RESEARCH PROJECT REPORT

Project Title:

PRODUCTION OF SALABLE RARE EARTHS PRODUCTS FROM COAL AND COAL BYPRODUCTS IN THE U.S. USING ADVANCED SEPARATION PROCESSES

PHASE 1

SUBMITTED TO:

U.S. Department of Energy
National Energy Technology Laboratory
626 Cochrans Mill Road
Pittsburgh, PA 15236-0940
Program Manager: Charles Miller

PROJECT REPORT INFORMATION:

Federal Grant Number: DE-FE0029956
DUNS Number: 07 042 4858
Marshall Miller & Associates, Inc., Project Control Number: MMA29956
Project Grant Period: 9/1/2017 – 5/31/2019
Submission Date: 09/13/2019

PRINCIPAL INVESTIGATOR AND AUTHORIZED REPRESENTATIVE:

Steven A, Keim, Ph.D., P.E.
Phone Number: +540-605-9004
Fax Number: +276-322-5460
E-Mail: Steve.Keim@mma1.com

SUBMITTED BY:

Marshall Miller & Associates, Inc.
582 Industrial Park Road
Bluefield, VA 24605

Steve A Keim

Steven A. Keim, Principal Investigator

Chapter 1.0
Phase 1 Executive Summary Report

TABLE OF CONTENTS

1.1 Executive Summary 1
  1.1.1 Introduction 1
  1.1.2 Feedstock Sampling, Identification, and Characterization 2
  1.1.3 System Design Preparation 3
    1.1.3.1 Summary Observations of the Lower Kittanning Seam Plant Rejects Prospect 4
    1.1.3.2 Summary Observations of the Investigation of the Pocahontas No. 3 Seam Plant Rejects Prospect 4
      1.1.3.2.1 Pocahontas No. 3 Seam Plant Circuit Samples TREE Content Results 4
      1.1.3.2.2 Pocahontas No. 3 Seam Preparation Plant Circuit Samples Heavy and Light REE Concentrations 5
    1.1.3.3 Summary Observations of the HHS Application to the Ferroglobe Blue Gem Seam and Jellico Coal Seam Rejects Samples 7
      1.1.3.3.1 Effects of Roasting on TREE Recovery from Three Float-sink Products of Rejects from Ferroglobe’s Gatliff Plant (Blue Gem Seam) 7
    1.1.3.4 Arq Corbin USA (Arq) – Summary Observations of HHS Treatment of Rejects Impoundment Samples 9
    1.1.3.5 Summary Observations of the Blue Diamond Coal Company (Operated by Blackhawk) Fire Clay (Hazard No. 4) Seam Examinations for REE Concentration Purposes. 9
      1.1.3.5.1 No. 76 Plant (Leatherwood) Fire Clay Seam (Hazard No. 4) Thickener Underflow Acid Leaching Testing Methodology 10
    1.1.3.6 Solvent Extraction Testing on a Variety of Feedstocks 12
    1.1.3.7 Detailed Flow Diagram 14
    1.1.3.8 Flowsheet Simulation to Evaluate Process Circuitry for REE Concentration from Coal-based Sources 18
      1.1.3.8.1 Flowsheet Development Software Selection 18
      1.1.3.8.2 Evaluation of LIMN 18
      1.1.3.8.3 Evaluation of REESim 18
    1.1.3.9 Summary Discussion of the Simulation Results for Project MMA 29956 18
    1.1.3.10 Availability of a 1 Short Ton per Hour Pilot Plant 20
      1.1.3.10.1 Pilot Plant Site Proposed for the 1 Short Ton per Hour REM Concentration Plant 21
    1.1.3.11 Risk and Fatal Flaw Analysis 23
      1.1.3.11.1 Summary of Significant Concerns 23
      1.1.3.11.2 Timing of Regulatory Permits, Agreements or Contracts 23
      1.1.3.11.3 Performance of the Proposed Solvent Extraction Process 23
      1.1.3.11.4 Roaster Performance with a Coal Preparation Plant Rejects Feedstocks 23
  1.1.4 Techno-Economic Analyses 23
    1.1.4.1 Summary of All Major Experimental Data, Engineering Analysis, Computations and Test Results (Fire Clay Seam (Hazard No. 4) Rejects. 24

1.1.4.2 Economic Assumptions....26
1.1.4.3 Economic Indicators for a Commercial-Scale Plant (500 Short Tons per Hour)....27
1.1.4.4 Influence of Optimal Roasting Conditions....28
1.1.4.5 Key Findings....29
1.1.4.6 Potential Pathway to Profitability....30
1.1.5 Permitting and Environmental Management for Phase 2 31
1.1.5.1 Sources of Run-of-Mine Fire Clay (Hazard No. 4) Coal from Which the Phase 1 Rejects Was Extracted....32
1.1.5.1.1 Mine No. 81 (Calvary Mine 81)....32
1.1.5.1.2 Mine No. 89 (Orchard Branch Mine 89)....32
1.1.5.2 Source of Rejects Derived from Processing the Fire Clay (Hazard No. 4) Seam During Phase 1....32
1.1.5.2.1 Blue Diamond Coal Company No. 76 Plant (Operated by Blackhawk Mining 32
1.1.5.3 Financial Viability of Blackhawk....32
1.1.5.4 Lessor/Lessee Relationships....33
1.1.5.5 Pilot Plant Permitting Plan for Phase 2....33
1.1.5.5.1 Likely Coal Mining Related Permit Obligations....33
1.1.5.5.2 Wastewater from the Pilot Plant Processing System....34
1.1.5.5.3 Solid Waste Generated by the Pilot Plant Processing System....34
1.1.5.6 Environmental Management Plan for the Pilot Plant Operations....34
1.1.5.6.1 Process Chemical Toxicity....35
1.1.5.6.2 Solid and Liquid Process Waste Streams....35
1.1.5.6.3 Air Quality Control Measures....35
1.1.5.6.4 Surface and Groundwater protection measures....35
1.1.5.6.5 Spill Prevention Control and Countermeasure Plan (SPCC)....35
1.1.5.6.6 Stormwater Pollution Prevention Plan and Stormwater Management Plan 36
1.1.5.7 Best Management Practices (BMPs)....36
1.1.5.7.1 Operational Source Control BMP....36
1.1.5.7.2 Structural Source Control BMPs....36
1.1.5.7.3 Treatment BMPs....36
1.1.5.7.4 Erosion and Sediment Control BMPs....37
1.1.5.8 Process Water Treatment....37
1.1.5.9 Environmental Monitoring....37
1.1.5.10 Environmental Management Plan Components for Phase 2....37
1.1.6 Pilot Plant Product Market Assessment and Purchase Agreement Sample.... 38
1.1.6.1 General Discussion of REE Demand Structure....38
1.1.6.2 General Discussion of the Supply Structure....38
1.1.6.3 Specific Discussion of the Market for the Pilot Plant Products....39
1.1.6.3.1 Gadolinium (Gd) Market Environment....39
1.1.6.3.2 Dysprosium (Dy) and Industrial Magnet Production....39
1.1.6.3.3 Yttrium (Y) Market Trends....39
1.1.6.4 REE Price Structure....40
1.1.6.5 Critical Factors Affecting the Rare Earth Market....40
1.1.6.6 Pilot Plant Products Sales Agreement Development....41

1.1.6.6.1 Establishment of Marketing/Tolling Agreements for Sales of REE Products 41
1.1.6.6.2 Informal MMA 29956 Pilot Plant Product Off-Take Arrangements 41
1.1.7 Environmental Critique of the Proposed 1 Short Ton per Hour Pilot Plant 41
1.1.7.1 List of Major Sources of Environmental Risks 41
1.1.7.2 Controls/Mitigation Strategies Proposed for each Item of Concern 41
1.1.7.2.1 Modification of Feed Materials 41
1.1.7.2.2 Concentration of Radioactive Materials 42
1.1.7.2.3 Release of Reagents 42
1.1.7.2.4 Elimination of Water Quality Impacts from Process-Water 42
1.1.7.2.5 Elimination of Air Quality Impact 42
1.1.7.3 Environmental Assessment/Environmental Impact Statement (EA/EIS) Response/Mitigation 43
1.1.8 Benchmark Process Audits 43
1.1.8.1 Deferred Audit of a USA-Based Toll Processor 44
1.1.8.2 MP Materials, Mountain Pass, California (Pending REE Producer and Feedstock Provider) 44
1.1.8.3 Arizona Pilot Plant (Hydro- Met Facility, Tucson, Arizona, Potential Feedstock Producer) 45
1.2 Acknowledgments 46
1.2.1 Principal Investigators 46
1.2.2 Participating and/or Supportive Entities 47

TABLES (IN THE TEXT)

Table 1-1: Feedstock Source Evaluation Sequence 2
Table 1-2: Basic Summary of In-Place Non-Coal Rock and REE+Y (REY) Short Tons 3
Table 1-3: Preparation Plant, Lower Kittanning Seam, REE Concentration of Tails of Plant Circuit Samples after HHS 4
Table 1-4: Elemental Analysis of the Final REO Generated from Pocahontas No. 3 Seam Middlings Reject Material 13
Table 1-5: REESim Predicted Assay Values for the Overall Plant Operations 19
Table 1-6: REESim Distribution Values for the Overall Plant Adjusted for Pilot Plant Experience 20
Table 1-7: Feedstock REE Concentration (Nominal) Used in Techno-Economic Analysis 24
Table 1-8: REE to REO Conversion Factors Used in Techno-Economic Analysis 25
Table 1-9: Overall Element-by-Element Recovery Values Used in Techno-Economic Analysis 25
Table 1-10: Summary of Economic Indicators for Commercial-Scale Plant (500 st/hr.) 28
Table 1-11: Project Economic Summary for Material Processed under Differing Roasting and Acid Application Conditions 29
Table 1-12: Economic Indicators for Commercial Plant under Improved Input Conditions 31

FIGURES (IN THE TEXT)

Figure 1-1: REE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 seam 5
Figure 1-2: HREE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Seam Preparation Plant 6
Figure 1-3: LREE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Seam Preparation Plant 6
Figure 1-4: Effects of Roasting on the TREE Recovery from (a) 1.8 SG Float, (b) 1.8-2.2 SG, and (c) 2.2 SG Sink Fraction of the Blue Gem Seam Coarse Reject Samples 8
Figure 1-5: Total REE Recovery Achieved by Leaching the Unroasted Products Generated of the No. 76 Plant (Leatherwood) Thickener Underflow Material Samples 11
Figure 1-6: TREE Recovery Achieved by Leaching the Roasted Products Generated from the Processing of the Leatherwood Thickener Underflow Material 12
Figure 1-7: Process Flow Diagram for REE Extraction from Coal-based Sources 15
Figure 1-8: Detailed Process Flow Diagram for the Feedstock Preparation Circuit 15
Figure 1-9: Detailed Process Flow Diagram for the Acid Leaching and Waste Treatment Circuit 16
Figure 1-10: Detailed Process Flow Diagram for the Rougher Solvent Extraction Circuit 16
Figure 1-11: Detailed Process Flow Diagram for the Cleaner Solevent Extraction Circuit 17
Figure 1-12: Detailed Process Flow Diagram for the Scandium Solvent Extraction Circuit 17
Figure 1-13: Previously Operated Pilot Plant Potentially Suited for Reutilization 21
Figure 1-14: Surveyed Map of the Truss Joist Facility 22

BIOGRAPHIES OF WRITERS

Chapter 2.0
Feedstock Sampling, Identification & Characterization

TABLE OF CONTENTS

2.1 Executive Summary 1

2.2 Introduction 2
2.2.1 Precedent Investigations 2
2.2.2 The Fire Clay Seam Initiatives 3

2.3 Geology: Fire Clay Seam Sample Collection, Site Characterization, and Deposit Modeling 4
2.3.1 Previous Fire Clay Seam Geological Investigations 4
2.3.2 Stratigraphy 4
2.3.3 Property Description 5
2.3.4 Data Sources 6
2.3.5 Modeling Methods and Features 7
2.3.5.1 Roof and Floor Material (Map 1) 8
2.3.5.2 Interval Partings between the Hazard No. 4 Coal and the Hazard No. 4 Flint Clay (Map 2) 8
2.3.5.3 Hazard No. 4 Flint Clay (Map 3) 8
2.3.5.4 Interval Partings between the Hazard No. 4 Flint Clay and the Hazard No. 4 Jack Rock Coal (Map 4) 8
2.3.5.5 Floor Material (Again, Map 1) 9

2.4 Resource Estimation, Concentrations, and Tonnage 9
2.4.1 Delineation of Zones of Confidence and Related Assumptions 9
2.4.2 Estimation of Tons of Non-coal Lithologies 9

2.5 Index of Maps and Exhibits 12

2.6 References Cited 12

2.7 Acknowledgments 13

TABLES (IN THE TEXT)

Table 2-1: Feedstock Source Evaluation Sequence 2
Table 2-2: Basic Summary of In-Place Non-Coal Rock and REY Short Tons 10
Table 2-3: Distribution of In-Place Y+REE Within Future Mining in Mine Nos. 81 and 89 and the "Study Area" 11

FIGURE (IN THE TEXT)

Figure 2-1: Potential REE Resource Material from a Mining Bench (not-to-scale) 7

EXHIBITS

MAPS

Chapter 3.0
System Design Package Preparation

TABLE OF CONTENTS

3.1 Summary of Investigations of REE Concentration Plant Feedstock Sources 1
3.1.1 Inventory of Coal and Coal Byproducts Feedstock Sources Examined and Tested in the Course of this Task 1
3.1.1.1 Evaluation of LIMN 2
3.1.2 West Virginia Mining Complex, Lower Kittanning Coal Seam 2
3.1.2.1 Lower Kittanning Coal Seam Mining and Preparation Plant Complex 2
3.1.3 West Virginia Mining Complex, Pocahontas No. 3 Seam Coal 2
3.1.3.1 Pocahontas No. 3 Seam Preparation Plant Survey – West Virginia 2
3.1.3.2 Pocahontas No. 3 Coal Seam Plant Circuit Samples TREE Content Results 4
3.1.3.3 Pocahontas No. 3 Coal Seam Preparation Plant Circuit Samples Heavy and Light REE Concentrations 5
3.1.3.4 Pocahontas No. 3 Coal Seam Plant Circuit Samples Normalized Ratio of HREE/LREE 6
3.1.3.4.1 Pocahontas No. 3 Coal Seam Preparation Plant Circuit Samples TREE in Relation to Process Solid Types 7
3.1.3.5 Pocahontas No. 3 Coal Seam Preparation Plant Circuit Samples REE Concentrations versus Ash Content 9
3.1.3.6 Pocahontas No. 3 Seam Preparation Plant Circuit Sample Character versus Coal Mining Methods, Plant Low-Ash Middlings and Plant High-Ash Middlings Sample Test Results 14
3.1.3.6.1 Pocahontas No. 3 Coal Seam Preparation Plant Circuit High Ash and Low Ash Middlings Sample Treatment Protocols 15
3.1.3.6.2 Pocahontas No. 3 Coal Seam Preparation Plant High Ash and Low Ash Middlings Samples Decarbonization Results 17
3.1.3.7 Pocahontas No. 3 Coal Seam Preparation Plant Circuits Low Ash Middlings HHS Coal and Tailings REE Results 19
3.1.3.8 Pocahontas No. 3 Coal Seam Preparation Plant Circuit High Ash Middlings HHS Coal and Tailings REE Results 21
3.1.3.9 Pocahontas No. 3 Coal Seam Rejects REE Concentration by Optical Sorting 24
3.1.3.9.1 Dual X-Ray Transmission Sorting (DXRT) 24
3.1.4 Ferroglobe Preparation Plant Circuit Sampling Program 27
3.1.4.1 Ferroglobe High Ash (Jellico) and Low Ash (Blue Gem) Preparation Plant Circuit Samples 27
3.1.4.1.1 Ferroglobe High Ash and Low Ash Sample Flotation Feed Testing and Screenbowl Effluent 28
3.1.4.1.2 Ferroglobe Screenbowl Centrifuge Effluent Sample Testing 33
3.1.4.1.3 Ferroglobe Spiral Stream Testing 33
3.1.4.2 Ferroglobe - Blue Gem Seam Coal Coarse Rejects 38

3.1.4.3 Ferroglobe - Jellico Seam Coal Clean Coal Sieve Overflow Sample Testing 38
3.1.4.4 Ferroglobe - Summary Observations of the HHS of the Blue Gem and Jellico Coal Seam Rejects Samples..40
3.1.5 Arq Corbin LLC Plant Material – HHS Prospect 41
3.1.5.1 Background to the Arq Investigation...41
3.1.5.2 Arq Project – HHS Test Results of an "As-Received" Sample of Coal Rejects Impoundment...41
3.1.5.2.1 Arq Project – HHS Test Results of Milled Sample of Coal Rejects Impoundment...41
3.1.5.2.2 Arq Project – Sample Ground to D80 Passing 8 Microns 42
3.1.5.2.3 Arq Project – HHS Test Results of Milled Sample of Coal Rejects Impoundment with Dispersant Additive...43
3.1.5.3 Arq Project – HHS Test Results of Attrition Grinding of Sample of Coal Rejects...43
3.1.6 Blue Diamond Coal Company: Fire Clay Coal Seam (Hazard 4) Seam Rejects.. 44
3.1.6.1 Fire Clay Seam (Hazard 4) Coal Reject Feedstock...44
3.1.6.2 Hazard 4 Thickener Underflow Froth Flotation Test Results 44
3.1.6.3 HHS of the Hazard 4 Seam Thickener Underflow of the Leatherwood Plant..........46
3.1.6.3.1 Results of Standard HHS Testing Procedures of the Leatherwood Plant, Hazard 4, Thickener Underflow 47
3.1.6.3.2 Supplemental Treatment of Leatherwood Plant Thickener Underflow Prior to HHS to Improve REM Recovery...49
3.1.6.3.2.1 HHS Feedstock Preparation by Blunging... 49
3.1.6.3.2.2 Oil Agglomeration and removal of Coal Particles from the HHS Feedstock.. 52

3.2 Roasting and Acid Leaching Experiments 57
3.2.1 Roasting 57
3.2.2 Acid Leaching.. 57
3.2.3 Leachability Tests of the Rejects from the Pocahontas No. 3 Coal Seam............ 59
3.2.3.1 Unroasted Pocahontas No. 3 Coal Seam Channel Sample Segments (Roof, Coal, and Floor) 59
3.2.3.2 Unroasted Pocahontas No. 3 Coal Seam Plant Coarse Rejects..................................62
3.2.3.2.1 Summary Observations of the Leaching Tests of the Unroasted Pocahontas No. 3 Coal Seam Coarse Rejects 64
3.2.3.3 Acid Leaching Response to Roasting Decarbonized Pocahontas No. 3 Coal Seam Preparation Plant Middlings Material 68
3.2.3.3.1 Step 1: Comparison of REE Recovery from Unroasted and Roasted Decarbonized Middlings Samples Prior to Acid Leaching..............................68
3.2.3.3.2 Step 2: Comparison of TREE Recovery from Acid Leaching Unroasted and Roasted Pocahontas No. 3 Seam Middlings Samples..............69
3.2.3.4 Distribution of TREE, HREE, LREE and CREE Content in Acid Leachates from Unroasted and Roasted Decarbonized Samples of Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit 69
3.2.3.4.1 Results: Effects on TREE, HREE, LREE and CREE 70
3.2.3.4.2 Results: Discussion on the Recovery of CREE 70

3.2.3.5 Acid Leaching of Unroasted and Roasted Coarse Rejects and Middlings from the Pocahontas No. 3 Coal Seam Preparation Plant..71
3.2.3.5.1 Impact of Roasting on Proximate Analyses of the Coarse Rejects Sample Prior to Acid Leaching..71
3.2.3.5.2 Acid Leaching Kinetics: Effects of Roasting Coarse Rejects and Middlings Circuit Samples from the Pocahontas No. 3 Coal Seam Preparation Plant..72
3.1.6.3.2.3 Effects of Roasting Temperature..72
3.2.3.5.3 Effects of Inorganic Acid Type on REE Recovery from the Roasted Pocahontas No. 3 Coal Seam Preparation Plant Coarse Rejects and Middlings Circuit Product Samples..75
3.2.3.5.4 REE Recovery from Coarse Rejects and Middlings Circuit Samples at Optimum Temperature and Acid Type Conditions 76
3.2.3.5.5 Effect of Roasting Temperature, Acid Type, and Leach Tank Temperature on Ce, Y, Dy and Sc Recovery from the Pocahontas No. 3 Coal Seam Preparation Plant Middlings and the 2.2 Specific Gravity (SG) Float Fraction of its Coarse Rejects 76
3.2.3.5.6 Summary Observations of Acid Leaching Behavior of Unroasted and Roasted Samples..80
3.2.3.6 Acid Leaching Behaviors of other Major Elements and Trace Elements in Unroasted and Roasted Samples of the Pocahontas No. 3 Coal Seam Preparation Plant..80
3.2.3.6.1 Summary Observations of the Acid Leaching of other Major Elements and Trace Elements from Unroasted and Roasted Samples of the Pocahontas No. 3 Coal Seam Preparation Plant 90
3.2.3.7 Effects of Acid Leaching Tank Temperature and Acid Type on the Recovery of REE from Roasted Samples from the Pocahontas No. 3 Coal Seam Preparation Plant 91
3.2.3.7.1 Effects of Leaching Temperature 91
3.2.3.8 Effects of Acid Leaching Solution Temperature and Acid Type on the Recovery of other Major Elements Associated with REEs in the Roasted Samples from Pocahontas No. 3 Coal Seam Preparation Plant..94
3.2.3.9 Effects of Leachate Solution pH and Acid Type on the Recovery of REEs and Associated Elements from the Roasted Coarse Rejects Sample of the Pocahontas No. 3 Coal Seam Preparation Plant 96
3.2.3.9.1 Effects of Solution pH 96
3.2.3.9.2 Effects of Acid Concentration on the Leaching Recovery of LREEs and HREEs 96
3.2.3.9.3 Effects of Acid Concentration on the Leaching Recovery of other Elements Associated with REEs..97
3.2.3.10 Summary Observations of the Effects of Roasting Temperature, Leachate Solution pH and Acid Type on the Recovery of REEs and Associated Elements from the Roasted Coarse Rejects Sample of the Pocahontas No. 3 Coal Seam Preparation Plant 99
3.2.4 Leachability Tests of the Rejects from the Blue Gem Seam Coal........................ 99
3.2.4.1 Effects of Roasting on TREE Recovery from Three Float-sink Products of Rejects from Ferroglobe’s Gatliff Plant (Blue Gem Seam)..99

3.2.4.1.1 Summary Observations of the Effects Roasting on Total REE Recovery from Three Float-Sink Products of Rejects from Ferroglobe's Gatliff Plant 100
3.2.4.2 Effects of Roasting on HREE and LREE Leach Recovery from Three Float-sink Products of Rejects from Ferroglobe's Gatliff Plant (Blue Gem Seam) 101
3.2.4.2.1 Summary Observations of the Effects of Roasting on HREE and LREE Leach Recovery from Three Float-Sink Products of Rejects from Ferroglobe's Gatliff Plant 102
3.2.4.3 The Effects of Roasting on the Leaching of Sc from (a) 1.8 SG Float, (b) 1.8-2.2 SG, and (c) 2.2 SG Sink Fractions of the Blue Gem Seam Coarse Reject Samples 102
3.2.4.3.1 Summary Observations of the Effects of Roasting on Sc Recovery from Three Float-Sink Products of Rejects from Ferroglobe's Gatliff Plant 103
3.2.4.4 Summary of Methodology and Observations of the Leachability Tests of the Rejects from the Blue Gem Seam Coal 103
3.2.5 Leachability Tests of the Rejects from the Fire Clay (Hazard No. 4) Seam Coal 103
3.2.5.1 Fire Clay (Hazard No.4) Coal, the Host Land Company, and the Host Coal Mining and Processing Company. 103
3.2.5.2 Effects of Roasting on the Mineralogy of the Fire Clay (Hazard No. 4) Rejects Material 104
3.2.5.3 No. 76 Plant (Leatherwood) Fire Clay Seam (Hazard No. 4) Thickener Underflow Acid Leaching Testing Methodology 104
3.2.5.3.1 Acid Leaching Test Results of Unroasted Fire Clay Seam (Hazard No. 4) Flotation Middling Material from the No. 76 Plant (Leatherwood) Thickener Underflow 105
3.2.5.3.2 Acid Leaching Test Results of Roasted Fire Clay Seam (Hazard No. 4) Flotation Middling Material from the No. 76 Plant (Leatherwood) Thickener Underflow 106
3.2.5.4 No. 76 Plant (Leatherwood) Fire Clay Seam (Hazard No. 4) Coarse Rejects Acid Leaching Testing Methodology 106
3.2.5.4.1 Effect of Solids and Acid Concentration on Leaching Recovery and Kinetics for the REEs Associated with the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material) 107
3.2.5.4.2 Effect of Acid Concentration on Leaching Recovery and Kinetics for the REEs Associated with the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No 4) Coarse Rejects Material 108
3.2.5.4.3 Effect of Acid Concentration on Leaching Recovery and Kinetics of other Elements Associated with REE Mineral in the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material 109
3.2.5.4.4 Effects of Acid Leaching Time on the Recovery of REE and Associated Contaminant Elements 111
3.2.5.4.5 Effects of Percent Solids in the Leaching Tank Contents on Recoveries of REE and Other Associated Elements from the Fire Clay (Hazard No. 4) Seam Rejects when using Very Low Acid Concentrations 112
3.2.5.4.6 Evaluation of the Co-Extraction of Lithium (Li) and Cobalt (Co) with REEs from the Fire Clay (Hazard No. 4) Seam Coarse Rejects of the No. 76 Plant (Leatherwood) 113

3.2.5.5 Evaluation of the Effectiveness of Commercial Roasting Services with Respect to Acid Leaching of REEs from Roasted Fire Clay (Hazard No. 4) Seam Coarse Rejects....113
3.2.5.5.1 Description of Commercial Roasting Trials Provided by Nex-Gen Industries 113
3.2.5.5.2 Comparison Laboratory Testing of the Leaching Characteristics of the Commercially Roasted Material 115
3.2.5.5.3 Preliminary Conclusion of the Effects of Commercial Roasting Services by Nex-Gen Industries 117

3.3 Solvent Extraction.... 117
3.3.1 Solvent Extraction Rougher/Cleaner Testing 117
3.3.1.1 Extractant Concentration Effect on Solvent Extraction 117
3.3.1.2 Effect of Different Anions on Solvent Extraction....118
3.3.1.3 Bench-Scale Testing of the Solvent Extraction Proposed for this Project 118
3.3.1.3.1 Laboratory Set-up of Rougher Loading, Scrubbing and Stripping 118
3.3.1.3.2 Continuous Flow SX Testing using Pocahontas No. 3 Coal Seam Preparation Plant Reject Samples....119
3.3.1.4 Evaluation of Extraction Characteristics of the REEs and Contaminant Elements 123
3.3.1.4.1 Rougher Stage Activities....123
3.3.1.4.2 Extraction of Metal Ions in the Stripped Solution from the Rougher Stage 124
3.3.1.4.3 Cleaner Circuit Simulations....125
3.3.1.4.4 Precipitation of REO From Stripped Solution by use of Oxalic Acid 128
3.3.1.4.5 Analysis of the Final Rare Earth Oxide Product....128
3.3.1.4.6 Ascorbic Acid Optimization to Control Iron (Fe) Contamination of the Leach Liquor from the Pocahontas No. 3 Coal Rejects 129
3.3.1.4.7 Scandium (Sc) Recovery from Pocahontas No. 3 Seam Middlings Rejects 131
3.3.1.5 Selective Loading of REEs in Solvent from Ferroglobe's Blue Gem Seam Tailings Materials....132
3.3.1.6 Characteristics of Individual REEs with Extractants 133
3.3.1.6.1 Effects of Extractants and Parameters on Individual REEs Collected from W. Ky No. 13 (a.k.a., Baker) Coal Seam Acid Mine Drainage (AMD) 133
3.3.1.7 Exploration of the Separation of Individual REEs (Y, Dy, and Nd from W. Ky. No. 13 Coal Seam AMD) 136
3.3.1.7.1 Stage One 137
3.3.1.7.2 Stage Two 138
3.3.1.7.3 Stage Three....138
3.3.1.7.4 Procedure: Extraction Experiment, Stage One 138
3.3.1.7.5 Procedure: Distribution Isotherm, Stage Two 139
3.3.1.7.6 Extraction Experiment, Stage One....139
3.3.1.7.7 Effect of Extractant pH....140

3.3.1.7.8 Effect of Extractant Concentration of the Recovery of Dy and Y using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD 143
3.3.1.7.9 Distribution Isotherm, Stage Two 145
3.3.1.7.10 Distribution Coefficients 147
3.3.1.7.11 Saponification Refinements 149

3.4 Detailed Flow Diagram 153
3.4.1 Flow Diagram Description 153
3.4.1.1 Circuit 1 – Feed Preparation 154
3.4.1.2 Circuit 2 – Acid Leaching and Waste Treatment 155
3.4.1.3 Circuit 3 – Rougher Solvent Extraction 157
3.4.1.4 Circuit 4 – Cleaner Solvent Extraction 158
3.4.1.5 Circuit 5 – Scandium Solvent Extraction 159

3.5 Flowsheet Simulation to Evaluate Process Circuitry for REE Concentration from Coal-based Sources 161
3.5.1 Flowsheet Development Software Selection 161
3.5.1.1 Evaluation of LIMN 161
3.5.1.2 Evaluation of REESim 161
3.5.1.2.1 REESim Software Description 161
3.5.2 Components of REESim Circuit Simulation Workbook for Project MMA 29956 162
3.5.3 Component Flow Rates Generated by REESim for Project MMA 29956 164
3.5.4 Circuit-by-Circuit Performance Indicators 171
3.5.4.1 Summary Discussion of the Simulation Results for Project MMA 29956 171
3.5.5 Detailed Listing of Equipment Requirements of a 1 tph Feed Rate Pilot Plant 184
3.5.6 MATLAB App for SX Design 196

3.6 Flow Sheet of 1 tph Feed Pilot Plant 202
3.6.1 Engineering Drawings Needed for Construction/Fabrication and Installation of System 202
3.6.1.1 Utilization of the Flow Sheets from an Existing Pilot Plant 206
3.6.1.2 Previously Used Pilot Plant Construction Drawings Showing Spatial Layout of Equipment/Piping/Plumbing/Electrical/Mechanical 210
3.6.1.2.1 Crushing and Grinding Equipment and Layout 210
3.6.2 Design Sufficient for CapEx Estimates 216

3.7 Project Risk and Fatal Flaw Analysis 216
3.7.1 Risk and Fatal Flaw Analysis Methodology 216
3.7.2 Limits of the Analysis Procedure 217
3.7.2.1 Risk and Fatal Flaw Analysis Matrix 217
3.7.3 Summary of Significant Concerns 223
3.7.3.1 Timing of Regulatory Permits, Agreements or Contracts 223
3.7.3.2 Performance of the Proposed Solvent Extraction Process 223

3.7.3.3 Roaster Performance with a Coal Preparation Plant Rejects Feedstocks 223

3.8 Acknowledgments 223

3.9 References 224

TABLES (IN THE TEXT)

Table 3-1: Coal and Coal Byproduct Feedstock Source Evaluation Sequence 1
Table 3-2: West Virginia Mining Complex, Lower Kittanning Seam, REE Concentration of Tails of Plant Circuit Samples after HHS 2
Table 3-3: Listing of Process Streams Sampled for REE Analysis 3
Table 3-4: Pocahontas No. 3 Coal Seam Preparation Plant HAM and LAM Testing Matrix 15
Table 3-5: Pocahontas No. 3 Coal Seam Preparation Plant Decarbonization Test Summary 17
Table 3-6: Pocahontas No. 3 Coal Seam Preparation Plant LAM Decarbonization Results 17
Table 3-7: Pocahontas No. 3 Coal Seam Preparation Plant HAM 9.5 x 1 mm Float Sample Decarbonization Results.. 18
Table 3-8: Pocahontas No. 3 Coal Seam Preparation Plant HAM 9.5 x 1 mm Sink Sample Decarbonization Results.. 18
Table 3-9: Pocahontas No. 3 Coal Seam Preparation Plant HAM 1.0 x 0.15 mm Sample Decarbonization Results.. 18
Table 3-10: Pocahontas No. 3 Coal Seam Preparation Plant DriJet™ X-Ray Sorter Test Results 25
Table 3-11: Pocahontas No. 3 Coal Seam Preparation Plant Analysis of LAM Particles with Different Colors from X-Ray Scans 27
Table 3-12: Low (Blue Gem) and High Ash (Jellico) Test Matrix of Ferroglobe Flotation Circuit and Screenbowl Sampling 28
Table 3-13: Particle Size Analysis of Flotation Feed for Three (3) Percent Ash (Jellico) Run.............. 28
Table 3-14: HHS Test Results on Flotation Feed (As-Received) Material Targeting Three (3) Percent Product Ash (Jellico) Content 29
Table 3-15: Particle Size Analysis of the Flotation Feed Collected during the 1 Percent Ash (Blue Gem) Plant Run.. 29
Table 3-16: HHS Separation Performance Achieved on Flotation Feed (As-Received) Collected from the 1 percent Ash Run Results 29
Table 3-17: Particle Size Analysis of the Flotation Product Collected during the 3 Percent Ash (Jellico) Plant Run.. 30
Table 3-18: HHS Performance when Treating Flotation Product (As-Received) Collected during a One (1) Percent Ash Plant Run 30
Table 3-19: Size Analysis of Flotation Product for One (1) Percent Ash (Blue Gem) Run.................... 31
Table 3-20: HHS Performance when Treating Flotation Product (As-Received) Collected during a One (1) percent Ash (Blue Gem) Plant Run 31
Table 3-21: Particle Size Analysis of the Flotation Tailings Sample Collected during the Three (3) Percent Ash (Jellico Seam) Plant Run... 31
Table 3-22: HHS Performance when Treating Flotation Product (As-Received) 32
Table 3-23: Particle Size Analysis of the Flotation Tailings Sample Collected during the One (1) Percent Ash Plant Run... 32
Table 3-24: HHS Performance when Treating Flotation Product (As-Received) Collected during a One (1) Percent Ash Plant Run 32
Table 3-25: Particle Size Analysis of the Screenbowl Effluent Sample Collected during the Three (3) Percent Ash Plant Run 33

Table 3-26: Particle Size Analysis of the Screenbowl Effluent Sample Collected during the One (1) Percent Ash Plant Run 33
Table 3-27: Particle Size Analysis of Low Ash (Blue Gem) Spiral Concentrate 35
Table 3-28: HHS Results for the Low-Ash Blue Gem Undersize Material (-0.25 mm) 35
Table 3-29: HHS Results for the Low-Ash Blue Gem Undersize Material Ground to D80=95 µm 36
Table 3-30: HHS Results for the Low-Ash Blue Gem Undersize Material Ground to D80=24 µm 36
Table 3-31: HHS Results for the Low-Ash Blue Gem Oversize Material Ground to D80=270 µm 36
Table 3-32: HHS Results for Low-Ash Blue Gem 37
Table 3-33: HHS Results for Low Ash Blue Gem Oversize Ground to D80=35 µm 37
Table 3-34: Results Obtained from HHS Processing of Ferroglobe’s Blue Gem Seam Coal (Plus 0.25 mm Sieve Oversize Ground to a Feed D80 Size of 4 Microns) 37
Table 3-35: Blue Gem Seam - TREE, LREE, HREE, and Individual REE Contents on a Whole Sample Basis as well as Heavy to Light REE Content Ratio in the Float-Sink Products 38
Table 3-36: Size Analysis of Mid-Ash Jellico Spiral Concentrate 38
Table 3-37: HHS Results for Mid-Ash Jellico Undersize 39
Table 3-38: HHS Results Mid-Ash Jellico Undersize Ground to D80=75 µm 39
Table 3-39: HHS Results for Mid-Ash Jellico Undersize Ground to D80=30 µm 39
Table 3-40: HHS Results for Mid-Ash Jellico Oversize Ground to D80=230 µm 40
Table 3-41: HHS Results for Mid-Ash Jellico Oversize Ground to D80=75 µm 40
Table 3-42: HHS Results for Mid-Ash Jellico Oversize Ground to D80=28 µm 40
Table 3-43: Results Obtained from HHS Processing of Arq As-Received Pond Fines (d80=67 um) 41
Table 3-44: Results Obtained from HHS Processing of Arq Micronized Pond Fines (D80=12 um) 42
Table 3-45: Results Obtained from HHS Processing of Arq Micronized Pond Fines (d80=8 um) with Multiple Sequential Stages of Cleaning and Recleaning 42
Table 3-46: Results Obtained from HHS Processing of Arq Pond Fines (D80=5 microns) with Multiple Stages of Cleaning and Sodium Silicate Dispersant (for Selected Tests) 43
Table 3-47: Results Obtained from HHS Processing of Micronized (D80=4 um) Clean Coal Agglomerates from an Initial Stage of HHS Processing on As-Received Arq Pond Fines 44
Table 3-48: Flotation Test Results for the Leatherwood Thickener Underflow 46
Table 3-49: HHS REE Recovery Test Results for Leatherwood Plant Thickener Underflow, Hazard 4 Seam (Test 1 - NaOL) 47
Table 3-50: HHS REE Recovery Test Result for Leatherwood Thickener Plant Underflow, Hazard 4 Seam (Test 2 - KOHX) 48
Table 3-51: HHS Test Results for Leatherwood Thickener Underflow (Test 3 – NaOL) 48
Table 3-52: HHS REE Recovery Test Results for Leatherwood Plant Thickener Underflow, Hazard 4 Seam (Test 4 – KOHX) 48
Table 3-53: Leatherwood Plant Thickener Underflow Sample HHS Decarbonization Results (Hazard 4 Seam) 49
Table 3-54: HHS REE Results of Blunged Leatherwood Thickener Underflow 51
Table 3-55: Leatherwood Thickener Underflow Coal Addition HHS REE Test “a” Results 53
Table 3-56: Leatherwood Thickener Underflow Coal Addition HHS REE Test “b” Results 54
Table 3-57: Leatherwood Thickener Underflow Coal Addition HHS REE Test “c” Results 56
Table 3-58: Pocahontas No. 3 Coal Seam Channel Sample C-1 59
Table 3-59: Ash Content and TREE Content Results of Unroasted and Roasting Products of Minus (-)0.18 mm Pocahontas No. 3 Coal Seam Middling Decarbonized Flotation Tailings 68

Table 3-60: Ash Content and REEs Composition Results of Roasting Products from Minus (-)10 micron Pocahontas No. 3 Coal Seam Middling Decarbonized Flotation Tailings 68
Table 3-61: Dry volatile, Fixed Carbon and Ash Contents of the Unroasted and Roasted Pocahontas No 3 Coal Seam Coarse Rejects Samples (Two hours of Roasting)................ 71
Table 3-62: Contents of Al, Fe, Ca, Mg (%) and Trace Elements (ppm) in the Pocahontas No. 3 Coal Seam Preparation Plant Samples Roasted at 600℃ for Two (2) Hours...................... 81
Table 3-63: Results of XRD Examination of the Roasted Fireclay (Hazard No. 4) Rejects Crystalline Structure... 104
Table 3-64: Elemental Analysis of the Final REO Generated from Pocahontas No. 3 Coal Seam Preparation Plant Middlings Reject Material 129
Table 3-65: Extraction Parameters (Based on REO Collected from W. Ky. No. 13 Coal Seam AMD) 137
Table 3-66: Extractant Dosage Utilized 140
Table 3-67: Log of the Distribution Coefficients of 5 Percent Depha 10 Percent TBP System (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)........ 148
Table 3-68: Separation Coefficients of 5 Percent Depha 10 Percent TBP System (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)....................... 149
Table 3-69: Linearization of the Distribution Coefficients of Table 3-67, Including R^2 fit (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)........ 149
Table 3-70: Summary of ID Numbers, Worksheet/tab Names and Unit Operation Descriptions......... 163
Table 3-71: Mass Rate Summary for the Overall Plant.. 165
Table 3-72: Mass Rate Summary for the Feed Preparation Circuit...................................... 166
Table 3-73: Mass Rate Summary for the Acid Leaching and Waste Treatment Circuit 167
Table 3-74: Mass Rate Summary for the Rougher Solvent Extraction Circuit 168
Table 3-75: Mass Rate Summary for the Cleaner Solvent Extraction Circuit...................................... 169
Table 3-76: Mass Rate Summary for the Scandium Solvent Extraction Circuit 170
Table 3-77: REESim Predicted Assay Values for the Overall Plant Operations.................................. 172
Table 3-78: REESim Distribution Values for the Overall Plant Adjusted for Pilot Plant Experience.. 173
Table 3-79: Assay Values of the Feed Preparation Circuit 174
Table 3-80: Distribution Values to the Feed Preparation Circuit 175
Table 3-81: Assay Values for the Acid Leaching and Waste Treatment Circuit.................................. 176
Table 3-82: Distribution Values for the Acid Leaching and Waste Treatment Circuit 177
Table 3-83: Assay Values for the Rougher Solvent Extraction Circuit.. 178
Table 3-84: Distribution Values for the Rougher Solvent Extraction Circuit 179
Table 3-85: Assay Values for the Cleaner Solvent Extraction Circuit 180
Table 3-86: Distribution Values for the Cleaner Solvent Extraction Circuit.. 181
Table 3-87: Assay Values for the Scandium Solvent Extraction Circuit.. 182
Table 3-88: Distribution Values for the Scandium Solvent Extraction Circuit 183
Table 3-89: Detailed Equipment List for Crushing and Grinding Operations (Cost Center 01)........... 185
Table 3-90: Detailed Equipment List for Roasting and Scrubbing Operations (Cost Center 02) 186
Table 3-91: Detailed Equipment List for Acid Leaching Operations (Cost Center 03) 187
Table 3-92: Detailed Equipment List for Solvent Extraction Operations (Cost Center 04).................. 188
Table 3-93: Detailed Equipment List for Washing and Tailings Operations (Cost Center 05)............. 188
Table 3-94: Detailed Equipment List for Water Treatment Operations (Cost Center 06)..................... 189
Table 3-95: Detailed Equipment List for Ancillary Operations (Cost Center 07)................................ 190

Table 3-96: Equipment Specifications for Crushing and Grinding (Cost Center 01)........................... 191
Table 3-97: Equipment Specifications for Roasting and Scrubbing (Cost Center 2)............................ 192
Table 3-98: Equipment Specifications for Acid Leaching (Cost Center 3).. 193
Table 3-99: Equipment Specifications for Solvent Extraction (Cost Center 04).................................. 194
Table 3-100: Equipment Specifications for Washing and Tailings (Cost Center 05) 194
Table 3-101: Equipment Specifications for Water Treatment (Cost Center 06) 195
Table 3-102: Equipment Specifications for Ancillary Operations (Cost Center 07).............................. 196
Table 3-103: Matrix Formulation for a Load, Scrub, Strip System each with 3 Stages as Shown in Figure 3-139.. 199
Table 3-104: Descriptions of the Proposed Pilot Plant Equipment 216
Table 3-105: Risk Analysis Related to Feedstock Material 217
Table 3-106: Risk Analysis Related to Crushing and Grinding Processes.. 217
Table 3-107: Risk Analysis Related to the Ore Roasting Process 218
Table 3-108: Risk Analysis Related to the Leaching Process 219
Table 3-109: Risk Analysis Related to the Roughing SX Process 220
Table 3-110: Risk Analysis Related to the Cleaning and Saponification SX Processes 221
Table 3-111: Risk Analysis Related to the Refining SX Process.. 221
Table 3-112: Risk Analysis Related to the Process Waste Handling 222
Table 3-113: Risk Analysis Related to the Regulatory Matters 222

FIGURES (IN THE TEXT)

Figure 3-1: Simplified Flowsheet for the Pocahontas No. 3 Seam Processing Facility........................... 3
Figure 3-2: TREE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant (Left Plot = whole-sample basis; Right Plot= ash-residue basis)... 5
Figure 3-3: HREE concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant (Left Plot= whole-sample basis; Right Plot = ash-residue basis).. 5
Figure 3-4: LREE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant (Left Plot= whole-sample basis; Right Plot = ash-residue basis).. 6
Figure 3-5: Ratios of Heavy-to-Light REEs versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant.. 7
Figure 3-6: Ratios of Critical-to-Noncritical REE versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant 7
Figure 3-7: TREE Concentrations versus Ash Contents for Coarse (>1 mm) Coal Samples only Collected from the Pocahontas No. 3 Coal Seam Preparation Plant..................................... 8
Figure 3-8: TREE Concentrations versus Ash Contents for Fine (<1 mm) Coal Samples Only Collected from the Pocahontas No. 3 Coal Seam Preparation Plant..................................... 8
Figure 3-9: Elemental Concentration versus Ash Content Plots for Sc, Y, La, Eu, Gd and Tb............. 10
Figure 3-10: Elemental Concentration versus Ash Content Plots for Ce, Pr, Nd, Dy, Ho and Eu 11
Figure 3-11: Elemental Concentration versus Ash Content Plots for Sm, Lu, Tm and Yb 12
Figure 3-12: Concentration of Thorium as a Function of Ash Content for the Pocahontas No. 3 Coal Seam Preparation Plant Samples 13
Figure 3-13: Correlation between TREE and Nd Concentrations for the Pocahontas No. 3 Coal Seam Preparation Plant Samples... 13
Figure 3-14: Measured versus Predicted TREE Concentrations Obtained Using Eq. [3.1] 14

