



Geological Survey of Finland

Circular Economy Solutions Unit

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Metallurgical testwork for the geometallurgical orientation study of the Konttijärvi Ni-Cu-PGE-(Au-Co) deposit

BATCircle Project Report 06 – WP1 Task 1.2

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Abstract This report summarises the main findings of an orientation study work performed on 10 previously selected ore types from the Konttijärvi project, as part of the BATCircle project. These 10 ore-types, which display distinct mineralogical properties, have been tested using various mineral processing methods in view investigation their processing behaviour as well as evaluating the suitability of each technique.			
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Contents

Documentation page

1	Introduction	4
2	Batcircle geometallurgical program	4
3	Konttijärvi Geometallurgical Experimental Objective	4
3.1	Questions to address :	4
4	Materials and experimental design	6
4.1	Samples	6
4.2	Experimental design	6
4.3	Chemical assays	9
4.4	Bulk Element XRF Characterization of each Konttijärvi Sample Orientation Sample	12
4.5	Semi Quantitative mineralogical analysis by X-Ray Diffraction (Q-XRD)	12
4.6	Automated Mineralogy	15
5	Sample Preparation	15
6	Magnetic separation	16
6.1	Methods/Procedure	16
6.2	Magnetic Separation Results	17
7	Gravity concentration	45
7.1	Methods/Procedure	45
7.2	Gravity Separation Results	45
8	Flotation	69
8.1	Methods/Procedure	69
8.2	Flotation Results	70
8.3	Palladium recovery flotation data correlations	87
8.4	Copper recovery flotation data correlations	92
8.5	Cobalt recovery flotation data correlations	94
8.6	Characterization of flotation tailings	96
9	Leaching	98
10	Sorting	99
11	Discussions	102
12	Conclusions	110
13	Future Work	110
14	References	113

15 Appendix A - Batcircle project summary	114
15.1 BATcircle Work Package Structure	114
15.2 Work Package 1 Summary	115
15.3 Work Package WP1.2 Structure and Deliverables	117
15.4 Work Package 1 Project consortium	117
15.5 Work Package WP1.2	117
15.6 Suhanko Arctic Platinum Oy	118
15.7 Konttijärvi deposit (SAP) BATCircle Geometallurgical Case Study	118
16 Appendix B – BATcircle Project WP1.2 Sample Labelling Protocol	119
17 Appendix C – Sample Grinding Size Distributions	121
18 Appendix D – Magnetic Separation Data	123
19 Appendix E – Gravity Separation Data	130
20 Appendix F – Flotation Separation Data	145

1 INTRODUCTION

This report is one in a series of reports to document work done in the development of the geometallurgy Konttijärvi case study, which is part of Work Package WP1.2 in the BATCircle project. A description and experimental approach of the BATCircle project is shown in Appendix A.

Suhanko is a polymetallic deposit, so there are number of valuable elements in this ore system. PGE (especially Pd) is the main commodity but significant amounts of battery metals including Ni, Cu and Co are to be found in the ore.

2 BATCIRCLE GEOMETALLURICAL PROGRAM

The BATCircle project Work Package 1.2 had two geometallurgical tasks. The first was a documented geometallurgical procedure (Michaux & O'Connor 2020). The second was to have this procedure shown in two case studies. One of this case studies is the Konttijärvi geometallurgical study. This report is the second for this case study, where the first was the characterization of samples collected, the data for which is used in this report (Michaux *et al* 2020).

Figure 1 shows the experimental design and workflow for this study.

3 KONTTIJÄRVI GEOMETALLURGICAL EXPERIMENTAL OBJECTIVE

In context of mineral processing, both leaching and flotation have been considered as process paths in previous work. Sulphide extraction to be considered is to target both copper (Cu) and nickel (Ni). The geometallurgical objective for this campaign is:

Geometallurgical studies to refine the metallurgical process response to maximize recovery of minerals into separate Cu/PGE and Ni/PGE/Co concentrates

3.1 Questions to address :

- Primary question – what is the most useful process path? Flotation, leaching, or a combination of both?
- Can gravity separation and magnetic separation augment the best process path?
- How does this change between rock types?
- How to achieve 80% for palladium recovery, where grain size is below 10 micron and mostly in pentlandite.
- Ni is concentrated at the lower end of the deposit. How to exploit this?
- A useful metallurgical target would be to try and suppress the pyrrhotite, as this would be a penalty element at the smelter.

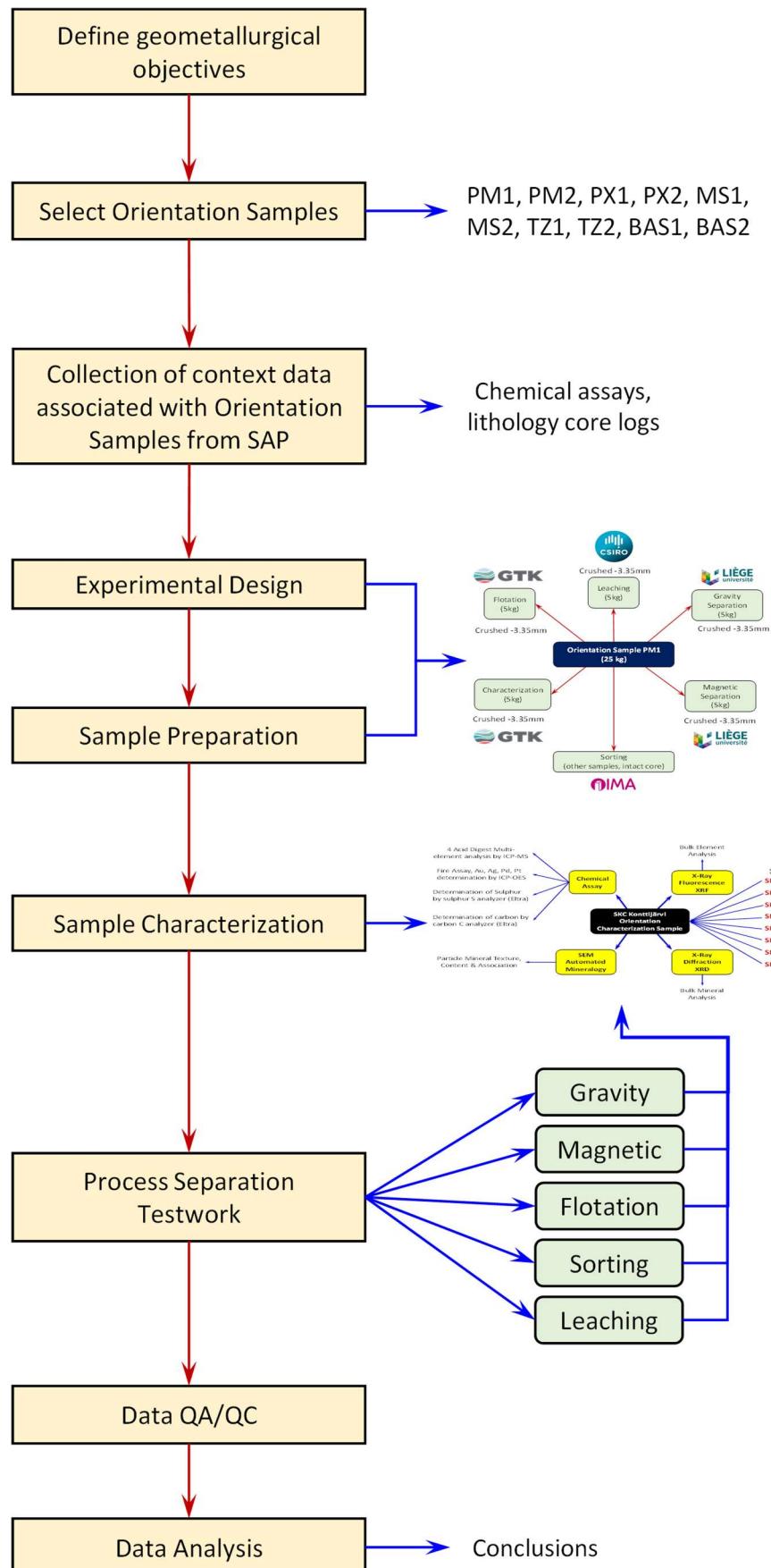


Figure 1. Konttijärvi geometallurgical program

4 MATERIALS AND EXPERIMENTAL DESIGN

4.1 Samples

Samples were collected in consultation with Suhanko Arctic Platinum (SAP) staff, which would be the foundation of the geometallurgical Orientation Study. These samples were to represent the rock texture extremes in which the rest of the deposit is made up of a combination of these samples. If these samples are characterized appropriately, process behaviour can be more effectively planned for.

The collected samples were transported to GTK-Mintec for sample preparation. Part of this sample preparation was to prepare a series of sub-samples, each of which were to be tasked for a specific process separation test. One of these sub-samples for each rock type was to be used to characterize the mineral composition and form of the rock as received (unprocessed). These characterization results would later be used to compare the products of process separation tests on parallel sister samples.

The ore type definition and sample selection procedure has been reported in (Michaux *et al* 2020). The labelling protocol used in this project is shown in Appendix B.

4.2 Experimental design

The experimental design of this study has several components. The first step was to all sample 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 3. One sub-sample for each Orientation sample was subject to characterization (Michaux *et al* 2020). Each remaining sub-sample was sent to the laboratory for process separation. 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 2).

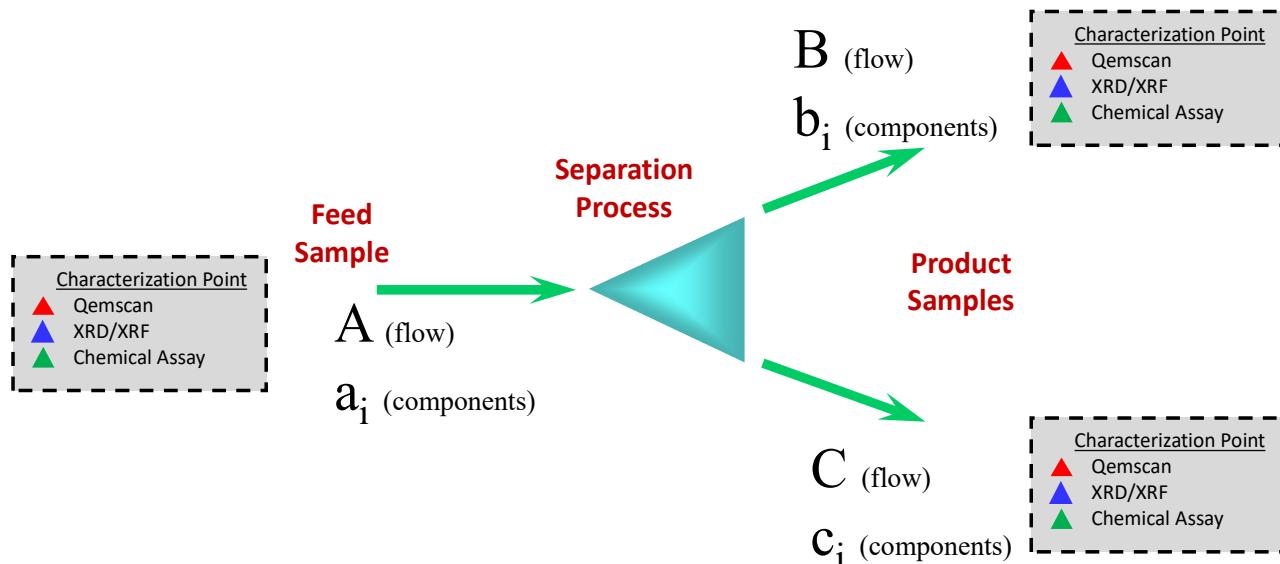


Figure 2. Each process test is done in a manner where a mineralogical mass balance could be conducted

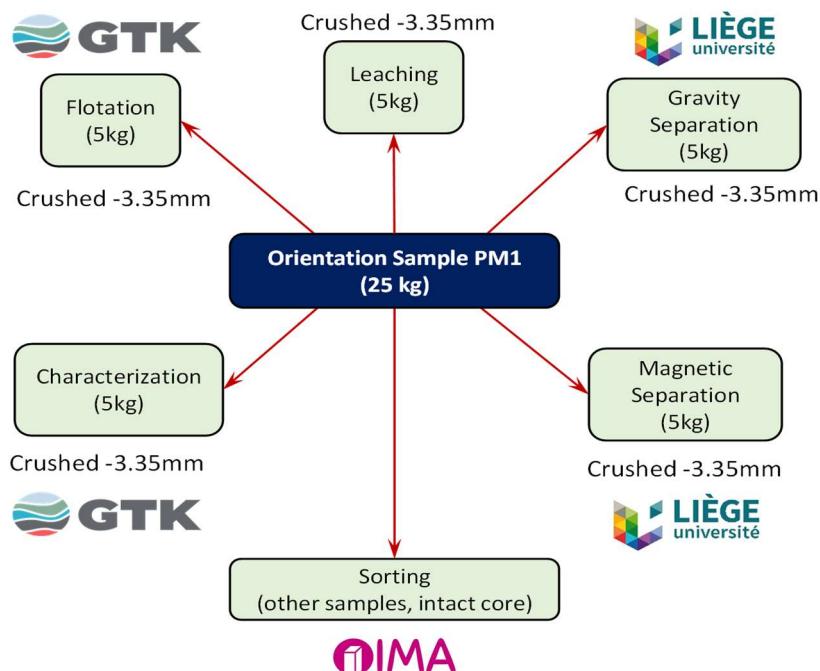


Figure 3. Experimental design sub-sampling

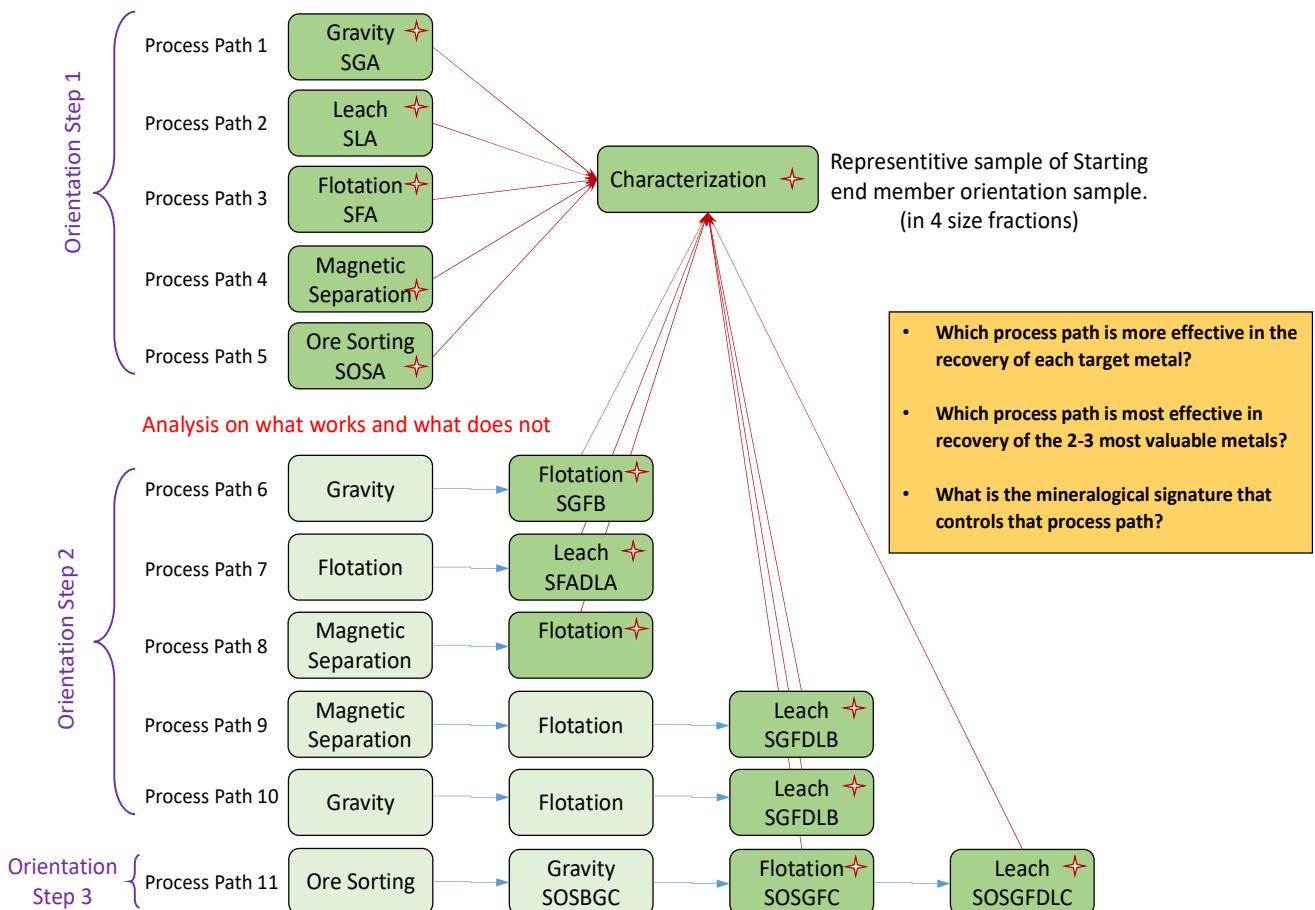


Figure 4. Process test experimental design

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 (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.

Due to project management 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. This data set would be used to examine which process path is most effective recovery for different target metals (PGE, Ni, Cu, Co).

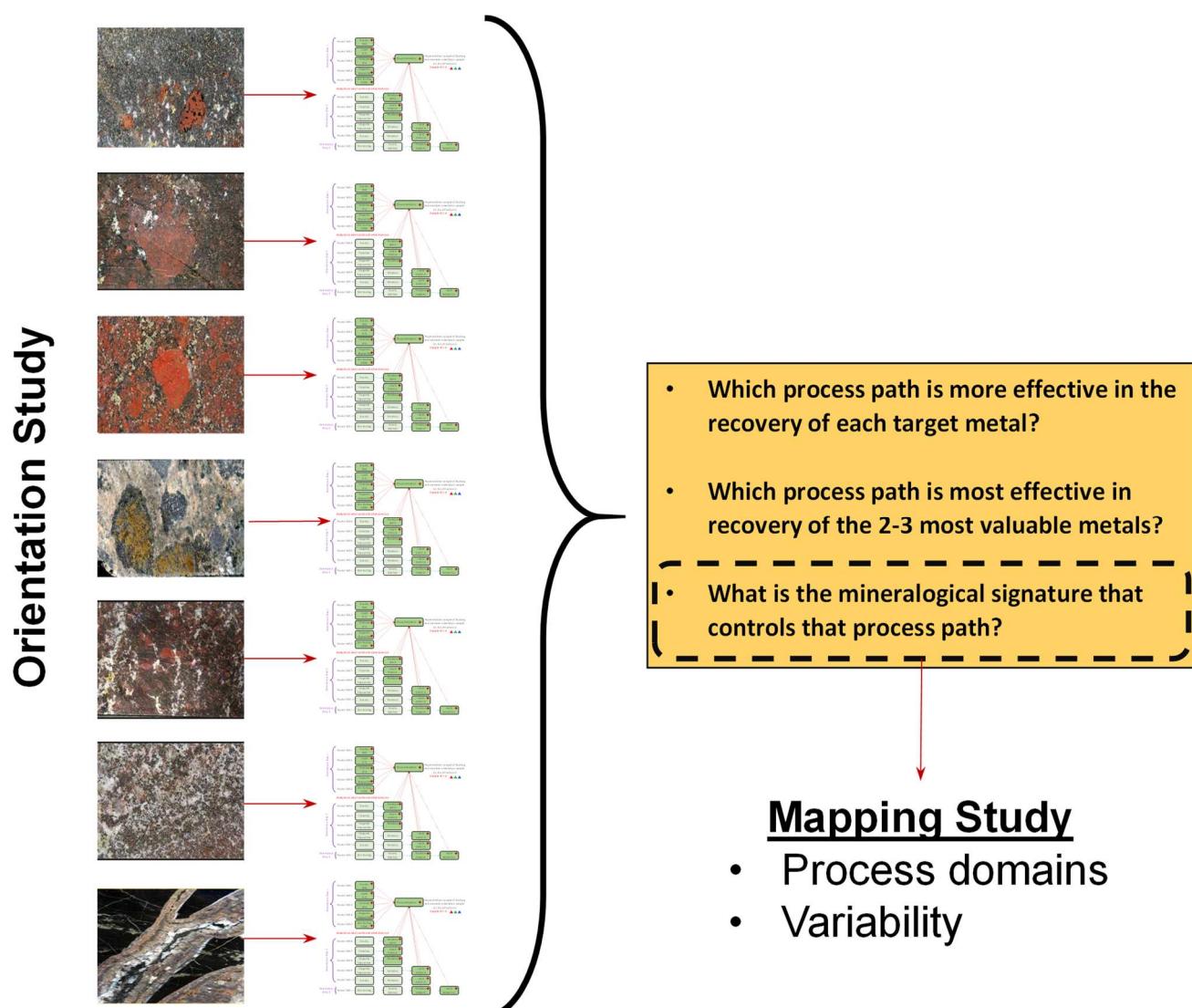


Figure 5. Different process paths compared across all Orientation Studies

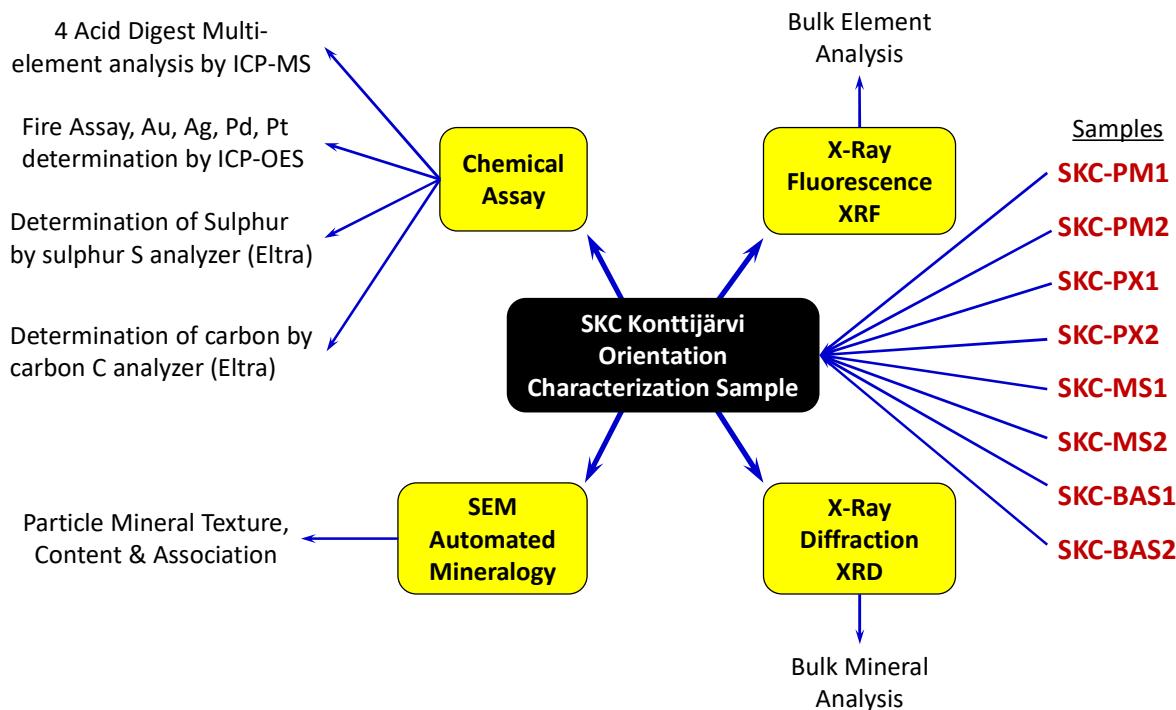


Figure 6. The four kinds of ore characterization methods used in this study

4.3 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.

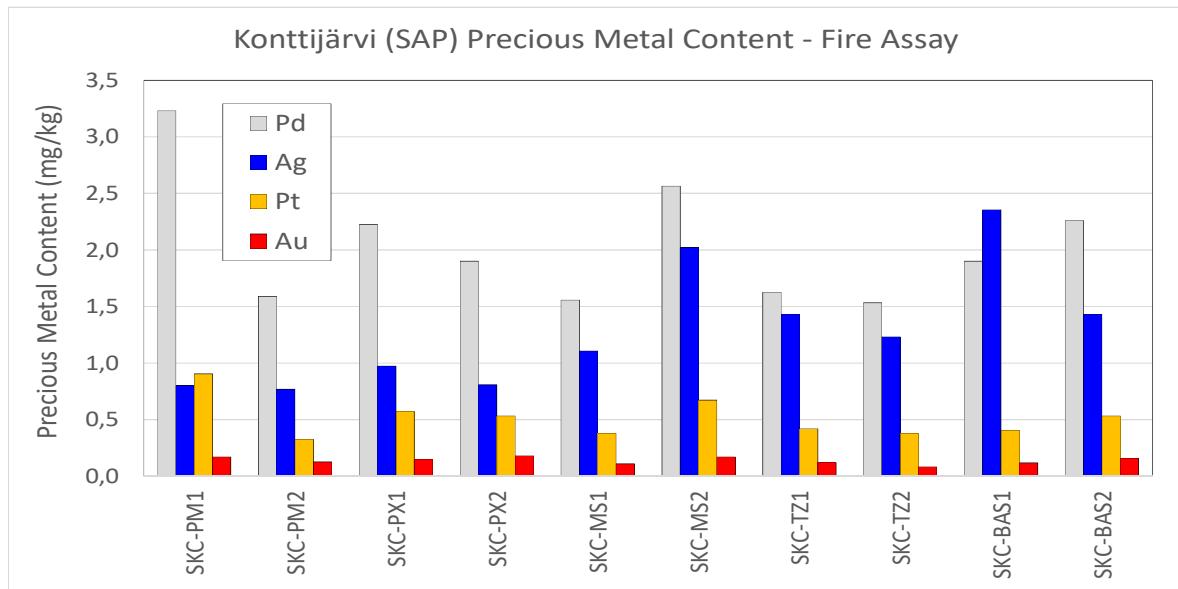


Figure 7. Precious metal content from chemical fire assay measurements for Konttijärvi Orientation Samples (Method 711P) (Michaux *et al* 2020)

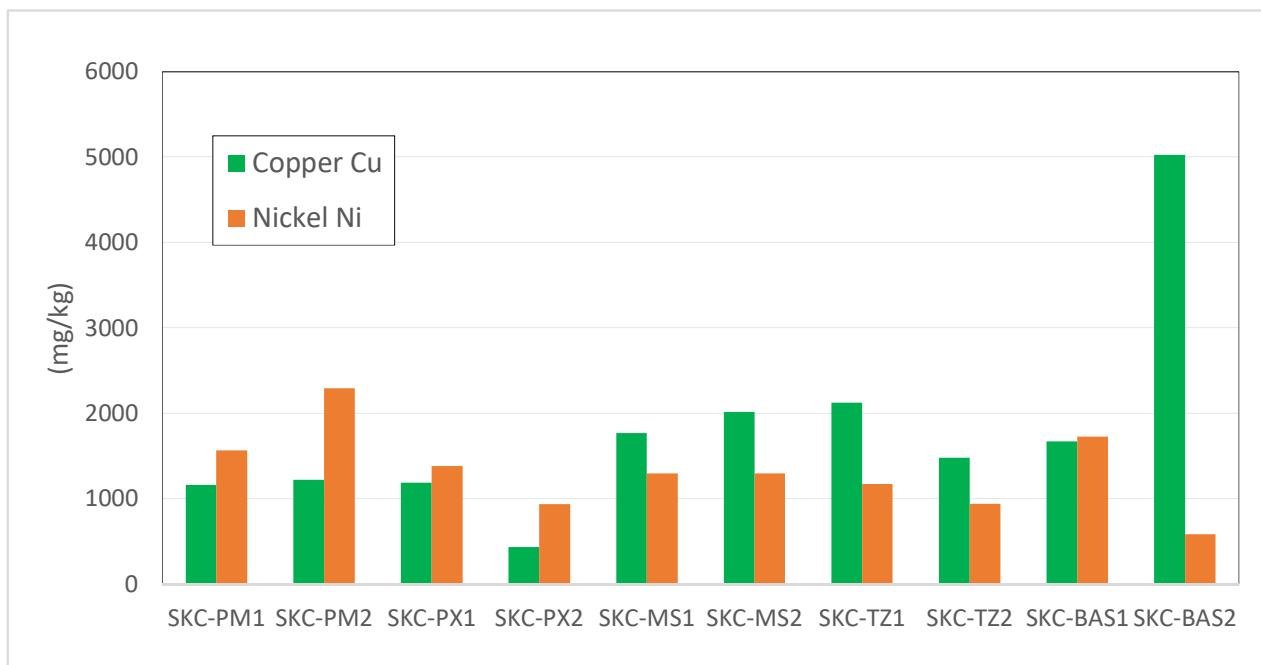


Figure 8. Copper and nickel content
Four-acid digestion and Multi-element analysis by ICP-OES-technique (Method 306P) (Michaux *et al* 2020)

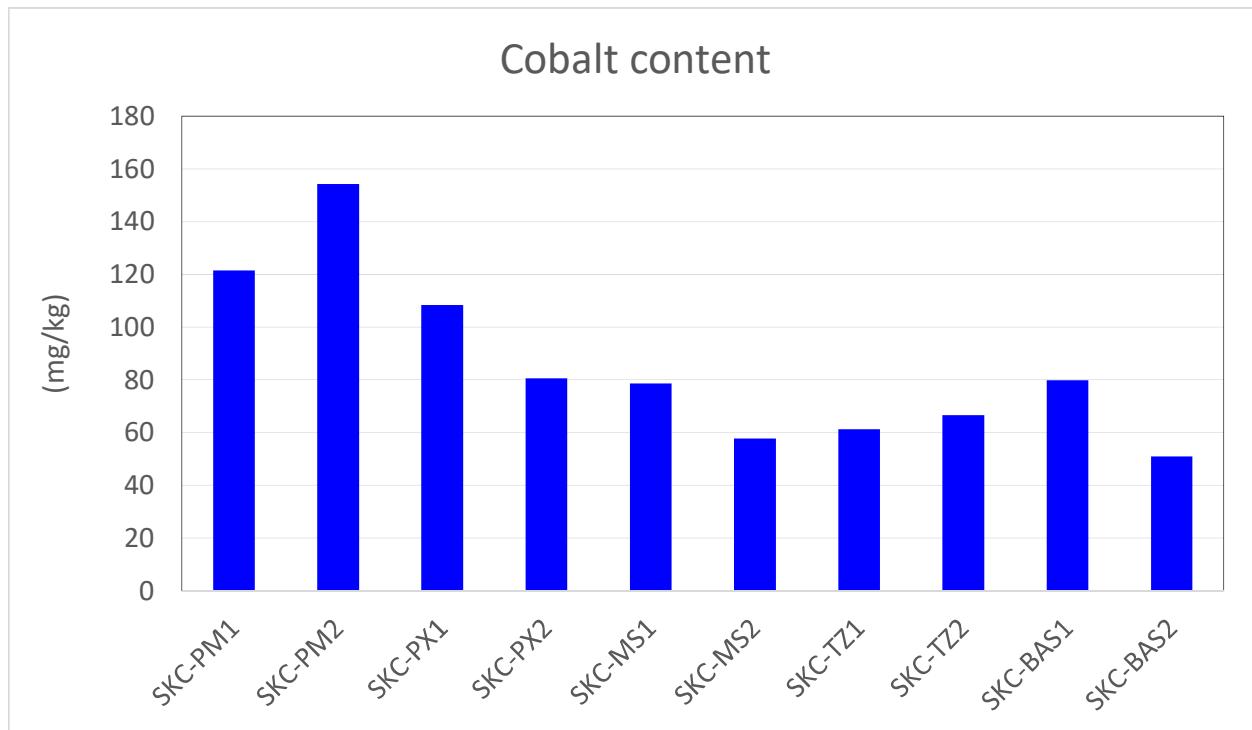


Figure 9. Cobalt content
Four-acid digestion and Multi-element analysis by ICP-OES-technique (Method 306P) (Michaux *et al* 2020)

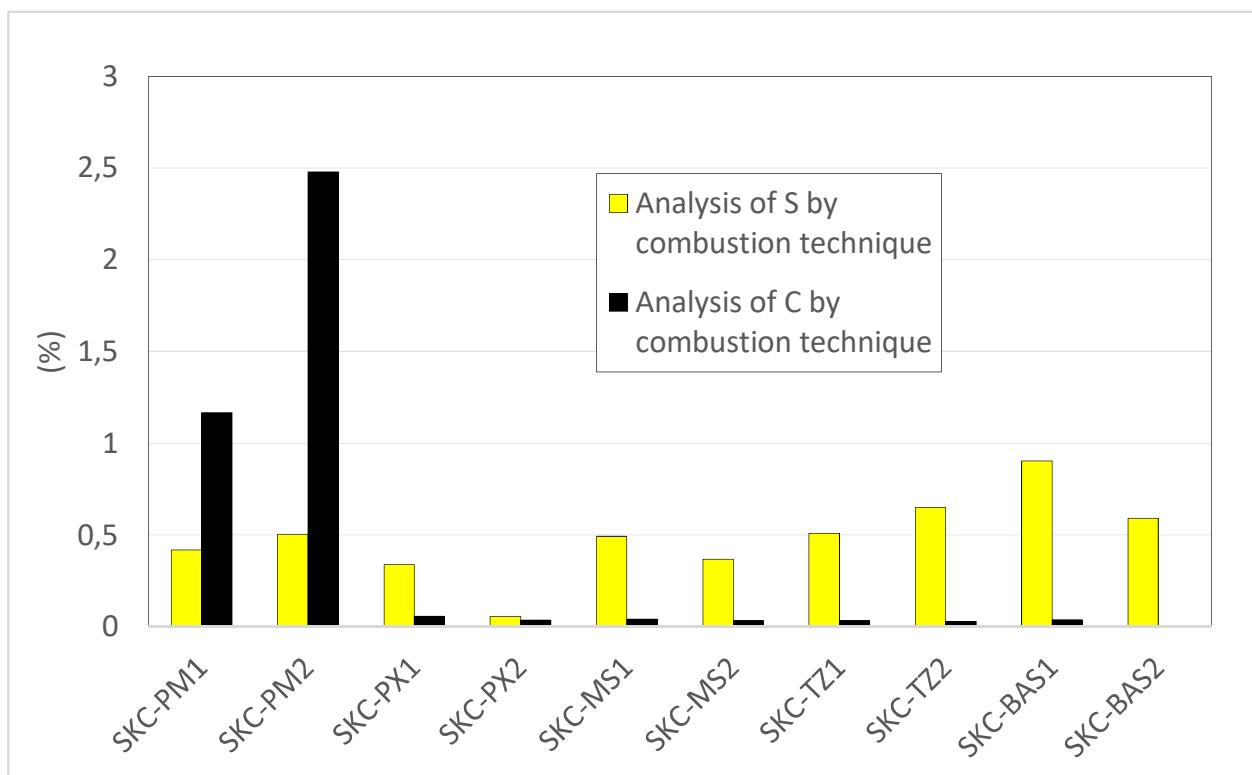


Figure 10. Determination of Sulphur (sulphur analyzer Method Eltra 810L) and Determination of carbon (carbon analyzer Method Eltra 811L) (Michaux *et al* 2020)

4.4 Bulk Element XRF Characterization of each Konttijärvi Sample Orientation Sample

The bulk mineralogy for the 10 SAP Orientation samples was characterized with XRF by X-Ray Minerals in the United Kingdom and Eurofins Labtium Oy in Finland (the XRF pellet method 180X was used). (see Appendix G)

XRF is an elemental analysis and can chemical elements are present and what are their concentrations. For example, the sample contains iron (Fe) and calcium (Ca) in a measured proportion. An X-ray fluorescence (XRF) spectrometer is an x-ray instrument used for routine, relatively nondestructive chemical analyses of rocks, minerals, sediments, and fluids. In a laboratory X-ray fluorescence (XRF) spectrometer the emission of characteristic "secondary" (or fluorescent) X-rays from a material that has been excited by being bombarded with high-energy X-rays (Fitton 1997, Potts 1987 and Rollinson 1993).

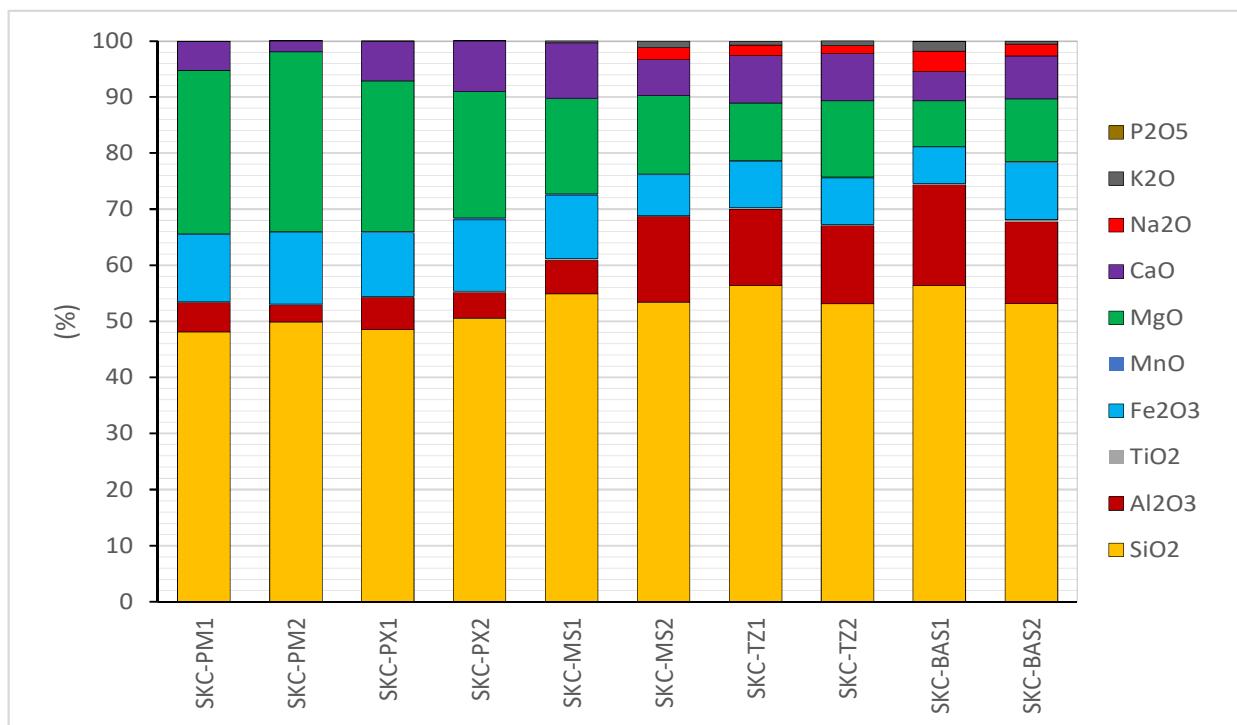


Figure 11. Mineral content from XRF Multi-element analysis (X-Ray Minerals) (Michaux et al 2020)

4.5 Semi Quantitative mineralogical analysis by X-Ray Diffraction (Q-XRD)

XRD analyses were conducted by X-Ray Mineral Services (Wales, UK). A whole-rock and clay fraction XRD 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 CuK α 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° (2θ), 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 BGMIN 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 (Chang-Zong *et al* 2015).

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 µm 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 CuK α radiation at 40 kV and 20/25 mA. Clay filters were scanned from 3 to 35° (2θ) at a step size of 0.05° and 2 s step time. 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%.

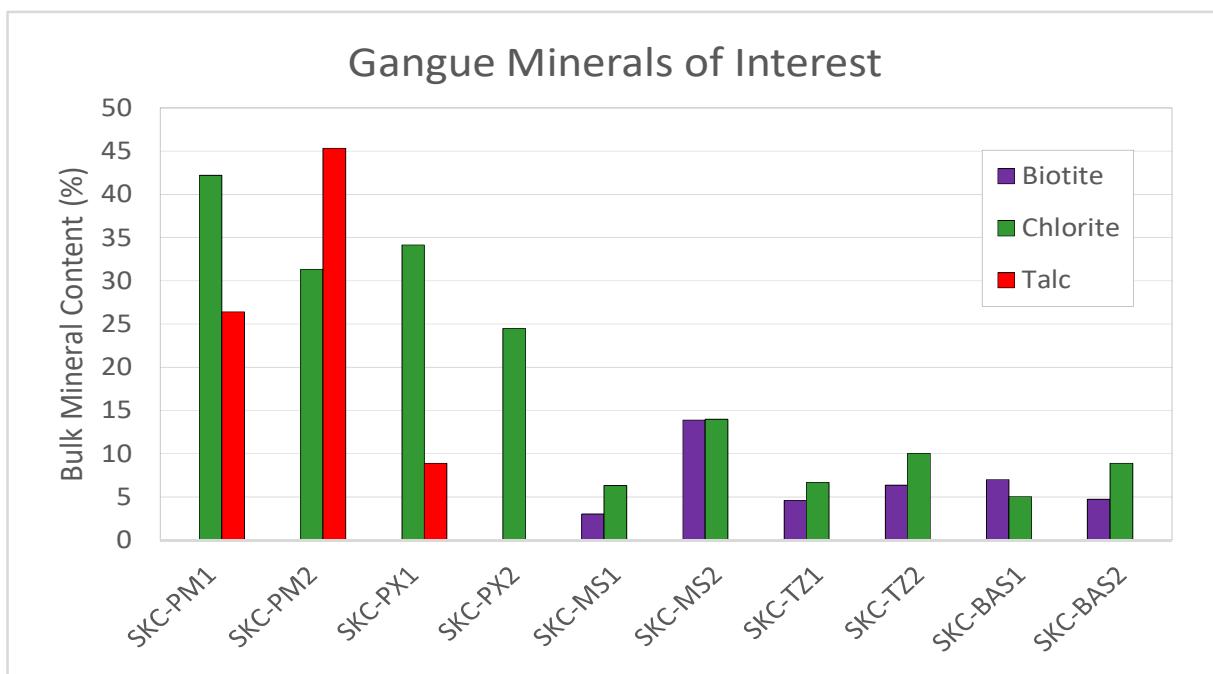


Figure 12. Gangue minerals of interest that may hamper recovery (XRD) (Michaux *et al* 2020)

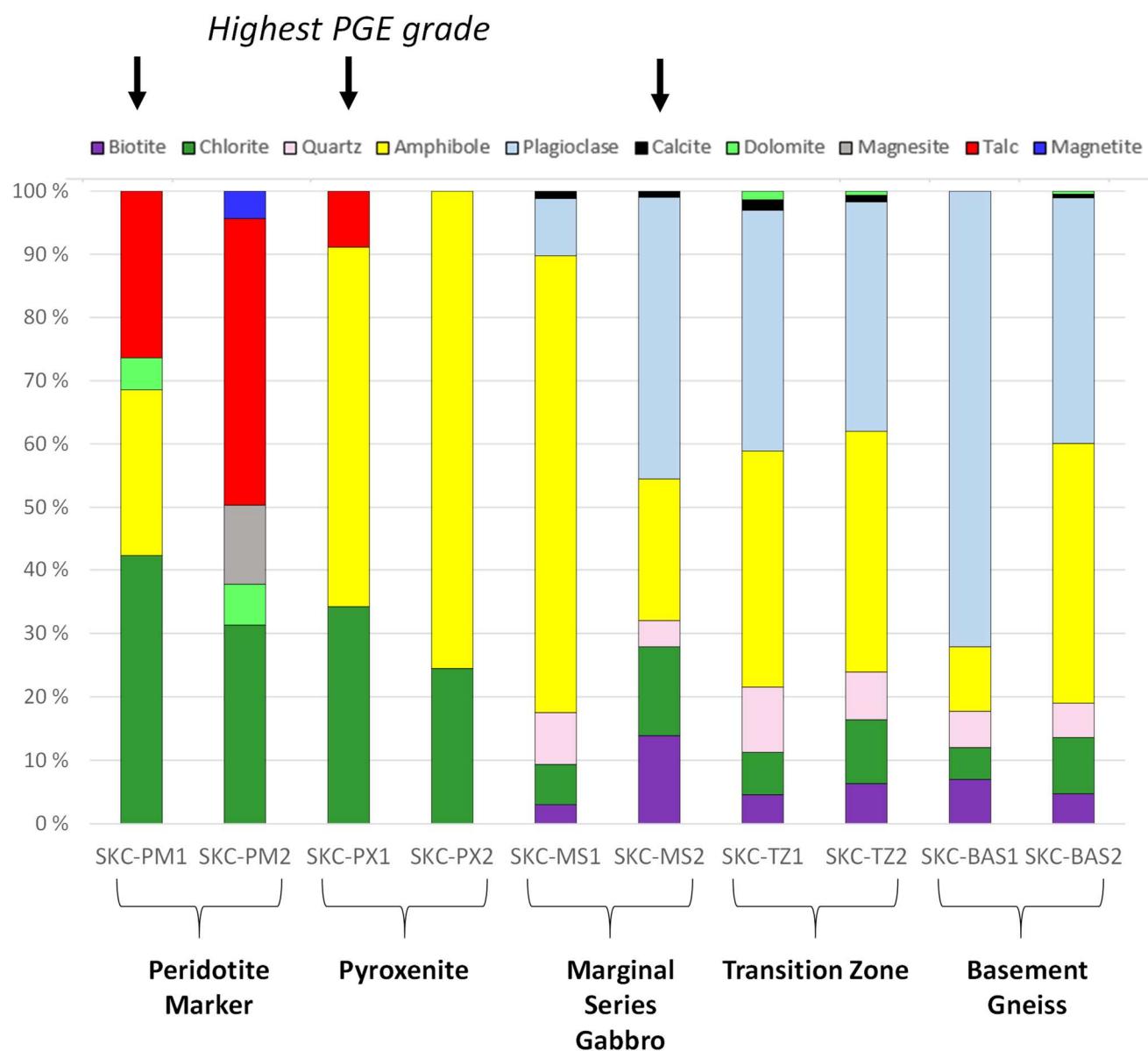


Figure 13. XRD measurement results for Konttijärvi Orientation Samples (Michaux et al 2020)

4.6 Automated Mineralogy

SEM Automated mineralogy is a sophisticated methodology that measures individual particles in context of what minerals are present and in what textural structure are they assembled together. The micro-texture of the 10 SAP Orientation samples was characterized using automated mineralogy in a Scanning Electron Microscope (SEM) by GK-Mintec in Finland (Michaux *et al* 2020).

Automated mineralogy as a characterization tool has advanced considerably. Reliable instrumentation, continuously updated software capabilities and faster data acquisition. Samples are mounted into polished resin blocks and are mapped using a scanning electron microscope (SEM). This method can be good for mapping micro-textures, but it is not as effective for gangue mineralogy in some cases, or with mineralogy with very similar back scattered electron grey scales. However, it is best practice to use complimentary analysis to establish good mineral chemistry with probe work and LA-ICP-MS or Raman, depending on the MOI (Hrstka *et al* 2018, Aylmore *et al* 2018, and Anderson *et al* 2014).

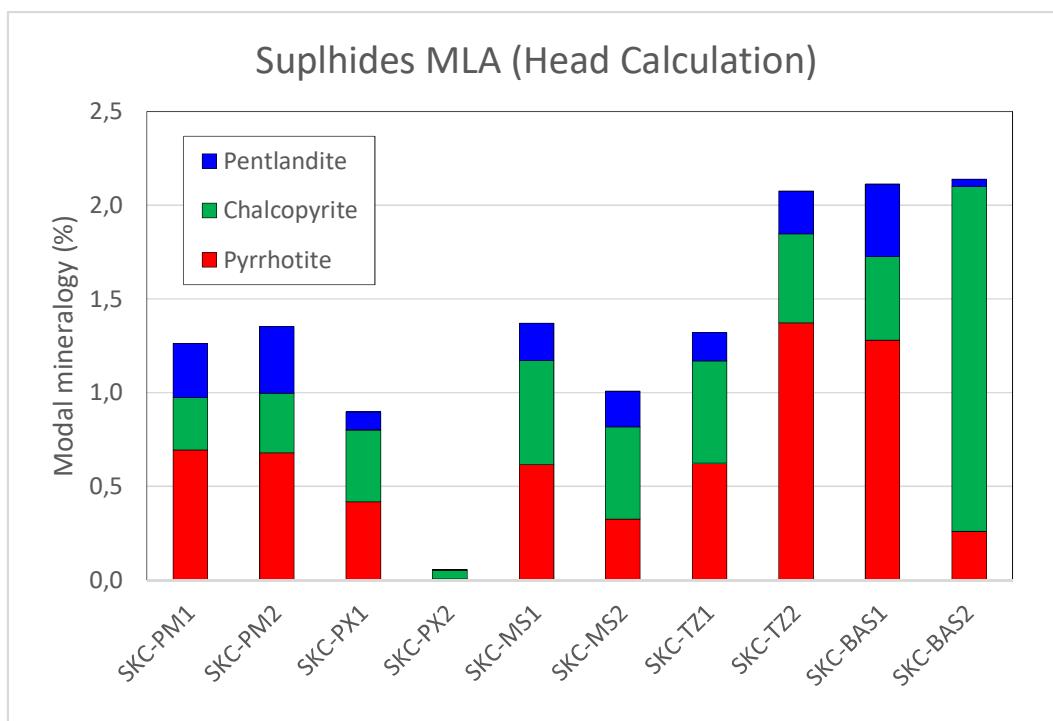


Figure 14. Sulphide minerals measured in Konttijärvi samples (MLA modal mineralogy head calculation) (Michaux *et al* 2020)

5 SAMPLE PREPARATION

Each Orientation Sample prepared for process separation test, was previously crushed to 99% passing 3.35 mm and then representatively split into sub products (Michaux *et al* 2020). To prepare each sub-product, further grinding in a rod mill was required to achieve appropriate liberation. Based on past GTK experience with this deposit and examination of the MLA data (Michaux *et al* 2020), a target grind size (P_{80}) of 75 micron was selected. Each sample was put in a rod mill in a set procedure, and sub-sampled until the target P_{80} was reached. Each product was then either directly separated or placed in a freezer to reduce oxidization. Appendix C shows this sample grinding data.

6 MAGNETIC SEPARATION

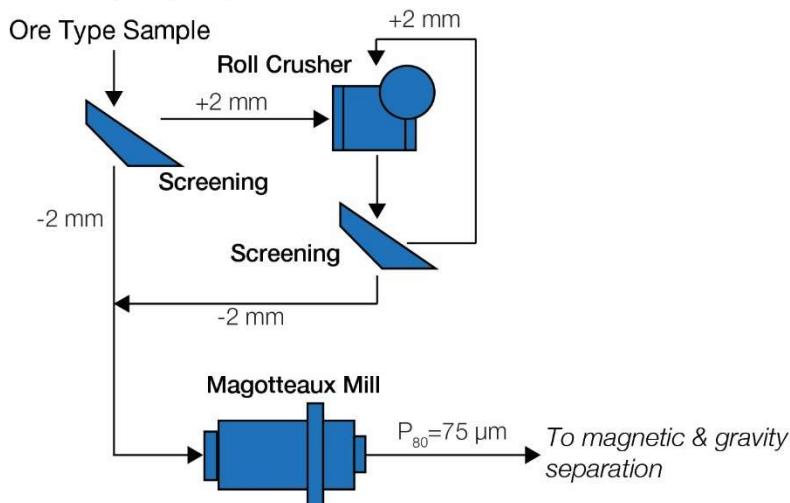
6.1 Methods/Procedure

Magnetic separation tests on the 10 ore types have been performed at the University of Liege, Belgium. The sample preparation involved crushing and milling of each sample to reach a P_{80} of 75 μm , following the flowsheet in [Error! Reference source not found.](#)¹⁶. The magnetic separation tests were carried out on a Wet High Intensity Magnetic Separator from Carpcor Inc (model CC WHIMS 2X 4XL) equipped with grooved plates as matrix and operated at 25 wt% solid with samples of 1 kg each. The samples were processed following the flowsheet in [Error! Reference source not found.](#)¹⁷, 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 15. Photographs of the magnetic separation test work.

a. Sample preparation



b. Magnetic separation

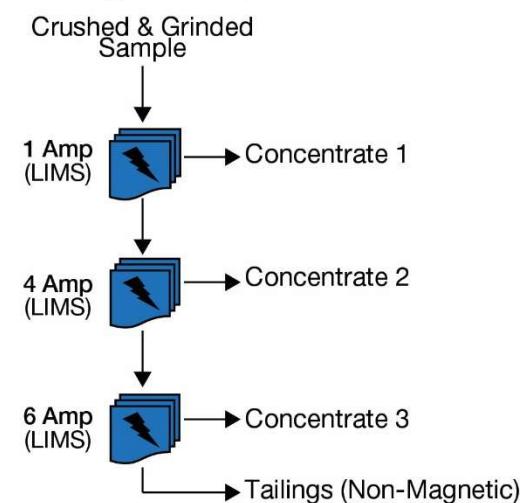


Figure 16. Flowsheets showing the (a) sample preparation procedure and (b) magnetic separation procedure

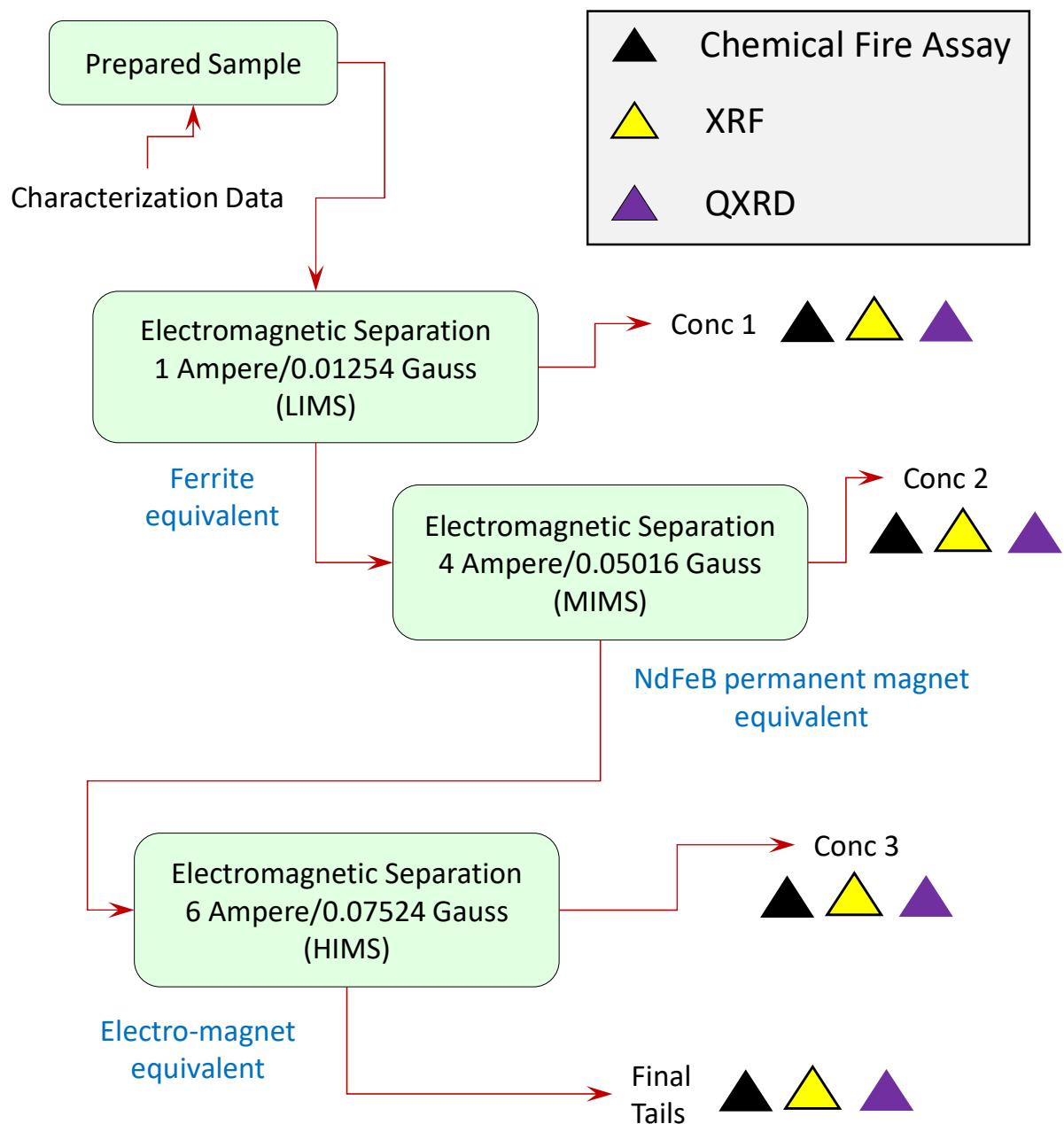


Figure 17. Magnetic separation experimental flowsheet

6.2 Magnetic Separation Results

Figure 18 and Table 1 show the mass pull for all the magnetic fractions for each ore type. Ore types PM1 and PM2 have a mass of minerals susceptible for magnetic separation of 28% and 35% respectively. The remaining ore types are Approximately 10% or lower.

Magnetic separation data is shown in Appendix D.

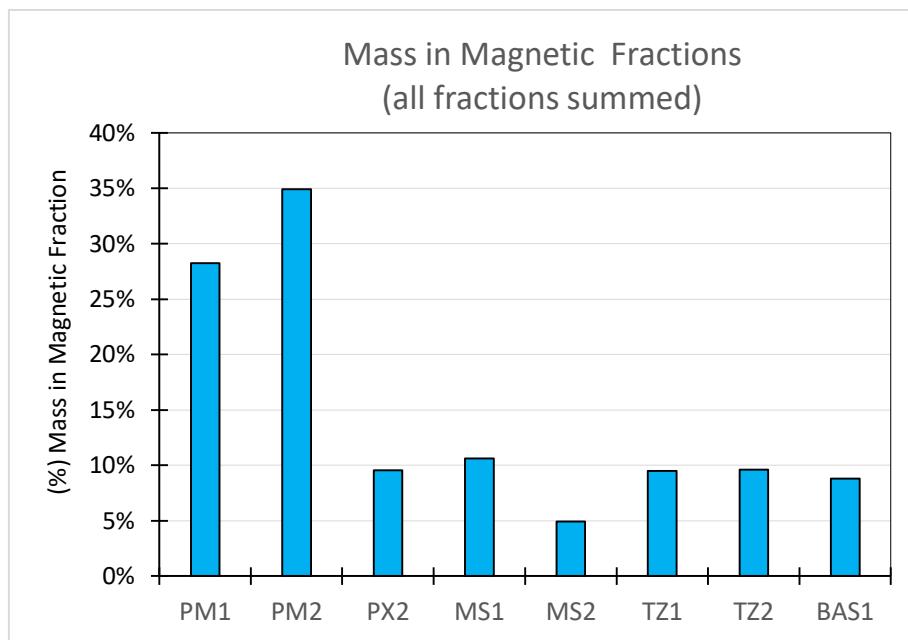


Figure 18. Mass separated with magnetic separation

Table 1. Magnetic separation mass pull in product fractions

MAGNETIC SEPARATION MASS PULL	PM1 (%)	PM2 (%)	PX1 (%)	PX2 (%)	MS1 (%)	MS2 (%)	TZ1 (%)	TZ2 (%)	BAS1 (%)	BAS2 (%)
1 Amp	4.66%	23.12%	-	1.18%	1.39%	1.11%	1.38%	2.94%	3.34%	-
4 Amp	21.85%	6.39%	-	3.86%	3.39%	1.68%	2.33%	3.13%	2.73%	-
6 Amp	1.75%	5.42%	-	4.54%	5.86%	2.16%	5.76%	3.56%	2.72%	-
Non-Mag	71.74%	65.08%	-	90.43%	89.36%	95.06%	90.52%	90.37%	91.21%	-

The bulk mineralogy of the magnetic separation products for each ore type obtained through semi-quantitative XRD is shown in Figure 19 to 31. The magnetic fractions are mostly composed of chlorite, amphibole, talc, and plagioclase, and to a lesser extent dolomite, quartz, magnesite and pyrrhotite.

The proportion of each mineral varies depending on the product and the ore type but globally there is no clear mineral separation pattern, where one mineral type is dominant in any particular fraction.

Mineral recoveries for the gangue minerals considered problematic are shown in Figures 19 to 22. 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.

Pyrrhotite is recovered in the 1 amp concentrate. From the MLA work done, pyrrhotite is often associated with precious metals, which could be the reason why precious metals are reporting to magnetic fractions in Figures 19 to 31. Notes for Figures 19 to 31:

1. Plagioclase 1 is probably andesine
2. Plagioclase 2 is probably albite
3. Chlorite is clinochlore
4. Amphibole 1 is probably actinolite
5. Amphibole 2 is probably anthophyllite-gedrite
6. Samples SKC-PM1 and SKM-PM1 contain a second species of amphibole that could not be quantified separately (probably cummingtonite)
7. Traces of graphite possibly present in samples SKC-MS1, MS2, TZ1, TZ2, BAS1 and BAS2

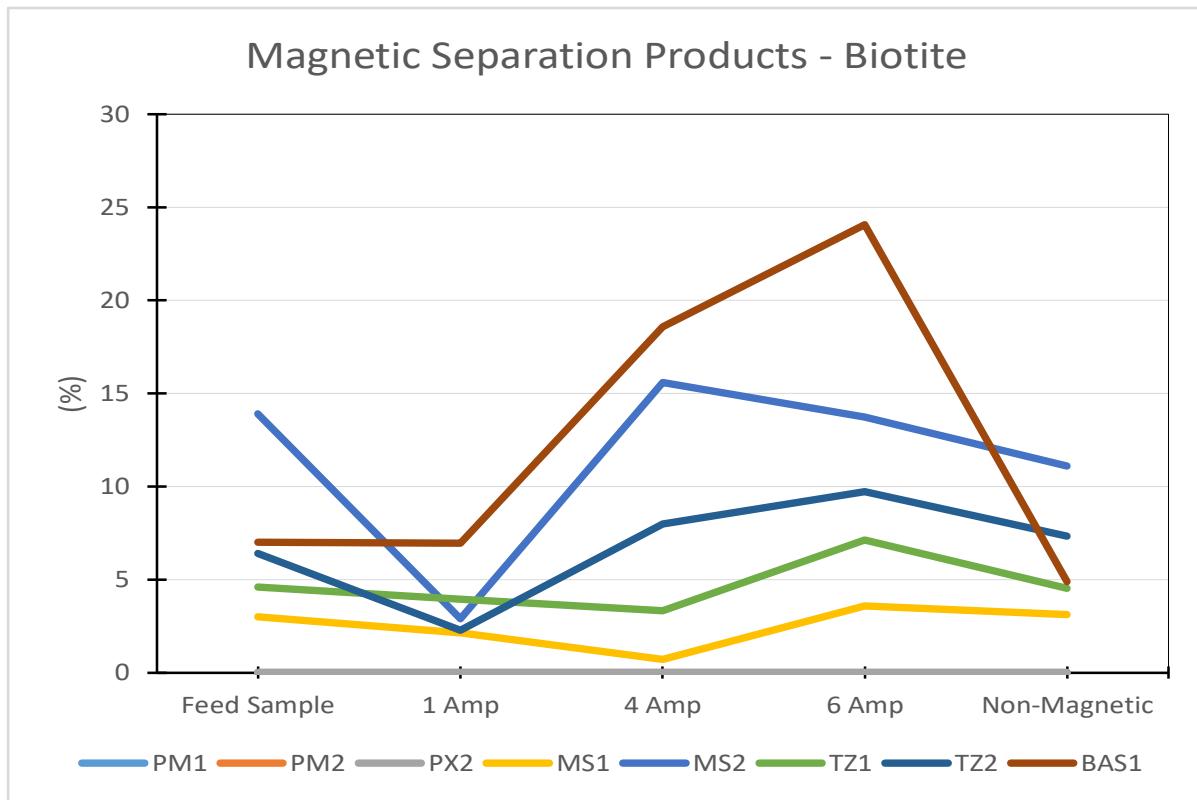


Figure 19. Semi-quantitative mineralogy (QXRD) of the magnetic products separation products Biotite

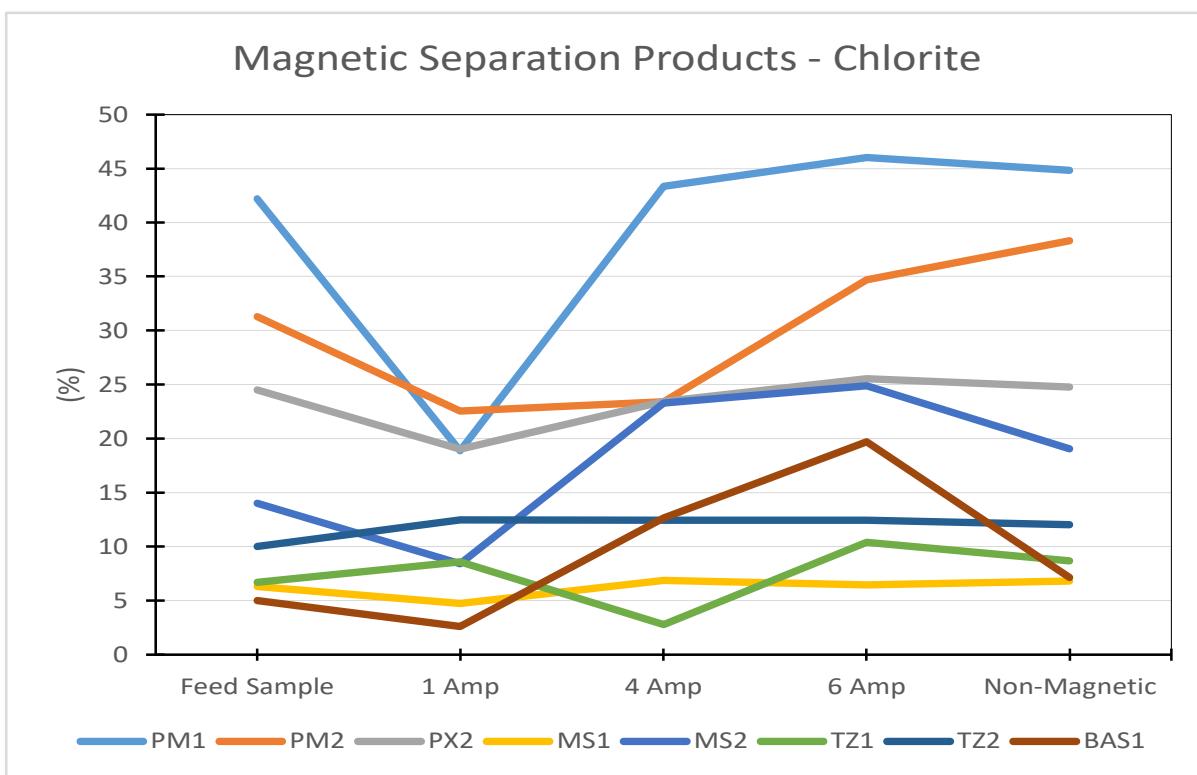


Figure 20. Semi-quantitative mineralogy (QXRD) of the magnetic products separation products Chlorite

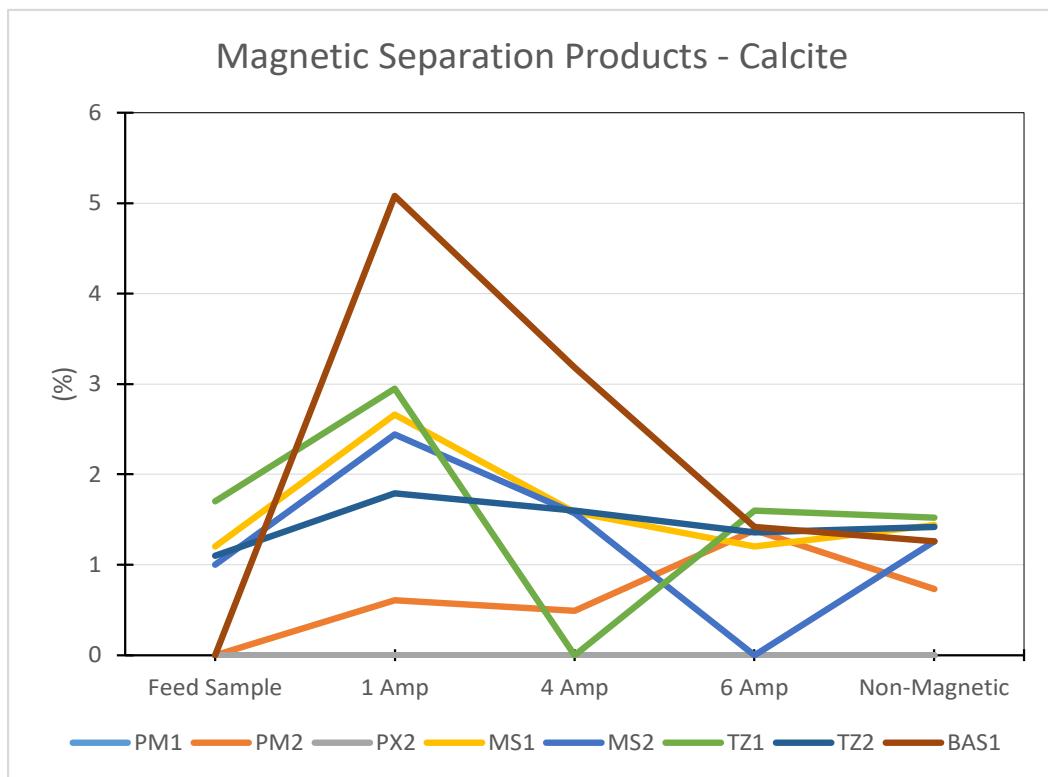


Figure 21. Semi-quantitative mineralogy (QXRD) of the magnetic products separation products Calcite

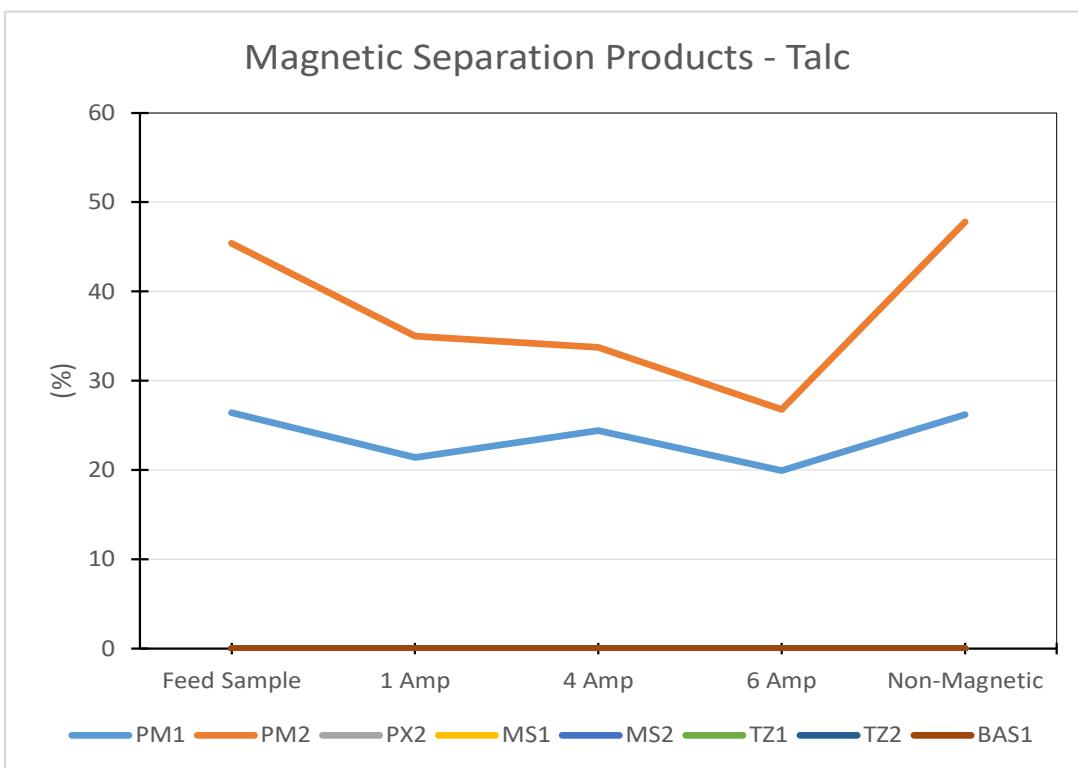


Figure 22. Semi-quantitative mineralogy (QXRD) of the magnetic products separation products Talc

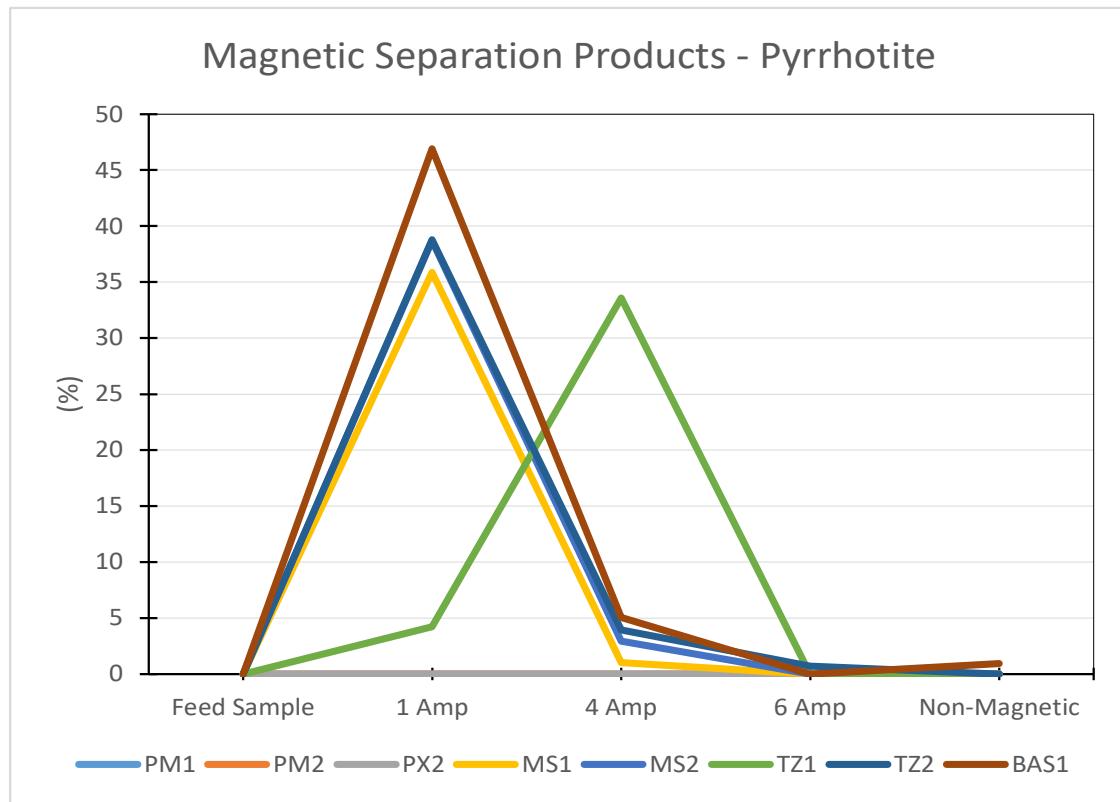


Figure 23. Semi-quantitative mineralogy (QXRD) of the magnetic products separation products Pyrrhotite

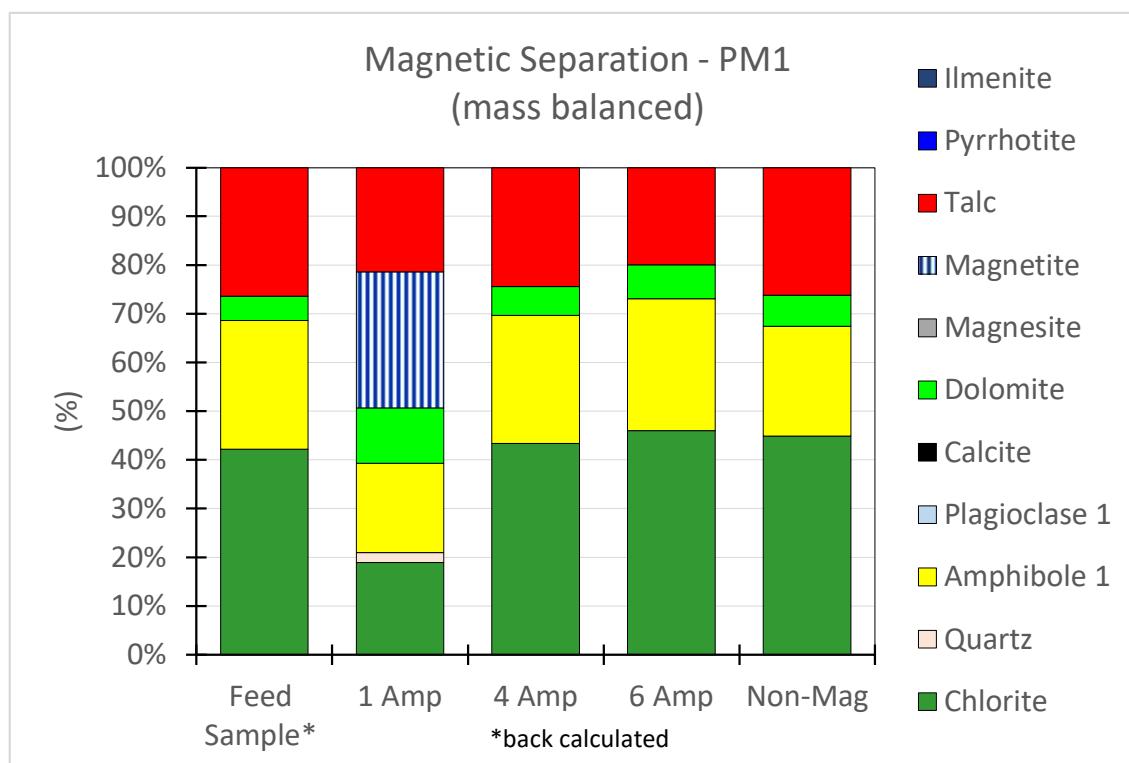


Figure 24. Semi-quantitative mineralogy (QXRD) of the magnetic separation products PM1

