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RESEARCH REPORT

June 6, 2018 C/MT/2017/8

Hannukainen Mining Oy Pilot Plant Operation, 11. – 29.9. 2017

Tapio Knuutinen

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Rautajärvi Pilot Plant

Abstract

A metallurgical test work was carried on Hannukainen iron ore samples at the pilot plant of GTK in September 11 - 29, 2017.

The objectives of the pilot plant were:

- To study the effect of different particle size distributions in processing
- To produce concentrate for further tests
- Mineralogical examination of the process products
- To collect tailings streams for environmental studies
- Process water analysis

Following table summarizes the results achieved in copper and magnetite concentration WITH Ve2 and VE3 samples.

September 25, 2017 - 13:40	Mass		Cu		Fe		S	
Feed: VE3	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%
CuCC3	3.1	0.4	17.70	78.2	36.1	0.8	46.64	23.5
WLC4	60.3	8.0	0.000	0.0	69.8	31.0	0.05	0.5
Tailings Low-S (WLT1-3)	665.0	88.6	0.006	5.4	12.4	61.1	0.17	18.0

September 26, 2017 - 20:00-20:45	Mass		Cu		Fe		S	
Feed: VE2	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%
CuCC3	3.7	0.5	16.75	66.3	67.9	74.7	47.50	6.6
WLC4	208.4	37.8	0.001	0.2	69.6	65.0	0.05	0.4
Tailings Low-S (PyTails+WLT4)	430.4	57.4	0.018	8.1	10.1	19.5	2.37	38.5

September 28, 2017 - 01:30-03:00	Mass		Cu		Fe		S	
Feed: VE2	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%
CuCC3	4.4	0.6	19.55	81.6	34.2	0.6	42.20	7.4
WLC4	221.3	29.5	0.00	0.3	69.6	64.9	0.10	0.9
Tailings Low-S (PyTails+WLT4)	388.0	51.7	0.01	2.8	8.2	13.3	0.37	5.7

September 28, 2017 - 10:00	Mass		Cu		Fe		S	
Feed: VE2	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%
CuCC3	5.0	0.7	17.45	78.6	35.3	0.7	45.53	7.7
WLC4	274.5	36.5	0.00	0.5	69.9	72.4	0.05	0.5
Tailings Low-S (PyTails+WLT4)	348.3	46.4	0.01	3.0	7.8	10.3	0.31	3.6

September 28, 2017 - 10:00	Mass		Cu		Fe		S	
Feed: VE2	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%
CuCC3	3.3	0.4	21.61	68.7	33.1	0.4	45.24	5.4
WLC4	254.0	33.8	0.00	0.3	69.9	70.8	0.18	1.7
Tailings Low-S (PyTails+WLT4)	387.6	51.6	0.02	5.6	9.0	13.9	1.66	23.4

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June 6, 2018

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Table of Content

June 6, 2018

List of Appendices

Appendix 1 – Flotation Test Report

Appendix 2 – Sampling LOG, XRF Assays and Screen Assays

Appendix 3 – Mineralogical Studies

Appendix 4 – Mass and Water Balance

List of Abbreviations (used in balance calculations and flow sheets)

RMD – Rod Mill Discharge
BMD – Ball Mill Discharge
Skako O/S – Primary Screen Over Size
Skako U/S – Primary Screen Under Size
Derrick O/S – Secondary Screen Over Size
Derrick U/S – Secondary Screen Under Size
CuRC – Copper Rougher Concentrate
CuRT – Copper Rougher Tailings
CuCC – Copper Cleaner Concentrate (number refers to cleaning stage)
CuCT – Copper Cleaner Tailings (number refers to cleaning stage)
CuCSC – Copper Cleaner Scavenger Concentrate
CuCST – Copper Cleaner Scavenger Tailings
WLC – Wet LIMS Concentrate (number refers the stage)
WLT – Wet LIMS Tailings (number refers the stage)
WLT 1-3 – Combined LIMS Tailing
CyUF – Cyclone Under Flow
CyOF – Cyclone Over Flow
PyRC – Pyrite Rougher Concentrate
PyRT – Pyrite Rougher Tailings
PyCC – Pyrite Cleaner Concentrate
PyCT – Pyrite Cleaner Tailings
PoRC – Pyrrhotite Rougher Concentrate
PoRT – Pyrrhotite Rougher Tailings
PoCC – Pyrrhotite Cleaner Concentrate
PoCT – Pyrrhotite Cleaner Tailings

June 6, 2018

1 INTRODUCTION

A metallurgical test work was carried on Hannukainen iron ore samples at the pilot plant of GTK in September 11 - 29, 2017. The Client delivered two head samples – VE2 and VE3 – approximately 150 tons of both feeds for the test work in September 2017. VE2 represents the main ore body and the VE3 near cut-off part of the Hannukainen ore body.

The pilot plant was operated with the sample VE2 during September 11 - 21 and 26 - 29 and with the sample VE3 during September 22 - 25.

In addition, the client delivered sub samples, ca. 150 kg, of both feeds in August 2017 for a bench scale test work.

The Hannukainen flowsheet comprised crushing, grinding, classification, flotation and magnetic separation stages.

The objectives of the pilot plant were:

- To study the effect of different particle size distributions in processing
- To produce concentrate for further tests
- Mineralogical examination of the process products
- To collect tailings streams for environmental studies
- Process water analysis

2 SAMPLE PREPARATION AND CRUSHING

The head samples were crushed individually to -100 mm with a jaw crusher (Lokomo C63) in closed circuit with a vibrating screen (Lokomo B230) and sampled for pre-concentration tests (sorting, magnetic separation, dense media separation). Approximately ten tons of both feeds were sampled during the pre-crushing stage and stored in big bags.

After the pre-crushing and sampling stage, the remaining samples, -100 mm, were crushed individually to 100 % passing 6 mm using the jaw crusher in the primary stage and a cone crusher (Lokomo G108) in the secondary stage in closed circuit with the Lokomo screen (B230). The crushed VE2 sample was reported into homogenization stock pile prior to wet processing and the crushed VE3 sample was stored for further processing.

Head samples were sampled during the crushing stage for laboratory tests and chemical and mineralogical analyses.

3 PILOT PLANT FEED SAMPLES

Head samples were sampled during the fine crushing stage. The collected sub samples were prepared for chemical and mineralogical analyses by dividing and crushing in multiple stages. The head assays are presented in Table 1.

Table 1. Head Assays of VE2 and VE3 Samples

Contents (%)	VE2 S17063029	VE3 S17067435	VE3 S17067436
SiO2	29.2	40.7	41.0
Al2O3	6.31	10.1	10.1
V2O3	0.013	0.013	0.013
MgO	4.76	5.83	5.80
CaO	9.03	16.2	16.3
Na2O	1.93	1.56	1.56
Cu	0.128	0.07	0.067
Ni	0.008	0.007	0.007
Cl	0.295	0.209	0.209
Fe	33.0	16.7	16.3
Eltra S	4.16	1.36	1.34
Satmagan	35.31	9.14	8.63
Au g/t	0.05	0.04	0.03

According to the Satmagan measurements, magnetite content in VE2 sample was 35.5% while in sample VE3 the corresponding value was 8.8% as average. So in that sense, VE3 sample can be considered as low grade ore that requires pre-concentration before feeding the beneficiation process.

The modal mineralogies of the head samples are presented in Table 2. The chemical formulas for different minerals are presented in Table 2 and in Appendix 3.

Table 2. Modal mineralogy of VE2 and VE3

Mineral	Chemical Formula (ideal)	VE2 - Pilot Feed	VE3 - Pilot Feed
		Wt%	Wt%
Quartz	SiO_2	0.10	0.81
Plagioclase	$Na_{0.5}Ca_{0.5}Si_3AlO_8$	1.11	11.62
K_feldspar	$KAlSi_3O_8$	0.13	0.48
Clinopyroxene	$(Ca,Mg,Fe,Al)_2(Si,Al)_20_6$	25.30	36.48
Ferrotschermakite	$(Ca,Fe)_2(Al_2Si_6O_{22})(OH)_2$	5.77	16.92
Actinolite	$Ca2Mg3Fe^{2+}{}_2Si_8O_{22}(OH)_2$	1.12	3.08
Tremolite	$Ca_2Mg_5(Si_8O_{22})(OH)_2$	0.00	0.00
Andradite	$Ca_3Fe^{2+}{}_2(SiO_4)_3$	0.08	
Marialite	$Na_4Al_3Si_9O_{24}Cl$	7.99	0.60
Epidote	$Ca_2Al_2(Fe^{3+},Al)(SiO_4)(Si_2O_7)O(OH)$	1.13	12.75
Chlorite	$(Mg,Fe)_3(Si,Al)_4O_{10}(OH)_2\cdot(Mg,Fe)_3(OH)_6$	0.13	1.16
Biotite	$K(Mg,Fe)_3(OH,F)_2(AlSi_3O_{10})$	0.03	0.05
Titanite	$CaTiSiO_5$	0.28	0.71
Clay	-	0.22	0.39
Berthierine	$(Fe^{2+},Fe^{3+},Al,Mg)_{2\text{-}3}(Si,Al)_2O_5(OH)_4$	0.99	1.63
Apatite	$Ca_5(PO_4)(F,Cl,OH)$	0.31	0.24
Allanite	$(Ce,Ca,Y)_2(Al,Fe^{3+})_3(SiO_4)_3(OH)$	0.08	0.48
Calcite	$CaCO_3$	0.01	0.61
Pyrite	Fe_2S_2	4.80	2.67
Pyrrhotite	$Fe^{2+}{}_{0.95}S$	2.50	0.31
Alteration of Pyrrhotite	-	0.03	
Chalcopyrite	$CuFe^{2+}S_2$	0.26	0.40
Magnetite	$Fe^{3+}{}_2Fe^{2+}O_4$	44.61	8.39
Uraninite	UO_2	0.00	
Goethite	$Fe^{3+}O(OH)$	2.88	
Process_metal	-	0.01	
Unknown	-	0.13	0.21
Total		100.00	100.00

4 BENCH-SCALE TESTS

4.1 Bench- scale Flotation Test Work

In total, thirteen (13) bench scale tests were conducted with VE2 and VE3 samples. The main objectives of the bench scale tests were to optimize the reagent consumptions in different flotation stages, determine optimal grinding fineness in copper cleaning flotation and study pyrite and pyrrhotite flotation.

The main results of the tests are presented in Tables 3, 4, 5 and 6. The complete flotation test work report is presented in Appendix 1.

The feed grades of the VE2 batch sample and the pilot feed sample differed to each other. The iron grade of the batch feed was higher (ca. 39.5 % as average) whereas the copper grade was lower (0.096 % as average).

The main results of the bench scale tests (tests 0-3) with the VE2 batch feed sample are presented in Table 3.

The best results were achieved in test 3 where the copper grade of CuCC4 was 16.5 % with 67.6 % recovery. The gold content in the copper concentrate was 7.11 g/t but the recovery only 5.4 %. The iron grade of WLC4 was 65.1 % with 79 % recovery. Regrinding was not applied between magnetic separation stages as was done in the pilot process.

Table 3. The Main Results of VE2 Batch Sample (Tests 0-3)

Test No.	Product	Mass Rec%	Cu%	RCu%	Fe%	RFe%	S%	RS%	Sat%	RSat%	Au (g/t)	RAu%	d80 (µm)
Test 0	**Feed**		0.098		39.9		3.13		47.79				**148**
	CuRC	4.6	1.970	91.6	42.0	4.8	32.80	47.7	12.12	1.2	-	-	
	PyRC	3.7	0.080	2.9	41.4	3.9	25.30	29.6	16.13	1.2			
	PoRC	1.0	0.040	0.4	55.1	1.4	17.81	5.6	44.86	0.9			
	WLT1-3	37.7	0.010	3.9	9.0	8.5	0.96	11.6	0.80	0.6			
	WLC4	51.8	0.002	1.1	62.5	81.2	0.33	5.5	88.39	95.9			
Test 1	**Feed**		0.103		39.8		3.13		46.39		0.39		109
	CuRC	6.3	1.540	94.7	43.9	7.0	32.30	65.3	13.80	1.9	0.05	0.8	
	PyRC	2.5	0.067	1.6	41.4	2.6	18.80	15	22.00	1.2	2.56	16.4	
	PoRC	1.5	0.025	0.4	57.3	2.2	18.18	8.9	47.43	1.3	2.87	11.2	
	WLT1-3	40.8	0.007	2.8	8.8	9.0	0.72	9.4	0.69	0.6	0.59	61.8	
	WLC4	48.1	0.001	0.5	65.4	79.0	0.09	1.3	91.27	94.7	0.05	6.4	
Test 2	**Feed**		0.091		38.4		2.90		44.73		0.43		139
	CuRC	7.7	1.100	92.9	43.1	8.6	28.75	76.2	17.56	3.0	4.08	72.8	
	CuCC3	0.9	7.910	80.2	37.6	0.9	45.10	14.3	0.73	0.0	4.56	9.7	39
Test 3	**Feed**		0.091		39.7		3.12		46.56		0.49		109
	CuRC	4.9	1.720	93.2	41.0	5.1	30.15	47.8	11.86	1.3	1.44	14.6	
	CuCC4	0.4	16.500	67.6	35.3	0.3	41.00	4.9	0.69	0.0	7.11	5.4	39
	PyRC	1.6	0.097	1.7	32.7	1.3	17.80	9.0	10.92	0.4	0.85	2.7	
	PoRC	3.2	0.025	0.9	56.9	4.6	19.19	19.5	46.06	3.4	8.94	58.1	
	WLT1-3	41.6	0.009	4.1	9.5	9.9	1.68	22.4	0.69	0.6	0.14	12.2	
	WLC4	48.1	0.000	0.1	65.1	79.0	0.08	1.2	91.52	94.6	0.05	4.9	

The main results of the flotation tests conducted with pilot process samples are presented in Table 4. A sample of Derrick u/s was taken on Sept. 13 for a bench scale test (Test 4). The Cu-grade of the copper concentrate was 16.5 % at 51 % recovery. Respectively, the Fe-grade of the iron concentrate was only 64.7 %.

Feed for test 5 was taken from the Cu rougher flotation conditioner, after the addition of flotation reagents. Before flotation, pH was increased to 8 with the main idea to study whether pyrite content in rougher concentrate could be decreased at higher pH. The results were

unsuccessful, Cu-grade of the copper concentrate was only 9.3 % at 69.4 % recovery. Respectively, the sulphur grade of the copper concentrate was high as 46.9 % which indicates high pyrite content in the concentrate.

Test 6 was a cleaning test for pyrite rougher concentrate taken from the pilot process. After two-stage cleaning, the sulphur grade of the concentrate was ca. 53 %.

In tests 7 and 8, the main idea was to test different flotation parameters in pyrite flotation. Feed for the tests was pyrite rougher concentrate from the pilot process. The sulphur grades of the tailings of tests 7 and 8 were 1.3 % and 1.25 %.

Tests 9 to 11 were performed on copper rougher tailings. In test 9, low intensity magnetic separation was applied first and pyrite flotation was conducted for the WLIMS tails. With this modification, the sulphur grade decreased to 0.7 % in the pyrite rougher tailings. Respectively, in tests 10 and 11, pyrite flotation was placed before LIMS, as in the base case flowsheet. The sulphur grade of 0.9 % in the rougher pyrite tailings was resulted in test 10.

June 6, 2018

Table 4. The Main Results of Bench Tests with Pilot Process Samples (Tests 4-11)

Test No.	Product	Mass Rec%	Cu%	RCu%	Fe%	RFe%	S%	RS%	Sat%	RSat%	Au (g/t)	RAu%
Test 4	**Feed**		0.128		34.7		3.97		37.96		0.19	
	CuRC	7.1	1.610	88.9	42.5	8.6	32.02	57	11.89	2.2	1.17	42.8
Feed - VE2	**CuCC3**	0.4	16.500	51.00	34.4	0.4	43.70	4.4	0.54	0.0	5.95	12.1
Pilot Derrick u/s	**PyRC**	2.4	0.149	2.7	34.0	2.3	21.60	12.8	9.75	0.6	0.34	4.1
Sept 13, 2017	**PoRC**	2.1	0.040	0.7	54.0	3.3	15.70	8.4	45.48	2.5	0.51	5.6
	WLT1-3	49.0	0.020	7.7	9.2	13	1.73	21.4	0.62	0.8	0.13	32.8
	WLC4	39.0	0.000	0	64.7	72.6	0.04	0.4	91.39	93.8	0.06	12.1
Test 5	**Feed**		0.130		32.1		3.72		33.60		0.15	
	CuRC	9.1	1.300	90.6	41.4	11.7	30.92	70.6	12.57	3.4	0.66	38.7
Feed - VE2	**CuCC4**	1.0	9.250	69.40	38.1	1.2	46.92	11.5	0.67	0.0	2.65	16.7
Cu flotation feed	**PyRC**	4.5	0.090	3.1	32.4	4.6	17.54	20	12.70	1.7	0.33	9.7
Sept 14, 2017	**PoRC**	1.1	0.034	0.3	49.2	1.7	2.90	9.3	47.70	1.5	1.02	7.2
	WLT1-3	49.7	0.014	5.4	8.3	12.9	0.49	6.4	0.58	0.9	0.12	38.8
	WLC4	35.1	0.002	0.5	63.1	69.1	0.04	0.4	88.00	92.4	0.02	4.6
Test 6	**Feed**		1.610		45.2		51.21					
Feed - VE2	**PyCC2**	87.0	1.650	87.5	44.5	85.8	52.78	89.7				
PyCC1 - Sept. 20, 2017 - 2:00PM	**PyCC3**	71.5	1.470	64.00	44.2	70	53.60	74.8				
Test 7	**Feed**		0.028		30.1		2.48					
Feed - VE2 PyRT - Sept. 20, 2017 - 2:00PM	**PySCT**	94.9	0.018	60.5	29.6	93.2	1.32	50.6				
Test 8	**Feed**		0.028		30.7		2.57					
Feed - VE2 PyRT Sept. 20, 2017 - 2:00PM	**PySCT**	93.7	0.018	61.2	30.1	91.8	1.25	45.6				
Test 9	**Feed**		0.022		27.0		2.04					
Feed - VE2	**WLC3**	34.7	0.007	10.9	60.4	77.7	2.18	37.2	79.75	98.5		
CuRT	**WLT1-3**	65.3	0.030	89.10	9.2	22.3	1.96	62.8	0.63	1.5		
Sept. 27, 2017	**PyRT4**	57.7	0.013	33.8	8.0	17.1	0.71	20.1	0.62	1.3		
Test 10	**Feed**		0.031		27.85		2.39					
Feed - VE2	**PyRC1-5**	6.3	0.346	70.1	40.38	9.2	24.09	63.9				
CuRT - Sept. 27, 2017	**PyRT5**	93.7	0.010	29.9	27.00	90.8	0.92	36.1				
Test 11	**Feed**						2.36					
Feed - VE2 CuRT	**PyRT4**	95.2					1.03	41.6				

The main objective of test 12 was to study whether calcium bearing minerals could be removed by flotation before the pyrite and pyrrhotite flotation stages. Lilaflot 1400 was used as collector and the used dosage was 250 g/t added in three stages (50 g/t, 100 g/t and 100 g/t). The recovery of calcium oxide was 4.4 % with 9.4 % grade. Feed for the test was VE2 pilot plant feed.

Table 5. The Main Results of test 12

Test No.	Product	Mass Rec%	Cu%	RCu%	Fe%	RFe%	S%	RS%	atmagan	Satmagar	CaO%	RCaO%	d80 (µm)
Test 12	**Feed**												90
	CuRC	4.6	1.930	91.3	38.0	4.4	30.50	49.5	5.87	0.6	3.98	2.2	
Feed - VE2	**SiliRC**	3.9	0.071	2.90	27.6	2.7	9.14	12.8	15.36	1.3	9.39	4.4	
Pilot Feed	**PyRC**	4.8	0.056	2.80	38.4	4.7	12.20	20.8	26.08	2.7	6.41	3.7	
	WLT1-3	38.8	0.006	2.40	8.2	8.1	0.23	3.1	0.76	0.6	15.5	81.2	
	PoRC	2.8	0.020	0.60	61.8	4.4	12.10	12.2	62.63	3.8	1.35	0.5	
	WLC4	44.7	0.000	0.00	66.8	75.6	0.10	1.6	94.21	90.9	1.36	7.3	

The VE3 pilot sample was used as feed in test 13. The primary grinding fineness was d80 of 124 µm. The Cu-grade of the copper concentrate was 13.2 % at 71 % recovery. Respectively, the gold content was 4.8 g/t at 15.3 % recovery. The Fe-grade of WLC4 was 64.2 % at 35.8 % recovery.

Table 6. Results of Test 13

Test No.	Product	Mass Rec%	Cu%	RCu%	Fe%	RFe%	S%	RS%	Sat%	RSat%	Au (g/t)	RAu%	d80 (µm)
Test 13	**Feed**		0.066		16.5		1.36		9.25		0.11		124
	CuRC	4.9	1.150	84.8	29.7	8.8	23.60	84.7	4.91	2.6	0.06	26.1	Regrinding
Feed - VE3	**CuCC3**	0.4	13.200	71.00	34.7	0.7	40.90	10.7	0.36	0.0	4.84	15.3	
Pilot Feed	**PyRC**	2.5	0.154	5.8	20.3	3.1	3.65	6.7	9.99	2.7	0.24	5.3	
	PoRC	0.6	0.033	0.3	59.6	2	5.15	2.1	68.70	4.2	1.2	6	
	WLT1-3	82.7	0.007	8.8	10.0	50.1	0.10	6.4	0.19	1.7	0.08	58.9	
	WLC4	9.2	0.002	0.3	64.2	35.8	0.04	0.3	89.35	88.8	0.03	2.5	

4.2 Pre-Concentration Test Work on Coarse Feed Fractions

Two different pre-concentration tests were conducted for the coarse feed, namely dry magnetic separation and dense media separation.

In dry magnetic separation, the -100 mm crushed ore was gradually crushed into fractions -100 mm, -60 mm and into -40 mm. After each crushing, the fraction -10 mm was screened away, to be reported in grinding. In each test, the amount of fresh -10 mm fraction was recorded, and it was added to the previous fines in the same proportion. The XRF assays were done in each test for magnetic fraction (Mags), non-magnetic fraction (Non-Mags) and for -10 mm fraction.

Separation was done with a belt separator, and the magnetic field was 0.23T. Tables 7 and 8 present the balance calculations for VE2 and VE3 samples with different feed sizes.

June 6, 2018

Table 7. Dry Separation Balances, VE2 Sample

10 - 100 mm	VE2		Fe		Cu		SiO2		Satmagan		Au	
Product	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	g/t	Rec%
Mags	2554	73.8	31	75.6	0.129	65.8	30.6	72.2	34.12	78.5	0.055	73.1
Non-Mags	309	8.9	12	3.5	0.161	9.9	46	13.1	7.2	2.0	0.08	12.9
-10 mm	600	17.3	36.3	20.8	0.203	24.3	26.5	14.7	36.16	19.5	0.045	14.0
Calc Feed	**3463**	**100.0**	**30.2**	**100.0**	**0.145**	**100.0**	**31.3**	**100.0**	**32.1**	**100.0**	**0.055**	**100.0**

10 - 60 mm	VE2		Fe		Cu		SiO2		Satmagan		Au	
Product	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	g/t	Rec%
Mags	1823	69.8	31.3	69.4	0.112	58.5	30.3	69.7	34.5	71.5	0.040	61.3
Non-Mags	136	5.2	8.34	1.4	0.155	6.0	51.2	8.8	2.7	0.4	0.055	6.3
-10 mm	201	7.7	37.8	9.2	0.160	9.2	25.2	6.4	41.3	9.4	0.090	15.2
Cumulative -10mm	654	25.0	36.8	29.2	0.190	35.5	26.1	21.5	37.7	28.0	0.059	32.4
Calc Feed	**2613**	**100.0**	**31.5**	**100.0**	**0.134**	**100.0**	**30.3**	**100.0**	**33.7**	**100.0**	**0.045**	**100.0**

10 - 40 mm	VE2		Fe		Cu		SiO2		Satmagan		Au	
Product	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	g/t	Rec%
Mags	614	63.0	28.8	60.0	0.097	49.0	32.3	64.9	31.5	61.7	0.060	57.3
Non-Mags	43	4.4	8.02	1.2	0.166	5.9	51.3	7.2	2.7	0.4	0.055	3.7
-10 mm	74	7.6	33.4	8.4	0.117	7.1	28.9	7.0	36.2	8.5	0.145	16.7
Cumulative -10mm	318	32.6	36.0	32.6	0.173	32.6	26.8	27.8	37.4	37.9	0.079	39.0
Calc Feed	**975**	**100.0**	**30.2**	**100.0**	**0.125**	**100.0**	**31.3**	**100.0**	**32.2**	**100.0**	**0.066**	**100.0**

Table 8. Dry Separation Balances, VE3 Sample

10 - 100mm	VE3		Fe		Cu		SiO2		Satmagan		Au	
Product	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	g/t	Rec%
Mags	1976	49.7	18.9	59.6	0.062	52.5	39	47.2	13.51	76.3	0.035	56.5
Non-Mags	1665	41.9	11.6	30.8	0.036	25.7	43.8	44.7	2.83	13.5	0.025	34.0
-10 mm	332	8.4	18.2	9.6	0.153	21.8	40.1	8.2	10.73	10.2	0.035	9.5
Calc Feed	**3973**	**100.0**	**15.8**	**100.0**	**0.059**	**100.0**	**41.1**	**100.0**	**8.8**	**100.0**	**0.031**	**100.0**

10 - 60mm	VE3		Fe		Cu		SiO2		Satmagan		Au	
Product	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	g/t	Rec%
Mags	1855	53.3	18.4	61.3	0.053	48.1	39.5	51.7	13.1	75.4	0.020	47.0
Non-Mags	1087	31.2	10.5	20.5	0.031	16.5	43.4	33.3	1.2	4.2	0.020	27.5
-10 mm	248	7.1	19.6	8.7	0.113	13.7	38.7	6.8	13.9	10.7	0.040	12.6
Cumulative -10mm	539	15.5	18.8	18.2	0.135	35.5	39.4	15.0	12.2	20.4	0.037	25.5
Calc Feed	**3481**	**100.0**	**16.0**	**100.0**	**0.059**	**100.0**	**40.7**	**100.0**	**9.3**	**100.0**	**0.023**	**100.0**

10 - 40mm	VE3		Fe		Cu		SiO2		Satmagan		Au	
Product	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	g/t	Rec%
Mags	1166	48.4	17.7	54.1	0.067	47.1	40.0	47.1	11.6	64.5	0.040	51.2
Non-Mags	678	28.1	10.2	18.1	0.028	11.4	44.3	30.3	0.8	2.5	0.025	18.6
-10 mm	194	8.0	18.2	9.3	0.096	11.2	39.2	7.7	12.1	11.2	0.070	14.9
Cumulative -10mm	567	23.5	18.6	23.5	0.121	23.5	39.4	22.6	12.2	33.0	0.048	30.2
Calc Feed	**2411**	**100.0**	**15.8**	**100.0**	**0.069**	**100.0**	**41.1**	**100.0**	**8.7**	**100.0**	**0.038**	**100.0**

Dense media separations were conducted for different size fractions of the screened -100 mm crushed ore samples. -10 mm material was taken out, and size fractions 60-100 mm, 40-60

mm and 10-40 mm were subjected into dense media separation. Tests were done with Erickson cone using a ferrosilicon density of 3.2 kg/l. Tables 9 and 10 summarize the dense media separation results for VE2 and VE3 samples.

