ABSTRACT
Relatively high minerals and metals content characterize historical tailings due to less efficient extraction methods and/or relatively low metal prices at the time. Repositories of such tailings pose environmental risks but could also become metals and minerals resources. An example of such tailings is the Yxsjöberg historical tungsten ore tailings in the Smaltjärnen tailings repository in Sweden.

The Smaltjärnen tailings repository was sampled by collecting drill core samples from different locations. The collected drill core samples were characterized physically (colour, texture, moisture content and particle size distribution) and chemically (elemental composition and distribution, and mineralogical composition). Feasible physical separation methods (magnetic and gravity separation) were pre-selected based on the tailings characteristics and the knowledge of processes from which the Yxsjöberg historical tailings were produced.

In this paper, results from three drill cores each representing a different location on the tailings repository are presented. The tailings mass distribution was high in the coarser particle size fractions of +300 µm and +149 µm. Tungsten (W) and Copper (Cu) were the metals of interest with one location having higher concentrations than the other two at 0.20 %WO₃ and 0.14 %Cu. Sulphur (S) was recovered in the magnetic fractions of the LIMS and HIMS. Using the Knelson concentrator, W recovery was enhanced. These results are fundamental in the development of methods for separation of minerals and extraction of metals of interest from the historical tailings in order to leave behind an inert and environmentally safe residue.

1 Introduction
Primary ores have been the main source of minerals and metals of economic interest but with decreasing ore grades and increasing production costs, secondary sources such as historical tailings are being considered. Relatively high minerals and metals content characterize historical tailings due to less efficient extraction methods and/or relatively low metal prices at the time. Repositories of such tailings pose environmental risks but could also become metals and minerals resources. An example of such tailings is the Yxsjöberg historical tungsten ore tailings in the Smaltjärnen tailings repository in Sweden. This is one of the historical mine wastes case study sites for the REMinE (Improve Resource Efficiency and Minimize Environmental Footprint) project. In this project, the approach to dealing with these wastes is holistic in that not only are the tailings being considered for reprocessing for their valuable minerals and metals but also that their effective extraction must leave behind an inert and environmentally safe residue. (Alakangas, Sandström et al. 2016)

In the past, tailings have been studied mainly for purposes of improving tailings management with regard to physical and chemical stability, which in turn reduce environmental risks
associated with tailings storage (Bussière 2007). However, in recent years, historical tailings are being considered for recycling as a supply risk-reducing measure after critical raw materials have been identified (European Commission 2017). Tungsten (W), Copper (Cu), Fluorspar (CaF₂), Sulphur (S), Tin (Sn), Zinc (Zn), Beryllium (Be) and Bismuth (Bi) are the eight main elements found in the Yxsjöberg historical ore tailings (Hällström, Alakangas et al. 2018) and are among the European Union (EU) critical raw materials cited for supply risk and economic importance. Of these eight elements, four (Cu, S, Sn and Zn) are classified as non-critical raw materials with less supply risk but of high economic importance (European Commission 2017). This shows the importance of studying this historical tailings ore repository.

The Yxsjöberg historical tungsten ore tailings repository is located in the old mining area of Bergslagen in Central Sweden where the Yxsjö Mines were in operation from 1935 to 1963, with average grade of the crude ore estimated at 0.3-0.4 %WO₃, 0.2 %Cu and 5-6 %fluorite (Hübner 1971, Rothelius 1957). The exploited minerals were scheelite for W, chalcopyrite for Cu and Fluorite. Figure 1 below shows the location of the Smaltjärnen tungsten ore tailings repository and the decommissioned processing plant. This repository is estimated to have about 2.2 million tons of tailings and covers an area of 26 hectares, with elemental concentrations of 1-2 %S, 0.02-0.2 %Cu, 0.02-0.3 %W, 0.02-0.04 %Sn and 0.02-0.03 %Be (Alakangas, Sandström et al. 2016).

Figure 1: Smaltjärnen tungsten ore tailings repository in Yxsjöberg, Sweden.

