semi-global model comprising the whole deposit area was generated. This model based on the solid called Envelope.dtm, that covered the deposit, and on geostatistical parameters applied to interpolate block grades inside the Envelope (Figure 13 and Figure 14).

Third, hard grade boundaries were used to outline orebodies on vertical sections at ten meter intervals and in places at five meter intervals. First and second approach “preliminary” outlines were used as guiding lines in problematic cases.

Nominal cut-off grade, 50 g/t Ag, was used but with a tendency to accept silver grades of 30 to 50 g/t in cases where the grade pattern appeared vague or gradual.
Figure 10  3D model of the Silver Mine deposit

Measured and indicated resource is red; blue is the extension of mineralization inferred with “Sampo” soundings; green lines indicate drill-holes. Production tunnel has been mined down to 350 meters level (Figure 10).
A block diagram of the Silver Mine deposit looking east; the six ore layers indicated with different colours; local coordinate system. Production tunnel tangents ore layers in the footwall (Figure 11).
Figure 12  Stereographic diagram of planar and linear elements measured in the Decline. Local coordinate system.

Figure 13  A,B Deposit enveloping solid (Envelope.dtm, transparent blue), planar structural controls of the Deposit (green and brown), Mine Decline (brown) and drill holes (black).
Figure 14  Blockmodel inside the Envelope.dtm solid, front section Y=11340, horizontal section Z=−100.
6.4 Database compilation

Updated Global Database Statistics is presented in Table 2 below.

Table 8 Global statistics of all sample assays

<table>
<thead>
<tr>
<th>Variable</th>
<th>Ag (ppm)</th>
<th>Au (ppm)</th>
<th>Cu (ppm)</th>
<th>Mn (ppm)</th>
<th>Pb (ppm)</th>
<th>S (ppm)</th>
<th>Zn (ppm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Number of samples</td>
<td>14707</td>
<td>14707</td>
<td>14421</td>
<td>13816</td>
<td>14705</td>
<td>6127</td>
<td>14707</td>
</tr>
<tr>
<td>Minimum value</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Maximum value</td>
<td>4700</td>
<td>28.62</td>
<td>91650</td>
<td>44000</td>
<td>99999</td>
<td>189000</td>
<td>114000</td>
</tr>
<tr>
<td>Mean</td>
<td>34.80197</td>
<td>0.11804</td>
<td>75.24346</td>
<td>1653.12845</td>
<td>1342.187277</td>
<td>13555.82716</td>
<td>3291.644658</td>
</tr>
<tr>
<td>Median</td>
<td>12</td>
<td>0.022</td>
<td>36</td>
<td>1236</td>
<td>499</td>
<td>11600</td>
<td>1250</td>
</tr>
<tr>
<td>Variance</td>
<td>9 245.02</td>
<td>0.27</td>
<td>71 648.79</td>
<td>2 619 699.06</td>
<td>10 817 058.41</td>
<td>119 560.975</td>
<td>92 156 145.77</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>96.15</td>
<td>0.52</td>
<td>267.67</td>
<td>1 618.55</td>
<td>3 288.93</td>
<td>10 628.12</td>
<td>6 495.86</td>
</tr>
<tr>
<td>Coefficient of variation</td>
<td>2.76</td>
<td>4.31</td>
<td>3.56</td>
<td>0.98</td>
<td>2.45</td>
<td>0.78</td>
<td>1.97</td>
</tr>
<tr>
<td>Skewness</td>
<td>14.52</td>
<td>25.90</td>
<td>10.94</td>
<td>5.05</td>
<td>10.69</td>
<td>6.03</td>
<td></td>
</tr>
<tr>
<td>Kurtosis</td>
<td>460.09</td>
<td>1 037.07</td>
<td>2 495.03</td>
<td>69.90</td>
<td>205.19</td>
<td>27.96</td>
<td>58.79</td>
</tr>
</tbody>
</table>

6.5 Data processing

6.5.1 Statistical analysis

Updated Mineralisation Intercept Statistics (or sample and composite statistics inside ore solids) is presented in Tables 3 and 4 below.

Table 9 Statistics of all sample assays inside ore solids

<table>
<thead>
<tr>
<th>Variable</th>
<th>Ag (ppm)</th>
<th>Au (ppm)</th>
<th>Cu (ppm)</th>
<th>Mn (ppm)</th>
<th>Pb (ppm)</th>
<th>S (ppm)</th>
<th>Zn (ppm)</th>
</tr>
</thead>
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<td>4301</td>
<td>4195</td>
<td>4054</td>
<td>4299</td>
<td>1615</td>
<td>4301</td>
</tr>
<tr>
<td>Minimum value</td>
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<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Maximum value</td>
<td>4700</td>
<td>28.62</td>
<td>122800</td>
<td>44000</td>
<td>99999</td>
<td>189000</td>
<td>114000</td>
</tr>
<tr>
<td>Mean</td>
<td>85.32</td>
<td>0.27</td>
<td>122.97</td>
<td>2 028.28</td>
<td>2 865.03</td>
<td>17 854.8</td>
<td>27 577.1</td>
</tr>
<tr>
<td>Median</td>
<td>43</td>
<td>0.1</td>
<td>57</td>
<td>1505</td>
<td>1370</td>
<td>15300</td>
<td>3140</td>
</tr>
<tr>
<td>Variance</td>
<td>24 879.5</td>
<td>0.8</td>
<td>124 297.1</td>
<td>4 106 715.7</td>
<td>27 800 062.0</td>
<td>162 867 818.4</td>
<td>93 399 977.9</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>157.7</td>
<td>0.9</td>
<td>352.6</td>
<td>5 276.2</td>
<td>9 627.4</td>
<td>3 482.6</td>
<td>9 664.4</td>
</tr>
<tr>
<td>Coefficient of variation</td>
<td>1.8</td>
<td>3.3</td>
<td>5.6</td>
<td>7.5</td>
<td>4.0</td>
<td>4.3</td>
<td></td>
</tr>
<tr>
<td>Skewness</td>
<td>9.7</td>
<td>16.4</td>
<td>19.7</td>
<td>5.6</td>
<td>7.5</td>
<td>4.0</td>
<td></td>
</tr>
<tr>
<td>Kurtosis</td>
<td>200.5</td>
<td>396.4</td>
<td>569.8</td>
<td>7.0</td>
<td>94.6</td>
<td>38.6</td>
<td>29.7</td>
</tr>
</tbody>
</table>

Sichel-t value 84.18

Table 10 Statistics of 1 m composite samples inside ore solids and inside Blockmodel 2013_8e

<table>
<thead>
<tr>
<th>Variable</th>
<th>Ag (ppm)</th>
<th>Au (ppm)</th>
<th>Cu (ppm)</th>
<th>Mn (ppm)</th>
<th>Pb (ppm)</th>
<th>S (ppm)</th>
<th>Zn (ppm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Number of samples</td>
<td>6 628</td>
<td>6 628</td>
<td>6 473</td>
<td>6 202</td>
<td>6 627</td>
<td>1 788</td>
<td>6 628</td>
</tr>
<tr>
<td>Minimum value</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Maximum value</td>
<td>1 200</td>
<td>12</td>
<td>9200</td>
<td>44000</td>
<td>90 000</td>
<td>158 564</td>
<td>66 000</td>
</tr>
<tr>
<td>Mean</td>
<td>83.90</td>
<td>0.26</td>
<td>121.83</td>
<td>2 048.54</td>
<td>2 983.84</td>
<td>17 617.8</td>
<td>6 440.68</td>
</tr>
<tr>
<td>Median</td>
<td>47.0</td>
<td>0.02</td>
<td>62.0</td>
<td>1 521.6</td>
<td>15 800.0</td>
<td>3 400.0</td>
<td></td>
</tr>
<tr>
<td>Variance</td>
<td>14 504.7</td>
<td>0.4</td>
<td>88 344.3</td>
<td>3 500 168.0</td>
<td>26 294 717.0</td>
<td>144 367 725.1</td>
<td>76 253 203.7</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>82.15</td>
<td>0.24</td>
<td>1 941.6</td>
<td>2 688.42</td>
<td>2 913.75</td>
<td>17 095.69</td>
<td>6 158.54</td>
</tr>
<tr>
<td>Coefficient of variation</td>
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<td>2.4</td>
<td>4.0</td>
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<td>0.7</td>
<td>1.4</td>
<td></td>
</tr>
<tr>
<td>Skewness</td>
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<td>8.9</td>
<td>18.4</td>
<td>4.9</td>
<td>7.4</td>
<td>3.2</td>
<td>3.3</td>
</tr>
<tr>
<td>Kurtosis</td>
<td>28.3</td>
<td>118.5</td>
<td>602.0</td>
<td>63.6</td>
<td>93.4</td>
<td>27.0</td>
<td>17.1</td>
</tr>
</tbody>
</table>

Sichel-t value 84.18
6.6 Top cutting

All sample data contained within the ore zone wireframes were selected for further data processing. This was done using the Surpac procedure, Drillhole 3DM Intersection. This data was used to determine top cut-off grade values for Ag, Au, Pb and Zn.

Main emphasis was on log-normal probability plots (Figure 15), Top cut off grades are in g/t: Ag 1 200, Au 12, Pb 90 000, Zn 66 000.

As to silver, two other methods for top cut off determination were checked:

WAI (2012) made use of deciles and accepted the then applied top cut off value of 700 g/t for silver. This value represents the 99.6 decile of global silver samples and the 99.0 decile of silver samples inside ore solids. Respectively the presently used value, 1 200 g/t, represents the 99.9 decile of global silver samples and the 99.7 decile of silver samples inside ore solids. This criterion would support a lower top cut value for silver than 1 200 g/t. On the other hand, the Sichel-t value of (non-cut) silver samples inside ore solids is higher than sample mean. This would imply that even the top cut off value of 1 200 g/t might be low.

![Figure 15 Log-normal probability plot of silver samples inside ore solids.](image_url)

6.7 Missing assays

Absent values were converted to zero values.

6.8 Compositing

The composite length of 1 m was applied.

6.9 Data processing summary
Top cuts have been applied of 1 200 g/t Ag, 12 g/t Au, 90 000 g/t Pb and 66 000 g/t Zn to remove outlier grades and normalise the dataset.

6.10 Variography

Variograms were produced for logarithmic grade values. Variography for silver produced parameters to be used in the grade interpolation of all elements: Ag, Au, Cu, Mn, Pb, S, and Zn (Figure 15 and Figure 16).

![Variogram Image]

**Figure 16** Best continuity for logarithmic Ag grade values: 140/55.

6.10.1 Variogram parameters

Parameters are shown in Table below.

**Table 11** Surpac style parameters for interpolation inside ore solids

<table>
<thead>
<tr>
<th>Sotkamo Deposit Variogram Orientations</th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Axis</td>
<td>Azimuth</td>
<td>Plunge</td>
<td>Range m</td>
</tr>
<tr>
<td>Major</td>
<td>140</td>
<td>-55</td>
<td>80</td>
</tr>
<tr>
<td>Semi-Major</td>
<td>274.3</td>
<td>-26.06</td>
<td>60</td>
</tr>
<tr>
<td>Minor</td>
<td>15.5</td>
<td>-21.6</td>
<td>7</td>
</tr>
</tbody>
</table>

**Parameters for Interpolation**

Angles of Rotation (Surpac)

| First Axis                            | 140 |
| Second Axis                           | -55 |
| Third Axis                            | 40  |

**Anisotropy Factors**

| Semi_Major Axis                      | 1.3 |
| Minor Axis                           | 10  |
6.11 Block modelling

A summary of the parameters used in the model prototype is shown in Table 12. A parent cell size of 5m x 2m x 5m (along strike, across strike and vertical) was selected. No rotation has been applied to the model, but simple sub-celling was applied.

Table 12 Silver Mine block model parameters

<table>
<thead>
<tr>
<th>Type</th>
<th>Y</th>
<th>X</th>
<th>Z</th>
</tr>
</thead>
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<td>Minimum Coordinates</td>
<td>11000</td>
<td>4650</td>
<td>-650</td>
</tr>
<tr>
<td>Maximum Coordinates</td>
<td>11700</td>
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<td>0</td>
</tr>
<tr>
<td>User Block Size</td>
<td>5</td>
<td>2</td>
<td>5</td>
</tr>
<tr>
<td>Min. Block Size</td>
<td>2.5</td>
<td>1</td>
<td>2.5</td>
</tr>
<tr>
<td>Rotation</td>
<td>0.000</td>
<td>0.000</td>
<td>0.000</td>
</tr>
</tbody>
</table>

6.12 Density

A density of 2.80 t/m³ has been applied to the resource block model.

6.13 Grade estimation

Grade estimation was carried out using Inverse Power Distance Squared (IPD2) as the principle interpolation method. Four estimation passes were run with each one consecutively larger to ensure (1) that all blocks within the mineralised envelopes were estimated and (2) that local peaks were not entirely away.

A summary of the estimation parameters used is summarised below in Table 13.
6.14 Model validation

Visual validation was done to assess successful application of the estimation passes and to ensure that as far as the data allowed, all blocks within mineralisation domains were estimated. Statistical validation ensures that the model estimates perform as expected.

The modal validation methods carried out included a visual assessment of grade; global statistical grade validation and comparative grade profiles (SWATH) were constructed.


Total volume of ore solids is 2 648 449 m$^3$, while the total volume of blocks inside ore solids is 2 623 956 m$^3$ plus 24 188 m$^3$ above the bedrock surface. Blocks fill the solids well (Figure 17).

<table>
<thead>
<tr>
<th>Sotkamo Silver Estimation Search Parameters</th>
</tr>
</thead>
<tbody>
<tr>
<td>Phase</td>
</tr>
<tr>
<td>-------</td>
</tr>
<tr>
<td>Search 1</td>
</tr>
<tr>
<td>Search 2</td>
</tr>
<tr>
<td>Search 3</td>
</tr>
<tr>
<td>Search 4</td>
</tr>
</tbody>
</table>

Figure 17  Section Y = 11480
6.14.2 Global grade validation

Tables 5 and 6 illustrate relations between mean global sample grades and sample grades inside ore solids also with top cuts for Ag, Au, Pb and Zn. Respectively, parameters compares mean composite sample grades with block grades.

6.14.3 SWATH analysis

SWATH plots have been generated from the model by averaging both the composites and blocks along northings, eastings and vertically. The dimensions of each panel are controlled by the dimensions of the block size. Each estimated grade should exhibit a close relationship to the composite data upon which the estimation is based. Examples of the SWATH plots produced are shown in Figure 18, Figure 19, Figure 20 and Figure 21.

![Taivaljärvi 2013: Ag composites vs. blocks and tonnage](image)

**Figure 18** Ag Northing SWATH-diagrams
Figure 19  
Au Northing SWATH-diagrams

Figure 20  
Pb Northing SWATH-diagrams
Figure 21  Zn Northing SWATH-diagrams

6.14.4 Validation summary

The results are acceptable.

6.15 Resource classification

The resources of the Silver Mine Ag-Zn-Pb-Au deposit are classified in accordance with guidelines of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves [JORC Code (2012)]. Criteria for defining resource categories were also in part derived from the geostatistical studies. In conjunction with the search estimation parameters, the estimated results were reviewed and the final classifications adjusted slightly through the use of wireframe solids. An overview of the classification distribution within the deposit is shown in Figure 22.

- **Measured** resources – belong to interpreted principal mineralised zone, based on a drill grid of 10m to 20m along strike, where grade continuity is confirmed
- **Indicated** resources – belong to interpreted principal mineralised zone, based on a drill grid of 20m to 30m along strike; grade continuity can be considered to be confirmed
- **Inferred** resources – can belong to main mineralised zones or peripheral zones but within defined mineralised solids and not estimated in the Measured and Indicated searches.
6.16 Mineral resource estimate

The resources of the Silver Mine Ag-Zn-Pb-Au deposit are classified in accordance with guidelines of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves [JORC Code (2012)]. The grades in the final resource block model were derived from surface and underground diamond drill hole sample composites based on Inverse Power Distance Squared (IPD2) estimation method for Ag, Au, Cu, Mn, Pb, S and Zn. A complete block model was built reflecting both the remaining in-situ parts, as well the parts which have already been mined out as part of the exploration decline and cross cut. The Mineral Resource Estimate is shown in Table below.

### Table 14 Mineral Resource Estimate of the Silver Mine Deposit

<table>
<thead>
<tr>
<th>JORC Classification</th>
<th>Volume m$^3$</th>
<th>Tonnage t</th>
<th>Density t/m$^3$</th>
<th>Ag g/t</th>
<th>Au g/t</th>
<th>Cu g/t</th>
<th>Mn %</th>
<th>Pb %</th>
<th>S %</th>
<th>Zn %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Measured</td>
<td>1,197,000</td>
<td>3,851,000</td>
<td>2.8</td>
<td>80.6</td>
<td>0.24</td>
<td>107</td>
<td>0.2</td>
<td>0.9</td>
<td>1.8</td>
<td>0.6</td>
</tr>
<tr>
<td>Indicated</td>
<td>950,000</td>
<td>2,660,000</td>
<td>2.8</td>
<td>87.0</td>
<td>0.24</td>
<td>125</td>
<td>0.2</td>
<td>0.3</td>
<td>1.6</td>
<td>0.7</td>
</tr>
<tr>
<td>Total</td>
<td>2,147,000</td>
<td>6,511,000</td>
<td>2.8</td>
<td>83.4</td>
<td>0.24</td>
<td>115</td>
<td>0.2</td>
<td>0.3</td>
<td>1.7</td>
<td>0.6</td>
</tr>
<tr>
<td>Inferred</td>
<td>477,000</td>
<td>1,340,000</td>
<td>2.8</td>
<td>76</td>
<td>0.2</td>
<td>99</td>
<td>0.2</td>
<td>0.2</td>
<td>1.6</td>
<td>0.5</td>
</tr>
</tbody>
</table>

Figure 22 The Silver Mine Deposit classified resource: 0 air, 1 measured, 2 indicated, 3 inferred
7 ORE RESERVE

The Ore Reserve Estimate has been audited by Outotec Mr Pekka Lovén, MAusIMM (CP), MSc (Mining), who is Competent Person in accordance with the JORC Code (2012). The audit report is presented in Appendix 4.