Table 9. DMS Separation Balances, VE2 Sample

	VE2		Fe		Cu		SiO2		Satmagan		Au		Fraction
Product	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	
Sink	36.6	14.6	40.3	19.8	0.192	20.6	23.5	10.8	46.0	21.1	0.100	12.2	60 - 100 mm
Float	36.5	14.6	14.7	7.2	0.145	15.5	42.1	19.4	11.0	5.0	0.140	17.0	
Sink	52.85	21.1	42.3	30.0	0.125	19.3	22.1	14.7	48.9	32.3	0.170	30.0	40 - 60 mm
Float	33.8	13.5	13.6	6.2	0.097	9.6	44.2	18.8	11.3	4.8	0.090	10.1	
Sink	49.85	19.9	41.4	27.7	0.129	18.8	22.7	14.3	47.8	29.8	0.110	18.3	10 - 40 mm
Float	33.8	13.5	12	5.4	0.123	12.2	46	19.6	8.2	3.5	0.090	10.1	
-10 mm	7.45	3.0	37.6	3.8	0.181	3.9	25.6	2.4	37.8	3.5	0.090	2.2	-10 mm
Calc Feed	**250.85**	**100.0**	**29.7**	**100.0**	**0.136**	**100.0**	**31.6**	**100.0**	**31.9**	**100.0**	**0.120**	**100.0**	
Sink + -10mm	146.75	58.5	41.3	81.2	0.146	62.7	22.8	42.2	47.2	86.7	0.128	62.7	
Float	104.1	41.5	13.5	18.8	0.122	37.3	44.0	57.8	10.2	13.3	0.108	37.3	

With dense media separation as pre-concentration method, it was possible to increase the Fe grade of VE2 sample till 41.3% with a recovery of 81.2%, and the mass rejection was 41.5% from the feed.

Table 10. DMS Separation Balances, VE3 Sample

	VE3		Fe		Cu		SiO2		Satmagan		Au		Fraction
Product	kg	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	Grade%	Rec%	
Sink	29.6	9.7	21.3	13.3	0.049	10.2	37.2	8.7	16.9	19.3	0.070	11.4	60 - 100 mm
Float	81.75	26.8	12.3	21.2	0.035	20.0	43.5	28.2	4.6	14.5	0.040	17.9	
Sink	31.5	10.3	23.8	15.8	0.079	17.4	35.6	8.9	19.0	23.0	0.090	15.5	40 - 60 mm
Float	78.95	25.8	12.9	21.5	0.024	13.3	43.1	27.0	5.0	15.1	0.050	21.6	
Sink	34.4	11.3	20.6	15.0	0.088	21.2	37.5	10.2	13.8	18.3	0.080	15.1	10 - 40 mm
Float	40.4	13.2	11.6	9.9	0.032	9.1	44.3	14.2	4.0	6.3	0.070	15.5	
-10 mm	8.9	2.9	17.7	3.3	0.142	8.9	40.6	2.9	10.1	3.5	0.060	2.9	-10 mm
Calc Feed	**305.5**	**100.0**	**15.5**	**100.0**	**0.047**	**100.0**	**41.3**	**100.0**	**8.5**	**100.0**	**0.060**	**100.0**	
Sink + -10mm	104.4	34.2	21.5	47.4	0.079	57.6	37.1	30.7	15.9	64.1	0.078	44.9	
Float	201.1	65.8	12.4	52.6	0.030	42.4	43.5	69.3	4.6	35.9	0.050	55.1	

Dense media separation did not work with VE3 sample as well as with VE2 sample. Fe grade raised from 15.5% to 21.5%, but the recovery remained low, being only 47.4%. The mass rejection from the feed was 65.8%.

5 PILOT PLANT OPERATION

5.1 Circuit Configuration

The present pilot plant campaign for Hannukainen ore samples was completed within three weeks period, starting on 11th October and ending on 29th October, 2017. Daily running times varied from 8 to 24 hours. Various flow sheet options were constructed and tested regarding the magnetite beneficiation circuit, as will be described in the following chapters.

The pilot plant process was operated completely continuously from crushed ore sample to final products. The pilot plant processing stages were:

Rod mill – Ball Mill grinding and two-stage screen classification of the ore down to flotation fineness, followed by a copper rougher flotation, where chalcopyrite was floated. The rougher concentrate was reground in two ball mills and cleaned three to four times. The first cleaner tailing was fed to scavenger stage, from where the scavenger concentrate was returned to Cu re-grinding and the tails were taken out from the circuit into pyrite flotation, and the later cleaner tailings were returned back to preceding cleaner stages.

The copper rougher flotation tailing and the first cleaner scavenger tailing were introduced in the beginning of the pilot plant run into pyrite flotation circuit with one cleaning stage. Another process alternative was also tested, where the Cu flotation circuit tailing was fed directly into magnetic separation circuit.

The magnetic separation circuit consisted of a three-stage wet low-intensity magnetic separation followed by a four-stage flotation process for pyrrhotite removal. The two first magnetic separations were done with Roxon separators with the same physical parameters as the latter Sala separator. The third separation stage was done with a Sala WS903 CC separator. All separators were installed with counter-current tanks. The tailings from second and third separators were combined and pumped back to the first separator in the beginning, later the circuit was operated open. In the beginning of the pilot plant operation, the magnetic separation stages were operated coarse, but later a re-grinding stage was added in order to achieve acceptable grade in magnetite concentrate.

The final concentrate from the LIMS separation, LC3, was conditioned with flotation reagents and subjected to reverse flotation, where pyrrhotite was removed as flotation concentrate, PoRC. The flotation concentrate was pumped into a Supaflo HR thickener.

The flotation tailing (cleaned magnetite concentrate), PoRT, was introduced to the fourth LIMS separation stage using a Sala WS 903 DM separator. The tailing of the last separation stage was combined to other magnetic separation tailing fractions and pumped to tailings. The magnetite concentrate, LC4, was collected into mixing tanks and filtered for further test work.

In pilot plant flotation test work, the following reagents were used in different flotation stages:

Reagent	Abbreviation	Purpose	Supplier	
Danafloat 245	DF245	Collector	Cheminova	
Danafloat 507	DF507	Collector	Cheminova	
Sodiun Ethyl Xanthate	SEX	Collector	Kemcore International Ltd	
Potassium Amyl Xanthate	PAX	Collector	General Quimica S.A.	Senmin, Hoechst
Sodium Isobutyl Xanthate	SIBX	Collector	General Quimica S.A.	
Dowfroth 250	Dow 250	Frother	Dow Europe GmbH	
Methyl Isobutyl Carbinol	MIBC	Frother	Flomin Inc	
Flotanol C7	C7	Frother	Clariant	
Carboxy Methyl Cellulose	CMC	Depressant	Noviant Oy	Charles Tennant & Co.
Calsium Hydroxide	Ca(OH)2	ph Regulator	Nordkalk	
Sulfuruc Acid	H2SO4	ph Regulator	Algol	

5.2 11.9.2017 Monday, VE2 Ore type

On the first day of the pilot run, the process was operated from 10:10 AM till 20:00 PM. Feed sample was VE-2. The purpose of the first operating day was to fill up the circuit, ensure the proper mechanical operation and the gain the first operating data from grinding circuit and magnetic separation products. Sampling Log, screen assays and XRF assays are presented in Appendix 2.

For grinding, the ore blend was unloaded from the stockpile into the daily feed bin, equipped with load cells and a variable speed driven belt feeder. The feed rate was monitored by the declining weight of the bin and by manual measurements. Feed for the plant was -6 mm crushed ore, and the feed rate was 750 kg/h. Ore and 700 l/h water were fed to the rod mill. Rod mill was working in closed circuit with a Skako vibrating scree, equipped with # 1 mm screen deck, and the mill discharge was returned to the screening. The screen under size was fed to secondary screening, Derrick #200 µm vibrating screen. Secondary screen under flow was fed to copper flotation. Secondary screen O/S was fed to ball mill, and the ball mill discharge was pumped back to primary screening. The pilot plant flow sheet on the 11rd of September is presented in Figure 1.

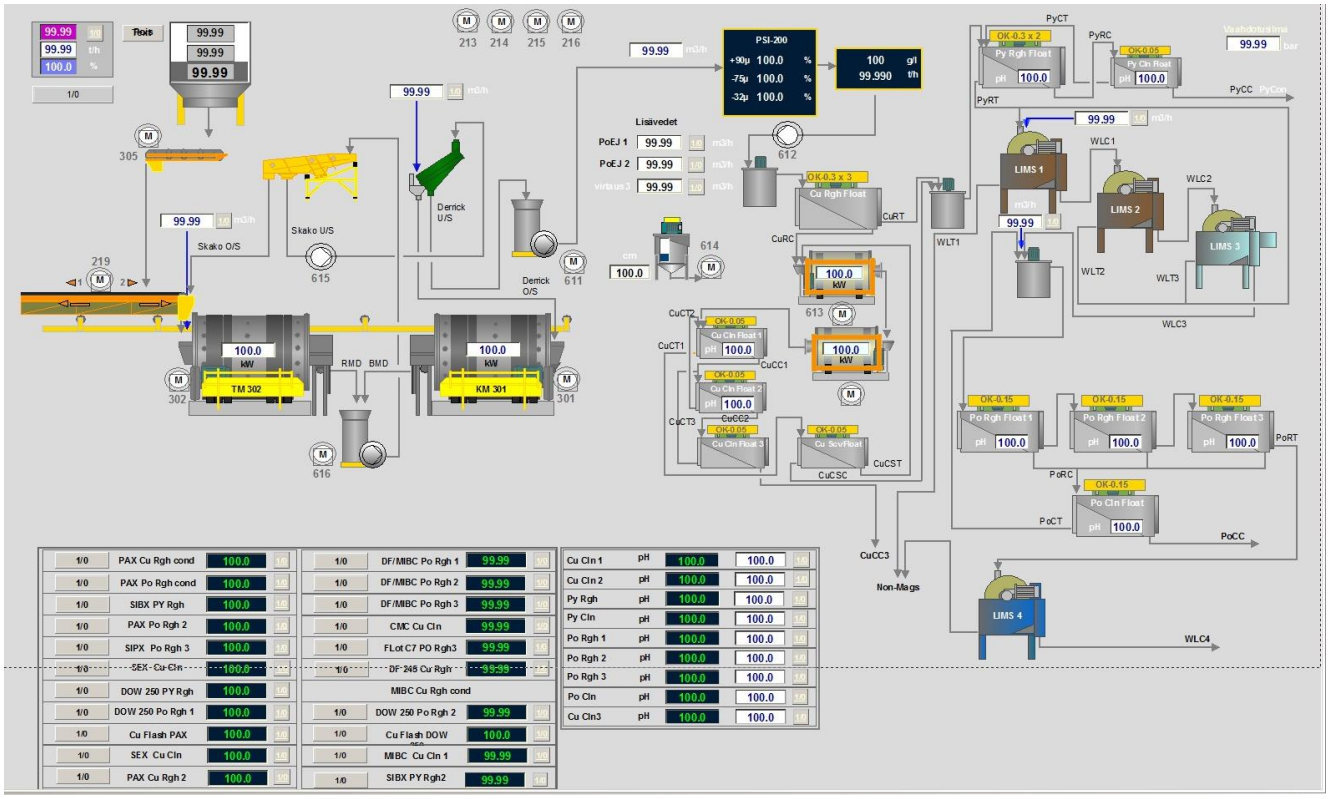

Figure 1. The Basic Pilot Plant Flow Sheet.

The rod mill used was SALA SSR D1000 x 1800 RP-DR with an inner diameter of 0,84 m and an inner length of 1,75 m, which makes 1,13 BMU (Base Mill Units). The mill discharge was overflow type, and the linings were made of rubber with a lifter height of 35 mm. The rod charge consisted of Ø 75 mm and Ø 50 mm rods with a power draw of 5,9 kW. The mill was equipped with 15 kW motor, and the idling power was measured to be 2,0 kW. The rod charge was ca. 600 kg. The volumetric load of rods was 10 %. The mill revolution speed was 32 rpm, being 71 % of the critical speed.

The rod mill product was pumped to a #1,0 mm Skako vibrating screen, 600 mm x 800 mm, with a screening area of 0,48 m^2. The screen overflow was returned to rod mill. The screen U/S was fed to secondary screening with a Derrick J24-36MS-1 vibrating screen, 600 mm x 600 mm, screening area 0,36 m^2.

The ball mill was a SALA SSR D1000 x 1500 BG-WR with an inner diameter of 0,84 m and inner length of 1,35 m (0,87 BMU) with a grate discharge with 24 openings of 8 mm x 30 mm in six segments. Linings were made of rubber, and the lifter height was 70 mm. The mill was equipped with 15 kW motor, and the idling power was measured to be 2.0 kW. The steel ball charge (Ø 25-40 mm) was ca. 750 kg with the volumetric filling of 19 %, giving 7.8 kW power draw. Mill speed was 35.7 rpm, or 80 % from the critical speed.

June 6, 2018

In the beginning, the classifying screen U/S was finer than the targeted value (P(80) 120 µm) having a P(80) value of 81 µm and the circulating load in grinding was increasing, so the ball load and classifying screen size (200 µm -> 250 µm) were adjusted. In the end of the day, the P(80) value was 137 µm.

During the day, the copper flotation circuits were only tested, and the reagent dosage pumps were calibrated. The following paragraphs describe the initial configuration of the circuits.

Derrick screen U/S was fed to copper rougher flotation through a flow meter and particle size analyser. P(80) of the copper flotation feed was in the end of the day 137 µm, and the feed pulp density was 26.5 %. The Cu rougher feed was conditioned in a 200 litre conditioning tank with PAX dosage of 80 g/t for 5 minutes before fed to flotation. The copper rougher flotation circuit consisted of two 2 x 500 litre OK flotation machines with the total cell volume of 1000 litres. The volumetric feed rate was 2.3 m^3/h, so the mean residence time in copper roughers was 26 min. MIBC was added to flotation cell, 55 g/t.

Cu rougher concentrate was pumped to a regrinding mill together with 175 g/t milk of lime. P(80) of the regrinding mill discharge was 24 µm. The copper cleaning circuit consisted of four identical 1 x 50 litre OK flotation machines in three stages. The first cleaner tailings had 1 x 50 litre scavenging flotation, and the cleaner scavenger tails were removed to pyrite flotation. The first cleaner concentrate was further cleaned twice, and the cleaner tails were returned back to the preceding cleaning stage. pH in the cleaner circuit was controlled with a weak solution of milk of lime. The pH levels were 11,8 in the first cleaning, 11,8 in the second and 11,9 in the third cleaning stage. 2 g/t CMC was added to the first cleaner, no other reagents were used in the Cu cleaning circuit.

During the afternoon it was noted that the coarse flotation feed caused sanding in the rougher flotation cells, so it was decided to replace the two 500 litre units with three 2 x 150 litre flotation machines. Also the thin froth made it difficult to operate the larger flotation cells.

The same change was done in pyrite flotation, where 2 x 500 litre machine was replaced with 2 x 2 x 150 litre units.

Due to the coarse flotation feed, the classifying screen was decided to change from 250 µm to 215 µm.

Figure 2 presents the main grinding circuit particle size distributions.

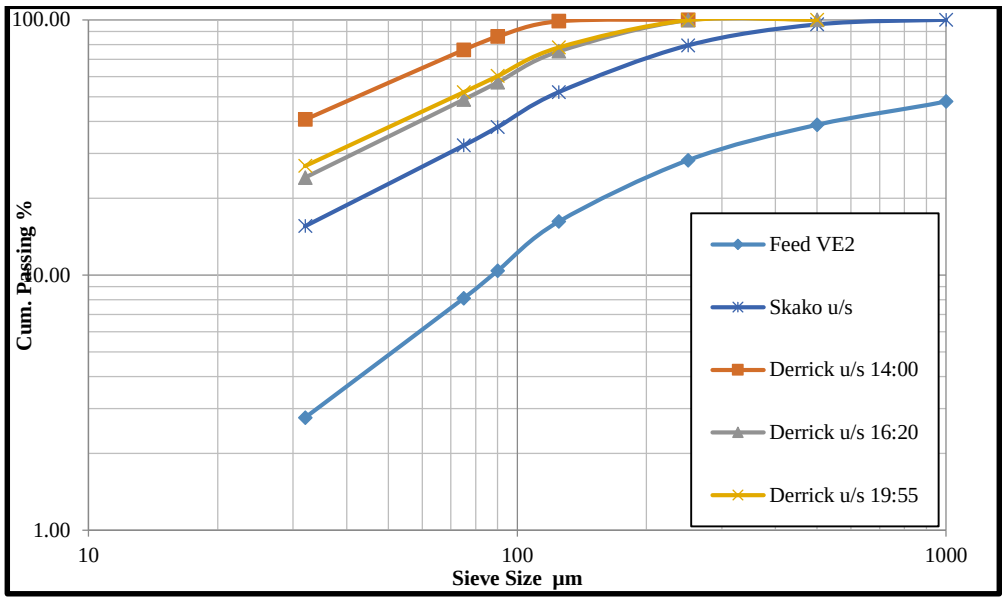

Figure 2. Process PSD's 11.9.2017.

At 19:00, samples from Cu rougher and first cleaner were collected. Table 11 presents the mass balance calculation for copper.

Table 11. Cu Mass Balance 11.9.2017 Hour 19:00

	Cu %	Rec.	Mass
Feed	0.14	100.0	100.0
CuRC	2.86	62.6	3.1
CuRT	0.054	37.4	96.9
CuCC1	9.04	40.5	0.4
CuCT1	1.27	22.1	2.7

5.3 12.9.2017 Tuesday, VE2 Ore type

As stated in previous chapter, some changes in flow sheet were done before start.

Cells in flotation circuit:

- Cu roughers, 3 * 2 * 150 litres,
- Pyrite roughers, 2 * 2 * 150 litres.

Derrick screen replacement 250 µm -> 215 µm.

The daily operating hours were 8:50 – 21:20.

June 6, 2018

After the replacement of the secondary screen mesh, the P(80) value for the copper rougher flotation feed was 114 µm. Figure 3 presents the main grinding circuit particle size distributions together with Cu re-grinding PSD's.

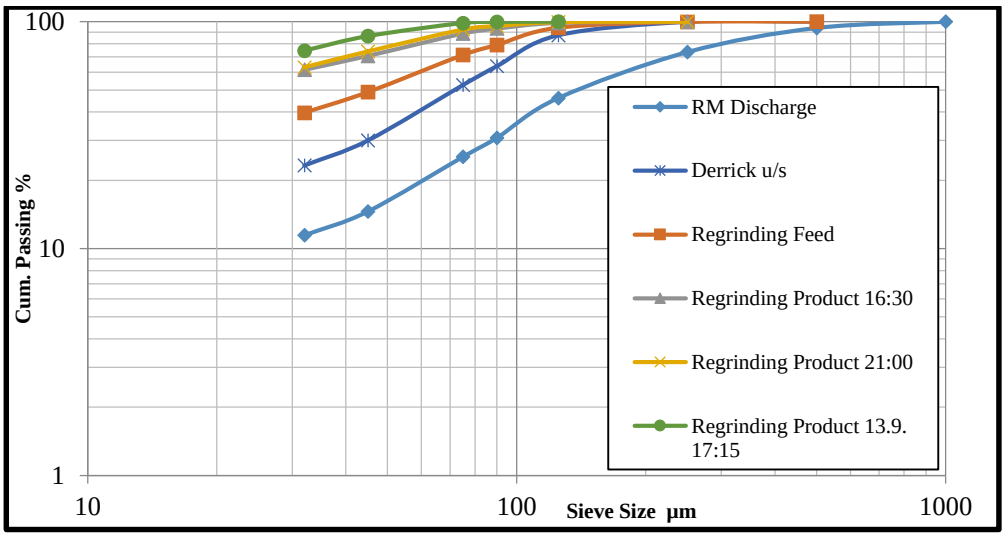

Figure 3. Process PSD's 12.9.2017.

The feed for the Cu re-grinding had a P(80) value of 92 µm, and the corresponding values for the cleaner feed were 60 and 55 µm, respectively. On the following day a second mill was taken in operation in re-grinding, and P(80) value after that decreased to 38 µm.

The pilot plant run was continued to fill up the new copper and pyrite flotation cells. The Cu rougher tailings and cleaner scavenger tailings were pumper into a 200 litre conditioning tank prior the pyrite flotation. The reagent dosages, calculated against pilot plant feed (0.75 tph) were as presented in Table 12.

June 6, 2018

Table 12. Reagents 12.9.

Date:		12.9.	12.9.
Time:		16:30	19:00
Reagent	Feed Point	g/t	g/t
CMC	**Cu Cln 1**		2
PAX	**Cu Rgh 1**	84	120
MIBC	**Cu Rgh 1**	40	40
DF 245	**Cu Rgh 1**	44	60
PAX	**Po. Flot 1**		200
PAX	**Po. Flot 2**		100
DF/MIBC	**Po. Rgh 1**		15
DF/MIBC	**Po. Rgh 2**		40
SIBX	**Po. Rgh 3**		100
SIBX	**Py Rgh**	148	148
DOW 250	**Py Rgh**	30	30

The main samples from the copper flotation were collected at 16:50, and the corresponding Cu-balance is presented in Table 13.

Table 13. Cu Mass Balance 12.9.2017 Hour 16:50

Product	kgh	wt%	Cu %	Rec%	Fe %	Rec%	S %	Rec%	Satmagan %	Rec%	SiO2 %	Rec%
Derrick u/s	750.0	100.0	0.14	100.0	35.47	100.0	3.77	100.0	39.21	100.0	27.17	100.0
CuRC	29.7	4.0	2.96	81.9	42.03	4.7	37.34	39.1	9.97	1.0	7.03	1.0
CuRT	720.3	96.0	0.03	18.1	35.20	95.3	2.39	60.9	40.41	99.0	28.00	99.0
CuCC3	7.0	0.9	11.20	73.2	38.90	1.0	47.50	11.8	0.92	0.0	0.32	0.0
CuCST	22.7	3.0	0.41	8.7	43.00	94.3	34.20	49.1	12.77	99.0	9.10	99.0

The Cu grade in rougher concentrate was 2.96%, and the corresponding values in cleaner concentrates were: CuCC2 9.53%, CuCC3 11.2%. The copper recovery in the final concentrate was 73.2%. Copper losses in rougher tailings was 18.1% and 8.7% in the first cleaner scavenger tailings (CuCST).

After filling up the preceding flotation stages, the three-stage LIMS (low intensity magnetic separation) circuit was taken in operation at 17:25. The circuit was operated closed, and the cleaner tails were returned back to first LIMS. Pyrite rougher tailings were pumped to three-stage wet low intensity magnetic separation. The first magnetic concentrate was cleaned twice, and the cleaner tailings were combined and pumped back to the first separator. In the magnetic separation circuit, three wet low-intensity magnetic separator were working in series, cleaning the magnetic product of the previous stage.

The pulp densities in separators were controlled by washing sprays and by additions of fresh water into the concentrate pumps of separators one and two. In separation stages 1 and 2, Roxon low-intensity magnetic separators were used. The drum diameter was 900 mm, and the

drum width was 540 mm. Drum speed was 23 rpm. In the third LIMS separation, Sala WS903 separator with drum diameter of 900 mm and width of 300 mm was used. The drum speed was 23 rpm, controlled with a frequency controller. All separators had counter-current tanks. After the one rougher and two cleaning steps of the LIMS concentrate, LC3 was subjected to pyrrhotite flotation process.

20:30 samples were collected from copper-, pyrite- and LIMS circuits. Copper circuit had three cleaning stages. The following table presents the main balances in each stage, Cu-balance in copper flotation, S-balance in pyrite flotation and Fe-balance in magnetic separation. XRF assays are presented in Appendix 2.

Table 14. Stage-wise Mass Balances 12.9.2017 Hour 20:30

Copper Flotation	Cu %	Cu Rec.	Au g/t	Au Rec
Feed	0.14	100.0	0.05	100.0
CuRC	3.01	80.2	0.66	42.4
CuRT	0.03	19.8	0.03	57.6
CuCC3	8.84	69.7	1.79	34.2
CuCT1	0.56	10.5	0.18	8.1
Pyrite Flotation	S %	Rec.		
Feed	3.36	100.0		
PyRC	40.88	17.9		
PyRT	2.80	82.1		
LIMS	Fe %	Rec.		
Feed	32.50	100.0		
WLC3	65.50	80.2		
WLT1	10.70	19.8		

5.4 13.9.2017 Wednesday, VE-2 Ore type

In the morning, 6:00 – 9:00 AM frequency controllers were installed to pyrrhotite flotation cells, because all cells were sanded with coarse LIMS concentrate.

Operating hours were 9:00 – 20:50. Base case flowsheet, so pyrrhotite cleaner tailings (PoCT) was returned back to rougher flotation. Still problems with sanding in the cells.

The grinding circuit was sampled at 12:15 for particle size distribution determinations. The obtained PSD curves are presented in Figure 4. Screening tables are presented in Appendix 2 together with XRF assays and sampling log. Flotation feed (Derrick screen u/s) had a P(80) value of 110µm.

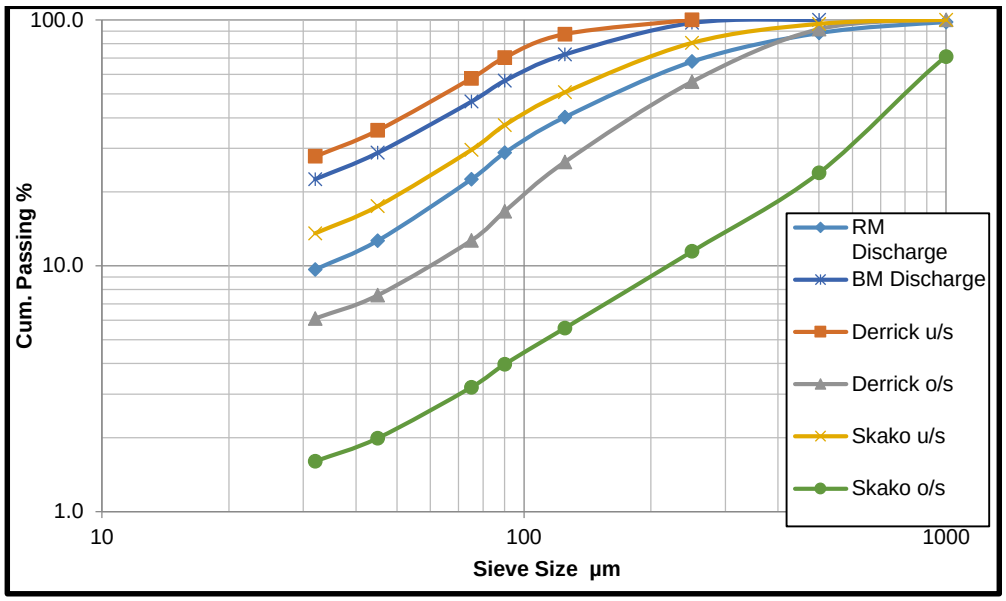

Figure 4. Process PSD's 13.9.2017.

Table 15. Reagents 13.9.

Date:		13.9.	13.9.
Time:		11:00	14:45
Reagent	Feed Point	g/t	g/t
CMC	Cu Cln 1	1	2
PAX	Cu Rgh 1	124	132
MIBC	Cu Rgh 1	40	36
DF 245	Cu Rgh 1	64	68
DOW 250	Po Rgh 1		44
DOW 250	Po Rgh 2		48
PAX	Po. Flot 1	224	320
PAX	Po. Flot 2	96	200
DF/MIBC	Po. Rgh 1	17	20
DF/MIBC	Po. Rgh 2	57	58
SIBX	Po. Rgh 3	100	136
DF/MIBC	Po. Rgh 3	56	60
Flotanol C7	Po. Rgh 3	60	62
SIBX	Py Rgh	176	304
DOW 250	Py Rgh	32	64

At 16:20, the second grinding mill was added to re-grinding of copper rougher concentrate (CuRC), because the P(80) value of the ground product was rather coarse, being 55µm. After installation, the copper cleaner feed had a P(80) of 38µm. Manual flow rate and pulp density measurements from different process stages were done at 18:00, and the results are presented in Table 16.