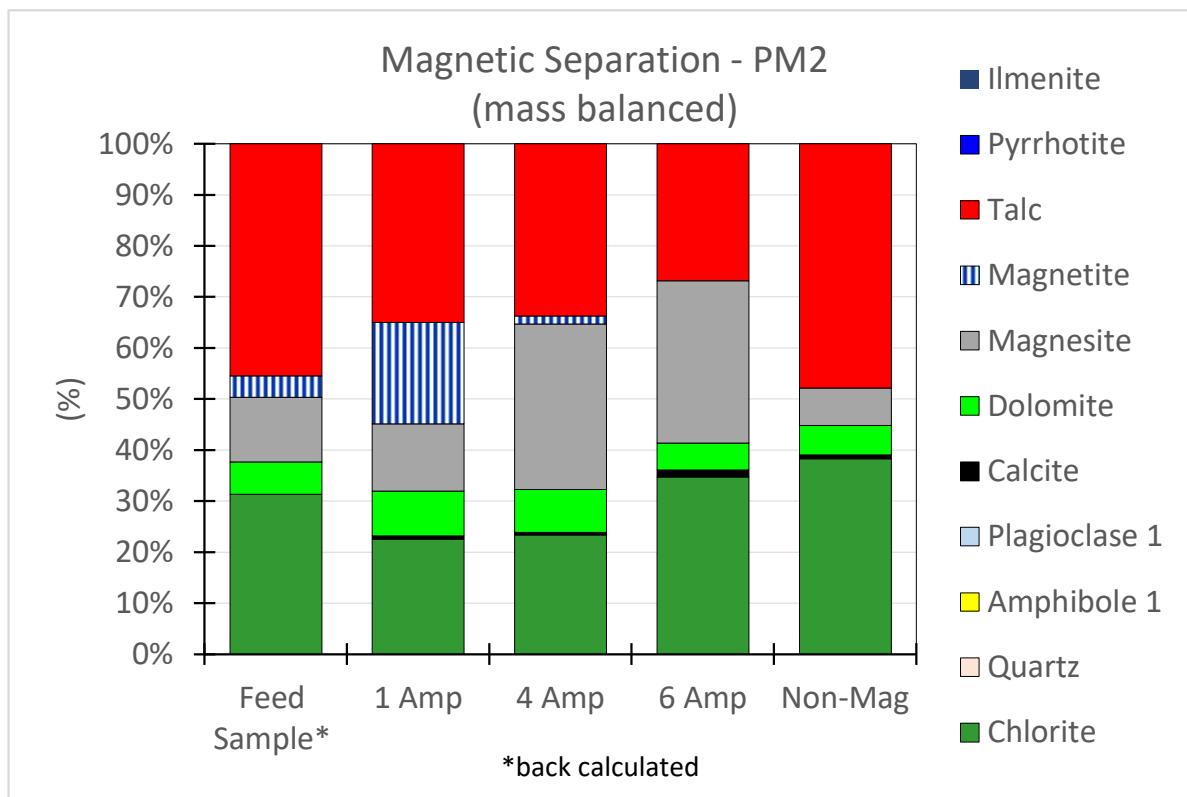


Figure 25. Semi-quantitative mineralogy (QXRD) of the magnetic separation products PM2

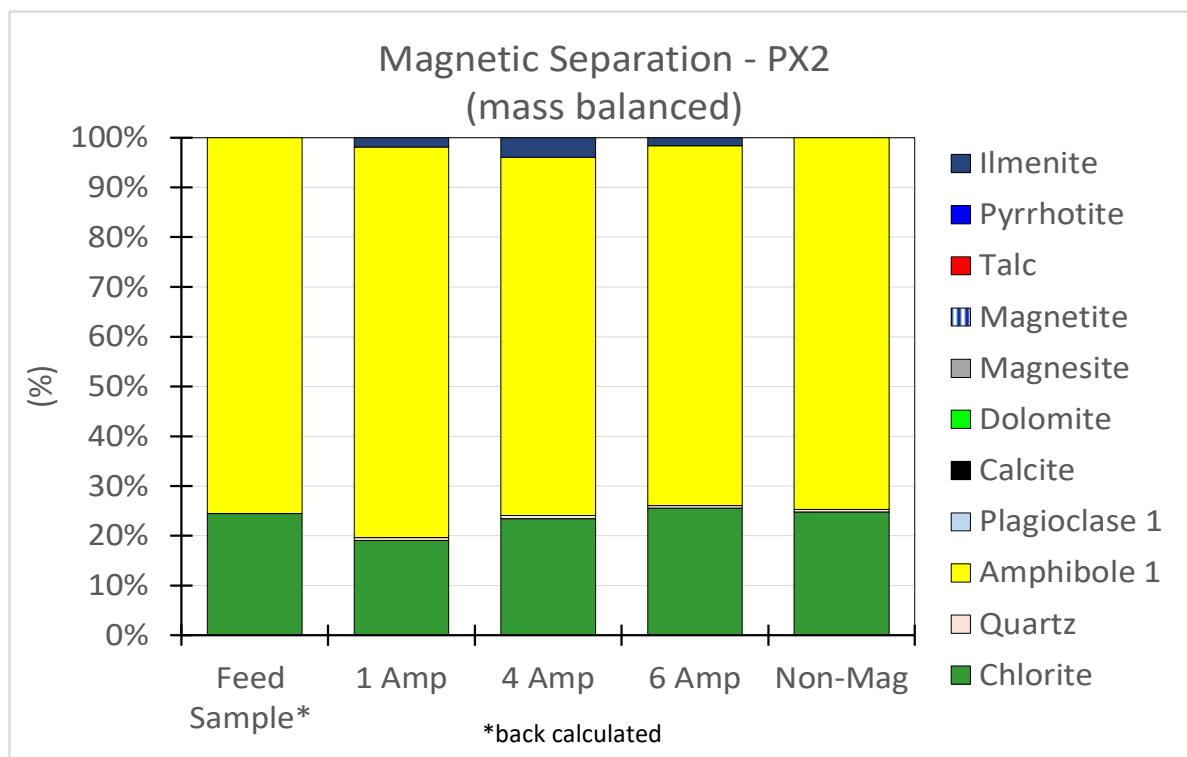


Figure 26. Semi-quantitative mineralogy (QXRD) of the magnetic separation products PX2

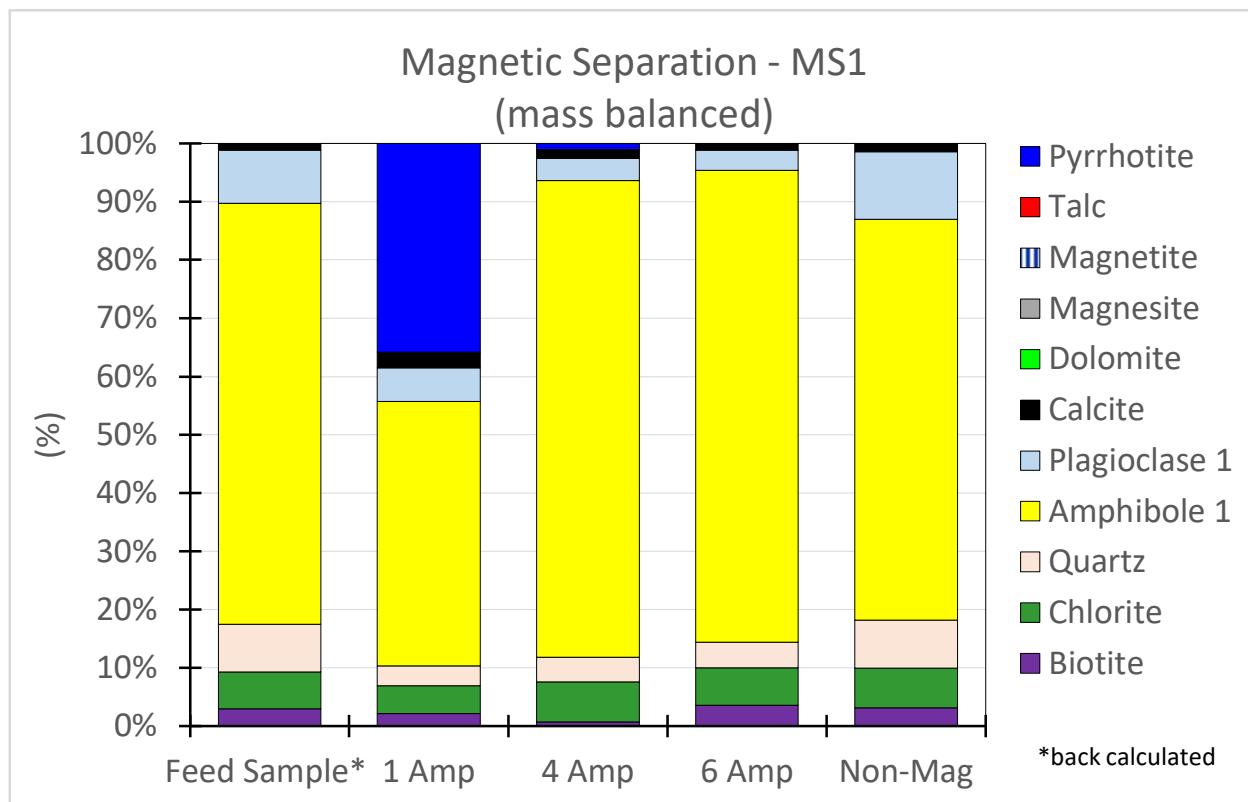


Figure 27. Semi-quantitative mineralogy (QXRD) of the magnetic separation products MS1

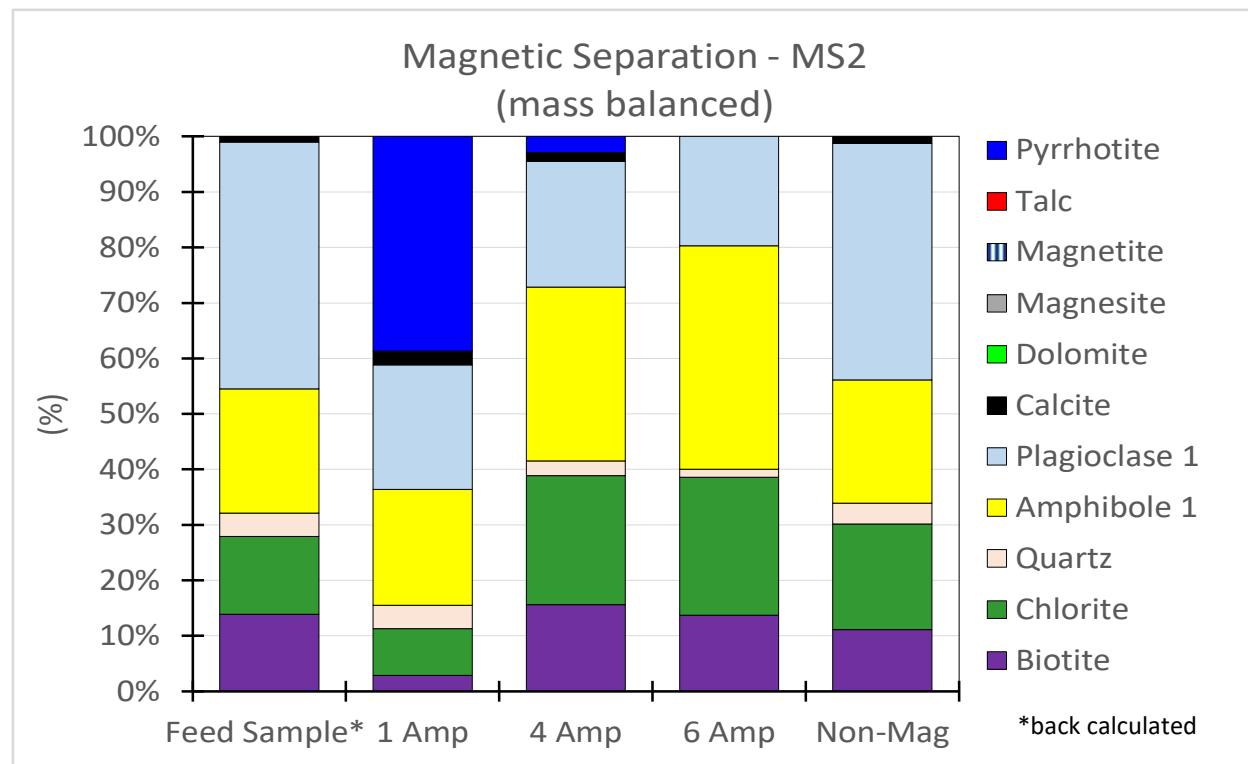


Figure 28. Semi-quantitative mineralogy (QXRD) of the magnetic separation products MS2

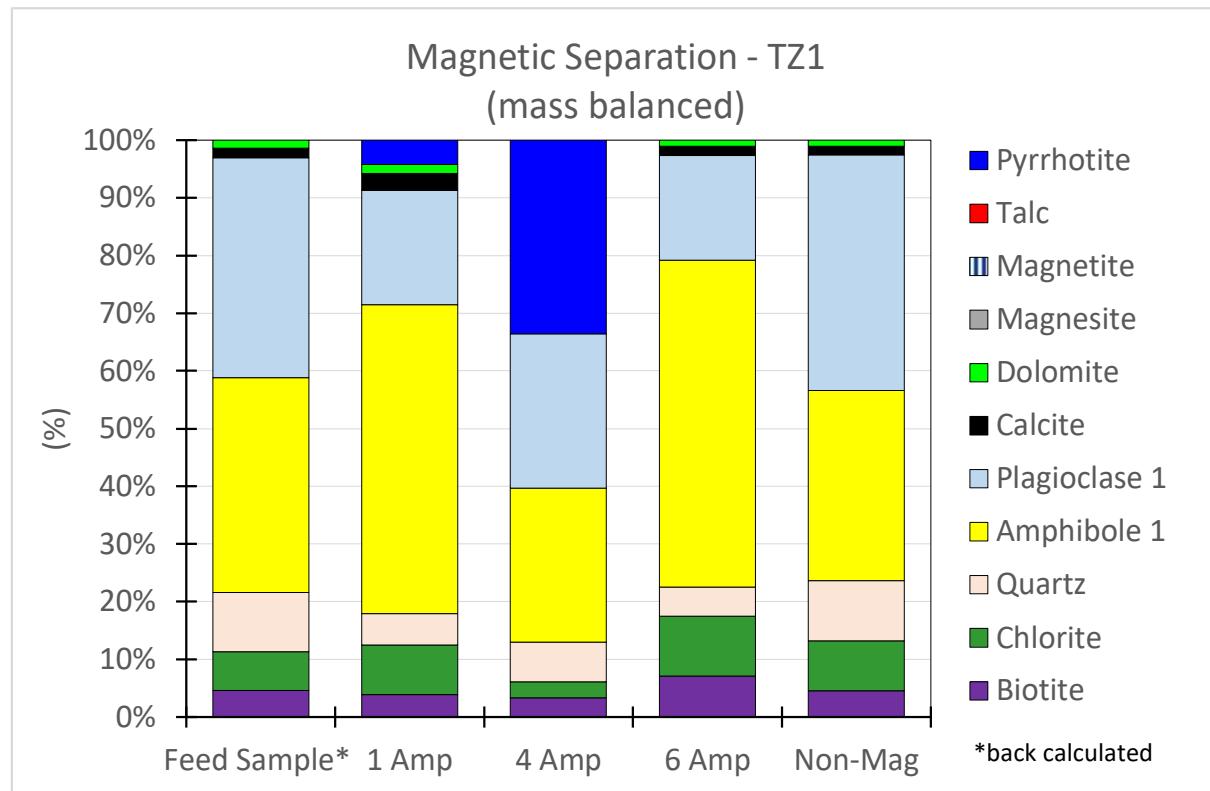


Figure 29. Semi-quantitative mineralogy (QXRD) of the magnetic separation products TZ1

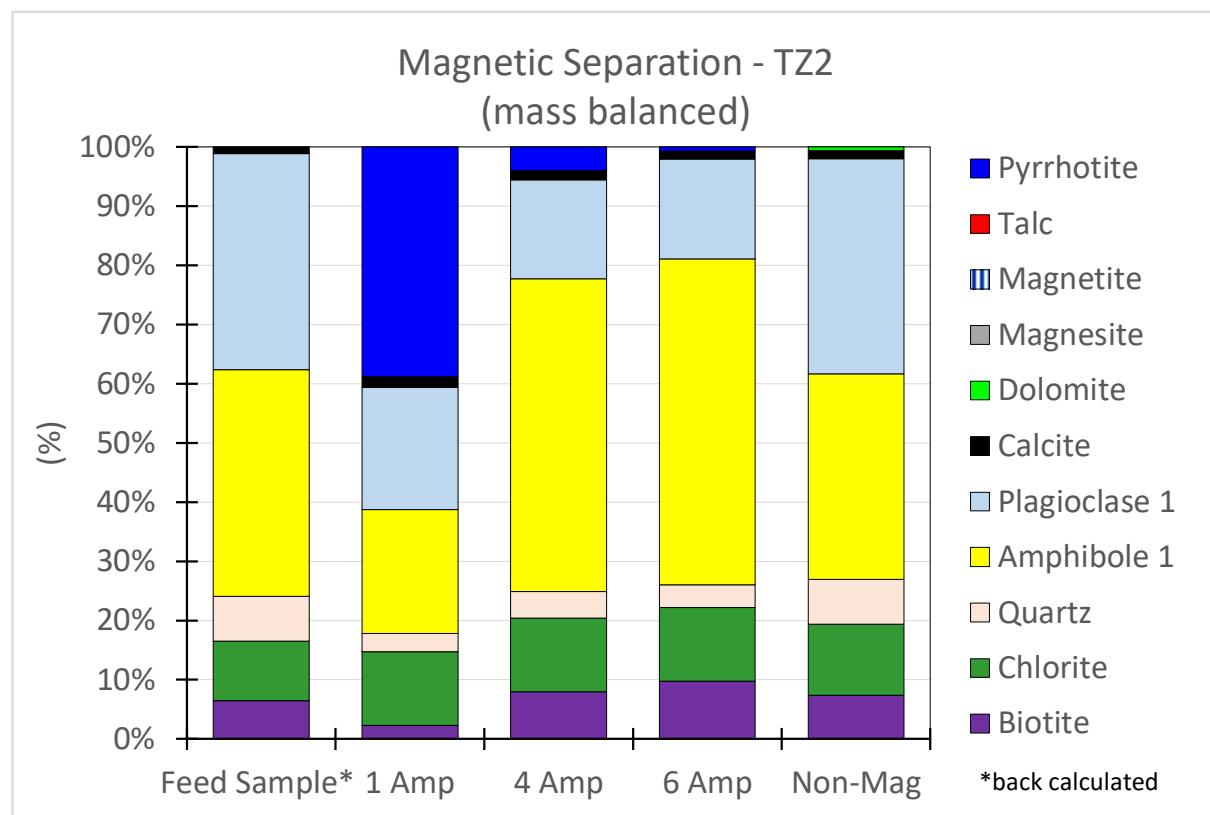


Figure 30. Semi-quantitative mineralogy (QXRD) of the magnetic separation products TZ2

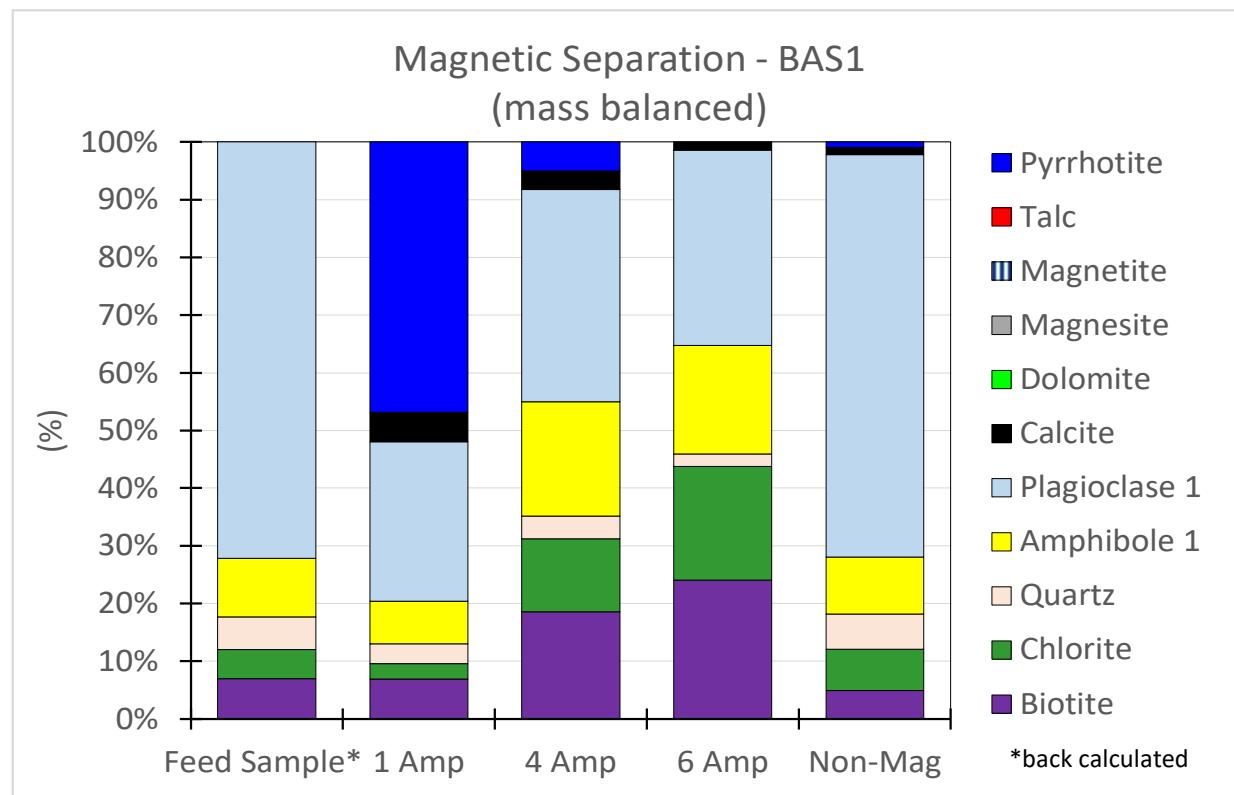


Figure 31. Semi-quantitative mineralogy (QXRD) of the magnetic separation products BAS1

Figure 32 to 47 shows the outcomes of an analysis done with Spotfire software (by John Martin, Hafren Scientific). Patterns in the magnetic separation data for everything collected was examined and used to create these charts, including chemical assays (fire assays method 711P, Eltra S, Eltra C and ICP method 306P), QXRD and XRF.

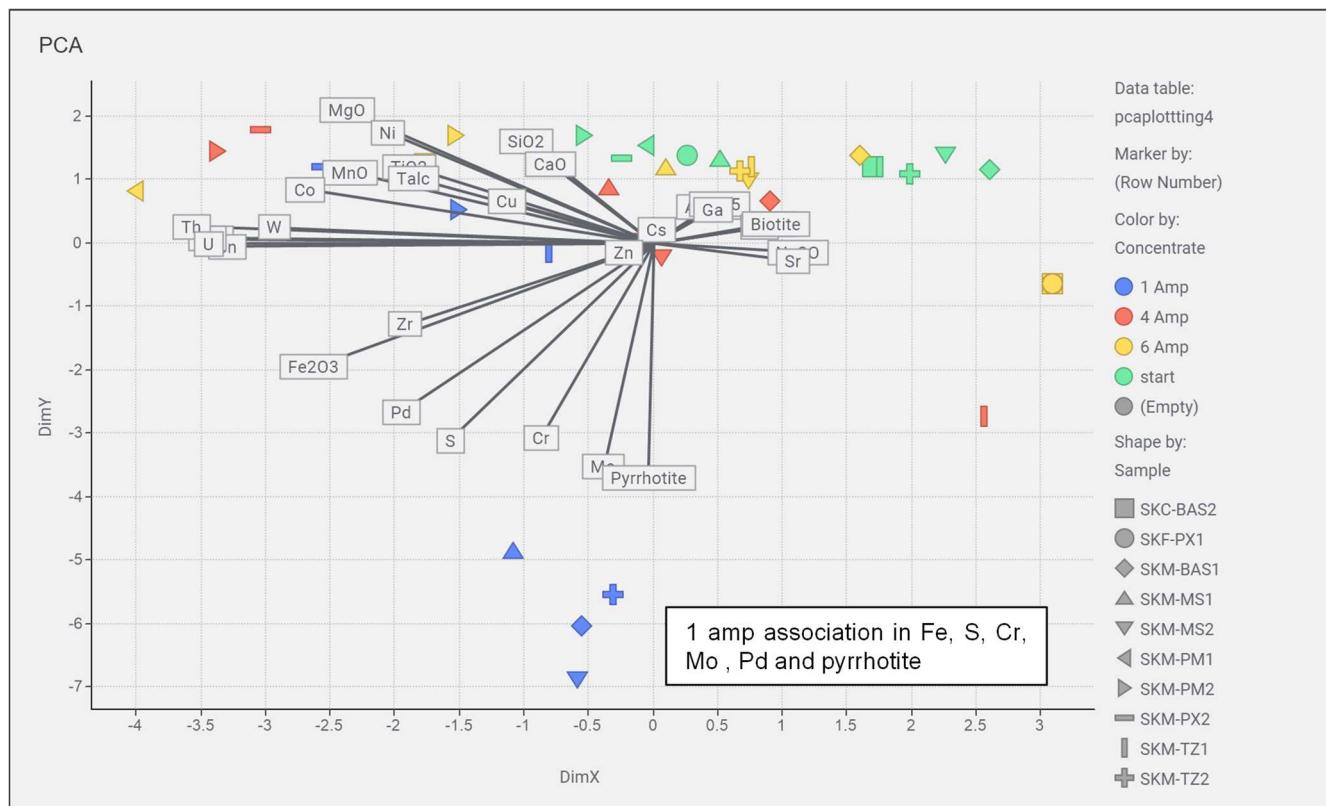


Figure 32. Principle Component Analysis of magnetic separation data
(Analysis done with Spotfire software by John Martin, Hafren Scientific)

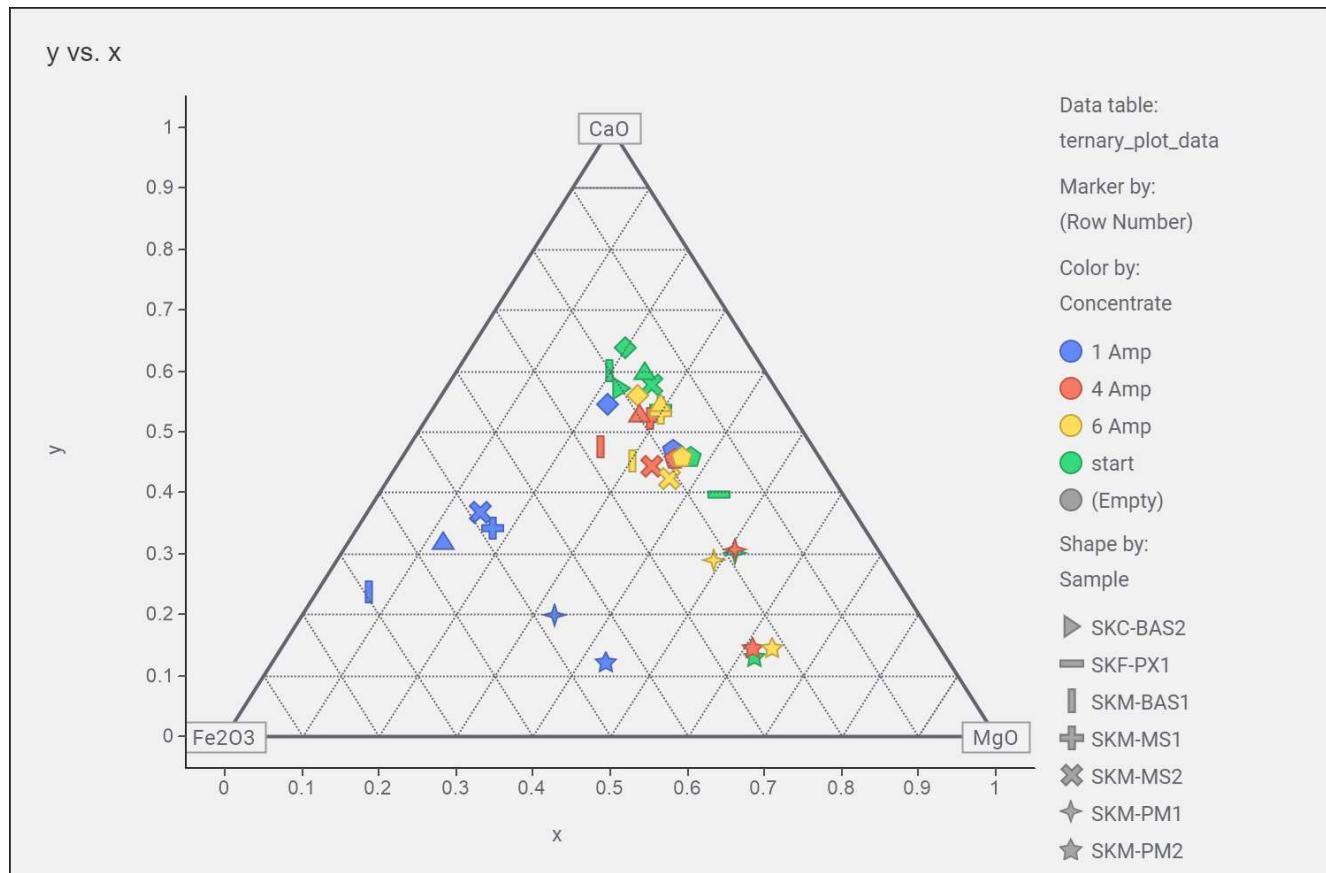
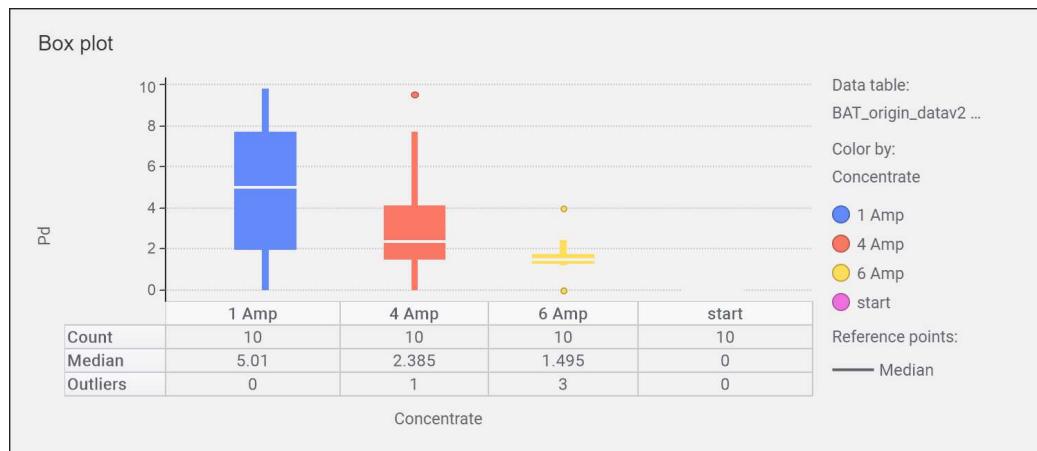


Figure 33. Magnetic separation recovery, on a trinary plot between Fe_2O_3 , MgO and CaO
(Analysis done with Spotfire software by John Martin, Hafren Scientific)



*Figure 34. Palladium recovery from magnetic separation (all samples)
(Analysis done with Spotfire software by John Martin, Hafren Scientific)*

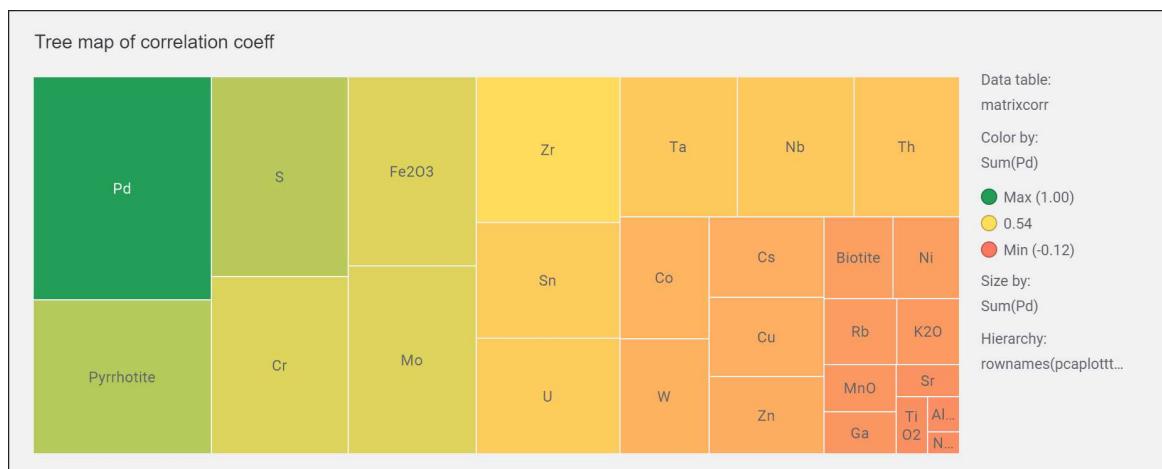


Figure 35. Tree map of correlation coefficient for Palladium recovery to characterization elements in feed samples (all samples) (Analysis done with Spotfire software by John Martin, Hafren Scientific)

Figures 36, 39, 42 and 45 show an analysis of the patterns observed in the magnetic separation within individual samples.

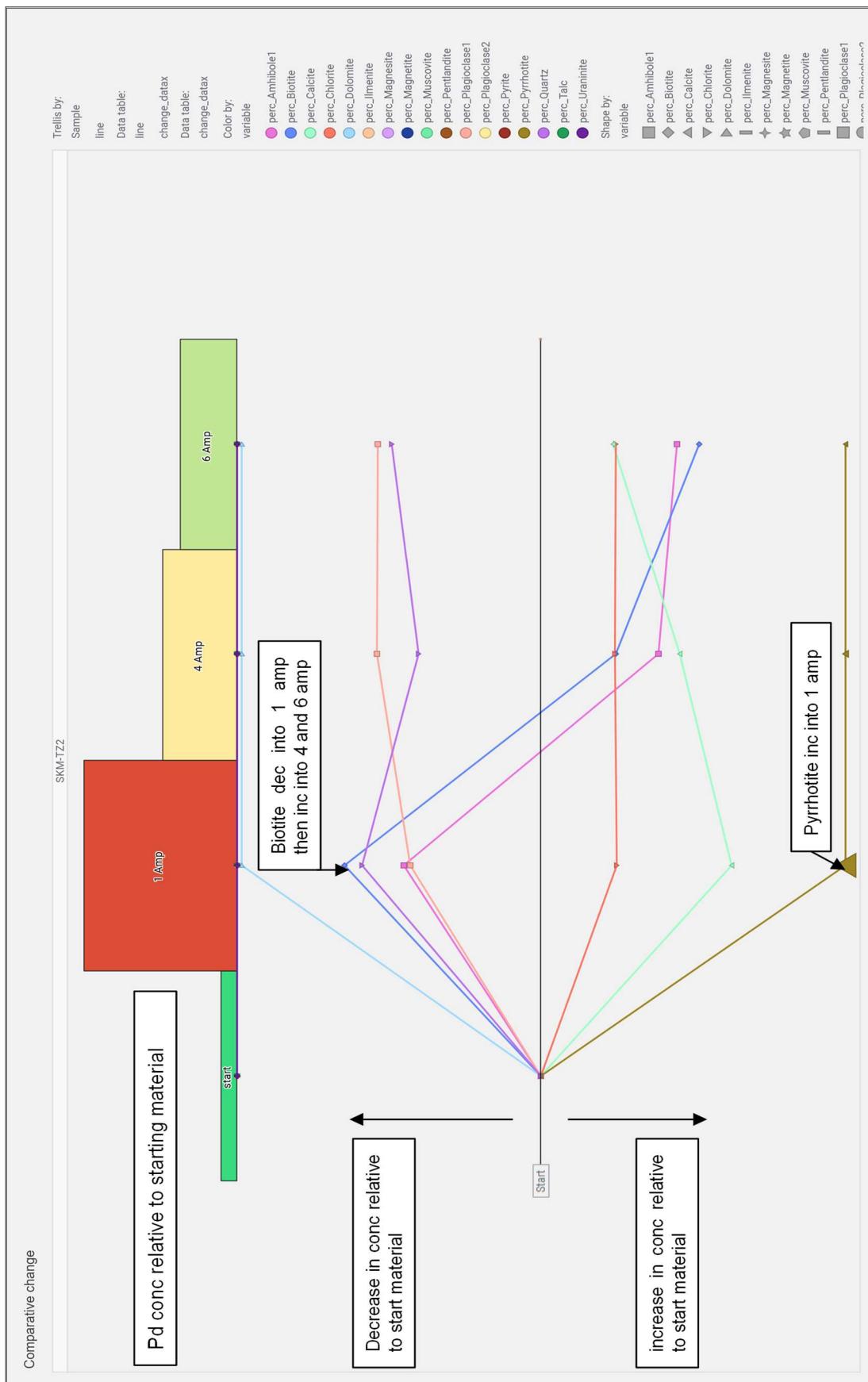


Figure 36. Element data relationship relative to Palladium concentrate for sample SKM-TZ2
(Analysis done with Spotfire software by John Martin, Hafren Scientific)

The deportation of each element to the different magnetic separation fractions (1 Amp, 4 Amp, 6 Amp and Non-magnetic) follow different patterns. Figures 37, 40, 43 and 46 show all the deportation of the measured elements into 5 clusters.

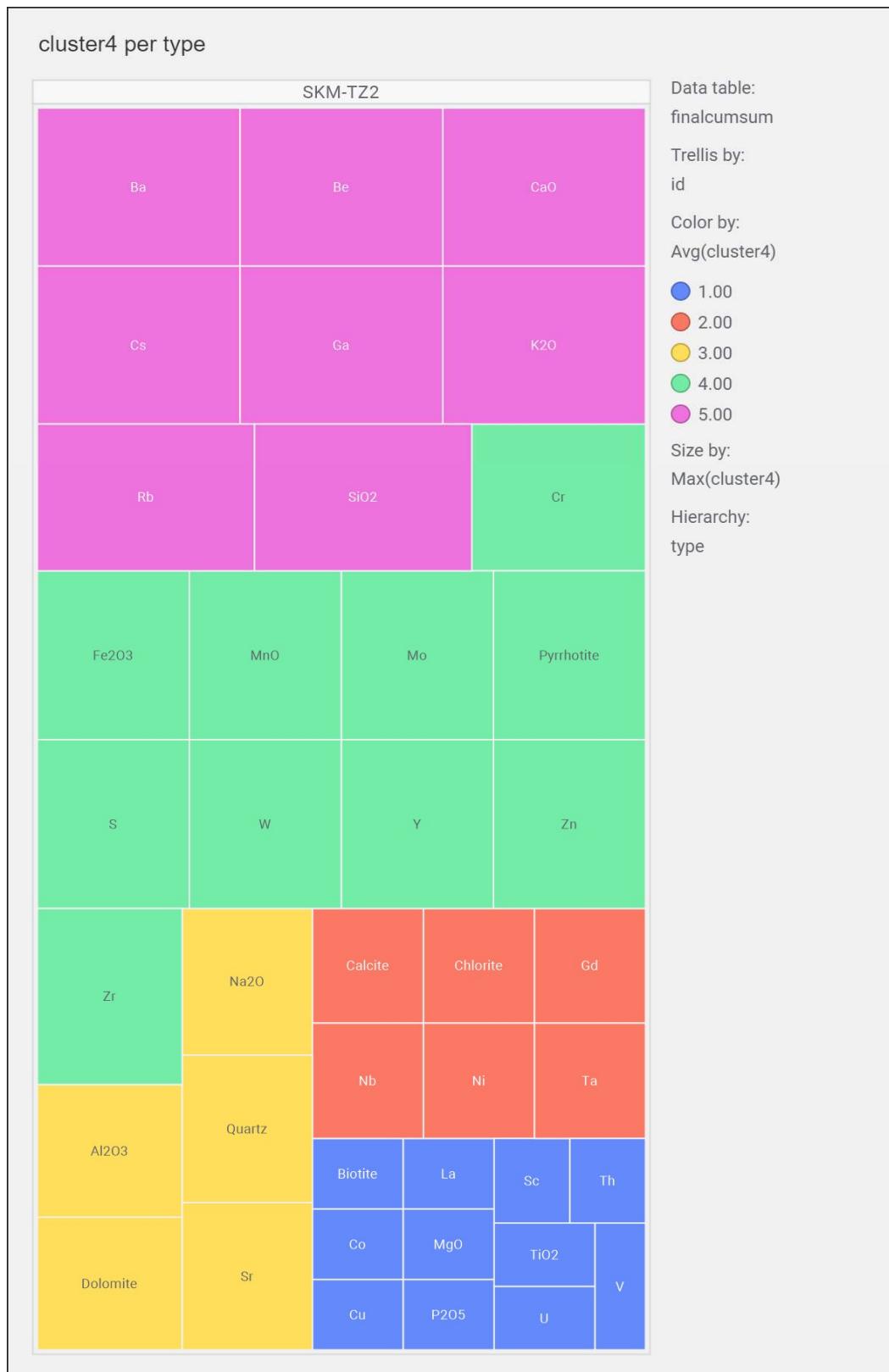


Figure 37. Element data correlation coefficient tree diagrams for elements across for magnetic separation fractions for sample SKM-TZ2 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

Figures 38, 41, 44 and 47 show how these five different clusters present with different forms. It is postulated that each cluster represents a different mineral grouping.

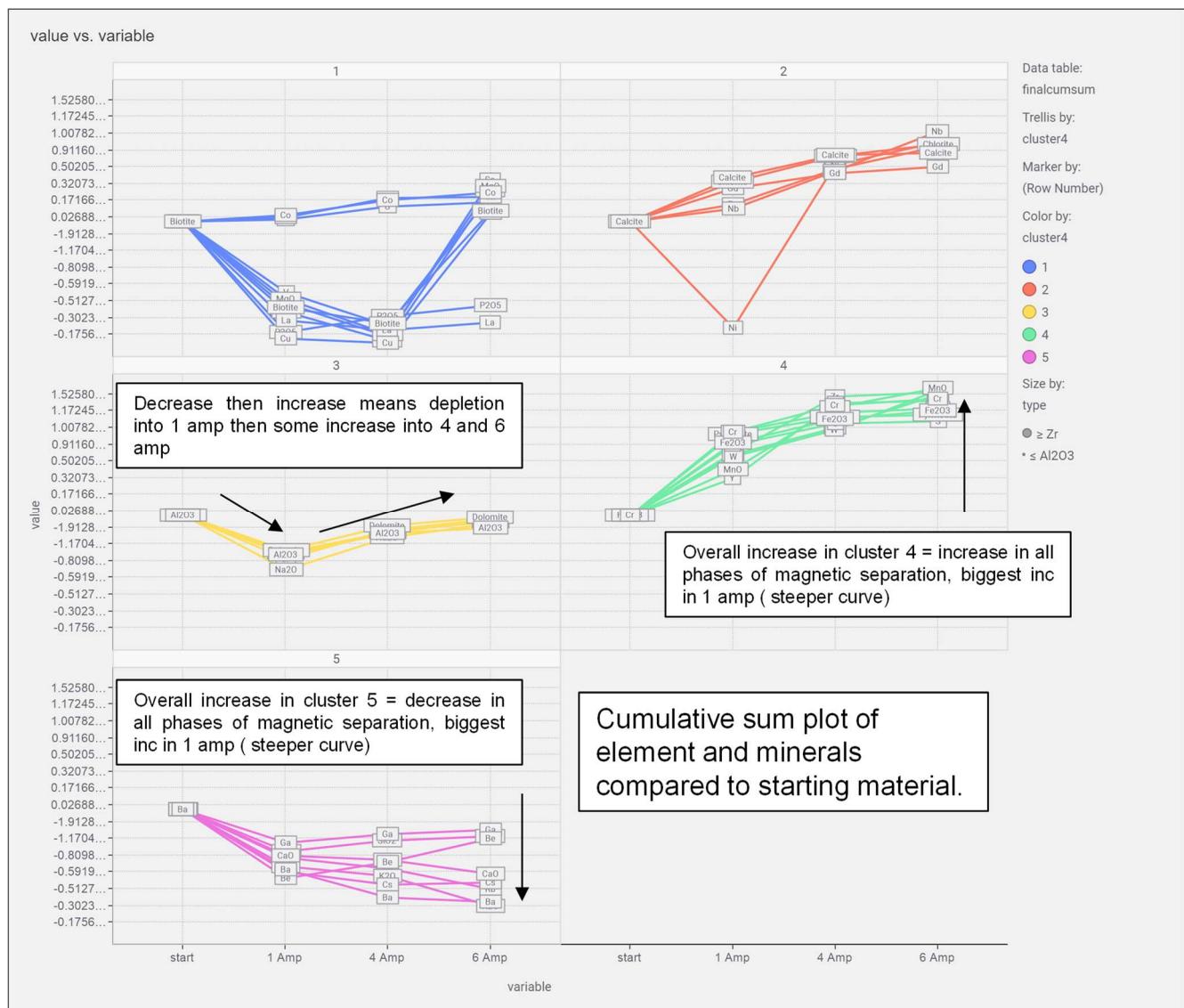


Figure 38. Element data relationship tree diagrams for elements across for magnetic separation fractions for sample SKM-TZ2 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

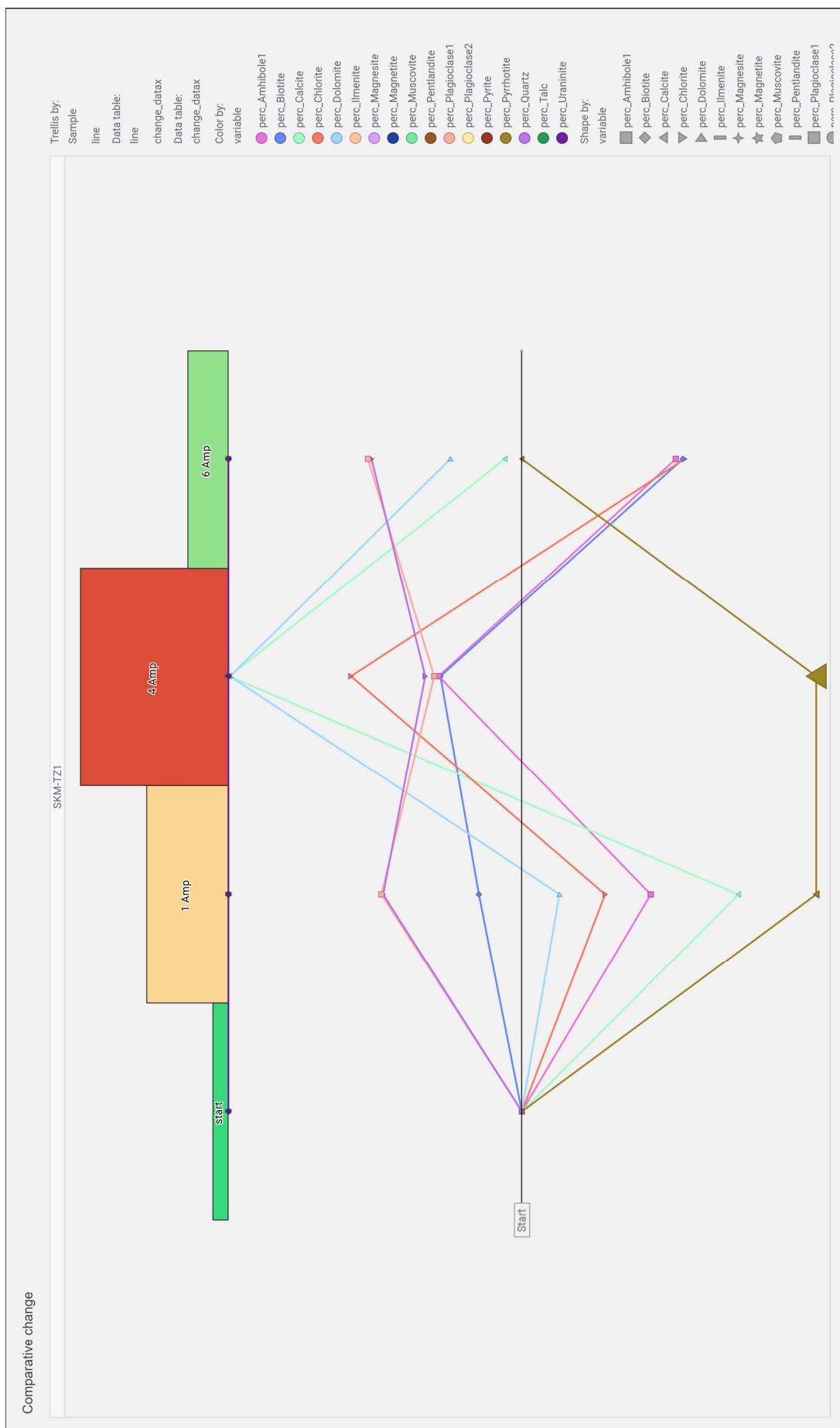


Figure 39. Element data relationship relative to Palladium concentrate for magnetic separation for sample SKM-TZ1
(Analysis done with Spotfire software by John Martin, Hafren Scientific)

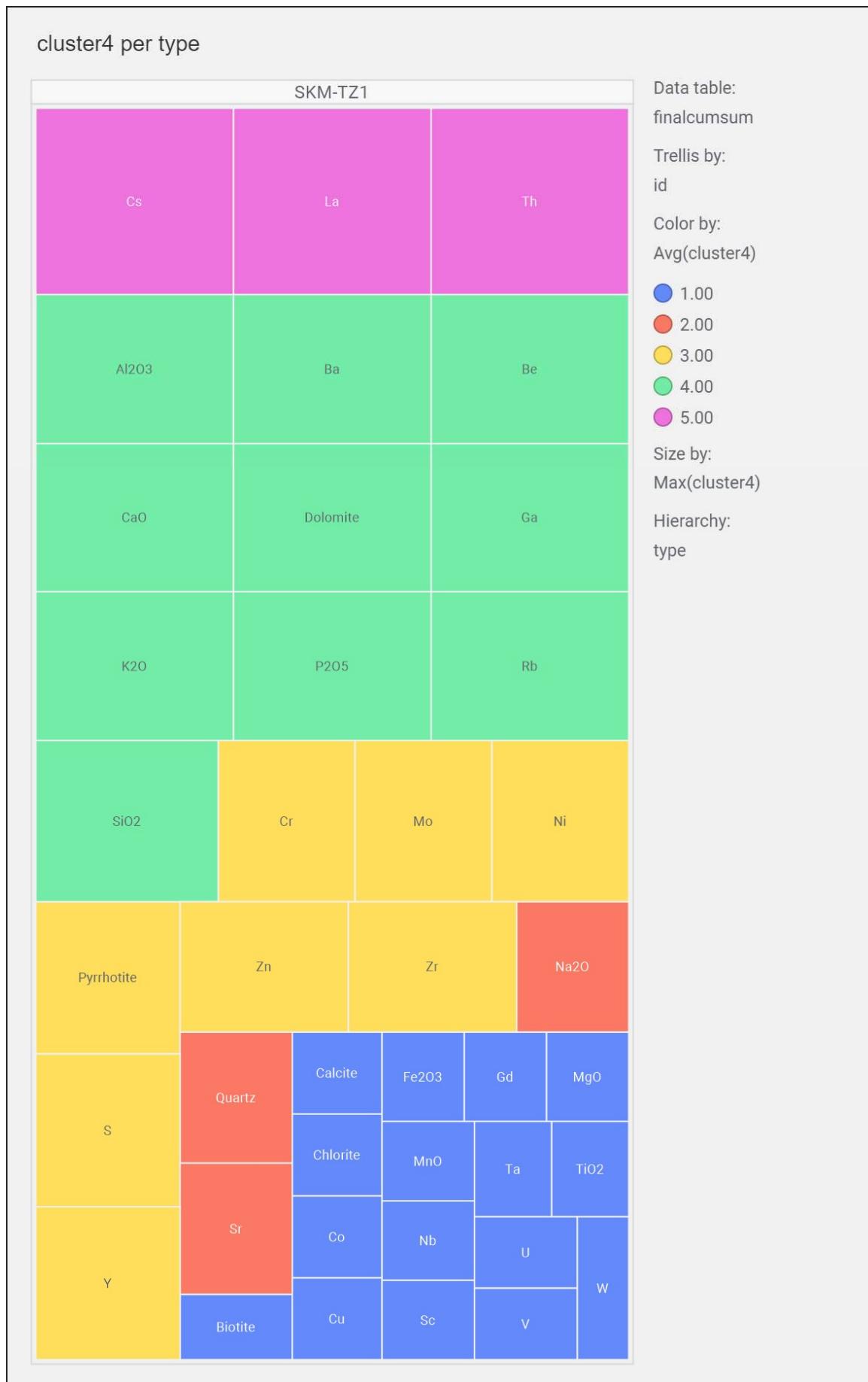


Figure 40. Element data correlation coefficient tree diagrams for elements across for magnetic separation fractions for sample SKM-TZ1 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

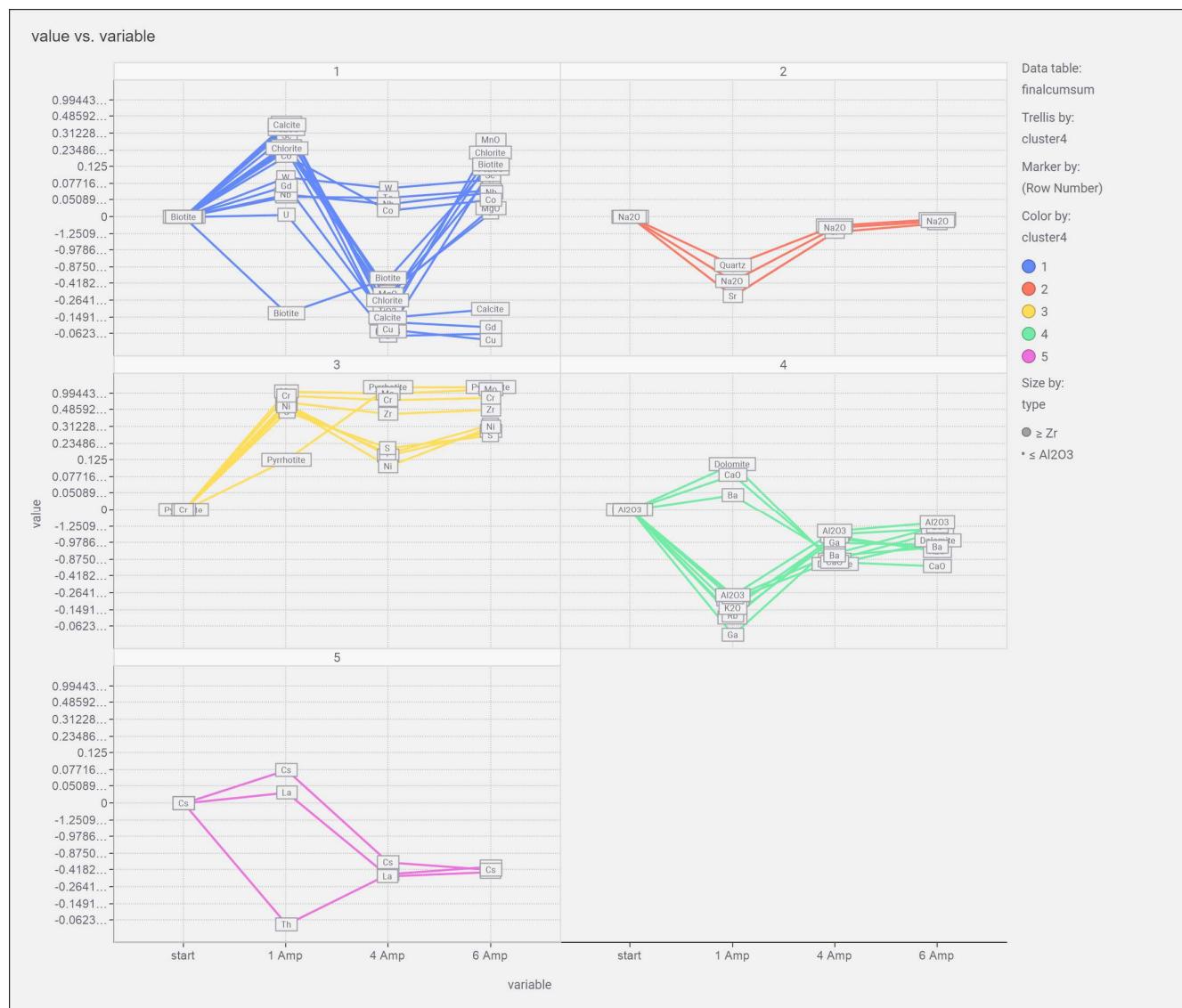


Figure 41. Element data relationship tree diagrams for elements across for magnetic separation fractions for sample SKM-TZ1 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

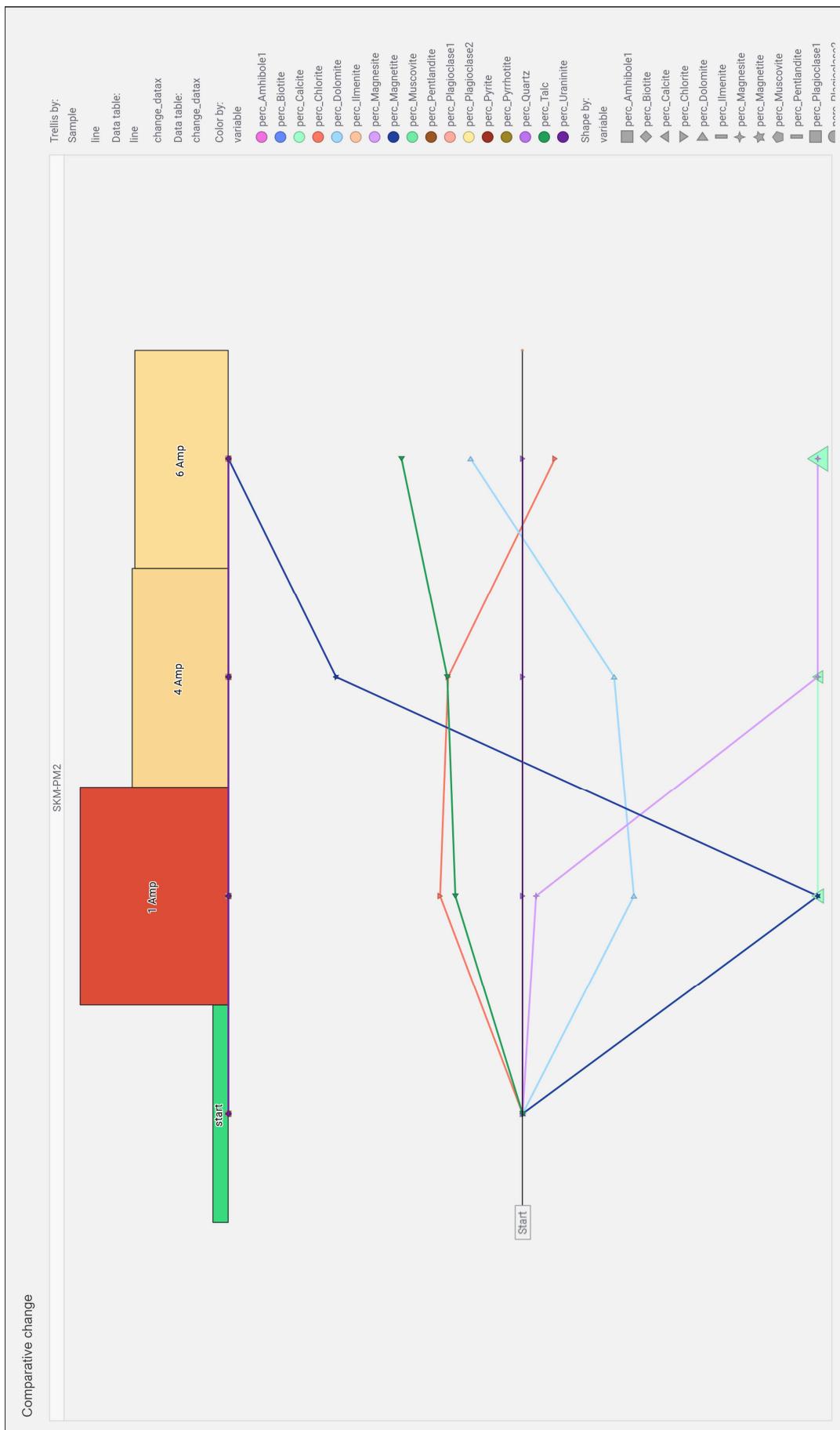


Figure 42. Element data relationship relative to Palladium concentrate for magnetic separation for sample SKM-PM2
(Analysis done with Spotfire software by John Martin, Hafren Scientific)

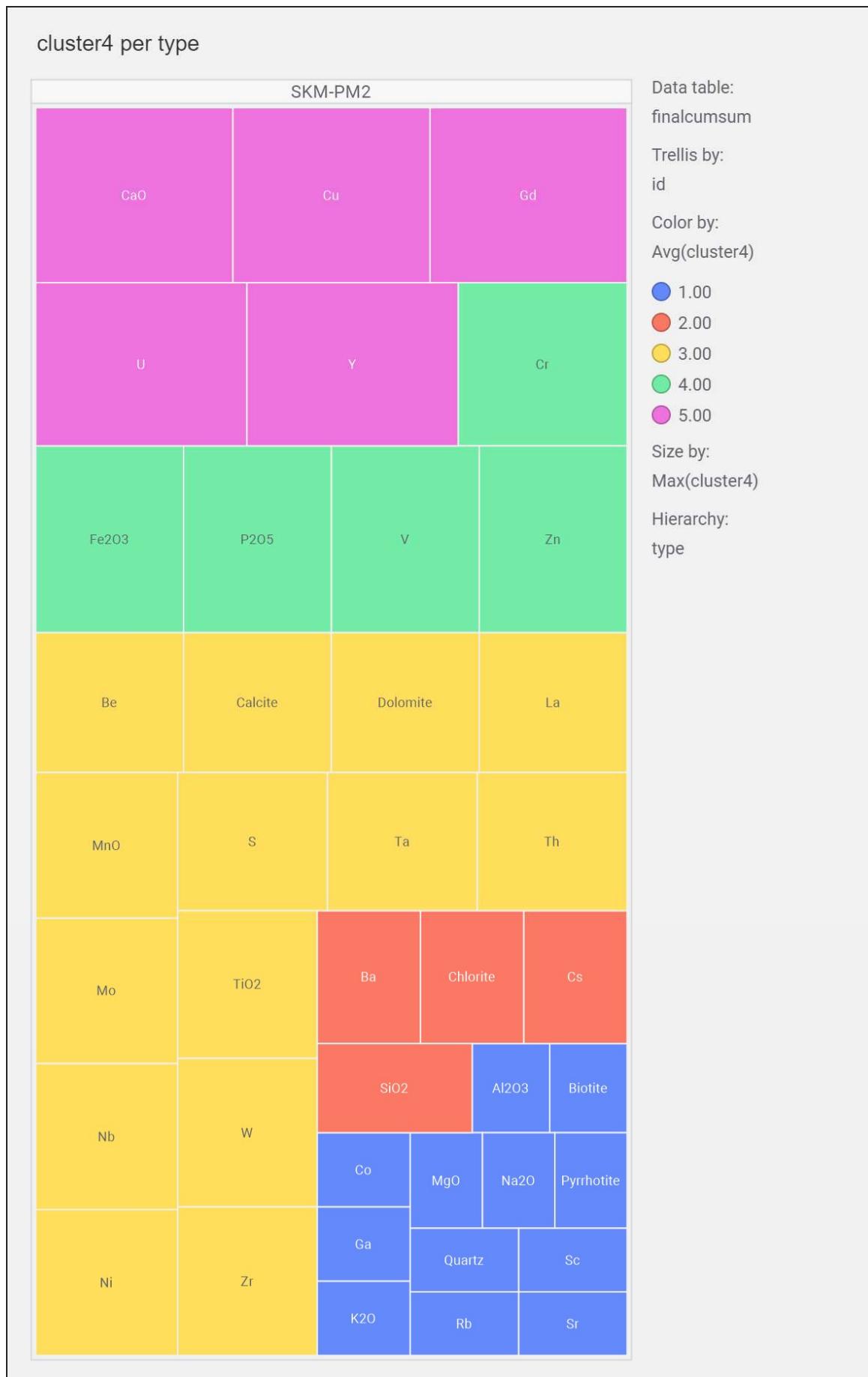


Figure 43. Element data correlation coefficient tree diagrams for elements across for magnetic separation fractions for sample SKM-PM2 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

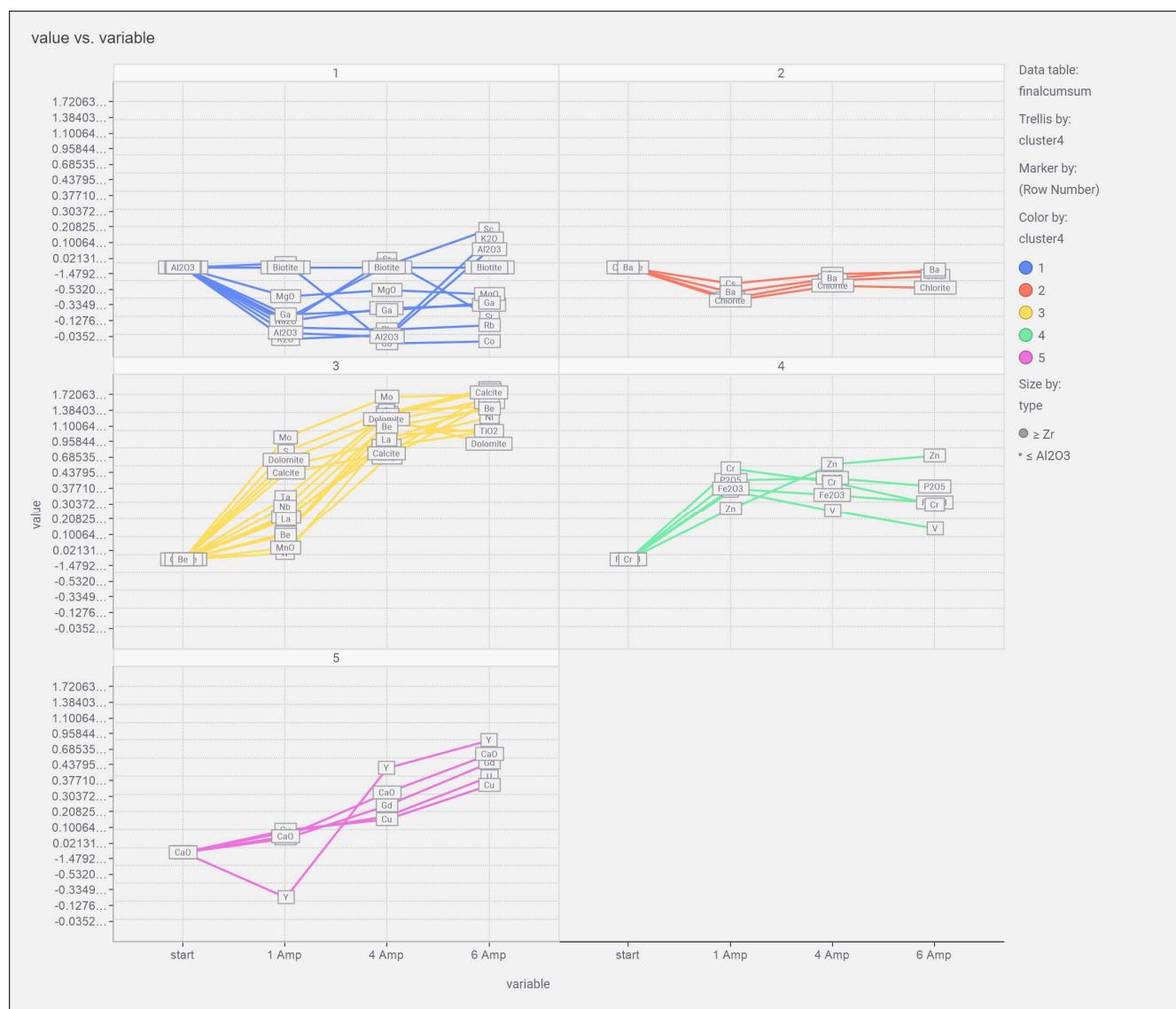


Figure 44. Element data relationship tree diagrams for elements across for magnetic separation fractions for sample SKM-PM2 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

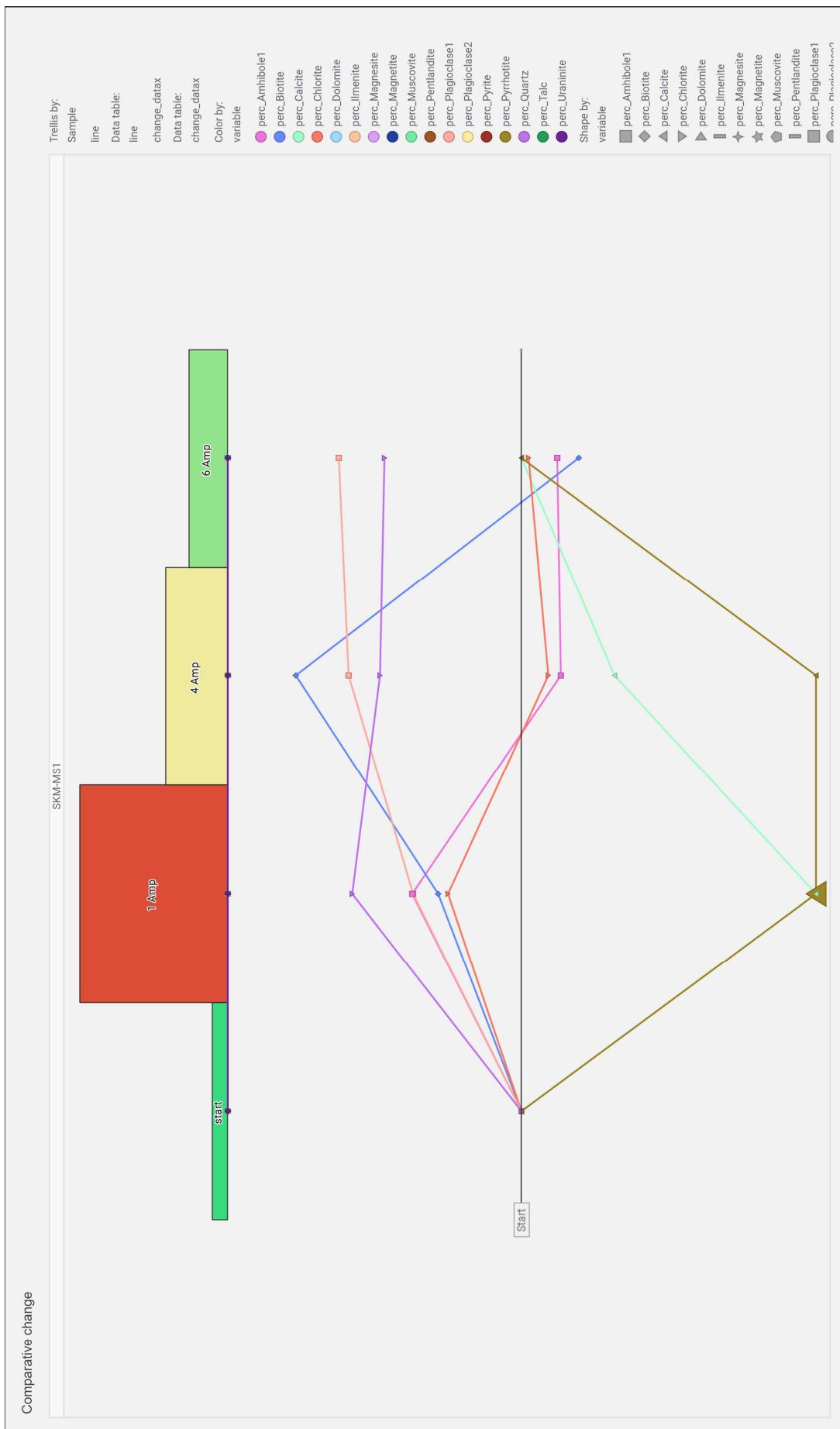


Figure 45. Element data relationship relative to Palladium concentrate for magnetic separation for sample SKM-MS1
(Analysis done with Spotfire software by John Martin, Hafren Scientific)



Figure 46. Element data correlation coefficient tree diagrams for elements across for magnetic separation fractions for sample SKM-MS1 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

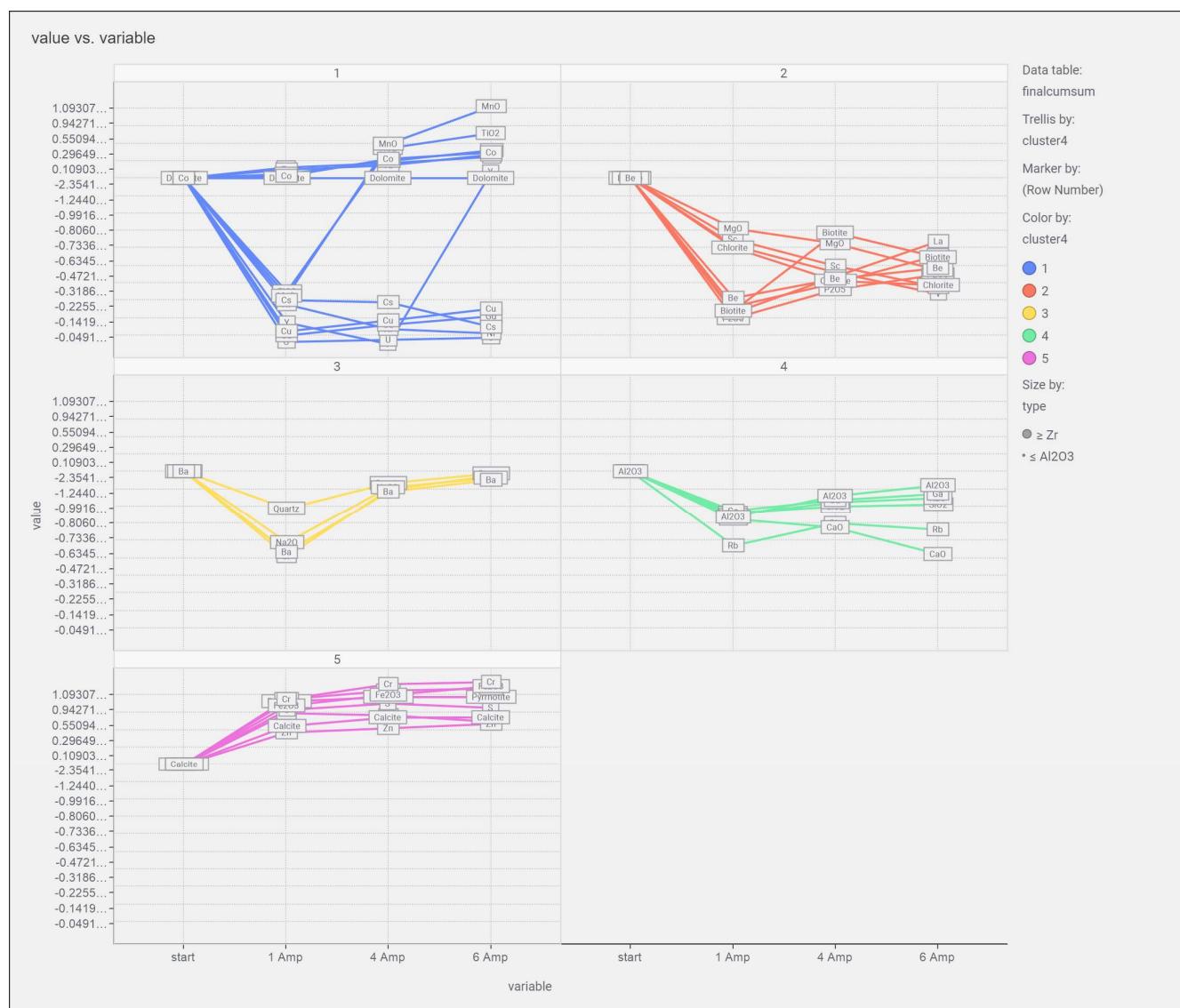


Figure 47. Element data relationship tree diagrams for elements across for magnetic separation fractions for sample SKM-MS1 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

Figures 48 to 56 shows the data patterns in precious metal deportation in magnetic separation for these samples.

