An Investigation into the Processing of

PEA RIDGE IRON TAILINGS

prepared for

JAMES KENNEDY

Project 16446-001 – Final Report
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NOTES

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Table of Contents

List of Tables

List of Figures

Introduction

At the request of Mr. James Kennedy, an executive summary was prepared summarizing testwork programs conducted by SGS on the Pea Ridge iron ore tailing deposit between 2013 and 2015. Specifically, the following 6 final reports formed the basis of this document:

- 13856-001. Scoping Level study on a Pea Ridge Iron Tail Sample. Sept 16, 2014;
- 13856-002. An Investigation by High Definition Mineralogy into the Mineralogical Characteristics of Two Metallurgical Samples from the Pea Ridge Projects, USA. June 5, 2013;
- 13856-003. Hydrometallurgical Experimental Tests on a Missouri Deposit. March 21, 2014;
- 13856-004. An Investigation into Beneficiation Testwork on a Pea Ridge Iron Tails Sample. Mar 23, 2015;
- 13856-005 PR1. An Investigation into Hydrometallurgical Flowsheet Development on Pea Ridge Flotation Samples. May 29, 2015;
- 13856-006. An Investigation into the Production of REE / Phosphate Concentrate on an Iron Tailings Sample from Pea Ridge. April 2, 2015.

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Testwork Summary

1. Pea Ridge Iron Tailing Resource Estimate

ThREE provided SGS with a resource estimate report by Pincock Allen and Holt (PAH) (Aug 27, 2012). This is summarized in Table 1, which shows a resource estimate of ~ 24.5 M metric tonnes grading 10% apatite and ~ 0.49% TREO.

Table 1: PAH Resource Estimate

Cut-off	FeT %	Short Tons	Fe(M lb)	Fe (Mag) %	S %	LREO g/t	HREO g/t	Y2O3 g/t	Apatite %	TREO g/t
0	19.8	27,216,513	10,772.30	3.43	0.11	3,953	328	648.1	10	
10	19.8	27,093,313	10,745.20	3.44	0.11	3,952	328	647.8	10	4,928
25	27.9	8,233,827	4,586.20	6.12	0.11	3,609	284	556.2	9.7	
Cut-off	FeT	Metric Tonne	Fe (tonnes)							
10	19.8	24,578,647	4,866,572							

2. Overall Flowsheet and Description

The goal of the metallurgical testwork program was the development of a flowsheet to recover REE, phosphate, magnetite, hematite, and pyrite from Pea Ridge iron tailings. The composition of a representative iron tailings sample received at the SGS Lakefield site is presented in Table 2 (elemental assays) and in Table 3 (XRD). TREE levels were 4,349 g/t, the iron oxide assay was 28.5% Fe_2O_3, the phosphate assay was 5.07% P_2O_5, and the sulphur assay was 2.2% S. The sample had a P_{80} of ~103 µm with ~33.4% of the mass distributed in the -20 µm fraction. The P_{80} of the ground product varied from 42 to 48 µm. Saved fractional analyses for the ground sample are provided in Table 4. Cerium, yttrium, aluminum oxide, magnesium oxide and potassium oxide were enriched in the -20 µm fraction, whereas phosphate had a slightly higher grade in the coarser fraction.

Table 2: Head Sample Analysis

REE ICP Scan, g/t																				
La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
922	1720	149	593	94.4	7.9	90.5	13.8	84.7	18.1	50.8	7.3	48.7	8.2	541	<25	12	72	3486	863	4349

Whole Rock Analysis, %														S Analysis %
SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO	Cr_2O_3	V_2O_5	LOI	Sum	S
40.9	5.58	28.5	3.3	7.88	0.38	3.23	0.51	5.07	0.07	0.02	0.03	3.2	98.6	2.23

The results of semi-quantitative mineralogy on the sample are presented in Table 3. The iron in the sample is present primarily as hematite, but also as pyrite and magnetite. The main gangue minerals are quartz and potassium-feldspar, followed by chlorite, mica, amphibole and plagioclase. Monazite is the main rare earth carrying mineral, followed by bastnaesite and a minor amount of xenotime. The REE minerals are generally fine-grained and are mainly hosted in coarser apatite grains. Therefore, processing this ore requires apatite recovery in order to recover the REE.

Table 3: Semi-Quantitative XRD Results

Mineral	Iron Tailings Comp (wt %)
Quartz	24.1
Hematite	20.9
Potassium-feldspar	14.3
Apatite	11.4
Chlorite	5.8
Mica	5.4
Pyrite	5.4
Magnetite	3.6
Amphibole	3.2
Plagioclase	2.8
Kaolinite	1.8
Dolomite	1.3
Total	100

A standard jar mill test was performed by Metso on a ~16 kg subsample of iron tailings. The specific energy for grinding the feed to a P_{80} of 40 µm with a 19.1 mm media charge was 4.47 kWh/t.

3. Mineral Processing Flowsheet – Production of Magnetite, Hematite, Pyrite and a REO Containing Phosphate Concentrate

3.1. Grinding Circuit

The feed material was stage-ground to 270 mesh (53 µm) as shown in **Figure 1**. From preliminary mineralogy analysis, it was concluded that there would be no further significant improvement in the liberation of the minerals by grinding the sample to a finer size.

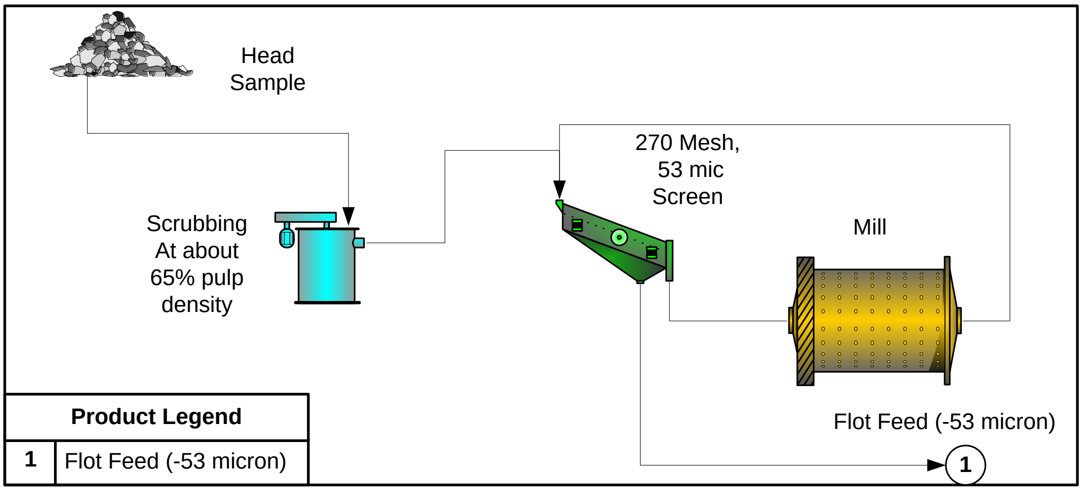


Figure 1: Grinding

Table 4: Saved Fraction Analysis on the Head Sample After Stage Grinding to -53 µm