June 6, 2018

Table 16. Process Flow Rate Measurements, 13.9. Hour 18:00

	Slurry SG g/l (**kg**)	Bucket l	Time s (**min**)	Flow m3/h	Solids %	Solids t/h
Crushed Ore	6.10		30.0	0.24	100.0	0.732
RM Disch.	1590	3.4	12.1	1.01	49.9	0.803
BM Disch.	1367	3.4	9.2	1.33	36.1	0.657
Skako O/S	2438	1.0	52.0	0.07	79.3	0.134
Skako U/S	1505	3.4	6.4	1.91	45.1	1.299
Derrick O/S	1266	1.0	3.0	1.20	28.3	0.429
Derrick U/S	1217	3.4	7.2	1.70	24.0	0.496
WLC1	1560	1.0	6.9	0.52	45.3	0.369
WLT1	1040	3.4	2.6	4.71	5.8	0.282
WLC2	1600	1.0	8.6	0.42	47.4	0.317
WLT2	1010	3.4	11.5	1.06	1.5	0.016
WLC3	1480	1.0	9.5	0.38	41.0	0.230
WLT3	1010	3.4	15.5	0.79	1.5	0.012
WLT 2+3	1010	3.4	7.1	1.72	1.5	0.026
PoFlot Feed	1180	10.0	41.0	0.88	19.3	0.200
PoFlot Tails 1	1150	10.0	31.0	1.16	16.5	0.220
PoFlot Tails 2	1070	10.0	22.7	1.59	8.3	0.140
PoFlot Tails 3	1120	10.0	22.0	1.64	13.5	0.248
WLC4	1150	3.4	12.7	0.96	16.5	0.183
WLT4	1000	10.0	14.8	2.43	0.0	0.000
	Power kW, Gross	Idle kW	Power kW, Net	Energy kWh/t, Net	%	Solids t/h
RM302	**5.30**	**2.0**	3.3	4.5	28.2	
BM301	**8.40**	**2.0**	6.4	8.7	54.7	
Cu Regrind	**2.4**	**0.4**	2.0	2.7	17.1	
Grinding	**16.1**	**4.4**	11.7	16.0	100.0	

The circulating load in rod mill circuit was 18% and in ball mill circuit 58%. The energy consumption in grinding circuit was 13.2 kWh/t, 4.5 kWh/t in rod mill and 8.7 kWh/t in ball mill.

During the day, two sets of plant survey samples were collected, at 18:00-18:30 and 21:05. During earlier sampling mixture of Danafloat 245 and MIBC was used in copper and pyrrhotite flotation circuits, while during later sampling the mixture was replaced by Danafloat 507. The reagent dosages were as presented in Table 15 at 14:45. The corresponding mass balance calculations are presented in Tables 17 and 18.

June 6, 2018

Table 17. Mass Balance 13.9. Hour 18:00 – 18:30

Stream	Total solids t/h	Rec %	Cu %	Cu Rec %	Fe %	Fe Rec %	S %	S Rec %	Satmagan %	Satmagan Rec %	SiO2 %	SiO2 Rec%
Derrick us	***740.0***	100.0	0.136	100.0	34.3	100.0	3.79	100.0	35.6	100.0	28.9	100.0
CuRC	***32.5***	4.4	2.663	86.0	43.5	5.6	40.89	47.5	7.8	1.0	3.6	0.6
CuRT	***707.5***	95.6	0.020	14.0	33.9	94.4	2.08	52.5	36.9	99.0	30.0	99.4
CuCC3	***14.9***	2.0	5.395	79.9	41.9	2.5	46.84	24.9	2.0	0.1	1.7	0.1
CuCST	***17.6***	2.4	0.351	6.1	44.8	3.1	35.86	22.5	12.7	0.9	5.3	0.4
Py Feed	***725.1***	98.0	0.028	20.1	34.1	97.5	2.90	75.1	36.3	99.9	29.4	99.9
PyRC	***21.6***	2.9	0.401	8.6	49.7	4.2	35.20	27.1	18.7	1.5	5.6	0.6
PyRT	***708.6***	95.8	0.018	12.6	33.7	94.3	2.08	52.7	36.7	98.8	30.0	99.6
PyCC	***16.5***	2.2	0.463	7.6	50.4	3.3	38.19	22.4	17.0	1.1	4.1	0.3
PYCT	***5.1***	0.7	0.202	1.0	47.6	1.0	25.56	4.7	24.3	0.5	10.7	0.3
WLC 3	***304.8***	41.2	0.003	0.9	65.1	78.2	1.45	15.8	83.9	97.2	5.7	8.1
WLT1	***403.9***	54.6	0.029	11.7	10.1	16.1	2.56	36.9	1.1	1.7	48.4	91.5
PoCC	***13.9***	1.9	0.026	0.4	58.5	3.2	29.42	14.6	38.2	2.0	3.0	0.2
PoRT	***290.8***	39.3	0.002	0.5	65.4	75.0	0.11	1.1	86.1	95.1	5.8	7.9
WLC 4	***290.8***	39.3	0.002	0.5	65.4	75.0	0.11	1.1	86.1	95.1	5.8	7.9
WLT4	***0.0***	0.0	0.102	0.0	12.8	0.0	1.51	0.0	7.5	0.0	44.0	0.0

Au content in the CuCC3 was 3.8 g/t, and RAu was 58%.

Table 18. Mass Balance 13.9. Hour 21:05

Stream	Total solids t/h	Rec %	Cu %	Cu Rec %	Fe %	Fe Rec %	S %	S Rec %	Satmagan %	Satmagan Rec %	SiO2 %	SiO2 Rec %
Derrick us	***750.1***	100.0	0.138	100.0	30.9	100.0	3.53	100.0	35.3	100.0	30.2	100.0
CuCC3	***9.9***	1.3	8.279	79.2	40.4	1.7	47.52	17.8	8.3	0.3	0.6	0.0
Py Feed	***740.2***	98.7	0.029	20.8	30.8	98.3	2.95	82.2	35.7	99.7	30.6	100.0
PyCC	***11.7***	1.6	0.590	6.7	48.8	2.5	42.06	18.5	13.4	0.6	3.2	0.2
PyRT	***728.5***	97.1	0.020	14.1	30.5	95.8	2.32	63.7	36.0	99.1	31.1	99.8
WLC3	***262.0***	34.9	0.004	1.1	65.9	74.4	1.49	14.7	85.3	84.4	5.8	6.7
WLT1	***466.5***	62.2	0.029	13.0	10.6	21.4	2.78	49.0	8.4	14.7	45.3	93.1
PoCC	***13.8***	1.8	0.040	0.5	59.2	3.5	25.90	13.5	41.4	2.2	3.6	0.2
PoRT	***248.2***	33.1	0.002	0.5	66.2	70.9	0.13	1.2	87.7	82.2	5.9	6.4
WLC4	***247.9***	33.1	0.002	0.5	66.3	70.9	0.13	1.2	87.8	82.2	5.8	6.4
WLT4	***0.3***	0.0	0.020	0.0	14.0	0.0	1.24	0.0	8.4	0.0	44.0	0.0

During the day, the Fe content in final magnetite concentrate (WLC4) varied between 65.4 - 66.3% Fe with recoveries of 71 – 75%. According to the Satmagan measurements, the recoveries of magnetite were notably higher, being in the range of 80 – 95%. The Au content in CuCC3 was 4.3 g/t with RAu of 63.7%. Copper concentrate had low quality with ca. 80%

recovery. The high Fe and S values in copper concentrates indicated that pyrite is floating together with copper.

5.5 14.9.2017 Thursday, VE-2 Ore type

Operation hours were 8:20 – 20:05. At 11:30 it was noted that the pH in Cu roughers was low, being ca. 5. Milk of lime was added to ball mill, in the beginning dosage was 700 g/t, and since 13:05, the dosage was increased to 1700 g/t in order to keep pH at level of 7. Low pH caused pyrite accumulation in Cu cleaner circuit. In order to improve the quality of the final magnetite concentrate, the collector dosages in pyrrhotite flotation were increased. At same time the depressant dosage in cupper cleaner was also increased in order to improve the copper cleaner concentrate quality. Reagent dosages are presented in Table 19.

Table 19. Reagents 14.9.

Date:		14.9.
Time:		8:50
Reagent	Feed Point	g/t
CMC	Cu Cln 1	5
PAX	Cu Rgh 1	132
MIBC	Cu Rgh 1	36
DF 507	Cu Rgh 1	68
DOW 250	Po Rgh 1	40
DOW 250	Po Rgh 2	50
PAX	Po. Flot 1	280
PAX	Po. Flot 2	200
DF/MIBC	Po. Rgh 1	20
DF/MIBC	Po. Rgh 2	60
SIBX	Po. Rgh 3	140
DF/MIBC	Po. Rgh 3	60
Flotanol C7	Po. Rgh 3	60
SIBX	Py Rgh	300
DOW 250	Py Rgh	60

Manual flow rate and pulp density measurements from different process stages were done at 12:30, and the results are presented in Table 20.

The circulating load in rod mill circuit was 33% and in ball mill circuit 43%. The energy consumption in grinding circuit was 13.0 kWh/t, 4.4 kWh/t in rod mill and 8.6 kWh/t in ball mill

June 6, 2018

Table 20. Process Flow Rate Measurements, 14.9. Hour 12:30

	Slurry SG g/l (**kg**)	Bucket l	Time s (**min**)	Flow m3/h	Solids %	Solids t/h
Crushed Ore	6.20		30.0	0.25	100.0	0.744
RM Disch.	1775	3.4	11.0	1.11	58.7	1.160
BM Disch.	1360	3.4	8.1	1.51	35.6	0.732
Skako O/S	2200	1.0	23.4	0.15	73.4	0.248
Skako U/S	1280	3.4	5.7	2.15	29.4	0.809
Derrick O/S	1220	1.0	3.3	1.09	24.3	0.323
Derrick U/S	1310	30.0	42.3	2.55	31.8	1.064
WLC1	1440	1.0	8.2	0.44	38.6	0.244
WLT1	1055	3.4	2.0	6.12	7.8	0.505
WLC2	1495	1.0	8.5	0.42	41.8	0.265
WLT2	1005	3.4	12.0	1.02	0.7	0.008
WLC3	1435	1.0	9.3	0.39	38.3	0.213
WLT3	1005	3.4	14.7	0.83	0.7	0.006
WLT 2+3	1010	3.4	6.2	1.97	1.5	0.030
	Power kW, Gross	Idle kW	Power kW, Net	Energy kWh/t, Net	%	Solids t/h
RM302	**5.30**	**2.0**	3.3	4.4	27.5	
BM301	**8.40**	**2.0**	6.4	8.6	53.3	
Cu Regrind	**2.7**	**0.4**	2.3	3.1	19.2	
Grinding	**16.4**	**4.4**	12.0	16.1	100.0	

After addition of lime, the pH levels in copper rougher flotation were as follows:

- CuRgh1: 5.95
- CuRgh2: 6.1
- CuRgh3: 6.2

The flotation circuit was sampled at 17:30 – 18:30. The calculated mass balance is presented in Table 21, and the assay data is presented in Appendix 2.

Cu grade in copper third cleaner concentrate was 11.8% Cu, 45% S and 38.4% Fe. Cu recovery in rougher flotation was 96.5%, and the main Cu loss was in cleaner scavenger tails. Au content in the Cu rougher concentrate was 0.69 g/t with a recovery of 60%. CuCC3 contained 4.95g/t Au with the recovery of 56%.x The assay values for the magnetite concentrate were 0.096% S, 66.5% Fe and 4.93% SiO_2. Fe recovery in magnetite concentrate (WLC4) was 71.6%. Magnetite recovery was better, 94.1%

Table 21. Mass Balance 14.9. Hour 17:30 – 18:30

Stream	Solids kg/h	Rec %	Cu %	Rec %	Fe %	Rec %	S %	Rec %	Satmagan %	Rec %	SiO2 %	Rec %
Derrick us	***749.9***	100.0	0.130	100.0	32.47	100.0	4.25	100.0	34.35	100.0	28.13	100.0
CuRC	***32.2***	4.3	2.916	96.5	43.05	5.7	38.02	38.5	7.21	0.9	4.96	0.8
CuRT	***717.7***	95.7	0.005	3.5	31.99	94.3	2.73	61.5	35.57	99.1	29.18	99.2
PyFeed	***743.2***	99.1	0.024	18.7	32.42	98.9	3.88	90.5	34.65	100.0	28.38	100.0
PyRT	***656.5***	87.5	0.024	16.1	33.97	91.6	1.42	29.2	39.23	100.0	31.45	97.9
PyRC	***86.7***	11.6	0.029	2.6	20.69	7.4	22.50	61.3	0.00	0.0	5.17	2.1
CuCC3	***6.7***	0.9	11.736	81.3	38.36	1.1	44.95	9.5	0.86	0.0	0.57	0.0
CuCST	***25.5***	3.4	0.582	15.2	44.29	4.6	36.18	29.0	8.89	0.9	6.12	0.7
WLC3	***292.3***	39.0	0.005	1.4	65.28	78.4	1.50	13.8	87.47	99.3	5.02	6.9
PoRC	***29.1***	3.9	0.046	1.4	56.04	6.7	14.09	12.9	45.57	5.1	4.47	0.6
PoRT	***263.3***	35.1	0.000	0.1	66.30	71.7	0.11	0.9	92.10	94.1	5.08	6.3
WLT4	***1.0***	0.1	0.024	0.0	16.40	0.1	2.62	0.1	10.35	0.0	41.88	0.2
WLC4	***262.3***	35.0	0.000	0.0	66.50	71.6	0.10	0.8	92.41	94.1	4.93	6.1
WLT1-3	***364.1***	48.6	0.039	14.7	8.82	13.19	1.35	15.46	0.50	0.71	52.68	90.91

At 19:20 it was noticed that the rod mill had been filled up with coarse material. The rod charge was increased and the milling circuit was emptied by running it without any fresh feed.

5.6 15.9.2017 Friday, VE-2 Ore type

Because of the sanding problems in all rougher flotation circuits, it was decided to change the Derrick screen into finer one, 215 ->180 µm.

Low intensity magnetic separation (LIMS) circuit was opened and all tailings were rejected.

Operating hours 7:50 – 20:50. Due to the increasing circulation in ball mill circuit, grinding media was added to the mill.

Flash flotation cell in grinding circuit was operated 19:20 – 20:50.

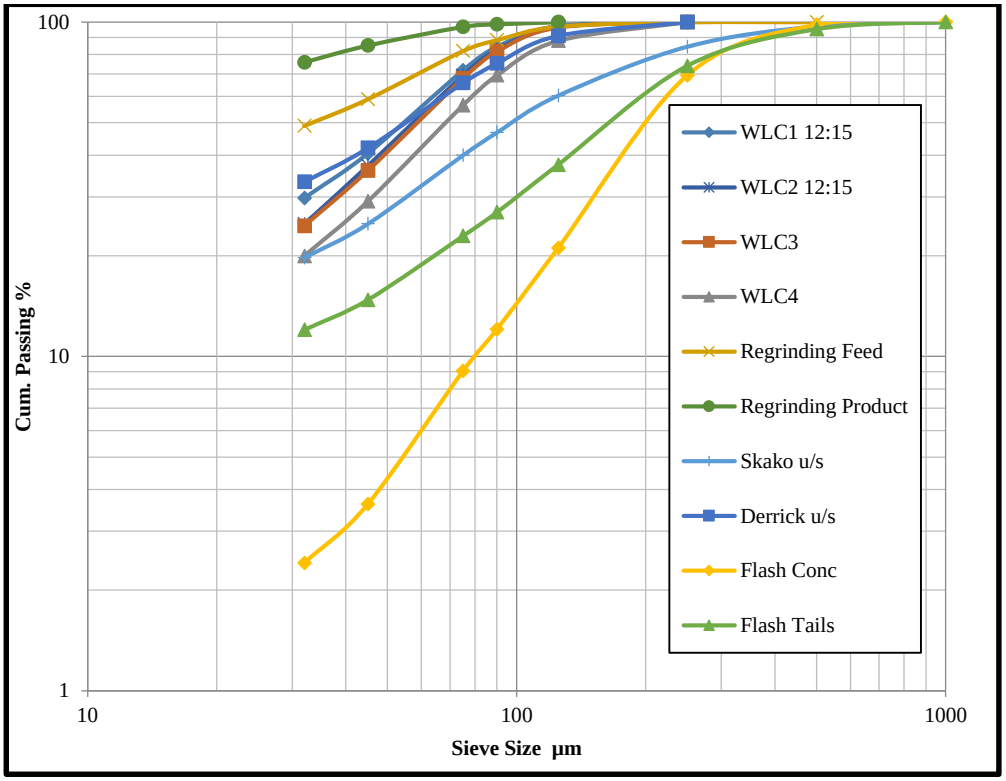

Figure 5. Process PSD's 15.9.2017.

The P(80) value of the flotation feed was 101 µm. The screen assays are presented in Appendix 2.

Table 22. Reagents 15.9.

Date:		15.9.
Time:		19:20
Reagent	Feed Point	g/t
CMC	Cu Cln 1	5
PAX	Cu Rgh 1	130
MIBC	Cu Rgh 1	35
DF 507	Cu Rgh 1	70
PAX	Flash Flot	60
DOW 250	Flash Flot	28
DOW 250	Po Rgh 1	44
DOW 250	Po Rgh 2	48
PAX	Po. Flot 1	300
PAX	Po. Flot 2	200
DF507	Po. Rgh 1	20
DF507	Po. Rgh 2	60
SIBX	Po. Rgh 3	140
DF507	Po. Rgh 3	60
Flotanol C7	Po. Rgh 3	60
SIBX	Py Rgh	300
DOW 250	Py Rgh	60

pH levels in copper rougher flotation were as follows:

- CuRgh1: 7.4
- CuRgh2: 7.6
- CuRgh3: 7.6

Au content in the Cu CC3 was 2.01 g/t.

When the Flash flotation was used, the Au content in the Flash concentrate was 0.61 g/t, and in the CuCC3 5.34 g/t. Rau was ca. 45%

5.7 18.9.2017 Monday, VE-2 Ore type

Operation between 12:00 – 24:00. Pyrrhotite cleaner flotation was operated open, and the cleaner tails were taken out as tailings. Very diluted cleaner tailing caused low solids concentration in pyrrhotite roughers, being 17 – 22 % solids, while the aim was to operate with higher pulp density.

Table 23. Reagents 18.9.

Date:		18.9.
Time:		13:00
Reagent	Feed Point	g/t
CMC	Cu Cln 1	12
PAX	Cu Rgh 1	52
MIBC	Cu Rgh 1	32
DF 245	Cu Rgh 1	68
DOW 250	Po Rgh 1	48
DOW 250	Po Rgh 2	44
PAX	Po. Flot 1	304
PAX	Po. Flot 2	192
DF507	Po. Rgh 1	19
DF507	Po. Rgh 2	55
SIBX	Po. Rgh 3	192
DF507	Po. Rgh 3	56
Flotanol C7	Po. Rgh 3	62
SIBX	Py Rgh	392
DOW 250	Py Rgh	62

pH levels in copper rougher flotation were as follows:

- CuRgh1: 7.4
- CuRgh2: 7.6
- CuRgh3: 7.7
- CuCln1: 12.2
- CuCln3: 9.8

June 6, 2018

At 19:00, Cu cleaner scavenger was operated without air, and the CuClnT1 was directed to pyrite rougher flotation. Grinding circuit particle size distributions were determined, and the distribution curves are presented in Figure 6. Detailed screen assays are presented in Appendix 2.

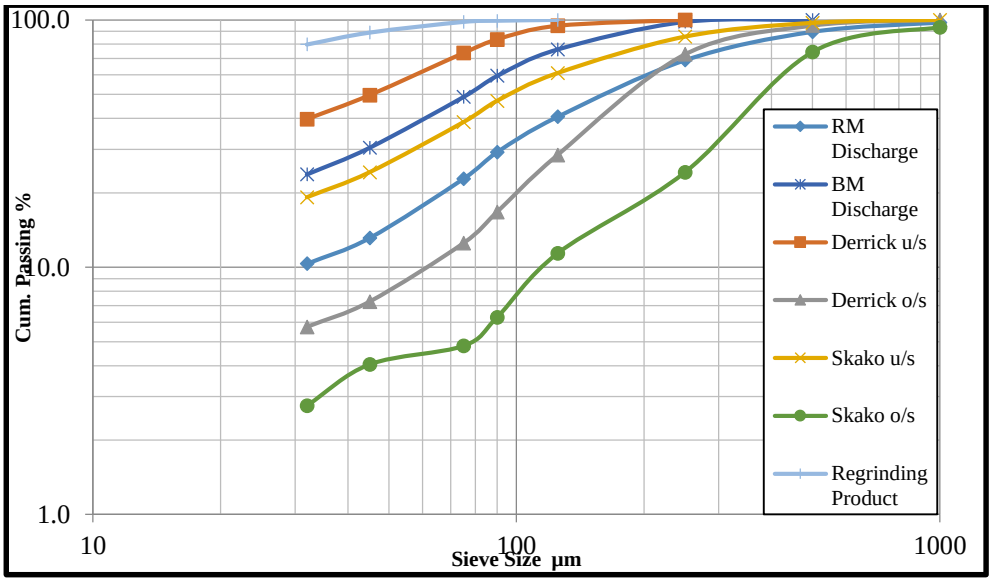

Figure 6. Process PSD's 18.9.2017.

Au content in CuRC was 0.95 g/t and in CuRT 0.03 g/t, so the recovery in Cu rougher flotation RAu was 46%.

5.8 19.9.2017 Tuesday, VE-2 Ore type

Running time during the day was 00:00 – 24:00.

Table 24. Reagents 19.9.

Date:		19.9.	19.9.	19.9.
Time:		1:00	7:30	10:30
Reagent	Feed Point	g/t	g/t	g/t
SEX	Cu Cln 1	5	6	6
CMC	Cu Cln 1	2	2	2
PAX	Cu Rgh 1	44	48	48
MIBC	Cu Rgh 1	32	50	40
DF 245	Cu Rgh 1	66	66	16
DOW 250	Po Rgh 1	52	54	54
DOW 250	Po Rgh 2	48	48	48
PAX	Po. Flot 1	272	292	292
PAX	Po. Flot 2	192	192	192
DF507	Po. Rgh 1	19	20	20
DF507	Po. Rgh 2	52	58	58
SIBX	Po. Rgh 3	204	200	200
DF507	Po. Rgh 3	54	58	58
Flotanol C7	Po. Rgh 3	63	60	60
SIBX	Py Rgh	404	392	120
DOW 250	Py Rgh	64	110	48

Because of the low Fe grade (ca. 65% Fe) in the magnetite concentrate, Mineral Liberation Analysis was done for the WLC4 (13.9. Hour 18:00 – 18:45). Following observations were made:

- The main sulphur carrier is pyrite, amount of what was measured to be 0.17 wt-%. Both pure particles were detected and associated with magnetite as mixed grains. The liberation for pyrite was ca. 60%
- The silicate content in the sample was 10.3 wt-%, mostly as mixed grains with magnetite. Silicate liberation was only 30%.
- The magnetite content was 89 wt-%, and the liberation was ca. 93%.

For the above mentioned reasons it was decided to start the installation of a re-grinding mill for the third magnetic separation concentrate (WLC3). Detailed MLA report is presented in Appendix 3.

June 6, 2018

Table 25. Process Flow Rate Measurements, 19.9. Hour 02:30 – 03:30

	Slurry SG g/l (**kg**)	Bucket l	Time s (**min**)	Flow m3/h	Solids %	Solids t/h
Crushed Ore	6.50		30.0	0.26	100.0	0.780
RM Disch.	1610	3.4	11.7	1.05	51.0	0.858
BM Disch.	1430	3.4	11.3	1.08	40.4	0.626
Skako O/S	2210	1.0	237.0	0.02	73.6	0.025
Skako U/S	1420	3.4	7.6	1.61	39.8	0.910
Derrick O/S	1500	3.4	10.8	1.13	44.8	0.762
Derrick U/S	1190	30.0	51.3	2.11	21.5	0.538
Derrick U/S	1190			2.50	21.5	0.639
WLC1	1515	1.0	7.8	0.46	42.9	0.300
WLT1	1100	3.4	3.9	3.14	13.6	0.471
WLC2	1590	1.0	7.3	0.49	46.9	0.368
WLT2	1010	3.4	11.9	1.03	1.5	0.015
WLC3	1510	1.0	10.2	0.35	42.7	0.228
WLT3	1005	3.4	14.9	0.82	0.7	0.006
PoFlot Feed	1270	1.0	4.2	0.86	26.9	0.292
PoFlot Tails 1	1165	3.4	12.1	1.01	17.9	0.211
PoFlot Tails 2	1170	3.4	10.5	1.17	18.4	0.250
PoFlot Tails 3	1215	3.4	11.0	1.11	22.4	0.302
WLC4	1670	1.0	15.6	0.23	50.7	0.195
WLT4	1000	3.4	12:0	1.02	0.0	0.000
	Power kW, Gross	Idle kW	Power kW, Net	Energy kWh/t, Net	%	Solids t/h
RM302	**5.50**	**2.0**	3.5	4.5	25.9	
BM301	**9.50**	**2.0**	7.5	9.6	55.6	
Cu Regrind	**2.7**	**0.2**	2.5	3.2	18.5	
Grinding	**17.7**	**4.4**	13.5	17.3	100.0	

Energy consumption in primary and secondary grinding stages was 14.1 kWh/t. Rod mill circulating load was low, while in ball mill circuit the circulating load was 98%.

At 9:35, Sala tower mill was introduced to grind LIMS3 Mags (WLC3). The mill was operated in closed circuit with 87µm Sweco screen. Media charge dia. was ca. 10 mm and the mill power draw was 4.1 kW. The aim of re-grinding was to lower the silicate and sulphur grades in magnetite concentrate. (Actually: SiO_2 content was lowered from 4.6 -> 2.4%, S-content 0.137 -> 0.091%)

At 18:15, the tower mill was moved to grind LIMS2 Mags (WLC2), in order to achieve higher pulp density in Pyrrhotite flotation feed and to remove silicates already before flotation stage. At same time, the Fe content in final magnetite concentrate, WLC4, increased from level 65 – 66 % Fe to 70 % Fe. The sulphur grade was now close to target, being 0.067 wt-% S. SiO_2 content was now 2.34%.

June 6, 2018

Particle size determinations were made for the main process streams. The detailed screen assays are presented in Appendix 2, and graphically in Figure 7.

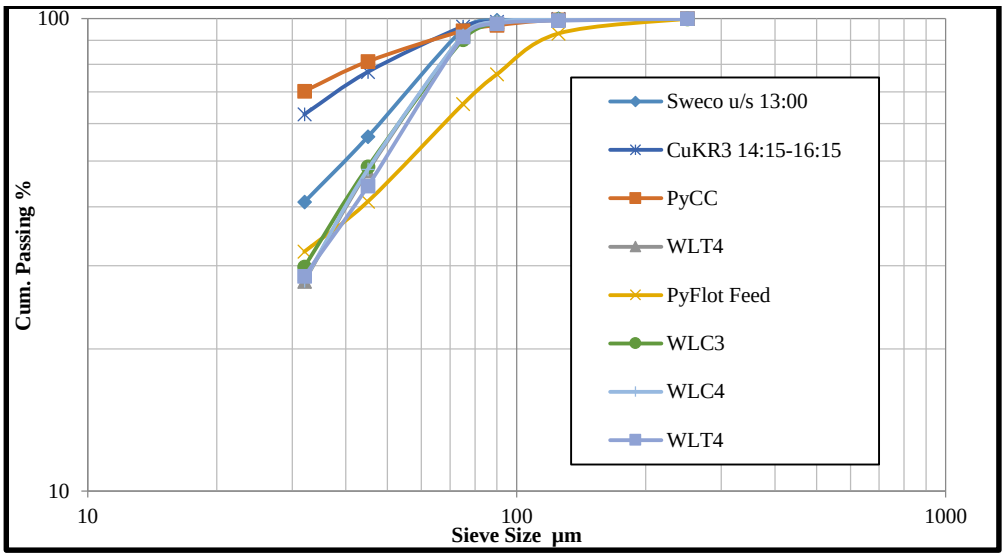

Figure 7. Process PSD's 19.9.2017.

P(80) value in magnetic separation (Sweco screen u/s) was now 63µm.

