



Geological Survey of Finland

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Metallurgical testwork for the geometallurgical orientation study of the Mawson Gold's Rajapalot Au-Co project *BATCircle Project Report 05 – WP1 Task 1.2*

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Abstract <p>This report summarises the main findings of an orientation study work performed on 3 previously selected ore types from the Rajapalot project. These 3 ore-types (MP, AY & PAL1), which display distinct mineralogical properties, have been tested using various mineral processing methods, including magnetic separation, gravity concentration and cyanide leaching (for gold only) with a view to investigating their processing behaviour, as well as evaluating the potential of each technique.</p> <p>Overall, flotation appears to be the most efficient technique both for gold and cobalt recovery, with recovery rates and concentrate grades above 90% and 100 g/t for gold, and between 23-63% and above 1% for cobalt, with best performances for cobalt obtained with the MP and AY ore types. Magnetic and gravity concentration only yielded very low recoveries for gold below 50% for all ore types, but cobalt recovery as high as 71% has been obtained during the magnetic separation of the PAL1 ore type. This is a result of a mineralogical control over cobalt recovery which is dictated by the cobalt deportment in linnæite (PAL1) or cobaltite (AY or MP). Indeed, cobalt recovery in the magnetic fraction can be accounted for by linnæite locked in pyrrhotite, while flotation is selectively floating cobaltite with mineral recoveries between 78%-93%. For the penalty elements, arsenic is almost exclusively hosted in cobaltite, and it will therefore follow the latter in the flotation concentrate, while uranium tends to follow the same pattern as gold during magnetic and gravity concentration, but is not recovered during flotation, making flotation the ideal method to selectively recover gold and cobaltite against uraninite.</p> <p>This study has highlighted that the 3 selected ore types can be considered as individual geometallurgical ore types with distinct process behaviour when submitted to the same process. This is partly due to a clear mineralogical control over cobalt recovery which is constrained by cobalt deportment in cobaltite or linnæite. Considerations for future testwork and process development are also presented.</p>			
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1 INTRODUCTION

Current production of battery raw materials is concentrated in a few countries outside the EU, especially for cobalt and graphite, with about 70% of the global cobalt supply coming from the Democratic Republic of Congo (DRC) and 64% of the global graphite supply from China (USGS, 2020). Hence, the effective and efficient recovery of these minerals is fast becoming a strategic priority for Europe and Finland in particular (Dehaine et al., 2020). Finland is one of the most important producers of battery raw materials for the EU market, meeting 66% of the EU demand for cobalt ores and concentrates and 16% of the demand for nickel (European Commission, 2018).

This report is one of a series of reports to document the work done in the development of the geometallurgy of the Rajapalot case study, which is part of Work Package WP1.2 in the BATCircle project. Rajapalot is a gold-cobalt project, where cobalt is considered as a by-product, and is located 35 km west of Rovaniemi, Lapland, Finland. A description and experimental approach of the BATCircle project is shown in Appendix 13.1. This report is the third for this case study, where the first was the ore type definition and sample selection procedure (Farajewicz and Cook, 2019), while the second detailed the geochemical and mineralogical characterisation of these ore types (Dehaine et al., 2021a).

2 BATCIRCLE GEOMETALLURGICAL PROGRAM

2.1 BATCircle geometallurgical program

The BATCircle project Work Package 1.2 had two geometallurgical tasks. The first was a documented geometallurgical procedure (Michaux, S. and O'Connor, 2020). The second was to have this procedure shown in two case studies. One of these case studies is the Rajapalot project geometallurgical study.

2.2 Geometallurgical orientation study objective

In context of mineral processing, gravity, flotation, and leaching have been considered as process paths in previous work. The geometallurgical objective for this campaign is:

Geometallurgical orientation study to refine the metallurgical process response and maximize recovery of gold and cobalt minerals into two separate concentrates

Questions to address:

- Primary question – what are the most adapted methods and what is the most efficient process path? Flotation, leaching, or a combination of both?
- Can gravity separation and magnetic separation improve the performance of the best process path?
- Does the process performance vary between the different ore types?
- How to achieve maximum gold and cobalt recovery at the same time?
- How to remove and control penalty elements (arsenic and uranium) along the process path?
- Is there any mineralogical control over process performance for the target metals and penalty elements?

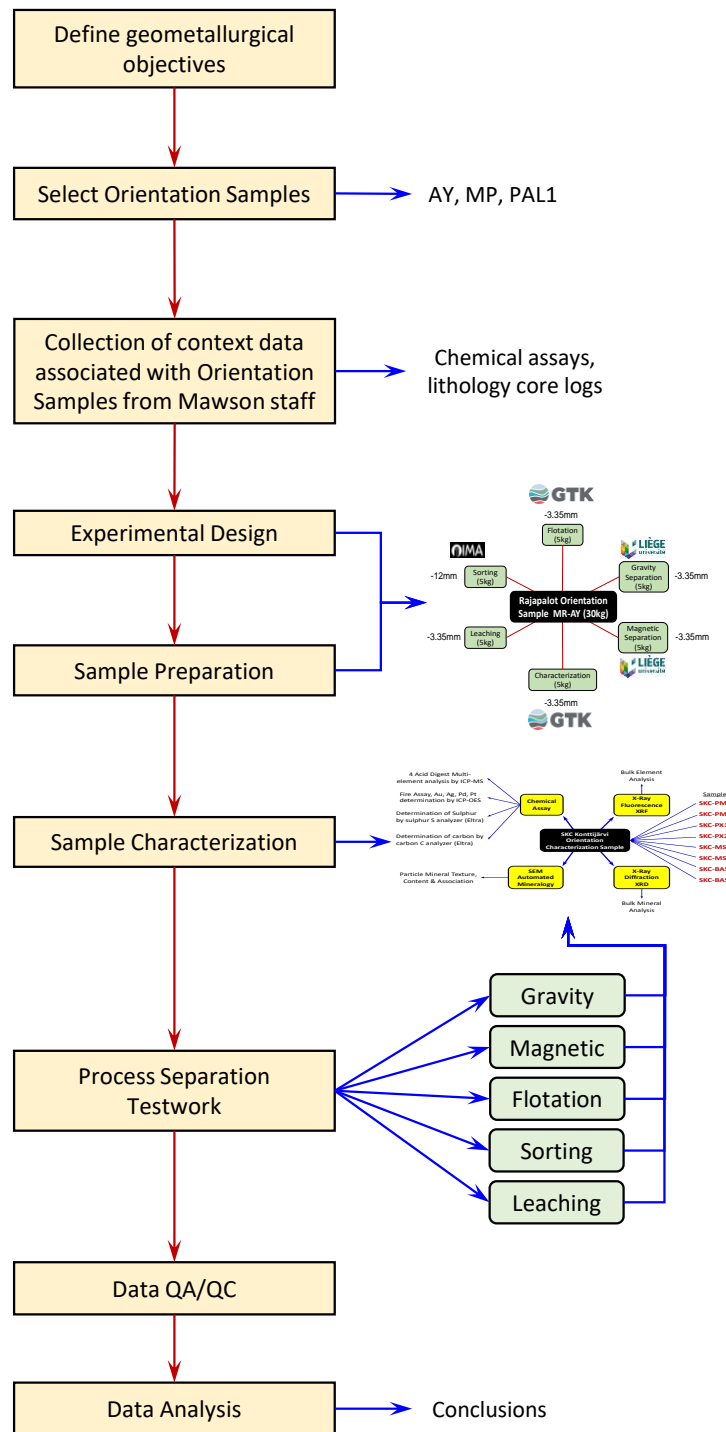


Figure 1. Rajapalot case study geometallurgical program.

3 MATERIALS AND EXPERIMENTAL DESIGN

3.1 Materials

The ore type definition and sample selection procedure has been reported in Farajewicz and Cook, 2019, while detailed geochemical and mineralogical characterisation of these ore types can be found in a previous report (Dehaine et al., 2021a). The characterisation results suggest that the 3 ore types have distinct mineralogical characteristics that are likely to influence process performance. An

overview of these key ore properties is given in Table 1. As such the ore types selected based on geological criteria do represent distinctive geometallurgical ore types that are likely to behave differently if processed through the same route, and hence must be considered separately for the subsequent geometallurgical program and test work planned.

Table 1. Overview of the main ore properties for the 3 geometallurgical ore types of the Rajapalot project.

Property	Ore Type		
	MPC-PAL1	MRC-AY	MRC-MP
Grades			
Au (g/t)	4.37	3.91	13.33
Co (%)	0.05	0.13	0.12
Mineralogy			
Main gangue	Plagioclase > Amphiboles	Plagioclase > Micas+ Qtz	Micas+ Qtz > Plagioclase
Problematic gangue (wt%)	Chlorite (~12%)	Chlorite (~7%), Micas (~9%)	Chlorite (~7%), Micas (~33%)
Pyrrhotite (wt%)	~12%	~14%	~13%
Cobalt deportment	Linnaeite >> Cobaltite	Cobaltite > Linnaeite	Cobaltite >> Linnaeite
Cobaltite liberation	~23%	~65%	~47%
Linnaeite liberation	~7%	~14%	~4%
Gold liberation	~60%	NA	NA

3.2 Experimental design

The experimental design of this study has several components. The first step was to collate all samples collected for each Orientation Sample, crush the drill core to 99% passing 3.35mm, which were then subdivided in a representative manner (using a sample riffler), as shown in Figure 2. One sub-sample for each Orientation sample was subjected to characterization (Dehaine et al., 2021a). Each remaining sub-sample was sent to a laboratory for process separation.

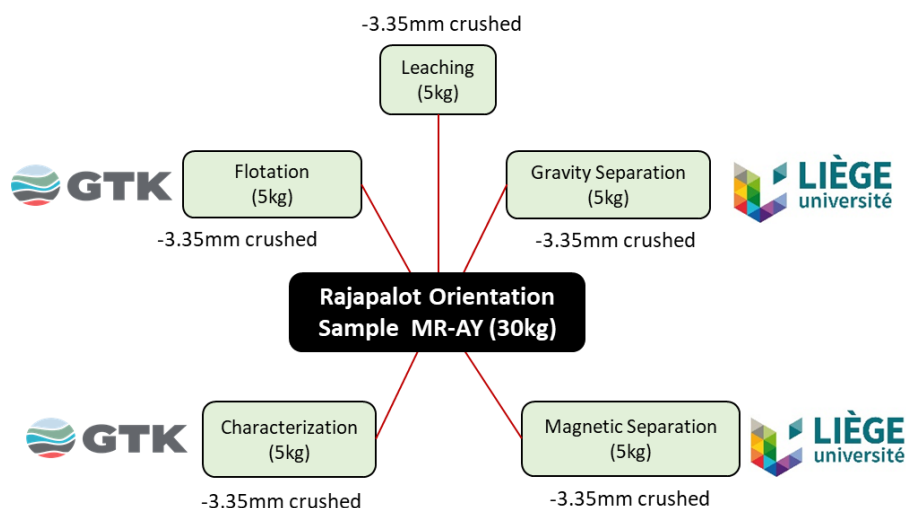


Figure 2. Experimental design sub-sampling.

The products of each process test were then characterized, in a manner where the feed samples and product samples could be used for a mineral mass balance (Figure 3).

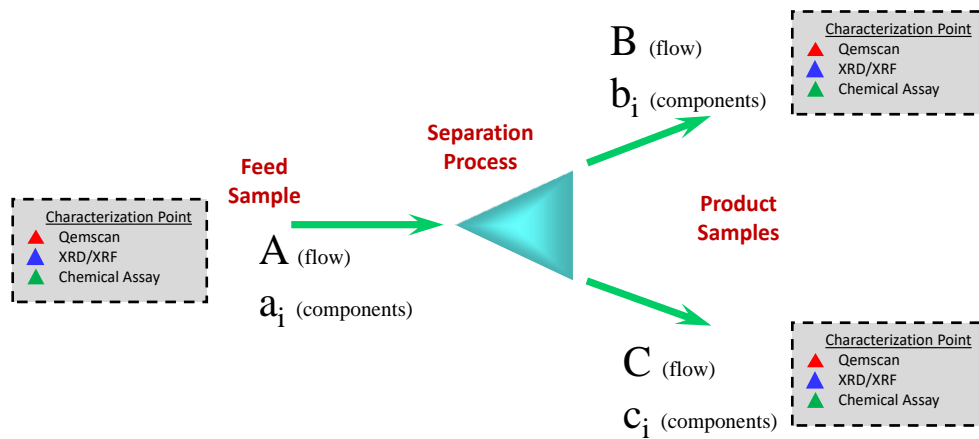


Figure 3. Each process test is done in a manner where a mineralogical mass balance could be conducted, when possible.

Figure 4 shows how the process separation tests would be used, where different process paths are compared. In Orientation Step 1, the process separation method (gravity, flotation, etc.) is tested on the sample as received. At the end of Step 1, an examination of what works for the best recovery and what does not work in context of the study objectives. Orientation Step 2 (also shown in Figure 4) is the testing of different combinations of process steps. Orientation Step 3 would be an experimental test of the process path combination of choice.

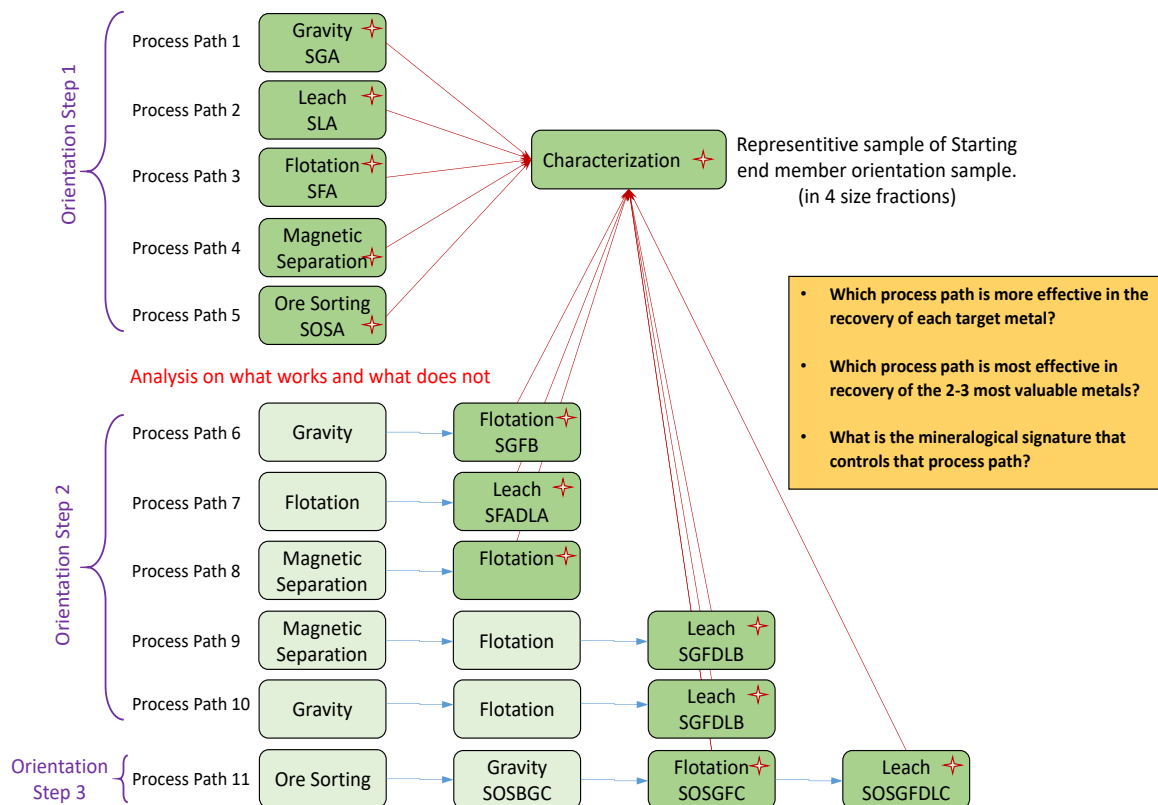


Figure 4. Process test experimental design.

Due to difficulties related to the Covid-19 quarantine, this report only shows only Orientation Step 1.

Figure 5 shows the outcomes of the process tests in each Orientation Sample together in one data set. These data set would be used to examine which process path is most effective recovery for different target metals (Au, Co).

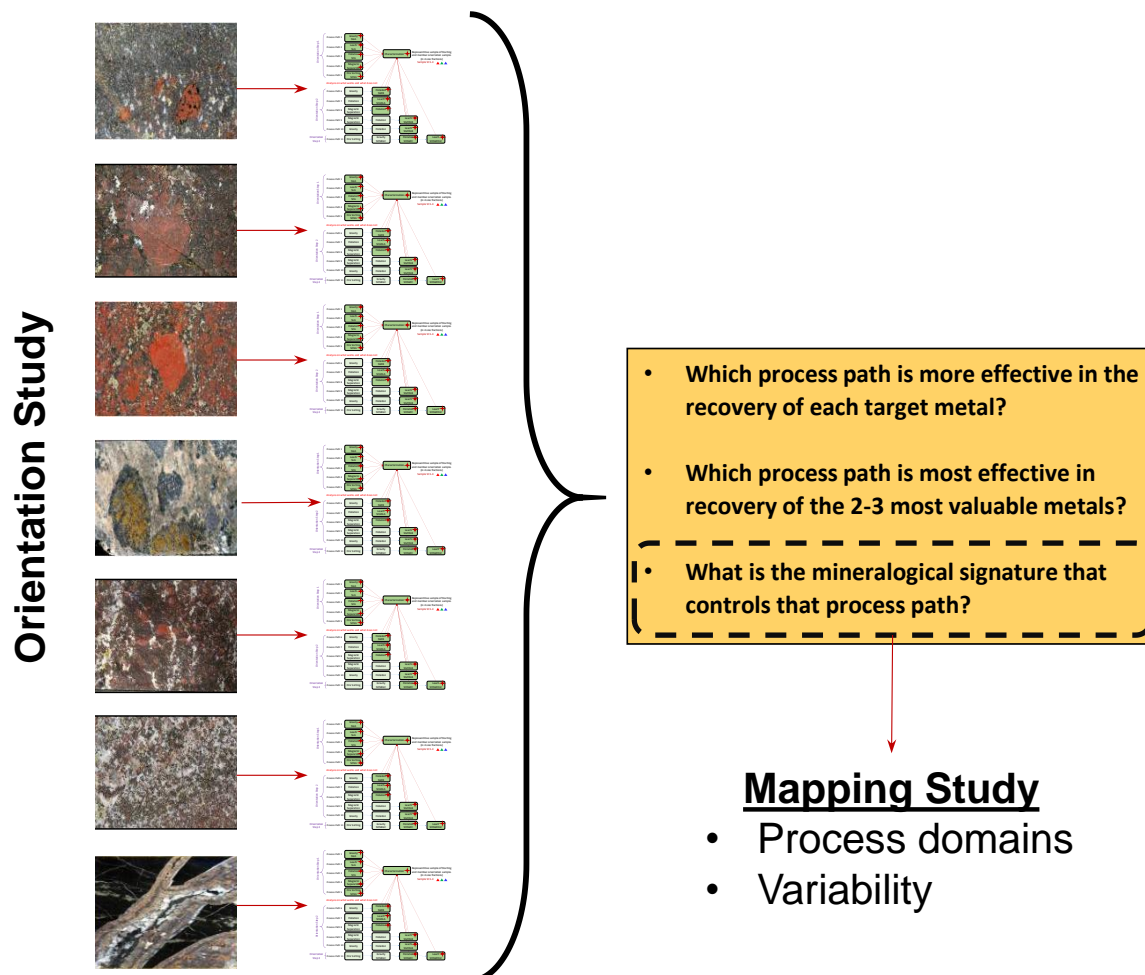


Figure 5. Different process paths compared across all Orientation Studies.