The Yxsjö processing plant had three main divisions namely gravity separation, magnetic separation and flotation, from which tailings were generated and discharged onto the Smaltjärnen tailings repository as shown in Figure 2 below. The scheelite grain size in the ore was 0.2-4.0 mm and 90 wt% of the ore was liberated at 0.4 mm grain size. However, scheelite and fluorite minerals are very fragile hence slimes are very easily formed by crushing and grinding. The concentrate recoveries were 50-70 wt% for scheelite, 50 wt% for fluorite and 25 wt% for chalcopyrite meaning a good amount these valuable minerals ended up in the tailings. (Rothelius 1957)
The amount of minerals and/or metals that remain in tailings depends mainly on the separation and extraction methods employed, and the processing efficiencies. The design and selection of these processing methods is based primarily on the mineralogical characteristics of ores (Tungpalan, Wightman et al. 2015). According to (Petruk 2000), applied mineralogy investigations are performed to determine mineralogical characteristics such as the following:

i. Identities of major, minor and trace minerals,
ii. Compositions of minerals that bear on the process,
iii. Quantities of minerals
iv. Particle and grain size distributions and textures of the minerals,
v. Mineral liberations,
vi. Surface coatings on minerals.

These mineralogical characteristics are also essential when considering tailings for reprocessing as a secondary source of minerals and metals of economic interest, and/or environmental impacts. Separation and extraction methods for tailings reprocessing may differ from those initially used for treatment of primary ores due to mineral variations. For instance for high density ores such as scheelite ores, gravity separation and flotation methods are conventionally used but depending on their mineral associations such as magnetite (ferromagnetic minerals) and pyrrhotite (paramagnetic minerals) magnetic separation is also used as was the case for Yxsjöberg scheelite ore (Rothelius 1957, Yang, Niemistö February, 2017). However, to reprocess tailings from scheelite ores, enhanced gravity separation would be required due to high fines content. Multigravity separators such as the Knelson concentrator provide the enhanced gravity separation for very fine particles (Das, Sarkar 2018).

Therefore, presented in this paper is a fundamental study of the Yxsjöberg historical tungsten ore tailings. The study has been done in order to develop effective methods for separating minerals and extracting metals and metalloids of interest from these historical tailings so that an inert and environmentally safe residue is left behind.

2 Material and methods

2.1 Sampling and sample preparation

The Smaltjärnen historical tailings repository was sampled by collecting drill core samples from nine different locations as shown in Figure 3 below. The black dots represent sampling in the first campaign while the red dots in the second campaign. The numbering of the sampling locations was based on the sequence of sampling in a sampling campaign. Since there were only two locations sampled in the first campaign, the first four locations were distinguished by _1 and _2 meaning 1_1 and 2_1 were in the first campaign while 1_2 and 2_2 in the second
campaign. For locations 1_1 and 2_1, a Cobra hand drill with an open gauge steel tube (40 mm diameter and 1 m length) for sample holding was used. For locations 1_2 to 7, a percussion drill rig from Envix Nord AB was used, which uses plexiglass tubes (40 mm diameter and 1.2 m length each tube) for sample holding. The sampled locations were selected based on the visual variability of the tailings in terms of colour and texture, and the proximity to former discharge points. It should be noted that sampling location 1_2 was also in very close proximity to the sampling location P4 referred to in the geochemical characterization of these historical tailings (Hällström, Alakangas et al. 2018).

![Figure 3: Drill core sampled locations on the Smaltjärnen historical tailings repository.](image)

However, the sampling points in the selected locations were randomly picked and a summary of the information about three sampling points that will be discussed in this paper is given in Table 1 below.