Figure 3-15: (Left) Laboratory Cone Crusher, (Right) Laboratory Disk Grinder ... 16
Figure 3-16: (Left) Laboratory Ball Mill, (Right) Laboratory Attrition Mill ... 16
Figure 3-17: Pocahontas No. 3 Coal Seam Preparation Plant Low Ash Middlings (LAM) HHS Coal TREE Analysis ... 20
Figure 3-18: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Rougher Concentrate (A) ... 21
Figure 3-19: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Rougher Concentrate (B) ... 22
Figure 3-20: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Scavenger Concentrate ... 22
Figure 3-21: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Coal Product (A) ... 23
Figure 3-22: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Coal Product (B) ... 23
Figure 3-23: Pocahontas No. 3 Coal Seam Preparation Plant Individual REE Concentrations in DriJet™ X-Ray Sorter Products ... 25
Figure 3-24: Pocahontas No. 3 Coal Seam Preparation Plant X-Ray scans of LAM DriJet™ Sorter Testing ... 26
Figure 3-25: Pocahontas No. 3 Coal Seam Preparation Plant Handpicked Particles from LAM Sample ... 26
Figure 3-26: Pocahontas No. 3 Coal Seam Preparation Plant Individual Analysis of LAM Particles with Different Colors from X-Ray Scans ... 27
Figure 3-27: Testing Procedure for Cleaned Coal Spiral Concentrate Samples ... 34
Figure 3-28: Procedure Used to Process the Leatherwood Thickener Underflow Sample to Assess REE Recovery Potential Using a Rougher-Scavenger Froth Flotation Circuit ... 45
Figure 3-29: Procedure for Leatherwood Plant Thickener Underflow HHS REE Recovery, Hazard 4 Seam (Test 1 – NaOL Collector) ... 46
Figure 3-30: Procedure for Leatherwood Plant HHS REE Recovery, Hazard 4 Seam (Test 2 – KOHX Collector). ... 46
Figure 3-31: Procedure for Leatherwood Plant Thickener Underflow HHS REE Recovery, Hazard 4 Seam (Test 3 – NaOL Collector) ... 47
Figure 3-32: Procedure for Leatherwood Plant Thickener Underflow HHS REE Recovery, Hazard 4 Seam (Test 4 – KOHX Collector) ... 47
Figure 3-33: (Left) Blunging Impeller, (Right) Blunging Mixing Vessel ... 49
Figure 3-34: Blunged Sample in the HHS Procedure ... 50
Figure 3-35: TREE Whole Sample Basis Leatherwood Plant Thickener Underflow; Blunged HHS Feedstock Testing Results (Hazard 4 Seam) ... 51
Figure 3-36: TREE Ash Basis Leatherwood Plant Thickener Underflow; ... 52
Figure 3-37: TREE Whole Sample Basis Leatherwood Thickener Underflow Coal Addition Test “a” ... 53
Figure 3-38: TREE Ash Basis Leatherwood Thickener Underflow Coal Addition Test “a” ... 54
Figure 3-39: TREE Whole Sample Basis Leatherwood Thickener Underflow Coal Addition Test “b” ... 55
Figure 3-40: TREE Ash Leatherwood Thickener Underflow Coal Addition Test “b” ... 55
Figure 3-41: TREE Whole Basis Leatherwood Thickener Underflow Coal Addition Test “c” ... 56
Figure 3-42: TREE Ash Basis Leatherwood Thickener Underflow Coal Addition Test “c” ... 57
Figure 3-43: Apparatus used for the Acid Leaching Tests ... 58

Figure 3-44: TREEs Leaching Recovery versus Leaching Time of Unroasted Pocahontas No. 3 Coal Seam Channel Sample Stratigraphic Segments 60
Figure 3-45: REEs Leaching Recovery of the West Virginia Pocahontas No. 3 Coal Seam Channel Samples after Standard Acid Leaching for Five (5) Hours 61
Figure 3-46: Typical REE recovery after Five (5) hours Acid Leaching of Selected West Virginia Pocahontas No 3 Coal Seam Channel Samples 61
Figure 3-47: Sample Characterization Flowsheet for Coarse Rejects Samples from the Pocahontas No. 3 Coal Seam Preparation Plant 62
Figure 3-48: TREE Leaching Recovery versus Leaching Time of Float Products (Pocahontas No. 3 Coal Seam) 63
Figure 3-49: TREE Leaching Recovery versus Leaching Time of Sink Products (Pocahontas No. 3 Coal Seam) 64
Figure 3-50: REEs Leaching Recovery of Sink-float Products after Standard Acid Leaching for Five (5) Hours (Pocahontas No. 3 Coal Seam) 65
Figure 3-51: Sc Recovery after Five (5) hours Acid Leaching in Sink-float Products (Pocahontas No. 3 Coal Seam) 66
Figure 3-52: Y Recovery after Five (5) Hours Acid Leaching in Sink-float Products (Pocahontas No. 3 Coal Seam) 66
Figure 3-53: Ce Recovery after 5 hours Acid Leaching in Sink-float Products (Pocahontas No. 3 Coal Seam) 67
Figure 3-54: Nd Recovery after Five (5) hours Acid Leaching in Sink-float Products (Pocahontas No. 3 Coal Seam) 67
Figure 3-55: TREE Leaching Recovery versus Leaching Time of the Pocahontas No. 3 Coal Seam Unroasted and Roasted Decarbonize Flotation Tailings Samples 69
Figure 3-56: REEs Leaching Recovery of Pocahontas No. 3 Coal Seam Decarbonized Flotation Tailings and Roasted Samples after Standard Acid Leaching for five (5) Hours 70
Figure 3-57: Typical REE Recovery after Five (5) hours Acid Leaching in Selected Pocahontas No. 3 Coal Seam Decarbonized Flotation Tailings and Roasted Samples 71
Figure 3-58: Effects of Roasting Temperature on REE Recovery from the Pocahontas No. 3 Coal Seam Preparation Plant Coarse Rejects (2.2 Float, (-) 80-mesh) 73
Figure 3-59: Effects of Roasting Temperature on REE Recovery from the Pocahontas No. 3 Coal Seam Preparation Plant Middlings Product 75
Figure 3-60: Recovery of Individual REEs from the Coarse Rejects and Middlings Circuit Samples Which Were roasted at 600℃ for Two (2) Hours 76
Figure 3-61: Effect of Roasting Temperature and Acid-Type on Ce Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Float of Coarse Rejects, (a) and (b) at 75℃; and the Middlings (c) and (d) at 25℃ 77
Figure 3-62: Effect of Roasting Temperature and Acid-Type on Y Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Float of Coarse Rejects, (a) and (b) at 75℃; and the Middlings (c) and (d) at 25℃ 78
Figure 3-63: Effect of Roasting Temperature and Acid-Type on Dy Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Float of Coarse Rejects, (a) and (b) at 75℃; and the Middlings (c) and (d) at 25℃ 79
Figure 3-64: Effect of Roasting Temperature and Acid-Type on Sc Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Float of Coarse Rejects, (a) and (b) at 75^{O}C; and the Middlings (c) and (d) at 25^{O}C 80

Figure 3-65: Effect of Roasting Temperature and Acid-Type on Al Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 81
Figure 3-66: Effect of Roasting Temperature and Acid-Type on Fe Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 82
Figure 3-67: Effect of Roasting Temperature and Acid-Type on Ca Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 83
Figure 3-68: Effect of Roasting Temperature and Acid-Type on Mg Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 84
Figure 3-69: Effect of Roasting Temperature and Acid-Type on Cu Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG. Coarse Rejects Float Fraction and the Middlings Product 85
Figure 3-70: Effect of Roasting Temperature and Acid-Type on Co Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 86
Figure 3-71: Effect of Roasting Temperature and Acid-Type on Zn Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 87
Figure 3-72: Effect of Roasting Temperature and Acid-Type on V Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 88
Figure 3-73: Effect of Roasting Temperature and Acid-Type on P Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 89
Figure 3-74: REE Recovery as a Function of Time for both Pocahontas No. 3 Coal Seam Preparation Plant Coarse Rejects and Middlings Roasted Samples (750℃ for 2 hours) at 75℃ and 25℃ 92
Figure 3-75: Recovery of “Extractable REEs” as a Function of Time for both Pocahontas No. 3 Coal Seam Preparation Plant Coarse Rejects and Middlings Roasted Samples (750℃ for 2 hours) at 75℃ and 25℃ 93
Figure 3-76: Recovery of Major Elements Associated with REEs in the Roasted Coarse Rejects and Roasted Middlings Samples of the Pocahontas No. 3 Coal Seam Preparation Plant (750℃ for 2 hours) 95
Figure 3-77: Effects of Acid Concentration on the Leaching Recovery of REEs from Roasted Coarse Rejects from the Pocahontas No. 3 Coal Seam Preparation Plant 97
Figure 3-78: Effects of Acid Concentration on the Leaching Recovery of other Elements Associated with REEs in the Roasted Coarse Rejects Samples of the Pocahontas No. 3 Coal Seam Preparation Plant 98
Figure 3-79: Effects of Roasting on the TREE Recovery from (a) 1.8 SG Float, (b) 1.8-2.2 SG, and (c) 2.2 SG Sink Fraction of the Blue Gem Seam Coarse Reject Samples 100
Figure 3-80: Effects of Roasting on the LREE and HREE Recovery from Blue Gem Seam Rejects: (a, b) 1.8 SG Float; (c, d) 1.8-2.2 SG, and (e, f) 2.2 SG Sink Fractions 101

Figure 3-81: Effects of Roasting on Sc Recovery from (a) 1.8 SG Float, (b) 1.8-2.2 SG, and (c) 2.2 SG Sink Fractions of Blue Gem Coal Coarse Rejects 102
Figure 3-82: Total REE Recovery Achieved by Leaching the Unroasted Products Generated of the No. 76 Plant (Leatherwood) Thickener Underflow Material Samples............ 105
Figure 3-83: TREE Recovery Achieved by Leaching the Roasted Products Generated from the Processing of the Leatherwood Thickener Underflow Material 106
Figure 3-84: Effect of Solids and Acid Concentration on Leaching Recovery and Kinetics for the REEs Associated with the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material 107
Figure 3-85: Effect of Acid Concentration on Leaching Recovery and Kinetics for the REEs Associated with the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay Coarse Rejects Material (Solids Concentration = 20 percent by Weight) 108
Figure 3-86: Effect of Acid Concentration on Leaching Recovery and Kinetics of Fe for the Roasted 1.8 x 2.2 SG fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material (Solids Concentration = 200 gm/L) 109
Figure 3-87: Effect of Acid Concentration on Leaching Recovery and Kinetics of Al for the roasted 1.8 x 2.2 SG fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material; Solids Concentration = 200 gm/L............ 110
Figure 3-88: Effect of Acid Concentration on Leaching Recovery and Kinetics of Ca for the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material (Solids Concentration = 200 gm/L) 110
Figure 3-89: REE and Contaminant Element Recovery as a Function of Leach Time When Using an Acid Concentration of 1M H_2SO_4 and a Solids Concentration of 200 Grams per Liter of Solution 111
Figure 3-90: Effects of Percent Solids on Weak Acid Leaching of Fire Clay (Hazard No. 4) Material............ 112
Figure 3-91: Effect of Acid Concentration and Solids Concentration in Leachate Solution on Li and Co Extraction............ 113
Figure 3-92: Roasting Equipment Operated by Nex-Gen Industries 114
Figure 3-93: Roasted Products Generated by the Commercial Roasting of Fire Clay (Hazard No. 4) Seam Crushed Coarse Rejects Material 114
Figure 3-94: Leaching Characteristics of Fire Clay (Hazard No. 4) Seam Coarse Rejects that was not treated by the Commercial Roaster (Acid Concentration = 1M H_2SO_4) 115
Figure 3-95: Leaching Characteristics of Roasted ‘Gray’ Fire Clay (Hazard No. 4) Seam Crushed Coarse Rejects Treated by the Commercial Roaster (Acid Concentration = 1M H_2SO_4) 116
Figure 3-96: Leaching Characteristics of Roasted ‘Brown’ Fire Clay (Hazard No. 4) Seam Crushed Coarse Rejects Treated by the Commercial Roaster (Acid Concentration = 1M H_2SO_4) 116
Figure 3-97: Impact of D2EHPA Concentration on the Metal Ion Extraction at pH 2 using Solvent Extraction of an Artificial Mixture 117
Figure 3-98: Effect of Anionic Species on the Extraction of Metal from an Artificial Mixture............ 118
Figure 3-99: Lab Scale Solvent Extraction Setup for Conducting the Continuous SX Tests at the Hydrometallurgical Laboratory at the University of Kentucky 119
Figure 3-100: Column Flotation Setup used to Decarbonize the Various Coal Rejects Prior to Leaching............ 120

Figure 3-101: Leaching Reactor used in the Hydrometallurgy Lab to Leach the Solids from Various Coal Reject Sources 121
Figure 3-102: Laboratory Schematic for the Rougher Circuit as Implemented on Bench-Scale SX-Setup for Various Coal Rejects Sources 122
Figure 3-103: Bench-Scale Setup for Implementing the Rougher and the Cleaner Circuit of Solvent Extraction of Various Coal Rejects Sources 123
Figure 3-104: Recovery Values of Metal Ions in the Organic Phase for the Leach Liquor with Five (5) Percent DEHPA Solution (Pocahontas No. 3 Coal Seam Preparation Plant Reject Material) 124
Figure 3-105: Extraction Curves of Metal Ions in the Stripped Solution Obtained from the Rougher Stage of the Extraction Process (Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit Rejects) 125
Figure 3-106: Laboratory Schematic for the Cleaner Circuit as Implemented on Bench-Scale of the Pocahontas No. 3 Rejects Samples 126
Figure 3-107: Concentration of Contaminants and TREE in the SX Feed for the Rougher Stage of Extraction (Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit Rejects) 127
Figure 3-108: Concentration of contaminants and TREE in the Stripped Solution Obtained from the Rougher Stage of Extraction (Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit Rejects) 127
Figure 3-109: Concentration of Contaminants and TREE in the stripped Solution Obtained from the Cleaner Stage of Extraction (Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit Rejects) 128
Figure 3-110: Elemental Distribution of REEs in the Final REO Generated from the Pocahontas No. 3 Coal Seam Preparation Plant Middlings Reject Material 129
Figure 3-111: Impact of Different Dosing of Ascorbic Acid on Extraction Curves of Iron with Five (5) Percent DEHPA and 10 percent TBP (Artificial Solution) 130
Figure 3-112: Effect of Molarity of NaOH used for Saponification for the Recovery of Scandium (Sc) (Pocahontas No. 3 Coal Seam Middlings Rejects) 132
Figure 3-113: Distribution of REEs in Ferroglobe's Gatliff Plant (Blue Gem Seam) Feed versus Stripping Showing Selective Loading of Heavy REEs in the Organic Phase 133
Figure 3-114: Aqueous Rare Earth Element Concentration for Test Solutions (Using REO from W. Ky. No. 13 Coal Seam AMD 134
Figure 3-115: Extraction versus pH for Blended Extraction with Prepared Solution (Using REO from W. Ky. No. 13 Coal Seam AMD) 135
Figure 3-116: Extraction versus pH for Blended Extraction with Prepared Solution for High Concentration Rare Earth Elements (using REO from W. Ky. No. 13 Coal Seam AMD)135
Figure 3-117: Comparison of the UK extractant Blend to Cyanex 572 Manufacturer's Data (using REO from W. Ky. No. 13 Coal Seam AMD) 136
Figure 3-118: Mixing (Left) and Phase Separation (Right). 139
Figure 3-119: Extraction Curve Using Two (2) Percent DEHPA (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) 140
Figure 3-120: Concentration of Individual REE in Feed Samples used for the Test (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) 141
Figure 3-121: Extraction Curve using 2 Percent DEHPA and 10 Percent TBP (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) 142

Figure 3-122: Extraction Curve Using 2 Percent Cyanex (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) 143
Figure 3-123: Effect of Extractant Dosage on Percent Extraction for Cyanex (at pH 0.5) and DEHPA (at pH 0.47) (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)........ 144
Figure 3-124: Effect of Extractant Dosage on Percent Extraction for Cyanex (at pH 0.99) and DEHPA (at pH 1.001) (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)........ 144
Figure 3-125: Effect of Extractant Dosage on Percent Extraction for Cyanex (at pH 1.47) and DEHPA (at pH 1.542) (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)........ 145
Figure 3-126: Feed Composition of Sample Used for Distribution Isotherm (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) 146
Figure 3-127: Distribution Isotherm for Loading of Y and Dy Mixture (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) 147
Figure 3-128: Failure of Solvent Extraction in the Saponification Stage of a Mixed Coal-Related Source Organic from a Pilot Scale Circuit Showing the Formation of a Gel........ 150
Figure 3-129: Post Organic and Solution in a Separatory Funnel after Shakeout 151
Figure 3-130: Distribution of REEs and U+Th in the Various Saponification Stages (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)........ 152
Figure 3-131: Distribution of Iron in the Various Saponification Stages (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) 152
Figure 3-132: Process Flow Diagram for REE Extraction from Coal-based Sources 154
Figure 3-133: Detailed Process Flow Diagram for the Feedstock Preparation Circuit........ 155
Figure 3-134: Detailed Process Flow Diagram for the Acid Leaching and Waste Treatment Circuit........ 156
Figure 3-135: Detailed Process Flow Diagram for the Rougher Solvent Extraction Circuit........ 157
Figure 3-136: Detailed Process Flow Diagram for the Cleaner Solevent Extraction Circuit 159
Figure 3-137: Detailed Process Flow Diagram for the Scandium Solvent Extraction Circuit........ 160
Figure 3-138: Identification Numbers Assigned to Facilitate the Flow Diagram Simulation Work 162
Figure 3-139: A Solvent Extraction System Diagram Showing Loading, Scrubbing and Stripping Arranged so that part of the Scrubbing Stream is Refluxed into the Scrubbing Stages 197
Figure 3-140: Representation of a Single Stage........ 197
Figure 3-141: MATLAB App Showing the Calculation of SX Parameters 201
Figure 3-142: Previously Operated Pilot Plant Potentially Suited for Reutilization........ 202
Figure 3-143: Pilot Plant Original Flow Sheet........ 203
Figure 3-144: P&ID Diagram of the Crushing Segment of the Previously Used Pilot Plant 205
Figure 3-145: Previously Used Pilot Plant Crushing and Grinding P&ID Diagram (Repeated Diagram) 207
Figure 3-146: Previously Used Pilot Plant Roasting Circuit P&ID........ 208
Figure 3-147: Previously Used Pilot Plant Roasting Gas Scrubbing P&ID 209
Figure 3-148: Surveyed Map of the Truss Joist Facility........ 211
Figure 3-149: Location of Crushing and Roasting Equipment in Relation to the Truss Joist Facility 212

Figure 3-150: 3D View of the Proposed 1 tph Pilot Plant's Crushing and Grinding Circuit showing Jaw Crusher, Conveyors, Roller Crusher, Screens Ball Mill and Storage Bins 213
Figure 3-151: Alternate View of the Proposed 1 tph Pilot Plant's Crushing and Grinding Circuit........ 213
Figure 3-152: General Arrangement of the Proposed Relocated 1 tph Pilot Plant Crushing and Grinding Circuit 214
Figure 3-153: Additional Equipment Details of a Proposed Relocated 1 tph Pilot Plant Crushing and Grinding Circuit 215

APPENDIX

3-1 .... Flow Sheets
3-2 .... REESim
3-3 .... REESim Output

Chapter 4.0
Techno-Economic Analysis

TABLE OF CONTENTS

4.1 General Approach to the Techno-Economic Analysis 1
4.2 Technical Review 3
4.2.1 Summary of All Major Experimental Data, Engineering Analysis, Computations and Test Results (Fire Clay Seam (Hazard No. 4) Rejects) 3
4.2.2 Synopsis of Individual and Combined Capabilities of Unit Operations Regarding Yield, Purity, Throughput 6
4.2.3 Review of Scale-Up Procedures and Reliability of Estimation Procedures 6
4.2.4 Listing of All Process Assumptions, Design Deficiencies and Deleterious Elements that Could Have Impact on Production 8
4.2.4.1 Technical Assumptions Regarding Process Design 8
4.2.4.2 Target Marketable Oxides of Sc, Dy, Gd, and Mixed Rare Earths 8
4.2.4.3 Economic Assumptions 8
4.2.5 Fatal Flaw Analysis 10
4.3 Economic Evaluation 10
4.3.1 Tabular Summary of +10 percent to 15 percent CapEx for Pilot-Scale and Full-Scale Commercial Installation of Circuitry/Ancillary Operations 10
4.3.1.1 Pilot-Scale Installation 10
4.3.1.2 Commercial-Scale Installation 23
4.3.2 Tabular Listing of all Expected Operational and Maintenance Costs Including Labor, Electric, Reagents, and Other Consumables 26
4.3.2.1 Labor Costs 27
4.3.2.2 Miscellaneous Fixed Costs 27
4.3.2.3 Electric Power Costs 28
4.3.2.4 Reagent Costs 28
4.3.2.5 Other Direct Costs 29
4.3.2.6 Overhead Costs 29
4.3.2.7 Operating Cost Summary 29
4.3.3 Cost/Benefit Analysis that Highlights Annual Production Projections, Cash Flow Forecasts and Other Key Economic Performance Indicators (IRR, NPV, Payback) 30
4.3.4 Sensitivity Analysis on Pricing, Feedstock Quality, Product Purity, CapEx, Open, Transportation Costs, etc. 32
4.3.4.1 Analysis of Feed Rate 32
4.3.4.2 Analysis of Feed Grade and REE Price 33
4.3.4.3 Analysis of Technical Performance Measures 34

4.3.4.4 Potential Pathway to Profitability 35

4.4 Key Findings 36

4.5 Acknowledgments 37

4.6 Special Note and Addendum to the Techno-Economic Analysis 38

4.6.1 Justification of the 357 ppmw Feed Grade of the Economic Model 38

4.6.2 Influence of Optimal Roasting Conditions 39

TABLES (IN THE TEXT)

Table 4-1: Worksheets Descriptions for REE-Econ Software 2
Table 4-2: Listing of Process Modules Used in Techno-Economic Analysis 2
Table 4-3: Listing of Accounting Codes Used in Techno-Economic Analysis 3
Table 4-4: Feedstock REE Concentration (Nominal) Used in Techno-Economic Analysis 4
Table 4-5: REE to REO Conversion Factors Used in Techno-Economic Analysis 5
Table 4-6: Overall Element-by-Element Recovery Values Used in Techno-Economic Analysis 5
Table 4-7: Recovery, Purity, and Throughput Data for Individual Unit Operations 6
Table 4-8: Overall Process Yield to Final Product Streams Used in Techno-Economic Analysis 6
Table 4-9: Scale-Up Criteria used for Commercial-Scale Cost Analysis 7
Table 4-10: Used Equipment Costs for Equipment Components in Process Section 1: Crushing and Grinding 11
Table 4-11: Shipping and Refurbishment Costs for Process Section 1: Crushing and Grinding 12
Table 4-12: Used Equipment Costs for Equipment Components in Process Section 2: Roasting 13
Table 4-13: Shipping and Refurbishment Costs for Process Section 2: Roasting 14
Table 4-14: Used Equipment Costs for Equipment Components in Process Section 3: Leaching 15
Table 4-15: Shipping and Refurbishment Costs for Process Section 3: Leaching 16
Table 4-16: Used Equipment Costs for Equipment Components in Process Section 4: Wastewater Treatment 17
Table 4-17: Shipping and Refurbishment Costs for Process Section 4: Wastewater Treatment 18
Table 4-18: Used Equipment Costs for Equipment Components in Process Section 5: Leaching Ancillary 19
Table 4-19: Shipping and Refurbishment Costs for Process Section 5: Leaching Ancillary 19
Table 4-20: Used Equipment Costs for Components in Process Section 6: SX Ancillary 20
Table 4-21: Shipping and Refurbishment Costs for Process Section 6: SX Ancillary 21
Table 4-22: Combined Equipment Costs for all Process Sections 21
Table 4-23: Combined Shipping and Refurbishment Costs for all Process Sections 22
Table 4-24: Quoted Construction Costs Associated with Disassembly of Existing 22
Table 4-25: Total Capital Cost Estimate for 2,000 lb./hr. Pilot-Scale Plant 23
Table 4-26: Capital Cost Model Parameters Used in Techno-Economic Analysis 24
Table 4-27: Lang Factor Estimation for Commercial-Scale Installation Costs 24
Table 4-28: Itemized List of Capital Costs for Commercial-Scale Plant (500 st/hr.) 25
Table 4-29: Module-Level Capital Costs for Commercial-Scale Plant (500 st/hr.) 26
Table 4-30: Labor Cost Estimate for Commercial-Scale Plant (500 st/hr.) 27
Table 4-31: Cost Basis for Miscellaneous Fixed Costs 27
Table 4-32: Cost Summary for Miscellaneous Fixed Costs at Commercial-Scale (500 st/hr.) 27
Table 4-33: Electric Power Cost Summary for Commercial-Scale Plant (500 st/hr.) 28

Table 4-34: Unit Reagent Costs....28
Table 4-35: Reagent Cost Summary for Commercial-Scale Plant (500 st/hr.)....29
Table 4-36: Waste Disposal Cost Summary for Commercial-Scale Plant (500 st/hr.)....29
Table 4-37: Operating Cost Summary by Process Module for Commercial-Scale Plant (500 st/hr.)....30
Table 4-38: Operating Cost Summary by Accounting Code for Commercial-Scale Plant (500 st/hr.)....30
Table 4-39: REE Price Deck Used for Commercial-Scale Economic Analysis....31
Table 4-40: Summary of Economic Indicators for Commercial-Scale Plant (500 st/hr.)....32
Table 4-41: Economic Indicators for Commercial Plant under Improved Input Conditions....36
Table 4-42: REE (ppmw) Data Source Inventory (Fire Clay Seam)....38
Table 4-43: Project Economic Summary for Material Processed under Differing Roasting and Acid Application Conditions....40

FIGURES (IN THE TEXT)

Figure 4-1: Sensitivity of Plant Capital Cost with Respect to Plant Feed Rate....33
Figure 4-2: Sensitivity of Plant Feed Grade and REE Price Multiple with Respect to Project NPV....33
Figure 4-3: Tornado Analysis Showing the Sensitivity of Plant Operating Cost with respect to Several Technical Performance Measures....34
Figure 4-4: Sensitivity of Reagent Consumption with Respect to this Project NPV....35

Chapter 8.0
Benchmark Process Audits

TABLE OF CONTENTS

8.1 Introduction **1**
8.2 Benchmark Visit Sites **4**
8.2.1 MP Materials, Mountain Pass, California 4
8.2.2 Previously Operated Pilot Plant (Trip 1) 10
8.2.2.1 Introduction **10**
8.2.2.2 Process Description **11**
8.2.2.3 Process and Construction Drawings **12**
8.2.2.4 Equipment and Condition **25**
8.2.3 Previously Operated Pilot Plant (Trip No. 2) 31
8.2.3.1 Appraisal of the Arizona Pilot Plant **31**
8.2.3.2 Salvage Equipment Power and Original Cost **31**
8.3 Acknowledgments **35**

TABLES (IN THE TEXT)

Table 8-1: Summary of Benchmark Process Audits 4
Table 8-2: List of Original P&ID Diagrams for the Previously Operated Plant 23
Table 8-3: Subsequent Drawings Indicating Modifications to the Roasting Gas Scrubbing and Burner Systems of the Previously Operated Pilot Plant 23
Table 8-4: General Layouts and Elevations of the Crushing/Grinding and Roasting Sections of the Previously Operated Pilot Plan 23
Table 8-5: Electrical and Control Drawings of the Previously Operated Pilot Plant 24
Table 8-6: Steel and Foundation Drawings of the Previously Operated Pilot Plant 24
Table 8-7: Roaster-Specific Drawings of the Previously Operated Pilot Plant 25
Table 8-8: Appraised Value, Effective February 12, 2019, of the Previously Operated Pilot Plant 31
Table 8-9: List of Proposed Previously Operated Pilot Plant Salvaged Equipment Summary Data showing Description, Equipment Identification., Referenced Drawing, Power, Original Cost and Weight 32

FIGURES (IN THE TEXT)

Figure 8-1: General Area Map Depicting the Disposition of the MP Materials' MP Mining Operations, LLC Site at Mountain Pass, California 2
Figure 8-2: General Location Map of the Previously Operated Nickel-laterite Ore Hydro-met Pilot Plant near Tucson, Arizona 3
Figure 8-3: Previously Operated Nickel-laterite Ore Hydro-met Pilot Plant near Tucson, Arizona 4
Figure 8-4: Photograph of Mountain Pass Ore 6
Figure 8-5: Ball Mill at Mountain Pass 7
Figure 8-6: Historic Solvent Extraction Train 8
Figure 8-7: Alternate Solvent Extraction Vessel Design 8
Figure 8-8: Newest Solvent Extraction Train 9

Figure 8-9: Overlook of Separations and Chemical Facility 9
Figure 8-10: Previously Operated Pilot Plant - High Level Flow Sheet 11
Figure 8-11: Previously Operated Pilot Plant Drawing 3166-EF-11. Crushing and Drying 14
Figure 8-12: Previously Operated Pilot Plant Drawing 3166-EF-21 - Roaster 15
Figure 8-13: Previously Operated Pilot Plant Drawing 3166-EF-22 - 2-Stage Roaster Scrubbing System 16
Figure 8-14: Previously Operated Pilot Plant Drawing 25801-PI-1 - Modified Dry Scrubbing System 17
Figure 8-15: Previously Operated Pilot Plant Drawing 3166-EF-31 - Leaching Circuit (Section 3) 18
Figure 8-16: Previously Operated Pilot Plant Drawing 3166-EF-41 - Counter Current Decantation (Section 4) 19
Figure 8-17: Previously Operated Pilot Plant Drawing 3166-EF-51 - Counter Current Decantation (Section 4) 20
Figure 8-18: Previously Operated Pilot Plant Drawing 3166-EF-61 - Solvent Extraction Pre-Treatment 21
Figure 8-19: Previously Operated Pilot Plant Drawing 3166-EF-71 - Solvent Extraction and Electrowinning of Ni 22
Figure 8-20: Aerial View of Jaw Crusher and Supporting Conveyors of the Previously Operated Pilot Plant 26
Figure 8-21: Rotary Kiln Dryer Showing Baghouse and Conveyors of the Previously Operated Pilot Plant 26
Figure 8-22: Screen and Roller Crusher with Supporting Conveyors of the Previously Operated Pilot Plant 27
Figure 8-23: Panorama Showing the Top of Roaster, Gas Scrubbing, and Ore Bins of the Previously Operated Pilot Plant 27
Figure 8-24: Top Shell of the Roaster of the Previously Operated Pilot Plant Showing Corrosion due to HCl 28
Figure 8-25: Alternate View of the Roaster Showing Shell Damage. 28
Figure 8-26: Interior View of Roaster of the Previously Operated Pilot Plant Showing Example of Serviceable Rakes 29
Figure 8-27: Lower Hearth Gear and Conditions of the Previously Operated Pilot Plant 29
Figure 8-28: Picture Showing Gas Scrubbing Equipment of the Previously Operated Pilot Plant 30
Figure 8-29: Aerial View of Gas Scrubbing Equipment and Portions of the Leaching Circuit of the Previously Operated Pilot Plant 31

APPENDIX

8-1 .... DARCO Appraisal

Chapter 1.0

PHASE 1 EXECUTIVE SUMMARY REPORT

Chapter 1.0
Phase 1 Executive Summary Report

TABLE OF CONTENTS

1.1 Executive Summary 1
1.1.1 Introduction 1
1.1.2 Feedstock Sampling, Identification, and Characterization 2
1.1.3 System Design Preparation 3
1.1.3.1 Summary Observations of the Lower Kittanning Seam Plant Rejects Prospect 4
1.1.3.2 Summary Observations of the Investigation of the Pocahontas No. 3 Seam Plant Rejects Prospect 4
1.1.3.2.1 Pocahontas No. 3 Seam Plant Circuit Samples TREE Content Results 4
1.1.3.2.2 Pocahontas No. 3 Seam Preparation Plant Circuit Samples Heavy and Light REE Concentrations 5
1.1.3.3 Summary Observations of the HHS Application to the Ferroglobe Blue Gem Seam and Jellico Coal Seam Rejects Samples 7
1.1.3.3.1 Effects of Roasting on TREE Recovery from Three Float-sink Products of Rejects from Ferroglobe's Gatliff Plant (Blue Gem Seam) 7
1.1.3.4 Arq Corbin USA (Arq) – Summary Observations of HHS Treatment of Rejects Impoundment Samples 9
1.1.3.5 Summary Observations of the Blue Diamond Coal Company (Operated by Blackhawk) Fire Clay (Hazard No. 4) Seam Examinations for REE Concentration Purposes. 9
1.1.3.5.1 No. 76 Plant (Leatherwood) Fire Clay Seam (Hazard No. 4) Thickener Underflow Acid Leaching Testing Methodology 10
1.1.3.6 Solvent Extraction Testing on a Variety of Feedstocks 12
1.1.3.7 Detailed Flow Diagram 14
1.1.3.8 Flowsheet Simulation to Evaluate Process Circuitry for REE Concentration from Coal-based Sources 18
1.1.3.8.1 Flowsheet Development Software Selection 18
1.1.3.8.2 Evaluation of LIMN 18
1.1.3.8.3 Evaluation of REESim 18
1.1.3.9 Summary Discussion of the Simulation Results for Project MMA 29956 18
1.1.3.10 Availability of a 1 Short Ton per Hour Pilot Plant 20
1.1.3.10.1 Pilot Plant Site Proposed for the 1 Short Ton per Hour REM Concentration Plant 21
1.1.3.11 Risk and Fatal Flaw Analysis 23
1.1.3.11.1 Summary of Significant Concerns 23
1.1.3.11.2 Timing of Regulatory Permits, Agreements or Contracts 23
1.1.3.11.3 Performance of the Proposed Solvent Extraction Process 23
1.1.3.11.4 Roaster Performance with a Coal Preparation Plant Rejects Feedstocks 23
1.1.4 Techno-Economic Analyses 23
1.1.4.1 Summary of All Major Experimental Data, Engineering Analysis, Computations and Test Results (Fire Clay Seam (Hazard No. 4) Rejects. 24

1.1.4.2 Economic Assumptions....26
1.1.4.3 Economic Indicators for a Commercial-Scale Plant (500 Short Tons per Hour)....27
1.1.4.4 Influence of Optimal Roasting Conditions....28
1.1.4.5 Key Findings....29
1.1.4.6 Potential Pathway to Profitability....30
1.1.5 Permitting and Environmental Management for Phase 2.... 31
1.1.5.1 Sources of Run-of-Mine Fire Clay (Hazard No. 4) Coal from Which the Phase 1 Rejects Was Extracted....32
1.1.5.1.1 Mine No. 81 (Calvary Mine 81)....32
1.1.5.1.2 Mine No. 89 (Orchard Branch Mine 89)....32
1.1.5.2 Source of Rejects Derived from Processing the Fire Clay (Hazard No. 4) Seam During Phase 1....32
1.1.5.2.1 Blue Diamond Coal Company No. 76 Plant (Operated by Blackhawk Mining 32
1.1.5.3 Financial Viability of Blackhawk....32
1.1.5.4 Lessor/Lessee Relationships....33
1.1.5.5 Pilot Plant Permitting Plan for Phase 2....33
1.1.5.5.1 Likely Coal Mining Related Permit Obligations....33
1.1.5.5.2 Wastewater from the Pilot Plant Processing System....34
1.1.5.5.3 Solid Waste Generated by the Pilot Plant Processing System....34
1.1.5.6 Environmental Management Plan for the Pilot Plant Operations....34
1.1.5.6.1 Process Chemical Toxicity....35
1.1.5.6.2 Solid and Liquid Process Waste Streams....35
1.1.5.6.3 Air Quality Control Measures....35
1.1.5.6.4 Surface and Groundwater protection measures....35
1.1.5.6.5 Spill Prevention Control and Countermeasure Plan (SPCC)....35
1.1.5.6.6 Stormwater Pollution Prevention Plan and Stormwater Management Plan 36
1.1.5.7 Best Management Practices (BMPs)....36
1.1.5.7.1 Operational Source Control BMP....36
1.1.5.7.2 Structural Source Control BMPs....36
1.1.5.7.3 Treatment BMPs....36
1.1.5.7.4 Erosion and Sediment Control BMPs....37
1.1.5.8 Process Water Treatment....37
1.1.5.9 Environmental Monitoring....37
1.1.5.10 Environmental Management Plan Components for Phase 2....37
1.1.6 Pilot Plant Product Market Assessment and Purchase Agreement Sample.... 38
1.1.6.1 General Discussion of REE Demand Structure....38
1.1.6.2 General Discussion of the Supply Structure....38
1.1.6.3 Specific Discussion of the Market for the Pilot Plant Products....39
1.1.6.3.1 Gadolinium (Gd) Market Environment....39
1.1.6.3.2 Dysprosium (Dy) and Industrial Magnet Production....39
1.1.6.3.3 Yttrium (Y) Market Trends....39
1.1.6.4 REE Price Structure....40
1.1.6.5 Critical Factors Affecting the Rare Earth Market....40
1.1.6.6 Pilot Plant Products Sales Agreement Development....41

1.1.6.6.1 Establishment of Marketing/Tolling Agreements for Sales of REE Products 41
1.1.6.6.2 Informal MMA 29956 Pilot Plant Product Off-Take Arrangements 41
1.1.7 Environmental Critique of the Proposed 1 Short Ton per Hour Pilot Plant 41
1.1.7.1 List of Major Sources of Environmental Risks 41
1.1.7.2 Controls/Mitigation Strategies Proposed for each Item of Concern 41
1.1.7.2.1 Modification of Feed Materials 41
1.1.7.2.2 Concentration of Radioactive Materials 42
1.1.7.2.3 Release of Reagents 42
1.1.7.2.4 Elimination of Water Quality Impacts from Process-Water 42
1.1.7.2.5 Elimination of Air Quality Impact 42
1.1.7.3 Environmental Assessment/Environmental Impact Statement (EA/EIS) Response/Mitigation 43
1.1.8 Benchmark Process Audits 43
1.1.8.1 Deferred Audit of a USA-Based Toll Processor 44
1.1.8.2 MP Materials, Mountain Pass, California (Pending REE Producer and Feedstock Provider) 44
1.1.8.3 Arizona Pilot Plant (Hydro- Met Facility, Tucson, Arizona, Potential Feedstock Producer) 45
1.2 Acknowledgments 46
1.2.1 Principal Investigators 46
1.2.2 Participating and/or Supportive Entities 47

TABLES (IN THE TEXT)

Table 1-1: Feedstock Source Evaluation Sequence 2
Table 1-2: Basic Summary of In-Place Non-Coal Rock and REE+Y (REY) Short Tons 3
Table 1-3: Preparation Plant, Lower Kittanning Seam, REE Concentration of Tails of Plant Circuit Samples after HHS 4
Table 1-4: Elemental Analysis of the Final REO Generated from Pocahontas No. 3 Seam Middlings Reject Material 13
Table 1-5: REESim Predicted Assay Values for the Overall Plant Operations 19
Table 1-6: REESim Distribution Values for the Overall Plant Adjusted for Pilot Plant Experience 20
Table 1-7: Feedstock REE Concentration (Nominal) Used in Techno-Economic Analysis 24
Table 1-8: REE to REO Conversion Factors Used in Techno-Economic Analysis 25
Table 1-9: Overall Element-by-Element Recovery Values Used in Techno-Economic Analysis 25
Table 1-10: Summary of Economic Indicators for Commercial-Scale Plant (500 st/hr.) 28
Table 1-11: Project Economic Summary for Material Processed under Differing Roasting and Acid Application Conditions 29
Table 1-12: Economic Indicators for Commercial Plant under Improved Input Conditions 31

FIGURES (IN THE TEXT)

Figure 1-1: REE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 seam 5
Figure 1-2: HREE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Seam Preparation Plant 6
Figure 1-3: LREE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Seam Preparation Plant 6
Figure 1-4: Effects of Roasting on the TREE Recovery from (a) 1.8 SG Float, (b) 1.8-2.2 SG, and (c) 2.2 SG Sink Fraction of the Blue Gem Seam Coarse Reject Samples........ 8
Figure 1-5: Total REE Recovery Achieved by Leaching the Unroasted Products Generated of the No. 76 Plant (Leatherwood) Thickener Underflow Material Samples 11
Figure 1-6: TREE Recovery Achieved by Leaching the Roasted Products Generated from the Processing of the Leatherwood Thickener Underflow Material 12
Figure 1-7: Process Flow Diagram for REE Extraction from Coal-based Sources 15
Figure 1-8: Detailed Process Flow Diagram for the Feedstock Preparation Circuit........ 15
Figure 1-9: Detailed Process Flow Diagram for the Acid Leaching and Waste Treatment Circuit 16
Figure 1-10: Detailed Process Flow Diagram for the Rougher Solvent Extraction Circuit........ 16
Figure 1-11: Detailed Process Flow Diagram for the Cleaner Solevent Extraction Circuit 17
Figure 1-12: Detailed Process Flow Diagram for the Scandium Solvent Extraction Circuit 17
Figure 1-13: Previously Operated Pilot Plant Potentially Suited for Reutilization 21
Figure 1-14: Surveyed Map of the Truss Joist Facility........ 22

BIOGRAPHIES OF WRITERS

1.1 Executive Summary

1.1.1 Introduction

The entirety of the report that follows this brief Executive Summary Chapter is the product of a Funding Announcement by the **US Department of Energy *(DOE)***, Office of Fossil Energy, originally issued 09/28/2016, otherwise known as Funding Opportunity Number: DE-FOA-0001627, under the title of "Production of Rare Earth Elements from Domestic U.S. Coal and Coal By-Products" The solicitation was subsequently amended several times ending in amendment 000005. Ultimately a consortium designated "Minerals Refining and Rare Earth Consortium" (to which many of the remaining participants recited in this report joined) filed a proposal on December 8, 2016. On June 9, 2017, the contracting office of the US DOE, **National Energy Technology Laboratory *(NETL)***, alerted Dr. Keim, the Authorized Representative of the December 08, 2016, proposal, that the consortium proposal had been selected.

In summary, this report presents the results of the efforts in meeting the stated project objectives as outlined in DE-FOA-0001627 to wit:

"The main objective of the proposed work is to design and construct a demonstration plant that can extract rare earth elements (REEs) from domestic U.S. coal and coal by-products. The plant will produce salable rare earth elements and intermediates, including at a minimum three individual REEs (preferably heavy rare earths) at 90-99% purities, with output capacities in excess of 10 pounds per day of composite rare earth products (reported as oxides). In addition, the plant will produce high value premium solid fuels, magnetite and other materials to increase the economic viability of the plant and to create jobs. If successful, the proposed work will help develop a domestic supply chain of the critical materials that are essential for the development of renewable energy resources, high-tech industries and a strong national defense."

The principal investigators immediately launched the tasks outlined in the Statement of Project Objectives *(SOPO)* in accordance to its Project Management Plan *(PMP)* based on their collective knowledge of the subject matter and the expectations generated by preliminary laboratory results. Events and circumstances related in part to the vagaries of the coal mining, processing, and marketing industry triggered a series of carefully deliberated step-wise transitions from the original Lower Kittanning coal source in West Virginia to an ultimate stopping point with the Fire Clay Seam (Hazard No. 4) rejects from **Blue Diamond Coal Company** in Kentucky (Operated by **Blackhawk Mining LLC**). These transitional steps are discussed in substantial detail in the body of this report. Notably, substantial data of potential future significance, both in terms of sample collection, processing, and testing, were collected in the process. Unfortunately, as reported in the Techno-Economic Chapter of this report, none of these efforts resulted in an anticipated economically viable venture. Thus, the principals associated with this project elected not to transition into a "Pilot Plant" operation. A proposal for funding that pilot plant was therefore not requested.

The outcome of this initiative under DE-FOA-0001627 notwithstanding, the following portions of this Executive Summary outline the activities of the investigators. That said, the reader is encouraged to examine the main body of this report in which each chapter is assembled as a free-standing product, complete with Table of Contents, for practical and easy reference purposes.

1.1.2 Feedstock Sampling, Identification, and Characterization

The investigations leading up to the discussion of the Fire Clay Seam in this report are summarized according to historical order in the following table.

Table 1-1: Feedstock Source Evaluation Sequence

Sequence	Company	Facility Name	Location			MSHA ID	Coal Seam	Comments
			Geographical Description	North Coordinates	West Coordinates			
1	Northern Appalachian Coal Producer	Mine	West Virginia	39° 20'N	79° 59'W	Multiple	Lower Kittanning	The initial collaborative effort could not be supported by the coal mining and processing company.
		Processing Plant	West Virginia	39° 20'N	79° 59'W	Multiple		
2	Southern Appalachian Coal Producer	Mine	West Virginia	37° 44'N	81° 14'W	Multiple	Pocahontas No. 3	After testing rejects and channel sample response to HHS cleaning and REE content, this initiative was terminated in light of the fact that the middlings material could not be processed to produce a clean coal product with desired quality.
		Processing Plant	West Virginia	37° 44'N	81° 14'W	Multiple		
3	Ferroglobe (d.b.a. Alden Resources, LLC)	Gatliff Tipple	Gatliff, Whitley County, KY	36° 40.698'N	84° 1.051'W	15-09938	Blue Gem and Jellico	Multiple segments of the plant circuit were sampled and analyzed for standard coal quality parameters, response to HHS cleaning and REE content. This initiative was terminated because of relatively low REE content in the samples tested.
		Mine #3, Bain Branch	Bryants Store, Knox County, KY	36° 46.748'N	83° 53.239'W	15-17691	Blue Gem	
		Mine #5, Log Cabin	Bryants Store, Knox County, KY	36° 46.684'N	83° 54.824'W	15-18426	Blue Gem	
		Contractors	Not Available	Not Available	Not Available	N.A.	Jellico	
4	Arq Corbin, LLC (d.b.a. Arq Limited)	Corbin Project, LLC	Corbin, Knox County, KY	36° 56.048'N	84° 5.110'W	15-02134	Mixed Coal Preparation Plant Rejects	A series of HHS studies were conducted in small grab samples of the rejects material the initiative was dropped when collaboration with the host company was not successful
5	Blue Diamond Coal Company and Blue Diamond Mining LLC, (d.b.a. Blackhawk Mining, LLC)	Blue Diamond No. 76 Plant (a.k.a. Leatherwood)	Slemp, Perry County, KY	37° 03.967'N	83° 7.233'W	15-16520	Fire Clay	Participation by both Kentucky River Properties LLC (Lessor to Blue Diamond) as well as the Plant and mine operator (d.b.a. Blackhawk Mining LLC) resulted in an aggressive campaign of drilling, channel sampling, plant rejects sampling and testing.
		Blue Diamond Mine No. 81	Smilax, Leslie County, KY	37° 8.152'N	83° 14.497'W	15-12753		
		Blue Diamond Mine No. 89	Viper, Perry County, KY	37° 10.347'N	83° 6.683'W	15-19405		

Chapter 2 details the modeling methods and tonnages of in-place non-coal reject material from the Fire Clay (Hazard No. 4) coal seam on a sizeable coal-bearing property (the "study area") owned primarily by **Kentucky River Properties, LLC (*KRP*)** and operated by **Blue Diamond Coal Company** (through **Blackhawk Mining, LLC (*Blackhawk*))** in Leslie County, Kentucky. The purpose of Chapter 2 is to classify and quantify the in-place coarse reject material from coal-mining activities which *may* yield extractable and economic quantities of rare earth elements (*REEs*).