Products	Weight	Assays: %														Distribution %													
	%	Fe	S	Ce	Y	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO	Fe	S	Ce	Y	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO
Ground Sample -53+45 µm	13.2	22.7	2.54	0.11	0.03	40.5	4.39	32.4	1.75	8.02	0.31	2.81	0.25	5.37	0.05	14.2	17.8	8.2	7.7	13.5	10.6	14.2	7.1	13.8	18.1	11.8	6.1	14.0	8.9
Ground Sample -45+38 µm	11.6	22.9	2.33	0.11	0.04	40.7	4.39	32.7	1.90	7.63	0.19	2.80	0.27	5.09	0.05	12.6	14.4	7.2	9.0	11.9	9.3	12.6	6.8	11.6	9.8	10.4	5.8	11.7	7.9
Ground Sample -38+20 µm	20.1	21.3	2.23	0.14	0.05	41.1	4.65	30.4	2.33	7.83	0.21	2.95	0.32	5.25	0.06	20.3	23.8	16.0	19.5	20.8	17.1	20.3	14.5	20.6	18.7	18.9	11.9	20.8	16.3
Ground Sample -20 µm	55.0	20.3	1.51	0.22	0.06	38.8	6.27	29.0	4.22	7.50	0.22	3.37	0.75	4.94	0.09	52.9	44.1	68.6	63.9	53.8	63.0	52.9	71.6	54.0	53.5	59.0	76.2	53.6	66.9
Head (Calc.)	100	21.1	1.89	0.177	0.052	39.7	5.48	30.2	3.24	7.65	0.23	3.15	0.54	5.08	0.07	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Head (Direct)		19.9	2.23	0.17	0.05	40.9	5.58	28.5	3.30	7.88	0.38	3.23	0.51	5.07	0.07														

3.2. Pyrite Flotation

The pyrite circuit focused on yielding a high-grade pyrite concentrate, which was intended to recover valuable elements such as gold, silver, cobalt, molybdenum and copper. Key factors in the design of the pyrite flotation circuit were the minimization of REE/phosphate losses to the pyrite concentrate and the achievement of a high grade pyrite concentrate. It was determined in the testwork (based on a locked cycle test) that at least three, but most likely five cleaning stages would be required to achieve these targets.

Based on the locked cycle test, it was possible to recover 61% of the sulphur into a pyrite concentrate with a 47.4% S grade, which corresponds to a pyrite grade of ~90%, in ~2.2% of the weight. Cerium and yttrium losses to the final pyrite concentrate averaged 0.2%.

A number of samples were submitted to different analytical labs for extra analyses, with specific attention on gold analyses. The results are presented in Table 5. Assuming that sulphur is mainly associated with pyrite, then these results suggest that the head sample contained 3.7% pyrite. If it is further assumed that all the gold is associated with pyrite, then these results suggest that if all the pyrite is collected into a concentrate, the gold might be upgraded by 26 times. In such an ideal pure pyrite concentrate, the expected gold assay would be 0.187 g/t. The pyrite concentrate was also submitted for gold assay and was found to contain 0.15 g/t Au. Thus, it can be concluded that gold is mainly associated in pyrite. Such a low grade gold concentrate, is not financially viable for extraction.

Pyrite may be used in a roaster to convert its sulphur contents into sulphuric acid. If all the pyrite is recovered and processed into sulphuric acid, the potential exists to off-set the net sulphuric acid consumption in the latter stages of the process by around 22% (equivalent to ~ 130,000 t/a H_2SO_4 at an annual throughput of ~ 4.9 Mt/a tails).

Table 5: Extra Analysis of Head Sample

Element	Iron Tail Head Sample		
	ALS Lab	SGS Lab	American Lab
S %	2.05	1.94	2.06
S ppm	20,700	17,600	20,321
Sn ppm	14.3	< 20	9.4
Ag ppm	<0.5	< 0.5	<0.1
Cu ppm	310	322	277
Co ppm	143	149	121
Ni ppm	40	61	43
Mo ppm	35	32	28
Au ppb	7	6	7

3.3. REE/Phosphate Flotation

REE/Phosphate flotation tests processed the combined pyrite tailings to recover the majority of the REE/phosphates with minimal iron oxide losses. Based on the locked cycle test and using two-stages of cleaning in the REE/phosphate circuit, it was possible to produce a REE/phosphate cleaner concentrate containing 9,291 g/t Ce, 2,827 g/t Y and 30.6% P_2O_5 with recoveries of ~87% of the cerium, 84% of the yttrium and 95% of the phosphorous, in 16.5% of the weight.

In one of the batch flotation tests, a third cleaning stage was used to reject more gangue minerals, thereby improving the concentrate grade. In this test, the 3rd cleaner concentrate graded 10,160 g/t Ce, 3,228 g/t Y, and 33.8% P_2O_5 with recoveries of ~77% of the cerium, 79% of the yttrium and 89% of the phosphorous, in ~13.8% of the weight. From further analysis of all the batch and locked cycle test results, it was concluded that it might be possible to produce an even higher grade concentrate with a mass pull of about 11 to 13%, with a grade of 32-34% P_2O_5 and phosphate recovery of about 86-88%, and with yttrium and cerium recoveries of 74-78% and 76-78%, respectively. In such a concentrate, yttrium and cerium assays would be about 3,300 g/t Y and 10,500 g/t Ce. It is believed that this is most likely the highest phosphate concentrate grade that can be generated.

The remaining sulphides in the pyrite tailings split between the REE/phosphate concentrate and tailings, with 18.6% of the sulphur in the feed reporting to the tailings, based on the locked cycle test results. Small amounts of iron-bearing minerals also reported to the REE/phosphate concentrate, which contained 4.8% of the iron in the feed.

The locked cycle test (LCT) flowsheet for both pyrite and phosphate flotation is shown in Figure 2.