Table 15 Ore Reserve Estimate

<table>
<thead>
<tr>
<th></th>
<th>Open pit</th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Tonnes</td>
<td>Ag ppm</td>
<td>Au ppm</td>
<td>Pb %</td>
<td>Zn %</td>
</tr>
<tr>
<td>Proved</td>
<td>697 000</td>
<td>100</td>
<td>0.28</td>
<td>0.27</td>
<td>0.57</td>
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<tr>
<td>Probable</td>
<td>7 000</td>
<td>90</td>
<td>0.51</td>
<td>0.49</td>
<td>0.94</td>
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<tr>
<td>Total</td>
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<td>100</td>
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<td>0.27</td>
<td>0.57</td>
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<tr>
<td>Waste</td>
<td>3 149 000</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Waste/ore</td>
<td>4.47</td>
<td></td>
<td></td>
<td></td>
<td></td>
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<tr>
<td>Overburden removal</td>
<td>306 000</td>
<td>m3</td>
<td></td>
<td></td>
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<table>
<thead>
<tr>
<th></th>
<th>Underground mine</th>
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<th></th>
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<th></th>
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</thead>
<tbody>
<tr>
<td></td>
<td>Tonnes</td>
<td>Ag ppm</td>
<td>Au ppm</td>
<td>Pb %</td>
<td>Zn %</td>
</tr>
<tr>
<td>Proved</td>
<td>1 103 000</td>
<td>97</td>
<td>0.29</td>
<td>0.37</td>
<td>0.76</td>
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<tr>
<td>Probable</td>
<td>1 529 000</td>
<td>106</td>
<td>0.29</td>
<td>0.36</td>
<td>0.74</td>
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<td>Total</td>
<td>2 632 000</td>
<td>103</td>
<td>0.29</td>
<td>0.36</td>
<td>0.75</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th></th>
<th>Total reserves</th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Tonnes</td>
<td>Ag ppm</td>
<td>Au ppm</td>
<td>Pb %</td>
<td>Zn %</td>
</tr>
<tr>
<td>Proved</td>
<td>1 600 000</td>
<td>98</td>
<td>0.29</td>
<td>0.33</td>
<td>0.69</td>
</tr>
<tr>
<td>Probable</td>
<td>1 536 000</td>
<td>106</td>
<td>0.29</td>
<td>0.36</td>
<td>0.74</td>
</tr>
<tr>
<td>Grand Total</td>
<td>3 336 000</td>
<td>102</td>
<td>0.29</td>
<td>0.34</td>
<td>0.71</td>
</tr>
</tbody>
</table>

The ore reserves have been estimated according to Australian JORC code (2012). Estimates are done with a “net smelter return cut-off” of 35 €/ton for underground-mining and 25 €/ton for open-pit mining. Ore recovery is estimated to be 90%. A side-rock dilution of 15% in underground and 10% in open-pit mining has been applied. Silver-content in diluting side-rock is estimated to be 31.3 g/ton Ag.

During the first four planned production-years the silver-content will vary between 120 to 130 g/ton and gold-content will be about 0.4 g/ton. During the third planned production-year the plan-capacity is planned to increase from 350 000 tons to 450 000 tons/year. Planned production-time based on presently known reserves will be 8 to 9 years.

Beside these ore reserves there are earlier published mineral resources of 1.33 million tons inferred resources with 75 g/ton silver and about 0.5 million tons containing more than 100 g/ton Ag. A substantial portion of these resources are expected to be converted
to ore reserves in a cost-effective way, with reasonable amount of drilling from existing decline.

The mineable deposit is approximately 450 m long, and extends presently approximately 500 m from top to bottom. Below the known deposit exists zone of exploration potential down to at least 1400 meters, this zone with low resistivity has been defined by geophysical deep-penetrating Sampo-survey. This depth potential will be investigated and eventually developed for mining while production in upper parts has commenced.

Below the present ore resource bottom level high silver content in drill cores have been found for example DH 106 23.4 meters at 148 g/t Ag; including 4 meters at 575 g/t Ag.

Based on geophysical survey mineralised zone is wider in depth. Lower apparent resistivity of deep parts indicates larger amount of ore minerals.
8 MINE DESIGN AND PLANNING

8.1 Introduction

In this study all existing information during exploration and mine planning history at Sotkamo Silver mine has been gathered in. Based on up-dated Mineral Resource a new mine plan has been prepared and has been reviewed by JORC-competent Mining Engineer.

As parts of the mine plan new mine design, mining schedule and Ore Reserve Estimate have been prepared in accordance with the guidelines of the JORC Code (2012) for the Sotkamo Silver Mine.

The mine design for the Silver Mine deposit was prepared using Surpac 6.5 software. It allows the user to design mine excavations such as development and stoping and the planner to prepare mining schedule and production rates for Life of Mine.

The outcoming mining schedule and production plan includes tonnages, grade, and development metres over the Life of Mine. Factors which have been taken into account include: Safety and schedule issues, deposit geometry, rock properties, shears and other weak structures, needed production rate, mine infrastructure.

8.2 Rock stability

8.2.1 Rock properties and stress status

Acid volcanic rock consisting mostly of quartz and sericite is clearly deformed and average slightly weaker than Finnish bedrock average.

Rock quality parameters (Report of Kajaani Oy 1986):

- Uniaxial Compress Strength 60 - 120 MPa
- Tensile strength (stress strength) 10-15 MPa
- Drilling index DRI 50 – 60 (average- fair good)
- Sharpening index of drill rigs aver. 145 m

Stress strength value is low what is typical for foliated rocks, test direction defines entirely the result value. Lower than average Uniaxial Compress strength is explained by grain size properties of the rock, which in turn improves drill ability and sharpening index. Regarding drill ability strong deviation/curving of drill holes in the tunnel is remarkable. Curving is caused mostly by drill pressure and rotating speed, but also weak/hard mineral combination (sericite/quartz) and strong foliation of the rock influence curving of holes.

Rock bolts and generally reinforcement of the tunnel rock during tunnel mining in 1988-1990 was not done systematically while rock was seen stable and need for reinforcement was seen low. During and after dewatering of the production tunnel down to 350 meters depth rock-falls have hardly been seen, and stability of tunnels and ventilation shaft is good.
Wardell Armstrong International (2012) states that rock can be classified from good to fair where good is best estimate of the scale. Anyhow, in order to be conservative they use term fair.

Based on tectonic observations three main joint directions can be defined:

- First main discontinuity 53/005 degrees (dip/direction)
- Second main discontinuity 79/298 degrees
- Third main discontinuity 23/177 degrees

Rock quality designation figures describing the rock mass are following, these figures are based on observations from diamond drill holes during exploration of the deposit:

RQD
- 87 -91 (best estimate)
- 50 – 60 (conservative estimate)

Q-coefficient
- 87 (best estimate)
- 50 (conservative estimate)

Stress survey performed by Suomen Malmi Oy (Finnexploration) in 1990 stress parameters of Silver Mine is typical for Finnish Bedrock in depth 0 to 300 meters. Stress can be estimated to increase linear while depth increases. (Horizontal stress average = 8,0 \( +0,06 \times \text{depth} \)). Results of survey by Suomen Malmi Oy are following:

- 1. main stress 7 – 27 MPa
- 2. main stress 3 -18 MPa
- 3. main stress 1 – 16 MPa

8.3 Stability and need for reinforcement

8.3.1 Open pit

According to analysis by WAI possible caving types for open pit are described in the following Figure 23.
Open pit area has hardly any need for reinforcement due to small depth and main ramp is on area which can be seen for safe area. Ramp area walls will be cleaned from loose rocks carefully and for this like also other areas alternative change plans in case weakness areas will be observed.
8.3.2 Underground Mine

Based on rock quality analysis and experience from tunnel need for rock support is small in the beginning of mining activity. Most likely is that tunnels showing weakness zones must be supported and areas with wider roof area will be supported systematically (mucking areas, crossings etc.).

Need to support stopes is also probably low, but anyhow in certain areas like + 120 level cable bolting is required. Geology and especially biotite-sericite-schists will be mapped, this while their stability properties are expected to be lower than in other areas.

8.3.3 Risk assessment and working in risk areas

Based on existing rock mechanical information caving-in risk can be estimated to be small in open pit and also in the underground mine. This view is supported also by observations in the existing tunnel: need for reinforcement is small and remarkable weakness zoned have not been seen. Normal rock jointing and discontinuities can be seen in the tunnel and in the ventilation rise, anyhow these can be managed with minor reinforcement work. Discontinuities cause in places water leaking into the tunnel and rise. Open pit mining and topsoil removal while mining commences can possibly increase amount of incoming water for some time.

Underground stopes will be at the begin small in size (20 000– 30 000 tons, 7 000 – 10 000 m³) and open stopes will be filled. Mining and production starts thus in small-scale and is expected to be easy to control. Empty spaces will be small (ca 10 000 m³) and thus also caving will be small. While making test stopes and collecting practical information stope sizes can be enlarged or modified to correspond observed rock behaviour.

Empty stope spaces cause naturally stresses in nearby tunnels; these will be observed carefully during ore mining and other operations in area. Risk areas will be supported by necessary means after rock falls or caving-ins and area can be defined as non-go area. In certain areas ore stopes are next to present production tunnel; tunnel is secured for safe traffic and while mining next to tunnel by-pass tunnels have been planned.

8.3.4 Follow-up and simulation of rock mechanics

While underground mining proceeds stress filed in the area will also change. Stress simulation associated with experiences form mining gives good background for further mine planning. Systematic and regular surveys in open pit and underground mine give good background for forecasting of caving-ins or other risks

8.4 Underground mining methods

8.4.1 Development

Current development at the Silver Mine consists of a 5m wide by 5m high, 2 570m long decline extending from surface to a depth of 350m, and a 2.2m diameter vertical ventila-
tion raise extending from circa 340 meters depth to surface, shaft is connected to decline at lowest level with a gently rising 70 meters long tunnel.

The mine design has been done by JK-Kaivosuunnittelu using Surpac 6.5 mine planning tool. Lowest mining depth is planned to be -450 m which requires extending of the existing tunnel during later production years. Mining method is planned to be longitudinal bench and fill stoping/sub-level stoping with 20 meters intervals, stopes are back-filled partly with cemented back-fill where necessary in order to give way mining of neighbouring stopes. The mine design also consists of level drifting, and other necessary associated development.

Figure 24 shows the principle of underground development in area of first underground mining. The proposed development is shown in red, existing tunnel is shown brown.

Figure 24 Proposed Development (red) for mining in levels -180-120

Diversion loops will need to be constructed between 140-160 m and 280-320 m levels to replace the existing ramp where it is located in close proximity to proposed stoping on these levels. Anticipated by-pass tunnels are presented in Figure 25, shown green in the picture; also tunnel extension for mining of deeper parts is presented.
Figure 25  Diversion Loop (green) in the Footwall between 120-160 m and 280-320 m levels

All development operations at the Silver Mine are performed by sub-contractor using conventional drill and blast tunnelling techniques, utilising electric-hydraulic face drilling jumbos for blasthole drilling, and diesel powered LHDs for mucking. The principal explosive used is pneumatically loaded ANFO emulsion with Nonel detonators. All vertical development for ventilation will be constructed using raise boring.

8.4.2 Stoping

Ore stoping just like all mining work in the mine are performed by contractors. Mine planning, mine survey and control are performed by Silver personnel. Most stoping at the Silver Mine has been planned to be done as longitudinal bench and fill, using cemented backfill where necessary. All stopes are not necessarily back-filled; also use of non-cemented backfill will be used where possible. Underground stoping will be started at levels 180 and 160 and mined upwards towards future open pit bottom. In places where ore is wide enough (20 – 30 m) transverse bench and fill will be considered. UG drifting and open pit barren rock provide enough material for back-fill purposes, tailings sand will also be used in back-fill, testing of suitability of these materials is on-going.
Figure 26  Silver Mine Stoping Method, example stope C180-1

Figure 27  Stope C180-1, Blasthole ring 10, Looking to NW
Figure 28  Section lines of Stope C180-1 (160 level coloured black), Planview
Figure 29  Section 4, Slot raise (coloured red) and widening blast holes
Figure 30  Section 10, Blast hole ring
The sequence of the mining operations is as follows:

- An access tunnel is driven from the footwall decline to the upper level of the orebody (= drilling level)
- An access tunnel is driven from the footwall decline to the lower level of the orebody (= loading level, level interval appr. 20 m, drift gradient mainly 3%)

The drifts are expanded to full stope width if the width of orebody is 5 – 6 meters and after this appropriate ground support is installed if seen necessary

- Blast holes are drilled mainly downwards from the upper level (upwards drilling only when there is not any upper level or a better loading cone is needed at loading level)
- A slot-raise is developed between the levels (in example stope C180-1 slot raise and widening blast holes are blasted leaving some 5 meters deck in first phase)
- Rings of blast holes are loaded and fired, starting at the slot-raise and retreating back to mucking point (Figure 28, Figure 29 and Figure 30; C180-1 slot raise and rings 1 – 5 are blasted breakthrough altogether in one blast, next blast is for ex. rings 6 – 8)
- Ore is mucked using LHDs (remote-controlled when needed) and loaded onto trucks for transport to the crusher
- Primary stopes are backfilled mainly with cemented slurry mixed with barren rock and secondary stopes are backfilled only with barren rock. Backfill (cemented) is placed in the stope using drillholes and pipes from surface to the drilling level and waste rock is unloaded from drilling level or through the slot raise of overlaying stope
- When the stope is completely backfilled the top of the fill forms the mucking floor of the overlying stope.

The maximum stope dimensions will be 20 m high, length some 60 -70 m along strike, and the width depending the orebody up to 30 meters. Production will follow basically a primary-secondary stope sequence.

All operations are conducted using electric-hydraulic production drilling jumbos, diesel LHDs and diesel dump trucks.
8.4.3 Losses and dilution

During the mine design process, losses and dilution were applied to the mined tonnages at the Silver Mine as outlined in Chapter 8.5.2 below.

8.4.4 Backfill

Most primary and secondary stopes in the Silver Mine will be backfilled with rock fill (CRF). Secondary stopes will be backfilled with un-cemented waste rock and/or tailings sand. Suitability tests to use tailings sand are on-going at Oulu University.

8.5 Surface mining

8.5.1 Open pit design

The open pit mine design has been prepared by JK-Kaivossuunnittelu. Overall lifetime of the pit is nine years, it will be mined in two stages, first “Starter pit” which reaches a depth of 40 meters in five years’ time and is followed by final open pit reaching a depth of 80 meters. The open pit is shown in Figure 31.
The benches have been designed with a height of 10m. The overburden varies between approximately 2-5m. The maximum slope angle is 80 degrees. The maximum pit depth is 80m below surface.

The haul road is 16m wide, narrowing to 10m in the base of the pit. It is at a maximum gradient of 1:7.

- Mining is done as contract work, drill- and blast mining where ore boundaries are defined by company geologists.

All production drill holes are sampled and analysed in order to improve ore model and to separate ore and barren rock. Marginal ore will be stockpiled in own storage to be used later in case of possible breakdowns or when it is.

The pit is mined using excavator and articulated trucks or mine dumpers.

- The pit is mined during winter seasons, total amount of ore will be 703 000 t and of waste rock 2.2 Million tons
8.5.2 Losses and dilution

During the mine design process, ore dilution factor was applied to the mined tonnages at the Silver Mine as outlined below.

- Mining Dilution: Underground mining: 15%, Open pit mining: 10%.
- Mining Recovery of planned stopes: 90%.

Mine plans have been reviewed by an external expert.

The review work has been done by Pekka Lovén, MSc(Mining), MAusIMM(CP) of Outotec (Finland) Oy, a Competent Person as defined by Joint Ore Reserves Committee (JORC, 2012). His review report is presented in Appendix 4.

8.6 Infrastructure

The location and specification of the surface infrastructure associated with the underground mine is presented in Chapter 10 and Appendix 5 Mine Area Layout.

8.6.1 Ore and barren rock extraction/transportation

Ore and barren rock extraction at the Silver Mine is carried out using trackless equipment, principally diesel powered dump trucks and LHDs, to transport ore up the decline to surface Run of Mine ore storage adjacent to the underground mine portal.

8.6.2 Ventilation

Primary ventilation incorporates a 2.2m diameter down-draft ventilation shaft, with the exhaust airflow exiting up the decline. A fan will be mounted on the shaft collar, and during normal operation will blow air down the shaft at 100 m³/s. In an emergency situation, the fan direction can be reversed to exhaust air up the shaft. The fan will also incorporate a heat exchanger in order to warm the air during the winter.

Secondary ventilation will consist of secondary fans and flexible ventilation ducts in each active heading.

8.6.3 Electricity

Electrical power will be required underground for ventilation, pumping and drilling. The main power consumers will be the electro-hydraulic drilling jumbos and the primary ventilation fan. Other significant power consumers will include the other drilling equipment, water pumps, and secondary ventilation fans.

Electrical power will be supplied through a single armoured 20kV cable, running through a drill hole from surface. At intervals along the decline, portable power transformers will be located, fed from the main HV cable. Portable switchgear will be located close to each active heading or stope to power the drills and the secondary ventilation fans.
8.6.4 Pumping

Underground pumping stations will be located on the 335, 250, 165 and 75 levels. Each pumping station will be equipped with a raw water pool with sedimentation basin, and local level controls. Underground water will be pumped from the lower level pump station up to the next level, and so on until ground level. Mine head water will be pumped with immersion pumps to the nearest underground pumping station.

Open pit and underground water will be pumped to the tailings pond using the open pit pump station.

8.6.5 Crushing and fine ore storage

The mine crushing and conveyors will operate at 250t/h. It will consist of a jaw crusher, a vibrating screen, and a cone crusher, linked with belt conveyors.