**Table 1: Sampling points 1_1, 1_2 and 6 summary information.**

<table>
<thead>
<tr>
<th>Sampling Point</th>
<th>Sampling Points GPS Coordinates</th>
<th>Altitude (m)</th>
<th>Number of drill holes</th>
<th>Drill hole depth (m)</th>
<th>Number of drill cores</th>
</tr>
</thead>
<tbody>
<tr>
<td>1_1</td>
<td>60.041669, 14.771031</td>
<td>307</td>
<td>1</td>
<td>3.0</td>
<td>3</td>
</tr>
<tr>
<td>1_2</td>
<td>60.041778, 14.775325</td>
<td>306</td>
<td>1</td>
<td>6.0</td>
<td>5</td>
</tr>
<tr>
<td>6</td>
<td>60.042656, 14.775217</td>
<td>303</td>
<td>1</td>
<td>2.2</td>
<td>2</td>
</tr>
</tbody>
</table>

Screening of each drill core was done and based on the differences in colour and texture the cores were split into smaller samples representing tailings layers in a particular location of the repository. The thickness of each observed compacted layer was noted to have its approximate depth in the tailings repository. Drill core vertical profiles showing the different tailings layers in the selected locations of the repository were thus determined as shown in the Figure 4 below. Each colour represents a layer of a drill core and its compacted length (depth) in the tailings repository. The number of observed layers in sampling points 1_1, 1_2 and 6 were 10, 11 and 3 respectively, with sampling point 1_2 having the longest drill core of approximately 500 cm.
In order to further characterize the tailings samples physically (moisture content, texture and particle size distribution) and chemically (elemental composition and distribution, and mineralogical composition), the samples were dried at 105°C for 48 hours. The dried tailings samples were then split into smaller samples using a riffle splitter. Figure 5 below shows the dried tailings samples for drill cores 1_1, 1_2 and 6, with each sample representing a layer in the drill core. Therefore, in addition to the numbering of the sampling locations, a layer number was added to distinguish the layers from top to bottom of a drill core meaning 1_2-1 is the first (top) layer in drill core 1 of the second campaign.

![Sampled tailings drill cores vertical profiles](image)

**Figure 4:** Vertical profiles for sampling points 1_1, 1_2 and 6.

![Dried tailings samples](image)

**Figure 5:** Dried tailings samples for drill core 1-1 with layers 1_1-1 to 1_1-10, drill core 1_2 with layers 1_2-1 to 1_2-11, and drill core 6 with layers 6-1 to 6-3, showing variability in colour and texture.

Replicate dry samples of approximately 150 grams each representing the tailings layers in a particular drill core were sieved using a sieve shaker (RO-TAP® Sieve Shaker model RX-29) with five sieves to obtain six size fractions. However, for sampling point 1_2, the quantities of some four layers were less than 150 grams hence the samples were sieved without replication, which for the bigger samples was necessary to check that the split samples were representative of the whole layer sample.
2.2 Characterization

The six size fractions in microns (µm) to be obtained from the dry sieving were +1190, +600, +300, +149, +75 and -75. Particle size distribution curves for each drill core layer and subsequently the entire drill core were determined. Using the Rosin-Rammler-Sperling-Bennett (RRSB) distribution, the particle size of which 80% of the tailings amount passes (D_{80}) was determined for each drill core layer.

Small samples of approximately 15 grams were split from each particle size fraction and bulk (drill core 1_2 four layers that were less than 150 grams) samples of each drill core layer, and were chemically analyzed at ALS Scandinavia AB using the inductively coupled plasma – sector field mass spectrometry (ICP-SFMS) method. The analysis protocol included 10 oxides and 22 elements namely SiO₂, Al₂O₃, CaO, Fe₂O₃, K₂O, MgO, MnO, Na₂O, P₂O₅, TiO₂, As, Ba, Be, Bi, Cd, Co, Cr, Cu, Hg, Mo, Nb, Ni, Pb, S, Sc, Sn, Sr, V, W, Y, Zn and Zr.

For tailings mineralogical composition, X-ray Diffraction (XRD) analysis was done using the diffractometer at LTU in the Division of Chemical Engineering. Drill core 1_2 layers 1_2-1, 1_2-4, 1_2-8 and 1_2-9 were selected for this analysis as they had the highest mass distributions in this longest drill core as shown in Figure 6 below. Small (less than 50 g) samples of these selected layers were pulverized in a ring mill for 15 seconds to obtain the powder samples required for XRD analysis. The raw data obtained from the diffractometer was evaluated using the PANalytical X’Pert HighScore Plus v3.0 software. The XRD evaluation was based on the chemical analysis results as a guide for possible elements in the proposed phases to obtain the best matches.