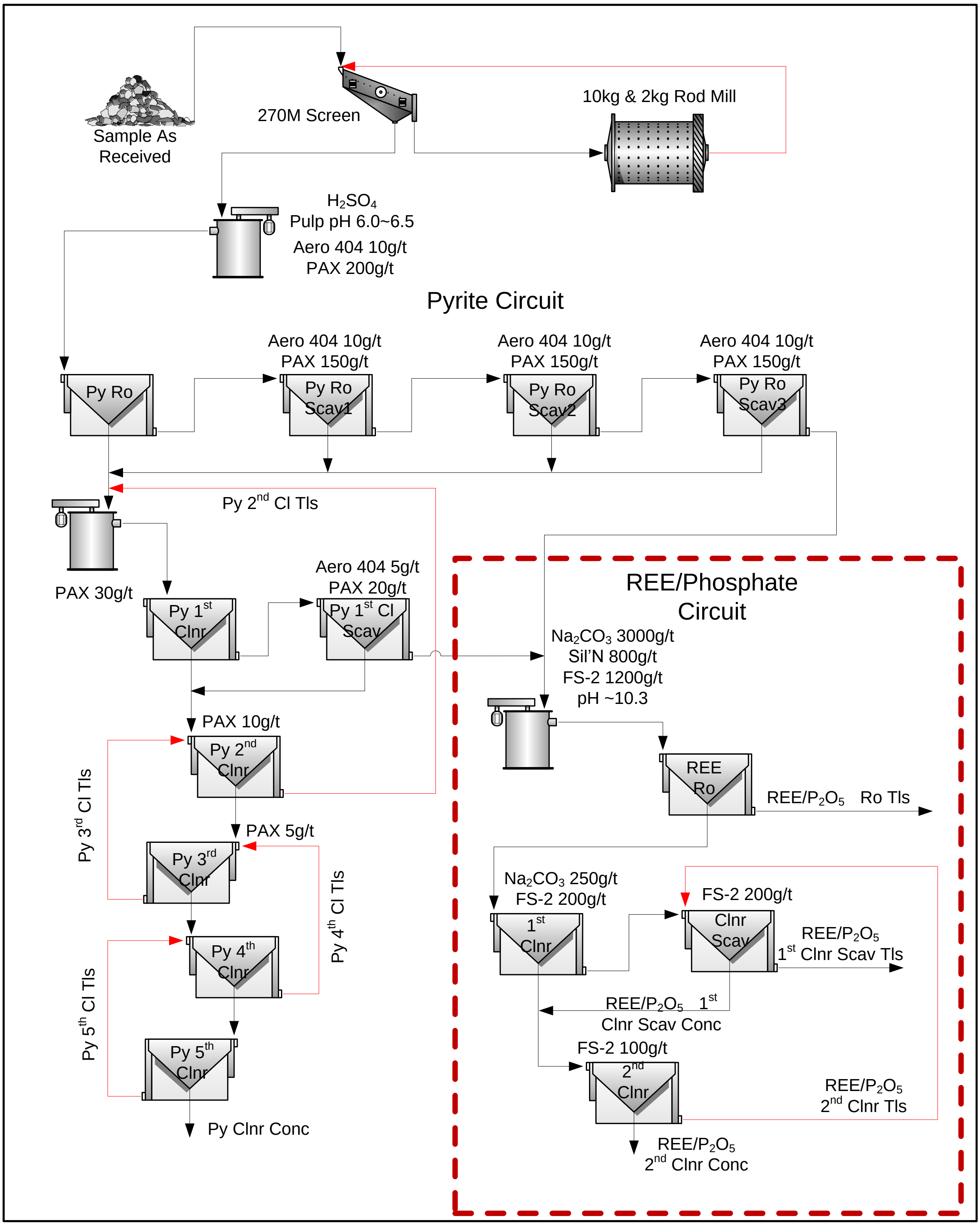


Figure 2: Locked Cycle Test Flowsheet

3.3.1. Mineralogical Analysis on the Flotation Concentrate (F7) Products

Mineralogical analysis using QEMSCAN^{TM} was performed on one of the initial REE/phosphate rougher concentrates in test F7 (13856-001). Phosphate rougher flotation concentrate and gravity tailings were submitted for QEMSCAN analysis, and the results are summarized as follows:

- The rougher phosphate concentrate consisted of apatite (62%), monazite (1.5%), bastnaesite/synchysite (0.9%), and xenotime (0.1%). The main mineral impurities included Fe-Oxides (7.8%), silicate minerals (20%), and minor carbonates (~2%).
- The gravity tail was composed mainly of silicate minerals (~80%) and Fe-Oxides (17%). Losses of apatite accounted for <3%, while REE minerals were of trace amounts (<0.4%). Xenotime was scarce in the tail and it was difficult to extract any meaningful data.
- Liberated monazite and bastnaesite/synchysite (and those in association with apatite and each other) were 79% and 35%, respectively, in the tail. Thus, with further reagent optimization, it is likely possible to recover more of the REE minerals to the phosphate concentrate.
- The electron microprobe analysis showed that xenotime contained 35.7% of the HREE and 38.3% of the (total) TREE, monazite contained 1.00% and 55.9%, respectively, and apatite contained 0.38% and 0.90%, respectively. A summary of the microprobe analysis is shown in Table 6.

Table 6: Summary of Results from Microprobe Analysis

No	Mineral	La	Ce	Pr	Nd	Sm	Gd	Dy	Y	Ttoal REE(+ Y)	Total LREE	Total HREE (+ Y)
22	Apatite	0.07	0.24		0.15		0.06		0.38	0.90	0.52	0.38
19	Monazite	15.7	27.5	2.34	7.96	0.75	0.59	0.17	0.83	55.9	54.9	1.00
4	Xenotime		0.07		0.20		2.37	3.86	31.9	38.3	2.64	35.7

- Based on the mass of the minerals and the average REE-Y values from the electron microprobe, apatite accounted for a significant proportion of yttrium (79% in the concentrate). It also carried significant amounts of LREE. Essentially, apatite must be recovered along with the monazite, xenotime, and bastnaesite/synchysite for optimum REE-Y recovery.
- Free and liberated monazite, bastnaesite/synchysite and xenotime increase with decreasing grain size, up to 53 µm. Additional grinding might not be required.
- Theoretical grade-recovery calculations indicate the potential to recover a significant portion of the REE minerals. However, it must be noted that the assays of REE minerals in the feed were fairly low.
- The SEM examination of the concentrate sample revealed that the REE minerals were generally fine-grained. The most striking feature is that apatite hosted a number of fine-grained monazite and xenotime particles, while bastnaesite/synchysite showed a weaker affiliation.

3.4. Magnetite Circuit

To recover magnetite, the combined REE/phosphate flotation tailings were passed through a Low Intensity Magnetic Separator (LIMS), with the LIMS rougher concentrate cleaned three times.

The LIMS magnetics product recovered approximately 14.8% of the iron with 62.7% Fe grade in 5.9% of the feed weight. The LIMS rougher stage recovered 93.4% of the magnetite (stage recovery) at a P_{80} of 41 µm and a magnetic intensity of about 1,000 Gauss. The losses of magnetite in the LIMS rougher tailings were likely due to liberation and/or insufficient magnetic intensity in the LIMS operation. The size fractional analysis results of the LIMS concentrate show that the +38 µm fraction graded 60.4% Fe (74% magnetite) and 11.5% SiO_2, while the -20 µm fraction graded 65.0% Fe (83.1% magnetite) and 5.64% SiO_2.

Due to the high silicate content of the LIMS concentrate, reverse silicate flotation was necessary. A number of flotation tests were conducted on the combined LIMS magnetic concentrate to reject silicate. It was possible to produce an iron product with over 65% Fe grade after floating/removing silicate minerals. The best metallurgical result for silica flotation was achieved with Lila Flot D817 collector. The magnetite concentrate float tails represented 93.5% stage recovery for iron, with a grade of 69.7% Fe and 1.67% SiO_2 in 85% of the weight of the silica flotation feed. The results proved that the LIMS magnetic concentrate could be easily upgraded to over 69% Fe by using reverse silica flotation. The magnetite recovery circuit is shown in Figure 3.