The crushing plant will operate 8 hours per day, 5 days per week, which corresponds to an average of 168t/h crushed fine ore feed capacity, producing 350 000 – 450 000 t/a of 25 mm crushed fine ore storage. The crushed ore will be stored in a 6 000t silo with 6 days of buffer storage to allow the concentrator plant to run when the crushing plant is not operating.

8.6.6 Mining equipment

The mining production fleet will be provided by the contractor for the open pit and underground operations.

8.6.7 Technical services, personnel and contractors

Technical services and project management will be provided by Sotkamo Silver.

8.7 Mining plan and schedule

8.7.1 Open pit

Open pit mining will be done in two stages. Starter pit will be mined in four years, on top of that pit expansion into final pit four additional years down to +80 meters depth. Open pit mine is presented in Figure 33.

Open pit mining plan is presented in Table below.
Table 16  Open pit mining plan

<table>
<thead>
<tr>
<th>Year</th>
<th>Open pit/Mining Ore</th>
<th>Ageq</th>
<th>Calc</th>
<th>Ag</th>
<th>Au</th>
<th>Pb</th>
<th>Zn</th>
<th>Waste Tonnes</th>
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<td>3</td>
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<td>7294</td>
<td>500 000</td>
<td></td>
</tr>
<tr>
<td>8</td>
<td>65 569</td>
<td>99,78</td>
<td>85,12</td>
<td>0.27</td>
<td>2429</td>
<td>5715</td>
<td>200 000</td>
<td></td>
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<tr>
<td>9 **</td>
<td>99 466</td>
<td>61,91</td>
<td>53,30</td>
<td>0.16</td>
<td>1716</td>
<td>3609</td>
<td>94 630</td>
<td></td>
</tr>
<tr>
<td>10 **</td>
<td>97 343</td>
<td>61,91</td>
<td>53,30</td>
<td>0.16</td>
<td>1716</td>
<td>3609</td>
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<tr>
<td>Total</td>
<td>703 790</td>
<td>115,39</td>
<td>99,84</td>
<td>0.28</td>
<td>2706</td>
<td>5700</td>
<td>3 149 140</td>
<td></td>
</tr>
</tbody>
</table>

* starter pit ends  
** marginal ore A

8.7.2 Underground

Underground ores down to - 160 level below surface for two first years are easily accessible from the existing tunnel, they are 10 – 30 meters wide and are mined by transverse sub-level stoping with back-fill.

During later production years UG mining down to - 430 level below surface is done as longitudinal sub-level stoping with back-fill. Several parallel ore lenses can be mined using same access tunnels.

Underground mining plan is presented in Table below and Figure 34.

Table 17 Underground mining plan

| Year | Underground mine/Mining Ore | Ageq | Calc | Ag | Au | Pb | Zn | Drifting Stope: Drifting Drifting Drifting Total mining |
|------|-----------------------------|------|------|----|----|----|----|--------------|--------------|--------------|--------------|--------------|--------------|
| 1    | 285 873                     | 235,75 | 214,23 | 0.36 | 3801 | 5681 | C180-1       | 1250 | 87500       | 1000               | 355 873      |
| 2    | 267 145                     | 145,46 | 128,41 | 0.29 | 4327 | 8251 | C160, A140, A120, B240, B220 | 1500 | 105000      | 140,120,240,220 | 372 145      |
| 3    | 223 292                     | 153,69 | 136,41 | 0.29 | 3085 | 6842 | H340, H320, H300, B180, B160 | 2200 | 154000      | 420,400, 380, 360 | 377 292      |
| 4    | 370 822                     | 137,68 | 122,34 | 0.34 | 4455 | 9329 | C420-C340, I410-I350 | 2500 | 175000      | 545 822      |
| 5*   | 424 344                     | 100,82 | 84,75  | 0.29 | 3500 | 7475 | A160-A120, C320, C300, B180 | 2500 | 175000      | 599 344      |
| 6    | 390 000                     | 100,82 | 84,75  | 0.29 | 3500 | 7475 | B180          | 2000 | 140000      | 530 000      |
| 7    | 390 000                     | 100,82 | 84,75  | 0.29 | 3500 | 7475 | B180          | 1500 | 105000      | 495 000      |
| 8    | 390 000                     | 100,82 | 84,75  | 0.29 | 3500 | 7475 | B180          | 1500 | 105000      | 495 000      |
| 9 ** | 281 173                     | 92,77  | 78,58  | 0.24 | 3450 | 7977 | Unplanned reserve | 1250 | 38500       | 318 673      |
| 10 **| 281 173                     | 92,77  | 78,58  | 0.24 | 3450 | 7977 | Unplanned reserve | 1250 | 38500       | 318 673      |
| Total| 2 631 648                   | 118,89 | 102,52 | 0.29 | 3629 | 7514 | 15000         | 105000       | 3 681 648    |              |
Figure 33  Open pit mining plan
Figure 34 Underground mining plan
9 MINERAL PROCESSING AND METALLURGICAL TESTING

9.1 Introduction

The Silver Mine deposit, consisting of four ore bodies with different base metal and precious metal ratios, has been categorised into three types of mineralisation. These are referred to as:

1. Silver mineralisation;
2. Silver-zinc-lead mineralisation; and
3. Low-grade footwall mineralisation.

In 2009 GTK commenced a programme of metallurgical testing to evaluate the processing requirements of the Silver Mine mineralisation, to produce saleable silver-lead and zinc concentrates. The programme consisted of bench scale batch and locked cycle testing, culminating in 2011 with pilot plant testing of 88 tn of ore.

Based on the GTK investigation, a concentration plant was designed to treat 350 000 t/a of ore with average grading, 105 g/t silver, 0.4 g/t gold, 0.8% zinc and 0.4% lead. The ore will be processed using silver-lead flotation consisting of rougher, scavenger and cleaning stages. The silver-lead tailings will be further treated in a flotation circuit consisting of rougher, scavenger and cleaning stages for zinc recovery. Zinc tailings will pass to a pyrite flotation section prior to final tailings disposal.

The average (dry) concentrate production is forecast to be:

- Ag-Au-Pb Concentrate 1 550 t/a
- Zn Concentrate 3 850 t/a
- Pyrite Concentrate 9 150 t/a

9.2 GTK metallurgical testing 2009

9.2.1 Test sample

In 2009 sample of site stockpile ore from the Silver Mine deposit was sent to GTK Mineral Processing Laboratory for bench scale testing. The objective of the study was to investigate if ageing of the ore in the stockpile had affected the beneficiation characteristics of the sample. The sample had been stored in open air at the Silver Mine site since 1991.

The test performed consisted of crushing grinding and flotation. The regime utilised for the bench-scale test was said to follow the processes developed for the Silver Mine ore by VTT in the late 80’s.
9.2.2 Bench scale test results

The results are summarised as follows:

- A lead recovery of 83.9% was obtained to a lead cleaner concentrate grading 60.1% Pb;
- A silver recovery of 74.1% was achieved to the lead cleaner concentrate grading at 1.4% Ag;
- A gold recovery of 73.5% to the lead cleaner concentrate was achieved at a grade of 43.6 g/t;
- A zinc recovery into the zinc cleaner concentrate was 88% at a 52.6% zinc grade; and
- The calculated head grades of the sample are reported as 1.45% Pb, 3.52% Zn, 382 g/t Ag and 1.2 g/t Au.

GTK concluded that, based on the results, ageing had not affected the processing characteristics of the ore sample and that the results were comparable to the earlier tests.

9.3 GTK metallurgical testing 2011

9.3.1 Test samples

Three ore samples, 60kg each, were submitted to GTK Mineral Processing Laboratory in February 2011. The samples were labelled based on the silver grade and designated as follows:

- Ag 50;
- Ag 100; and
- Ag 150.

Each sample was crushed to 100% passing 4mm, homogenised and split into representative sub-samples for head assays and flotation testing. The crushed samples were stored in a freezer to avoid oxidation.

A blend of the three composites was prepared and split into sub-samples for head assays and locked cycle testing.

The flotation test programme consisted of bench-scale rougher and cleaner flotation tests, incorporating one stage of cleaning in the lead and zinc flotation circuits for Tests 1 and 2 and two stages of cleaning for Test 3. One locked cycle test was performed on the blend sample in order to investigate the effect of re-circulating streams on final concentrates recoveries and grades. Two stages of cleaning were performed in each circuit. There was no regrinding prior to cleaner flotation.

Rougher flotation was conducted using the primary grind size of 75% passing 75μm.

The samples submitted for the test programme were expected to be representative of the production head grades. Each composite was analysed for silver and gold by Fire Assay,
for lead, zinc and iron by nitric acid digestion and atomic adsorption spectrometer, for sulphur by Eltra and for about 40 elements by XRF.

9.3.2 Batch flotation tests results

Representative 5kg feed batches were milled to 75% passing 75μm and subjected to flotation testing.

The reagents used in lead flotation were zinc sulphate (ZnSO₄) and Aerophine 3418A in rougher flotation and sodium cyanide (NaCN) and Aerophine 3418A in the cleaner stages. Lead flotation was conducted at a nominal pH8.2.

Reagents used in zinc flotation were copper sulphate (CuSO₄) and sodium isobutyl xanthate (SIBX). Zinc flotation was conducted at pH11.5.

Methyl isobutyl carbinol (MIBC) was used as the frother and lime as the pH modifier in the lead cleaner flotation and zinc flotation.

9.3.3 Lead concentrates grades and recoveries

The results of the batch flotation tests are summarised as follows:

- Lead recoveries in the highest grade composite were significantly higher, with Ag150 achieving 60.7% lead recovery to a concentrate grading 67.2% Pb;
- The Ag50 and Ag100 samples were amenable to upgrading to a 50%Pb concentrate grade after two cleans; and
- Lead flotation performance in the lower grade Ag50 sample was better than that of the Ag100 sample, with potentially around 69% lead recovery to a concentrate grading 50% Pb possible, when compared with the Ag100 sample, with around 57% lead recovery to a 50 %Pb concentrate.

9.3.4 Silver deportment to lead concentrate

Batch testing of the three ore types demonstrated that:

- Silver recoveries to the rougher concentrates were comparable for the lower grade ore types at around 70%. Rougher recovery for the Ag150 sample was higher at 75%;
- Without regrinding prior to cleaner floatation, silver minerals in the Ag100 sample appear to be more readily recoverable, suggesting a greater degree of locking in the Ag50 and Ag150 samples at the d75=75μm primary grind size; and
- After two stages of cleaner flotation, silver recoveries ranged from 37.2% (Ag100) to 45.2% (Ag150) into lead concentrates grading from 1.1% Ag (Ag50) to 2.3% Ag (Ag100).
9.3.5 Gold deportment to lead concentrate

Batch testing of the three ore types demonstrated that:

- The trend of gold grades and recoveries were similar to that of silver for each ore type;
- Gold recoveries to the lead rougher concentrates were generally between 70 and 80%; and
- The highest gold grade was obtained in the Ag100 sample.

9.3.6 Zinc concentrates grades and recoveries

Batch testing of the three ore types demonstrated that:

- The best zinc rougher recoveries were obtained with the Ag50 and Ag150 samples;
- The 50% zinc grade target was achieved after two cleaning stages in the Ag100 and Ag150 samples; and
- After two stages of cleaning the Ag50 sample achieved a 78.3% zinc recovery to a concentrate grading 48.7% Zn.

9.3.7 Locked cycle flotation test results

The locked cycle test was carried out over five cycles. The cleaner tailings were returned to the previous stages on the next cycle as in a continuous flotation circuit. The second lead and zinc concentrates as well as the lead and zinc tailings from each cycle were assayed.

The results of the locked cycle test, performed at a 75% passing 75 µm primary grind size without rougher concentrate regrind and two stages of cleaning are summarised as follows:

- 86.7% lead recovery to a concentrate grading 54.6% Pb was achieved;
- Silver grade in the lead concentrate was approximately 1.7% Ag at 69.5% recovery;
- Approximately 7% of the silver was recovered to the zinc concentrate grading 544 g/t Ag;
- 90.8% zinc recovery was achieved to a concentrate 51% Zn;
- Using Test 3 conditions, reagent consumption in the locked cycle test was comparable with that observed in the open circuit test.

9.3.8 Bench-scale tests conclusions

The results of the bench-scale flotation tests demonstrated that two cleaning stages, in both the lead and zinc circuits, were required to achieve the target 50% Pb and 50% Zn grades.

The locked cycle test results showed that lead, zinc, silver and gold can be upgraded into saleable concentrates showing good recoveries and acceptable grades.
The locked cycle calculated head grades for lead and zinc were higher than that of the single Ag100 sample and so cannot be scaled up directly.

The lead and zinc open-circuit final grades are higher than the locked cycle test and, as expected, recoveries in the locked cycle test are higher. The lower locked cycle concentrate grades may be as a consequence of the particular lead and zinc mineralisation style in the Ag50 component of the blend.

Lead and silver recoveries into the lead concentrate were 87% and 70% respectively. Zinc recovery into the zinc concentrate was 91% and silver recovery 6.7%.

The total silver recovery obtained to the two concentrates in the locked cycle test was $69.5\% + 6.7\% = 76.2\%$. Payment terms for silver in zinc concentrate will be less favourable than that for the lead concentrate.

9.4 Pilot plant test programme

9.4.1 Test samples

GTK conducted pilot plant testing on the Silver Mine Ag-Pb-Zn ore type over the period August 30 to October 2 2011. Sotkamo Silver provided a blend of higher and lower grade sub-ore samples totalling 88 tn for the test work programme. Metallurgical testing was divided into two campaigns consisting of:

1. 58t, August 30 – September 9; and
2. 30t, October 3 – October 7.

The primary objective of the pilot plant was to generate data for the Silver Mine Mill design and confirm the results of the earlier bench and mini-pilot (locked cycle) scale studies on the ore.

Metallurgical testing of the Feed 1 sample consisted of flotation-gravity and flotation only regimes. Testing of the Feed 2 sample was confined to flotation only.

9.5 Pilot plant gravity and flotation results

The results of the pilot plant tests are summarised as follows:

- Gravity processing prior to flotation did not have any significant advantages over flotation only. The lead grade of the gravity concentrate was very high but silver and gold grades were comparable to that obtained in flotation concentration;
- Gravity and flotation concentration did not produce a precious metal concentrate;
- The finer grind produced higher grade silver-lead concentrates but the respective recoveries were higher at the coarser grind size;
- Silver and gold recoveries were independent of grind size;
- The Aerophine 3418A collector showed better flotation response than the Cheminova Danafloat 271; and
The sulphur content of the tails could be reduced from 0.7 – 1.1 % to 0.1 – 0.2 % S which is of benefit to tailings disposal. Silver grade in the sulphur (pyrite) concentrate ranged from 200 – 300 g/t Ag.

9.6 Flowsheet description

9.6.1 General

Metallurgical testing of the Silver Mine ore has demonstrated that the silver, lead and zinc minerals are amenable to separation and concentration to produce saleable concentrates.

Pilot plant testing produced three concentrates; namely:

1. Ag-Au-Pb Concentrate;
2. Zinc Concentrate; and
3. Sulphur (pyrite) concentrate.

The flotation process requires grinding of the ore, followed by silver-lead flotation consisting of rougher and scavenger stages followed by three-stage cleaning. The silver-lead flotation tailings are then transferred to the zinc flotation circuit consisting of rougher and scavenger stages and three stages of cleaning. Tailings from zinc flotation then undergo pyrite flotation, which consists of a rougher and cleaning stage.

Silver Mine will have a concentration capacity of 350 000 - 450 000 t/a. The run of mine will be mined from the open pit and the underground mines. The crushing plant will have a capacity of 250 t/h and the concentration plant capacity of 45-55 t/h.

The silver mine concentration process will produce Ag-Au-Pb concentrates (1 550 t/a), Zn concentrates (3 850 t/a) and pyrite concentrates (9 150 t/a). The total design concentrator capacity is approximately 14 550 t/a.

9.6.2 Crushing

The ROM ore is transported from the underground mine and the open pit to the ROM ore storage area. The ROM ore is sorted to the ore and the barren rock piles from where the ore is transported to the ore crushing plant and the barren rock is transported to the barren rock storage area. The ROM ore is crushed and fed to the concentrator plant via crushed fine ore storage with conveyor belts.

9.6.3 Grinding and classification

A conventional Rod mill – Ball mill grinding circuit is used in the concentrator plant of the Silver Mine. The fine ore is fed to the rod mill via a mill feed conveyor which is loaded with four feeders. Feeders rotation speed is controlled by the mill feed conveyor weight control loop.

The grinding circuit consists of a rod mill in an open circuit and a ball mill in a closed circuit with a vibrating screen and a hydro cyclone. The nominal ore feed rate is 40 t/h
and maximum capacity is 45-55 t/h. The product grind size after rod milling is 4.76mm and, after ball milling, on average 73 μm.

The rod mill is equipped with a trommel. The trommel oversize (pebbles and steel scats) will be collected from the rod mill to the container. The trommel undersize will be forwarded to the mills discharge sump and pumped to the vibrating screen. The main purpose of the screen is to prevent blockages in the hydro cyclone feed. The screen underflow is pumped to a hydro cyclone and the overflow passes through a flow meter and a particle size analyzer to the flotation circuit. The size of the cyclone apex can be changed from 7 to 15mm, depending on the targeted grinding product particle size distribution. The screen overflow (O/F) and the cyclone underflow (U/F) are fed to the ball mill. The ball mill feed rate is 146.1 t/h.

The net grinding energy was 10.1 kWh/t for the coarse grind, 10.7 kWh/t for the middle grind and 12.2 kWh/t for the fine grind. The work index for Silver Mine ore varied from 11.3 kWh/t to 11.9 kWh/t.

The cyclone overflow is forwarded to the Ag-Au-Pb-flotation feed conditioner.