![Figure 6: Mass distribution (%) of tailings layers in drill core 1_2.](image)

A ZEISS Axiophot Fluorescent Microscope mounted with an Infinity HD Camera was used for optical microscopy to study the scheelite (mineral of interest) particles in the tailings with regard to occurrence, texture and liberation. The 11 layers of drill core 1_2 were used as bulk samples for this study by splitting 3 grams from each layer and preparing epoxy polished samples.
Scanning Electron Microscope (SEM) with Energy-Dispersive X-ray Spectroscopy (EDS) and Back Scattered Electron (BSE) imaging was used to study the mineral phases in drill core layers 1_2-4 and 1_2-9, which were the two layers with the highest mass distributions in drill core 1_2. The same epoxy polished samples used for optical microscopy were used but only after carbon coating them to avoid sample surface charging. The ZEISS MERLIN™ FE-SEM (Field Emission – Scanning Electron Microscope) at LTU in the Division of Chemical Engineering was used. BSE images were captured and analyzed using the INCA-Oxford Instruments software.

2.3 Physical separation

Magnetic and gravity separation were pre-selected as feasible physical separation methods based on the knowledge from literature, tailings characteristics and knowledge of processes from which the Yxsjöberg historical tailings were produced.

2.3.1 Magnetic separation

The two layers (1_2-4 and 1_2-8) of drill core 1_2 were used with each layer being tested as a separate sample. The sample specifications for this separation method were having at least 150 grams with particle size greater than 75 µm. The Low Intensity Magnetic Separator (LIMS) and High Intensity Magnetic Separator (HIMS) in the mineral processing laboratory at LTU were used. The separation in both LIMS and HIMS was optimized by varying the operating variable parameters and observing the attained colour separation in the products. The operating variable parameters for LIMS were the drum speed and feed rate, while for HIMS it was the angle of the separation blades. Table 2 below summarizes the magnetic separation operating parameters and Figure 7 shows pictures of LIMS and HIMS.

Table 2: Magnetic separation operating parameters (Khavari 2018).

<table>
<thead>
<tr>
<th></th>
<th>Motor (0-100)</th>
<th>Vibrator (0-100)</th>
<th>Plane 1 (-5, +5)</th>
<th>Plane 2 (-5, +5)</th>
</tr>
</thead>
<tbody>
<tr>
<td>LIMS</td>
<td>100</td>
<td>70</td>
<td>Fixed</td>
<td>N.A</td>
</tr>
<tr>
<td>HIMS</td>
<td>Fixed</td>
<td>Fixed</td>
<td>+3</td>
<td>-1</td>
</tr>
</tbody>
</table>

Figure 7: Pictures of LIMS (left) and HIMS (right)

The flow chart in Figure 8 below summarizes the magnetic separation process. The products were chemically analyzed at ALS Scandinavia AB using the ICP-SFMS method with the same analysis protocol as mentioned in section 2.2 above.
2.3.2 Gravity separation

The two main layers (1_2-4 and 1_2-9) of drill core 1_2 were used for gravity separation tests, with each layer being tested as a separate sample. The sample specifications for this separation method were having not more than 2 kg with particle size less than 600 µm. A laboratory scale Knelson Concentrator at the Boliden AB laboratory was used with operating parameters set as shown in Table 3 below.

Table 3: Knelson concentrator operating parameters (Khavari 2018).

<table>
<thead>
<tr>
<th>Drum Speed (rpm)</th>
<th>Centrifugal force (G)</th>
<th>Fluidization water flow rate (L/min)</th>
<th>Fluidization water pressure (kPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1500</td>
<td>60</td>
<td>3.5</td>
<td>2</td>
</tr>
</tbody>
</table>

The picture of the Knelson concentrator and the flow chart in Figure 9 below summarizes the gravity separation process. The products were chemically analyzed at ALS Scandinavia AB using the ICP-SFMS method with the same analysis protocol as mentioned in section 2.2 above.