June 6, 2018

Table 26. Process Flow Rate Measurements, 19.9. Hour 16:45

	Slurry SG g/l (**kg**)	Bucket l	Time s (**min**)	Flow m3/h	Solids %	Solids t/h
Crushed Ore	6.30		30.0	0.25	100.0	0.756
RM Disch.	1630	3.4	11.8	1.04	52.0	0.879
BM Disch.	1430	3.4	9.1	1.35	40.4	0.778
Skako O/S	2230	1.0	230.0	0.02	74.2	0.026
Skako U/S	1570	3.4	3.9	3.14	48.8	2.406
Derrick O/S	1400	3.4	11.2	1.09	38.4	0.588
Derrick U/S	1210	3.4	3.9	3.14	23.3	0.886
Derrick U/S	1190			2.50	21.5	0.639
WLC1	1420	1.0	7.1	0.51	37.4	0.269
WLT1	1060	3.4	4.5	2.72	8.5	0.245
WLC2	1610	1.0	6.1	0.59	47.9	0.455
WLT2	1010	3.4	12.4	0.99	1.5	0.015
WLC3	1610	1.0	9.8	0.37	47.9	0.283
WLT3	1010	3.4	17.5	0.70	1.5	0.010
PoFlot Feed	1290	3.4	15.7	0.78	28.4	0.286
PoFlot Tails 1	1150	3.4	9.7	1.26	16.5	0.239
PoFlot Tails 2	1130	3.4	9.7	1.26	14.5	0.207
PoFlot Tails 3	1190	3.4	12.3	1.00	20.2	0.239
WLC4	1740	1.0	10.6	0.34	53.7	0.317
WLT4	1000	3.4	22.0	0.56	0.0	0.000
Sweco O/S	2880	1.0	37.4	0.10	82.5	0.229
Sweco U/S	1290	3.4	17.5	0.70	33.7	0.304
	Power kW, Gross	Idle kW	Power kW, Net	Energy kWh/t, Net	%	Solids t/h
RM302	**5.50**	**2.0**	3.5	4.6	21.0	
BM301	**9.50**	**2.0**	7.5	9.9	44.9	
Cu Regrind	**2.7**	**0.4**	2.3	3.0	13.8	
Fe Regrind	**4.1**	**0.7**	3.4	4.5	20.4	
Grinding	**21.8**	**5.1**	16.7	22.1	100.0	

By adding the SALA Tower Mill in grinding the WLC2, the total energy consumption increased to 22.1 kWh/t.

Some frother was added to second Cu cleaner flotation. The frother was MIBC with a dosage of 2 g/t.

After changes, the process was let to balance for several hours. Plant survey samples were collected at 21:00. The XRF assays are presented in Appendix 2 and the material balance is presented in Table 27.

Table 27. Mass Balance 19.9. Hour 21:00

Stream	Total solids t/h	Rec %	Cu %	Cu Rec %	Fe %	Fe Rec %	S %	S Rec %	Satmagan %	Satmagan Rec %	SiO2 %	SiO2 Rec %
Derrick us	***749.9***	100.0	0.131	100.0	38.07	100.0	3.87	100.0	39.76	100.0	27.6	100.0
CuRC	***18.6***	2.5	3.938	74.8	41.80	2.7	39.17	25.1	8.36	0.5	7.5	0.7
CuRT	***731.3***	97.5	0.034	25.2	37.97	97.3	2.98	74.9	40.56	99.5	28.1	99.3
CuCC4	***2.5***	0.3	21.557	54.2	34.59	0.3	41.64	3.5	0.67	0.0	0.3	0.0
CuCT1	***16.1***	2.2	1.252	20.6	42.90	2.4	38.80	21.6	9.54	0.5	8.6	0.7
PyFeed	***747.4***	99.7	0.060	45.8	38.08	99.7	3.75	96.5	39.89	100.0	27.7	100.0
PyCC	***21.6***	2.9	1.284	28.3	44.86	3.4	49.38	36.7	3.02	0.2	0.9	0.1
PyRT	***725.8***	96.8	0.024	17.5	37.88	96.3	2.39	59.8	40.98	99.8	28.4	99.9
WLC3	***364.8***	48.6	0.007	2.8	65.94	84.3	2.74	34.4	80.58	98.6	2.6	4.5
WLT1-3	***361.0***	48.1	0.040	14.7	9.51	12.0	2.04	25.4	0.97	1.2	54.6	95.4
PoCC	***101.8***	13.6	0.024	2.5	56.75	20.2	9.60	33.7	44.43	15.2	1.8	0.9
PoRT	***263.0***	35.1	0.001	0.3	69.50	64.0	0.08	0.7	94.57	83.4	2.9	3.7
WLC4	***262.0***	34.9	0.001	0.3	69.71	64.0	0.08	0.7	94.88	83.4	2.7	3.5
WLT4	***1.0***	0.1	0.029	0.0	16.40	0.1	1.36	0.0	12.10	0.0	42.6	0.2

The Cu grade in CuCC4 was 21.6% with a recovery of 54.2%. The final magnetite concentrate had Fe grade of 69.7% and the RFe was 64%. S grade of the magnetite concentrate was 0.08%. Satmagan balance yielded some 83.4% recovery for magnetite.

5.9 20.9.2017 Wednesday, VE-2 Ore type

Running time during the day was 00:00 – 17:40. During the night, frequent samplings were done from copper cleaner concentrate, pyrite cleaner concentrate and magnetite concentrate. XRF assays are collected into Appendix 2. The sulphur grade in WLC4 was low during the night, being 0.051% as average. Fe grade was 70%.

The magnetite concentrate (WLC4) collection into thickener was done 05:30 – 17:00.

Table 28 presents the reagent dosages during the day. Plant survey samples were collected at 12:00, and the mass balance calculation is presented in Table 30.

June 6, 2018

Table 28. Reagents 20.9.

Date:		20.9.	20.9.	20.9.
Time:		3:30	12:15	13:10
Reagent	Feed Point	g/t	g/t	g/t
SEX	Cu Cln 1	9	8	6
CMC	Cu Cln 1	2	2	2
MIBC	Cu Cln 1	2	2	2
MIBC	Cu Cln2	3	3	3
PAX	Cu Rgh 1	36	48	24
MIBC	Cu Rgh 1	38	50	50
DF 507	Cu Rgh 1	46	50	50
PAX	Cu Rgh 2		4	
PAX	Flash Flot			24
DOW 250	Flash Flot			20
DOW 250	Po Rgh 1	52	44	44
DOW 250	Po Rgh 2	48	48	48
PAX	Po. Flot 1	28	312	312
PAX	Po. Flot 2	224	232	232
DF507	Po. Rgh 1	24	20	20
DF507	Po. Rgh 2	56	54	54
SIBX	Po. Rgh 3	208	224	224
DF/MIBC	Po. Rgh 3	52	56	56
Flotanol C7	Po. Rgh 3	56	60	60
SIBX	Py Rgh	224	240	240
DOW 250	Py Rgh	48	68	68

Table 29. Process Flow Rate Measurements, 20.9. Hour 03:30

	Slurry SG g/l (**kg**)	Bucket l	Time s (**min**)	Flow m3/h	Solids %	Solids t/h
Crushed Ore	6.40		30.0	0.26	100.0	0.768
RM Disch.	1620	3.4	13.4	0.91	51.5	0.762
BM Disch.	1400	3.4	12.1	1.01	38.4	0.544
Skako O/S	2220	1.0	294.0	0.01	73.9	0.020
Skako U/S	1355	3.4	5.9	2.07	35.2	0.990
Derrick O/S	1300	1.0	4.4	0.82	31.0	0.330
Derrick U/S	1240	3.4	4.4	2.78	26.0	0.898
Derrick U/S	1240			2.43	26.0	0.784
WLC1	1420	1.0	7.4	0.49	37.4	0.258
WLT1	1075	3.4	4.1	2.99	10.5	0.336
WLC2	1455	1.0	7.1	0.51	39.5	0.291
WLT2	1010	3.4	12.9	0.95	1.5	0.014
WLC3	1524	1.0	10.4	0.35	43.4	0.229
WLT3	1005	3.4	8.6	1.42	0.7	0.011
PoFlot Feed	1335	1.0	4.9	0.73	31.7	0.311
PoFlot Tails 1	1230	1.0	3.6	1.00	23.6	0.291
PoFlot Tails 2	1235	1.0	3.6	1.00	24.0	0.297
PoFlot Tails 3	1310	3.4	9.6	1.28	29.9	0.499
WLC4	1725	1.0	12.6	0.29	53.1	0.262
WLT4	1000	3.4	13.7	0.89	0.0	0.000
Sweco O/S	1390	1.0	6.8	0.53	35.4	0.261
Sweco U/S	1150	3.4	7.9	1.55	19.6	0.349
	Power kW, Gross	Idle kW	Power kW, Net	Energy kWh/t, Net	%	Solids t/h
RM302	**5.50**	**2.0**	3.5	4.6	21.1	
BM301	**9.50**	**2.0**	7.5	9.8	45.2	
Cu Regrind	**2.7**	**0.4**	2.3	3.0	13.9	
Fe Regrind	**4.0**	**0.7**	3.3	4.3	19.9	
Grinding	**21.7**	**5.1**	16.6	21.6	100.0	

June 6, 2018

Table 30. Mass Balance 20.9. Hour 12:00

Stream	Total solids t/h	Rec %	Cu %	Cu Rec %	Fe %	Fe Rec %	S %	S Rec %
Derrick us	***750.1***	100.0	0.142	100.0	36.41	100.0	4.51	100.0
CuRC	***21.5***	2.9	4.147	83.3	41.36	3.3	41.04	26.0
CuRT	***728.6***	97.1	0.024	16.7	36.27	96.7	3.44	74.0
CuCC4	***2.6***	0.3	22.695	55.2	32.94	0.3	41.31	3.2
CuCT1	***18.9***	2.5	1.592	28.1	42.52	2.9	41.00	22.9
PyFeed	***747.5***	99.7	0.064	44.8	36.43	99.7	4.39	96.8
PyRC	***62.5***	8.3	0.472	27.6	37.88	8.7	28.82	53.2
PyRT	***684.9***	91.3	0.027	17.2	36.29	91.0	2.16	43.6
WLC3	***324.9***	43.3	0.009	2.7	68.05	81.0	2.21	21.2
WLT1-3	***360.0***	48.0	0.043	14.4	7.63	10.1	2.11	22.4
PoRC	***31.2***	4.2	0.056	1.6	55.02	6.3	19.91	18.3
PoRT	***293.7***	39.2	0.004	1.1	69.43	74.7	0.33	2.9
WLC4	***291.7***	38.9	0.004	1.0	69.76	74.5	0.29	2.5
WLT4	***2.0***	0.3	0.038	0.1	21.34	0.2	6.29	0.4

The copper concentrate had a grade of 22.7% Cu and RCu was 55.2%. The final magnetite concentrate had Fe grade of 69.8% and the RFe was 74.5%. RAu in Cu rougher flotation was 38.7% with a grade of 0.83 g/t Au. The Au content in CuCC4 was 1.83 g/t with a recovery of 10.5%.

Flash flotation was in use at 13:10 – 17:40. CuClnT1 was directed to Pyrite cleaner flotation, because it had higher sulphur content than the Cu rougher tails. Process samples were taken at 16:00 – 17:00, and the corresponding mass balance calculation is presented in Table 31.

The Flash flotation concentrate had a grade of 2.24% Cu with a recovery of 36.7%. Copper rougher tailing had 0.022% Cu, and the rougher flotation recovery was 48,3%, so the total RCu in copper rougher flotation was 85%.

WLC4 had 70% Fe with a recovery RFe of 70.5%. S-content was 0.08%. Magnetite recovery was 91.3%

During the day samples for environmental analyses were collected. At 17:00 – 17:40, WLC3 sample was collected for pyrrhotite laboratory flotation test work.

June 6, 2018

Table 31. Mass Balance 20.9. Hour 16:00 – 17:00

Stream	Total solids kg/h	Rec %	Cu %	Cu Rec %	Fe %	Fe Rec %	S %	S Rec %	Satma-gan %	Satmagan Rec %	SiO2 %	SiO2 Rec %
Feed	***750.1***	100.0	0.141	100.0	34.68	100.0	4.43	100.0	37.19	100.0	27.6	100.0
Skako os	***20.8***	2.8	0.111	2.2	33.97	2.7	2.03	1.3	34.79	2.6	32.9	3.3
Skako us	***1276.9***	170.2	0.095	115.1	31.25	153.4	3.46	132.8	33.35	152.6	30.8	189.7
Derick os	***544.0***	72.5	0.100	51.8	26.85	56.2	3.37	55.1	27.19	53.0	34.4	90.3
Derrick us	***732.9***	97.7	0.091	63.3	34.51	97.2	3.52	77.7	37.92	99.6	28.1	99.3
Flash Conc	***17.2***	2.3	2.247	36.7	41.78	2.8	43.15	22.3	6.37	0.4	7.9	0.7
Flash Tails	***526.8***	70.2	0.030	15.1	26.36	53.4	2.07	32.8	27.87	52.6	35.3	89.7
CuRC	***12.0***	1.6	4.245	48.3	40.29	1.9	32.86	11.9	10.19	0.4	12.4	0.7
CuRT	***720.9***	96.1	0.022	15.0	34.42	95.4	3.04	65.8	38.38	99.2	28.4	98.6
ReGrinding Discharge	***29.2***	3.9	3.068	85.0	41.17	4.6	38.92	34.2	7.94	0.8	9.8	1.4
CuCC4	***6.0***	0.8	9.360	53.3	40.70	0.9	46.18	8.3	1.48	0.0	0.9	0.0
CuCT1	***23.2***	3.1	1.440	31.7	41.29	3.7	37.04	25.8	9.61	0.8	12.1	1.3
PyFeed	***744.1***	99.2	0.066	46.7	34.63	99.1	4.10	91.7	37.48	100.0	27.9	100.0
PyRC	***13.5***	1.8	2.528	32.4	43.54	2.3	47.39	19.3	4.65	0.2	2.6	0.2
PyRT	***730.6***	97.4	0.021	14.3	34.47	96.8	3.29	72.4	38.09	99.7	28.3	99.8
WLC3	***419.1***	55.9	0.007	2.7	52.77	85.0	4.26	53.7	65.80	98.8	5.1	10.2
WLT1-3	***311.5***	41.5	0.039	11.6	9.85	11.8	2.00	18.7	0.81	0.9	59.6	89.6
PoRC	***148.8***	19.8	0.017	2.5	24.44	14.0	11.79	52.8	13.38	7.1	6.2	4.5
PoRT	***270.3***	36.0	0.001	0.3	68.36	71.0	0.11	0.9	94.66	91.7	4.4	5.8
WLC4	***262.1***	34.9	0.001	0.2	69.97	70.5	0.08	0.7	97.20	91.3	3.0	3.8
WLT4	***8.2***	1.1	0.012	0.1	17.04	0.5	0.89	0.2	13.66	0.4	49.7	2.0

The Cu recovery to secondary screen over size (Flash flotation feed) was 51.8%, from which the recovery into Flash flotation concentrate was 36.7%, yielding a stage recovery of 71%. However, the copper grade in Flash flotation concentrate was low, 2.2% Cu, so it had to be fed into copper cleaning circuit through re-grinding stage together with the rougher concentrate.

In this test period, the use of Flash flotation gave 1.7% increase in copper rougher flotation recovery (Table 30. 88.3% vs 85%), but the grade of the cleaner feed was lower than with rougher flotation alone.

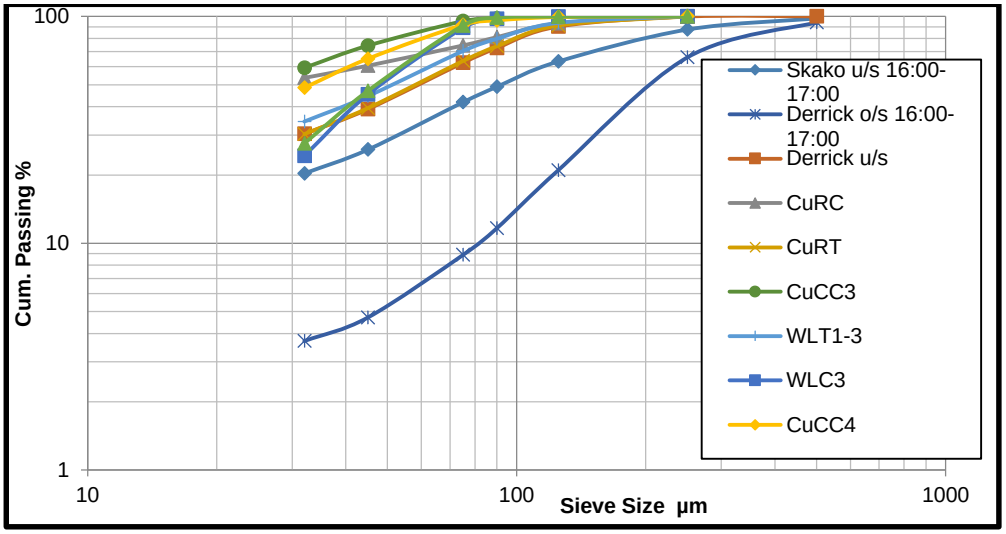

Figure 8. Process PSD's 20.9.2017.

5.10 21.9.2017 Thursday, VE-2 Ore type

Running time during the day was 08:45 – 16:15.

Table 32. Reagents 21.9.

Date:		21.9.
Time:		10:10
Reagent	Feed Point	g/t
PAX	Cu Rgh 1	48
MIBC	Cu Rgh 1	50
DF 245	Cu Rgh 1	46
DOW 250	Po Rgh 1	44
DOW 250	Po Rgh 2	44
PAX	Po. Flot 1	280
PAX	Po. Flot 2	224
DF/MIBC	Po. Rgh 1	19
DF/MIBC	Po. Rgh 2	55
SIBX	Po. Rgh 3	204
DF/MIBC	Po. Rgh 3	56
Flotanol C7	Po. Rgh 3	58
SIBX	Py Rgh	232
DOW 250	Py Rgh	60

The process was operated without Cu cleanings in order to collect larger Cu rougher concentrate samples for downstream testing. At same time, total LIMS tailing was collected.

5.11 22.9.2017 Friday, VE-3 Ore type

DF 507 was replaced by DF 245 in pyrrhotite flotation, and C7 was replaced by DOW250. Pyrite cleaner was not in use, and the CuClnT1 was directed to Pyrite rougher flotation.

Daily operating hours were 08:00 – 16:00.

Table 33. Process Flow Rate Measurements, 22.9. Hour 12:40

	Slurry SG g/l (**kg**)	Bucket l	Time s (**min**)	Flow m3/h	Solids %	Solids t/h
Crushed Ore	6.60		30.0	0.26	100.0	0.792
RM Disch.	1685	3.4	15.3	0.80	54.7	0.737
BM Disch.	1510	3.4	7.3	1.68	45.4	1.150
Skako O/S	2310	1.0	88.0	0.04	76.3	0.072
Skako U/S	1530	3.4	5.0	2.45	46.6	1.745
Derrick O/S	1440	3.4	5.1	2.40	41.1	1.420
Derrick O/S	1510	1.0	2.3	1.57	45.4	1.074
Derrick U/S	1215	3.4	4.6	2.66	23.8	0.769
Derrick U/S	1250			2.10	26.9	0.706
LIMS 1 Feed	1150	3.4	3.8	3.22	19.6	0.725
WLC1	1200	1.0	12.2	0.30	21.1	0.075
WLT1	1150	3.4	4.2	2.91	19.6	0.656
WLC3	1225	1.0	16.5	0.22	23.2	0.062
WLT3	1002	3.4	8.6	1.42	0.3	0.004
	Power kW, Gross	Idle kW	Power kW, Net	Energy kWh/t, Net	%	Solids t/h
RM302	**5.50**	**2.0**	3.5	4.4	26.3	
BM301	**9.50**	**2.0**	7.5	9.5	56.4	
Cu Regrind	**2.7**	**0.4**	2.3	2.9	17.3	
Grinding	**17.7**	**4.4**	13.3	16.8	100.0	

Figure 9 presents the particle size distributions in grinding circuit with VE-3 sample. Detailed screen assays are presented in Appendix 2.

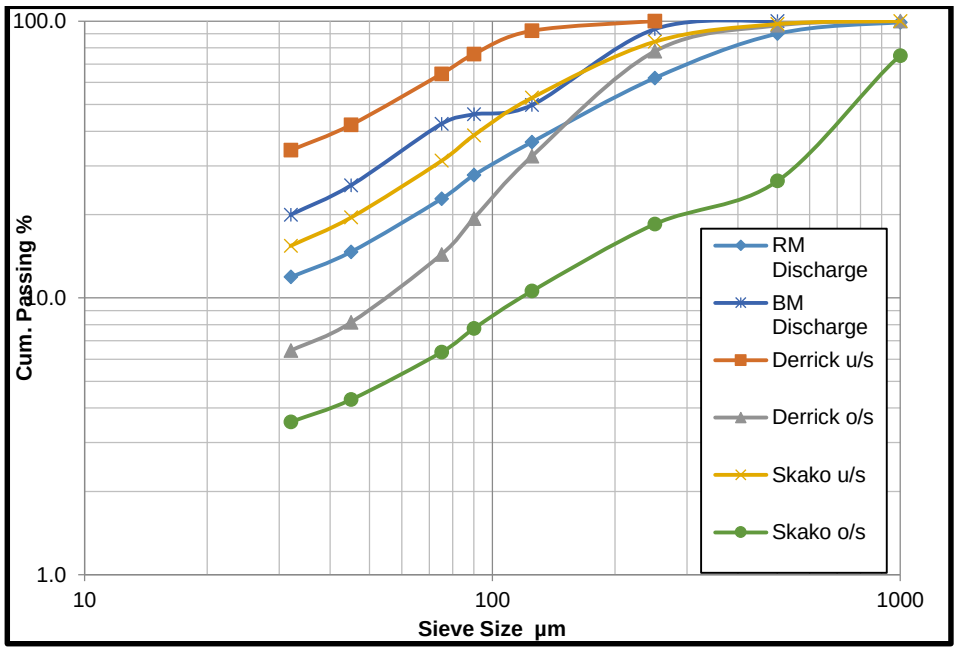

Figure 9. Process PSD's 22.9.2017.

After 13:50, Pyrrhotite flotation was operated with only one flotation machine. Process samples were taken at 14:50, and the XRF assays are presented in Appendix 2.

The Cu recovery in rougher flotation was 91.1%, and the grade of the second cleaner concentrate was 23.5% Cu. WLC4 had 71.7 % Fe with 0.138% S.

5.12 25.9.2017 Monday, VE-3 and VE-2 Ore types

CuClnSc-flotation was taken back in operation. The re-grinding was by-passed in LIMS circuit.

Operating hours were 07:45 – 21:00. The feed bin was empty at 15:40.

Figure 10 presents the particle size distributions of the process with VE-3 feed sample, screen assays are presented in Appendix 2.

Process samples were taken at 13:40. XRF assays are presented in Appendix 2 and the mass balance in Table 34.

June 6, 2018

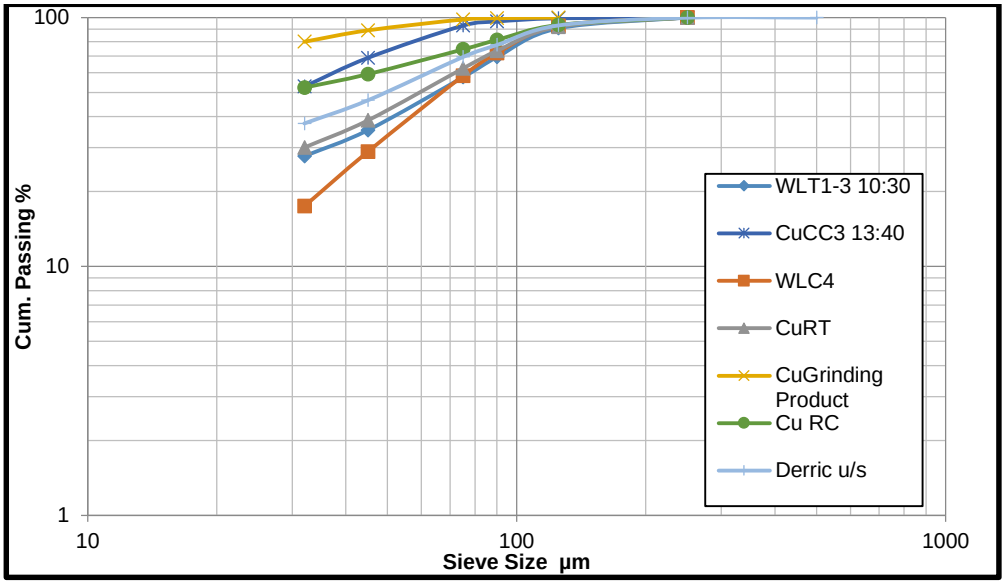

Figure 10. Process PSD's 25.9.2017.

Table 34. Mass Balance 25.9. Hour 13:40

Stream	Total solids t/h	Rec %	Cu %	Cu Rec %	Fe %	Fe Rec %	S %	S Rec %	Satmagan %	Satmagan Rec %	SiO2 %	SiO2 Rec %
Derrick us	***750.6***	100.0	0.093	100.0	18.05	100.0	0.82	100.0	8.6	100.0	39.3	100.0
CuRC	***19.1***	2.5	3.448	94.4	35.43	5.0	22.78	71.0	1.8	0.5	13.6	0.9
CuRT	***731.5***	97.5	0.005	5.6	17.59	95.0	0.24	29.0	8.8	99.5	40.0	99.1
CuCC1	***15.1***	2.0	5.949	129.3	40.51	4.5	47.83	118.2	0.2	0.1	2.0	0.1
CuCT1	***20.2***	2.7	1.520	44.0	35.86	5.3	25.14	82.7	1.8	0.5	13.8	0.9
CuCC2	***19.4***	2.6	11.019	307.0	38.06	5.5	45.99	145.8	0.2	0.0	1.0	0.1
CuCT2	***12.1***	1.6	2.948	51.0	41.63	3.7	48.13	94.8	0.3	0.0	2.4	0.1
CuCC3	***3.1***	0.4	17.699	78.2	36.11	0.8	46.64	23.5	0.2	0.0	0.4	0.0
CuCT3	***16.3***	2.2	9.759	228.8	38.43	4.6	45.87	122.4	0.2	0.0	1.1	0.1
CuCSC	***4.1***	0.6	4.672	27.7	38.04	1.2	52.02	35.1	0.5	0.0	4.3	0.1
CuCST	***16.0***	2.1	0.706	16.2	35.30	4.2	18.19	47.6	2.1	0.5	16.2	0.9
PyFeed	***681.0***	90.7	0.022	21.6	12.99	65.3	0.59	65.6	0.4	4.1	43.1	99.3
PyRC	***0.8***	0.1	1.258	1.5	43.23	0.3	48.87	6.6	2.8	0.0	2.5	0.0
PyRT	***680.2***	90.6	0.021	20.2	12.95	65.0	0.53	59.0	0.4	4.0	43.1	99.3
PyCC	***0.8***	0.1	1.272	1.5	43.48	0.3	49.33	6.6	2.7	0.0	2.1	0.0
PyCT	***0.0***	0.0	0.195	0.0	23.79	0.0	12.17	0.0	7.1	0.0	34.0	0.0
WLC1	***76.6***	10.2	0.004	0.5	61.36	34.7	0.96	12.0	81.0	96.1	8.2	2.1
WLT1	***654.9***	87.3	0.005	5.1	12.47	60.3	0.16	16.9	0.3	3.3	43.7	97.0
WLC2	***68.2***	9.1	0.002	0.2	67.49	34.0	1.00	11.1	90.8	96.0	3.7	0.8
WLT2	***8.4***	1.1	0.026	0.3	11.54	0.7	0.66	0.9	1.2	0.2	45.4	1.3
WLC3	***66.6***	8.9	0.001	0.1	68.99	33.9	1.01	10.9	93.0	95.9	3.0	0.7
WLT3	***1.7***	0.2	0.014	0.0	6.87	0.1	0.72	0.2	2.0	0.1	31.9	0.2
WLT1-3	***665.0***	88.6	0.006	5.4	12.45	61.1	0.17	18.0	0.3	3.5	43.7	98.5
WLC4	***60.3***	8.0	0.000	0.0	69.78	31.0	0.05	0.5	95.5	89.3	2.6	0.5
WLT4	***0.7***	0.1	0.010	0.0	33.10	0.2	0.51	0.1	35.0	0.4	29.8	0.1
PoRC	***5.6***	0.7	0.014	0.1	65.00	2.7	11.44	10.4	72.6	6.3	3.2	0.1
PoRT	***61.0***	8.1	0.000	0.0	69.35	31.2	0.05	0.5	94.8	89.7	2.9	0.6

RCu in rougher flotation was 94.4% and the final copper concentrate was 17.7 % Cu with 78.2% recovery. Gold grade in CuCC3 was 3.51 g/t and the RAu was 29% A lot of iron was

lost in preceding processing stages, but according to the Satmagan measurements, the recovery of magnetite to WLC4 was 89.3%. Fe grade was as with VE-2 sample, being 69.8% Fe.