3.3 Chemical assays

Whole rock geochemical assays were performed by Eurofin Labtium (Kuopio, Finland) following two methods: A four acid digestion (Method code 306P /306M) to get accurate estimate of the gold and cobalt content of each sample, as well as Inductively Coupled Plasma Optical-Emission Spectrometry (ICP-OES, Thermo Fischer ICap 6500 Duo) for major elements and Mass Spectrometry (ICP-MS, Thermo Electron iCAP Qc) for trace elements.

For the four-acid digestion method, a representative aliquot of each sample was prepared in pulp (0.2 g) and digested with a mixture of concentrated acids (hydrofluoric acid, perchloric acid, hydrochloric acid and nitric acid) in a teflon tube by heating on a heating block. The solution is then evaporated to dryness and the solid residue is dissolved in nitric acid. The final solution is diluted with water prior to instrumental measurements using ICP-OES and/or ICP-MS.

3.4 Quantitative mineralogical analysis by X-Ray Diffraction (QXRD)

QXRD analyses were conducted by X-Ray Mineral Services (Wales, UK). A whole-rock and clay fraction QXRD analysis was carried out for each sample. The samples were first disaggregated using a pestle and mortar. A 2 g split of this material was used for the whole rock analysis: the samples were 'micronised' using a McCrone Micronising Mill to obtain a powder with a particle size between 5 - 10 μm . The slurry was dried overnight at 80°C, re-crushed to a finer powder and back-packed into a steel sample holder, producing a randomly orientated sample. The whole-rock samples were scanned on a PANalytical X'Pert PRO diffractometer using a $\text{CuK}\alpha$ radiation at 40 kV and 40 mA. The diffractometer is equipped with Automatic Divergence Slits (10 mm irradiated area), sample spinner and PIXcel 1-D detector. Scan parameters are from 4.5 to 75° (2 θ), at a step size of 0.013 and nominal time per step of 0.2 s (continuous scanning mode). The goal of the whole-rock sample preparation is to have a random orientation of the grains, allowing unbiased phase quantification and minimizing the error caused by preferred orientation of certain minerals (*e.g.*, mica flakes, feldspar, calcite). The study of a randomly oriented powder will give an approximate proportion of clay minerals present in the sample.

Qualitative analysis on whole rock samples was carried out using two commercial software packages associated with the ICDD database: Traces (v.6) by GBC Scientific Equipment and HighScore Plus (v.4) by PANalytical. Quantitative X-ray Diffraction (QXRD) was performed using the Rietveld method with BGMN Autoquan software. The samples studied here contain minerals such as chlorite, biotite, talc and amphibole in abundant quantities – these minerals' grains tend to have strong preferred orientation normal to the surface of the sample. The preferred orientation creates a systematic error in the observed diffraction peak intensities, but the Rietveld method has proven to be effective even with samples that contain abundant minerals with a platy or fibrous texture.

For each sample, a 5 g split of the disaggregated material was taken and weighed accurately. The weight was recorded in a central register for later reference. Separating the $-2 \mu\text{m}$ fraction was achieved by ultrasound and centrifugation. The total weight of clay extracted was determined by removing a 20-25 g aliquot of the final clay suspension and evaporating to dryness at 80°C. The initial and final weights of the beaker used were also recorded in the register. The clay XRD mount was obtained by filtering the clay suspension through a Millipore glass micro-fibre filter and drying the filtrate on the filter paper. The samples were analysed as an untreated clay, after overnight saturation with ethylene glycol vapour and following heating at 380°C for 2 hours, with a further heating to 550°C for one hour. Clay filters were scanned on a Philips PW1730 diffractometer using a $\text{CuK}\alpha$ radiation at 40 kV and 20/25 mA. The results of the clay fraction analysis are considered semi-quantitative, whereas the whole-rock are quantitative as they are achieved with a different method that is standard-less (the Rietveld method).

Detection limits of XRD vary depending on the mineral's crystallinity, whether other minerals in the sample produce overlapping diffraction peaks, and how the XRD experiment has been set up. The average detection limit for the analyses presented is approximately 2%.

3.5 Cobalt minerals quantification

Because of the aforementioned detection limit, QXRD does not allow quantification of the amount of cobalt minerals (cobaltite, linnaeite, cobaltian-pyrite), except for cobaltite, which in some concentrates, where it is enriched. To overcome this issue, the amount of cobaltite and linnaeite in the samples is approximated using the average composition of each of the minerals obtained by EMPA (Dehaine et al., 2021a), and by assuming that all the arsenic is hosted in cobaltite, and that all the cobalt is hosted either in cobaltite or linnaeite.

4 MAGNETIC SEPARATION

4.1 Methods/Procedure

Magnetic separation and gravity separation tests on the 3 ore types have been conducted at the University of Liege, Belgium. The sample preparation involves crushing and milling of each sample to reach a P80 of 75 μm , following the flowsheet in Figure 6a. Magnetic separation tests were carried out on a Wet High Intensity Magnetic Separator from Carpco Inc (model CC WHIMS 2X 4XL) equipped with grooved plates as matrix at 25 wt% solid with samples of 1 kg each (Figure 7). The sample were processed following the flowsheet in Figure 6b, whereby the pulp was submitted successively to a 1 Amp, 4 Amp and 6 Amp magnetic separation. The so-called concentrates correspond to the magnetic fraction whereas the final tailings correspond to the non-magnetic fraction of the sample.

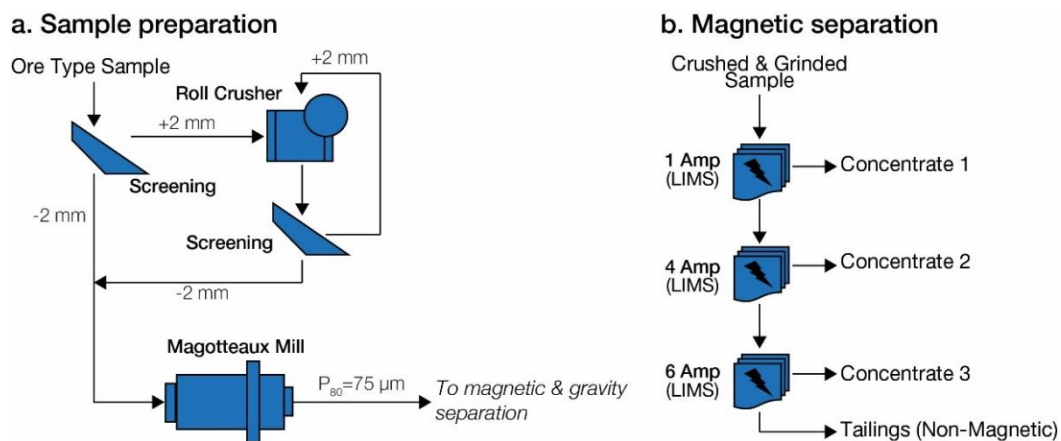


Figure 6. Flowsheets showing the (a) sample preparation procedure and (b) magnetic separation procedure.



Figure 7. Photographs of the magnetic separation test work.

4.2 Results

Results in terms of yield, grade and recovery for target metals (gold and cobalt) and penalty elements (arsenic and uranium) are given in Figure 8, Figure 9 and Figure 10 respectively. The total magnetic fraction (all magnetic concentrates) represents between 25 wt% (MP) and 49 wt% (PAL1) of the total sample. Overall, gold recovery is rather limited in the magnetic products with, however, a relatively high gold grade at 38 g/t Au obtained in the 4 Amp (4A) concentrate for the MP ore type. Cobalt seems to be concentrated in the magnetic fractions with recoveries ranging from 34 to 71% in the 1 amp (1A) product, and decreasing cobalt grade from the 1A to the non-magnetic product with grades above 2000 ppm in the 1A product for MP an AY ore types (Figure 9). Arsenic recovery in the magnetic fraction is very limited and uranium seem to follow the same pattern as gold (Figure 10).

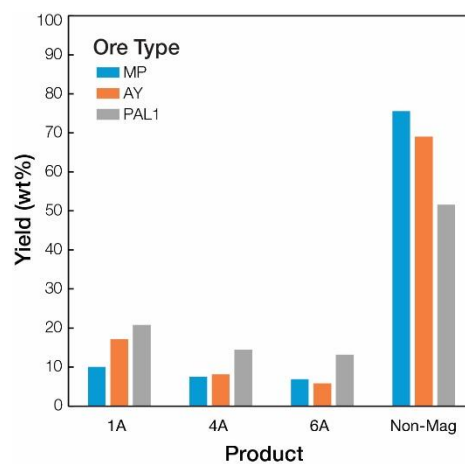


Figure 8. Mass balance between the magnetic separation products for the 3 ore types.

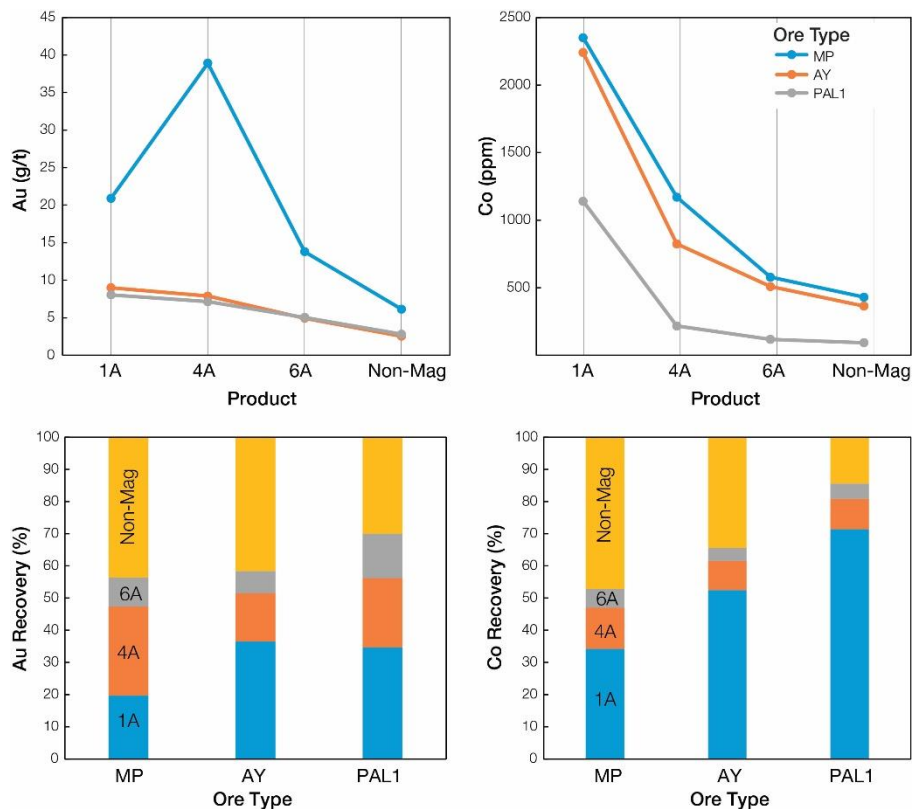


Figure 9. Target elements (Au, Co) grade and recovery in the magnetic separation products for the 3 ore types.

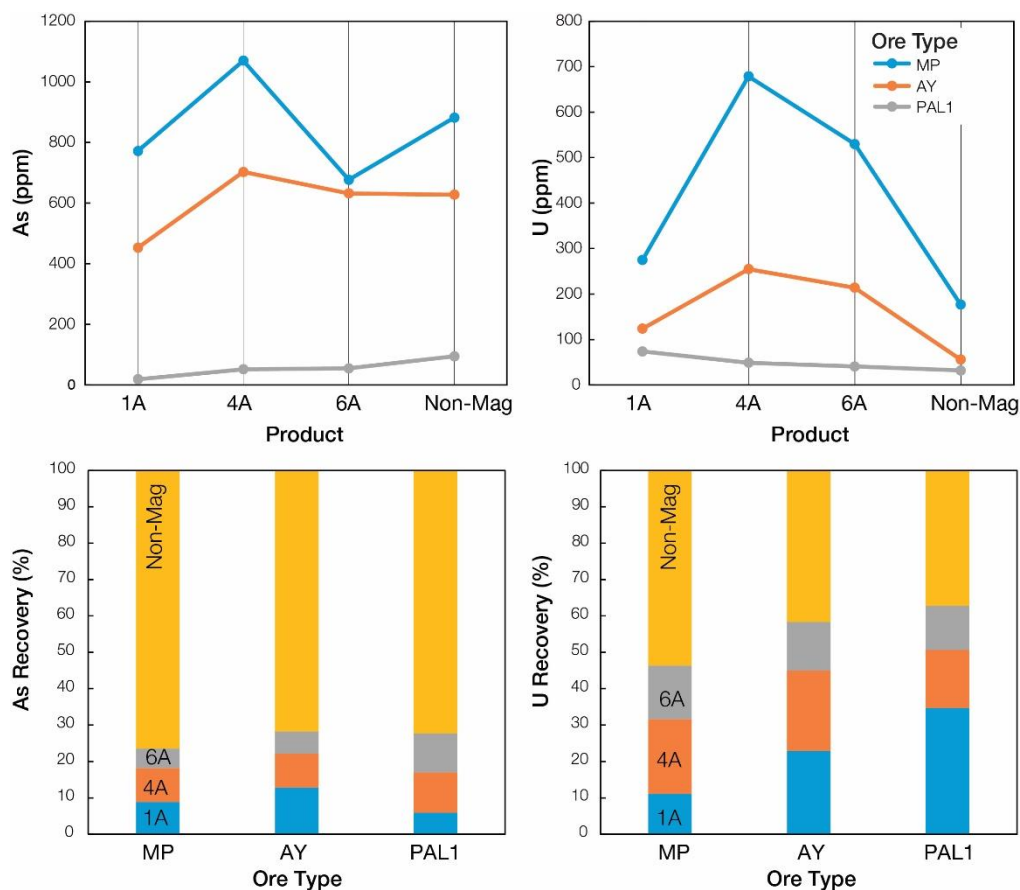


Figure 10. Penalty elements (As, U) grade and recovery in the magnetic separation products for the 3 ore types.

The bulk mineralogy of the magnetic separation products for each ore type obtained through quantitative XRD is shown in Figure 11. The magnetic fractions are mostly composed of pyrrhotite, biotite, chlorite, muscovite, amphiboles and to a lesser extent quartz and feldspars. The proportion of each mineral varies depending on the product and the ore type but globally the 1A concentrate is mostly composed of pyrrhotite (between 45-78 wt%). The amount of pyrrhotite decreases in the 4A and 6 amp (6A) concentrates. Mineral recoveries for the gangue minerals considered problematic are shown in Figure 12. Note that only minerals for which a complete mass balance can be calculated are shown, *e.g.*, muscovite is considered problematic but below the detection limit in most of the products. Most of the pyrrhotite is recovered in the 1A concentrate, with recoveries of up to 90% in the case of the AY ore type. Biotite and chlorite seem to be poorly recovered in the magnetic fractions.

In terms of cobalt minerals, cobaltite and linnaeite show a completely opposite behaviour (Figure 13). Indeed, almost all the linnaeite is recovered in the 1A magnetic concentrate while most of the cobaltite is recovered in the non-magnetic fraction. Thus, the cobalt recovery in the magnetic fractions observed in Figure 9 is mostly due to linnaeite. Since linnaeite is known to be diamagnetic (Dehaine et al., 2021b), and that it is always locked in pyrrhotite regardless of the ore type (Dehaine et al., 2021a), the observed linnaeite recovery in the magnetic fraction is therefore due to the magnetic separation of pyrrhotite in which linnaeite grains are locked.