The Property has three areas of interest:

1. Mine projections from the active Mine No. 89;
2. Mine projections from the active Mine No. 81; and
3. A future mining area, also known throughout this report as the "study area."

The "study area" does not yet have specific mine projections, but the coal has been modeled and has some reject areas defined. The presumed coarse reject material is comprised of several stratigraphic and lithologic intervals:

1. Roof material;
2. Partings between the Hazard No. 4 coal seam and the regionally persistent Flint Clay;
3. The Flint Clay;
4. Partings between the Flint Clay and the Hazard No. 4 Jack Rock coal horizon (where present); and
5. Floor material.

Rock tonnages for each of these individual horizons have been estimated and converted to tons of REEs.

The following table summarizes the basic findings of this geologic study.

Table 1-2: Basic Summary of In-Place Non-Coal Rock and REE+Y (REY) Short Tons

Location	In-Place Non-Coal Rock Tons	Sc	Y	Gd	Dy	Total REY Tons
		(Tons In-Place)				(Whole Sample Basis)
Total Mine No. 81 (Measured + Indicated + Inferred)	2,338,000	40	73	24	18	832
Total Mine No 89 (Measured + Indicated)	491,000	8	15	6	4	150
Study Area, Subtotal KRP Controlled (Measured)	45,699,000	643	1,381	489	363	15,520
Study Area Subtotal Non KRP (Measured)	13,196,000	186	396	140	104	4,459
Study Area - Total All Holes & Ownership (Indicated)	56,619,000	791	1,733	611	458	19,408
Study Area - All Tonnages and Ownership (Measured + Indicated)	**115,513,000**	**1,620**	**3,510**	**1,240**	**926**	**39,387**
All Areas - All Tonnages - All Ownerships	**118,343,000**	**1,667**	**3,598**	**1,270**	**947**	**40,370**

The reader should be aware, that the preceding table presents a snapshot of the estimated in-situ quantity of REE within the "study area". The table was made to conform to the target marketable products recited in the "Techno-Economic Analysis" portion of this report. Thus, upon receipt of an appropriate request, a multi-tabbed Excel workbook can be provided to the reader. The workbook presents a significantly more detailed examination of all the REE elements identified for the subject "study area".

1.1.3 System Design Preparation

The following describes the evolution of the search and investigation for one or more reliable sources of long-term supplies of coal and coal byproducts that exhibit the appropriate REE content to support a future rare earth oxide/rare earth mineral *(REO/REM)* concentration enterprise. The description includes a system design and process simulation. The results were used to execute a techno-economic analysis of the selected REO/REM concentration process.

1.1.3.1 Summary Observations of the Lower Kittanning Seam Plant Rejects Prospect

The initial coal rejects prospect in West Virginia, controlled a large resource/reserve of both steam and metallurgical quality coal in the Lower Kittanning Seam.

The Lower Kittanning coal bed and the associated mining complex played a key role in the compilation of a proposal to DOE/NETL in December 2016 (FOA-0001627) that was subsequently funded as Project MMA 29956. The selection of this mining complex and its participation in the proposal pursuant to FOA-0001627 was based on prior hydrophobic-hydrophilic separation *(HHS)*[1] and REE work on products extracted from the Lower Kittanning rejects circuits of the operator's coal handling and preparation plant by the **University of Kentucky *(UK)*, *Virginia Tech (VT)***, and **Minerals Refining Company**[2] ***(MRC)***. A sample of the results of that work is presented in the following table.

Table 1-3: Preparation Plant, Lower Kittanning Seam,
REE Concentration of Tails of Plant Circuit Samples after HHS

Sample Site	HHS Product	Product Ash (%)	REE (ppm)	
			Ash Basis	Whole-Basis
6-in Cyclone O/F	Tails	89.50	385.72	345.80
Thickener U/F	Tails	90.32	376.75	340.28
Spiral Rejects	Tails	89.19	351.67	313.66
Spiral Middlings	Tails	90.31	341.04	307.99
Reflux (Off Sieve)	Tails	88.90	252.84	224.77

Source: Dr. Yoon, Ph.D., Virginia Tech

However, events at the coal mine and preparation plant that were beyond the control of the project investigators required additional investigations to identify alternative feedstock sites.

1.1.3.2 Summary Observations of the Investigation of the Pocahontas No. 3 Seam Plant Rejects Prospect

1.1.3.2.1 Pocahontas No. 3 Seam Plant Circuit Samples TREE Content Results

The following figures provide plots of the total rare earth element (*TREE*) concentration versus dry ash content for each sample analyzed from the preparation plant. For convenience, the TREE values have been plotted both as a parts-per-million *(ppm)* concentration on a whole-sample basis (*ppmw*) and on an ash-residue basis (*ppma*). The plotted data points are interesting in that it appears that two grouping of data points can be identified in each plot. For the whole-sample data listed in the left side of the graph, the two groupings of data can be largely represented using two straight lines. The lower line falls roughly in line with trends observed for many other coal feedstocks in that the high ash (i.e., 100 percent ash) falls in the

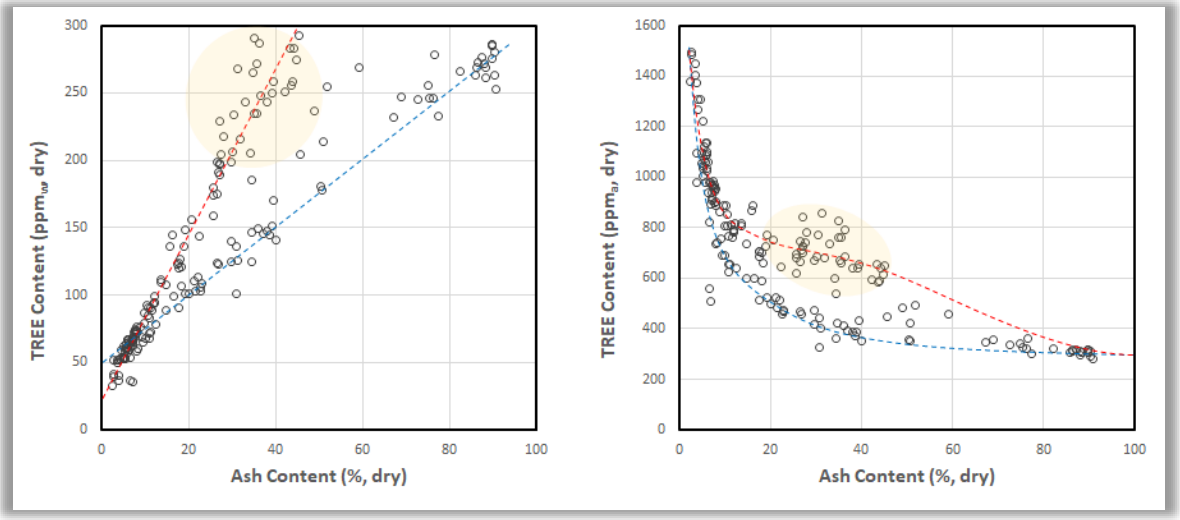
---

[1] The patented Hydrophobic-Hydrophilic Separation (*HHS*) process was developed at Virginia Tech by Dr. Roe-Hoan Yoon and research partner Dr. Gerald Luttrell to capture the coal fines and reduce the waste

[2] MRC Massey Building, 5002 Monument Avenue, Richmond, Virginia 23230. MRC has developed a patented HHS technology that recovers micron-sized particles of coal that are currently disposed of as part of a coal preparation plant waste. The tailings of that HHS process provide access to REE compounds with a significantly reduced interference from carbonaceous particles, thus, the interest to project MMA 29956. The market value of the extremely low ash metallurgical grade coal product captured by HHS can be used to offset the cost of REE concentration.

range of 250-350 ppmw. In contrast, the upper line reaches this TREE concentration value at a relatively low ash content of approximately 30-45 percent ash. Samples falling in this range, which is shaded in the graphs, represent splits of coal/rock that are elevated in TREEs. The enhanced concentration can also be observed in the ash-basis concentration plot shown in the right side of the graphs. In this case, numerous samples falling in the upper grouping have ash-based TREE concentrations in the 600 to 800 ppma range with corresponding ash contents of 20-45 percent. The TREE concentrations for the lower grouping fall in the range of 400 ppma over the same range of ash values.

Figure 1-1: REE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 seam
(Left Plot = whole-sample basis; Right Plot= ash-residue basis)



1.1.3.2.2 Pocahontas No. 3 Seam Preparation Plant Circuit Samples Heavy and Light REE Concentrations

Further examination of the REE concentration data from this particular preparation plant shows that the twin groupings of REEs occur both for the heavy rare earth elements (*HREEs*) and the light rare earth elements (*LREEs*). These trends, which are illustrated in the following two graphs, indicate that this particular coal contains splits that are elevated in both heavy and light REEs.

Figure 1-2: HREE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Seam Preparation Plant
(Left Plot= whole-sample basis; Right Plot = ash-residue basis)

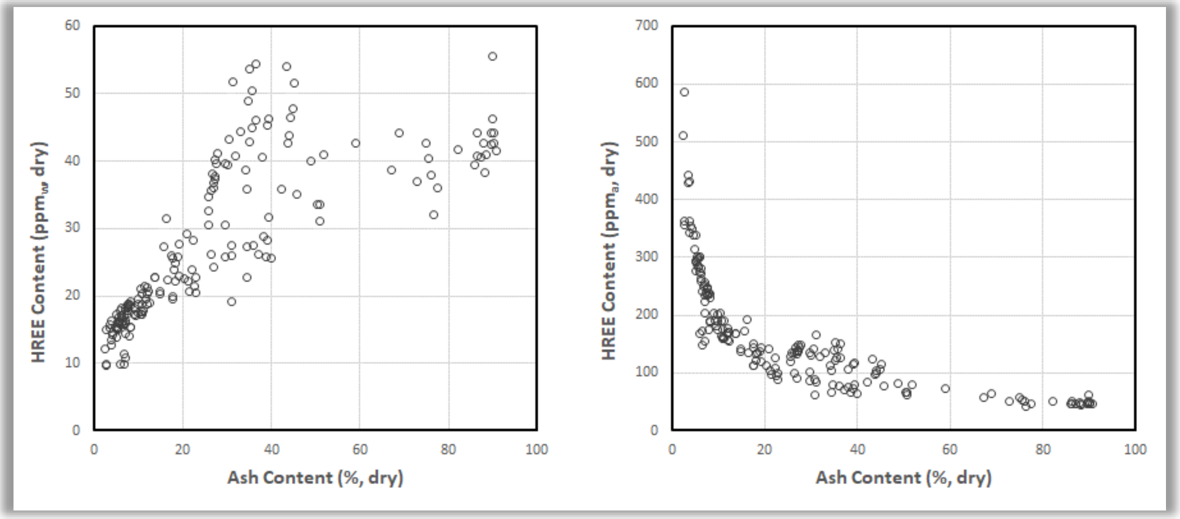
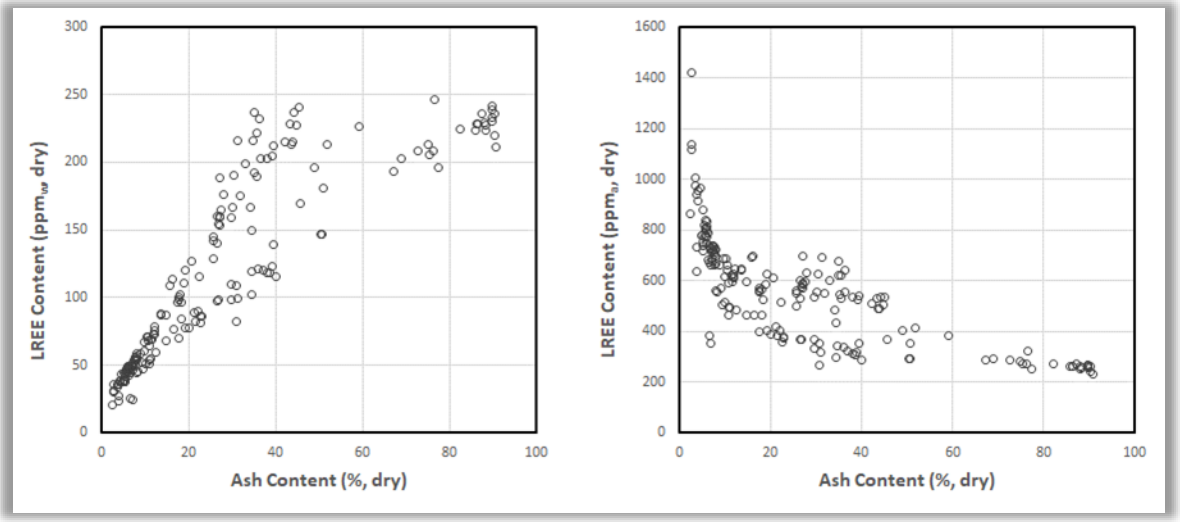


Figure 1-3: LREE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Seam Preparation Plant
(Left Plot= whole-sample basis; Right Plot = ash-residue basis)



The major findings from the roasting and leaching studies performed on the coarse rejects and middlings material collected from Pocahontas No. 3 Seam are enumerated as follows.

1. Total REE recovery values were increased significantly by roasting at 600℃ for two hours, i.e., about 80 percent and 75 percent of the TREEs were leached from the coarse rejects and middlings, respectively, when using 1.2 M HCl and 75℃ leaching temperature.
2. For the coarse rejects samples, H_2SO_4 performed as efficiently as HCl while, for the middlings material, recovery decreased by about 15 absolute percentage points when using HCl.

3. Roasting at 500℃ and 600℃ was more efficient for LREEs compared to HREEs. For example, a Ce recovery of 96 percent was realized from treating the roasted coarse rejects samples using 1.2 M HCl while only about 35 percent of Y and Dy were leached.
4. Under the same roasting and leaching conditions, LREEs recovery from the coarse rejects samples were higher than the middlings, while HREEs were relatively easier to be leached from the middlings. About 60 percent of the HREEs were recovered from the 600℃ roasted middlings.
5. Based on the effects of roasting on leaching kinetics, three groups of elements were studied, i.e.: Fe, Cu and Co (maximum recovery occurred at 400℃); Ca (maximum recovery occurred between 400℃ and 600℃, more than 80 percent was leached for the non-roasted samples); and REEs, Al, Mg, Zn and V (maximum recovery occurred at 500℃ or 600℃).
6. Two factors apparently contributed significantly to the improved REE recovery by roasting, i.e., liberation of ash materials and REE mineralogy transformation.
7. REE mineral particles may be completely and/or partially locked by the other mineral particles such as clays. However, this observation alone may not be a dominant factor in limiting REEs acid leachability.
8. For the samples roasted at 750℃, the REEs recovery were sensitive to acid leaching solution temperature as well as solution pH. Significant decreases in recovery occurred under any given solution pH when the acid leaching solution temperature was decreased from 75℃ to 25℃.

However, in spite of considerable efforts to identify REE concentrations in many portions of the Pocahontas No. 3 seam, events at the coal mine and preparation plant that were beyond the control of the project investigators compelled that alternative feedstock sites be secured.

1.1.3.3 Summary Observations of the HHS Application to the Ferroglobe Blue Gem Seam and Jellico Coal Seam Rejects Samples

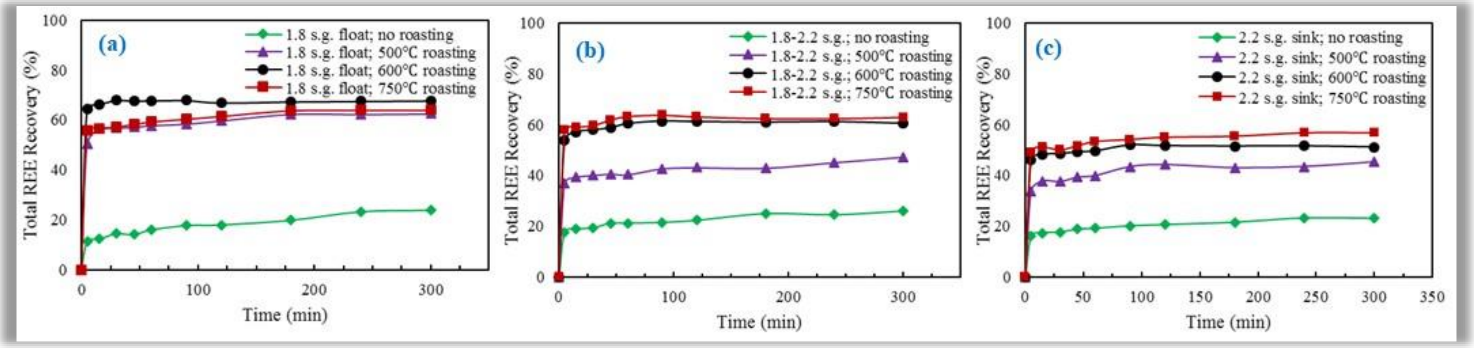
Altogether, of the tested samples collected from the Ferroglobe Plant, the flotation circuit streams, and particularly those from the flotation product stream, tested very well in the HHS process. All tests generated a low ash, low-moisture product that met the target market specifications. The oversize material from both the Blue Gem and Jellico spiral concentrate samples also performed extremely well. Ultra-low ash, high-value products were produced from these coal processing rejects streams with little to no grinding necessary. The performance on the -0.25 millimeter *(mm)* undersize material was promising as well, and capable of producing extremely low ash coal products; however, this material requires the addition of a dispersant to aid in the selective agglomeration step.

Overall, the Blue Gem and Jellico feedstocks collected during this test program were considered ideal for upgrading ultrafine coal using the HHS process.

1.1.3.3.1 Effects of Roasting on TREE Recovery from Three Float-sink Products of Rejects from Ferroglobe's Gatliff Plant (Blue Gem Seam)

The effects of roasting on TREE recovery from three float-sink products (i.e., 1.8 specific gravity *(SG)* float, 1.8-2.2 SG, and 2.2 SG sink) were assessed and compared to the performance on unroasted samples as shown in the following figure.

Figure 1-4: Effects of Roasting on the TREE Recovery from (a) 1.8 SG Float, (b) 1.8-2.2 SG, and (c) 2.2 SG Sink Fraction of the Blue Gem Seam Coarse Reject Samples



1.1.3.3.2 Summary Observations of the Effects Roasting on TREE Recovery from Three Float-Sink Products of Rejects from Ferrogloble's Gatliff Plant

1. For the non-roasted samples, REE recovery between the density fractions were within 25 percent of each other.
2. Roasting significantly improved recovery with the highest (55-65 percent of REEs) occurring between 600°C and 750°C.
3. It should be noted that the 1.8-2.2 SG and 2.2 SG sink samples contained very high ash contents (83.50 percent and 92.28 percent). Accordingly, it is believed that the improvement in REE recovery is not correlated with the organic matter removal and micro-dispersed ash material in organic matrix. Rather, the increase in REE recovery is more likely explained by the thermal decomposition of rare earth minerals (mainly phosphates) at high temperatures.
4. The results shown in the preceding figure as graphs (b) and (c) indicate recovery from the 500°C roasted samples was less than that of the 600°C and 750°C samples. It is, therefore, apparent that the decomposition of rare earth minerals for improved leach recovery in the Blue Gem coarse rejects is favorable in temperatures above 600°C.
5. A comparison between graphs (a) and (b) of the preceding figure shows that REE recovery of the 500°C roasted 1.8 SG float material was higher than that of the 1.8-2.2 SG (63 percent versus 47 percent).
6. The difference in recovery between 500°C and 600°C roasted samples was minimal for the 1.8 SG float sample.
7. More than 90 percent of the organic matter in the 1.8 SG float sample was removed by roasting at 500°C for two hours. The improved recovery by roasting at 500°C is partially due to the release of REEs associated with organic matter and/or micro-dispersed ash material in the organic matrix. Previous studies have reported that REEs in the finely dispersed ash material in coal have higher leachability.
8. For the 1.8 SG roasted sample, the recovery values achieved with the 600°C roasted material was a little higher than the 750°C, while the 1.8-2.2 SG and 2.2 SG sink samples show an opposite trend. This finding may be due to the different mineralogy between 1.8 SG float and the other two fractions.
9. The test results also indicate that there is a temperature limit where leach recovery values begin to decrease with increasing temperature. This indicates that morphology changes are temperature

dependent. Roasting at 750℃ and above may begin to sinter the ash material in the 1.8 SG float fraction which is known to reduce leach recovery.

In light of a limited prospect to make a combined HHS and REE recovery viable at this site, further examination was suspended in favor or an alternative site.

1.1.3.4 Arq Corbin USA (Arq) – Summary Observations of HHS Treatment of Rejects Impoundment Samples

Arq is a private technology-led energy group, headquartered in London, UK, with operations in Lexington and Corbin, Kentucky; Truro, UK; and Queensland, Australia. Among other things, Arq specializes in converting coal fines from coal waste dumps into particles small enough to be blended with fuel-oil products (see Arq Technology™). Through its USA subsidiary, Arq Corbin, LLC, Arq is in the process of installing a processing/conversion facility in the city of Corbin, Kentucky, adjacent to large coal waste embankments and attendant lagoons adjacent to the site of a former coal handling and preparation plant (locally known as the "U.S. Steel Property". The Arq Technology™ process of extracting and milling coal from coal waste dumps is expected to generate a highly decarbonized waste stream that could be beneficial for REE concentration purposes.

A sample from Arq's rejects impoundment was micronizing using ceramic media instead of iron media to minimize any increase in feed ash due to media wear and corrosion. This test procedure generated product ash values between 1.0-1.2 percent ash. These purity levels were very close to the <1 percent ash specification targeted for this particular feedstock. In fact, one test (Run 24) produced a 0.96 percent ash product after two-stages of agglomerate cleaning. These results suggest that grinding of an agglomerated product is the most effective way of generating very low ash products with the HHS process.

As this Arq project site was in the midst of construction and because no appropriate arrangements could be made with the owner to reach an agreement in principal for further investigations, the consideration of this feedstock was terminated.

1.1.3.5 Summary Observations of the Blue Diamond Coal Company (Operated by Blackhawk) Fire Clay (Hazard No. 4) Seam Examinations for REE Concentration Purposes.

The experimental work performed on material collected from Fire Clay (Hazard No. 4) Seam sources involved the collection of samples from the rejects (coarse and thickener underflow) and middlings circuits of Blue Diamond Coal Co's No. 76 Plant[3] (a.k.a. Blackhawk Mining's Leatherwood Preparation Plant) located near the community of Slemp, Perry County, Kentucky.

The plant produces a low ash coal product from a primary dense medium vessel and a medium-ash content product from a secondary dense medium vessel, dense medium cyclone circuit and spiral concentrator circuit. Reject streams include the coarse reject belt material (plus (+)150 micrometer or micron (µm)) and fine reject thickener underflow slurry (minus (-)150 µm).

The plant is primarily supplied run-of-mine *(ROM)* coal feedstock from "company operated" mines in the Fire Clay Seam (Hazard No. 4) but it also takes ROM coal from other seams mined by captive or independent commercial mine operators. The plant operator will often selectively process the Fire Clay (Hazard No. 4) Seam for sale to a higher value pulverized coal injection *(PCI)* metallurgical coal market.

---

[3] Blue Diamond Coal Company, No. 76 Plant, (a.k.a. Leatherwood Plant) MSHA ID 15-16520, controlled by Blackhawk Mining LLC, 48 Beech Fork Road, Slemp, Perry County, Kentucky, 41763.

Other cleaned coal products and high ash products of the Fire Clay (Hazard No. 4) are typically sold into the steam coal markets.

The discussions that follow are based on samples from the rejects and middlings circuits of the No. 76 Plant while cleaning the Fire Clay (Hazard No. 4) Seam feedstock. As addressed in other chapters of this report, the Fire Clay (Hazard No. 4) Seam (to include its coal benches and its attendant partings, “riders”, roof and floor lithologies) presents a relatively well-known supply of rare earth elements. Most of the REEs are concentrated in the rejects and middlings circuits of the No. 76 Plant.

The following figures and narratives are examples of one of many tests conducted on the Fire Clay (Hazard No. 4) Seam rejects materials. (A complex series of test results are recited in the body of the report.)

1.1.3.5.1 No. 76 Plant (Leatherwood) Fire Clay Seam (Hazard No. 4) Thickener Underflow Acid Leaching Testing Methodology

To assess the recoverability of the REEs from the Leatherwood thickener underflow material, leaching studies were conducted on the flotation middling and tailing products as well as the material finer than 45 µm (-325-mesh).

Standard leaching conditions were used in the tests which included the use of a 1.2 M[4] sulfuric acid solution at a temperature of 75°C.

Leaching experiments were conducted on roasted and unroasted samples to assess the effect on recovery.

Roasting was used as a pre-treatment step prior to leaching at a temperature of 600°C for a period of two hours.

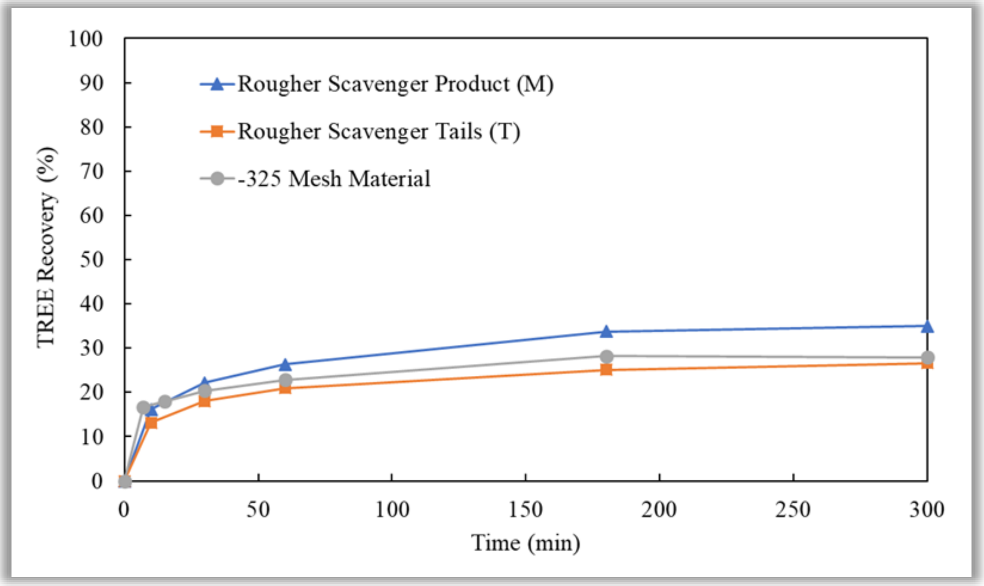
1.1.3.5.1.1 Acid Leaching Test Results of Unroasted Fire Clay Seam (Hazard No. 4) Flotation Middling Material from the No. 76 Plant (Leatherwood) Thickener Underflow

The TREE recovery achieved after leaching the unroasted flotation middling material for five hours was 35 percent as shown in the following figure.

---

[4] Molarity (**M**) is mols/liter.

Figure 1-5: Total REE Recovery Achieved by Leaching the Unroasted Products Generated of the No. 76 Plant (Leatherwood) Thickener Underflow Material Samples



Approximately one-half of the recovered REEs was extracted within the first 10 minutes.

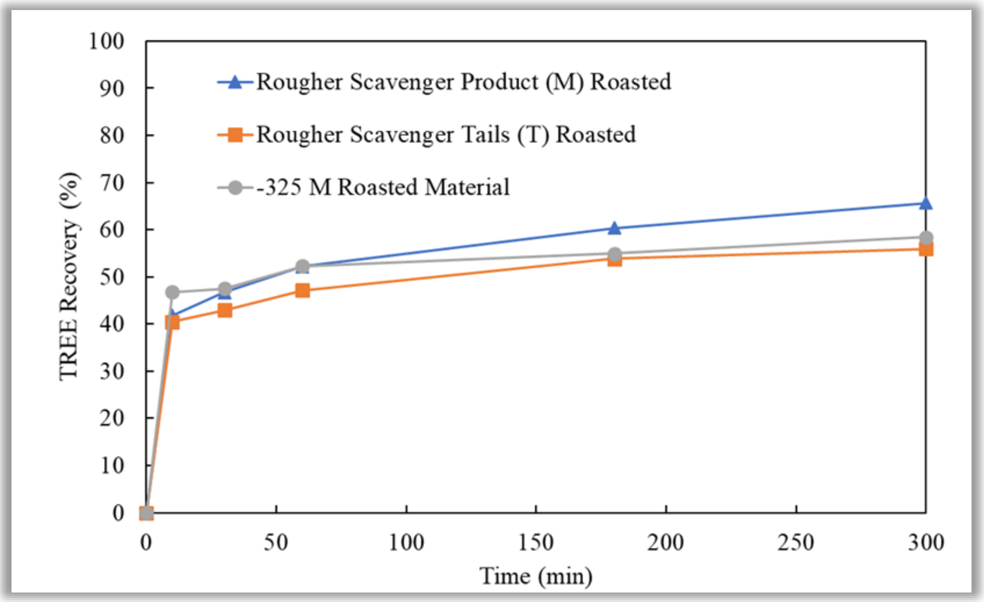
The REE recovery rates and final values for the flotation tailings and -45µm fraction were lower which agrees with previously reported findings.

After 300 minutes of leaching, recovery values obtained from the treatment of the tailings and -45µm size fraction reached 28 percent.

1.1.3.5.1.2 Acid Leaching Test Results of Roasted Fire Clay Seam (Hazard No. 4) Flotation Middling Material from the No. 76 Plant (Leatherwood) Thickener Underflow

After roasting, leaching rates and overall recovery improved significantly for all three samples as shown in the following figure.

Figure 1-6: TREE Recovery Achieved by Leaching the Roasted Products Generated from the Processing of the Leatherwood Thickener Underflow Material



TREE recovery values greater than 40 percent were achieved within only 10 minutes of leaching for each of the three roasted materials which signifies a substantial increase in the release rate of the REEs as compared to the test results on the unroasted samples. Given that the roasting process was conducted under 600^{o}C which is below the ash fusion point, the significantly enhanced leach rates and improved REE recovery values was likely due to the decomposition of the clay structure and phosphate minerals from which highly soluble rare earth minerals were released. After 10 minutes of leaching, recovery values gradually increased and reached levels of 58 percent to 65 percent after five hours of leaching.

1.1.3.6 Solvent Extraction Testing on a Variety of Feedstocks

Solvent Extraction Experiments included the following initiatives:

1. Extractant concentration effect on solvent extraction;
2. Effect of different ions on solvent extraction.

Bench-scale testing of the solvent extraction proposed for this project included the following activities:

1. Rougher loading scrubbing and stripping;
2. Continuous flow solvent extraction *(SX)* testing using No. 3 Pocahontas Seam Plant Rejects.

The investigators also evaluated the extraction characteristics of the REEs and contaminant elements by conducting the following experiments on the Pocahontas No. 3 Seam middlings reject material.

1. Rougher stage activities;
2. Extraction of metal ions in the stripped solution from the rougher stage;
3. Cleaner circuit simulations;
4. Precipitation of REO from stripped solutions by use of oxalic acid;
5. Analyses of the final REO product;
6. Ascorbic acid optimization to control iron (Fe) contamination of the leach liquor; and
7. Scandium Recovery.

The results of this investigation are summarized in the following table.

Table 1-4: Elemental Analysis of the Final REO Generated from Pocahontas No. 3 Seam Middlings Reject Material

Element	REE in Solution (ppm)	REO (ppm)
Lanthanum	26,040	30,519
Cerium	265,200	325,745
Praseodymium	41,176	49,741
Neodymium	184,080	214,637
Samarium	63,536	73,676
Europium	10,896	12,616
Yttrium	67,736	8,6018
Gadolinium	45,152	52,042
Terbium	3,110	3,468
Dysprosium	28,744	32,989
Holmium	3,466	3,970
Erbium	6,274	7,173
Thulium	712.8	814
Ytterbium	2,494	2,838
Lutetium	658	749
Total	**749,274**	**89,6995**

The investigators examined the results of the selective loading of REEs in the solvent from Ferroglobe's Blue Gem Seam preparation plant tailings material. The laboratory team also evaluated the characteristics of individual REEs relative to a choice of extractants from a simulated acid mine drainage. The investigators explored the separation of individual REEs from rejects material collected from processing the Fire Clay (Hazard No. 4) seam. The investigations included an evaluation of the effects of extractant concentration on the recovery rates of Y, Dy and Nd from Western Kentucky coal seam rejects. Distribution coefficients were derived and tabulated. Refinements to the saponification process were investigated.

1.1.3.7 Detailed Flow Diagram

The data collected from the flowsheet development activities (ultimately heavily reliant on tests of a Fire Clay Seam (Hazard No. 4) rejects feedstock) were used to construct a process flow diagram for REE extraction. For engineering purposes, the flow diagram was subdivided into five functional circuits, i.e.:

- Circuit 1 – Feed Preparation;
- Circuit 2 – Acid Leaching and Waste Treatment;
- Circuit 3 – Rougher Solvent Extraction;
- Circuit 4 – Cleaner Solvent Extraction; and
- Circuit 5 – Scandium Solvent Extraction.

For each circuit, experimental data was collected and evaluated for use in the flow diagram development. Economic factors were also considered in the flow diagram development, as described in later sections of this document. The technical work focused on detailed engineering activities including:

1. Calculations of mass flow rates for solid, liquid and component (e.g., ash, REEs, elemental impurities) species entering and exiting each circuit;
2. Preparation of a general listing of required processing equipment (type, size, capacity, power and consumables) for each circuit;
3. Preparation of a generalized process flowsheet (flow diagram) showing the unit-to-unit arrangement of all unit operations and connecting/recirculated streams for each circuit; and
4. Numerical values for the performance indicators used in the flowsheet development work (i.e., mass splits, liquid splits, component assays, etc.) were obtained from the aforementioned battery of experimental characterization studies and separation/extraction tests described in this report.

The following figures depict the final versions of the flow diagrams identified by the process engineering team as the most viable configuration for REE extraction.

Figure 1-7: Process Flow Diagram for REE Extraction from Coal-based Sources

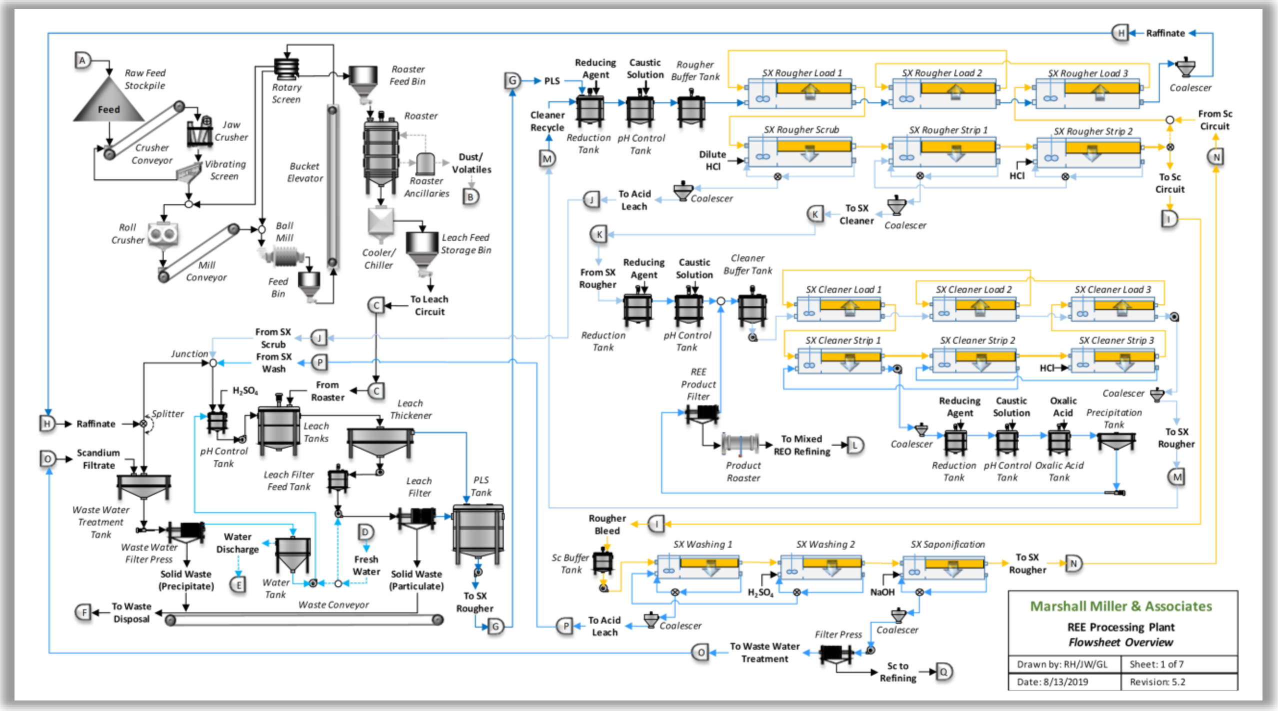


Figure 1-8: Detailed Process Flow Diagram for the Feedstock Preparation Circuit

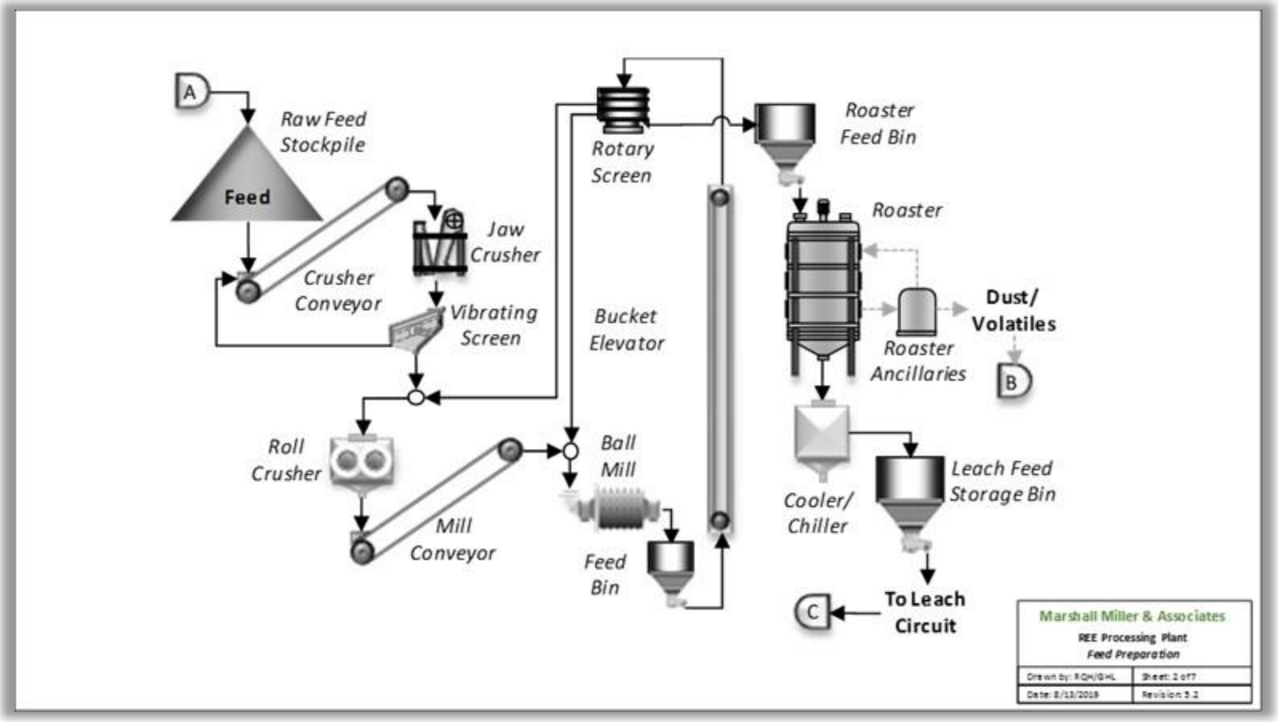


Figure 1-9: Detailed Process Flow Diagram for the Acid Leaching and Waste Treatment Circuit

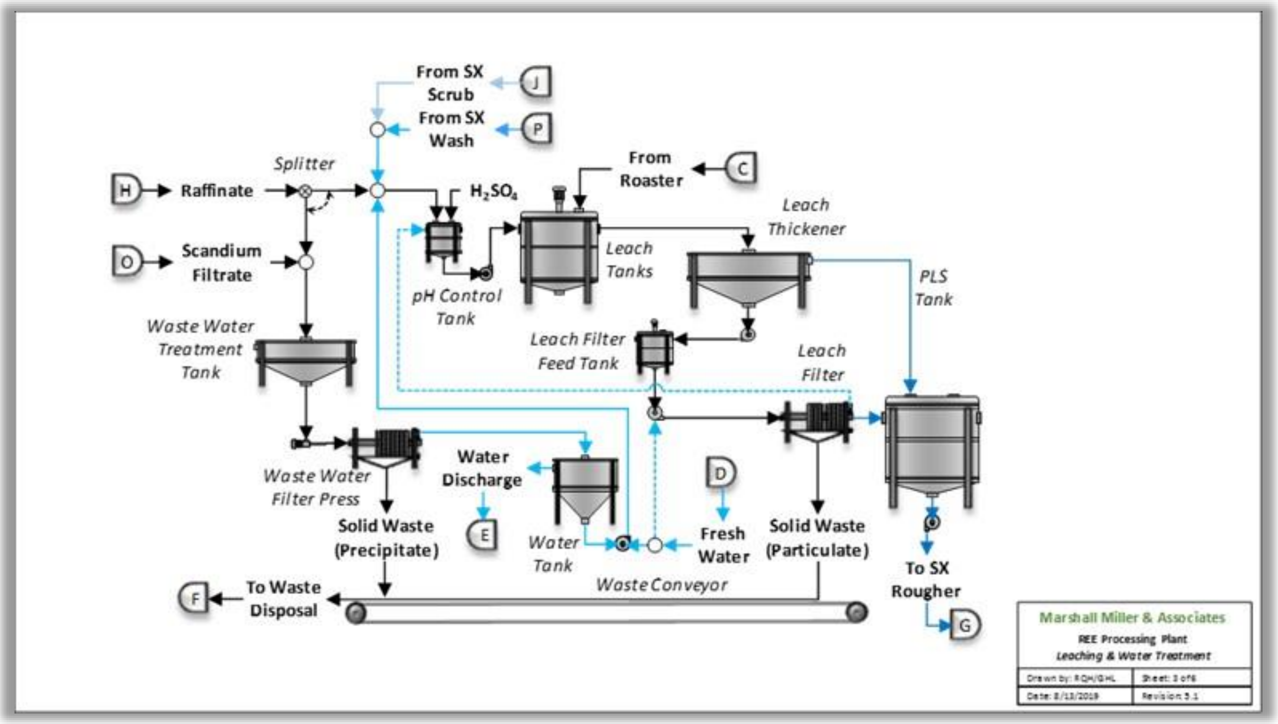


Figure 1-10: Detailed Process Flow Diagram for the Rougher Solvent Extraction Circuit

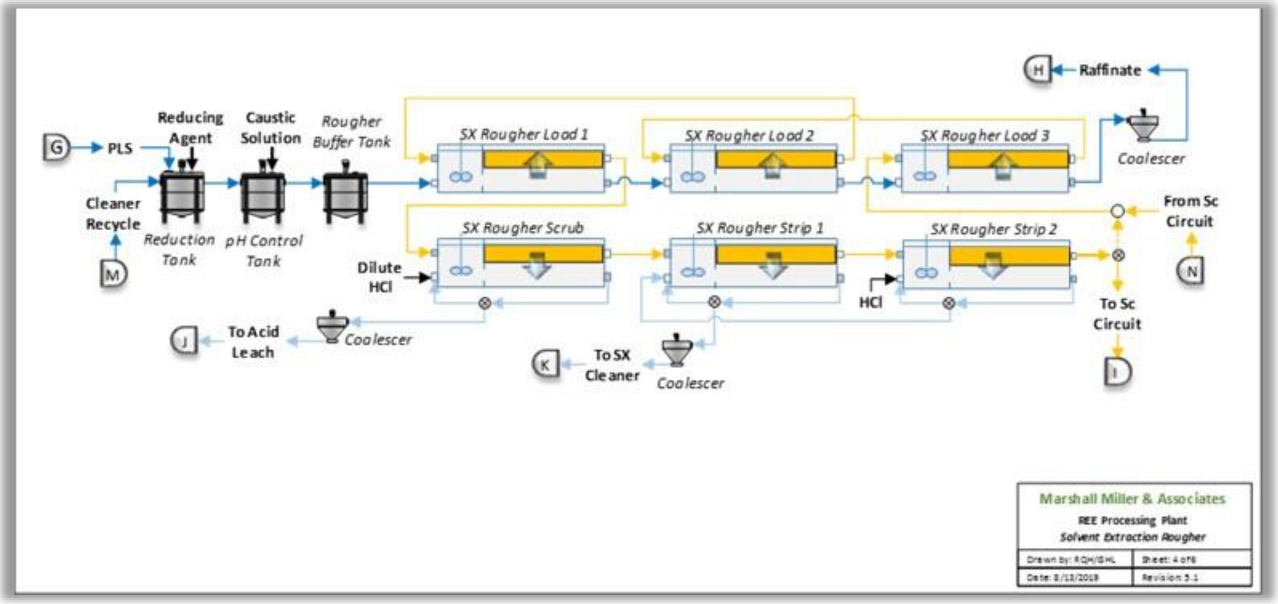


Figure 1-11: Detailed Process Flow Diagram for the Cleaner Solevent Extraction Circuit

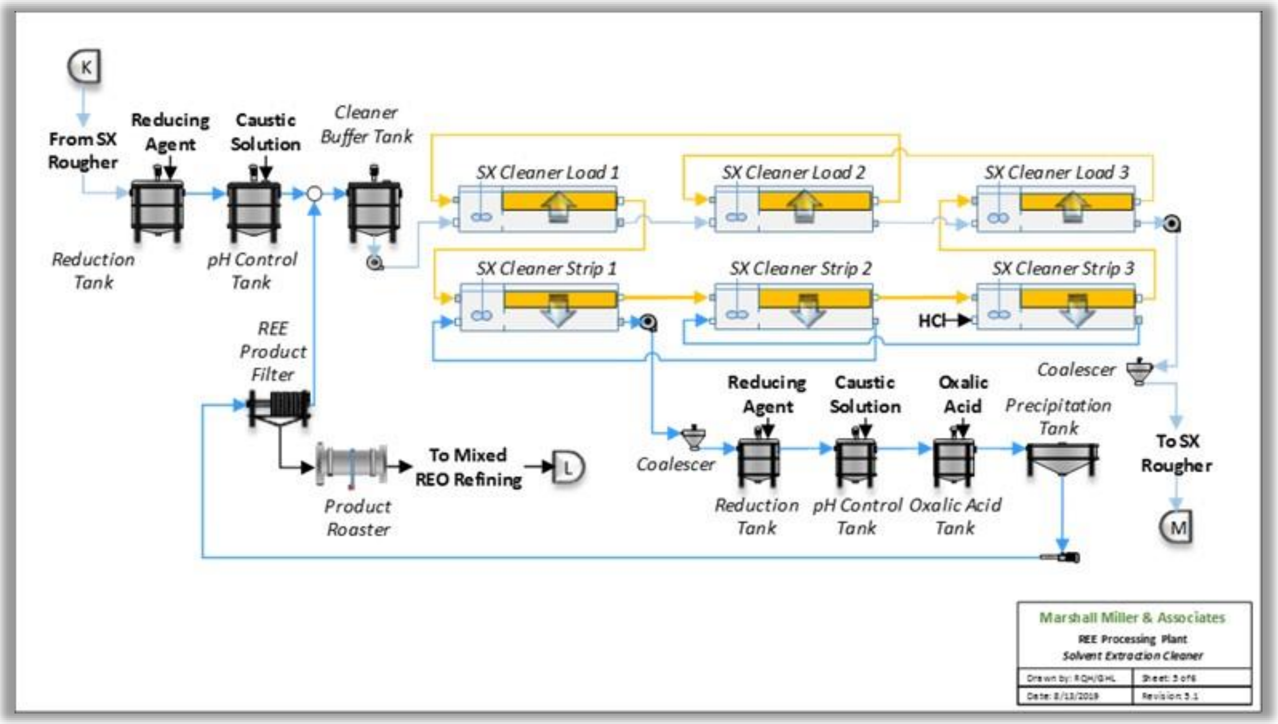
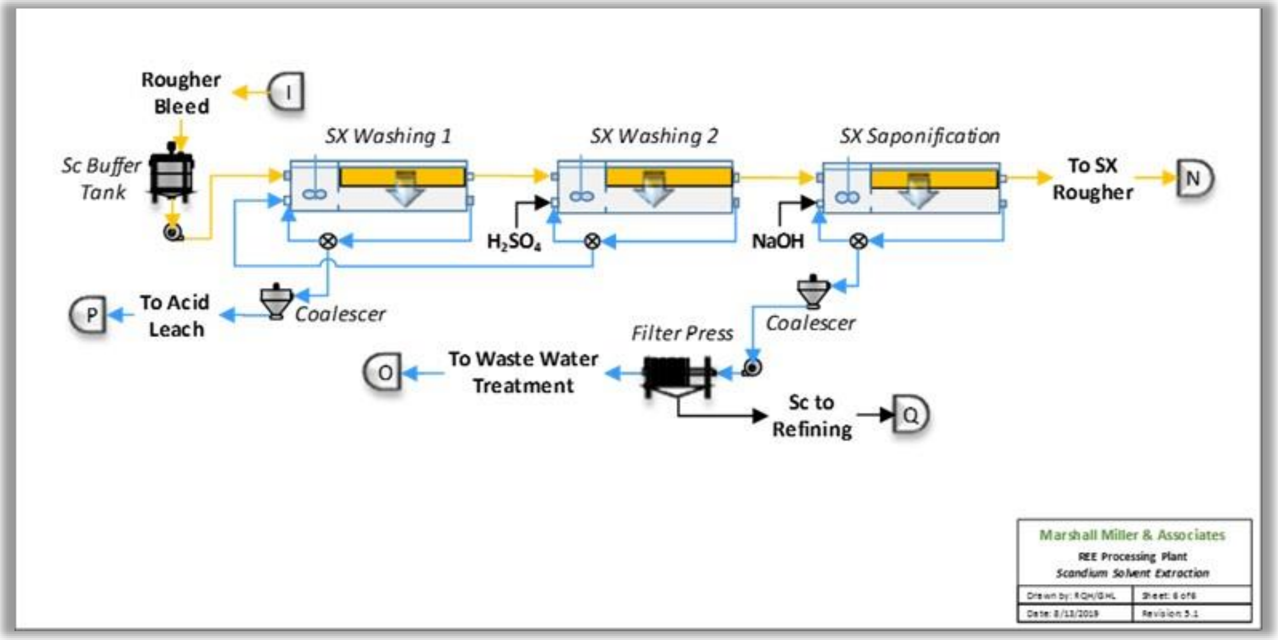


Figure 1-12: Detailed Process Flow Diagram for the Scandium Solvent Extraction Circuit



1.1.3.8 Flowsheet Simulation to Evaluate Process Circuitry for REE Concentration from Coal-based Sources

1.1.3.8.1 Flowsheet Development Software Selection

In order to streamline the flowsheet development work, the project team utilized a process flowsheet simulation tool.