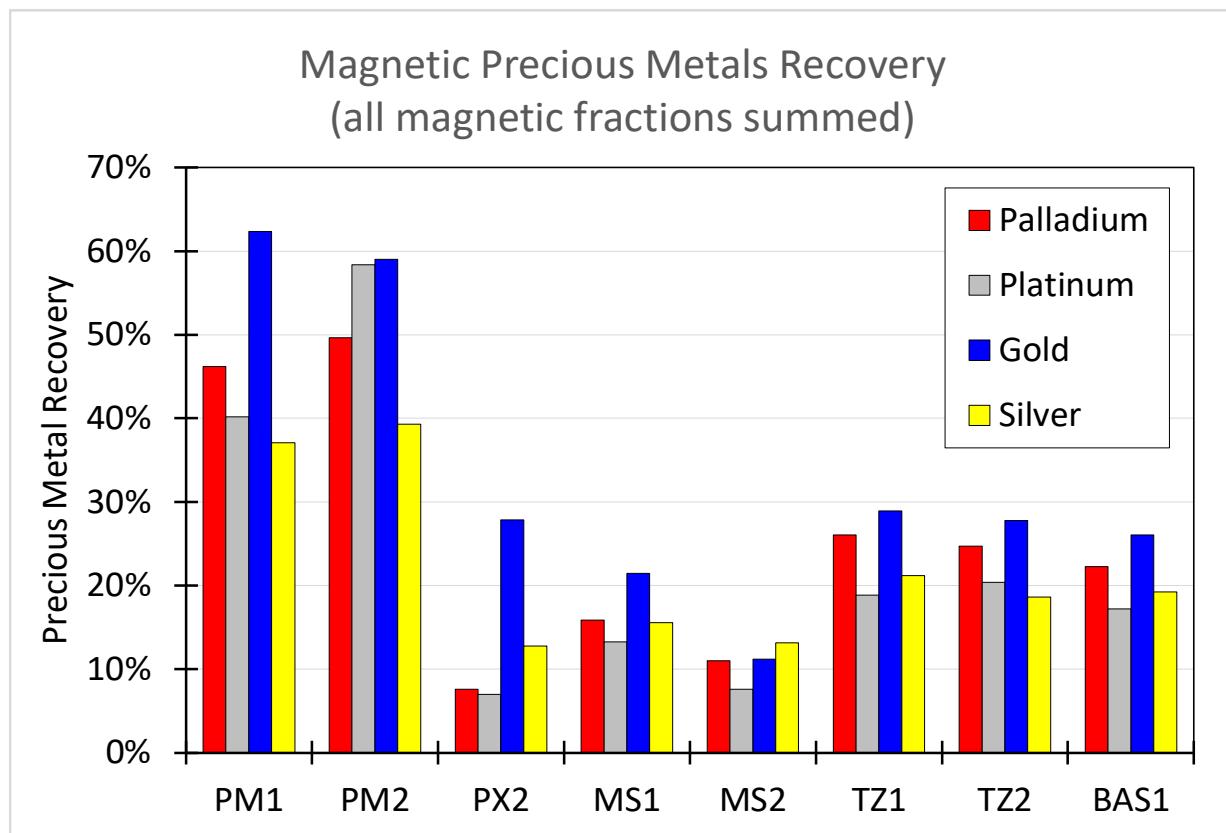


Figure 48. Precious metals recovered with magnetic separation (all magnetic fractions combined)
(Chemical Assay Method 711P)

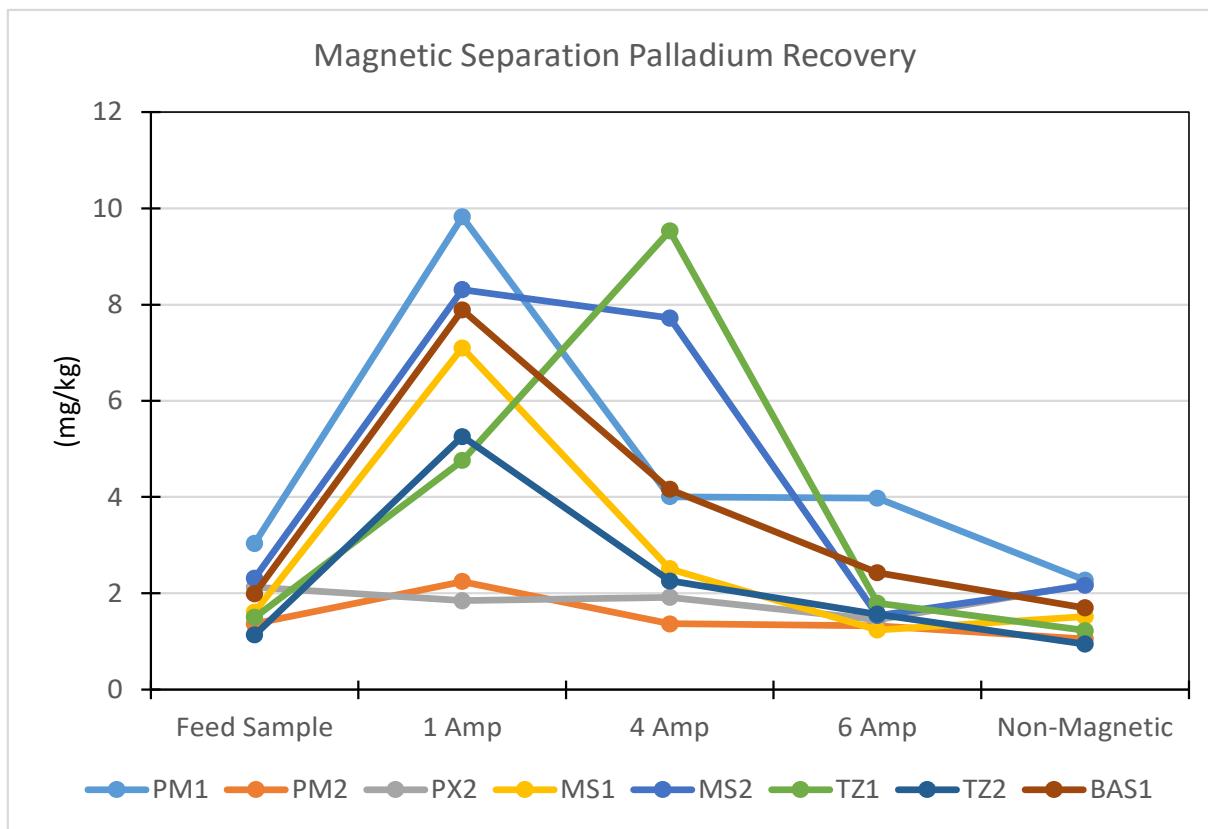


Figure 49. Palladium recovery from magnetic separation (Chemical Assay Method 711P)

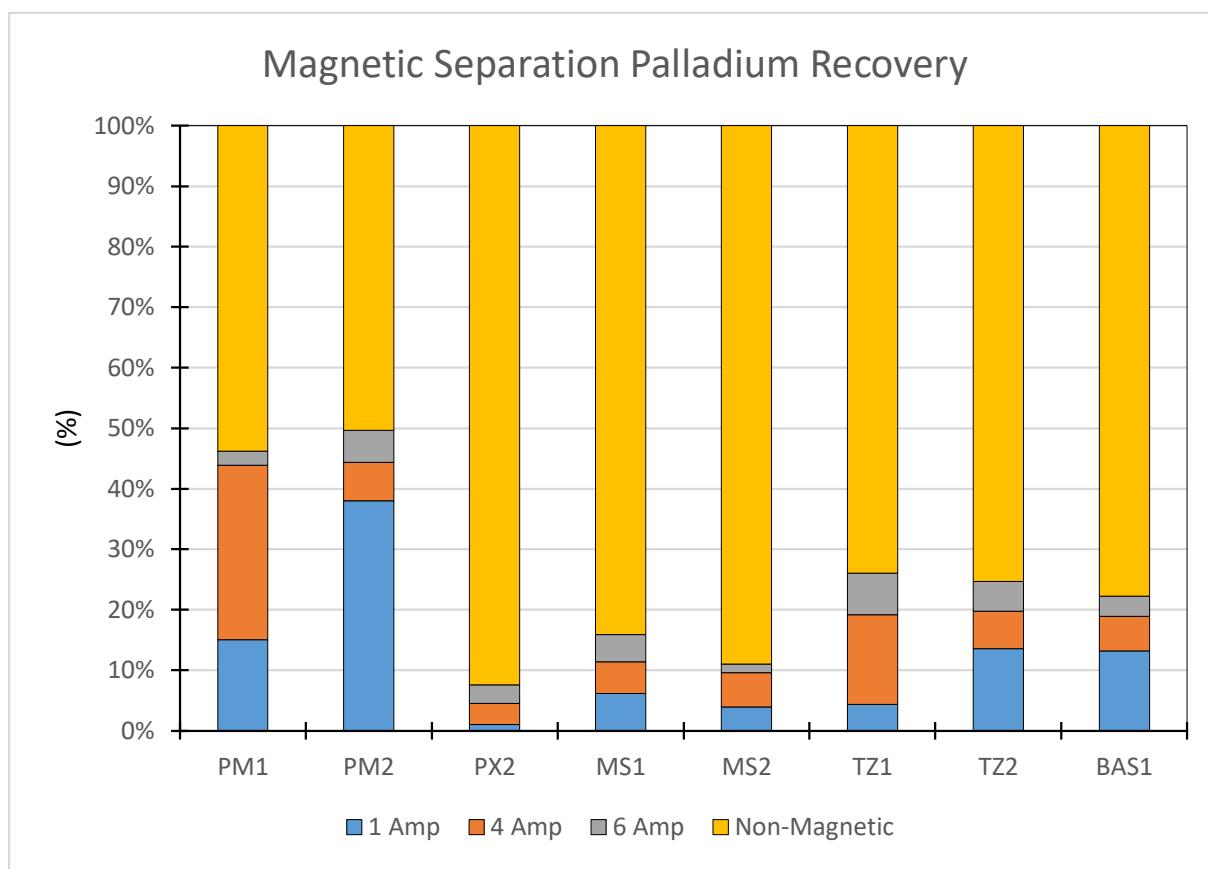


Figure 50. Palladium recovery from magnetic separation (Chemical Assay Method 711P)

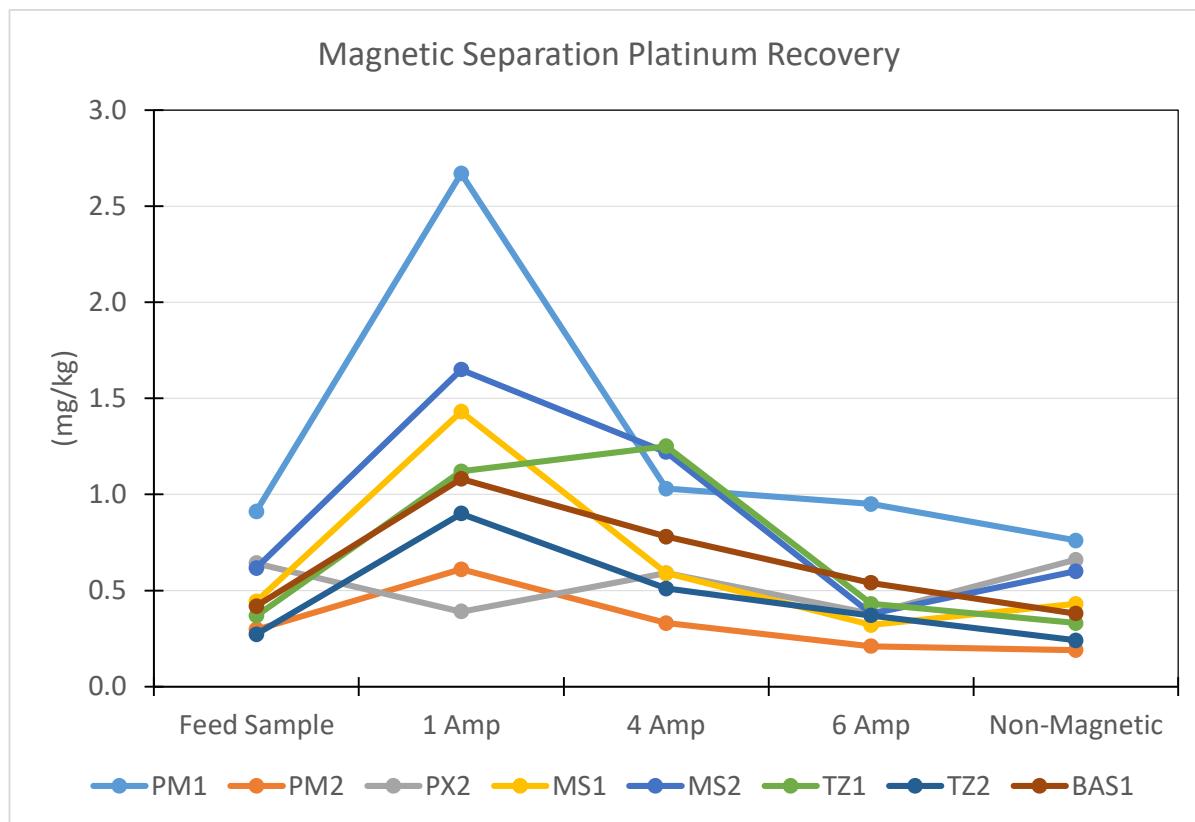


Figure 51. Platinum recovery from magnetic separation (Chemical Assay Method 711P)

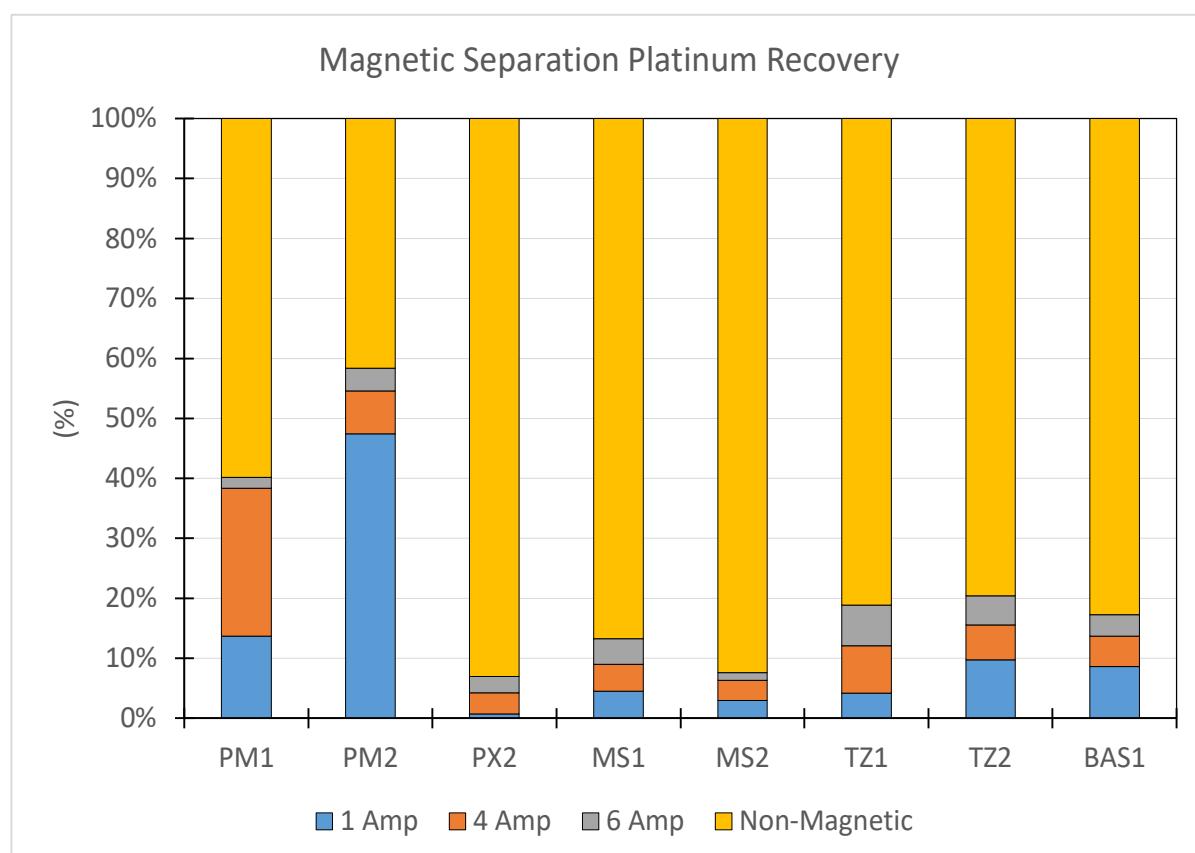


Figure 52. Platinum recovery from magnetic separation (Chemical Assay Method 711P)

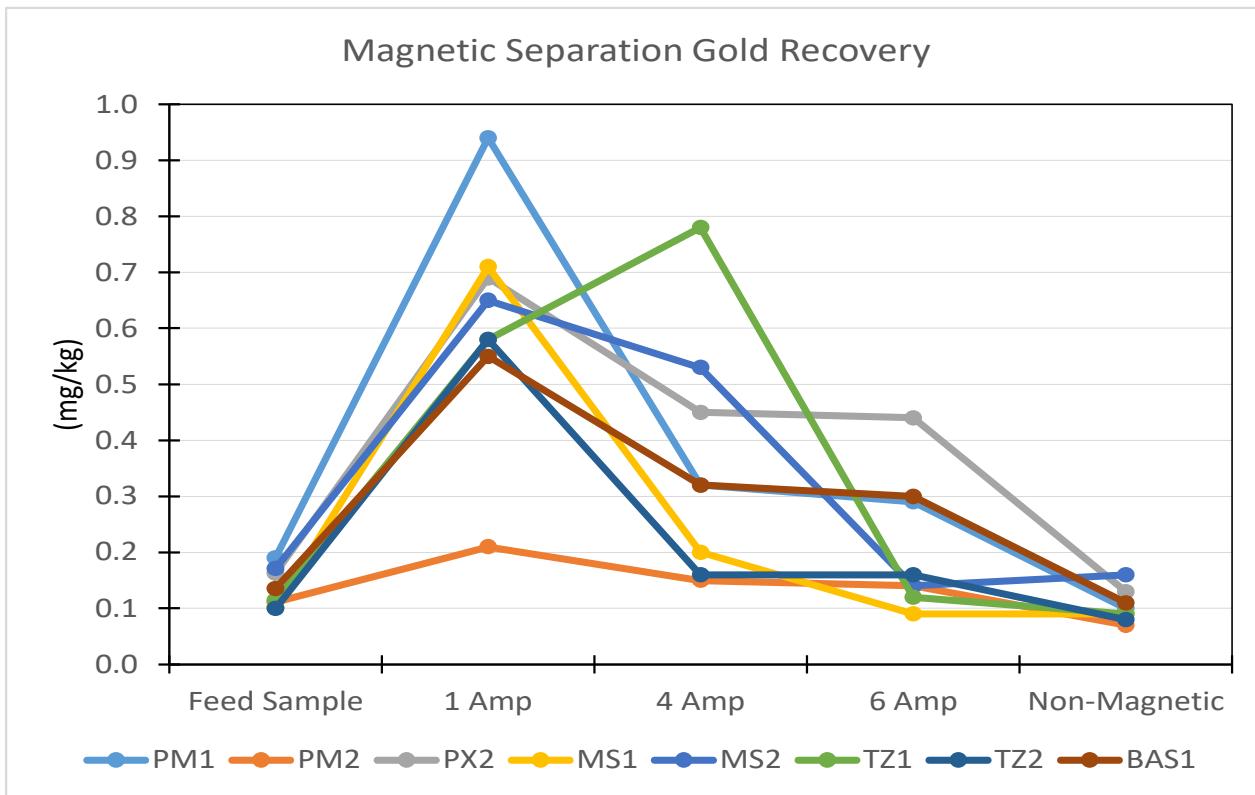


Figure 53. Gold recovery from magnetic separation (Chemical Assay Method 711P)

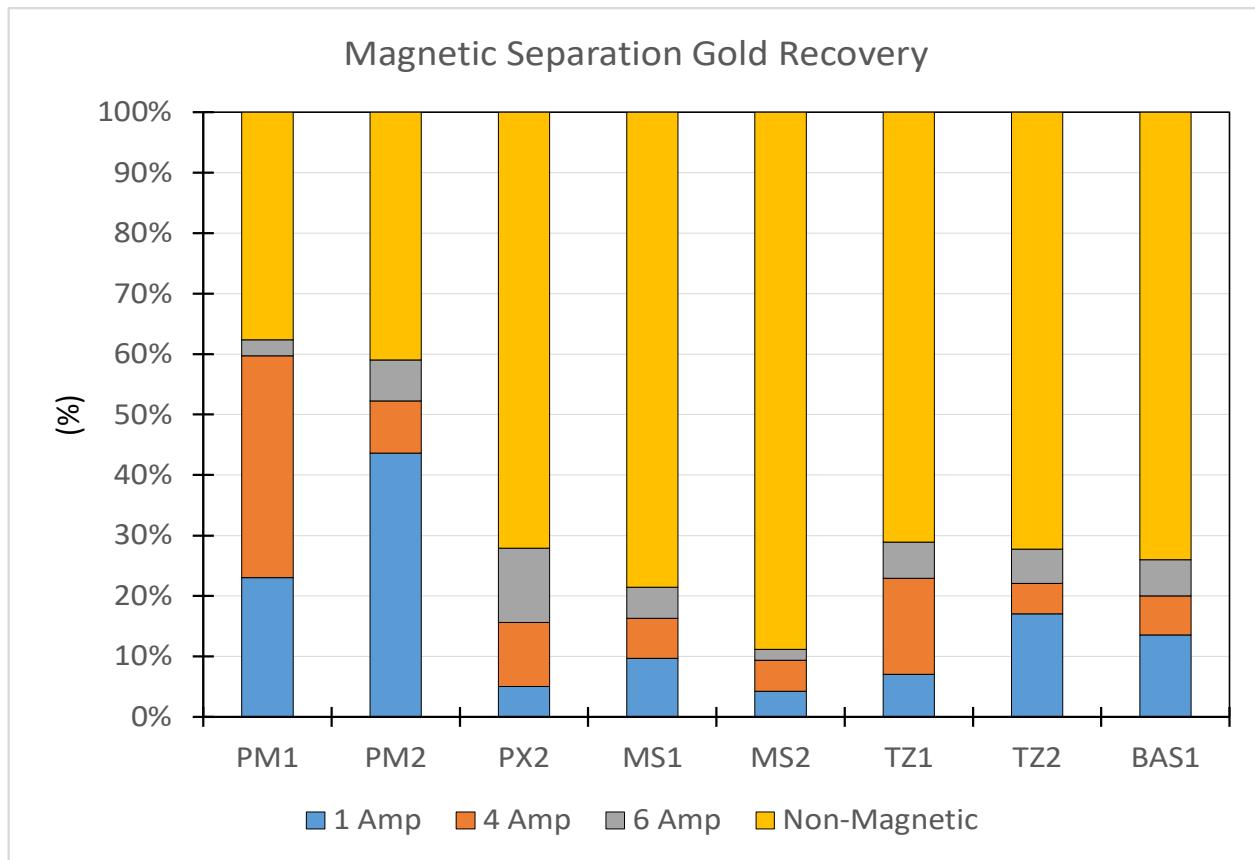


Figure 54. Gold recovery from magnetic separation (Chemical Assay Method 711P)

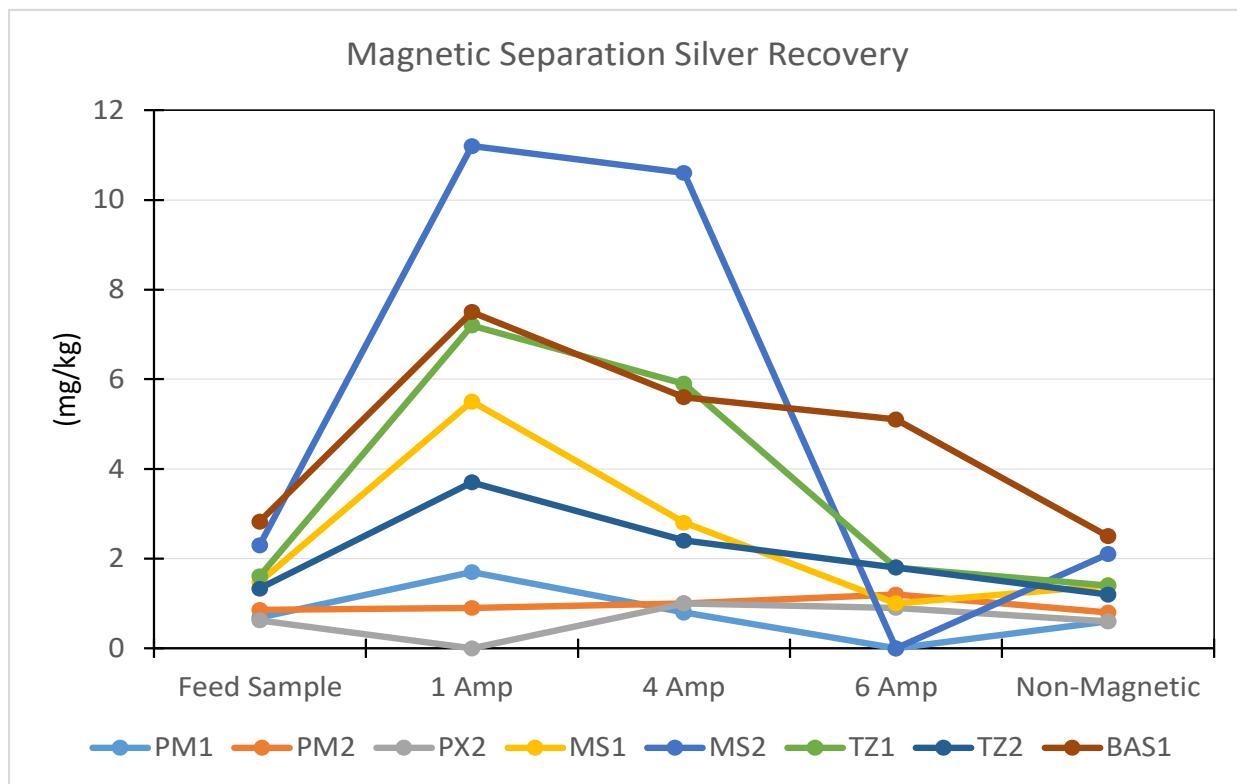


Figure 55. Silver recovery from magnetic separation (Chemical Assay Method 711P)

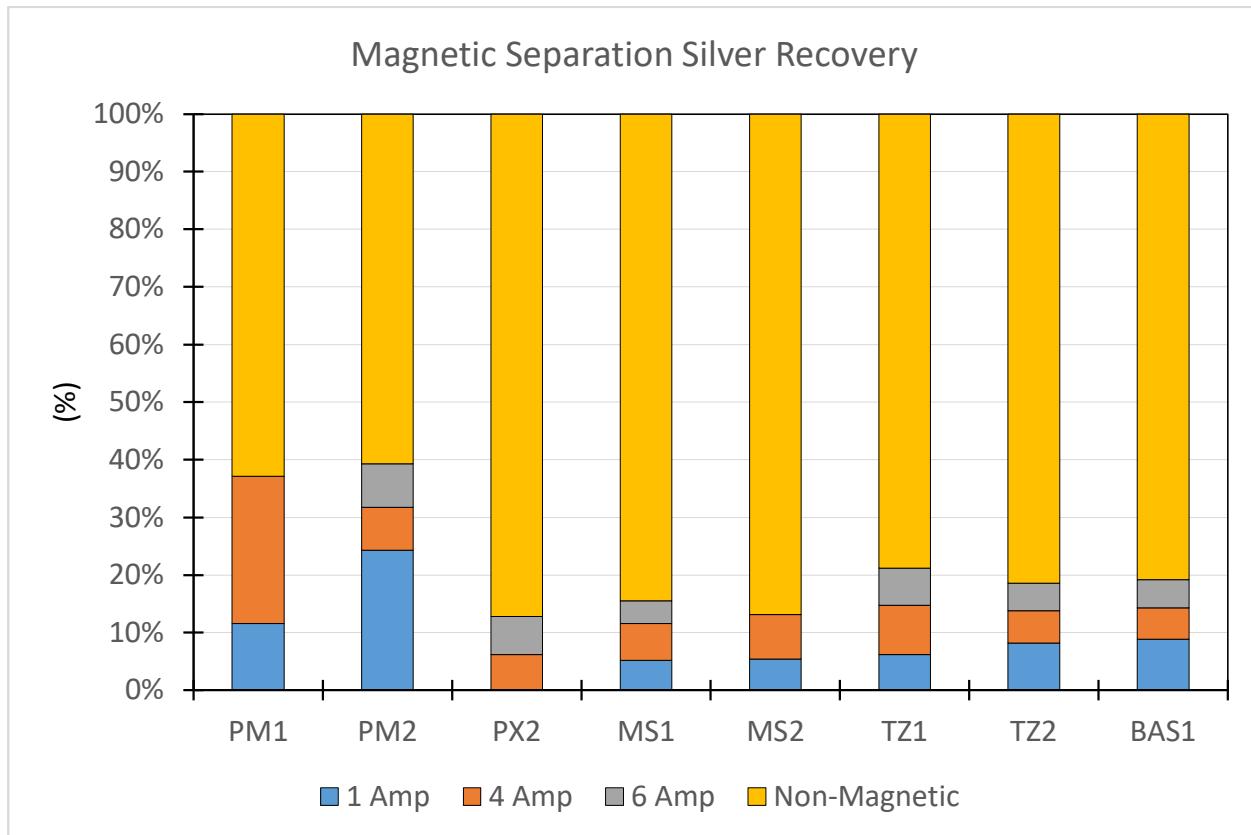


Figure 56. Silver recovery from magnetic separation (Chemical Assay Method 711P)

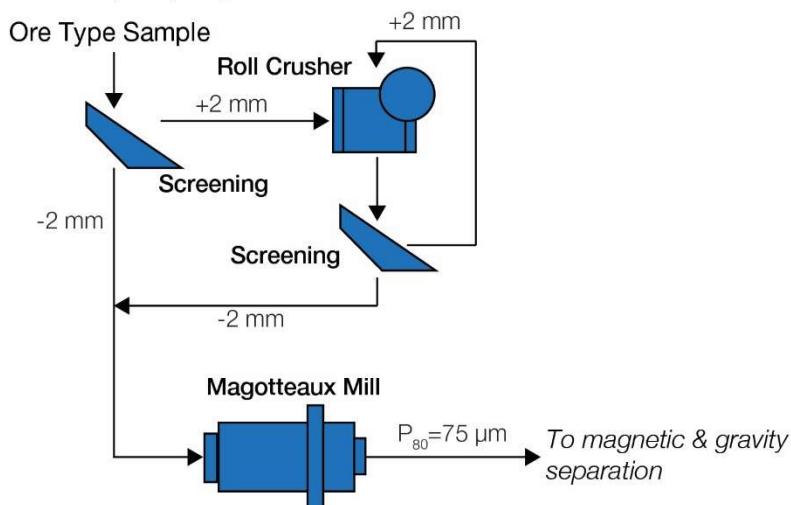
Due to sample mass restrictions, a 4 acid digest chemical assay measurement was not done for the magnetic separation products.

7 GRAVITY CONCENTRATION

7.1 Methods/Procedure

As for the magnetic separation test work, the gravity separation tests on the 10 ore types have been performed 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 P_{80} of 75 μm , following the flowsheet in Figure 57. Gravity concentration tests were carried out on a KHD Humbold Wedag laboratory shaking table (Type MN 929/2) equipped with a rubber deck with carved ripples with a 5 kg sample. The pulp density was set at 10-15wt% solids

a. Sample preparation



b. Gravity separation

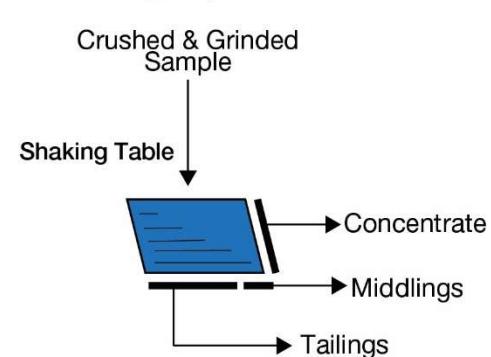


Figure 57. Flowsheets showing the (a) sample preparation procedure and (b) gravity concentration procedure



Figure 58. Photographs of the shaking table test work

7.2 Gravity Separation Results

Figure 59 and Table 2 show the mass pull for all the gravity separation fractions for each ore type.

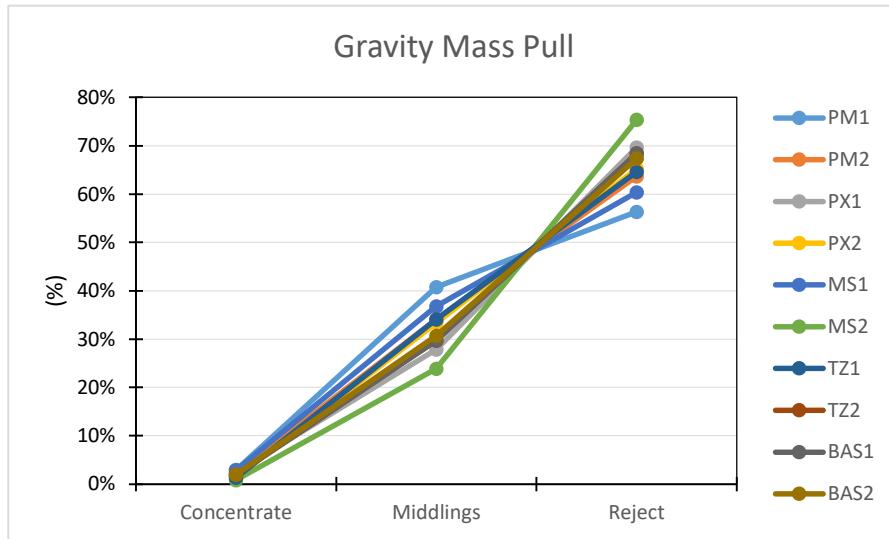


Figure 59. Mass in separated products with gravity separation

Table 2. Gravity separation mass pull in product fractions

GRAVITY SEPARATION MASS PULL	PM1 (%)	PM2 (%)	PX1 (%)	PX2 (%)	MS1 (%)	MS2 (%)	TZ1 (%)	TZ2 (%)	BAS1 (%)	BAS2 (%)
Concentrate	2.9%	2.2%	1.86%	2.50%	1.4%	2.9%	0.8%	1.3%	1.9%	2.0%
Middlings	41%	34%	27.24%	27.86%	33%	37%	24%	34%	30%	31%
Reject Tails	56%	64%	70.90%	69.64%	65%	60%	75%	65%	68%	67%

The bulk mineralogy of the gravity separation products for each ore type obtained through semi-quantitative XRD is shown in Figure 60 to 69. The gravity fractions are mostly composed of chlorite, amphibole, talc, and plagioclase, and to a lesser extent dolomite, quartz, magnesite and pyrrhotite. The proportion of each mineral varies depending on the product and the ore type but globally there is no clear mineral separation pattern, where one mineral type is dominant in any particular fraction.

Pyrrhotite, pyrite, chalcopyrite and pentlandite are recovered in the gravity concentrate. From the MLA work done, sulfide minerals are often associated with precious metals, which could be the reason why precious metals are reporting to gravity concentrate fractions in Figures 60 to 69.

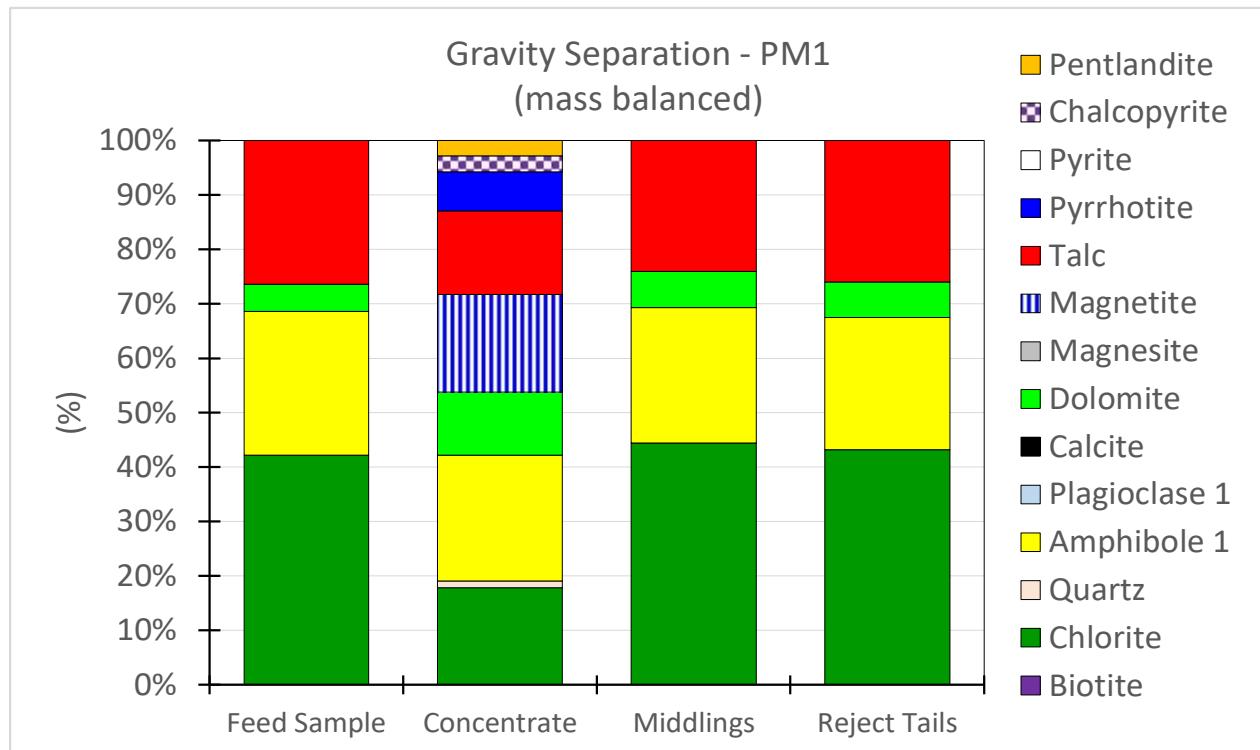


Figure 60. Semi-quantitative mineralogy (QXRD) of the gravity separation products PM1 (X-Ray Mineral Services)

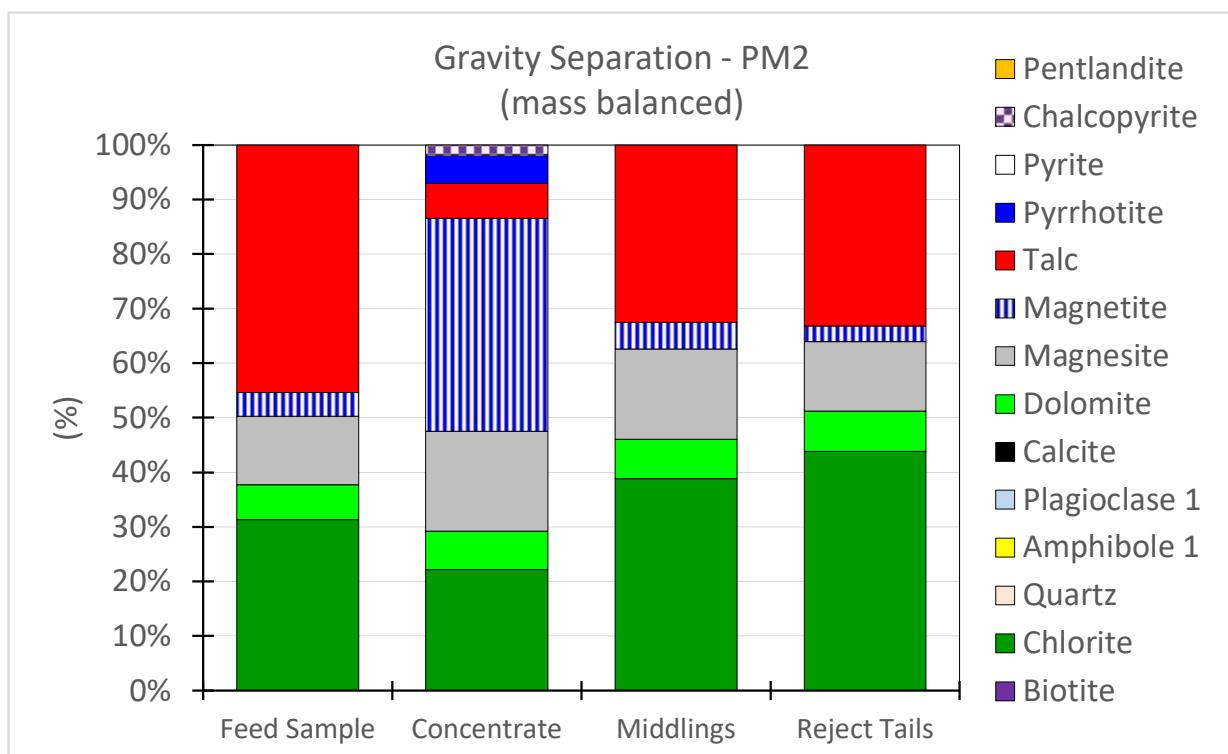


Figure 61. Semi-quantitative mineralogy (QXRD) of the gravity separation products PM2 (X-Ray Mineral Services)

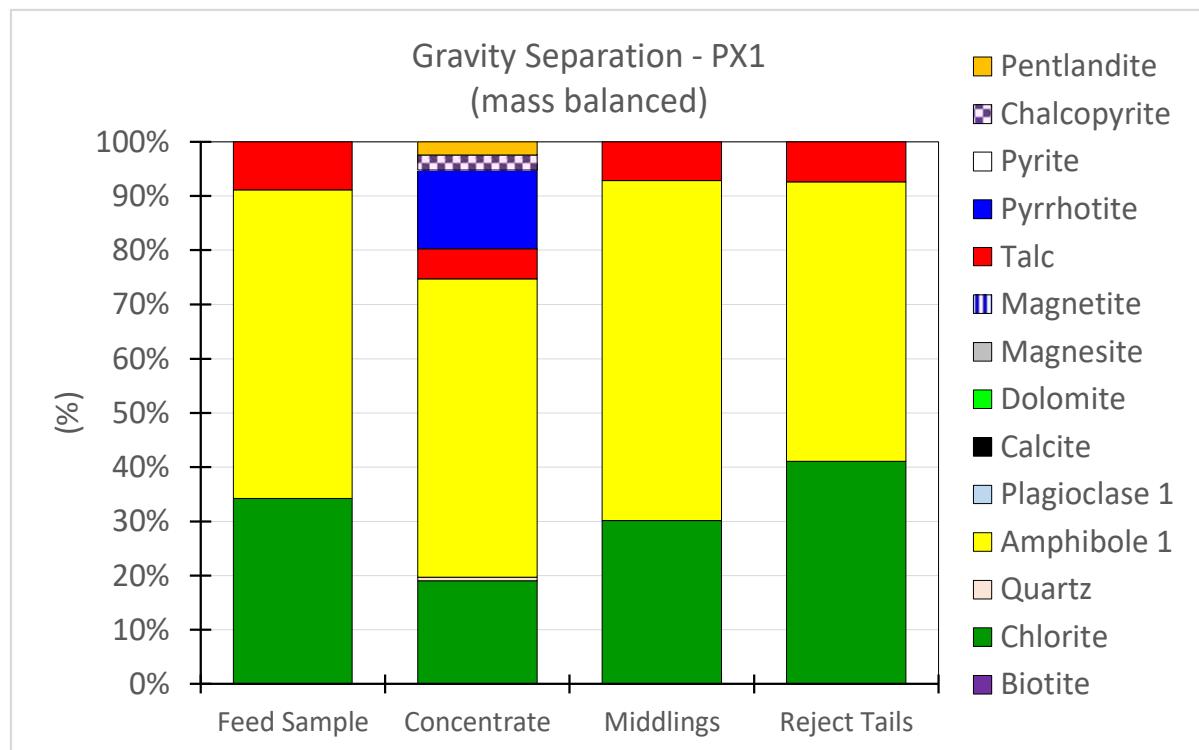


Figure 62. Semi-quantitative mineralogy (QXRD) of the gravity separation products PX1 (X-Ray Mineral Services)

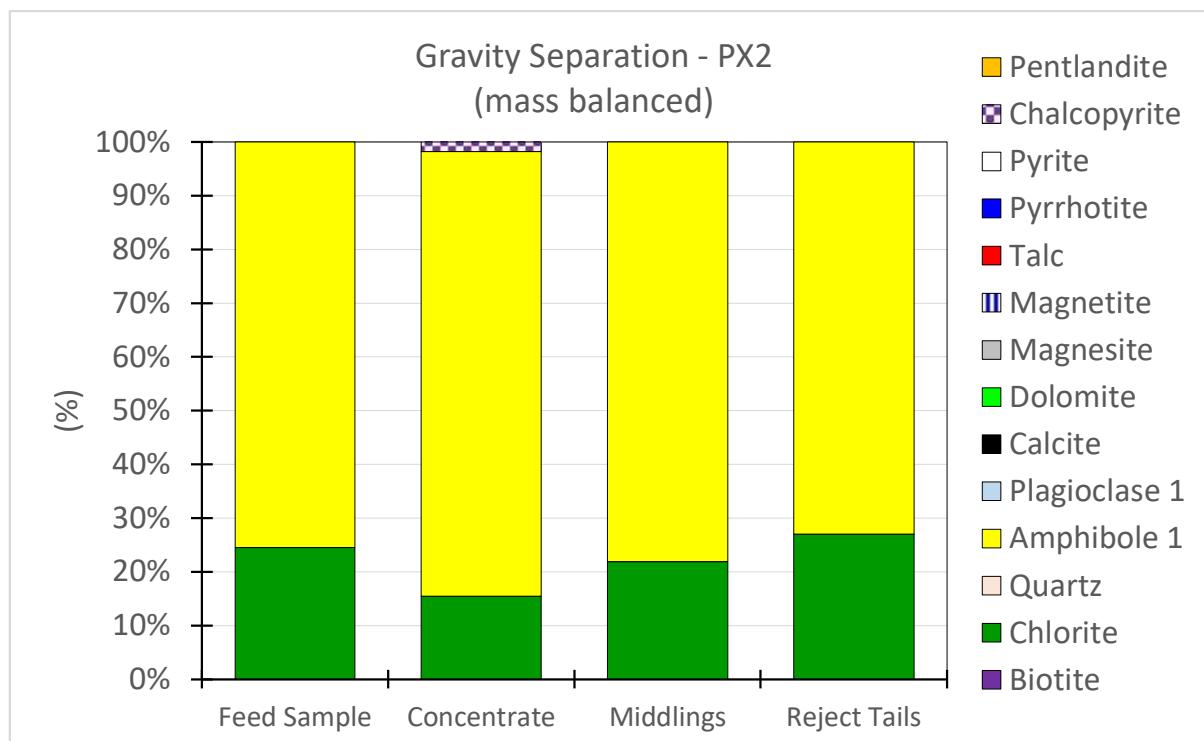


Figure 63. Semi-quantitative mineralogy (QXRD) of the gravity separation products PX2 (X-Ray Mineral Services)

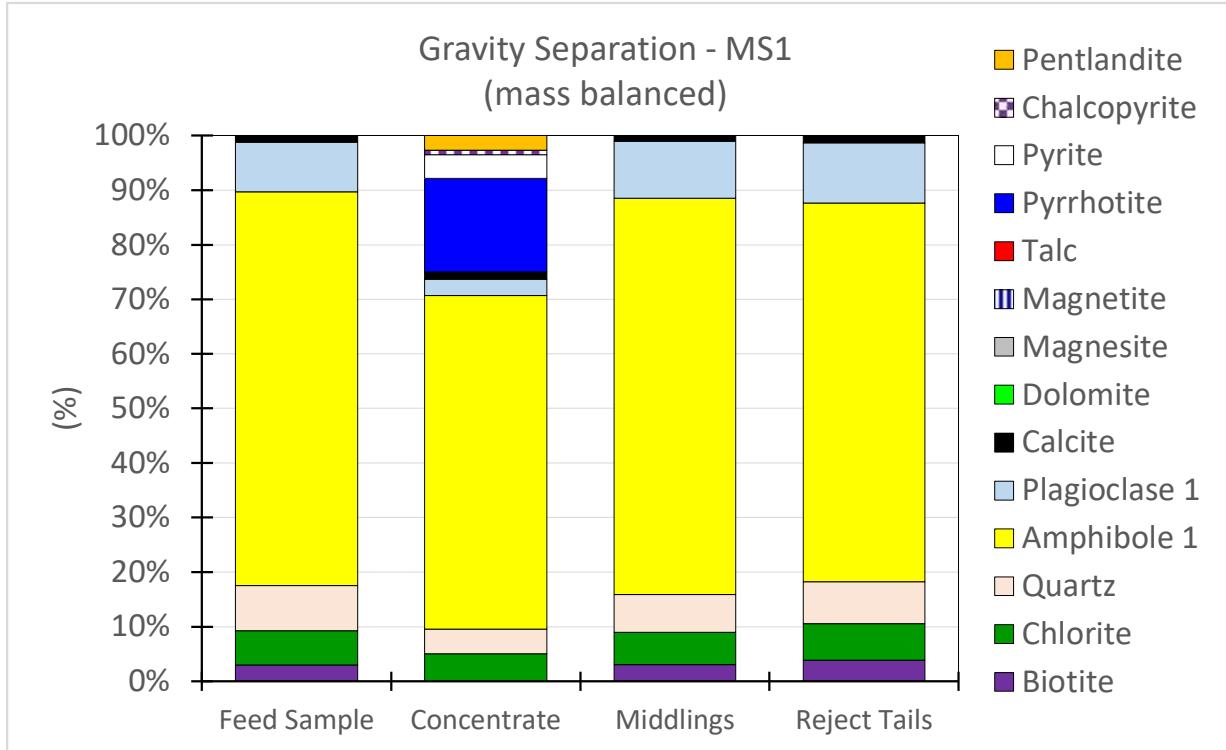


Figure 64. Semi-quantitative mineralogy (QXRD) of the gravity separation products MS1 (X-Ray Mineral Services)

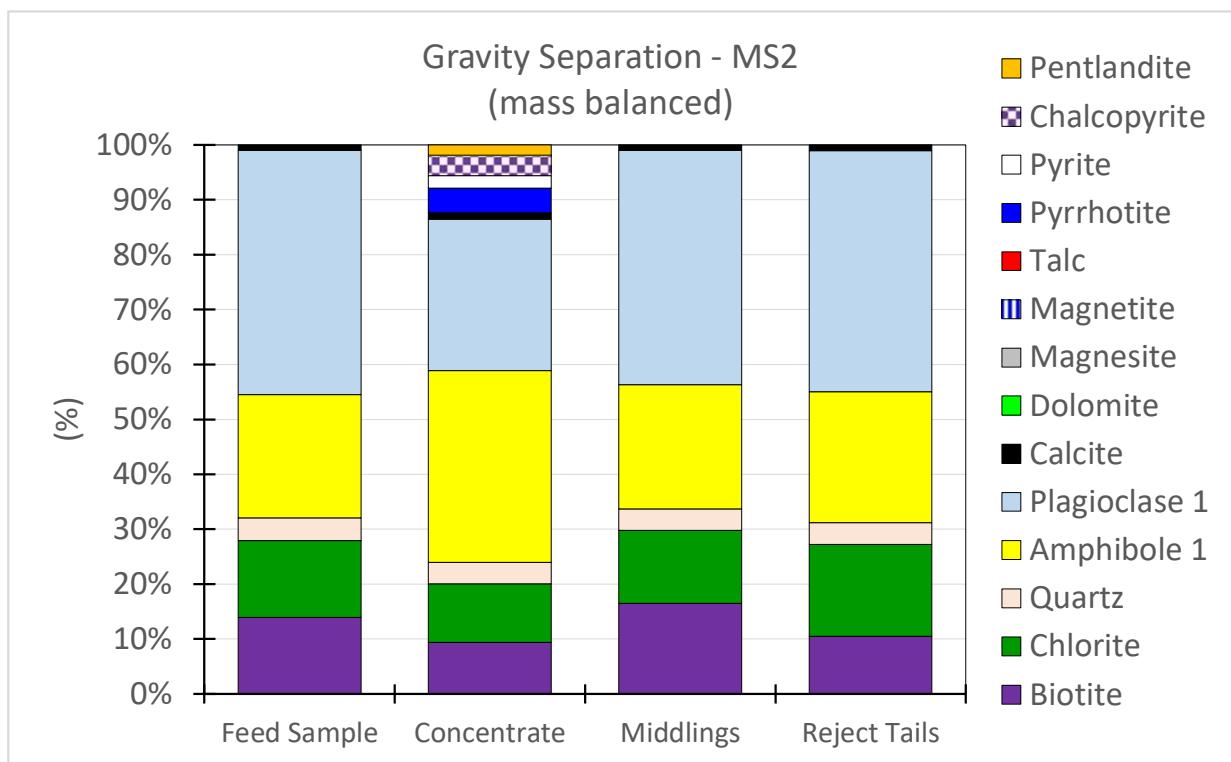


Figure 65. Semi-quantitative mineralogy (QXRD) of the gravity separation products MS2 (X-Ray Mineral Services)

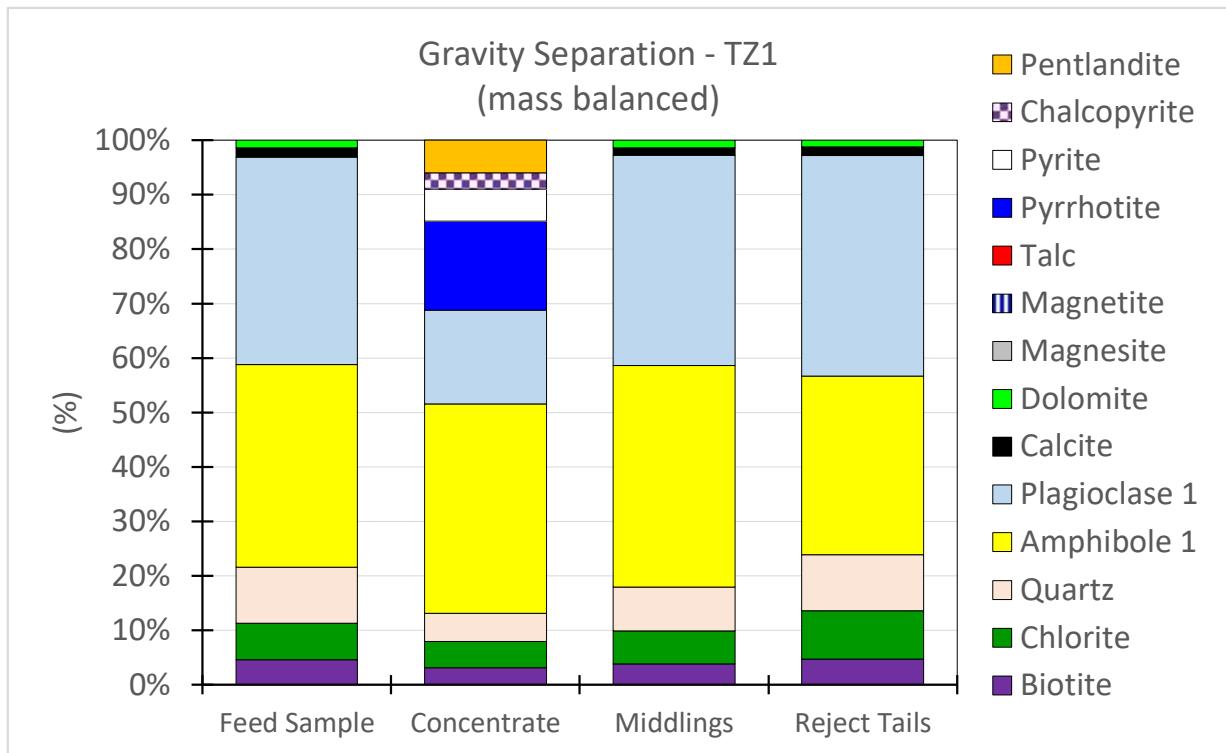


Figure 66. Semi-quantitative mineralogy (QXRD) of the gravity separation products TZ1 (X-Ray Mineral Services)

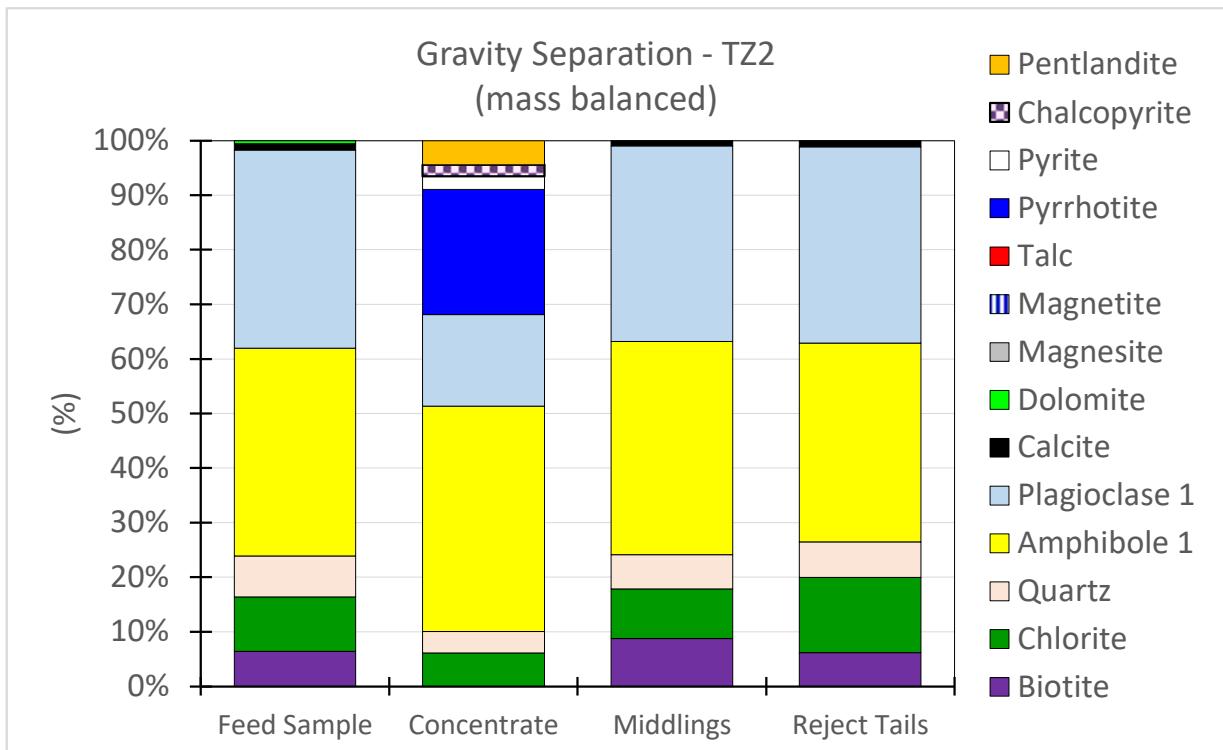


Figure 67. Semi-quantitative mineralogy (QXRD) of the gravity separation products TZ2 (X-Ray Mineral Services)

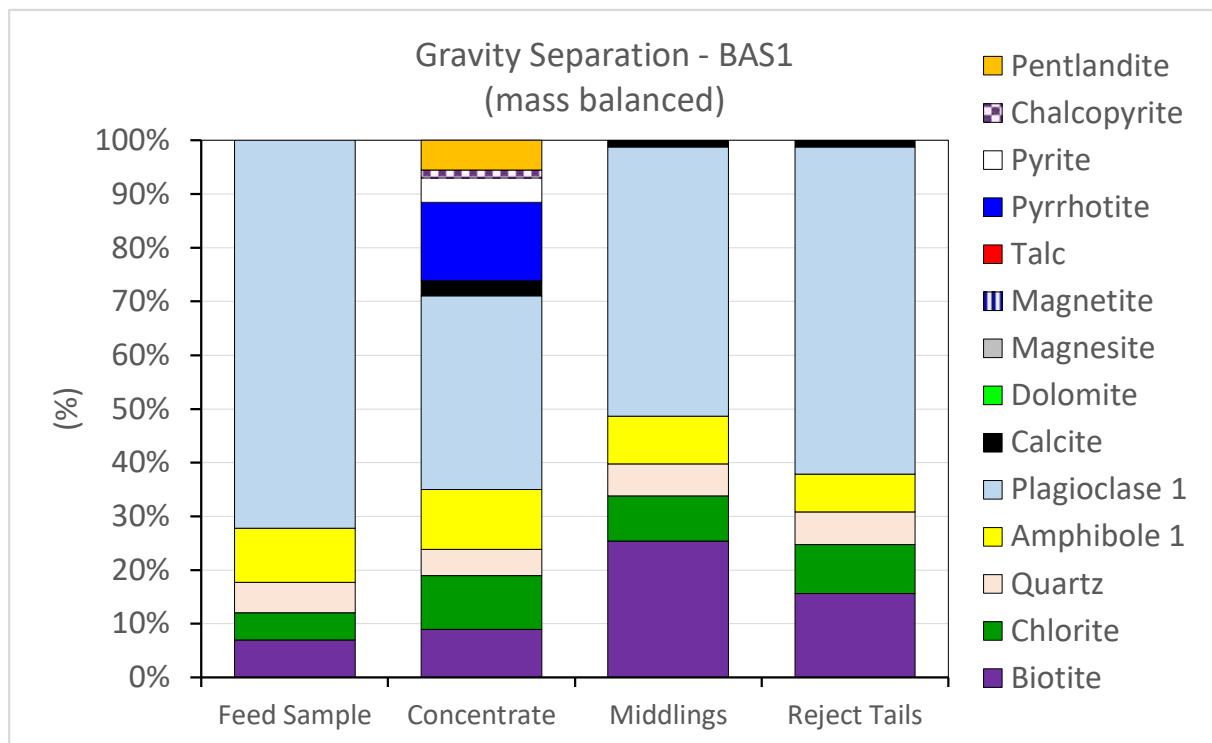


Figure 68. Semi-quantitative mineralogy (QXRD) of the gravity separation products BAS1 (X-Ray Mineral Services)

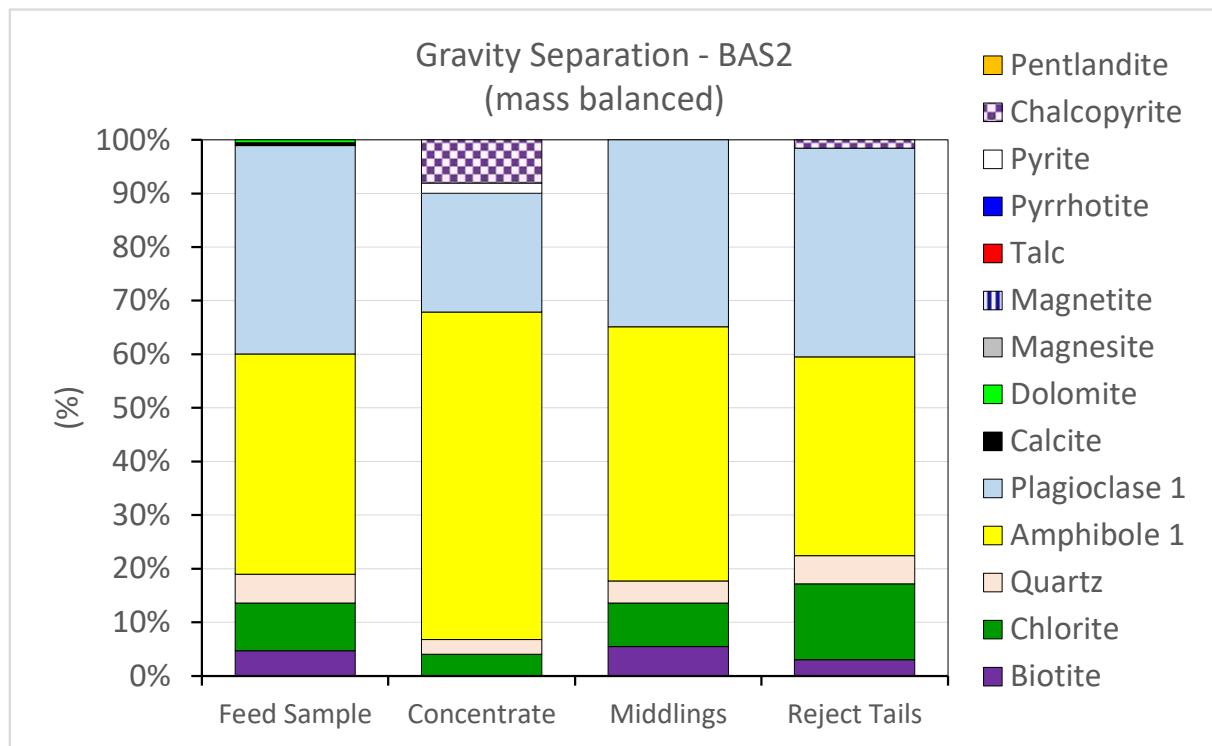


Figure 69. Semi-quantitative mineralogy (QXRD) of the gravity separation products BAS2 (X-Ray Mineral Services)

Figure 70 to 77 shows the outcomes of an analysis done with Spotfire software (by John Martin, Hafren Scientific). Patterns in the gravity separation data for everything collected was examined and used to create these charts, including chemical assays (fire assays method 711P, Eltra S, Eltra C and ICP method 306P), QXRD and XRF.

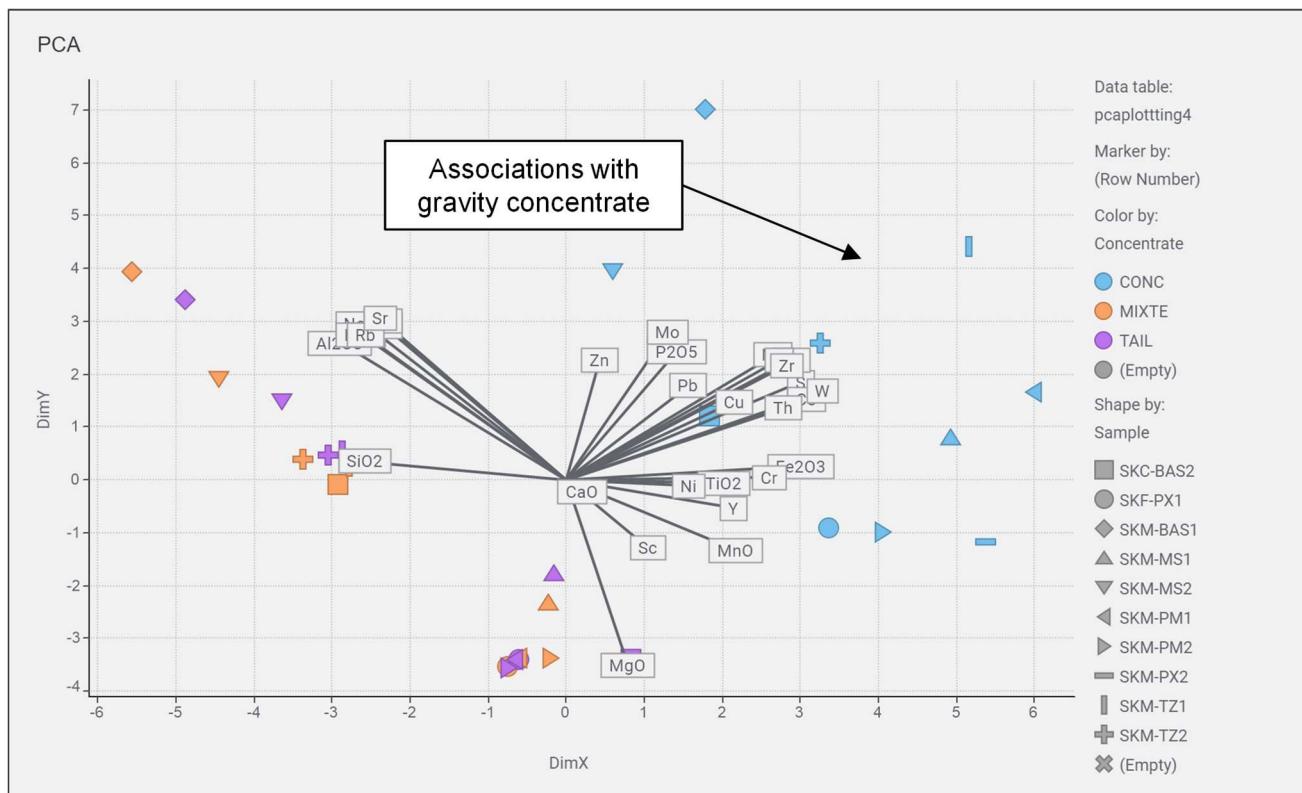


Figure 70. Principle Component Analysis of gavity separation data
 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

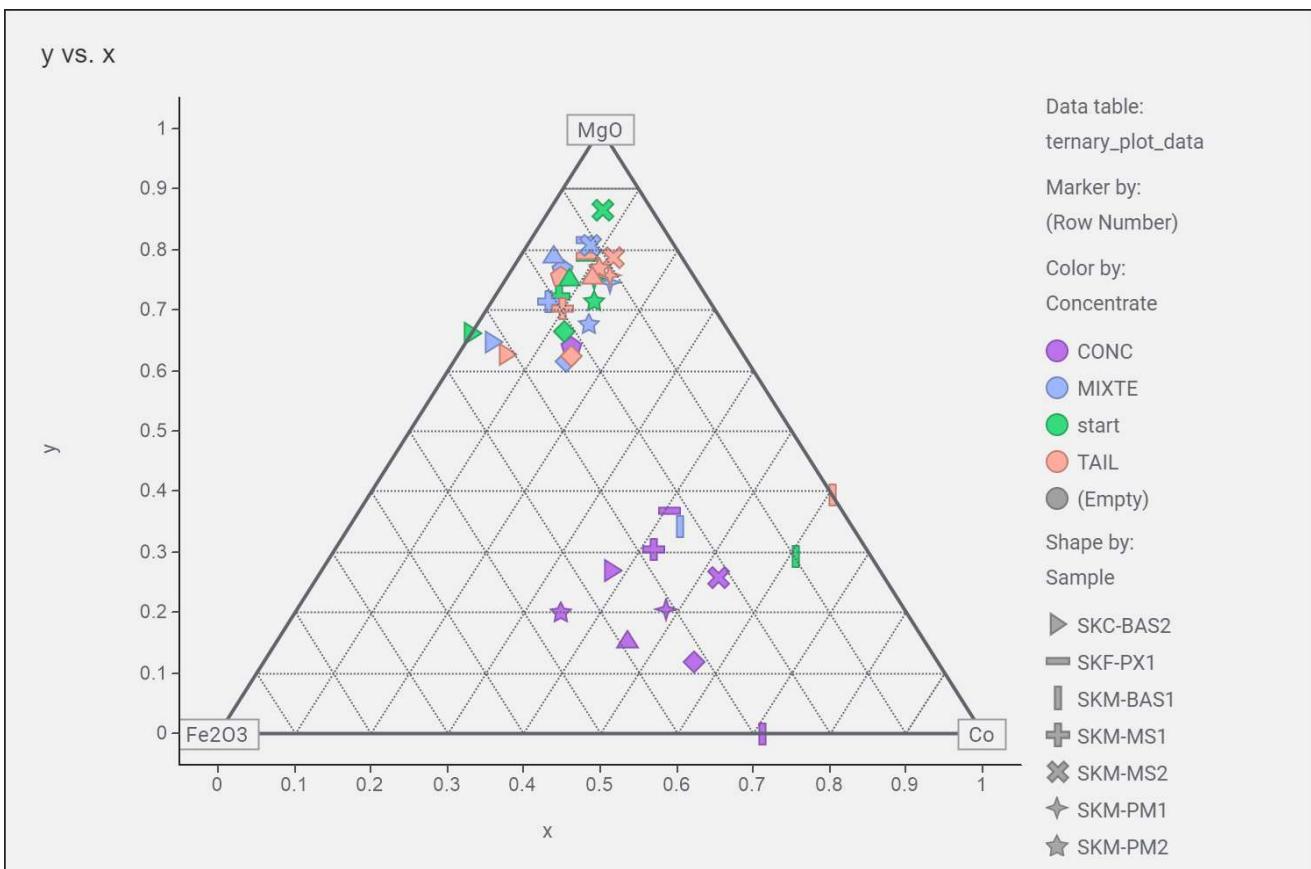


Figure 71. Gravity separation recovery, on a trinairy plot between Fe_2O_3 , MgO and Co
 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

Figures 72 and 75 show an analysis of the patterns observed in the gravity separation within individual samples.

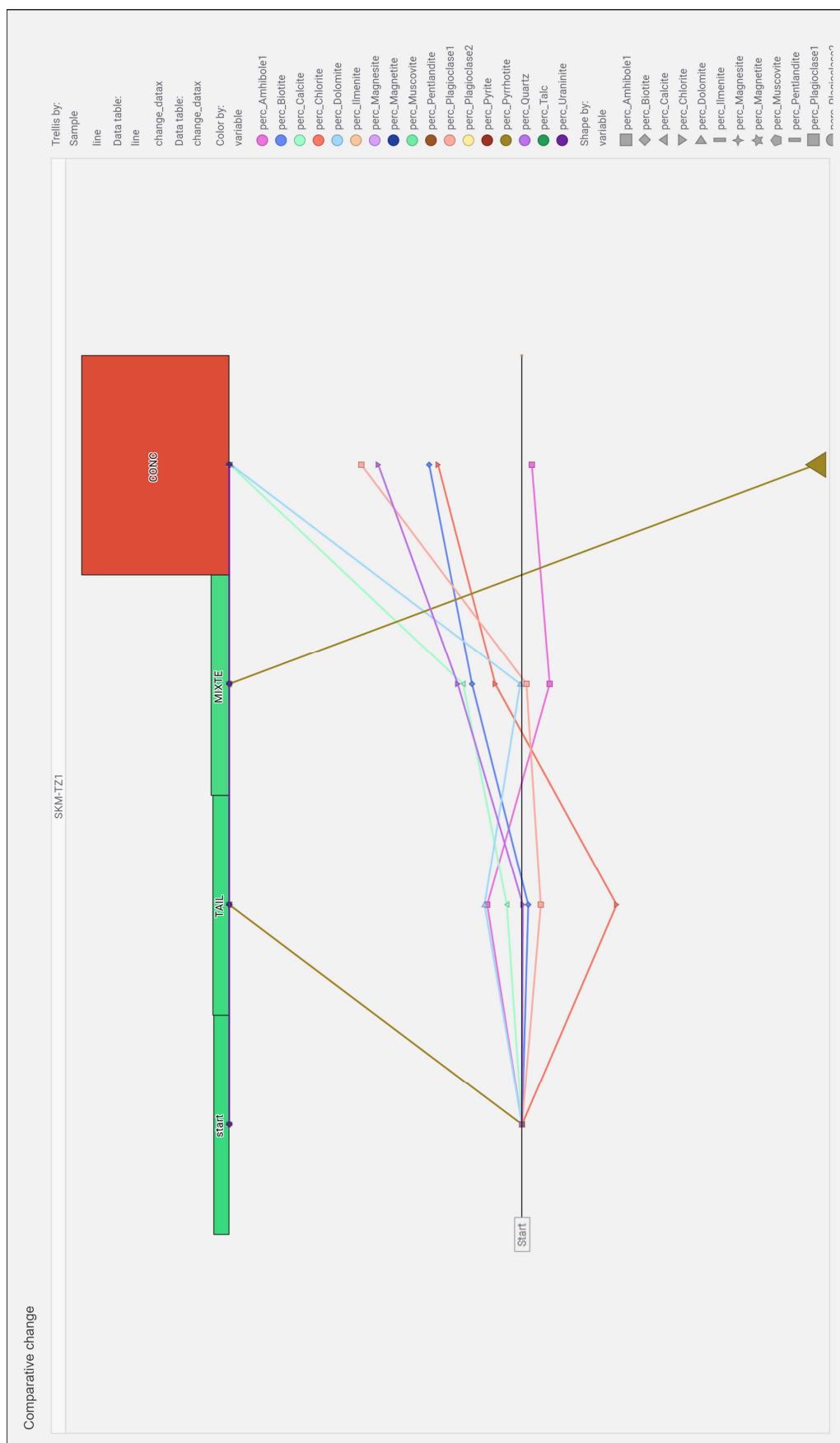


Figure 72. Element data relationship relative to Cobalt concentrate for gravity separation for sample SKM-TZ1
(Analysis done with Spotfire software by John Martin, Hafren Scientific)

The deportation of each element to the different magnetic separation fractions (concentrate, middlings and reject tailings) follow different patterns. Figures 73 and 76 show all the deportation of the measured elements into 5 clusters.