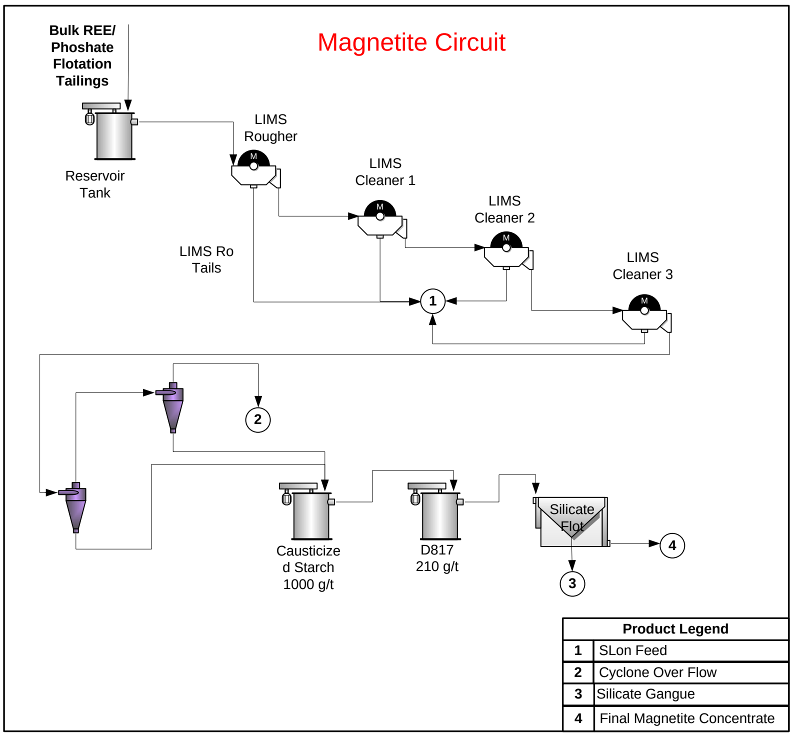


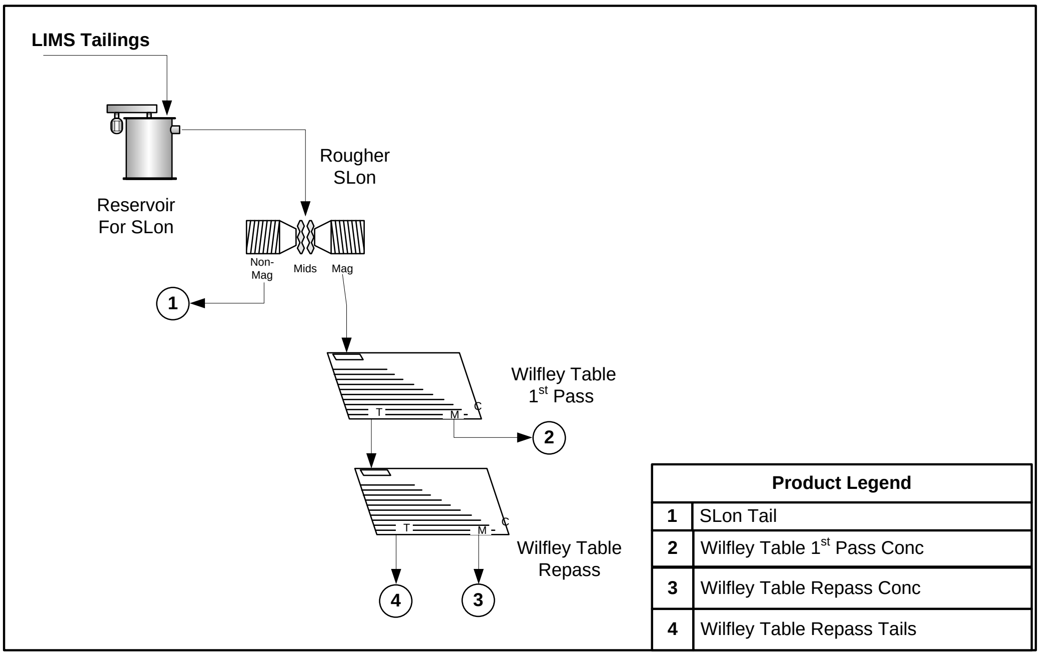
Figure 3: Magnetite Recovery Circuit

3.5. Hematite Circuit

A SLon Wet High Intensity Magnetic Separator (WHIMS) was used to process the combined LIMS tailings, with the goal of producing a hematite concentrate and recovering the remaining iron in the tailings. The iron grade was improved from 21.6% Fe in the SLon feed to 36.5% Fe in the SLon rougher concentrate with 89.5% stage iron recovery. This improved further to 40% Fe in the cleaner concentrate with 80.4% stage iron recovery. The SLon did not produce a high grade hematite concentrate due to the relatively high levels of impurities such as 26.8% SiO_2, 4.50% Al_2O_3, 2.89% MgO and 2.72% K_2O reporting to the SLon concentrate. Based on the results of the SLon first cleaning stage, applying additional cleaning stages most likely will not increase the iron grade significantly. As a result, a reverse silica flotation test was conducted on the SLon rougher concentrate with the collector Lila Flot D817 to reject the silica. The final flotation product (silica float tails) achieved 61.2% Fe grade with 63.3% stage recovery for iron in 43.1% of the weight. The silica grade in the product was however still high, at 7.71% SiO_2. Applying additional flotation stages may reject more silica and achieve a higher iron grade in the concentrate, but will likely result in considerable iron losses. Reverse silica flotation performance was inferior to gravity separation.

As the WHIMS test failed to produce a high enough grade iron concentrate, a gravity separation test was performed by separately passing the WHIMS cleaner concentrate and tailings over a Wilfley table (WT). The collected tailings from the Wilfley table were scavenged once. The combined gravity concentrate graded 67.8% Fe and 1.29% SiO_2, with 60.5% stage recovery for iron. This clearly indicates that a WT can produce a high grade hematite concentrate with low silicate content.

Since both the SLon 1st Cleaner concentrate and tailings were processed by the WT and the combined concentrate from the WT had acceptable concentrate grade and recovery, the SLon cleaner stage might not be required and the SLon rougher concentrate can potentially be directly processed by gravity separation, as shown in Figure 4.



Product Legend	
1	SLon Tail
2	Wilfley Table 1st Pass Conc
3	Wilfley Table Repass Conc
4	Wilfley Table Repass Tails

Figure 4: Combined SLon and Gravity Operation to Produce Hematite Concentrate

3.6. Conceptual Beneficiation Flowsheet, Product Recoveries and Reagent Consumptions

The conceptual flowsheet developed in the testwork program is presented in Figure 5 and consists ofpyrite flotation, REE/phosphate flotation, a magnetite circuit and a hematite circuit. All the expected mass pulls and grades are shown in the flowsheet. Flotation reagents and corresponding dosages in different flotation stages are presented in Table 7.

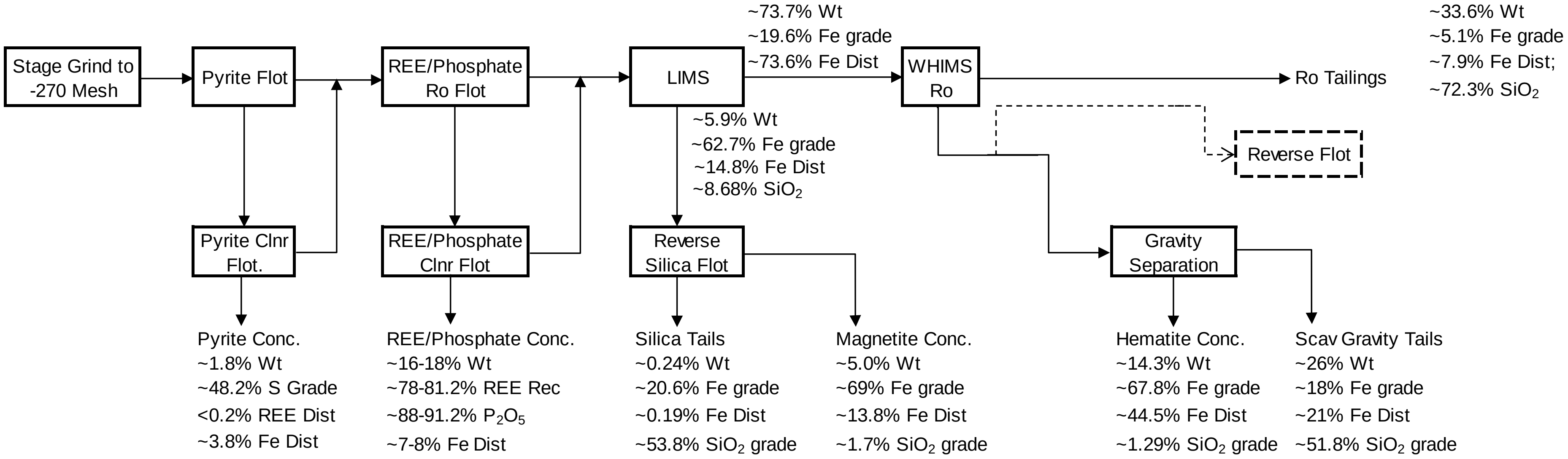
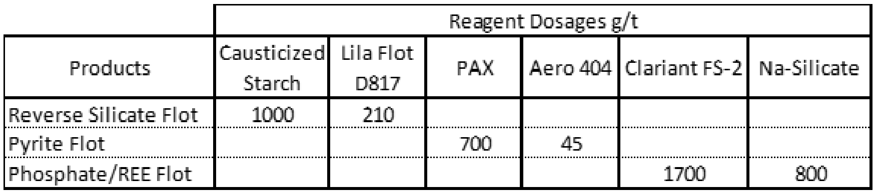