5.13 26.9.2017 Tuesday, VE-2 Ore type

Copper cleaner-scavenger flotation was in operation, and CuCST was reported to Pyrite cleaner. Pyrite cleaner tails were directed to be mixed with Pyrrhotite cleaner concentrate. Tower mill in closed circuit with Sweco screen was re-installed to grind LIM 2 Mags (WLC2).

Operating hours were 07:30 – 21:00. Reagent dosages during the day are presented in Table 35.

Table 35. Reagents 26.9.

Date:		26.9.	26.9.
Time:		7:30	19:00
Reagent	Feed Point	g/t	g/t
SEX	Cu Cln 1	6	7
CMC	Cu Cln 1	2	2
MIBC	Cu Cln 1	3	2
PAX	Cu Rgh 1	50	52
MIBC	Cu Rgh 1		48
PAX	Cu Rgh 2	5	6
DOW 250	Po Rgh 1	40	48
DOW 250	Po Rgh 2	40	44
PAX	Po. Flot 1	100	96
PAX	Po. Flot 2	100	92
DF245/MIBC	Po. Rgh 1	40	20
DF245/MIBC	Po. Rgh 2	40	19
SIBX	Po. Rgh 3	100	88
DF/MIBC	Po. Rgh 3	40	19
DOW 250	Po. Rgh 3	40	19
SIBX	Py Rgh	200	196
DOW 250	Py Rgh	80	82

Table 36 presents the manual flow rate measurement from the process at 11:30.

June 6, 2018

Table 36. Process Flow Rate Measurements, 26.9. Hour 11:30

	Slurry SG g/l (**kg**)	Bucket l	Time s (**min**)	Flow m3/h	Solids %	Solids t/h
Crushed Ore	6.50		30.0	0.26	100.0	0.780
RM Disch.	1610	3.4	11.3	1.08	51.0	0.889
BM Disch.	1435	3.4	9.4	1.30	40.8	0.762
Skako O/S	2200	1.0	112.0	0.03	73.4	0.052
Skako U/S	1460	3.4	6.6	1.85	42.4	1.147
Derrick O/S	1370	3.4	8.5	1.44	36.3	0.717
Derrick U/S	1215	3.4	6.2	1.97	23.8	0.571
Derrick U/S	1215			2.44	23.8	0.705
WLC1	1360	1.0	9.5	0.38	33.4	0.172
WLT1	1070	3.4	5.5	2.23	9.8	0.234
WLC2	1360	1.0	8.8	0.41	33.4	0.186
WLT2	1005	3.4	12.8	0.96	0.7	0.007
WLC3	1475	1.0	10.6	0.34	40.7	0.204
WLT3	1005	3.4	13.2	0.93	0.7	0.007
PoFlot Feed	1250	1.0	4.7	0.77	25.3	0.242
PoFlot Tails 1	1195	1.0	5.6	0.64	20.6	0.158
PoFlot Tails 2	1250	1.0	7.2	0.50	25.3	0.158
PoFlot Tails 3	1310	1.0	59.7	0.06	29.9	0.024
WLC4	1200	1.0	50.0	0.07	21.1	0.018
WLT4	1000	1.0	8.1	0.44	0.0	0.000
	Power kW, Gross	Idle kW	Power kW, Net	Energy kWh/t, Net	%	Solids t/h
RM302	**5.20**	**2.0**	3.2	4.1	20.1	
BM301	**9.15**	**2.0**	7.2	9.2	44.8	
Cu Regrind	**2.7**	**0.4**	2.3	2.9	14.4	
Fe Regrind	**4.0**	**0.7**	3.3	4.2	20.7	
Grinding	**21.1**	**5.1**	16.0	20.4	100.0	

Plant survey samples were collected at 20:00 – 20:45, particle size distribution determinations were made to main products as presented in Figure 11. Detailed screen assays are presented in Appendix 2. The corresponding mass balance calculation is presented in Table 37. XRF assays are presented in Appendix 2.

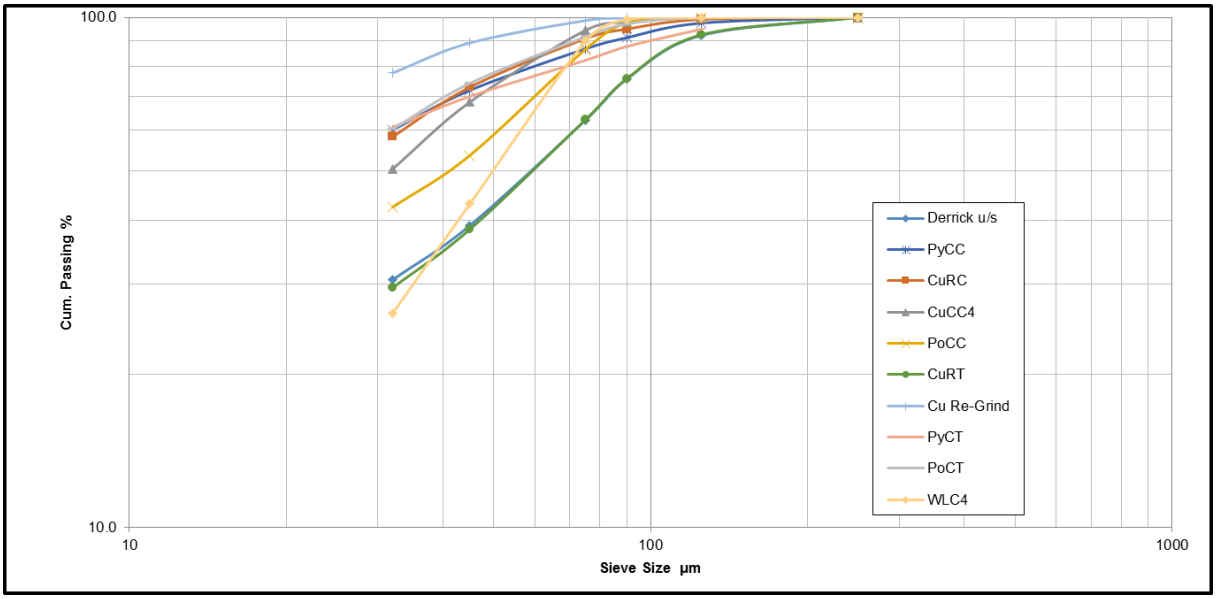

Figure 11. Process PSD's 26.9.2017.

Table 37. Mass Balance 26.9. Hour 20:00 – 20:45

Stream	Total solids t/h	Rec %	Cu %	Cu Rec %	Fe %	Fe Rec %	S %	S Rec %	Satmagan %	Satmagan Rec %	SiO2 %	SiO2 Rec %
Derrick us	***750.0***	100.0	0.125	100.0	29.76	100.0	3.53	100.0	30.7	100.0	28.8	100.0
CuRC	***12.5***	1.7	5.837	78.1	42.30	2.4	44.26	20.9	7.8	0.4	5.0	0.3
CuRT	***737.5***	98.3	0.028	21.9	29.55	97.6	2.84	79.1	31.1	99.6	29.2	99.7
CuCC3	***3.7***	0.5	16.748	66.3	36.44	0.6	47.51	6.6	0.5	0.0	0.3	0.0
CuCST	***8.8***	1.2	1.250	11.8	44.77	1.8	42.90	14.3	10.9	0.4	6.9	0.3
WLT1-3	***492.1***	65.6	0.040	21.0	10.41	23.0	3.28	61.0	1.1	2.4	42.1	95.8
WLC3	***245.4***	32.7	0.004	0.9	67.91	74.7	1.96	18.2	91.3	97.1	3.5	3.9
PoRC	***35.0***	4.7	0.019	0.7	60.70	9.5	13.40	17.7	62.8	9.5	5.7	0.9
PoRT	***210.4***	28.1	0.001	0.2	69.11	65.2	0.06	0.4	96.0	87.6	3.1	3.0
WLT4	***2.0***	0.3	0.005	0.0	19.80	0.2	0.39	0.0	18.6	0.2	40.3	0.4
WLC4	***208.4***	27.8	0.001	0.2	69.59	65.0	0.05	0.4	96.8	87.4	2.7	2.6
CyOF	***61.5***	8.2	0.181	11.9	11.64	3.2	8.59	19.9	1.5	0.4	48.7	13.9
CyUF	***430.6***	57.4	0.020	9.1	10.24	19.7	2.52	41.0	1.1	2.0	41.1	81.9
PyRT	***428.4***	57.1	0.018	8.1	10.08	19.3	2.38	38.4	1.0	1.9	41.3	81.8
PyRC	***2.2***	0.3	0.424	1.0	41.20	0.4	31.14	2.6	13.6	0.1	14.1	0.1

The copper concentrate had a grade of 16.7% Cu and RCu was 66.3%. The final magnetite concentrate had Fe grade of 69.6% and the RFe was 65%. The magnetite recovery was 87.4% according to the Satmagan measurements. S content in the final concentrate was 0.05%.

5.14 27.9.2017 Wednesday, VE-2 Ore type

Plant was operated in three shifts from 07:50 till 29.9.2017 hour 13:00.

At 12:40, LIMS circuit was moved to handle copper rougher tails. After that separate flotation lines were arranged for WLC3 (pyrrhotite flotation) and combined LIMS Non-Mags WLT1-3 (pyrite flotation). The Non-Mags were dewatered with cyclones before Pyrite flotation line.

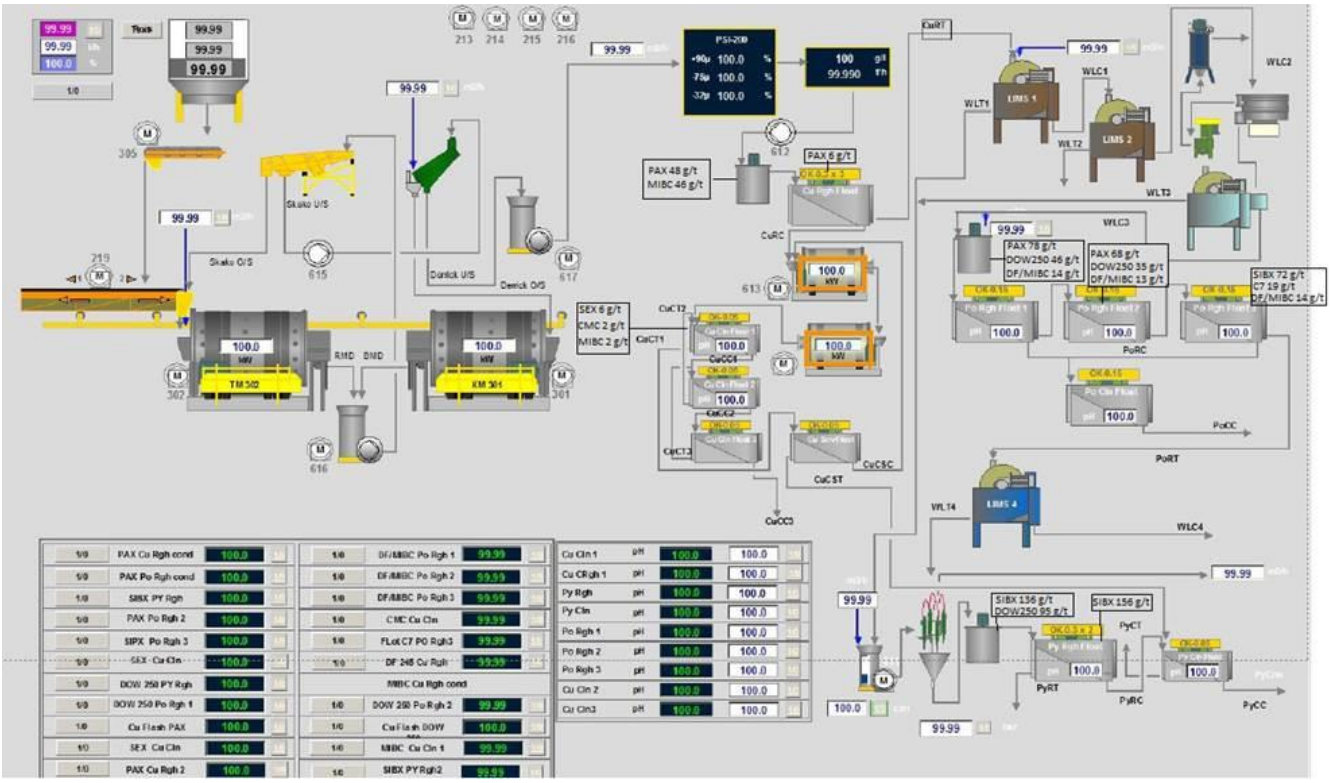

Figure 12. Process Flow Sheet 27.9.2017.

Table 38 shows the flotation reagent dosages.

June 6, 2018

Table 38. Reagents 27.9.

Date:		27.9.	27.9.
Time:		7:30	16:00
Reagent	Feed Point	g/t	g/t
SEX	Cu Cln 1	6	6
CMC	Cu Cln 1	2	2
MIBC	Cu Cln 1	2	
PAX	Cu Rgh 1	48	48
MIBC	Cu Rgh 1	46	46
PAX	Cu Rgh 2	6	6
DOW 250	Po Rgh 1	46	46
DOW 250	Po Rgh 2	34	36
PAX	Po. Flot 1	80	76
PAX	Po. Flot 2	68	68
DF245/MIBC	Po. Rgh 1	14	14
DF245/MIBC	Po. Rgh 2	14	13
SIBX	Po. Rgh 3	64	80
DF/MIBC	Po. Rgh 3	13	15
DOW 250	Po. Rgh 3	19	19
SIBX	Py Rgh	152	120
DOW 250	Py Rgh	104	86
SIBX	Py Rgh 1/2	192	120

Table 39. Process Flow Rate Measurements, 27.9. Hour 15:30

	Slurry SG g/l (**kg**)	Bucket l	Time s (**min**)	Flow m3/h	Solids %	Solids t/h
Crushed Ore	6.50		30.0	0.26	100.0	0.780
RM Disch.	1635	3.4	12.5	0.98	52.2	0.836
BM Disch.	1390	3.4	8.2	1.49	37.7	0.783
Skako O/S	2220	1.0	190.0	0.02	73.9	0.031
Skako U/S	1435	3.4	5.3	2.31	40.8	1.351
Derrick O/S	1355	1.0	3.0	1.20	35.2	0.573
Derrick U/S	1220	3.4	5.9	2.07	24.3	0.614
Derrick U/S	1215			2.44	23.8	0.705
WLC1	1430	1.0	7.8	0.46	38.0	0.251
WLT1	1125	3.4	4.8	2.55	16.7	0.478
WLC2	1450	3.4	27.1	0.45	39.2	0.257
WLT2	1005	3.4	13.5	0.91	0.7	0.007
WLC3	1555	1.0	10.0	0.36	45.1	0.252
WLT3	1005	3.4	10.3	1.19	0.7	0.009
PoFlot Feed	1285	1.0	5.3	0.68	28.0	0.245
PoFlot Tails 1	1190	1.0	4.7	0.77	20.2	0.184
PoFlot Tails 2	1180	3.4	12.6	0.97	19.3	0.221
PoFlot Tails 3	1300	1.0	9.7	0.37	29.1	0.141
WLC4	1720	1.0	19.3	0.19	52.9	0.170
WLT4	1002	3.0	14.7	0.73	0.3	0.002
Cy OF	1000	3.4	2.2	5.56	0.0	0.000
Cy UF	1220	3.4	9.8	1.25	27.0	0.412
	Power kW, Gross	Idle kW	Power kW, Net	Energy kWh/t, Net	%	Solids t/h
RM302	**5.20**	**2.0**	3.2	4.1	20.1	
BM301	**9.15**	**2.0**	7.2	9.2	44.8	
Cu Regrind	**2.7**	**0.4**	2.3	2.9	14.4	
Fe Regrind	**4.0**	**0.7**	3.3	4.2	20.7	
Grinding	**21.1**	**5.1**	16.0	20.4	100.0	

June 6, 2018

During the day, process was followed by frequent samplings from main streams, as presented in following:

	Time:	13:00			Time:	15:15			Time:	17:30			Time:	21:25
27.9.	CuRC	CuRT	CuCC3	WLC4	CuRT	CuCC3	PyRT	WLC4	PyRT	CuRT	WLC4	CuCC3	PyRT	WLC4
Cu	4.81	0.027	16	0.004	0.022	15.1	0.019	0.003	0.009	0.029	0.002	23	0.021	0.003
Fe	44.6	30	35.7	69.4	28.7	37.5	8.43	69.3	8.56	30.2	69.4	32.3	8.1	69.5
Eltra S	**37.7**	**2.39**	**45.2**	**0.061**	**2.45**	**46.7**	**0.78**	**0.064**	**0.802**	**2.59**	**0.061**	**40.6**	**0.5**	**0.087**
Satmagan	**10.65**	**37.51**	**0.26**	**95.26**	**29.55**	**0.26**	**1.18**	**95.67**	**1.33**	**32.77**	**96.59**	**0.38**	**1.08**	**96.94**

Full XRF assays are presented in Appendix 2.

5.15 28.9.2017 Thursday, VE-2 Ore type

Process samples were collected at 01:30 – 03:00. Full assays are presented in Appendix 2. The mass balance is presented in Table 40.

Table 40. Mass Balance 28.9. Hour 01:30 – 03:00

Stream	Total solids t/h	Rec %	Cu %	Cu Rec %	Fe %	Fe Rec %	S %	S Rec %	Satmagan %	Satmagan Rec %	SiO2 %	SiO2 Rec %
Derrick us	***750.0***	100.0	0.140	100.0	31.63	100.0	3.34	100.0	34.3	100.0	29.8	100.0
CuRC	***17.5***	2.3	5.266	87.5	43.67	3.2	43.65	30.5	6.2	0.4	2.8	0.2
CuRT	***732.5***	97.7	0.018	12.5	31.34	96.8	2.38	69.5	35.0	99.6	30.5	99.8
CuCC3	***4.4***	0.6	19.555	81.6	34.17	0.6	42.20	7.4	0.3	0.0	0.1	0.0
CuCST	***13.1***	1.7	0.471	5.9	46.86	2.6	44.14	23.1	8.1	0.4	3.7	0.2
WLT1-3	***464.5***	61.9	0.024	10.7	10.16	19.9	2.36	43.8	1.1	2.0	46.2	96.0
WLC3	***268.0***	35.7	0.007	1.9	68.04	76.9	2.40	25.7	93.6	97.5	3.2	3.8
PoRC	***44.7***	6.0	0.035	1.5	62.71	11.8	13.87	24.8	77.6	13.5	3.9	0.8
PoRT	***223.3***	29.8	0.002	0.4	69.11	65.1	0.10	0.9	96.8	84.0	3.0	3.0
WLT4	***2.0***	0.3	0.009	0.0	15.30	0.1	0.70	0.1	0.9	0.0	43.2	0.4
WLC4	***221.3***	29.5	0.002	0.3	69.60	64.9	0.10	0.9	97.7	84.0	2.7	2.6
CyOF	***52.5***	7.0	0.066	3.3	16.35	3.6	4.08	8.6	1.7	0.3	38.4	9.0
CyUF	***412.0***	54.9	0.019	7.4	9.37	16.3	2.15	35.3	1.1	1.7	47.2	87.0
PyRT	***386.0***	51.5	0.008	2.8	8.12	13.2	0.37	5.7	1.1	1.6	48.8	84.2
PyRC	***26.0***	3.5	0.186	4.6	28.05	3.1	28.55	29.6	1.3	0.1	23.8	2.8

The grade of CuCC3 was 19.8% Cu with RCu of 81.6%. Magnetite concentrate had Fe grade of 69.6% with 64.9% recovery. The recovery of magnetite was calculated to be 84%. With the new flow sheet option, the sulphur grade of the pyrite tailings could be lowered to 0.37% S. When taking into account the tailings streams (PyRT + WLT4), the S-content would be 0.36%.

Tailings were collected into thickener during the night, and also sample collection for WLC4 was done to another thickener.

The second plant survey of the process was done at 10:00 (28.9.2017). XRF assays are presented in Appendix 2 and the process mass balance in Table 41. A mass and water balance was also evaluated, and the balance is presented in Appendix4.

June 6, 2018

Table 41. Mass Balance 28.9. Hour 10:00

Stream	Total solids t/h	Rec %	Cu %	Cu Rec %	Fe %	Fe Rec %	S %	S Rec %	Satmagan %	Satmagan Rec %	SiO2 %	SiO2 Rec %
Derrick us	***751.0***	100.0	0.149	100.0	35.3	100.0	3.83	100.0	39.3	100.0	27.57	100.0
CuRC	***23.1***	3.1	4.405	91.3	40.3	3.5	45.56	36.6	7.4	0.6	5.52	0.6
CuRT	***727.9***	96.9	0.013	8.7	35.1	96.5	2.50	63.4	40.3	99.4	28.27	99.4
CuCC1	***15.9***	2.1	9.181	130.5	39.1	2.3	47.37	26.1	1.3	0.1	0.57	0.0
CuCT1	***22.5***	3.0	2.281	46.0	41.4	3.5	44.51	34.8	8.2	0.6	5.80	0.6
CuCST	***18.1***	2.4	0.819	13.3	41.7	2.9	44.95	28.3	9.4	0.6	6.98	0.6
CuCSC	***4.3***	0.6	8.412	32.6	40.2	0.7	42.70	6.4	3.6	0.1	0.82	0.0
CuCC2	***20.6***	2.7	15.635	289.0	34.5	2.7	42.54	30.5	0.8	0.1	0.40	0.0
CuCT2	***10.9***	1.4	5.385	52.4	40.8	1.7	47.16	17.8	1.7	0.1	0.75	0.0
CuCC3	***5.0***	0.7	17.450	78.0	35.3	0.7	47.81	8.3	0.4	0.0	0.19	0.0
CuCT3	***15.6***	2.1	15.055	211.0	34.2	2.0	40.86	22.2	1.0	0.1	0.46	0.0
WLC1	***375.0***	49.9	0.008	2.7	59.5	84.2	1.95	25.4	77.4	98.3	8.85	16.0
WLT1	***352.9***	47.0	0.019	5.9	9.2	12.3	3.09	38.0	1.0	1.2	48.91	83.4
WLC2	***365.7***	48.7	0.008	2.5	60.7	83.8	1.95	24.8	79.2	98.1	7.92	14.0
WLT2	***9.3***	1.2	0.028	0.2	12.6	0.4	2.01	0.7	4.3	0.1	45.36	2.0
WLC3	***322.1***	42.9	0.005	1.4	67.7	82.3	2.09	23.4	89.8	98.0	3.04	4.7
WLT3	***43.7***	5.8	0.027	1.1	8.9	1.5	0.88	1.3	0.9	0.1	43.95	9.3
WLT1-3	***405.8***	54.0	0.020	7.2	9.3	14.2	2.83	39.9	1.0	1.4	48.30	94.7
PyRC	***32.9***	4.4	0.083	2.4	19.0	2.4	27.20	31.1	1.0	0.1	37.20	5.9
CuCln1 Feed	***27.5***	3.7	5.037	124.0	40.3	4.2	45.11	43.1	6.8	0.6	4.78	0.6
CuCln2 Feed	***31.5***	4.2	12.098	341.4	36.7	4.4	44.14	48.3	1.1	0.1	0.52	0.1
PyRT	***346.4***	46.1	0.010	3.0	7.7	10.1	0.31	3.8	1.0	1.2	50.19	84.0
PyCC	***8.0***	1.1	1.407	10.0	41.6	1.3	43.12	11.9	3.4	0.1	6.33	0.2
PyCT	***43.1***	5.7	0.148	5.7	24.4	4.0	31.73	47.5	4.1	0.6	30.18	6.3
PoRC	***45.6***	6.1	0.023	0.9	56.8	9.8	14.45	22.9	50.5	7.8	5.41	1.2
PoRT	***276.4***	36.8	0.002	0.5	69.6	72.5	0.05	0.5	96.3	90.2	2.65	3.5
PoCC	***10.6***	1.4	0.046	0.4	60.2	2.4	26.51	9.8	52.2	1.9	2.48	0.1
PoCT	***35.0***	4.7	0.016	0.5	55.7	7.4	10.78	13.1	50.0	5.9	6.31	1.1
WLC4	***274.5***	36.5	0.002	0.5	69.9	72.4	0.05	0.5	96.8	90.0	2.39	3.2
WLT4	***2.0***	0.3	0.004	0.0	22.3	0.2	0.32	0.0	24.3	0.2	38.20	0.4
PyCln1 Feed	***51.0***	6.8	0.345	15.8	27.1	5.2	33.51	59.4	4.0	0.7	26.46	6.5
CyUF	***379.3***	50.5	0.016	5.4	8.7	12.5	2.64	34.9	1.0	1.3	49.07	89.9
Cy OF	***26.5***	3.5	0.077	1.8	17.1	1.7	5.49	5.1	1.5	0.1	37.32	4.8

The grade of CuCC3 was 17.5% Cu with RCu of 78%. Au content in CuRC was 1.11 g/t and the RAu was 48.8%. CuCC3 Au grade was 2.56 g/t with a recovery of 33.1%.

Magnetite concentrate had Fe grade of 69.9% with 72.4% recovery. Sulphur content was 0.05 %. The recovery of magnetite was calculated to be 90%. The sulphur grade of the pyrite tailings was 0.3% S. The reagent dosages in flotation circuit during sampling are presented in Table 42. The feed rates are given against the plant feed and against particular processing stage.

June 6, 2018

Table 42. Reagents 28.9.

Hannukainen pilot 2017 chemicals and dosages			
Ore type VE2			
Date:		28.9.	28.9.
Time:		10:00	10:00
Reagent	Feed Point	g/t*	g/t**
SEX	**Cu Cln 1**	6	151
CMC	**Cu Cln 1**	2	53
MIBC	**Cu Cln 1**	2	56
PAX	**Cu Rgh 1**	48	48
MIBC	**Cu Rgh 1**	46	46
PAX	**Cu Rgh 2**	6	54
DOW 250	**Po Rgh 1**	46	107
DOW 250	**Po Rgh 2**	35	189
PAX	**Po. Rgh 1**	78	182
PAX	**Po. Rgh 2**	68	340
DF/MIBC	**Po. Rgh 1**	14	33
DF/MIBC	**Po. Rgh 2**	13	96
SIBX	**Po. Rgh 3**	72	168
DF/MIBC	**Po. Rgh 3**	14	96
Flotanol C7	**Po. Rgh 3**	19	44
SIBX	**Py Rgh**	136	269
DOW 250	**Py Rgh**	95	188
SIBX	**Py Rgh 1/2**	156	309

*Values calculated against pilot ore mass feed
**Values calculated against mass feed of particular unit

Finnish name of the cell	Short name	Chemical	Dosage 28.9.2017 g/t*	Dosage 28.9.2017 g/t**
Kupari kertausvaahdotus1	**Cu Cln 1**	Ca(OH)2	681	18409
Kupari kertausvaahdotus2	**Cu Cln 2**	Ca(OH)2	148	3514
Kupari kertausvaahdotus3	**Cu Cln 3**	Ca(OH)2	58	4114
Kupari esivaahdotus	**Cu Rgh**	Ca(OH)2	942	942
Pyrrotiitti esivaahdotus1	**Po Rgh 1**	H2SO4	75	175
Pyrrotiitti esivaahdotus2	**PoRgh 2**	H2SO4	147	518
Pyrrotiitti esivaahdotus3	**PoRgh 3**	H2SO4	150	351
Pyriitti kertausvaahdotus	**Py Cln**	H2SO4	1381	20303
Pyriitti esivaahdotus	**Py Rgh**	H2SO4	389	770

*Values calculated against pilot ore mass feed
**Values calculated against mass feed of particular unit

It was noticed that the S-grade was increasing in WLC4, so the SIBX dosage in last pyrrhotite flotation cell was increased from 75 g/t to 100 g/t.

At 13:50, a secondary Lime feeding point was installed after the grinding circuit.

14:45 – 16:45 changes in cyclone configurations.