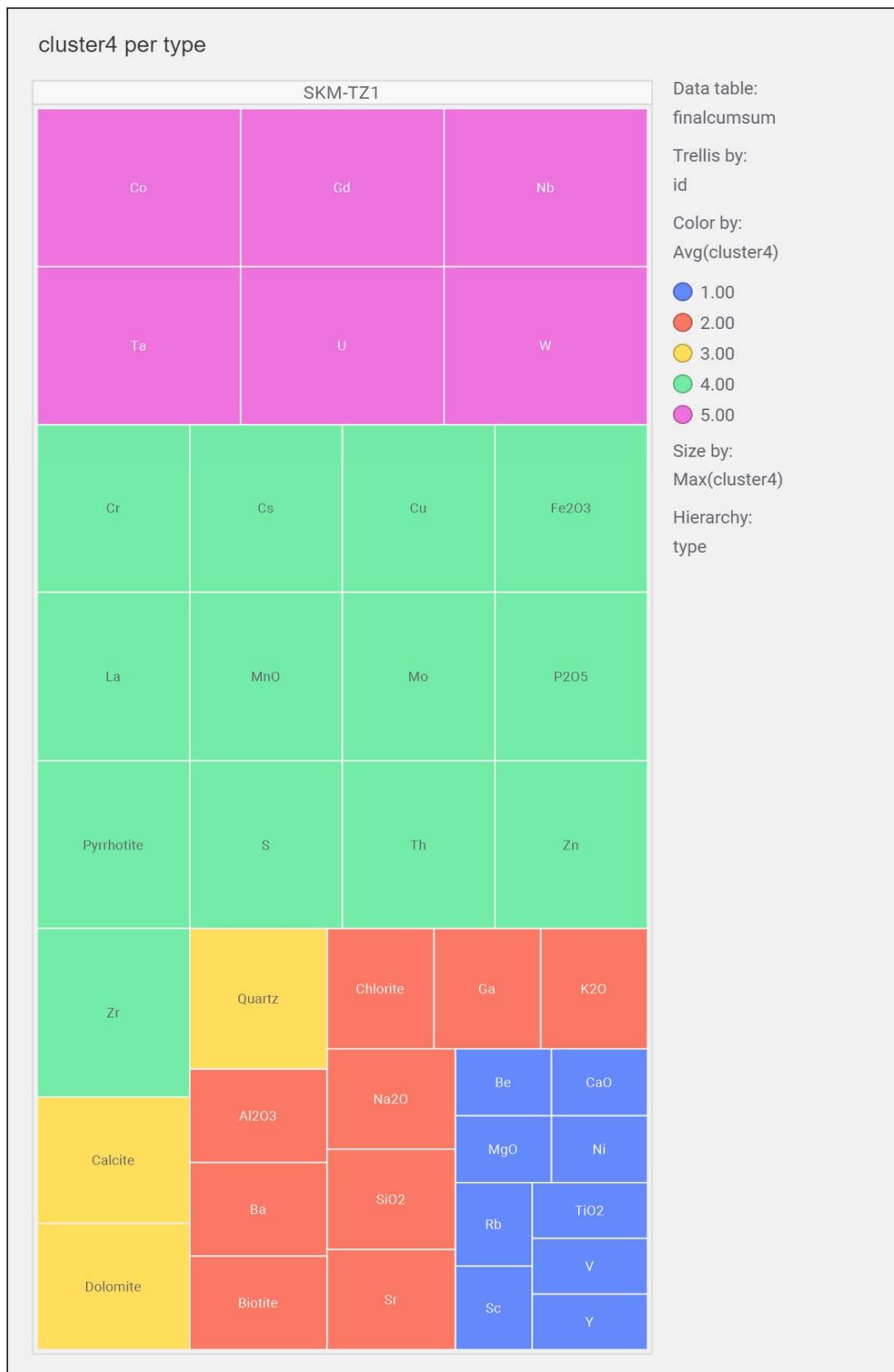


Figure 73. Element data correlation coefficient tree diagrams for elements across for gravity separation fractions for sample SKG-TZ1 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

Figures 74 and 77 show how these five different clusters present with different forms. It is postulated that each cluster represents a different mineral grouping.

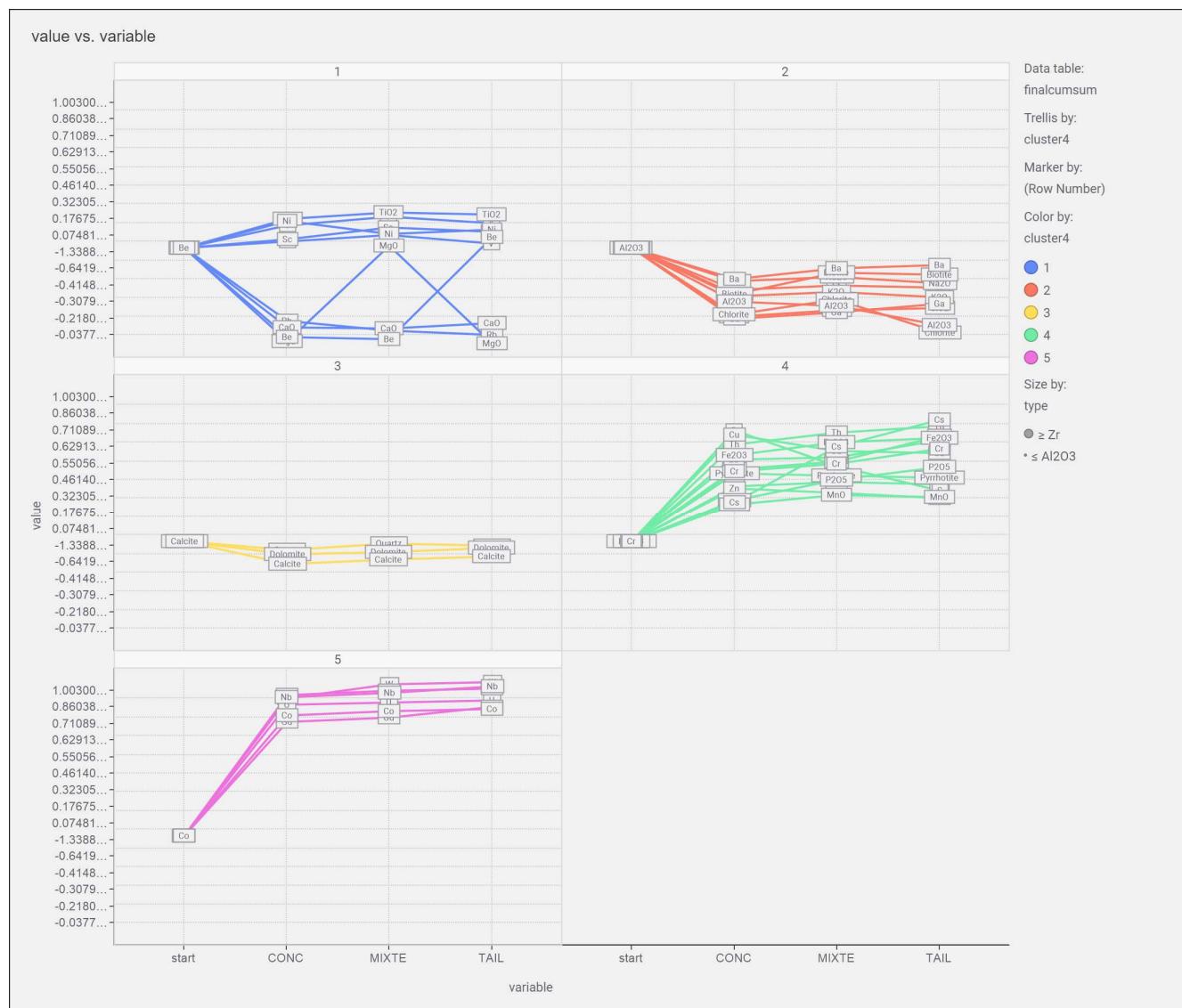


Figure 74. Element data relationship tree diagrams for elements across for gravity separation fractions for sample SKG-TZ1 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

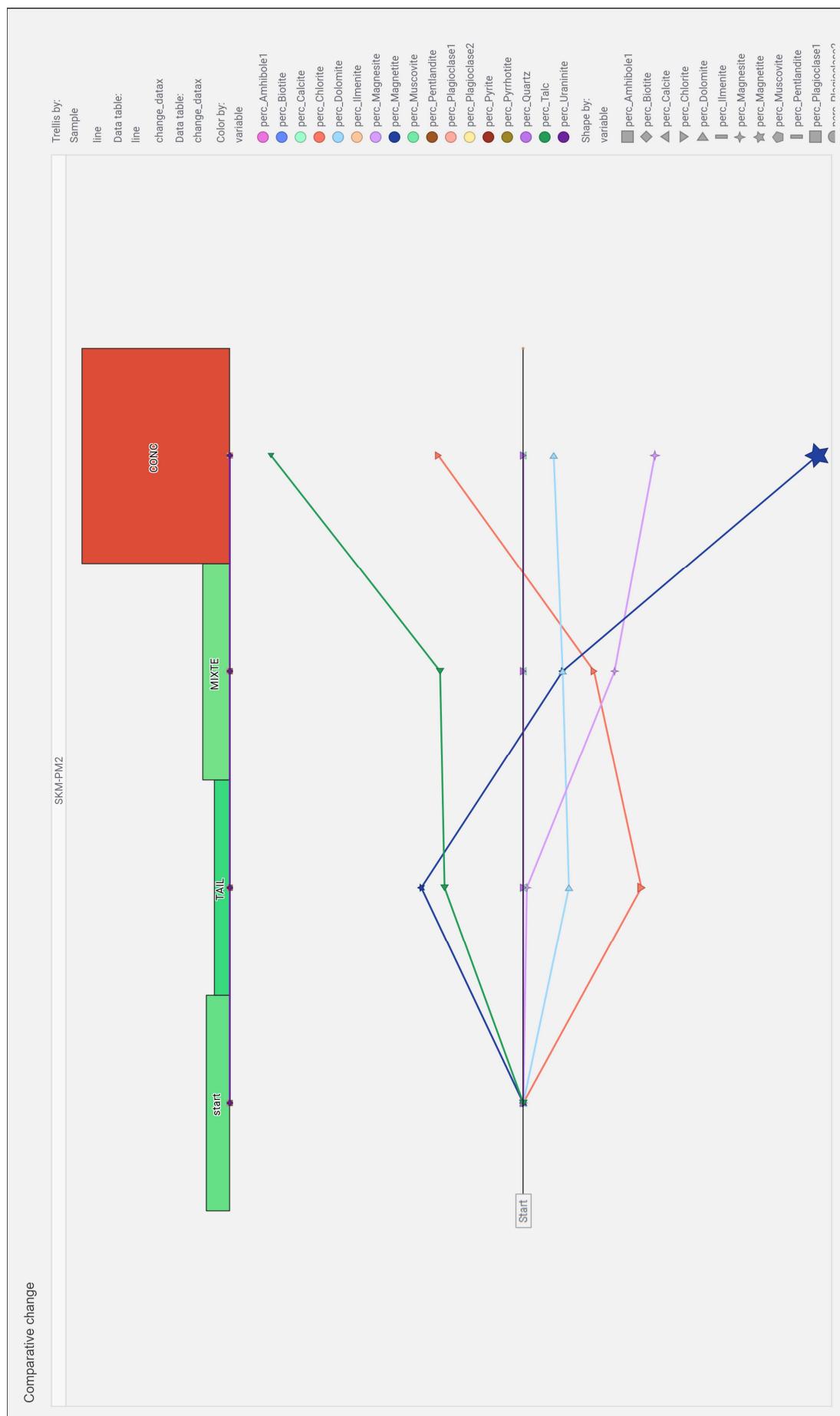


Figure 75. Element data relationship relative to Cobalt concentrate for gravity separation for sample SKG-PM2
(Analysis done with Spotfire software by John Martin, Hafren Scientific)

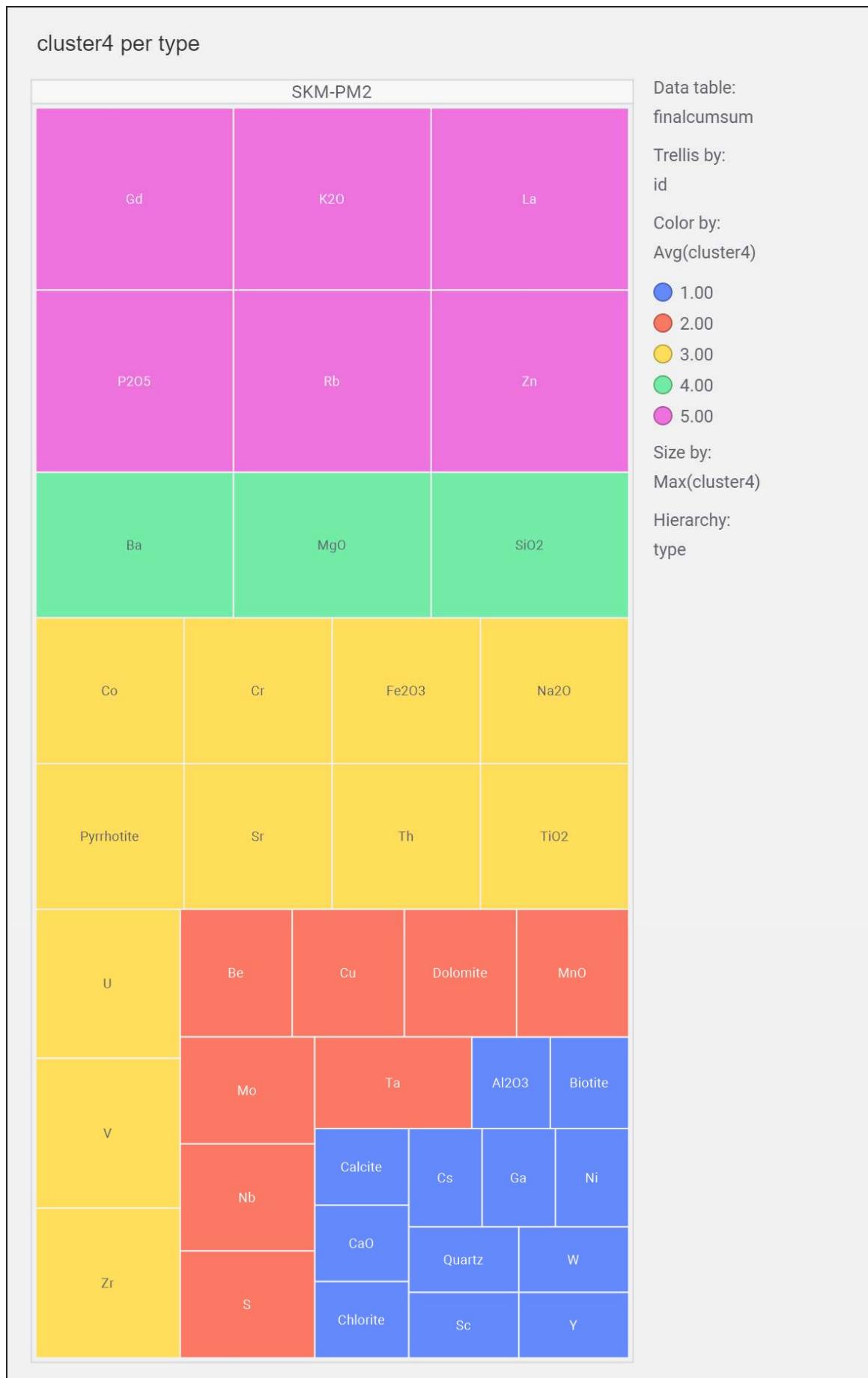


Figure 76. Element data correlation coefficient tree diagrams for elements across for gravity separation fractions for sample SKG-PM2 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

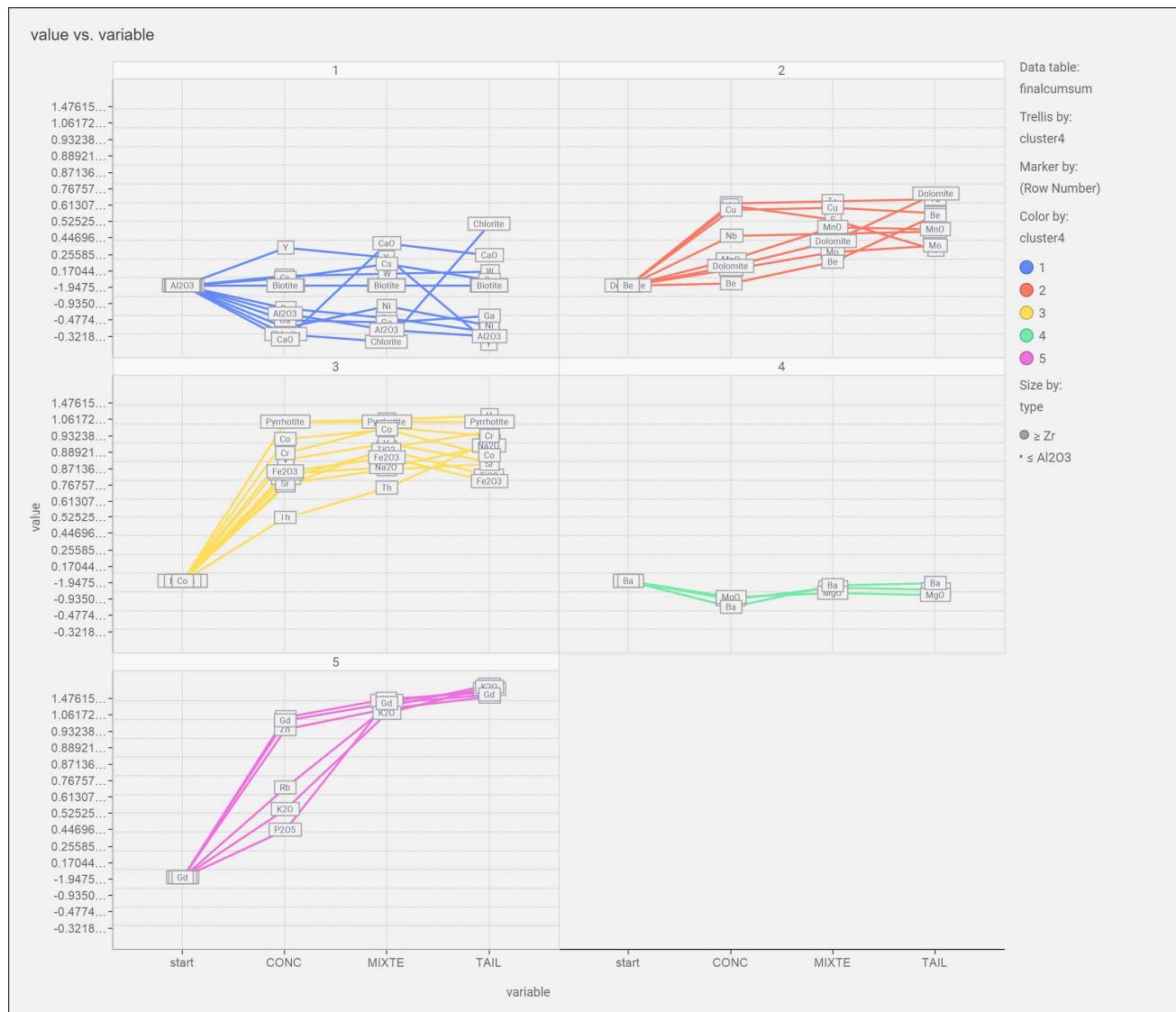


Figure 77. Element data relationship tree diagrams for elements across for gravity separation fractions for sample SKG-PM2 (Analysis done with Spotfire software by John Martin, Hafren Scientific)

Table 3 shows the sulfide mineral content in the gravity concentrate fraction. All of these minerals were below detection limit in the feed samples, which highlights the effectiveness in concentration.

Table 3. Sulfide mineral content in gravity concentrate fraction
(Semi-quantitative mineralogy (QXRD) (X-Ray Mineral Services))

Mineral in gravity concentrate	PM1 (%)	PM2 (%)	PX1 (%)	PX2 (%)	MS1 (%)	MS2 (%)	TZ1 (%)	TZ2 (%)	BAS1 (%)	BAS2 (%)
Pyrrhotite	7.15	5.22	14.50		17.15	4.51	16.40	22.92	14.62	
Pyrite					4.34	2.29	5.80	2.48	4.57	1.90
Chalcopyrite	2.93	1.86	2.77	1.80	0.80	3.68	3.04	2.10	1.45	8.07
Pentlandite	2.84		2.40		2.72	1.91	5.97	4.40	5.50	

Sum total 12.92% 7.08% 19.67% 1.80% 25.01% 12.39% 31.21% 31.90% 26.14% 9.97%

Figures 78 to 86 and Tables 5 to 8 show the precious metal recovery. Overall, some precious metals are susceptible to gravity recovery methods, but not enough to be considered as the primary method of extraction. It could be possible to use gravity separation as a secondary extraction method, however. Recovery ranges between 65% and 30%, with mass pull in the concentrate + middling fractions combined ranging from 25% to 45%.

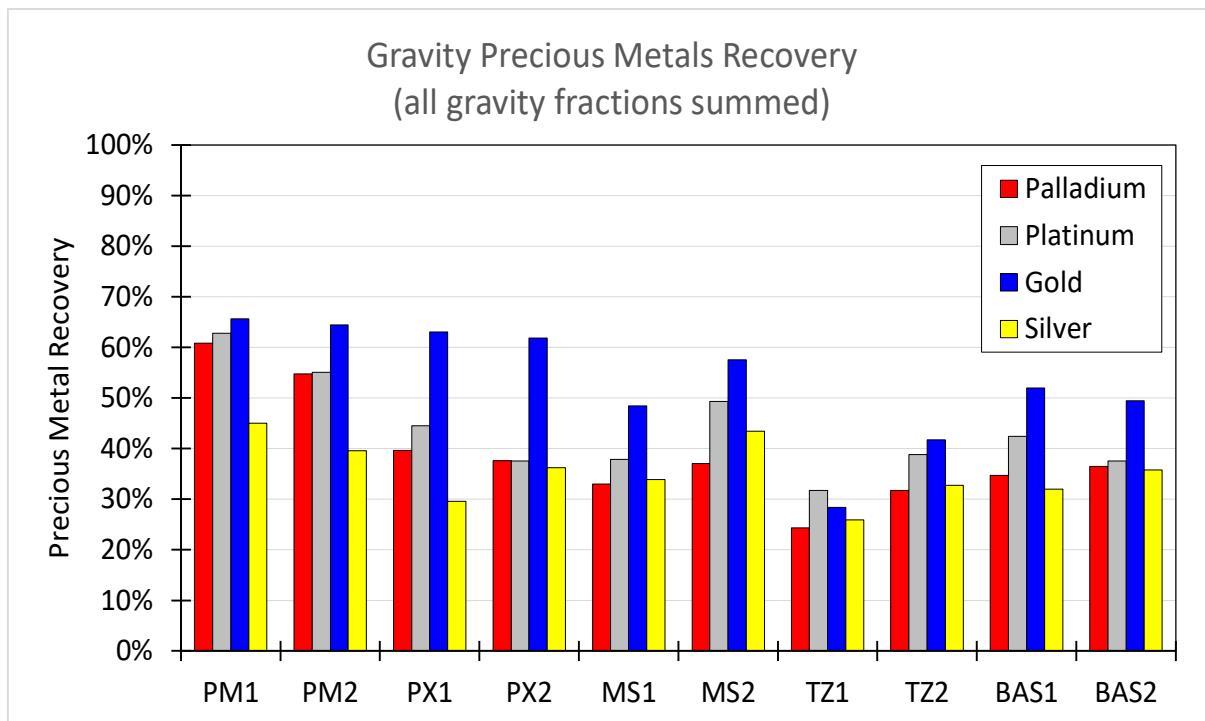


Figure 78. Mass balanced precious metal recovery from gravity separation (concentrate and middling fractions combined)
(Chemical Assay Method 711P)

Figures 87 to 92 and Tables 9 to 11 show the copper, nickel, and cobalt recovery. Again, it can be seen that some of the target valuable metals are recovered but not enough to consider gravity separation as a primary process separation method. Especially when it is remembered that these recoveries are associated with a large portion of the original sample.

Figures 93 to 96 and Tables 12 to 13 show the sulfur and carbon in each of the gravity separation fractions. Sulfur recovery was shown to be very good, with 80-90% of sulfur from the feed sample being deported to the gravity concentrate. This could have implications for tailings management.

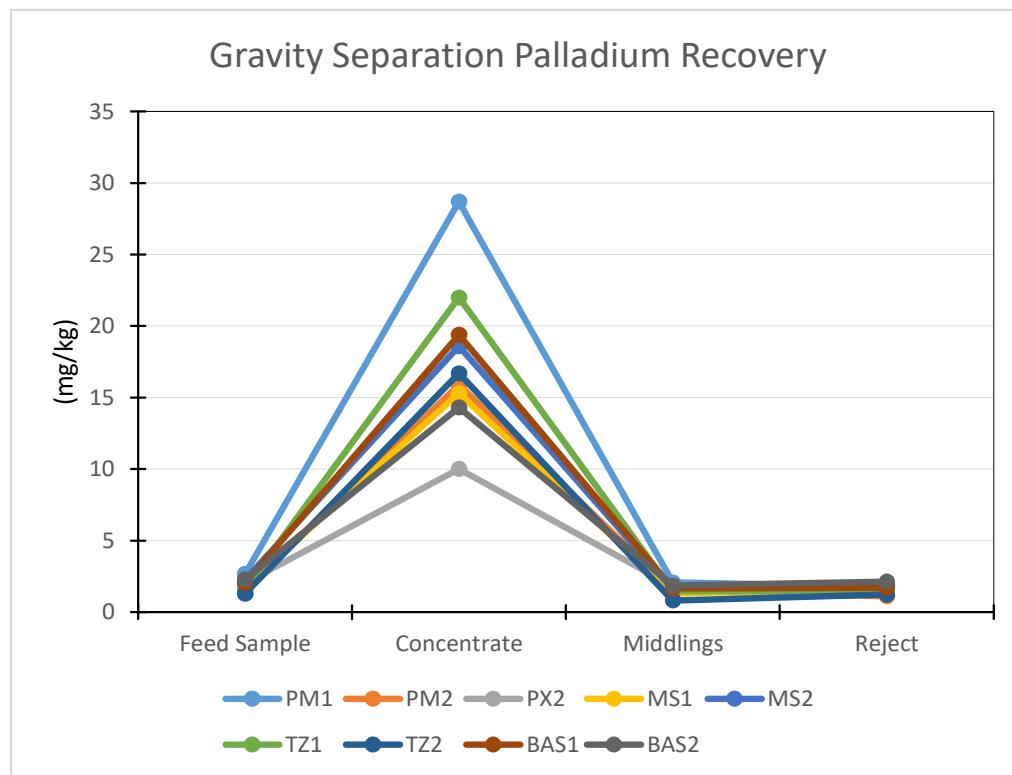


Figure 79. Palladium recovery from gravity separation (Chemical Assay Method 711P)

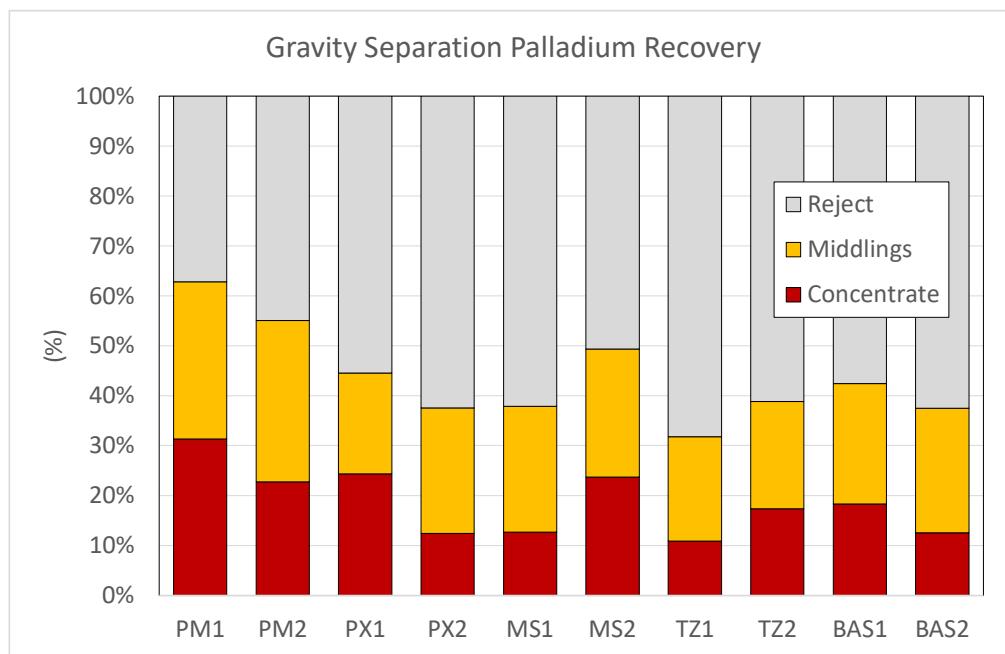


Figure 80. Mass balanced Palladium recovery from gravity separation (Chemical Assay Method 711P)

Table 4. Palladium recovery from gravity separation (Chemical Assay Method 711P)

Palladium Content (Fire assay, 711P)	PM1 (mg/kg)	PM2 (mg/kg)	PX1 (mg/kg)	PX2 (mg/kg)	MS1 (mg/kg)	MS2 (mg/kg)	TZ1 (mg/kg)	TZ2 (mg/kg)	BAS1 (mg/kg)	BAS2 (mg/kg)
Feed Sample	2.7	1.6	2.0	2.0	1.7	2.3	1.7	1.3	2.0	2.28
Concentrate	28.7	15.9	25.9	10.0	15.3	18.6	22.0	16.7	19.4	14.30
Middlings	2.1	1.5	1.5	1.8	1.3	1.6	1.5	0.8	1.7	1.86
Reject Tails	1.8	1.1	1.6	1.8	1.6	1.9	1.5	1.2	1.7	2.12

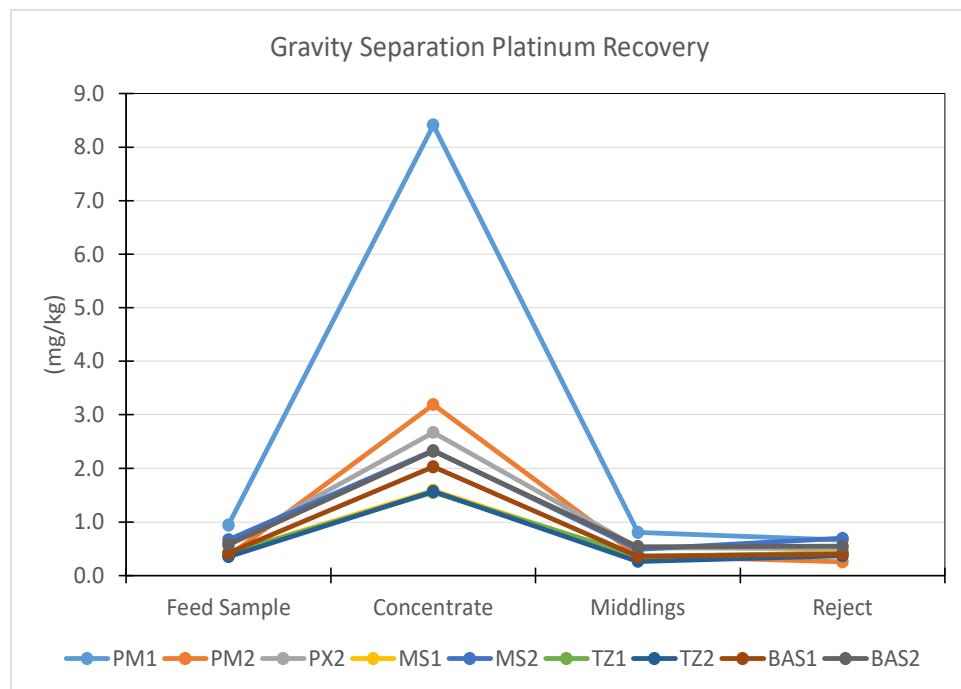


Figure 81. Platinum recovery from gravity separation (Chemical Assay Method 711P)

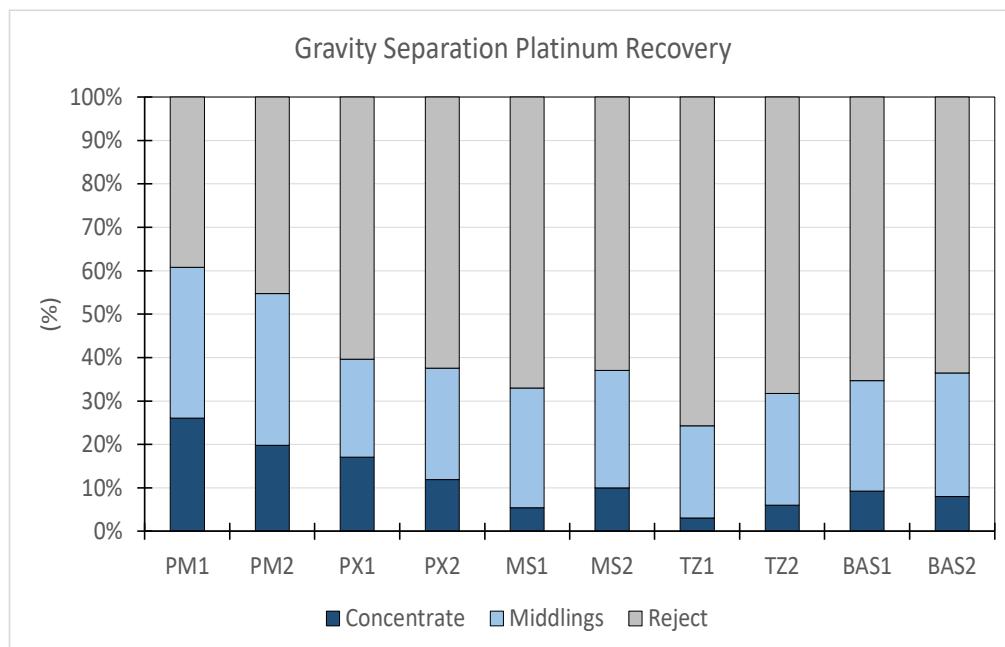


Figure 82. Mass balanced Platinum recovery from gravity separation (Chemical Assay Method 711P)

Table 5. Platinum recovery from gravity separation (Chemical Assay Method 711P)

Platinum Content (Fire assay, 711P)	PM1 (mg/kg)	PM2 (mg/kg)	PX1 (mg/kg)	PX2 (mg/kg)	MS1 (mg/kg)	MS2 (mg/kg)	TZ1 (mg/kg)	TZ2 (mg/kg)	BAS1 (mg/kg)	BAS2 (mg/kg)
Feed Sample	0.94	0.36	0.57	0.56	0.42	0.67	0.41	0.35	0.42	0.58
Concentrate	8.41	3.19	5.23	2.67	1.59	2.33	1.55	1.57	2.03	2.32
Middlings	0.81	0.37	0.47	0.52	0.35	0.49	0.37	0.26	0.36	0.54
Reject Tails	0.66	0.26	0.49	0.50	0.43	0.70	0.41	0.37	0.40	0.55

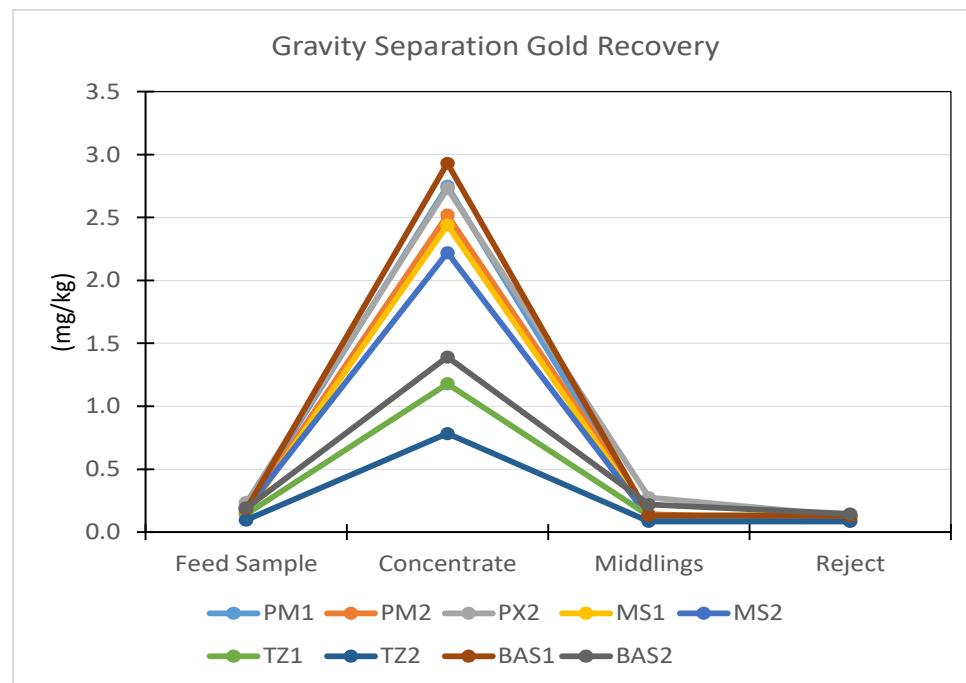


Figure 83. Gold recovery from gravity separation (Chemical Assay Method 711P)

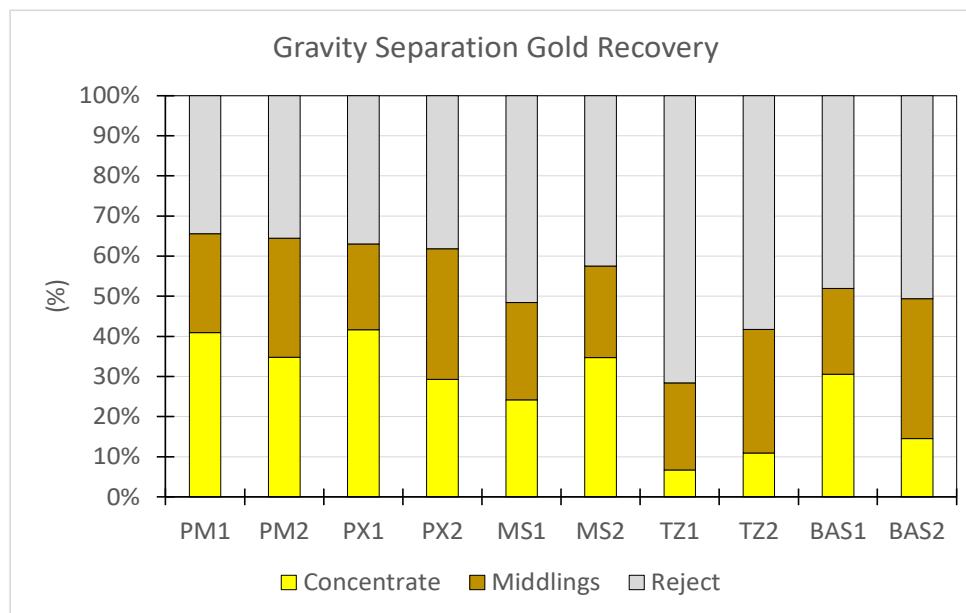


Figure 84. Mass balanced Gold recovery from gravity separation (Chemical Assay Method 711P)

Table 6. Gold recovery from gravity separation (Chemical Assay Method 711P)

Gold Content (Fire assay, 711P)	PM1 (mg/kg)	PM2 (mg/kg)	PX1 (mg/kg)	PX2 (mg/kg)	MS1 (mg/kg)	MS2 (mg/kg)	TZ1 (mg/kg)	TZ2 (mg/kg)	BAS1 (mg/kg)	BAS2 (mg/kg)
Feed Sample	0.20	0.16	0.17	0.23	0.14	0.18	0.14	0.10	0.18	0.19
Concentrate	2.75	2.52	3.73	2.73	2.44	2.22	1.18	0.78	2.93	1.39
Middlings	0.12	0.14	0.13	0.27	0.11	0.11	0.13	0.09	0.13	0.22
Reject Tails	0.12	0.09	0.09	0.13	0.11	0.13	0.14	0.09	0.13	0.14

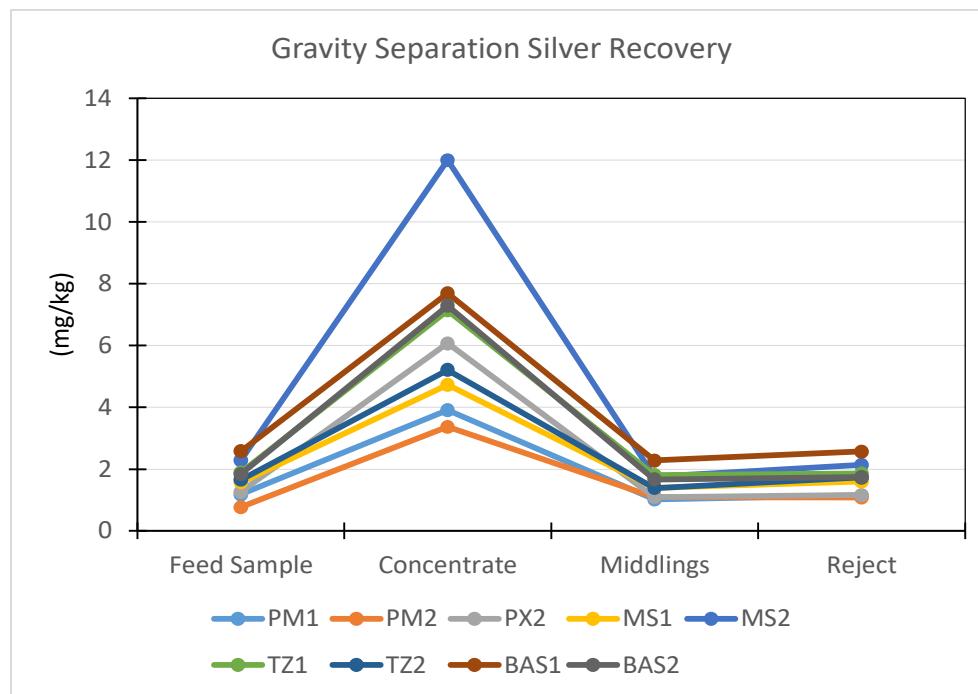


Figure 85. Silver recovery from gravity separation (Chemical Assay Method 711P)

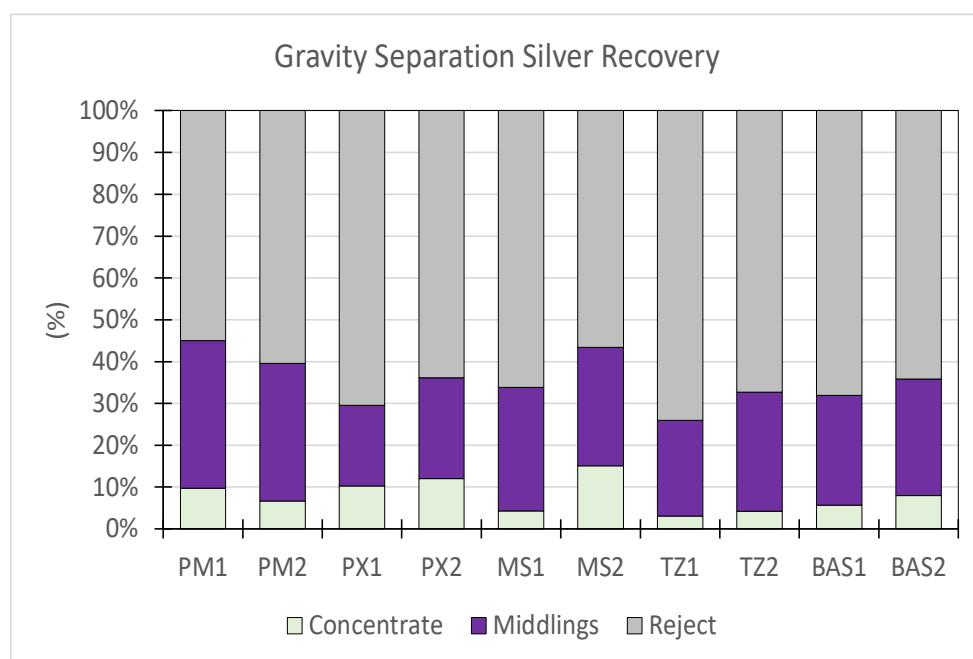


Figure 86. Mass balanced Silver recovery from gravity separation (Chemical Assay Method 711P)

Table 7. Silver recovery from gravity separation (Chemical Assay Method 711P)

Silver Content (Fire assay, 711P)	PM1 (mg/kg)	PM2 (mg/kg)	PX1 (mg/kg)	PX2 (mg/kg)	MS1 (mg/kg)	MS2 (mg/kg)	TZ1 (mg/kg)	TZ2 (mg/kg)	BAS1 (mg/kg)	BAS2 (mg/kg)
Feed Sample	1.18	0.77	1.28	1.27	1.58	2.28	1.89	1.66	2.58	1.84
Concentrate	3.91	3.37	7.04	6.07	4.73	12.00	7.14	5.21	7.69	7.29
Middlings	1.02	1.10	0.91	1.10	1.40	1.76	1.81	1.39	2.28	1.67
Reject Tails	1.15	1.08	1.27	1.16	1.60	2.14	1.86	1.73	2.57	1.75

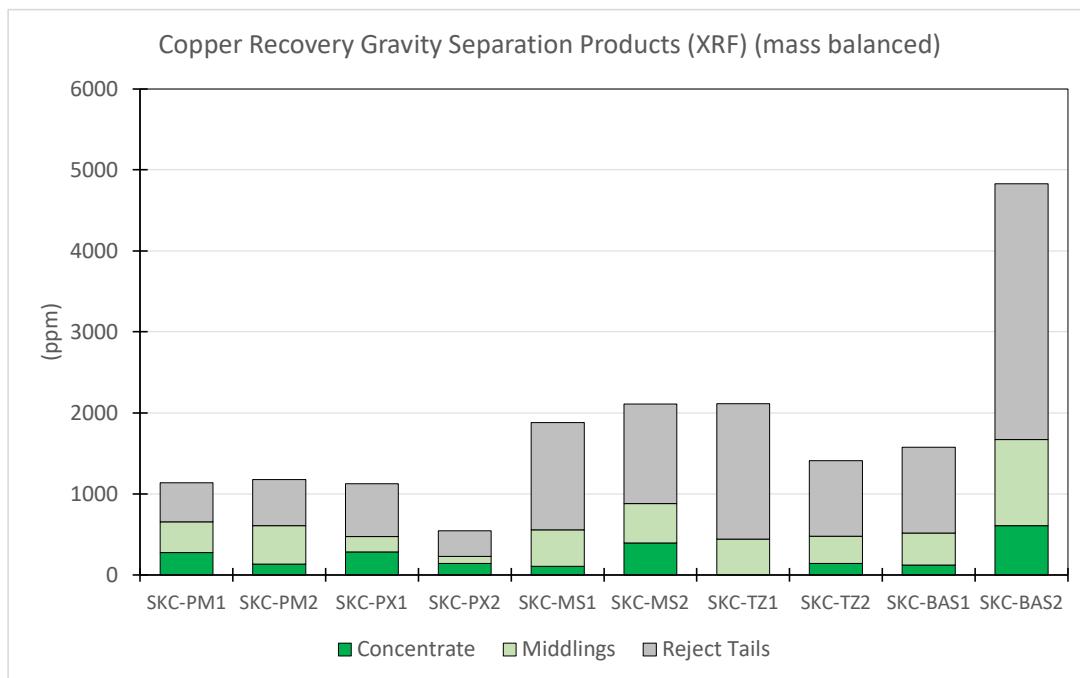


Figure 87. Mass balanced Copper recovery from gravity separation (XRF) (Measurement X-Ray Mineral Services)

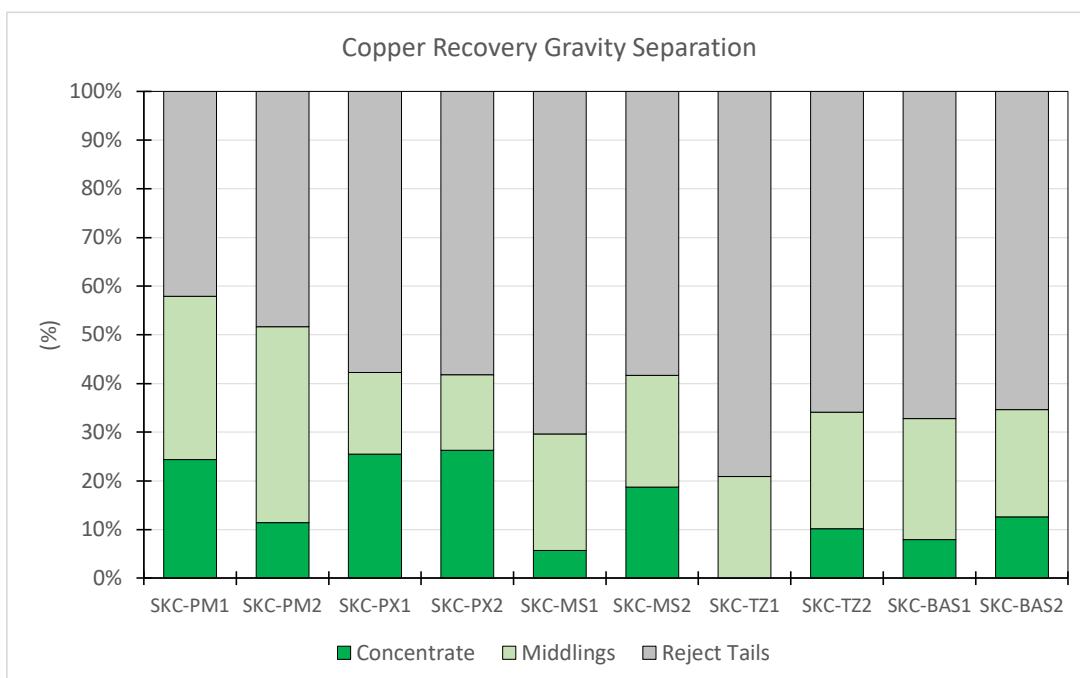


Figure 88. Mass balanced Copper recovery from gravity separation (XRF) (Measurement X-Ray Mineral Services)

Table 8. Mass balanced Copper recovery from gravity separation (XRF) (Measurement X-Ray Mineral Services)

Copper content (XRF)	SKC-PM1 (ppm)	SKC-PM2 (ppm)	SKC-PX1 (ppm)	SKC-PX2 (ppm)	SKC-MS1 (ppm)	SKC-MS2 (ppm)	SKC-TZ1 (ppm)	SKC-TZ2 (ppm)	SKC-BAS1 (ppm)	SKC-BAS2 (ppm)
*Feed Sample Concentrate	1137	1177	1124	543	1882	2112	2114	1411	1577	4829
Middlings	278	134	287	143	108	396	0	143	125	610
Reject Tails	380	474	189	84	450	485	441	338	392	1061
	479	569	649	316	1324	1232	1672	930	1060	3158

*Back calculated

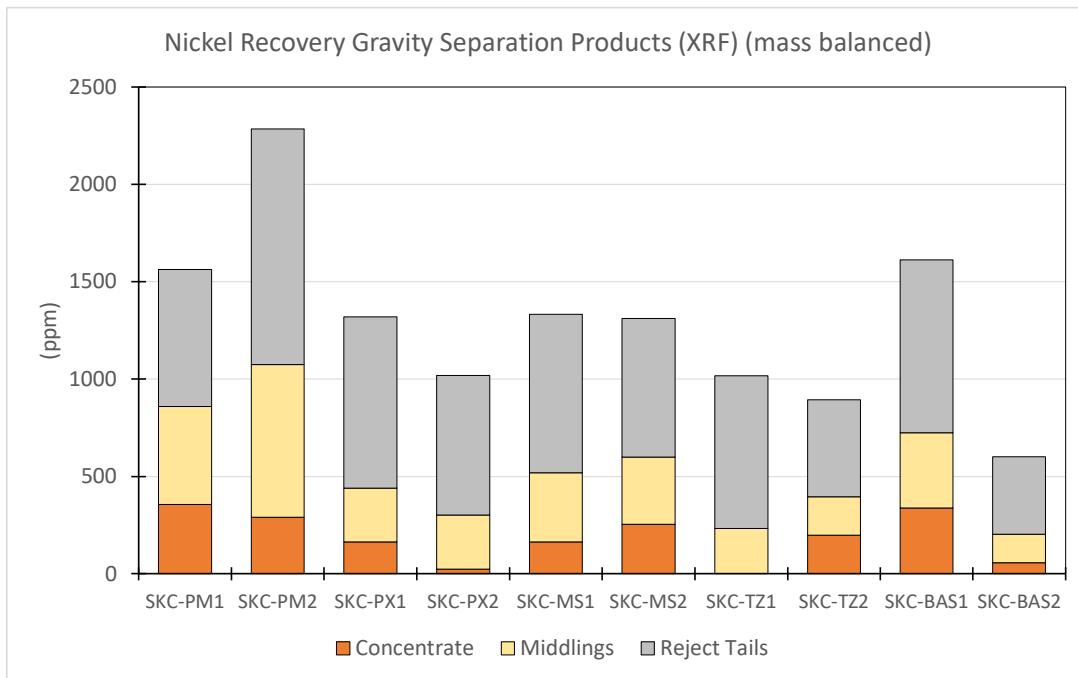


Figure 89. Mass balanced Nickel recovery from gravity separation (XRF) (Measurement X-Ray Mineral Services)

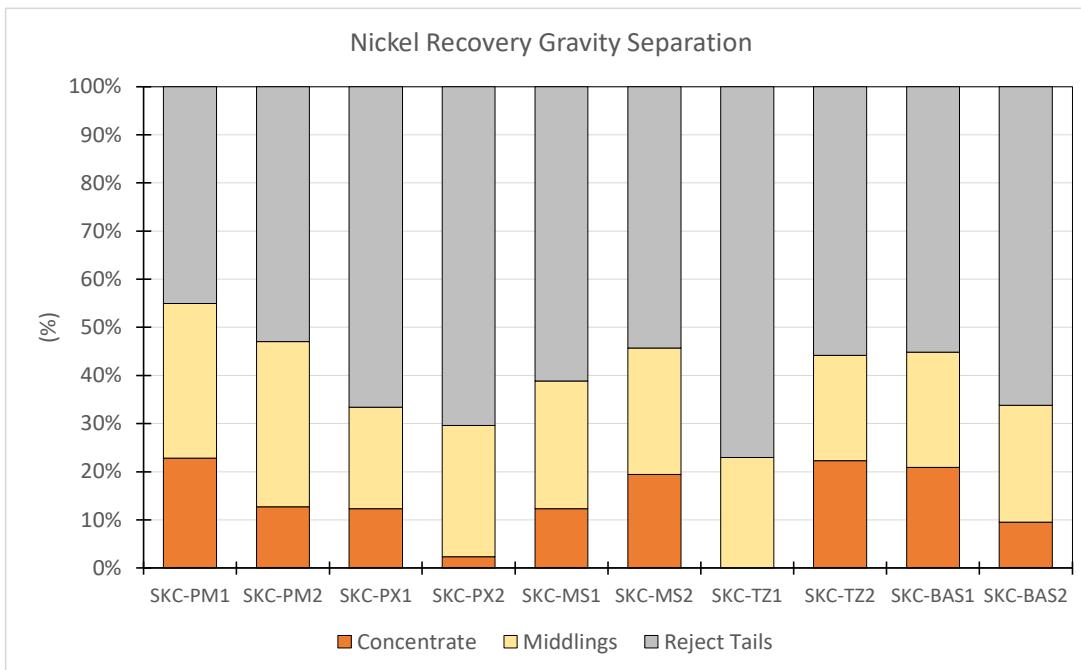


Figure 90. Mass balanced Nickel recovery from gravity separation (XRF) (Measurement X-Ray Mineral Services)

Table 9. Mass balanced Nickel recovery from gravity separation (XRF) (Measurement X-Ray Mineral Services)

Nickel content (XRF)	SKC-PM1 (ppm)	SKC-PM2 (ppm)	SKC-PX1 (ppm)	SKC-PX2 (ppm)	SKC-MS1 (ppm)	SKC-MS2 (ppm)	SKC-TZ1 (ppm)	SKC-TZ2 (ppm)	SKC-BAS1 (ppm)	SKC-BAS2 (ppm)
*Feed Sample	1562	2284	1319	1019	1333	1312	1016	893	1612	600
Concentrate	357	290	162	24	164	255	0	199	337	57
Middlings	501	784	278	278	353	344	233	196	386	146
Reject Tails	704	1210	879	717	815	712	783	498	889	397

*Back calculated

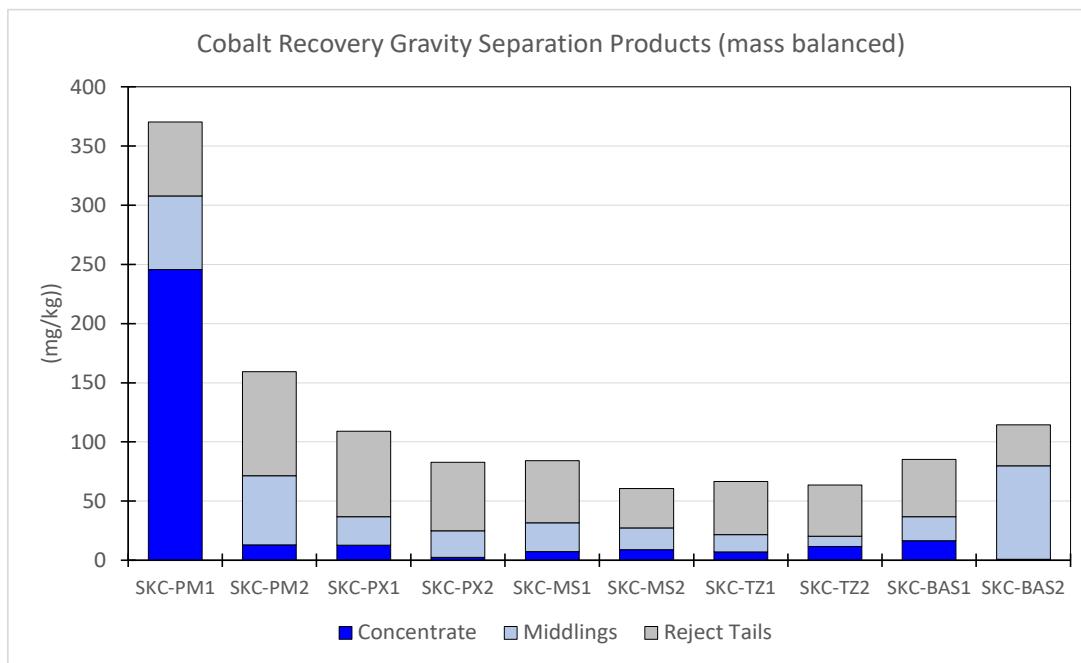


Figure 91. Mass balanced Cobalt recovery from gravity separation (4 Acid digest chemical assay, method 306P)

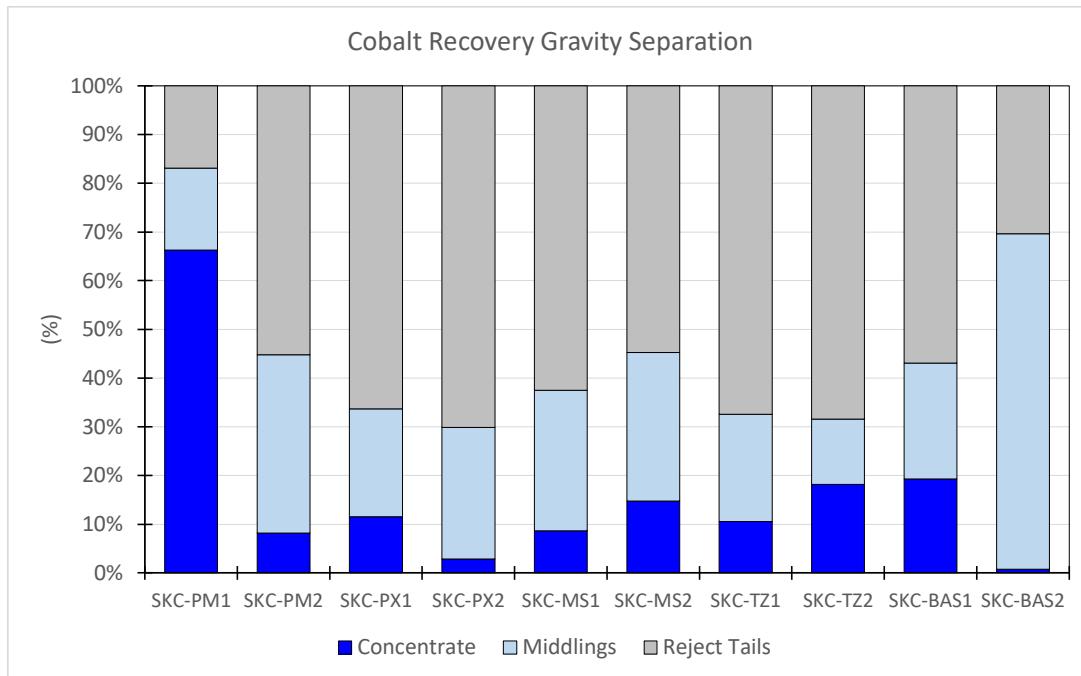


Figure 92. Mass balanced Cobalt recovery from gravity separation (4 Acid digest chemical assay, method 306P)

Table 10. Mass balanced Cobalt recovery from gravity separation (4 Acid digest chemical assay, method 306P)

Cobalt content (Multi element ICP 306P)	SKC-PM1 (mg/kg)	SKC-PM2 (mg/kg)	SKC-PX1 (mg/kg)	SKC-PX2 (mg/kg)	SKC-MS1 (mg/kg)	SKC-MS2 (mg/kg)	SKC-TZ1 (mg/kg)	SKC-TZ2 (mg/kg)	SKC-BAS1 (mg/kg)	SKC-BAS2 (mg/kg)
*Feed Sample	370	159	109	83	84	61	66	64	85	114
Concentrate	245	13	13	2	7	9	7	12	16	1
Middlings	63	58	24	22	24	18	15	9	20	79
Reject Tails	63	88	72	58	53	33	45	44	48	35

*Back calculated

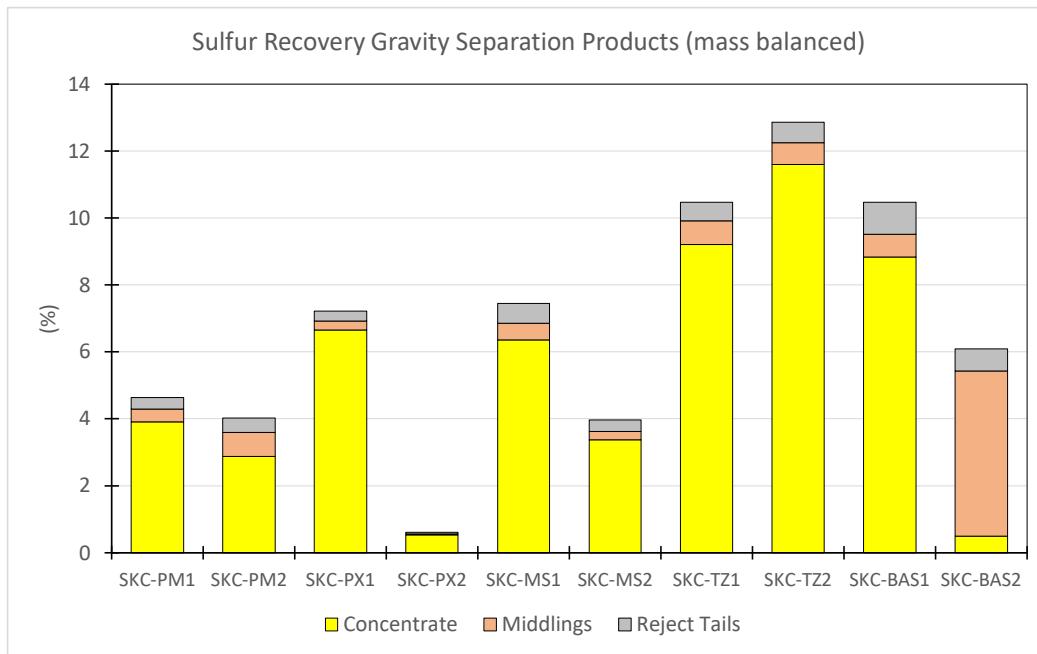


Figure 93. Mass balanced Sulfur recovery from gravity separation (ELTRA-S chemical assay, method 810L)

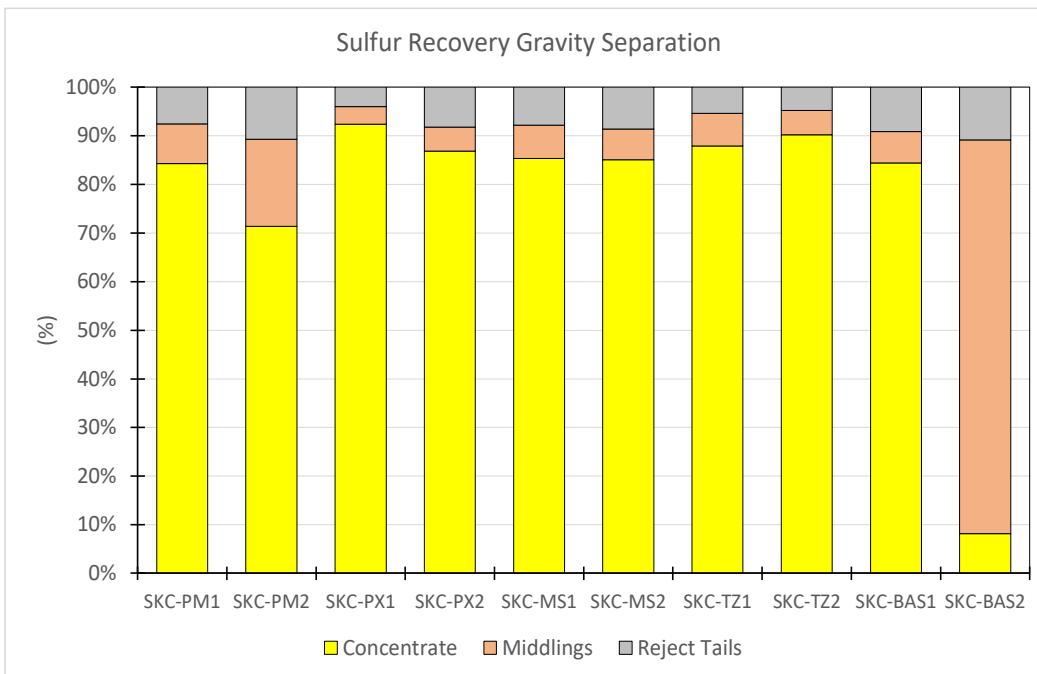


Figure 94. Mass balanced Sulfur recovery from gravity separation (ELTRA-S chemical assay, method 810L)

Table 11. Mass balanced Sulfur recovery from gravity separation (ELTRA-S chemical assay, method 810L)

Sulfur content (Eltra S assay)	SKC-PM1 (mg/kg)	SKC-PM2 (mg/kg)	SKC-PX1 (mg/kg)	SKC-PX2 (mg/kg)	SKC-MS1 (mg/kg)	SKC-MS2 (mg/kg)	SKC-TZ1 (mg/kg)	SKC-TZ2 (mg/kg)	SKC-BAS1 (mg/kg)	SKC-BAS2 (mg/kg)
*Feed Sample	4.64	4.02	7.21	0.61	7.44	3.96	10.47	12.86	10.47	6.09
Concentrate	3.91	2.87	6.66	0.53	6.35	3.37	9.21	11.6	8.84	0.5
Middlings	0.38	0.72	0.26	0.03	0.51	0.25	0.7	0.65	0.67	4.93
Reject Tails	0.35	0.43	0.29	0.05	0.58	0.34	0.56	0.61	0.96	0.66

*Back calculated

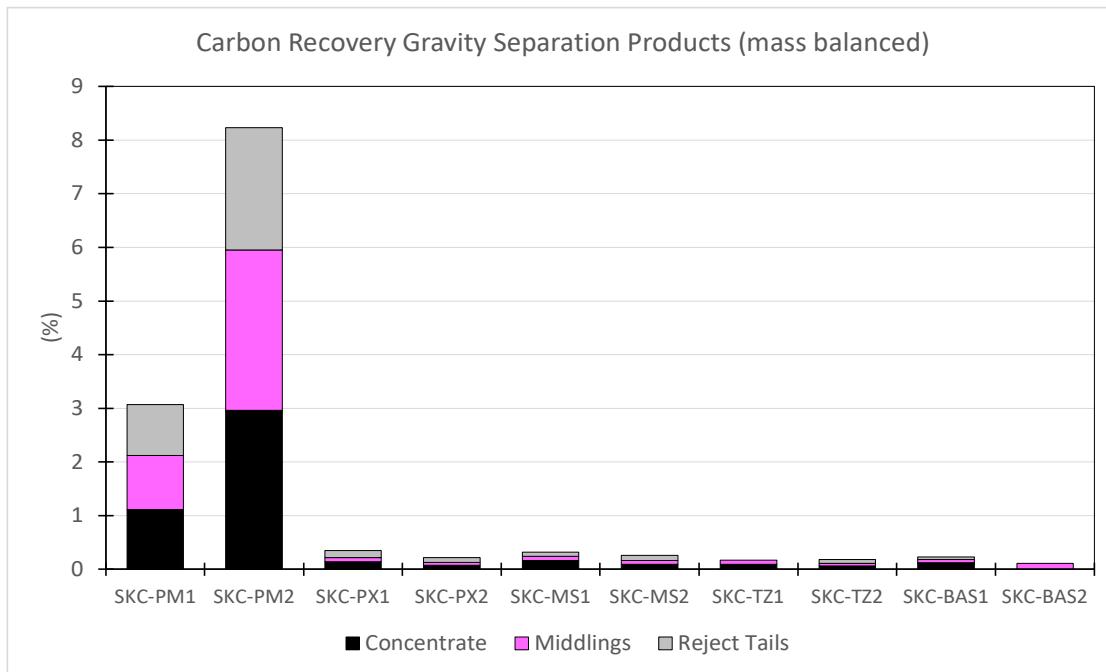


Figure 95. Mass balanced Carbon recovery from gravity separation (ELTRA-C chemical assay, method 811L)

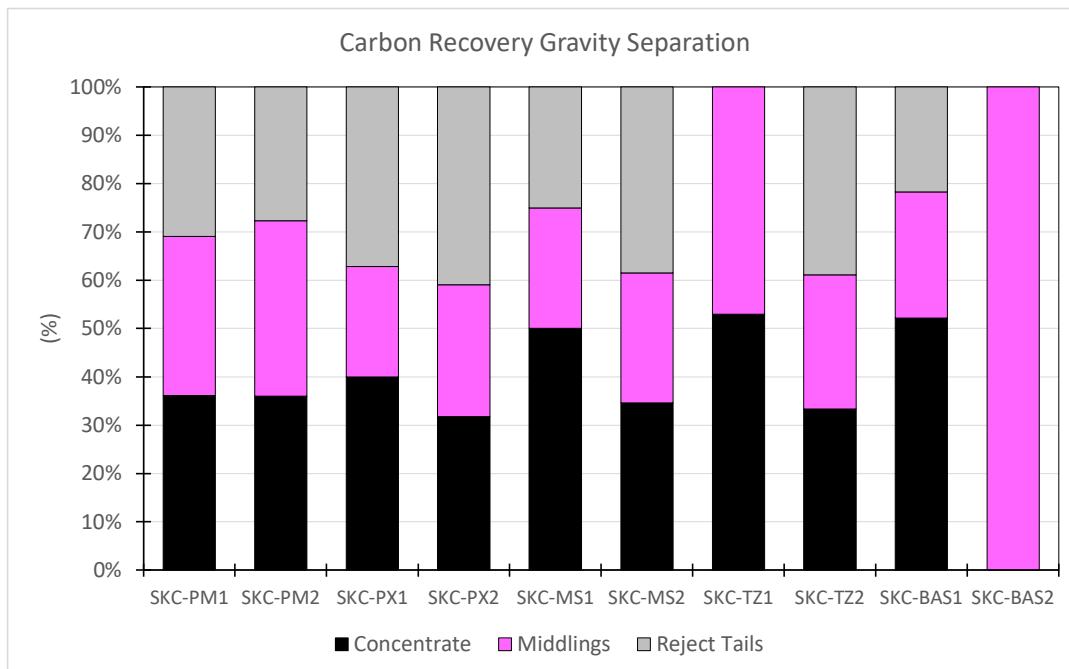


Figure 96. Mass balanced Carbon recovery from gravity separation (ELTRA-C chemical assay, method 811L)

Table 12. Mass balanced Carbon recovery from gravity separation (ELTRA-C chemical assay, method 811L)

Carbon content (Eltra C assay)	SKC-PM1 (mg/kg)	SKC-PM2 (mg/kg)	SKC-PX1 (mg/kg)	SKC-PX2 (mg/kg)	SKC-MS1 (mg/kg)	SKC-MS2 (mg/kg)	SKC-TZ1 (mg/kg)	SKC-TZ2 (mg/kg)	SKC-BAS1 (mg/kg)	SKC-BAS2 (mg/kg)
*Feed Sample	3.07	8.23	0.35	0.22	0.32	0.26	0.17	0.18	0.23	0.11
Concentrate	1.11	2.96	0.14	0.07	0.16	0.09	0.09	0.06	0.12	
Middlings	1.01	2.99	0.08	0.06	0.08	0.07	0.08	0.05	0.06	0.11
Reject Tails	0.95	2.28	0.13	0.09	0.08	0.1		0.07	0.05	

*Back calculated

8 FLOTATION

8.1 Methods/Procedure

Bulk flotation tests were done for the nine of the Konttijärvi Orientation Samples. The general simplified flotation flowsheet is presented in Figure 97 and the used test conditions in Table 14. The crushed -3.35 mm feed material was ground with the mild steel rod mill, the used ore feed batch was 1 kg, solids content in grinding about 53 % and rod charge 8 kg. The target grinding closing size was a D₈₀ of 60 µm. The actual resulting grinding closing size varied slightly depending on the ore sample with a range of D₈₀ 48-64 µm.

Flotation test 1 was carried out in a Outotec-GTK LabCellTM laboratory machine. Tests 2-10 were carried out in a Outokumpu type GTK flotation machine, which was very similar to the commercial model ("Outotec-GTK LabCellTM"). The rougher flotation was done in a 4 liter cell, with a mixing speed of 1800 rpm. The solids content in rougher stage was approximately 25 % (1kg/4 liter). The cleaner stages were done with 2.5 liter or 1.5 liter cell and mixing speeds were 1500 rpm and 1100 rpm respectively. The solids contents varied depending on the sample and cleaning stage. The air flow was 3L/min in rougher stage when 4 liter cell was used and 2 liter/min in the cleanings when 2.5 liter cell was used and 1.5 liter/min when 1.5 liter cell was used (Table 13. Konttijärvi flotation experimental test conditions).

After the grinding, the rougher bulk flotation was performed in four stages. The rougher concentrates RC1-4 were combined for the cleaning flotation stage. For most of the samples only one cleaning was done due to the rather small amount of rougher concentrate. In the final cleaning flotation stage, several samples were taken as time series, in order to see flotation kinetics which was challenging due the small amount of concentrates.