3 Results and discussion

3.1 Characterization

The physical and chemical characterization of the historical tailings showed variability in the layers of the three selected drill cores thus showing the tailings variability in the different locations of the repository. In terms of moisture content, the expectation was that the moisture content in the tailings samples would increase with increasing depth in the tailings repository. However, this was not the case as some tailings layers had higher moisture content than the layers underneath them as shown below in Figure 10.

This moisture variation was mainly attributed to the varying textures (particle sizes) in the tailings layers. It was observed that the layers with finer particle sizes had higher moisture content than the layers with coarser particles with an exception of the bottom layer (6-3) of drill
core 6. Only about 27 wt% of layer 6-3 had particle size greater than 1190 µm, which contributed to the high D_{80} value hence the high moisture content in this layer was attributed to the 73 wt% passing 1190 µm.

In order to appreciate the particle size variation in the three selected locations of the repository, the determined D_{80} values for the layers of the drill cores were plotted against the depth as shown in Figure 11 below. It was observed that drill core 1_1 had coarser particles than drill cores 1_2 and 6.

![Figure 10: Moisture content variation at different depths.](image)

![Figure 11: Variation of D_{80} values with depth.](image)

The drill core 1_1 layer at depth between 30 and 70 cm shows a significantly coarser (D_{80} = 1065 µm) tailings layer than the other layers in the same drill core. The bottom layer in drill core 6 was the coarsest (D_{80} = 1830 µm) but had material that was not typical of the tailings hence only chemical analysis could confirm the observation. The longest drill core 1_2 had two
layers that were significantly fine with $D_{80}$ values of 259 and 227 $\mu$m at depths 303 and 483 cm, respectively. This variation in texture of the tailings in the different locations would have an effect on the choice of methods to be considered for reprocessing. For instance in location 1_1 further grinding may be required as the minerals of interest may be locked up the coarse particles.

With regard to the six particle size fractions, the tailings weight distribution in each size fraction for each drill core was determined. It was observed that in all drill cores the dominating particle size fractions were +300 $\mu$m and +149 $\mu$m meaning most of the tailings particles were less than 600 $\mu$m but greater than 149 $\mu$m as can be seen in Figure 12 below. Drill cores 1_2 and 6 had more fines (-75 $\mu$m) than drill core 1_1. This variation in particle size is indicative of varying process parameters such as grinding size and/or changes in the mineralogy during the production period of 1936 to 1963.

![Particle Size Distribution (PSD) Curves](image.png)

**Figure 12:** PSD curves for the three drill cores (right).

Apart from the tailings weight distribution, elemental concentrations and distributions were also determined in the six particle size fractions. The main oxides were $\text{SiO}_2$, $\text{Al}_2\text{O}_3$, $\text{CaO}$ and $\text{Fe}_2\text{O}_3$ while W, Cu, S, Sn, Zn, Be and Bi were the main elements. However, it should be noted that Fluorine (F) was one of the main elements in the tailings (Hällström, Alakangas et al. 2018) but was not analyzed in the selected chemical analysis protocol for this study. From the elemental concentrations in the particle size fractions, weighted average concentrations in the layers of each drill core were determined and subsequently in the entire drill core. **Figure 13** below is an example of elemental concentrations and mass distributions in drill core 1_2. Tungsten (W) and Cu, which were the metals of interest, were observed to have high concentrations 2329 mg/kg and 1427 mg/kg respectively, in the fine (-75 $\mu$m) particle size fraction. However, in terms of mass distribution, all elements were high in particle size fractions +300 $\mu$m and +149 $\mu$m as was the case for the tailings weight distribution. Therefore, for purposes of reprocessing these historical tailings, mineral liberation needs to be considered for such coarse tailings particles.
Elemental concentrations and mass distributions were also varying at different depths of the drill cores. The highest WO$_3$ and Cu concentrations were observed in drill core 1_1 as shown in Figure 14 below. In all the three drill cores, the highest WO$_3$ concentration was between 88 and 153 cm depth. These highest WO$_3$ concentrations were 0.27, 0.22 and 0.16 % in drill cores 1_1, 1_2 and 6, respectively. For Cu, the depths of highest concentrations were different for each drill core; it was near the surface between 30 and 70 cm for drill core 1_1, much deeper for drill core 1_2 between 313 and 365 cm, and for drill core 6 it was between 111 and 153 cm.