June 6, 2018

Table 43. Process Flow Rate Measurements, 28.9. Hour 15:30

	Slurry SG g/l (**kg**)	Bucket l	Time s (**min**)	Flow m3/h	Solids %	Solids t/h
Crushed Ore	6.20		30.0	0.25	100.0	0.744
RM Disch.	1645	3.4	11.2	1.09	52.7	0.948
BM Disch.	1410	3.4	6.7	1.83	39.1	1.007
Skako O/S	2120	1.0	321.0	0.01	71.0	0.017
Skako U/S	1490	3.4	3.1	3.95	44.2	2.602
Derrick O/S	1265	1.0	3.2	1.13	28.2	0.401
Derrick U/S	1235	3.4	5.2	2.35	25.6	0.744
Derrick U/S	1235			2.32	25.6	0.733
WLC1	1520	1.0	8.5	0.42	43.2	0.278
WLT1	1090	3.4	4.9	2.50	12.4	0.337
WLC2	1425	1.0	7.4	0.49	37.7	0.261
WLT2	1014	3.4	12.8	0.96	2.1	0.020
WLC3	1530	1.0	10.3	0.35	43.8	0.234
WLT3	1012	3.4	10.1	1.21	1.8	0.022
WLT1-3	1055	10.0	7.8	4.62	7.8	0.381
PoFlot Feed	1310	1.0	5.3	0.68	29.9	0.266
PoFlot Tails 1	1170	1.0	4.6	0.78	18.4	0.168
PoFlot Tails 2	1120	1.0	7.2	0.50	13.5	0.076
PoFlot Tails 3	1665	1.0	23.8	0.15	50.5	0.127
WLC4	1590	1.0	9.9	0.36	46.9	0.271
WLT4	1002	3.0	18.1	0.60	0.3	0.002
Cy OF	1005	10.0	10.2	3.53	0.7	0.026
Cy UF	1160	3.4	6.0	2.04	20.7	0.490
CuRgh Feed	1255	3.4	5.3	2.31	27.3	0.792
CuRC	1090	1.0	24.4	0.15	11.1	0.018
CuRT	1170	3.4	3.1	3.95	19.5	0.903
CuCST	1045	1.0	7.6	0.47	5.8	0.029
PyRT	1190	3.4	8.0	1.53	23.9	0.436
PyCC	1055	1.0	57.9	0.06	7.0	0.005
PyCT	1030	1.0	4.1	0.88	3.9	0.035
PoCC	1045	1.0	28.9	0.12	5.4	0.007
PoCT	1005	1.0	3.8	0.95	0.6	0.006
PyFeed	1160	1.0	2.8	1.29	18.5	0.277
CuCSC	1180	1.0	141.0	0.03	20.5	0.006
	Power kW, Gross	Idle kW	Power kW, Net	Energy kWh/t, Net	%	Solids t/h
RM302	**5.20**	**2.0**	3.2	4.3	18.5	
BM301	**9.10**	**2.0**	7.1	9.5	41.0	
Cu Regrind	**2.7**	**0.4**	2.3	3.1	13.3	
Fe Regrind	**5.4**	**0.7**	4.7	6.3	27.2	
Grinding	**22.4**	**5.1**	17.3	23.3	100.0	

PSD determinations were done for the main streams. Screen assays are presented in Appendix 2 and graphically in Figure 12. XRF assays were done for the fractions, and the assaying results are presented in Tables 44 -48.

Table 44. CuCC3, element distributions in size fractions.

Sieve opening	CuCC3			Cu		Fe		S		Satmagan		SiO2	
(µm)	Weight g	Cum.pass. %	Fraction %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %
75	3.8	96.0	4.0	6.00	1.5	43.80	5.1	50.20	4.6	0.41	3.9	0.59	12.8
45	18.7	76.3	19.7	9.39	11.1	39.90	22.5	48.90	22.0	0.26	12.1	0.21	22.2
32	15.3	60.2	16.1	12.90	12.5	37.50	17.3	45.20	16.6	0.26	9.9	0.19	16.5
-32	57.2		60.2	20.80	75.0	32.00	55.2	41.30	56.8	0.52	74.1	0.15	48.5
Tot.	**95.0**		**100.0**	**16.69**	**100.0**	**34.92**	**100.0**	**43.78**	**100.0**	**0.42**	**100.0**	**0.19**	**100.0**

In copper cleaner concentrate, the highest Cu grade was obtained in the finest fraction with the recovery of 75%. The P(80) value of the concentrate was 51 µm.

Table 45. WLC4, element distributions in size fractions.

Sieve opening	WLC4			Cu		Fe		S		Satmagan		SiO2	
(µm)	Weight g	Cum.pass. %	Fraction %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %
75	7.4	91.0	9.0	0.048	25.25	68.1	8.74	0.236	26.59	94.09	8.69	3.58	14.04
45	38.6	44.4	46.6	0.008	21.87	68.9	45.94	0.056	32.79	96.33	46.22	3.09	62.97
32	14.4	27.0	17.4	0.013	13.27	70.8	17.63	0.071	15.52	98.68	17.68	1.64	12.48
-32	22.3		27.0	0.025	39.60	71.7	27.70	0.074	25.10	98.63	27.42	0.89	10.51
Tot.	**82.7**		**100.0**	**0.02**	**100.00**	**69.92**	**100.00**	**0.08**	**100.00**	**97.16**	**100.00**	**2.29**	**100.00**

The final magnetite concentrate had a P(80) value of 68 µm. Iron is distributed along the size fraction, and the highest Fe grades are obtained in the fine end. The impurities, sulphur and silica can be found mostly in the coarse fractions.

Table 46. PyCC, element distributions in size fractions.

Sieve opening	PyCC1			Cu		Fe		S		Satmagan		SiO2	
(µm)	Weight g	Cum.pass. %	Fraction %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %
		100.0	0.0		0.00		0.00		0.00		0.00		0.00
75	4.2	78.8	21.2	0.2	3.18	39.3	19.95	43.6	20.90	0.72	4.52	8.7	29.34
45	3.0	63.6	15.2	0.458	5.24	43.7	15.93	46.1	15.87	0.98	4.42	5.96	14.43
32	1.8	54.2	9.4	0.84	5.93	44.6	10.04	46.3	9.84	2.05	5.70	4.9	7.33
-32	10.6		54.2	2.1	85.65	41.6	54.09	43.5	53.40	5.31	85.36	5.66	48.90
Tot.	**19.6**		**100.0**	**1.33**	**100.00**	**41.71**	**100.00**	**44.18**	**100.00**	**3.37**	**100.00**	**6.28**	**100.00**

P(80) was 78 µm. Copper is concentrated in fine fraction, other elements are following the particle size distribution.

Table 47. PoCC, element distributions in size fractions.

Sieve opening	PoCC			Cu		Fe		S		Satmagan		SiO2	
(µm)	Weight g	Cum.pass. %	Fraction %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %
75	3.0	89.9	10.1	0.075	16.11	58.8	9.56	33.2	16.43	28.73	5.49	1.54	6.50
45	8.3	61.8	28.1	0.059	35.18	58.6	26.46	30.8	42.32	32.3	17.14	2.33	27.32
32	3.0	51.6	10.2	0.048	10.34	60.4	9.86	26.6	13.21	51.08	9.80	2.34	9.92
-32	15.3		51.6	0.035	38.37	65.2	54.12	11.1	28.04	69.24	67.56	2.61	56.26
Tot.	**29.6**		**100.0**	**0.05**	**100.00**	**62.21**	**100.00**	**20.44**	**100.00**	**52.92**	**100.00**	**2.40**	**100.00**

P(80) of pyrrhotite cleaner concentrate was 65 µm.

Table 48. PyRT, element distributions in size fractions.

Sieve opening	PyRT			Cu		Fe		S		Satmagan		SiO2	
(µm)	Weight g	Cum.pass. %	Fraction %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %	Grade %	Rec %
500		100.0											
90	12.5	74.1	25.9	0.01	23.35	7.92	25.24	0.28	20.20	1.28	29.06	49.80	26.34
75	6.6	60.5	13.6	0.007	9.50	7.84	13.07	0.262	10.03	1.28	15.20	49.6	13.72
45	13.0	33.8	26.8	0.007	18.79	7.73	25.48	0.274	20.75	0.98	23.01	49.3	26.97
32	4.6	24.2	9.5	0.01	9.56	7.71	9.05	0.27	7.28	0.92	7.69	48.2	9.39
-32	11.7		24.2	0.016	38.80	9.12	27.16	0.61	41.73	1.18	25.04	47.7	23.58
Tot.	**48.4**		**100.0**	**0.01**	**100.00**	**8.13**	**100.00**	**0.35**	**100.00**	**1.14**	**100.00**	**48.98**	**100.00**

P(80) value for the pyrite rougher tailings was 102 µm, and the highest S content was in the finest fraction.

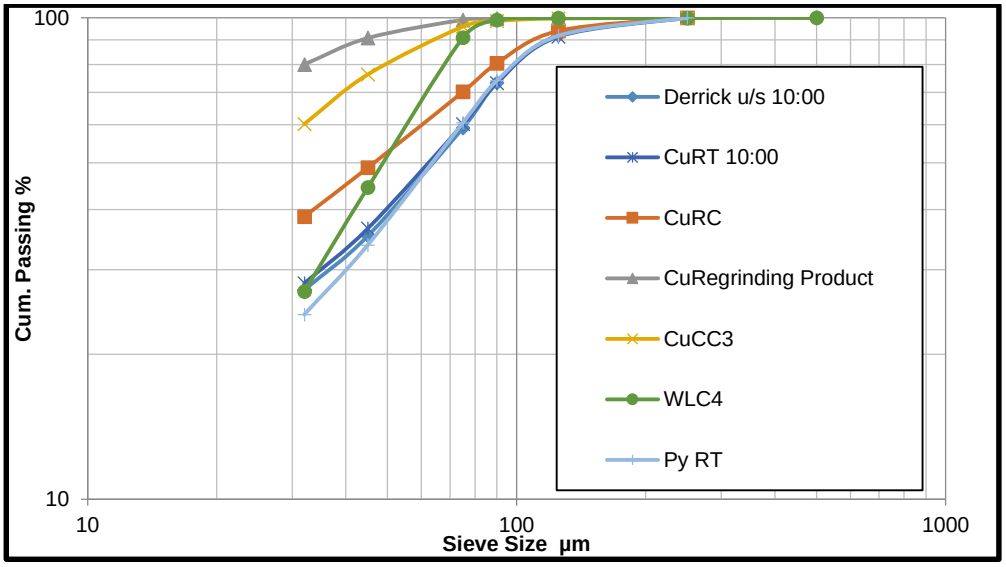

Figure 12. Process PSD's, 28.9.2017

5.16 29.9.2017 Friday, VE-2 Ore type

Table 49. Mass Balance 29.9. Hour 00:30 – 03:00

Stream	Total solids t/h	Rec %	Cu %	Cu Rec %	Fe %	Fe Rec %	S %	S Rec %	Satmagan %	Satmagan Rec %	SiO2 %	SiO2 Rec %
Derrick us	***751.7***	100.0	0.137	100.0	33.36	100.0	3.65	100.0	36.1	100.0	28.9	100.0
CuRC	***24.6***	3.3	3.695	88.3	39.25	3.8	44.84	40.2	11.1	1.0	12.5	1.4
CuRT	***727.1***	96.7	0.017	11.7	33.16	96.2	2.26	59.8	37.0	99.0	29.4	98.6
CuCC3	***3.3***	0.4	21.612	68.7	33.13	0.4	45.24	5.4	0.4	0.0	0.2	0.0
CuCST	***21.3***	2.8	0.944	19.5	40.19	3.4	44.77	34.8	12.8	1.0	14.3	1.4
WLT1-3	***442.3***	58.8	0.025	10.5	10.18	18.0	2.46	39.6	1.1	1.8	46.5	94.8
WLC3	***284.8***	37.9	0.004	1.2	68.85	78.2	1.95	20.2	92.7	97.2	2.9	3.8
PoRC	***29.1***	3.9	0.032	0.9	62.86	7.3	17.40	18.5	61.5	6.6	3.1	0.4
PoRT	***255.7***	34.0	0.001	0.3	69.53	70.9	0.19	1.7	96.3	90.6	2.8	3.4
WLT4	***1.7***	0.2	0.010	0.0	17.20	0.1	1.36	0.1	13.1	0.1	42.1	0.3
WLC4	***254.0***	33.8	0.001	0.3	69.89	70.8	0.18	1.7	96.8	90.5	2.6	3.0
CyOF	***48.3***	6.4	0.082	3.9	17.24	3.3	4.93	8.7	1.5	0.3	37.2	8.3
CyUF	***394.0***	52.4	0.017	6.7	9.31	14.6	2.15	30.9	1.0	1.5	47.7	86.5
PoCC	***4.1***	0.5	0.064	0.3	61.89	1.0	24.48	3.7	45.0	0.7	1.4	0.0
PoCT	***25.0***	3.3	0.026	0.6	63.02	6.3	16.24	14.8	64.3	5.9	3.4	0.4
PyRT	***385.9***	51.3	0.015	5.6	8.93	13.7	1.66	23.4	1.0	1.5	48.1	85.6
PyRC	***8.1***	1.1	0.132	1.0	27.40	0.9	25.60	7.6	1.1	0.0	25.2	0.9

The grade of CuCC3 was 21.6% Cu with RCu of 68.7%. Magnetite concentrate had Fe grade of 69.9% with 70.9% recovery. Sulphur content was 0.18 %. The recovery of magnetite was calculated to be 90.5%. The sulphur grade of the pyrite tailings was 1.66% S.

At 06:45 third rougher stage for Pyrite flotation was taken in operation in order to decrease the S-content in pyrite tails. The last process samples were taken 12:00 – 13:00. XRF assays are presented in Appendix 2. The main assays are presented below:

29.9. 12:00-13:00	CuCC3 S17069731	WLC4 S17069732	PyCC S17069733	PoCC S17069734
Cu	19	0.003	3.25	0.05
Fe	35.1	70.1	44.2	62.1
Eltra S	**42.6**	**0.035**	**45.5**	**20.7**
Satmagan	**0.41**	**98.27**	**3.83**	**50.36**

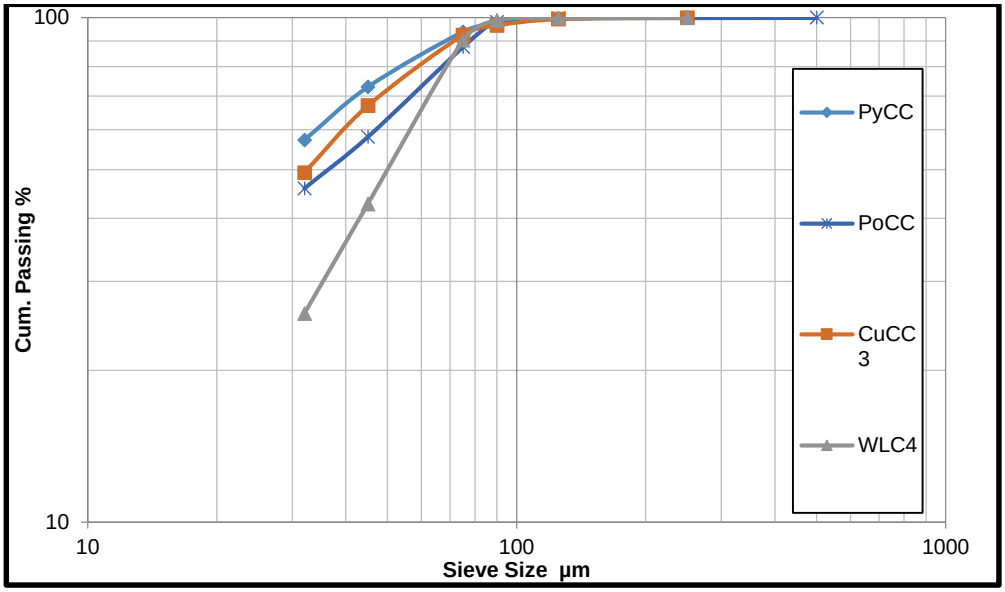

Figure 13. Process PSD's, 29.9.2017

5.17 Cobalt in Flotation Process

Co assays were added to HSC mass balances to evaluate how cobalt is behaving in the process. The cobalt balances are presented in the following Tables 50 - 59.

Table 50. Cobalt Mass Balance 14.9. Hour 17:30 – 18:30

Stream	Total solids kg/h	Rec %	Co %	Co Rec %
Derrick us	***749.9***	100.0	0.037	100.0
CuRC	***32.2***	4.3	0.273	32.1
CuRT	***717.7***	95.7	0.026	68.0
CuCST	***25.5***	3.4	0.265	24.6
CuCC3	***6.7***	0.9	0.302	7.4
PyFeed	***743.2***	99.1	0.034	92.1
PyRT	***656.5***	87.5	0.028	67.9
PyRC	***86.7***	11.6	0.078	24.6
WLC3	***292.3***	39.0	0.018	18.9
WLT1-3	***364.1***	48.6	0.037	49.1
PoRC	***29.1***	3.9	0.069	7.3
PoRT	***263.3***	35.1	0.012	11.5
WLT4	***1.0***	0.1	0.025	0.1
WLC4	***262.3***	35.0	0.012	11.5

June 6, 2018

Table 51. Cobalt Mass Balance 19.9. Hour 21:00

Stream	Total solids kg/h	Rec %	Co %	Co Rec %
Derrick us	***749.9***	100.0	0.038	100.0
CuRC	***18.6***	2.5	0.269	17.6
CuRT	***731.3***	97.5	0.032	82.1
CuCC4	***2.5***	0.3	0.189	1.6
CuCT1	***16.1***	2.2	0.281	15.9
PyFeed	***747.4***	99.7	0.035	91.8
PyCC	***21.6***	2.9	0.359	27.2
PyRT	***725.8***	96.8	0.025	63.7
WLC3	***364.8***	48.6	0.028	35.8
WLT1-3	***361.0***	48.1	0.021	26.6
PoCC	***101.8***	13.6	0.071	25.4
PoRT	***263.0***	35.1	0.011	10.2
WLC4	***262.0***	34.9	0.011	10.1
WLT4	***1.0***	0.1	0.031	0.1

Table 52. Cobalt Mass Balance 20.9. Hour 12:00

Stream	Total solids t/h	Rec %	Co %	Co Rec %
Derrick us	***750.1***	100.0	0.033	100.0
CuRC	***21.5***	2.9	0.295	25.8
CuRT	***728.6***	97.1	0.025	74.3
CuCC4	***2.6***	0.3	0.173	1.8
CuCT4	***18.9***	2.5	0.188	14.5
PyFeed	***747.5***	99.7	0.031	94.5
PyRC	***62.5***	8.3	0.045	11.5
PyRT	***684.9***	91.3	0.030	83.8
WLC3	***324.9***	43.3	0.023	30.5
WLT1-3	***360.0***	48.0	0.037	54.3
PoRC	***31.2***	4.2	0.072	9.2
PoRT	***293.7***	39.2	0.018	21.6
WLC4	***291.7***	38.9	0.018	21.4
WLT4	***2.0***	0.3	0.034	0.3

Table 53. Cobalt Mass Balance 20.9. Hour 16:00 -17:00

Stream	Total solids t/h	Rec %	Co %	Co Rec %
Feed	***750.1***	100.0	0.040	100.0
Skako os	***20.8***	2.8	0.040	2.8
Skako us	***1276.9***	170.2	0.029	123.4
Derick os	***544.0***	72.5	0.026	47.9
Derrick us	***732.9***	97.7	0.031	76.0
Flash Conc	***17.2***	2.3	0.313	17.9
Flash Tails	***526.8***	70.2	0.017	29.8
CuRC	***12.0***	1.6	0.220	8.8
CuRT	***720.9***	96.1	0.028	67.3
ReGrinding Discharge	***29.2***	3.9	0.263	25.6
CuCC4	***6.0***	0.8	0.342	6.8
CuCT1	***23.2***	3.1	0.242	18.7
PyFeed	***744.1***	99.2	0.030	74.4
PyRC	***13.5***	1.8	0.291	13.1
PyRT	***730.6***	97.4	0.025	61.4
WLC3	***419.1***	55.9	0.018	25.1
WLT1-3	***311.5***	41.5	0.035	36.3
PoRC	***148.8***	19.8	0.049	24.3
PoRT	***270.3***	36.0	0.001	0.9
WLC4	***262.1***	34.9	0.001	0.6
WLT4	***8.2***	1.1	0.012	0.3

June 6, 2018

Table 54. Cobalt Mass Balance 25.9. Hour 10:30

Stream	Total solids kg/h	Rec %	Co %	Co Rec %
Derrick us	***750.0***	100.0	0.026	100.0
CuRC	***23.0***	3.1	0.179	21.1
CuRT	***727.0***	96.9	0.021	78.3
CuCC2	***2.2***	0.3	0.174	2.0
CuSCT	***20.8***	2.8	0.179	19.1
PyFeed	***747.8***	99.7	0.022	84.4
PyRC	***10.4***	1.4	0.162	8.6
PyRT	***737.4***	98.3	0.020	75.6
WLC3	***72.3***	9.6	0.018	6.7
WLT1-3	***665.1***	88.7	0.020	68.2
PoRC	***5.4***	0.7	0.023	0.6
PoRT	***66.9***	8.9	0.018	6.2
WLC4	***63.0***	8.4	0.018	5.8
WLT4	***3.9***	0.5	0.013	0.3

Table 55. Cobalt Mass Balance 25.9. Hour 10:30

Stream	Total solids kg/h	Rec %	Co %	Co Rec %
Derrick us	***750.0***	100.0	0.026	100.0
CuRC	***23.0***	3.1	0.179	21.1
CuRT	***727.0***	96.9	0.021	78.3
CuCC2	***2.2***	0.3	0.174	2.0
CuSCT	***20.8***	2.8	0.179	19.1
PyFeed	***747.8***	99.7	0.022	84.4
PyRC	***10.4***	1.4	0.162	8.6
PyRT	***737.4***	98.3	0.020	75.6
WLC3	***72.3***	9.6	0.018	6.7
WLT1-3	***665.1***	88.7	0.020	68.2
PoRC	***5.4***	0.7	0.023	0.6
PoRT	***66.9***	8.9	0.018	6.2
WLC4	***63.0***	8.4	0.018	5.8
WLT4	***3.9***	0.5	0.013	0.3

June 6, 2018

Table 56. Cobalt Mass Balance 25.9. Hour 13:40

Stream	Total solids kg/h	Rec %	Co %	Co Rec %
Derrick us	***750.6***	100.0	0.029	100.0
CuRC	***19.1***	2.5	0.178	15.6
CuRT	***731.5***	97.5	0.025	84.0
CuCC1	***15.1***	2.0	0.227	15.8
CuCT1	***20.2***	2.7	0.203	18.8
CuCC2	***19.4***	2.6	0.193	17.2
CuCT2	***12.1***	1.6	0.252	14.0
CuCC3	***3.1***	0.4	0.158	2.2
CuCT3	***16.3***	2.2	0.218	16.4
CuCSC	***4.1***	0.6	0.243	4.6
CuCST	***16.0***	2.1	0.196	14.4
PyFeed	***681.0***	90.7	0.027	84.5
PyRC	***0.8***	0.1	0.251	1.0
PyRT	***680.2***	90.6	0.024	75.0
PyCC	***0.8***	0.1	0.253	0.9
PyCT	***0.0***	0.0	0.068	0.0
WLC1	***76.6***	10.2	0.015	5.3
WLT1	***654.9***	87.3	0.013	39.1
WLC2	***68.2***	9.1	0.018	5.6
WLT2	***8.4***	1.1	0.014	0.5
WLC3	***66.6***	8.9	0.009	2.8
WLT3	***1.7***	0.2	0.019	0.1
WLT1-3	***665.0***	88.6	0.013	39.7
WLC4	***60.3***	8.0	0.018	5.0
WLT4	***0.7***	0.1	0.013	0.0
PoRC	***5.6***	0.7	0.028	0.7
PoRT	***61.0***	8.1	0.014	3.9

Table 56. Cobalt Mass Balance 26.9. Hour 20:00 – 20:45

Stream	Total solids kg/h	Rec %	Co %	Co Rec %
Derrick us	***750.0***	100.0	0.027	100.0
CuRC	***12.5***	1.7	0.270	16.7
CuRT	***737.5***	98.3	0.023	83.8
CuCC3	***3.7***	0.5	0.253	4.6
CuCST	***8.8***	1.2	0.277	12.0
WLT1-3	***492.1***	65.6	0.030	73.4
WLC3	***245.4***	32.7	0.010	12.5
PoRC	***35.0***	4.7	0.048	8.3
PoRT	***210.4***	28.1	0.004	4.2
WLT4	***2.0***	0.3	0.015	0.1
WLC4	***208.4***	27.8	0.004	4.1
CyOF	***61.5***	8.2	0.027	8.2
CyUF	***430.6***	57.4	0.031	65.3
PyRT	***428.4***	57.1	0.030	63.5
PyRC	***2.2***	0.3	0.151	1.6

June 6, 2018

Table 57. Cobalt Mass Balance 28.9. Hour 01:30 – 03:00

Stream	Total solids kg/h	Rec %	Co %	Co Rec %
Derrick us	***750.0***	100.0	0.029	100.0
CuRC	***17.5***	2.3	0.279	22.8
CuRT	***732.5***	97.7	0.023	77.2
CuCC3	***4.4***	0.6	0.197	4.0
CuCST	***13.1***	1.7	0.306	18.7
WLT1-3	***464.5***	61.9	0.026	55.2
WLC3	***268.0***	35.7	0.018	22.1
PoRC	***44.7***	6.0	0.051	10.6
PoRT	***223.3***	29.8	0.011	11.5
WLT4	***2.0***	0.3	0.018	0.2
WLC4	***221.3***	29.5	0.011	11.3
CyOF	***52.5***	7.0	0.048	11.7
CyUF	***412.0***	54.9	0.023	43.4
PyRT	***386.0***	51.5	0.016	28.8
PyRC	***26.0***	3.5	0.121	14.7

Table 58. Cobalt Mass Balance 28.9. Hour 10:00

Stream	Total solids kg/h	Rec %	Co %	Co Rec %
Derrick us	***751***	100.0	0.036	100.0
CuRC	***23.13***	3.080	0.296	25.3
CuRT	***728***	96.9	0.028	75.4
CuCln1 Feed	***38.3***		**0.277**	39.3
CuCC1	***15.85***	2.111	0.306	17.9
CuCT1	***22.47***	2.992	0.257	21.4
CuCST	***18.14***	2.416	0.255	17.1
CuCSC	***4.33***	0.576	0.280	4.5
CuCln2 Feed	***31.49***		**0.271**	31.6
CuCC2	***20.62***	2.746	0.230	17.5
CuCT2	***10.87***	1.447	0.349	14.0
CuCln3 Feed	***20.62***		**0.221**	16.9
CuCC3	***4.99***	0.664	0.234	4.3
CuCT3	***15.63***	2.082	0.217	12.5
WLC1	***375***	49.9	0.014	19.4
WLT1	***353***	47.0	0.025	32.6
WLC2	***366***	48.7	0.021	28.4
WLT2	***9.30***	1.239	0.025	0.9
WLC3	***322***	42.9	0.020	23.8
WLT3	***43.7***	5.81	0.016	2.6
WLT1-3	***406***	54.0	0.035	52.5
PyRC	***32.9***	4.38	0.205	24.9
PyRT	***346***	46.1	0.018	23.1
PyCC	***7.96***	1.060	0.251	7.4
PyCT	***43.1***	5.73	0.110	17.5
PoRC	***45.6***	6.08	0.046	7.8
PoRT	***276.4***	36.8	0.001	1.0
PoCC	***10.64***	1.417	0.070	2.8
PoCT	***35.0***	4.66	0.035	4.5
WLC4	***274.5***	36.5	0.004	4.1
WLT4	***1.958***	0.2608	0.007	0.1
PyCln1 Feed	***51.0***	6.79	0.139	26.2
CyUF	***379***	50.5	0.034	47.7
Cy OF	***26.54***	3.53	0.051	5.0

Table 59. Cobalt Mass Balance 29.9. Hour 00:30 – 03:00

Stream	Total solids kg/h	Rec %	Co %	Co Rec %
Derrick us	***751.7***	100.0	0.033	100.0
CuRC	***24.6***	3.3	0.205	20.3
CuRT	***727.1***	96.7	0.027	79.4
CuCC3	***3.3***	0.4	0.177	2.3
CuCST	***21.3***	2.8	0.210	18.0
WLT1-3	***442.3***	58.8	0.034	59.9
WLC3	***284.8***	37.9	0.017	19.5
PoRC	***29.1***	3.9	0.059	6.9
PoRT	***255.7***	34.0	0.012	12.4
PoCC	***4.1***	0.5	0.081	1.3
PoCT	***25.0***	3.3	0.055	5.5
WLT4	***1.7***	0.2	0.021	0.1
WLC4	***254.0***	33.8	0.012	12.3
CyOF	***48.3***	6.4	0.030	5.8
CyUF	***394.0***	52.4	0.034	54.0
PyRT	***385.9***	51.3	0.032	49.8
PyRC	***8.1***	1.1	0.112	3.7

6 MINERALOGY

6.1 Mineralogy of VE2 Pilot Samples

The samples were measured using MLA-instrument (FEI Quanta 600 + EDAX Apollo XL EDX detectors). The measurement method was XBSE. Microscope specimens were vertical polished sections.