**9.6.4 Ag-Au-Pb flotation**

The silver-lead flotation circuit comprises rougher and scavenger stages, and three-stage cleaning. The cyclone overflow will be conditioned in the conditioning tank before transferring to the bank of rougher/scavenger cells of the silver-lead flotation circuit. This bank will comprise 3 x 20 m³ cells. The concentrate from the 1st and the 2nd rougher cells is transferred to the 1st cleaning cell. The concentrate from the 3rd rougher/scavenger cell is pumped back to the 1st rougher cell.

The silver-lead flotation circuit includes three cleaning stages. This finishing bank will comprise 3 x 10 m³ cells. The concentrate of the 1st cleaning cell of the silver-lead – flotation is transferred to the 2nd cleaning cell and tailings to the silver-lead flotation conditioner. The concentrate of the 2nd cleaning cell is transferred to the 3rd cleaning cell and tailings to the 1st cleaning cell. The concentrate of the 3rd cleaning cell is forwarded to the silver-lead concentrate thickener and tailing to the 2nd cleaning cell.

Residence times for the roughing and scavenging duties is 27 minutes. Cell levels will be controlled by means of pinch valves and level transmitters installed in each tank cell and on each bank of trough cells. Air control in all cases will be automatic valves located on the air inlet line to each cell. Slurry pH will be measured on line by pH meters installed in the feed box of the first rougher cell and it will be used to control lime addition.

**9.6.5 Zn flotation**

The zinc flotation circuit comprises of rougher and scavenger stages and a three-stage cleaning.

The rougher and scavenger cells of the zinc flotation circuit will comprise of 3 x 20 m³ cells. The concentrate from the 1st rougher cell is forwarded to the 2nd cleaning cell. The concentrate from the 2nd and the 3rd rougher cells is forwarded to the 1st cleaning cell.
The zinc flotation circuit also includes three stages of cleaning. This bank will comprise of 3 x 10m³ cells. The concentrate of the 1st cleaning cell of the zinc flotation is transferred to the 2nd cleaning cell and tailings to the 2nd rougher cell. The concentrate of the 2nd cleaning cell is forwarded to the 3rd cleaning cell and tailings to the 1st cleaning cell. The concentrate of the 3rd cleaning cell is forwarded to the zinc concentrate thickener and tailing to the 2nd cleaning cell.

Residence times for the roughing and scavenging duties is ca. 40 minutes. Cell levels will be controlled by means of pinch valves and level transmitters installed in each tank cell and on each bank of trough cells. Air control in all cases will be automatic valves located on the air inlet line to each cell. Slurry pH will be measured on line by pH meters installed in the feed box of the first rougher cell and it will be used to control lime addition.

Specialised froth pumps, designed according to the industry standard froth factors, will be used for all concentrate duties. Variable speed drives will be installed on appropriate flotation feed and tailings pumps to allow control of overall flow rates through the circuit.

9.6.6 Pyrite flotation

The pyrite flotation circuit comprises a rougher stage and a cleaning stage.

The pyrite flotation rougher stage comprises of 1 x 120m³ cell. The concentrate from the 1st rougher cell is forwarded to the 1st cleaning cell.

The pyrite flotation circuit cleaning stage comprises 1 x 10m³ cell. The concentrate of the 1st cleaning cell of the pyrite-flotation is forwarded to the pyrite concentrate thickener and tailings to the 1st rougher cell.

Residence times for the roughing and scavenging duties is 63 minutes. Cell levels will be controlled by means of pinch valves and level transmitters installed in each tank cell and on each bank of trough cells. Air control in all cases will be automatic valves located on the air inlet line to each cell.

Specialised froth pumps, designed according to the industry standard froth factors, will be used for all concentrate duties. Variable speed drives will be installed on appropriate flotation feed and tailings pumps to allow control of overall flow rates through the circuit.

The pyrite flotation tailings will be directed at the rate of 130.43m³/h (24.7% solids) to the tailings sump from where they will be pumped to the Tailings Management Facility (TMF).
9.6.7 Concentrate thickening and filtering

9.6.7.1 Lead-silver-gold concentrate

The silver-lead concentrate is pumped to the 15m³ batch operating silver-lead concentrate cone thickener equipped with a rake mixer. The thickener underflow at 55-60% solids is fed to the silver-lead concentrate tank and pumped to the vacuum filter. The silver-lead concentrate tank is equipped with an agitator. The thickener overflow reports to the circulation water tank.

The thickened slurry is pumped by peristaltic pump to a drum vacuum filter. The filtrate water will be pumped to the circulation water tank. The filter cake with a moisture content of about 10% will be discharged from the vacuum filter directly onto a conveyor, which will transport it to the silver-lead concentrate storage room.

The silver-lead concentrate handling circuit will produce separate dewatered silver-lead concentrate with moisture levels suitable for road and rail transport. The silver-lead concentrate handling circuit will operate in conjunction with the silver-lead flotation circuit.

9.6.7.2 Zinc concentrate

The zinc concentrate is pumped to the 15m³ batch operating zinc concentrate cone thickener equipped with a rake mixer. The thickener underflow at 55-60% solids is fed to the zinc concentrate tank and pumped to the vacuum filter.

The zinc concentrate tank is equipped with an agitator. The thickener overflow is transferred to the circulation water tank. The thickened slurry is pumped by peristaltic pump to a drum vacuum filter. The filtrate water will be pumped to the circulation water tank. The filter cake with a moisture content of about 10% will be discharged from the vacuum filter directly onto a conveyor which will carry it to the zinc concentrate storage room.

The zinc concentrate handling circuit will produce separate dewatered zinc concentrate with moisture levels suitable for road and rail transport. The zinc concentrate handling circuit will operate in conjunction with the zinc flotation.

9.6.7.3 Pyrite concentrate

The pyrite concentrate is pumped to the 15m³ batch operating pyrite concentrate cone thickener equipped with a rake mixer. The thickener underflow at 55-60% solids is fed to the pyrite concentrate tank and pumped to the vacuum filter.

The pyrite concentrate tank is equipped with an agitator. The thickener overflow is directed to the circulation water tank. The thickened slurry is pumped by peristaltic pump to a drum vacuum filter. The filtrate water will be pumped to the circulation water tank. The filter cake with a moisture content of about 10% will be discharged from the vacuum filter directly onto a conveyor which will transport it to the pyrite concentrate storage room.
The pyrite concentrate handling circuit will produce separate dewatered pyrite concentrate with moisture levels suitable for road and rail transport. The pyrite concentrate handling circuit will operate in conjunction with the pyrite flotation.

9.6.7.4 Concentrate storage and transportation

The silver-lead, zinc and pyrite concentrate will be conveyed to concentrate storage rooms from the vacuum filters with the moisture levels suitable for road transportation.

The concentrates will be removed from the concentrate rooms by the wheel loader to the transporting truck.

9.6.7.5 Tailings disposal and water re-circulation

The pyrite flotation tailings (final tailings) will be transported at the rate of 130.43 m$^3$/h (24.7% solids) to the tailings sump from where it is pumped directly to the TMF.

The values are averages of the GTK pilot test samples.
10 PROCESS INFRASTRUCTURE

10.1 General

The Process concept and production figures are based on GTK study presented in Chapter 8.

The Crushing plant will use 2-stage crushing equipped with a Jaw crusher, vibrating screen and a Cone crusher.

The process of the concentrator plant comprises grinding circuit equipped with rod mill and ball mill, classification circuit equipped with screen and cyclone and finally three flotation circuits followed by concentrate thickening and filtering.

The ore crushing will be executed by a contract work which includes the ore crushing equipment (crushers, screen, conveyors to the crushed ore storage and crushers’ auxiliary systems), the dust collecting and cleaning system of the ore crushing. Electric power supply to the crushing plant is provided and installed by Sotkamo Silver.

The crushing cost effect is included in operational cost calculation.

The Grinding circuit ore goes through the 3-stage flotation. Nearly all noble metals and lead are separated in the first stage of Silver-lead flotation. In the second stage of the flotation, i.e. Zinc-flotation, zinc is separated from the ore slurry. And the third flotation stage is for the Pyrite-flotation separation.

Flotation concentrates are dried in their own dewatering circuits equipped with a concentrate thickener and vacuum filter equipment. Thickener and vacuum filter water are forwarded to the concentrator plant circulating water tank.

Dried concentrates are conveyed to the concentrate storage rooms situated in the Process Plant Building (PPB). Concentrates will then be transported by trucks to the smelting plants.
10.2 Mine

10.2.1 Mine design criteria

The Mine operation time is 16 h/d and 5 days per week, totally 80 h/week.

10.2.2 UG and Open pit water

UG mine consists of four UG mine pumping stations (1 – 4). Pumping stations are located at four levels of the UG mine:

- pumping station 4 at the level 75
- pumping station 3 at the level 165
- pumping station 2 at the level 250
- pumping station 1 at the level 335

Pumping stations are equipped with raw water pools with sedimentation basin and local level controls. UG mine water is pumped from the lower level pumping station to the next level pumping station and so on and finally to the ground level. Mine head water is pumped with the immersion pumps to the nearest UG mine pumping stations.

UG and open pit drainage waters will be pumped to the tailings pond. This is shown in Chapter 11.10.5 Figure 41 Water treatment and balance flow sheet.

UG and open pit raw water pumping stations are equipped with level control switches, local control panels and reserve pumps and can be controlled from the pumping station’s local control panels.

10.2.3 Heating and ventilation

The UG mine ventilation and air heating design criteria is 100 m$^3$/s. The ventilation fan is two directional. Normal blowing direction is to the UG mine and in the emergency case the blower can operate in the opposite direction. Mine head ventilation is done with flexible ducts and fans.

UG mine is ventilated and ventilation air heated up to +5 °C by the LPG propane container heating plant which is placed at the top of the ventilation shaft. Maximum capacity of the plant is 4.6 MW when the temperature difference is 37 °C. This is also net energy demand situation.

Estimated annual energy demand is 5 000 MWh and pressure difference 3 500 Pa.

The UG mine potable water to the water containers, refuge and break rooms of the UG mine is transported by trucks.

The UG mine sanitary water system is equipped with two sanitary water tanks at the UG mine levels 225 and 330 which are emptied regularly. UG mine sanitary water tanks are transported by trucks and purged to the sanitary water purifying tank of the mine area.
10.2.4 Electricity and process control

Electrical power will be required underground for ventilation, pumping and drilling. The main consumers of the power are the electro-hydraulic drilling jumbo and ventilation fan. Other significant power demands will be at the other drilling equipment, water pumps, and mine head ventilation fans.

Electrical power will be provided via a single armoured 20kV voltage cable that will run along the drilling hole. At intervals along the ramp, portable power centre (PPC) consisting of skid mounted transformer with all associated switching and safety equipment will be located and fed from the main HV cable. It is estimated that 1 PPC on the level 240 will be required to service the mine. As development and mining proceeds the PPCs can be relocated to (the most) optimal locations to suit the current mining activities. Portable switchgear will be located close to each working head or stope to enable the drills to plug in when working on a particular sub-level. Auxiliary ventilation fans will also be connected to these portable switches to provide adequate ventilation at the working faces.

All pumping stations will be controlled from the DCS.

The total estimated installed load of UG and OP mine will be 962 kW.

10.2.5 Ore transportation and crusher supply

The ore is transported from the UG and OP mine to the Run of Mine (ROM) ore storage.

The ROM ore storage size will contain total 40 000 tons. Storage area foundation will be built leak-proof with 1.5 mm HDPE-liner and drainage water will be discharged to the settling pond 4.

From the ROM ore storage the ore is transported to the jaw crusher by the wheel loaders.

10.3 Crushing plant

10.3.1 Crushing plant design criteria

Operating data for the mine crushing and conveying plant is 250 t/h and grain size for the crushed ore is initially P<sub>80</sub> 25 mm.

Jaw crusher nominal product size is P<sub>80</sub> with the close side setting (CSS) 100 mm and 250 t/h capacity.

Belt conveyors’ feeding capacities are as follows: 250 t/h conveyor 1 250 t/h conveyor 2 and 450 t/h conveyor 3 (capacity of conv. 2&3 consists of the jaw and cone crusher capacity).

Vibrating Screen’s capacity is 450 t/h and screen under size (U/S) ore (P<sub>80</sub> 25 mm) is conveyed to the fine ore storage with the belt conveyor 6 and conveyor 7 with the capacity of 250 t/h. Vibrating Screen over size (O/S) ore is conveyed to the cone crusher.
Cone crusher nominal product size is $P_{80} = 25$ mm (CSS) and nominal capacity is 197 t/h (max. 250 t/h). The feeding capacity of the cone crusher belt conveyor (conveyor 4) is 200 t/h.

The crushing plant operating time will be 5 days per week and 16h/d which corresponds to average 168 t/h crushed ore feed capacity to fine ore storage (350 000 t/a). It has been assumed that the crushing equipment availability will be 90%.

The crushed ore ($P_{80} = 25$ mm) will be stored into the 3500 ton fine ore storage with 3 days buffer storage to enable the concentrator plant to run when the crushing plant is not operating.

Operating data for the mill feed is 350 000 – 450 000 t/a and grain size for the mill feed is $P_{80} = 25$ mm.

The Ore feeders’ (2 pcs, later on 4 pcs) capacities (each) are 70 t/h and mill feed conveyor capacity is 100 t/h.

The Mill feed operating time will be 7 days per week and 24 hours per day corresponding to 45 - 55 t/h crushed ore feed capacity to ball mill (350 000 – 450 000 t/a). It has been assumed that the mill feed equipment availability will be 95%.

Principle of the crushing and mill feed process is shown in Figure 36.

### 10.3.2 Crushing and Fine Ore Storage

The crushing plant comprises a complete mine handling with the following main equipment: receiving hopper and feeder (frequency converted), a jaw crusher (1-stage crusher), iron separation, vibrating screening (2-deck screen), a cone crusher (2-stage crusher) and belt conveyors, fine ore storage’s belt feeders and all auxiliary equipment.

From the Jaw crusher ore is conveyed via belt conveyors and iron separation to the vibrating screen. Two deck vibrating screen U/S ore ($< 25$ mm) is conveyed via belt conveyors to the crushed fine ore storage.

Vibrating screen O/S fraction ($> 25$ mm) is forwarded to the cone crusher. From the cone crusher ore is forwarded back to the vibrating screen.

Jaw and Cone crusher are equipped with own emergency stops and local control panels. All conveyors are also equipped with own emergency stops, local control switches and the overhead covers. The system profile is presented in Appendix 6 Crushing and Conveyor system profile.

The following crushing plant equipment items are placed in crushing area for dust formation and noise minimization: a jaw crusher, vibrating screening and a cone crusher.

Crushed ore will be conveyed to the fine ore storage, which is equipped with canopy to prevent the crushed ore storage from wetting and freezing.
10.3.3 Mill feed

The crushed ore will be conveyed from fine ore storage to the concentrator plant rod mill.

The concentrator plant rod mill feed comprises four crushed ore storages’ feeders (each frequency controlled), rod mill feed conveyor, belt conveyor weight meter and all auxiliary equipment.

Ore is conveyed to the rod and ball mill by the ore feeders and mill feed conveyor. The rod mill feed conveyor is equipped with a belt conveyor weight meter. Normally one or two feeders are running with the constant speed for the base load of the mill feed and one of the feeders is used for the control purpose.

The ball mill product grind size is $P_{80}$ 85 µm. The ball mill is equipped with variable speed drive and the designed capacity is 45 - 55 t/h.

All ore feeders and the rod mill feed conveyor are equipped with own emergency stops and local control switches. Ore feeders and rod mill feed conveyor maintenance run is performed from the local control switch and main control from the Concentrator plant DCS. Rod mill feed conveyor is equipped with an overhead cover.

10.3.4 Utilities

10.3.4.1 Compressed Air

A compressed air will be used for the process control and maintenance of the equipment. The service will be outsourced.

10.3.5 Electrification

Electric power supply to the crushing plant is provided and installed by Sotkamo Silver.

The total estimated installed load will be 507 kW.

10.3.5.1 Variable Speed Drives for Ore storage and conveyor system

The drives would be either included in the MCC's for the smaller sizes or in free standing IP20 cabinets for inclusion in the electrical room.

The units will be microprocessor based employing vector control without sensors, with the switched mode power supply fed from the DC link. There will be a keypad on the front of the cabinet for inverter function programming and a LED display.

10.3.5.2 Cables

Plant cables will be armoured with aluminium or copper XLPE/PVC insulated. Office, light power and lighting cable will run in conduit unarmoured.
10.3.5.3 Control and instrumentation for crushed fine ore storage and conveyor system

The power for the control system hardware and the field instruments will be supplied from Uninterrupted Power Supplies (UPS) in the electrical room in the PPB.

The UPSs will be supplied from 400V feeder compartments in the MCC.

A distributed control system consisting of Distributed Control System (DCS) will be used.

The status of the drives indicating MCC ready, emergency stops ready, field local start and drive running will be fed to the DCSs. A run signal will be transferred from the DCS to each drive.

The instrumentation will be microprocessor based and will supply data to the control network for mass balance measurement, level, density, flow, pressure and temperature measurement.

The operator interface to the plant will be at the control room located in the PPB. The system will have an Ethernet interface for connection to the plant control system network.

All drives will be started and stopped from the DCS. Some drives will have a field start/stop station. All stops and emergency stops and personal protection devices, like conveyor pull-keys, will be hardwired into the MCC starter circuits. Conveyors, crushers, mills, agitators, etc. will only be started after the audio/visual start up warning has been activated for 15 seconds.

10.3.6 Process control

10.3.6.1 Crushing plant process control system

Jaw and Cone crusher and their auxiliary devices main process control is conducted from the local control panels from the field and alternative control panel from the Concentrator plant DCS.

Crushing plant conveyors main control is conducted from the Concentrator plant DCS and maintenance run is conducted from the local control switches.

The Crushing plant will be controlled from the Central Control Room (CCR) in the PPB. The control system will be designed for plant control.

The control and instrumentation philosophy of the crushing plant is to provide minimum instrumentation as possible, if there is no other economically or process justified reasons requiring more sophisticated automation.