These highest Cu concentrations were 1742, 1147 and 1178 mg/kg in drill cores 1_1, 1_2 and 6, respectively. However, it is important to know the mass distribution of these metals in the layers of the drill cores because the concentrations may be high but the actual quantities would be small when the total metal content in the drill core is considered. Figure 15 below shows the WO$_3$ and Cu mass distributions in the three drill cores.
From the Figure 15 above, it is observed that for drill core 6, most (~70%) of the WO₃ and Cu is contained in the upper layer (0-111 cm) while for drill core 1_2 it is between 174 and 295 cm with 32 wt% of the total WO₃ and 30 wt% of the total Cu. For drill core 1_1, the depth (30-70 cm) with the highest Cu concentration also had the highest mass distribution of Cu (24 wt%) and W (21 wt%).

Since these tailings have been in this repository for a very long time, sulphur depletion due to oxidation was expected with the depletion decreasing from top to bottom though this would also depend on other factors such as particle size and initial quantity. Therefore, both concentration and mass distribution of sulphur in each drill core was studied. In all the three drill cores, the highest S concentration was between 88 and 153 cm depth as was the case for WO₃. These highest S concentrations were 1.89, 1.85 and 3.05 % in drill cores 1_1, 1_2 and 6, respectively, as shown in Error! Reference source not found. (left) below.

In Figure 16 (right) above, it is observed that for drill core 6, 57 % of the S is contained in the upper layer (0-111 cm) while for drill core 1_2 it is between 174 and 295 cm with 28 wt% of the total S. For drill core 1_1, the depth (150-190 cm) had the highest mass distribution of S at 24 wt%. The chemical analysis also confirmed the observation that the material in bottom layer of drill core 6 was not typical of the tailings as the concentration and distribution of all the main elements was extremely low. Observing the depletion trend for drill core 1_2 in Figure 16 (right), there would be three possible main deposition and oxidation periods where S is seen to be having a significant stepwise increase in depth; the first being for the depth 296-483 cm, second 145-295 cm and third 0-144 cm. These trends also show possible mineralogical variations with regard to sulphur content over the production period of 1936 to 1963.

These variations in the elemental concentrations and distributions at different depths of the locations would have an effect on the grades and recoveries of the concentrates that may be produced from the reprocessing of these tailings. Consequently, process parameters would need to be varied depending on which location and depth tailings are obtained from. For instance in location 1_2, in order to get much of W and Cu out this area, tailings at deeper depths (below 174 cm) must be treated. Blending of tailings during reprocessing may also be necessary for process and product optimization.

Taking the average WO₃ and Cu concentration of these three tailings locations to be 0.15 % and 0.11 % respectively, and applying them to the estimated 2.2 million tons of tailings in this repository gives approximately 3300 tons of WO₃ and 2512 tons of Cu.

Table 4 below summarizes the weighted average elemental concentrations in the Three (3) drill cores. Drill core 1_1 had higher concentrations in WO₃, Cu, Sn, Zn, Be and Bi than drill cores
1_2 and 6, while drill core 6 had the highest S concentration. Taking the average WO$_3$ and Cu concentration of these three tailings locations to be 0.15 % and 0.11 % respectively, and applying them to the estimated 2.2 million tons of tailings in this repository gives approximately 3300 tons of WO$_3$ and 2512 tons of Cu.

Table 4: Weighted average elemental concentrations in the Three (3) drill cores.