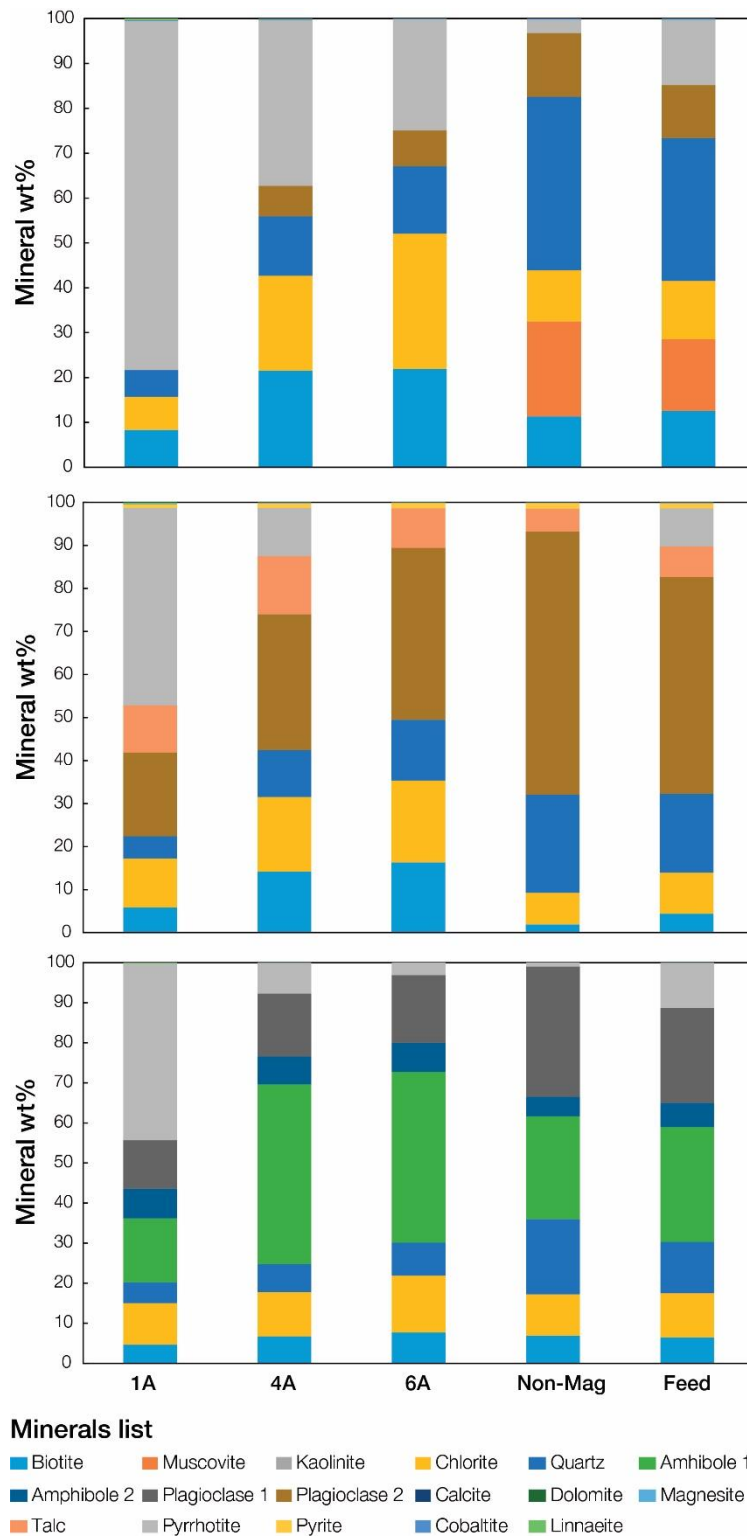


Figure 11. Quantitative mineralogy (QXRD) of the magnetic products separation products for the 3 ore types: MP (top), AY (middle) and PAL1 (bottom).

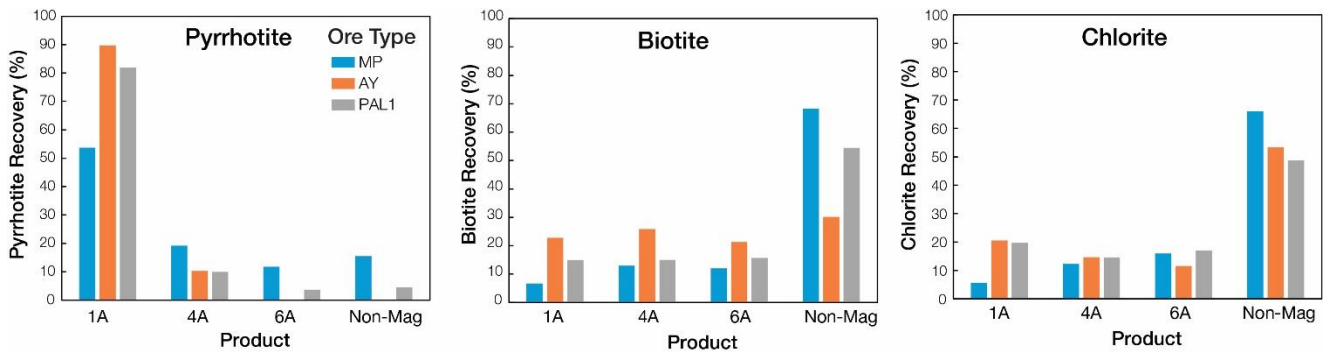


Figure 12. Problematic gangue minerals recovery in the magnetic separation products for the 3 ore types. Pyrrhotite content below the detection limit for AY ore type 6 amp and non-magnetic products.

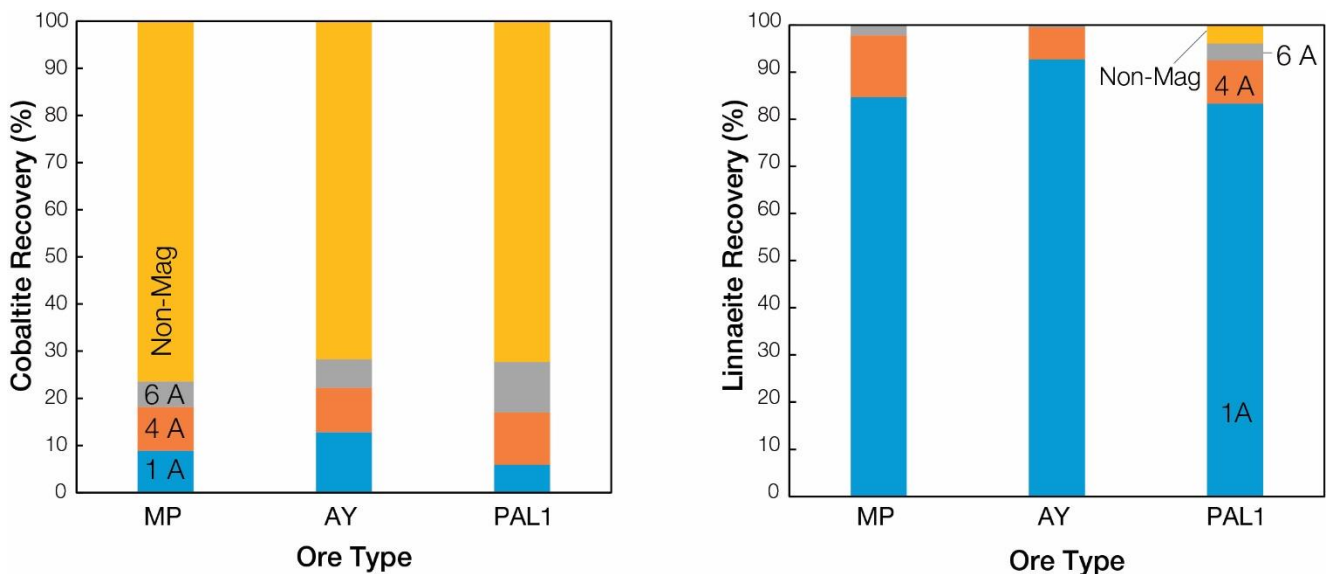


Figure 13. Cobaltite (left) and linnaeite (right) recovery in the magnetic separation products for the 3 ore types. Note that these results are obtained using chemical assays and minerals stoichiometry (see section 3.5).

5 GRAVITY CONCENTRATION

5.1 Methods/Procedure

As for the magnetic separation test work, the gravity separation tests on the 3 ore types have also been conducted at the University of Liege, Belgium. The same sample went through the same sample protocol preparation involving crushing and milling of each sample to reach a P80 of 75 μm, following the flowsheet in Figure 14a. Gravity concentration tests were carried out on a KHD Humboldt Wedag shaking table is Type MN 929/2 with a 5 kg sample (Figure 15). The pulp density was set at 15wt% solids and the feed rate at 1 kg/min. Amplitude and frequency of the stroke, deck angle, and wash water flowrate were kept constant for all the tests at 7 mm, 6 Hz, 2.5° and 7.5 L/min respectively.

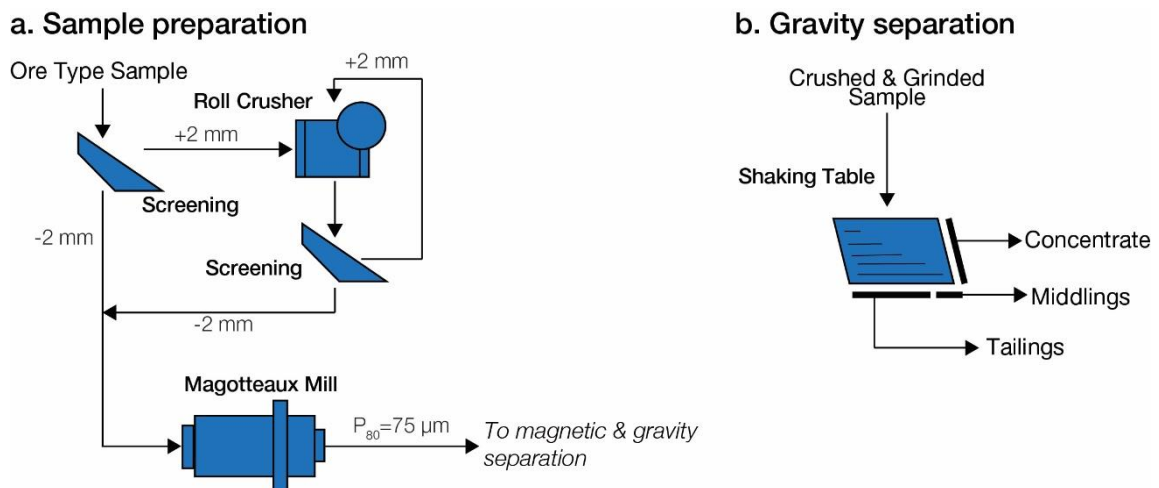


Figure 14. Flowsheets showing the (a) sample preparation procedure and (b) gravity concentration procedure.



Figure 15. Photographs of the shaking table test work.

5.2 Results

Results in terms of yield, grade and recovery for target metals (gold and cobalt) and penalty elements (arsenic and uranium) are given in Figure 16, Figure 17 and Figure 18 respectively. For all ore types, the yields are relatively low (1.3-2.2 wt%). Overall, gold recovery is relatively low (33-44 %) with the best concentration of up to 800 g/t obtained for the MP ore type (Figure 17). Cobalt recovery in the concentrate is marginal with most of the cobalt going to the tailings or middling. However, a clear concentration is observed for cobalt with MP and AY ore types with grades of up to 1.3 and 1 %Co respectively. Similarly, arsenic and uranium recovery in the gravity concentrate are relatively low. In terms of concentration arsenic seems to follow cobalt with higher grades obtained with MP and AY ore types, while the higher uranium grade, as for gold, is obtained with the MP ore type (Figure 18).

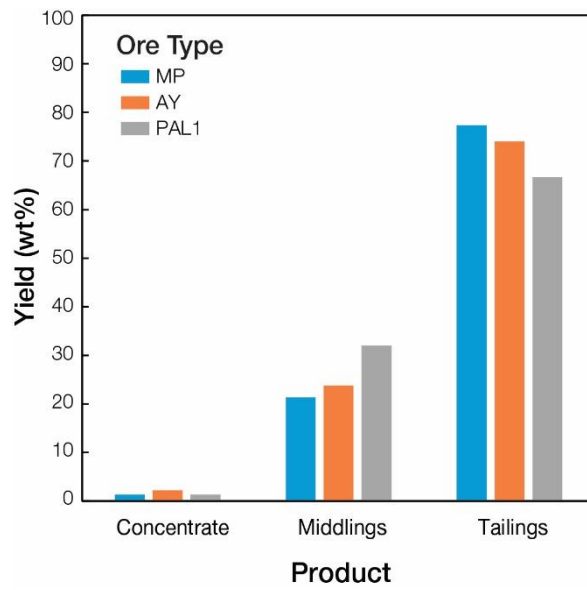


Figure 16. Mass balance between the gravity concentration products for the 3 ore types.

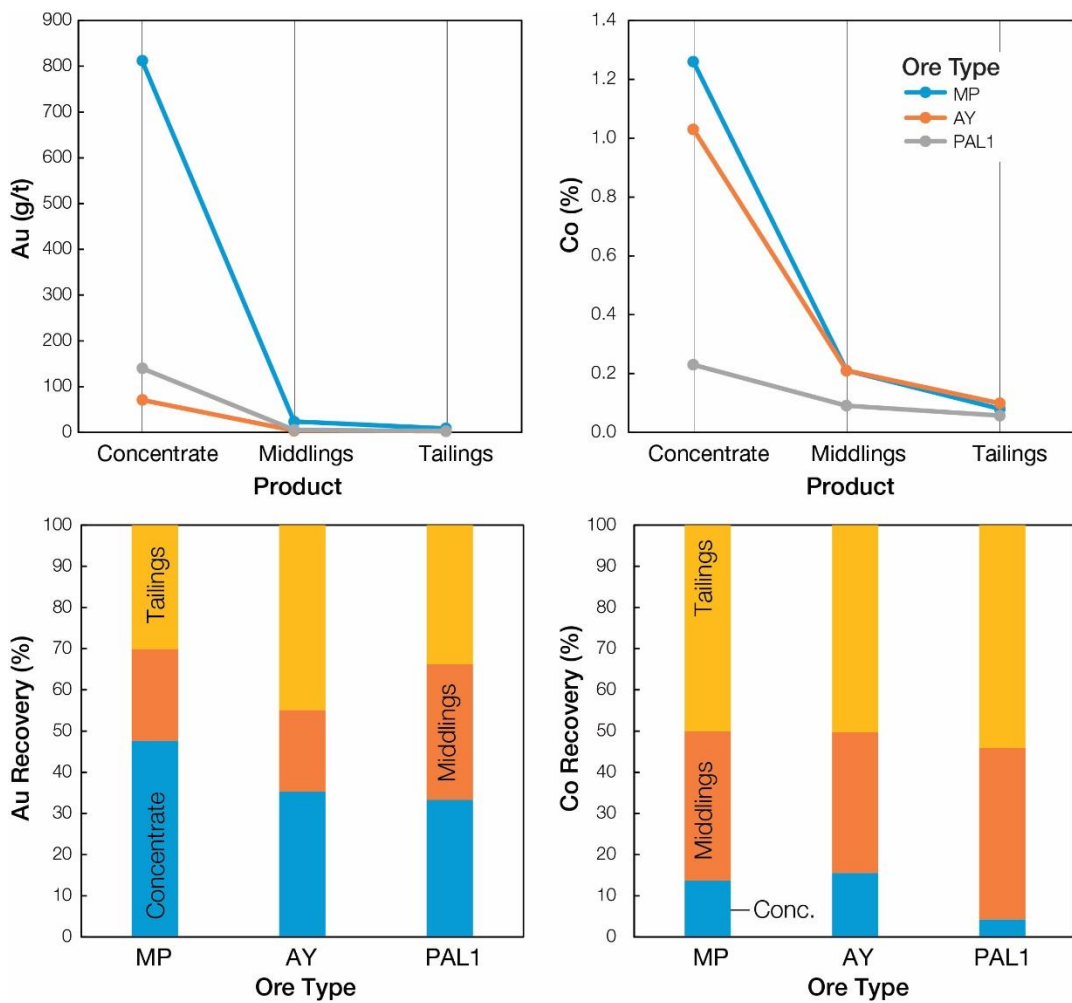


Figure 17. Target elements (Au, Co) grade and recovery in the gravity concentration products for the 3 ore types.

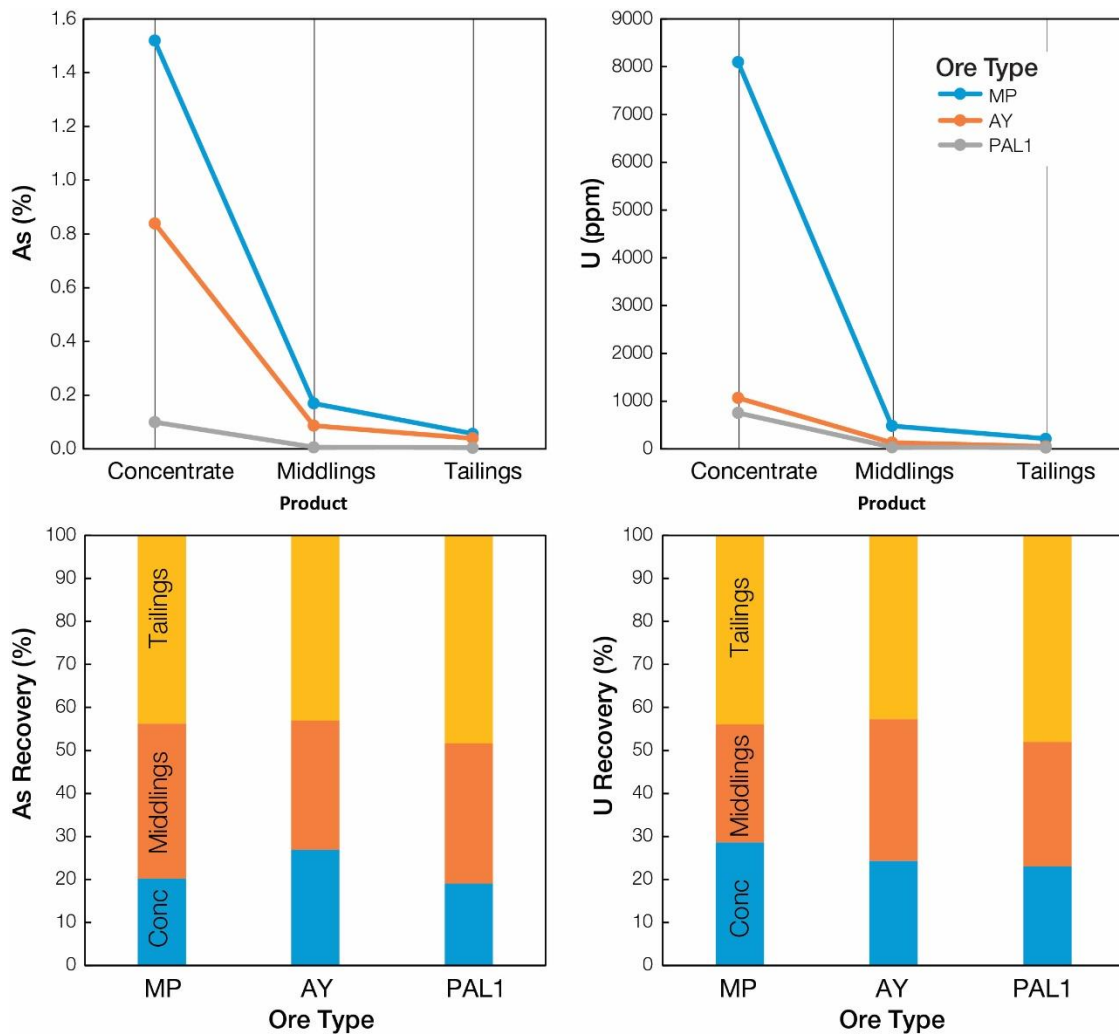


Figure 18. Penalty elements (As, U) grade and recovery in the gravity concentration products for the 3 ore types.