The control system will be based on a proved and reliable Distributed Control System (DCS) hardware and software, which will be installed in the Automation Room of the PPB.
The control system will interface with the plant instrumentation via DCSs and input/output (I/O) modules, which will be housed in remote I/O cabinets located in the plant.

The control system will control the plant's motors via drive interface modules - Simo-codes or similar (Profibus) - installed in individual MCC compartments.

The control system will communicate with the operator interfaces (computer screens & keyboards), the plant instrumentation (remote I/O cabinets) and the drives Micro Computer Control (MCC) via network cables on Profibus DP.

The power for the control system hardware and the field instruments will be supplied from 2 uninterruptible power supplies (UPS) - the main one and a standby - in the CCR.

10.3.6.2 Main controllable process variables and instruments

Crushing plant Instruments will be provided to monitor, control and record the main process variables:

- Rod mill feed conveyor tonnages
- Crushers’ power draw

DCS system software will be programmed to provide control for:

- Mill optimization - the tonnage to the ball mill will be measured with a weight meter on the rod mill feed conveyor.

10.3.6.3 Manual control

The following activities and parameters will be manually controlled or measured:

- Mill feed samples for moisture determination

10.3.7 Crushing plant balance

Crushing plant feed curve, Ore stockpile output curve and estimated balance are shown in Figure 35.
Figure 35  Crushing plant balance flow sheet
10.3.8 Crushing plant layout

The philosophy of the crushing plant layout is based on confining all major process equipment where it would be possible to minimize dust and noise in the crushing plant area and implement the easiest structure for common overhead crane for the equipment maintenance. The crushing plant will be surrounded by 8 m high constructed barren rock embankments.

Mill feeders are located below the crushed ore storage inside the conveyor tunnel. The crushed fine ore storage is covered with steel framework canopy.

10.3.9 Crushing plant flow sheet

Crushing plant flow sheet is shown in Figure 36.

Figure 36 Crushing and mill feed, process flow sheet
10.4 Concentrator plant

The concentrator plant design criteria, process concept and production figures are based on GTK study presented in Chapter 8. The following is based on quotation.

Sotkamo Silver has indicated that the annual capacity for the processing plant will be from 350 000 tpa to 450 000 tpa ore. This quotation covers the ore feeding into fine ore stockpile and into grinding, flotation, dewatering and reagent preparation and distribution in the processing area.

Process design is based on 2011 batch & pilot test work results, because the used test sample quality (metal grades) seems to be fairly close to expected ore reserve grades.

10.4.1 Process description

10.4.1.1 General

Test work has indicated that by using conventional selective flotation process separate Pb, Ag, (Au), Zn and S (pyrite, pyrrhotite) concentrates can be produced. The simplified process flow sheet will be summarized below:

- Ore will be initially crushed down to 25 mm (P80) by a contractor
- Crushed ore will be reclaimed from the ore stockpile by feeders into 2-stage grinding in rod and ball mills and ground down to P80=85 µm.
- Hydro cyclones will be used as classifiers to ensure suitable ore slurry with regards to particle size will be achieved for efficient mineral separation
- The flotation circuit consists of three consecutive flotation stages to recover lead, silver and gold into a precious metal into Pb concentrate, zinc into Zn concentrate and the rest of sulphide minerals into S concentrate to remove all remaining sulphides from the tailing
- All concentrates will be pumped into high rate thickeners, from where the thickened concentrates are pumped into storage tanks prior to filtration

10.4.1.2 Grinding and classification

The contractor will take care of the crushing of the ore down to 25 mm size (P80) and delivers the crushed ore (chute being contractor’s battery limit) on a fine ore conveyor belt discharging the ore into fine ore stockpile. The ore is reclaimed by variable speed feeders onto the belt conveyor, which feeds the ore into the mill. The ball mill is operating in closed circuit with hydro cyclones. Hydro cyclones overflow (25-30% solids) at P80=73 µm fineness will enter the Pb circuit conditioner. The speed of the ball (VSD) mill can be adjusted to suit the energy needs in grinding.

10.4.1.3 Flotation

The design of the flotation circuit is based on 350 000 tpa capacity, but space will be allowed to increase flotation equipment volume at a later stage should needs arise.
Pb flotation consists of a 20m³ conditioner and 4 x 20m³ tank cells for rougher and scavenger flotation with residence time being 27 minutes. Pb rougher concentrate will be cleaned in three consecutive stages while cleaner tails are returned downstream. Scavenger concentrate and the 1st cleaner tails are returned to the head of Pb flotation circuit. All Pb cleaner circuit cells are 5 m³ tank cells and they allow long residence times in flotation.

Pb flotation tailing is pumped into Zn circuit flotation conditioner (20 m³). Zn flotation circuit consists of rougher and scavenger flotation (5 x 20m³ tank cells). Zn rougher concentrate is transferred into Zn 2nd cleaner flotation (1 x 5m³) and Zn scavenger concentrate respectively into Zn 1st cleaner flotation (2 x 5m³). 3rd Zn cleaner is carried out in 1 x 5m³ tank cell. Zn cleaner circuit is configured by pumping concentrates upstream (similar to Pb cleaner circuit) and cleaner tails flow by gravity downstream.

The tailing from Zn flotation is pumped into S flotation conditioner. S flotation circuit consists of a conditioner 20m³, 3 x 50m³ rougher, scavenger cells. S rougher concentrate is not cleaned, but pumped into thickener.

The pilot test work ore sample had following grades:

- Pb 0.35%
- Ag 98 g/t
- Au 0.46 g/t
- Zn 0.79%
- S 1.24%

GTK has studied Silver Mine mineralogy and based on those findings the silver minerals occur mostly liberated in flotation feed. Due to Ag-minerals occurrence mode they will report mostly into Pb concentrate; same comment applies to chalcopyrite. Ore contains also carbonates up to 5-9% (mostly ankerite Ca-Fe carbonate).

### 10.4.1.4 Dewatering

All concentrates will be pumped into thickeners. Thickener underflows, typically at least 60% solids, will be pumped into agitated storage tanks prior to filtration. Pb, Zn and S concentrates will be filtered in a joint Pressure Filter and conveyed into concentrate storage.

### 10.4.1.5 Reagents

Flotation process requires following reagents:

- Aerophine 3418A (collector)
- Sodiumisobutylxanthate (SIBX) (collector)
- Frother, MIBC
- Frother, Dow 250
- ZnSO₄ (Zinc depressant)
- CuSO₄ (Zinc activator)
• Dextrin (modifier in Pb flotation)
• Ca(OH)$_2$ (pH control)

Table 18 Flotation reagents dosage

<table>
<thead>
<tr>
<th>Consumption, g/t</th>
<th>g/t ore feed</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Pb-flotation</strong></td>
<td></td>
</tr>
<tr>
<td>Aerophine 3418A</td>
<td>13</td>
</tr>
<tr>
<td>MIBC, g/t</td>
<td>47</td>
</tr>
<tr>
<td>ZnSO$_4$, g/t</td>
<td>725</td>
</tr>
<tr>
<td>Dextrin, g/t</td>
<td>21</td>
</tr>
<tr>
<td>Ca(OH)$_2$</td>
<td>200</td>
</tr>
<tr>
<td><strong>Zn-flotation</strong></td>
<td></td>
</tr>
<tr>
<td>CuS 04, g/t</td>
<td>153</td>
</tr>
<tr>
<td>SIBX g/t</td>
<td>23</td>
</tr>
<tr>
<td>MIBC, g/t</td>
<td>47</td>
</tr>
<tr>
<td>Ca(OH)$_2$, g/t</td>
<td>2,000</td>
</tr>
<tr>
<td><strong>S-flotation</strong></td>
<td></td>
</tr>
<tr>
<td>SIBX g/t</td>
<td>23</td>
</tr>
<tr>
<td>Frother, Dow 250</td>
<td>61</td>
</tr>
</tbody>
</table>

The estimated consumption rates come from the GTK pilot test work.

Flocculant addition is needed for the efficient thickening for the concentrates. The dosage rate for flocculant is 20 g/t feed into each concentrate thickener.

10.4.2 Concentrating process flow sheet

The Concentrating process flow sheet is presented in the Figure 37.

The Process plant layout is presented in Appendix 7 and 8.
Figure 37  Concentrating process flow sheet
10.4.3 Utilities

10.4.3.1 Process Water

Process circulation water will be stored in a 45 m$^3$ circulation water tank within the process plant building (PPB) and is serviced with an external line from the Tailings Management Facility pump station 3 (PS 3). In the normal run all process water comes to the concentrator plant from settling pond 2’s circulation PS 3. Besides individual lines are directed to circulation water tank from the concentrate thickeners and concentrate vacuum filter.

From Olkilahti pump station (PS 1) raw water is used for the circulation water tank feed, if the circulation water pump PS 3 is not in operation, for example, in maintenance situations. Normally, Olkilahti PS 1 raw water is recommended to be used only for tailings pond start-up situations and settling pond circulation water reservoir water quality problems.

The circulation water tank’s circulation water is pumped to the concentrator plant circulation water network with the frequency controlled pump.

Circulation water tank is equipped also with a fire water pump (diesel pump) serving as the plant fire water network and washing water pump serving as the plant washing water network.

Circulation water is used for:

- Rod and ball mill dilutions and consistency control water
- Flotation cells chute washing
- Pump sumps level control
- Plant washing water posts
- Fire water

10.4.3.2 Raw Water

Raw water is pumped from the Olkilahti PS 1 or mine area PS 4 to 15 m$^3$ Raw Water tank within the PPB. Normally, mine area raw water PS 4 is used for the raw water tank feed. If raw water PS 4 is not in operation, for example, in maintenance situations or poor water quality/reserve, Olkilahti raw water from PS 1 can be used as a raw water back-up pumping station.

The raw water tank’s water is pumped to concentrator plant raw water network with the frequency controlled pump. The chemical system raw water pump of the raw water tank is serving chemical dilution only. Sealing water is made from the plant raw water with the sealing water filters. Sealing water is pumped to the plant sealing water network with the raw water tank sealing water pumps.
Raw water is used for:

- Slurry and vacuum pumps sealing water (water after sealing water filters)
- Reagents dilutions
- Flotation cells chute washing (optional)

10.4.3.3 Compressed Air

Two compressors will be installed in the concentrator plant to supply instrument air to the concentrator plant automation valves and process equipment and mill air for maintenance use.

Mill air and instrument air will be provided by means of duty and standby compressors. Air compressors’ are air cooled. All pressure air (instrument and mill air) will be filtered and dried.

Pressure air centre is equipped also with 2 – 3 pressure air reservoir tanks.

The concentrator plant laboratory will be connected to concentrator plant pressure air system.

A small mobile air compressor can be also used in the workshop for maintenance of equipment.

10.4.3.4 Heating and ventilation

The Process plant building (PPB) will be heated electrically and equipped with mechanical ventilation with heat recovery. The maximum amount of ventilation is dimensioned according to heat generation of the process.

The non-heated concentrate storage will be equipped with exhaust ventilation.

The PPB is equipped also with several local exhaust ventilations, which are controlled locally when necessary. Electric equipment rooms and CCR is equipped with mechanical ventilation with heat recovery and cooling.

The waste and chemical waters of the laboratory are collected to cesspit. The cesspit is equipped with level alarm. The waste and chemical waters of the laboratory are pumped to container and transported to hazardous disposal plant.

10.4.4 Electricity

10.4.4.1 Concentrator plant Load

The total installed load will be 2 215 kW.
10.4.4.2 Motor Control Centres

The free standing sheet steel cabinets are to be protected up to IP30, front connected units arranged in vertical tiers with top and bottom cable ways. The units shall be of withdrawable or fixed type according to the rating. The MCCs will be mounted in electrical rooms with closed ventilation system. Intelligent motor controller and protection unit for each starter with link to DCS shall be included.

10.4.4.3 Variable Speed Drives

The drives would be either included in the MCCs for the smaller sizes or in free standing IP20 cabinets for inclusion in the electrical room.

The units will be microprocessor based employing vector control without a sensor, with the switched mode power supply fed from the DC link. There will be a keypad on the front of the cabinet for inverter function programming and a LED display.

10.4.4.4 Mill Starter

There will be inverters to give 'soft starting' to the squirrel cage motors on the rod and ball mill drives. The inverters supplies will be taken directly from circuit breakers on the main switchboard.

10.4.4.5 Cables

Plant cables will be armoured with aluminium and copper XLPE/PVC insulated. Office, light power and lighting cable will be running in conduit unarmoured.

10.4.4.6 Control and instrumentation

The power for the control system hardware and the field instruments will be supplied from Uninterrupted Power Supplies (UPS) in the electrical room in the PPB.

The UPSs will be supplied from 400V feeder compartments in the MCC.

A distributed control system will be used consisting of Distributed Control System (DCS).

The status of the drives indicating MCC ready, emergency stops ready, field local start and drive running will be fed to the DCSs. A run signal will be supplied from the DCS to each drive.

The instrumentation will be microprocessor based and will supply data to the control network for mass balance measurement, level, density, flow, pressure and temperature.

The operator interface to the plant will be at the control room located in the PPB. The system will have an Ethernet interface for connection to the plant control system network.
All drives will be started and stopped from the DCS. Each drive will have a field start/stop station. All stops and emergency stops and personal protection devices, like conveyor pull keys, will be hardwired into the MCC starter circuits. Conveyors, crushers, mills, agitators etc. will only be started after the audio/visual start up warning has been activated for 15 seconds.

10.4.5 Process control

10.4.5.1 Concentrator plant process control system

Rod and Ball mills are controlled from the concentrator plant DCS and their auxiliary devices control is happening from the Mills’ local control panels and alternative control of the auxiliary devices of the Mills is done from the DCS.

Concentrator plant slurry and water pumps, flotation machines’ rotors and equipment and all utilities’ systems’ main control is monitored from the Concentrator plant’s DCS.

Local control panels and switches are used for the process controls of the equipment, only if it’s the easiest and reasonable way to carry out process control. These kinds of process systems are, for example, pressure air centre, concentrate filtering and chemical systems.

The control and instrumentation philosophy of the concentrator plant will be to keep such instrumentation to a minimum, except for the cases where there is a valid economic or process-justified reason for more sophisticated automation.

The plant will be controlled from a Central Control Room (CCR) in the PPB. The control system will be designed for plant control by 2 operators (the other operator station together with engineering station).

The control system will be based on proven and reliable Distributed Control System (DCS) hardware and software, which will be installed in the Automation Room of the PPB.

The operator interface with the plant will be via Industrial PC, Keyboard, Mouse and LCD computer screens and printer.

The control system will interface with the plant instrumentation via DCS's and input/output (I/O) modules, which will be housed in remote I/O cabinets located in the plant.

The control system will control the plant's motors via drive interface modules - Simocodes or similar (Profibus) - installed in the individual MCC compartments.

The control system will communicate with the operator interfaces (computer screens & keyboards), the plant instrumentation (remote I/O cabinets) and the drives Micro Computer Control (MCC) via network cables on Profibus DP.
The power for the control system hardware and the field instruments will be supplied from 2 uninterruptible power supplies (UPS) - one main and one standby - in the CCR.

10.4.5.2 Main controllable process variables and instruments

Instruments will be provided to monitor, control and record the main process variables:

- Mill power
- Cyclone feed density
- Water flows to the concentrator plant and to the Mills
- Cyclone pressure
- Main equipment power
- Courier analyzer
- Particle size analyser

DCS system software will be programmed to provide control over the following:

- Water addition to the mill will be proportional to the ore feed rate to maintain a constant slurry density in the mill.
- The level in the mill discharge sump will be maintained by dilution to prevent pump cavitation.
- The speed of the cyclone feed will be variable to enable the feed to the cyclones to be maintained at constant density and cyclone feed density will be measured for this purpose
- The flotation cells will be provided with automatic control of slurry level and air addition rate.
- The Thickeners will be provided with slurry density measurement

Courier analyser system will be used for process management, monitoring and control. The analyser contains following:

- Element concentrations measurement (XRF)
- Flash flotation control
- Primary rougher flotation control
- Rougher control
- Cleaner control
- Scavenger control
- Flotation circuit control

10.4.5.3 Manual control

The following activities and parameters will be manually controlled or measured:

- Collection of samples from the concentrate storages for the determination of moisture and metal contents
10.4.5.4 Process control system architecture

Plant automation system architecture is presented in Appendix 9 and shown in the Figure 38.
Figure 38  Automation system architecture
10.4.6 IT-Architecture

Local Area Network (Office network) will be built as star topology where the PPB is a central point. Optical fibre cables will be built from the PPB to the Crushing and Mine Area.

Printers are centralized model. Office PCs consist of a required operation system and office programs. Local server will be situated in the PPB.

10.4.7 Laboratory facilities

The geological sample handling laboratory (G-lab) is situated inside the drill core storage. The G-lab will be an enclosed, insulated and heated room inside the storage. Geological samples include, for example, rock boulders, drill cores, RC- and chip samples. In the G-lab there has to be a saw for cutting the cores and also a jaw crusher and, for example, a ring mill for pulverizing the samples. Also equipment for drying the samples is needed. Riffle splitters are needed to split the pulverized samples. Roll tables are needed for drill core logging. There has to be also enough space for occasional drill core cases. General furnishing with tables, lockers and chairs, is also naturally needed.

The process sample handling laboratory (P-lab) is situated inside the PPB adjacent to the sample analysing laboratory rooms. Process samples are mainly in the slurry form and are from different points of grinding and flotation processes. Also some other samples, for example, raw materials, chemicals, may be handled in this laboratory. In the P-lab there have to be vacuum filtration units for dewatering of slurry samples. Also heating plates and oven are needed for drying. Sieving machine, sieves and balances are needed for particle size analysis. Sample splitters and spinning riffles are needed for sample dividing. Furnishing has to fit a normal wet laboratory style. Services including all laboratory equipment of the P-lab will be outsourced.