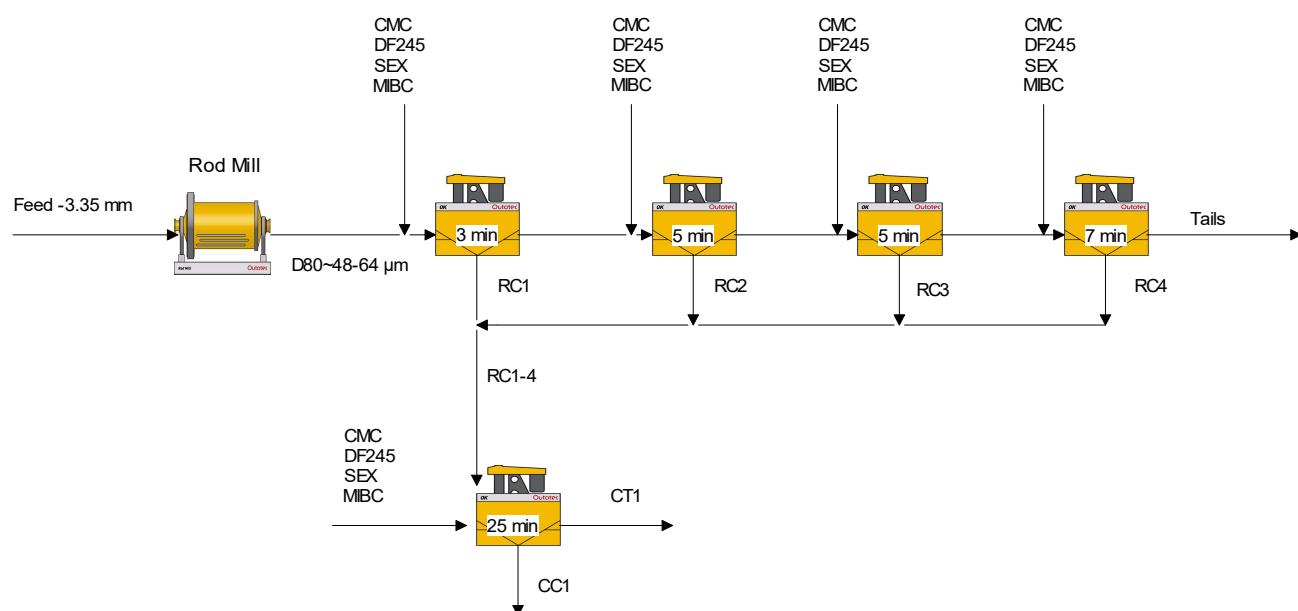


Figure 97. Konttijärvi flotation tests, general flowsheet, flotation tests 3-10

During all flotation tests, the pH was measured with a pH meter (Metrohm 877 Titrino Plus). An overview of the flotation procedure with the timing followed for each of the 10 flowsheets can be found in (Table 13). The flotation chemicals were fed before every flotation stage as in the Figure 97. It should be noted that flotation conditions were not optimized for the all separate ore samples.

Generally, the similar chemical dosages were used for the all tested samples. Chemicals used:

- Sodium carboxymethyl cellulose, (trademark is CMC Celect HPD, producer CP Kelco, Finland), was used to depress silicates
- Sodium Ethyl Xanthate (SEX) provided by Supplier and Danafloat F245 (Sodium 0, 0-diisobutyl dithiophosphate) was used as collectors
- Methyl Isobutyl Carbinol (MIBC), was used as a frother in all the experiments

Table 13. Konttijärvi flotation experimental test conditions

Test	Sample	Grinding		Flotation					Reagents			
		Time min	D80 μm	Product	Time min	Cell I	Rounds rpm	Air l/min	CMC g/t	SEX g/t	DF245 g/t	MIBC g/t
1	SKF-PM2	62	64	RC1-4	25	4	1800	3	200	60	60	40
				CC1	15	2.5	1500	2	10	2	2	3
				CC2	13	2.5	1500	2	10	1	1	
				CC3	13	2.5	1500	2	10	1	1	
				Total	66				230	64	64	43
2	SKF-PM2	62	64	RC1-4	20	4	1800	3	500	60	60	40
				CC1	8	2.5	1500	2	50	7	7	8
				CC2	17	1.5	1100	1.5	20	15	15	15
				Total	45				570	82	82	63
3	SKF-MS1	40	63	RC1-4	20	4	1800	3	500	60	60	40
				CC1	25	1.5	1100	1.5	10	45	45	40
				Total	45				510	105	105	80
4	SKF-MS2	40	56	RC1-4	20	4	1800	3	500	60	60	40
				CC1	25	1.5	1100	1.5	10	45	45	50
				Total	45				510	105	105	90
5	SKF-TZ1	48	62	RC1-4	20	4	1800	3	500	60	60	40
				CC1	25	1.5	1100	1.5	10	45	45	50
				Total	45				510	105	105	90
6	SKF-TZ2	47	62	RC1-4	20	4	1800	3	500	60	60	40
				CC1	25	1.5	1100	1.5	10	45	45	50
				Total	45				510	105	105	90
7	SKF-BAS1	37	63	RC1-4	20	4	1800	3	500	60	60	40
				CC1	25	1.5	1100	1.5	10	45	45	50
				Total	45				510	105	105	90
8	SKF-BAS2	39	63	RC1-4	20	4	1800	3	500	60	60	40
				CC1	25	1.5	1100	1.5	10	45	45	50
				Total	45				510	105	105	90
9	SKF-PX1	34	48	RC1-4	20	4	1800	3	500	60	60	40
				CC1	25	1.5	1100	1.5	10	45	45	50
				Total	45				510	105	105	90
10	SKF-PX2	25	53	RC1-4	20	4	1800	3	500	60	60	40
				CC1	25	1.5	1100	1.5	10	50	50	50
				Total	45				510	110	110	90

8.2 Flotation Results

The copper and nickel grades and recoveries in bulk flotation tests are presented in Tables 14, 15 and 16 and in Figures 98 to 118. In Test 1 with sample SKF-PM2, the mass pull into rougher concentrate and 3rd cleaner concentrate was very high. The silicate gangue minerals were not depressed effectively and for this reason the copper and nickel grades were very low in rougher concentrate, Cu 0.16 % and Ni 0.29 %. The 2nd flotation test was done with same ore sample and the depressant dosage (CMC) was

increased from 200 g/t to 500 g/t in rougher flotation which improved slightly the quality of the rougher concentrate. The copper recovery was 86 % with 0.26 % grade and nickel recovery 53 % with 0.38 % grade. After the 2nd cleaning it was achieved 3.9 % copper grade with 80.5 % recovery and 3 % nickel grade with 27 % recovery. The flotation tests with other ore samples were done based on the experiences from Test 2.

Generally, if the Test 1 will be excluded, the copper recoveries in rougher flotation concentrates were 82.6 – 90.6 % and nickel recoveries varied very much 4.9 – 74.5 %. After the cleanings, copper grades were 2.3 – 14.6 % with 77.1 – 87.5 % recoveries. The corresponding nickel recoveries were 1 – 71.3 % with 0.1 – 4.1 % grades.

Table 14. Konttijärvi flotation copper, nickel and cobalt grades and recoveries

Test	Sample	Product	Mass %	Cu		Ni		Co	
				%	Rec. %	%	Rec. %	%	Rec. %
1	SKF-PM2	RC1-4	61.0	0.16	91.1	0.29	75.2	0.02	67.1
		CC3	20.2	0.43	78.4	0.31	26.1	0.02	21.5
2	SKF-PM2	RC1-4	33.8	0.26	85.9	0.38	52.6	0.08	71.3
		CC2	2.1	3.88	80.5	3.03	27.0	0.20	11.2
3	SKF-MS1	RC1-4	5.7	2.27	87.2	1.10	46.9	0.09	14.3
		CC1	2.0	6.27	85.4	2.81	42.5	0.17	10.3
4	SKF-MS2	RC1-4	5.3	2.66	90.3	0.94	39.6	0.06	12.8
		CC1	1.0	13.71	87.1	3.93	31.0	0.19	7.9
5	SKF-TZ1	RC1-4	4.8	2.95	82.6	1.26	54.7	0.08	15.1
		CC1	1.5	8.76	77.1	3.35	45.8	0.19	11.1
6	SKF-TZ2	RC1-4	4.6	2.20	88.5	1.05	58.6	0.09	11.5
		CC1	1.5	6.57	85.9	2.97	54.2	0.23	9.3
7	SKF-BAS1	RC1-4	6.0	2.25	89.3	1.99	74.5	0.12	24.7
		CC1	2.8	4.75	87.9	4.08	71.3	0.23	21.9
8	SKF-BAS2	RC1-4	6.2	6.44	90.6	0.31	32.3	0.05	14.0
		CC1	2.6	14.61	87.5	0.60	26.5	0.09	10.3
9	SKF-PX1	RC1-4	6.9	1.30	90.6	0.61	29.8	0.06	14.8
		CC1	1.0	8.58	86.4	2.82	19.9	0.25	9.2
10	SKF-PX2	RC1-4	4.6	0.41	83.3	0.11	4.9	0.02	5.5
		CC1	0.8	2.29	81.6	0.12	1.0	0.02	0.6

Table 15. Konttijärvi flotation palladium, platinum, gold and silver grades and recoveries

Test	Sample	Product	Mass %	Au		Ag		Pd		Pt	
				g/t	Rec. %	g/t	Rec. %	g/t	Rec. %	g/t	Rec. %
1	SKF-PM2	RC1-4	61.0	0.2	87.3	1.3	64.5	1.9	88.0	0.4	77.9
		CC3	20.2	0.4	69.0	1.8	30.1	3.6	54.7	0.7	49.3
2	SKF-PM2	RC1-4	33.8	0.3	82.1	1.1	53.5	2.9	80.1	0.5	63.9
		CC2	2.1	3.9	74.6	10.4	31.3	41.6	72.6	5.9	52.9
3	SKF-MS1	RC1-4	5.7	0.9	48.7	10.6	44.3	17.1	65.2	3.6	54.9
		CC1	2.0	2.5	45.3	26.9	40.0	44.9	60.8	9.4	50.4
4	SKF-MS2	RC1-4	5.3	2.7	71.3	23.7	56.9	30.1	84.5	5.8	63.2
		CC1	1.0	12.9	64.5	110.0	49.5	141.3	74.0	26.3	53.3
5	SKF-TZ1	RC1-4	4.8	1.1	48.3	12.7	47.6	19.4	66.0	3.8	52.9
		CC1	1.5	3.0	40.9	31.9	37.6	52.0	55.6	9.9	43.5
6	SKF-TZ2	RC1-4	4.6	1.2	58.9	13.6	52.6	19.4	82.5	3.9	61.5
		CC1	1.5	3.4	55.2	36.3	45.5	56.7	78.5	11.1	56.6
7	SKF-BAS1	RC1-4	6.0	1.3	66.9	25.5	77.9	21.5	75.6	3.3	60.0
		CC1	2.8	2.6	63.3	50.3	71.4	43.8	71.6	6.5	54.5
8	SKF-BAS2	RC1-4	6.2	1.4	74.8	12.0	61.2	18.9	75.6	4.8	74.2
		CC1	2.6	2.8	65.6	24.9	54.2	38.4	65.5	9.7	63.6
9	SKF-PX1	RC1-4	6.9	1.1	79.9	6.8	45.6	19.2	76.7	5.1	67.7
		CC1	1.0	6.5	69.8	39.0	37.7	112.5	65.0	29.5	56.4
10	SKF-PX2	RC1-4	4.6	3.3	40.7	8.4	42.4	21.0	68.2	6.1	61.5
		CC1	0.8	12.9	28.3	39.0	34.8	108.7	62.1	31.3	55.8

Table 16. Metal flotation kinetics for the Konttijärvi deposit

Sample	Cell Residence Time (Minutes)	Cu Recovery (%)	Ni Recovery (%)	Pd Recovery (%)	Pt Recovery (%)	Co Recovery (%)	Au Recovery (%)	Ag Recovery (%)
PM2	2	62.71	10.68	45.71	30.38	4.33	56.14	16.89
Test 2	7	77.03	22.52	66.86	47.81	9.24	71.77	26.39
SKF-PM2	12	79.81	25.73	71.45	51.71	10.63	73.94	30.17
2nd cleaning	17	80.53	27.00	72.58	52.88	11.22	74.57	31.28
PX1	5	49.13	0.84	13.24	11.88	1.06	15.72	12.21
Test 9	15	69.64	2.31	26.47	23.11	2.15	30.85	20.59
SKF-PX1	20	79.12	8.52	45.61	39.94	4.87	54.40	28.99
1st cleaning	25	86.39	19.89	64.98	56.44	9.24	69.82	37.65
PX2	15	74.96	0.31	52.80	48.75	0.31	25.32	27.42
Test 10	20	80.56	0.59	60.13	54.12	0.46	27.12	31.74
SKF-PX2	25	81.58	0.97	62.07	55.77	0.64	28.29	34.83
1st cleaning								
MS1	5	46.18	2.55	7.26	12.36	1.19	9.63	12.83
Test 3	15	73.59	18.61	31.38	30.91	4.72	29.80	25.78
SKF-MS1	20	84.11	40.47	58.03	48.30	9.11	43.18	36.44
1st cleaning	25	85.39	42.52	60.76	50.40	10.28	45.35	39.96
MS2	5	64.53	3.77	23.46	23.19	2.53	28.28	22.97
Test 4	15	79.78	9.56	40.16	36.45	4.34	52.78	37.12
SKF-MS2	25	87.06	30.99	74.03	53.29	7.92	64.51	49.50
1st cleaning								
TZ1	5	44.87	5.06	9.00	14.04	2.41	11.63	11.23
Test 5	15	60.01	10.82	17.24	21.64	4.38	19.50	18.22
SKF-TZ1	20	69.29	29.62	38.00	33.11	7.92	30.87	27.72
1st cleaning	25	77.14	45.80	55.58	43.50	11.08	40.90	37.56
TZ2	5	51.29	8.23	20.56	20.15	2.21	15.57	12.93
Test 6	15	73.58	29.70	51.74	40.75	5.46	32.06	24.80
SKF-TZ2	20	80.99	44.20	68.31	49.61	7.60	41.33	34.34
1st cleaning	25	85.86	54.17	78.52	56.64	9.31	55.20	45.53
BAS1	5	59.86	11.33	15.15	16.35	6.13	23.73	20.72
Test 7	15	76.60	33.03	35.80	30.08	12.25	40.65	36.27
SKF-BAS1	20	86.22	68.53	68.38	51.06	20.55	59.96	65.21
1st cleaning	25	87.85	71.28	71.62	54.45	21.90	63.28	71.37
BAS2	5	59.28	4.61	25.08	29.45	3.03	27.28	30.76
Test 8	15	72.81	8.11	37.11	42.37	4.75	40.67	40.91
SKF-BAS2	20	79.94	16.28	52.92	53.75	7.35	53.79	47.84
1st cleaning	25	87.49	26.51	65.54	63.56	10.28	65.62	54.23

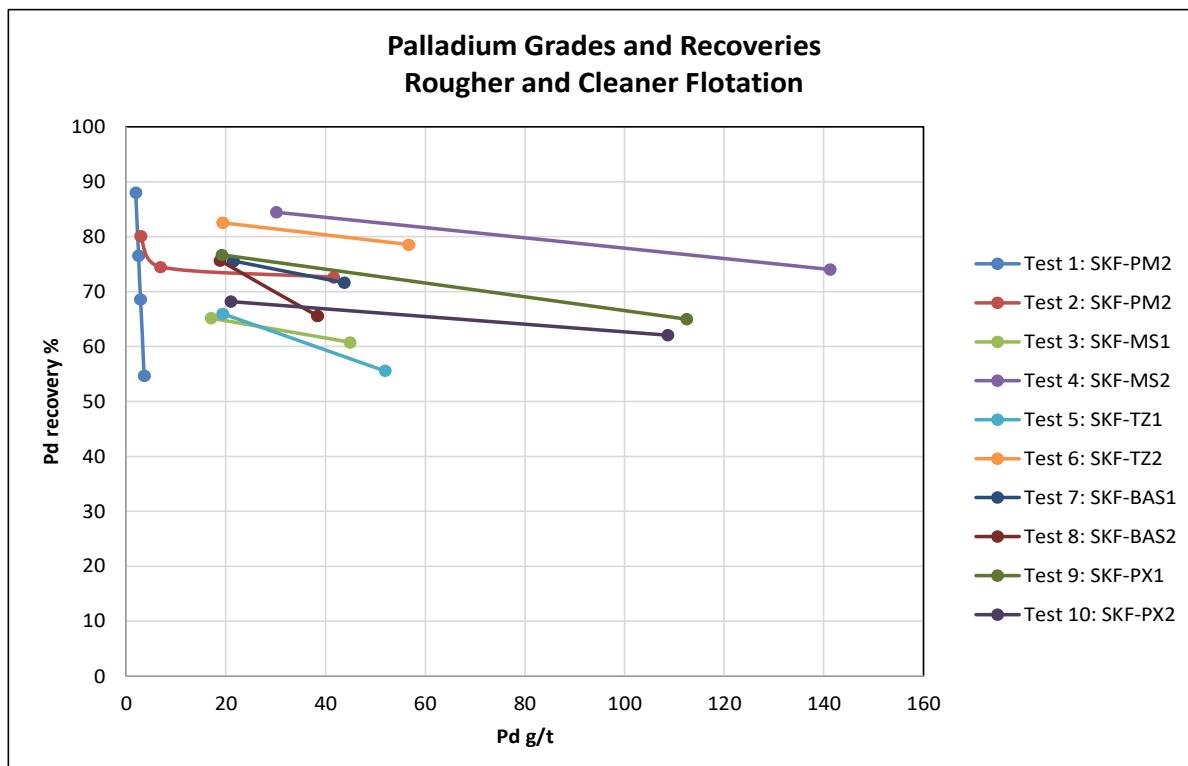


Figure 98. Palladium grades and recoveries in bulk flotation tests

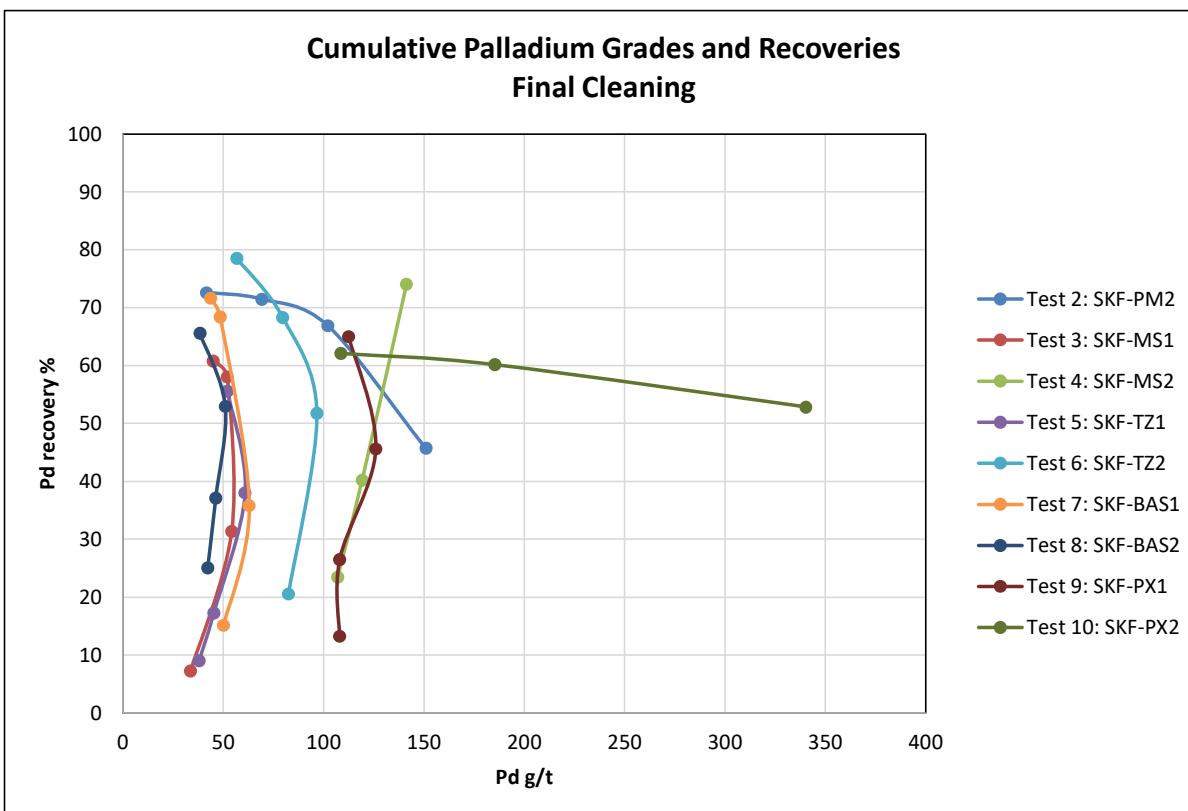


Figure 99. Cumulative palladium grades and recoveries in final cleaner flotation

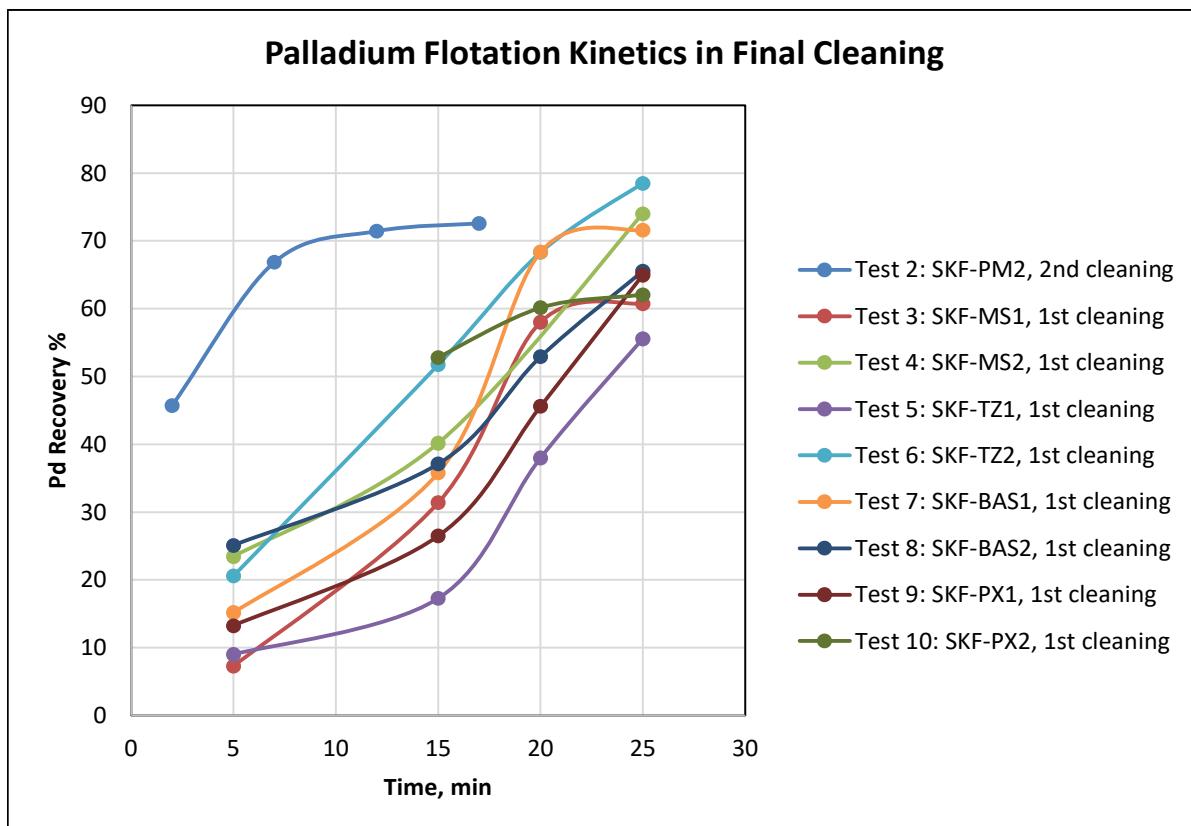


Figure 100. Palladium flotation kinetics in the final cleaning

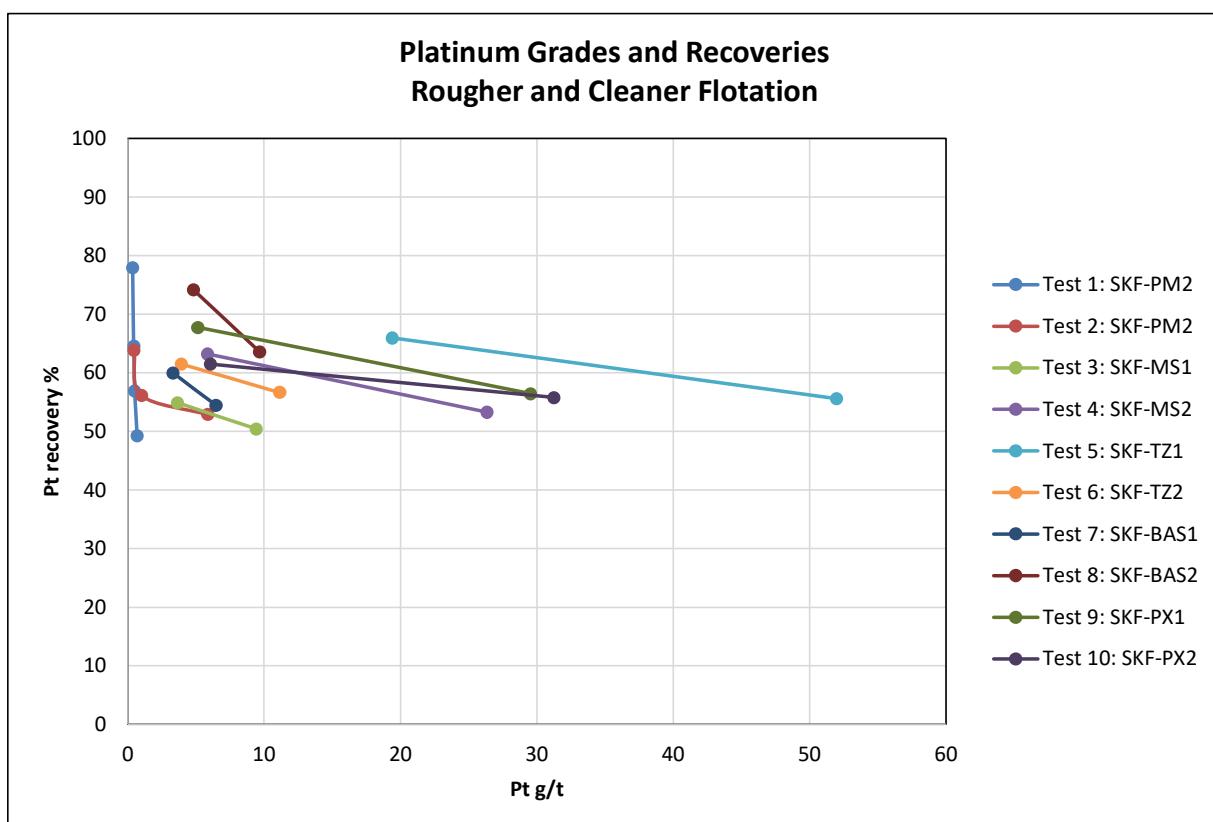


Figure 101. Platinum grades and recoveries in bulk flotation tests

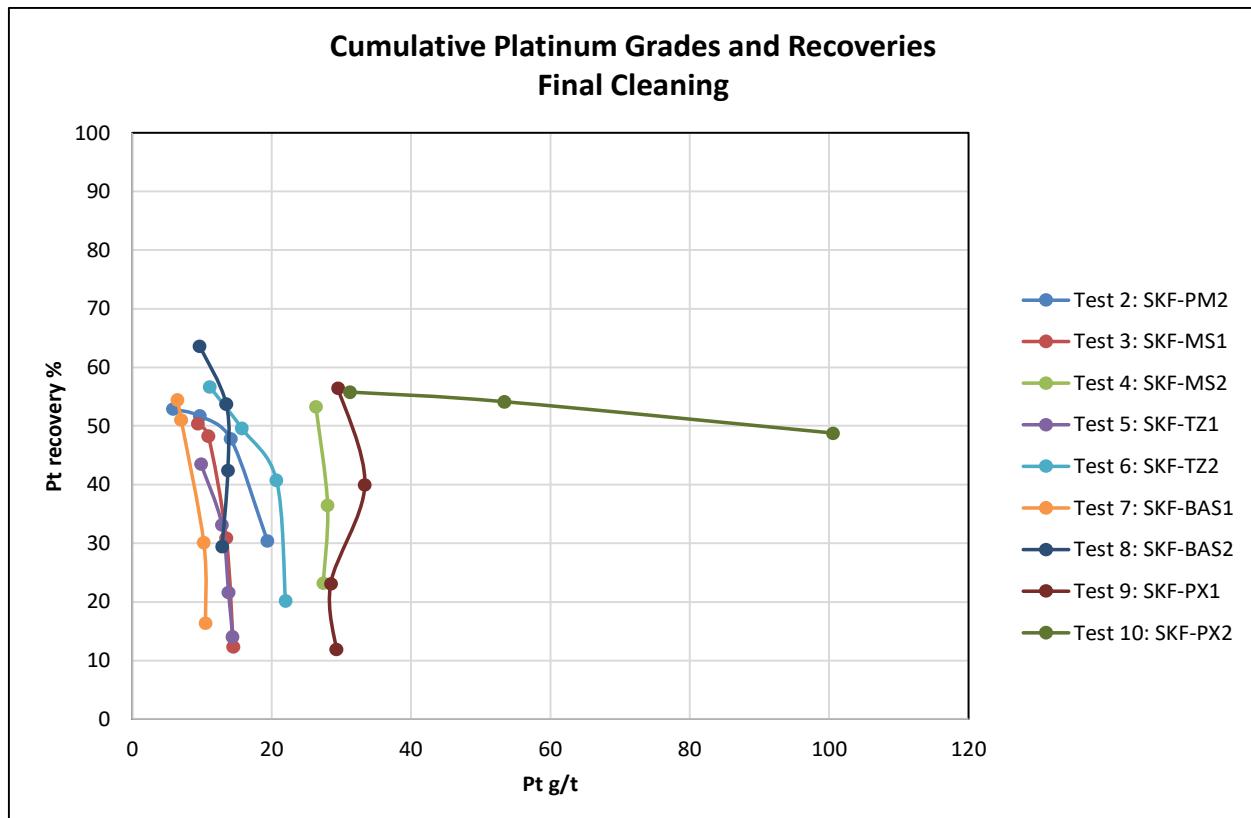


Figure 102. Cumulative platinum grades and recoveries in final cleaner flotation

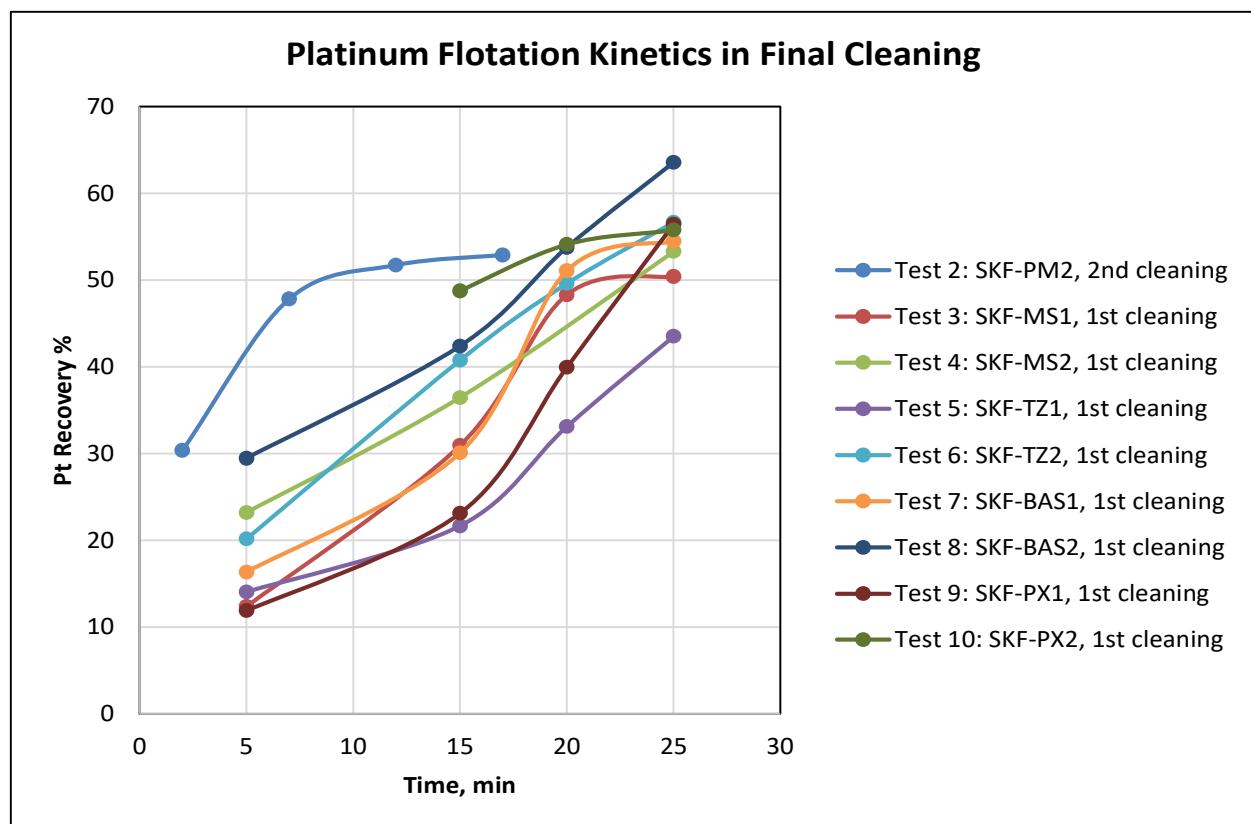


Figure 103. Platinum flotation kinetics in the final cleaning

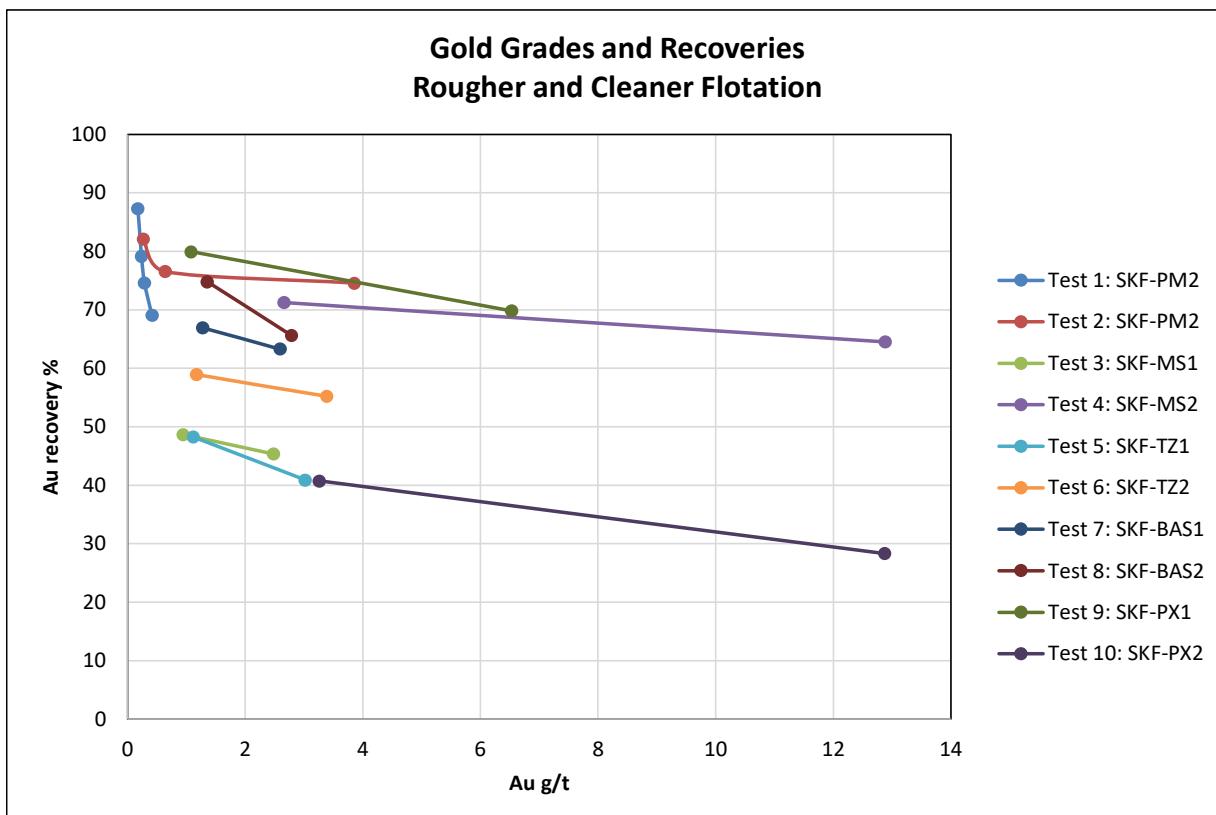


Figure 104. Gold grades and recoveries in bulk flotation tests

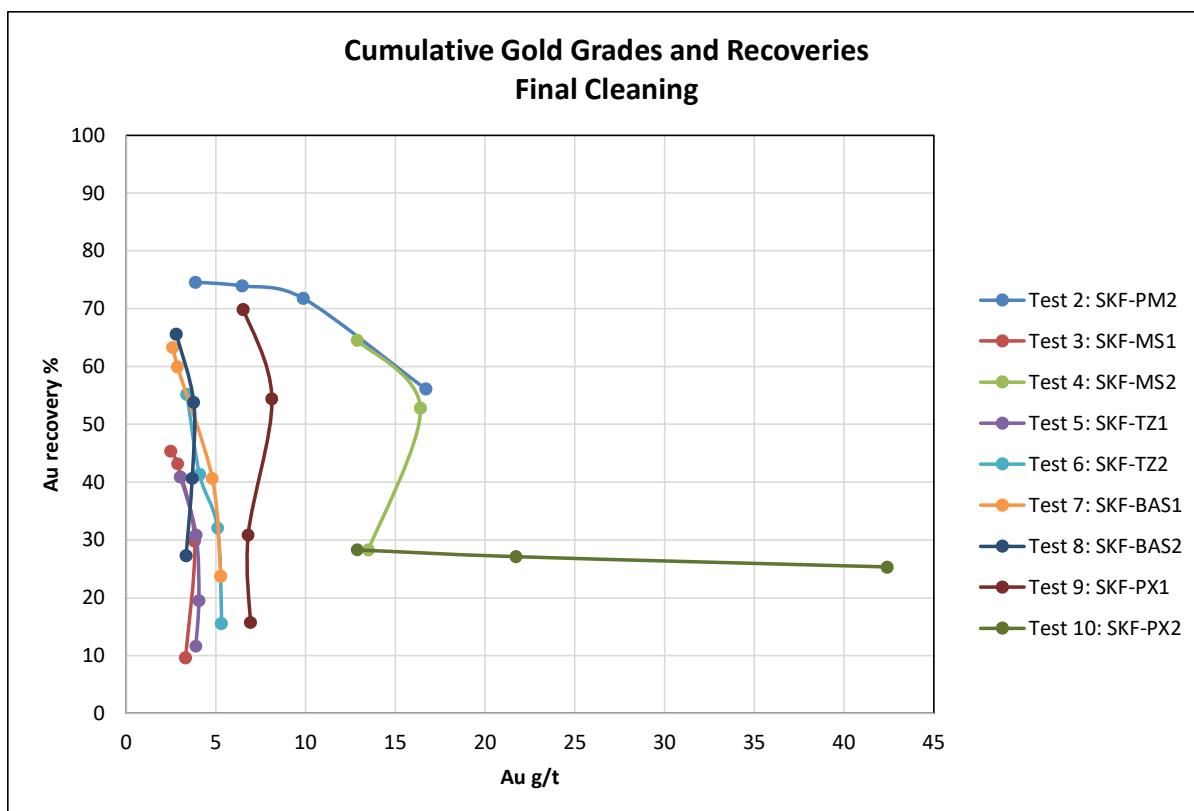


Figure 105. Cumulative gold grades and recoveries in final cleaner flotation

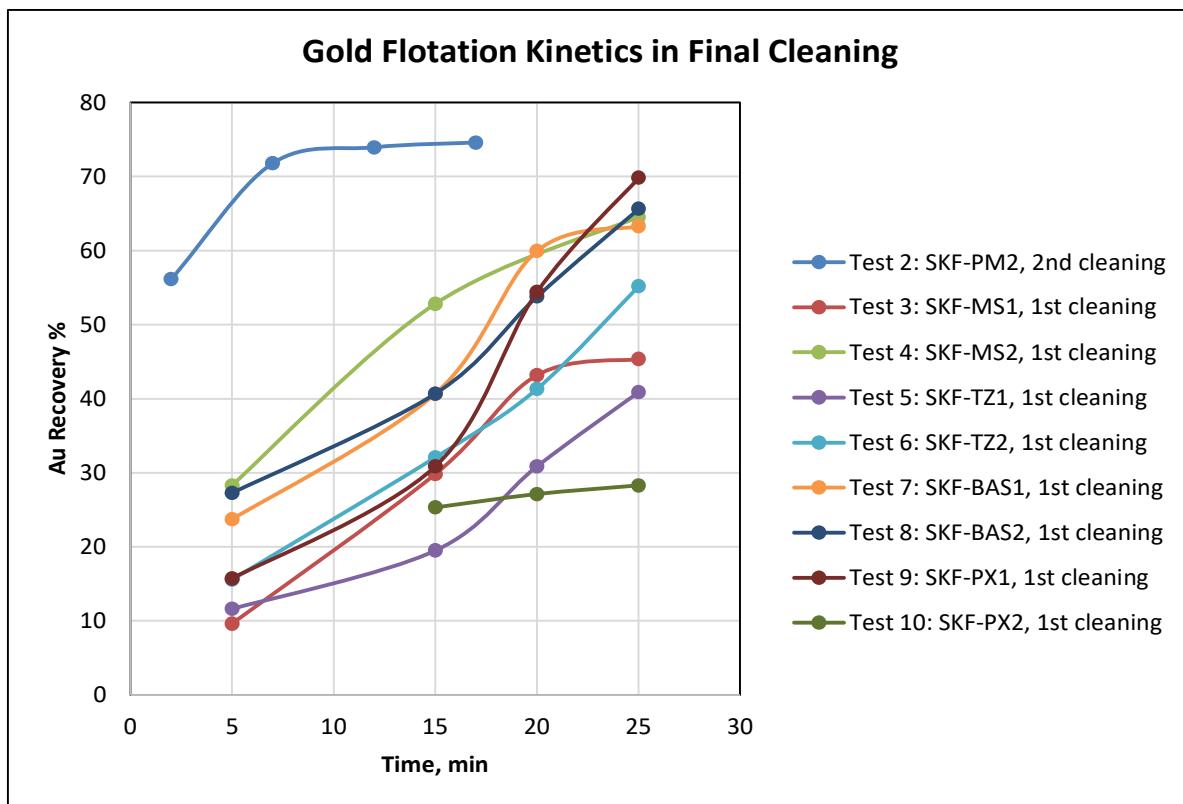


Figure 106. Gold flotation kinetics in the final cleaning

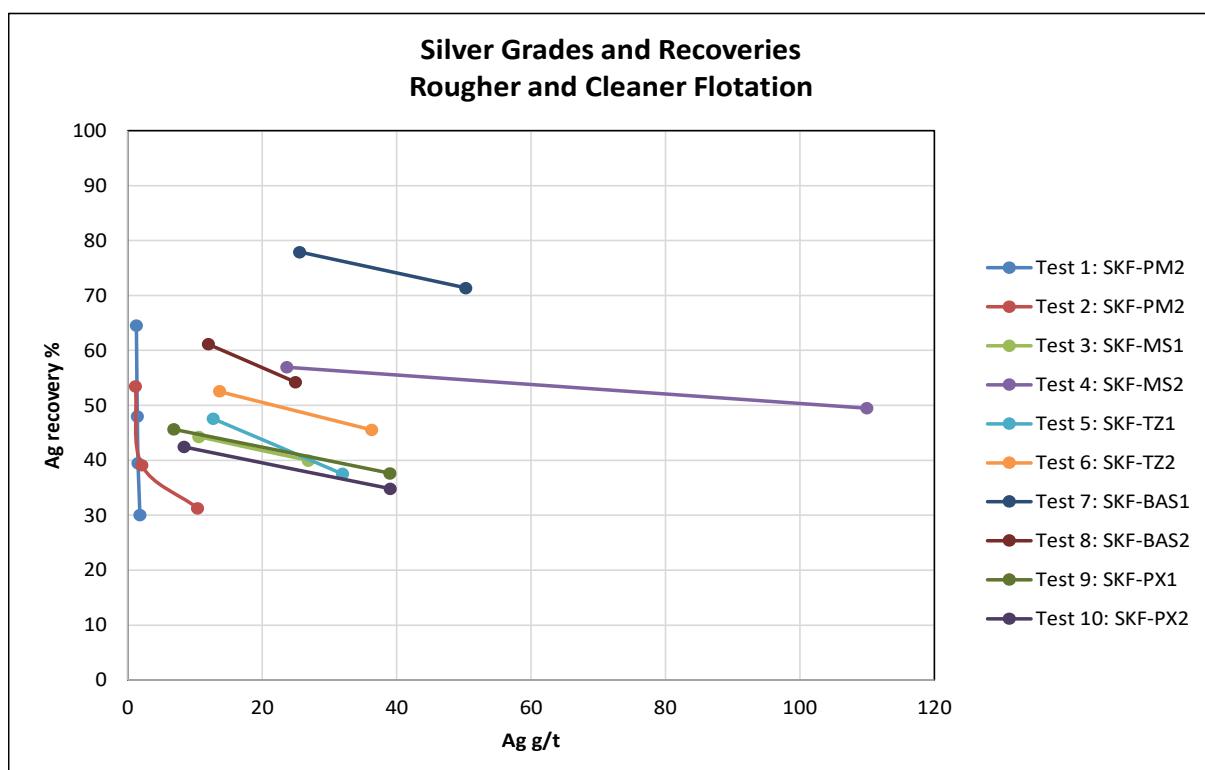


Figure 107. Silver grades and recoveries in bulk flotation tests

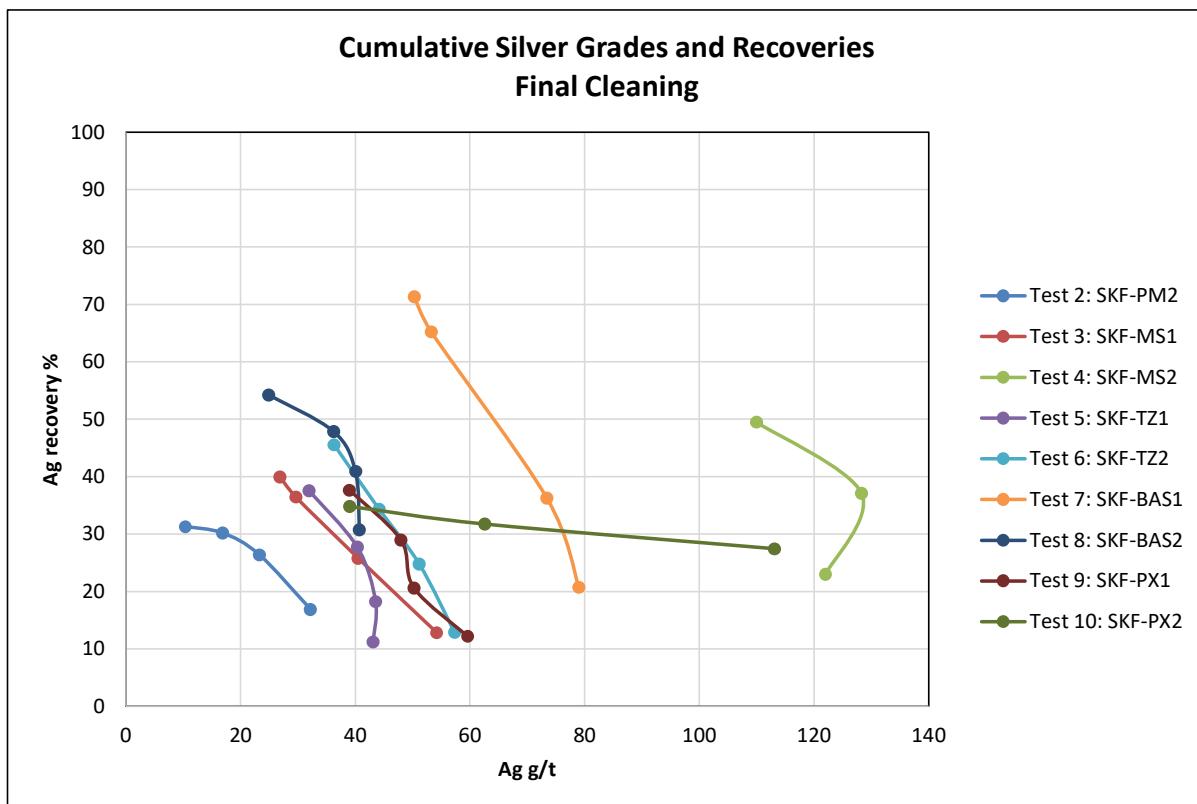


Figure 108. Cumulative silver grades and recoveries in final cleaner flotation

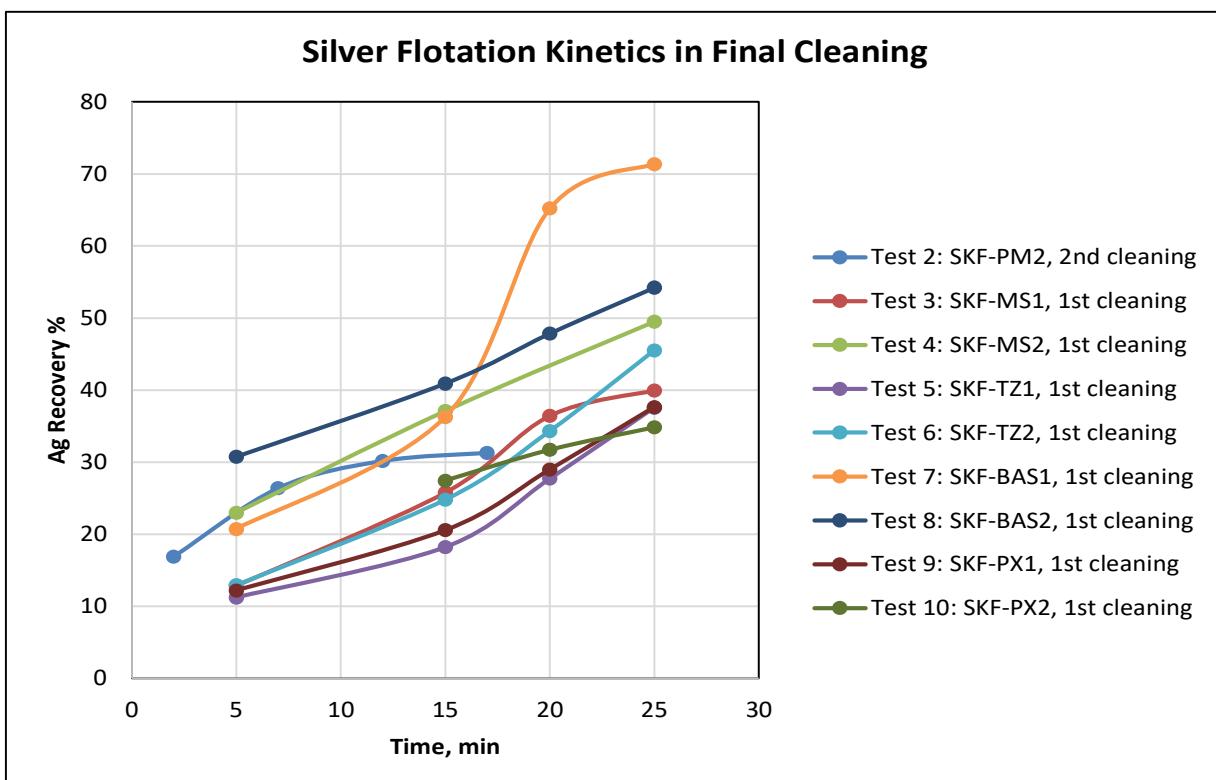


Figure 109. Silver flotation kinetics in the final cleaning

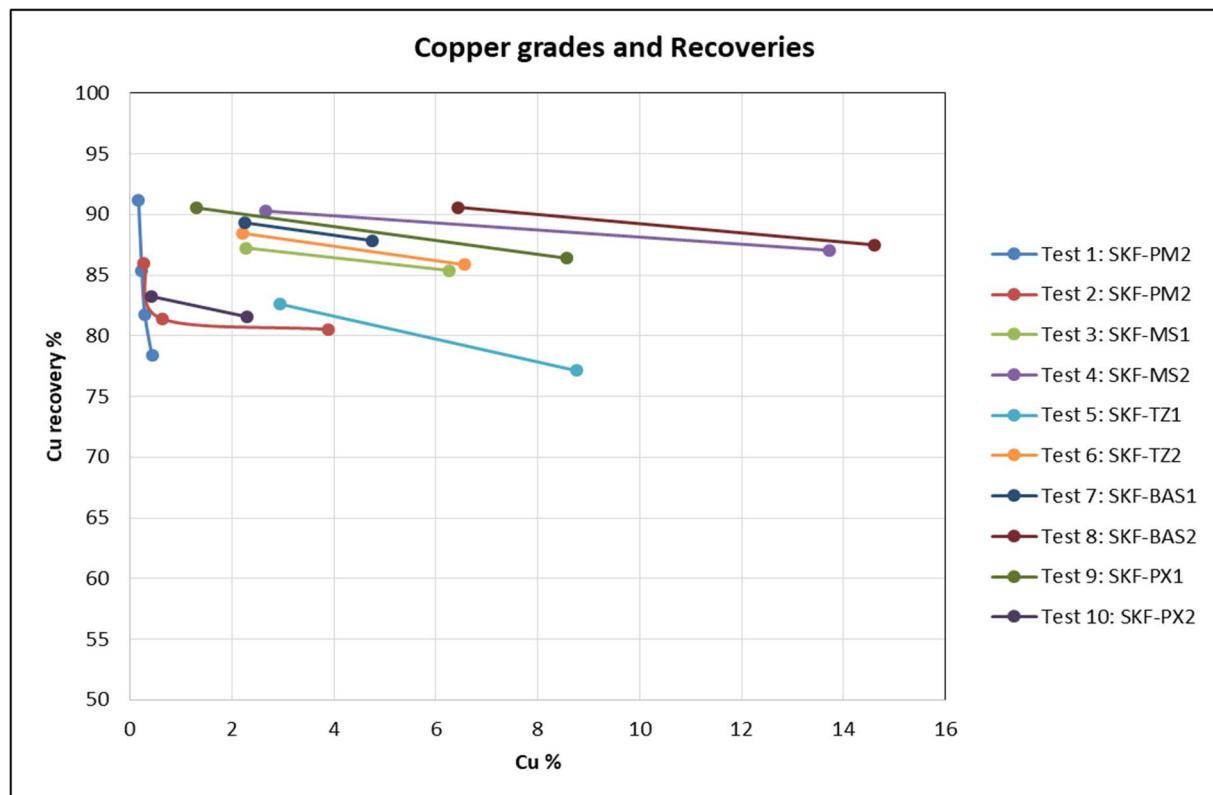


Figure 110. Copper grades and recoveries in bulk flotation tests

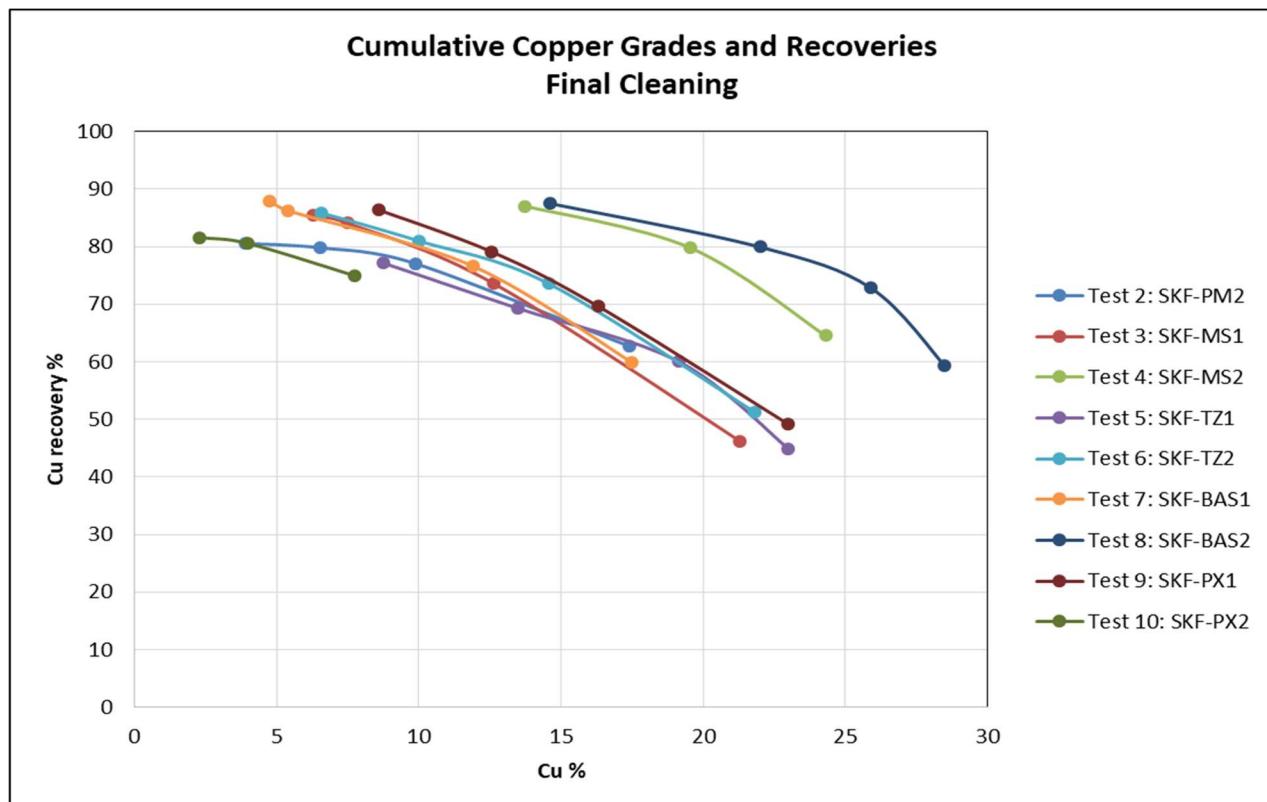


Figure 111. Cumulative copper grades and recoveries in final cleaner flotation

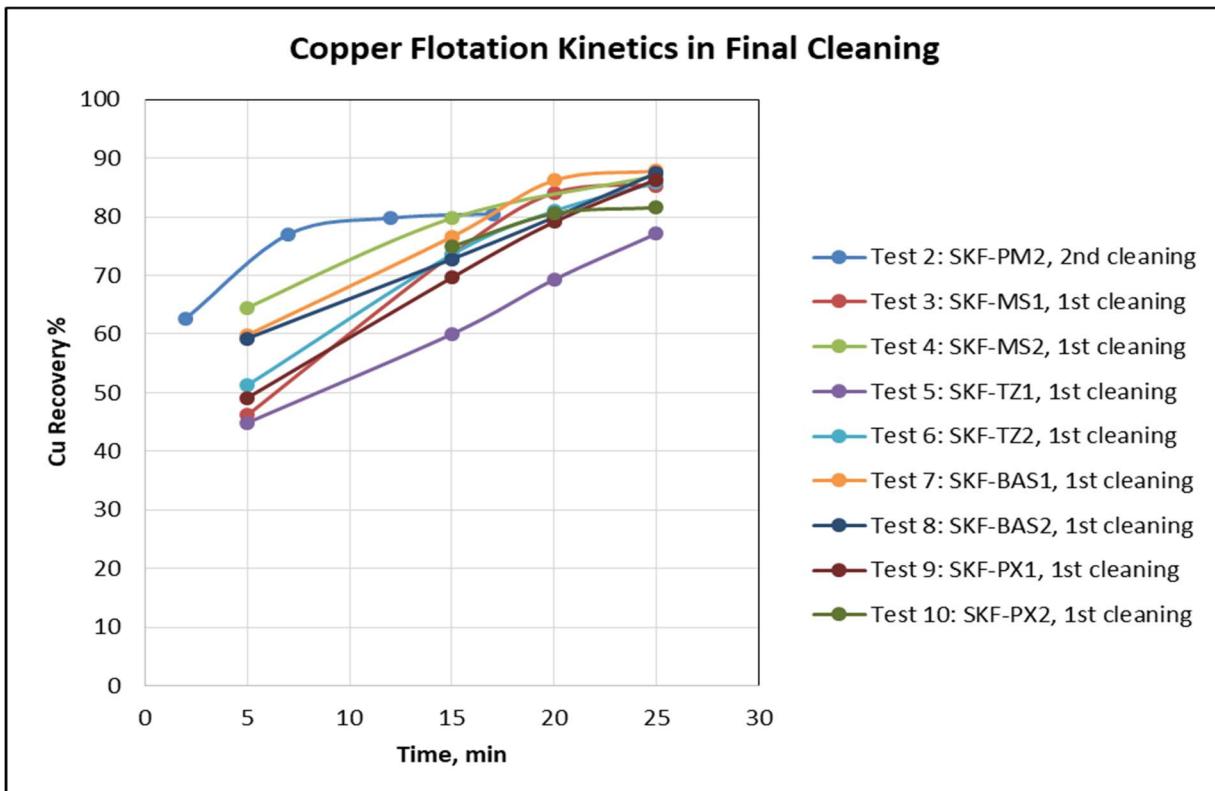


Figure 112. Copper flotation kinetics in the final cleaning

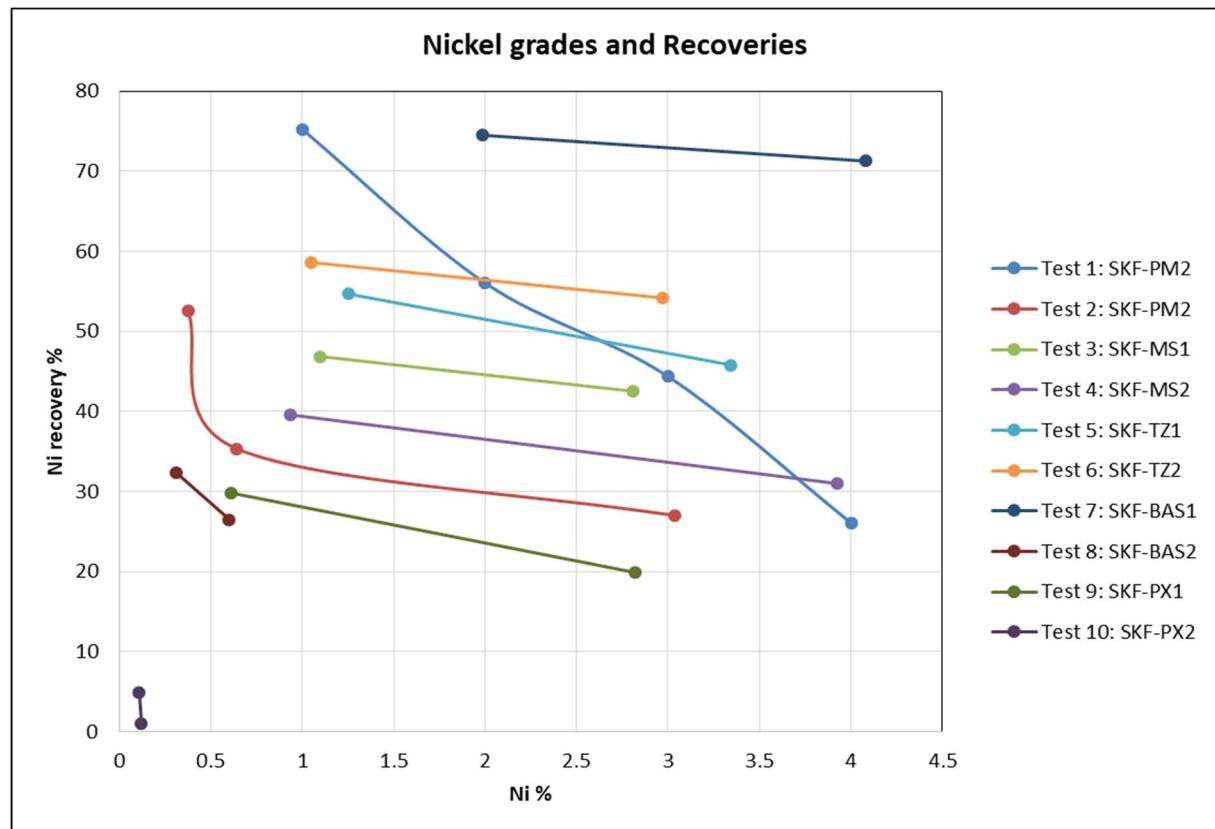


Figure 113. Nickel grades and recoveries in bulk flotation tests

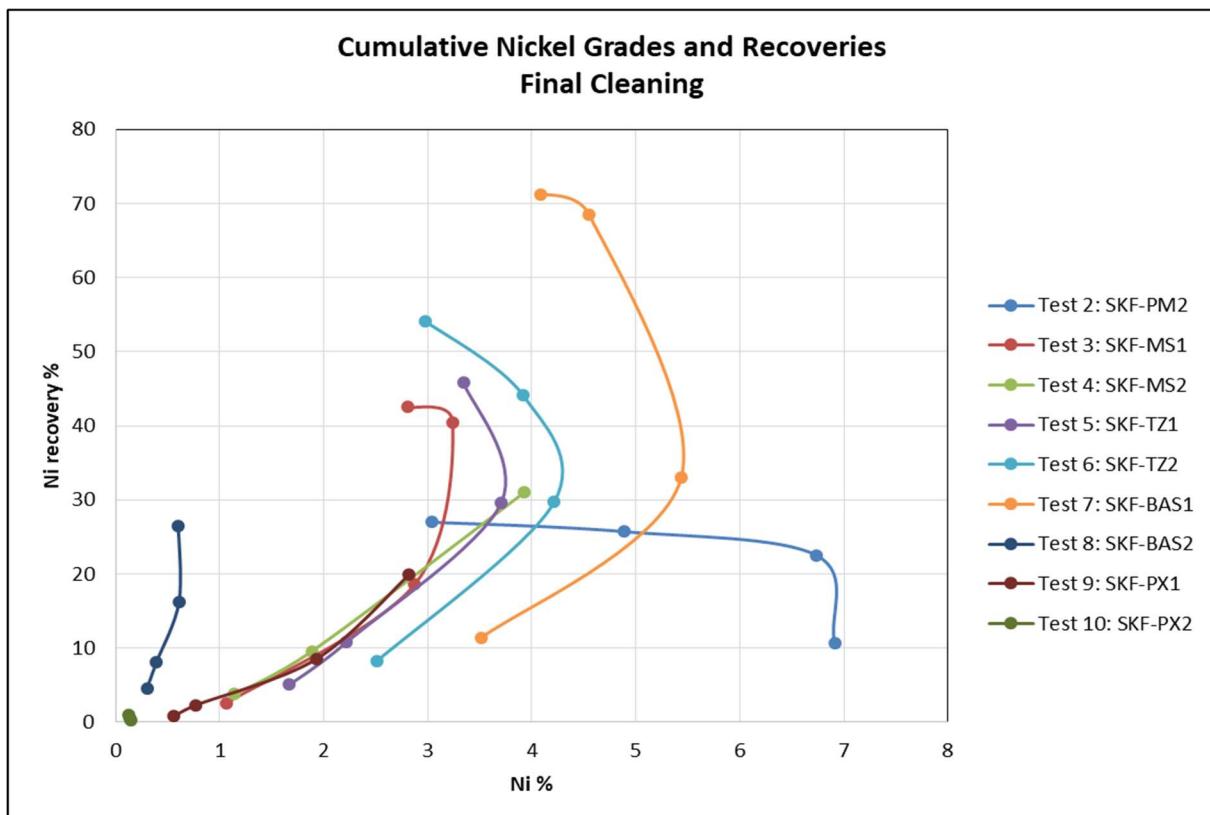


Figure 114. Cumulative Nickel grades and recoveries in final cleaner flotation

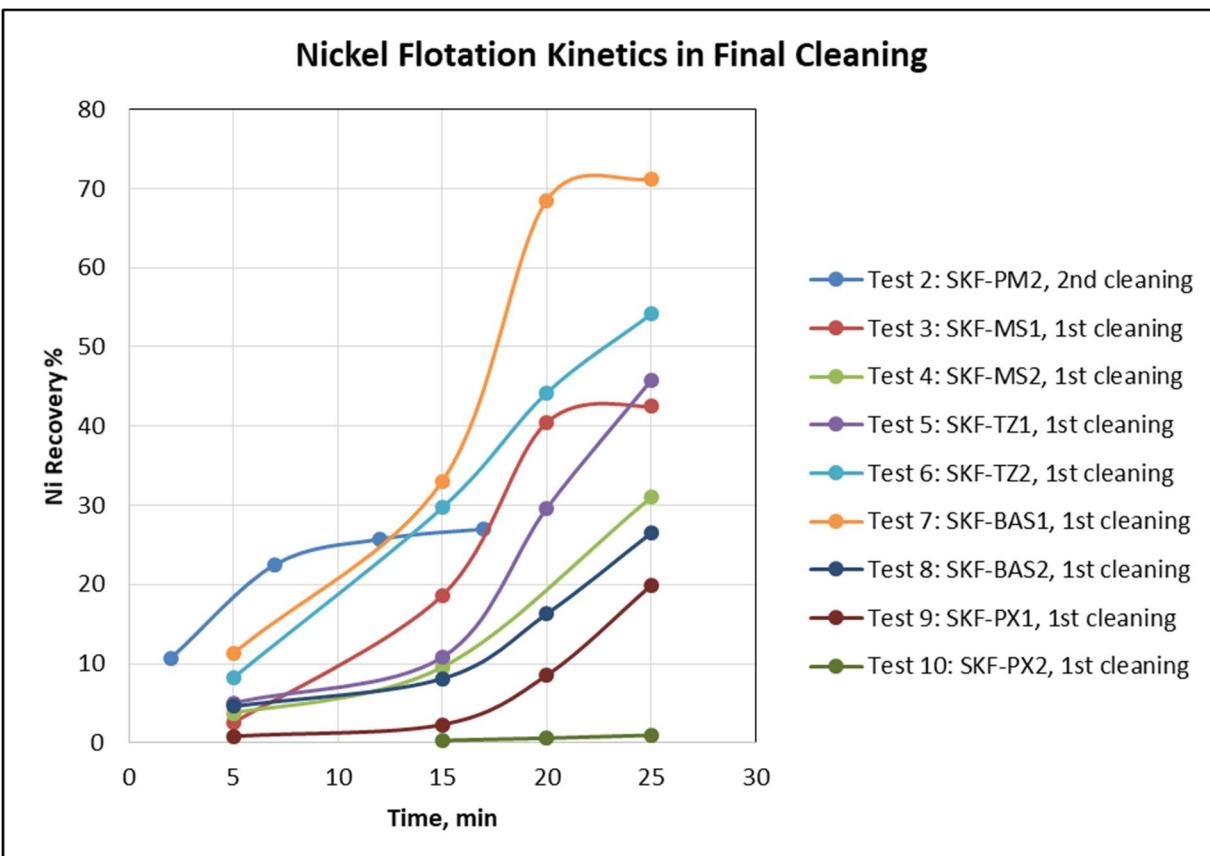


Figure 115. Nickel flotation kinetics in the final cleaning

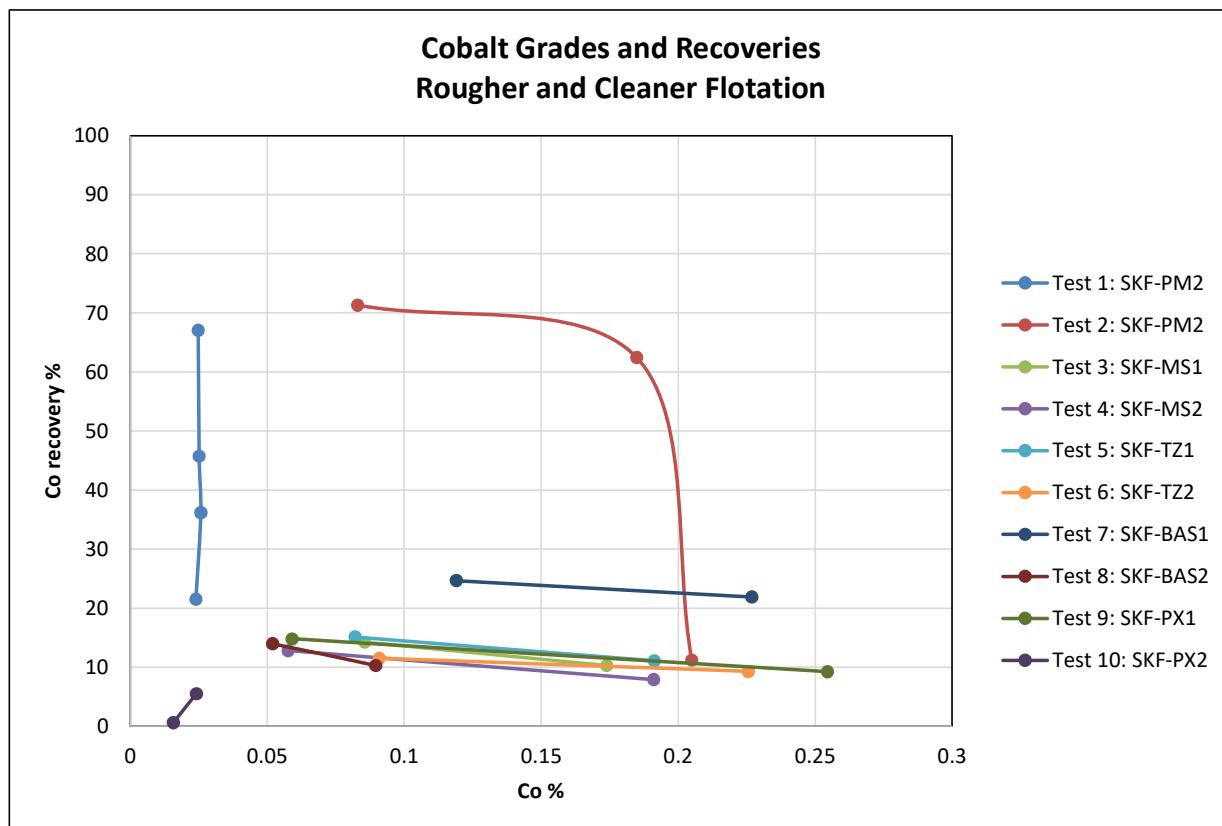


Figure 116. Cobalt grades and recoveries in bulk flotation tests

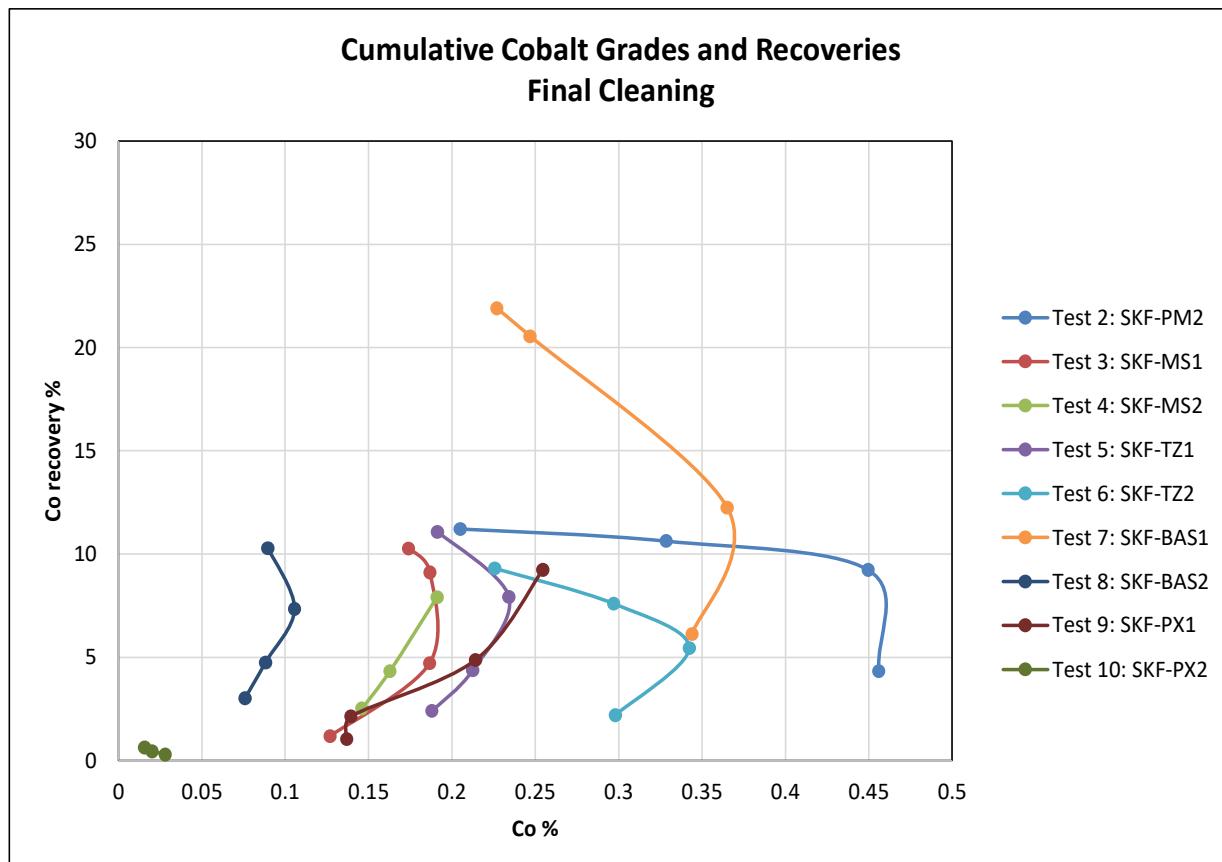


Figure 117. Cumulative Cobalt grades and recoveries in final cleaner flotation

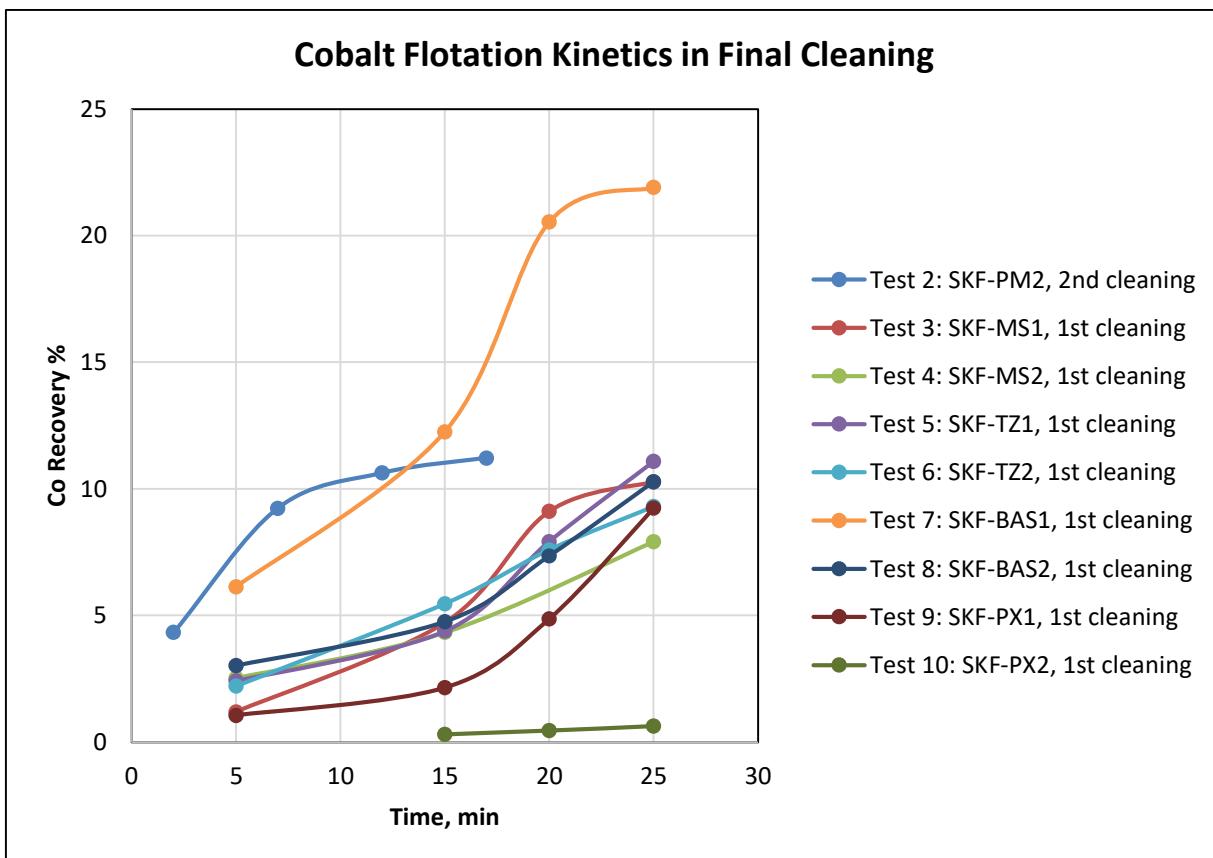


Figure 118. Cobalt flotation kinetics in the final cleaning

The copper and nickel flotation kinetics in the final cleaning flotation stage are presented in Figures 112 and 115. Generally, the copper minerals were floating faster in comparison to nickel minerals. After 15 minutes flotation time, most of copper was floated, thus from nickel only about half.