<table>
<thead>
<tr>
<th>Core</th>
<th>WO$_3$ (%)</th>
<th>Cu (mg/kg)</th>
<th>S (%)</th>
<th>Sn (mg/kg)</th>
<th>Zn (mg/kg)</th>
<th>Be (mg/kg)</th>
<th>Bi (mg/kg)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drill core 1_1</td>
<td>0.20</td>
<td>1418</td>
<td>1.32</td>
<td>602</td>
<td>319</td>
<td>282</td>
<td>538</td>
</tr>
<tr>
<td>Drill core 1_2</td>
<td>0.12</td>
<td>981</td>
<td>1.33</td>
<td>558</td>
<td>251</td>
<td>241</td>
<td>368</td>
</tr>
<tr>
<td>Drill core 6</td>
<td>0.13</td>
<td>1026</td>
<td>1.73</td>
<td>517</td>
<td>275</td>
<td>223</td>
<td>435</td>
</tr>
</tbody>
</table>

In optical microscopy, the fluorescence property of Scheelite mineral was used to identify the scheelite particles and the blue filter was found to be effective for distinguishing them from other minerals of similar luminance and brightness such as Bismuthinite (Khavari 2018). Figure 17 (left) shows how scheelite particles appeared under a microscope in normal light and with the blue filter. The distinction between scheelite and bismuthinite particles was verified using SEM-EDS chemical composition.

Figure 17: Scheelite identification (left), scheelite texture – fine, coarse particles (centre), scheelite liberation – liberated, non-liberated particles (centre), and scheelite particle with a rim around it (right). (Khavari 2018)

The scheelite particles in the polished samples were both fine and coarse, and liberated and non-liberated, meaning mineral liberation analysis would be essential in order to develop effective separation methods. Some had rims, which were identified as hydrous ferric oxides (Hällström, Alakangas et al. 2018) hence the recovery of such scheelite particles may be hindered in processes like flotation where reagents need to have contact with the mineral particle. Hence, pre-treatment methods such as scrubbing may be required.

The XRD and SEM results were complementary to the geochemical characterization results obtained by our project partners in REMinE who did a more detailed analysis. The main minerals in which the main elements W, Cu, S, Sn, Zn, Be, Bi and F were contained were scheelite, chalcopyrite, pyrrhotite, cassiterite, danalite (both Zn and Be), bismuthinite and fluorite, respectively. (Hällström, Alakangas et al. 2018, Khavari 2018)
3.2 Physical separation

The tailings characterization results showed that the concentration of W was high in the fines (-75 µm) as could be seen earlier in Figure 13 (left). Therefore, being found in the high-density mineral scheelite, enhanced gravity separation by use of a Knelson concentrator would be appropriate. The tailings also had high sulphur concentration, which was mainly in the mineral pyrrhotite a paramagnetic mineral, and had magnetite a ferromagnetic mineral hence magnetic separation by use of LIMS and HIMS was appropriate. This magnetic separation would enhance the separation of minerals like scheelite and chalcopyrite from pyrrhotite, which is the main Fe-sulphide mineral in the tailings responsible for Acid Mine Drain (AMD) (Hällström, Alakangas et al. 2018).

3.2.1 Magnetic separation

The two layers (1_2-4 and 1_2-8) of drill core 1_2 were used with each layer being tested as a separate sample. For both samples, the amount recovered in LIMS was very low at 4.2 wt% and 5.0 wt% respectively. This indicates a low amount of ferromagnetic minerals in the tailings. In the HIMS, the magnetic fraction had the highest mass recoveries at 87.3 wt% and 85.8 wt% respectively thus indicating a high amount of paramagnetic minerals. The recoveries of S, W and Cu in the magnetic fraction of the HIMS shown in Table 5 below were of interest in assessing the separation. Sulphur was mostly recovered in the magnetic fractions of both separators with only about 1.3 wt% in the non-magnetic fraction as desired which would be good environmentally. However, W and Cu, which were desired to be recovered in the non-magnetic fraction of the HIMS ended up in the magnetic fraction meaning the separation was not achieved. This would be due to the non-liberation of scheelite and chalcopyrite minerals from the paramagnetic minerals and/or the liberated small particles are entrapped and entrained.

Table 5: Recoveries in the magnetic fraction of HIMS.