The sample analysing laboratory (A-lab) is situated inside the PPB adjacent to P-lab rooms. Samples for the analysing laboratory are pre-treated in G-lab or in P-lab. Also, some other samples (raw materials, environmental samples) arrive to A-lab. Analysis equipment includes AAS, XRF, ICP-OES and UV-VIS. Services including all laboratory equipment of the A-lab will be outsourced.

10.4.8 Mobile equipment

Mobile equipment in and around the plant area will comprise a 2t gas operated hydraulic forklift. The wheel loader will be used for the loading of the concentrates.

The light vehicle will be required for the maintenance inspection of the Mine Area.
11 PROJECT INFRASTRUCTURE

11.1 Mine area general infrastructure

Existing infrastructure considers 1.8 km long forest road as an access road from road 9005 to the decline access. From the decline access there is narrow, 3 m wide and 600 m long, side rock paved dumper road to the ventilation shaft. 20 kV back-up power line accesses the ventilation shaft from NNE. The 110 kV/45 kV transmission line alongside the planned access road R1 to the 45/20 kV main substation is built.

In the western side of decline access there is former side rock area holding approximately 80 000 m³ rock which will be levelled down to +211 ASL as substructure of the process plant area and excessive masses will be used to build the substructure of the new access road.

Process plant area will be 5.5 ha large and will include process plant building (PPB), crushing plant, office/laboratory building, drill core storage with workshop, Run of Mine (ROM) ore storage, 45/20 kV main substation, bio-chemical sewage disposal, parking area and truck park. In the East of the decline there will be a 2 ha area for contractors. Plant area will be situated on 600 m SW of ventilation shaft.

PPB will be 1 939 m² heated and insulated building with steel framework, which includes central control room (CCR) and all related process equipment. Concentrate ore storage will be 995 m² and situated as an adjacent non-heated building with similar structure.

The primary crushing area is designed to be surrounded by 8 m high constructed side rock embankments. The primary crusher will be located approximately 100 m SE of the PPB next to the ROM. Crushing will be operated by the contractor.

Major part of the existing access road operates as a public road and upgrading it can cause unexpected expenses due to fairly deep peat layers in its middle part. The new access road R1, 970 m, from north will be built straight from the road 9005.

Tailings Management Facility (TMF) will consist of 28 ha tailings pond, 1.05 ha settling pond 1, 5.1 ha settling pond 2, 1.5 ha settling pond 3 and 2.3 ha wetland treatment area. TMF will be located SW from the process plant area. Settling pond 4 will be 810 m² and situated 100 m NE from ROM.

Bio-chemical sewage disposal will be located between process plant area and TMF.

Explosives container will be located in 100 m SE from the ventilation shaft.

11.2 Power supply and distribution

Designing of the distribution of 20 kV and 690V network, transformers and frequency converters is based on the quotation.
11.2.1 Total power consumption

Assumed total consumption will be about 4.1MW.

- Underground Mine / kW 1 162
- Mine area / kW 206
- Crushing / kW 507
- Concentrator (PPB) / kW 2 215

All “stand by loads” are separated, but rest loads are assumed to be running with electrical power equal than their mechanical rated power. Assumed loads have rather high margin.

Mine is running with full load 947kW about 80h/week. Rest about 88h/week load is about 500kW lower.

Some loads of Mine Area and Crushing plant may still have some changes on the final design, e.g. pumps and building electrification.

11.2.2 Recommended transformer capacities

Dimensioning of the main transformers will be based on maximum average power, including possible growth in the future.

- Underground Mine 1.0 to 1.25 MVA
- PS 1 Olkilahdi 0.1 to 0.2 MVA
- Mine area 0.5 to 0.63 MVA
- Crushing & Concentrator 2.5 to 3.15 MVA (combined)

For lighting and other building electrification purposes is required 230/400V voltage distribution with 315…630kVA transformer.

11.2.3 Transmission line 110 kV/45kV, main incomer

Total length of the transmission line is 20.2 km and starting point is at UPM’s hydropower-station in Katerma. The line ends up at the process plant area of the Silver Mine. The connection from Katerma to the UPM’s line is with earth cable 110 kV/45 kV to transformer station. In the Silver Mine area, there is a 45 kV/ 20 kV transformer station.

11.3 Electricity

The mine area total power for installed total load in mine area will be 206 kW.

11.3.1 Electric network layout

Main switchgear will be with main incomer and back-up incomer.

Medium voltage main switchgear will be in the PPB because of the largest loads. Also MCC of building electrification will be located in the PPB.
Distribution transformers located far from main medium voltage switchgear will be connected via RMUs as a chain.

Closed ring connections are not required, but some cable routes to mine may have increased risk of damages and some routes might be critical due to difficulties in repairing possible damages on the cable.

Incomers or incomers and a sectionalizer have to be equipped with interlocking which prevents parallel connection of these two lines. Lay-out of electric network is presented in Figure 39.
Figure 39 Electric network lay-out
11.3.2 Protection

Each part of the network has to be equipped with an independent short circuit and earth fault protections. For short-circuit protection three phase over current relays are needed. For earth fault protection either earth fault current protection or residual voltage protection or both are needed.

Both short-circuit and earth fault protections have to be coordinated with externally connected networks. Utility companies will give more accurate instructions or requirements.

11.3.3 Earth fault protection at 20kV

Both incoming feeders and all outgoing feeders at main switchgear will be equipped with sensitive directional earth fault protections. Feeders at RMUs don’t need earth fault protection.

Operation of earth fault protection depends on the magnitude of earth fault current. With very low current identification of the faulty feeder might be difficult or even impossible. Safety can still be guaranteed with residual voltage based protection, which trips incoming feeders.

Earth fault current is received from system grounding or capacitances of wires. Shielded cables generate higher earth fault current than overhead lines. System grounding can be executed by grounding neutral point of main transformer or by grounding neutral point of separate grounding transformer. That current is preferred to be limited to 5-10A by a separate resistor.

Earth fault currents of 20kV wires:

- Overhead line 0.067 A/km
- PAS-wire 0.054 A/km
- Cables 1.3-2.1 A/km (typical cross sections)

Earth fault current is the lowest on isolated networks with short cable length.

Protection would then be able to detect earth faults on all the other feeders but mine feeder. Residual voltage protection with two steps might be suitable for that feeder. With time delay, which is coordinated with earth fault protections of other feeders, it would first trip the mine feeder and after some delay the incomer. If both the mine and the pumping station have shielded cables, earth fault current would be high enough for protection to operate properly.

System grounding via 5-10A resistor would provide a lot of current for protection of all feeders to operate properly.
11.3.4 Re-closing

Relaying of line will have re-closing operations. Re-closing will be activated on earth faults and on short circuits. Re-closings automatically connect voltage back to the line after fault. Quite commonly insulation level of faulty overhead line will be restored during short breaks. Typical reason for this kind of faults is lightning and also some sticks or birds on the lines. Re-closing schemes have typically one to three fast auto re-closings shots with 300…1 000 ms dead time and 100…500 ms burning time and one long time shot with dead time of 30…60 s.

It still should be noted that during the first dead time of auto re-closing all processes will be lost. Anyhow the line would automatically become available, if re-closings were successful. If the last re-closing fails, the line will be left dead and fault has to be localised by using fault patrol.

11.3.5 Cables

Because short-circuit current level are so low, also quite low cross sections can be used without jeopardising their thermal strength.

11.3.6 RMU’s disconnectors

Distribution transformers which are close to the main switchboard don’t need a separate disconnector in front of them. Anyhow distribution transformers on the pumping station and down on the mine would need a disconnector or load disconnector in front of them. Down on the mine RMU with a load disconnector is preferred. Gas insulated construction is preferred. If connection line to pumping will be installed to poles, overhead line or pole mounted cable, disconnector on the pole would be acceptable. With underground cable RMU with a load disconnector is preferred.

11.3.7 Spare feeders

Reservation for additional loads in the future can be the base for spare capacity on MCCs, installed spare feeders on switch boards or for just empty space for additional equipment. On the cable network any cable which has enough capacity can be cut and can be divided to two or more distribution transformers by new RMU-unit.

11.3.8 Single line diagrams

Silver Mine 20kV single line diagram shown in the Figure 40.
11.4 Fuel supply

Plant fuel supply will be provided with the contract work with the local suppliers. Minor fuel tank 1 m$^3$ will be located close to the PPB. A diesel and fuel oil station (storage max 9.9 m$^3$) will be located near the ROM pad to enable the fuelling up of mine trucks and the ROM Loader.

11.5 Heat production and distribution

Heating for buildings will be carried out by electric heating, e.g. remote control room, laboratory, office and concentrator in the case of shutdown. The concentrating process itself is heat generating and heat recovery will be utilized.

11.6 ICT, Communications and Video surveillance

The mine area will be connected with 100 MB internet connection. The Offices, control rooms and laboratories are connected to the mill LAN network. The mill area consists also WLAN network. The connections outside are protected with virus protection and firewall. Mill network can be accessed outside mill area via VPN connection.
The process surveillance and area monitoring will be done with CCTVs (which are mainly remote controlled). The location of CCTV and monitored points are:

- main access gate
- process plant area camera
- open pit
- UG mine entrance
- crushed fine ore storage
- mill feed
- flotation cells, 3 pcs
- filters, 3 pcs
- concentrate rooms, 3 pcs
- tailings pond

The telephone connections are based on mobile phones. A separate base station will be located to the area.

11.7 Access road R1 and other roads

The new access road will be 7 m wide paved with 150 mm thick crushed aggregate layer. Substructure will be constructed displacing the peat with coarse side rock. Peat thickness varies between 0 – 4.7 m. Peat thickness data is based on GTK peat report. Vertical alignment of the road from natural ground level will be 1.5 m and road slopes 1:2.

All other roads R2 - R13 will be built with same principle and gradually at need with other construction. The location of roads is presented on Appendix 5 Mine area layout.

11.8 Process Plant Area Buildings

11.8.1 Crushed fine ore storage

The basement of the storage will consist of concrete bed and foundations. Foundations will consist of tunnel for feed conveyor to the PPB. Dimensions of the tunnel will be 40 m length, 4 m width and 3 m height. Fine ore storage will be covered by canopy with steel framework and PVC coated polyester fabric cladding.

11.8.2 Process plant building (PPB)

The PPB will include the concentrator and all other related process equipment. An elevated control room, electrical switch room and concentrate storage will be built inside the PPB. The plant office and reception with laboratory will be located adjacent to the PPB.

The PPB will be serviced by a 15 t overhead maintenance crane extending the full length of the building.

The PPB will be an insulated building with steel framework and PVC coated polyester fabric cladding.
11.8.3 Office building with laboratory

Office building and facilities will consist of leased barrack with steel cladding. The laboratory will be adjacent with office building.

11.8.4 Drill core storage and workshop

Core storage and workshop, total of 300 m², will consist of leased building with steel framework and PVC coated polyester fabric cladding. The floor will be asphalt; core logging will be handled in the insulated and heated part of the storage total of 160 m².

11.8.5 Explosives container

Explosive storage will be in an insulated 40 ft steel container by contractor.

11.8.6 Fencing

The 20/45 kV main substation will be fenced with 2 m high chain wire fence and will have locked gate.

11.8.7 Site safety and security

First aid facilities will be located in the office building. There will be basic wall mounted first aid kits through the PPB.

The process plant area, mine area and the access control point will be well-lighted. The access road is monitored from the PPB control room and will be equipped with gate.

11.8.8 General maintenance

Security, cleaning service, snow removal, cesspit emptying and waste management will be done by local suppliers.
11.9 Tailings management facility (TMF)

The Tailings Management Facility (TMF) will consist of a 28 ha tailings pond, 1.05 ha settling pond 1, 5.1 ha settling pond 2, 1.5 ha settling pond 3, approximately 810 m² settling pond 4 and 2.3 ha wetland treatment area. TMF will be located in 500 m SW from the process plant area except SP 4 which is located in 100 m NE.

The facility works as a settlement pond system to separate the tailings from the accompanying water and also as a final storage of the majority of tailings. The TMF in its final size is designed to hold tailings of 9 years operation.

At the lowest natural ground level of the TMF the pond dam is designed to be constructed from +210m ASL to an elevation of +218.90 m ASL. The tailings pond HW-level will be +217.40 ASL and +217.20 ASL in the settling pond.

The TMF is designed according to the Environmental Permit and the Pond dam structures have to fulfil the requirements of the Dam Safety Instructions. On the TMF area is located a few good building stone deposits which will be utilized for construction.

TMF is presented in Appendix 10 Tailings Management Facility.

11.9.1 Tailings pond (TP)

The Tailings pond will be constructed in two stages. TP 1st stage will consist 14.7 ha (420 000 m³) and TP 2nd 10.3 ha (150 000 m³). The volumes are defined from HW-level.

According to the Environmental Permit as large area as possible of the bed layer of the pond is to build up of natural compressed or constructed peat layer. Alternative compaction structure can be carried out e.g. bentonite mat to fulfil the standards of the Permit.

The slopes of the dam will be 1:2.4-3.0 on the upstream and 1:2 on the downstream. The bentonite mat is required as the compaction structure of the upstream slope. The core of the dam will build up of till with the crest of 4m wide.

The tailings in the tailings pond will be covered with water to prevent dust emissions. The tailings will be pumped from the plant as slurry via an overland pipeline. The pipeline will be located around the dam and will beach tailings slurry through outlets. The tailings will be discharged from the perimeter wall through a spigot or equivalent delivery system with multiple open-end delivery stations to form a beach that slopes towards the centre of the facility.

Each of the open end delivery stations will comprise a slotted pipeline that extends down to the bottom of the facility and a valve to control the flow.

Two standard structures of the dam are presented in the Appendix 11 Standard cross-section of TP and SP dam and Appendix 12 Standard cross-section of TP dam at till slope.
11.9.2 Settling pond 1 (SP1)

The pond will be excavated on a till hill to ensure that purged water’s quality is according to environmental permit regulations. The pond area is 1.05 ha and volume 23 000 m$^3$.

The natural peat layer of the bottom of the pond will be replaced with bentonite mat which is also required as the compaction structure of the upstream slope of the dam.

The dam will be built with similar structure as the tailings pond dam.

11.9.3 Settling pond 2 (SP2)

The pond 2 area is 5.1 ha and volume 190 000 m$^3$.

The natural peat layer of the bottom of the pond can be utilized but it has to be weighted e.g. geotextile/barren rock structure to prevent peat for rising or moving. The dam will be built with similar structure as the tailings pond dam.

11.9.4 Settling pond 3 (SP3)

The pond 3 area is 1.5 ha and volume 25 000 m$^3$. The natural till layer of the bottom of the pond can be utilized and the structure of the dam can be completely constructed of till with water permeability not more than natural till at the pond area.

11.9.5 Settling pond 4 (SP4)

The pond 4 area is 810 m$^2$ and volume 540 m$^3$. SP 4 will be excavated on a till hill in the south of barren rock area along the road R4. At the SW end of the pond there will be built pump station 2.

11.9.6 Wetland treatment

Wetland treatment area is 2.3 ha and on natural state with thick peat layer. From Wetland treatment water will be discharged to a ditch leading to Koivupuro trickle.

11.9.7 Marginal ore storage

Marginal ore storage will be 0.8 ha large and locate along west side of R4. The bed layer of the area can be built up of natural compressed (or constructed) peat layer. The margin ore storage area drainage water will be discharged to SP 4.

11.9.8 Barren rock area

Barren rock area will be 1.2 ha large and locate along west side of R4. The requirements of the foundations of the area are similar to Margin ore storage. The barren rock area drainage water will be discharged to SP 4.
11.9.9 Overburden disposal area

Overburden disposal area will be 2 ha large and located north of margin ore storage. Overburden can be built up on the natural peat.

11.9.10 Noise barrier

The noise barrier will be built on the NE side of the open pit. The most of the barrier material will be utilized overburden from the open pit.

11.9.11 Drainage

All drainage water from the process plant area, margin ore storage and barren rock area will be discharged to the SP 4 and then pumped to the tailings pond or SP 3. Drainage water from open pit and UG will be pumped to the tailings pond or SP 3.

11.10 Water circulation and supply

The plant area water system comprises Raw Water, Process Circulation Water and Potable Water. There will be water pumping stations in five different locations:

- Olkilahiti
- Settling Pond 4
- UG mine
- Open pit
- Tailings Management Facility (SP 2)

Water will be supplied also from smaller water sources: mine area raw water and potable water pumping stations, where water is collected from ground water.

Raw Water will be supplied from the Tipasjärvi Lake. Process Water will be extracted from the plant and recycled. The water balance is made up of mostly process water and any deficit will be obtained from decanted tailings water.

Potable water will be trucked in or pumped from an on-site well and stored in a tank within the PPB as required. Sanitary purified water is conducted to through the Wetland treatment to Koivupuro trickle.

The plant area water system water circulation control will be implemented so that the concentrator plant internal water circulation is as effective as possible. Most of the concentrator plant circulation water is pumped with solids to the tailings pond, where water is clarified from the solids.

Tailings pond’s water is conducted through the settling ponds from where circulation water is pumped back to the concentrator plant. Most of the settling ponds’ water consists of the tailings pond’s decanted water and UG mine raw water.
Tailing pond’s function is also to be long period settling pool for the extra water. Mine extra water is over flowed through the tailings and settling ponds to the wetland treatment and to Koivupuro trickle.

11.10.1 Process water

Process circulation water will be stored in a 45 m³ circulation water tank within the PPB.

In the normal run the most of the process water comes to the concentrator plant from the settling pond 2’s circulation PS 3. Circulation water consists of the settling pond 2 and TP’s rainwater, UG mine raw water and tailing pond’s clarification water.

Settling pond 4 pumping station’s water (PS 2) and open pit pumping station’s water pumped to the tailings pond or SP 3.

Olkilahti PS 1 raw water for the circulation water tank feed is used, if circulation water PS 3 is not in operation, for example, in maintenance situations.

Mine area process water pumping stations (1…3) are equipped with level control switches, local control panels and reserve pumps and pumps can be controlled from the PPB’s DCS or pumping station’s local control panels.