Figure 5: Conceptual Global Flowsheet

Table 7: Flotation Reagents Used in Different Flotation Stages



	Reagent Dosages g/t					
Products	Causticized Starch	Lila Flot D817	PAX	Aero 404	Clariant FS-2	Na-Silicate
Reverse Silicate Flot	1000	210				
Pyrite Flot			700	45		
Phosphate/REE Flot					1700	800

4. Hydrometallurgical Flowsheet – Processing of REO Containing Phosphate Concentrate into Separate REO and Calcium Phosphate Concentrates

A block process flow diagram of the hydrometallurgical flowsheet for treating the REO/phosphate flotation concentrate is shown in Figure 6.

4.1. Circuit Descriptions

The flowsheet consists of various unit operations. They are briefly explained below.

4.1.1. Pre-leaching (PL)

Flotation (REO / Phosphate) concentrate is leached with concentrated hydrochloric acid (HCl) in a Pre-Leach (PL) circuit. In this operation, approximately 70% of the concentrate mass is dissolved, including dissolution of 99% P, 30% light rare earth elements (LREE) and ~70% heavy rare earth elements (HREE).

4.1.2. Caustic Crack (CC)

PL residue is treated with concentrated (50-70%) sodium hydroxide solutions in a caustic crack circuit (CCr) which metathesizes insoluble rare earth fluorides and phosphates into acid soluble hydroxides. This circuit can be operated “dry” on a steel belt furnace or in low pressure autoclaves. At high NaOH concentrations (60-70%), this circuit can also be operated in atmospheric agitated reactors. The latter process was used in much of the testwork.

4.1.3. Water Wash (WW)

Caustic crack discharge pulp is diluted with water in the Water Wash (WW) circuit to facilitate residue filtration. This leaves excess caustic and spent caustic (as sodium silicates or sodium phosphates) in solution and REE hydroxides in the cake. The cake is well washed to remove acid consuming excess caustic and is then advanced to the secondary acid leach. The wash liquor advances to a silicon removal circuit (SiR) to regenerate caustic.

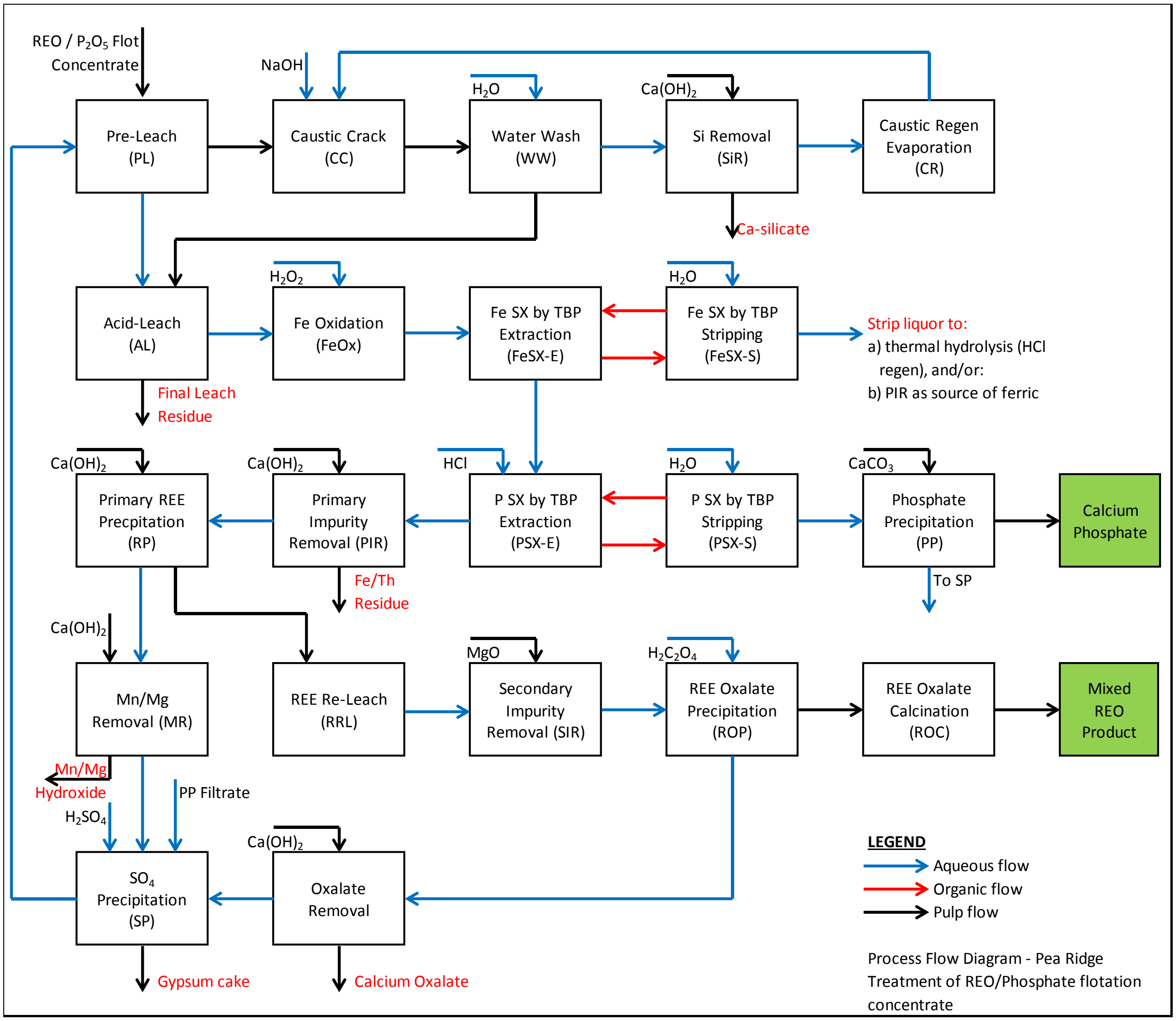


Figure 6: Process Flow Diagram – Hydrometallurgical Treatment of REO/Phosphate Flotation Concentrate

4.1.4. Silicon Removal (SiR)

Excess caustic and spent caustic contained in the WW filtrate is treated in a silicon removal (SiR) circuit with hydrated lime to produce calcium silicate, which is filtered off and removed from the circuit. Spent caustic (sodium silicate and sodium phosphate) reacts to form free caustic which, after evaporation of excess water, is returned to the caustic crack stage. Some of the aluminium and uranium is also removed in the SiR operation. The Si-free liquor subsequently advances to the evaporation operation.

4.1.5. Caustic Regeneration (CR)

SiR filtrate is evaporated to a target NaOH strength of 50-60% and re-used in the caustic crack circuit. Lost sodium (through cake moisture losses) is replaced by the addition of fresh caustic. Locked cycle testing showed a buildup of sulphur and potassium levels in the caustic liquor. This will require a bleed treatment to control these elements.

4.1.6. Acid Leach (AL)

The WW residue is leached in the acid leach circuit (AL) using excess acidity from the PL circuit. Overall (combined PL and AL) extraction were excellent at 99% P and 97% total rare earth elements (TREE).