<table>
<thead>
<tr>
<th>Drill core layer</th>
<th>%S</th>
<th>%W</th>
<th>%Cu</th>
</tr>
</thead>
<tbody>
<tr>
<td>1_2-4</td>
<td>45.2</td>
<td>80.6</td>
<td>90.8</td>
</tr>
<tr>
<td>1_2-8</td>
<td>35.4</td>
<td>83.6</td>
<td>90.3</td>
</tr>
</tbody>
</table>

3.2.2 Gravity separation

The two main layers (1_2-4 and 1_2-9) of drill core 1_2 were used for gravity separation tests, with each layer being tested as a separate sample. The separation cycles were two and three respectively. The recovery of W in each cycle product was assessed as shown in
Table 6 below. It was observed that concentrate W recovery was decreasing with the increasing number of separation cycles. This is due to the decreasing amount of dense and coarse particles that contain W. The high W recovery in the tailings is an indication of high W in the fine fractions. Comparison between the two layers shows a better recovery for layer 1_2-4, which was coarser with higher W content than layer 1_2-9.
Table 6: Tungsten recoveries in the Knelson concentrator products.

<table>
<thead>
<tr>
<th></th>
<th>Cycle 1 Concentrate 1 (%W)</th>
<th>Cycle 2 Concentrate 2 (%W)</th>
<th>Cycle 3 Concentrate 3 (%W)</th>
<th>Tailings (%W)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drill core layer 1_2-4</td>
<td>35.1</td>
<td>12.6</td>
<td>N/A</td>
<td>52.4</td>
</tr>
<tr>
<td>Drill core layer 1_2-9</td>
<td>26.7</td>
<td>15.7</td>
<td>9.5</td>
<td>48.1</td>
</tr>
</tbody>
</table>

4 Conclusions

The characterization of the Yxsjöberg historical tungsten ore tailings was done as a fundamental study that will help to develop effective methods for separating minerals and extracting metals of interest from these historical tailings so that an inert and environmentally safe residue is left behind. There was particle size and elemental composition variability in the three repository locations. The tailings mass distribution was high in the coarser particle size fractions of +300 µm and +149 µm hence mineral liberation analysis would be essential in order to develop effective separation methods. Therefore, grinding of the coarser particle size fractions should be considered with economic viability. Tungsten (W) and Copper (Cu) were the metals of interest with location 1_1 (60.041669, 14.771031) having higher concentrations than the other two at 0.20 %WO₃ and 0.14 %Cu. The average WO₃ and Cu concentration of the three tailings locations was 0.15 % and 0.11 % respectively, and applying them to the estimated 2.2 million tons of tailings in this repository gives approximately 3300 tons of WO₃ and 2512 tons of Cu.

Tungsten recovery in the Knelson concentrator was enhanced but the mass recovery was low. Process optimization for these tailings would be required as the parameter settings that were used were optimized for gold recovery. For magnetic separation, W and Cu, which were desired to be recovered in the non-magnetic fraction of the HIMS ended up in the magnetic fraction hence the desired separation was not achieved. Therefore, the liberation of scheelite and chalcopyrite mineral particles need to be assessed and possibly have another separation step prior to magnetic separation which could take care of the entrapment and entrainment of liberated small particles.

Acknowledgements

Our sincere gratitude goes to the late Niclas Strandberg (AB Yxsjö Industriservice) who was the owner of the Yxsjöberg tailings repository for his support during the sampling campaigns. We also thank our REMinE project partners from Luleå University of Technology (LTU) in Sweden, University of Porto in Portugal and the Research and Development National Institute for Metals and Radioactive Resources (INCDMRR) in Romania. Special thanks goes to Lina Hällström, Erdogan Kol and Musah Salifu for their help and support during the sampling campaigns and for very insightful discussions. Thanks to Gunnar Frohm and Johan Hansson from Boliden Mines for allowing us to use their Knelson concentrator and for helping us during the laboratory exercise, and to Dr. Mehdi Parian for making it possible for us to get to Boliden and supporting us throughout. Special thanks to Dr. Cecilia Lund for advising and guiding us on sample preparation for optical microscopy, and to Dr. Bertil Pålsson for helping us plan and conduct physical separation tests. This work was funded by VINNOVA [Grant numbers 215 06 631].
References