1.1.3.8.2 Evaluation of LIMN[5]

The simulation initially made use of the LIMN flowsheet processor to create, model and analyze different flowsheet configurations. However, as work progressed, it became apparent that the use of this particular tool was limited due to a lack of hydrometallurgical models and constraints associated with software licensing and distribution.

1.1.3.8.3 Evaluation of REESim

To avoid the LIMN software use and application issues, the engineering team switched to a spreadsheet-based simulation package that was being developed under another DOE sponsored project entitled "*Pilot-Scale Testing of an Integrated Circuit for the Extraction of Rare Earth Minerals and Elements from Coal and Coal Byproducts Using Advanced Separation Technologies* (DE-FE0027035). One of the key tasks to be completed under this on-going project involved the development of a flowsheet simulation tool that can be applied to develop, design and evaluate process circuitry for REE concentration from coal-based sources. This software tool, which is currently referred to as REESim, was ideally suited for use in the current project.

1.1.3.9 Summary Discussion of the Simulation Results for Project MMA 29956

In general, the simulation results can be used to derive several key observations related to the final process flow diagram. These insights include the following:

1. In the feed preparation circuit, the roasting operation is an essential step in generating a dry solid feedstock that responds well to REE leaching. This operation also ensures that essentially no water enters with the dry feed, which is very beneficial in establishing proper water balances for the facility.
2. Extraction efficiencies for acid leaching achieved an average value of only 23.96 percent for the REE of interest in this project.
    a) The lowest recovery of 9.81 percent was obtained for dysprosium, while a high of 63.9 percent was realized for terbium.
    b) Scandium, which has the highest market value of the elements under consideration, is associated with an extraction recovery of only 18.1 percent during acid leaching.

---

[5] The LIMN "The Flowsheet Processor" software has been developed by David Wiseman since starting his company in early 1994.

3. The simulation data indicated that the circuits in the proposed flow process diagram should be able to produce an REO product with a purity level of 97.4 percent or higher. The same simulation indicates a scandium purity of 99.999 percent.
    a) Note: Incorporation of actual experience with low roasting conversion and poor acid leaching efficiencies into REESim reveals a predicted plant recovery of TREEs of only 17.03 percent. Predicted Scandium recovery can be even poorer, at 0.77 percent.
4. The recycling of raffinate back to the primary pH control tank for acid leaching was found to be necessary to maintain proper flow and acid balances in the process facility.
    a) A recirculation rate of approximately 85-90 percent was established as a reasonable value for balancing flow/acid demands against a build-up of unwanted impurities in the pregnant leach solution *(PLS)*.

See the following tables for a summary of the plant operation simulations.

Table 1-5: REESim Predicted Assay Values for the Overall Plant Operations

1-5 - TOTAL PLANT										
	IN	**OUT**								
	Raw Feed	**Roast Dust/Vol.**	**AL Filter Cake**	**WT Filter Cake**	**REO Product**	**REO Dust**	**Sc Filter Cake**	**Feed In**	**Product Out**	**Delta --**
Circuit Performance Indicators										
Yield (%)	100.00	1.00	92.84	6.15	0.01	0.00	0.00	100.00	100.00	--
Ash (%)	81.12	81.12	79.87	100.00	100.00	100.00	100.00	81.12	81.12	--
Impurity Assays (%, dry)										
Unknown %	40.78	40.78	43.48	0.00	0.00	0.00	0.00	40.78	40.78	0.00
Al %	11.84	11.84	10.80	27.46	0.02	0.02	0.00	11.84	11.84	0.00
Ca %	1.34	1.34	0.11	19.89	2.52	2.52	0.00	1.34	1.34	0.00
Fe %	7.38	7.38	4.39	52.62	0.03	0.03	0.00	7.38	7.38	0.00
Si %	19.75	19.75	21.06	0.00	0.00	0.00	0.00	19.75	19.75	0.00
Trace Assays (%, dry)										
Sc %	0.0016	0.0016	0.0014	0.0004	0.1324	0.1324	99.9993	0.0016	0.0016	0.0000
Y %	0.0033	0.0033	0.0031	0.0001	5.9397	5.9397	0.0000	0.0033	0.0033	0.0000
La %	0.0062	0.0062	0.0048	0.0109	16.4162	16.4162	0.0001	0.0062	0.0062	0.0000
Ce %	0.0133	0.0133	0.0103	0.0130	42.5305	42.5305	0.0003	0.0133	0.0133	0.0000
Pr %	0.0015	0.0015	0.0012	0.0019	4.5631	4.5631	0.0000	0.0015	0.0015	0.0000
Nd %	0.0059	0.0059	0.0048	0.0002	20.6566	20.6566	0.0001	0.0059	0.0059	0.0000
Pm %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Sm %	0.0010	0.0010	0.0009	0.0000	2.8354	2.8354	0.0000	0.0010	0.0010	0.0000
Eu %	0.0002	0.0002	0.0002	0.0000	0.2412	0.2412	0.0000	0.0002	0.0002	0.0000
Gd %	0.0008	0.0008	0.0008	0.0005	1.3721	1.3721	0.0000	0.0008	0.0008	0.0000
Tb %	0.0001	0.0001	0.0000	0.0002	0.9480	0.9480	0.0000	0.0001	0.0001	0.0000
Dy %	0.0007	0.0007	0.0007	0.0000	1.0564	1.0564	0.0000	0.0007	0.0007	0.0000
Ho %	0.0001	0.0001	0.0001	0.0000	0.0000	0.0000	0.0000	0.0001	0.0001	0.0000
Er %	0.0004	0.0004	0.0004	0.0000	0.0000	0.0000	0.0000	0.0004	0.0004	0.0000
Tm %	0.0001	0.0001	0.0000	0.0001	0.1593	0.1593	0.0000	0.0001	0.0001	0.0000
Yb %	0.0003	0.0003	0.0003	0.0000	0.4560	0.4560	0.0000	0.0003	0.0003	0.0000
Lu %	0.0000	0.0000	0.0000	0.0000	0.1279	0.1279	0.0000	0.0000	0.0000	0.0000
Th %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
U %	0.0000	0.0000	0.0000	0.0000	0.0001	0.0001	0.0000	0.0000	0.0000	0.0000
TREE%	0.0357	0.0357	0.0291	0.0274	97.4349	97.4349	99.9999	0.0357	0.0357	--

Note: The raw feed parameters are based on a Fire Clay Seam coarse rejects sample collected from the Leatherwood Plant on 11-16-2018. The analytical information from this sample (as well as others collected from this plant) is located on the RESEARCHER DATABASE.

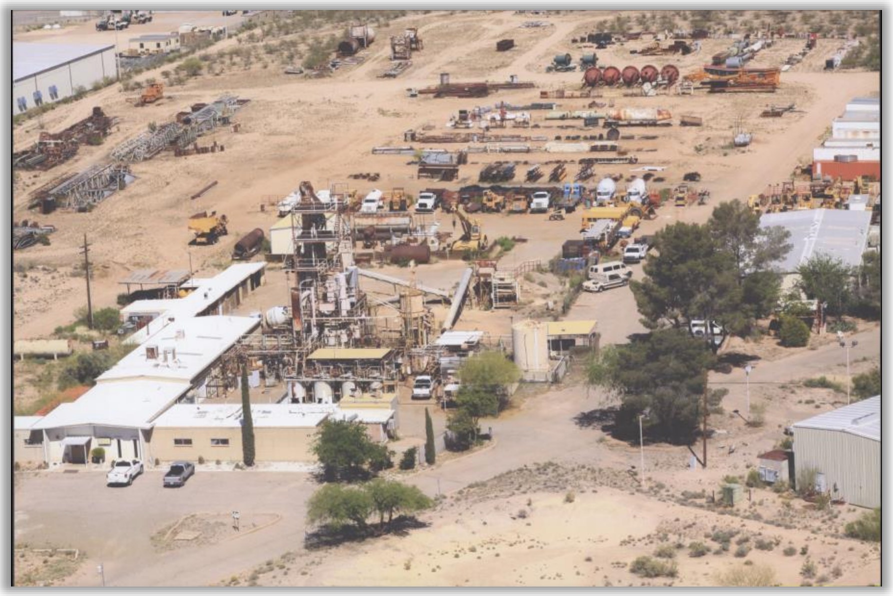
Table 1-6: REESim Distribution Values for the Overall Plant Adjusted for Pilot Plant Experience

1-5 - TOTAL PLANT										
	IN	**OUT**								
	Raw Feed	**Roast Dust/Vol.**	**AL Filter Cake**	**WT Filter Cake**	**REO Product**	**REO Dust**	**Sc Filter Cake**	**Feed In**	**Product Out**	**Delta --**
Mass Distributions										
Total	100.00	1.00	92.84	6.15	0.01	0.00	0.00	100.00	100.00	--
Ash	100.00	1.00	99.00	0.00	0.00	0.00	0.00	100.00	100.00	--
Impurity Distributions										
Unknown	100.00	1.00	82.63	1.50	0.51	0.03	14.33	100.00	100.00	0.00
Al	100.00	1.00	87.24	0.11	11.07	0.58	0.00	100.00	100.00	0.00
Ca	100.00	1.00	71.03	10.70	16.41	0.86	0.00	100.00	100.00	0.00
Fe	100.00	1.00	71.89	6.04	20.01	1.05	0.00	100.00	100.00	0.00
Si	100.00	1.00	71.73	7.77	18.53	0.98	0.00	100.00	100.00	0.00
Trace Element Distributions										
Sc	100.00	1.00	82.63	1.50	0.51	0.03	14.33	100.00	100.00	0.00
Y	100.00	1.00	87.24	0.11	11.07	0.58	0.00	100.00	100.00	0.00
La	100.00	1.00	71.03	10.70	16.41	0.86	0.00	100.00	100.00	0.00
Ce	100.00	1.00	71.89	6.04	20.01	1.05	0.00	100.00	100.00	0.00
Pr	100.00	1.00	71.73	7.77	18.53	0.98	0.00	100.00	100.00	0.00
Nd	100.00	1.00	75.89	0.21	21.76	1.15	0.00	100.00	100.00	0.00
Pm	100.00	1.00	99.00	0.00	0.00	0.00	0.00	100.00	100.00	0.00
Sm	100.00	1.00	80.74	0.16	17.19	0.90	0.00	100.00	100.00	0.00
Eu	100.00	1.00	89.20	0.09	9.22	0.49	0.00	100.00	100.00	0.00
Gd	100.00	1.00	84.90	3.43	10.14	0.53	0.00	100.00	100.00	0.00
Tb	100.00	1.00	36.90	10.08	49.42	2.60	0.00	100.00	100.00	0.00
Dy	100.00	1.00	88.48	0.38	9.63	0.51	0.00	100.00	100.00	0.00
Ho	100.00	1.00	99.00	0.00	0.00	0.00	0.00	100.00	100.00	0.00
Er	100.00	1.00	99.00	0.00	0.00	0.00	0.00	100.00	100.00	0.00
Tm	100.00	1.00	71.22	7.21	19.54	1.03	0.00	100.00	100.00	0.00
Yb	100.00	1.00	89.16	0.70	8.68	0.46	0.00	100.00	100.00	0.00
Lu	100.00	1.00	75.16	6.24	16.72	0.88	0.00	100.00	100.00	0.00
Th	100.00	1.00	93.70	5.23	0.07	0.00	0.00	100.00	100.00	0.00
U	100.00	1.00	90.05	2.59	6.04	0.32	0.00	100.00	100.00	0.00
	100.00	1.00	75.70	4.72	17.03	0.90	0.65	100.00	100.00	0.00

1.1.3.10 Availability of a 1 Short Ton per Hour Pilot Plant

Due to the short time frame allotted to the Project in Phase 2, the procurement and fabrication of equipment for the project was the primary concern. From the knowledge and network of the design team, a previously operated Hydro-Met Pilot Plant was located in the Western United States that may be suitable for re-engineering and restoration. This could represent a significant time and labor savings for the project to decrease risks. The following figure shows an aerial photograph of the previously operated Hydro-Met plant at its site in the Western United States.

Figure 1-13: Previously Operated Pilot Plant Potentially Suited for Reutilization



1.1.3.10.1 Pilot Plant Site Proposed for the 1 Short Ton per Hour REM Concentration Plant

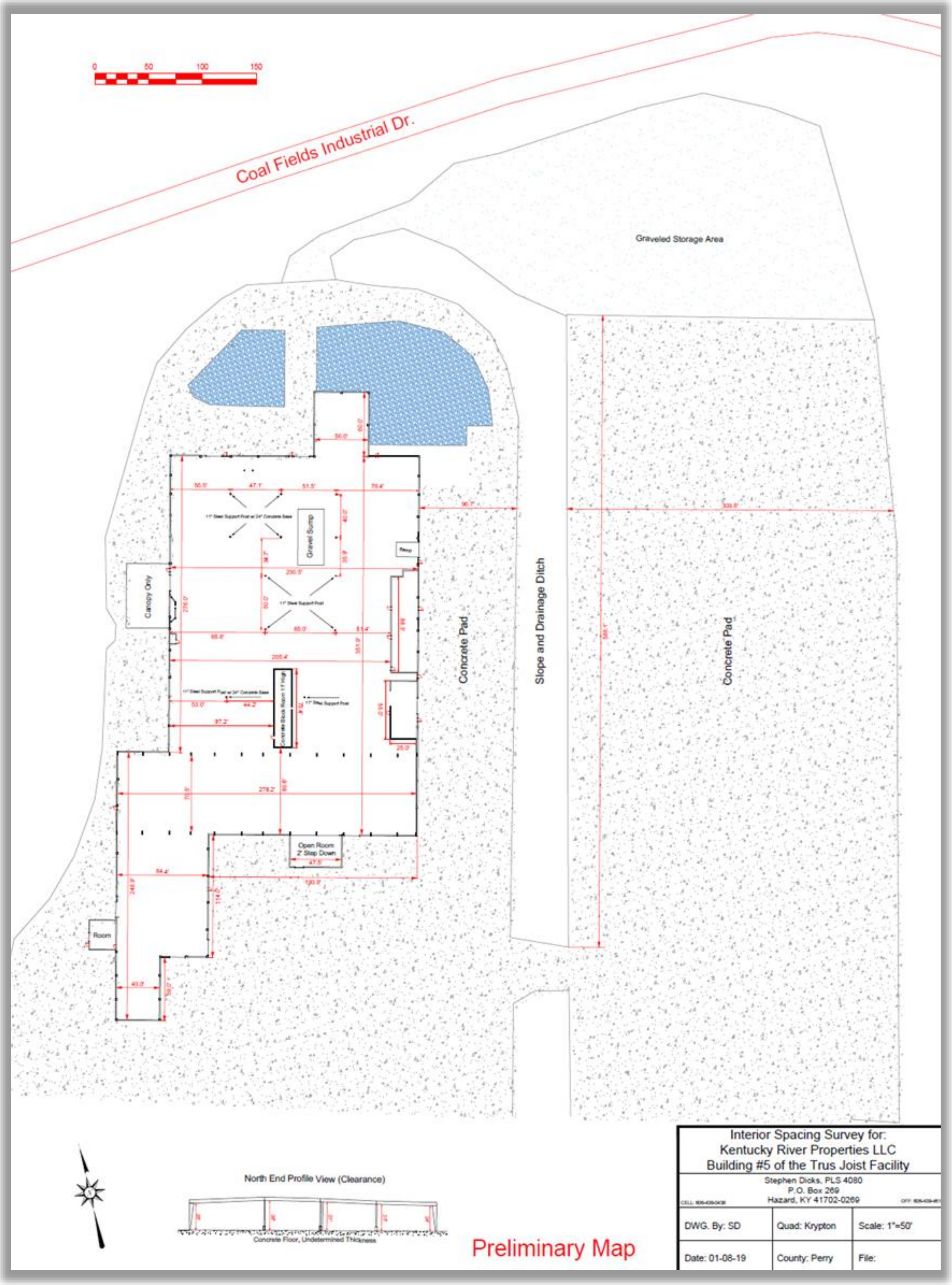
In its collaborative effort in support of the project, KRP offered the use of its sizeable industrial complex building on Trus Joist Lane of the Coal Fields Industrial Park near Chavies, Kentucky.

A survey of the Trus Joist facility[6] was conducted previously to determine an accurate representation of the interior of the facility and to assist the team designing the Pilot Plant layout. The site and building map of the surveyed Trus Joist facility is shown in the following figure.

---

[6] Trus Joist is a former wood laminate beam fabrication facility located on 610 Trus Joist Lane in the Coal Fields Industrial Park, near Chavies, Kentucky. The site is now owned by a subsidiary of KRP and is the proposed site for a 1 tph Pilot Plant.

Figure 1-14: Surveyed Map of the Truss Joist Facility



1.1.3.11 Risk and Fatal Flaw Analysis

As part of the scope of work a fatal flaw analysis was performed. In this case, the analysis identifies what the team feels are the most likely and most severe failures that may occur in the technical components of the project.

The technique utilized was a failure mode and effects analysis *(FMEA)*. A FMEA is a matrix of the analysis of potential failures, their severity, what causes them and what may be done to detect and prevent them. The FMEA was populated with the following number scheme in order of increasing severity. These are 1,3,7, and 10. The highest ranking, 10 is reserved for significant safety items. The ratings for failure severity, occurrence, and detection are multiplied together to determine the risk priority number *(RPN)*. The higher the RPN the greater the risk to the project.

1.1.3.11.1 Summary of Significant Concerns

The following items of particular concern appear to be present, given the resultant RPN numbers generated by the risk and fatal flaw analysis.

1.1.3.11.2 Timing of Regulatory Permits, Agreements or Contracts

The most significant risk facing the project was determined to be the rate at which the permits could be filed, negotiated and granted. In that regard, securing the appropriate environmental protection permits is expected to require extensive base-line surveys, planning, document processing, and regulatory agency(ies) evaluation(s), as discussed in other chapters of this report. The critical nature of these permits cannot be overstated as consultation with the appropriate regulators reveals that no phase of the project construction will be allowed prior to appropriate regulatory approvals.

1.1.3.11.3 Performance of the Proposed Solvent Extraction Process

The next most severe potential fatal flaw relates to the risks associated with the limited testing of the proposed SX process. This abbreviated testing of the SX process was an artifact of the protracted evaluation of various sources of coal preparation plant related feedstocks during the preliminary phases of this project. The remaining schedule did not provide the project team sufficient time to adequately test the roughing, cleaning, saponification, and recycling circuits of the proposed REE pilot plant. In many instances, preliminary or best-available data was utilized for the design reported herein. These designs are preliminary and as expressed in the analysis will require further testing and validation to remove risk.

1.1.3.11.4 Roaster Performance with a Coal Preparation Plant Rejects Feedstocks

The other significant risk identified was the operation of the roaster. The roaster is a multiple stage rotary hearth design. This means that the internal rakes will cause material to cascade from one stage to the next. In a heated, oxidizing environment, the risk of creating a coal dust explosion is not readily known. For this reason, additional study and consultation with experts is required prior to construction to lower the risk.

1.1.4 Techno-Economic Analyses

To assess the feasibility of recovering REEs from coal rejects, a detailed technical review and economic evaluation were conducted. The primary input data for this analysis consisted of experimental testing conducted at both the bench-scale and small pilot-scale. In addition to the experimental data, other inputs to the techno-economic analysis were derived from detailed flowsheet simulations and engineering designs.

The actual cost analysis was conducted at both a large pilot-scale and a commercial-scale. The large pilot-scale utilizes a nominal capacity of 2,000 lb./hr. (1-short ton [st/hr.] or 0.91[metric-ton] mt/hr.), and these results were used as the basis for the engineering work documented throughout this report. To assess the future commercial viability of the process technology, the analysis was also conducted at a commercial-scale with a nominal feed rate capacity of 500 short tons per hour.

To facilitate the technical and economic review, the engineering team used a spreadsheet-based economic assessment tool that has been developed under another DOE sponsored project entitled "*Pilot-Scale Testing of an Integrated Circuit for the Extraction of Rare Earth Minerals and Elements from Coal and Coal Byproducts Using Advanced Separation Technologies* (DE-FE0027035). One of the key tasks to be completed under this ongoing project involved the development of an economic evaluation tool that can be applied to process circuitry for REE concentration from coal-based sources. This software, which is currently referred to as REE-Econ, was ideally suited for use in the current project.

1.1.4.1 Summary of All Major Experimental Data, Engineering Analysis, Computations and Test Results (Fire Clay Seam (Hazard No. 4) Rejects.

The following economic analysis is based on tests performed on roasted[7] Fire Clay Seam (Hazard No. 4) rejects at a 0.25 ton per hour proof-of-concept pilot plant located in West Kentucky and operated by the University of Kentucky. Prior to conducting the techno-economic analysis, the experimental data was reviewed and analyzed by the engineering team. The results of this technical analysis were integrated into the flowsheet simulations conducted using the REESim™ software. The results pertinent to the techno-economic analysis, including REE feed concentrations, overall REE recovery values and element to oxide conversion factors, are shown in the following tables.

**Table 1-7: Feedstock REE Concentration (Nominal)
Used in Techno-Economic Analysis**

REE	Feed Conc. (ppm)	Distribution (%)
Scandium (Sc)	16.25	5.40
Yttrium (Y)	33.45	9.31
Lanthanum (La)	62.37	16.26
Cerium (Ce)	132.50	34.56
Praseodymium (Pr)	15.36	6.50
Neodymium (Nd)	59.19	15.33
Samarium (Sm)	10.29	3.53
Europium (Eu)	1.63	0.46
Gadolinium (Gd)	8.44	2.84
Terbium (Tb)	1.20	0.25
Dysprosium (Dy)	6.84	1.73
Holmium (Ho)	1.26	0.50
Erbium (Er)	3.61	1.55
Thulium (Tm)	0.51	0.35
Ytterbium (Yb)	3.28	1.13
Lutetium (Lu)	0.48	0.30
Total:	**356.64**	**100.00**

---

[7] Personal Communication with Dr. Honaker, Ph.D. The Nex-Gen roasting facilities located in the Coal Fields Industrial Park near Chavies, Kentucky, was used in this process. At that stage of the project, the required control of roasting temperature and roasting atmosphere had not yet been determined. Evidence now shows that this sample had been overheated. Leaching results from this roasted sample were subsequently significantly less than optimal.

Table 1-8: REE to REO Conversion Factors Used in Techno-Economic Analysis

Compound	% REE in Pure REO	Source
Sc_2O_3	65.2%	[1]
Y_2O_3	78.7%	[2]
La_2O_3	85.3%	[2]
CeO_2	81.4%	[2]
Pr_6O_{11}	82.8%	[2]
Nd_2O_3	85.7%	[2]
Sm_2O_3	86.2%	[2]
Eu_2O_3	86.4%	[2]
Gd_2O_3	86.8%	[2]
Tb_4O_7	85.0%	[2]
Dy_2O_3	87.1%	[2]
Ho_2O_3	87.3%	[1]
Er_2O_3	87.5%	[2]
Tm_2O_3	87.6%	[1]
Yb_2O_3	87.8%	[2]
Lu_2O_3	87.9%	[1]

Sources:

1) *https://www.sigmaaldrich.com*
2) *https://www.bluelinecorp.com/re-calculator*

Table 1-9: Overall Element-by-Element Recovery Values Used in Techno-Economic Analysis

REE	Recovery to Various Product Streams (%)				
	Sc_2O_3	Dy_2O_3	Gd_2O_3	Mixed REO	Losses
Scandium (Sc)	14.3%	0.0%	0.0%	0.5%	85.2%
Yttrium (Y)	0.0%	0.0%	0.0%	11.1%	88.9%
Lanthanum (La)	0.0%	0.0%	0.0%	16.4%	83.6%
Cerium (Ce)	0.0%	0.0%	0.0%	20.0%	80.0%
Praseodymium (Pr)	0.0%	0.0%	0.0%	18.5%	81.5%
Neodymium (Nd)	0.0%	0.0%	0.0%	21.8%	78.2%
Samarium (Sm)	0.0%	0.0%	0.0%	17.2%	82.8%
Europium (Eu)	0.0%	0.0%	0.0%	9.2%	90.8%
Gadolinium (Gd)	0.0%	0.0%	10.1%	0.0%	89.9%
Terbium (Tb)	0.0%	0.0%	0.0%	49.4%	50.6%
Dysprosium (Dy)	0.0%	9.6%	0.0%	0.0%	90.4%
Holmium (Ho)	0.0%	0.0%	0.0%	0.0%	100.0%
Erbium (Er)	0.0%	0.0%	0.0%	0.0%	100.0%
Thulium (Tm)	0.0%	0.0%	0.0%	19.5%	80.5%
Ytterbium (Yb)	0.0%	0.0%	0.0%	8.7%	91.3%
Lutetium (Lu)	0.0%	0.0%	0.0%	16.7%	83.3%
Product Purity (%)					
Oxide Basis	95.0%	99.0%	99.0%	--	--
REE Basis	61.9%	86.2%	85.9%	81.0%	--

1.1.4.2 Economic Assumptions

Several economic assumptions were also applied to facilitate the commercial-scale techno-economic analysis. These assumptions are in the following list. Many of the items, including those regarding the financing structure, escalation rates, tax calculations, and operating period have been supplied by NETL in the "*Guidance for Development of Techno-Economic Analyses"* document.

- All amounts are in US dollars.
- The total operational period for the plant is 20 years.
- The plant feed rate is fixed at 500 short TPH (st/hr.), with a feedstock concentration of 357 ppm REE on a whole sample basis.
- Inflation has been applied to sales revenue and operating costs using a fixed rate of three percent.
- Capital costs are spread over a period not to exceed three years, and the allocation between those three years is 10 percent, 60 percent, and 30 percent for years one through three, respectively. Thus, the total analysis period (capital purchase plus operating) is not to exceed 23 years.
- During the capital expenditure period, capital costs escalate at a constant rate of 3.6 percent.
- The project is debt financed for 50 percent of the total overnight[8] capital requirement. The remaining 50 percent is financed by equity.
- The debt repayment terms include: Six (6) percent interest rate, 10-year loan period, and no grace period on debt repayment. The repayment uses a standard amortization schedule with constant payments throughout the payoff period.
- Working capital is not included in this estimate and will instead be borne by the operating entity at no cost to the project.
- The combined federal and local tax rate is fixed at 26 percent. This value is lower than the one required in the NETL guidance document; however, it reflects a recent reduction in corporate tax rates.
- All capital is depreciable, using a 150 percent declining balance depreciation schedule over 20 years. The depreciation method was NOT changed to straight line when conditions favored the switch.
- The mineral depletion rate for REEs is 14 percent. Depletion is charged at the appropriate rate times the net sales revenue after deducting royalties and any severance tax, provided that the total amount calculated by depletion rates does not exceed 50 percent of the taxable income before depletion.
- The plant is part of a larger entity with sufficient revenue to offset negative taxable income. Thus, losses are not carried forward and are instead calculated as a "negative tax" that indicates the reduction in tax burden required for overall entity.
- The land and mineral rights are leased, and royalties will be paid to the landowner.
- Royalties are charged at standard rate of 6.5 percent of total sales revenue.

---

[8] The overnight market is the component of the money market involving the shortest term loan. Lenders agree to lend borrowers funds only "overnight" i.e., the borrower must repay the borrowed funds plus interest at the start of business the next day. Given the short period of the loan, the interest rate charged in the overnight market, known as the overnight rate is, generally speaking, the lowest rate at which banks lend money.

- All process modules are located at the same facility with no significant transportation cost between modules.
- The plant operation schedule is fixed at eight (8) hours per shift, three (3) shifts per day, 336 operational days per year.
- Equipment costs were determined by scale-factor analysis. Installation costs were determined by an overall Lang[9] factor.
- Reagents, consumables, power, and labor were determined by itemized cost analysis.
- Other operating costs, including maintenance and supply, QAQC, sales, admin, property tax, insurance, and plant overhead were all calculated using proportional factors.
- The rare earth price deck is primarily based on a standard price deck provided by NETL. Elements not included in this price deck were derived from **Asian Metal, Inc**. ***(AMI)***[10]

The REE sales price was based on a 100 percent price realization for the purified oxide products highlighted in this project:

Scandium oxide (Sc_2O_3)
Dysprosium oxide (Dy_2O_3
Gadolinium oxide (Gd_2O_3)

All other products were assumed to be sold as bulk mixed rare earth oxides *(MREOs)* at a 65 percent price realization.

1.1.4.3 Economic Indicators for a Commercial-Scale Plant (500 Short Tons per Hour)

The following table includes the key output parameters from this overall analysis. Unfortunately, these results show that the overall REE recovery process is not economically viable, as the process costs greatly exceed the processing revenue in each year.

---

[9] The Lang Factor is an estimated ratio of the total cost of creating a process within a plant, to the cost of all major technical components. It is widely used in industrial engineering to calculate the capital and operating costs of a plant. The factors were introduced by H. J. Lang and Dr Michael Bird in *Chemical Engineering* magazine in 1947, as a method for estimating the total installation cost for plants and equipment.

[10] Asian Metal, Inc., No. 6 Jianguomenwai Avenue, Suite 16B, Tower C, SK Plaza, Beijing, 100022, China, http://www.asianmetal.com/RareEarthsPrice/RareEarths.html

Table 1-10: Summary of Economic Indicators for Commercial-Scale Plant (500 st/hr.)

Parameter	Unit	Value
Technical Results		
Plant Feed Rate	sTPH*	500
Plant Feed Grade	ppm	357
Overall REE Production	kg/hr.	35.7
Sc_2O_3 Production	kg/hr.	1.7
Dy_2O_3 Production	kg/hr.	0.3
Gd_2O_3 Production	kg/hr.	0.5
MREO Production	kg/hr.	33.2
Economic Results		
Overnight Capital Cost	$	129,580,060
Constant Dollar OpEx	$ / yr.	519,567,924
(unit conversion)	$/ st feed	128.86
(unit conversion)	$ /kg REE	2,251.97
Constant Dollar Revenue	$ / yr.	59,268,751
Net Present Value (@10% discount)	$	(3,387,608,423)
Internal Rate of Return	%	N/A
Payback Period	Operating Years	N/A

While many factors contribute to this outcome, the final unprofitable result can be principally attributed to the high chemical consumption values, the low REE feed grades, and the low REE recovery values, which all contribute to high operating costs on a $/kg basis. In addition, low market prices for several REE compounds also limits the economic feasibility of the venture. Lastly, the final project flowsheet analyzed in this chapter does not produce marketable byproducts other than the three refined REEs and the bulk MREO product.

Modifications to the flowsheet or the feedstock selection may provide an opportunity to produce other byproducts such as recoverable coal or non-REE critical materials whose sales will offset the cost of the REE concentration process. If produced in a synergistic manner, these additional sources of revenue may greatly improve the economic viability of the REE recovery process.

1.1.4.4 Influence of Optimal Roasting Conditions

The techno-economic analysis presented in the preceding sections of this summary "*Baseline Case"*) used test data from a bulk sample of a commercially roasted feedstock that was later determined to be significantly overheated. Additional laboratory data revealed that superior outcomes can be achieved by carefully controlling the roasting temperature during leaching. When properly roasted, fair leaching recoveries can be achieved even at relatively low acid doses, indicating that there is an optimal leaching pH which balances REE recovery with respect to the costs of acid doses. The research team then analyzed both a high-recovery/high-acid concentration leaching scenario (0.5 M, 53 percent TREE recovery) as well as a low-recovery/low-acid concentration leaching scenario (0.05M, 24 percent TREE recovery) of properly roasted feedstock in relation to the poorly roasted feedstock of the base case. The results of these analyses are shown in the following table.

Table 1-11: Project Economic Summary for Material Processed under Differing Roasting and Acid Application Conditions

Parameter	Unit	Base Case[1]	Case A[2]	Case B[3]
Technical Results				
Plant Feed Rate	TPH	500.00	500.00	500
Plant Feed Grade	ppm	356.64	356.64	357
Overall REE Production	kg/hr.	35.67	60.03	35.7
Sc_2O_3 Production	kg/hr.	1.71	3.75	1.7
Dy_2O_3 Production	kg/hr.	0.35	0.67	0.3
Gd_2O_3 Production	kg/hr.	0.45	0.26	0.5
MREO Production	kg/hr.	33.17	55.35	33.2
Economic Results				
Overnight Capital Cost	$	129,580,060.04	129,795,550.36	129,580,060
Constant Dollar OPEX	$/yr.	519,567,923.88	520,867,729.67	213,644,168
(unit conversion)	$/t feed	128.86	129.18	52.99
(unit conversion)	$ /kg REE	2,251.97	1,346.79	926.00
Constant Dollar Revenue	$/yr.	59,268,750.74	128,781,032.77	59,268,751
Net Present Value (@10% discount)	$	(3,387,608,423.25)	(2,943,021,695.42)	(1,251,087,493)
Internal Rate of Return	%	N/A	N/A	N/A
Payback Period	Operating Years	N/A	N/A	N/A

1. Base Case = Poor roasting performance, high acid concentration in leaching, low REE Recovery.
2. Case A = Optimal roasting, high acid concentration in leaching, high REE recovery.
3. Case B = Optimal roasting, low acid concentration in leaching, low REE recovery.

While the reductions in Cases A and B are not sufficient to prompt economic viability, the results do represent significant gains in processing knowledge and provide some input on the appropriate pathway to process optimization.

1.1.4.5 Key Findings

In general, the results from the techno-economic assessment can be used to derive several key findings related to the economic performance of the proposed extraction process. These key findings include:

1. The process equipment to be used in the Pilot-Scale Facility represents a total equipment cost of over $10.3 million (including $1.8 million in used equipment and $8.5 million in new equipment), but the used equipment will require an additional investment of $1.5 million to disassemble, transport, rehabilitate, refurbish, reassemble, and restart. Construction of the new process buildings is expected to be $1.6 million, bringing the total plant value to $13.4 million.
2. Capital cost estimates at both the pilot-scale and commercial-scale show that the roasting operation is the most significant capital investment category, representing nearly one-third of the total capital expenditure.
3. A scale-factor economic analysis of a potential 500 st/hr. production facility shows that the proposed process circuitry is too costly to be economically viable without government subsidy or significant technical improvements.
4. Sensitivity analysis of the cost model shows that process recovery, plant feed grade, and reagent consumption are critical components influencing overall profitability.
5. As such, additional research and development should seek to identify higher-grade REE resources, as well as process alternatives that reduce acid and base consumption while improving recovery.

6. Additional revenue from other byproducts such as recoverable coal or critical materials should be considered.

1.1.4.6 Potential Pathway to Profitability

Altogether, the sensitivity analysis indicates that a potential pathway to profitability must include:

1. Improved control of the feedstock roasting temperature.
2. Increases to the plant feed grade to reflect more recent information;
3. Reduction in reagent consumption; and/or
4. Improvements to process recovery.

To illustrate the combined influence of these three parameters, an “ideal case” model scenario was evaluated. In this scenario:

1. The overall reagent consumption was reduced by 75% percent relative to baseline values;
2. The REE feed grade was increased to 450 ppm;
3. The overall recovery was increased to 53 percent (three times the baseline value); and
4. All other input parameters and model assumptions remain unchanged from their baseline values.

(Despite the relatively high influence of REE prices, this pricing factor was not considered in the ideal case analysis, since these prices are not within the control of plant operator [i.e., guaranteed price subsidies were not considered a viable option to attain profitability]).

The results of this analysis are shown in the following table and indicate that these changes greatly improve the economic outcomes for the project.

Table 1-12: Economic Indicators for Commercial Plant under Improved Input Conditions

Parameter	Unit	Value
Technical Results		
Plant Feed Rate	sTPH*	500
Plant Feed Grade	ppm	450
Overall REE Production	kg/hr.	134.7
Sc_2O_3 Production	kg/hr.	6.5
Dy_2O_3 Production	kg/hr.	1.5
Gd_2O_3 Production	kg/hr.	1.7
MREO Production	kg/hr.	125.0
Economic Results		
Overnight Capital Cost	$	129,056,413
Constant Dollar OpEx	$ / yr.	156,556,687
(unit conversion)	$/ st feed	38.83
(unit conversion)	$ /kg REE	179.97
Constant Dollar Revenue	$ / yr.	220,333,864
Net Present Value (@10% discount)	$	254,248,143
Internal Rate of Return	%	36%
Payback Period	Operating Years	3.5

**Short tons per hour*

1.1.5 Permitting and Environmental Management for Phase 2

As addressed in other sections of this report, Phase 1 entailed collecting and analyzing coal rejects produced during the preparation (cleaning) of coal for sale using modern processes. The discarded material (rejects) from modern preparation plants contain very little remnant coal and comprise various mineral compounds that would otherwise interfere with the market purpose of the cleaned coal. Studies have indicated that the rejects of certain coal beds, in this case the Fire Clay coalbed of eastern Kentucky, is relatively rich in yttrium (*Y*) and a variety of REE. This combination of elements will hereinafter be denoted as Y+REE.

Tests conducted on crushed and milled Fire Clay coalbed rejects donated by **Blue Diamond Mining Company** (a.k.a. **Blue Diamond Coal Company** or "***Blue Diamond***") (as subsidiary of Blackhawk) by UK and VT demonstrated that expert modification of existing ore preparation, leaching, and solvent extraction methods might eventually result in the feasible production of marketable grade concentrates of Y+REE.

The collaborative efforts of Blackhawk, Blackhawk's lessor, KRP, **Marshall Miller & Associates (*MM&A*)**, VT, and UK, resulted in a focus on the rejects from the Fire Clay Seam mined and processed in the Letcher and Perry County, Kentucky, facilities of Blackhawk. In addition to establishing a REE enriching feedstock source through Blackhawk, this initiative extended conceptually to the installation of an REE Pilot Plant (as part of Phase 2) at a large idled industrial facility owned by KRP at the Coalfields Industrial Park near Chavies, Kentucky. Subject to a successful Pilot Plant venture, the concepts included the installation and operation of a full-sized REE enrichment facility at or near the coal processing plant of Blackhawk at Slemp, Kentucky.

1.1.5.1 Sources of Run-of-Mine Fire Clay (Hazard No. 4) Coal from Which the Phase 1 Rejects Was Extracted

Blue Diamond supplied coal preparation plant rejects from an undifferentiated mix of ROM coal recovered from its Calvary Mine No. 81 and Orchard Branch Mine No. 89. These are fully mechanized room-and-pillar mines using continuous mining machines and shuttle cars. Both mines practice "selective second mining."

1.1.5.1.1 Mine No. 81 (Calvary Mine 81)

Mine No. 81 has a long production history and has been one of the mainstays of Blue Diamond in spite of the vagaries of the metallurgical and thermal coal markets. It has portals located near Smilax and Slemp, Kentucky. The mine's safety record is good. MSHA's most recent analyses of Calvary Mine No. 81 safety records reveal that the mine does not fit the profile of an entity with a pattern of violations *(POV)*. The last complete five-year history of Safety Statistics collected by the **Kentucky Division of Mine Safety and Training *(KDMS&T)*** reveal that the mine exhibits a relatively good reportable accident profile and has not incurred any events that resulted in a fatality. Calvary Mine No. 81 operates under KDNR Permit Numbers: 866-5161, 866-5163, and 897-5150.

1.1.5.1.2 Mine No. 89 (Orchard Branch Mine 89)

Orchard Branch Mine No. 89 is a recent addition to Blue Diamond. It is located near Viper, Kentucky. The historical production from this mine is good. The mine's current safety record is very good. MSHA's most recent analyses of Mine No. 89 safety records reveal that the mine does not fit the profile of an entity with a POV. The last complete five-year history of Safety Statistics collected by the KDMS&T reveal that the mine exhibits an "average-to-fair" accident profile but that it has not incurred any events that result in a fatality. Orchard Branch Mine No. 89 operates under KDNR Permit Numbers: 897-5113, and 897-5136.

1.1.5.2 Source of Rejects Derived from Processing the Fire Clay (Hazard No. 4) Seam During Phase 1

1.1.5.2.1 Blue Diamond Coal Company No. 76 Plant (Operated by Blackhawk Mining

Coal is transported by belt from Mine No. 81 and trucked from Mine No. 89 to the No. 76 Preparation Plant located near Slemp, Kentucky. This plant has a long operating history and has been repeatedly updated as well as improved. It is considered one of the most modern coal preparation plants in Kentucky. The preparation plant's current safety record is good. MSHA's most recent analyses of this facility's safety records reveal that the facility does not fit the profile of an entity with a POV. (The KDMS&T does not maintain records of coal handling and preparation plants.) No 76 Plant operates under KDNR Permit Numbers: 897-0596, 897-0597, 897-8062, and 897-9008.

1.1.5.3 Financial Viability of Blackhawk

Blackhawk was founded in 2010 and expanded by buying-up active coal reserves and mining operations in the wake of other coal companies' Chapter 11 bankruptcies. Eventually Blackhawk operated 19 underground mines and six surface mines in Kentucky and West Virginia. However, as a result of these transactions as well as subsequent activities taken to position Blackhawk as one of the largest metallurgical coal producers in the nation, Blackhawk accumulated a substantial debt. Significantly, the anticipated improvement in the metallurgical coal market in 2015 did not occur until late 2016. The alternate steam-making coal market continues to be depressed in the light of competition from natural gas and lower than expected electricity demand growth. Consequently, Blackhawk filed for Chapter 11 bankruptcy[11] on July 19, 2019.

Prior to the filing, the company negotiated a "Restructuring Support Agreement" with its lenders. Blackhawk expects that the court-supervised "pre-packaged" bankruptcy proceedings will be completed in short order, i.e., 60 days more or less[12].

Blackhawk reports that it has sufficient liquidity to continue normal mining operations and to meet its obligations in the ordinary course of doing business. As such, Blackhawk will continue to pay employee wages and provide healthcare and other benefits without interruption in the ordinary course of business and to pay suppliers and vendors in full under normal terms for goods and services provided both prior to and after the Chapter 11 filing date[13].