4.1.7. Iron Oxidation (FeOx) and Solvent Extraction of Ferric Iron (FeSX)

Ferrous iron contained in the AL liquor is oxidized to ferric iron which is amenable to solvent extraction with TBP. While the testwork used hydrogen peroxide to oxidize the ferrous to ferric, it is believed that this can be accomplished with oxygen or even air in a commercial operation, at the cost of extended retention time requirement.

Ferric iron was successfully (> 98%) removed from the AL liquor using 100% TBP to form an iron loaded organic phase, which was subsequently stripped with water to form an acidic ferric chloride liquor. The ferric chloride liquor is required elsewhere in the flowsheet to improve selective removal of thorium from the phosphate SX raffinate. FeSX raffinate – containing all REE and phosphate – advances to a second solvent extraction step to extract phosphoric acid.

4.1.8. Solvent Extraction of Phosphate (PSX) and Phosphate Precipitation (PP)

Phosphate is removed form the FeSX raffinate by a second SX step (also 100%TBP) and stripped with water to make a dilute phosphoric acid solution. This is then neutralized with limestone to make a crude calcium phosphate product ($CaHPO_4$, 40% P_2O_5). High phosphate extraction (90%) is feasible and REE co-extraction (in the bench tests) is estimated at ~10%. REE co-extraction under continuous conditions is expected to be much lower. Uranium co-extraction approaches 100% in the combined iron and phosphate SX circuits. Hydrochloric acid is added to the extraction to maintain acidity and avoid precipitation of REE phosphates.

4.1.9. Primary Impurity Removal (PIR)

Phosphate SX raffinate is treated with ferric chloride (Fe SX strip liquor) and limestone in a primary impurity removal (PIR) circuit to remove thorium, scandium, and residual iron. Ferric chloride addition targets a Fe:P ratio of 1. This promotes the formation and precipitation of $FePO_4$, which is known to incorporate thorium phosphate. Thorium removal efficiency was excellent at 100% and REE losses were negligible (0-2%). PIR filtrate advances to primary rare earth precipitation.

4.1.10. REE Precipitation (RP)

Hydrated lime is used to precipitate REE contained in the PIR filtrate as a crude REO concentrate. This precipitate contains 25% TREE and 11%F. REE precipitation efficiencies are 99% or higher. The solids advance to a re-leach circuit to be able to conduct further purification steps on a more concentrated liquor. The filtrate advances to residual impurity removal (manganese / magnesium removal) and ultimately to a gypsum precipitation circuit to produce hydrochloric acid for recycle and gypsum as one of the primary waste streams.

4.1.11. REE Precip Re-Leach (RRL), Secondary Impurity Removal (SIR), and Rare Earth Oxalate Precipitation (ROP)

The crude RP precipitate is re-leached with hydrochloric acid and treated with magnesia to remove any residual thorium. SIR filtrate is subsequently treated with oxalic acid to form a 41% TREE product, which is calcined to decompose the oxalates to form a final REO oxide product of 96% TREO purity.

4.1.12. Manganese and Magnesium Precipitation (MR) and Sulphate Precipitation / Acid Regeneration (SP)

RP filtrate is treated with hydrated lime to remove manganese and magnesium. Precipitation efficiencies of 94 and 100% were determined under optimum conditions. The MR filtrate was combined with filtrates from phosphate precipitation and (neutralized) REE oxalate precipitation filtrate. This combined filtrate consisting primarily of calcium chloride was reacted with concentrated sulphuric acid to produce hydrochloric acid for recycle to the pre-leach and a gypsum cake for back fill or sold as a possible by-product.

4.2. Products and Recoveries (REO and Phosphate Concentrates)

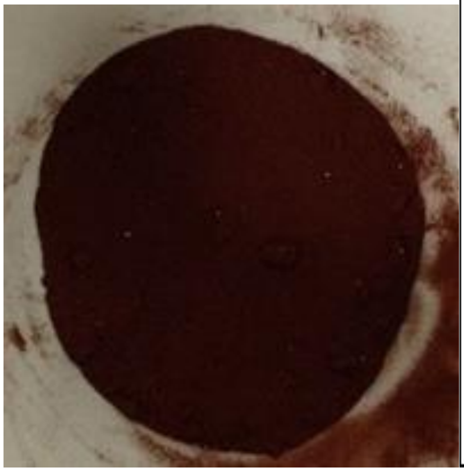
Overall recoveries (from ore to final product) of REO and calcium phosphate were calculated to be 69% and 74%, respectively. Individual circuit recoveries are shown in Table 8. Key impurities deport to different streams. Uranium is extracted by TBP and will end up in the iron or phosphate strip liquor and uranium extracted in the PSX will end up in the phosphate product unless removed. Thorium mostly reports to the primary impurity removal solids as a 0.1 - 0.2% Th precipitate, which would likely be mixed with other residue streams and diluted down significantly.

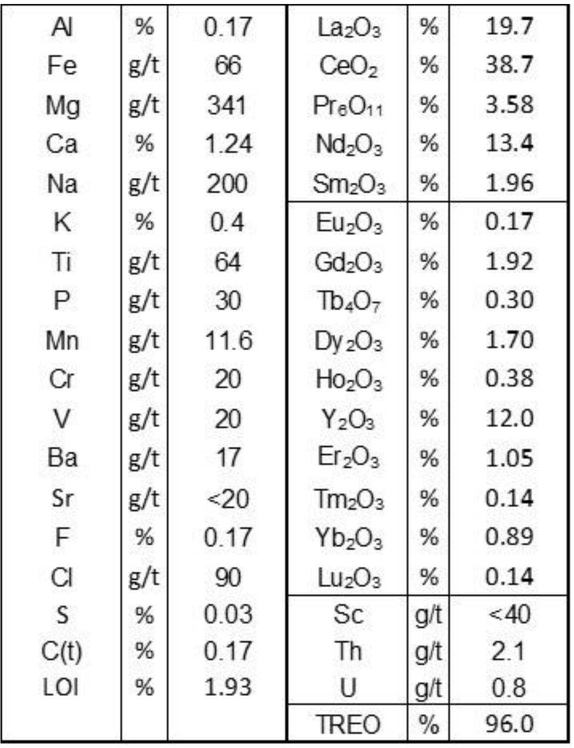
Table 8: Individual and Overall Circuit Recoveries (based on testwork)

Circuit Recoveries	Product	REO	P_2O_5	Notes
Beneficiation	Phos/REE Flot Con	81.2	91.2	
Leach Circuit (PL-CC-AL)	Preg Liquor	98	99.5	
Caustic Wash / Silicon Removal	Calcium Silicate	0	1	
Iron SX (FeSX)	Raffinate	99	91	
Phosphate SX (PSX)	Raffinate / Strip	89	90	REO recoveries can be increased by including scrubbing circuit
Phosphate Precipitation (PP)	**Calcium Phosphate**	n/a	92	
Primary Impurity Removal	PIR Filtrate	99	n/a	
REE Precipitation (RP)	Crude RP Precipitate	99	n/a	
REE Re-Leach (RRL)	Preg Liquor	100	n/a	Assumed that any REE losses in RRL are recovered by recycling
Secondary Th Removal	Purified Preg Liquor	100	n/a	Assumed that REE losses in SIR are recovered by recycling
REE Oxalate Precipitation (ROP)	ROP Solids	100	n/a	Assumed that REE losses in ROP are recovered by recycling
REO Calcination	**REO Calcine Product**	100	n/a	
Recovery Into Product		69%	74%	

4.2.1. REO Concentrate

A photograph and the composition of the REO concentrate is shown in Figure 7. The direct REO assay shows a grade of 96% TREO. Compared to concentrates produced in other projects, the Pea Ridge REO concentrate is clean, with only very low levels of deleterious elements, such as only 2.1 g/t Th and 0.8 g/t U. Key impurities include 1.2% calcium, 0.17% aluminium and 0.4% potassium. When the REO grade is determined by calculating 100-Impurities (%), a grade of around 98% is obtained.