The bulk mineralogy of the gravity concentration products for each ore type obtained through quantitative XRD is shown in Figure 19. The gravity concentrates are mostly composed of pyrrhotite and, to a lesser extent, pyrite, amphibole, chlorite, magnetite, ilmenite, and cobaltite. The proportion of each mineral varies depending on the product and the ore type but globally the gravity concentrate is mostly composed of pyrrhotite (between 55-78 wt.%). Mineral recoveries for the gangue minerals considered problematic are shown in Figure 20. In terms of recovery, most of these minerals are reporting to the middlings or tailings for all the ore types.

Cobalt minerals recovery, as for cobalt recovery, is very limited with most of the cobalt recovery in the concentrate attributed to cobaltite and, to a lesser extent, linnaeite locked in pyrrhotite for all ore types (Figure 21). However, in terms of grade, cobaltite concentration of up to 3.5 % cobaltite is reach with the MP ore type, followed by the AY ore type with 2 %cobaltite and finally the PAL1 ore type with 0.2 % cobaltite in the concentrate (Figure 21).

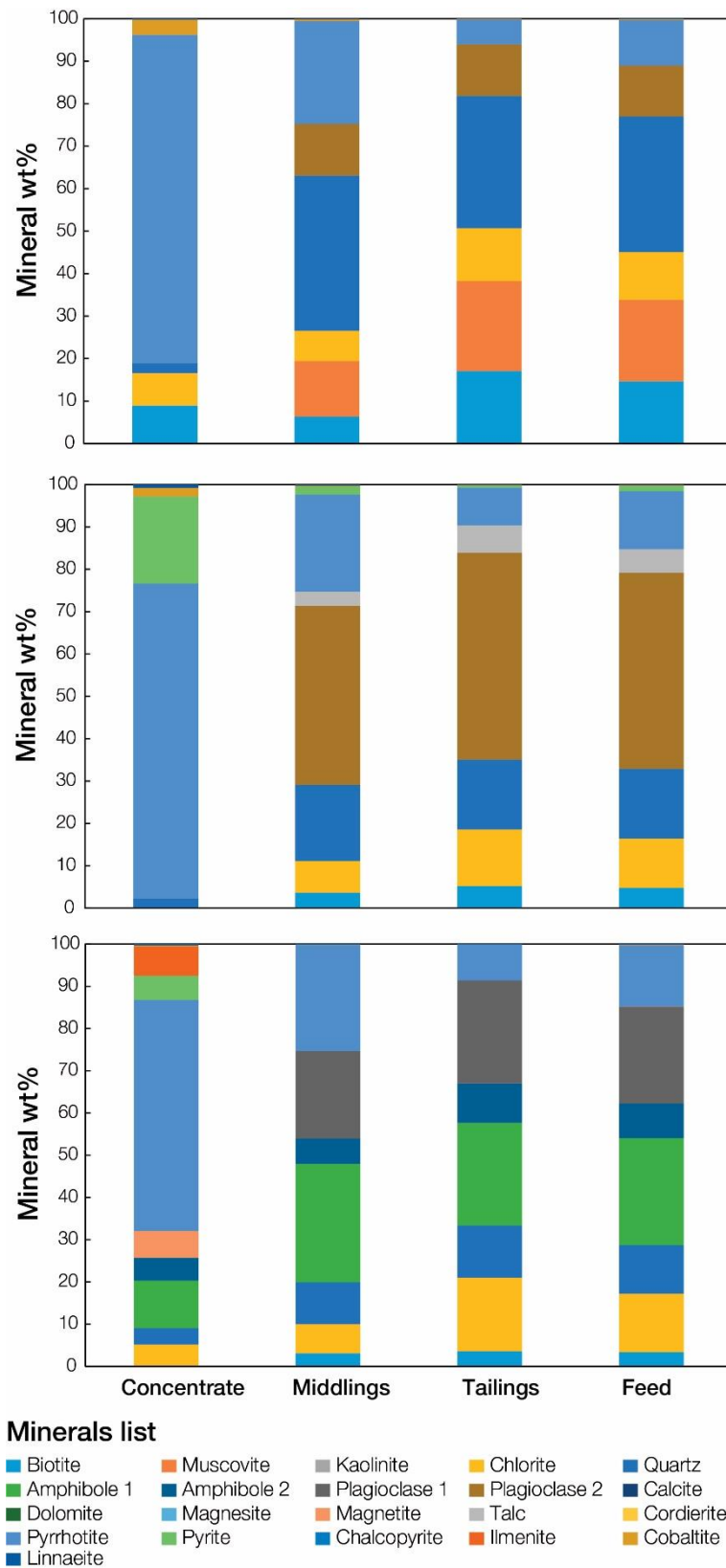


Figure 19. Quantitative mineralogy (QXRD) of the gravity concentration products for the 3 ore types: MP (top), AY (middle) and PAL1 (bottom).

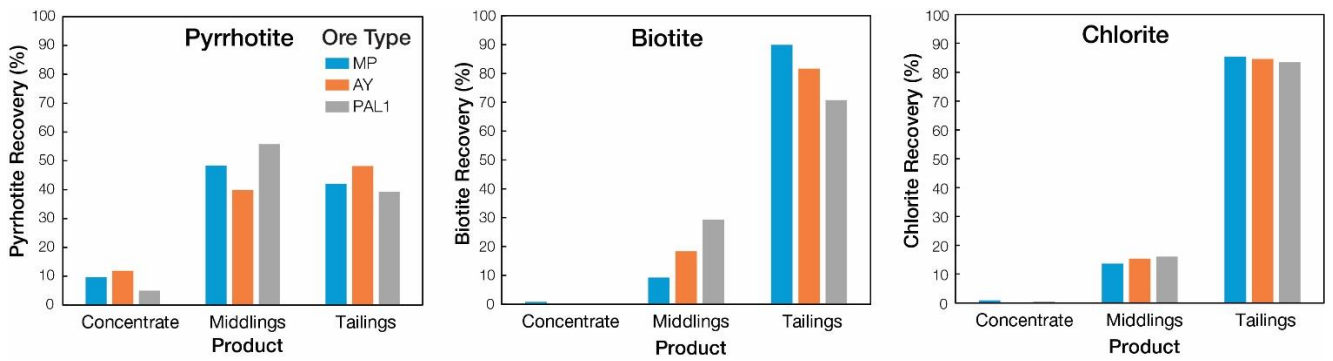


Figure 20. Problematic gangue minerals recovery in the gravity concentration products for the 3 ore types. Biotite content below the detection limit for MP and PAL1 ore type in the concentrate.

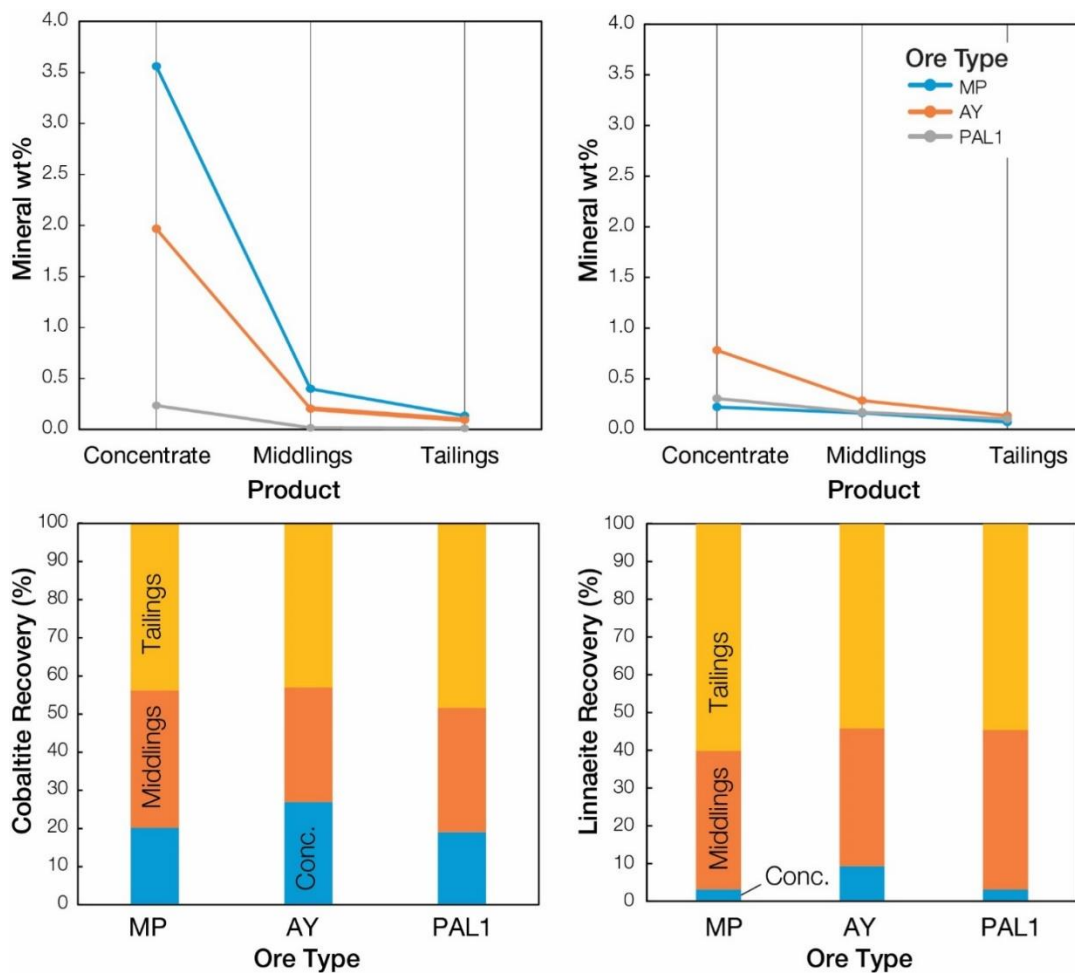


Figure 21. Cobaltite (left) and linnaeite (right) grade (top) and recovery (bottom) in the gravity concentration products for the 3 ore types. Note that these results are obtained using chemical assays and minerals stoichiometry (see section 3.5).

6 FLOTATION

6.1 Methods/Procedure

Flotation tests on the 3 ore types have been conducted at GTK Mintec pilot plant laboratory in Outokumpu, Finland. A similar sample preparation procedure was applied to each ore type including a grinding the 2 kg samples in a rod mill with 1 L of water and a rod charge of 8 kg to reach a D80 of 75 μm . The flotation tests were carried out on the ground mixed ore sample using a Outotec-GTK LabCell™. For each experiment, the dry sample was mixed into the cell with water at 2100 rpm to obtain ~30 wt% solid density.

The rougher flotation was done in three stages in a 7 L cell and the concentrates were combined and sent to a cleaning stage in a 2.5 L cell during which concentrates were collected at different time interval to obtain study the flotation kinetics. Air flow rate was constant at the rougher and cleaner stages 5 and 2 L/min, respectively. During the whole experiment, pH was measured with a Metrohm 877 Titrino Plus pH meter. An overview of the flotation procedure with the timing followed for each of the five flowsheets can be found in Table 2.

Sodium Ethyl Xanthate (SEX) was used as the main collector, with a DanaFloat 245 (DF245) supplied by Cheminova A/S (Harbøre, Denmark) used as a co-collector. The latter mainly consists of sodium O,O-diisobutyl dithiophosphate and has previously been successfully used as a collector for bulk copper-cobalt flotation (Tijsseling et al., 2020, 2019). Methyl Isobutyl Carbinol (MIBC) was used as a frother in all the experiments. Technical-quality sodium silicate (Na_2SiO_3), Zeopol 33 (Huber), was used to depress silicate minerals.

Table 2. Flotation procedure for the experiments described in this study.

Stage	Duration (min)	Reagents (g/t)					Air Flow (L/min)	Rotor (rpm)	pH
		DF245	SEX	CaOH ₂	Na ₂ SiO ₃	MIBC			
Conditioning - Water							2100	8.7	
Conditioning - Sodium silicate	3				250		"	9	
Conditioning - collector & frother	2	10	20			20	"	9	
Rougher flotation 1	4						5	8.8	
Conditioning - collector & frother	2	10	20			10	"	8.8	
Rougher flotation 2	4						5	8.8	
Conditioning - collector & frother	2	10	30			10	"	"	
Rougher flotation 3	4						5	8.8	
Lime addition				15			1500	9	
Conditioning - collector & frother	2	10	10			10	"		
Cleaner flotation 1	1						2	"	
Cleaner flotation 2	2						2	"	
Cleaner flotation 3	2						2	"	
Cleaner flotation 4	3						2	8.6	

6.2 Preliminary Results

Two distinct flotation routes were first tested on a blend ore (mixture of leftover half-core samples from the 3 ore-types) before the actual experiment on the 3 ore types. The first route targeting a bulk gold-cobalt recovery used the reagent scheme described above while the second route, based on a US

patent (Moyer, 1948), is targeting specifically cobaltite through the use copper sulphate (CuSO_4) provided by J.T. Baker, as an activator (Figure 22).

The first bulk flotation test (test 11) yielded a concentrate at 142 g/t gold and 1.44 %Co with recoveries of 90% and 39% for gold and cobalt, respectively. In comparison, the second bulk flotation test (test 12) with copper sulphate as an activator yielded concentrate at 13 g/t gold and 0.62 % Co with recoveries of 74% and 66% for gold and cobalt respectively. Thus, while the addition of copper sulphate allowed to increase cobalt recovery from 39% to 66%, gold recovery dropped from 90% to 74%. Even if the feed material used for this second test had a 3-times lower gold content, the enrichment obtained with this second test still much below the one obtained for the first test both for gold and cobalt. For this reason, the first flotation route was selected for the standard flotation test applied to the 3 ore types.

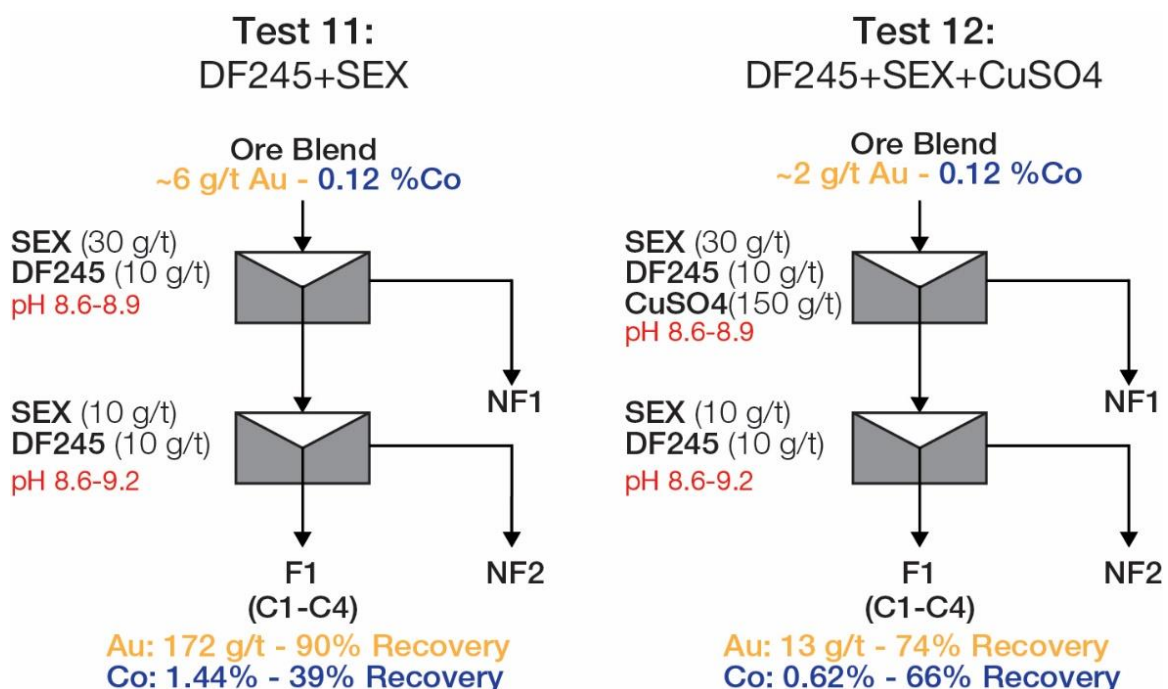


Figure 22. Overview of the two flotation procedures tested on the ore blend with flowsheets, operating conditions and results for gold and cobalt.

6.3 Results

Flotation results in terms of kinetics, grade and recovery are given in Figure 23 and Figure 24 respectively. Gold flotation kinetics vary depending on the ore type with PAL1 ore showing the fastest kinetic and highest recovery rate at 90 % gold recovery, MP ore showing a much slower gold flotation kinetics with a final gold recovery of 82%, while an intermediate gold flotation kinetic is observed for the AY ore with a final gold recovery at 76%. Cobalt flotation kinetics appear to be relatively slow for every ore type with however varying maximum cobalt recovery depending on the ore type. Indeed, maximum cobalt recovery of 23%, 46% and 63% are obtained for the PAL1, AY and MP ore types, respectively. Overall, the flotation seems to be relatively selective with regards to uranium which

displays maximum recoveries below 10% for every ore type. Flotation kinetics for arsenic, which reflects the behaviour of cobaltite, are quite fast for all type, except for the PAL1 ore for which linnaeite is the dominant cobalt mineral.