Given the foregoing corrective activities, it appears likely that Blackhawk's Fire Clay Seam operations in Leslie and Perry Counties will prove to be viable sources of rejects for an REE Pilot Plant.[14]

1.1.5.4 Lessor/Lessee Relationships

Blue Diamond conducts most of its coal mining operations on leased lands. In respect to the REE project, KRP owns the Fire Clay coalbed and leases the coal in that coalbed (among others on its lands) to Blue Diamond. The lease by and between KRP and Blue Diamond is dated the August 3, 1988 but is not recorded. By letter of commitment dated 11/21/2018, KRP declared its support to Phase 1 of the project on a cost-sharing basis. Subject to events beyond its control, KRP proposes to extend that commitment into a subsequent phase, should process simulation by UK support that transition. To that end, KRP made available access to the coal preparation plant rejects and to certain industrial facilities and lands in the Coalfields Industrial Park near Chavies, in Perry County, Kentucky, and has coordinated key elements of the Fire Clay coalbed exploration, mine face sampling at Blue Diamond mines, and coal rejects bulk sample collection at Blue Diamond's coal preparation plant.

1.1.5.5 Pilot Plant Permitting Plan for Phase 2

Although Phase 1 of DOE/NETL Project DE-FE0029956 has not demonstrated the feasible production of REOs, continuing effort on the subject under other commissions might yet result in a positive outcome. Consistent with Phase 1 scope of work, the second phase (Phase 2), should it occur, will include permitting and constructing a 1-tph raw feed REE concentration pilot plant. Preliminary arrangements have been made with KRP for the use of its idled industrial facility located in the Coal Fields Industrial Park near Chavies, Kentucky.

1.1.5.5.1 Likely Coal Mining Related Permit Obligations

A minor revision to Blue Diamond's No. 76 Plant will be required by KDNR to properly document the use of rejects as a raw feed to the REE Pilot Plant. Preparation, submittal and approval of this minor revision should take no more than 60 days.

Because of its association with a coal preparation plant, the REE Pilot Plant may have to be permitted as a coal operation. The REE Pilot Plant may either be permitted under a new KDNR permit or amended to an

---

[11] Kallanish Energy, (www.kallanishenergy.com) July 22, 2019. Another major coal producer files for bankruptcy.
[12] Ibid.
[13] July 16, 2019. Posted by Gary Slone of Mountain-Topmedia (gary@mountain-topmedia.com
[14] This perception is supported by Wood Mackenzie, a reputable global coal market analyst. See https://www.axios.com/coal-bankruptcies-are-piling-up, by Ben Ceman of Axios, July 19,2019.

existing Blackhawk Preparation Plant permit located nearby at the Coal Fields Industrial Park. Should it be required, the preparation, submittal and approval of a comprehensive Mine Permit Application *(MPA)* may take 12 months to complete. The comprehensive MPA requires baseline surface water and ground water data collection for a minimum of six months prior to initial submittal. Alternatively, should the nearby Blackhawk preparation plant agree to an amendment to its permit in order to incorporate the Pilot Plant operations, the preparation, submittal and approval process of that amendment may take six months to complete.

1.1.5.5.2 Wastewater from the Pilot Plant Processing System

Treated wastewater from the REE Pilot Plant Facility will be discharged into the public sanitary sewer system operated by the HPWD. The discharge must comply with all applicable pre-treatment standards promulgated under Section 307 (B) of the Clean Water Act *(CWA)*, which may be included in the permit issued to the publicly-owned treatment works *(POTW)*. Discharges into the POTW do not need a KPDES permit. Alternatively, the wastewater could be discharged to waters of the Commonwealth under coverage of an individual **Kentucky Pollutant Discharge Elimination System *(KPDES)*** permit.

1.1.5.5.3 Solid Waste Generated by the Pilot Plant Processing System

Solid waste from the REE Pilot Plant Facility may be disposed at the Blue Diamond’s existing rejects facility adjacent to the No. 76 coal handling and preparation plant under KDNR Permit No. 897-8062. This facility is approximately 38 miles from the REE Pilot Plant Facility. If the solid waste is disposed in the Blue Diamond’s facility, a minor revision to document the final location of disposal will be necessary. This could be included in the minor revision for the raw feed discussed previously. Alternatively, the solid process waste can be disposed at a qualified commercial landfill. The nearest landfill that will accept the solid waste from the REE Pilot Plant Facility is most likely the Laurel Ridge Landfill located near Lily, Kentucky that is situated approximately 75 miles from the proposed Pilot Plant site.

1.1.5.5.3.3 Air Emissions

The following processes associated with the REE Pilot Plant Facility will require a permit from the Division for Air Quality *(DAQ)*: raw feed stockpile, crushing, grinding, feedstock material roasting, leaching, extraction and treatment, and solvent extraction. The DAQ is a branch of KDEP. A permit from DAQ is required for the construction and operation of an air contaminant source and its air pollution control equipment. The type of permit required for the REE Pilot Plant Facility is dependent upon the amount of air pollutants the facility has the potential-to-emit *(PTE)*, if operated 24 hours per day, 7 days a week at maximum capacity for one year. Total air emissions are identified by the amount of regulated air pollutants *(RAPs)* the facility has the PTE.

1.1.5.6 Environmental Management Plan for the Pilot Plant Operations

The components of the environmental management for this pilot plant will include the following key elements:

> Solid and Liquid waste stream mass and volume;

> Solid and Liquid waste stream constituents and concentrations;

> Process chemicals consumption;

> Suitable air emission control technologies; and

> Suitable liquid containment control measures.

1.1.5.6.1 Process Chemical Toxicity

The consumable chemicals used in the process include acids and bases commonly used in chemical manufacturing and materials processing. The primary health risk to humans is chemical burns associated with physical contact with the materials. Environmental risks include alterations to media (soil and water) pH. High and low pH conditions can have detrimental effects to aquatic life. The materials are water-soluble and therefore are quite mobile in the environment. Spill prevention controls and countermeasures (*SPCC*) plans will be an essential component of the management plan for the Phase 2 operation.

The recyclable chemicals used in the solvent extraction process are Kerosene, Tributyl Phosphate *(TBP)* and 2-Di *(ethylhexyl)* Phosphoric Acid *(D2EPHA)*. Kerosene is a common hydrocarbon-based material. The TBP and D2EHPA are organophosphate compounds that are Category 4 Hazardous Classification under **Occupational Safety and Health Administration** ***(OSHA)*** guidance for oral toxicity. Both chemicals bind with soils and are not particularly mobile in the environment.

1.1.5.6.2 Solid and Liquid Process Waste Streams

Testing conducted during Phase 1 has indicated that the solid waste stream passes Toxic Characteristic Leaching Procedures *(TCLP)*. The water from the process was also tested for priority pollutant parameters and all values were below regulatory threshold levels. Solid waste will be disposed of in a third-party permitted solid waste landfill. Treated wastewater from the process will be disposed of in the municipal sanitary sewer system located at the facility.

1.1.5.6.3 Air Quality Control Measures

The process includes roasting of the feedstock material prior to REO extraction. The roasting process will include a scrubber system. The leaching, extraction and treatment portion of the process will be conducted indoors with fume hoods and positive (fan-driven) ventilation constructed over processing equipment to a central stack discharge. The solvent extraction portion of the process will have a separate fume hood and fan system that will report to a vapor phase activated carbon canister before being routed to the central discharge. The activated carbon system will remove all organics prior to discharge.

1.1.5.6.4 Surface and Groundwater protection measures

Coal mine rejects storage, crushing, grinding, milling and roasting will be conducted under shed-styled shelters (as appropriate) adjacent to fully-enclosed metal-clad buildings. All these facilities will be mounted on substantial pre-existing concrete pavement. Water run-off from this portion of the system will be captured and treated as needed in a pre-existing collection basin prior to discharge.

1.1.5.6.5 Spill Prevention Control and Countermeasure Plan (SPCC)

The leaching of the roasted feedstock and subsequent solvent extraction of the REE compounds will be conducted indoors. All the processes will be conducted on sealed, impermeable concrete surfaces. Liquid tight berms will be constructed within the leaching and solvent extraction process areas. The volume contained in each berm area will be equal or greater than the volume of liquid contained in the process equipment within the bermed area. Each bermed area will contain a collection sump and pump. Any process spillage can be pumped back to the process, to water treatment, or to temporary storage.

1.1.5.6.6 Stormwater Pollution Prevention Plan and Stormwater Management Plan

Stormwater Pollution Prevention Plans *(SWPPPs)* and Stormwater Management Plans *(SWMPs)* are site-specific, written documents that:

1. Identify potential sources of stormwater pollution on a construction, industrial and/or municipal site;
2. Describe stormwater control measures and Best Management Practices *(BMPs)* that will be used to reduce or eliminate pollutants in stormwater discharges from the project site; and
3. Identify the procedures the operator of the project site will implement to comply with the terms and conditions of the site-specific general permit.

In anticipation of transitioning into Phase 2 of this project (Pilot Plant operations), steps were taken to compile a formal draft SWPPP for the site at 610 Trus Joist Lane, Chavies, Kentucky. However, the completion of this relatively extensive "Draft" document (along with its exhibits) was halted upon an investigative/researcher team consensus that the project should not move forward into Phase 2.

1.1.5.7 Best Management Practices (BMPs)

1.1.5.7.1 Operational Source Control BMP

1.1.5.7.1.1 Generally Applicable BMPs:

1. Good Housekeeping
2. All Loading/Unloading Areas
3. Tanker Truck Transfer Areas to Above Ground Storage Tanks
4. Roof/Building Drains:
5. Preventive Maintenance
6. Spill Prevention and Emergency Clean-up
7. Employee Training
8. Inspections, Reporting, and Recordkeeping
9. Inappropriate (Non-permitted) Discharges

1.1.5.7.2 Structural Source Control BMPs

1. Generally Applicable Structural Source Control BMPs:
2. All Loading/Unloading Areas:
3. Loading and Unloading Docks:
4. Tanker Truck Transfer Areas to Above Ground Storage Tanks

1.1.5.7.3 Treatment BMPs

The REE Pilot Plant will be in an existing industrial complex adjacent to the Coal Fields Industrial Park in Perry County, Kentucky (formerly the Trus Joist Industrial Facility). Stormwater runoff and associated erosion and sediment control is provided by the East and South Stormwater Ponds for the existing facility. The proposed Pilot Plant will be situated either inside the confines of Building #5 or immediately adjacent to it. All areas associated with Pilot Plant construction are either covered by roof or concrete pavement.

Therefore, the existing runoff reporting to the East and South Stormwater Ponds is unchanged by construction of the proposed pilot plant. No additional treatment BMPs should be required.

1.1.5.7.4 Erosion and Sediment Control BMPs

Stormwater runoff and associated erosion and sediment control is provided by the East and South Stormwater Ponds for the existing facility. The proposed Pilot Plant will be situated either inside the confines of Building #5 or immediately adjacent to it. All areas associated with Pilot Plant construction is either covered by roof or concrete pavement. Therefore, the existing runoff reporting to the East and South Stormwater Ponds is unchanged by construction of the proposed Pilot Plant. No additional erosion and sediment control BMPs should be required.

1.1.5.8 Process Water Treatment

Process wastewater is expected to contain dissolved metals concentrations (primarily Fe, Al, Mn) and thus, treatment is likely. During the treatment process the pH of will be raised through the addition of NaOH. The raise in pH and the addition of aeration will facilitate the oxidation and precipitation of dissolved metals. A filter press will remove the precipitated solids from the water for subsequent solid waste disposal at a solid waste landfill. The treated water will be discharged into the municipal sewer system serving the site.

The organic compounds involved in the solvent extraction process will not be discharged but will be captured and recycled. Any spillage within the process area will be contained within the bermed enclosure and pumped back into the system. Alternatively, spillage can be routed through a liquid phase activated carbon canister to the water treatment system for discharge to the municipal sewer system.

1.1.5.9 Environmental Monitoring

In addition to the testing conducted in association with the disposal of liquid and solid waste, portable instrumentation will be utilized on-site to assure proper operation of environmental control measures.

1.1.5.10 Environmental Management Plan Components for Phase 2

The Environmental management plan for the operation of the Phase 2 operation will include the following components:

- Environmental Health and Safety Training Program
  - OSHA training of plant workers
  - Site-specific safety training of workers on the process and its hazards
  - Record Keeping (MSDS, Worker hours, etc.)
- Spill Prevention Control and Countermeasures
  - Training of all employee on the project SPCC plan
  - On-site spill response equipment and supplies
- Surface Water Protection Plan
  - Training of all employees on SWPPP plan
  - Process environmental monitoring (treated water)
  - Stormwater control and monitoring (Crushing, Grinding, Roasting Area)

- > Air Quality Protection Plan
    - Monitoring, operation reporting in compliance with Air Permit
    - Monitoring and record keeping of Air Phase Activated Carbon system

1.1.6 Pilot Plant Product Market Assessment and Purchase Agreement Sample

1.1.6.1 General Discussion of REE Demand Structure

Overall global REE demand is probably in the range of 120,000-170,000 metric tons *(mt)* per year as REOs. The reason that the range is so wide is that there is a substantial amount that enters unofficially or through the black market and is, thus, exceedingly difficult to quantify. In addition, each REE has its own market demand and dynamics.

Each element's market is affected in various ways by demand from a variety of applications and uses, as well as substitution and increasing efficiency in the use of elements within applications. Each of these markets fluctuate, in terms of supply, demand and price.

It is important to understand that the distribution of the rare earths as they are extracted does not match the demand of the marketplace. Certain elements are often stockpiled and sold intermittently.

1.1.6.2 General Discussion of the Supply Structure

China is the main producer, processor, supplier, and consumer of REE materials in the global marketplace. REE mining and processing in China generally (though not always) takes place in four main regions: 1) Inner Mongolia, 2) Sichuan, 3) Shandong, and 4) South China.

In addition to the REE material mined in-country, China also imports a large amount of REE material to be processed. Feedstock is imported from Southeast Asia, Africa, and the Americas. Some of this feedstock enters the country legitimately, but much of it comes in illegally or is mislabeled in order to pass through customs as it is illegal to import certain REE materials that have a history of being associated with high radioactivity levels[15].

As mentioned above, there are numerous feedstock sources outside of China, and much of this material makes its way to China for processing, as there are few processing facilities outside of China in large-scale operations.

Notable quantities of REE materials come from **Lynas Corporation** ***(Lynas)***'s Mount Weld deposit in Western Australia. The ore is mined in Australia, made into a mineral concentrate, and then shipped to Lynas' facility in Malaysia for further processing.

There are mining operations in Russia (as a byproduct), Africa, and a number of other places worldwide.

---

[15] Rare earth minerals are processed primarily from ores and minerals that naturally contain uranium and thorium. Normal processing activities for rare earth minerals typically includes the separation and removal of uranium and thorium. However, that process is not always successful and some rare earth concentrates may thus also incorporate aggregations of uranium and/or thorium that exceed national and international limits or result in the imposition of severe regulatory constraints on subsequent process-tailings disposal. Heavy mineral sands *(HMS)* operations can also produce byproduct REE concentrates, but these are often highly radioactive. See also discussions on Technologically Enhanced Naturally Occurring Radioactive Material *(TENORM)*.

There are a number of junior mining companies (most listed on the Toronto Venture Exchange or Australian Stock Exchange) pursuing rare earths as well.

1.1.6.3 Specific Discussion of the Market for the Pilot Plant Products

The following discusses the market demand for gadolinium (Gd), dysprosium (Dy), and yttrium (Y), the proposed products of a future Pilot Plant based on the findings of Project MMA 29956.

1.1.6.3.1 Gadolinium (Gd) Market Environment

Demand for gadolinium is fairly inconsistent. Buyers do not generally make purchases on a regular basis, but as needed, and often with large gaps of time (sometimes quarters or years) between purchases. As with all REE markets, particular specifications are required for each end user. Although, there are a number of niche markets for Gd. Two key markets for gadolinium are as follows.

1.1.6.3.1.1 Gd as a Medical Contrast Agent for Magnetic Resonance Imaging (MRI)

Gadolinium is a preferred contrast agent for MRI procedures, being used in approximately 30 million procedures per year. It is usually taken intravenously. There is considerable controversy regarding the safety of Gd as a contrast agent, but it still maintains market position.

1.1.6.3.1.2 Gd as a Nuclear Reactor Control

Because Gd has a high thermal cross section, it is used as a neutron poison in nuclear moderator systems. Gd is processed into nitrate form for this application.

1.1.6.3.2 Dysprosium (Dy) and Industrial Magnet Production

The primary use of Dy is in neodymium-iron-boron (*NdFeB*) permanent magnets as it enables retention of magnetic capabilities at higher temperatures. While the Dy content in NdFeB may have declined, the magnet market continues to grow, and demand for Dy is steady.

1.1.6.3.2.1 Dy in Permanent Magnets

Before the REE crisis of 2010-2011, Dy content in NdFeB magnets was as high as 8-11-percent, depending on the manufacturer. Because of excessive pricing (Dy reached almost $3000/kg in 2011), manufacturers quickly incorporated new techniques, such as grain boundary diffusion, to reduce the amount of Dy required. With these efforts, Dy content was reduced to 2 to 3-percent, and in some cases eliminated entirely. As prices dropped and then stabilized after the crisis, the market for Dy has also stabilized.

1.1.6.3.3 Yttrium (Y) Market Trends

Overall demand for yttrium has declined since the significant reduction in usage in the phosphor market due to the transition from fluorescent lighting to LEDs, which use far less REEs, by volume. Besides the phosphor market, yttrium is also used increasingly in the ceramic market, as well as in lasers.

1.1.6.3.3.1 Y and Phosphor Compounds

Yttrium is used for phosphors in the lighting industry, including fluorescent (shrinking market), and LEDs (growing market, but uses less REEs).

1.1.6.3.3.2 Y and Ceramics Products

Yttrium is used in ceramics across a number of applications, perhaps the most significant of which is the yttria-stabilized zirconia *(YSZ)* market. YSZ (found in the following figure) provides a material that is lightweight, exhibits very high strength, and resists wear and tear. It is used in dental implants, oil drilling, electronics, thermal barrier coatings *(TBC)*, among other uses.

1.1.6.3.3.3 Y and Laser Production

Neodymium-doped yttrium aluminum garnet (*Nd-YAG*) lasers are dominant in the solid-state laser market. These lasers first came to market in 1964, and they have held a prominent position since. There are other laser materials available, and there are a number of other rare earths used to dope YAG lasers, such as ytterbium, thulium, holmium, and erbium.

1.1.6.4 REE Price Structure

Prices for rare earths vary widely, even within the market for a particular element. Customer specifications and the processing required to meet these specs most often determine the final delivered price. Issues of concern for customers may include purity (usually 99.5 to 99.9999-percent) but may also include characteristics such as particle size and surface area. Purity is not just a matter of the purity level of the desired elements but also the identity and content of the deleterious elements.

Pricing services, such as AMI, are providing the market with insight into the prices of each element. These prices are gathered from buyers and sellers who conduct their business privately, and all have differing motives in reporting prices. There is no central exchange for these products, and there is no standard purchase agreement form. Each client has its own specifications, and, therefore, the pricing services generally serve as negotiating benchmarks, not final contract prices.

The REE market deals primarily in kilograms *(kg)* and metric ton *(mt)* and generally priced in US dollars *(USD)*. Please refer to the summary of the Techno-Economic Analyses for a recitation of the market prices used in this study.

1.1.6.5 Critical Factors Affecting the Rare Earth Market

There are many factors that affect the REE market, from global macroeconomic and political issues, to industry-specific issues regarding individual applications, substitution, and potential processing technologies. Key factors include, but are not limited to:

1. Geopolitical Issues
2. Trade Disputes
3. Territorial Disputes
4. Political Movements
5. Natural Disasters
6. Substitutions
7. New Technology in Applications
8. New Technologies in Processing
9. Individual REE Market Growth/Decline

1.1.6.6 Pilot Plant Products Sales Agreement Development

1.1.6.6.1 Establishment of Marketing/Tolling Agreements for Sales of REE Products

In order to become a supplier of REE products, a prospective producer must typically undergo a lengthy qualification process as demanded by the final end user. The effort can take from 2-24 months and may cost hundreds of thousands of dollars depending on the required customer specifications. Generally, end users do not want to undergo this process unless they know that the supplier has the potential to produce a substantial amount of product over a long period of time from an industrial-scale facility. Because the MMA 29956 program is a research project of very limited REE material output, it will not be possible to undertake the typical full-scale qualification.

1.1.6.6.2 Informal MMA 29956 Pilot Plant Product Off-Take Arrangements

Based on discussions with **Blue Line Corporation *(Blue Line)***[16], it is anticipated that the project may be able to sell its three targeted products, but most likely, at a discounted market price. For the purpose of Project MMA 29956, Blue Line has indicated that it has the ability to blend small batches of REE product into larger batches as long as Blue Line can ensure (through testing) that the material is consistent and will meet the required specifications of its facilities capabilities and those set by its customers.

1.1.7 Environmental Critique of the Proposed 1 Short Ton per Hour Pilot Plant

1.1.7.1 List of Major Sources of Environmental Risks

The major sources of environmental risks identified at this time with the operation of the proposed 1 short ton per hour pilot plant are:

1. Modification of Feed Materials
2. Concentration of Radioactive Materials
3. Release of Reagents
4. Water Quality Impact
5. Air Quality Impact

1.1.7.2 Controls/Mitigation Strategies Proposed for each Item of Concern

1.1.7.2.1 Modification of Feed Materials

The feedstock stockpile (coal preparation plant rejects) residence time at the Pilot Plant site is expected to be very short, and the quantities relatively small. Based on information provided by the coal preparation plant operator, leachates from the Fire Clay Seam rejects comply with water discharge standards.[17]

The larger publicly recognized concern, however, focuses on the solid rejects generated as a result of processing for the recovery of rare earth compounds and their subsequent long-term disposal. Given the information on hand, however, the investigative team anticipates that the roasting and leaching processes proposed for this project will greatly decrease the propensity of the solid rejects from the REE concentration process to contaminate water sources through a natural leaching process. Nevertheless, TCLP tests will be conducted on the solid reject materials to detect and manage any issues not yet identified.

---

[16] Blue Line Corp, 3443 East Commerce Street, San Antonio, Texas 78220-1322, Tel: +1 (210) 225-0400, http://www.bluelinecorp.com

[17] Personal communication with Craig Daniels, PG (TN) of KRP, August 23, 2019.

1.1.7.2.2 Concentration of Radioactive Materials

Since this project entails leaching and concentration of REEs from high ash content reject material generated by a Fire Clay Seam preparation plant, there is a concern that the radioactive compounds of uranium and thorium which are naturally present in extremely low concentrations in the high ash coal preparation plant rejects will be subject to concentration in the tailings and/or REE compounds generated by the REE beneficiation process. As a result of this possibility, this project will quantify and understand the risks presented by these two radioactive elements. The general approach will be:

1. Prevent the concentration of these elements where possible;
2. If concentrations are indicated, dilute the occurrence to background levels and admix with the process tailings;
3. If incorporation with tailings is not possible, dilution and capture through alternate means will be undertaken.

Results from the work of this project indicate that given the leaching processes proposed, a significant majority of the radionuclides remain with the feed material. In other words, uranium and thorium do not materially report to the cleaning circuit. This was validated by independent analysis of REE concentrate produced by the proposed roughing circuit. However, if concentrations of undesirable radioactive elements prove to be unavoidable before the application of abatement protocols, appropriate firms specializing in radioactive element monitoring will be contacted to provide oversight and as needed, monitoring of REE facility personnel.

1.1.7.2.3 Release of Reagents

Inadvertent release of reagents will be controlled in a number of methods both procedurally and operationally. The most likely mode of occurrence of inadvertent release of reagents will be through leaks and spills. Should they occur, a number of containment methods will be in-place and subsequently utilized to manage these occurrences. Containment facilities will be appropriately sized to comport with proven industry practice. They will be expertly installed, and routinely maintained so that overflow of the containment facilities themselves is improbable.

1.1.7.2.4 Elimination of Water Quality Impacts from Process-Water

Consistent with current small scale REE compound production procedures adopted for Phase 1 of project MMA 29956, all excess process water will be neutralized, treated and filtered prior to discharge. These small scale REE compound production tests reveal that excess process water is completely manageable. Among other things, the water was tested for priority pollutant parameters and all values were below regulatory threshold levels. That said, monitoring the pilot plant effluent will be included as part of the project mission and steps will be taken as warranted by the facts to assure that discharge water quality comports with regulatory standards. Periodically, laboratory test results of water samples will be verified by 3rd party laboratories to ensure water will meet the standards of the permitted disposal methods.

1.1.7.2.5 Elimination of Air Quality Impact

In all respects, adherence to environmental protection regulations will be a significant part of the operation of the Pilot Plant.

The primary concern of this project with regard to impacting air quality is related to CO_2, sulfur, and dust generated in course of the operation of a crushed ore roaster. In that regard, the exhaust from the roaster will be treated by both dry and wet scrubbing to remove acid-forming compounds and particulate matter.

Furthermore, flows from the process ventilation associated with the leaching component of the REE compound production process may be combined with the roaster exhaust scrubbing system.

Volatile organic compounds *(VOCs)* related to the kerosene found in the solvent extraction circuit will be captured through activated carbon filters.

1.1.7.3 Environmental Assessment/Environmental Impact Statement (EA/EIS) Response/Mitigation

The Health, Safety and Environmental *(HSE)* Control Plan is based on the "Six Sigma principles" and FMEA. This type of methodology is common with projects in manufacturing and ISO 9001 standards.

In terms of the Pilot Plant proposed for this project, the FMEA defines the following parameters.

1. Possible failure modes of the process;
2. Likely outcomes of those failures;
3. Likelihood of occurrence;
4. Possibility of detection;
5. Severity of the occurrence.

Whereas the FMEA explores and defines potential failures and their effects, the HSE Control Plan defines the methods that the process controls. Control plans form the foundation of SOPO development and risk mitigation for this project.

All regulatory submittals required to address the topic of a categorical exemption (not requiring an EA/EIS[18] or mitigation) may be required. Nevertheless, the project managers understand that an EA of the operational design and environmental management plan may be required by authorities having jurisdiction over that matter.

<u>1.1.8 Benchmark Process Audits</u>

The Statement of Project Objectives included the performance of benchmark process audits to provide:

1. An understanding of industrial "best practices" in similar plants and
2. The ability to test design assumptions against operational experience.

The benchmark audits were to include:

1. An operating REEs producer utilizing a similar proposed flowsheet;
2. Toll processor; and
3. Feedstock provider.

---

[18] Environmental Assessment/Environmental Impact Statement (EA/EIS)

1.1.8.1 Deferred Audit of a USA-Based Toll Processor

An audit of a toll processor was not undertaken by the MMA 29956 team primarily because of the lack of refining capacity in the United States. For instance, during the course of executing Project MMA 29956, the researchers became aware that only MP Materials, **Rare Earth Salts**[19], and **Ucore Rare Metals, Inc. (*Ucore*)**[20] were known to possess the capacity needed for processing REEs for the improvement of purity. In pursuit of that matter, a number of discussions were held with Ucore. However, Ucore would not allow research team members to visit its REE processing facility in Salt Lake City, Utah. Next, although it was originally proposed that Blue Line would serve as the toll refiner for the project team it was discovered through conversations with Blue Line that the company did not possess the required in-house refining capability to sustain a tolling operation. Rare Earth Salts was not visited as their process differed from the circuit proposed by the team. (Due to process economics requiring vertical integration, the team decided to incorporate refining circuits as part of the project design effort, a concept not compatible with Rare Earth Salts.)

1.1.8.2 MP Materials, Mountain Pass, California (Pending REE Producer and Feedstock Provider)

Members of this research team spent three days in February 2018 at the **MP Materials *(Mountain Pass)*** rare earth mine and REM concentrating facilities located near the unincorporated community of Mountain Pass in San Bernardino County, California. The Mountain Pass operation is comprised of an open cast REE mine, physical concentration plant, REE recovery plant and a refining circuit that produces individual REE concentrates. It has been operated by privately held **Molycorp Minerals LLC *(Molycorp)***, when the operation filed for Chapter 11 bankruptcy. Affiliates of two investment fund advisors and a Chinese entity took over the operation in July 2017 under the name of MP Materials[21]. The mine operating entity is listed as **MP Mining Operations, LLC (*MPMO*)**[22]. At the time of MM&A research team's visit in February 2018, MPMO was entering its start-up period. Notwithstanding these events, the visit satisfied the objectives of benchmark auditing both from the perspective of a REE producer and a feed stock provider.

Although the team conducted and documented its learnings and discussions, they were legally bound as a covenant to the visit not to release the details. Nevertheless, the trip proved to be both essential and very instructive. The trip to the processing facilities of the MP Materials mine was essential to the team's understanding of the practical implications of feedstock leaching and subsequent recovery of REEs from the leachate using solvent extraction. Significant insights collected from the Mountain Pass visit were applied to the flowsheet development of Project MMA 29956.

---

[19] Source: Rare Earth Salts is a privately held industrial and applications technology company focused on the separation and refining of all 16 REEs to high purity from various feedstocks; 5331 Element Ave. Beatrice, NE 68310, Tel: +1 402-806-4400, https://rareearthsalts.com/

[20] Source: Ucore Rare Metals Inc. is a development-phase company focused on rare metals resources, extraction and beneficiation technologies; 210 Waterfront Drive, Suite 106 Bedford, Nova Scotia, Canada B4A 0H3 Tel: (902) 482-5214, www.ucore.com

[21] Sources: https://en.wikipedia.org/wiki/Mountain_Pass_rare_earth_mine and https://mpmaterials.com/

[22] Sources: Nuclear Regulatory Commission, "Order Approving Direct Transfer of Two Export Licenses", November 27, 2017, https://www.nrc.gov/docs/ML1732/ML17325B702.pdf and "Mountain Pass Sells for $20.5M" https://www.mining

1.1.8.3 Arizona Pilot Plant (Hydro- Met Facility, Tucson, Arizona, Potential Feedstock Producer)

A Pilot Plant located at 6701 South Wilmot Road, Tucson, Arizona 85756-2930 was found to be suitable for refit and revitalization for the extraction of REEs. The current owner of the plant is **Cimetta Engineering & Construction Co. Inc.** ***(Cimetta Engineering)***[23]. Mr. Cimetta has used the comminution and the hydrometallurgical portions of the plant for his engineering consulting business. The facility (the "Arizona Plant")[24] is for sale.

Further research showed that the Pilot Plant utilized a modified version of the "Caron" process for the reduction and extraction of Ni from nickel laterite. The process is included in the book Extractive Metallurgy of Nickel, Cobalt and Platinum Group Metals by Crundwell et. al.

The researchers elected to undertake several visits to the Arizona Plant. The visits identified the equipment and process, determined plant suitability and affirmed the availability of the plant for purchase. The visits also sought to determine, to the extent possible, the detailed engineering parameters associated with the Pilot Plant and its appraised value. In aggregate, both visits, bolstered by the assistance of a very cooperative owner of the facility, provided surprisingly detailed knowledge of relevant process units and circuits as well as potential access to equipment that could be used in Phase 2 of the project.

A significant appeal of the Arizona Pilot Plant was the existing particle sizing equipment, properly scaled equipment and the reduction in lead time needed for acquisition if the project proceeded to Phase 2 of the MMA 29956 project. Further, in the weeks after the initial visit, the owner was able to locate a significant amount of engineering documentation.

During the second visit of the process audit performed on the Arizona Pilot Plant, the team was able to gain access to binders, index cards and drawings describing much of the original costs, specifications and vendors of the equipment. This data was compiled as the basis for a detailed equipment selection. Since all the plant equipment was designed and sized for the correct capacity, the data is valuable to the team as a starting point for the plant equipment costing. It also provides the means for estimating shipping costs.

As part of the second visit to the Arizona plant to continue the benchmark audit process, an appraisal was conducted by Roger A. Daugherty, President of **Darco Energy Management Corporation** ***(Darco)***. The estimate included the cost to replace the equipment and fair market value at removal. The definition of values was uniform with those used by the **American Society of Appraisers (*ASA*).** The assets appraised were the personal property and engineering drawings. The appraised values are effective February 12, 2019.

---

[23] Cimetta Engineering and Construction Co., Inc., is listed as a "general contractor" with an address at 6701 Wilmot Road, Tucson, AZ. *Source: Better Business Bureau.*

[24] Cimetta Engineering & Construction Co. Inc., 6701 S. Wilmot Road Tucson, Arizona 85756, Tel: +1 (520) 574-2930.

1.2 Acknowledgments

The following individuals contributed materially to this undertaking, either by direct participation in the analytical procedures or in the pursuit of a wide variety of directly associated supporting tasks. Their efforts, and those of their staff, in support of this project is particularly commendable.

A host of other talented individuals, too many to be listed here, also played a wide variety of indirect activities that affected the course of this investigation. These are recognized below through their company names.

1.2.1 Principal Investigators

Barker, Steve, Kentucky River Properties, LLC, President, CEO, and General Counsel, MMA 29956 Principal Liaison, Exploration, and Facilitator (Hazard No. 4 Seam Rejects Feedstock), Lexington, Kentucky

Cox, Clinten, The Anchor House, President, MMA 29956 Market Analyst, Evanston, Illinois

Garland, Jared, Cumberland Mine Service, Inc., Project Engineer, MMA 29956 Construction Estimator, Totz, Kentucky

Honaker, Rick, Ph.D., P.E., University of Kentucky, Professor of Mining Engineering, MMA 29956 Principal Investigator and Pilot Plant Manager, Lexington, Kentucky

Keim, Steven, Ph.D., P.E., Marshall Miller & Associates, Inc., Senior Vice President, MMA 29956 Authorized Representative and Project Director/Principal Investigator, Blacksburg, Virginia

Lord Ted, C.P.G., Marshall Miller & Associates, Inc., Senior Geologist, MMA 29956 Project Geologist, Bluefield, Virginia.

Luttrell, Gerald, Ph.D., P.E., Virginia Tech, Massey Professor, MMA 29956 HHS Expert and Flow Sheet Analyst, Blacksburg, Virginia

Meek, Allen, P.E., Marshall Miller & Associates, Inc., Senior Project Manager, MMA 29956 Environmental Protection Expert, Nicholasville, Kentucky

Naumann, Hans E., P.E., Marshall Miller & Associates, Inc., Principal Engineer, MMA 29956 Mining Engineer, Lexington, Kentucky

Noble, Aaron, Ph.D., P.E., Virginia Tech, Associate Professor, MMA Project 29956 Financial Analyst, Blacksburg, Virginia

Werner, Joshua, Ph.D., P.E., Mechanical Engineer, Assistant Professor, University of Kentucky, Department of Mining Engineering, Lexington, Kentucky

Whittaker, Matt, P.E., Marshall Miller & Associates, Inc. Senior Principal – Civil Engineering, MMA 29956 Permit Specialist, Bluefield, Virginia

Yoon, Roe-Hoan, Ph.D., P.E, Virginia Tech, Director of CAST and Professor, MMA 29956 HHS Expert, Blacksburg, Virginia

1.2.2 Participating and/or Supportive Entities

> Alliance Resource Partners, LP/Alliance Coal, LLC/Webster County Coal LLC/Dotiki No. 3 Shop. Providence, Kentucky

> Arq USA – Corbin Project, LLC, Corbin, Kentucky

> Blackhawk Mining, LLC (Formally listed on permits as Blue Diamond Coal Company), Lexington, Kentucky

> Blue Line Corporation, San Antonio, Texas

> Cimetta Engineering and Construction Co, Inc., Tucson, Arizona

> Darco Energy Management Corporation, Bluffton, South Carolina

> Ferroglobe dba Alden Resources LLC, Corbin, Kentucky

> KGS, Lexington, Kentucky

> Lower Kittanning Seam Mining and Preparation Plant, West Virginia

> Mineral Labs, Inc, Salyersville, Kentucky

> Mineral Refining Company, LLC, Richmond, Virginia

> MP Materials, Mountain Pass, California

> Nex-Gen Industries, Chavies, Kentucky

> Outotec, USA, 8280 Stayton Dr., Jessup, Maryland

> Pocahontas No. 3 Coal Seam Mining and Preparation Plant, West Virginia

> SGS Canada Inc., Lakefield - Ontario, Canada

> West Virginia University, Morgantown, West Virginia

Chapter 2.0

FEEDSTOCK SAMPLING, IDENTIFICATION & CHARACTERIZATION

Chapter 2.0
Feedstock Sampling, Identification & Characterization

TABLE OF CONTENTS

2.1 Executive Summary **1**
2.2 Introduction **2**
2.2.1 Precedent Investigations 2
2.2.2 The Fire Clay Seam Initiatives 3
2.3 Geology: Fire Clay Seam Sample Collection, Site Characterization, and Deposit Modeling **4**
2.3.1 Previous Fire Clay Seam Geological Investigations 4
2.3.2 Stratigraphy 4
2.3.3 Property Description 5
2.3.4 Data Sources 6
2.3.5 Modeling Methods and Features 7
2.3.5.1 Roof and Floor Material (Map 1) **8**
2.3.5.2 Interval Partings between the Hazard No. 4 Coal and the Hazard No. 4 Flint Clay (Map 2) **8**
2.3.5.3 Hazard No. 4 Flint Clay (Map 3) **8**
2.3.5.4 Interval Partings between the Hazard No. 4 Flint Clay and the Hazard No. 4 Jack Rock Coal (Map 4) **8**
2.3.5.5 Floor Material (Again, Map 1) **9**
2.4 Resource Estimation, Concentrations, and Tonnage **9**
2.4.1 Delineation of Zones of Confidence and Related Assumptions 9
2.4.2 Estimation of Tons of Non-coal Lithologies 9
2.5 Index of Maps and Exhibits **12**
2.6 References Cited **12**
2.7 Acknowledgments **13**

TABLES (IN THE TEXT)

Table 2-1: Feedstock Source Evaluation Sequence 2
Table 2-2: Basic Summary of In-Place Non-Coal Rock and REY Short Tons 10
Table 2-3: Distribution of In-Place Y+REE Within Future Mining in Mine Nos. 81 and 89 and the "Study Area" 11

FIGURE (IN THE TEXT)

Figure 2-1: Potential REE Resource Material from a Mining Bench (not-to-scale) 7

EXHIBITS

MAPS

2.1 Executive Summary

This report details the modeling methods and tonnages of in-place non-coal reject material from the Hazard No. 4 coal seam on a property (*Property*) owned by **Kentucky River Properties, LLC (*KRP*)** and operated by **Blackhawk Mining, LLC (*Blackhawk*)** in Leslie County, Kentucky. The purpose of this study is to classify and quantify the in-place coarse reject material from coal-mining activities which *may* yield extractable and economic quantities of rare earth elements (*REEs*).

The Property has three areas of interest:

1. mine projections from the active Mine No. 89;
2. mine projections from the active Mine No. 81; and
3. a future mining area, also known throughout this report as the "study area."

The "study area" does not yet have specific mine projections, but the coal has been modeled and has some reject areas defined. The presumed coarse reject material is comprised of several stratigraphic and lithologic intervals:

1. roof material;
2. partings between the Hazard No. 4 coal seam and the regionally persistent Flint Clay;
3. the Flint Clay;
4. partings between the Flint Clay and the Hazard No. 4 Jack Rock coal horizon (where present); and
5. floor material.

Rock tonnages for each of these individual horizons have been estimated and converted to tons of rare earth elements (REEs).

The table below summarizes the basic findings of this geologic study.

Location	In-Place Non-Coal Rock Tons	Sc	Y	Gd	Dy	Total REY Tons
		(Tons In-Place)				(Whole Sample Basis)
Total Mine No. 81 (Measured + Indicated + Inferred)	2,338,000	40	73	24	18	832
Total Mine No 89 (Measured + Indicated)	491,000	8	15	6	4	150
Study Area, Subtotal KRP Controlled (Measured)	45,699,000	643	1,381	489	363	15,520
Study Area Subtotal Non KRP (Measured)	13,196,000	186	396	140	104	4,459
Study Area - Total All Holes & Ownership (Indicated)	56,619,000	791	1,733	611	458	19,408
Study Area - All Tonnages and Ownership (Measured + Indicated)	**115,513,000**	**1,620**	**3,510**	**1,240**	**926**	**39,387**
All Areas - All Tonnages - All Ownerships	**118,343,000**	**1,667**	**3,598**	**1,270**	**947**	**40,370**

Upon receipt of an appropriate request a multi-tabbed Excel workbook can be provided to the reader. That workbook presents a significantly more detailed examination of all the REE elements identified for the subject "study area". Portions of that workbook are reproduced at the end of this section.

These short tons are in-situ and are reported as the total short tons of REEs + Y. They are not reflective of corresponding oxides.

Because of unfavorable economics discussed in Techno-Economic Chapter of this report, REE tons were not classified as "reserve". As such, in-place resources have been presented, analogous to resource terminology as utilized by international (Joint Ore Reserve Committee (JORC) and Canadian National Instrument 43-101) resource and reserve reporting standards.

Future drilling and testing *could* reclassify some or all of these resources as reserves, assuming favorable REE concentrations, plant efficiencies, and economics. **Marshall Miller & Associates (*MM&A*)** recommends an aggressive exploration campaign to further define the distribution and concentration of potentially extractable REEs.

2.2 Introduction

2.2.1 Precedent Investigations

The investigations leading up to the discussion of the Fire Clay seam in this report are summarized according to historical order in the following table.

Table 2-1: Feedstock Source Evaluation Sequence

			Location			MSHA ID	Coal Seam	Comments
Sequence	Company	Facility Name	Geographical Description	North Coordinates	West Coordinates			
1	Northern Appalachian Coal Producer	Mine	West Virginia	39° 20’N	79° 59’W	Multiple	Lower Kittanning	The initial collaborative effort could not be supported by the coal mining and processing Company.
		Processing Plant	West Virginia	39° 20’N	79° 59’W	Multiple		
2	Southern Appalachian Coal Producer	Mine	West Virginia	37° 44’N	81° 14'W	Multiple	Pocahontas No. 3	After testing refuse and channel sample response to HHS cleaning and REE content, this initiative was terminated in light of the fact that the middlings material could not be processed to produce a clean coal product with desired quality.
		Processing Plant	West Virginia	37° 44’N	81° 14'W	Multiple		
3	Ferroglobe (d.b.a. Alden Resources, LLC)	Gatliff Tipple	Gatliff, Whitley County, KY	36° 40.698’N	84° 1.051'W	15-09938	Blue Gem and Jellico	Multiple segments of the plant circuit were sampled and analyzed for standard coal quality parameters, response to HHS cleaning and REE content. This initiative was terminated because of relatively low REE content in the samples tested.
		Mine #3, Bain Branch	Bryants Store, Knox County, KY	36° 46.748’N	83° 53.239'W	15-17691	Blue Gem	
		Mine #5, Log Cabin	Bryants Store, Knox County, KY	36° 46.684’N	83° 54.824'W	15-18426	Blue Gem	
		Contractors	Not Available	Not Available	Not Available	N.A.	Jellico	
4	Arc Corbin, LLC (d.b.a. Arq Limited)	Corbin Project, LLC	Corbin, Knox County, KY	36° 56.048’N	84° 5.110'W	15-02134	Mixed Coal Preparation Plant Refuse	A series of HHS studies were conducted in small grab samples of the refuse material the initiative was dropped when collaboration with the host company was not successful
5	Blue Diamond Coal Company and Blue Diamond Mining LLC, (d.b.a. Blackhawk Mining, LLC)	Blue Diamond No. 76 Plant (a.k.a. Leatherwood)	Slemp, Perry County, KY	37° 03.967’N	83° 7.233'W	15-16520	Fire Clay	Participation by both Kentucky River Properties LLC (Lessor to Blue Diamond) as well as the Plant and mine operator (d.b.a. Blackhawk Mining LLC) resulted in an aggressive campaign of drilling, channel sampling, plant refuse sampling and testing.
		Blue Diamond Mine No. 81	Smilax, Leslie County, KY	37° 8.152’N	83° 14.497'W	15-12753		
		Blue Diamond Mine No. 89	Viper, Perry County, KY	37° 10.347'N	83° 6.683'W	15-19405		

2.2.2 The Fire Clay Seam Initiatives

This report details the modeling; methods and tonnages of in-place non-coal reject material from the Hazard No. 4 (a.k.a. Fire Clay) coal seam on a Property owned by **Kentucky River Properties, LLC *(KRP)*** and operated by **Blackhawk Mining, LLC**[1] in Leslie County, Kentucky. The purpose of this study is to classify and quantify the in-place coarse reject material from coal-mining activities which may yield extractable and economic quantities of REE.

The Property has three areas of interest:

1. mine projections from the active Mine No. 89;
2. mine projections from the active Mine No. 81; and
3. a future mining area (which does not yet have specific mine projections).

The presumed coarse reject material is comprised of several stratigraphic and lithologic intervals:

1. roof material;
2. non-coal partings from within the Hazard No. 4 upper coal seam;
3. partings between the Hazard No. 4 coal seam and the regionally persistent Flint Clay;
4. the Flint Clay;
5. partings between the Flint Clay and the Hazard No. 4 Jack Rock coal horizon;
6. non-coal partings within the Hazard No. 4 Jack Rock coal; and
7. floor material.

Tonnages for each of these individual horizons have all been calculated and will later be used to estimate in-place REE content using the results from several sample sites on the Property.

Mine No. 81 contains 2,338,000 tons, Mine No. 89 contains approximately 491,000 tons, and the future mining site contains 115,343,00 tons of combined non-coal material. Dr. Rick Honaker from the UK College of Engineering will apply appropriate REE concentrations and estimate in-place REE content.

As a United States strategic initiative, the **US Department of Energy *(DOE)*** is looking at the possibility of establishing economically viable sources of REEs at home to help liberate the dependency on foreign nations, such as China, for these crucial industrial elements. Several coal seams in the eastern United States are documented to contain elevated concentrations of REEs. The Hazard No. 4 coal bed in eastern Kentucky is the subject of this investigation. This project is a combined effort from **Marshall Miller & Associates *(MM&A)***, DOE, KRP, UK College of Engineering, and the study of several **Virginia Tech *(VT)*** graduate studies.

MM&A's involvement in this phase of the project is to classify and quantify the potentially REE-bearing in situ coarse reject material of the Hazard No. 4 coal seam as a biproduct of coal-mining activities on the KRP Property, operated by Blackhawk in Leslie County, Kentucky. This study is *not* a remodel of KRP coal reserves. It serves only to classify and quantify the non-coal material within the previously modeled

---

[1] Blackhawk Mining, LLC is the "current controller" listed in **Mine Safety and Health Administration *(MSHA)*** reports. **Blue Diamond Coal Co. *(Blue Diamond)*** is listed by MSHA as the "operator" of the coal mines as well as coal handling and preparation plant referenced herein. Blackhawk and Blue Diamond are often interchangeably recited in the industry. However, in deference to local nomenclature, this geologic report will refer to "Blackhawk" as the mine and plant "operator".

reserve boundaries. This report breaks down the non-coal fractions within the mining section by stratigraphic interval and reports the corresponding tonnages. REE concentrations will be applied to these figures by Dr. Rick Honaker from UK to approximate the likely REE tonnages of in-place material.