11.10.2 Raw water

Raw water is pumped from Olkilahti (PS 1) to Raw Water tank (15 m³) within the PPB.

Normally Olkilahti PS 1 raw water is used only for TMF start-up situations and TMF circulation water quality problems.

PS 1 equipped with level control switch, local control panel, reserve pump and pump can be controlled from the PPB’s DCS or pumping station’s local control panel.

11.10.3 Chemical feeding stations

Reservation for stations will be designed in following positions:

- TP → SP2
- SP2 → SP1
- SP3

11.10.4 Sewage disposal, bio-chemical

Solid waste will be transported to the municipal waste treatment. Purified sewage water will be discharged to the Wetland treatment.

11.10.5 Water treatment and balance flow sheet

Water treatment and balance flow sheet is presented in the Figure 41.
Figure 41  Water treatment and balance flow sheet
12 ENVIRONMENTAL STUDIES, LEGISLATION AND PERMITTING

12.1 Environmental Permit

Sotkamo Silver has received the Environmental Permit for Silver Mine in April 2013. The Environmental Permit concerns the following activities as applied:

- Total mining volume is under 550 000 t/a (ore, barren rock and building stone)
- Crushing ore, barren rock and building stone
- Concentrating
- Production of the metal and pyrite concentrates
- Discharging of the purified process and waste water to Koivupuro stream
- Mining waste treatment and piling and related auxiliary activities

12.1.1 General

The Permit holder has to sufficiently be aware of the environmental impacts and risks of the operation by monitoring the process, emissions and effects of the operation constantly.

12.1.2 Water supply and discharge

For Sotkamo Silver has been granted right for water supply from Olkilahti, Pieni Tipasjärvi Lake max 23 m³/h. Also ground water pumping from open pit and underground mine is allowed for drainage purpose.

The water circulation control will be implemented so that the concentrator plant internal water circulation is as effective as possible.

The purified process and waste water is allowed to discharge through Wetland treatment to a ditch leading to Koivupuro trickle.

12.1.3 Tailings Management Facility

At the operational stage there has to be constantly sufficient capacity of water reservation and purification for once every 20 years rainfall recurrence interval. The concentration of sulphur in tailing has to be as low as possible, annual average value < 0.3%.

The requirements of the structure of the ponds and dams are presented in chapter 10.9.

12.1.4 Dust generation and emissions to air

The dust generation targets of the crusher and concentrator has to be equipped with local dust extraction and filtering system. In the open-air the crushed ore conveyors have to be covered and equipped with dust extraction system.

At the open pit, piles and tailings pond the dust spreading to environment has to be prevented by watering. At the open pit the dust generation targets can also be covered.
12.1.5 Noise and vibration

The open pit and building stone blasting has to be mainly executed at the day time with prior notice.

Equivalent noise levels at the nearest households for day time (07:00 to 22:00) has to be <55dB and < 50dB for night time (22:00 to 07:00).

It is recommended that at the summer-time 1.5. – 31.8. equivalent noise levels at the nearest leisure households for day time (07:00 to 22:00) would be <45dB and < 40dB for night time (22:00 to 07:00).

The blasting vibration effect has to be minimized by blasting technical solutions. The vibration velocity at the nearest households for night time (22:00 to 07:00) has to be < 2 mm/s.

12.1.6 ROM and margin ore storage

On the ROM storage is allowed to store max 40 000 t at the time before crushing.

On the margin ore storage is allowed to store max 100 000 t at the time.

12.1.7 Barren rock and overburden

Technically suitable barren rock is not waste, if it’s utilized for the mine area construction immediately or after short period storing. The sulphur content has to be < 0.3% and without acidification potential.

The barren rock with sulphur content < 0.8% can be utilized for the mine area construction if it’s placed constantly under surface or groundwater level.

Removed overburden is not waste, if it’s utilized for the mine area construction immediately or after short period storing. The metal content of the overburden may not exceed the requirements of the law.

12.1.8 Chemicals and fuel

The chemical and fuel storages are to build to meet the standards and are inspected regularly to reduce risk of spillages.

12.1.9 Monitoring and reporting

The detailed monitoring program is to be prepared in accordance with the permit. The reports must be provided to Kainuu Centre for Economic Development, Transport and the Environment (ELY) at determinate times.
12.1.10 Mine closure and rehabilitation plan

The detailed mine closure and rehabilitation plan is to be prepared in accordance with the principles and technical details of the permit and submitted to the Regional State Administrative Agency (AVI) for acceptance.

12.1.11 Environmental Bond

The requirement of the environmental bonds is defined in the Environmental Permit:

- Construction of the Water treatment system 200 000 €
- Post-closure Water treatment and monitoring 500 000 €
- Ore storage areas as per 1. operational year plan 4 €/m²
- Barren rock area as per 1. operational year plan 2 €/m²
- Tailings pond area as per 1. operational year plan 4 €/m²
- Rehabilitation in case of changing or cancelling the permit 200 000 €
- Expenses in case of changing or cancelling the permit 10 000 € (AVI)

12.2 Land Use and Building

Sotkamo Silver has purchased the whole mining concession and some adjacent areas, total 428 ha. Land use and building development is controlled by Land Use and Building Act. The purchased areas are presented in Appendix 13 Map of Land Use and Appendix 14 Map of Real Estate.

12.3 Mine Safety Permit

Sotkamo Silver has received a Mine Safety Permit for the Silver Mine according to new safety requirements on 15th January 2014. The Company will be the first mining-company in Finland to start-up operation under the new mine-safety permit.

All new mines have to seek authorisation and old mines will be forced to upgrade its operations to meet new requirements. This new permit is required along with a mining license to be engaged in mining. It is approved by the Finnish Safety and Chemicals Agency (TUKES).

The Mining Authority inspects mines regularly, usually once a year. The mining operator shall designate an authorised person who is in charge of safety. This person of the Silver Mine has been authorised by TUKES.

The Mine Safety Permit includes the following activities:

- Information of activities provided by contractor
- Mine construction project plan and schedule
- Overall plan
- Operation’s risk assessment
- Operating principles for the prevention of accidents and documentation
- Internal emergency plan
12.4 Equator Principles

The Equator Principles is a risk management framework, adopted by financial institutions, for determining, assessing and managing environmental and social risk in projects and is primarily intended to provide a minimum standard for due diligence to support responsible risk decision-making.

Sotkamo Silver is following instructions of the Equator Principles.

12.5 Social Licence to Operate

According to Fraser Institute: “The Social Licence to Operate (SLO) refers to the acceptance within local communities of both mining companies and their projects. In order to obtain an SLO it is necessary to develop good relationships with all stakeholders, especially with local communities. The credibility of mining companies is based on mutual respect, honesty, open dialogue, transparency, timely responses to community concerns, information disclosure, and constancy and predictability in the companies’ ethical behaviour. Obtaining an SLO is essential for reducing the risks of social conflicts and for enhancing a company’s reputation.”

Sotkamo Silver has developed SLO especially with local people and communities for several years and obtained good transparency and trust.
13 PROJECT EXECUTION

Project execution section focuses on giving the reader brief information about the future project execution and contract strategy as well as the preliminary construction schedule.

13.1 Execution and contract strategy

Since Sotkamo Silver is a relatively small mining company with limited resources, EPCM (Engineering Procurement Contract Management) approach is chosen as the current execution strategy. EPC is possible approach to build up the beneficiation process plant excluding civil works. EPC is used in electrical and automation including financing.

13.2 Preparation for execution phase

The target for the Preparation for Execution (Pre-Ex) phase is to improve the definition of the project to justify CAPEX and OPEX estimates at ±10% accuracy level and to have all required permits in place. Selected Long Lead Items (LLI) may have to be purchased or at least preparations for purchasing being completed during Pre-Ex.

13.2.1 Project management

In addition to planning and following-up the activities described in the following subsections, the Project Management must plan, recruit, establish and organize the Owner’s project team, both for the Pre-Ex phase itself and later for the Execution phase.

Owner’s activities related to financing, potential partners, off-take agreements etc. are not discussed in this BFS Update.

13.2.2 Pre-engineering

The main technical and engineering activities in the Pre-Ex phase will be:

- Additional geotechnical and hydrological sampling, testing and engineering related to the mine area, processing plant area and TMF area.
- Additional geological drilling to improve the understanding of the deposit, move resources to a higher status of definition and possible increase the resource and reserve estimate.
- Field work related to the environmental permitting process.
- Laboratory and pilot scale beneficiation tests in order to investigate the effects of variations in ore quality and to optimize the chemical recipes and costs.
- Basic Engineering, in the first phase to support the preparation of ITB for the EPCM contract and engineering of LLIs, and in the second phase to prepare for Detailed Engineering.
13.2.3 Permitting, land use plans and CSR activities

Efforts and activities to improve the local knowledge, understanding and support for the project must be substantially increased. A communication strategy must be developed and implemented.

Mining Permit and Environmental Permit has been received as well as the Mining Safety Permit.

In Pre-Ex phase the other permit applications will be prepared and handled to authorities.

13.2.4 Procurement

The main procurement activities during Pre-Ex will be:

- Preparation of ITB for the EPCM contract
- Bidding process and EPCM contract award
- RFQs, bid evaluation and possible contracts for LLIs.

13.3 Execution phase

Sotkamo Silver’s final investment decision is scheduled after securing the project financing.

Execution planning will mostly be the responsibility of the EPCM contractor. However, there are certain elements that need to be considered by the Owner at an early stage.

Scheduled duration of the construction phase including commissioning is 12 months. The first sections are scheduled to be ready for commissioning 10 months after start of construction.

13.4 Production start and ramp-up

After commissioning is completed the production will be started and the following ramp-up schedule is foreseen:

- The total ramp-up schedule is 4 months, based on mining company’s personnel own experience and also on experience from other mining projects and process plant tuning time.
- During the first four months it’s estimated that ore feed to crusher and mill will be 73 000 tn and Ag-Au-Pb concentrate production 470 tn.
- During the first full year it’s estimated that after 8 months the average production level 2250 t/a of silver-lead concentrate production can be achieved.
13.5 Implementation schedule

The preliminary main schedule for the project is shown below in Figure 42.

Figure 42 Silver Mine execution schedule
14 MARKET STUDIES

Sotkamo Silver has valid long-term agreements for selling zinc and lead concentrate.

For selling the lead concentrate Sotkamo Silver has contracted Letter of Intent (LOI) with the Smelter Company located in Central Europe. The commercial and legal terms of the LOI are legally binding.

For selling the zinc concentrate Sotkamo Silver has contracted the Sales Agreement with Boliden Commercial AB, Sweden.
15 CAPITAL AND OPERATIONAL COSTS

15.1 Capital cost estimate

The investment estimate of Silver Mine Bankable Feasibility Study Update is based on the quotations, the technical concept and scope presented in this report and on the corresponding layout drawings.

The estimate covers Silver Mine with the following departments:

- Underground and Open pit (incl. water pumping, electrification, automation and HVAC)
- Crushing plant
- Concentrator plant
- Mine Area, Environmental management and Utilities (Mine area i.e. roads and earthwork, water pumping stations, potable water and sanitary water systems and all other mine area piping and cabling, platforms for the main transformers, Mine closure costs etc.)
- Indirect costs
- Contingency

A detailed investment cost calculation for the Silver Mine has been prepared, presented in Appendix 15.

The costs are based on enquiries and CTS’s cost file information. Main machinery costs are based on enquiries.

The estimate on installation costs is based on the typical installation cost levels and price indication from machine suppliers and CTS reference information. The costs are complete contractor costs including installation material, wages, fringe benefits, insurance expenses, contractor's overhead and profit.

The cost estimates of machinery and equipment correspond to the anticipated price level of the offers with fixed prices on the Q3/2013.

All costs are specified in Euro’s without VAT.

The total investment cost calculation summary is presented in the Table 19.
### Table 19  Investment cost summary

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<td></td>
</tr>
<tr>
<td>Year 3</td>
<td>0</td>
<td>0</td>
<td>1,594</td>
<td>206</td>
<td>563</td>
<td>377</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>2,740</td>
<td></td>
</tr>
<tr>
<td>Year 4</td>
<td>0</td>
<td>719</td>
<td>44</td>
<td>0</td>
<td>450</td>
<td>261</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>1,474</td>
<td></td>
</tr>
<tr>
<td>Year 5</td>
<td>0</td>
<td>0</td>
<td>44</td>
<td>0</td>
<td>450</td>
<td>261</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>755</td>
<td></td>
</tr>
<tr>
<td>Year 6</td>
<td>0</td>
<td>0</td>
<td>22</td>
<td>0</td>
<td>225</td>
<td>131</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>377</td>
<td></td>
</tr>
<tr>
<td><strong>TOTAL PROJECT CAPEX</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>32,350</td>
</tr>
</tbody>
</table>

#### 15.2 Calculation basics for disciplines

##### 15.2.1 Civil work

The construction amounts have been calculated and cost estimate is based on CTS’s or other engineering company cost file information. Quotations has been asked and received for civil works. These have been used for earth works and road construction.

Construction costs includes buildings, machine foundations mine area infrastructure incl. roads, earthwork, piping and cabling costs, overburden dump capital costs, mine closure costs, etc.

Environmental costs and mine closure costs are included in civil work costs and are based on CTS’s cost file information. Part of the environmental investment costs will be realized later.

Mine site rehabilitation consists mainly of earth construction and other work done right after mine closure, and also includes monitoring and work carried out during the active and passive care stages.
15.2.2 Machinery

Equipment costs for crushing plant, concentrator plant, water pumping stations and laboratory equipment are based on enquiries and other equipment are based on CTS’s cost file information.

Pumps, tanks, agitators, conveyors and steel structures as well as equipment insulation are presented under machinery costs.

15.2.3 Piping

The piping costs are based on the estimated piping lengths and are based on CTS’s cost file information. The costs include pipes, fittings, pipe supports and insulation.

15.2.4 Electrification

Equipment and materials for electrification have been specified. The cost estimate is based on CTS’s cost file information. Price indication for 45kV and 20kV voltage systems is from ABB.

15.2.5 Automation

Equipment and materials for automation have been specified. The cost estimate is based on CTS’s cost file information or quotation. DCS cost is based on enquiry.

15.2.6 HVAC

Ventilation volumes are calculated by CTS and estimate is based on CTS’s cost file information.

Potable water and sanitary water systems, electric heating and fire protection facilities are included in HVAC costs.

15.2.7 Spare Parts

The spare part costs are estimated mainly on a percentage basis on the estimated material costs of process and auxiliary equipment, electrification, process control and HVAC. Main machinery spare part costs are based on enquiries.

15.3 Capital cost estimate per departments

15.3.1 Mining

Contracted outsourced mining is chosen as mining method for the Silver Mine. Therefore no investments will be spent to mining fleet.

The investment cost of Open pit and Underground mine consist following items:

124
15.3.2 Crushing plant

The crushing itself is outsourced and therefore the cost of crusher is not included in Capital cost.

The cost covers the civil works of crushed ore storage, feeders and conveyors from storage to mill and electrical and automation of those.

15.3.3 Concentrator plant

Beneficiation plant civil work is mainly building foundation and secondary structures of beneficiation building. The building itself is handled as leased equipment and included in OPEX except of installation.

The main machinery includes the ball and rod mills with cyclones, flotation cells thickeners, filtering equipment and other beneficiation process equipment.

In the first phase the capacity is limited to 350 000 tpa mill feed due to EP requirements. It will be increased to 450 000 tpa in third production year after the update of EP. These costs are handled as sustaining capital cost and can been seen as a peak of investments in year three in the Table 19 Investment cost summary.
Table 22 Capital cost breakdown, concentrator plant

<table>
<thead>
<tr>
<th>Cost break down</th>
<th>Costs x 1 000 €</th>
</tr>
</thead>
<tbody>
<tr>
<td>Civil work</td>
<td>2 253</td>
</tr>
<tr>
<td>Main machinery</td>
<td>7 580</td>
</tr>
<tr>
<td>Auxiliary machinery</td>
<td>2 286</td>
</tr>
<tr>
<td>Piping</td>
<td>928</td>
</tr>
<tr>
<td>Electrification and automation</td>
<td>4 945</td>
</tr>
<tr>
<td>HVAC</td>
<td>440</td>
</tr>
<tr>
<td>Production increase Y3 (included in above numbers)</td>
<td>(1 703)</td>
</tr>
<tr>
<td><strong>TOTAL</strong></td>
<td><strong>18 432</strong></td>
</tr>
</tbody>
</table>

15.3.4 Mine area and utilities

The civil work covers the cost of production area, access road, the fabrication of crushed rock for layers, office foundation and HVAC, TMF area work. The office is purchased as leased facility, but the foundations with electrical and HVAC costs are included in Capex.

Machinery includes the raw water pumping station with pipeline as well as TMF process pumping stations and pipes.

In electrification and automation there is the 20 kV distribution system, area lighting, pumping stations, camera monitoring system and office equipment. HVAC includes the office and sanitary waste water system.

Table 23 Capital cost breakdown, mine area and utilities

<table>
<thead>
<tr>
<th>Cost break down</th>
<th>Costs x 1 000 €</th>
</tr>
</thead>
<tbody>
<tr>
<td>Process plant area works and roads</td>
<td>1 171</td>
</tr>
<tr>
<td>Office</td>
<td>437</td>
</tr>
<tr>
<td>TMF area civil</td>
<td>2 950</td>
</tr>
<tr>
<td>Area and TMF pumps and piping</td>
<td>316</td>
</tr>
<tr>
<td>Electrification and automation</td>
<td>1 190</td>
</tr>
<tr>
<td>HVAC office and other</td>
<td>336</td>
</tr>
<tr>
<td><strong>TOTAL</strong></td>
<td><strong>6 400</strong></td>
</tr>
</tbody>
</table>

15.3.5 Indirect Costs

The project indirect costs are estimated based on percentage basis on the estimated material costs of process and auxiliary equipment, electrification, process control and HVAC.