The cumulative grades and recoveries of copper and nickel in the final cleaning flotation stage are shown in Figures 111 and 114. It can be noticed that best copper grade recovery curve was with the sample SKF-BAS2, the copper grade was high 28 % with 60 % recovery after 5 minutes cleaner flotation, finally after 25 minutes flotation, the cumulative copper recovery was 87 % with 14.6 % grade. Thus, it should be noted that the copper feed grade was the highest too in SKF-BAS2.

The nickel grades were generally rather low with all samples in the final cleaning flotation stage.

The copper and nickel feed grades versus recoveries in rougher flotation are plotted in Figure 119. It can be noticed that the nickel recovery is more related to the feed grade than copper recovery. Generally, the trend was the higher the feed grades the higher the recovery.

Regarding the precious metals, gold, silver, palladium and platinum, the recoveries varied quite a lot. In rougher flotation gold recoveries were 41 – 82 %, silver 42 – 78 %, palladium 65 – 85 % and platinum 53 – 68 % (Test 1 will be excluded because of the very high mass pull).

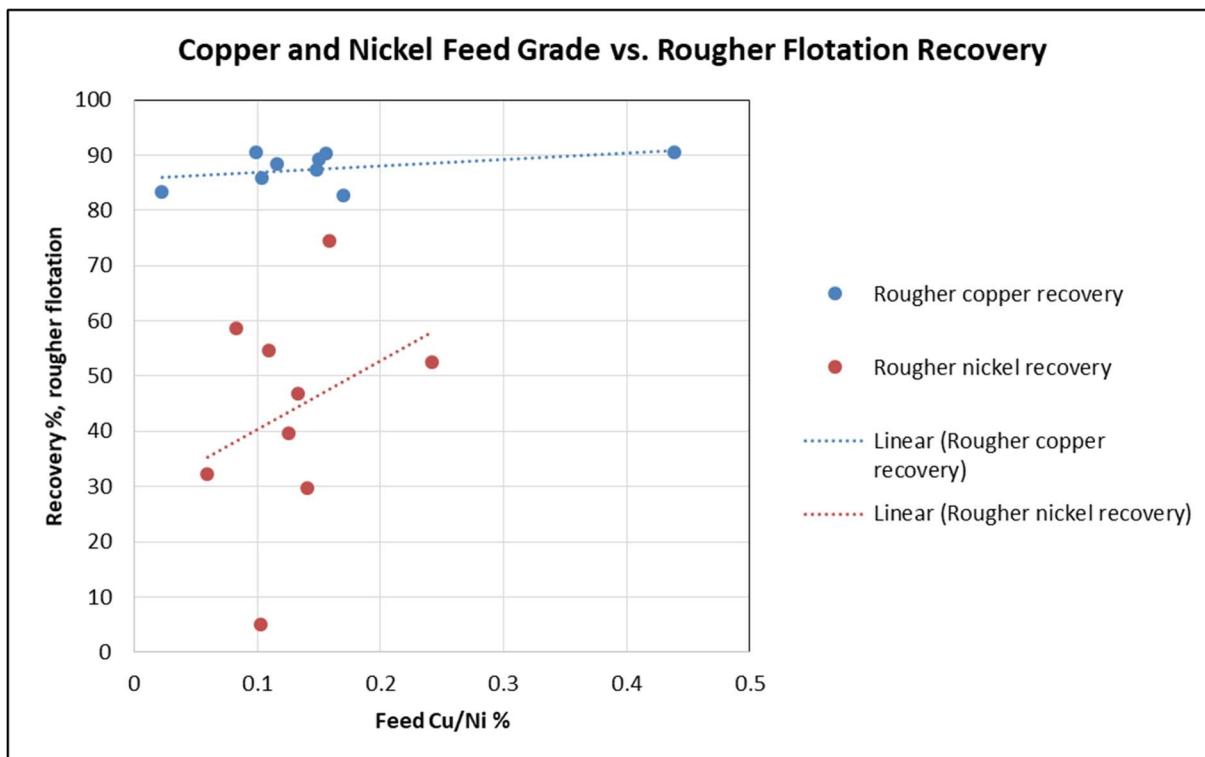


Figure 119. Copper and nickel feed grades vs. recoveries in rougher flotation

Figures 120 to 125 show how each of the 25-minute residence time recovery for each metal as they correlate with each other. No clear correlation is observed for most of the metals suggest different mineral phases are associated with each metal but cobalt recovery do show a good correlation with nickel recovery (Figure 125). This is often observed for magmatic Ni sulfide deposits for which both cobalt and nickel are hosted in pentlandite (Dehaine et al., 2021). Provided that all the nickel is deported in pentlandite, the ratio between cobalt and nickel recovery can even be used as a proxy to assess the proportion of the cobalt content of the ore deported in pentlandite, and the proportion deported in other minerals (sulfides, silicates).

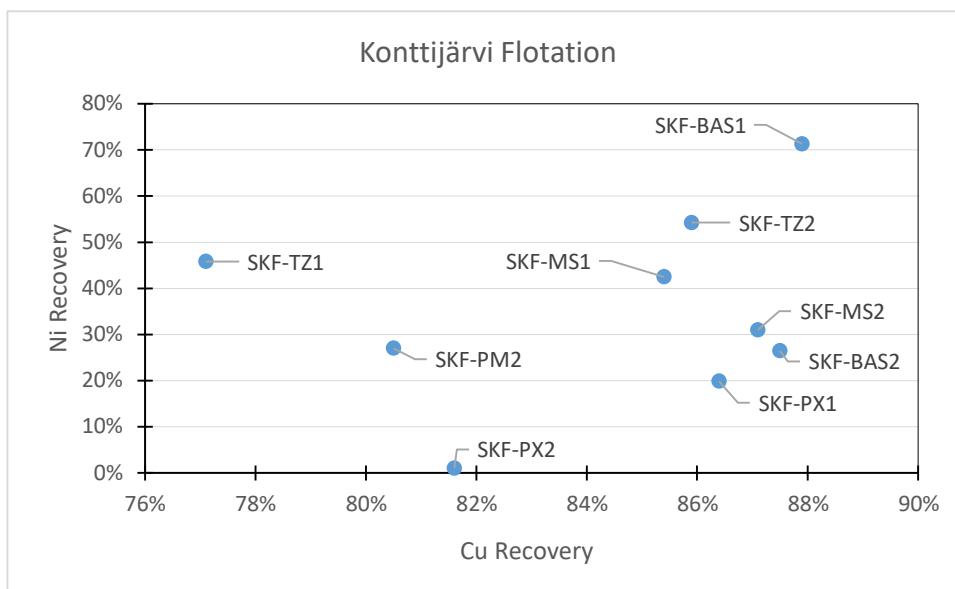


Figure 120. Konttijärvi flotation recovery Nickel vs. Copper

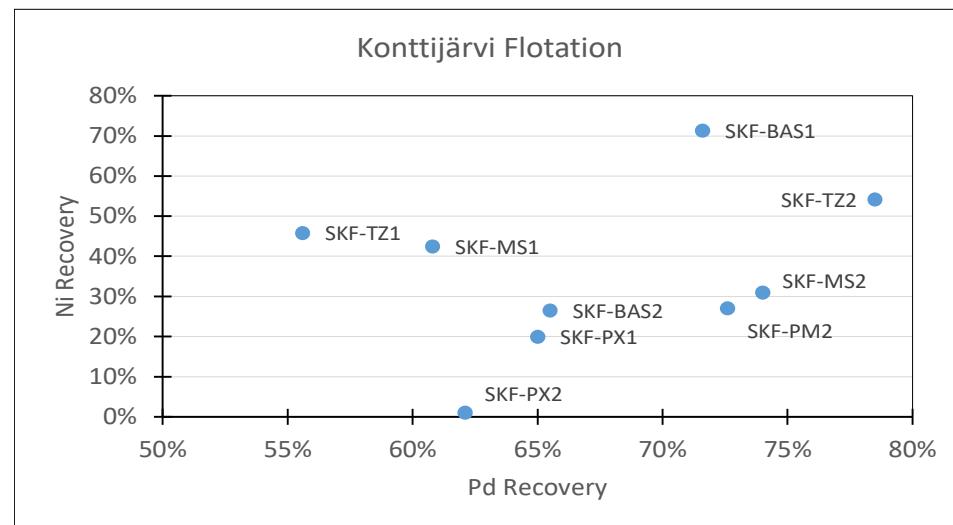


Figure 121. Konttijärvi flotation recovery Palladium vs. Nickel

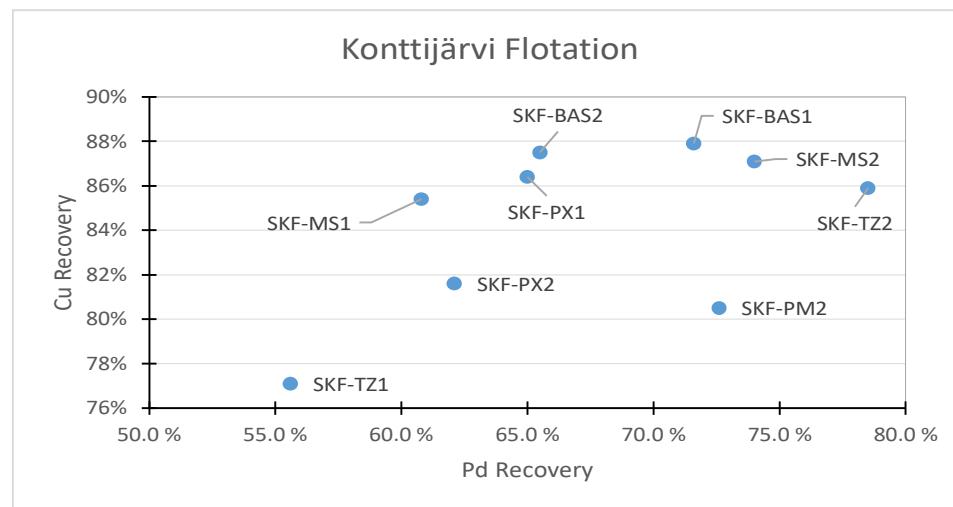


Figure 122. Konttijärvi flotation recovery Palladium vs. Copper

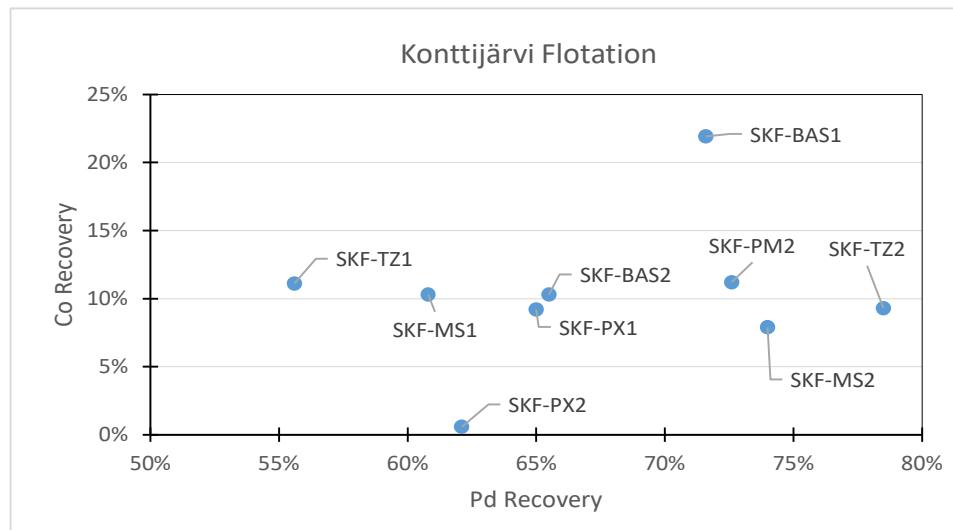


Figure 123. Konttijärvi flotation recovery Palladium vs. Cobalt

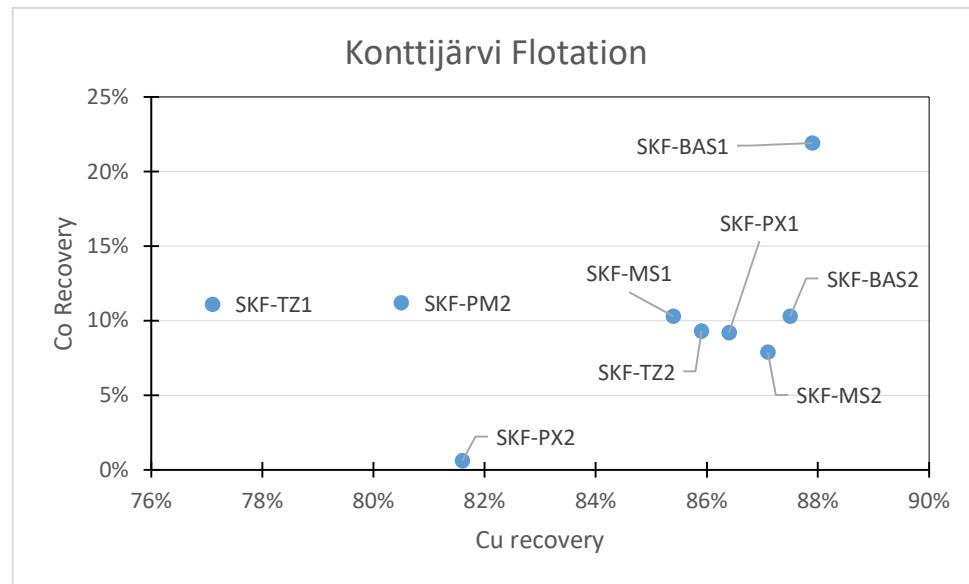


Figure 124. Konttijärvi flotation recovery Copper vs. Cobalt

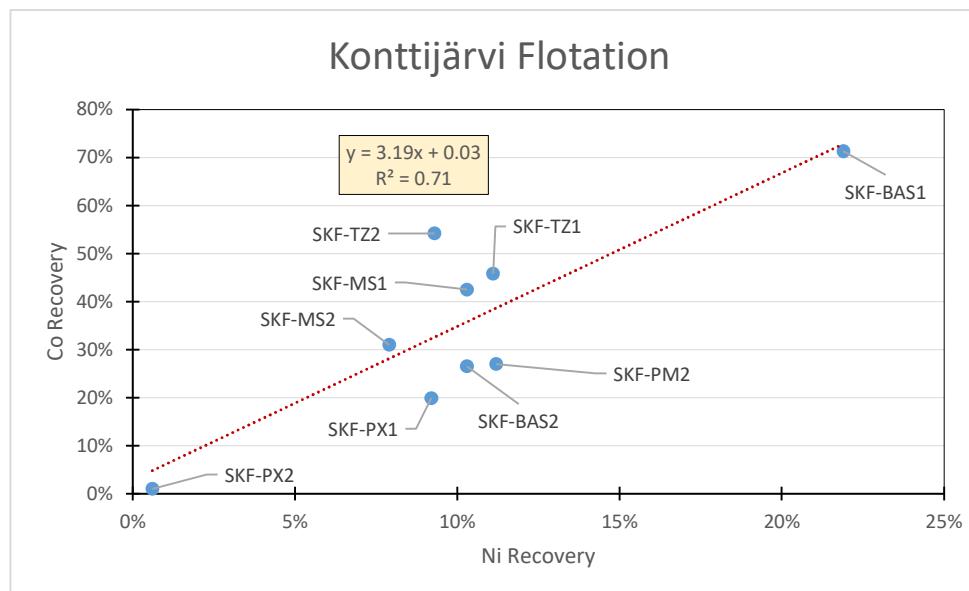


Figure 125. Konttijärvi flotation recovery Cobalt vs. Nickel

8.3 Palladium recovery flotation data correlations

The most valuable precious metal in the Konttijärvi (metal value in context of metal grade) could be palladium. Flotation is one method of recovering palladium. This section is the outcome of examining palladium recovery performance in flotation in a cross correlation matrix with all other characterization measurements of the feed samples. This work was done in collaboration with John Martin (Hafren Scientific Group & X-Ray Mineral Services), using the Spotfire software.

A number of relationships have been found. As the number of samples is relatively low, these conclusions are not considered to be statistically significant, but could be used as a hypothesis in future experimental design.

Figure 126 show a Principle Component Analysis plot of all characterization data of the feed sample and the average (for all data) palladium recovery at all cell residence times measured, to indicate elemental associations. On the same plot, the relative positioning of each individual sample.

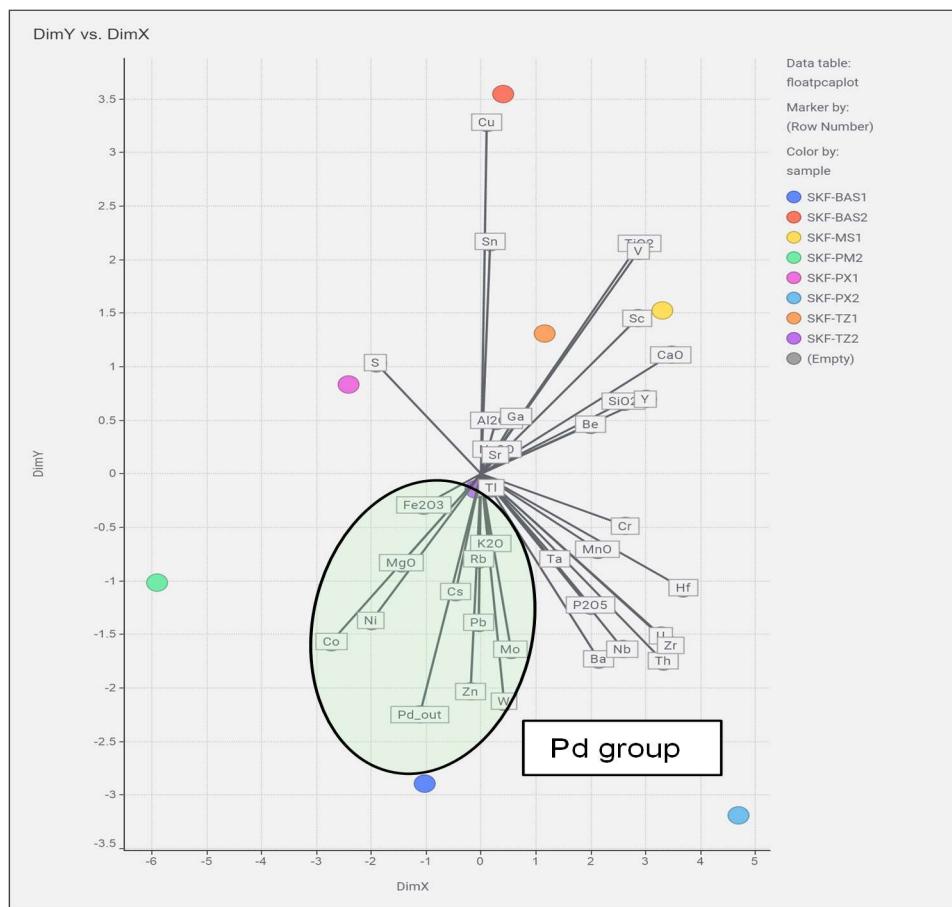


Figure 126. Principal Component Analysis of palladium recovery after 20 minutes cell residence time and all other feed characteristics to indicate elemental associations. Note that due to the small number of samples, these results are only indicative (Analysis done with Spotfire software by John Martin, Hafren Scientific)

This data was then put in cross correlation matrix to examine possible relationships with palladium recovery. The relationships found were most stable when comparing the Pd recovery data of 20 minutes cell residence time. Figure 127 and 128 show the correlations with palladium recovery (20 minutes in cell), where the larger squares are the stronger correlations. These squares are also colour coded in a range from purple for a strong correlation to yellow for a very weak correlation.

Figure 127 shows the statistically balance correlation spectrum with the highest (best) three palladium recoveries (20 minutes in cell). Figure 128 shows the statistically balance correlation spectrum with the lowest (worst) three palladium recoveries (20 minutes in cell).

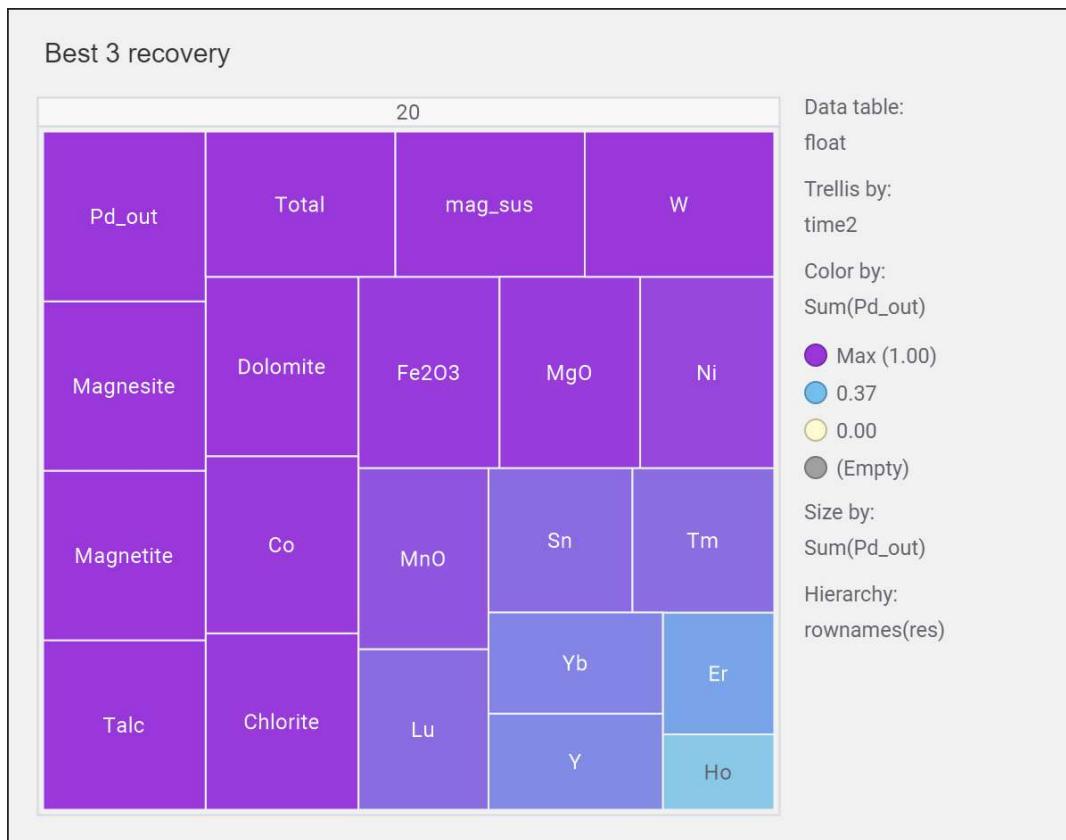


Figure 127. Tree map of correlation coefficient summary between the highest 3 palladium recoveries after 20 minutes cell residence time and all other feed characteristics (Analysis done with Spotfire software by John Martin, Hafren Scientific)

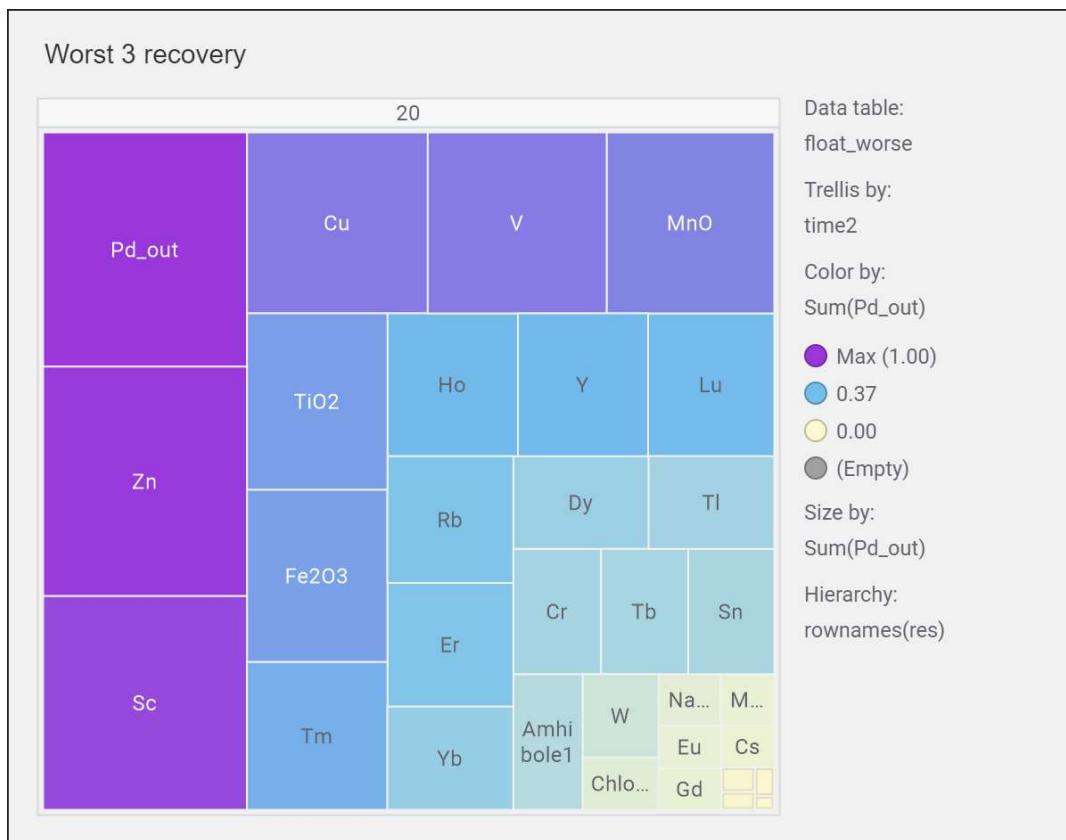


Figure 128. Tree map of correlation coefficient summary between the lowest 3 palladium recoveries after 20 minutes cell residence time and all other feed characteristics (Analysis done with Spotfire software by John Martin, Hafren Scientific)

Given what was learned from Figures 127 and 128 several elements were selected and plotted against palladium recovery (20 minutes cell residence time). Figure 129 shows palladium recovery plotted against ferric oxide (Fe_2O_3), nickel (Ni) and molybdenum (Mo). Ferric iron (Fe_2O_3) and Nickel (Ni) are good indicators of Pd recovery at 20 mins in most samples. Molybdenum (Mo) is the best predictor in BAS-1 and TZ2 samples.

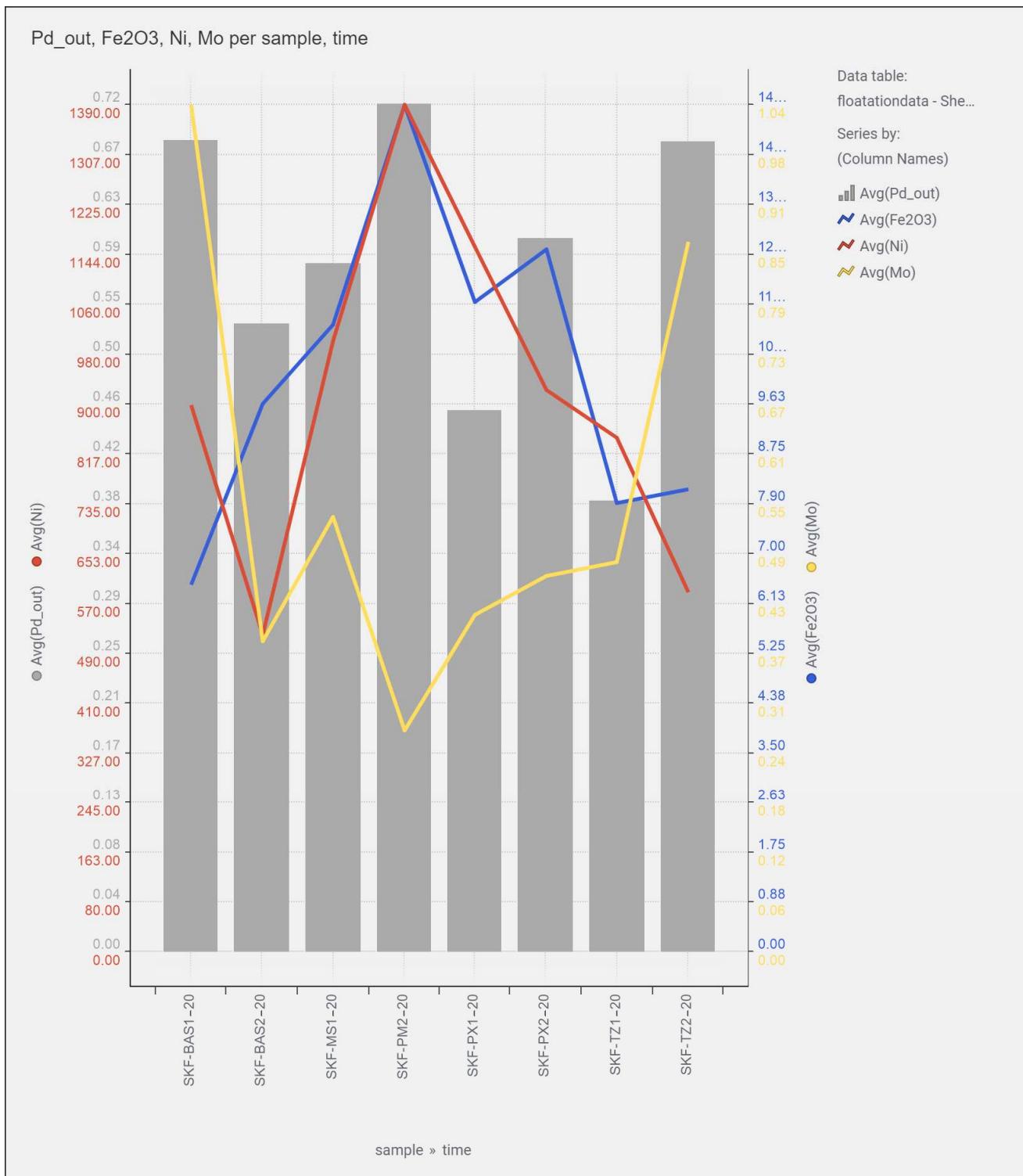


Figure 129. Palladium recoveries after 20 minutes cell residence time plotted against Fe_2O_3 , Ni and Mo (Analysis done with Spotfire software by John Martin, Hafren Scientific)

Figure 130 shows palladium recovery plotted against biotite, chlorite, and plagioclase. Biotite and Plagioclase are indicators of Pd at 20 mins cell residence time in most samples. Chlorite is the inverse of these two minerals.

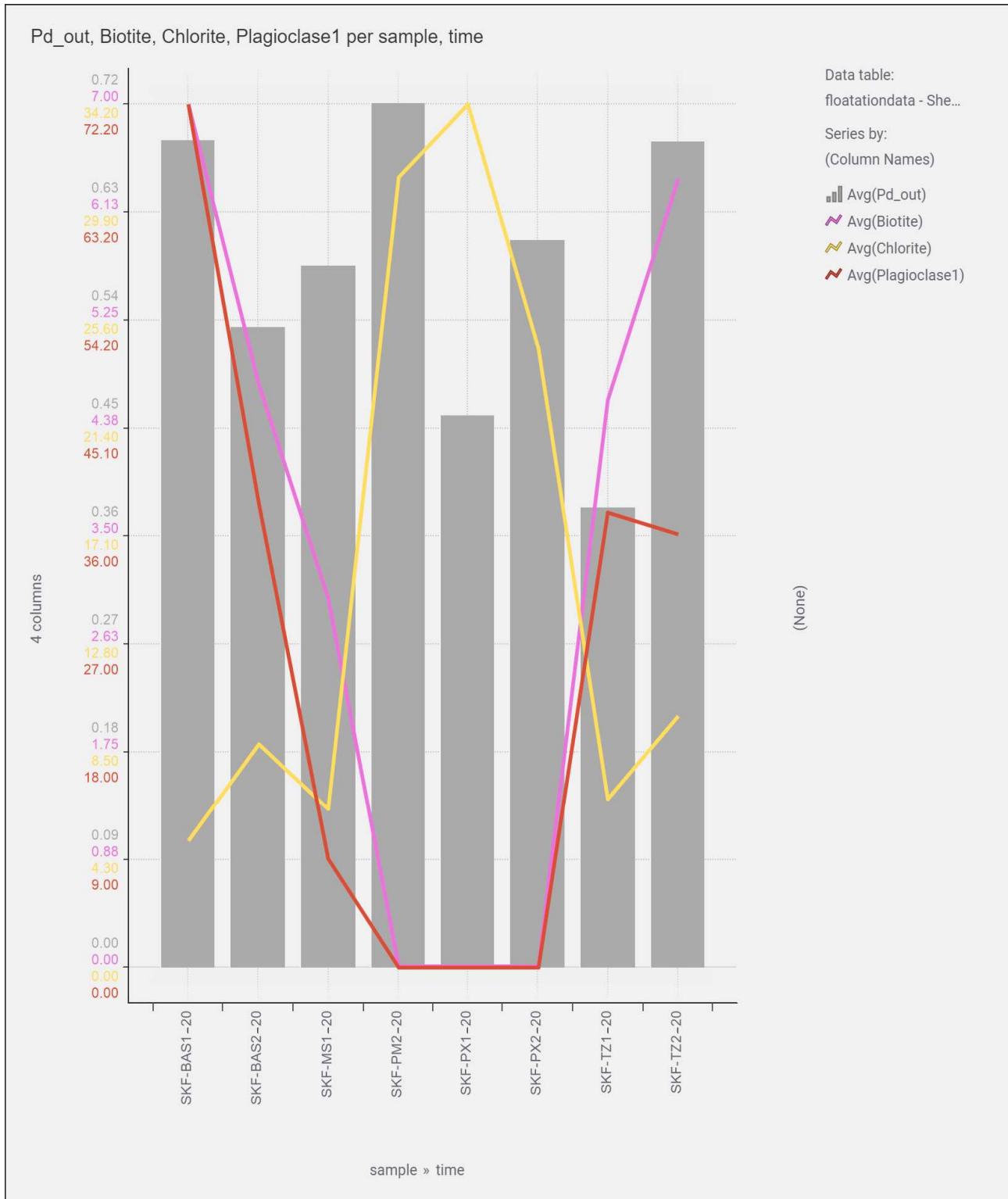


Figure 130. Palladium recoveries after 20 minutes cell residence time plotted against biotite, chlorite, and plagioclase
(Analysis done with Spotfire software by John Martin, Hafren Scientific)

8.4 Copper recovery flotation data correlations

The most valuable base metal in the Konttijärvi (metal value in context of metal grade) could be copper. Flotation is one method of recovering copper. This section is the outcome of examining copper recovery performance in flotation in a cross correlation matrix with all other characterization measurements of the feed samples. This work was done in collaboration with John Martin (Hafren Scientific Group & X-Ray Mineral Services), using the Spotfire software.

A number of relationships have been found. Again, as the number of samples is relatively low, these conclusions are not considered to be statistically significant, but could be used as a hypothesis in future experimental design.

Figure 131 show a Principle Component Analysis plot of all characterization data of the feed sample and the average (for all data) copper recovery at all cell residence times measured, to indicate elemental associations. On the same plot, the relative positioning of each individual sample. Copper (Cu) recovery is strongly associated with Mo, W and Zn.

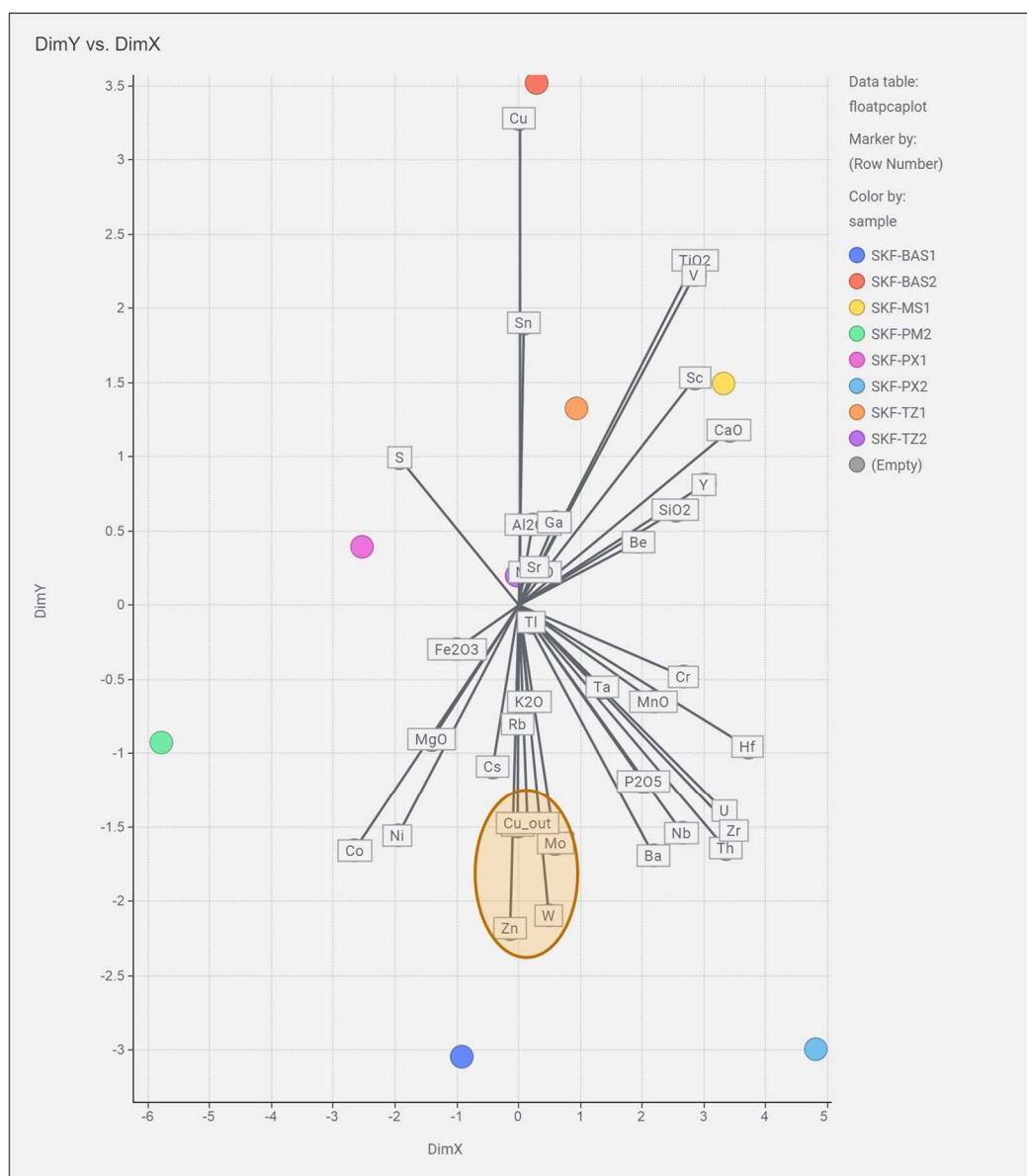


Figure 131. Principal Component Analysis of copper recovery after 20 minutes cell residence time and all other feed characteristics to indicate elemental associations. Note that due to the small number of samples, these results are only indicative (Analysis done with Spotfire software by John Martin, Hafren Scientific)

Given what was learned from Figures 131 several elements were selected and plotted against copper recovery (20 minutes cell residence time). Figure 132 shows copper recovery plotted against Zinc (Zn), Molybdenum (Mo) and Barium (Ba). The most useful of these relationships could be Mo.

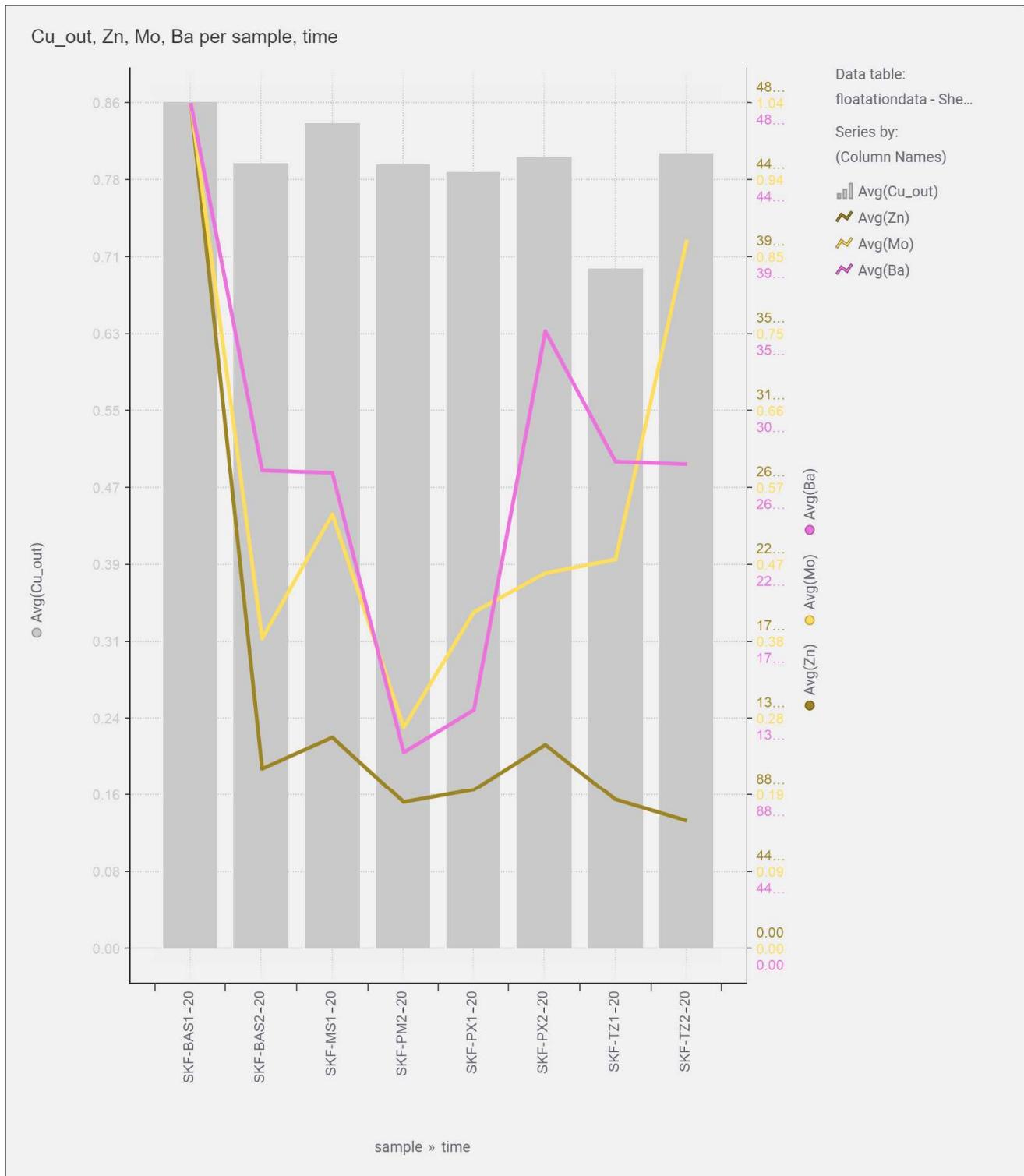


Figure 132. Copper recoveries after 20 minutes cell residence time plotted against Zinc (Zn), Molybdenum (Mo) and Barium (Ba) (Analysis done with Spotfire software by John Martin, Hafren Scientific)

8.5 Cobalt recovery flotation data correlations

Flotation is one method of recovering cobalt. This section is the outcome of examining cobalt recovery performance in flotation in a cross-correlation matrix with all other characterization measurements of the feed samples. This work was done in collaboration with John Martin (Hafren Scientific Group & X-Ray Mineral Services), using the Spotfire software.

A number of relationships have been found. Again, as the number of samples is relatively low, these conclusions are not considered to be statistically significant, but could be used as a hypothesis in future experimental design.

Figure 133 show a Principle Component Analysis plot of all characterization data of the feed sample and the average (for all data) cobalt recovery at all cell residence times measured, to indicate elemental associations. On the same plot, the relative positioning of each individual sample.

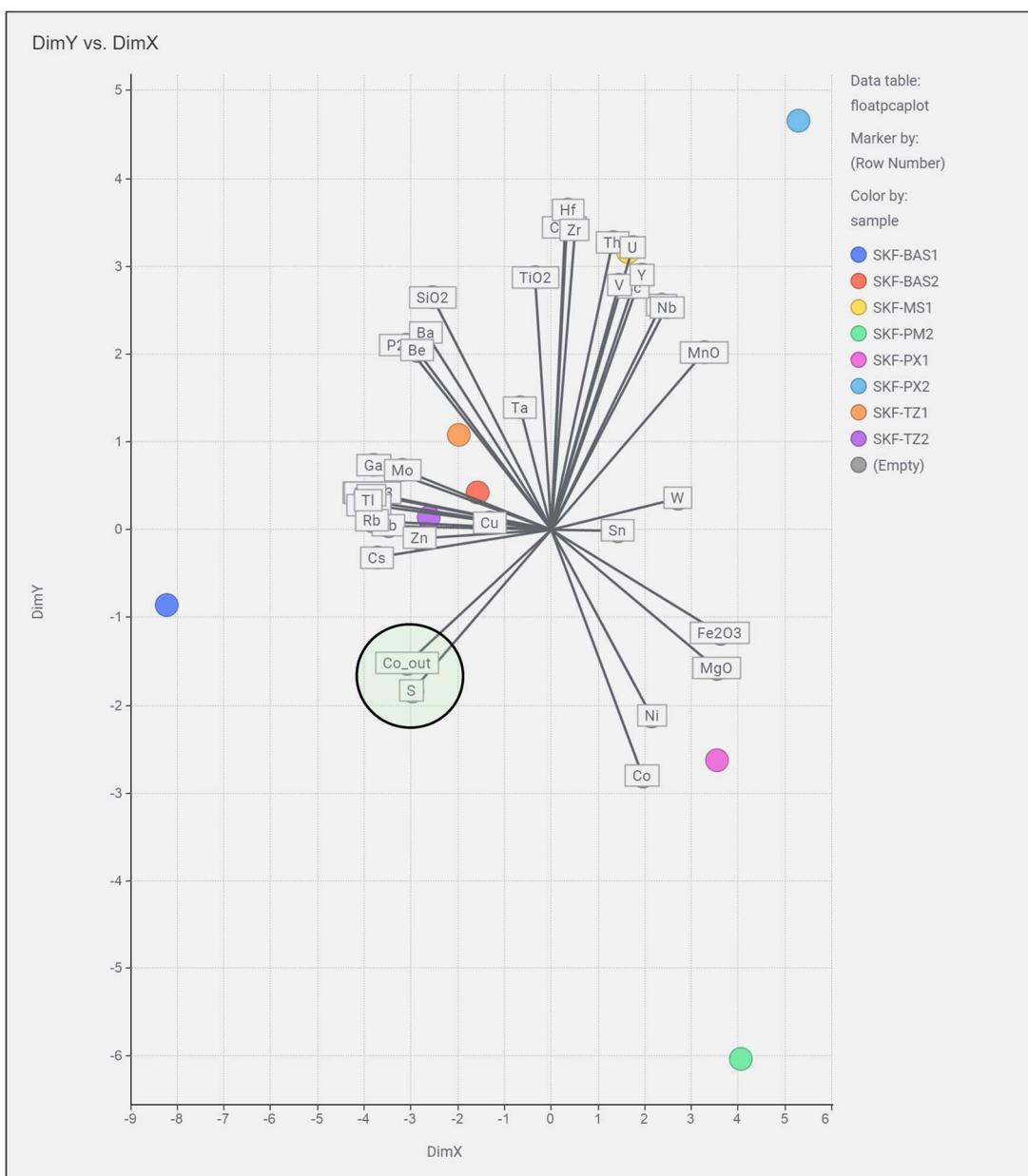


Figure 133. Principal Component Analysis of cobalt recovery after 20 minutes cell residence time and all other feed characteristics to indicate elemental associations. Note that due to the small number of samples, these results are only indicative (Analysis done with Spotfire software by John Martin, Hafren Scientific)

Given what was learned from Figures 133 several elements were selected and plotted against cobalt recovery (20 minutes cell residence time). Cobalt recovery is strongly associated with Sulfur (S). Figure 134 shows copper recovery plotted against Sulfur (S) and MnO.

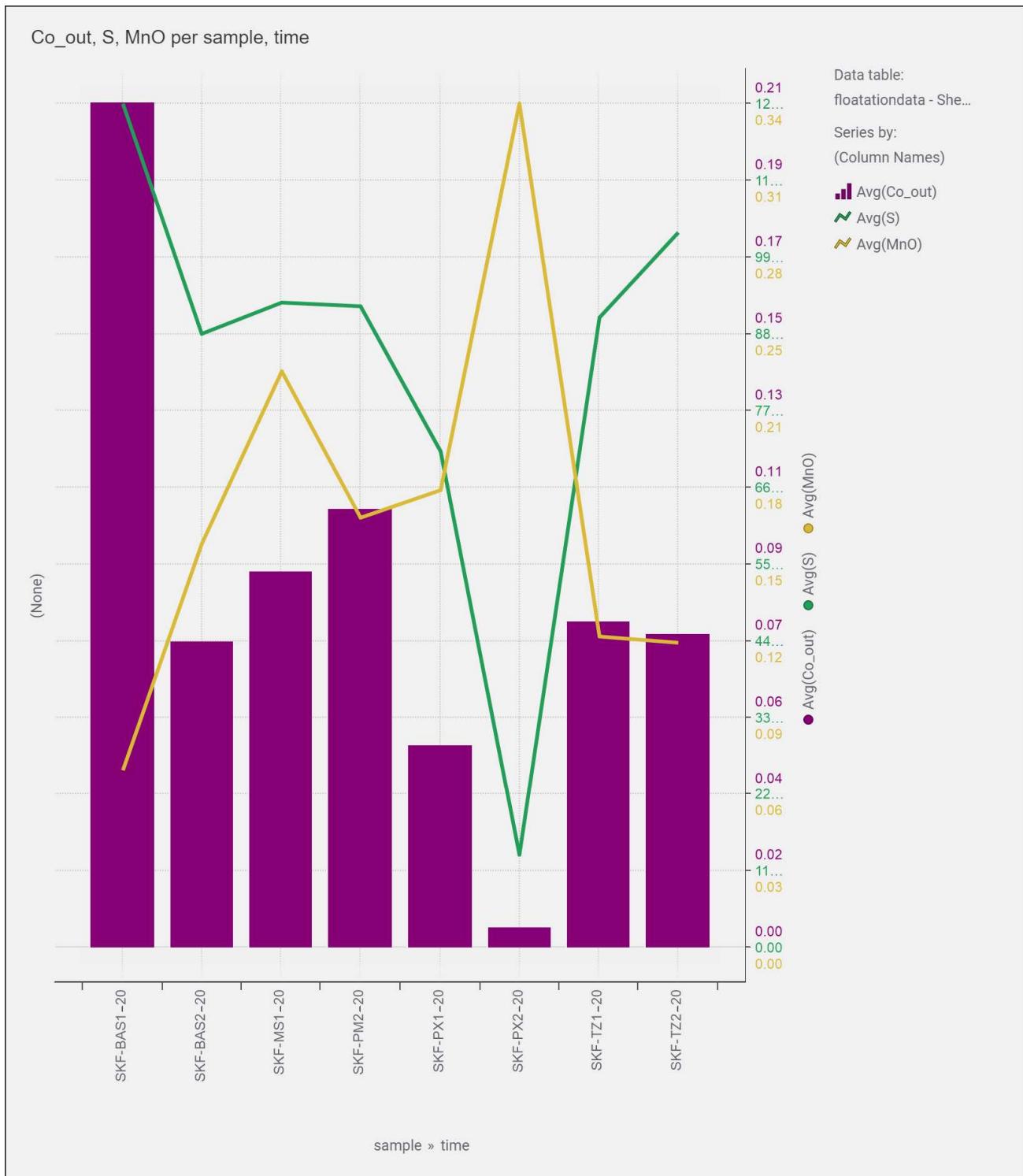


Figure 134. Cobalt recoveries after 20 minutes cell residence time plotted against Sulfur (S) and MnO
(Analysis done with Spotfire software by John Martin, Hafren Scientific)

8.6 Characterization of flotation tailings

The tailings of the flotation tests were characterized. Tables 17, 18 and 19 show the results. This data would be useful in planning tailings management. ICP measurement data for flotation tails is shown in Appendix F.

Table 17. Characterization of flotation tailings PM2, PX1 and PX2

Sample	Product	Units	PM2		PX1		PX2	
			Rougher Tails RC1-4	Final Conc Tails CT2	Rougher Tails RC1-4	Final Conc Tails CT1	Rougher Tails RC1-4	Final Conc Tails CT1
Label								
Mass	(g)		656.5	110.2	928.8	58.5	949.9	37.9
Fraction of feed sample	(%)		66.2	11.1	93.1	5.9	95.4	3.8
Cu (XRF)	(%) in sample		0.02	0.01	0.01	0.07	0.00	0.01
	Recovery (%)		14.1	0.9	9.4	4.2	16.7	1.7
Ni (XRF)	(%) in sample		0.17	0.18	0.11	0.24	0.10	0.11
	Recovery (%)		47.4	8.3	70.2	9.9	95.1	4.0
Co (XRF)	(%) in sample		0.02	0.18	0.025	0.026	0.020	0.026
	Recovery (%)		28.7	51.2	85.2	5.6	94.5	4.9
Fe (XRF)	(%) in sample		14.3	4.7	8.3	9.3	9.1	9.5
	Recovery (%)		81.3	4.5	90.7	6.4	95.1	4.0
S (Eltra)	(%) in sample		0.40	0.05	0.11	0.61	0.01	0.05
	Recovery (%)		54.9	1.2	31.3	10.9	31.4	4.9
SiO ₂ (XRF)	(%) in sample		36.1	61.4	48.6	46.3	49.4	48.9
	Recovery (%)		56.0	16.0	93.9	5.6	95.5	3.8
MgO (XRF)	(%) in sample		30.0	30.2	23.6	23.5	20.2	20.4
	Recovery (%)		66.5	11.2	93.6	5.9	95.4	3.8
Au (FA+AAS)	(g/t)		0.03	0.02	0.02	0.16	0.23	1.21
	Recovery (%)		17.9	2.0	20.1	10.1	59.3	12.4
Ag (FA+AAS)	(g/t)		0.50	0.50	0.60	1.40	0.55	1.82
	Recovery (%)		46.5	7.8	54.4	8.0	57.6	7.6
Pd (FA+AAS)	(g/t)		0.37	0.21	0.43	3.43	0.48	2.28
	Recovery (%)		19.9	1.9	23.3	11.7	31.8	6.1
Pt (FA+AAS)	(g/t)		0.13	0.07	0.18	1.00	0.18	0.69
	Recovery (%)		36.1	3.3	32.3	11.3	38.5	5.7

Table 18. Characterization of flotation tailings MS1, MS2 and TZ1

Sample	Units	MS1		MS2		TZ1	
		Rougher Tails RC1-4	Final Conc Tails CT1	Rougher Tails RC1-4	Final Conc Tails CT1	Rougher Tails RC1-4	Final Conc Tails CT1
Mass	(g)	941.1	36.6	937.6	42.6	948.8	32.5
Fraction of feed sample	(%)	94.3	3.7	94.7	4.3	95.2	3.3
Cu (XRF)	(%) in sample	0.02	0.08	0.02	0.12	0.03	0.29
	Recovery (%)	12.8	1.9	9.7	3.2	17.4	5.5
Ni (XRF)	(%) in sample	0.08	0.16	0.08	0.25	0.05	0.30
	Recovery (%)	53.1	4.4	60.4	8.6	45.3	8.9
Co (XRF)	(%) in sample	0.031	0.037	0.022	0.027	0.023	0.032
	Recovery (%)	85.7	4.0	87.2	4.9	84.9	4.0
Fe (XRF)	(%) in sample	8.0	8.9	4.9	5.9	5.6	7.4
	Recovery (%)	91.0	3.9	90.7	4.9	90.3	4.1
S (Eltra)	(%) in sample	0.17	0.41	0.07	0.52	0.14	1.16
	Recovery (%)	31.3	2.9	19.2	6.6	29.9	8.4
SiO ₂ (XRF)	(%) in sample	54.1	53.0	52.6	51.5	54.7	52.3
	Recovery (%)	95.2	3.6	95.4	4.2	96.1	3.1
MgO (XRF)	(%) in sample	14.1	14.3	9.6	9.8	7.5	7.5
	Recovery (%)	95.1	3.7	95.2	4.4	96.2	3.3
Au (FA+AAS)	(g/t)	0.06	0.10	0.06	0.31	0.06	0.25
	Recovery (%)	51.3	3.3	28.7	6.7	51.7	7.4
Ag (FA+AAS)	(g/t)	0.80	1.60	1.00	3.80	0.70	3.90
	Recovery (%)	55.7	4.3	43.1	7.4	52.4	10.0
Pd (FA+AAS)	(g/t)	0.55	1.79	0.31	4.58	0.50	4.45
	Recovery (%)	34.8	4.4	15.5	10.4	34.0	10.4
Pt (FA+AAS)	(g/t)	0.18	0.46	0.19	1.13	0.17	0.98
	Recovery (%)	45.1	4.5	36.8	9.9	47.1	9.4

Table 19. Characterization of flotation tailings TZ2, BAS1 and BAS2

Sample	Units	TZ2		BAS1		BAS2	
		Rougher Tails RC1-4	Final Conc Tails CT1	Rougher Tails RC1-4	Final Conc Tails CT1	Rougher Tails RC1-4	Final Conc Tails CT1
Mass	(g)	946.1	31.1	936.7	31.7	935.5	35.3
Fraction of feed sample	(%)	95.4	3.1	94.0	3.2	93.8	3.5
Cu (XRF)	(%) in sample	0.01	0.10	0.02	0.07	0.04	0.38
	Recovery (%)	11.5	2.6	10.7	1.5	9.4	3.1
Ni (XRF)	(%) in sample	0.04	0.12	0.04	0.16	0.04	0.10
	Recovery (%)	41.4	4.5	25.5	3.2	67.7	5.8
Co (XRF)	(%) in sample	0.034	0.026	0.023	0.025	0.021	0.024
	Recovery (%)	88.5	2.2	75.3	2.8	86.0	3.7
Fe (XRF)	(%) in sample	5.4	7.1	3.9	5.1	6.7	7.6
	Recovery (%)	88.0	3.8	78.8	3.6	89.1	3.8
S (Eltra)	(%) in sample	0.20	0.79	0.17	0.67	0.09	0.68
	Recovery (%)	32.7	4.2	19.3	2.6	13.9	4.0
SiO ₂ (XRF)	(%) in sample	52.4	50.3	55.6	53.9	52.3	51.0
	Recovery (%)	96.4	3.0	95.8	3.1	95.2	3.5
MgO (XRF)	(%) in sample	9.7	9.7	5.1	5.1	8.5	8.0
	Recovery (%)	96.5	3.2	96.2	3.2	95.9	3.4
Au (FA+AAS)	(g/t)	0.04	0.11	0.04	0.13	0.03	0.29
	Recovery (%)	41.1	3.7	33.1	3.6	25.2	9.2
Ag (FA+AAS)	(g/t)	0.60	2.70	0.46	4.02	0.50	2.36
	Recovery (%)	47.4	7.0	22.1	6.5	38.8	6.9
Pd (FA+AAS)	(g/t)	0.20	1.40	0.44	2.10	0.40	4.38
	Recovery (%)	17.5	4.0	24.4	3.9	24.4	10.1
Pt (FA+AAS)	(g/t)	0.12	0.46	0.14	0.57	0.11	1.20
	Recovery (%)	38.5	4.9	40.0	5.5	25.8	10.6

9 LEACHING

The Konttijärvi Orientation samples were representatively sub-divided after crushing, where one of the sub-products was assigned for leaching test work. These samples were sent CSIRO in Australia for this work to be done.

Due to logistical restrictions associated with the COVID-19 pandemic, the samples have been held in Australian customs. Currently it has been requested that the samples are returned to Finland as soon as practical.

As such, the data results for the leaching of these samples was not available at the time of the writing of this report. It recommended that this work is completed, as the fundamental experimental question for this deposit involves a comparison between flotation recovery and leaching recovery.

10 SORTING

Some of each Orientation Sample was prepared to be subject to sorting techniques to determine the potential for this to be used. The samples prepared were too small in mass (only 5kg) and were selected in a fashion where mineralogy was very similar internally. While it was possible for the system to determine the difference between samples, it was decided to source new material. SAP staff supplied a length of drill core in trays that crosses domains. Mine On-Line Service IMA (BATCircle partner) measured the core samples while they still were in the core trays, with the core intact, and returned the sample undamaged to SAP on completion of the study. This section is a description of that work done, in collaboration with IMA Chairman, Ilpo Auranen.

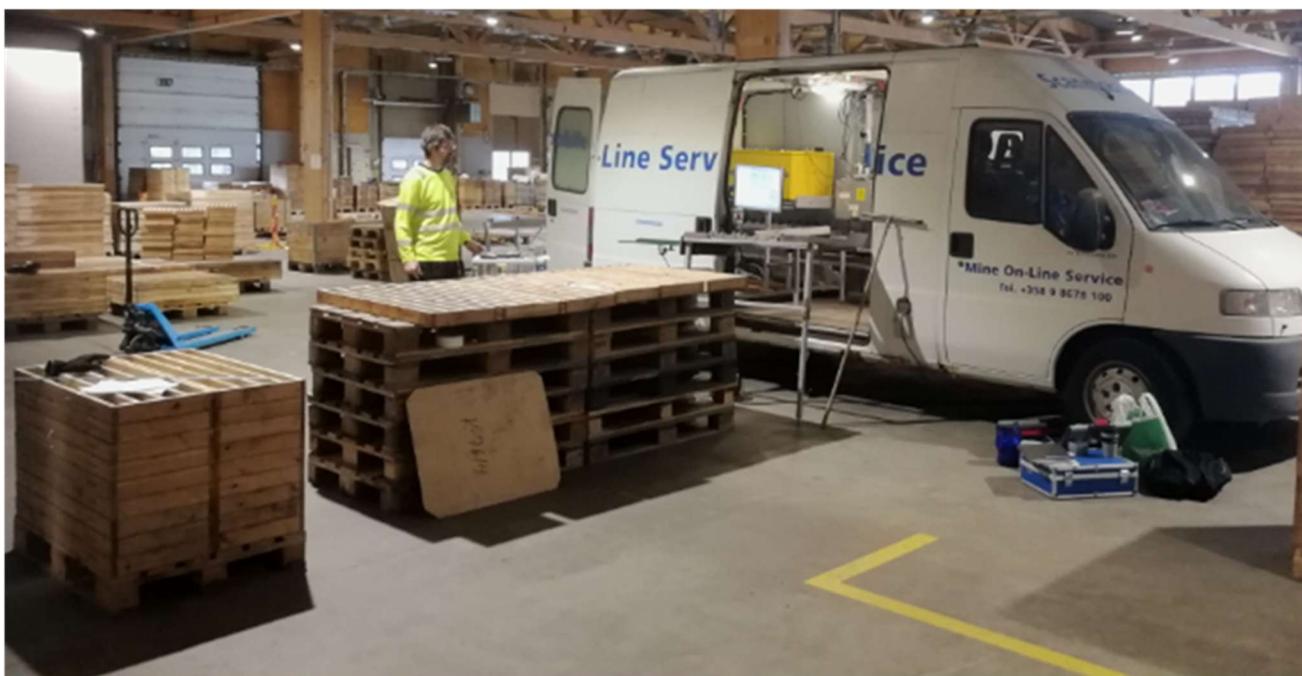


Figure 135. IMA Mine On-Line Service Scanmobile drill core analysis laboratory (Ilpo Auranen IMA)

The IMA Scanmobile Unit used an XRF based sensor suite to examine the metal content in the intact core. This method reveals and quantifies unrecognized ore recovery and waste rejection potential for ore sorting. This could be very important, especially in small, marginal orebodies. If waste could be rejected, less ore is put in the mill, thus reducing operating costs without losing metal content.

Evidence of ore/waste heterogeneity is masked by:

- Common drill core sampling and analysis practices where analysis technology and related costs is setting limits to sample size/length thus averaging the ore grade on core length (drill chip clippings for example)
- A general unawareness of heterogeneity present within orebodies which could be exploited to recover ore from waste or alternately reject waste from ore by ore sorting

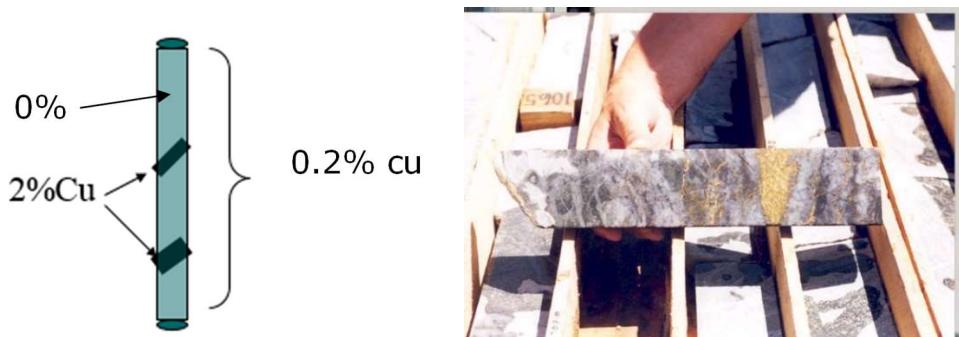


Figure 136. Heterogeneity present within the orebody which can be exploited (Ilpo Auranen IMA)

The Scanmobile unit did fast scanning analysis of drill cores in short length averages (2- 20cm range) on the samples on the core still in the core tray. The core trays were put on a roller belt and sequentially passed under the sensor instrumentation.

The elements in the ore were measured by XRF scanning, and minerals were measured by Digital Imaging simultaneously. An automatic estimation of RQD-value and veins calculation for rock strength was also done.



Figure 137. scanning analysis of drill cores in short length averages (2-20cm range) (Ilpo Auranen IMA)

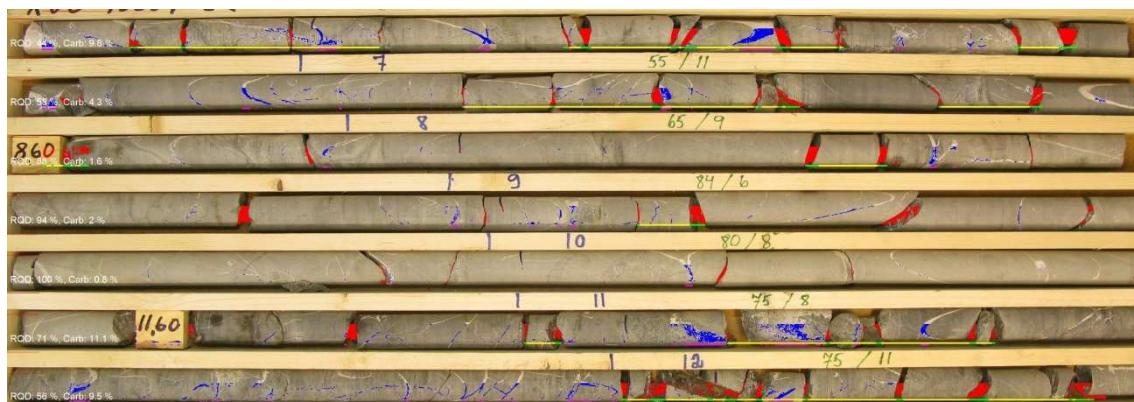


Figure 138. Scanmobile RQD index (red) and carbonates (white) calculation (Ilpo Auranen IMA)

The instrument package reported data via a proprietary software (Remolog™), which is a web based logging tool and a remote logging tool. An example of Remolog's RQD and white mineral calculation is shown in Figure 138.

Samples used for this study were taken from drill hole KOJ-598 (Figure 139). Three drill core holes were analysed with Scanmobile (SM) dense scanning XRF-analysis system. Elements were analysed by scanning XRF-analysis with 20 cm length for each analysis. Decisions made for ore sorting was based on this 20 cm sample length. Laboratory analysis grades were used for grade calculations.

A Palladium cut-off grade of 0.5 ppm was used for the ore sorting. The sample Palladium grades varied from 0 ppm to 10.4 ppm.



Figure 139. Samples from drill hole KOJ-598

Waste (as defined as below 0.5 ppm Pd) was identified by the Scanmobile unit (Figure 135). Summing results, 15% of total core length was “sorted out” as waste. This could represent a 15% reduction minerals processing tonnage processed by the mill. By ‘removing this low grade material from the feed, the average ore grade as mill head grade, would increase by 12%.

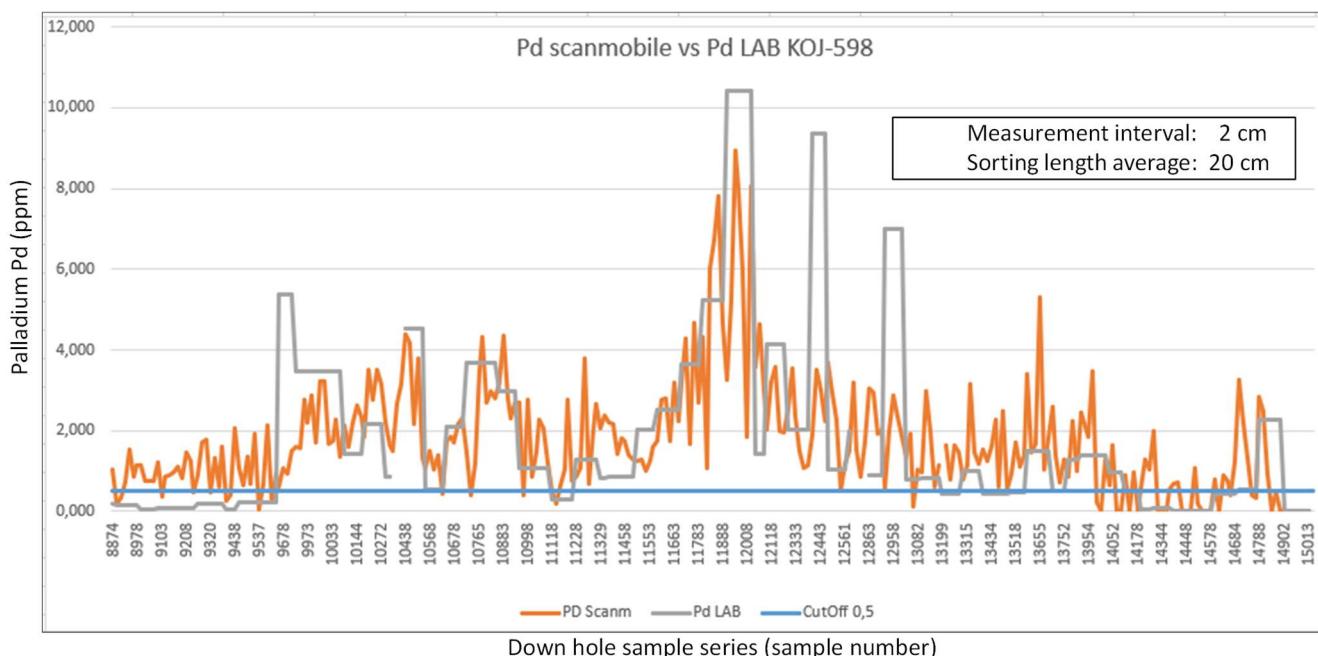


Figure 140. Palladium sorting potential measurement on intact drill core KOJ-598 (Ilpo Auranen IMA)

11 DISCUSSIONS

Figure 141 shows the average recovery across all samples for each of the valuable metals.

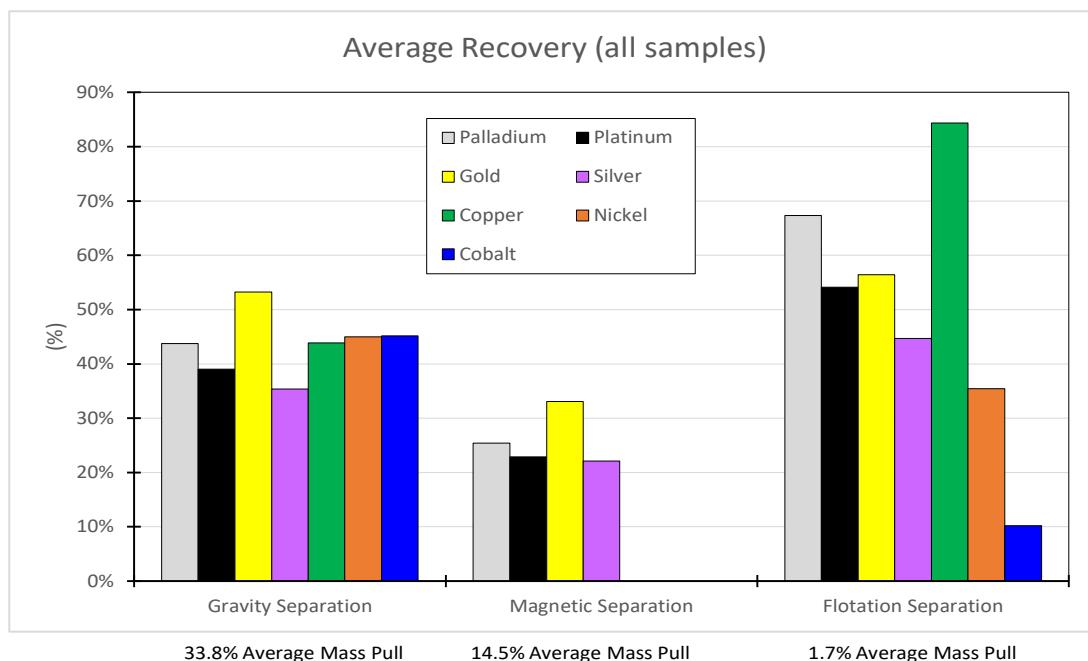


Figure 141. Average metal recovery and mass pull, all samples

Flotation is clearly the best primary recovery method. As shown in sample PM2, talc content does not seem to affect flotation. Flotation is able to concentrate the greatest recovery of methods examined into a mass of 1-2% of the feed.