The measured samples are presented in Table 60.

Table 60. Measured samples

Product	Date	Time	Microscope specimen no.
CuKR3	28.9.17	10:00	OK15518
2SY	26.9.17	10:00	OK15519
PyKR1	28.9.17	10:00	OK15520
PyKJ1	28.9.17	10:00	OK15521
PyEJ	28.9.17	10:00	OK15522
PoKR	28.9.17	10:00	OK15523
PoKJ	28.9.17	10:00	OK15524
LIMS M4	28.9.17	10:00	OK15525
LIMS EM4	28.9.17	10:00	OK15526

The modal mineralogy is presented in Table 61. The particle sizes are presented in Figures 14 – 22.

Table 61. Mineral contents of VE2 Process Products

Mineral		CuKR3	2SY	PYKR1	PYKJ1	PYEJ	POKR	POKJ	LIMS_M4	LIMS_EM4
		CuCC3	CyO/F	PyCC	PyCT	PyRT	PoCC	PoCT	WLC4	WLT4
Quartz	SiO_2	0.00	0.61	0.14	0.79	1.41	0.00	0.00	0.00	0.06
Plagioclase	$Na_{0.5}Ca_{0.5}Si_3AlO_8$	0.00	3.21	0.44	2.45	4.05	0.03	0.15	0.07	0.72
K_feldspar	$KAlSi_3O_8$	0.01	1.17	0.08	1.03	1.73	0.02	0.13	0.00	0.08
Clinopyroxene	$(Ca,Mg,Fe,Al)_2(Si,Al)_20_6$	0.01	16.75	3.96	27.19	53.41	1.77	5.41	2.08	49.80
Ferrotschermakite	$(Ca,Fe)_2(Al_2Si_6O_{22})(OH)_2$	0.00	19.51	1.08	6.07	7.43	0.67	1.31	0.49	5.84
Actinolite	$Ca2Mg3Fe^{2+}{}_2Si_8O_{22}(OH)_2$	0.01	8.36	0.33	1.50	1.10	0.30	0.56	0.09	1.36
Epidote	$Ca_2Al_2(Fe^{3+},Al)(SiO_4)(Si_2O_7)O(OH)$	0.01	2.22	0.69	2.44	3.67	0.08	0.21	0.13	1.63
Tremolite	$Ca_2Mg_5(Si_8O_{22})(OH)_2$	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Vesuvianite		0.00	0.08	0.01	0.14	0.10	0.00	0.01	0.00	0.07
Andradite	$Ca_3Fe^{2+}{}_2(SiO_4)_3$	0.00	0.11	0.00	0.04	0.06	0.00	0.01	0.03	0.10
Marialite	$Na_4Al_3Si_9O_{24}Cl$	0.01	17.99	2.55	13.02	19.12	0.60	2.03	0.91	14.42
Chlorite	$(Mg,Fe)_3(Si,Al)_4O_{10}(OH)_2\cdot(Mg,Fe)_3(OH)_6$	0.01	0.84	0.20	0.49	0.81	0.04	0.11	0.02	0.19
Biotite	$K(Mg,Fe)_3(OH,F)_2(AlSi_3O_{10})$	0.00	0.06	0.01	0.08	0.04	0.01	0.01	0.00	0.04
Titanite	$CaTiSiO_5$	0.01	1.33	0.15	0.83	0.76	0.16	0.25	0.05	0.42
Clay		0.00	1.94	0.12	0.41	0.88	0.06	0.10	0.01	0.16
Berthierine	$(Fe^{2+},Fe^{3+},Al,Mg)_{2-3}(Si,Al)_2O_5(OH)_4$	0.01	11.07	0.65	2.06	1.50	0.34	0.59	0.09	0.94
Apatite	$Ca_5(PO_4)(F,Cl,OH)$	0.00	0.31	0.04	0.19	0.49	0.00	0.01	0.00	0.10
Allanite	$(Ce,Ca,Y)_2(Al,Fe^{3+})_3(SiO_4)_3(OH)$	0.00	1.55	0.01	0.09	0.02	0.01	0.01	0.00	0.04
Calcite	$CaCO_3$	0.04	0.04	0.00	0.00	0.07	0.00	0.00	0.05	0.00
Pyrite	Fe_2S_2	**46.21**	**5.77**	**73.39**	**30.91**	**0.33**	**1.21**	**0.21**	**0.02**	**0.01**
Pyrrhotite	$Fe^{2+}{}_{0.95}S$	**0.37**	**0.29**	**8.52**	**4.32**	**0.02**	**47.58**	**18.40**	**0.11**	**0.56**
Alteration of Pyrrhotite		0.05	2.78	1.86	1.57	0.08	4.83	2.59	0.01	0.07
Chalcopyrite	$CuFe^{2+}S_2$	**52.88**	**0.15**	**4.19**	**0.51**	**0.00**	**0.07**	**0.00**	**0.00**	**0.00**
Magnetite	$Fe^{3+}{}_2Fe^{2+}O_4$	**0.01**	**0.28**	**0.69**	**1.59**	**0.61**	**37.95**	**63.23**	**94.71**	**22.07**
Uraninite	UO_2	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Goethite	$Fe^{3+}O(OH)$	0.02	2.55	0.55	1.78	2.07	4.07	4.34	0.97	1.16
Process_metal		0.02	0.00	0.03	0.09	0.11	0.17	0.25	0.13	0.05
Unknown		0.05	0.89	0.21	0.32	0.09	0.04	0.06	0.01	0.04
Total		100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00

June 6, 2018

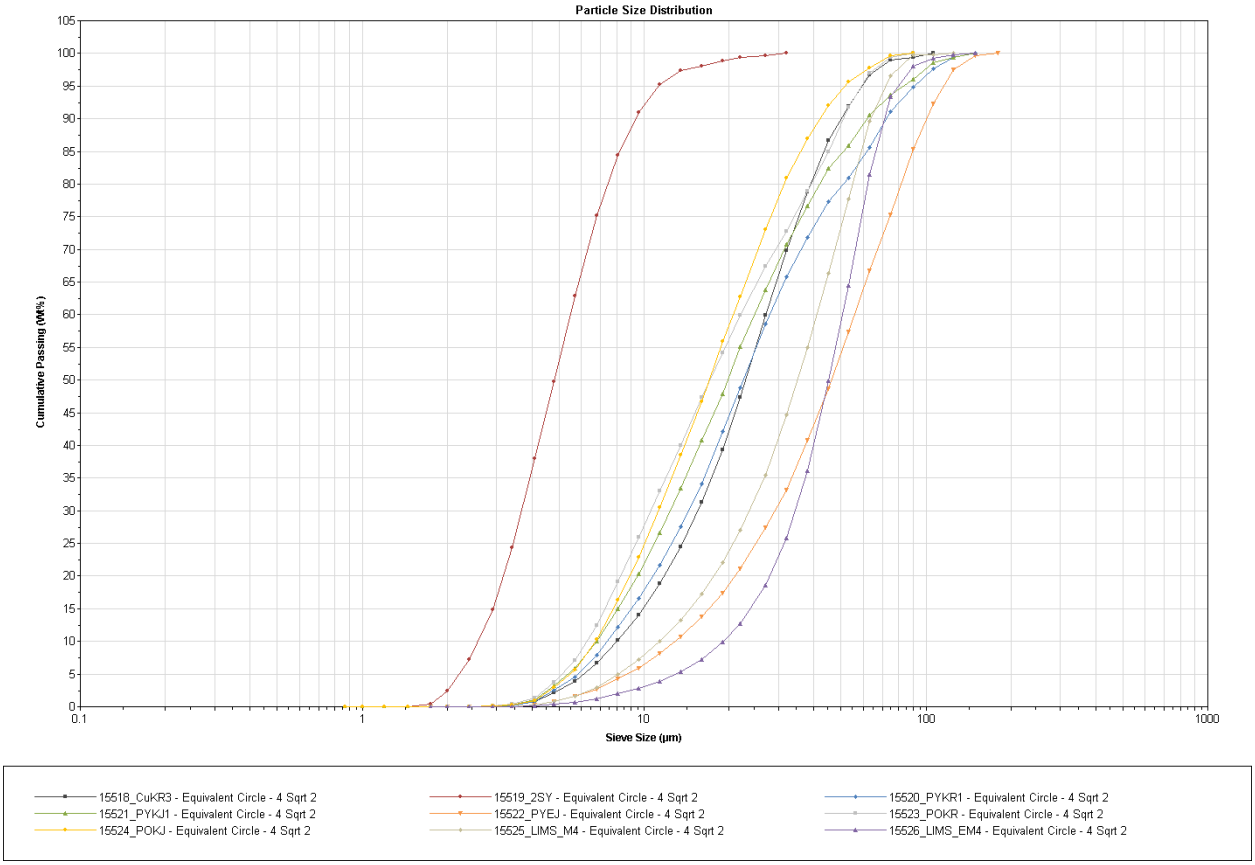

Figure 14. Particle Size of the Samples.

June 6, 2018

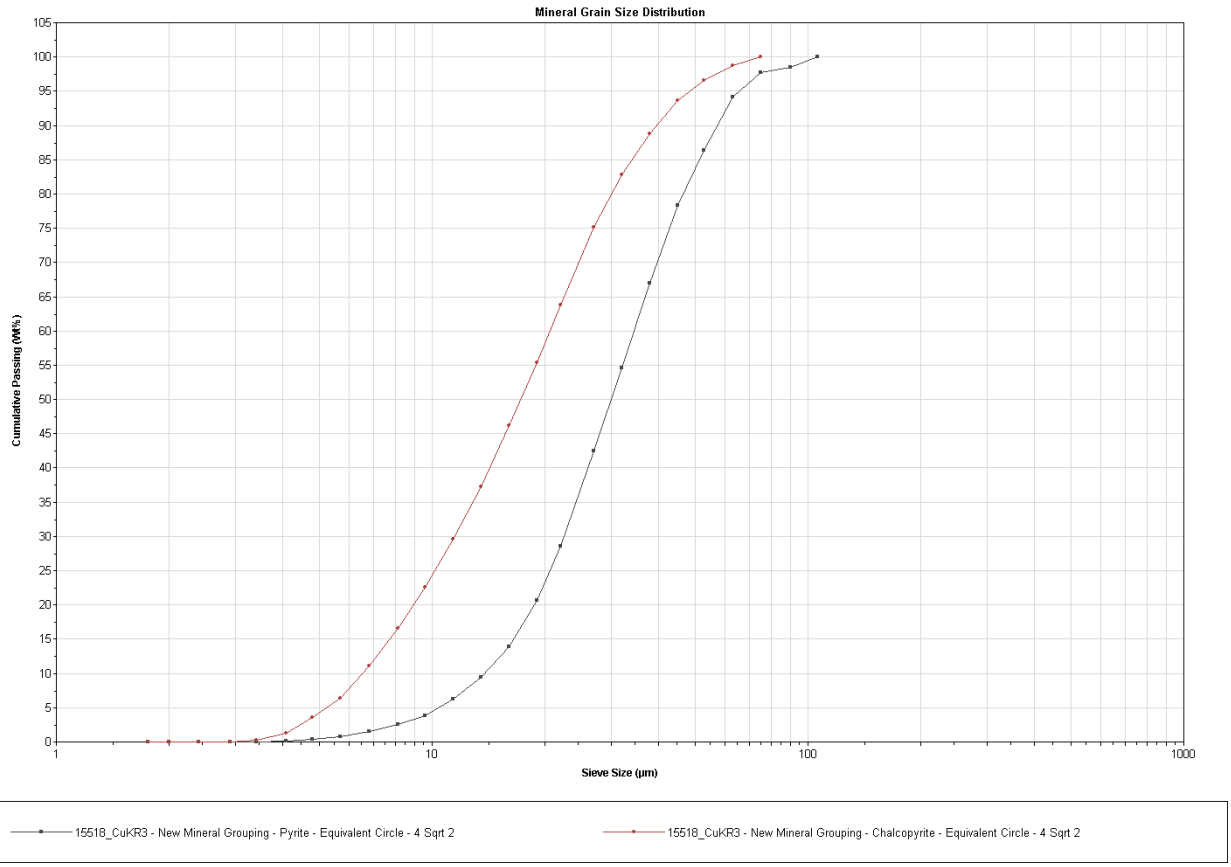

Figure 15. Particle Size of Pyrite and Chalcopyrite in CuCC3 (CuKR3).

June 6, 2018

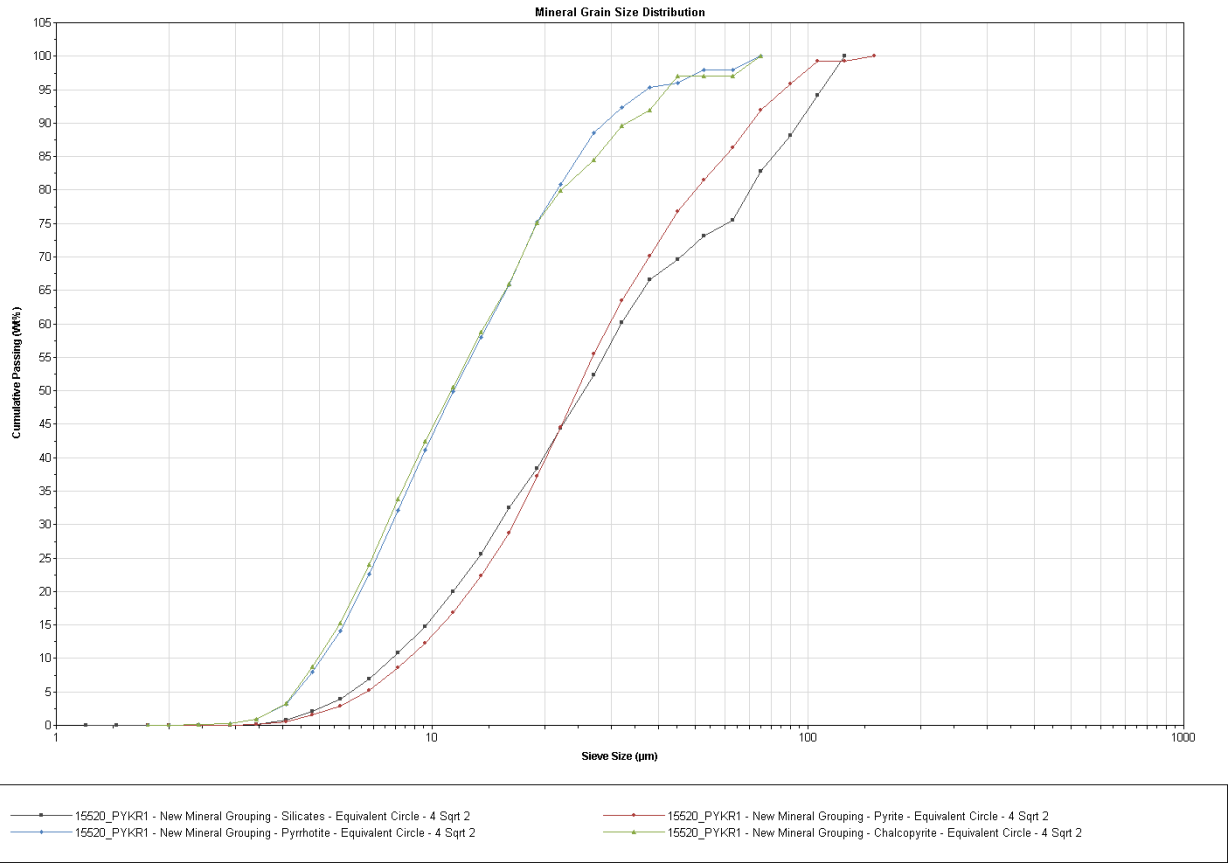

Figure 16. Particle Size of Silicates, Pyrite, Pyrrhotite and Chalcopyritein PyCC1 (PyKR1)

June 6, 2018

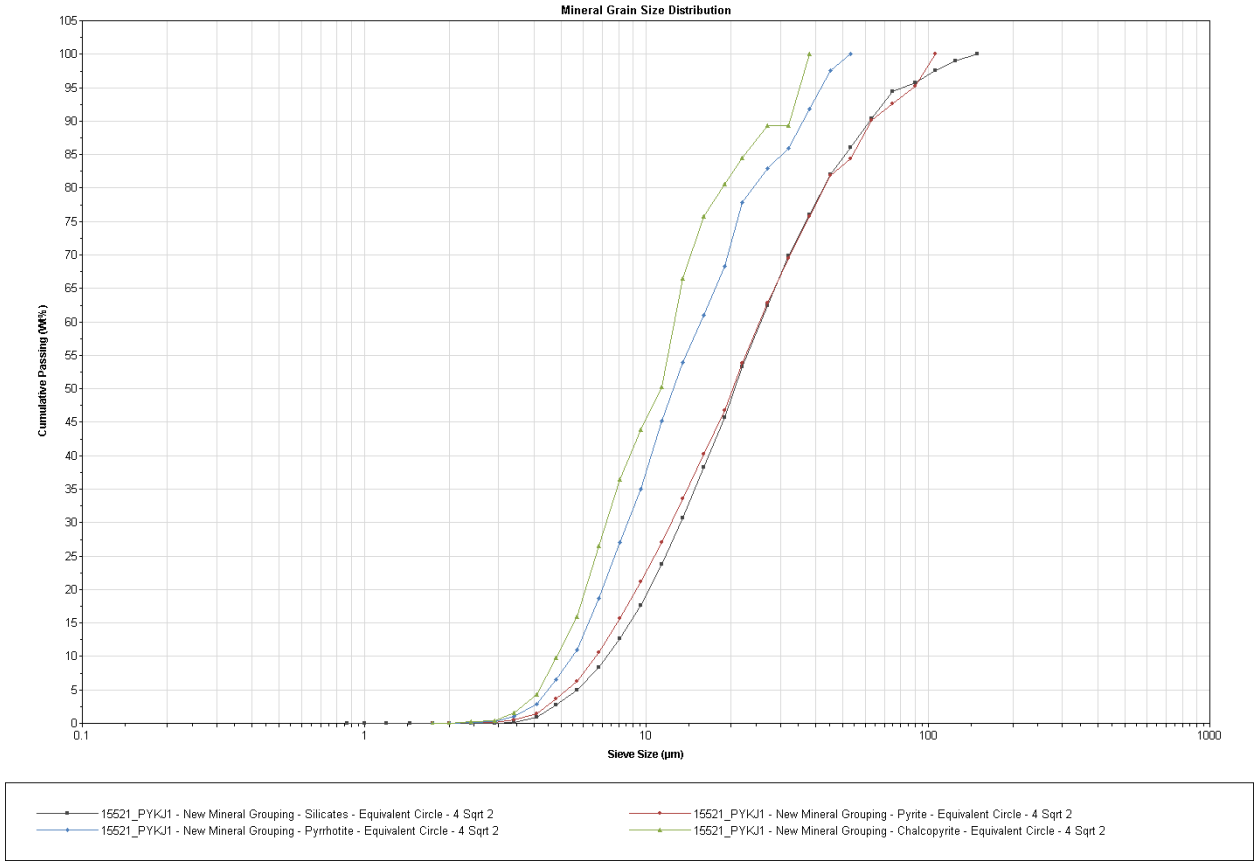

Figure 17. Particle Size of Silicates, Pyrite, Pyrrhotite and Chalcopyrite in PyCT (PyKJ1)

June 6, 2018

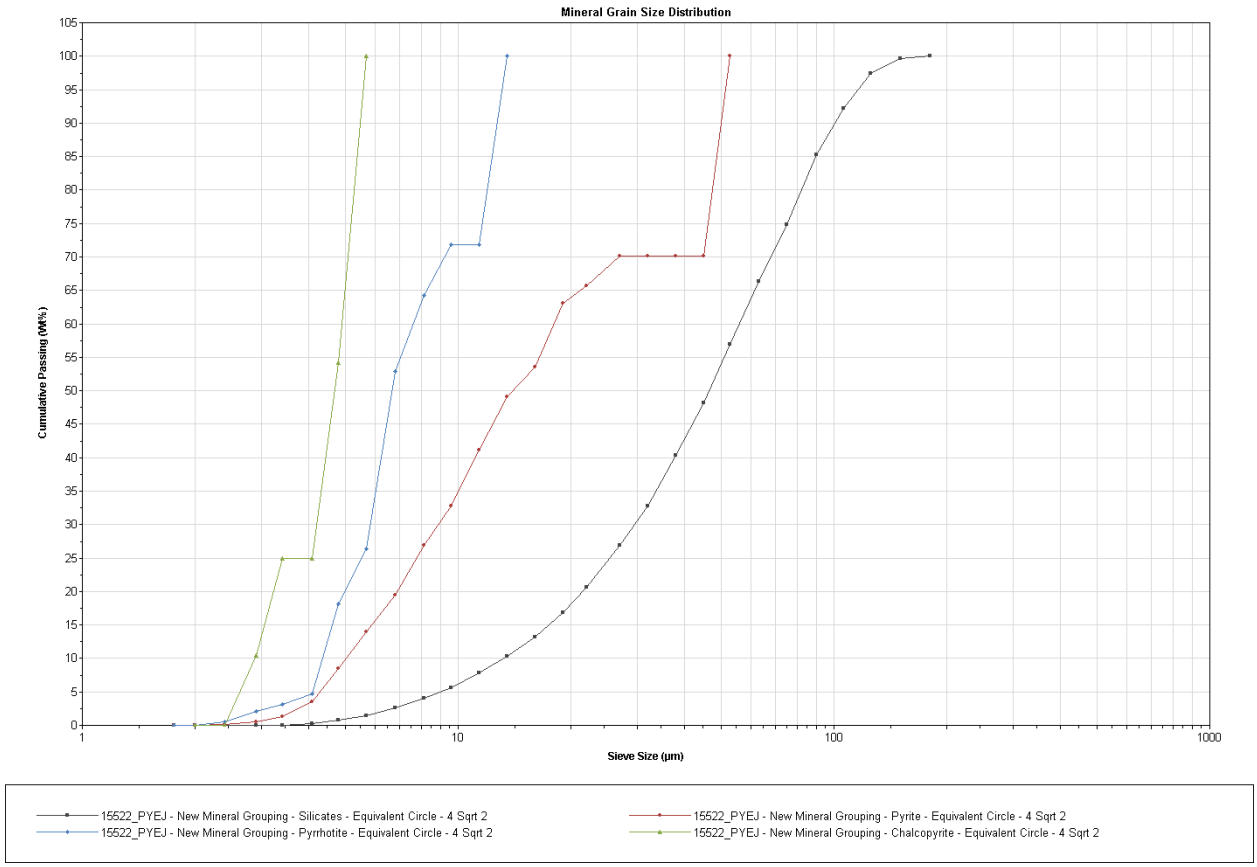

Figure 18. Particle Size of Silicates, Pyrite, Pyrrhotite and Chalcopyrite in PyRT (PyEJ)

June 6, 2018

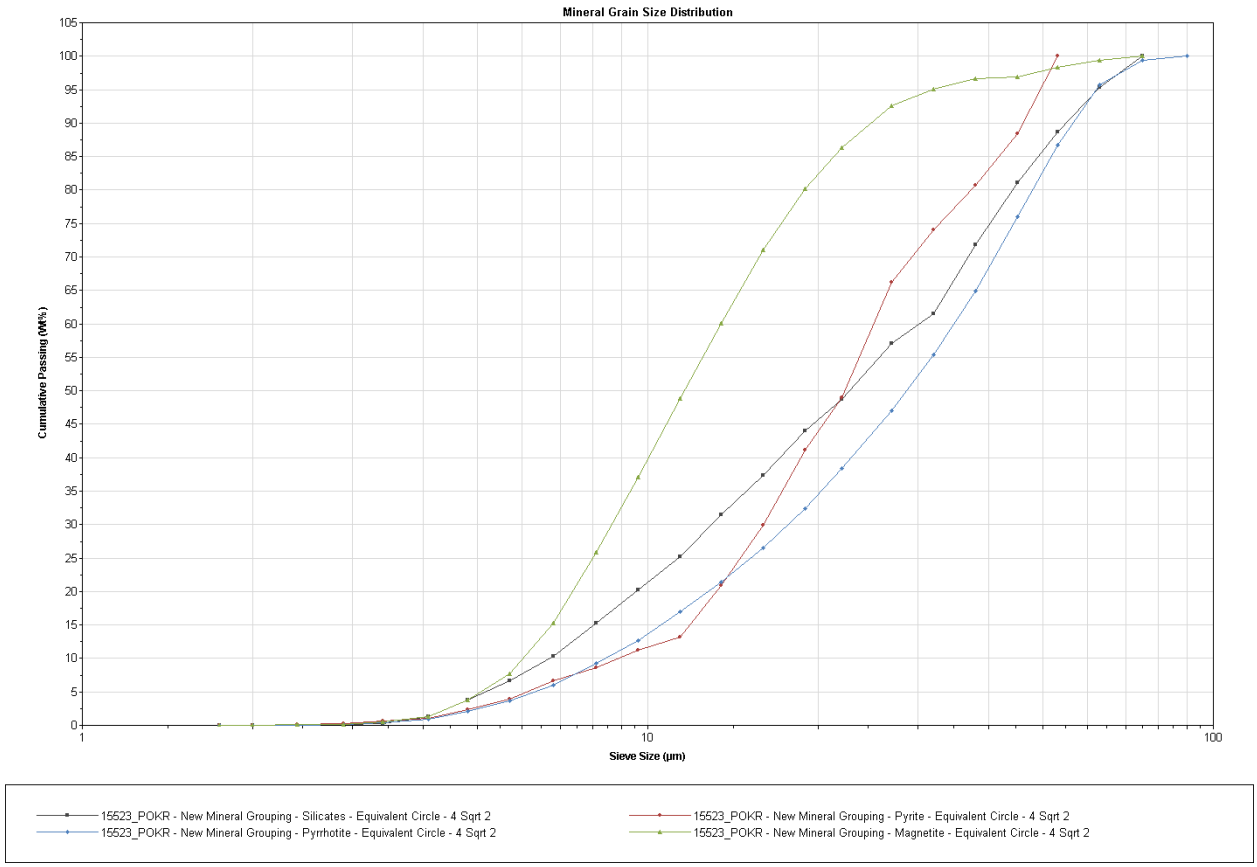

Figure 19. Particle Size of Silicates, Pyrite, Pyrrhotite and Magnetite in PoCC (PoKR)

June 6, 2018

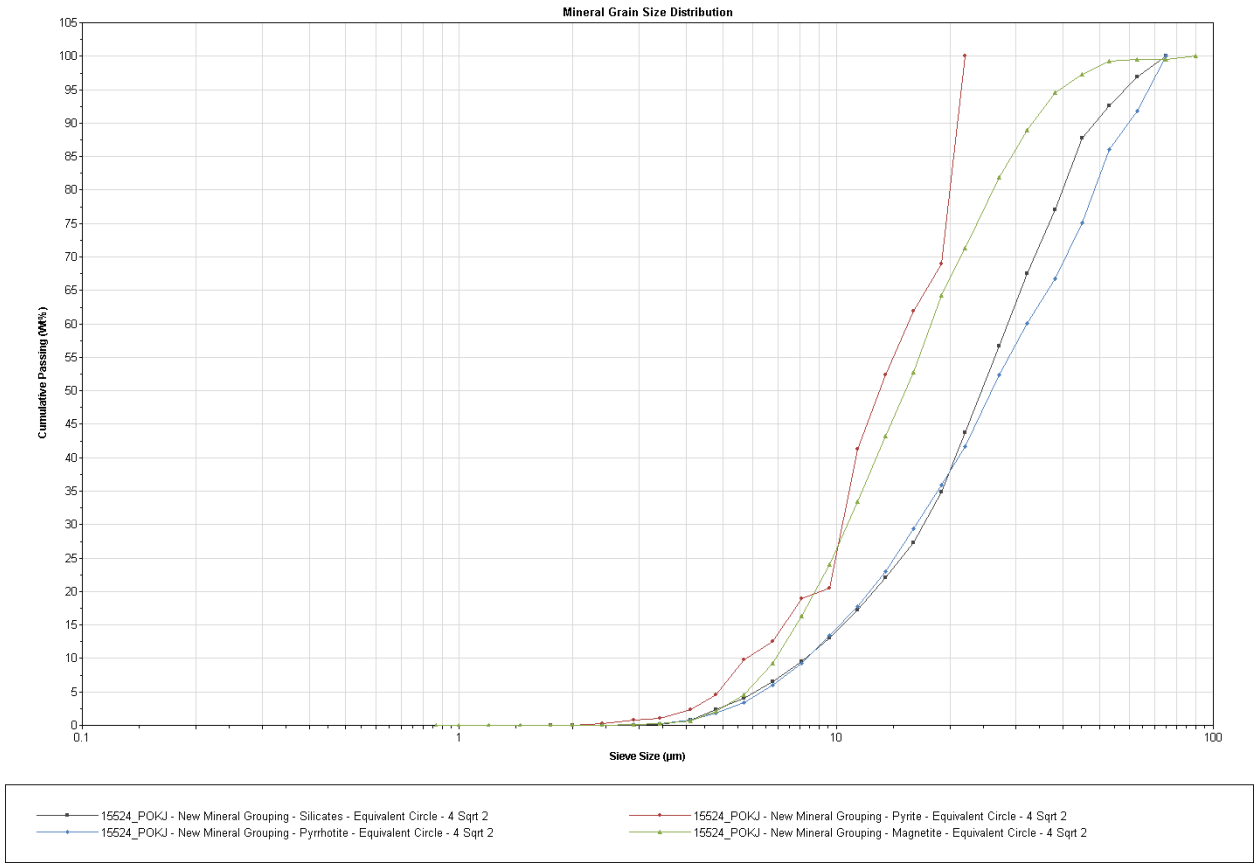

Figure 20. Particle Size of Silicates, Pyrite, Pyrrhotite and Magnetite in PoCT (PoKJ).