2.3 Geology: Fire Clay Seam Sample Collection, Site Characterization, and Deposit Modeling

2.3.1 Previous Fire Clay Seam Geological Investigations

The presence of REEs in various Appalachian coal horizons has long been the subject of study, primarily from the academic realm. Several key investigators have devoted many years and resources to the documentation of the lateral and vertical distribution of REEs in and around coal beds. For further reading, a list of relevant academic publications is found in the works cited section at the end of this Section.

In addition to the body of published academic works, a team of senior students in Virginia Tech's Mine Engineering program researched, developed, and wrote a report exploring the practical viability of REE extraction from previously mined partings and reject material found in refuse impoundments on the Property.

MM&A previously modeled and calculated reserves for the Property for the **James River Coal Company**[2] ***(James River)*** in 2004. That report is titled "*Evaluation of James River Coal Company Reserves as of March 31, 2004*." With permission from the current mine operator and land company, much of the correlative work was carried over from that project to the present assignment and in doing so, provided a solid baseline for adding correlations for new drill holes and datapoints. It must be noted that the scope of work on this project deviates from standard "Reserve" and "Reserve-Base" studies. Whereas the James River reports focus on recoverable coal tonnages, this exercise focuses primarily on characterizing and quantifying REE-bearing reject material within the mineable Hazard No. 4 coal seam.

2.3.2 Stratigraphy

The Hazard No. 4 (Fire Clay) coal seam is a Middle Pennsylvanian aged member of the Hyden Formation of the Breathitt Group. The coal is located stratigraphically between the Kendrick Shale Member and the Magoffin Member. The Hazard No. 4 coal typically contains upper and lower horizons, separated by a regionally extensive Flint Clay and shale parting known as the Jack Rock Parting (Greb, Hiett, Weisenfluh, Andrews, & Sergeant, 1999). The Flint Clay is a distinctively hard, brown-to-gray clay (tonstein) that is volcanic in origin (Greb et al., 1999; Eble et al. 1999). To briefly describe the deposit and geological events involved in REE deposition, roughly 311 million years ago, a volcano erupted somewhere east of the study site and spewed ash over a large geographic region (Rice, Kosanke, & Henry, 1994). This ash fell on active peat bogs, percolated through it, and formed what is known as "Tonstein," or "Flint Clay". This discrete layer of mostly kaolinite clays contains the altered remnants of the volcanic ash and serves as the original source material for the bulk of the REE content of interest to this project. Through various forces, such as burial, compaction, biological and geochemical processes, much of the REE content has leached out of the volcanic layer into the over- and underlying strata.

---

[2] James River is the past operator on the properties held by KRP. Blackhawk is the current mine operator.

Nomenclature of seam names is far from standardized across the region, but for modeling purposes, MM&A staff utilized the following seam-tag naming convention (See the figure of this chapter for graphic detail):

> The upper coal horizon is "Hazard 4" (H4).
> The Jack Rock Parting is "Hazard 4 Flint Clay" (H4FC).
> The lower coal seam is the "Hazard 4 Jack Rock" (H4JR).

The coal produced from the Hazard No. 4 seam is typically a high-volatile bituminous coal that is generally low in ash (averaging roughly 10-percent regionally) and sulfur content (averaging 1-percent) (Greb et al. 1999). The lower coal seam generally has higher ash content than the higher seam (Greb et al. 1999). The coal appears continuously across most of the coal field and is thickest and most continuous to the southeast toward Pine Mountain and thins to the northeast in a series of elongated pods separated by thin or absent coal. This trend is reasonably typical of major coal beds in the Eastern Kentucky Coal Field (Greb et al., 1999).

Mines in the area have historically mined coal seams within the Hyden Formation of the Breathitt Group including the Leatherwood, Alma, and Hazard No. 4 (Fire Clay) seams. The coal reserves within the project area are in the Hazard No. 4 seam. Coal from this seam has been widely mined and is usually one of the top producers of met-grade coal in eastern Kentucky.

Specific to this Property, the Hazard 4 coal (H4) is persistent. The Flint Clay appears nearly everywhere. There are, however, a few areas where the Flint Clay appears to be absent. It is unknown if the cause of the absence is through primary or secondary geological forces, or if the Flint Clay was misidentified by the exploratory drillers. Although the Flint Clay's origin is volcanic in nature and appears in consistent thicknesses across the property, MM&A opted to honor original lithologies, placing a zero-thickness "tag" where the Flint Clay was omitted from original logs.

Also relevant to this study is the presence or absence of the lower coal unit (Hazard 4 Jack Rock coal). Although this Jack Rock coal appears regionally, there is a large swath of land in the middle of the study area where it pinches out. On the reserve-base maps, these areas are marked by Hazard 4 Jack Rock "pinch out" lines. There are two such lines on the maps. The H4JR coal appears in the southern and northern parts of the Property but is missing from a large portion of the center of the Property. Historically, this lower coal unit is high in ash and brings down the overall quality of the coal. There seems to be, however an inverse relationship between coal quality and REE-recovery potential. Where present, the H4JR coal appears to be high in REEs and is more likely to release coarse fractions into the reject material from which REEs can be extracted.

2.3.3 **Property Description**

The project area is in the Eastern Kentucky Coal Field in the Central Appalachian Coal basin in Leslie County, Kentucky. Approximate location of the Property is 90 miles southeast of Lexington, Kentucky and 60 miles northwest of Kingsport, Tennessee. This study encompasses three areas of interest. On the northern part of the Property, Mine Nos. 81 and 89 are active and will provide the coarse refuse material for the REE project for the next few years. There are also several abandoned mines on the Property: the BL4 Mine to the south; Leeco No. 3 Mine; and Mine No. 74 South to the north and northeast. The future mining site is an area of approximately 27,850 acres, of which most of the land is owned and controlled by KRP. Within this future site, there are approximately 11,150 acres of modeled coal reserves, accounting for adverse tracts and seam thinning. MM&A did not independently verify land and mineral rights as part of this study. Underground mining methods of the Hazard No. 4 coal seam are employed on this Property.

2.3.4 **Data Sources**

The primary data for modeling the potential REE-bearing refuse material is the collection of drillholes from various owners and operators of the Property.

For correlation purposes, MM&A utilized an in-house drill hole database comprised of: 1) previously drilled exploration holes, carried over from the 2004 James River Coal Reserve study; 2) several exploration holes drilled since the 2004 MM&A report; 3) four recently drilled exploration holes (12/2018 – 01/2019); and 4) in-mine measurements. There have been 590 drill holes across the Property correlated and used for modeling.

Four recently drilled exploration holes commissioned primarily for this project (RELW18-01, 02, 03, and RELW19-01; circa 2018 and 2019) were all drilled within the future mining site and sampled both for coal quality and REE content. Sampling and testing included additional fractions from the floor underlying the Hazard No. 4 coal horizon.

Furthermore, two channel samples, taken in the proximity of the next two years of underground mining by Blackhawk, were also collected for the benefit of this project (RECH-01 and RECH-02, circa 2019). The carefully segregated samples included roof and floor strata that had been exposed during the underground coal mining process. All samples were individually analyzed for the traditional coal quality properties. Additional tests were undertaken to for REE content characterization purposes. These two channel samples supplement a channel sample (RECH-A) collected in 2017, also from the underground Hazard No. 4 coal seam mine of Blackhawk.

Lastly, Hazard No. 4 coal preparation plant refuse samples collected from the No. 76 Plant[3] (a.k.a. "Leatherwood") operated by Blackhawk were correlated to the appropriate sites in each of Blackhawk's underground mines by date marks on the mine progress maps maintained by Blackhawk.

REE distribution and concentrations relating to lateral extent and vertical mobility specific to this Property are still poorly understood due to the low number of coal seam horizon samples analyzed for REE content. In summary, the inventory of REE content sources includes:

> Nine coarse refuse samples.
> Three channel samples (which collected roof and flint clay parting in Blackhawk Mine Nos. 81 and 89).
> Four core samples drilled from the future mining site.

The few numbers of datapoints on the Property are *not* statistically significant, and any predicted REE content is to be classified as inferred resource. Additional sampling and testing of the Hazard No. 4 coal seam via drill hole (cores) and channel samples will be precursor activities to full commercial exploitation of the Property's Hazard No. 4 coals for REE extraction purposes.

---

[3] This coal handling and preparation plant is formally known as the No. 76 Plant of Blue Diamond Mining LLC, but it is operated by Blackhawk. The plant is located at the junction of Beech Fork and Leatherwood Creek (hence the name "Leatherwood") in Perry County, Kentucky.

2.3.5 Modeling Methods and Features

The main purpose of this exercise was to classify and quantify the potential REE-bearing coarse reject material from the mining face. The method of determining likely saleable tons of REE resource tonnages involves:

1. modeling the thickness of the non-coal strata;
2. calculating tonnages of rock from each of these strata;
3. applying appropriate REE concentrations to convert from tons of rock to tons of REE;
4. applying the conversion factor to convert the tons of REE to tons of REE oxides; and
5. applying appropriate mine and plant recoveries.

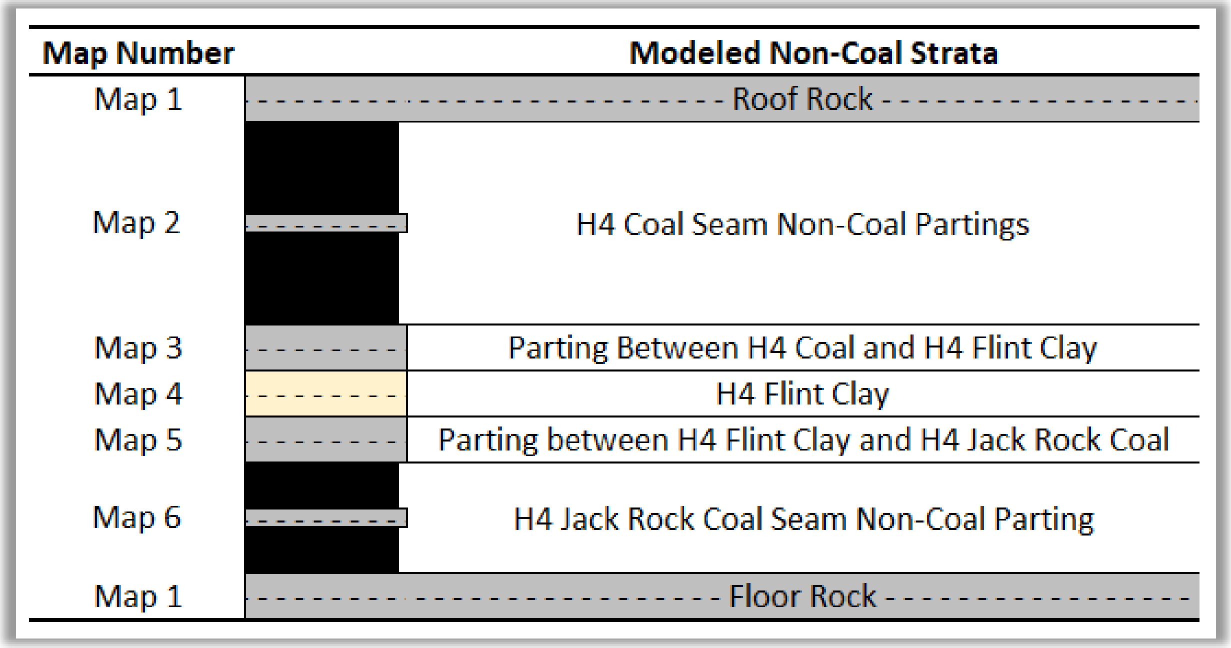
MM&A has modeled the deposit and reported short tons of REE. However, because of the unfavorable economics of this project at this time, the REE short tons were not classified as "reserve". Thus, in-place "resources" have been presented, analogous to resource terminology as utilized by international (Joint Ore Reserve Committee (JORC) and Canadian National Instrument 43-101) "resource" and "reserve" reporting standards.

The non-coal strata from within the mining face include:

1. roof material;
2. interval partings between the Hazard No. 4 coal and the Hazard No. 4 Flint Clay;
3. Hazard No. 4 Flint Clay;
4. interval partings between the Hazard No. 4 Flint Clay and the Hazard No. 4 Jack Rock coal; and
5. floor material (same thickness grid as roof material).

Thickness grids were generated for each of the potential REE-bearing non-coal units from within the effective mining section (*see the following Figure*).

Figure 2-1: Potential REE Resource Material from a Mining Bench (not-to-scale)



Notes: -Thicknesses not to scale
- Not all bore holes and in-mine measurements display each of these strata

These discrete horizons account for all the non-coal strata from within the effective mining section that are likely to end up in the coarse refuse after extraction and processing. MM&A opted to subdivide the non-coal strata to this extent to allow for maximum flexibility in assigning and assessing REE content based on trends in core- and channel-sample analyses. Exhibits included in the Appendix provide detailed cross sections cutting through the modeled areas and show the distribution of the various coal and non-coal strata in the mining face.

Carlson Software (formerly SurvCAD) was used to model and generate tonnage estimates. The Approximation/interpolation Based On Smoothing *(ABOS)* method of gridding[4] was employed in this model because it honored datapoints and grouped trends well. Resource estimates were calculated only from within reserve polygons from the currently operating mines (Mine Nos. 81 and 89) and a future mining site south of Mine No. 81. Geologic cross sections were generated to show trends and lithologies across the projected mining areas (see the exhibits of the Appendix).

The following are detailed descriptions and methods on each of the modeled thickness grids.

2.3.5.1 Roof and Floor Material (Map 1)

A minimum mining height was assumed to be 60 inches (5 feet). In cases where total seam was 56 inches or greater, it was assumed that 2 inches of floor and 2 inches of roof material would be taken. In cases where the total seam was less than 56 inches, the difference between 60 inches and the total seam was divided by two and attributed to the floor and roof respectively. There is a single thickness grid representing both the floor and the roof, which are counted twice in the resource tables in the attached Excel file. There is a heavy presence of sandstone above the coal seam; however, the immediate roof in the majority of the drill holes is mostly comprised of shaley lithologies.

2.3.5.2 Interval Partings between the Hazard No. 4 Coal and the Hazard No. 4 Flint Clay (Map 2)

The Flint Clay does not always lie directly beneath the H4 interval of the Hazard No. 4 coal seam. There are often shaley partings and other minor lithologies that separate the two seams. To capture this potential REE-bearing material in the resource calculations, a thickness interval was extracted between the H4 and the H4FC.

2.3.5.3 Hazard No. 4 Flint Clay (Map 3)

The Flint Clay is generally pervasive and predictable as it was deposited as a blanket of volcanic ash. Across the coal reserve projections, the thickness averages 0.42 feet. Some areas appear to lack Flint Clay lithology. As previously mentioned, this could be attributed either to primary or secondary geological processes or to a misidentification by an exploratory drilling team. The Flint Clay's original REE has mostly been leached into the overlying and underlying coals and clays, however it is material that will appear in the coarse refuse after production.

2.3.5.4 Interval Partings between the Hazard No. 4 Flint Clay and the Hazard No. 4 Jack Rock Coal (Map 4)

On occasion there are additional partings between the Flint Clay and the H4 Jack Rock coal seam. Quite often these partings are an extension of clay-rich lithologies such as fire clays, clay stones, and shales, with occasional occurrences of shale with coal streaks, sandy shale, and sandstones.

---

[4] Definition of gridding: the construction of a rectangular matrix of numbers from a set of scattered data point.

2.3.5.5 Floor Material (Again, Map 1)

Calculations for floor material are the same as those applied to the roof material. Lithology in the floor is different than in the roof. It is quite common for coals to be underlain by clay-rich strata known as fireclays. This deposit is no different. Typical lithologies for the floor material include Fire Clay, shale, shale with coal streaks, and occasional sandy units. Core and channel samples used for REE analysis demonstrated slightly different REE concentrations between the roof and floor material. While the modeled tonnages of rock are the same for the roof and floor material, the different corresponding REE concentrations were applied separately to the roof and the floor tonnages.

2.4 Resource Estimation, Concentrations, and Tonnage

2.4.1 Delineation of Zones of Confidence and Related Assumptions

Once thickness grids were generated for each of the strata, circles of influence were drawn around each of the datapoints, representing certain degrees of confidence in the thicknesses represented.

- Measured arcs encompass any area within ¼ mile (1,320 feet) of a recorded observation.
- Indicated areas encompass any area between ¼ mile and ¾ mile (3,960 feet) from a recorded observation.
- Inferred areas are anything beyond ¾ mile from a recorded observation.

SPECIAL NOTE: While tonnages are reported as measured, indicated, and inferred, the associated level of confidence only applies to thickness of the strata and the tons of rock therein. It is *not* any indication of the level of confidence in the concentrations or tons of REE contained therein.

A uniform density of 162 lbs./ft^3 was applied to each of the strata, as most lithologies represented are shale, flint clay, and fireclay.

The rock tonnage calculation assumed 100% mine recovery.

2.4.2 Estimation of Tons of Non-coal Lithologies

As mentioned previously, MM&A modeled the thickness of non-coal material from the mining face. Tonnages of rock were then converted to short tons of REE (not REE oxides) using the concentrations provided by UK from the various sample locations.

Due to the few REE sample locations, rather than gridding quality, MM&A opted to apply the following assumptions regarding the concentrations used to convert rock tons to REE tons:

- Mines Nos. 81 and 89 used the REE results from their corresponding channel samples.
- The study area south of the active mines have four REE sample locations (from each of the four recently drilled exploration holes).
  - Within ¼-mile of these exploration holes, the REE results specific to that sample location was applied to those corresponding tons.
  - Outside the ¼-mile radius, a generalized weighted average of the strata-specific REEs from the four drill holes was used.

All tonnages are categorized by strata as to be able to apply the corresponding REE concentrations. In the few instances where a sample location lacked one or more of the modeled strata (i.e. interval between H4 coal and H4 Flint Clay), the weighted average used in the study area for that specific strata was used.

The Excel workbook compiled by MM&A in the course of this geologic evaluation contains summary and detailed summary tabs indicating in-place measured, indicated, and inferred rock tonnages and corresponding REE tonnages (total and individual elements) by strata, area, and ownership. The workbook is available upon appropriate request.

The following tables provide a basic break down of the total in-place rock and key element tonnages.

Table 2-2: Basic Summary of In-Place Non-Coal Rock and REY Short Tons

Location	In-Place Non-Coal Rock Tons	Sc	Y	Gd	Dy	Total REY Tons
		(Tons In-Place)	(Tons In-Place)	(Tons In-Place)	(Tons In-Place)	(Whole Sample basis)
Total Mine No. 81 (Measured + Indicated + Inferred)	2,338,000	40	73	24	18	832
Total Mine No 89 (Measured + Indicated)	491,000	8	15	6	4	150
Study Area, Subtotal KRP Controlled (Measured)	45,699,000	643	1,381	489	363	15,520
Study Area Subtotal Non KRP (Measured)	13,196,000	186	396	140	104	4,459
Study Area - Total All Holes & Ownership (Indicated)	56,619,000	791	1,733	611	458	19,408
Study Area - All Tonnages and Ownership (Measured + Indicated)	**115,513,000**	**1,620**	**3,510**	**1,240**	**926**	**39,387**
All Areas - All Tonnages - All Ownerships	**118,343,000**	**1,667**	**3,598**	**1,270**	**947**	**40,370**

Table 2-3: Distribution of In-Place Y+REE Within Future Mining in Mine Nos. 81 and 89 and the "Study Area"

Area	Confidence Category	Sc (tons)	Y (tons)	La (tons)	Ce (tons)	Pr (tons)	Nd (tons)	Sm (tons)	Eu (tons)	Gd (tons)	Tb (tons)	Dy (tons)	Ho (tons)	Er (tons)	Tm (tons)	Yb (tons)	Lu (tons)	Total REY Tons (Ash Basis)	REY Concentration (Whole sample basis in ppm)	Total In-Place REY Tons (Whole sample basis)
Mine No. 81	Measured	10.58	19.29	39.14	87.44	10.24	40.40	8.74	1.08	6.35	0.32	4.72	1.06	3.47	0.79	2.61	0.45	236.66	357.35	219.22
	Indicated	26.77	49.72	100.54	225.91	26.29	104.13	22.51	2.76	16.42	0.83	12.28	2.75	8.92	2.04	6.66	1.12	609.62	358.19	563.83
	Inferred	2.30	4.36	8.77	19.84	2.29	9.12	1.97	0.24	1.44	0.07	1.09	0.24	0.78	0.18	0.58	0.10	53.36	359.13	49.26
Total Mine No. 81 (Measured + Indicated + Inferred)		**40**	**73**	**148**	**333**	**39**	**154**	**33**	**4**	**24**	**1**	**18**	**4**	**13**	**3**	**10**	**2**	**900**	**358**	**832**
Mine No. 89	Measured	6.86	13.74	27.62	66.34	7.21	28.29	6.30	0.88	5.11	0.27	3.20	0.89	2.71	0.47	1.53	0.19	171.59	355.50	134.56
	Indicated	0.81	1.62	3.25	7.81	0.85	3.33	0.74	0.10	0.60	0.03	0.38	0.10	0.32	0.05	0.18	0.02	20.20	357.59	15.84
Total Mine No 89 (Measured + Indicated)		**8**	**15**	**31**	**74**	**8**	**32**	**7**	**1**	**6**	**0**	**4**	**1**	**3**	**1**	**2**	**0**	**192**	**356**	**150**
KRP Study Area, Controlled by RELW18-01	Measured	15.59	29.16	60.71	142.08	16.26	63.10	13.63	1.67	10.48	0.89	7.55	1.74	5.20	0.93	3.87	0.67	372.69	356.51	346.08
KRP Study Area, Controlled by RELW18-02	Measured	14.43	26.79	55.87	131.10	15.05	57.87	12.97	1.59	9.85	0.76	7.10	1.59	4.53	0.76	3.55	0.65	343.92	364.07	322.33
KRP Study Area, Controlled by RELW18-03	Measured	11.18	24.36	48.91	107.42	12.85	52.04	12.20	1.34	8.59	0.82	6.25	1.40	4.02	0.67	3.32	1.19	296.11	355.87	256.43
KRP Study Area, Controlled by RELW19-01	Measured	7.55	27.04	45.85	111.61	12.88	54.60	12.31	1.14	9.34	0.52	7.76	1.63	4.31	0.51	3.07	0.62	300.76	521.08	270.53
KRP Study Area Exclusive of Drill Hole Control	Measured	593.85	1,273.61	2,533.51	5,806.03	680.44	2,698.37	606.88	69.58	450.41	36.29	334.81	73.62	214.40	34.01	166.37	40.13	15,590.10	359.40	14,324.99
Subtotal KRP Study Area	***Measured***	***643***	***1,381***	***2,745***	***6,298***	***737***	***2,926***	***658***	***75***	***489***	***39***	***363***	***80***	***232***	***37***	***180***	***43***	***16,904***	***362***	***15,520***
Non-KRP Study Area, Controlled by RELW18-02	Measured	0.05	0.11	0.22	0.52	0.06	0.22	0.05	0.01	0.04	0.00	0.03	0.01	0.02	0.00	0.01	0.00	1.34	379.40	1.25
Non-KRP Study Area, Controlled by RELW18-03	Measured	2.46	5.76	11.29	24.34	2.95	12.03	2.79	0.31	1.95	0.19	1.48	0.32	0.95	0.16	0.78	0.28	67.92	347.21	61.37
Non-KRP Study Area, Controlled by RELW19-01	Measured	0.10	0.33	0.57	1.37	0.16	0.67	0.15	0.01	0.11	0.01	0.09	0.02	0.05	0.01	0.04	0.01	3.69	496.15	3.33
Non-KRP Study Area Exclusive of Drill Hole Control	Measured	183.50	389.79	776.89	1,777.89	208.92	827.47	186.29	21.48	138.06	11.17	102.16	22.45	65.73	10.41	51.10	12.43	4,778.98	357.51	4,393.45
Subtotal Non-KRP Study Area	***Measured***	***186***	***396***	***789***	***1804***	***212***	***840***	***189***	***22***	***140***	***11***	***104***	***23***	***67***	***11***	***52***	***13***	***4852***	***357***	***4459***
Total Study Area KRP and Non-KRP	**Measured**	**829**	**1,777**	**3,534**	**8,102**	**950**	**3,766**	**847**	**97**	**629**	**51**	**467**	**103**	**299**	**47**	**232**	**56**	**21,756**	**361**	**19,980**
Subtotal of KRP Controlled Study Area	Indicated	582.33	1,283.06	2,543.64	5,850.56	680.30	2,706.08	606.74	68.48	452.05	36.05	340.14	74.99	214.98	34.25	165.89	39.06	15,655.91	364.53	14,366.50
Subtotal of Non-KRP Controlled Study Area	Indicated	208.70	449.59	890.78	2,043.56	239.32	950.29	213.70	24.44	158.77	12.76	118.34	26.01	75.60	12.00	58.63	14.10	5,488.79	360.52	5,041.21
Total Study Area, KRP and Non-KRP	**Indicated**	**791**	**1,733**	**3,434**	**7,894**	**920**	**3,656**	**820**	**93**	**611**	**49**	**458**	**101**	**291**	**46**	**225**	**53**	**21,145**	**363**	**19,408**
Study Area - All Tonnages and Ownership (Measured + Indicated)		**1,620**	**3,510**	**6,968**	**15,996**	**1,869**	**7,423**	**1,668**	**190**	**1,240**	**99**	**926**	**204**	**590**	**94**	**457**	**109**	**42,900**	**362**	**39,387**
All Areas - All Tonnages - All Ownerships		**1,667**	**3,598**	**7,148**	**16,404**	**1,916**	**7,608**	**1,708**	**195**	**1,270**	**101**	**947**	**209**	**606**	**97**	**468**	**111**	**43,992**	**341**	**40,370**

2.5 Index of Maps and Exhibits

- Map 1. Thickness of Hazard No. 4 Seam Roof/Floor
- Map 2. Thickness of Parting between Hazard No. 4 Coal and Hazard No. 4 Flint Clay
- Map 3. Thickness of Hazard No. 4 Flint Clay
- Map 4. Thickness of Parting between Hazard No. 4 Flint Clay and Hazard No. 4 Jack Rock Coal
- Exhibit 1. Geological Cross Section A-A'
- Exhibit 2. Geological Cross Section B-B'
- Exhibit 3. Geological Cross Section C-C'
- Exhibit 4. Geological Cross Section D-D'

2.6 References Cited

Eble, C. F., Hower, J. C., & Andrews, W. M. Jr., 1999. Compositional variations in the Fire Clay Coal bed of eastern Kentucky: geochemistry, petrography, palynology, and paleoecology: University of Kentucky Report of Investigations 14, Series XI.

Ekmann, J. M., 2012. *Rare earth elements in coal deposits – a prospectivity analysis*. Adapted from poster presentation given at AAPG Eastern Section meeting, Cleveland, Ohio, p. 22-26.

Greb. S.F., 2003. *Mining geology of the Fire Clay coal*. Kentucky Geological Survey Map and Chart. 30, Retrieved from https://uknowledge.uky.edu/kgs_mc/30.

Greb, S. F., Eble, C. F., & Hower, J. C., 1999. *Depositional history of the Fire Clay Coal bed (Late Duckmantian), eastern Kentucky, USA*. International Journal of Coal Geology 40, p. 255-280.

Greb, S. F., Hiett, J. K., Andrews, R. E., & Sergeant, R. E., 1999. *Geology of the Fire Clay Coal in part of the eastern Kentucky coal field*: University of Kentucky Report of Investigations 2, Series XII.

Greb, S. F., Hiett, J. K., Weisenfluh, G. A., Andrews, R. E., & Sergeant, R. E., 1999. *Available resources of the Fire Clay Coal in part of the eastern Kentucky coal field*: University of Kentucky Report of Investigations 3, Series XII.

Hower, J. C., Ruppert, L. F., & Eble, C. F., 1999. *Lanthanide, yttrium, and zirconium anomalies in the Fire Clay coal bed, eastern Kentucky*. International Journal of Coal Geology 39, p. 141-153.

Rice, C. L., Kosanke, R. M., and Henry, T. W., 1994. *Revision of nomenclature and correlations of some Middle Pennsylvanian units in the northwestern part of the Appalachian basin, Kentucky, Ohio, and West Virginia*. Geological Society of America, Special Paper 294.

2.7 Acknowledgments

Special acknowledgment, however, is due to the following individuals and / or entities whose openness and willingness to share information proved invaluable to this undertaking:

Steven Keim, Ph.D., P.E., Principal Investigator of MMA-29956 and contributor to this chapter; Marshall Miller & Associates, Inc, 1700 Kraft Drive, Ste. 2050, Blacksburg, Virginia 24060-0012.

Ted Lord, MBA, Principal author of this chapter; Senior Geologist, Marshall Miller & Associates, Inc., 582 Industrial Park Road, Bluefield, Virginia, 24605-9364.

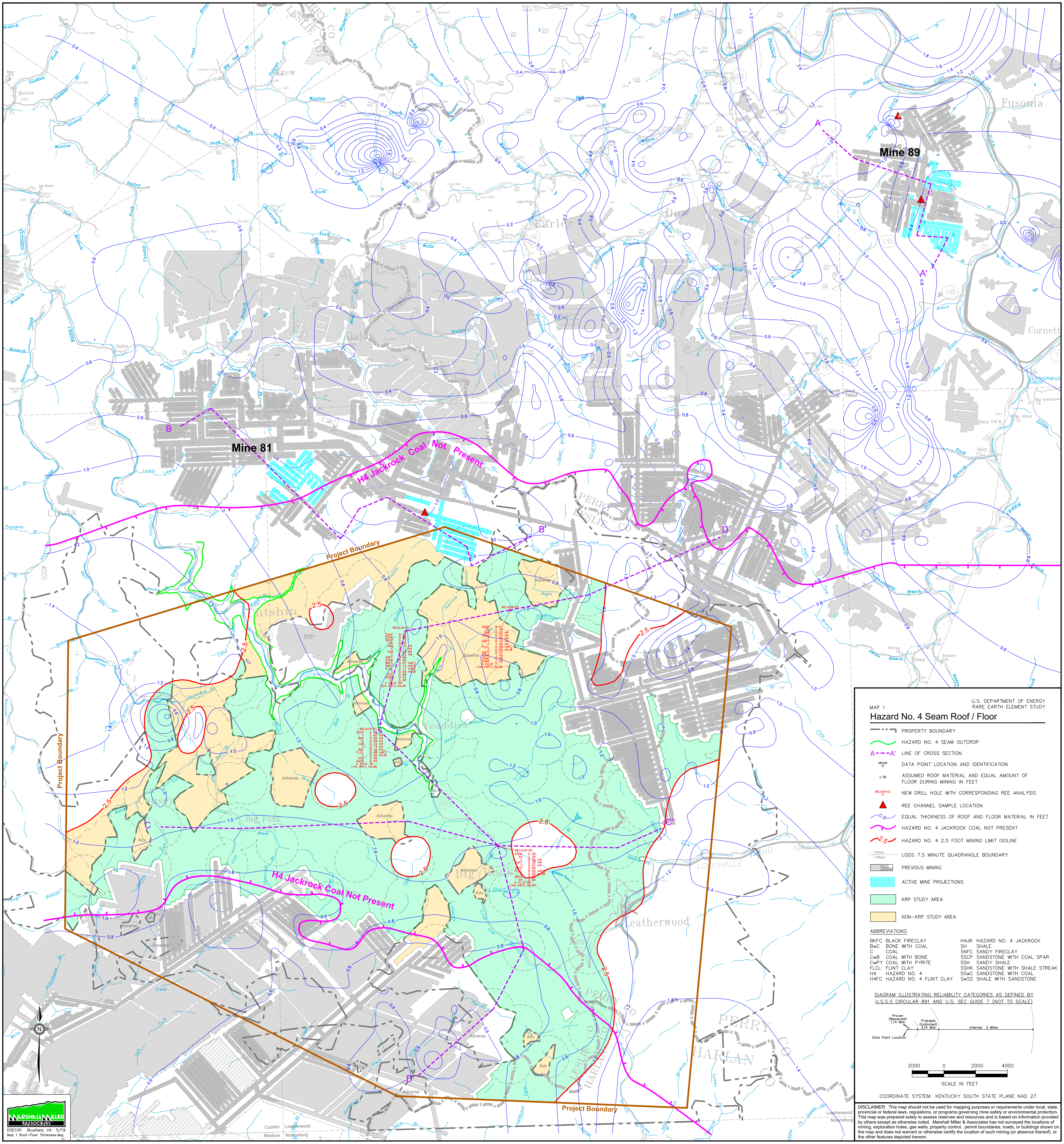
Steve G. Barker, Esq., Interested Party and Sub-Awardee to this project who contributed his regional and local knowledge of the coal-bearing property involved in this investigation. President, CEO, and General Counsel, Kentucky River Properties, LLC, 360 East Vine Street, Suite 310, Lexington, Kentucky, 40507-1522.

Chuck Mullins, P.E., L.S, Facilitator for the collection of facts relevant to the assembly of this chapter: Chief Engineer, Kentucky River Properties, LLC, 300 Black Gold Boulevard, Hazard, Kentucky,41701-2605.

Craig Daniels, P.G.(TN), Contributor to this chapter through management of geological exploration campaigns, refuse sample handling, and coordination of laboratory analyses; Chief Geologist, Kentucky River Properties, LLC, 300 Black Gold Boulevard, Hazard, Kentucky,41701-2605.

Cortland F. Eble, Ph.D.: Contributor to this chapter by his participation in stratigraphic sample collection, processing, and analyses from lands relevant to this project as well as providing advice and commentaries; Kentucky Geologic Survey, c/o University of Kentucky, 237D Mining and Mineral Resources Building, Lexington, KY 40506-0540.

Hans E. Naumann, Jr., B.S., M.S., P.E.: Associate and facilitator of the assembly of this chapter: Principal Consultant, Marshall Miller & Associates, Inc., 744 Lakeshore Drive, Lexington, Kentucky, 40502-3119.



MAP 1

U.S. DEPARTMENT OF ENERGY
RARE EARTH ELEMENT STUDY

Hazard No. 4 Seam Roof / Floor

▬ ▪ ▪ ▬	PROPERTY BOUNDARY
〰	HAZARD NO. 4 SEAM OUTCROP
A- - -A'	LINE OF CROSS SECTION
KR129 ○	DATA POINT LOCATION AND IDENTIFICATION
1.05	ASSUMED ROOF MATERIAL AND EQUAL AMOUNT OF FLOOR DURING MINING IN FEET
RELW18-01 ○	NEW DRILL HOLE WITH CORRESPONDING REE ANALYSIS
▲	REE CHANNEL SAMPLE LOCATION
0.8	EQUAL THICKNESS OF ROOF AND FLOOR MATERIAL IN FEET
〰	HAZARD NO. 4 JACKROCK COAL NOT PRESENT
2.5	HAZARD NO. 4 2.5 FOOT MINING LIMIT ISOLINE
Vicco / Tilford	USGS 7.5 MINUTE QUADRANGLE BOUNDARY
▨	PREVIOUS MINING
▦	ACTIVE MINE PROJECTIONS
🟩	KRP STUDY AREA
🟨	NON–KRP STUDY AREA

ABBREVIATIONS

BKFC	BLACK FIRECLAY	H4JR	HAZARD NO. 4 JACKROCK
BwC	BONE WITH COAL	SH	SHALE
C	COAL	SNFC	SANDY FIRECLAY
CwB	COAL WITH BONE	SSCP	SANDSTONE WITH COAL SPAR
CwPY	COAL WITH PYRITE	SSH	SANDY SHALE
FLCL	FLINT CLAY	SSHK	SANDSTONE WITH SHALE STREAK
H4	HAZARD NO. 4	SSwC	SANDSTONE WITH COAL
H4FC	HAZARD NO. 4 FLINT CLAY	SwSS	SHALE WITH SANDSTONE

DIAGRAM ILLUSTRATING RELIABILITY CATEGORIES AS DEFINED BY U.S.G.S CIRCULAR 891 AND U.S. SEC GUIDE 7 (NOT TO SCALE)

Proven (Measured) 1/4 Mile
Probable (Indicated) 3/4 Mile
Inferred 3 Miles
Data Point Location

2000 0 2000 4000

SCALE IN FEET

COORDINATE SYSTEM: KENTUCKY SOUTH STATE PLANE NAD 27

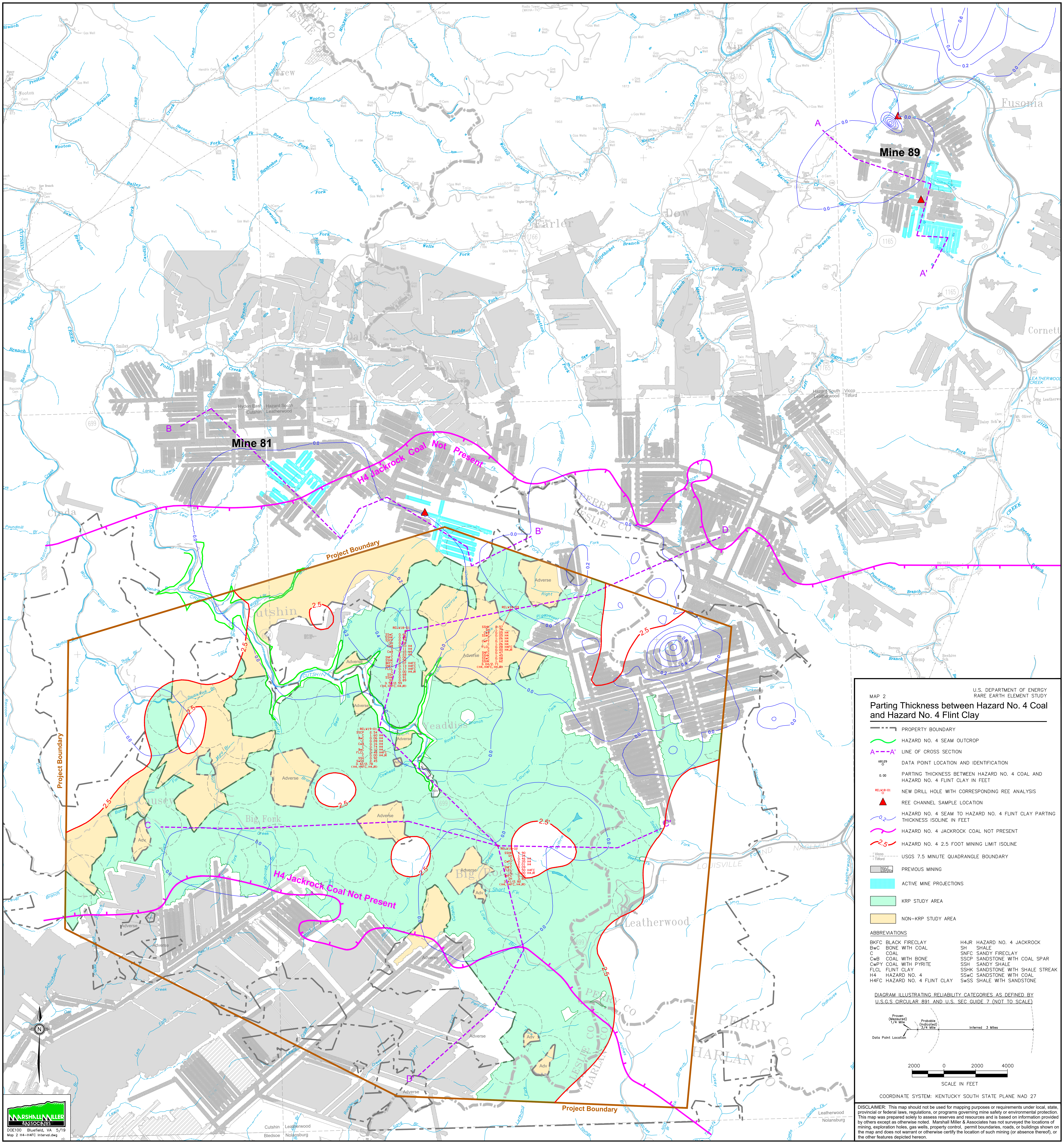
DISCLAIMER: This map should not be used for mapping purposes or requirements under local, state, provincial or federal laws, regulations, or programs governing mine safety or environmental protection. This map was prepared solely to assess reserves and resources and is based on information provided by others except as otherwise noted. Marshall Miller & Associates has not surveyed the locations of mining, exploration holes, gas wells, property control, permit boundaries, roads, or buildings shown on the map and does not warrant or otherwise certify the location of such mining (or absence thereof), or the other features depicted hereon.

MARSHALL MILLER & ASSOCIATES®
DOE100 Bluefield, VA 5/19
Map 1 Roof–Floor Thickness.dwg
mh

MAP 2

U.S. DEPARTMENT OF ENERGY
RARE EARTH ELEMENT STUDY

Parting Thickness between Hazard No. 4 Coal and Hazard No. 4 Flint Clay



- PROPERTY BOUNDARY
- HAZARD NO. 4 SEAM OUTCROP
- A- - -A' LINE OF CROSS SECTION
- KR129 ○ DATA POINT LOCATION AND IDENTIFICATION
- 0.00 PARTING THICKNESS BETWEEN HAZARD NO. 4 COAL AND HAZARD NO. 4 FLINT CLAY IN FEET
- RELW18-01 ○ NEW DRILL HOLE WITH CORRESPONDING REE ANALYSIS
- ▲ REE CHANNEL SAMPLE LOCATION
- 0.2 HAZARD NO. 4 SEAM TO HAZARD NO. 4 FLINT CLAY PARTING THICKNESS ISOLINE IN FEET
- HAZARD NO. 4 JACKROCK COAL NOT PRESENT
- 2.5 HAZARD NO. 4 2.5 FOOT MINING LIMIT ISOLINE
- Vicco Tilford USGS 7.5 MINUTE QUADRANGLE BOUNDARY
- ▨ PREVIOUS MINING
- ▦ ACTIVE MINE PROJECTIONS
- 🟩 KRP STUDY AREA
- 🟨 NON–KRP STUDY AREA

ABBREVIATIONS

BKFC	BLACK FIRECLAY
BwC	BONE WITH COAL
C	COAL
CwB	COAL WITH BONE
CwPY	COAL WITH PYRITE
FLCL	FLINT CLAY
H4	HAZARD NO. 4
H4FC	HAZARD NO. 4 FLINT CLAY
H4JR	HAZARD NO. 4 JACKROCK
SH	SHALE
SNFC	SANDY FIRECLAY
SSCP	SANDSTONE WITH COAL SPAR
SSH	SANDY SHALE
SSHK	SANDSTONE WITH SHALE STREAK
SSwC	SANDSTONE WITH COAL
SwSS	SHALE WITH SANDSTONE

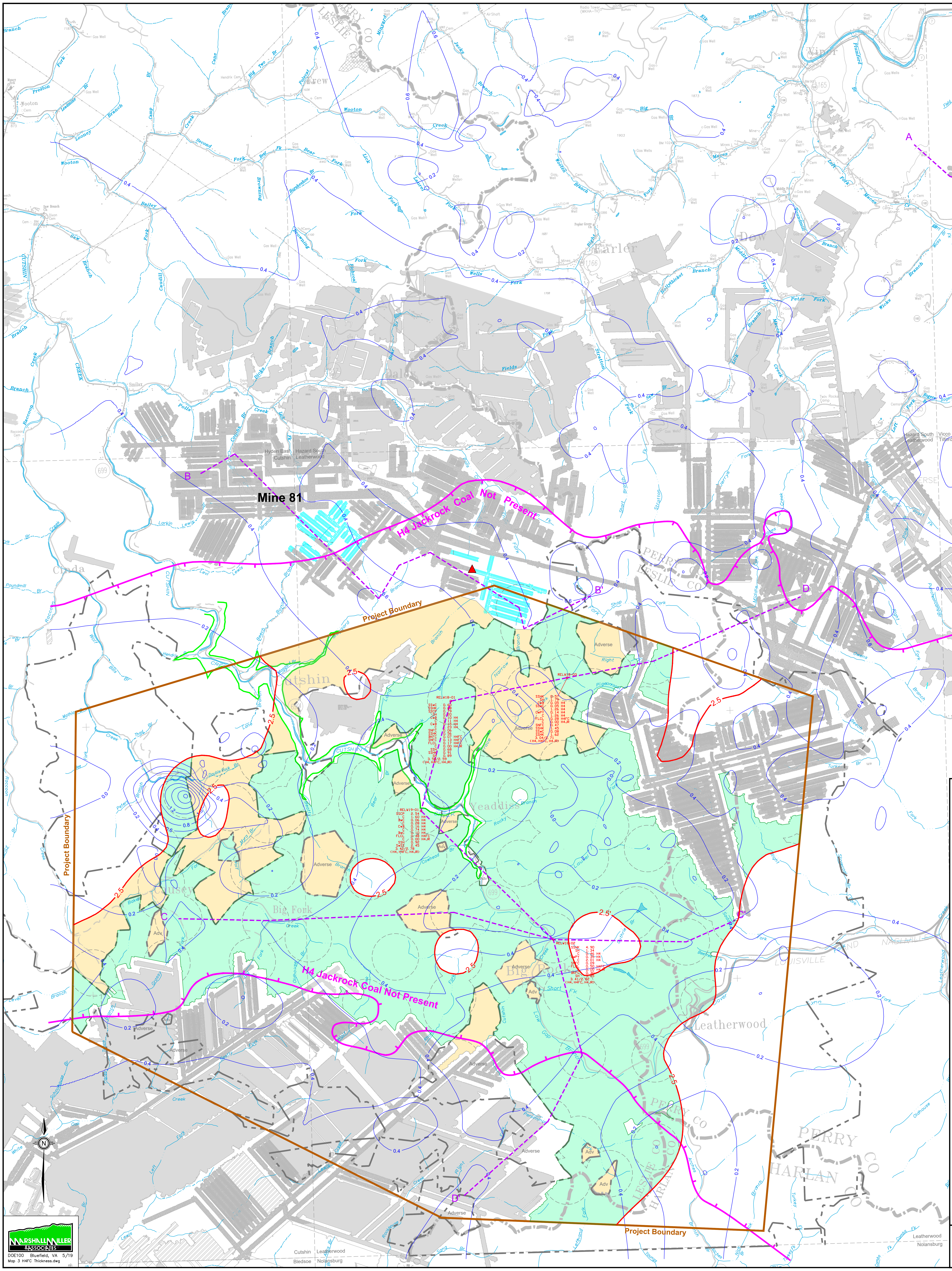
DIAGRAM ILLUSTRATING RELIABILITY CATEGORIES AS DEFINED BY U.S.G.S CIRCULAR 891 AND U.S. SEC GUIDE 7 (NOT TO SCALE)

SCALE IN FEET

COORDINATE SYSTEM: KENTUCKY SOUTH STATE PLANE NAD 27

DISCLAIMER: This map should not be used for mapping purposes or requirements under local, state, provincial or federal laws, regulations, or programs governing mine safety or environmental protection. This map was prepared solely to assess reserves and resources and is based on information provided by others except as otherwise noted. Marshall Miller & Associates has not surveyed the locations of mining, exploration holes, gas wells, property control, permit boundaries, roads, or buildings shown on the map and does not warrant or otherwise certify the location of such mining (or absence thereof), or the other features depicted hereon.