Gravity and magnetic separation are able to separate some minerals, but not enough to justify the process for this purpose alone. The recovery is too low, and the mass pull into the gravity and magnetic fractions is too high to be of practical use. However, these methods could be used to clean the feed to the flotation circuit, or they could be used to clean the tailings to mitigate acid mine drainage risk.

As shown in Figures 93 & 94, gravity separation is very effective in deporting sulfur content, with an average of 85.3% of sulfur (across all samples) reporting to the gravity concentrate, with an average mass pull of 2% of feed. Gravity separation also concentrated sulfide minerals (pyrrhotite, pyrite, chalcopyrite and pentlandite), where the gravity concentrate was on average 17.81% sulfide minerals (Table 3).

Gravity separate concentrate also contains an average of 40.8% of the feed sample carbon (Figures 95 & 96), with samples PM1 and PM2 containing more carbon than other samples.

The gravity separation method was also more effective in the extraction of nickel and cobalt than flotation for the same ore types. For cobalt, average flotation recovery was 10.2% with an average mass pull of 1.7%, whereas cobalt recovery in the summed concentrate and middling's fractions was 45.1% with an average mass pull of 33.8%. Cobalt recovery in just the gravity concentrate product was 16.1% with an average mass pull of 2.0%, which is still more effective than flotation.

For nickel, average flotation recovery was 35.5% with an average mass pull of 1.7%, whereas nickel recovery in the summed concentrate and middling's fractions was 45.0% with an average mass pull of 39.5%. Nickel recovery in just the gravity concentrate product was 13.5% with an average mass pull of 2.0%, which is still more effective than flotation.

Magnetic separation had a similar profile but was not as effective as gravity separation.

Figures 142 to 148 and Tables 20 to 26 show the metal recoveries for all metals of interest.

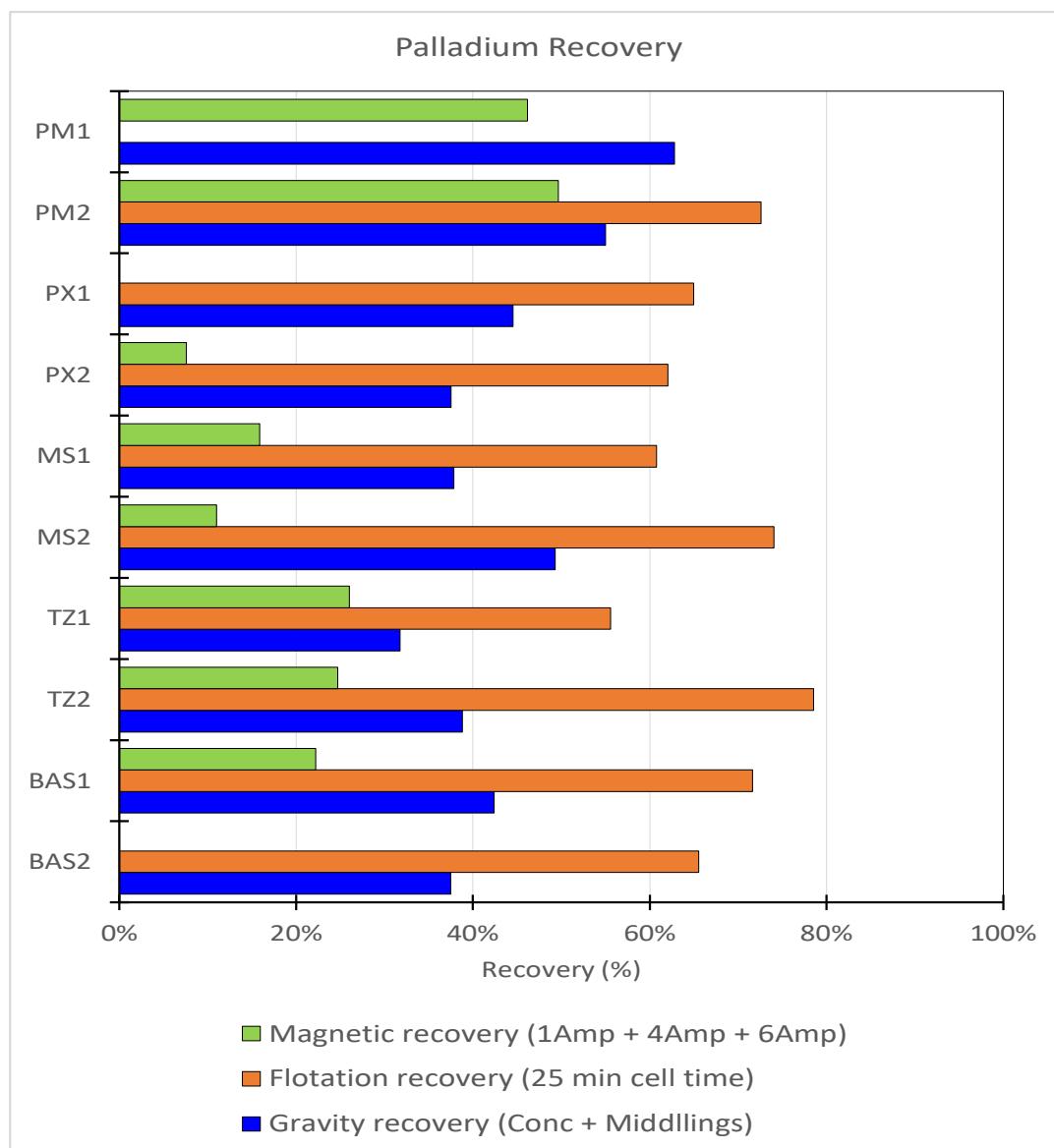


Figure 142. Palladium recovery magnetic, gravity and flotation compared

Table 20. Palladium recovery magnetic, gravity and flotation compared

PALLADIUM	Gravity Recovery (Conc + Middlings) (%)	Mass Pull (wt%)	Magnetic Recovery (1Amp + 4Amp + 6Amp) (%)	Mass Pull (wt%)	Flotation Recovery (25 min cell time) (%)	Mass Pull (wt%)
PM1	62.8%	43.7%	46.2%	28.3%	72.6%	2.1%
PM2	55.0%	36.3%	49.6%	34.9%	65.0%	1.0%
PX1	44.5%	29.1%			62.1%	0.8%
PX2	37.5%	30.4%	7.6%	9.6%	60.8%	2.0%
MS1	37.9%	34.8%	15.9%	10.6%	74.0%	1.0%
MS2	49.3%	39.6%	11.0%	4.9%	55.6%	1.5%
TZ1	31.8%	24.7%	26.0%	9.5%	78.5%	1.5%
TZ2	38.8%	35.4%	24.7%	9.6%	71.6%	2.8%
BAS1	42.4%	31.6%	22.2%	8.8%	65.5%	2.6%
BAS2	37.5%	32.7%				

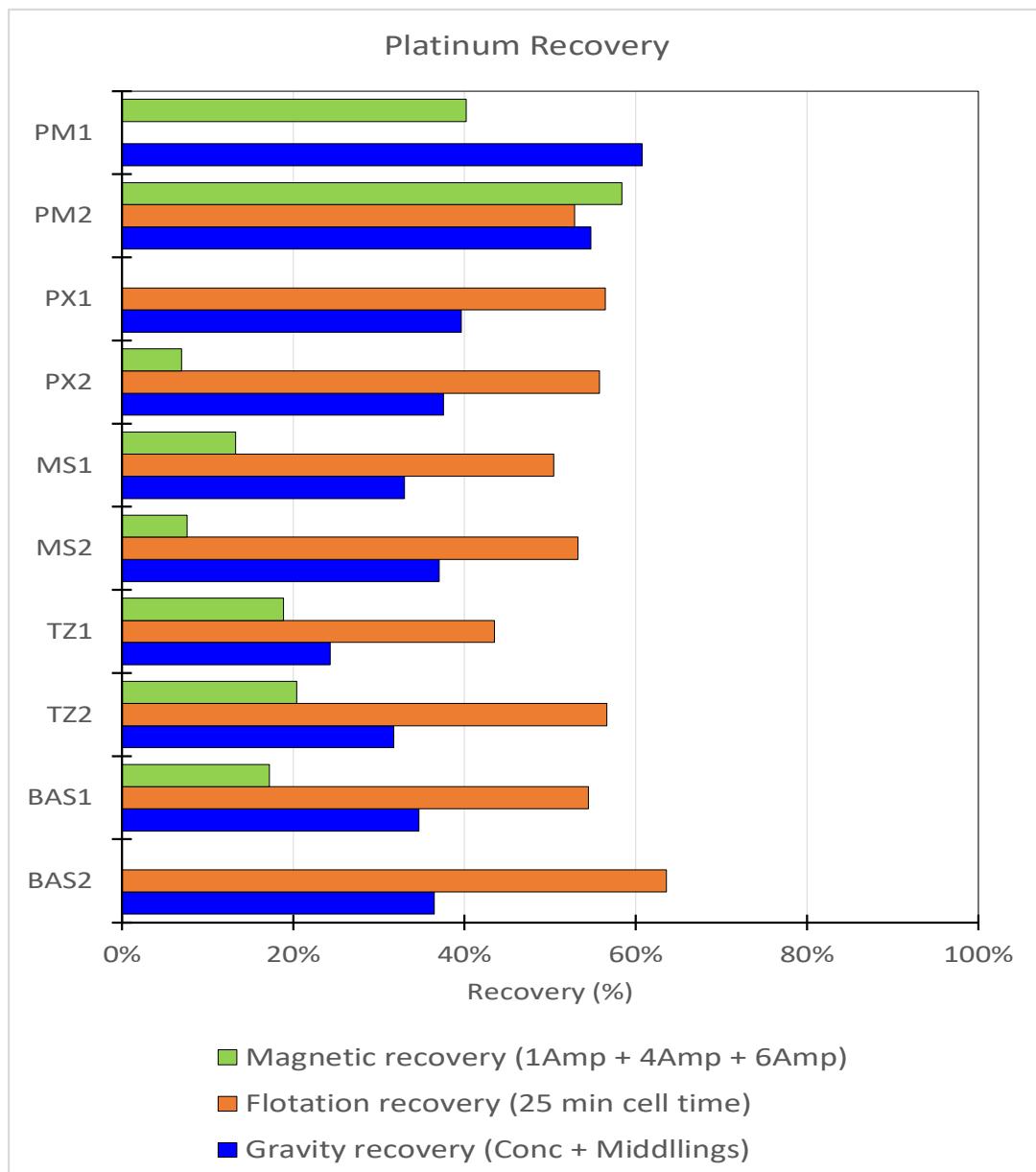


Figure 143. Platinum recovery magnetic, gravity and flotation compared

Table 21. Platinum recovery magnetic, gravity and flotation compared

PLATINUM	Gravity Recovery (Conc + Middlings) (%)	Mass Pull (wt%)	Magnetic Recovery (1Amp + 4Amp + 6Amp) (%)	Mass Pull (wt%)	Flotation Recovery (25 min cell time) (%)	Mass Pull (wt%)
PM1	60.8%	43.7%	40.2%	28.3%	52.9%	2.1%
PM2	54.8%	36.3%	58.4%	34.9%	56.4%	1.0%
PX1	39.7%	29.1%			55.8%	0.8%
PX2	37.6%	30.4%	7.0%	9.6%	50.4%	2.0%
MS1	33.0%	34.8%	13.2%	10.6%	53.3%	1.0%
MS2	37.0%	39.6%	7.6%	4.9%	43.5%	1.5%
TZ1	24.3%	24.7%	18.9%	9.5%	56.6%	1.5%
TZ2	31.7%	35.4%	20.4%	9.6%	54.5%	2.8%
BAS1	34.7%	31.6%	17.2%	8.8%	63.6%	2.6%
BAS2	36.5%	32.7%				

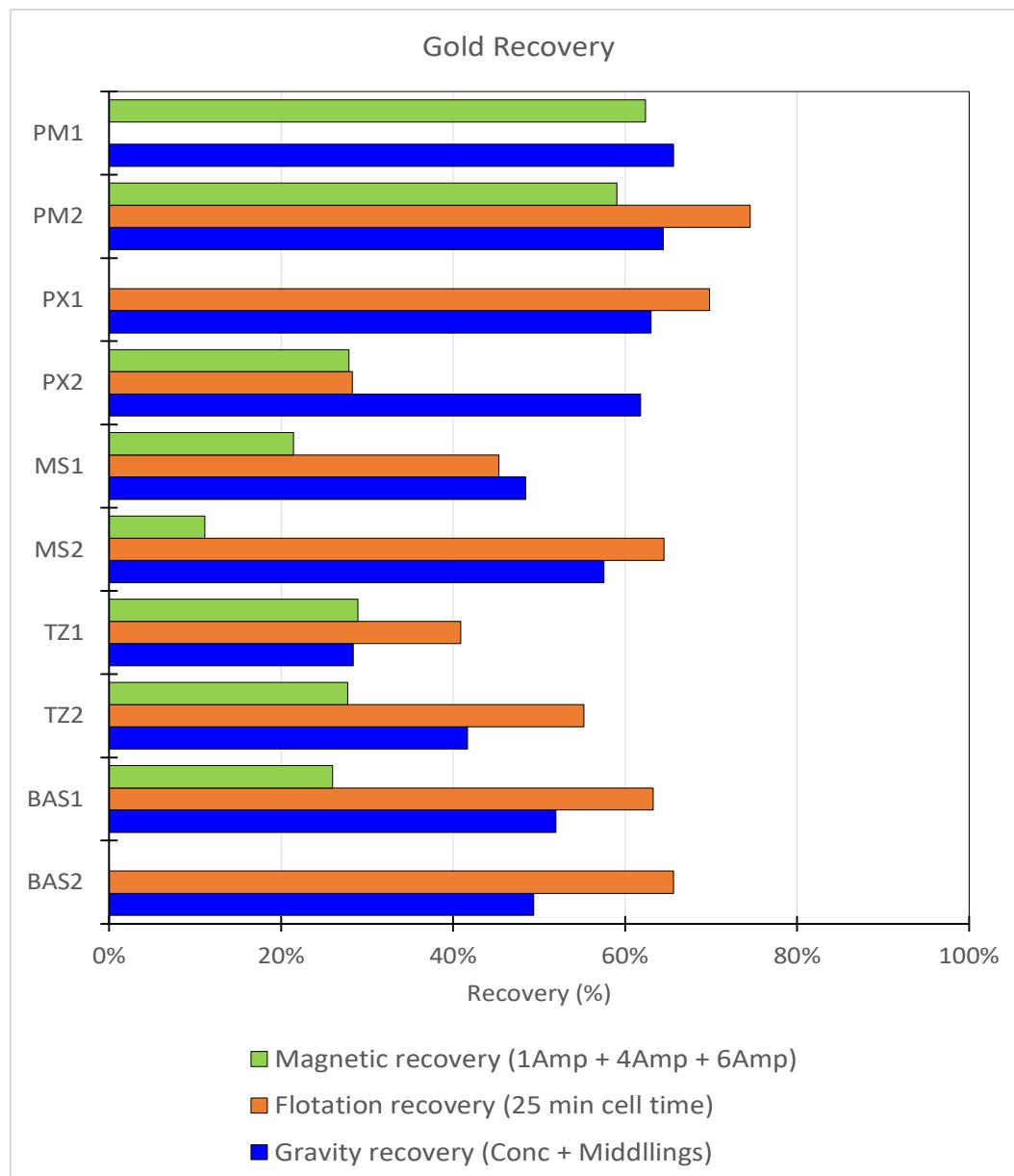


Figure 144. Gold recovery magnetic, gravity and flotation compared

Table 22. Gold recovery magnetic, gravity and flotation compared

GOLD	Gravity Recovery (Conc + Middlings) (%)	Mass Pull (wt%)	Magnetic Recovery (1Amp + 4Amp + 6Amp) (%)	Mass Pull (wt%)	Flotation Recovery (25 min cell time) (%)	Mass Pull (wt%)
PM1	65.6%	43.7%	62.4%	28.3%		
PM2	64.4%	36.3%	59.1%	34.9%	74.6%	2.1%
PX1	63.0%	29.1%			69.8%	1.0%
PX2	61.8%	30.4%	27.9%	9.6%	28.3%	0.8%
MS1	48.4%	34.8%	21.4%	10.6%	45.3%	2.0%
MS2	57.5%	39.6%	11.2%	4.9%	64.5%	1.0%
TZ1	28.4%	24.7%	28.9%	9.5%	40.9%	1.5%
TZ2	41.7%	35.4%	27.7%	9.6%	55.2%	1.5%
BAS1	51.9%	31.6%	26.0%	8.8%	63.3%	2.8%
BAS2	49.4%	32.7%			65.6%	2.6%

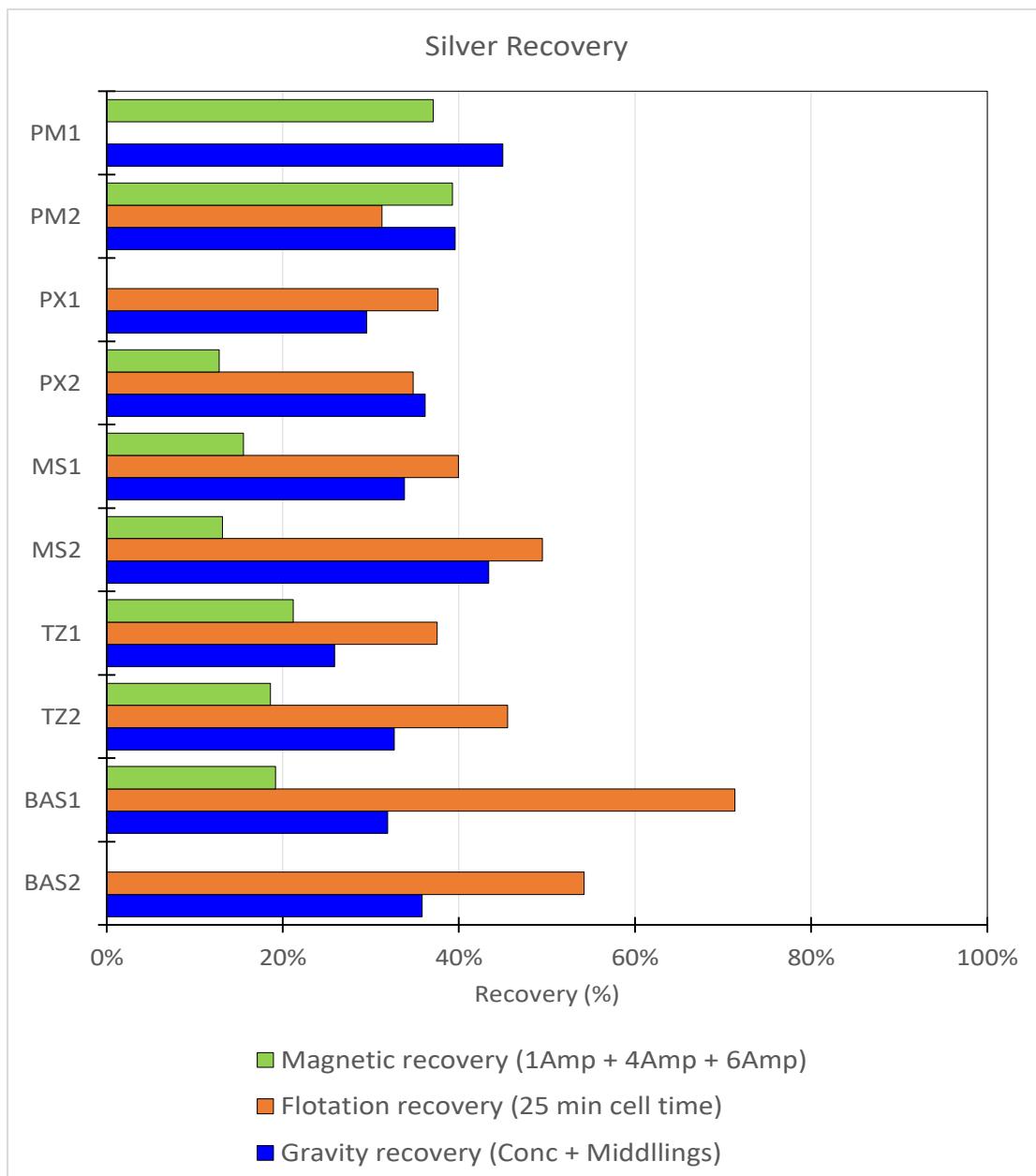


Figure 145. Silver recovery magnetic, gravity and flotation compared

Table 23. Silver recovery magnetic, gravity and flotation compared

SILVER	Gravity Recovery (Conc + Middlings) (%)	Mass Pull (wt%)	Magnetic Recovery (1Amp + 4Amp + 6Amp) (%)	Mass Pull (wt%)	Flotation Recovery (25 min cell time) (%)	Mass Pull (wt%)
PM1	45.0%	43.7%	37.1%	28.3%		
PM2	39.6%	36.3%	39.3%	34.9%	31.3%	2.1%
PX1	29.5%	29.1%			37.7%	1.0%
PX2	36.2%	30.4%	12.8%	9.6%	34.8%	0.8%
MS1	33.9%	34.8%	15.5%	10.6%	40.0%	2.0%
MS2	43.4%	39.6%	13.1%	4.9%	49.5%	1.0%
TZ1	25.9%	24.7%	21.2%	9.5%	37.6%	1.5%
TZ2	32.7%	35.4%	18.6%	9.6%	45.5%	1.5%
BAS1	31.9%	31.6%	19.2%	8.8%	71.4%	2.8%
BAS2	35.8%	32.7%			54.2%	2.6%

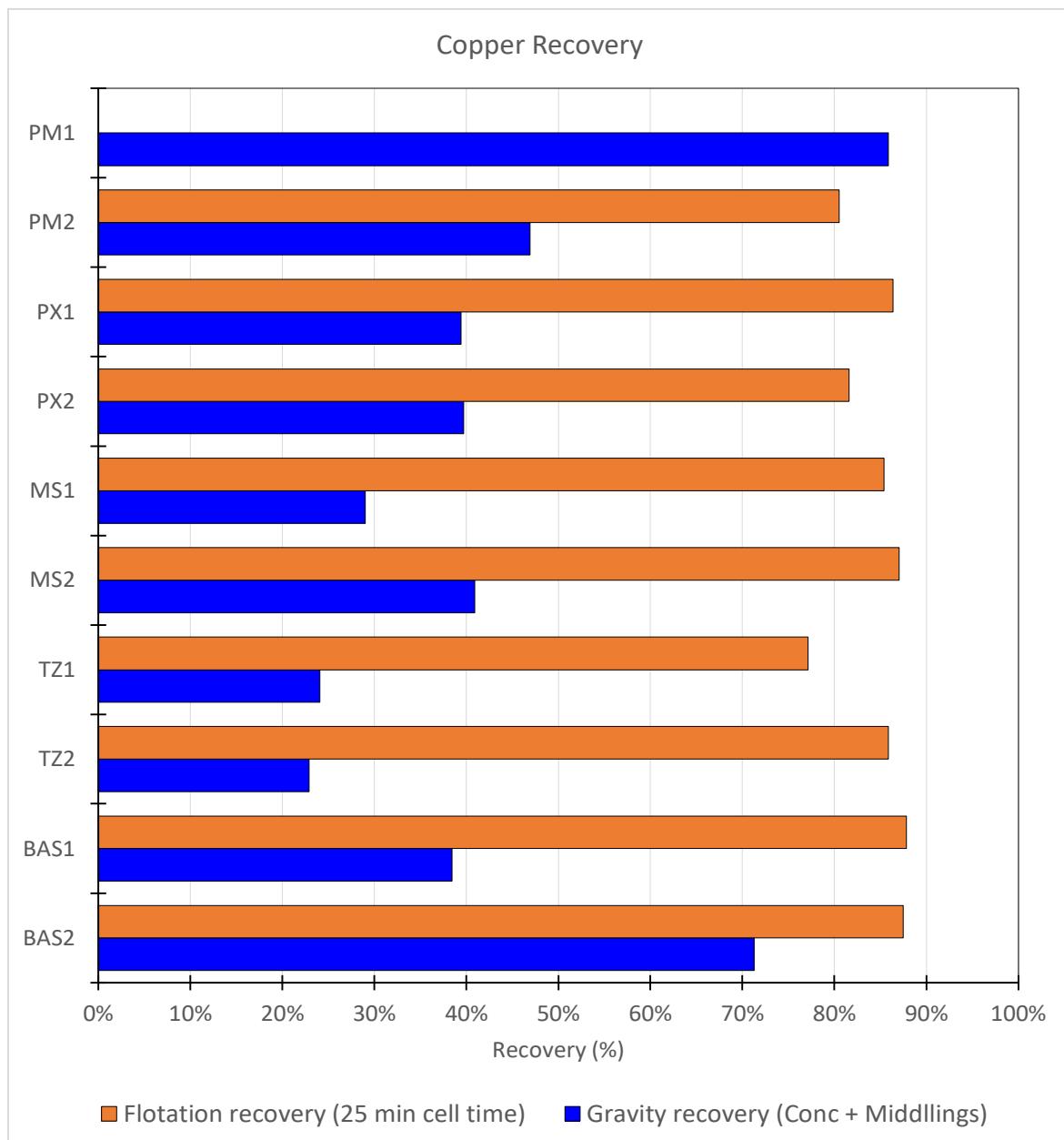


Figure 146. Copper recovery gravity and flotation compared

Table 24. Copper recovery gravity and flotation compared

COPPER	Gravity Recovery (Conc + Middlings) (%)	Mass Pull (wt%)	Flotation Recovery (25 min cell time) (%)	Mass Pull (wt%)
PM1	85.9%	43.7%	80.5%	
PM2	46.9%	36.3%	86.4%	2.1%
PX1	39.4%	29.1%	81.6%	1.0%
PX2	39.7%	30.4%	85.4%	0.8%
MS1	29.0%	34.8%	87.1%	2.0%
MS2	40.9%	39.6%	77.1%	1.0%
TZ1	24.1%	24.7%	85.9%	1.5%
TZ2	22.9%	35.4%	87.9%	1.5%
BAS1	38.5%	31.6%	87.5%	2.8%
BAS2	71.3%	32.7%	89.0%	2.6%

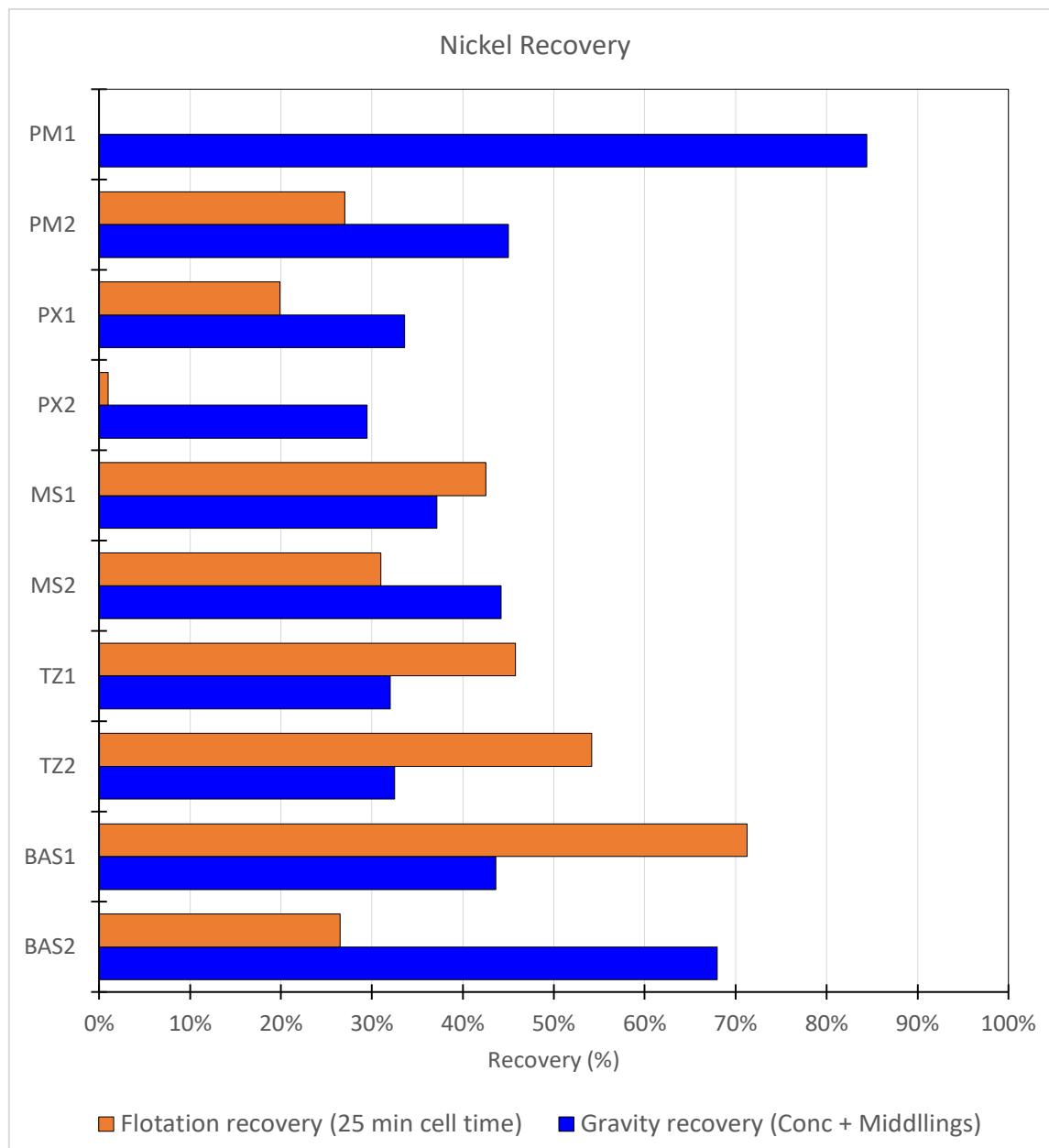


Figure 147. Nickel recovery gravity and flotation compared

Table 25. Nickel recovery gravity and flotation compared

NICKEL	Gravity Recovery (Conc + Middlings) (%)	Mass Pull (wt%)	Flotation Recovery (25 min cell time) (%)	Mass Pull (wt%)
PM1	84.4%	43.7%		
PM2	45.0%	36.3%	27.0%	2.1%
PX1	33.6%	29.1%	19.9%	1.0%
PX2	29.5%	30.4%	1.0%	0.8%
MS1	37.1%	34.8%	42.5%	2.0%
MS2	44.2%	39.6%	31.0%	1.0%
TZ1	32.0%	24.7%	45.8%	1.5%
TZ2	32.5%	35.4%	54.2%	1.5%
BAS1	43.6%	31.6%	71.3%	2.8%
BAS2	68.0%	32.7%	26.5%	2.6%

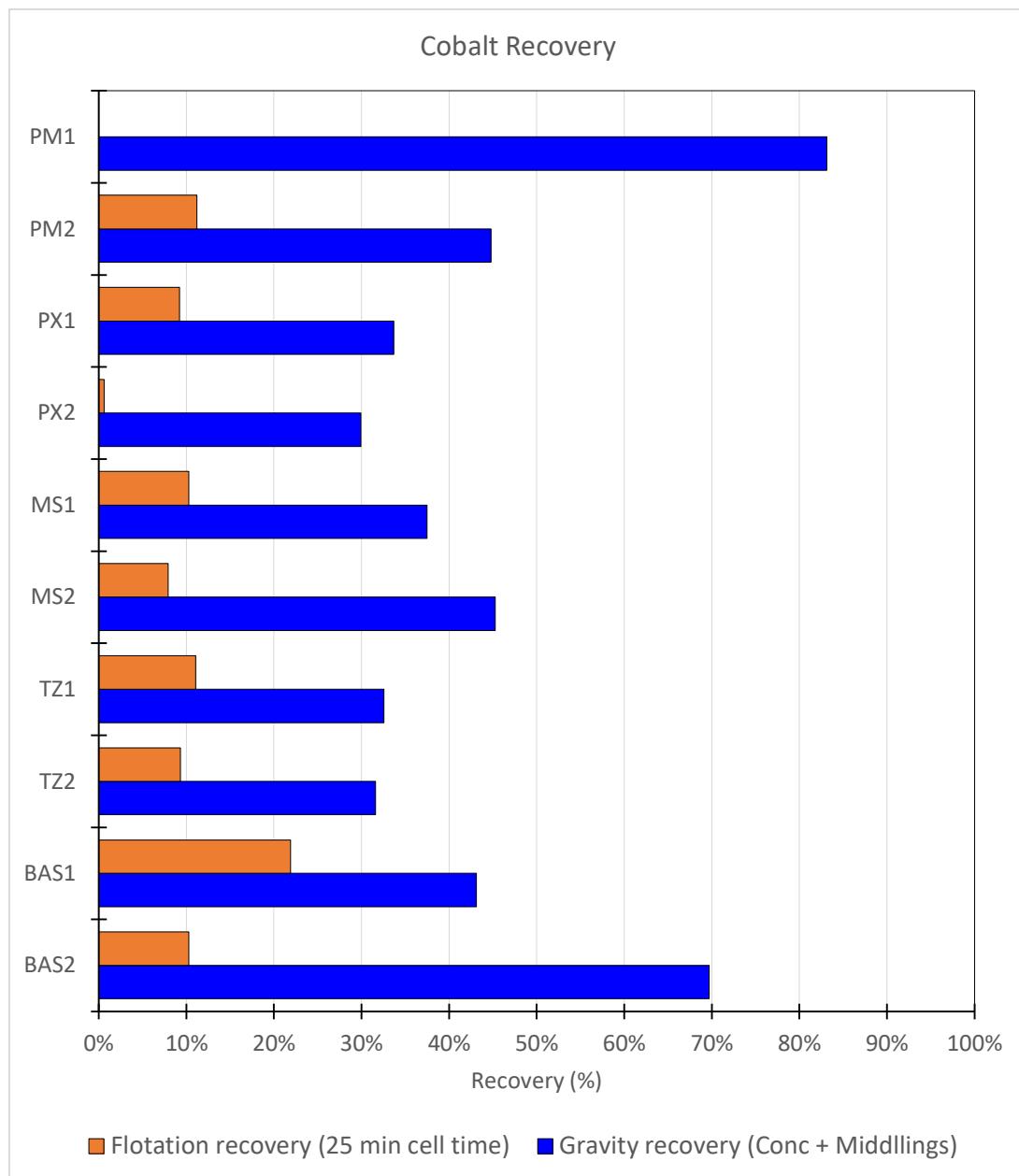


Figure 148. Cobalt recovery gravity and flotation compared

Table 26. Cobalt recovery gravity and flotation compared

COBALT	Gravity Recovery (Conc + Middlings) (%)	Mass Pull (wt%)	Flotation Recovery (25 min cell time) (%)	Mass Pull (wt%)
PM1	83.1%	43.7%		
PM2	44.8%	36.3%	11.2%	2.1%
PX1	33.7%	29.1%	9.2%	1.0%
PX2	29.9%	30.4%	0.6%	0.8%
MS1	37.5%	34.8%	10.3%	2.0%
MS2	45.3%	39.6%	7.9%	1.0%
TZ1	32.6%	24.7%	11.1%	1.5%
TZ2	31.6%	35.4%	9.3%	1.5%
BAS1	43.1%	31.6%	21.9%	2.8%
BAS2	69.7%	32.7%	10.3%	2.6%

12 CONCLUSIONS

The following conclusions have been made from this study.

- The characterization methods of XRD, XRF, ICP, Eltra C, Eltra S and MLA all were useful.
- Flotation was the most effective separation process to recover Cu, Pd, Pt, Au and Ag. It was less effective for Ni and Co.
- Each of these metals had different flotation kinetics. Copper floats relatively quickly compared to other metals, where different Orientation Samples have slightly different signatures. As the Konttijärvi deposit is a lens structure with each of the ore types in a known layered form, this difference could be exploited.
- The gravity separation process recovered some of each of metals of interest, but it was not effective enough (recovery less than flotation with an associated high mass pull). Gravity separation was very effective at the recovery of sulfur and was reasonably useful for carbon recovery.
- Magnetic separation also recovered some of each of the metals of interest but was not effective enough to be considered further.
- The trial of sorting technology on intact drill core showed that heterogeneity of palladium grade does exist in this deposit. Using a measurement interval of 2cm and a sorting decision interval of 20cm, it was theoretically possible to remove 15 % of the core mass as waste, which would have theoretically resulted in a 12% upgrade in mill feed head grade.

13 FUTURE WORK

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

- The planned leaching process separation tests should be carried out on the prepared Orientation Sample sub-samples that are currently being returned to SAP. This would complete the Geometallurgy campaign Stage 1. For each sample, three leaching tests should be considered and planned for.
 1. Direct cyanide leaching for precious metals Pd, Pt, Au and Ag
 2. Cyanide leaching with activated carbon (Carbon in Leach, CIL)
 3. Acid leaching for base metals Cu, Ni and Co
- Geometallurgy campaign Stage 2 should then be undertaken. It is recommended that two process paths be tested, and each process optimized based on what has been learned so far.
 1. Flotation Copper(-Au) > Flotation Ni-PGES(-Au-Co) > Flotation for sulfur removal (Figure 149)
 2. Flotation Copper(-Au) > Flotation Ni-PGES(-Au-Co) > Gravity for sulfur/Fe/C removal (Figure 150)

Figures 149 shows a proposed flowsheet to test and optimize in future testwork. Copper (and sometimes gold) have kinetics to float faster than other metals. So, Stage 1 flotation is to be optimized for Cu-Au recovery (roughing stage followed with a cleaning stage). A Stage 2 of flotation is proposed to be optimized for Ni-PGES(-Au-Co) recovery, which all have very similar kinetics. A third Stage 3 of flotation is proposed to be optimized for sulfur extraction, to produce a cleaner final tailings. This process would have to be optimized for each ore type as they all have slightly different kinetics (PM2 for example has fast kinetics for all metals). The grade recovery data shown is a result of the existing testwork, the numbers would be mass balanced to adjust to a new process.

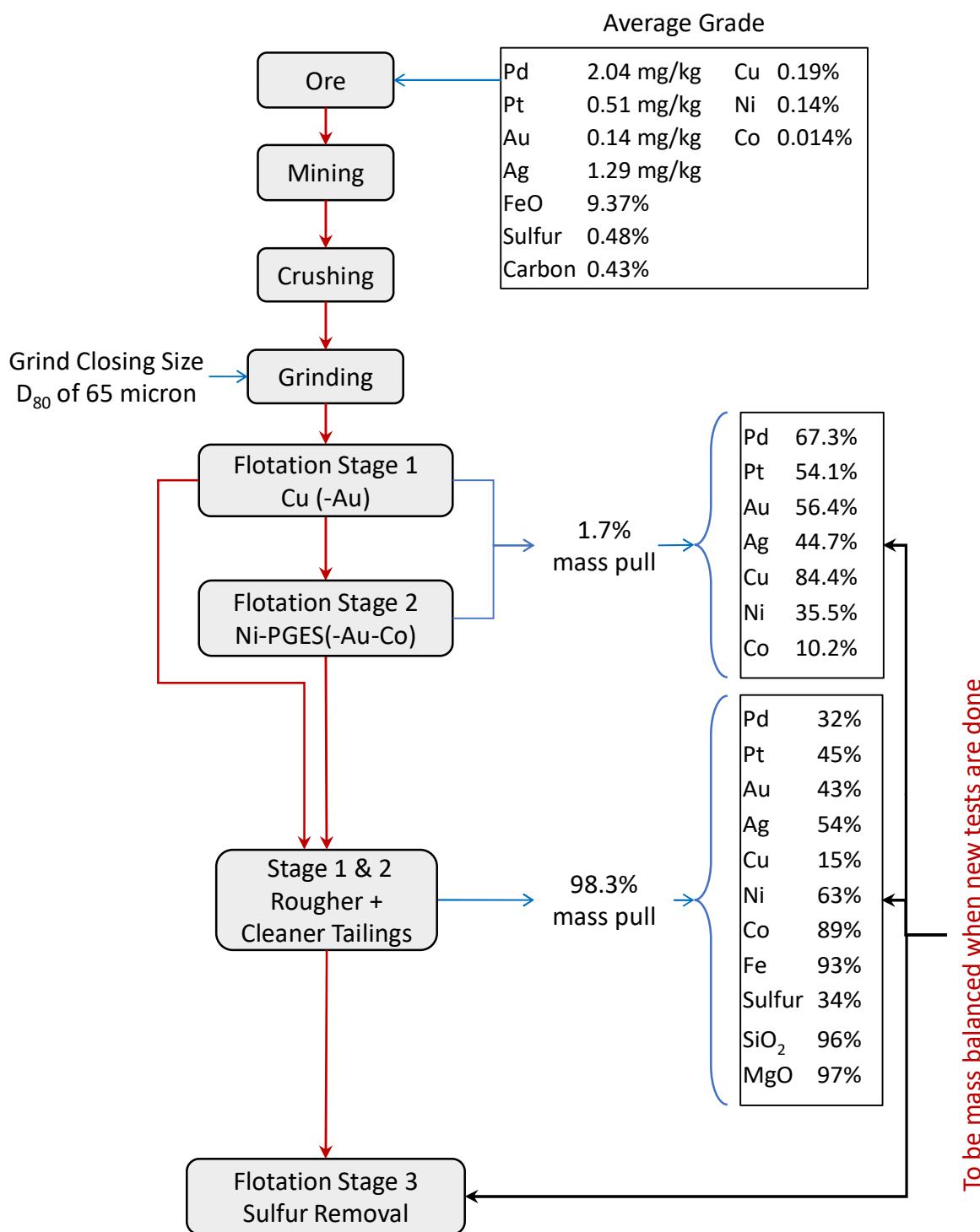


Figure 149. Possible process flow sheet to test for and optimize in future testwork - 1

What is proposed in Figure 150 is a flotation flow sheet the same as Figure 149, but instead of Stage 3 flotation (designed for sulfur cleaning), it is followed with a gravity separation to clean the tailings of sulfur, Fe and Carbon. The grade recovery data shown is a result of the existing testwork, the numbers would be mass balanced to adjust to a new process.

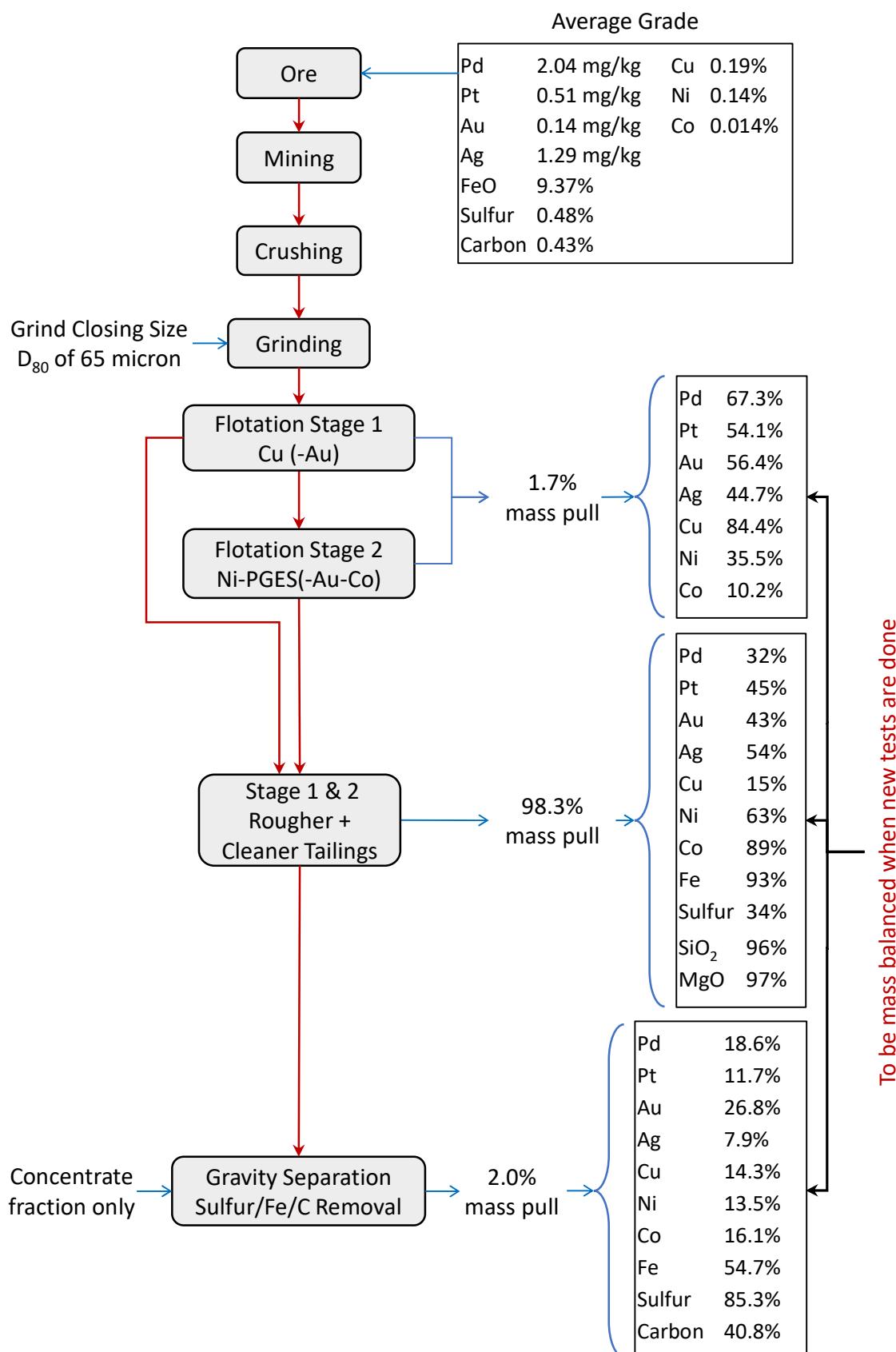


Figure 150. Possible process flow sheet to test for and optimize in future testwork - 2

14 REFERENCES

1. Anderson, K.F.E., et al., *Quantitative mineralogical and chemical assessment of the Nkout iron ore deposit, Southern Cameroon*. Ore Geology Reviews, 2014. **62**: p. 25-39.
2. Aylmore, M.G., et al., *Applications of advanced analytical and mass spectrometry techniques to the characterisation of micaceous lithium-bearing ores*. Minerals Engineering, 2018. **116**: p. 182-195.
3. Chang-Zhong L, Lingmin Z, Kaimin S, (2015). *Quantitative X-ray Diffraction (QXRD) analysis for revealing thermal transformations of red mud*, Chemosphere, Volume 131, Pages 171-177, ISSN 0045-6535
4. Fitton, G., (1997), *X-Ray fluorescence spectrometry*, in Gill, R. (ed.), *Modern Analytical Geochemistry: An Introduction to Quantitative Chemical Analysis for Earth, Environmental and Material Scientists*: Addison Wesley Longman, UK.
5. Hrstka, T., et al., (2018), *Automated mineralogy and petrology – applications of TESCAN Integrated Mineral Analyzer (TIMA)*. Journal of Geosciences, 2018. **63**(1): p. 47-63.
6. Potts, P.J., (1987), *A Handbook of Silicate Rock Analysis*: Chapman and Hall.
7. Jeans, C. (1998). MOORE, D. M. & REYNOLDS, R. C., Jr. 1997. X-Ray Diffraction and the Identification and Analysis of Clay Minerals, 2nd ed. xviii 378 pp. Oxford, New York: Oxford University Press. Price £27.95 (spiral-bound paperback). ISBN 0 19 508713 5. Geological Magazine, 135(6), 819-842. doi:10.1017/S0016756898501501
8. Michaux, S., O'Connor, L., (2020): How to Set Up and Develop a Geometallurgical Program, GTK Open Work File Report 72/2019, ISBN 978-952-217-409-3, http://tupa GTK.fi/raportti/arkisto/72_2019.pdf
9. Michaux, S., Butcher, A., Dehaine, Q., Knauf, O., Sardisco, L., Hughes, C., and Morgan, L. (2020): Characterization for Konttijärvi Geometallurgical Orientation Study Samples, GTK BATCircle project report 03 for WP1.2
10. Rollinson, H., (1993), *Using Geochemical Data: Evaluation, Presentation, Interpretation*: John Wiley, NY.
11. Dehaine, Q., Tijsseling, L.T., Glass, H.J., Törmänen, T., Butcher, A.R., 2021. Geometallurgy of cobalt ores: A review. Minerals Engineering 160, 106656. <https://doi.org/10.1016/j.mineng.2020.106656>

15 APPENDIX A - 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 A1. The BATcircle Consortium

Official BATcircle Project website: <https://www.batcircle.fi/>

15.1 BATcircle Work Package Structure

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 characterizations.

WP5 – Business Potential

Strengthening the cooperation within the overall ecosystem.

WP6 – Project management**WP7 – Development of the European BATcircle****15.2 Work Package 1 Summary**

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. Analyze 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

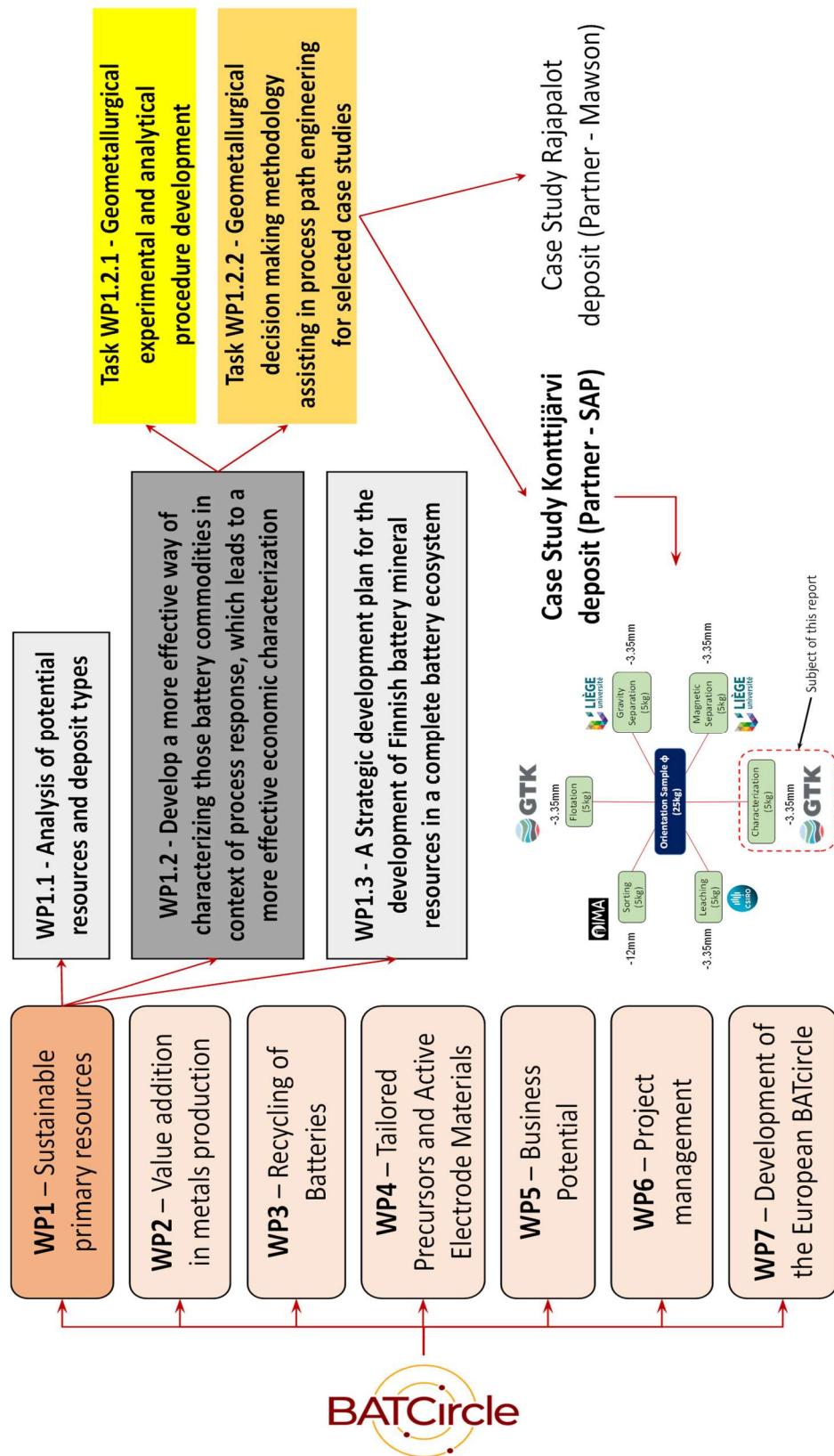


Figure A2. Map of the BATCircle project to WP1.2 to this report

15.3 Work Package WP1.2 Structure and Deliverables

Table A1. 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 A2. 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. Flotability 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 SAP case study, Konttijärvi deposit series.

15.4 Work Package 1 Project consortium

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

15.5 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 characteristics and acquired raw material specification requirements, a concept or protocol for proper geometallurgical study is developed for each deposit type.

15.6 Suhanko Arctic Platinum Oy

The Konttijärvi deposit is owned by Suhanko Arctic Platinum (SAP). SAP consists of three large project areas in Northern Finland: Suhanko, Narkaus and Penikat. There are large, untapped deposits of platinum, palladium and gold (PGE, Au) in the area with significant amounts of copper and nickel as a by-product. The deposits are located close to the ground surface and their parts extending to the rock surface continue under a thin ground cover for several kilometers.

SAP Website

<https://suhanko.com/indexFI.html>

15.7 Konttijärvi deposit (SAP) BATCircle Geometallurgical Case Study

This geometallurgical approach will be applied to two case studies. One of those BATCircle WP1.2 case studies is the Konttijärvi deposit, owned by Suhanko Arctic Platinum (SAP).

- Economic minerals in order of importance Palladium (2g/t), Pt (0.5g/t), Cu (0.16%), Ni (0.08%), Au (0.1g/t), Co, Ag, Rhodium
- The platinum group elements (PGE) are the most valuable, palladium (Pd) in particular
- Both leaching and flotation have been considered as process paths
- Sulphide extraction to be considered is both copper (Cu) and nickel (Ni)

16 APPENDIX B – BATCIRCLE PROJECT WP1.2 SAMPLE LABELLING PROTOCOL

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

The owner of the case study deposit

- Suhanko Artic Platinum (SAP)

The name of the case study deposit

- Konttijärvi K

Process separation methods

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

Orientation Sample rock type labels

- Peridotite Marker 1 rock type PM1
- Peridotite Marker 2 rock type PM2
- Pyroxenite ore 1 rock type PX1
- Pyroxenite ore 2 rock type PX2
- Marginal Series 1 rock type MS1
- Marginal Series 2 rock type MS2
- Transition Zone 1 rock type TZ1
- Transition Zone 1 rock type TZ2
- Basement rock type 1 BAS1
- Basement rock type 2 BAS2

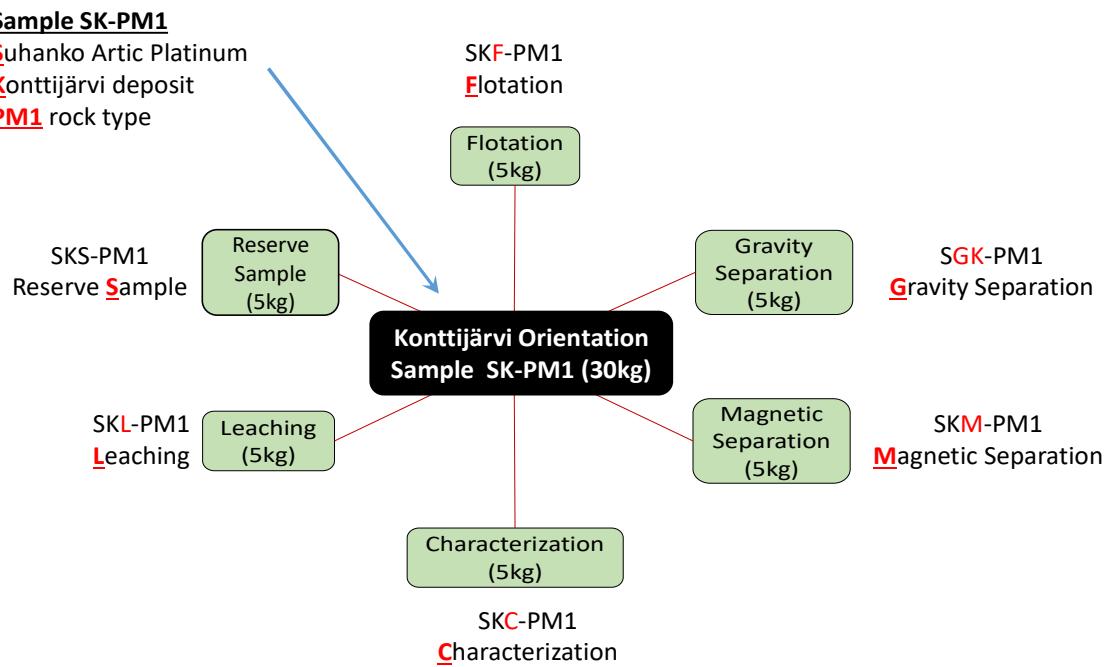


Figure B1. BATCircle Project sample labelling protocol for process separation sub-samples for each Orientation Sample

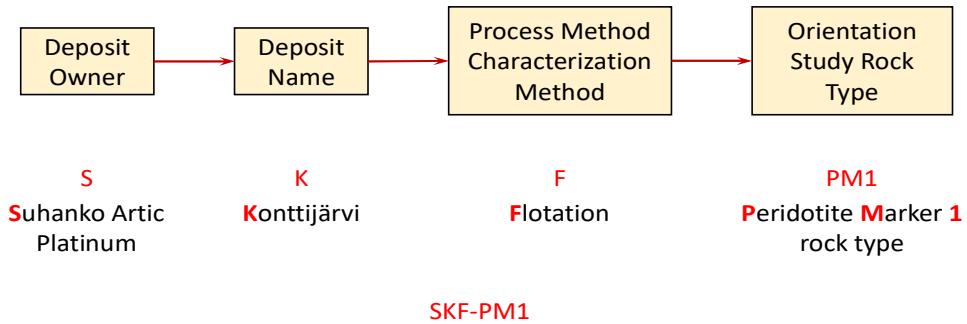


Figure B2. BATCircle Project sample labelling protocol example for sample SK-PM1 to be sent to flotation, SKF-PM1

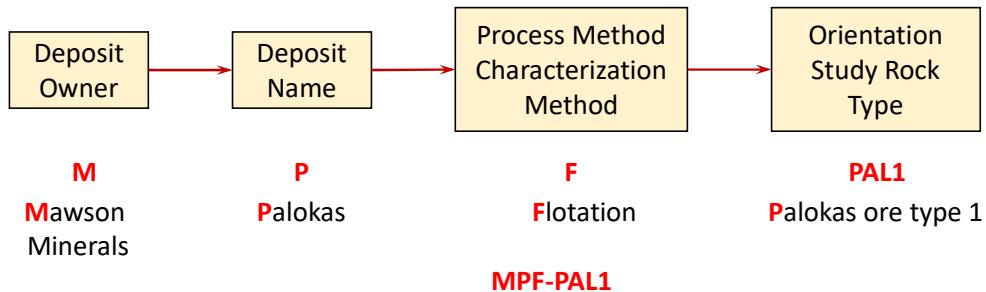


Figure B3. BATCircle Project sample labelling protocol example for sample MP-PAL1 to be sent to flotation, MPF-PAL1

Rod mill griding size distributions in preparation for test work

17 APPENDIX C – SAMPLE GRINDING SIZE DISTRIBUTIONS

Table C27. Rod mill griding size distributions in preparation for test work

Sieve Size (μm)	SKC-PM1			SKC-PM2			SKC-PX1			SKC-PX2			SKC-MS1		
	Mass (g)	Cumulative (%) Passing (%)	Mass on Sieve (%)	Mass (g)	Cumulative (%) Passing (%)	Mass on Sieve (%)	Mass (g)	Cumulative (%) Passing (%)	Mass on Sieve (%)	Mass (g)	Cumulative (%) Passing (%)	Mass on Sieve (%)	Mass (g)	Cumulative (%) Passing (%)	Mass on Sieve (%)
-100	100.0			100.0				100.0			100.0			100.0	
-150	10.13	76.6	9.0	10.36	76.1	9.0	10.42	77.2	9.1	8.7	85.3	7.6	14.9	67.8	12.0
-125	6.39	71.0	5.7	6.36	70.6	5.5	6.65	71.4	5.8	5.9	80.2	5.1	9.0	60.6	7.2
-90	12.07	60.3	10.7	12.57	59.6	10.9	12.63	60.3	11.1	10.7	70.8	9.3	16.2	47.5	13.0
-75	5.76	55.2	5.1	5.64	54.7	4.9	6.15	54.9	5.4	5.1	66.4	4.4	7.3	41.7	5.9
-45	12.95	43.7	11.5	17.18	39.8	14.9	12.91	43.6	11.3	10.7	57.2	9.3	14.5	30.0	11.6
-32	8.08	36.5	7.2	7.40	33.4	6.4	7.39	37.2	6.5	7.3	50.8	6.3	8.5	23.2	6.8
-32	41.22			36.5	38.35	0.0	33.4	42.46	0.0	37.2	58.5	50.8	28.8	0.0	23.2
Yht	112.85			100.0	114.98		100.0	114.26		100.0	115.0		100.0	124.4	

Task. d_{80}
(μm)

163

159

124

182



Mintec

min : Huom. : wet sieve 32µm, Ro-Tap 10

SEULA-ANALYYSTITAULUKKO

Table C28. Rod mill griding size distributions in preparation for test work



MMA
Minotec

Huom.: wet sieve 32µm, Ro-Tap 10 min

Näyteli edot BatCircle WP1

Hankk een nimi jaBF BATCircle M WP1 Task 1.2 Geometallurgy (50404-n:o :4015922)

Sieve Size (µm)	SKC-TZ1			SKC-MS2			SKC-TZ2			SKC-BAS1			SKC-BAS2			
	Mass (g)	Cumulative % Passing (%)	Mass on Sieve (%)	Mass (g)	Cumulative % Passing (%)	Mass on Sieve (%)	Mass (g)	Cumulative % Passing (%)	Mass on Sieve (%)	Mass (g)	Cumulative % Passing (%)	Mass on Sieve (%)	Mass (g)	Cumulative % Passing (%)	Mass on Sieve (%)	Mass (g)
150	14.60	69.1	12.4	12.30	73.1	11.0	15.20	69.1	12.9	13.0	72.0	11.3	13.1	72.5	11.6	
125	8.90	61.5	7.6	7.40	66.5	6.6	9.20	61.3	7.8	8.2	64.9	7.1	7.9	65.5	7.0	
90	16.70	47.3	14.2	13.80	54.2	12.3	16.00	47.7	13.6	16.3	50.8	14.1	15.3	51.9	13.6	
75	7.30	41.1	6.2	6.30	48.6	5.6	7.10	41.7	6.0	7.8	44.0	6.8	7.1	45.6	6.3	
45	15.00	28.4	12.7	14.40	35.7	12.9	14.80	29.2	12.5	17.8	28.6	15.4	15.4	32.0	13.6	
32	7.40	22.1	6.3	7.90	28.7	7.1	7.60	22.7	6.4	9.6	20.3	8.3	8.2	24.7	7.3	
-32	26.00	0.0	22.1	32.10	0.0	28.7	26.80	0.0	22.7	23.4	0.0	20.3	27.9	0.0	24.7	
Yht.	117.70		100.0	112.00		100.0	118.00		100.0	115.4		100.0	112.9		100.0	100.0

Task.
 d_{80}
(µm)

169
176
182

171
175
182

122/145

18 APPENDIX D – MAGNETIC SEPARATION DATA

Table D1. Magnetic separation mass balance (Fire Assay)

Ore Type	Product	Mass (g)	Yield (wt%)	Assays				Metal Recovery (%)			
				Pd (mg/kg)	Pt (mg/kg)	Au (mg/kg)	Ag (mg/kg)	Pd	Pt	Au	Ag
PM1	1 Amp	45.5	4.66%	9.82	2.67	0.94	1.70	15.1%	13.7%	23.0%	11.6%
	4 Amp	213.2	21.85%	4.01	1.03	0.32	0.80	28.8%	24.7%	36.7%	25.5%
	6 Amp	17.08	1.75%	3.98	0.95	0.29	0.00	2.3%	1.8%	2.7%	0.0%
	Non-Mag	699.95	71.74%	2.28	0.76	0.10	0.60	53.8%	59.8%	37.6%	62.9%
	Feed*	975.73	100.00%	3.04	0.91	0.19	0.68	100%	100%	100%	100%
PM2	1 Amp	228.88	23.12%	2.25	0.61	0.21	0.90	38.0%	47.5%	43.6%	24.3%
	4 Amp	63.24	6.39%	1.37	0.33	0.15	1.00	6.4%	7.1%	8.6%	7.4%
	6 Amp	53.61	5.42%	1.33	0.21	0.14	1.20	5.3%	3.8%	6.8%	7.6%
	Non-Mag	644.27	65.08%	1.06	0.19	0.07	0.80	50.4%	41.6%	40.9%	60.7%
	Feed*	990	100.00%	1.37	0.30	0.11	0.86	100%	100%	100%	100%
PX2	1 Amp	11.64	1.18%	1.85	0.39	0.69	0	1.0%	0.7%	5.0%	0.0%
	4 Amp	38.14	3.86%	1.92	0.59	0.45	1.00	3.5%	3.5%	10.6%	6.2%
	6 Amp	44.85	4.54%	1.45	0.38	0.44	0.90	3.1%	2.7%	12.2%	6.6%
	Non-Mag	894.19	90.43%	2.18	0.66	0.13	0.60	92.4%	93.0%	72.1%	87.2%
	Feed*	988.82	100.00%	2.13	0.64	0.16	0.62	100%	100%	100%	100%
MS1	1 Amp	13.72	1.39%	7.10	1.43	0.71	5.50	6.1%	4.5%	9.7%	5.2%
	4 Amp	33.31	3.39%	2.51	0.59	0.20	2.80	5.3%	4.5%	6.6%	6.4%
	6 Amp	57.67	5.86%	1.24	0.32	0.09	1.00	4.5%	4.2%	5.2%	4.0%
	Non-Mag	879.22	89.36%	1.52	0.43	0.09	1.40	84.1%	86.8%	78.6%	84.5%
	Feed*	983.92	100.00%	1.61	0.44	0.10	1.48	100%	100%	100%	100%
MS2	1 Amp	11.04	1.11%	8.31	1.65	0.65	11.2	4.0%	3.0%	4.2%	5.4%
	4 Amp	16.64	1.68%	7.72	1.22	0.53	10.6	5.6%	3.3%	5.2%	7.7%
	6 Amp	21.43	2.16%	1.54	0.38	0.14	0	1.4%	1.3%	1.8%	0.0%
	Non-Mag	944.25	95.06%	2.17	0.60	0.16	2.10	89.0%	92.4%	88.8%	86.9%
	Feed*	993.36	100.00%	2.32	0.62	0.17	2.30	100%	100%	100%	100%
TZ1	1 Amp	13.51	1.38%	4.76	1.12	0.58	7.20	4.4%	4.2%	7.0%	6.2%
	4 Amp	22.8	2.33%	9.53	1.25	0.78	5.90	14.8%	7.9%	15.9%	8.6%
	6 Amp	56.31	5.76%	1.80	0.43	0.12	1.80	6.9%	6.7%	6.0%	6.4%
	Non-Mag	884.53	90.52%	1.23	0.33	0.09	1.40	74.0%	81.1%	71.1%	78.8%
	Feed*	977.15	100.00%	1.51	0.37	0.11	1.61	100%	100%	100%	100%
TZ2	1 Amp	28.37	2.94%	5.26	0.90	0.58	3.70	13.6%	9.7%	17.0%	8.2%
	4 Amp	30.21	3.13%	2.26	0.51	0.16	2.40	6.2%	5.9%	5.0%	5.6%
	6 Amp	34.33	3.56%	1.57	0.37	0.16	1.80	4.9%	4.8%	5.7%	4.8%
	Non-Mag	872.01	90.37%	0.95	0.24	0.08	1.20	75.3%	79.6%	72.3%	81.4%
	Feed*	964.92	100.00%	1.14	0.27	0.10	1.33	100%	100%	100%	100%
BAS1	1 Amp	33.11	3.34%	7.89	1.08	0.55	7.50	13.2%	8.6%	13.6%	8.9%
	4 Amp	27.07	2.73%	4.16	0.78	0.32	5.60	5.7%	5.1%	6.4%	5.4%
	6 Amp	26.94	2.72%	2.43	0.54	0.30	5.10	3.3%	3.5%	6.0%	4.9%
	Non-Mag	903.8	91.21%	1.70	0.38	0.11	2.50	77.8%	82.8%	74.0%	80.8%
	Feed*	990.92	100.00%	1.99	0.42	0.14	2.82	100%	100%	100%	100%

*Back-calculated feed.