Indirect costs include engineering costs, temporary constructions, project management, installation supervision, training and other indirect costs. Percentage of 5 to 6 % is used in different discipline.
15.3.6 Contingency

The estimate contingency of 5% is defined as an allowance for unforeseen elements of cost within the defined project scope. This is calculated from the estimated material costs of process and auxiliary equipment, electrification, process control and HVAC.

As such, it is a cost included in the cost estimate to cover events or incidents which will probably occur during the course of the project but, at the time of estimate preparation, are not quantifiable. The cost reflects a measure of the level of uncertainties related to an established scope of work. It is an integral part of the estimate and typically it is applied to all parts of the estimate, i.e., direct and indirect costs. It should be assumed that the contingency will be spent.

Typical uncertainties applicable to contingency include:

- Insufficient information due to incomplete engineering;
- Areas or systems with a reasonable probability of change occurring during the detail design stage;
- Equipment or material costs obtained by ratio or updated from historical costs or previous estimates;
- Variations in labour productivity and costs;
- Raw materials cost variations such as the steel price;
- Lack of data on economic conditions.

The contingency does not cover scope changes. Additional items that contingency does not cover include: labour unrest, adverse weather conditions, changes in government policies, force majeure or acts of God. These risks are covered by a separate project risk allocations.

15.3.7 Exclusions

The following costs are not included in the estimate:

- interest during the implementation period
- production losses
- working capital, which is taken into account in financial analyze
- licence fees
- financing costs
- cost escalation and exchange rate fluctuations
- commodity price fluctuations
15.4 Operation costs

The estimate covers Silver Mine with the following departments:

- Underground and Open pit (incl. water pumping, electrification, automation and HVAC)
- Crushing plant
- Concentrator plant
- Mine Area, Environment and Utilities (Mine area i.e. roads and earthwork, water pumping stations, potable water and sanitary water systems and all other mine area piping and cabling, platforms for the main transformers, Mine closure costs etc.)
- Indirect costs

An operational cost calculation for the Silver Mine has been prepared, presented in Appendix 16.

The costs are based on Quotations, CTS’s consumption calculations and SoSi information of the unit prices.

All costs are specified in Euro’s without VAT. The inflation factor is not in use.

The production cost through the various stages of production (mining, crushing, and concentration) infrastructure and administration has been estimated at Euro 1 930 € per tonne of concentrate and 38.83 € per tonne of ore.

No provisions are made in the estimate towards future escalation in costs.

It is assumed that all the plant will operate 365 days/year and with 82% availability for crushing operation and 90% availability for rest of the plant. Mine and crusher are operated 5 days per week.

No provision was made for depreciation, financing expenses, royalties, directors’ and auditor’s fees, sales expenses such as discounts/rebates/sales commissions, etc.

15.4.1 Operation cost summary

A breakdown of operating costs by area is shown in Table 24. These costs, however, exclude depreciation, interest, sales and administrative expenses. The figures presented average year 4.
Table 24  Operation cost breakdown (year 4)

<table>
<thead>
<tr>
<th>Cost break down</th>
<th>Total 1 000 €/y</th>
<th>€/tn of ore</th>
<th>€/tn of products</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining</td>
<td>10 210</td>
<td>22.48</td>
<td>1 117</td>
</tr>
<tr>
<td>Crusher</td>
<td>992</td>
<td>2.18</td>
<td>109</td>
</tr>
<tr>
<td>Beneficiation</td>
<td>4 399</td>
<td>9.68</td>
<td>481</td>
</tr>
<tr>
<td>Site general, environmental and utilities</td>
<td>2 038</td>
<td>4.49</td>
<td>223</td>
</tr>
<tr>
<td>TOTAL</td>
<td>17 639</td>
<td>38.83</td>
<td>1 930</td>
</tr>
</tbody>
</table>

The operation cost distribution for different cost items has been calculated and indicative factors, cost per ore and cost per total concentrate has been prepared, presented in Table 25.

Table 25  Operation cost per cost item (operation year 4 production 450 000 tpa)

<table>
<thead>
<tr>
<th>Mine</th>
<th>Crushing</th>
<th>Beneficiation</th>
<th>Site general</th>
<th>Total 1000 €</th>
<th>€/tn of ore</th>
<th>€/tn of product (AgAuPb)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Personnel</td>
<td>499.6</td>
<td>0.0</td>
<td>1 151.8</td>
<td>931.2</td>
<td>2 582.6</td>
<td>5.69</td>
</tr>
<tr>
<td>Electricity and heat</td>
<td>701.9</td>
<td>47.1</td>
<td>1 116.4</td>
<td>98.0</td>
<td>1 868.5</td>
<td>4.32</td>
</tr>
<tr>
<td>Contractor</td>
<td>8 833.2</td>
<td>908.5</td>
<td>0.0</td>
<td>425.0</td>
<td>10 163.3</td>
<td>22.38</td>
</tr>
<tr>
<td>Maintenance and spares</td>
<td>175.4</td>
<td>7.2</td>
<td>1 099.5</td>
<td>296.4</td>
<td>1 142.3</td>
<td>3.48</td>
</tr>
<tr>
<td>Consumables</td>
<td>0.0</td>
<td>0.0</td>
<td>823.6</td>
<td>10.1</td>
<td>830.7</td>
<td>1.84</td>
</tr>
<tr>
<td>Miscellaneous</td>
<td>0.0</td>
<td>29.6</td>
<td>207.3</td>
<td>277.6</td>
<td>514.5</td>
<td>1.13</td>
</tr>
<tr>
<td>Total</td>
<td>10 210.1</td>
<td>992.3</td>
<td>4 398.7</td>
<td>2 038.3</td>
<td>17 639.4</td>
<td>37.65</td>
</tr>
</tbody>
</table>

15.4.2 Open pit and underground mine

Quotation has been asked from several contractors for underground mining, overburden removal, drifting and open pit mining. The contractor is responsible of quarrying and haulage of ore to crusher. The used unit prices are presented in Table 26.

Table 26  Mining unit prices op mining ore

<table>
<thead>
<tr>
<th>Cost break down</th>
<th>Tonnage Y4</th>
<th>unit price €/tn</th>
<th>Total €</th>
</tr>
</thead>
<tbody>
<tr>
<td>Overburden removal</td>
<td></td>
<td></td>
<td>1.77</td>
</tr>
<tr>
<td>OP waste mining</td>
<td>550 000</td>
<td>1.7</td>
<td>935 000</td>
</tr>
<tr>
<td>OP ore mining</td>
<td>84 891</td>
<td>2.04</td>
<td>173 200</td>
</tr>
<tr>
<td>UG drifting</td>
<td>175 000</td>
<td>14.68</td>
<td>2 569 000</td>
</tr>
<tr>
<td>UG ore mining</td>
<td>369 344</td>
<td>13.96</td>
<td>5 156 000</td>
</tr>
<tr>
<td>TOTAL</td>
<td>1 179 235</td>
<td>avg 7.49</td>
<td>8 833 200</td>
</tr>
</tbody>
</table>

Overburden removal is included in Capex.
In first two years the ore quarrying is limited to 350 000 tpa due to environmental permit regulations and will after that rise to 450 000 tpa after the environmental permit is updated.

Mining company is responsible of open pit and underground dewatering. The operation cost of these is included in energy price and maintenance.

15.4.3 Crushing plant

The ore crushing cost have been calculated based on quotation. Mining company is responsible to supply the electricity.

The superstructure of crushed ore storage is a leased building. The yearly cost is based on quotation and are 29 600 €.

Spare parts are calculated for conveyor and belt feeders from storage to mill.

15.4.4 Rental buildings

The superstructure of process plant building has a steel frame and light weight insulated claddings. A quotation of leased contract for this building has been received and used as operation cost. The yearly leasing cost is 130 600 €. The same solution is used for maintenance/core storage building and office. The yearly leasing cost for these are 83 600 €.

15.4.5 Electrification

Equipment and materials for electrification have been specified. The cost estimate is based on CTS’s cost file information or quotation. Price indication for 45kV and 20kV voltage systems is from ABB.

15.4.6 Electricity

The electricity amounts have been calculated and SoSi electricity price indication has been used in estimation. The mining company has made an agreement with UPM Kymmenen of the network connection fee and energy price. In the calculations the fixed fee for connection is 40 700 €/year. The energy charge is 43.00 €/MWh.

15.4.7 Heat

The necessary heat for underground mine is produced with LNG (liquid natural gas). Quotation has been asked for suppliers to invest for necessary storage and boiler unit to produce heat from LNG. This heated air is then blown via shaft to UG.

The yearly fixed fee of equipment is 119 600 €/year and energy production fee 63.49 €/MWh.

Other buildings and facilities are heated with electrical heaters. The heat energy amounts have been calculated by CTS.
15.4.8 Chemical consumptions

The chemical consumption has been calculated based on GTK report chemical consumptions and chemical prices are based on price indication from supplier. The annual floatation chemical usage is 891.8 tn and cost 559 545 € for 350 000 tpa production level.

For the 450 000 tpa production it is calculated that 720 000 € is used per year for these same chemicals. Minor part of chemicals is estimated to be used in TMF area annual cost of 10 100 €.

15.4.9 Beneficiation consumables

The ball and rod mill’s will use 5 kg of ball’s or rod bars per year per tn of ore feed to mill based on GTK test work. The annual cost of these are 72 900 € for smaller production and is increasing based on factor when the production level is changed.

15.4.10 Salaries and other personnel costs

The personnel costs have been calculated based on SoSi informed personnel amounts and salaries price indication. Mining company own personnel yearly salary cost are 2 582 600 €

15.4.11 Spare parts

The spare part costs has been calculated based on investment cost file information and the costs are estimated mainly on a percentage basis on the estimated material costs of process and auxiliary equipment, electrification, process control and HVAC. The yearly spare and wear part cost are 190 600 €.

15.4.12 Maintenance

The maintenance costs are evaluated and based on CTS cost file information estimates from equipment supplier. Total annual maintenance cost for whole operation is 1170 000 €.

15.4.13 ICT and Telephone

The telephone and ICT costs are evaluated are based on CTS cost file information. The total annual cost are 45 000 €.

15.4.14 General Maintenance

The maintenance costs covering security, snow removal, cleaning, cesspit emptying and waste management are evaluated and based on CTS cost file information and are 44 000 €/year.
15.4.15 Environment monitoring

The environment monitoring costs are evaluated and based on CTS cost file information.

15.4.16 Laboratory services

Silver Mine has asked quotation of total laboratory services. Based on this the annual fee of laboratory services is 309,600 €.

The environmental monitoring is calculated as outsourced service as annual cost of 71,400 € based on CTS cost file information.

15.4.17 Mine closure

In the operation cost calculation Mine closure cost is calculated as yearly amount 150,000 €, which is collected to a bond to cover the mine closure cost.

15.4.18 Exclusions

The following costs are not included in the estimate:

- product transportation, included in product price
- land lease
- insurance costs
16 FINANCIAL ANALYSIS

16.1 Basis of economic evaluation

Capital and operating cost estimates were generated and inputted to an economic model along with other economic inputs including product prices and exchange rates. Production data from the mining plan were also input. The economic model was then used to evaluate different scenarios in order to determine the viability of the project.

Taxation and depreciation was also taken into account in the model. An annual discount rate of 8% is used. The Net Present Value calculation assumes cash flows occur at the middle of each year, and are discounted back to 1 January 2015.

Also a model where the construction year and first two production years were calculated as monthly basis was modelled. This was then chosen as the final model for the financial analyses.

The model contains one construction year (Y1) and 8 production years. The model incorporated flexibility for the key drivers of the project enabling analysis and the sensitivity of key variables.

USD/Euro exchange rate 0.76 was used for product pricing. All cost information is input into the model in real terms as Q3/2013.

16.2 Sotkamo Silver base case

An international market report was used to evaluate the future product prices. For this BFS Update the following price scenarios was used:

<table>
<thead>
<tr>
<th>Metal</th>
<th>Down side</th>
<th>Base case</th>
<th>Up side</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ag USD/oz</td>
<td>15.0</td>
<td>21.5</td>
<td>28.0</td>
</tr>
<tr>
<td>Au USD/oz</td>
<td>1 000.0</td>
<td>1 380.0</td>
<td>1 500.0</td>
</tr>
<tr>
<td>Zn USD/t</td>
<td>2 000.0</td>
<td>2 615.0</td>
<td>2 800.0</td>
</tr>
<tr>
<td>Pb USD/t</td>
<td>2 000.0</td>
<td>2 380.0</td>
<td>2 500.0</td>
</tr>
<tr>
<td>EUR/USD</td>
<td>0.65</td>
<td>0.76</td>
<td>0.90</td>
</tr>
</tbody>
</table>

The historical silver price development is illustrated in the Figure 43. In the financial model Zinc and Lead prices are based on long term forecast. The prices used are presented in table 28. In financial calculations the Silver and Gold prices has been kept stable based on each case.
The IRR/NPV sensitivity analyses were done only for Silver pricing and EUR/USD rate.

16.3 Sensitivity analysis

The sensitivity analyses were performed for Silver pricing and EUR/USD exchange rate. Each variable was individually increased and lowered between “up-side” “down side” cases, whilst leaving the other base case parameters unchanged to test the sensitivity of the model to changing economic and operational conditions. The resulting effect on Net present value (NPV), internal rate of return (IRR) and investment payback time after tax is presented in Table 29. Same factors before tax are presented in table 30.
Table 29  Resulting effects on NPV, IRR and payback time (tax included)

<table>
<thead>
<tr>
<th>Financial sensitivity</th>
<th>NPV 1000 €</th>
<th>IRR %</th>
<th>Payback time (Y)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Base case</td>
<td>23 333</td>
<td>30.24</td>
<td>2.5</td>
</tr>
<tr>
<td>Down side Ag 15.0</td>
<td>392</td>
<td>8.39</td>
<td>-</td>
</tr>
<tr>
<td>Down side EUR/USD 0.65</td>
<td>12 510</td>
<td>20.10</td>
<td>3.4</td>
</tr>
<tr>
<td>Up side Ag 28.0</td>
<td>46 781</td>
<td>51.47</td>
<td>1.7</td>
</tr>
<tr>
<td>Up side EUR/USD 0.90</td>
<td>37 090</td>
<td>42.79</td>
<td>1.9</td>
</tr>
</tbody>
</table>

The payback time is calculated from the process start-up. In the EUR/USD cases Ag price 21.50 USD/oz was used.

Table 30  Resulting effects on NPV, IRR and payback time before tax

<table>
<thead>
<tr>
<th>Financial sensitivity</th>
<th>NPV 1000 €</th>
<th>IRR %</th>
<th>Payback time (Y)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Base case</td>
<td>29 514</td>
<td>35.37</td>
<td>2.2</td>
</tr>
<tr>
<td>Down side Ag 15.0</td>
<td>1 012</td>
<td>9.01</td>
<td>7.6</td>
</tr>
<tr>
<td>Up side Ag 28.0</td>
<td>58 841</td>
<td>61.01</td>
<td>1.5</td>
</tr>
</tbody>
</table>

16.4  Cash flow

For Base Case an annualized cash flow position is shown in Figure 44 and a detailed cash flow analysis is given in Table 31.
Table 31  Detailed cash flow analysis, Base Case (after tax)

<table>
<thead>
<tr>
<th>Production year</th>
<th>AgPb Con</th>
<th>Zn Con</th>
<th>Revenue 1000 €</th>
<th>Total mining 1000 tpa</th>
<th>Capex 1000 €</th>
<th>Opex 1000 €</th>
<th>Annual total cash flow 1000 €</th>
<th>Cumulative cash flow 1000 €</th>
</tr>
</thead>
<tbody>
<tr>
<td>-1</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>547</td>
<td>-23 903</td>
<td>-</td>
<td>-23 903</td>
<td>-23 903</td>
</tr>
<tr>
<td>1</td>
<td>2 253</td>
<td>3 509</td>
<td>27 095</td>
<td>546</td>
<td>-1 877</td>
<td>-13 364</td>
<td>8 152</td>
<td>-15 751</td>
</tr>
<tr>
<td>2</td>
<td>2 230</td>
<td>4 002</td>
<td>27 812</td>
<td>546</td>
<td>-1 224</td>
<td>-13 767</td>
<td>10 532</td>
<td>-5 219</td>
</tr>
<tr>
<td>3</td>
<td>2 564</td>
<td>5 055</td>
<td>31 782</td>
<td>1 134</td>
<td>-2 740</td>
<td>-17 510</td>
<td>8 860</td>
<td>3 641</td>
</tr>
<tr>
<td>4</td>
<td>2 346</td>
<td>6 794</td>
<td>30 916</td>
<td>1 179</td>
<td>-1 474</td>
<td>-17 639</td>
<td>9 521</td>
<td>13 163</td>
</tr>
<tr>
<td>5</td>
<td>1 749</td>
<td>5 588</td>
<td>23 507</td>
<td>1 225</td>
<td>-755</td>
<td>-17 911</td>
<td>4 606</td>
<td>17 769</td>
</tr>
<tr>
<td>6</td>
<td>1 721</td>
<td>5 627</td>
<td>23 238</td>
<td>1 090</td>
<td>-377</td>
<td>-17 060</td>
<td>4 778</td>
<td>22 547</td>
</tr>
<tr>
<td>7</td>
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16.5 Conclusions

The updated Bankable Feasibility Study (BFS) has demonstrated both the technical and economic viability of the project. The results of Base Case profitability calculation are presented in Appendix 17 Profitability calculation, Base Case.
17 RISKS

17.1 Investment risk

The sensitivity analysis has been evaluated on five key variables within the cash flow: metal price (revenue), operating costs, capital expenditure, discount factor and exchange rate.

The sensitivity analysis suggests that the Silver Mine operations are most sensitive to Ag price and USD/EUR currency volatility.

The US Dollar/Euro exchange rate is an important factor at the Silver Mine as the majority of costs are incurred in Euros whereas the majority of revenue is either generated in, or linked to the US Dollar.

17.2 Project risk

In the Silver Mine Project can be mentioned few identified risks:

- Appeals against Building Permit