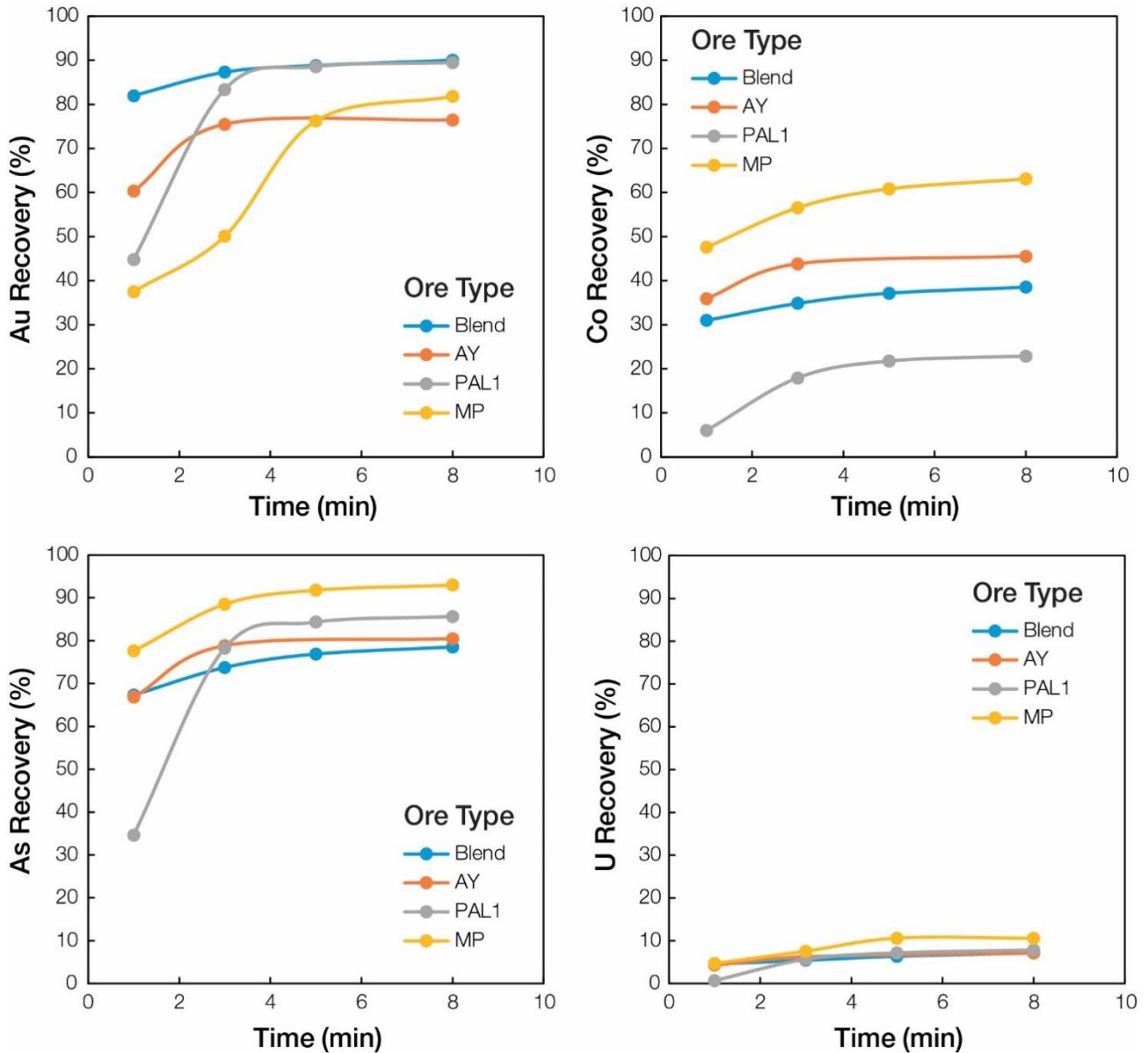


Figure 23. Flotation kinetic curves for gold (top left), cobalt (top right), arsenic (bottom left) and uranium (bottom right).

The distinct process behaviour for gold and cobalt can also be observed in the grade-recovery graphs (Figure 24). Indeed, each ore type yield a clear distinct performance in terms of recovery for both gold and cobalt. In terms of grade, the best results are obtained with PAL1 for gold (436 g/t Au) and with MP for cobalt (2.9 % Co).

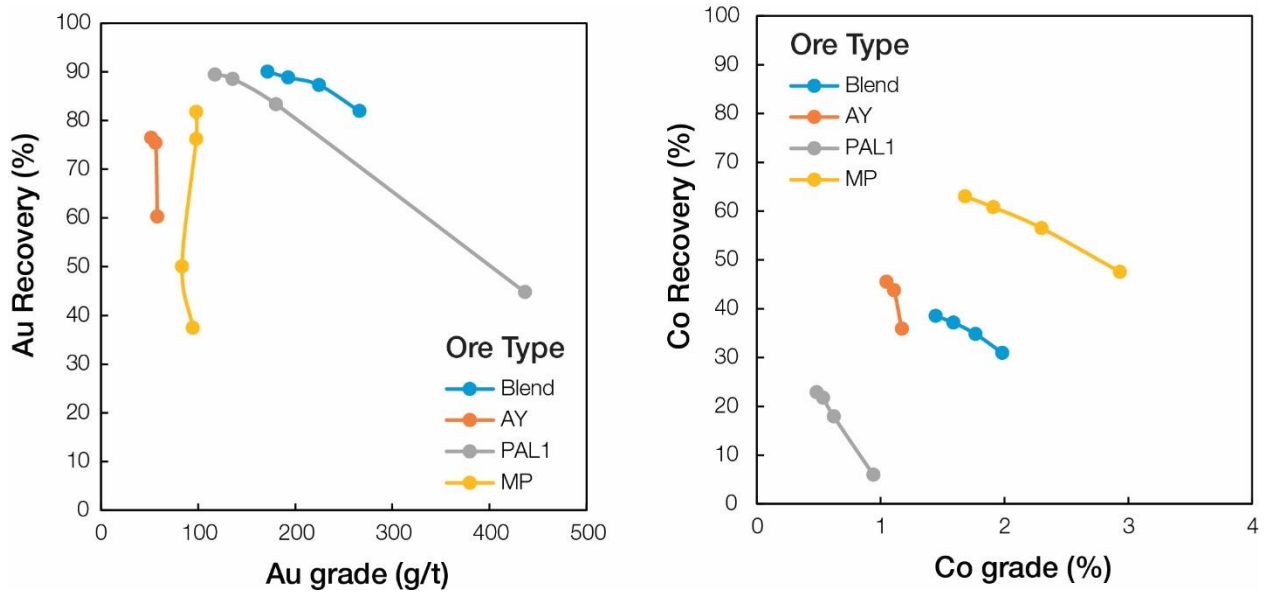


Figure 24. Grade vs recovery curves for gold (left) and cobalt (right).

Figure 25 shows the kinetic curves for cobaltite and linnaeite. Results suggest that most of the cobalt recovery observed in Figure 23 can be attributed to cobaltite which displays maximum recoveries between 78%-93% while linnaeite recovery is limited (18%-27%). This can be explained by the differences in the degree of liberation of these two minerals, as linnaeite is mostly locked in pyrrhotite.

While cobalt recovery may seem relatively low, these results suggest that the recovery of the available cobalt minerals (e.g., cobaltite), as opposed to the non-available cobalt minerals (e.g., linnaeite), is relatively good.

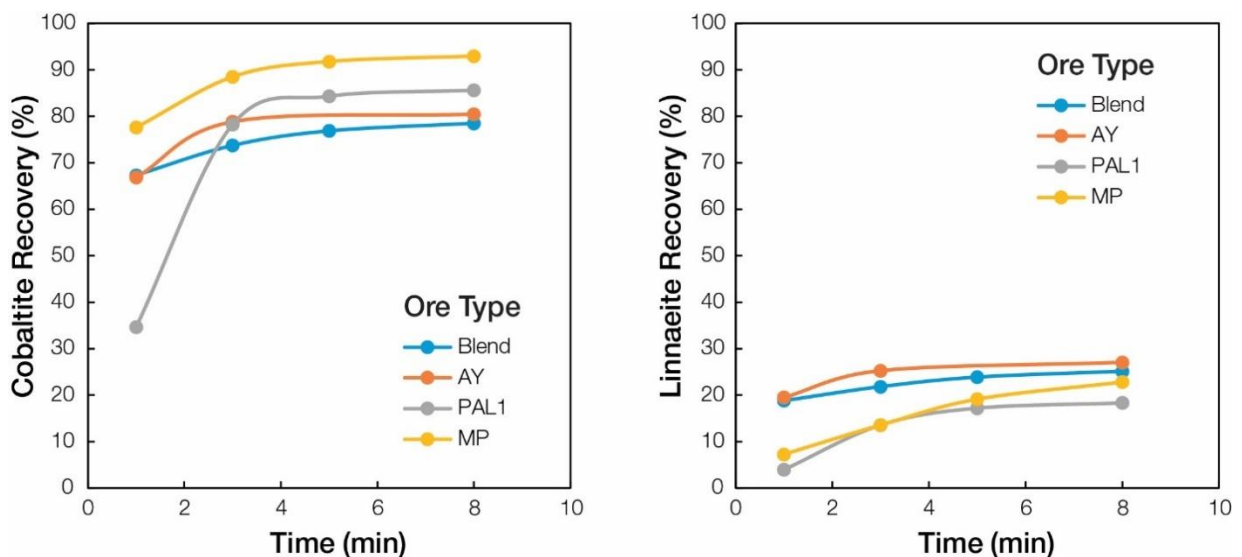


Figure 25. Kinetic curves for cobaltite (left) and linnaeite (right). Note that these results are obtained using chemical assays and minerals stoichiometry (see section 3.5).

Due to low sample mass of some concentrates, no additional QXRD could be conducted, but the chemistry suggests that the concentrates are mostly composed of pyrrhotite, amphiboles and potentially micas and chlorite.

7 LEACHING

7.1 Methods/Procedure

Each of the three Rajapalot Orientation Samples were crushed (99% passing 3.35mm) and subdivided into sub-samples. One of those sub-samples was prepared for leaching response (approximately 3kg each). These three sub-samples for leaching response were ground to a D_{80} of 75 microns in a rod mill at GTK Mintec, and samples were packed in nitrogen to reduce oxidization. The resulting samples were sent to Outotec for leaching process separation. These samples were then further sub-divided representatively for the following characterization and test work. Each of the three Orientation Sample sub-samples were subject to two leaching tests (totalling 6 tests).

1. Direct cyanidation leaching
2. Cyanide leaching with activated carbon (Carbon in Leach, CIL)

Cyanide leaching tests were done in a 2 L agitated reactor using high 2-3 g/L NaCN concentration and air feed for 24 hours at ambient temperature. Two hours pre-oxidization by air was done before starting cyanide leaching. Tests were done at a pH of 11-12. It should be emphasized that the purpose of the conducted tests was only to show the potential for gold and silver extraction. Detailed parametrization of the cyanide leaching conditions requires additional cyanide leaching tests with the selected parameter variation.

Preliminary cyanide leaching tests were done. The purpose of the preliminary tests was to determine the cyanide leachable gold and silver. The preliminary tests do not allow determining cyanide leaching in detail. Cyanide leaching tests were done in a 2 L agitated glass reactor. The aqueous slurry pH was increased to 11-12 and air was fed (0.5 l/min) for two hours. After pre-aeration cyanidation was started. In the tests with activated carbon, carbon was added before cyanide addition. Air was fed under the impeller during the cyanidation. Initial solid content varied 32-37% w/w in the tests being 31-36% at the completion end.

Sodium cyanide was added to maintain its concentration between 2.2-3.5 g/L. The total cyanidation time was 24 hours in all tests. Liquid samples were taken during the test. The final slurry was filtrated, cake washed with distilled water, dried, and subjected to following analyses. In the tests with activated carbon, carbon was sieved out from the rest of the washed solids and dried separately.

- Liquid
 - During the test: Au, Ag
 - At the end of cyanidation: Ag, Au, Ca, Co, Cu, Fe, Ni, Zn
- Solid
 - At the end of cyanidation: Ag, Au, C (total)
- Activated Carbon
 - At the end of cyanidation: Ag, Au

7.2 Results

The test log sheets are shown in Appendix 13.7. Dissolved oxygen was on the saturated level during the leaching in all tests. Pre-aeration was used in the tests. Based on the results it should be considered if pre-aeration was able to leave out when designing the industrial processing. The following results were recorded.

Table 3. Realized parameters in the pressure leaching tests.

Test	Sample	Activated carbon	Ca(OH) ₂ addition		NaCN Consumption		Solids	Solids in slurry
			Initial	During test	Initial	During test		Initial
			kg/t(solid)		kg/t(solid)			g
1	MRC-AY	No	0.6	3.1	7.4	1.24	603	31.7
2	MRC-MP	No	0.7	2.3	6.0	1.18	750	36.6
3	MPC-PAL1	No	0.6	2.6	7.5	1.04	601	31.9
4 (CIL)	MRC-AY	Yes	0.5	0.3	6.0	1.84	745	33.6
5 (CIL)	MRC-MP	Yes	0.7	0.1	6.2	1.22	726	32.6
6 (CIL)	MPC-PAL1	Yes	0.8	0.6	6.0	1.35	744	33.5

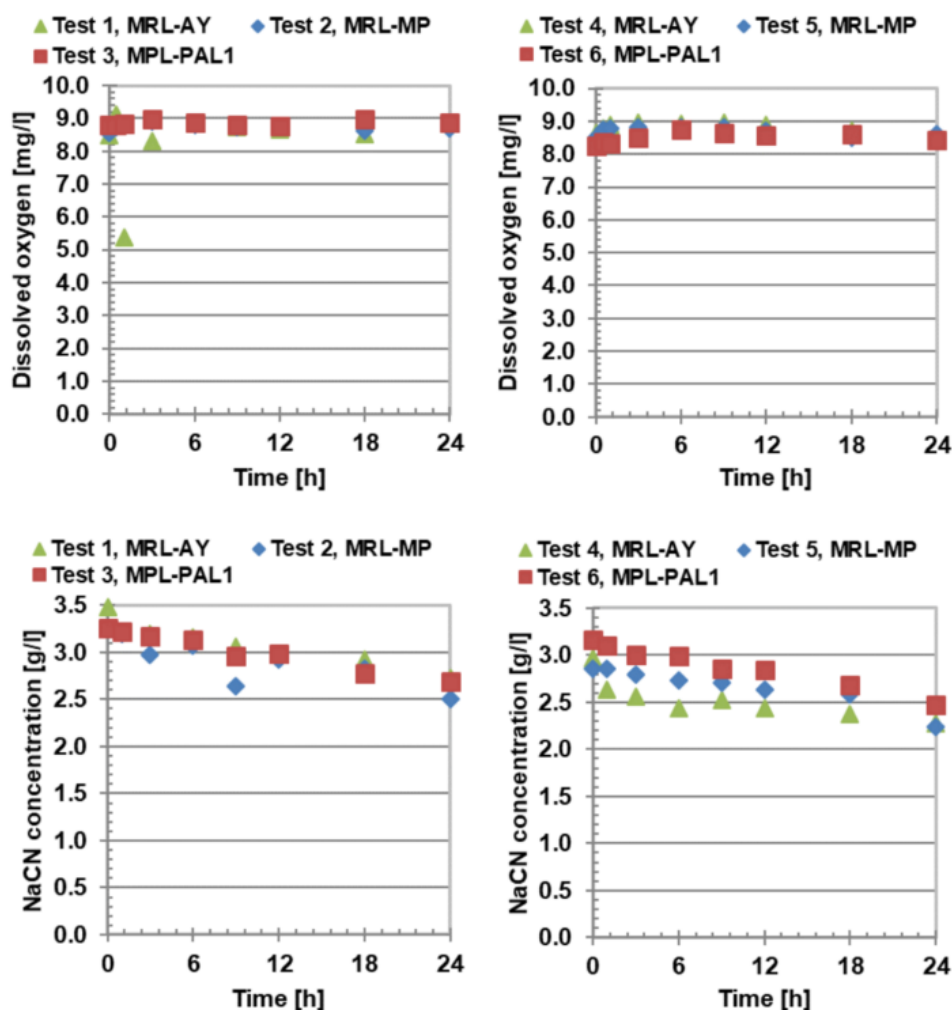


Figure 26. Dissolved oxygen and NaCN profiles in the cyanide leaching tests.

There was no attempt to optimize lime consumption. In spite of that, lime consumption was at low or moderate levels in all tests. The tests were done under high pH conditions, 11.0 - 11.8, which increase

lime consumption. Somewhat lower lime consumption should be possible at lower pH, for example 10.0 - 10.5.

There was also no attempt to optimize or minimize cyanide consumption, but relatively high cyanide concentration was used in the tests to ensure effective leaching. It should be noted that lower cyanide level may somewhat decrease the leaching kinetics. NaCN levels remained on high levels, 2.2 – 3.5 g/l, in all tests. Typical industrial scale NaCN concentration values are closer to 0.1 – 0.5 g/l at the end of the process circuit. Despite relatively high NaCN levels, the realised sodium cyanide consumptions during the leaching tests were low varying between 1 – 2 kg/t (solid) in tests.

All slurries were pre-aerated two hours before cyanide leaching. The purpose of the cyanide leaching was to determine especially cyanide leachable gold and also silver fraction, not to determine cyanide leaching parameters for industrial processing. The effect of the activated carbon on gold extraction was tested by making tests with and without activated carbon. All analysis results are shown in Appendix 13.7.

Gold and silver extractions are shown in Table 4. It can be seen from the results that excellent gold extractions can be achieved with all samples in direct cyanidation. The gold extraction values were over 97% for each sample. Thus, there was no sign of pre-robbing for any of the samples. In addition, dissolved gold levels up during 1-3 as shown in Figure 27, seems to level up with MRL-MP (test 2) but shows slight increase still after 24 hours for other samples. However, the slope of the increase is small and only minor increase in extraction can be expected with longer retention times.

Surprisingly, using activated carbon decreased gold extractions showing values between 75.5 – 89.2% depending on the sample. It would have been expected to achieve similar extractions with carbon as without, but instead, passivation / decreasing kinetics took place with carbon. In case of further developing cyanide leaching, the use of activated carbon should be carefully studied. However, gold was absorbed fully onto carbon, except in the test 5, MRL-MP, low dissolved residue of 0.02 mg/l was realized. Initial evaluation suggests using Carbon in Pulp (CIP) type process instead of Carbon in Leach (CIL).

Table 4. Au and Ag leaching extraction using back calculated head grades

Test	Sample	Au rec %	Ag rec %
1	MRL-AY	97.3	23.1
2	MRL-MP	97.7	24.7
3	MPL-PAL1	98.0	17.4
4 (CIL)	MRL-AY	85.3	18.1
5 (CIL)	MRL-MP	75.5	13.5
6 (CIL)	MPL-PAL1	89.2	13.7

Silver extractions showed low values in all tests being between 17.4 – 24.7% in direct cyanidation and, analogously to gold, even lower with activated carbon. Dissolved Ag during the tests 1-3, as shown in

Figure 27, seems to level up with MRL-MP (test 2) but shows slight increase still after 24 hours for other samples.