*Table D2. Semi-quantitative mineralogy (QXRD) of the magnetic products separation products
(Measurement by X-Ray Mineral Services)*

Sample Label	Sample Sub-product	Biotite (%)	Chlorite (%)	Quartz (%)	Amphibole 1 (%)	Plagioclase 1 (%)	Calcite (%)	Dolomite (%)	Magnesite (%)	Magnetite (%)	Talc (%)	Pyrrohotite (%)	Ilmenite (%)	Total (%)
SKC-PM1	Characterization	42.2		26.4			5.0	11.4		26.4				100
	1 Amp	18.9	2.1		18.4			5.9		27.9	21.4			100
	4 Amp	43.3		26.3			7.0			24.4				100
	6 Amp	46.0		27.1			6.3			20.0				100
	Non-Mag	44.8		22.6						26.2				100
SKC-PM2	Characterization		31.3				TR	6.4	12.6	4.3	45.4			100
	1 Amp		22.6				0.6	8.8	13.2	19.9	35.0			100
	4 Amp		23.4				0.5	8.4	32.4	1.6	33.7			100
	6 Amp		34.7				1.4	5.3	31.9	0.0	26.8			100
	Non-Mag		38.3				0.7	5.8	7.4	0.0	47.8			100
SKC-PX2	Characterization		24.5	TR	75.5									100
	1 Amp		19.0	0.7	78.4									100
	4 Amp		23.4	0.6	72.0									100
	6 Amp		25.6	0.5	72.3									100
	Non-Mag		24.8	0.5	74.7									100
SKC-MS1	Characterization	3.0	6.3	8.2	72.2	9.1	1.2					TR		100
	1 Amp	2.2	4.7	3.5	45.4	5.8	2.7					35.9		100
	4 Amp	0.7	6.9	4.3	81.8	3.8	1.6					1.0		100
	6 Amp	3.6	6.5	4.4	80.9	3.5	1.2							100
	Non-Mag	3.1	6.8	8.2	68.9	11.5	1.4							100

TR = Trace Result just under detection limit

*Table D3. Semi-quantitative mineralogy (QXRD) of the magnetic products separation products
(Measurement by X-Ray Mineral Services)*

Sample Label	Sample Sub-product	Biotite (%)	Chlorite (%)	Quartz (%)	Amphibole 1 (%)	Plagioclase 1 (%)	Calcite (%)	Dolomite (%)	Magnesite (%)	Magnetite (%)	Talc (%)	Pyrrhotite (%)	Ilmenite (%)	Total (%)
SKC-MS2	Characterization	13.9	14.0	4.2	22.4	44.5	1.0	TR			TR			100
	1 Amp	2.9	8.4	4.2	20.9	22.5	2.4				38.7			100
	4 Amp	15.6	23.3	2.7	31.3	22.6	1.6				2.9			100
	6 Amp	13.7	24.9	1.4	40.3	19.7	0.0							100
SKM-MS2	Non-Mag	11.1	19.0	3.8	22.2	42.6	1.3							100
	Characterization	4.6	6.7	10.3	37.2	38.1	1.7	1.4			TR			100
	1 Amp	3.9	8.6	5.4	53.5	19.8	3.0	1.6			4.2			100
	4 Amp	3.3	2.8	6.9	26.7	26.7					33.6			100
SKC-TZ1	6 Amp	7.1	10.4	5.0	56.6	18.2	1.6	1.1						100
	Non-Mag	4.5	8.7	10.5	33.0	40.8	1.5	1.0						100
	Characterization	6.4	10.0	7.5	38.1	36.3	1.1	0.6			TR			100
	1 Amp	2.3	12.5	3.1	20.9	20.7	1.8				38.8			100
SKM-TZ1	4 Amp	8.0	12.4	4.5	52.8	16.7	1.6				4.0			100
	6 Amp	9.7	12.5	3.8	55.1	16.9	1.4				0.7			100
	Non-Mag	7.3	12.0	7.6	34.7	36.3	1.4	0.6						100
	Characterization	6.4	10.0	7.5	38.1	36.3	1.1	0.6			TR			100
SKC-TZ2	1 Amp	2.3	12.4	4.5	52.8	16.7	1.6				38.8			100
	4 Amp	8.0	12.5	3.8	55.1	16.9	1.4				4.0			100
	6 Amp	9.7	12.0	7.6	34.7	36.3	1.4				0.7			100
	Non-Mag	7.3	12.0	7.6	34.7	36.3	1.4							100
SKC-BAS1	Characterization	7.0	5.0	5.7	10.1	72.2	TR				TR			100
	1 Amp	7.0	2.6	3.5	7.4	27.6	5.1				46.9			100
	4 Amp	18.6	12.6	4.0	19.8	36.8	3.2				5.1			100
	6 Amp	24.1	19.7	2.2	18.8	33.8	1.4				1.0			100
SKM-BAS1	Non-Mag	4.9	7.2	6.1	9.9	69.7	1.3							100

TR = Trace Result just under detection limit

Table D4. XRF Trace Elements in the magnetic separation products (Measurement by X-Ray Mineral Services)

XRF Trace Elements

Sample	Concentrate	V	Cr	Co	Ni	Cu	Zn	Ga	Ge	As	Se	Rb	Sr	Y	Zr	Nb	Mo	Sn	Sb	Cs	Ba	La	Ce	Nd	Hf	Pb	Th	U
		ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
SKC-PM1	Characterization	65	746	77	1420	1000	80	6	4	ND	3	5	ND	29	ND													
	1 Amp	420	3650	400	2620	3590	90	542	ND	ND	14	56	1230	ND														
	4 Amp	61	711	146	1550	1180	75	42	ND	ND	8	4	ND	33	145	ND	ND	38	69	31	ND	62	103	11	ND	ND	ND	31
	6 Amp																	8	14	ND								
	Non-Mag	59	699	123	1340	965	81	20	ND	ND	4	3	ND	31	41	ND	ND	241	ND									
SKC-PM2	Characterization	66	378	94	2210	1110	88	5	ND	ND	2	11	ND	5	31	ND												
	1 Amp	171	1120	460	2760	1380	105	20	ND	ND	8	42	ND	30	24	ND	ND	85	ND									
	4 Amp	62	357	238	2010	1140	92	43	ND	ND	8	12	ND	6	33	145	ND	225	ND	ND	8	17	7	ND	ND	ND	9	ND
	6 Amp	49	327	194	1950	1260	85	32	ND	ND	5	9	ND	8	30	98	ND	198	ND	ND	7	13	ND	ND	ND	7	ND	ND
	Non-Mag	37	250	151	2000	1040	94	9	ND	ND	ND	ND	ND	30	14	ND	ND	148	ND	ND	ND	11	ND	8	ND	ND	ND	
SKC-PX2	Characterization	92	1250	89	956	363	114	7	ND	ND	5	ND	9	44	ND													
	1 Amp	206	1270	178	988	419	114	36	ND	ND	4	6	ND	9	45	127	ND	175	ND									
	4 Amp	133	1290	173	967	379	116	38	ND	ND	4	5	ND	10	46	123	ND	197	ND									
	6 Amp																	28	ND	166	ND							
	Non-Mag	94	1210	158	960	418	111	16	ND	ND	4	ND	11	46	28	ND												
SKC-MS1	Characterization	117	850	53	1220	1740	113	11	3	ND	4	12	114	11	41	ND												
	1 Amp	141	1060	113	1180	1240	123	12	ND	ND	3	13	42	13	39	7	ND	10	ND									
	4 Amp																	ND										
	6 Amp																	12	ND									
	Non-Mag	116	825	80	1080	1240	113	11	2	ND	ND	3	11	129	12	41	ND											
SKC-MS2	Characterization	38	449	ND	1190	1870	87	17	ND	ND	4	39	465	ND	43	ND												
	1 Amp	51	460	24	1210	1980	90	17	ND	ND	3	35	467	ND	44	ND												
	4 Amp																											
	6 Amp																											
	Non-Mag																											
SKC-TZ1	Characterization	81	347	ND	1060	2110	85	20	2	ND	3	12	489	7	42	ND												
	1 Amp	106	637	93	1380	2000	132	22	2	ND	3	17	274	8	37	ND	ND	18	ND									
	4 Amp																	6	ND									
	6 Amp																	518	ND									
	Non-Mag	77	336	21	876	2080	88	20	ND	ND	3	11	518	6	43	ND	ND	5	ND									
SKC-TZ2	Characterization	85	601	25	810	1440	80	16	ND	ND	3	26	479	ND	37	ND												
	1 Amp																											
	4 Amp																											
	6 Amp																											
	Non-Mag																											
SKC-BAS1	Characterization	38	262	10	1480	1610	400	29	ND	9	4	48	649	ND	45	ND												
	1 Amp																											
	4 Amp																											
	6 Amp																											
	Non-Mag	ND	235	28	1140	1460	385	28	ND	9	ND	46	670	ND	49	ND	ND	4	ND									

ND = Not determined (abundance < LOD (limit of detection), see below)

Note:

Element	V	Cr	Co	Ni	Cu	Zn	Ga	Ge	As	Se	Rb	Sr	Y	Zr	Nb	Mo	Sn	Sb	Cs	Ba	La	Ce	Nd	Hf	Pb	Th	U
LOD	37	39	10	11	10	7	3	2	5	2	3	30	5	5	20	2	3	56	6	9	2	4	5	2	2	2	

Table D5. ICP measurement data for magnetic separation products (Measurement by X-Ray Mineral Services)

Oxide Element	Feed	PM1						PM2						PX2					
		Sample	1 Amp	4 Amp	6 Amp	Non-Mag		Sample	1 Amp	4 Amp	6 Amp	Non-Mag		Sample	1 Amp	4 Amp	6 Amp	Non-Mag	
Al2O3	%	6.3	4.4	6.0	6.3	6.3		4.4	4.0	4.5	4.7	4.7		5.5	3.9	4.7	5.0	4.9	
SiO2	%	44.4	30.5	41.8	40.0	42.2		40.6	28.6	27.6	27.2	43.1		49.7	47.6	47.6	49.0	47.8	
TiO2	%	0.1	0.5	0.2	0.6	0.1		0.1	0.2	0.4	0.2	0.0		0.2	1.0	2.2	0.9	0.1	
Fe2O3	%	12.5	34.3	11.9	14.5	10.9		14.9	29.8	14.5	13.3	8.9		12.3	13.1	13.3	12.9	12.0	
MnO	%	0.2	0.2	0.2	0.2	0.2		0.2	0.2	0.3	0.4	0.1		0.3	0.5	0.6	0.5	0.3	
MgO	%	24.3	16.9	23.4	23.0	23.5		27.2	20.8	25.8	27.6	26.6		20.0	17.7	18.0	18.9	19.1	
CaO	%	5.4	4.0	5.4	5.2	5.3		2.2	2.2	2.4	2.4	2.1		9.2	9.1	8.8	9.0	8.7	
Na2O	%	0.0	0.1	0.1	0.0	0.0		0.1	0.0	0.0	0.0	0.0		0.1	0.1	0.1	0.1	0.1	
K2O	%	0.0	0.0	0.0	0.0	0.0		0.0	0.0	0.0	0.0	0.0		0.0	0.0	0.0	0.0	0.0	
P2O5	%	0.0	0.0	0.0	0.0	0.0		0.0	0.0	0.0	0.0	0.0		0.0	0.0	0.0	0.0	0.0	
Cl	%	na	na	na	na	na		na	na	na	na	na		na	na	na	na	na	
S	ppm	306	2573	736	1439	294		460	1970	1540	651	326		64	323	184	179	223	
As	ppm	na	na	na	na	na		na	na	na	na	na		na	na	na	na	na	
Ba	ppm	282	25	27	46	29		108	58	45	16	17		367	42	33	42	32	
Ce	ppm	2	84	15	23	6		3	6	14	11	3		12	13	13	12	10	
Co	ppm	106	139	120	104	123		130	133	122	130	121		72	86	84	84	85	
Cr	ppm	603	1494	583	625	593		193	404	167	144	145		1123	1508	1190	1210	1151	
Cs	ppm	0	1	0	0	0		0	0	0	0	0		0	0	0	0	0	
Cu	ppm	908	1545	1058	943	808		855	1013	1011	1144	1069		451	213	366	332	363	
Ga	ppm	6	5	5	6	6		5	4	4	4	4		7	5	6	6	6	
Ge	ppm	na	na	na	na	na		na	na	na	na	na		na	na	na	na	na	
Hf	ppm	0	9	1	4	1		0	1	2	1	0		1	3	2	2	2	
La	ppm	1	43	7	11	3		1	3	7	5	2		5	6	8	6	5	
Mo	ppm	0	36	2	6	1		0	6	3	1	1		0	63	9	5	1	
Nb	ppm	1	1358	155	508	36		1	36	121	112	18		2	191	133	117	25	
Nd	ppm	2	54	7	12	3		2	3	6	5	2		6	16	8	7	6	
Ni	ppm	1177	1631	1451	1347	1255		1383	2178	1969	1876	2021		926	1126	910	902	907	
Pb	ppm	1	3	8	2	2		0	1	1	1	2		1	2	2	2	2	
Rb	ppm	1	3	2	1	1		1	1	1	1	1		1	2	2	1	1	
Sb	ppm	na	na	na	na	na		na	na	na	na	na		na	na	na	na	na	
Se	ppm	na	na	na	na	na		na	na	na	na	na		na	na	na	na	na	
Sn	ppm	0	4753	589	1190	193		0	120	224	192	131		0	514	229	250	198	
Sr	ppm	17	13	10	12	11		12	10	14	10	9		19	14	9	12	9	
Th	ppm	0	9	1	2	1		0	0	2	1	0		2	3	2	2	2	
U	ppm	0	16	2	4	1		0	0	1	1	1		1	4	2	2	2	
V	ppm	63	251	52	53	57		57	159	34	24	23		112	186	120	111	103	
Y	ppm	3	23	3	8	3		5	3	9	7	2		10	12	12	12	12	
Zn	ppm	76	134	81	110	85		81	98	95	90	95		106	116	110	117	114	
Zr	ppm	16	84	16	45	11		14	23	47	13	10		36	76	69	58	62	

Note:
Some of the major elements totals are less than 100% because sulphur in the samples may be lost in part during fusion for ICP analysis and is therefore not measured

Table D6. ICP measurement data for magnetic separation products (Measurement by X-Ray Mineral Services)

Oxide/Element	%	MS1					MS2					TZ1				
		Feed Sample	1 Amp	4 Amp	6 Amp	Non-Mag	Feed Sample	1 Amp	4 Amp	6 Amp	Non-Mag	Feed Sample	1 Amp	4 Amp	6 Amp	Non-Mag
Al2O3	7.2	4.5	6.2	6.0	7.8	17.3	10.9	18.1	14.3	16.0	12.3	na	13.2	16.6	16.6	
SiO2	54.7	33.5	52.4	53.7	55.8	54.4	34.7	52.1	46.8	54.5	47.8	na	50.4	54.9	54.9	
TiO2	0.3	0.2	0.5	0.4	0.3	0.1	0.1	0.2	0.3	0.3	0.2	0.4	na	0.3	0.2	
Fe2O3	11.0	30.3	13.5	12.3	10.6	6.6	26.1	6.5	12.1	11.6	7.9	14.4	na	11.5	7.2	
MnO	0.2	0.2	0.3	0.3	0.2	0.1	0.2	0.1	0.2	0.2	0.1	0.2	na	0.2	0.1	
MgO	14.1	8.0	14.7	15.3	13.4	7.8	5.6	8.9	12.7	14.4	6.9	10.0	na	11.4	7.1	
CaO	9.8	5.9	10.2	10.4	9.4	6.7	5.4	6.5	6.6	6.3	8.5	9.3	na	9.5	8.2	
Na2O	1.1	0.7	0.6	0.6	1.2	3.4	1.8	3.6	1.6	1.3	2.9	1.6	na	1.5	3.1	
K2O	0.3	0.2	0.3	0.3	0.4	1.2	0.6	1.1	1.7	1.7	0.6	0.5	na	0.7	0.6	
P2O5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	na	0.0	0.0	
Cl	na	na	na	na	na	na	na	na	na	na	na	na	na	na	na	
S	464	4730	648	363	930	289	3983	359	952	491	447	1200	na	574	536	
As	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	
Ba	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	
Ce	12	9	9	9	12	10	10	10	9	8	10	11	na	10	10	
Co	69	76	109	82	77	51	39	48	86	83	55	112	na	70	45	
Cr	806	3975	1359	1070	824	458	9195	495	2175	1501	318	1812	na	615	424	
Cs	0	0	0	0	0	2	1	2	2	2	1	1	na	1	1	
Cu	1235	815	1147	1003	1742	1212	911	1517	1639	1553	1259	2224	na	1448	1506	
Ga	10	7	10	10	10	16	13	16	18	19	19	19	na	20	19	
Ge	na	na	na	na	na	na	na	na	na	na	na	na	na	na	na	
Hf	1	2	1	1	1	1	2	1	2	2	1	1	3	na	1	
La	5	4	4	4	5	5	5	5	5	3	5	5	6	na	5	
Mo	1	441	59	10	3	1	919	4	185	87	0	171	na	14	16	
Nb	1	9	14	8	3	1	3	1	6	5	1	15	na	4	2	
Nd	8	13	7	7	9	5	10	5	6	5	5	11	na	7	7	
Ni	1005	606	1180	1026	1145	1003	507	1098	1722	1609	843	1891	na	1229	899	
Pb	5	23	2	3	9	11	8	11	6	5	9	7	na	7	10	
Rb	10	6	10	11	11	43	22	40	62	67	13	11	na	15	13	
Sb	na	na	na	na	na	na	na	na	na	na	na	na	na	na	na	
Se	na	na	na	na	na	na	na	na	na	na	na	na	na	na	na	
Sn	1	12	13	8	13	0	6	2	9	10	1	26	na	18	3	
Sr	122	70	55	46	128	418	319	448	254	203	468	300	na	270	487	
Th	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
U	ppm	1	0	1	1	0	0	0	0	0	0	0	na	0	0	
V	ppm	143	82	158	164	135	54	34	55	86	85	121	na	136	85	
Y	ppm	12	11	12	12	3	8	3	8	8	6	14	na	8	14	
Zn	ppm	106	157	131	123	115	79	125	79	131	130	75	184	na	121	80
Zr	ppm	29	83	27	26	28	20	114	27	102	82	24	128	na	32	164

Note:
Some of the major elements totals are less than 100% because sulphur in the samples may be lost in part during fusion for ICP analysis and is therefore not measured

Table D7. ICP measurement data for magnetic separation products (Measurement by X-Ray Mineral Services)

	BAS1						
	Feed Sample	1 Amp	4 Amp	6 Amp	4 Amp	6 Amp	Non-Mag
Al2O3	16.7	8.4	12.6	12.6	16.9	17.7	20.3
SiO2	51.8	29.0	47.6	48.6	51.8	48.0	55.0
TiO2	0.2	0.1	0.3	0.3	0.2	0.1	0.2
Fe2O3	8.1	32.6	12.2	10.2	6.6	38.4	12.0
MnO	0.1	0.2	0.2	0.2	0.1	0.1	0.1
MgO	9.1	5.1	12.0	13.3	9.2	4.6	4.6
CaO	%	8.6	5.2	8.9	9.4	9.0	5.3
Na2O	%	2.3	1.3	1.4	1.4	2.4	5.0
K2O	%	0.8	0.4	0.9	0.9	0.8	1.5
P2O5	%	0.0	0.0	0.0	0.0	0.0	0.0
Cl	%	na	na	na	na	na	na
S	ppm	505	4351	1223	511	747	611
As	ppm	na	na	na	na	na	na
Ba	ppm	265	131	301	278	352	497
Ce	ppm	8	6	10	8	9	15
Co	ppm	55	65	86	80	45	71
Cr	ppm	599	4115	1475	1073	617	229
Cs	ppm	1	1	1	1	2	1
Cu	ppm	886	593	1099	1168	1203	969
Ga	ppm	17	10	15	16	16	24
Ge	ppm	na	na	na	na	na	na
Hf	ppm	0	3	3	1	3	1
La	ppm	4	2	5	4	5	8
Mo	ppm	1	672	92	31	4	1
Nb	ppm	1	12	54	86	6	1
Nd	ppm	4	22	10	8	5	7
Ni	ppm	587	324	1454	1120	714	898
Pb	ppm	6	3	12	15	11	32
Rb	ppm	28	14	30	31	28	51
Sb	ppm	na	na	na	na	na	24
Se	ppm	na	na	na	na	na	na
Sn	ppm	0	10	40	68	13	0
Sr	ppm	484	249	277	258	490	623
Th	ppm	1	1	1	1	1	1
U	ppm	0	0	1	0	0	0
V	ppm	84	47	107	117	84	37
Y	ppm	4	7	12	5	11	3
Zn	ppm	69	120	112	110	78	454
Zr	ppm	19	94	119	23	136	23

	TZ2				
	Feed Sample	1 Amp	4 Amp	6 Amp	
Oxide/Element					
Al2O3	%	16.7	8.4	12.6	12.6
SiO2	%	51.8	29.0	47.6	48.6
TiO2	%	0.2	0.1	0.3	0.2
Fe2O3	%	8.1	32.6	12.2	10.2
MnO	%	0.1	0.2	0.2	0.1
MgO	%	9.1	5.1	12.0	13.3
CaO	%	8.6	5.2	8.9	9.4
Na2O	%	2.3	1.3	1.4	1.4
K2O	%	0.8	0.4	0.9	0.9
P2O5	%	0.0	0.0	0.0	0.0
Cl	ppm	na	na	na	na
S	ppm	na	na	na	na
As	ppm	na	na	na	na
Ba	ppm	265	131	301	278
Ce	ppm	8	6	10	8
Co	ppm	55	65	86	80
Cr	ppm	599	4115	1475	1073
Cs	ppm	1	1	1	1
Cu	ppm	886	593	1099	1168
Ga	ppm	17	10	15	16
Ge	ppm	na	na	na	na
Hf	ppm	0	3	3	1
La	ppm	4	2	5	4
Mo	ppm	1	672	92	31
Nb	ppm	1	12	54	86
Nd	ppm	4	22	10	8
Ni	ppm	587	324	1454	1120
Pb	ppm	6	3	12	15
Rb	ppm	28	14	30	31
Sb	ppm	na	na	na	na
Se	ppm	na	na	na	na
Sn	ppm	0	10	40	68
Sr	ppm	484	249	277	258
Th	ppm	1	1	1	1
U	ppm	0	0	1	0
V	ppm	84	47	107	117
Y	ppm	4	7	12	5
Zn	ppm	69	120	112	110
Zr	ppm	19	94	119	23

	BAS1				
	Feed Sample	1 Amp	4 Amp	6 Amp	
Oxide/Element					
Al2O3	%	16.7	8.4	12.6	12.6
SiO2	%	51.8	29.0	47.6	48.6
TiO2	%	0.2	0.1	0.3	0.2
Fe2O3	%	8.1	32.6	12.2	10.2
MnO	%	0.1	0.2	0.2	0.1
MgO	%	9.1	5.1	12.0	13.3
CaO	%	8.6	5.2	8.9	9.4
Na2O	%	2.3	1.3	1.4	1.4
K2O	%	0.8	0.4	0.9	0.9
P2O5	%	0.0	0.0	0.0	0.0
Cl	ppm	na	na	na	na
S	ppm	na	na	na	na
As	ppm	na	na	na	na
Ba	ppm	265	131	301	278
Ce	ppm	8	6	10	8
Co	ppm	55	65	86	80
Cr	ppm	599	4115	1475	1073
Cs	ppm	1	1	1	1
Cu	ppm	886	593	1099	1168
Ga	ppm	17	10	15	16
Ge	ppm	na	na	na	na
Hf	ppm	0	3	3	1
La	ppm	4	2	5	4
Mo	ppm	1	672	92	31
Nb	ppm	1	12	54	86
Nd	ppm	4	22	10	8
Ni	ppm	587	324	1454	1120
Pb	ppm	6	3	12	15
Rb	ppm	28	14	30	31
Sb	ppm	na	na	na	na
Se	ppm	na	na	na	na
Sn	ppm	0	10	40	68
Sr	ppm	484	249	277	258
Th	ppm	1	1	1	1
U	ppm	0	0	1	0
V	ppm	84	47	107	117
Y	ppm	4	7	12	5
Zn	ppm	69	120	112	110
Zr	ppm	19	94	119	23

Note:

Some of the major elements totals are less than 100% because sulphur in the samples may be lost in part during fusion for ICP analysis and is therefore not measured

19 APPENDIX E – GRAVITY SEPARATION DATA

Table E1. Gravity separation mass balance (Fire Assay)

Ore Type	Product	Mass (g)	Yield (wt%)	Assays				Metal Recovery (%)			
				Pd (mg/kg)	Pt (mg/kg)	Au (mg/kg)	Ag (mg/kg)	Pd	Pt	Au	Ag
PM1	Concentrate	80	2.93%	28.70	8.41	2.75	3.91	31.3%	26.0%	40.9%	9.7%
	Middlings	1115	40.77%	2.07	0.81	0.12	1.02	31.5%	34.7%	24.7%	35.3%
	Reject Tails	1540	56.31%	1.77	0.66	0.12	1.15	37.2%	39.2%	34.4%	55.0%
	Feed*	2735	100.00%	2.68	0.94	0.20	1.18	100%	100%	100%	100%
PM2	Concentrate	60	2.25%	15.90	3.19	2.52	3.37	22.7%	19.8%	34.7%	6.7%
	Middlings	910	34.08%	1.49	0.37	0.14	1.10	32.3%	35.0%	29.7%	32.9%
	Reject Tails	1700	63.67%	1.11	0.26	0.09	1.08	45.0%	45.2%	35.6%	60.4%
	Feed*	2670	100.00%	1.57	0.36	0.16	1.14	100%	100%	100%	100%
PX1	Concentrate	55	1.86%	25.9	5.23	3.73	7.04	24.3%	17.1%	41.6%	10.3%
	Middlings	805	27.24%	1.47	0.472	0.131	0.91	20.2%	22.6%	21.4%	19.3%
	Reject Tails	2095	70.90%	1.55	0.485	0.087	1.27	55.5%	60.3%	37.0%	70.5%
	Feed*	2955	100.00%	1.98	0.57	0.17	1.28	100%	100%	100%	100%
PX2	Concentrate	70	2.50%	10	2.67	2.73	6.07	12.4%	11.9%	29.2%	12.0%
	Middlings	780	27.86%	1.82	0.517	0.273	1.10	25.1%	25.7%	32.6%	24.2%
	Reject Tails	1950	69.64%	1.81	0.503	0.128	1.16	62.5%	62.4%	38.2%	63.8%
	Feed*	2800	100.00%	2.02	0.56	0.23	1.27	100%	100%	100%	100%
MS1	Concentrate	45	1.43%	15.30	1.59	2.44	4.73	12.7%	5.4%	24.2%	4.3%
	Middlings	1050	33.33%	1.30	0.35	0.11	1.40	25.2%	27.5%	24.3%	29.6%
	Reject Tails	2055	65.24%	1.64	0.43	0.11	1.60	62.1%	67.0%	51.6%	66.1%
	Feed*	3150	100.00%	1.72	0.42	0.14	1.58	100%	100%	100%	100%

* Back-calculated feed.

Table E2. Gravity separation mass balance (Fire Assay)

Ore Type	Product	Mass (g)	Yield (wt%)	Assays			Metal Recovery (%)				
				Pd (mg/kg)	Pt (mg/kg)	Au (mg/kg)	Ag (mg/kg)	Pd	Pt	Au	Ag
MS2	Concentrate	85	2.87%	18.6	2.33	2.22	12	23.7%	10.0%	34.7%	15.1%
	Middlings	1090	36.76%	1.57	0.489	0.114	1.76	25.6%	27.0%	22.8%	28.3%
	Reject Tails	1790	60.37%	1.89	0.695	0.129	2.14	50.7%	63.0%	42.5%	56.6%
	Feed*	2965	100.00%	2.25	0.67	0.18	2.28	100%	100%	100%	100%
TZ1	Concentrate	25	0.82%	22.00	1.55	1.18	7.14	10.9%	3.1%	6.7%	3.1%
	Middlings	730	23.86%	1.45	0.37	0.13	1.81	20.9%	21.2%	21.7%	22.8%
	Reject Tails	2305	75.33%	1.50	0.41	0.14	1.86	68.2%	75.7%	71.6%	74.1%
	Feed*	3060	100.00%	1.66	0.41	0.14	1.89	100%	100%	100%	100%
TZ2	Concentrate	40	1.34%	16.70	1.57	0.78	5.21	17.3%	6.0%	11.0%	4.2%
	Middlings	1020	34.06%	0.81	0.26	0.09	1.39	21.5%	25.7%	30.7%	28.5%
	Reject Tails	1935	64.61%	1.22	0.37	0.09	1.73	61.2%	68.3%	58.3%	67.3%
	Feed*	2995	100.00%	1.29	0.35	0.10	1.66	100%	100%	100%	100%
BAS1	Concentrate	50	1.92%	19.40	2.03	2.93	7.69	18.3%	9.3%	30.6%	5.7%
	Middlings	775	29.69%	1.65	0.36	0.13	2.28	24.1%	25.4%	21.4%	26.2%
	Reject Tails	1785	68.39%	1.71	0.40	0.13	2.57	57.6%	65.3%	48.1%	68.1%
	Feed*	2610	100.00%	2.03	0.42	0.18	2.58	100%	100%	100%	100%
BAS2	Concentrate	60	2.00%	14.30	2.32	1.39	7.29	12.5%	8.0%	14.5%	7.9%
	Middlings	920	30.67%	1.86	0.54	0.22	1.67	25.0%	28.5%	34.9%	27.9%
	Reject Tails	2020	67.33%	2.12	0.55	0.14	1.75	62.5%	63.5%	50.6%	64.2%
	Feed*	3000	100.00%	2.28	0.58	0.19	1.84	100%	100%	100%	100%

* Back-calculated feed.

*Table E3. Semi-quantitative mineralogy (QXRD) of the gravity products separation products
(Measurement by X-Ray Mineral Services)*

Sample Label	Sample Sub-product	Biotite (%)	Chlorite (%)	Quartz (%)	Amphibole 1 (%)	Plagioclase 1 (%)	Calcite (%)	Dolomite (%)	Magnesite (%)	Magnetite (%)	Talc (%)	Pyrrohotite (%)	Pyrite (%)	Chalcopyrite (%)	Pentlandite (%)	Total (%)
SKC-PM1	Feed Sample Concentrate	42.2	TR	26.4			5.0	11.6		TR	26.4					100
	Middlings	17.9	1.2	23.2				6.7		TR	18.0	7.2				100
	Reject Tails	44.4	TR	24.8				6.5			24.1					100
		43.2	TR	24.3							26.0					100
SKC-PM2	Feed Sample Concentrate	31.3	TR				TR	6.4	12.6	4.3	45.4					100
	Middlings	22.2	TR				TR	7.1	18.2	39.1	6.4	5.2			TR	100
	Reject Tails	38.8	TR				TR	7.3	16.5	4.9	32.5				TR	100
		43.8	TR				TR	7.4	12.8	2.8	33.2					100
SKC-PX1	Feed Sample Concentrate	34.2	TR	57.0			TR									100
	Middlings	15.5	0.6	54.8												96
	Reject Tails	30.1	TR	62.7											TR	100
		41.0	TR	51.6												100
SKC-PX2	Feed Sample Concentrate	24.5	TR	75.5												100
	Middlings	15.4		82.8												100
	Reject Tails	21.9		78.1												100
		27.0		73.0												100
SKC-MS1	Feed Sample Concentrate	3.0	6.3	8.2	72.2	9.1	1.2									100
	Middlings	TR	5.0	4.5	61.2	3.0	1.3									100
	Reject Tails	3.1	5.9	6.9	72.6	10.5	1.0									100
		3.9	6.7	7.7	69.4	11.1	1.3									100

TR = Trace Result just under detection limit

*Table E4. Semi-quantitative mineralogy (QXRD) of the gravity products separation products
(Measurement by X-Ray Mineral Services)*

Sample Label	Sample Sub-product	Biotite (%)	Chlorite (%)	Quartz (%)	Amphibole 1 (%)	Plagioclase 1 (%)	Calcite (%)	Dolomite (%)	Magnesite (%)	Magnetite (%)	Talc (%)	Pyrrophyte (%)	Pyrite (%)	Chalcopyrite (%)	Pentlandite (%)	Total (%)
SKC-MS2	Feed Sample	13.9	14.0	4.2	22.4	44.5	1.0	TR								100
	Concentrate	9.4	10.7	3.9	34.9	27.6	1.1									100
SKG-MS2	Middlings	16.5	13.3	3.9	22.6	42.7	1.0									100
	Reject Tails	10.6	16.7	4.0	23.8	44.0	1.1									100
SKC-TZ1	Feed Sample	4.6	6.7	10.3	37.2	38.1	1.7	1.4								100
	Concentrate	3.1	4.8	5.3	38.4	17.2	TR	TR								100
SKG-TZ1	Middlings	3.8	6.1	8.0	40.7	38.6	1.4	1.4								100
	Reject Tails	4.7	8.9	10.3	32.7	40.5	1.6	1.2								100
SKC-TZ2	Feed Sample	6.4	10.0	7.5	38.1	36.3	1.1	0.6								100
	Concentrate	TR	6.1	4.0	41.2	16.8	TR	TR								100
SKG-TZ2	Middlings	8.7	9.1	6.3	39.0	35.9	1.0	TR								100
	Reject Tails	6.2	13.7	6.5	36.4	35.9	1.2	TR								100
SKC-BAS1	Feed Sample	7.0	5.0	5.7	10.1	72.2	TR	TR								100
	Concentrate	8.9	10.1	4.9	11.1	36.1	2.8	TR								100
SKG-BAS1	Middlings	25.4	8.4	6.0	8.9	50.1	1.2	TR								100
	Reject Tails	15.6	9.2	6.1	7.1	60.9	1.3	TR								100
SKC-BAS2	Feed Sample	4.7	8.9	5.4	41.1	38.9	0.6	0.5								100
	Concentrate	TR	4.0	2.8	61.1	22.2	TR									100
SKG-BAS2	Middlings	5.5	8.2	4.1	47.5	34.8	TR									100
	Reject Tails	3.0	14.2	5.2	37.1	39.0	TR									100

TR = Trace Result just under detection limit

Table E5. XRF Trace Elements in the gravity separation products (Measurement by X-Ray Mineral Services)

XRF Trace Elements		Client: Geological Survey of Finland																										
Sample	Concentrate	V	Cr	Co	Ni	Cu	Zn	Ga	Ge	As	Se	Rb	Sr	Y	Zr	Nb	Mo	Sn	Sb	Cs	Ba	La	Ce	Nd	Hf	Pb	Th	U
SKC-PM1	Feed Sample	65	746	77	1420	1000	80	6	4	ND	3	5	ND	29	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND		
SKC-PM1	Concentrate	473	2810	2280	12200	9490	239	59	ND	86	39	44	68	41	80	ND	320	3	ND	24	46	257	ND	535	112	26		
SKC-PM1	Middleings	61	742	153	1230	933	98	6	ND	6	3	5	ND	29	ND	ND	5	ND	7	9	30	ND	12	2	ND	ND		
SKC-PM1	Reject Tails	61	740	160	1250	850	91	6	ND	7	3	4	ND	ND	28	ND	ND	5	ND	ND	ND	28	5	13	2	ND		
SKC-PM2	Feed Sample	66	378	94	2210	1110	88	5	ND	ND	2	11	ND	5	31	ND	ND	ND	ND	ND	ND	ND	ND	ND	5	ND		
SKC-PM2	Concentrate	7080	10500	1460	12900	5980	156	41	ND	ND	23	28	34	ND	40	51	ND	242	ND	ND	11	16	22	ND	397	375	66	
SKC-PM2	Middlelings	60	449	80	2300	1390	98	6	ND	ND	3	15	ND	5	31	ND	ND	8	ND	ND	ND	12	12	ND	11	8	ND	ND
SKC-PM2	Reject Tails	51	327	30	1900	893	92	5	2	ND	2	6	ND	ND	28	ND	ND	6	ND	ND	ND	ND	ND	ND	6	ND	ND	
SKC-PX1	Feed Sample	70	914	80	1260	1040	89	6	ND	ND	3	3	ND	6	32	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	5	ND	
SKC-PX1	Concentrate	93	1370	410	8730	15400	213	20	ND	ND	27	44	20	ND	6	58	32	ND	152	ND	ND	14	28	85	ND	61	12	5
SKC-PX1	Middlelings	63	809	47	1020	692	79	5	ND	ND	3	ND	ND	6	32	ND	ND	3	ND	ND	ND	ND	ND	ND	ND	ND	ND	
SKC-PX1	Reject Tails	69	977	42	1240	915	92	7	FeNi	ND	2	3	ND	ND	31	ND	ND	3	ND	ND	ND	ND	ND	ND	ND	ND		
SKC-PX2	Feed Sample	92	1250	89	956	363	114	7	ND	ND	5	ND	9	14	44	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	
SKC-PX2	Concentrate	194	942	ND	947	5710	150	17	ND	ND	22	11	7	ND	14	90	66	ND	182	ND	ND	12	27	59	ND	22	2	ND
SKC-PX2	Middlelings	95	1180	71	999	303	108	6	ND	ND	2	ND	4	ND	11	43	ND	3	ND	ND	ND	6	11	ND	ND	ND	ND	ND
SKC-PX2	Reject Tails	87	1230	70	1030	454	111	7	ND	ND	5	ND	5	ND	9	42	ND	ND	5	ND	ND	ND	ND	12	ND	ND	ND	ND
SKC-MS1	Feed Sample	117	850	53	1220	1740	113	11	3	ND	4	12	114	11	41	ND	ND	ND	ND	ND	ND	110	8	16	ND	11	ND	ND
SKC-MS1	Concentrate	87	1310	923	11500	7530	217	43	ND	40	53	23	57	12	44	73	ND	386	ND	ND	10	19	66	ND	141	13	6	
SKC-MS1	Middlelings	122	936	33	1060	1350	114	10	3	ND	4	11	100	13	41	ND	6	ND	99	ND	13	ND	12	ND	ND	ND	ND	
SKC-MS1	Reject Tails	113	938	17	1250	2030	121	12	3	ND	4	12	115	10	40	ND	ND	8	ND	ND	114	6	14	ND	ND	14	ND	

ND = Not determined (abundance < LOD (limit of detection), see below)

Note:

Element	V	Cr	Co	Ni	Cu	Zn	Ga	Ge	As	Se	Rb	Sr	Y	Zr	Nb	Mo	Sn	Sb	Cs	Ba	La	Ce	Nd	Hf	Pb	Th	U
LOD	37	39	10	11	10	7	3	2	5	2	3	30	5	5	5	20	2	3	56	6	9	2	4	5	2	2	

Table E6. XRF Trace Elements in the gravity separation products (Measurement by X-Ray Mineral Services)

XRF Trace Elements																															
Sample	Concentrate	V	Cr	Co	Ni	Cu	Zn	Ga	Ge	As	Se	Rb	Sr	Y	Zr	Nb	Mo	Sn	Sb	Cs	Ba	La	Ce	Nd	Hf	Pb	Th	U			
		ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm			
SKC-MMS2	Feed Sample	38	449	ND	1190	1870	87	17	ND	4	39	465	ND	43	ND	ND	ND	ND	17	ND	ND	24	ND	ND	ND	ND	ND	ND			
SKG-MMS2	Concentrate	66	811	ND	8890	13800	190	45	ND	13	41	38	455	ND	61	ND	281	ND	ND	241	12	22	38	ND	50	ND	ND	ND	ND		
	Middlings	42	491	ND	937	1320	84	18	ND	ND	2	49	467	ND	45	ND	4	ND	425	ND	16	ND	25	ND	ND	ND	ND	ND	ND		
	Reject Tails	41	553	ND	1180	2040	89	18	ND	ND	3	34	465	ND	46	ND	13	ND	323	7	17	12	ND	28	ND	ND	ND	ND	ND		
SKC-TZ1	Feed Sample	81	347	ND	1060	2110	85	20	2	ND	3	12	489	7	42	ND	ND	ND	ND	ND	13	ND	ND	19	ND	ND	ND	ND	ND	ND	
SKG-TZ1	Concentrate	84	427	ND	977	1850	81	20	ND	ND	4	11	496	7	40	ND	ND	7	ND	ND	212	ND	9	ND	ND	20	ND	ND	ND	ND	ND
	Middlings	60	475	ND	1040	2220	85	21	ND	ND	3	12	517	6	44	ND	ND	8	ND	ND	236	ND	14	9	ND	21	ND	ND	ND	ND	ND
SKC-TZ2	Feed Sample	85	601	25	810	1440	80	16	ND	ND	3	26	479	ND	37	ND	ND	ND	ND	206	ND	10	ND	15	ND	ND	ND	ND	ND	ND	
SKG-TZ2	Concentrate	48	1560	1460	14900	10700	174	66	ND	10	52	40	257	ND	43	ND	41	539	ND	ND	77	9	18	33	ND	96	26	10	ND	ND	
	Middlings	70	640	ND	575	993	67	17	ND	ND	3	29	485	ND	39	ND	4	ND	ND	223	6	15	ND	ND	16	ND	ND	ND	ND	ND	
	Reject Tails	76	737	ND	771	1440	77	18	ND	ND	2	24	476	6	39	ND	ND	6	ND	ND	191	ND	ND	12	ND	28	ND	ND	ND	ND	ND
SKC-BAS1	Feed Sample	38	262	10	1490	1610	400	29	ND	9	4	48	649	ND	45	ND	ND	ND	ND	446	6	22	6	ND	106	ND	ND	ND	ND	ND	
SKG-BAS1	Concentrate	ND	774	1490	17600	6530	1720	84	ND	79	42	51	535	ND	55	ND	30	516	5	ND	265	10	22	34	ND	405	11	10	ND	ND	
	Middlings	ND	369	ND	1300	1320	377	30	ND	7	4	61	616	ND	45	ND	7	ND	ND	541	11	24	11	ND	84	ND	ND	ND	ND	ND	
	Reject Tails	ND	402	ND	1300	1550	367	28	ND	9	3	43	646	ND	50	ND	7	ND	ND	425	ND	12	ND	ND	90	ND	ND	ND	ND	ND	
SKC-BAS2	Feed Sample	117	484	ND	517	4870	128	19	Mo	ND	4	19	375	7	30	ND	ND	ND	ND	ND	183	ND	6	ND	13	ND	ND	ND	ND	ND	ND
SKG-BAS2	Concentrate	138	979	ND	2860	30500	398	42	Zn	20	34	24	194	8	35	43	ND	205	ND	83	13	20	74	ND	96	6	2	ND	ND	ND	
	Middlings	116	538	ND	475	3460	114	20	ND	ND	3	22	356	9	33	ND	4	ND	ND	216	6	11	ND	13	ND	22	ND	ND	ND	ND	
	Reject Tails	100	647	ND	590	4690	130	20	ND	ND	4	17	383	8	33	ND	5	ND	ND	171	ND	14	15	ND	14	ND	ND	ND	ND	ND	

ND = Not determined (abundance < LOD (limit of detection), see below)

Note:

Element	V	Cr	Co	Ni	Cu	Zn	Ga	Ge	As	Se	Rb	Sr	Y	Zr	Nb	Mo	Sn	Sb	Cs	Ba	La	Ce	Nd	Hf	Pb	Th	U
LOD	37	39	10	11	10	7	3	2	5	2	3	30	5	5	5	20	2	2	3	56	6	9	2	4	5	2	2

Table E7. Gravity separation products chemical assay (4 acid digest)

Parameter	Detection Limit	Method	PM1 Concentrate (mg/kg)	PM1 Middlings (mg/kg)	PM1 Reject Tails (mg/kg)		PM2 Concentrate (mg/kg)	PM2 Middlings (mg/kg)	PM2 Reject Tails (mg/kg)
Ag	0.1	306P	3.25	0.37	0.43		1.41	0.37	0.32
As	0.1	306P	57.7	5.39	4.6		24	1.91	1.72
Bi	0.1	306P	1.07	0.46	0.52		1.09	0.4	0.33
Cd	0.1	306P	0.85	0.2	0.19		0.43	0.19	0.14
Ce	0.1	306P	41.4	5.37	5.85		19.9	6.03	4.75
Dy	0.1	306P	23.1	1.83	1.75		6.19	0.87	0.76
Er	0.05	306P	0.9	0.4	0.38		0.51	0.34	0.28
Eu	0.05	306P	0.18	0.05	<0.05		0.12	0.06	<0.05
Gd	0.05	306P	16.2	2.97	3.6		3.51	1.7	1.32
Hf	0.5	306P	1.18	<0.5	<0.5		0.81	<0.5	<0.5
Ho	0.01	306P	0.66	0.18	0.2		0.2	0.13	0.1
La	0.1	306P	15.6	1.49	1.54		10.1	2.5	1.96
Lu	0.01	306P	0.06	0.05	0.04		0.07	0.06	0.04
Nb	1	306P	56.8	1.55	1.65		42.2	1.7	1.31
Nd	0.2	306P	227	30.2	32.8		56.6	15.2	11.5
Pr	0.1	306P	43.5	7.43	8.47		9.91	3.9	2.89
Sb	0.2	306P	4.37	0.64	0.82		0.92	<0.2	<0.2
Sm	0.01	306P	1.71	0.46	0.44		1.38	0.54	0.45
Sn	2	306P	52.9	<2	<2		15.6	<2	<2
Ta	0.2	306P	140	1.93	2.68		110	2.77	2.02
Tb	0.01	306P	1.05	0.22	0.22		0.4	0.14	0.11
Th	0.5	306P	1.51	<0.5	<0.5		1.23	<0.5	<0.5
Tl	0.5	306P	<0.5	<0.5	<0.5		<0.5	<0.5	<0.5
Tm	0.01	306P	0.11	0.09	0.09		0.13	0.09	0.09
U	0.2	306P	5.94	0.26	0.32		3.54	<0.2	0.22
Y	0.1	306P	3.31	2.45	2.31		3.04	1.97	1.56
Yb	0.1	306P	0.46	0.34	0.32		0.44	0.37	0.3
Al	50	306P	25000	34300	33500		13800	24100	23300
Ba	2	306P	54	4	4		11	<2	3
Be	0.5	306P	<0.5	<0.5	<0.5		<0.5	<0.5	<0.5
Ca	50	306P	39600	39800	36500		15300	18000	14700
Co	2	306P	602	111	111		578	171	138
Cr	2	306P	869	467	463		514	183	143
Cu	2	306P	6500	1100	956		3930	1460	1040
Fe	50	306P	219000	85500	82700		338000	117000	91100
K	100	306P	1130	<100	124		312	<100	<100
Li	2	306P	<2	<2	<2		<2	<2	<2
Mg	50	306P	94500	153000	151000		96900	165000	170000
Mn	2	306P	1530	1360	1250		1710	1450	1170
Mo	2	306P	7	<2	3.1		<2	<2	<2
Na	50	306P	2870	216	152		778	123	113
Ni	2	306P	8350	1320	1360		9070	2490	2020
P	50	306P	126	60	53		103	85	65
Pb	10	306P	388	10	13		53	<10	<10
Rb	2	306P	2	<2	<2		<2	<2	<2
S	50	306P	39000	3790	3440		28800	6860	4140
Sc	1	306P	14.5	19.2	18.1		8	12.7	12
Sr	1	306P	58.9	11.7	10.6		22.1	11.6	9.2
Ti	2	306P	4560	760	536		3090	887	558
V	2	306P	193	59.7	55.6		270	62.5	44.2
Zn	2	306P	145	82	78		79	76	76
Zr	2	306P	27.6	5	5		17.6	3.2	3.7
S *	0.01	810L	3.91	0.38	0.35		2.87	0.72	0.43
C *	0.05	811L	1.11	1.01	0.95		2.96	2.99	2.28

Table E8. Gravity separation products chemical assay (4 acid digest)

Parameter	Detection Limit	Method	PX1 Concentrate (mg/kg)	PX1 Middlings (mg/kg)	PX1 Reject Tails (mg/kg)		PX2 Concentrate (mg/kg)	PX2 Middlings (mg/kg)	PX2 Reject Tails (mg/kg)
Ag	0.1	306P	4.87	0.23	0.52		3.96	0.32	0.45
As	0.1	306P	32	0.8	1.29		8.18	0.16	0.21
Bi	0.1	306P	2.25	0.1	0.21		1.06	0.24	0.22
Cd	0.1	306P	1.06	0.14	0.19		0.21	<0.1	<0.1
Ce	0.1	306P	26.2	2.65	3.55		25.6	7.8	9.61
Dy	0.1	306P	11.1	0.75	0.88		9.64	1.68	1.75
Er	0.05	306P	0.85	0.46	0.42		1.69	1.16	1.11
Eu	0.05	306P	0.18	0.06	0.06		0.34	0.18	0.17
Gd	0.05	306P	7.02	0.82	1.17		7.19	1.52	1.82
Hf	0.5	306P	1.57	<0.5	0.5		1.07	0.87	0.91
Ho	0.01	306P	0.37	0.15	0.14		0.64	0.35	0.35
La	0.1	306P	10.6	1.01	1.34		10.3	3.24	4.07
Lu	0.01	306P	0.09	0.06	0.06		0.31	0.22	0.2
Nb	1	306P	32	<1	<1		21.8	<1	<1
Nd	0.2	306P	119	4.34	8.43		86.3	7.3	11.1
Pr	0.1	306P	22.5	0.99	2.01		16.4	1.69	2.71
Sb	0.2	306P	1.22	<0.2	0.22		0.54	<0.2	<0.2
Sm	0.01	306P	1.51	0.49	0.49		2.6	1.17	1.19
Sn	2	306P	5.11	<2	<2		2.34	<2	<2
Ta	0.2	306P	81.4	0.26	1.37		35.5	<0.2	<0.2
Tb	0.01	306P	0.95	0.11	0.12		0.7	0.23	0.24
Th	0.5	306P	2.07	0.53	0.55		3.42	1.78	2.3
Tl	0.5	306P	<0.5	<0.5	<0.5		<0.5	<0.5	<0.5
Tm	0.01	306P	0.13	0.11	0.1		0.3	0.24	0.22
U	0.2	306P	5.62	0.26	0.23		3.88	1.1	1.25
Y	0.1	306P	4.44	3.28	2.87		12.3	9.11	8.71
Yb	0.1	306P	0.6	0.45	0.4		1.88	1.4	1.32
Al	50	306P	14500	25700	36600		14800	24100	28300
Ba	2	306P	10	<2	3		5	3	4
Be	0.5	306P	<0.5	<0.5	<0.5		<0.5	<0.5	<0.5
Ca	50	306P	52800	57900	48600		68800	68400	62600
Co	2	306P	678	88.6	102		96.4	80.3	83.3
Cr	2	306P	897	540	651		738	549	673
Cu	2	306P	15000	754	1050		5040	318	468
Fe	50	306P	164000	73200	81000		98900	64000	73900
K	100	306P	209	139	<100		223	179	200
Li	2	306P	<2	<2	4		2	2	3
Mg	50	306P	102000	137000	139000		108000	122000	120000
Mn	2	306P	1720	1260	1330		5810	2190	2160
Mo	2	306P	7.6	<2	3.6		<2	<2	<2
Na	50	306P	575	284	318		549	541	476
Ni	2	306P	9620	1080	1320		939	981	1020
P	50	306P	208	<50	<50		173	<50	<50
Pb	10	306P	61	<10	<10		23	368	<10
Rb	2	306P	<2	<2	<2		<2	<2	<2
S	50	306P	64400	2610	2830		4720	98	301
Sc	1	306P	20	24	20.9		34.9	33.9	28.9
Sr	1	306P	12.8	6	6.2		10.5	8.1	7.8
Ti	2	306P	9260	805	604		7420	872	915
V	2	306P	77.8	58.1	72.9		36.9	77.2	85.6
Zn	2	306P	165	67	79		112	95	99
Zr	2	306P	48.8	8	8.1		30.8	25.3	25.2
S *	0.01	810L	6.66	0.26	0.29		0.53	0.03	0.05
C *	0.05	811L	0.14	0.08	0.13		0.07	0.06	0.09

Table E9. Gravity separation products chemical assay (4 acid digest)

Parameter	Detection Limit	Method	MS1 Concentrate (mg/kg)	MS1 Middlings (mg/kg)	MS1 Reject Tails (mg/kg)		MS2 Concentrate (mg/kg)	MS2 Middlings (mg/kg)	MS2 Reject Tails (mg/kg)
Ag	0.1	306P	2.38	0.69	1		8.32	1.35	1.7
As	0.1	306P	36.8	1.03	2.4		13.3	2.61	1.87
Bi	0.1	306P	3.43	0.33	0.41		1.15	0.32	0.25
Cd	0.1	306P	1.13	0.33	0.53		1.48	0.3	0.44
Ce	0.1	306P	20.4	10.8	13.8		19.9	10.8	11.2
Dy	0.1	306P	20.7	2.6	2.59		5.58	0.79	0.9
Er	0.05	306P	1.76	1.44	1.43		0.73	0.43	0.43
Eu	0.05	306P	0.64	0.54	0.61		0.87	0.7	0.66
Gd	0.05	306P	7.48	2.54	3.02		3.36	1.05	1.3
Hf	0.5	306P	1.38	0.74	0.79		2.16	0.75	0.67
Ho	0.01	306P	0.69	0.49	0.5		0.27	0.13	0.14
La	0.1	306P	8.53	4.37	5.57		9.89	5.33	5.46
Lu	0.01	306P	0.21	0.19	0.18		0.09	0.05	0.05
Nb	1	306P	43.8	1.23	12.1		51.7	1.66	3
Nd	0.2	306P	120	11.4	14.7		46.1	7.98	10.6
Pr	0.1	306P	13.7	2.29	3.15		8.49	1.96	2.63
Sb	0.2	306P	1.57	<0.2	0.34		0.55	0.34	0.22
Sm	0.01	306P	2.76	2.12	2.47		1.5	0.85	0.85
Sn	2	306P	17.9	<2	4.67		24.4	2.08	3.69
Ta	0.2	306P	99.1	1.04	8.24		119	1.99	4.89
Tb	0.01	306P	0.85	0.39	0.41		0.43	0.12	0.14
Th	0.5	306P	2.07	1.1	1.45		1.81	0.95	1.05
Tl	0.5	306P	<0.5	<0.5	<0.5		<0.5	<0.5	<0.5
Tm	0.01	306P	0.26	0.24	0.24		0.14	0.1	0.1
U	0.2	306P	5.84	0.72	0.95		2.44	0.35	0.39
Y	0.1	306P	12.2	11.5	11.2		4.87	2.96	2.94
Yb	0.1	306P	1.44	1.32	1.29		0.63	0.4	0.4
Al	50	306P	24400	34700	37000		78500	94500	93700
Ba	2	306P	64	105	120		281	430	321
Be	0.5	306P	0.6	<0.5	<0.5		<0.5	0.6	<0.5
Ca	50	306P	60100	70900	66200		55400	44700	44900
Co	2	306P	512	72.7	80.7		312	50.2	54.9
Cr	2	306P	979	768	729		531	367	403
Cu	2	306P	6740	1400	2110		11100	1430	2020
Fe	50	306P	154000	78800	80100		83100	46900	48100
K	100	306P	1610	2770	2830		7910	12300	8380
Li	2	306P	8	18	20		31	48	41
Mg	50	306P	75200	87300	84000		51000	52800	53200
Mn	2	306P	1920	1710	1700		1010	771	776
Mo	2	306P	35.7	3.6	11.1		27.2	7.2	14.9
Na	50	306P	3920	7160	7930		19300	26700	26900
Ni	2	306P	11100	1120	1380		7750	1010	1240
P	50	306P	335	118	138		464	110	130
Pb	10	306P	141	<10	15		45	21	26
Rb	2	306P	3	9	8		35	55	38
S	50	306P	62100	5020	5910		33400	2570	3560
Sc	1	306P	42.3	49.7	45.6		18.2	14.1	13.5
Sr	1	306P	60.4	103	119		450	458	454
Ti	2	306P	2640	1840	1710		1070	1110	827
V	2	306P	134	147	139		60.4	57.2	52.3
Zn	2	306P	171	97	103		106	72	71
Zr	2	306P	30	16.6	18.5		62.6	12.3	12.5
S *	0.01	810L	6.35	0.51	0.58		3.37	0.25	0.34
C *	0.05	811L	0.16	0.08	0.08		0.09	0.07	0.1

Table E10. Gravity separation products chemical assay (4 acid digest)

Parameter	Detection Limit	Method	TZ1 Concentrate (mg/kg)	TZ1 Middlings (mg/kg)	TZ1 Reject Tails (mg/kg)	TZ2 Concentrate (mg/kg)	TZ2 Middlings (mg/kg)	TZ2 Reject Tails (mg/kg)
Ag	0.1	306P	2.7	1.41	1.14	2.63	0.36	1.13
As	0.1	306P	19.9	1.46	1.28	14.4	0.45	4.92
Bi	0.1	306P	1.44	0.2	0.24	1.2	0.1	0.27
Cd	0.1	306P	1.51	0.36	0.41	1.19	0.12	0.32
Ce	0.1	306P	26.2	11	10.7	19.1	4.51	9.47
Dy	0.1	306P	18.3	1.48	1.35	2.24	0.47	0.97
Er	0.05	306P	1.06	0.82	0.71	0.66	0.3	0.56
Eu	0.05	306P	0.7	0.67	0.64	0.48	0.23	0.44
Gd	0.05	306P	4.23	1.45	1.46	3.77	0.54	1.18
Hf	0.5	306P	1.66	0.69	0.65	1.48	<0.5	0.6
Ho	0.01	306P	0.39	0.26	0.23	0.28	0.09	0.18
La	0.1	306P	12.3	5.58	5.49	8.5	2.14	4.5
Lu	0.01	306P	0.12	0.11	0.09	0.08	0.03	0.07
Nb	1	306P	121	2.03	3.53	67.4	<1	3.12
Nd	0.2	306P	103	7.88	9.07	33.9	3.25	7.68
Pr	0.1	306P	11.6	1.72	1.92	8.29	0.77	1.86
Sb	0.2	306P	0.24	<0.2	<0.2	0.42	<0.2	<0.2
Sm	0.01	306P	2.42	1.35	1.26	1.54	0.46	0.91
Sn	2	306P	22	<2	5.92	17.8	<2	5.94
Ta	0.2	306P	156	3.26	5.44	169	0.95	6.23
Tb	0.01	306P	0.68	0.22	0.2	0.34	0.07	0.14
Th	0.5	306P	2.62	1	1.12	1.96	<0.5	0.93
Tl	0.5	306P	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Tm	0.01	306P	0.17	0.16	0.14	0.13	0.09	0.12
U	0.2	306P	5.96	0.45	0.5	4.17	<0.2	0.38
Y	0.1	306P	6.85	6.27	5.4	4.5	2.06	3.99
Yb	0.1	306P	0.81	0.79	0.68	0.58	0.29	0.52
Al	50	306P	53400	81800	87500	45100	43000	83300
Ba	2	306P	114	225	251	91	121	200
Be	0.5	306P	<0.5	0.7	0.6	<0.5	<0.5	<0.5
Ca	50	306P	52700	62800	57400	49100	32200	57800
Co	2	306P	860	61.3	59.5	868	25.1	67.4
Cr	2	306P	895	299	353	1160	221	581
Cu	2	306P	9700	1950	2280	9590	543	1630
Fe	50	306P	170000	58600	55500	208000	28400	57700
K	100	306P	2820	4780	5180	2810	3810	5910
Li	2	306P	13	20	23	11	15	29
Mg	50	306P	42500	46100	42100	48600	29500	56800
Mn	2	306P	1540	1000	899	1050	469	906
Mo	2	306P	86.6	9.6	19.4	83.2	5.4	19.2
Na	50	306P	11500	21400	23400	8750	9150	17300
Ni	2	306P	19300	1110	1190	14300	336	982
P	50	306P	148	101	106	327	<50	119
Pb	10	306P	56	20	17	81	<10	28
Rb	2	306P	6	10	13	9	14	25
S	50	306P	83900	6840	5700	92100	3130	6330
Sc	1	306P	24.6	25.7	21.1	23.3	13	23.1
Sr	1	306P	298	494	517	239	249	466
Ti	2	306P	1590	1430	1300	1670	727	1240
V	2	306P	91.2	92.2	79.6	80.2	45.3	83.8
Zn	2	306P	163	68	66	108	37	82
Zr	2	306P	27.9	10.9	11.2	36.1	4.7	9.7
S *	0.01	810L	9.21	0.7	0.56	11.6	0.65	0.61
C *	0.05	811L	0.09	0.08	<0.05	0.06	0.05	0.07

Table E11. Gravity separation products chemical assay (4 acid digest)

Parameter	Detection Limit	Method	BAS1 Concentrate (mg/kg)	BAS1 Middlings (mg/kg)	BAS1 Reject Tails (mg/kg)		BAS2 Concentrate (mg/kg)	BAS2 Middlings (mg/kg)	BAS2 Reject Tails (mg/kg)
Ag	0.1	306P	6.56	2.13	1.88		0.79	4.51	1.15
As	0.1	306P	48.5	1.69	2.11		1.3	24.4	2.64
Bi	0.1	306P	1.07	0.19	0.14		0.31	1.55	0.36
Cd	0.1	306P	8.23	1.52	1.39		0.26	1.45	0.33
Ce	0.1	306P	27.1	15.6	15.1		7.42	12.2	8.35
Dy	0.1	306P	3.2	0.68	0.67		1.7	11.1	1.64
Er	0.05	306P	0.63	0.33	0.33		1.01	1.44	0.89
Eu	0.05	306P	0.85	0.7	0.71		0.66	0.67	0.65
Gd	0.05	306P	4.17	1.18	1.06		1.47	4.98	1.81
Hf	0.5	306P	1.31	0.54	0.6		<0.5	0.95	0.62
Ho	0.01	306P	0.28	0.11	0.1		0.32	0.52	0.29
La	0.1	306P	12.8	7.82	7.62		3.75	5.16	4.12
Lu	0.01	306P	0.07	0.04	0.04		0.14	0.19	0.12
Nb	1	306P	90.9	3.2	1.61		1.41	46.8	1.63
Nd	0.2	306P	42.5	11	10		6.71	76.9	10.1
Pr	0.1	306P	9.87	2.82	2.57		1.36	12.3	2.28
Sb	0.2	306P	2.51	0.31	0.26		0.22	1.14	0.39
Sm	0.01	306P	1.78	1.05	1		1.12	1.55	1.14
Sn	2	306P	25.3	3.9	<2		<2	26.8	3.47
Ta	0.2	306P	185	5.04	3.27		1.86	102	2.69
Tb	0.01	306P	0.33	0.11	0.11		0.24	0.72	0.24
Th	0.5	306P	1.62	0.89	0.79		0.57	0.93	0.74
Tl	0.5	306P	<0.5	<0.5	0.55		<0.5	<0.5	<0.5
Tm	0.01	306P	0.12	0.09	0.09		0.19	0.23	0.17
U	0.2	306P	10.3	0.32	0.39		0.22	3.52	0.29
Y	0.1	306P	4.24	2.3	2.26		7.65	9.91	6.55
Yb	0.1	306P	0.54	0.3	0.3		0.98	1.27	0.84
Al	50	306P	75700	105000	104000		80600	56300	86900
Ba	2	306P	278	435	562		211	92	179
Be	0.5	306P	<0.5	0.5	0.5		<0.5	0.5	<0.5
Ca	50	306P	33500	35700	34900		57700	59000	51300
Co	2	306P	858	68.1	70.7		43.9	257	51.5
Cr	2	306P	492	300	270		411	710	471
Cu	2	306P	5400	1580	1340		3670	27000	4990
Fe	50	306P	133000	43000	48800		68300	133000	68900
K	100	306P	7850	11500	16400		4890	2660	3900
Li	2	306P	20	39	57		30	24	27
Mg	50	306P	23600	27700	30200		50500	53100	49600
Mn	2	306P	752	546	592		1320	2040	1150
Mo	2	306P	47.1	21.8	10.6		6	42.9	19.4
Na	50	306P	26000	38500	36200		23300	13000	24900
Ni	2	306P	16800	1520	1460		517	3080	668
P	50	306P	257	158	141		62	150	80
Pb	10	306P	380	93	88		11	84	19
Rb	2	306P	31	47	68		19	6	16
S	50	306P	69700	7040	8890		5000	49400	6640
Sc	1	306P	8.5	6.8	7.5		28.9	35	23.5
Sr	1	306P	494	636	608		356	192	383
Ti	2	306P	1150	917	1240		2140	5630	1770
V	2	306P	36.5	35.1	42.7		145	175	125
Zn	2	306P	1480	349	367		88	226	98
Zr	2	306P	27.6	9.6	7.4		5.7	16.3	5.8
S *	0.01	810L	8.84	0.67	0.96		0.5	4.93	0.66
C *	0.05	811L	0.12	0.06	0.05		<0.05	0.11	<0.05

Table E12. ICP measurement data for gravity separation products (Measurement by X-Ray Mineral Services)

Oxide/Element	%	PM1				PM2				PX1				PX2				
		Feed	Sample	Concentrate	Middlings	Reject	Tails	Feed	Sample	Concentrate	Middlings	Reject	Tails	Feed	Sample	Concentrate	Middlings	Reject
Al ₂ O ₃	%	6.3	44.4	29.5	42.8	6.8	6.7	4.4	40.6	14.0	33.7	41.3	49.2	42.0	49.4	5.0	5.0	6.7
SiO ₂	%	0.1	0.9	0.1	0.1	0.1	0.1	0.1	0.5	0.2	0.2	0.1	0.1	1.5	0.2	0.2	0.1	0.1
TiO ₂	%	12.5	33.2	12.0	11.8	14.9	14.9	2.2	2.1	2.5	2.1	2.1	11.5	20.0	10.4	11.1	11.1	11.1
Fe ₂ O ₃	%	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
MnO	%	24.3	14.1	25.0	25.6	27.2	14.0	25.2	27.0	27.0	22.8	22.8	16.6	21.4	21.4	21.7	21.7	21.7
MgO	%	5.4	5.3	5.1	4.8	5.1	5.1	2.2	2.1	2.5	2.1	2.1	7.8	7.4	7.4	8.0	8.0	8.0
CaO	%	0.0	0.5	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Na ₂ O	%	0.0	0.0	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
K ₂ O	%	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
P ₂ O ₅	%	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Cl	%	na	na	na	na	na	na	na	na	na	na	na	na	na	na	na	na	na
S	ppm	306	3280	429	308	460	3200	617	206	357	2961	214	315	na	na	na	na	na
As	ppm	282	80	30	34	34	108	60	40	52	133	40	41	25	25	25	25	25
Ba	ppm	2	34	5	6	3	16	9	5	2	15	3	4	4	4	4	4	4
Ce	ppm	106	510	123	120	130	401	139	113	86	323	76	84	84	84	84	84	84
Co	ppm	603	1023	603	609	193	593	227	180	808	922	708	850	850	850	850	850	850
Cr	ppm	0	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Cs	ppm	908	3928	697	671	855	1785	913	799	850	5026	520	797	797	797	797	797	797
Cu	ppm	6	7	6	6	5	4	5	5	6	4	5	6	4	5	7	5	7
Ga	ppm	na	na	na	na	na	na	na	na	na	na	na	na	na	na	na	na	na
Ge	ppm	0	2	0	0	0	0	1	0	0	0	0	0	2	0	0	0	0
Hf	ppm	1	21	2	2	1	9	5	3	1	1	1	1	9	2	2	2	2
La	ppm	0	6	2	3	0	1	1	1	1	1	1	1	6	1	3	1	3
Mo	ppm	1	74	2	2	1	55	4	3	13	11	2	97	5	8	8	8	8
Nb	ppm	2	313	26	31	2	46	13	11	11	11	2	97	5	8	8	8	8
Nd	ppm	1177	2569	1059	1147	1383	762	1146	1633	1151	1151	651	946	946	946	946	946	946
Ni	ppm	1	253	7	10	0	19	1	1	1	1	1	17	1	1	1	1	1
Pb	ppm	17	70	13	12	12	22	13	12	10	10	10	13	9	7	7	7	7
Rb	ppm	0	2	0	0	0	0	1	1	0	0	0	1	1	0	0	0	0
Sb	ppm	0	9	0	0	0	0	5	0	0	0	0	4	0	0	0	0	0
Se	ppm	3	228	60	58	57	301	68	45	69	69	69	64	64	74	74	74	74
Sn	ppm	76	216	98	95	81	129	94	95	86	86	86	80	96	3	3	3	3
Sr	ppm	16	68	16	14	14	44	17	13	16	16	16	15	15	13	13	13	13
Th	ppm																	
U	ppm																	
V	ppm																	
Y	ppm																	
Zn	ppm																	
Zr	ppm																	

Note:
Some of the major elements totals are less than 100% because sulphur in the samples may be lost in part during fusion for ICP analysis and is therefore not measured

Table E13. ICP measurement data for gravity separation products (Measurement by X-Ray Mineral Services)

Oxide/Element	PX2				MS1				MS2			
	Feed Sample	Concentrate	Middlings	Reject Tails	Feed Sample	Concentrate	Middlings	Reject Tails	Feed Sample	Concentrate	Middlings	Reject Tails
Al2O3 %	5.5	2.8	4.5	5.1	7.2	4.6	6.2	7.0	17.3	15.2	19.3	18.7
SiO2 %	49.7	48.0	49.4	48.0	54.7	44.4	52.5	54.3	54.4	48.6	51.6	51.8
TiO2 %	0.2	3.7	0.2	0.2	0.3	0.5	0.3	0.3	0.1	0.2	0.2	0.1
Fe2O3 %	12.3	14.2	12.1	12.5	11.0	20.1	11.3	11.1	6.6	12.5	7.0	7.0
MnO %	0.3	0.8	0.3	0.3	0.2	0.3	0.2	0.2	0.1	0.1	0.1	0.1
MgO %	20.0	17.1	19.1	19.4	14.1	12.4	13.5	13.7	7.8	8.2	8.3	8.6
CaO %	9.2	9.4	9.2	8.7	9.8	8.5	9.6	9.2	6.7	7.7	6.2	6.2
Na2O %	0.1	0.1	0.1	0.1	1.1	0.5	0.9	1.1	3.4	2.4	3.5	3.5
K2O %	0.0	0.0	0.0	0.0	0.3	0.2	0.3	0.4	1.2	0.9	1.5	1.0
P2O5 %	0.0	0.1	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.1	0.0	0.0
Cl %	na	na	na	na	na	na	na	na	na	na	na	na
S ppm	64	262	98	124	464	3417	417	666	289	1672	179	258
As ppm	na	na	na	na	na	na	na	na	na	na	na	na
Ba ppm	367	44	33	54	282	70	170	131	410	290	431	329
Ce ppm	12	18	8	9	12	20	11	13	10	21	11	12
Co ppm	72	103	71	73	69	278	64	72	51	217	53	58
Cr ppm	866	1102	1136	806	1066	904	888	458	458	702	519	569
Cs ppm	0	0	0	0	0	1	0	1	2	2	2	2
Cu ppm	451	2963	247	357	1235	3544	962	1459	1212	5213	912	1271
Ga ppm	7	5	6	7	10	9	10	10	16	16	16	16
Ge ppm	na	na	na	na	na	na	na	na	na	na	na	na
Hf ppm	1	3	1	1	2	1	1	1	1	3	1	1
La ppm	5	13	3	5	5	10	5	7	5	10	6	6
Mo ppm	0	4	1	2	1	34	6	10	1	32	8	17
Nb ppm	2	81	2	3	1	88	3	3	1	107	2	6
Nd ppm	6	62	8	10	8	103	11	14	5	40	8	10
Ni ppm	926	802	988	1012	1005	2228	957	1128	1003	984	855	1002
Pb ppm	1	14	2	2	5	53	7	8	11	16	16	15
Rb ppm	1	3	3	1	10	7	10	13	43	32	54	39
Sb ppm	na	na	na	na	na	na	na	na	na	na	na	na
Se ppm	na	na	na	na	na	na	na	na	na	na	na	na
Sn ppm	0	253	4	7	1	322	6	8	0	202	6	13
Sr ppm	19	12	10	12	122	57	100	115	418	415	416	431
Th ppm	2	3	2	2	1	2	1	1	1	2	0	1
U ppm	1	3	1	1	1	4	1	1	0	2	0	0
V ppm	112	101	106	103	143	136	142	139	54	63	55	54
Y ppm	10	14	10	9	12	16	11	12	3	5	3	3
Zn ppm	106	130	108	112	106	183	112	117	79	134	79	82
Zr ppm	36	101	33	32	29	54	29	31	20	98	22	26

Note:
Some of the major elements totals are less than 100% because sulphur in the samples may be lost in part during fusion for ICP analysis and is therefore not measured

Table E14. ICP measurement data for gravity separation products (Measurement by X-Ray Mineral Services)

Oxide/Element	%	TZ1			TZ2			BAS1				
		Feed	Sample	Concentrate	Middlings	Reject	Tails	Feed	Sample	Concentrate	Middlings	Reject Tails
Al2O3	16.0	10.8	16.3	17.4	16.7	9.1	16.6	16.7	20.5	15.6	20.4	20.7
SiO2	54.5	40.6	52.4	54.0	51.8	37.8	51.6	52.5	55.1	44.1	53.4	54.5
TiO2	0.2	0.3	0.2	0.2	0.2	0.4	0.2	0.2	0.2	0.2	0.2	0.2
Fe2O3	7.9	20.1	8.4	7.9	8.1	22.8	8.1	7.8	6.4	18.5	7.4	6.1
MnO	0.1	0.2	0.1	0.1	0.1	0.2	0.1	0.1	0.1	0.1	0.1	0.1
MgO	6.9	6.8	7.3	6.8	9.1	7.7	9.3	9.4	4.6	3.9	5.1	4.7
CaO	8.5	7.2	8.5	7.8	8.6	6.8	8.4	7.9	5.3	4.7	4.8	4.8
Na2O	2.9	1.4	2.7	3.0	2.3	1.2	2.3	2.3	5.0	3.5	4.8	5.0
K2O	0.6	0.3	0.6	0.6	0.8	0.3	0.9	0.7	1.5	0.9	1.9	1.3
P2O5	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.1	0.0	0.0
Cl	%	na	na	na	na	na	na	na	na	na	na	na
S	ppm	447	3181	394	421	505	3283	525	303	611	2648	375
As	ppm	na	na	na	na	na	na	na	na	na	na	na
Ba	ppm	268	146	232	261	265	121	253	204	497	309	567
Ce	ppm	10	27	12	11	8	21	9	9	15	30	422
Co	ppm	55	323	60	57	55	264	50	60	71	387	65
Cr	ppm	318	1254	386	460	599	1102	684	749	229	586	356
Cs	ppm	1	1	1	1	1	1	1	1	2	2	410
Cu	ppm	1259	4096	1004	1215	886	2667	566	925	969	1780	2
Ga	ppm	19	15	19	19	17	10	16	16	24	21	25
Ge	ppm	na	na	na	na	na	na	na	na	na	na	na
Hf	ppm	1	4	1	1	0	2	1	1	1	3	1
La	ppm	5	13	6	6	4	10	5	5	8	14	8
Mo	ppm	0	89	9	21	1	67	9	19	1	44	22
Nb	ppm	1	271	3	3	1	139	2	3	1	133	3
Nd	ppm	5	174	8	9	4	31	6	7	7	104	11
Ni	ppm	843	1159	689	884	587	493	477	733	898	1126	931
Pb	ppm	9	19	8	13	6	25	7	18	32	130	46
Rb	ppm	13	10	14	15	28	13	32	27	51	33	48
Sb	ppm	na	na	na	na	na	na	na	na	na	na	na
Se	ppm	na	na	na	na	na	na	na	na	na	na	na
Sn	ppm	1	560	7	8	0	420	5	7	0	336	7
Sr	ppm	468	276	457	470	484	234	458	446	623	495	596
Th	ppm	1	2	1	1	1	2	1	1	2	1	1
U	ppm	0	7	0	0	0	6	0	0	0	13	0
V	ppm	85	93	89	80	84	81	86	82	37	41	37
Y	ppm	6	8	6	5	4	5	4	3	4	2	2
Zn	ppm	75	163	75	78	69	120	65	75	454	1556	351
Zr	ppm	24	95	26	25	19	59	20	19	23	85	25

Note:
Some of the major elements totals are less than 100% because sulphur in the samples may be lost in part during fusion for ICP analysis and is therefore not measured

Table E15. ICP measurement data for gravity separation products (Measurement by X-Ray Mineral Services)

Oxide/ Element		BAS2			
		Feed Sample	Concentrate	Middlings	Reject Tails
Al ₂ O ₃	%	16.3	11.3	15.4	17.0
SiO ₂	%	51.8	45.6	52.4	50.8
TiO ₂	%	0.3	0.9	0.3	0.3
Fe ₂ O ₃	%	9.6	17.4	9.8	9.8
MnO	%	0.2	0.3	0.2	0.2
MgO	%	7.8	8.8	8.1	8.1
CaO	%	7.5	8.0	7.7	6.9
Na ₂ O	%	3.1	1.7	2.9	3.2
K ₂ O	%	0.5	0.3	0.6	0.5
P ₂ O ₅	%	0.0	0.0	0.0	0.0
Cl	%	na	na	na	na
S	ppm	448	2318	250	325
As	ppm	na	na	na	na
Ba	ppm	268	116	237	210
Ce	ppm	8	14	8	9
Co	ppm	44	181	49	53
Cr	ppm	435	803	509	600
Cs	ppm	1	1	1	1
Cu	ppm	2298	9147	1761	2662
Ga	ppm	19	16	18	19
Ge	ppm	na	na	na	na
Hf	ppm	0	1	1	1
La	ppm	4	8	4	5
Mo	ppm	0	35	6	22
Nb	ppm	1	49	3	2
Nd	ppm	4	81	7	10
Ni	ppm	519	573	444	577
Pb	ppm	5	36	7	10
Rb	ppm	19	10	24	19
Sb	ppm	na	na	na	na
Se	ppm	na	na	na	na
Sn	ppm	1	200	5	7
Sr	ppm	367	179	327	357
Th	ppm	1	1	1	1
U	ppm	0	4	0	0
V	ppm	123	169	135	117
Y	ppm	7	9	7	6
Zn	ppm	105	240	100	109
Zr	ppm	15	39	17	18

Note:

Some of the major elements totals are less than 100% because sulphur in the samples may be lost in part during fusion for ICP analysis and is therefore not measured

20 APPENDIX F – FLOTATION SEPARATION DATA

Table F1. ICP Measurement data of flotation feed and tailing samples (Measurement by X-Ray Mineral Services)

Oxide/ Element	PM2		PX1		PX2		MS1		MS2	
	Feed	Sample								
Al ₂ O ₃	4.4	4.6	6.6	5.9	5.5	4.9	7.2	6.5	17.3	17.3
SiO ₂	40.6	39.7	49.2	47.0	49.7	48.8	54.7	53.5	54.4	50.3
TiO ₂	0.1	0.2	0.1	0.2	0.2	0.2	0.3	0.3	0.1	0.2
Fe ₂ O ₃	14.9	20.2	11.5	11.5	12.3	12.9	11.0	11.1	6.6	7.2
MnO	0.2	0.2	0.2	0.2	0.3	0.3	0.2	0.2	0.1	0.1
MgO	27.2	31.6	22.8	27.0	20.0	22.6	14.1	16.5	7.8	13.4
CaO	2.2	3.4	7.8	8.3	9.2	10.2	9.8	10.8	6.7	7.4
Na ₂ O	0.1	0.0	0.1	0.0	0.1	0.0	1.1	0.7	3.4	3.3
K ₂ O	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.3	1.2	0.9
P ₂ O ₅	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Cl	na	na								
S	ppm	460	508	357	182	64	46	46	304	289
As	ppm	na	na	na	na	na	na	na	na	na
Ba	ppm	108	0	133	0	367	0	282	160	410
Ce	ppm	3	0	2	0	12	0	12	0	10
Co	ppm	130	131	86	83	72	84	69	55	51
Cr	ppm	193	260	808	865	1123	1173	806	796	458
Cs	ppm	0	14	0	12	0	0	0	12	2
Cu	ppm	855	180	850	41	451	16	1235	148	1212
Ga	ppm	5	6	3	7	8	10	12	12	16
Ge	ppm	na	na	na	na	na	na	na	na	na
Hf	ppm	0	na	0	1	na	1	na	1	na
La	ppm	1	0	0	0	5	4	5	0	5
Mo	ppm	0	0	0	2	0	0	1	2	1
Nb	ppm	1	0	1	0	2	5	1	0	1
Nd	ppm	2	93	2	100	6	67	8	108	5
Ni	ppm	1383	1329	1151	907	926	886	1005	606	1003
Pb	ppm	0	0	1	0	1	0	5	5	11
Rb	ppm	1	2	2	0	1	0	10	14	43
Sb	ppm	na	na	na	na	na	na	na	na	na
Se	ppm	na	na	na	na	na	na	na	na	na
Sn	ppm	0	0	1	0	0	0	1	0	0
Sr	ppm	12	-17	10	0	19	0	122	175	418
Th	ppm	0	0	0	0	2	2	1	1	2
U	ppm	0	0	0	0	1	1	0	0	0
V	ppm	57	52	69	64	112	98	143	138	54
Y	ppm	5	1	4	1	10	12	13	3	0
Zn	ppm	81	80	86	76	106	109	106	90	79
Zr	ppm	14	15	16	15	36	31	29	28	23

Note:
Some of the major elements totals are less than 100% because sulphur in the samples may be lost in part during fusion for ICP analysis and is therefore not measured

Table F2. ICP Measurement data of flotation feed and tailing samples (Measurement by X-Ray Mineral Services)

Oxide/Element	TZ1		TZ2		BAS1		BAS2	
	Feed Sample	Test5/Tails	Feed Sample	Test6/Tails	Feed Sample	Test7/Tails	Feed Sample	Test8/Tails
Al2O3	16.0	15.6	16.7	16.4	20.5	20.1	16.3	15.8
SiO2	54.5	52.9	51.8	50.3	55.1	54.1	51.8	51.6
TiO2	0.2	0.2	0.2	0.2	0.2	0.2	0.3	0.3
Fe2O3	7.9	8.1	8.1	7.7	6.4	5.7	9.6	9.5
MnO	0.1	0.1	0.1	0.1	0.1	0.1	0.2	0.2
MgO	6.9	10.4	9.1	12.9	4.6	7.5	7.8	10.8
CaO	8.5	9.3	8.6	9.6	5.3	6.0	7.5	8.4
Na2O	2.9	2.9	2.3	2.1	5.0	5.2	3.1	3.1
K2O	0.6	0.5	0.8	0.6	1.5	1.2	0.5	0.4
P2O5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Cl	na							
S	ppm	447	230	505	302	611	273	448
As	ppm	na						
Ba	ppm	268	375	265	331	497	762	268
Ce	ppm	10	0	8	25	15	23	8
Co	ppm	55	43	55	29	71	31	44
Cr	ppm	318	331	599	593	229	246	435
Cs	ppm	1	0	1	0	2	0	1
Cu	ppm	1259	210	886	81	969	88	2298
Ga	ppm	19	21	17	18	24	22	19
Ge	ppm	na						
Hf	ppm	1	na	0	na	1	na	0
La	ppm	5	14	4	9	8	5	4
Mo	ppm	0	2	1	2	1	2	0
Nb	ppm	1	0	1	0	1	0	1
Nd	ppm	5	68	4	79	7	79	4
Ni	ppm	843	415	587	266	898	366	519
Pb	ppm	9	13	6	9	32	51	5
Rb	ppm	13	16	28	33	51	62	19
Sb	ppm	na						
Se	ppm	na						
Sn	ppm	1	0	0	0	0	0	1
Sr	ppm	468	875	484	855	623	1140	367
Th	ppm	1	2	1	1	3	1	3
U	ppm	0	0	0	0	0	0	0
V	ppm	85	84	78	37	39	123	111
Y	ppm	6	4	0	3	0	7	5
Zn	ppm	75	56	69	55	454	225	105
Zr	ppm	24	17	19	18	23	24	15

Note:

Some of the major elements totals are less than 100% because sulphur in the samples may be lost in part during fusion for ICP analysis and is therefore not measured