Al	%	0.17	La_2O_3	%	19.7
Fe	g/t	66	CeO_2	%	38.7
Mg	g/t	341	Pr_6O_{11}	%	3.58
Ca	%	1.24	Nd_2O_3	%	13.4
Na	g/t	200	Sm_2O_3	%	1.96
K	%	0.4	Eu_2O_3	%	0.17
Ti	g/t	64	Gd_2O_3	%	1.92
P	g/t	30	Tb_4O_7	%	0.30
Mn	g/t	11.6	Dy_2O_3	%	1.70
Cr	g/t	20	Ho_2O_3	%	0.38
V	g/t	20	Y_2O_3	%	12.0
Ba	g/t	17	Er_2O_3	%	1.05
Sr	g/t	<20	Tm_2O_3	%	0.14
F	%	0.17	Yb_2O_3	%	0.89
Cl	g/t	90	Lu_2O_3	%	0.14
S	%	0.03	Sc	g/t	<40
C(t)	%	0.17	Th	g/t	2.1
LOI	%	1.93	U	g/t	0.8
			TREO	%	96.0

Figure 7: REO Concentrate – Photograph and Composition

4.2.2. Phosphate Concentrate

A crude phosphate concentrate is produced by neutralizing the PSX phosphoric acid strip liquor with limestone. This produces a calcium phosphate concentrate of around 39.7% P_2O_5 (17.3% P) assumed to be $CaHPO_4$, which is also known as dicalcium phosphate. Key impurities in this product are REE (~ 0.14% TREE), 41 g/t Th, and 43 g/t U.

Table 9: Phosphate Product Composition

Si	%	0.2	La	g/t	242
Al	%	0.1	Ce	g/t	538
Fe	%	0.1	Pr	g/t	59
Mg	%	0.24	Nd	g/t	211
Ca	%	24.9	Sm	g/t	49.5
Na	%	<0.01	Eu	g/t	4.6
K	%	0.01	Gd	g/t	47.1
Ti	%	0.03	Tb	g/t	6.4
P	%	17.3	Dy	g/t	35.7
Mn	%	<0.01	Ho	g/t	6.8
Cr	%	<0.004	Y	g/t	190
V	%	<0.01	Er	g/t	18.7
			Tm	g/t	2.7
			Yb	g/t	24.6
			Lu	g/t	4.7
			Sc	g/t	<25
			Th	g/t	41.1
			U	g/t	42.9
			F	%	1.0
			Cl	g/t	93

4.3. Reagent Consumption

The hydrometallurgical flowsheet for the treatment of the apatite flotation concentrate is characterized by high gross reagent consumptions. Lower mass pulls (and associated higher concentrate grades) will lead to a reduction in reagent consumption. Gross consumption data is presented per circuit in Table 10. The flowsheet, as developed, incorporates several reagent regeneration circuits, specifically:

- Hydrochloric acid. The primary leach reagent is HCl. All acid neutralization is conducted by calcium based reagents (lime, limestone). This leads to the formation of a concentrated calcium chloride brine final liquor. Addition of sulphuric acid to this liquor leads to the formation of hydrochloric acid and gypsum. Acid recycling was tested in a single pre-leach test and no deleterious effect were observed in this test.

- Caustic. Caustic is used to metathesize insoluble REE fluorides and phosphates and render them acid soluble as REE hydroxides. This leads to the formation of sodium silicates and sodium phosphates, which by the addition of lime (in the SiR circuit) react to form calcium silicates and sodium hydroxide. Caustic recycling was tested during one locked cycle test series.

- Ferric chloride. Ferric is used in the primary impurity removal (PIR) circuit, but is also produced in the FeSX circuit. Ferric recycling was not tested in the overall test program.

Table 11 summarizes the potential generation of HCl and NaOH expressed in kg/t ore. It is assumed that 90% of all chloride and 90% of the NaOH can be recycled. Ultimately this value depends on the chloride and sodium losses (soluble and insoluble) in the various circuits. Such losses must be determined during filtration tests carried out during integrated pilot plant operations.

Assuming that reagent regeneration is feasible and does not lead to impurity buildup or other deleterious effects on metal recovery, a net overall reagent consumption can be determined. Such assumptions need to be confirmed and sustained during integrated pilot plant operations. Table 12 shows the overall net reagent consumption and calculated hydromet reagent costs using assumed reagent costs. This shows that an overall reagent cost of 44 $/t ore may be feasible. Sulphuric acid is the highest contributor to the reagent costs at 36.5% followed by lime (25%) and hydrochloric acid (11%).

As noted elsewhere in this report (Section 3.2), the opportunity exists to process the pyrite concentrate and convert its sulphur contents into sulphuric acid. Preliminary calculations show that the pyrite concentrate could possibly supply 22% of the overall required sulphuric acid.

Table 10: Reagent Consumption – Hydromet Flowsheet

Unit Ops	Reagent	Addition kg/t conc	Consumption kg/t conc	Consumption kg/t ore
PL	HCl	720	319	52.6
CC	NaOH	2000	216	35.6
SiR	$Ca(OH)_2$	311	311	51.3
AL	HCl		37	6.1
FeOx	H_2O_2	5.0	5.0	0.8
FeSX				
PSX	HCl	101.4	101	16.7
PP	$CaCO_3$	416.6	417	68.7
PIR	$FeCl_3$	22.8	23	3.8
	$CaCO_3$	87.5	87	14.4
RP	$Ca(OH)_2$	94.1	94	15.5
RL	HCl	13.3	13	2.2
SIR/ThR	MgO	4.2	4	0.7
ROP	$H_2C_2O_4$	10.7	11	1.8
MnR	$Ca(OH)_2$	39.5	40	6.5
SP / AR	H_2SO_4	645.1	645	106.4

Table 11: Reagent Generation

Unit Ops	Reagent	Generation kg/t conc	Generation kg/t ore	Assumptions
FeSX	$FeCl_3$	30.9	5.1	Sufficient for FeOX requirement
AR	HCl	423.6	69.9	Assume 90% of chloride reprocessed into HCl Remainder (10%) is lost as soluble losses (moisture in cakes)
SiR	NaOH	194.4	32.1	Assume 90% of NaOH regenerated in SiR

Table 12: Summarized Reagent Consumption

Reagent	Consumption kg/t concentrate	Consumption kg/t ore	Reagent Costs $/t (100% reagent)	Reagent Costs $/t (ore)	Relative Costs %
HCl	47.1	7.8	600	4.7	10.7%
H_2SO_4	645	106.4	150	16.0	36.5%
NaOH	21.6	3.6	470	1.7	3.8%
$Ca(OH)_2$	445	73.4	150	11.0	25.2%
H_2O_2	5.0	0.8	1200	1.0	2.3%
$CaCO_3$	504	83.2	60	5.0	11.4%
MgO	4	0.7	750	0.5	1.2%
$H_2C_2O_4$	11	1.8	2212	3.9	9.0%
Total				**43.7**	**100.0%**

5. Knowledge Gaps and Opportunities for Optimization

Various gaps exist in the knowledge of the overall flowsheet. They are summarized below:

- The hydrometallurgical flowsheet is relatively complex and involves various recycle streams. The effect of prolonged recycling of reagents and process streams on a) metal leach performance and b) buildup of impurities has not been tested. This would normally be investigated during integrated pilot plant operations of sufficient duration. This may lead to increased reagent requirements than those calculated above;

- The REO and phosphate products should be evaluated by potential off-take companies to ensure specifications are met;

- Rheological and solid/liquid separation properties of concentrates, tails, leach residues, and precipitates have not been determined. These are required to estimate capital costs for thickening and filtration equipment. Also, wash efficiencies impact overall metal recoveries and reagent availability for regeneration (HCl and NaOH);

- The bulk of the reagent consumption is related to the processing of the REE/phosphate concentrate. Therefore, the effect of reducing the flotation mass pull from say 18 to 13% may have a significant effect on the overall net consumption of reagents. Testwork should be conducted focussing on lowering the mass pull while keeping losses minimum. Higher grade concentrate samples should subsequently be evaluated in the hydrometallurgical flowsheet to confirm it can be processed successfully.