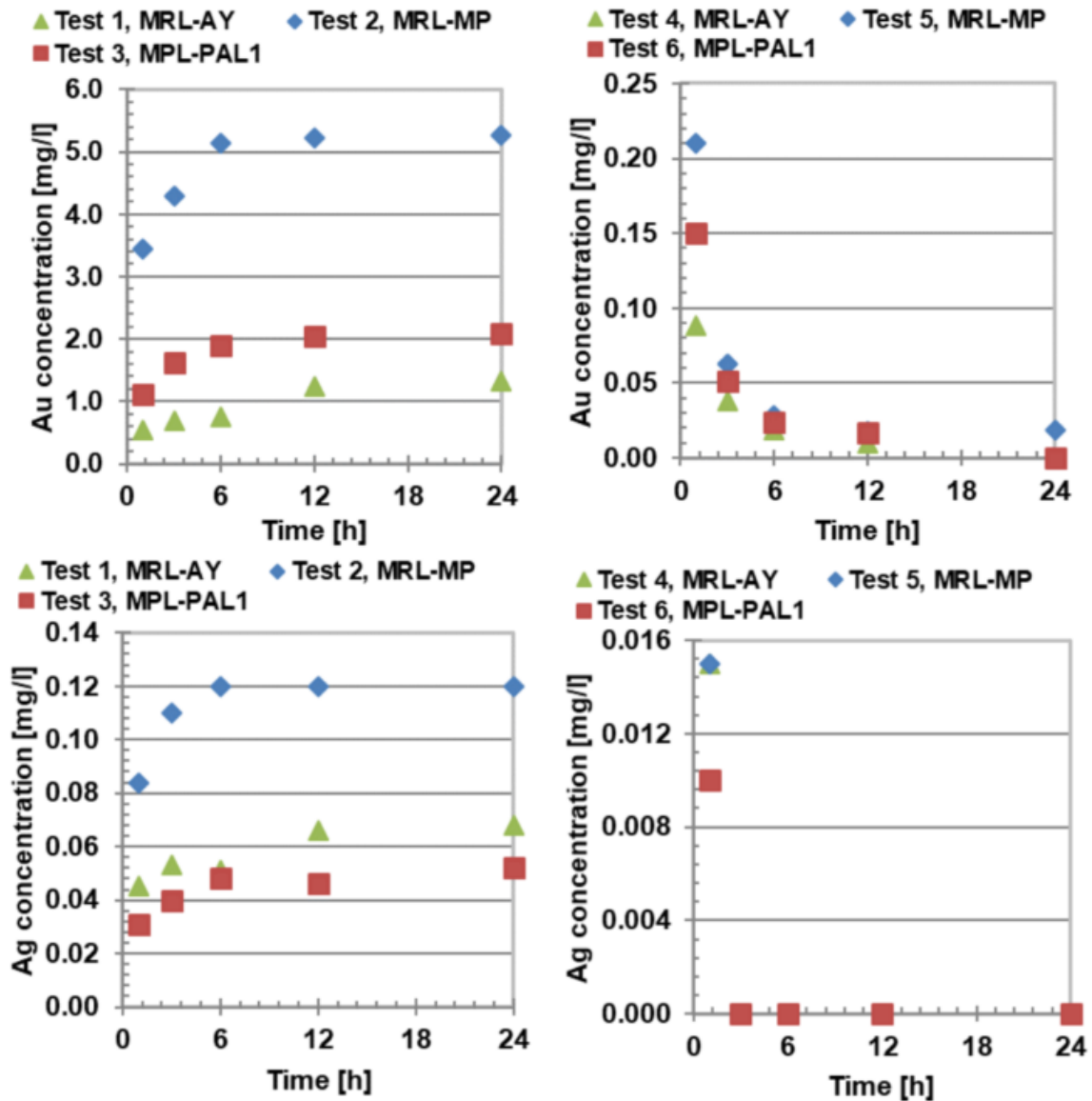


Figure 27. Au and Ag concentrations in solution during the direct cyanidation leaching tests (left) and CIL tests (right).

Cyanide leaching was tested for three samples, MRL-AY, MRL-MP and MRL-PAL1, and it is apparent that additional test work is required to determine design parameters for the cyanide leaching process. At least, maximal solid content, minimal cyanide level and carbon adsorption characteristics with CIP/CIL option need to be determined. The tests should be carried out using the solution composition expected to be used at the plant.

Final dissolved concentrations for other elements analysed are shown in Table 5. Leaching of other target elements is relatively low. For example, Co extraction was lower than 0.5% for all samples. Cu levels were relatively high compared to Au, though low in absolute measures. Cu in solution adsorbs

onto activated carbon though not as selectively as Au, but Cu adsorption needs to be considered in future testing and design of the process.

Table 5. Au and Ag leaching extractions using back calculated head grades.

Test	Sample	Al mg/l	Ca mg/l	Co mg/l	Cu mg/l	Fe mg/l	Ni mg/l	Zn mg/l
1	MRL-AY	2.3	2.3	2.6	7.8	63	0.5	<0.5
2	MRL-MP	3.1	2.9	3.7	15	85	0.9	0.8
3	MPL-PAL1	5.7	6.1	<0.5	13	70	0.5	<0.5
4 (CIL)	MRL-AY	2.0	3.0	3.0	12	57	<1.0	<1.0
5 (CIL)	MRL-MP	5.0	3.0	3.0	17	52	<1.0	1.0
6 (CIL)	MPL-PAL1	8.0	3.0	<2.0	20	73	<2.0	<2.0

8 DISCUSSIONS

8.1 Comparison of the individual methods

Figure 28 summarise the performance of all mineral processing techniques tested for the 3 ore types, in terms of concentrate and recovery for gold, cobalt, cobaltite and linnaeite. While all the test work described in this study has been conducted with constant operating conditions for the 3 ore types to see how the process performance varies between the ore types, these conditions were selected based the operators experience and prior knowledge about the samples. This mean that these operating conditions are not optimal and may be optimised through a dedicated follow-up test work program. This must be kept in mind when interpreting the results.

Flotation appears to be the most efficient technique for gold recovery with recovery rates above 90% and concentrate grades above 100 g/t with PAL1. Magnetic and gravity concentration only yielded very low recoveries below 50% for all ore types. While this was expected for magnetic separation, the results obtained with gravity concentration may seem disappointing. Beyond the sub-optimal operation conditions used for the test as mentioned above, this may be explained by an incomplete liberation of the gold. The fact that between 30-40% of the gold reported in the magnetic fractions may be, for instance, explained by the presence of non-liberated gold particles associated with pyrrhotite.

Similar cobalt recoveries have been obtained with magnetic separation and flotation with, however, higher concentrate grades for the latter, with grades above 1% Co. This is a result of a mineralogical control over cobalt recovery (see section below) whereby cobalt recovery in the magnetic fraction can be accounted for linnaeite locked in pyrrhotite while flotation is selectively floating cobaltite.

As for the penalty elements, arsenic being almost exclusively hosted in cobaltite, it will follow the later in the flotation concentrate. While uranium, tends to follow the same pattern as gold during magnetic and gravity concentration, it is not recovered during flotation, making flotation the ideal method to selectively recover gold and cobaltite against uraninite.

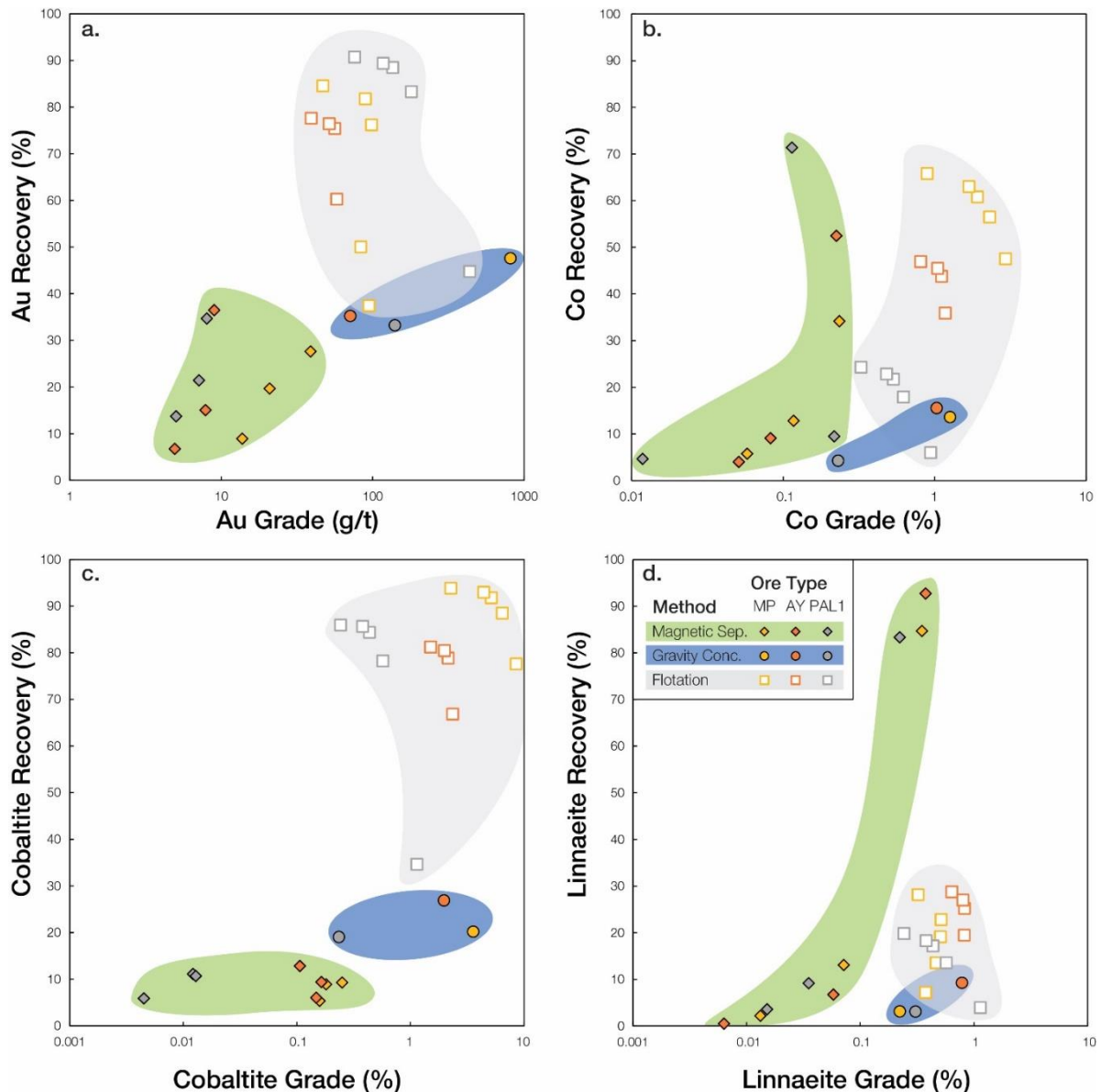


Figure 28. Recovery vs grade graph summarizing the performance of all mineral processing methods for the 3 ore types. (a) Gold, (b) Cobalt, (c) Cobaltite and, (d) Linnaeite. Note that these results are obtained using chemical assays and minerals stoichiometry (see section 3.5).

8.2 Ore types and Mineralogical control

The results described in this study clearly highlight the different process behaviours of the 3 ore types considered, therefore confirming that these can be considered as individual geometallurgical ore types. Roughly speaking, typical process performance in terms of grade and recovery ranks in the order PAL1>MP>AY for gold and MP>AY>PAL1 for cobalt. For the latter, the opposite is observed, however, with magnetic separation due to the mineralogical control explained below.

Indeed, the mineral processing tests results obtained in this study clearly highlight a mineralogical control over cobalt recovery. In particular, the department of cobalt between cobaltite and linnaeite seems to also explain, to some extent, the differences in cobalt recovery between ore types. In the

case of flotation for instance, the majority of the cobalt recovered in the concentrate is due to cobaltite which is largely recovered while linnaeite recovery remains limited (Figure 29, left). This also explains the better cobalt recovery obtained with the MP and AY ore types for which most of the cobalt is hosted in cobaltite. Consequently, the amount of cobaltite in the ore, or the proportion of cobalt deported in cobaltite, seems to control the overall cobalt recovery for every test conducted on the 3 ore types (Figure 29, right). The more cobaltite in the feed ore, the higher the cobalt recovery is with flotation and gravity concentration. The opposite is observed with magnetic separation because cobaltite being non-magnetic, it is the linnaeite locked in the pyrrhotite which is recovered (with high recovery rates for pyrrhotite). This trend will have to be confirmed by further test work.

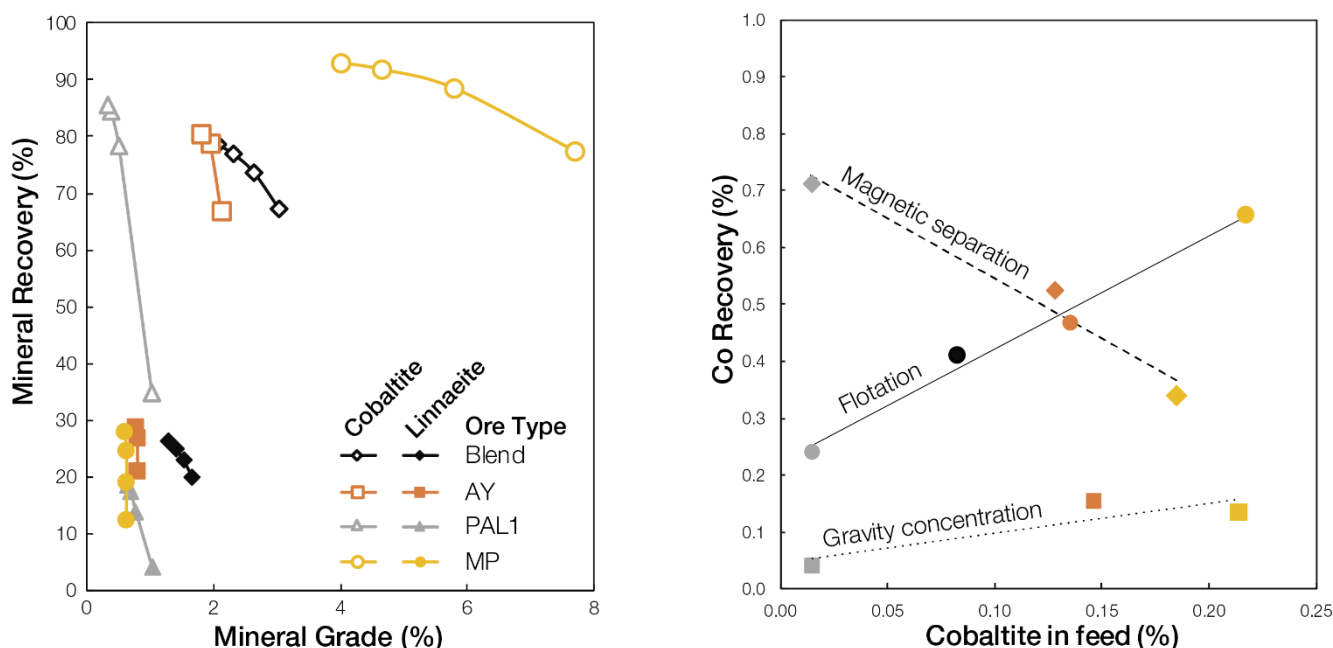


Figure 29. Illustration of the mineralogical control of cobaltite over cobalt recovery. (Left) Mineral grade vs recovery during flotation and (right) influence of the amount of cobaltite in the feed ore on cobalt recovery for various methods. Note that these results are obtained using chemical assays and minerals stoichiometry (see section 3.5).

8.3 Considerations for future test work and flowsheet development

While the test work results described in this study does not allow for the establishment of a definitive flowsheet for the Rajapalot Au-Co project, it provides grounds for future work and the next stage of the geometallurgical campaign whereby different process paths are tested.

Given the relatively good results obtained with flotation, a bulk Au-Co flotation process appears to be the most adapted process path. A simplified flowsheet proposal for this option is given in Figure 30. However, it may be of interest to remove most of pyrrhotite early in the process in a separate pyrrhotite concentrate. By doing so, the amount of sulphide in the ore will be significantly reduced prior to flotation as pyrrhotite can make up to 12 wt.% of the ore (Dehaine et al., 2021a). This could improve flotation efficiency and reduce reagents consumption. In addition, it would allow to recover most of the linnaeite which is locked in pyrrhotite in a separate stream that could potentially be further processed to recover the cobalt. A simplified flowsheet proposal for this option is given in Figure 31.

**Rajapalot Au-Co Project
Flotation-only Flowsheet Proposal**

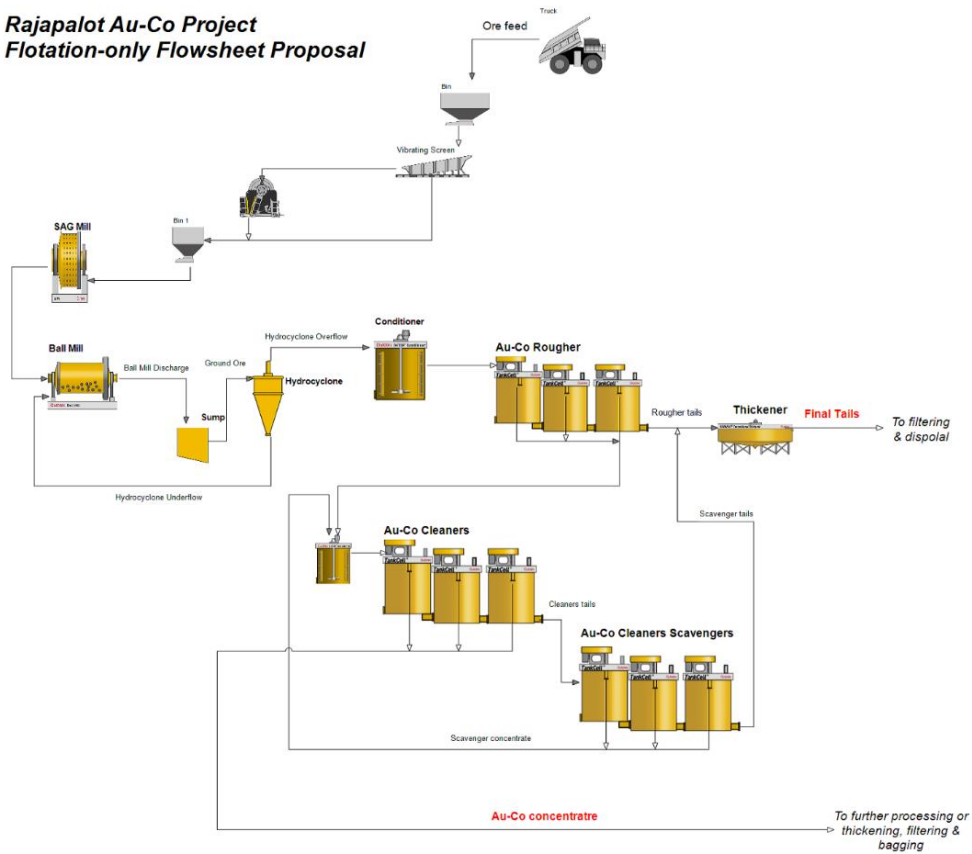


Figure 30. Simplified flotation-only flowsheet proposal for the Rajapalot Au-Co project.