MARSHALL MILLER & ASSOCIATES®
DOE100 Bluefield, VA 5/19
Map 2 H4–H4FC Interval.dwg
mh



U.S. DEPARTMENT OF ENERGY
RARE EARTH ELEMENT STUDY

MAP 3

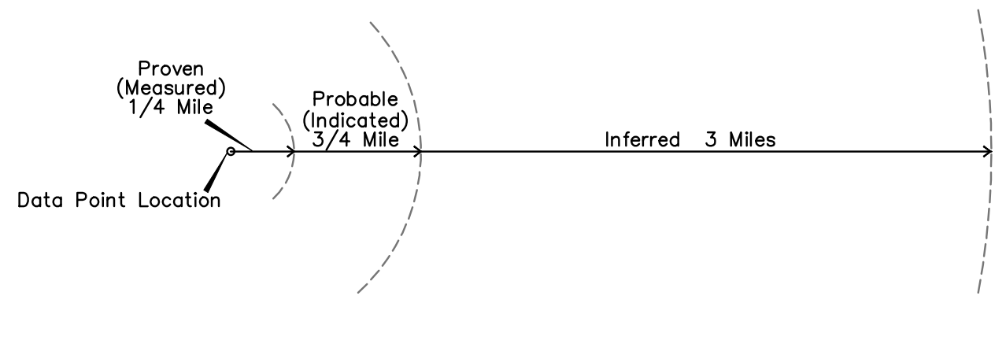
Hazard No. 4 Flint Clay Thickness

▬ · · ▬	PROPERTY BOUNDARY
〜	HAZARD NO. 4 SEAM OUTCROP
A- - -A'	LINE OF CROSS SECTION
CT467 ○	DATA POINT LOCATION AND IDENTIFICATION
H4FC 0.38	HAZARD NO. 4 FLINT CLAY THICKNESS IN FEET
RELW18-01 ○	NEW DRILL HOLE WITH CORRESPONDING REE ANALYSIS
▲	REE CHANNEL SAMPLE LOCATION
0.2	HAZARD NO. 4 FLINT CLAY THICKNESS ISOLINE IN FEET
〜	HAZARD NO. 4 JACKROCK COAL NOT PRESENT
2.5	HAZARD NO. 4 2.5 FOOT MINING LIMIT ISOLINE
Vicco Tilford	USGS 7.5 MINUTE QUADRANGLE BOUNDARY
▨	PREVIOUS MINING
▦	ACTIVE MINE PROJECTIONS
🟩	KRP STUDY AREA
🟨	NON-KRP STUDY AREA

ABBREVIATIONS

BKFC	BLACK FIRECLAY	H4JR	HAZARD NO. 4 JACKROCK
BwC	BONE WITH COAL	SH	SHALE
C	COAL	SNFC	SANDY FIRECLAY
CwB	COAL WITH BONE	SSCP	SANDSTONE WITH COAL SPAR
CwPY	COAL WITH PYRITE	SSH	SANDY SHALE
FLCL	FLINT CLAY	SSHK	SANDSTONE WITH SHALE STREAK
H4	HAZARD NO. 4	SSwC	SANDSTONE WITH COAL
H4FC	HAZARD NO. 4 FLINT CLAY	SwSS	SHALE WITH SANDSTONE

DIAGRAM ILLUSTRATING RELIABILITY CATEGORIES AS DEFINED BY U.S.G.S CIRCULAR 891 AND U.S. SEC GUIDE 7 (NOT TO SCALE)



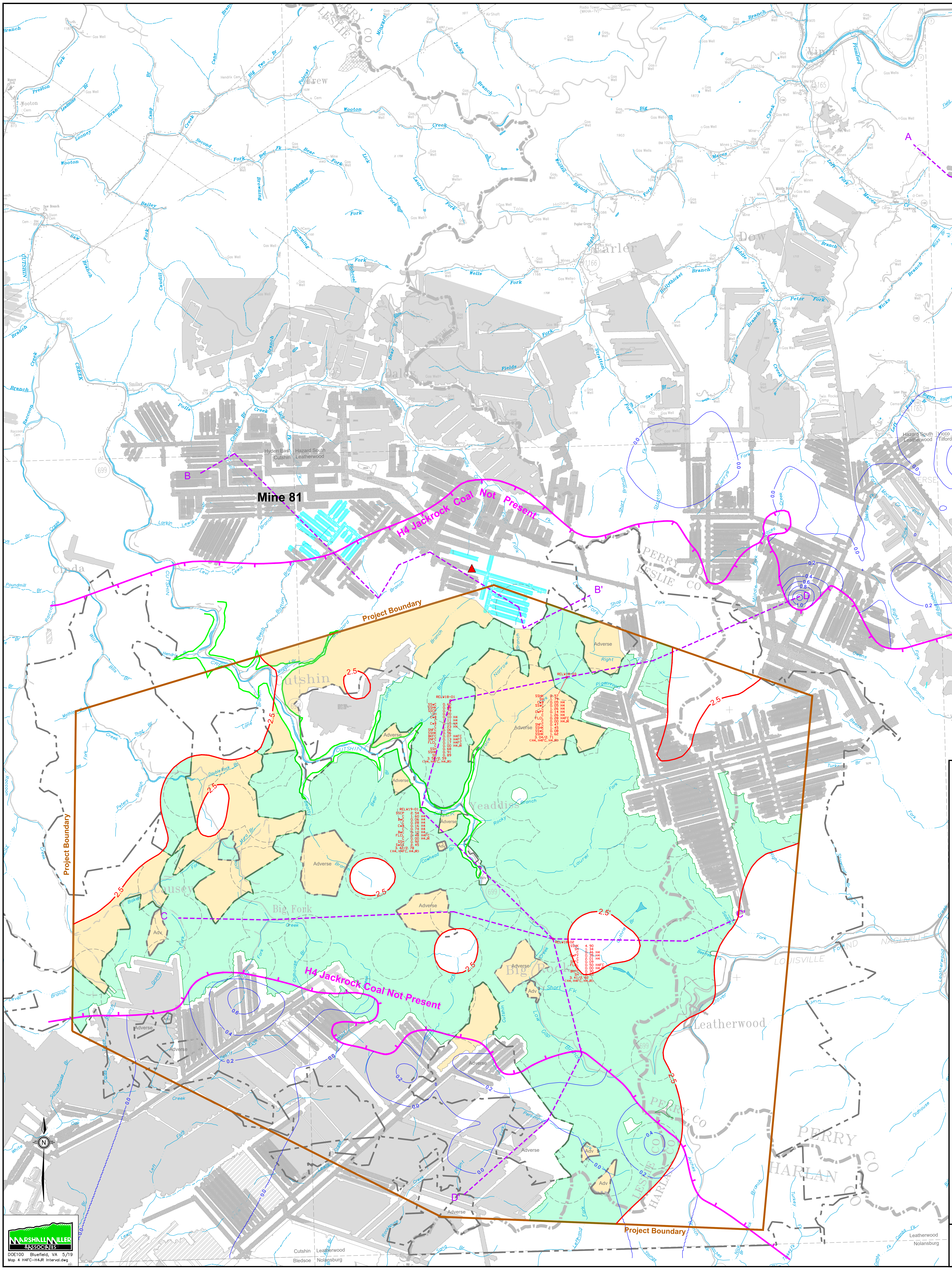
2000 0 2000 4000

SCALE IN FEET

COORDINATE SYSTEM: KENTUCKY SOUTH STATE PLANE NAD 27

DISCLAIMER: This map should not be used for mapping purposes or requirements under local, state, provincial or federal laws, regulations, or programs governing mine safety or environmental protection. This map was prepared solely to assess reserves and resources and is based on information provided by others except as otherwise noted. Marshall Miller & Associates has not surveyed the locations of mining, exploration holes, gas wells, property control, permit boundaries, roads, or buildings shown on the map and does not warrant or otherwise certify the location of such mining (or absence thereof), or the other features depicted hereon.

MARSHALL MILLER & ASSOCIATES®
DOE100 Bluefield, VA 5/19
Map 3 H4FC Thickness.dwg
mh



MAP 4

U.S. DEPARTMENT OF ENERGY
RARE EARTH ELEMENT STUDY

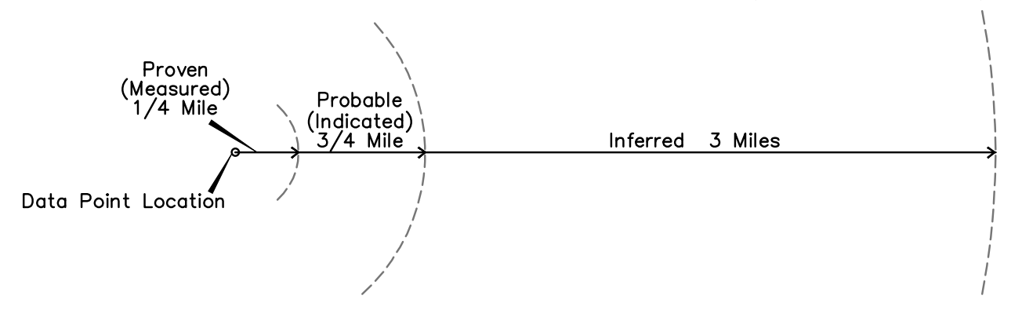
Parting Thickness between Hazard No. 4 Flint Clay and Hazard No. 4 Jack Rock Coal

▬ ▪ ▪ ▬	PROPERTY BOUNDARY
〰	HAZARD NO. 4 SEAM OUTCROP
A- - -A'	LINE OF CROSS SECTION
KR129 ○	DATA POINT LOCATION AND IDENTIFICATION
0.00	PARTING THICKNESS BETWEEN HAZARD NO. 4 FLINT CLAY AND HAZARD NO. 4 JACK ROCK COAL THICKNESS IN FEET
RELW18-01 ○	NEW DRILL HOLE WITH CORRESPONDING REE ANALYSIS
▲	REE CHANNEL SAMPLE LOCATION
0.2	HAZARD NO. 4 FLINT CLAY TO HAZARD NO. 4 JACKROCK PARTING THICKNESS ISOLINE IN FEET
〰	HAZARD NO. 4 JACKROCK COAL NOT PRESENT
2.5	HAZARD NO. 4 2.5 FOOT MINING LIMIT ISOLINE
Vicco Tilford	USGS 7.5 MINUTE QUADRANGLE BOUNDARY
▨	PREVIOUS MINING
▦	ACTIVE MINE PROJECTIONS
🟩	KRP STUDY AREA
🟨	NON-KRP STUDY AREA

ABBREVIATIONS

BKFC	BLACK FIRECLAY	H4JR	HAZARD NO. 4 JACKROCK
BwC	BONE WITH COAL	SH	SHALE
C	COAL	SNFC	SANDY FIRECLAY
CwB	COAL WITH BONE	SSCP	SANDSTONE WITH COAL SPAR
CwPY	COAL WITH PYRITE	SSH	SANDY SHALE
FLCL	FLINT CLAY	SSHK	SANDSTONE WITH SHALE STREAK
H4	HAZARD NO. 4	SSwC	SANDSTONE WITH COAL
H4FC	HAZARD NO. 4 FLINT CLAY	SwSS	SHALE WITH SANDSTONE

DIAGRAM ILLUSTRATING RELIABILITY CATEGORIES AS DEFINED BY U.S.G.S CIRCULAR 891 AND U.S. SEC GUIDE 7 (NOT TO SCALE)



2000 0 2000 4000

SCALE IN FEET

COORDINATE SYSTEM: KENTUCKY SOUTH STATE PLANE NAD 27

DISCLAIMER: This map should not be used for mapping purposes or requirements under local, state, provincial or federal laws, regulations, or programs governing mine safety or environmental protection. This map was prepared solely to assess reserves and resources and is based on information provided by others except as otherwise noted. Marshall Miller & Associates has not surveyed the locations of mining, exploration holes, gas wells, property control, permit boundaries, roads, or buildings shown on the map and does not warrant or otherwise certify the location of such mining (or absence thereof), or the other features depicted hereon.

MARSHALL MILLER & ASSOCIATES®

DOE100 Bluefield, VA 5/19
Map 4 H4FC–H4JR Interval.dwg
mh

Chapter 3.0

SYSTEM DESIGN PACKAGE PREPARATION

Chapter 3.0
System Design Package Preparation

TABLE OF CONTENTS

3.1 Summary of Investigations of REE Concentration Plant Feedstock Sources 1
- 3.1.1 Inventory of Coal and Coal Byproducts Feedstock Sources Examined and Tested in the Course of this Task 1
  - **3.1.1.1 Evaluation of LIMN 2**
- 3.1.2 West Virginia Mining Complex, Lower Kittanning Coal Seam 2
  - **3.1.2.1 Lower Kittanning Coal Seam Mining and Preparation Plant Complex 2**
- 3.1.3 West Virginia Mining Complex, Pocahontas No. 3 Seam Coal 2
  - **3.1.3.1 Pocahontas No. 3 Seam Preparation Plant Survey – West Virginia 2**
  - **3.1.3.2 Pocahontas No. 3 Coal Seam Plant Circuit Samples TREE Content Results 4**
  - **3.1.3.3 Pocahontas No. 3 Coal Seam Preparation Plant Circuit Samples Heavy and Light REE Concentrations 5**
  - **3.1.3.4 Pocahontas No. 3 Coal Seam Plant Circuit Samples Normalized Ratio of HREE/LREE 6**
    - 3.1.3.4.1 Pocahontas No. 3 Coal Seam Preparation Plant Circuit Samples TREE in Relation to Process Solid Types 7
  - **3.1.3.5 Pocahontas No. 3 Coal Seam Preparation Plant Circuit Samples REE Concentrations versus Ash Content 9**
  - **3.1.3.6 Pocahontas No. 3 Seam Preparation Plant Circuit Sample Character versus Coal Mining Methods, Plant Low-Ash Middlings and Plant High-Ash Middlings Sample Test Results 14**
    - 3.1.3.6.1 Pocahontas No. 3 Coal Seam Preparation Plant Circuit High Ash and Low Ash Middlings Sample Treatment Protocols 15
    - 3.1.3.6.2 Pocahontas No. 3 Coal Seam Preparation Plant High Ash and Low Ash Middlings Samples Decarbonization Results 17
  - **3.1.3.7 Pocahontas No. 3 Coal Seam Preparation Plant Circuits Low Ash Middlings HHS Coal and Tailings REE Results 19**
  - **3.1.3.8 Pocahontas No. 3 Coal Seam Preparation Plant Circuit High Ash Middlings HHS Coal and Tailings REE Results 21**
  - **3.1.3.9 Pocahontas No. 3 Coal Seam Rejects REE Concentration by Optical Sorting 24**
    - 3.1.3.9.1 Dual X-Ray Transmission Sorting (DXRT) 24
- 3.1.4 Ferroglobe Preparation Plant Circuit Sampling Program 27
  - **3.1.4.1 Ferroglobe High Ash (Jellico) and Low Ash (Blue Gem) Preparation Plant Circuit Samples 27**
    - 3.1.4.1.1 Ferroglobe High Ash and Low Ash Sample Flotation Feed Testing and Screenbowl Effluent 28
    - 3.1.4.1.2 Ferroglobe Screenbowl Centrifuge Effluent Sample Testing 33
    - 3.1.4.1.3 Ferroglobe Spiral Stream Testing 33
  - **3.1.4.2 Ferroglobe - Blue Gem Seam Coal Coarse Rejects 38**

3.1.4.3 Ferroglobe - Jellico Seam Coal Clean Coal Sieve Overflow Sample Testing 38
3.1.4.4 Ferroglobe - Summary Observations of the HHS of the Blue Gem and Jellico Coal Seam Rejects Samples..40
3.1.5 Arq Corbin LLC Plant Material – HHS Prospect 41
3.1.5.1 Background to the Arq Investigation..41
3.1.5.2 Arq Project – HHS Test Results of an “As-Received” Sample of Coal Rejects Impoundment...41
3.1.5.2.1 Arq Project – HHS Test Results of Milled Sample of Coal Rejects Impoundment..41
3.1.5.2.2 Arq Project – Sample Ground to D80 Passing 8 Microns 42
3.1.5.2.3 Arq Project – HHS Test Results of Milled Sample of Coal Rejects Impoundment with Dispersant Additive..43
3.1.5.3 Arq Project – HHS Test Results of Attrition Grinding of Sample of Coal Rejects...43
3.1.6 Blue Diamond Coal Company: Fire Clay Coal Seam (Hazard 4) Seam Rejects.. 44
3.1.6.1 Fire Clay Seam (Hazard 4) Coal Reject Feedstock..44
3.1.6.2 Hazard 4 Thickener Underflow Froth Flotation Test Results 44
3.1.6.3 HHS of the Hazard 4 Seam Thickener Underflow of the Leatherwood Plant..........46
3.1.6.3.1 Results of Standard HHS Testing Procedures of the Leatherwood Plant, Hazard 4, Thickener Underflow 47
3.1.6.3.2 Supplemental Treatment of Leatherwood Plant Thickener Underflow Prior to HHS to Improve REM Recovery..49
3.1.6.3.2.1 HHS Feedstock Preparation by Blunging... 49
3.1.6.3.2.2 Oil Agglomeration and removal of Coal Particles from the HHS Feedstock.. 52

3.2 Roasting and Acid Leaching Experiments 57
3.2.1 Roasting 57
3.2.2 Acid Leaching.. 57
3.2.3 Leachability Tests of the Rejects from the Pocahontas No. 3 Coal Seam............ 59
3.2.3.1 Unroasted Pocahontas No. 3 Coal Seam Channel Sample Segments (Roof, Coal, and Floor) 59
3.2.3.2 Unroasted Pocahontas No. 3 Coal Seam Plant Coarse Rejects..................................62
3.2.3.2.1 Summary Observations of the Leaching Tests of the Unroasted Pocahontas No. 3 Coal Seam Coarse Rejects 64
3.2.3.3 Acid Leaching Response to Roasting Decarbonized Pocahontas No. 3 Coal Seam Preparation Plant Middlings Material 68
3.2.3.3.1 Step 1: Comparison of REE Recovery from Unroasted and Roasted Decarbonized Middlings Samples Prior to Acid Leaching..............................68
3.2.3.3.2 Step 2: Comparison of TREE Recovery from Acid Leaching Unroasted and Roasted Pocahontas No. 3 Seam Middlings Samples..............69
3.2.3.4 Distribution of TREE, HREE, LREE and CREE Content in Acid Leachates from Unroasted and Roasted Decarbonized Samples of Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit 69
3.2.3.4.1 Results: Effects on TREE, HREE, LREE and CREE 70
3.2.3.4.2 Results: Discussion on the Recovery of CREE 70

3.2.3.5 Acid Leaching of Unroasted and Roasted Coarse Rejects and Middlings from the Pocahontas No. 3 Coal Seam Preparation Plant.......................................71
3.2.3.5.1 Impact of Roasting on Proximate Analyses of the Coarse Rejects Sample Prior to Acid Leaching...71
3.2.3.5.2 Acid Leaching Kinetics: Effects of Roasting Coarse Rejects and Middlings Circuit Samples from the Pocahontas No. 3 Coal Seam Preparation Plant...72
3.1.6.3.2.3 Effects of Roasting Temperature.. 72
3.2.3.5.3 Effects of Inorganic Acid Type on REE Recovery from the Roasted Pocahontas No. 3 Coal Seam Preparation Plant Coarse Rejects and Middlings Circuit Product Samples 75
3.2.3.5.4 REE Recovery from Coarse Rejects and Middlings Circuit Samples at Optimum Temperature and Acid Type Conditions 76
3.2.3.5.5 Effect of Roasting Temperature, Acid Type, and Leach Tank Temperature on Ce, Y, Dy and Sc Recovery from the Pocahontas No. 3 Coal Seam Preparation Plant Middlings and the 2.2 Specific Gravity (SG) Float Fraction of its Coarse Rejects 76
3.2.3.5.6 Summary Observations of Acid Leaching Behavior of Unroasted and Roasted Samples..80
3.2.3.6 Acid Leaching Behaviors of other Major Elements and Trace Elements in Unroasted and Roasted Samples of the Pocahontas No. 3 Coal Seam Preparation Plant...80
3.2.3.6.1 Summary Observations of the Acid Leaching of other Major Elements and Trace Elements from Unroasted and Roasted Samples of the Pocahontas No. 3 Coal Seam Preparation Plant 90
3.2.3.7 Effects of Acid Leaching Tank Temperature and Acid Type on the Recovery of REE from Roasted Samples from the Pocahontas No. 3 Coal Seam Preparation Plant 91
3.2.3.7.1 Effects of Leaching Temperature 91
3.2.3.8 Effects of Acid Leaching Solution Temperature and Acid Type on the Recovery of other Major Elements Associated with REEs in the Roasted Samples from Pocahontas No. 3 Coal Seam Preparation Plant.................................94
3.2.3.9 Effects of Leachate Solution pH and Acid Type on the Recovery of REEs and Associated Elements from the Roasted Coarse Rejects Sample of the Pocahontas No. 3 Coal Seam Preparation Plant 96
3.2.3.9.1 Effects of Solution pH 96
3.2.3.9.2 Effects of Acid Concentration on the Leaching Recovery of LREEs and HREEs 96
3.2.3.9.3 Effects of Acid Concentration on the Leaching Recovery of other Elements Associated with REEs...97
3.2.3.10 Summary Observations of the Effects of Roasting Temperature, Leachate Solution pH and Acid Type on the Recovery of REEs and Associated Elements from the Roasted Coarse Rejects Sample of the Pocahontas No. 3 Coal Seam Preparation Plant 99
3.2.4 Leachability Tests of the Rejects from the Blue Gem Seam Coal........................ 99
3.2.4.1 Effects of Roasting on TREE Recovery from Three Float-sink Products of Rejects from Ferroglobe’s Gatliff Plant (Blue Gem Seam)..99

3.2.4.1.1 Summary Observations of the Effects Roasting on Total REE Recovery from Three Float-Sink Products of Rejects from Ferrogloble's Gatliff Plant 100

3.2.4.2 Effects of Roasting on HREE and LREE Leach Recovery from Three Float-sink Products of Rejects from Ferroglobe's Gatliff Plant (Blue Gem Seam) 101

3.2.4.2.1 Summary Observations of the Effects of Roasting on HREE and LREE Leach Recovery from Three Float-Sink Products of Rejects from Ferrogloble's Gatliff Plant 102

3.2.4.3 The Effects of Roasting on the Leaching of Sc from (a) 1.8 SG Float, (b) 1.8-2.2 SG, and (c) 2.2 SG Sink Fractions of the Blue Gem Seam Coarse Reject Samples 102

3.2.4.3.1 Summary Observations of the Effects of Roasting on Sc Recovery from Three Float-Sink Products of Rejects from Ferrogloble's Gatliff Plant 103

3.2.4.4 Summary of Methodology and Observations of the Leachability Tests of the Rejects from the Blue Gem Seam Coal 103

3.2.5 Leachability Tests of the Rejects from the Fire Clay (Hazard No. 4) Seam Coal 103

3.2.5.1 Fire Clay (Hazard No.4) Coal, the Host Land Company, and the Host Coal Mining and Processing Company. 103

3.2.5.2 Effects of Roasting on the Mineralogy of the Fire Clay (Hazard No. 4) Rejects Material 104

3.2.5.3 No. 76 Plant (Leatherwood) Fire Clay Seam (Hazard No. 4) Thickener Underflow Acid Leaching Testing Methodology 104

3.2.5.3.1 Acid Leaching Test Results of Unroasted Fire Clay Seam (Hazard No. 4) Flotation Middling Material from the No. 76 Plant (Leatherwood) Thickener Underflow 105

3.2.5.3.2 Acid Leaching Test Results of Roasted Fire Clay Seam (Hazard No. 4) Flotation Middling Material from the No. 76 Plant (Leatherwood) Thickener Underflow 106

3.2.5.4 No. 76 Plant (Leatherwood) Fire Clay Seam (Hazard No. 4) Coarse Rejects Acid Leaching Testing Methodology 106

3.2.5.4.1 Effect of Solids and Acid Concentration on Leaching Recovery and Kinetics for the REEs Associated with the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material) 107

3.2.5.4.2 Effect of Acid Concentration on Leaching Recovery and Kinetics for the REEs Associated with the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No 4) Coarse Rejects Material 108

3.2.5.4.3 Effect of Acid Concentration on Leaching Recovery and Kinetics of other Elements Associated with REE Mineral in the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material....109

3.2.5.4.4 Effects of Acid Leaching Time on the Recovery of REE and Associated Contaminant Elements 111

3.2.5.4.5 Effects of Percent Solids in the Leaching Tank Contents on Recoveries of REE and Other Associated Elements from the Fire Clay (Hazard No. 4) Seam Rejects when using Very Low Acid Concentrations 112

3.2.5.4.6 Evaluation of the Co-Extraction of Lithium (Li) and Cobalt (Co) with REEs from the Fire Clay (Hazard No. 4) Seam Coarse Rejects of the No. 76 Plant (Leatherwood) 113

3.2.5.5 Evaluation of the Effectiveness of Commercial Roasting Services with Respect to Acid Leaching of REEs from Roasted Fire Clay (Hazard No. 4) Seam Coarse Rejects 113
3.2.5.5.1 Description of Commercial Roasting Trials Provided by Nex-Gen Industries 113
3.2.5.5.2 Comparison Laboratory Testing of the Leaching Characteristics of the Commercially Roasted Material 115
3.2.5.5.3 Preliminary Conclusion of the Effects of Commercial Roasting Services by Nex-Gen Industries 117

3.3 Solvent Extraction 117
3.3.1 Solvent Extraction Rougher/Cleaner Testing 117
3.3.1.1 Extractant Concentration Effect on Solvent Extraction 117
3.3.1.2 Effect of Different Anions on Solvent Extraction 118
3.3.1.3 Bench-Scale Testing of the Solvent Extraction Proposed for this Project 118
3.3.1.3.1 Laboratory Set-up of Rougher Loading, Scrubbing and Stripping 118
3.3.1.3.2 Continuous Flow SX Testing using Pocahontas No. 3 Coal Seam Preparation Plant Reject Samples 119
3.3.1.4 Evaluation of Extraction Characteristics of the REEs and Contaminant Elements 123
3.3.1.4.1 Rougher Stage Activities 123
3.3.1.4.2 Extraction of Metal Ions in the Stripped Solution from the Rougher Stage 124
3.3.1.4.3 Cleaner Circuit Simulations 125
3.3.1.4.4 Precipitation of REO From Stripped Solution by use of Oxalic Acid 128
3.3.1.4.5 Analysis of the Final Rare Earth Oxide Product 128
3.3.1.4.6 Ascorbic Acid Optimization to Control Iron (Fe) Contamination of the Leach Liquor from the Pocahontas No. 3 Coal Rejects 129
3.3.1.4.7 Scandium (Sc) Recovery from Pocahontas No. 3 Seam Middlings Rejects 131
3.3.1.5 Selective Loading of REEs in Solvent from Ferroglobe's Blue Gem Seam Tailings Materials 132
3.3.1.6 Characteristics of Individual REEs with Extractants 133
3.3.1.6.1 Effects of Extractants and Parameters on Individual REEs Collected from W. Ky No. 13 (a.k.a., Baker) Coal Seam Acid Mine Drainage (AMD) 133
3.3.1.7 Exploration of the Separation of Individual REEs (Y, Dy, and Nd from W. Ky. No. 13 Coal Seam AMD) 136
3.3.1.7.1 Stage One 137
3.3.1.7.2 Stage Two 138
3.3.1.7.3 Stage Three 138
3.3.1.7.4 Procedure: Extraction Experiment, Stage One 138
3.3.1.7.5 Procedure: Distribution Isotherm, Stage Two 139
3.3.1.7.6 Extraction Experiment, Stage One 139
3.3.1.7.7 Effect of Extractant pH 140

3.3.1.7.8 Effect of Extractant Concentration of the Recovery of Dy and Y using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD 143
3.3.1.7.9 Distribution Isotherm, Stage Two....145
3.3.1.7.10 Distribution Coefficients....147
3.3.1.7.11 Saponification Refinements....149

3.4 Detailed Flow Diagram....153
3.4.1 Flow Diagram Description....153
3.4.1.1 Circuit 1 – Feed Preparation....154
3.4.1.2 Circuit 2 – Acid Leaching and Waste Treatment....155
3.4.1.3 Circuit 3 – Rougher Solvent Extraction....157
3.4.1.4 Circuit 4 – Cleaner Solvent Extraction....158
3.4.1.5 Circuit 5 – Scandium Solvent Extraction....159

3.5 Flowsheet Simulation to Evaluate Process Circuitry for REE Concentration from Coal-based Sources....161
3.5.1 Flowsheet Development Software Selection....161
3.5.1.1 Evaluation of LIMN....161
3.5.1.2 Evaluation of REESim....161
3.5.1.2.1 REESim Software Description....161
3.5.2 Components of REESim Circuit Simulation Workbook for Project MMA 29956....162
3.5.3 Component Flow Rates Generated by REESim for Project MMA 29956....164
3.5.4 Circuit-by-Circuit Performance Indicators....171
3.5.4.1 Summary Discussion of the Simulation Results for Project MMA 29956....171
3.5.5 Detailed Listing of Equipment Requirements of a 1 tph Feed Rate Pilot Plant....184
3.5.6 MATLAB App for SX Design....196

3.6 Flow Sheet of 1 tph Feed Pilot Plant....202
3.6.1 Engineering Drawings Needed for Construction/Fabrication and Installation of System....202
3.6.1.1 Utilization of the Flow Sheets from an Existing Pilot Plant....206
3.6.1.2 Previously Used Pilot Plant Construction Drawings Showing Spatial Layout of Equipment/Piping/Plumbing/Electrical/Mechanical....210
3.6.1.2.1 Crushing and Grinding Equipment and Layout....210
3.6.2 Design Sufficient for CapEx Estimates....216

3.7 Project Risk and Fatal Flaw Analysis....216
3.7.1 Risk and Fatal Flaw Analysis Methodology....216
3.7.2 Limits of the Analysis Procedure....217
3.7.2.1 Risk and Fatal Flaw Analysis Matrix....217
3.7.3 Summary of Significant Concerns....223
3.7.3.1 Timing of Regulatory Permits, Agreements or Contracts....223
3.7.3.2 Performance of the Proposed Solvent Extraction Process....223

3.7.3.3 Roaster Performance with a Coal Preparation Plant Rejects Feedstocks 223

3.8 Acknowledgments 223

3.9 References 224

TABLES (IN THE TEXT)

Table 3-1: Coal and Coal Byproduct Feedstock Source Evaluation Sequence 1
Table 3-2: West Virginia Mining Complex, Lower Kittanning Seam, REE Concentration of Tails of Plant Circuit Samples after HHS 2
Table 3-3: Listing of Process Streams Sampled for REE Analysis 3
Table 3-4: Pocahontas No. 3 Coal Seam Preparation Plant HAM and LAM Testing Matrix 15
Table 3-5: Pocahontas No. 3 Coal Seam Preparation Plant Decarbonization Test Summary 17
Table 3-6: Pocahontas No. 3 Coal Seam Preparation Plant LAM Decarbonization Results 17
Table 3-7: Pocahontas No. 3 Coal Seam Preparation Plant HAM 9.5 x 1 mm Float Sample Decarbonization Results 18
Table 3-8: Pocahontas No. 3 Coal Seam Preparation Plant HAM 9.5 x 1 mm Sink Sample Decarbonization Results 18
Table 3-9: Pocahontas No. 3 Coal Seam Preparation Plant HAM 1.0 x 0.15 mm Sample Decarbonization Results 18
Table 3-10: Pocahontas No. 3 Coal Seam Preparation Plant DriJet™ X-Ray Sorter Test Results 25
Table 3-11: Pocahontas No. 3 Coal Seam Preparation Plant Analysis of LAM Particles with Different Colors from X-Ray Scans 27
Table 3-12: Low (Blue Gem) and High Ash (Jellico) Test Matrix of Ferroglobe Flotation Circuit and Screenbowl Sampling 28
Table 3-13: Particle Size Analysis of Flotation Feed for Three (3) Percent Ash (Jellico) Run 28
Table 3-14: HHS Test Results on Flotation Feed (As-Received) Material Targeting Three (3) Percent Product Ash (Jellico) Content 29
Table 3-15: Particle Size Analysis of the Flotation Feed Collected during the 1 Percent Ash (Blue Gem) Plant Run 29
Table 3-16: HHS Separation Performance Achieved on Flotation Feed (As-Received) Collected from the 1 percent Ash Run Results 29
Table 3-17: Particle Size Analysis of the Flotation Product Collected during the 3 Percent Ash (Jellico) Plant Run 30
Table 3-18: HHS Performance when Treating Flotation Product (As-Received) Collected during a One (1) Percent Ash Plant Run 30
Table 3-19: Size Analysis of Flotation Product for One (1) Percent Ash (Blue Gem) Run 31
Table 3-20: HHS Performance when Treating Flotation Product (As-Received) Collected during a One (1) percent Ash (Blue Gem) Plant Run 31
Table 3-21: Particle Size Analysis of the Flotation Tailings Sample Collected during the Three (3) Percent Ash (Jellico Seam) Plant Run 31
Table 3-22: HHS Performance when Treating Flotation Product (As-Received) 32
Table 3-23: Particle Size Analysis of the Flotation Tailings Sample Collected during the One (1) Percent Ash Plant Run 32
Table 3-24: HHS Performance when Treating Flotation Product (As-Received) Collected during a One (1) Percent Ash Plant Run 32
Table 3-25: Particle Size Analysis of the Screenbowl Effluent Sample Collected during the Three (3) Percent Ash Plant Run 33

Table 3-26: Particle Size Analysis of the Screenbowl Effluent Sample Collected during the One (1) Percent Ash Plant Run ... 33
Table 3-27: Particle Size Analysis of Low Ash (Blue Gem) Spiral Concentrate ... 35
Table 3-28: HHS Results for the Low-Ash Blue Gem Undersize Material (-0.25 mm) ... 35
Table 3-29: HHS Results for the Low-Ash Blue Gem Undersize Material Ground to D80=95 µm ... 36
Table 3-30: HHS Results for the Low-Ash Blue Gem Undersize Material Ground to D80=24 µm ... 36
Table 3-31: HHS Results for the Low-Ash Blue Gem Oversize Material Ground to D80=270 µm ... 36
Table 3-32: HHS Results for Low-Ash Blue Gem ... 37
Table 3-33: HHS Results for Low Ash Blue Gem Oversize Ground to D80=35 µm ... 37
Table 3-34: Results Obtained from HHS Processing of Ferroglobe’s Blue Gem Seam Coal (Plus 0.25 mm Sieve Oversize Ground to a Feed D80 Size of 4 Microns) ... 37
Table 3-35: Blue Gem Seam - TREE, LREE, HREE, and Individual REE Contents on a Whole Sample Basis as well as Heavy to Light REE Content Ratio in the Float-Sink Products ... 38
Table 3-36: Size Analysis of Mid-Ash Jellico Spiral Concentrate ... 38
Table 3-37: HHS Results for Mid-Ash Jellico Undersize ... 39
Table 3-38: HHS Results Mid-Ash Jellico Undersize Ground to D80=75 µm ... 39
Table 3-39: HHS Results for Mid-Ash Jellico Undersize Ground to D80=30 µm ... 39
Table 3-40: HHS Results for Mid-Ash Jellico Oversize Ground to D80=230 µm ... 40
Table 3-41: HHS Results for Mid-Ash Jellico Oversize Ground to D80=75 µm ... 40
Table 3-42: HHS Results for Mid-Ash Jellico Oversize Ground to D80=28 µm ... 40
Table 3-43: Results Obtained from HHS Processing of Arq As-Received Pond Fines (d80=67 um) ... 41
Table 3-44: Results Obtained from HHS Processing of Arq Micronized Pond Fines (D80=12 um) ... 42
Table 3-45: Results Obtained from HHS Processing of Arq Micronized Pond Fines (d80=8 um) with Multiple Sequential Stages of Cleaning and Recleaning ... 42
Table 3-46: Results Obtained from HHS Processing of Arq Pond Fines (D80=5 microns) with Multiple Stages of Cleaning and Sodium Silicate Dispersant (for Selected Tests) ... 43
Table 3-47: Results Obtained from HHS Processing of Micronized (D80=4 um) Clean Coal Agglomerates from an Initial Stage of HHS Processing on As-Received Arq Pond Fines ... 44
Table 3-48: Flotation Test Results for the Leatherwood Thickener Underflow ... 46
Table 3-49: HHS REE Recovery Test Results for Leatherwood Plant Thickener Underflow, Hazard 4 Seam (Test 1 - NaOL) ... 47
Table 3-50: HHS REE Recovery Test Result for Leatherwood Thickener Plant Underflow, Hazard 4 Seam (Test 2 - KOHX) ... 48
Table 3-51: HHS Test Results for Leatherwood Thickener Underflow (Test 3 – NaOL) ... 48
Table 3-52: HHS REE Recovery Test Results for Leatherwood Plant Thickener Underflow, Hazard 4 Seam (Test 4 – KOHX) ... 48
Table 3-53: Leatherwood Plant Thickener Underflow Sample HHS Decarbonization Results (Hazard 4 Seam) ... 49
Table 3-54: HHS REE Results of Blunged Leatherwood Thickener Underflow ... 51
Table 3-55: Leatherwood Thickener Underflow Coal Addition HHS REE Test “a” Results ... 53
Table 3-56: Leatherwood Thickener Underflow Coal Addition HHS REE Test “b” Results ... 54
Table 3-57: Leatherwood Thickener Underflow Coal Addition HHS REE Test “c” Results ... 56
Table 3-58: Pocahontas No. 3 Coal Seam Channel Sample C-1 ... 59
Table 3-59: Ash Content and TREE Content Results of Unroasted and Roasting Products of Minus (-)0.18 mm Pocahontas No. 3 Coal Seam Middling Decarbonized Flotation Tailings ... 68

Table 3-60: Ash Content and REEs Composition Results of Roasting Products from Minus (-)10 micron Pocahontas No. 3 Coal Seam Middling Decarbonized Flotation Tailings 68
Table 3-61: Dry volatile, Fixed Carbon and Ash Contents of the Unroasted and Roasted Pocahontas No 3 Coal Seam Coarse Rejects Samples (Two hours of Roasting)................ 71
Table 3-62: Contents of Al, Fe, Ca, Mg (%) and Trace Elements (ppm) in the Pocahontas No. 3 Coal Seam Preparation Plant Samples Roasted at 600℃ for Two (2) Hours...................... 81
Table 3-63: Results of XRD Examination of the Roasted Fireclay (Hazard No. 4) Rejects Crystalline Structure... 104
Table 3-64: Elemental Analysis of the Final REO Generated from Pocahontas No. 3 Coal Seam Preparation Plant Middlings Reject Material 129
Table 3-65: Extraction Parameters (Based on REO Collected from W. Ky. No. 13 Coal Seam AMD) 137
Table 3-66: Extractant Dosage Utilized 140
Table 3-67: Log of the Distribution Coefficients of 5 Percent Depha 10 Percent TBP System (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)........ 148
Table 3-68: Separation Coefficients of 5 Percent Depha 10 Percent TBP System (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)....................... 149
Table 3-69: Linearization of the Distribution Coefficients of Table 3-67, Including R^2 fit (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)........ 149
Table 3-70: Summary of ID Numbers, Worksheet/tab Names and Unit Operation Descriptions......... 163
Table 3-71: Mass Rate Summary for the Overall Plant.. 165
Table 3-72: Mass Rate Summary for the Feed Preparation Circuit...................................... 166
Table 3-73: Mass Rate Summary for the Acid Leaching and Waste Treatment Circuit 167
Table 3-74: Mass Rate Summary for the Rougher Solvent Extraction Circuit 168
Table 3-75: Mass Rate Summary for the Cleaner Solvent Extraction Circuit...................................... 169
Table 3-76: Mass Rate Summary for the Scandium Solvent Extraction Circuit 170
Table 3-77: REESim Predicted Assay Values for the Overall Plant Operations.................................. 172
Table 3-78: REESim Distribution Values for the Overall Plant Adjusted for Pilot Plant Experience... 173
Table 3-79: Assay Values of the Feed Preparation Circuit 174
Table 3-80: Distribution Values to the Feed Preparation Circuit 175
Table 3-81: Assay Values for the Acid Leaching and Waste Treatment Circuit.................................. 176
Table 3-82: Distribution Values for the Acid Leaching and Waste Treatment Circuit 177
Table 3-83: Assay Values for the Rougher Solvent Extraction Circuit... 178
Table 3-84: Distribution Values for the Rougher Solvent Extraction Circuit 179
Table 3-85: Assay Values for the Cleaner Solvent Extraction Circuit 180
Table 3-86: Distribution Values for the Cleaner Solvent Extraction Circuit.. 181
Table 3-87: Assay Values for the Scandium Solvent Extraction Circuit.. 182
Table 3-88: Distribution Values for the Scandium Solvent Extraction Circuit 183
Table 3-89: Detailed Equipment List for Crushing and Grinding Operations (Cost Center 01)........... 185
Table 3-90: Detailed Equipment List for Roasting and Scrubbing Operations (Cost Center 02) 186
Table 3-91: Detailed Equipment List for Acid Leaching Operations (Cost Center 03) 187
Table 3-92: Detailed Equipment List for Solvent Extraction Operations (Cost Center 04).................. 188
Table 3-93: Detailed Equipment List for Washing and Tailings Operations (Cost Center 05)............. 188
Table 3-94: Detailed Equipment List for Water Treatment Operations (Cost Center 06)..................... 189
Table 3-95: Detailed Equipment List for Ancillary Operations (Cost Center 07)................................ 190

Table 3-96: Equipment Specifications for Crushing and Grinding (Cost Center 01) 191
Table 3-97: Equipment Specifications for Roasting and Scrubbing (Cost Center 2) 192
Table 3-98: Equipment Specifications for Acid Leaching (Cost Center 3) 193
Table 3-99: Equipment Specifications for Solvent Extraction (Cost Center 04) 194
Table 3-100: Equipment Specifications for Washing and Tailings (Cost Center 05) 194
Table 3-101: Equipment Specifications for Water Treatment (Cost Center 06) 195
Table 3-102: Equipment Specifications for Ancillary Operations (Cost Center 07) 196
Table 3-103: Matrix Formulation for a Load, Scrub, Strip System each with 3 Stages as Shown in Figure 3-139 199
Table 3-104: Descriptions of the Proposed Pilot Plant Equipment 216
Table 3-105: Risk Analysis Related to Feedstock Material 217
Table 3-106: Risk Analysis Related to Crushing and Grinding Processes 217
Table 3-107: Risk Analysis Related to the Ore Roasting Process 218
Table 3-108: Risk Analysis Related to the Leaching Process 219
Table 3-109: Risk Analysis Related to the Roughing SX Process 220
Table 3-110: Risk Analysis Related to the Cleaning and Saponification SX Processes 221
Table 3-111: Risk Analysis Related to the Refining SX Process 221
Table 3-112: Risk Analysis Related to the Process Waste Handling 222
Table 3-113: Risk Analysis Related to the Regulatory Matters 222

FIGURES (IN THE TEXT)

Figure 3-1: Simplified Flowsheet for the Pocahontas No. 3 Seam Processing Facility 3
Figure 3-2: TREE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant (Left Plot = whole-sample basis; Right Plot= ash-residue basis) 5
Figure 3-3: HREE concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant (Left Plot= whole-sample basis; Right Plot = ash-residue basis) 5
Figure 3-4: LREE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant (Left Plot= whole-sample basis; Right Plot = ash-residue basis) 6
Figure 3-5: Ratios of Heavy-to-Light REEs versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant 7
Figure 3-6: Ratios of Critical-to-Noncritical REE versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant 7
Figure 3-7: TREE Concentrations versus Ash Contents for Coarse (>1 mm) Coal Samples only Collected from the Pocahontas No. 3 Coal Seam Preparation Plant 8
Figure 3-8: TREE Concentrations versus Ash Contents for Fine (<1 mm) Coal Samples Only Collected from the Pocahontas No. 3 Coal Seam Preparation Plant 8
Figure 3-9: Elemental Concentration versus Ash Content Plots for Sc, Y, La, Eu, Gd and Tb 10
Figure 3-10: Elemental Concentration versus Ash Content Plots for Ce, Pr, Nd, Dy, Ho and Eu 11
Figure 3-11: Elemental Concentration versus Ash Content Plots for Sm, Lu, Tm and Yb 12
Figure 3-12: Concentration of Thorium as a Function of Ash Content for the Pocahontas No. 3 Coal Seam Preparation Plant Samples 13
Figure 3-13: Correlation between TREE and Nd Concentrations for the Pocahontas No. 3 Coal Seam Preparation Plant Samples 13
Figure 3-14: Measured versus Predicted TREE Concentrations Obtained Using Eq. [3.1] 14