- While uranium and thorium separation from REE was successful, no testwork was carried out to control radionuclide levels in the REO product. Additional treatment may be required;

- Environmental characterization of process plant tailings is required.

During the coarse of the testwork various opportunities to improve the performance of the circuit have been identified. They are summarized below:

- Further optimization of the bulk REE/phosphate flotation conditions to reject more gangue minerals and target a higher grade concentrate. This should lead to a lower reagent consumption in the hydrometallurgical flowsheet;
- Exploring reduced reagent consumption in pyrite flotation;
- Modify the combined WHIMS + Gravity Separation flowsheet to increase overall hematite recovery;
- Re-evaluate and optimize the reverse silicate flotation condition on the SLon concentrate;
- Evaluation of hydro-separation / elutriation on the magnetic (LIMS and SLon) concentrates to reject silicate gangue minerals;
- Production of fluoride precipitate by reacting the PL liquor with sodium chemicals (such as NaCl or NaSiO3). Preliminary testwork showed that around 70% of the fluoride contained in the PL liquor could be precipitated into a compound assaying 53% F and 14.7 % Si. Removal of the fluoride ion in the PL liquor should lead to a lower fluoride content in the final RL liquor and may have benefits in terms of material of construction requirement due to the very corrosive nature of acidic liquors containing high levels of fluoride;
- Continued examinations to reduce overall acid addition to PL and AL. Lower acid additions also lead to lower lime requirements, so this should be a high priority in any follow-up testwork. Net reduced acid additions can be accomplished by operating at higher PL pulp densities. Preliminary testwork has shown that a reduction of 30-40% is feasible by increasing the PL pulp density, but additional work is required to determine the impact on downstream unit operations;
- Investigation into acid recovery from PL and/or AL solutions by means of evaporation and distillation
- The phosphate SX circuit (PSX) requires the addition of HCl to avoid formation of insoluble REE phosphates and also to enable the efficient loading of phosphoric acid onto the organic reagent. A trade-off study should be carried out between HCl addition, phosphate recovery, and impurity removal in the PIR circuit;
- Changes in the leach circuit should be accompanied with a re-evaluation of the downstream circuit. Specifically, the following options should be considered:
  - Separate treatment of the PL and AL liquors. If the acid leach is conducted with fresh acid, a much more concentrated liquor can be produced. This may have some benefits including a net reduction in overall reagent consumption. Testwork is required to confirm this.
  - Examination of alternative REE extraction schemes, such as solvent extraction (for example D2EHPA, P507, Cx572) to extract bulk REE or possibly separate HREE and LREE concentrates.

6. Revenue Streams and Reagent Costs

Using the PAH resource estimate, the metallurgical extractions determined during the testwork program and assumed metal pricing, potential revenue streams can be calculated for the REO, phosphate, magnetite, and hematite product streams. It should be noted that the following analysis does not constitute a formal preliminary economic analysis (PEA) and does not follow 43-101 reporting standards. Product pricing was obtained from publicly available information, but no formal market investigation was carried out. Actual pricing may differ based on penalties due to presence of impurities. All revenue streams are based on US dollars per metric tonne of ore (i.e. iron ore tailings).

In the case of REO, no separation charges were taken into account. In some studies, a separation charge of 5 US$ per kg individual LREO and 15 US$ per kg individual HREO has been used. This will make the low value REO (such as La, Ce, Sm) uneconomic to produce as separated elements. Summarized revenue streams are summarized in Table 13, which shows a potential revenue of 122 US$/t ore.

Additional investigations should examine the potential of alternative revenue generators such as base and precious metals contained in the pyrite, or aluminum and potassium chemicals. Additionally, it is worth investigating if higher value phosphate products (including phosphoric acid) can be produced from the dilute phosphoric acid stream out of the PSX processing step.

Summarized reagent costs (for hydromet flowsheet only) were included in Table 12, which indicated total reagent costs (required for production of REO and phosphate product) of ~ 44 US$/t ore, roughly 36% of the potential revenue. As the bulk of the reagent costs will be related to the hydromet flowsheet, it seems that additional studies are justified in developing the Pea Ridge iron ore tailings deposit. It is therefore recommended that ThREE enters a formal PEA study, and provided a positive outcome is reached, to advance further into a PFS.

To advance from PEA further into PFS or FS, the following will be required:

- All mineral resources should be upgraded to the indicated and measured categories. Inferred resources category cannot be used in a PFS/FS report. All inferred resources will be considered waste and cannot be assigned an economic value;
- A marketing study should be completed outlining the potential markets for the different saleable products that the project will generate, prices, treatment costs (if applicable), transportation, and logistics, etc.;
- Site and tailings geotechnical study should be completed;
- A hydrological study should be completed.
- Additional testwork will be required to fill some of the knowledge gaps. Specifically, for FS level investigations, integrated pilot plant operations and feed variability studies will be required.

Table 13: Revenue Streams

		Head grade g/t or %	Price* $/kg REO	In Ground Value, US$/t ore	Overall Recovery ore to product %	Potental revenue, US$/t ore
La_2O_3	g/t	1017	2	2	62.7	1
CeO_2	g/t	1986	2	4	65.1	3
Pr_6O_{11}	g/t	169	80	14	65.6	9
Nd_2O_3	g/t	650	60	39	65.8	26
Sm_2O_3	g/t	103	3	0	65.1	0
Eu_2O_3	g/t	9	300	3	65.4	2
Gd_2O_3	g/t	98	20	2	65.9	1
Tb_4O_7	g/t	15	650	10	66.0	7
Dy_2O_3	g/t	91	350	32	66.5	21
Ho_2O_3	g/t	19	40	1	66.7	1
Y_2O_3	g/t	646	15	10	65.7	6
Er_2O_3	g/t	55	40	2	67.3	1
Tm_2O_3	g/t	8	0	0	61.8	0
Yb_2O_3	g/t	52	30	2	66.6	1
Lu_2O_3	g/t	9	990	9	41.2	4
Total REO	g/t	4928		128		82.4
Fe(tot)	%	19.8				
Fe(tot) recovered as magnetite	%	14.6	200	8.0		8.0
Fe(tot) recovered as hematite	%	44.5	50	6.3		6.3
P_2O_5**	%	4.22**				
$CaHPO_4$	%	8.09	400	32.4	76.7	24.8
Total						**122**

* REO pricing based on Alkane Resources FS
Ferrous pricing based on trade-metal.com / high purity magnetite market
Phosphate (assumed as Feed grade DCP) pricing based on correspondence with market analyst and producers.

** Equivalent to 10% apatite

References

Pincock Allen and Holt (PAH). Esteban Acuna. Resource Estimate of the Pea Ridge Iron Ore Tailings Project. August 27, 2012