**Rajapalot Au-Co Project Magnetic &
Flotation Flowsheet Proposal**

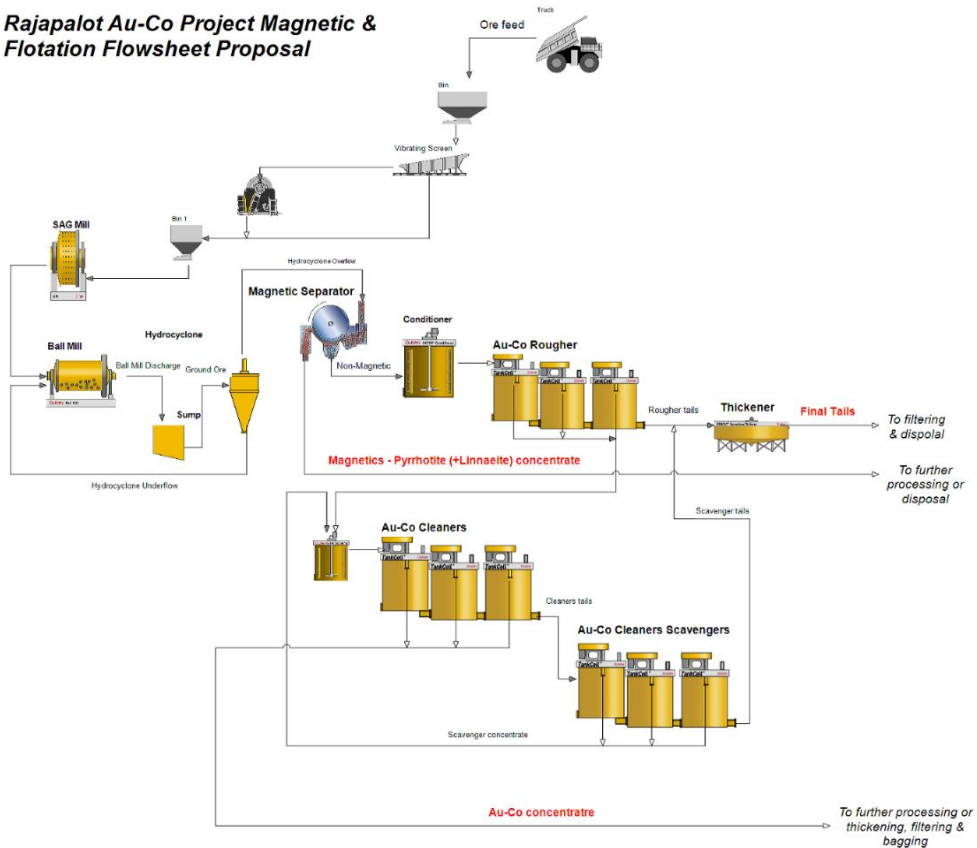


Figure 31. Simplified magnetic and flotation flowsheet proposal for the Rajapalot Au-Co project.

Based on the current results, a possible process path for the Rajapalot Au-Co project would include the following stages:

1. **Crushing & grinding** to liberate gold and cobaltite,
2. (Optional) **Magnetic separation** to recover pyrrhotite and linnaeite in a separate concentrate,
3. **Bulk Au-Co flotation** with at least a roughing and cleaning stages to recover gold and cobaltite in a single concentrate and remove uraninite,

The design and choice of the following stages are beyond the scope of this study and will depend on the decision whether or not the refining stages will happen on-site, if gold and cobalt are to be sold in a single or separate product, etc. Each possibility will lead to different products:

- a. After thickening, filtering and bagging, the Au-Co concentrate could be sold as a **Au-Co mixed product**,
- b. After an additional gold-cobalt separation stage, possibly by flotation, to obtain **separate gold and cobalt concentrates** to be sold as such or further refined,
- c. The thus obtained gold concentrate could then go through cyanidation and smelting to produce **gold doré** while the cobalt concentrate could as well undergo further hydrometallurgical treatment (leaching, solvent extraction, purification) to produce **cobalt sulphate**.

9 CONCLUSIONS

The metallurgical testwork described in this report have been conducted as part of the geometallurgical orientation study for the Mawson Gold Rajapalot Au-Co project. In terms of process performance, based on the standard tests performed, the following conclusions can be made:

- Magnetic separation can be used to selectively recover the pyrrhotite (up to 90% recovery) at relatively low amperage (LIMS equivalent) and with it the cobalt content associated with the locked linnaeite. Some gold may be, however, recovered in the magnetic fraction as well if not previously liberated.
- Gravity concentration yielded relatively poor performance both for gold and cobalt with recoveries below 44% for gold and 20% for cobalt. This again, may be due too poor liberation of gold and other target sulphides.
- Flotation is the most effective separation process to recover Au and Co associated with cobaltite. Results show recovery rates and concentrate grades above 90% and 100 g/t for gold and between 23-63% and above 1% for cobalt with, however, recovery rates between 78%-93% for cobaltite
- Only cyanide leaching for gold has been tested, yielding high extraction rates between 97.3 - 98%. Initial evaluation suggests using Carbon in Pulp (CIP) type process instead of Carbon in Leach (CIL).

In terms of process behaviour of the 3 ore types, the following conclusions can be made:

- Results confirmed the distinct gangue mineralogy between the ore types as well as the distinct cobalt deportment (cobaltite vs linnaeite),
- Each ore type showed a distinct process behaviour, in particular for cobalt, with clear distinct behaviour between PAL1 and (AY+MP) ore types,
- Consequently, these 3 ore types can be considered as individual geometallurgical ore types.

In terms of mineralogical control, the following conclusions can be made:

- There is a clear mineralogical control over cobalt recovery,
- This control is determined by the amount of cobaltite (or the proportion of cobalt therein) in the ore,
- This can be readily integrated into the 3D model using assays data as a proxy for the mineralogy (*e.g.*, arsenic can be used as a proxy for cobaltite).

10 FUTURE WORK

More work is required to complete this study. The following tasks are recommended to be done.

- The planned leaching process separation tests should also be carried out to recover cobalt from the prepared Orientation Sample sub-samples that have only been tested for cyanide leaching of gold. This would complete the Geometallurgy campaign Stage 1. For each sample, additional leaching tests should be considered and planned for the recovery of cobalt and other base metals (Ni, Cu). This may include nitric acid leaching or even nitrogen species catalysed (NSC) acid pressure leaching.
- Geometallurgy campaign Stage 2 should then be undertaken. It is recommended that the two aforementioned process paths (Figure 30 and Figure 31) should be tested, and each process optimized based on what has been learned so far.

11 ACKNOWLEDGMENTS

This research has been undertaken as part of the Finland-based circular ecosystem of battery metals consortium (BATCircle) project [Grant No. 4853/31/2018] funded by Business Finland (website: <https://batcircle.aalto.fi/en>). The authors wish to thank Walter Rivera from University of Liège in Belgium for conducting the magnetic and gravity separation tests. Tero Korhonen from GTK Mintec is also thanked for conducting the flotations tests. We also acknowledge the extensive work done by Chris Hughes, Luke Morgan and Lorenza Sardisco from XMS on XRF and QXRD analyses of the samples and products discussed in this report.

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13 APPENDICES

13.1 Appendix - BATCircle project summary

BATCircle is a project developed and funded by Business Finland (<https://www.businessfinland.fi/>). This is a 21-million-euro project with 23 consortium partners, and a duration of 24 months. The BATCircle project has been designed to be based around the concept of a Circular Ecosystem of Battery Metals in Finland. The concept includes both primary raw materials, downstream refining, and recycling in batteries. Most relevant operators in the existing battery business at all stages of the regional value chain are involved with this project in some form.



Figure A 1. The BATCircle Consortium - website: <https://batcircle.aalto.fi/en>

This project consists of four technical work packages (WP1-WP4), one for business studies (WP5) one for project management (WP6) and one for European co-operation (WP7).

- **WP1 – Sustainable Primary Resources:** The methods range from screening and efficient use of battery related multi-mineral resources
- **WP2 – Value Addition in Metals Production:** Improved metallurgical processes (pyro, hydro, mechanical) for refining
- **WP3 – Recycling of Batteries:** Recycling of battery metals as well as synthesis and characterization of advanced precursors
- **WP4 – Tailored Precursors and Active Electrode Materials:** Active materials for lithium batteries. These technical tasks are supplemented by business studies aiming at identifying new business opportunities within the battery ecosystem and providing new tools for structural and electrochemical characterisations
- **WP5 – Business Potential:** Strengthening the cooperation within the overall ecosystem
- **WP6 – Project management**
- **WP7 – Development of the European BATCircle**

Work Package 1 (WP1) is managed by GTK. The impact of the WP1 is to quantify the Finnish potential for development of battery minerals mining. A new way of characterising battery mineral deposits will be developed (geometallurgy developed fit for purpose in battery mineral systems). Then a theoretical future Finnish controlled battery ecosystem is to be developed in a series of strategic steps.

The key objectives of Work Package 1 are as follows:

1. Analyse known Finnish battery mineral deposits with special emphasis on cobalt and associated typically polymetallic deposits,
2. Develop a more effective way of characterizing those battery commodities in context of process response, which leads to a more effective economic characterization. Geometallurgy is the chosen approach to do this,
3. A strategic development plan for the development of Finnish battery mineral resources in a complete battery ecosystem.

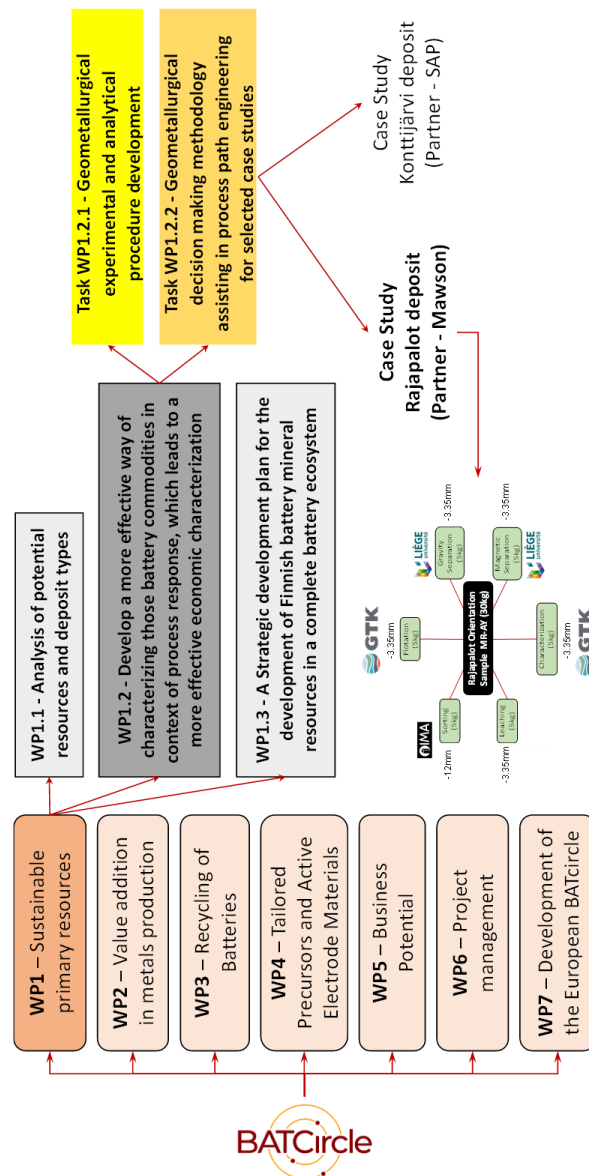


Figure A 2. Map of the BATCircle project to WP1.2 to this report.

13.2 Appendix - Work Package WP1.2 Structure and Deliverables

Table A 1. Work Package WP1.2 Objectives

WP Objectives
<ul style="list-style-type: none"> Analyse known Finnish battery mineral deposits like with special emphasis on cobalt and associated polymetallic deposits. Lithium and graphite deposits will be part of the work with smaller weight. Develop a more effective way of characterizing cobalt bearing battery commodities in context of process response, which leads to a more effective economic characterization. Geometallurgy is the chosen approach to do this. Also cobalt minerals flotation chemical environment is studied and preliminary hydrometallurgical process model will be built. A Strategic development plan for the development of Finnish battery mineral resources in a complete battery ecosystem.

Table A 2. Work Package WP1.2 Deliverables

Deliverables	By	Month
D1.1.1. A database analysis of Finnish cobalt, lithium and graphite deposits as well as technical and economic profiles about different type of deposits (study report).	GTK	M12
D1.2.1. A geometallurgical experimental and analytical procedure for cobalt related battery minerals that allows for process characterisation domaining on drill core scale samples to be validated with selected Case studies	GTK	M8
D1.2.2. A geometallurgical decision making methodology based on Case study laboratory validation. This assists in beneficiation process path development of cobalt bearing battery minerals.	GTK	M20
D1.2.3. Floatability model of Co-containing minerals based on bubble-particle attachment probability	Aalto	M20
D1.2.4. Adjustable Process model	Aalto	M24
D1.3.1. A Strategic development plan for the development of Finnish mineral resources in a complete battery ecosystem	GTK	M24

Deliverable D1.2.1 is:

Michaux, S., P. (2020): How to Set Up and Develop a Geometallurgical Program, GTK Open Work File Report, ISBN 978-952-217-409-3

Deliverable D1.2.2 is the procedure applied to two case studies. This report is part of the Mawson case study, Rajapalot deposit series.

Work Package 1 Project consortium

- GTK
- Aalto University
- SAP
- **Mawson Gold**
- Arctic Platinum
- FinnCobalt
- FMG

Work Package WP1.2

The title of WP1.2 is:

WP1.2 Develop a more effective way of characterizing those battery commodities in context of process response, which leads to a more effective economic characterization

Based on each deposit type geological and mineralogical characteristic and acquired raw material specification requirements, a concept or protocol for proper geometallurgical study is developed for each deposit type.

Mawson Gold Oy

The advanced exploration gold-cobalt prospects of the Rajapalot area were discovered by [Mawson OY](#) and is a strategic battery metal project for Finland. The project has been delineated with more than 80 kilometres of diamond drilling over the last 3 years and continues to increase in size. Mawson is advancing the project through the initial stages of land use planning and environmental permitting. Already a NI43-101 resource base comprising over 700,000 ounces gold-equivalent has been defined. (for detail read [Sept 2020 NI43-101 Inferred Resource](#)), making Rajapalot a top 10 European resource for cobalt alone. Drill hole samples from the prospects were used in BATCircle 1.0 geometallurgical studies.

Rajapalot deposit (Mawson) BATCircle Geometallurgical Case Study

This geometallurgical approach will be applied to two case studies. One of those BATCircle WP1.2 case studies is the Rajapalot deposit, owned by Mawson Oy.

Gold is free milling and liberates well, followed by gravity, leaching and or flotation separation methods. Cobalt occurs in multiple mineral species, and therefore requires detailed benchtop research to successfully provide locally-derived battery metal feedstock to the growing needs of the low carbon economy.

13.3 Appendix - BATCircle Project WP1.2 Sample Labelling Protocol

The following labelling protocol was adopted to keep track of what samples were and what has been done to them.

The owner of the case study deposit

- Mawson M

The name of the case study deposit

- Raja R
- Palokas P

Process separation methods

- Characterization (to compare all sub-products to)
- Flotation F
- Leaching L
- Gravity Separation G
- Magnetic Separation M
- Sample Reserve S

Orientation Sample rock type labels

- Muscovite/sericite-biotite schist (potassic), ductile, rich in sheet silicates MR-MP
- Albite breccia (sodic), brittle/competent, low in sheet silicates MR-AY
- Predominantly Mg-Fe amphibolite (MFA) with considerable chlorite content MP-PAL1

Sample MR-AY

- M**awson
- R**ajapalot region
- A**lbite
- Y**ellow colour in database

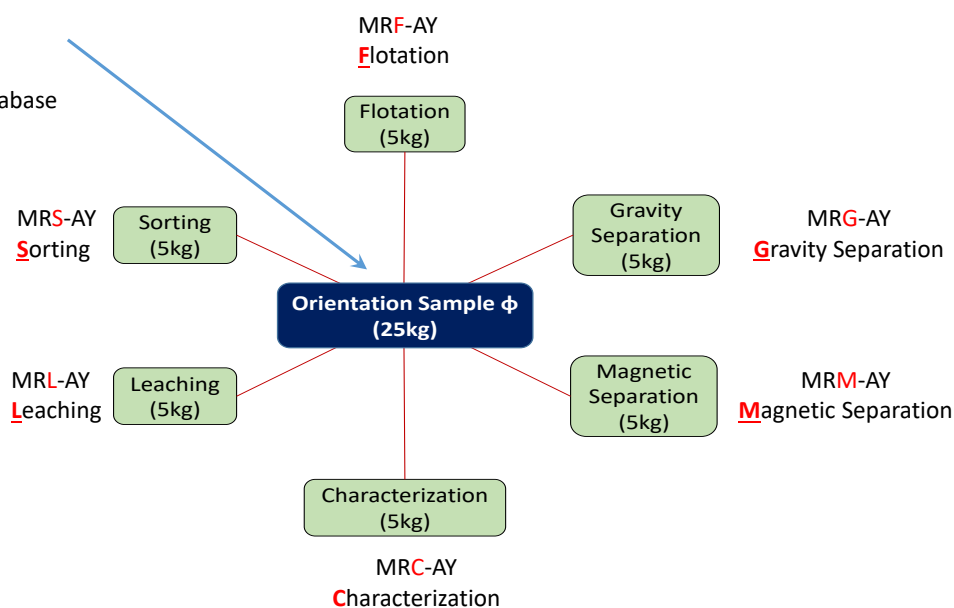


Figure A 3. BATCircle Project sample labelling protocol for process separation sub-samples for each Orientation Sample.

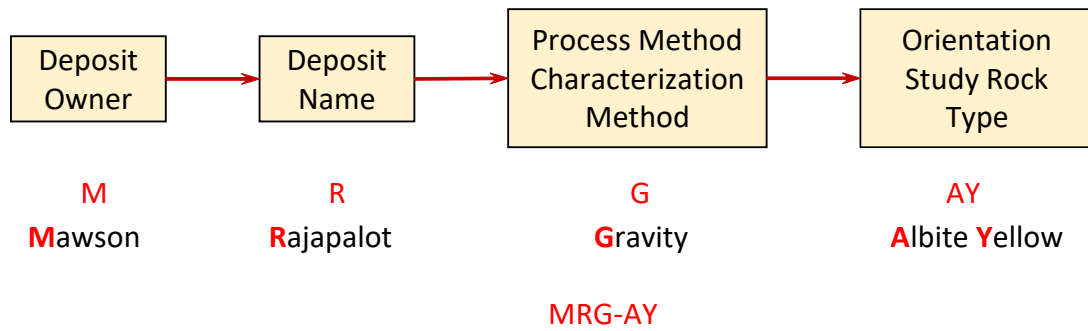


Figure A 4. BATCircle project sample labelling protocol example for sample MR-AY to be sent to gravity concentration - MRG-AY.