June 6, 2018

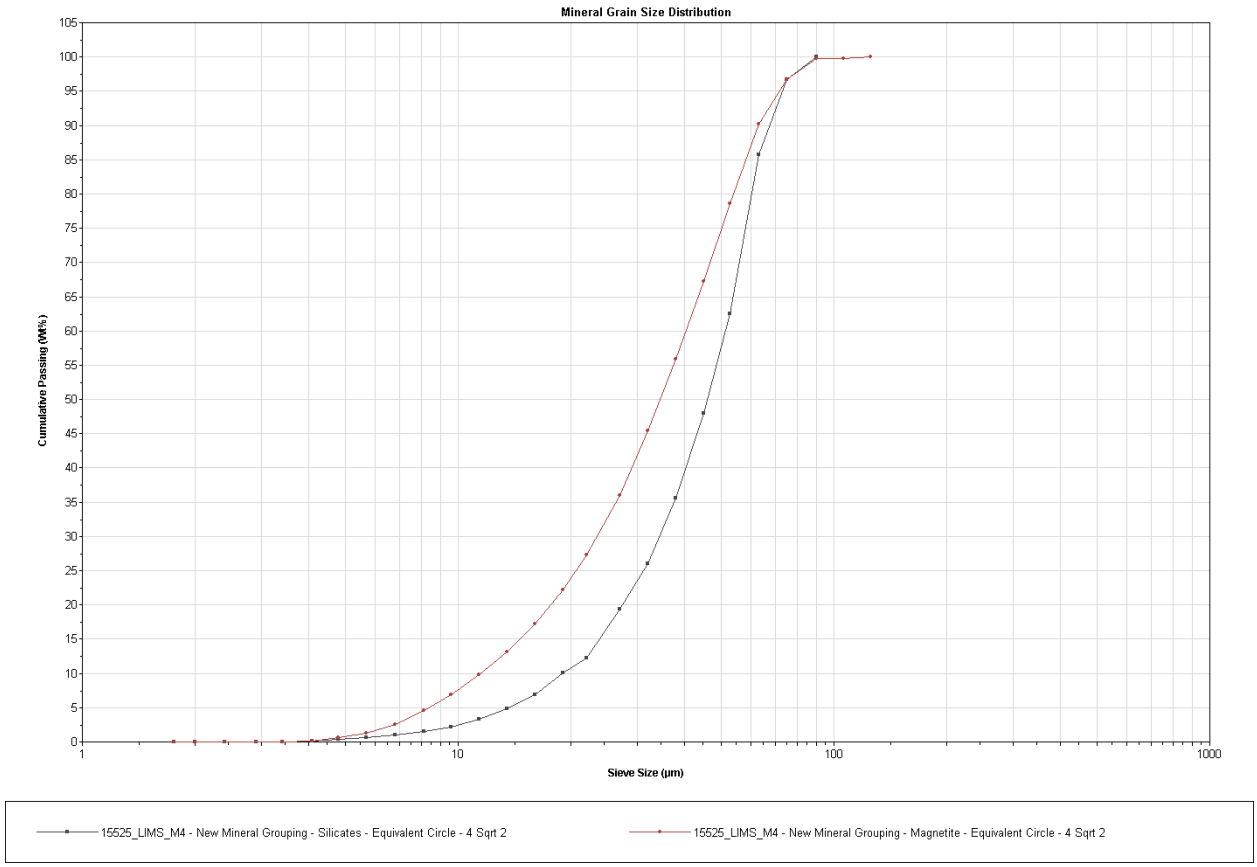

Figure 21. Particle Size of Silicates and Magnetite in WLC4 (LIMS M4)

June 6, 2018

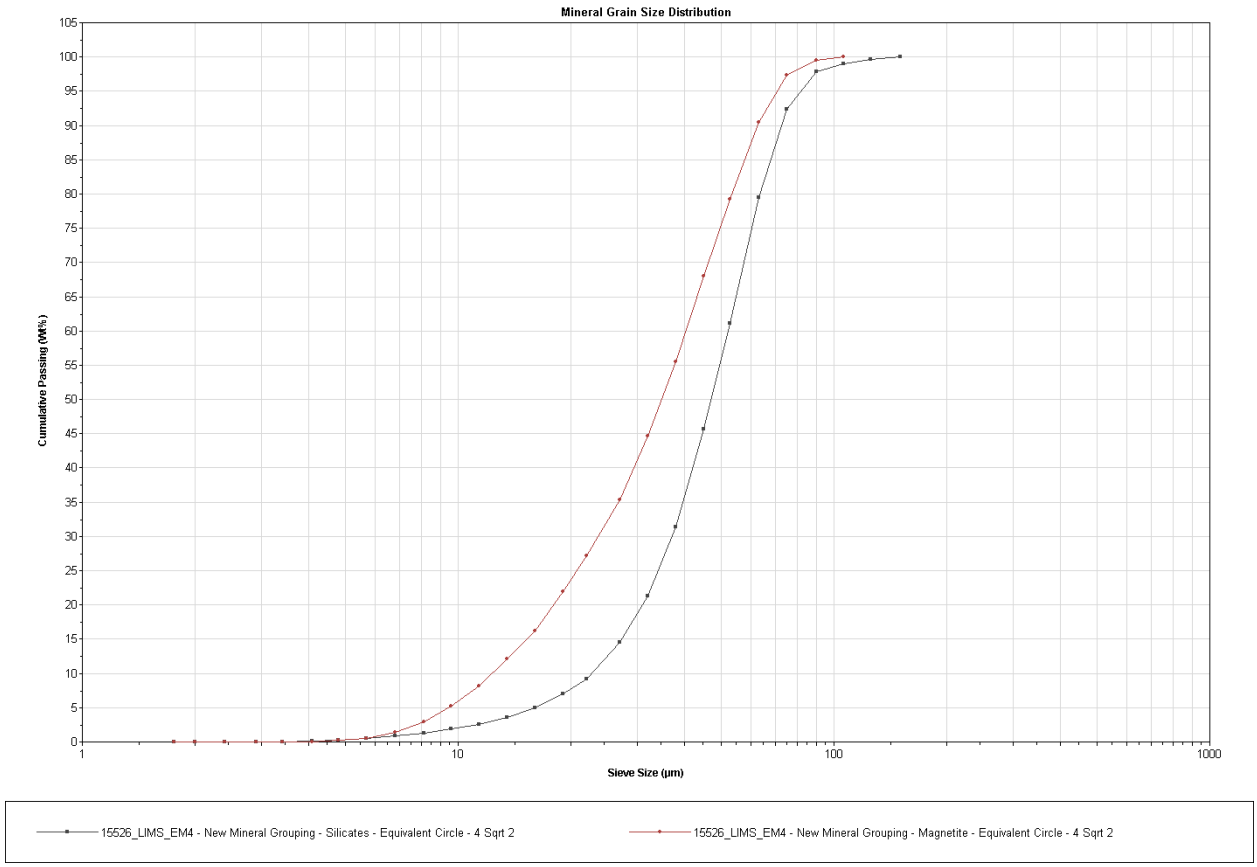

Figure 22. Particle Size of Silicates and Magnetite in WLT4 (LIMS EM4)

7 SUMMARY

In total, approximately 300 tons of VE2 and VE3 samples were delivered to GTK Mintec in August 2017 for the pilot test work. The calculated head grades of the feeds were as follows:

VE2 – Fe 33.0 %, S 4.2 %, Cu 0.128 %, Satmagan 35.3 and Au 0.05 g/t.
VE3 – Fe 16.5 %, S 1.4 %, Cu 0.069 %, Satmagan 8.9 and Au 0.04 g/t.

Based on the mineralogical measurements with MLA on the head samples, the main minerals of VE2 were Magnetite 44.6 %, Clinopyroxene 25.3 %, Ferrotschermakite 5.8 % and Pyrite 4.8 %. Respectively, the main minerals of VE 3 feed were Clinopyroxene 36.5 %, Amphiboles 16.9 %, Epidote 12.8 %, Plagioclase 11.6 % and Magnetite 8.4 %.

The pilot plant study took place between September 11 and 29 and was operated with the VE2 sample for 13 days, September 11 - 22 and 26 - 29, and the VE3 sample only for two (2) days, September 22 and 25.

Pilot plant study was commissioned on September 11, 2017, with the VE2 feed sample. The total grinding net energy of Hannukainen circuit with VE2 feed was 23.3 kWh/t including the primary grinding stage, regrinding of rougher copper concentrate and regrinding of WLC2.

During the first operating days, the Fe grade of WLC4 was around 66% to 67 %. A sample of the WLC4 was taken for MLA measurement and results showed that part of the magnetite was locked in silicate particles. The d80 value of the Derrick u/s was ca. 110 µm during the first piloting days. Due to this notification, regrinding with Sala Agitated mill for WLC2 was applied starting from September 19, 2017. After the regrinding was commissioned, the Fe grade of WLC4 increased to 69.6 %. Respectively, the SiO2 grade of WLC4 decreased from 4.6 % to 2.4 %. The d80 value of the regrinding was 63 µm and the net energy of the mill was 3.3 kW. On September 28 the Fe grade of the iron concentrate was 69.9 % with 72.4 % recovery. Respectively, the Satmgan value of the concentrate was 96.8 and with 90.0 % recovery. The same sample was measured with MLA and the content of magnetite was 94.7 %.

The main results of WLC4 of VE2 feed are presented in Figure 23.

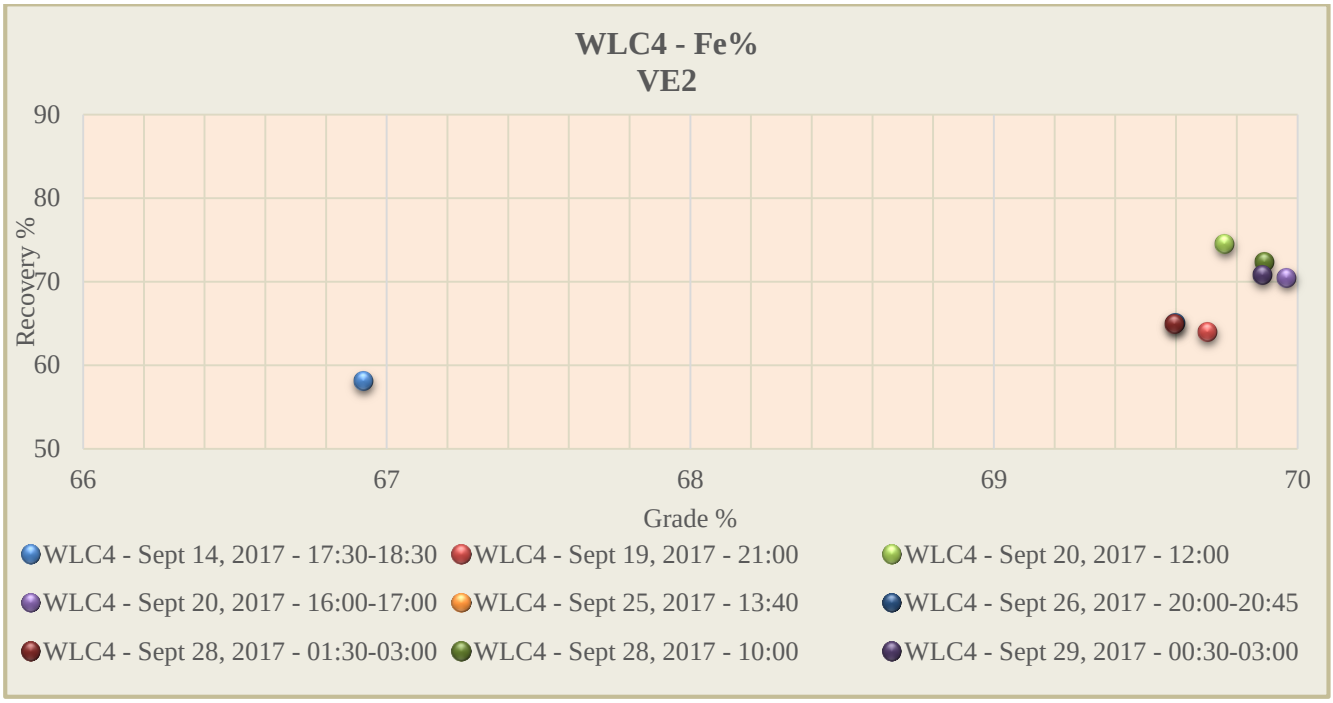

Figure 23. Iron Grade vs. Recovery of VE2 WLC4

The main results of Cu-concentrate of VE2 feed are presented in Figure 24. The best results were achieved during September 28 when the Cu grade of the CuCC3 was 19.6 % with 78.6 % recovery. MLA measurement was conducted on the same sample the results showed that the pyrite content in the concentrate was high as 46.7 %.

Flash flotation was tested in the grinding circuit for Derrick o/s on September 15 and 20. The Cu grade of the Flash concentrate was 2.4 % with 25 % recovery. However, the end-product, CuCC4, had only 9.7 % Cu grade with 73.5 % recovery. The mass recovery of the Flash concentrate was 1.3 %.

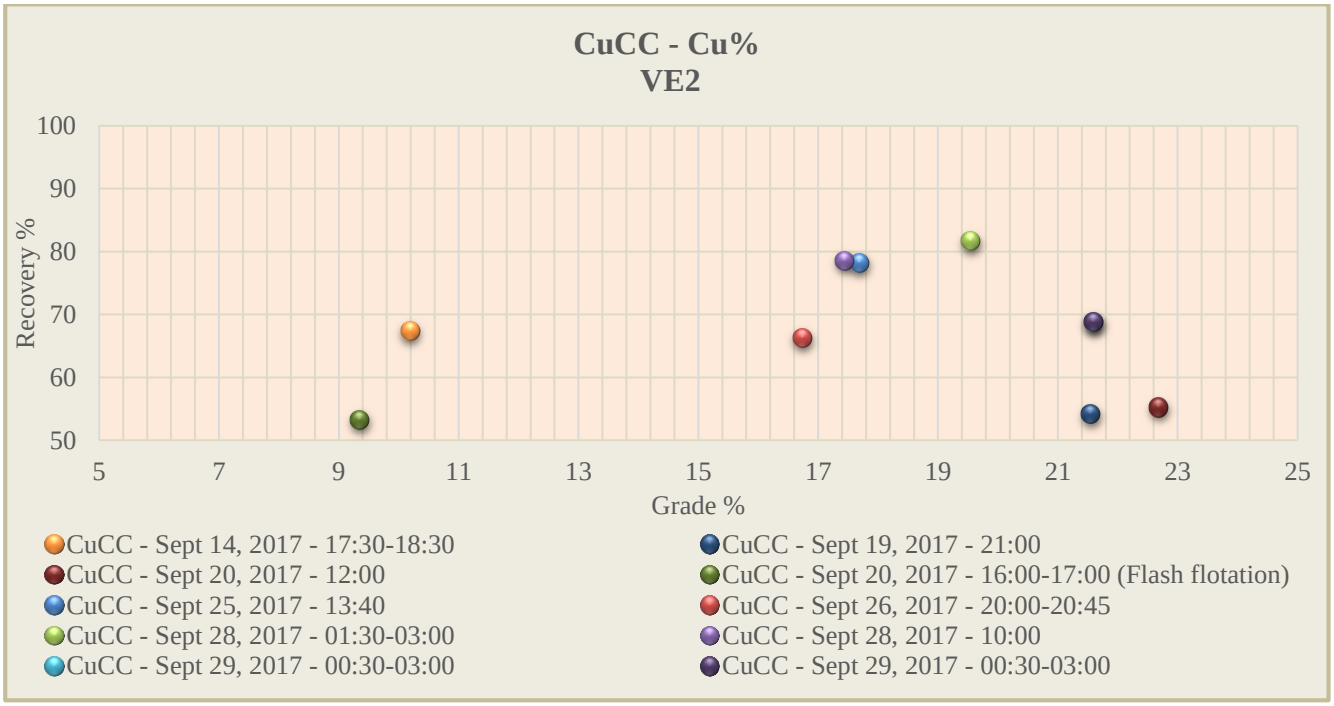

Figure 24. Copper Grade vs. Recovery of VE2 CuCC

The results of the tailings including the main low sulphur tailing streams, WLT1-3 or PyRT+WLT4, of the VE2 sample are presented in Figure 25. As can be seen from the Figure, the sulphur grade decreased along the test work. The target grade for sulphur was 0.3 %. At the beginning of the pilot trial, the flowsheet comrised grinding – Cu-flotation – Pyrite flotation – WLIMS – Pyrrhotite flotation and WLIMS stages. With this flowsheet, the suplhur grade of the WLT1-3 maintained at level of 1.7 to 2.4 %.

Due to the high sulphur content in tailings, the process was modified on September 27, 2017. The tailings of copper rougher flotation were directed to the WLIMS. The non-mag was directed to pyrite flotation through a dewatering stage with hydrocyclones to increase the solids concentration of flotation feed. With these modifications, the sulphur grade in the pyrite rougher tailings was managed to decrease close to the target grade, 0.3 %. A sample of pyrite rougher tailings was analysed for chemical composition per size fractions and the results show that ca. 42 % of the sulphur was in the -32µm size fraction. The main carrier of sulphur in the tailings was pyrite. Some of the sulphur was also removed in the dewatering stage to the cyclone overflow. The sulphur grade of the cyclone overflow on September 28 was 5.5 % with a recovery 5.1 %.

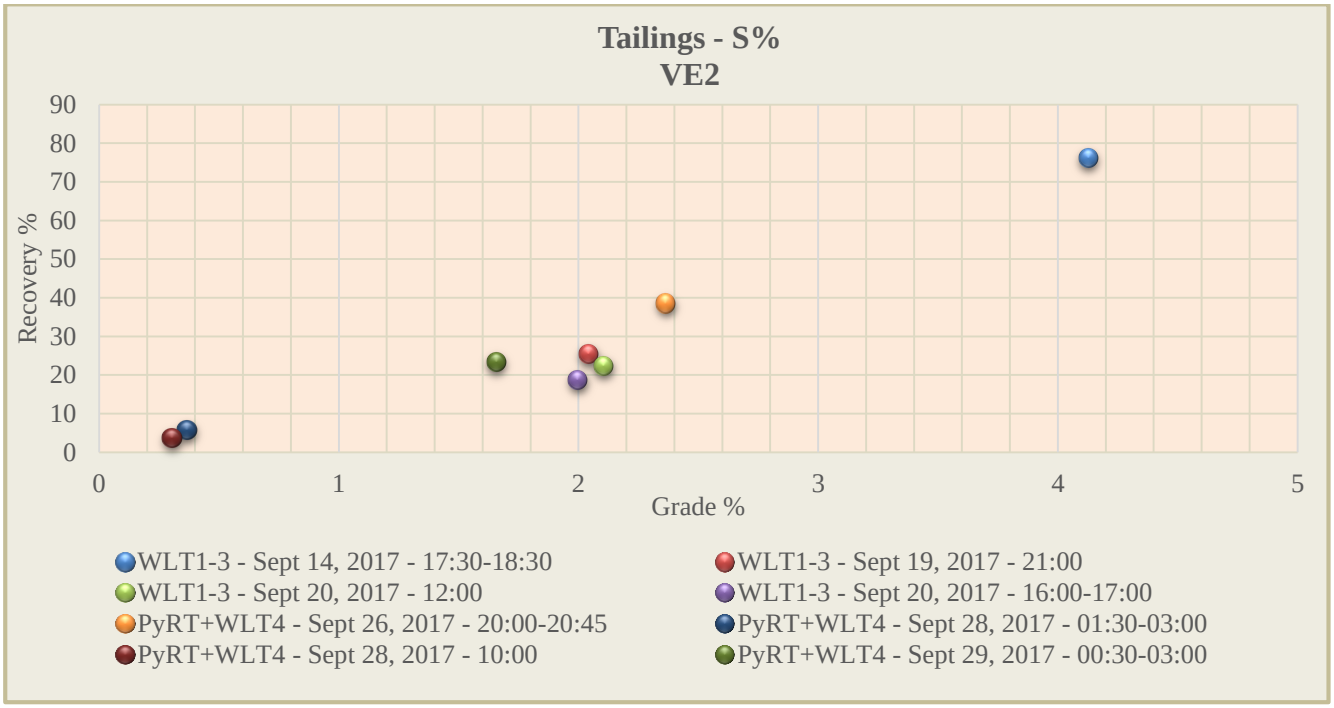

Figure 25. Sulphur Grade vs. Recovery of VE2 Tailings

The main results of the pyrite and pyrrhotite flotation products of VE2 feed are presented in Figure 26. The highest S grade in pyrite cleaning concentrate was resulted on September 19, 49.4 % grade with 36.7 % recovery. Only one cleaning stage was in use. A bench scale cleaning test was performed on a sample of pyrite cleaning concentrate, taken on September 20 from the pilot process. After two additional cleaning stages, the sulphur grade of the PyCC3 was ca. 53% (Table 4).

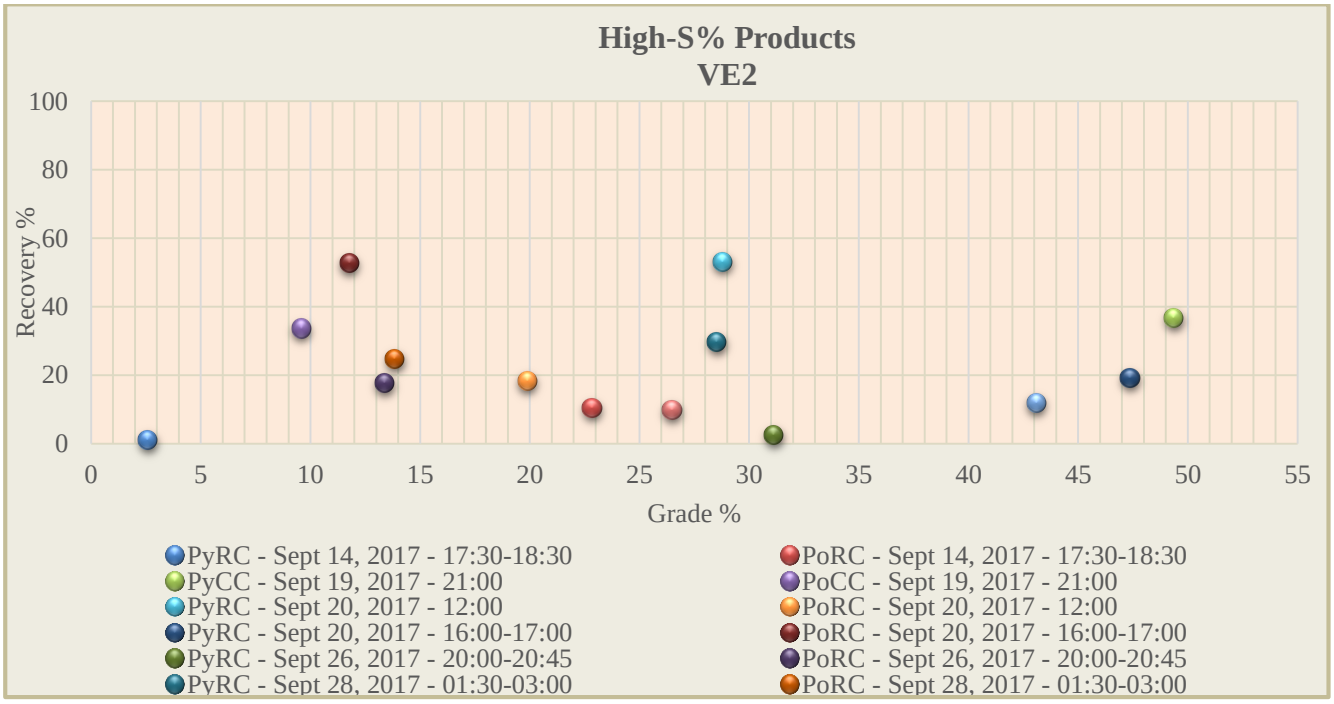

Figure 26. Main Results of Pyrite and Pyrrhotite Flotation with VE2 Feed

The pilot plant was operated with the VE3 sample only for two days. The feed contained much less of iron, copper and sulphur in comparison with the VE2 sample. During September 25, the Copper grade of CuCC3 was 17.7 % with 78.2 % recovery (Table 30). Respectively, the iron grade of the final LIMS concentrate was 69.8 % at 31 % recovery. However, according to the Satmagan measurements, the magnetite content of the iron concentrate was 95.5 with 89.3 % recovery.

8 COMPARISON TO EARLIER TEST WORK 2011

Table 62. Feed

Contents (%)	VE2 S17063029	2011
SiO2	29.2	29.6
Al2O3	6.31	5.29
MgO	4.76	5.66
CaO	9.03	8.37
Na2O	1.93	1.57
Cu	0.128	0.19
Fe	33.0	33.9
Eltra S	4.16	2.70
Satmagan	35.31	37.7

The sulphur content was notably higher in 2017 compared to the earlier study, and the magnetite content was a bit lower. Still the Fe grades were more or less equal. At 2011, no mineralogical examination was done for the feed, but it can be assumed that the pyrite content of the new sample was higher. The other difference is in copper content which was now lower than in 2011.

The copper flotation was now more difficult in terms of achieving the +25% Cu grades. It was also noted that the natural pH in copper flotation was now lower than 2011 (7 vs 5.4).

On the other hand, the pyrrhotite flotation was now more selective, and the iron losses were now smaller, indicating that the pyrrhotite content was now lower. The iron loss in pyrrhotite flotation was 2011 in the level of 30% while it was now in the range of 7 – 12%.

The total Fe recovery in final magnetite concentrate was now 70 – 74%, and in 2011 the corresponding figure was 54 – 64%.

During the pilot plant operation it was noted that the copper flotation could be operated in a bit coarser particle size, but the final purification of the magnetite concentrate required finer grind, P(80) of 65µm.

9 UNCERTAINTIES

The pilot plant tests were done from metallurgical point of view. Therefore chemical dosages were not optimized and they are only indicatory and serve as directional for the environmental modelling. Also no water circulation was done during the pilot scale tests. Typically the real

enrichment process will use less process chemicals because of the water circulation. The water circulation will also reduce the use of sulfuric acid and lime which are used in the water treatment. Cost effectiveness of the enrichment plant is also a big driver to minimize the chemical usage in the production phase. Because of these reasons the pilot scale test results should only be viewed directly from metallurgical point of view. As for the environmental side of the tests, further modelling should be carried out.