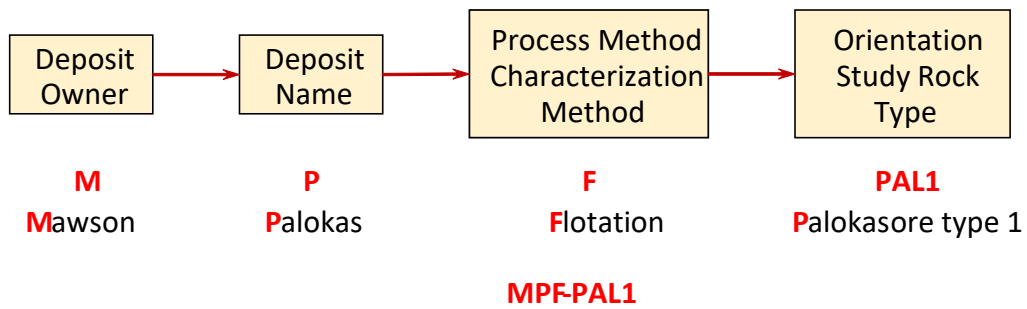


Figure A 5. BATCircle project sample labelling protocol example for sample MP-PAL1 to be sent to flotation - MPF-PAL1.

13.4 Appendix - Magnetic separation mass balance

Table A 3. Mass balance for the magnetic separation tests.

Ore Type	Product	Mass (g)	Yield (wt%)	Grade				Recovery (%)			
				Au (g/t)	Co (ppm)	As (ppm)	U (ppm)	Au	Co	As	U
MRM-MP	1 Amp	98.2	10.0	20.9	2350	772	275	20	34	9	11
	4 Amp	74.1	7.6	38.9	1170	1070	679	28	13	9	21
	6 Amp	67.6	6.9	13.8	578	677	530	9	6	5	15
	Non-Mag	740.7	75.5	6.2	430	882	177	44	47	76	54
	Feed*	980.5	100.0	10.6	688	871	249	100	100	100	100
MRM-AY	1 Amp	168.1	17.1	9.0	2240	453	124	36	52	13	23
	4 Amp	79.2	8.1	7.9	824	703	255	15	9	9	22
	6 Amp	56.7	5.8	4.9	509	632	214	7	4	6	13
	Non-Mag	678.2	69.1	2.6	364	628	56	42	34	72	42
	Feed*	982.2	100.0	4.2	731	604	93	100	100	100	100
MPM-PAL1	1 Amp	205.0	20.8	8.1	1140	19	74	35	71	-	35
	4 Amp	143.0	14.5	7.1	217	52	49	21	9	12	16
	6 Amp	129.7	13.2	5.0	118	55	41	14	5	11	12
	Non-Mag	508.8	51.6	2.8	93	95	32	30	14	77	37
	Feed*	986.4	100.0	4.8	332	64	44	100	100	100	100

*Back-calculated feed.

13.5 Appendix - Gravity concentration mass balance

Table A 4. Mass balance for the gravity concentration tests.

Ore Type	Product	Yield (wt%)	Grade				Recovery (%)			
			Au (g/t)	Co (%)	As (%)	U (ppm)	Au	Co	As	U
MRM-MP	Concentrate	1.3	812	126.0	152.0	8100	48	14	20	29
	Middlings	21.3	24	21.0	17.0	486	22	36	36	27
	Tailings	77.3	9	8.0	5.7	215	30	50	44	44
	Feed*	100.0	23	12.4	10.1	378	100	100	100	100
MRM-AY	Concentrate	2.2	71	103.0	84.0	1070	35	16	27	24
	Middlings	23.8	4	21.0	8.7	134	20	34	30	33
	Tailings	74.0	3	9.9	4.0	56	45	50	43	43
	Feed*	100.0	4	14.6	6.9	97	100	100	100	100
MPM-PAL1	Concentrate	1.3	140	23.0	10.0	756	33	4	19	23
	Middlings	32.0	6	9.1	0.7	39	33	42	33	29
	Tailings	66.7	3	5.7	0.5	31	34	54	48	48
	Feed*	100.0	6	7.0	0.7	43	100	100	100	100

*Back-calculated feed.

13.6 Appendix - Flotation mass balance

Table A 5. Mass balance for the flotation tests.

Ore Type	Flot min	Product	Weight		Co		As		U		Au	
			g	%	%	Rec%	%	Rec%	%	Rec%	g/t	Rec%
Blend		RC1-3	57.3	5.7	0.84	41.14	0.55	80.60	0.03	11.85	96.42	93.02
		Tails	942.8	94.3	0.07	58.86	0.01	19.40	0.01	88.15	0.44	6.98
	1	CC1/1	18.3	1.8	1.98	30.99	1.43	67.31	0.03	4.56	266.00	81.95
	2	CC1/2	4.8	0.5	0.94	3.86	0.52	6.42	0.02	0.90	66.19	5.35
	3	CC1/1-2	23.1	2.3	1.76	34.85	1.24	73.73	0.03	5.46	224.48	87.30
	2	CC1/3	4.3	0.4	0.63	2.32	0.28	3.14	0.03	0.94	21.07	1.53
	5	CC1/1-3	27.4	2.7	1.59	37.16	1.09	76.87	0.03	6.40	192.56	88.83
	3	CC1/4	3.8	0.4	0.42	1.36	0.17	1.65	0.03	0.77	18.67	1.19
	8	CC1/1-4	31.2	3.1	1.44	38.53	0.98	78.52	0.03	7.17	171.38	90.02
		CT1	26.1	2.6	0.12	2.61	0.03	2.08	0.02	4.68	6.81	2.99
	Feed	1000.1	100.0	0.12	100.00	0.04	100.00	0.01	100.00	5.94	100.00	
AY		RC1-3	161.3	8.1	0.81	46.96	0.64	81.23	0.01	9.63	39.28	77.67
		Tails	1836.1	91.9	0.08	53.04	0.013	18.77	0.008	90.37	0.99	22.33
	1	CC1/1	85	4.3	1.17	35.91	1	66.84	0.009	4.34	57.88	60.31
	2	CC1/2	24.6	1.2	0.89	7.91	0.62	11.99	0.013	1.92	50.30	15.17
	3	CC1/1-2	109.6	5.5	1.11	43.82	0.91	78.84	0.01	6.26	56.18	75.48
	2	CC1/3/1	6.2	0.3								
	3	CC1/1-3	115.8	5.8								
	3	CC1/3/2	4.9	0.2								
	5	CC1/3	11.1	0.6	0.431	1.73	0.186	1.62	0.014	0.93	7.24	0.99
	8	CC1/1-3	120.7	6.0	1.04	45.55	0.85	80.46	0.01	7.19	51.68	76.46
	CT1	40.6	2.0	0.096	1.41	0.024	0.77	0.01	2.44	2.43	1.21	
	Feed	1997.4	100.0	0.14	100.00	0.06	100.00	0.01	100.00	4.08	100.00	
PAL1		RC1-3	111.9	5.6	0.33	24.32	0.10	85.94	0.01	9.21	75.84	90.76
		Tails	1890.0	94.4	0.06	75.68	0.001	14.06	0.004	90.79	0.46	9.24
	1	CC1/1	9.6	0.5	0.94	6.02	0.485	34.64	0.006	0.71	436.30	44.80
	2	CC1/2	33.7	1.7	0.53	11.92	0.174	43.62	0.012	4.98	107.00	38.57
	3	CC1/1-2	43.3	2.2	0.62	17.94	0.24	78.26	0.01	5.69	180.01	83.36
	2	CC1/3	17.8	0.9	0.32	3.80	0.046	6.09	0.007	1.49	27.21	5.18
	5	CC1/1-3	61.1	3.1	0.53	21.74	0.19	84.35	0.01	7.18	135.49	88.54
	3	CC1/4	10.2	0.5	0.167	1.14	0.017	1.29	0.005	0.68	8.38	0.91
	8	CC1/1-4	71.3	3.6	0.48	22.88	0.16	85.64	0.01	7.86	117.31	89.45
		CT1	40.6	2.0	0.053	1.44	0.001	0.30	0.003	1.35	3.01	1.31
	Feed	2001.9	100.0	0.07	100.00	0.01	100.00	0.00	100.00	4.67	100.00	
MP		RC1-3	198.8	9.9	0.89	65.82	0.96	93.83	0.04	19.04	46.89	84.60
		Tails	1803.0	90.1	0.051	34.18	0.007	6.17	0.02	80.96	0.94	15.40
	1	CC1/1	43.7	2.2	2.93	47.59	3.63	77.60	0.048	4.71	94.50	37.48
	2	CC1/2	22.5	1.1	1.07	8.95	0.99	10.90	0.057	2.88	61.72	12.60
	3	CC1/1-2	66.2	3.3	2.30	56.54	2.73	88.50	0.05	7.59	83.36	50.09
	2	CC1/3	19.5	1.0	0.59	4.28	0.346	3.30	0.069	3.02	147.75	26.15
	5	CC1/1-3	85.7	4.3	1.91	60.81	2.19	91.80	0.06	10.61	98.01	76.24
	3	CC1/4	15.3	0.8	0.395	2.25	0.156	1.17	0.073	2.51	40.13	5.57
	8	CC1/1-4	101.0	5.0	1.68	63.06	1.88	92.96	0.06	13.12	89.24	81.81
		CT1	97.8	4.9	0.076	2.76	0.018	0.86	0.027	5.93	3.14	2.79
	Feed	2001.8	100.0	0.13	100.00	0.10	100.00	0.02	100.00	5.50	100.00	

13.7 Appendix - Leaching mass balance

Table A 6. Leaching log data Test 1 & 2.

TEST LOGSHEETS

Test 1: MRL-AY

Direct leaching	START	Unit	Value	FINAL	Unit	Value	Test phase	Time	Temp	pH	O2	Air	Ca(OH) ₂	NaCN	NaCN 56 g/L	NaCN 2.5 g/L	Sample
	Solid, wet	g	603.7	Slurry	g	1948		h	°C		mg/l	ml/min	g	g/L	ml	ml	ml
	Moisture	%	0.17				Pre-aeration	0.0	22	11.0		500	0.37				
	Solid, dry	g	602.7	Cake as dry	g	599.1		1.0	22	11.0		500	0.46				
								2.0	22	11.1		500	0.53				
Initial	Liquid	g	1297				Leaching	0.0	22	11.4	8.5	500		3.5	80	0.5	0.5
Initial + moisture	Liquid	g	1298			1349		0.5	22	11.3	9.1	500					
								1.0	22	11.2	5.4	500		3.2		18.0	18.0
	Liquid density	kg/l	1			1		3.0	22	11.1	8.3	500		3.2		16.5	16.5
	Solid content	% w/w	31.7			30.8		6.0	22	11.3		500	0.7	3.2		18.0	18.2
								9.0	22	11.1	8.7	500		3.1		1.0	1.0
								12.0	22	11.2	8.7	500	0.11	3.0			18.0
								18.0	22	11.1	8.5	500	0.08	2.9		1.0	1.0
								24.0	22	11.0	8.8	500		2.7			1.0

Test 2: MRL-MP

Direct leaching	START	Unit	Value	FINAL	Unit	Value	Test phase	Time	Temp	pH	O2	Air	Ca(OH) ₂	NaCN	NaCN 56 g/L	NaCN 2.5 g/L	Sample
	Solid, wet	g	751.8	Slurry	g	2092		h	°C		mg/l	ml/min	g	g/L	ml	ml	ml
	Moisture	%	0.20				Pre-aeration	0.0	22	11.4		500	0.56				
	Solid, dry	g	750.3	Cake as dry	g	748.6		1.0	22	11.2		500	0.66				
								2.0	21			500					
Initial	Liquid	g	1300				Leaching	0.0	21	11.6	8.6	500	0.83	3.2	80	0.5	0.5
Initial + moisture	Liquid	g	1302			1343		0.5	21	11.5	8.7	500					
								1.0	21	11.4	8.8	500		3.2		16.0	16.0
	Liquid density	kg/l	1			1		3.0	22	11.3	8.9	500		3.0		19.0	19.0
	Solid content	% w/w	36.6			35.8		6.0	22	11.2	8.8	500		3.1		18.5	18.5
								9.0	22	11.1	8.7	500		2.6		1.0	1.0
								12.0	22	11.2	8.8	500	0.08	2.9			15.0
								18.0	22	11.0	8.6	500		2.8			1.0
								24.0	22	11.0	8.7	500	0.19	2.5			1.0

Table A 7. Leaching log data Test 3 & 4.

Test 3: MPL-PAL1

Direct leaching	START	Unit	Value	FINAL	Unit	Value	Test phase	Time	Temp	pH	O2	Air	Ca(OH) ₂	NaCN	NaCN 56 g/L	NaCN 2.5 g/L	Sample	
	Solid, wet	g	602.4	Slurry	g	1920												
	Moisture	%	0.18				Pre-aeration	0.0	22	11.1		500	0.36					
	Solid, dry	g	601.3	Cake as dry	g	599.8		1.0	21	11.1	8.7	500	0.49					
								2.0	22	11.5		500	0.53					
Initial	Liquid	g	1280				Leaching	0.0	22	11.5	8.8	500		3.3	80			1.0
Initial + moisture	Liquid	g	1281	Liquid	g	1320		0.5	22	11.4	8.8	500						
								1.0	22	11.4	8.9	500				18.0		18.0
	Liquid density	kg/l	1	Liquid density	kg/l	1		3.0	22	11.3	9.0	500				18.0		18.0
	Solid content	% w/w	31.9	Solid content	% w/w	31.2		6.0	22	11.2	8.9	500				15.0		15.0
								9.0	22	11.1	8.8	500				1.0		1.0
				Solids loss in electrode wash				12.0	22	11.0	8.8	500				16.5		16.5
						0.8		18.0	22	11.0	9.0	500	0.56			1.0		1.0
								24.0	22	11.4	8.9	500						1.0

Test 4: MRL-AY

CIL	START	Unit	Value	FINAL	Unit	Value	Test phase	Time	Temp	pH	O2	Air	Ca(OH) ₂	NaCN	NaCN 56 g/L	NaCN 2.5 g/L	Sample	
	Solid, wet	g	746.4	Slurry	g	2300												
	Moisture	%	0.17				Pre-aeration	0.0	21	11.0		500	0.39					
	Solid, dry	g	745.2	Cake as dry	g	740.7		1.0	21	11.0		500	0.16					
	Carbon	g	20.1	Carbon	g	19.8		2.0	22	11.1		500	0.06					
Initial	Liquid	g	1472				Leaching	0.0	22	11.4	8.6	500						2.0
Initial + moisture	Liquid	g	1473	Liquid	g	1560		0.5	22	11.3	8.8	500				80		2.0
								1.0	22	11.2	8.9	500						17.0
	Liquid density	kg/l	1	Liquid density	kg/l	1		3.0	22	11.1	9.0	500				18.0		18.0
	Solid content	% w/w	33.6	Solid content	% w/w	32.2		6.0	22	11.3	8.9	500			1.8			20.0
								9.0	22	11.1	9.0	500				2.1		2.1
				Solids loss in electrode wash				12.0	22	11.2	8.9	500			1.5			24.0
						1.2		18.0	22	11.1	8.7	500			3.3			2.0
								24.0	22	11.0	8.5	500						17.0

Table A 8. Leaching log data Test 5 & 6.

Test 5: MRL-MP

CIL	START	Unit	Value	FINAL	Unit	Value	Test phase	Time	Temp	pH	O2	Air	Ca(OH) ₂	NaCN	NaCN 56 g/L	NaCN 2.5 g/L	Sample	
	Solid, wet	g	727.7	Slurry	g	2281												
	Moisture	%	0.20				Pre-aeration	0.0	22	11.0		500	0.52					
	Solid, dry	g	726.2	Cake as dry	g	724.6		1.0	21	11.0		500	0.09					
	Carbon	g	20.0	Carbon	g	19.8		2.0	22	11.1		500	0.01					
Initial	Liquid	g	1500				Leaching	0.0	22	11.4	8.5	500		2.9	80	1.0	1.0	
Initial + moisture	Liquid	g	1501	Liquid	g	1557		0.5	22	11.3	8.7	500						
	Liquid density	kg/l	1	Liquid density	kg/l	1		1.0	22	11.2	8.8	500		2.9		20.0	20.0	
	Solid content	% w/w	32.6	Solid content	% w/w	31.8		3.0	22	11.1	8.8	500		2.8		18.0	18.0	
								6.0	22	11.3	8.8	500		2.7		19.0	19.0	
								9.0	22	11.1	8.8	500		2.7		1.0	1.0	
								12.0	22	11.2	8.7	500		2.6		24.5	24.5	
								18.0	22	11.1	8.5	500		2.6		0.5	0.5	
								24.0	22	11.0	8.6	500		2.2			17.0	

Test 6: MPL-PAL1

CIL	START	Unit	Value	FINAL	Unit	Value	Test phase	Time	Temp	pH	O2	Air	Ca(OH) ₂	NaCN	NaCN 56 g/L	NaCN 2.5 g/L	Sample	
	Solid, wet	g	745.7	Slurry	g	2329												
	Moisture	%	0.18				Pre-aeration	0.0	24			500	0.60					
	Solid, dry	g	744.4	Cake as dry	g	740.7		1.0	24			500	0.15					
	Carbon	g	20.1	Carbon	g	20.1		2.0	24			500						
Initial	Liquid	g	1476				Leaching	0.0	24		8.3	500		3.2	86	10.0	10.0	
Initial + moisture	Liquid	g	1477	Liquid	g	1588		0.5	24		8.4	500						
	Liquid density	kg/l	1	Liquid density	kg/l	1		1.0	24		8.3	500		3.1		20.0	20.0	
	Solid content	% w/w	33.5	Solid content	% w/w	31.8		3.0	23		8.5	500		3.0		20.0	20.0	
								6.0	23		8.8	500		3.0		19.5	19.5	
								9.0	23		8.6	500		2.9				
								12.0	23		8.6	500		2.8		20.0	20.0	
								18.0	23		8.6	500		2.7			2.0	
								24.0	23		8.5	500	0.31	2.5			23.0	