Figure 3-15: (Left) Laboratory Cone Crusher, (Right) Laboratory Disk Grinder 16
Figure 3-16: (Left) Laboratory Ball Mill, (Right) Laboratory Attrition Mill 16
Figure 3-17: Pocahontas No. 3 Coal Seam Preparation Plant Low Ash Middlings (LAM) HHS Coal TREE Analysis 20
Figure 3-18: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Rougher Concentrate (A) 21
Figure 3-19: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Rougher Concentrate (B) 22
Figure 3-20: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Scavenger Concentrate 22
Figure 3-21: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Coal Product (A) 23
Figure 3-22: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Coal Product (B) 23
Figure 3-23: Pocahontas No. 3 Coal Seam Preparation Plant Individual REE Concentrations in DriJet™ X-Ray Sorter Products 25
Figure 3-24: Pocahontas No. 3 Coal Seam Preparation Plant X-Ray scans of LAM DriJet™ Sorter Testing 26
Figure 3-25: Pocahontas No. 3 Coal Seam Preparation Plant Handpicked Particles from LAM Sample 26
Figure 3-26: Pocahontas No. 3 Coal Seam Preparation Plant Individual Analysis of LAM Particles with Different Colors from X-Ray Scans 27
Figure 3-27: Testing Procedure for Cleaned Coal Spiral Concentrate Samples 34
Figure 3-28: Procedure Used to Process the Leatherwood Thickener Underflow Sample to Assess REE Recovery Potential Using a Rougher-Scavenger Froth Flotation Circuit 45
Figure 3-29: Procedure for Leatherwood Plant Thickener Underflow HHS REE Recovery, Hazard 4 Seam (Test 1 – NaOL Collector) 46
Figure 3-30: Procedure for Leatherwood Plant HHS REE Recovery, Hazard 4 Seam (Test 2 – KOHX Collector). 46
Figure 3-31: Procedure for Leatherwood Plant Thickener Underflow HHS REE Recovery, Hazard 4 Seam (Test 3 – NaOL Collector) 47
Figure 3-32: Procedure for Leatherwood Plant Thickener Underflow HHS REE Recovery, Hazard 4 Seam (Test 4 – KOHX Collector) 47
Figure 3-33: (Left) Blunging Impeller, (Right) Blunging Mixing Vessel 49
Figure 3-34: Blunged Sample in the HHS Procedure 50
Figure 3-35: TREE Whole Sample Basis Leatherwood Plant Thickener Underflow; Blunged HHS Feedstock Testing Results (Hazard 4 Seam) 51
Figure 3-36: TREE Ash Basis Leatherwood Plant Thickener Underflow; 52
Figure 3-37: TREE Whole Sample Basis Leatherwood Thickener Underflow Coal Addition Test “a” .. 53
Figure 3-38: TREE Ash Basis Leatherwood Thickener Underflow Coal Addition Test “a” 54
Figure 3-39: TREE Whole Sample Basis Leatherwood Thickener Underflow Coal Addition Test “b” .. 55
Figure 3-40: TREE Ash Leatherwood Thickener Underflow Coal Addition Test “b” 55
Figure 3-41: TREE Whole Basis Leatherwood Thickener Underflow Coal Addition Test “c” 56
Figure 3-42: TREE Ash Basis Leatherwood Thickener Underflow Coal Addition Test “c” 57
Figure 3-43: Apparatus used for the Acid Leaching Tests 58

Figure 3-44: TREEs Leaching Recovery versus Leaching Time of Unroasted Pocahontas No. 3 Coal Seam Channel Sample Stratigraphic Segments 60
Figure 3-45: REEs Leaching Recovery of the West Virginia Pocahontas No. 3 Coal Seam Channel Samples after Standard Acid Leaching for Five (5) Hours 61
Figure 3-46: Typical REE recovery after Five (5) hours Acid Leaching of Selected West Virginia Pocahontas No 3 Coal Seam Channel Samples 61
Figure 3-47: Sample Characterization Flowsheet for Coarse Rejects Samples from the Pocahontas No. 3 Coal Seam Preparation Plant 62
Figure 3-48: TREE Leaching Recovery versus Leaching Time of Float Products (Pocahontas No. 3 Coal Seam) 63
Figure 3-49: TREE Leaching Recovery versus Leaching Time of Sink Products (Pocahontas No. 3 Coal Seam) 64
Figure 3-50: REEs Leaching Recovery of Sink-float Products after Standard Acid Leaching for Five (5) Hours (Pocahontas No. 3 Coal Seam) 65
Figure 3-51: Sc Recovery after Five (5) hours Acid Leaching in Sink-float Products (Pocahontas No. 3 Coal Seam) 66
Figure 3-52: Y Recovery after Five (5) Hours Acid Leaching in Sink-float Products (Pocahontas No. 3 Coal Seam) 66
Figure 3-53: Ce Recovery after 5 hours Acid Leaching in Sink-float Products (Pocahontas No. 3 Coal Seam) 67
Figure 3-54: Nd Recovery after Five (5) hours Acid Leaching in Sink-float Products (Pocahontas No. 3 Coal Seam) 67
Figure 3-55: TREE Leaching Recovery versus Leaching Time of the Pocahontas No. 3 Coal Seam Unroasted and Roasted Decarbonize Flotation Tailings Samples 69
Figure 3-56: REEs Leaching Recovery of Pocahontas No. 3 Coal Seam Decarbonized Flotation Tailings and Roasted Samples after Standard Acid Leaching for five (5) Hours 70
Figure 3-57: Typical REE Recovery after Five (5) hours Acid Leaching in Selected Pocahontas No. 3 Coal Seam Decarbonized Flotation Tailings and Roasted Samples 71
Figure 3-58: Effects of Roasting Temperature on REE Recovery from the Pocahontas No. 3 Coal Seam Preparation Plant Coarse Rejects (2.2 Float, (-) 80-mesh) 73
Figure 3-59: Effects of Roasting Temperature on REE Recovery from the Pocahontas No. 3 Coal Seam Preparation Plant Middlings Product 75
Figure 3-60: Recovery of Individual REEs from the Coarse Rejects and Middlings Circuit Samples Which Were roasted at 600℃ for Two (2) Hours 76
Figure 3-61: Effect of Roasting Temperature and Acid-Type on Ce Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Float of Coarse Rejects, (a) and (b) at 75℃; and the Middlings (c) and (d) at 25℃ 77
Figure 3-62: Effect of Roasting Temperature and Acid-Type on Y Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Float of Coarse Rejects, (a) and (b) at 75℃; and the Middlings (c) and (d) at 25℃ 78
Figure 3-63: Effect of Roasting Temperature and Acid-Type on Dy Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Float of Coarse Rejects, (a) and (b) at 75℃; and the Middlings (c) and (d) at 25℃ 79
Figure 3-64: Effect of Roasting Temperature and Acid-Type on Sc Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Float of Coarse Rejects, (a) and (b) at 75^{O}C; and the Middlings (c) and (d) at 25^{O}C 80

Figure 3-65: Effect of Roasting Temperature and Acid-Type on Al Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 81
Figure 3-66: Effect of Roasting Temperature and Acid-Type on Fe Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 82
Figure 3-67: Effect of Roasting Temperature and Acid-Type on Ca Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 83
Figure 3-68: Effect of Roasting Temperature and Acid-Type on Mg Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 84
Figure 3-69: Effect of Roasting Temperature and Acid-Type on Cu Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG. Coarse Rejects Float Fraction and the Middlings Product 85
Figure 3-70: Effect of Roasting Temperature and Acid-Type on Co Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 86
Figure 3-71: Effect of Roasting Temperature and Acid-Type on Zn Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 87
Figure 3-72: Effect of Roasting Temperature and Acid-Type on V Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 88
Figure 3-73: Effect of Roasting Temperature and Acid-Type on P Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product 89
Figure 3-74: REE Recovery as a Function of Time for both Pocahontas No. 3 Coal Seam Preparation Plant Coarse Rejects and Middlings Roasted Samples (750℃ for 2 hours) at 75℃ and 25℃ 92
Figure 3-75: Recovery of “Extractable REEs” as a Function of Time for both Pocahontas No. 3 Coal Seam Preparation Plant Coarse Rejects and Middlings Roasted Samples (750℃ for 2 hours) at 75℃ and 25℃ 93
Figure 3-76: Recovery of Major Elements Associated with REEs in the Roasted Coarse Rejects and Roasted Middlings Samples of the Pocahontas No. 3 Coal Seam Preparation Plant (750℃ for 2 hours) 95
Figure 3-77: Effects of Acid Concentration on the Leaching Recovery of REEs from Roasted Coarse Rejects from the Pocahontas No. 3 Coal Seam Preparation Plant 97
Figure 3-78: Effects of Acid Concentration on the Leaching Recovery of other Elements Associated with REEs in the Roasted Coarse Rejects Samples of the Pocahontas No. 3 Coal Seam Preparation Plant 98
Figure 3-79: Effects of Roasting on the TREE Recovery from (a) 1.8 SG Float, (b) 1.8-2.2 SG, and (c) 2.2 SG Sink Fraction of the Blue Gem Seam Coarse Reject Samples 100
Figure 3-80: Effects of Roasting on the LREE and HREE Recovery from Blue Gem Seam Rejects: (a, b) 1.8 SG Float; (c, d) 1.8-2.2 SG, and (e, f) 2.2 SG Sink Fractions 101

Figure 3-81: Effects of Roasting on Sc Recovery from (a) 1.8 SG Float, (b) 1.8-2.2 SG, and (c) 2.2 SG Sink Fractions of Blue Gem Coal Coarse Rejects 102
Figure 3-82: Total REE Recovery Achieved by Leaching the Unroasted Products Generated of the No. 76 Plant (Leatherwood) Thickener Underflow Material Samples....... 105
Figure 3-83: TREE Recovery Achieved by Leaching the Roasted Products Generated from the Processing of the Leatherwood Thickener Underflow Material 106
Figure 3-84: Effect of Solids and Acid Concentration on Leaching Recovery and Kinetics for the REEs Associated with the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material 107
Figure 3-85: Effect of Acid Concentration on Leaching Recovery and Kinetics for the REEs Associated with the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay Coarse Rejects Material (Solids Concentration = 20 percent by Weight) 108
Figure 3-86: Effect of Acid Concentration on Leaching Recovery and Kinetics of Fe for the Roasted 1.8 x 2.2 SG fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material (Solids Concentration = 200 gm/L) 109
Figure 3-87: Effect of Acid Concentration on Leaching Recovery and Kinetics of Al for the roasted 1.8 x 2.2 SG fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material; Solids Concentration = 200 gm/L....... 110
Figure 3-88: Effect of Acid Concentration on Leaching Recovery and Kinetics of Ca for the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material (Solids Concentration = 200 gm/L) 110
Figure 3-89: REE and Contaminant Element Recovery as a Function of Leach Time When Using an Acid Concentration of 1M H_2SO_4 and a Solids Concentration of 200 Grams per Liter of Solution 111
Figure 3-90: Effects of Percent Solids on Weak Acid Leaching of Fire Clay (Hazard No. 4) Material....... 112
Figure 3-91: Effect of Acid Concentration and Solids Concentration in Leachate Solution on Li and Co Extraction....... 113
Figure 3-92: Roasting Equipment Operated by Nex-Gen Industries 114
Figure 3-93: Roasted Products Generated by the Commercial Roasting of Fire Clay (Hazard No. 4) Seam Crushed Coarse Rejects Material 114
Figure 3-94: Leaching Characteristics of Fire Clay (Hazard No. 4) Seam Coarse Rejects that was not treated by the Commercial Roaster (Acid Concentration = 1M H_2SO_4) 115
Figure 3-95: Leaching Characteristics of Roasted 'Gray' Fire Clay (Hazard No. 4) Seam Crushed Coarse Rejects Treated by the Commercial Roaster (Acid Concentration = 1M H_2SO_4) 116
Figure 3-96: Leaching Characteristics of Roasted 'Brown' Fire Clay (Hazard No. 4) Seam Crushed Coarse Rejects Treated by the Commercial Roaster (Acid Concentration = 1M H_2SO_4) 116
Figure 3-97: Impact of D2EHPA Concentration on the Metal Ion Extraction at pH 2 using Solvent Extraction of an Artificial Mixture 117
Figure 3-98: Effect of Anionic Species on the Extraction of Metal from an Artificial Mixture....... 118
Figure 3-99: Lab Scale Solvent Extraction Setup for Conducting the Continuous SX Tests at the Hydrometallurgical Laboratory at the University of Kentucky 119
Figure 3-100: Column Flotation Setup used to Decarbonize the Various Coal Rejects Prior to Leaching....... 120

Figure 3-101: Leaching Reactor used in the Hydrometallurgy Lab to Leach the Solids from Various Coal Reject Sources 121
Figure 3-102: Laboratory Schematic for the Rougher Circuit as Implemented on Bench-Scale SX-Setup for Various Coal Rejects Sources 122
Figure 3-103: Bench-Scale Setup for Implementing the Rougher and the Cleaner Circuit of Solvent Extraction of Various Coal Rejects Sources 123
Figure 3-104: Recovery Values of Metal Ions in the Organic Phase for the Leach Liquor with Five (5) Percent DEHPA Solution (Pocahontas No. 3 Coal Seam Preparation Plant Reject Material) 124
Figure 3-105: Extraction Curves of Metal Ions in the Stripped Solution Obtained from the Rougher Stage of the Extraction Process (Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit Rejects) 125
Figure 3-106: Laboratory Schematic for the Cleaner Circuit as Implemented on Bench-Scale of the Pocahontas No. 3 Rejects Samples 126
Figure 3-107: Concentration of Contaminants and TREE in the SX Feed for the Rougher Stage of Extraction (Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit Rejects) 127
Figure 3-108: Concentration of contaminants and TREE in the Stripped Solution Obtained from the Rougher Stage of Extraction (Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit Rejects) 127
Figure 3-109: Concentration of Contaminants and TREE in the stripped Solution Obtained from the Cleaner Stage of Extraction (Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit Rejects) 128
Figure 3-110: Elemental Distribution of REEs in the Final REO Generated from the Pocahontas No. 3 Coal Seam Preparation Plant Middlings Reject Material 129
Figure 3-111: Impact of Different Dosing of Ascorbic Acid on Extraction Curves of Iron with Five (5) Percent DEHPA and 10 percent TBP (Artificial Solution) 130
Figure 3-112: Effect of Molarity of NaOH used for Saponification for the Recovery of Scandium (Sc) (Pocahontas No. 3 Coal Seam Middlings Rejects) 132
Figure 3-113: Distribution of REEs in Ferroglobe’s Gatliff Plant (Blue Gem Seam) Feed versus Stripping Showing Selective Loading of Heavy REEs in the Organic Phase 133
Figure 3-114: Aqueous Rare Earth Element Concentration for Test Solutions (Using REO from W. Ky. No. 13 Coal Seam AMD 134
Figure 3-115: Extraction versus pH for Blended Extraction with Prepared Solution (Using REO from W. Ky. No. 13 Coal Seam AMD) 135
Figure 3-116: Extraction versus pH for Blended Extraction with Prepared Solution for High Concentration Rare Earth Elements (using REO from W. Ky. No. 13 Coal Seam AMD)135
Figure 3-117: Comparison of the UK extractant Blend to Cyanex 572 Manufacturer’s Data (using REO from W. Ky. No. 13 Coal Seam AMD) 136
Figure 3-118: Mixing (Left) and Phase Separation (Right). 139
Figure 3-119: Extraction Curve Using Two (2) Percent DEHPA (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) 140
Figure 3-120: Concentration of Individual REE in Feed Samples used for the Test (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) 141
Figure 3-121: Extraction Curve using 2 Percent DEHPA and 10 Percent TBP (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) 142

Figure 3-122: Extraction Curve Using 2 Percent Cyanex (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) ... 143
Figure 3-123: Effect of Extractant Dosage on Percent Extraction for Cyanex (at pH 0.5) and DEHPA (at pH 0.47) (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) ... 144
Figure 3-124: Effect of Extractant Dosage on Percent Extraction for Cyanex (at pH 0.99) and DEHPA (at pH 1.001) (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) ... 144
Figure 3-125: Effect of Extractant Dosage on Percent Extraction for Cyanex (at pH 1.47) and DEHPA (at pH 1.542) (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) ... 145
Figure 3-126: Feed Composition of Sample Used for Distribution Isotherm (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) ... 146
Figure 3-127: Distribution Isotherm for Loading of Y and Dy Mixture (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) ... 147
Figure 3-128: Failure of Solvent Extraction in the Saponification Stage of a Mixed Coal-Related Source Organic from a Pilot Scale Circuit Showing the Formation of a Gel ... 150
Figure 3-129: Post Organic and Solution in a Separatory Funnel after Shakeout ... 151
Figure 3-130: Distribution of REEs and U+Th in the Various Saponification Stages (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) ... 152
Figure 3-131: Distribution of Iron in the Various Saponification Stages (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD) ... 152
Figure 3-132: Process Flow Diagram for REE Extraction from Coal-based Sources ... 154
Figure 3-133: Detailed Process Flow Diagram for the Feedstock Preparation Circuit ... 155
Figure 3-134: Detailed Process Flow Diagram for the Acid Leaching and Waste Treatment Circuit ... 156
Figure 3-135: Detailed Process Flow Diagram for the Rougher Solvent Extraction Circuit ... 157
Figure 3-136: Detailed Process Flow Diagram for the Cleaner Solevent Extraction Circuit ... 159
Figure 3-137: Detailed Process Flow Diagram for the Scandium Solvent Extraction Circuit ... 160
Figure 3-138: Identification Numbers Assigned to Facilitate the Flow Diagram Simulation Work ... 162
Figure 3-139: A Solvent Extraction System Diagram Showing Loading, Scrubbing and Stripping Arranged so that part of the Scrubbing Stream is Refluxed into the Scrubbing Stages ... 197
Figure 3-140: Representation of a Single Stage ... 197
Figure 3-141: MATLAB App Showing the Calculation of SX Parameters ... 201
Figure 3-142: Previously Operated Pilot Plant Potentially Suited for Reutilization ... 202
Figure 3-143: Pilot Plant Original Flow Sheet ... 203
Figure 3-144: P&ID Diagram of the Crushing Segment of the Previously Used Pilot Plant ... 205
Figure 3-145: Previously Used Pilot Plant Crushing and Grinding P&ID Diagram (Repeated Diagram) ... 207
Figure 3-146: Previously Used Pilot Plant Roasting Circuit P&ID ... 208
Figure 3-147: Previously Used Pilot Plant Roasting Gas Scrubbing P&ID ... 209
Figure 3-148: Surveyed Map of the Truss Joist Facility ... 211
Figure 3-149: Location of Crushing and Roasting Equipment in Relation to the Truss Joist Facility ... 212

Figure 3-150: 3D View of the Proposed 1 tph Pilot Plant's Crushing and Grinding Circuit showing Jaw Crusher, Conveyors, Roller Crusher, Screens Ball Mill and Storage Bins 213
Figure 3-151: Alternate View of the Proposed 1 tph Pilot Plant's Crushing and Grinding Circuit 213
Figure 3-152: General Arrangement of the Proposed Relocated 1 tph Pilot Plant Crushing and Grinding Circuit 214
Figure 3-153: Additional Equipment Details of a Proposed Relocated 1 tph Pilot Plant Crushing and Grinding Circuit 215

APPENDIX

3-1 ..... Flow Sheets
3-2 ..... REESim
3-3 ..... REESim Output

3.1 Summary of Investigations of REE Concentration Plant Feedstock Sources

3.1.1 Inventory of Coal and Coal Byproducts Feedstock Sources Examined and Tested in the Course of this Task

The following narratives describe the evolution of the search and investigation for one or more reliable sources of a long-term supplies of coal and coal byproducts that exhibit the appropriate rare earth elements *(REE)* content to support a future REE concentration and marketing enterprise. Given the very short term of this assignment, this aspect of the project relied heavily on the investigative team members' prior knowledge of the relationship of REE content to certain coal beds as well as their ongoing relationships with the coal mining/processing owners-operators. Although many sites were considered, the following coal and coal byproducts producers eventually participated directly in this investigation by making one or more of their facilities accessible to the investigators. The table is arranged in general order of investigation – taken loosely, as often tasks at several sites were undertaken at the same time.

Table 3-1: Coal and Coal Byproduct Feedstock Source Evaluation Sequence

			Location					
Sequence	**Company**	**Facility Name**	**Geographical Description**	**North Coordinates**	**West Coordinates**	**MSHA ID**	**Coal Seam**	**Comments**
1	Northern Appalachian Coal Producer	Mine	West Virginia	39° 20'N	79° 59'W	Multiple	Lower Kittanning	The initial collaborative effort could not be supported by the coal mining and processing company.
		Processing Plant	West Virginia	39° 20'N	79° 59'W	Multiple		
2	Southern Appalachian Coal Producer	Mine	West Virginia	37° 44'N	81° 14'W	Multiple	Pocahontas No. 3	After testing rejects and channel sample response to HHS cleaning and REE content, this initiative was terminated in light of the fact that the middlings material could not be processed to produce a clean coal product with desired quality.
		Processing Plant	West Virginia	37° 44'N	81° 14'W	Multiple		
3	Ferroglobe (d.b.a. Alden Resources, LLC)	Gatliff Tipple	Gatliff, Whitley County, KY	36° 40.698'N	84° 1.051'W	15-09938	Blue Gem and Jellico	Multiple segments of the plant circuit were sampled and analyzed for standard coal quality parameters, response to HHS cleaning and REE content. This initiative was terminated because of relatively low REE content in the samples tested.
		Mine #3, Bain Branch	Bryants Store, Knox County, KY	36° 46.748'N	83° 53.239'W	15-17691	Blue Gem	
		Mine #5, Log Cabin	Bryants Store, Knox County, KY	36° 46.684'N	83° 54.824'W	15-18426	Blue Gem	
		Contractors	Not Available	Not Available	Not Available	N.A.	Jellico	
4	Arc Corbin, LLC (d.b.a. Arq Limited)	Corbin Project, LLC	Corbin, Knox County, KY	36° 56.048'N	84° 5.110'W	15-02134	Mixed Coal Preparation Plant Rejects	A series of HHS studies were conducted in small grab samples of the rejects material the initiative was dropped when collaboration with the host company was not successful
5	Blue Diamond Coal Company and Blue Diamond Mining LLC, (d.b.a. Blackhawk Mining, LLC)	Blue Diamond No. 76 Plant (a.k.a. Leatherwood)	Slemp, Perry County, KY	37° 03.967'N	83° 7.233'W	15-16520	Fire Clay	Participation by both Kentucky River Properties LLC (Lessor to Blue Diamond) as well as the Plant and mine operator (d.b.a. Blackhawk Mining LLC) resulted in an aggressive campaign of drilling, channel sampling, plant rejects sampling and testing.
		Blue Diamond Mine No. 81	Smilax, Leslie County, KY	37° 8.152'N	83° 14.497'W	15-12753		
		Blue Diamond Mine No. 89	Viper, Perry County, KY	37° 10.347'N	83° 6.683'W	15-19405		

3.1.1.1 Evaluation of LIMN

The simulation initially made use of the LIMN[1] flowsheet processor to create, model and analyze different flowsheet configurations. However, as work progressed, it became apparent that the use of this particular tool was limited due to a lack of hydrometallurgical models and constraints associated with software licensing and distribution.

3.1.2 West Virginia Mining Complex, Lower Kittanning Coal Seam

3.1.2.1 Lower Kittanning Coal Seam Mining and Preparation Plant Complex

This coal mining complex produces both steam and metallurgical quality coal from the Lower Kittanning Seam. It played a key role in the compilation of a proposal to **US Department of Energy (*DOE*)**/**National Energy Technology Laboratory (*NETL*)** in December 2016 (FOA-0001627) that was subsequently funded as project MMA 29956. The selection of this Lower Kittanning Seam mining and processing complex and its participation in the proposal pursuant to FOA-0001627 was based on prior HHS and REE work on products extracted from its Lower Kittanning rejects circuits by the **University of Kentucky *(UK)*, *Virginia Tech (VT)***, and **Minerals Refining Company**[2] ***(MRC)***. A sample of the results of that work is presented below.

Table 3-2: West Virginia Mining Complex, Lower Kittanning Seam, REE Concentration of Tails of Plant Circuit Samples after HHS

Sample Site	HHS Product	Product Ash (%)	REE (ppm)	
			Ash Basis	Whole-Basis
6-in Cyclone O/F	Tails	89.5	385.72	345.80
Thickener U/F	Tails	90.32	376.75	340.28
Spiral Rejects	Tails	89.19	351.67	313.66
Spiral Middlings	Tails	90.31	341.04	307.99
Reflux (Off Sieve)	Tails	88.90	252.84	224.77

Source: Dr. Yoon, Ph.D., Virginia Tech

However, events at the coal mine and preparation plant that were beyond the control of the project investigators compelled that alternative feedstock sites be secured.

3.1.3 West Virginia Mining Complex, Pocahontas No. 3 Seam Coal

3.1.3.1 Pocahontas No. 3 Seam Preparation Plant Survey – West Virginia

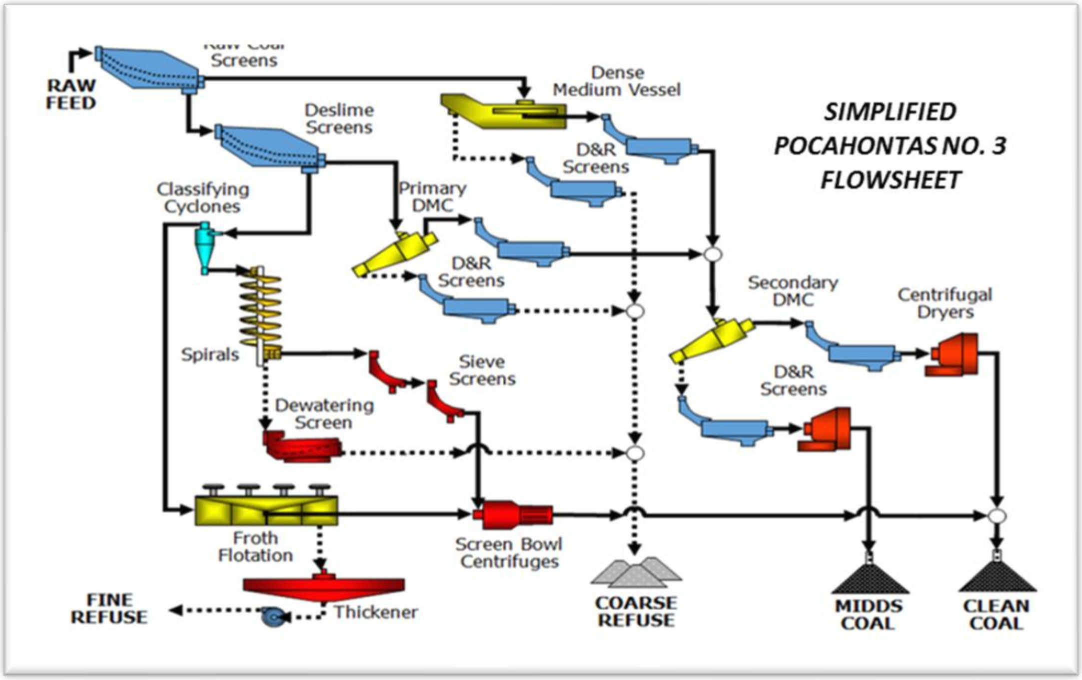
A previous study revealed that the Pocahontas No. 3 Seam coal processed at this preparation plant contains elevated levels of REEs. The plant is fed from an underground mining operation that utilizes continuous mining machines for coal extraction. The mine produces approximately 1 million tons annually of low-volatile metallurgical-quality bituminous coal. A simplified flowsheet for the processing facility is provided in the following figure.

---

[1] The LIMN "The Flowsheet Processor" software has been developed by David Wiseman since starting his company in early 1994.

[2] MRC Massey Building, 5002 Monument Avenue, Richmond, VA 23230. MRC has developed a patented HHS technology that recovers micron-sized particles of coal that are currently disposed of as part of a coal preparation plant waste. The tailings of that HHS process provide access to REE compounds with a significantly reduced interference from carbonaceous particles, thus, the interest to project.
MMA 29956. The market value of the extremely low carbon metallurgical grade coal product captured by HHS can be used to offset the cost of REE concentration.

Figure 3-1: Simplified Flowsheet for the Pocahontas No. 3 Seam Processing Facility



In order to identify the most promising process stream for REE recovery, 40 segments of the plant circuit were sampled. These sample points are identified in the following table. The collected samples were dried, weighed and further segregated into different particle size and density classes. The samples generated from this procedure were subsequently (1) pulverized to below 60-100-mesh, (2) split into small representative lots, (3) subjected to complete combustion to generate a digestible residual ash, (4) thermally digested/decomposed using an appropriate acid mixture, and (5) analyzed using an ICP spectrometer to determine the elemental concentrations of rare earth elements of interest to this project. In total, 188 individual samples were generated, prepared, and analyzed as part of the plant audit conducted under this sampling and characterization task.

Table 3-3: Listing of Process Streams Sampled for REE Analysis

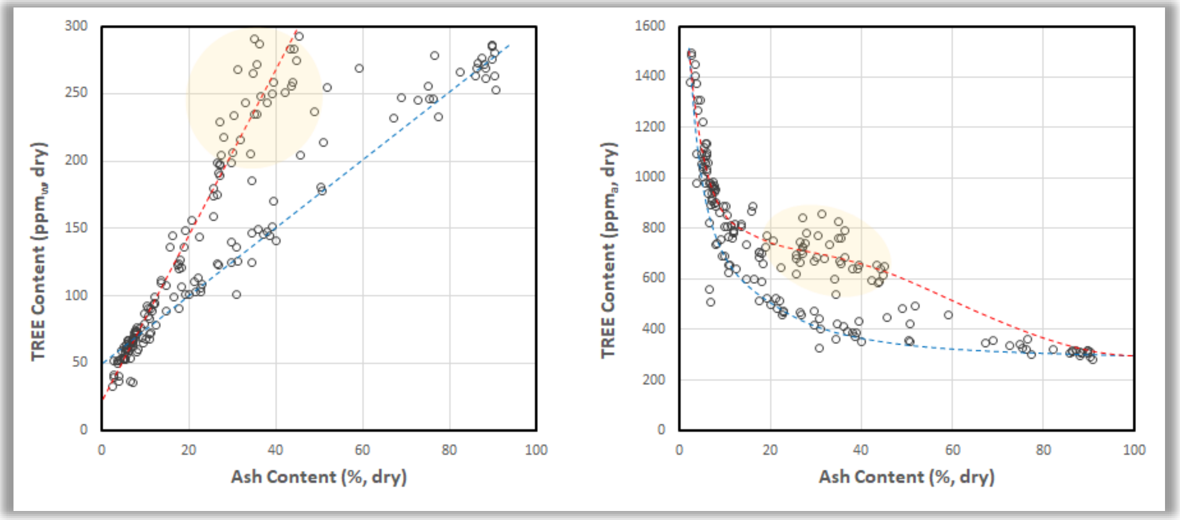
Stream	Stream Description
1	Raw Coal Screen - Top Deck
2	Heavy Media Vessel Feed - Bottom Deck RC Screen
3	Heavy Media Vessel Clean Coal - Top Deck
4	Heavy Media Vessel Rejects - Top Deck
5	Heavy Media Vessel Clean Coal - Bottom Deck
6	Heavy Media Vessel Rejects - Bottom Deck
7	Primary DM Cyclone Feed - Deslime Screen Discharge
8	Primary DM Cyclone Product - D & R Screen Discharge
9	Primary DM Cyclone Rejects - D & R Screen Discharge
10	Secondary DM Cyclone Feed
11	Secondary DM Cyclone Product - D & R Screen Discharge
12	Sec. DM Cyclone Midds Product - D & R Screen Discharge
13	15" Raw Coal Classifying Cyclone Feed

Stream	Stream Description
14	15" Raw Coal Classifying Cyclone Underflow
15	15" Raw Coal Classifying Cyclone Overflow
16	Spiral Feed
17	Spiral Clean Coal - Spiral Fine Wire Sieves Primary Feed
18	Spiral Rejects - Feed to HF Rejects Screen
19	Spiral Secondary Fine Wire Sieve Product
20	Spiral Combined Fine Wire Sieve Total Effluent
21	15" Clean Coal/Effluent Classifying Cyclone Feed
22	15" Clean Coal/Effluent Classifying Cyclone Underflow
23	15" Clean Coal/Effluent Classifying Cyclone Overflow
24	Effluent Fine Wire Sieve Feed
25	Effluent Fine Wire Sieve Product
26	Effluent Fine Wire Sieve Effluent
27	Conventional Flotation Feed
28	Conventional Flotation Concentrate
29	Conventional Flotation Tailings
30	#1 Primary DMC CSI-40 Product
31	#2 Primary DMC CSI-40 Product
32	Screen Bowl Feed #1
33	Screen Bowl Product #1
34	Screen Bowl Main Effluent #1
35	Screen Bowl Feed #2
36	Screen Bowl Product #2
37	Screen Bowl Main Effluent #2
38	Thickener Underflow
39	Sec. DMC Luco 1400 Dryer Product
40	Midds Eb-40 Dryer Product

3.1.3.2 Pocahontas No. 3 Coal Seam Plant Circuit Samples TREE Content Results

Figure 3-2 provides plots of the total rare earth element (*TREE*) concentration versus dry ash content for each sample analyzed from the preparation plant. For convenience, the TREE values have been plotted both as a parts-per-million *(ppm)* concentration on a whole-sample basis (*ppmw*) and on an ash-residue basis (*ppma*). The plotted data points are interesting in that it appears that two grouping of data points can be identified in each plot. For the whole-sample data listed in left side of the graph, the two groupings of data can be largely represented using two straight lines. The lower line falls roughly in line with trends observed for many other coal feedstocks in that the high ash (i.e., 100 percent ash) falls in the range of 250-350 ppmw. In contrast, the upper line reaches this TREE concentration value at a relatively low ash content of approximately 30-45 percent ash. Samples falling in this range, which is shaded in the graphs, represent splits of coal/rock that are elevated in TREEs. The enhanced concentration can also be observed in the ash-basis concentration plot shown in the right side of the graphs. In this case, numerous samples falling in the upper grouping have ash-based TREE concentrations in the 600 to 800 ppma range with corresponding ash contents of 20-45 percent. The TREE concentrations for the lower grouping fall in the range of 400 ppma over the same range of ash values.

Figure 3-2: TREE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant (Left Plot = whole-sample basis; Right Plot= ash-residue basis)



3.1.3.3 Pocahontas No. 3 Coal Seam Preparation Plant Circuit Samples Heavy and Light REE Concentrations

Further examination of the REE concentration data from this particular preparation plant shows that the twin groupings of REEs occurs both for the heavy rare earth elements (*HREEs*) and the light rare earth elements (*LREEs*). These trends, which are illustrated in the following two graphs, indicate that this particular coal contains splits that are elevated in both heavy and light REEs.

Figure 3-3: HREE concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant (Left Plot= whole-sample basis; Right Plot = ash-residue basis)

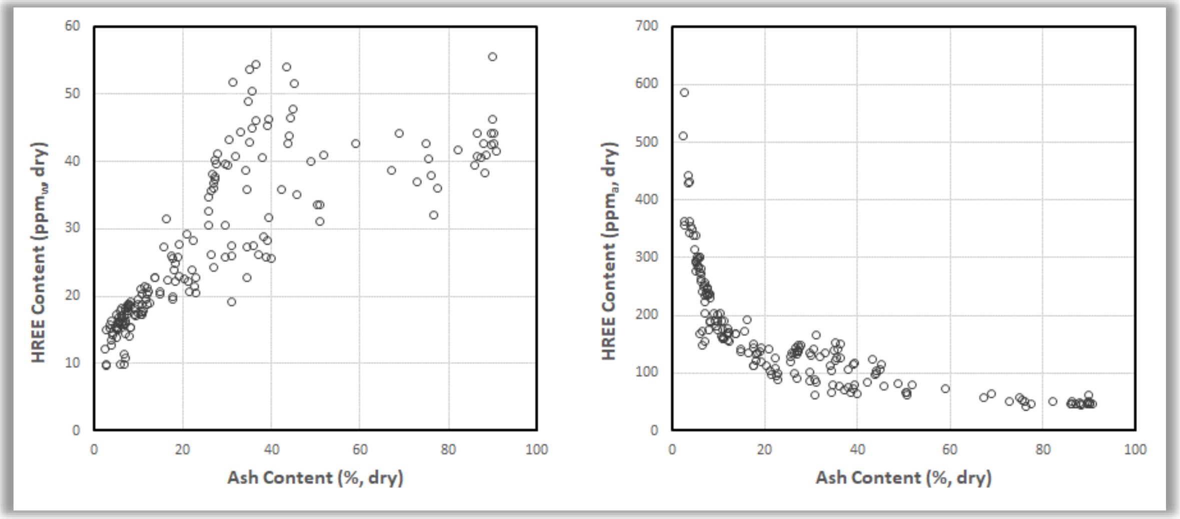
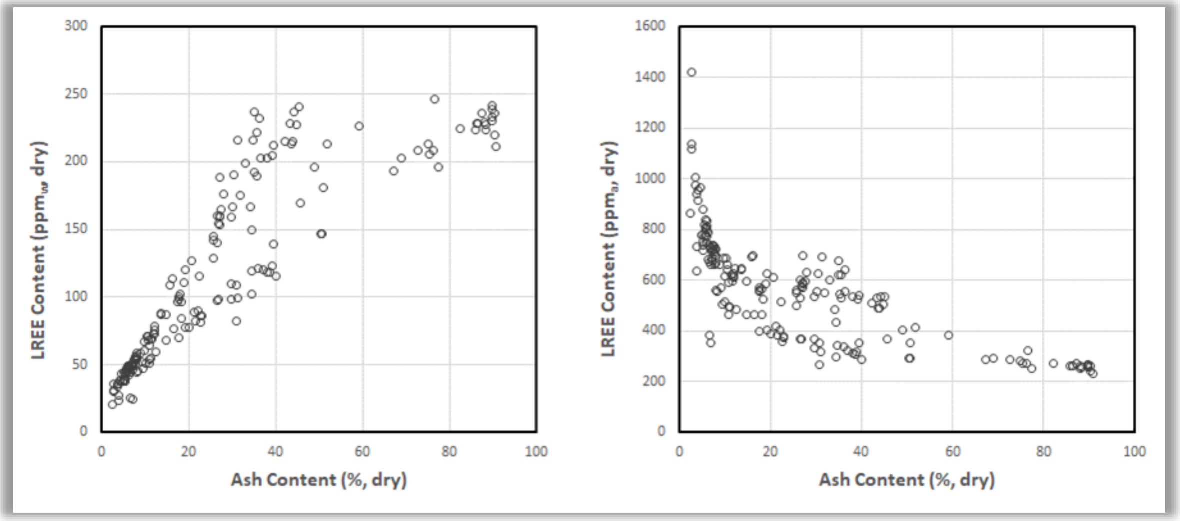


Figure 3-4: LREE Concentrations versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant (Left Plot= whole-sample basis; Right Plot = ash-residue basis)



3.1.3.4 Pocahontas No. 3 Coal Seam Plant Circuit Samples Normalized Ratio of HREE/LREE

However, as shown in the following graph, a plot of the normalized ratio of heavy-to-light REE concentrations versus ash content does not show more than one grouping of data. The ratio linearly increases from about 0.18 heavy-to-light REEs at 100 percent ash to about 0.25 heavy-to-light REEs at approximately 15 percent ash. At this point, the ratio increases sharply to a value approaching 0.6 heavy-to-light REEs at the lowest ash content. A similar shape of curve is observed when plotting the ratio of critical-to-noncritical REEs, as shown in Pocahontas No. 3 Coal Seam Preparation Plant Circuit Samples TREE in Relation to Process Solid Types. In this case, the ratio is about 0.34 at 100 percent ash and increases linearly to about 0.42 at 15 percent ash. The curve then turns sharply with further decreases in ash content, eventually reaching a value exceeding 0.7 critical-to-noncritical REEs for the lowest ash point.

Figure 3-5: Ratios of Heavy-to-Light REEs versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant

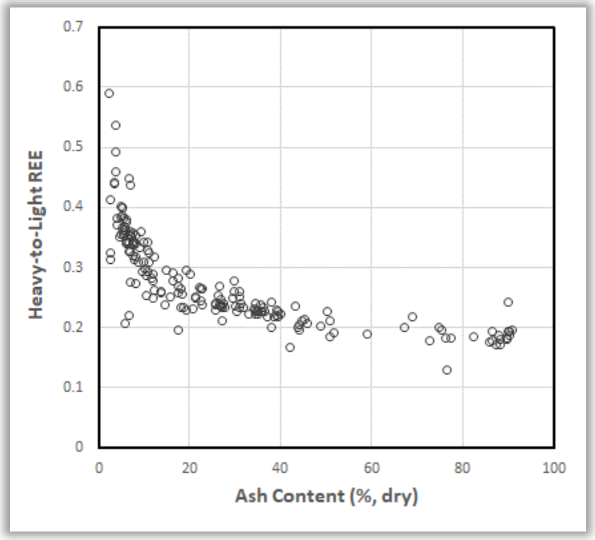
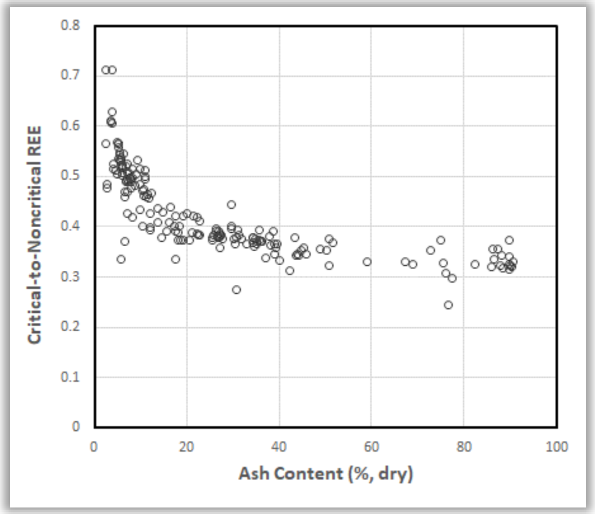


Figure 3-6: Ratios of Critical-to-Noncritical REE versus Ash Contents for Samples Collected from the Pocahontas No. 3 Coal Seam Preparation Plant



<u>*3.1.3.4.1*</u> <u>*Pocahontas No. 3 Coal Seam Preparation Plant Circuit Samples TREE in Relation to Process Solid Types*</u>

A detailed assessment of the sampling data from the preparation plant indicated that the two distinct groupings of TREE data could be explained by fundamental differences in the types of solids reporting to the coarse (plus 1 mm) and fine (minus 1 mm) product streams. These differences can be visually observed in the TREE versus ash plots for the coarse and fine circuits shown in the following two graphs TREE concentrations versus Ash Contents for Coarse (>1 mm) Coal Samples Only Collected from the Pocahontas No. 3 Coal Seam Preparation Plant.

Figure 3-7: TREE Concentrations versus Ash Contents for Coarse (>1 mm) Coal Samples only Collected from the Pocahontas No. 3 Coal Seam Preparation Plant

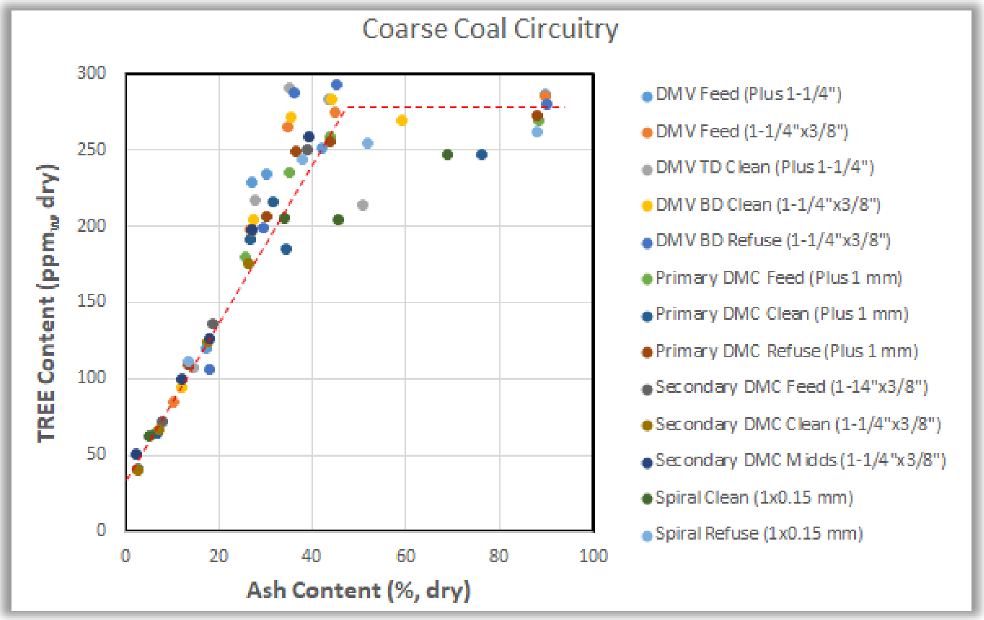
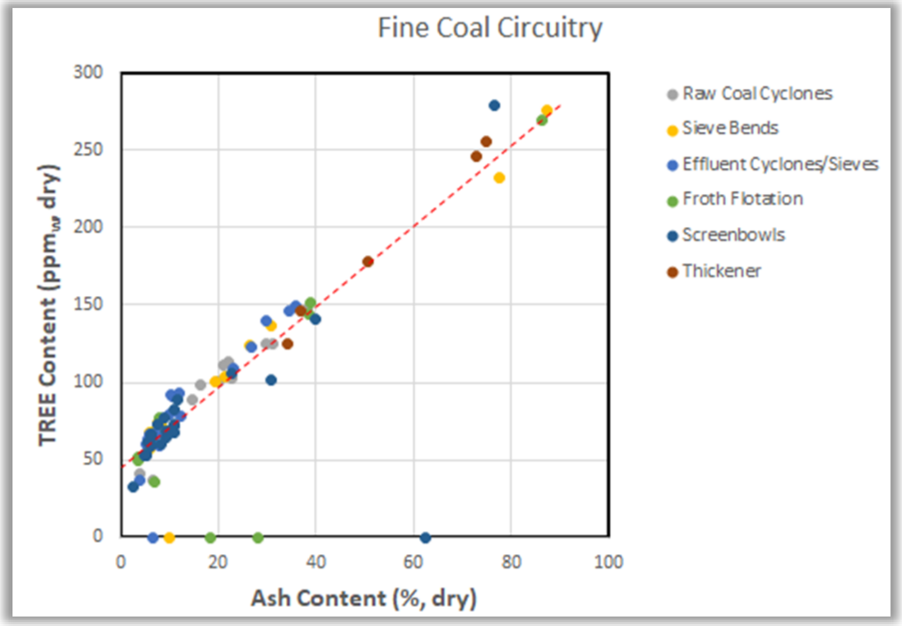

Coarse Coal Circuitry

Figure 3-8: TREE Concentrations versus Ash Contents for Fine (<1 mm) Coal Samples Only Collected from the Pocahontas No. 3 Coal Seam Preparation Plant


Fine Coal Circuitry

A side-by-side comparison of these two plots show that the REE concentration data for samples collected from the coarse coal circuit fall along a trend line that is considerably steeper than that obtained for samples collected from the fine coal circuit. This data suggests that "middlings" particles comprised of composite grains of coal and rock present in the coarse coal circuit have substantially higher concentrations of REEs compared to mixtures of more liberated coal and rock present in the finer size fractions.

3.1.3.5 Pocahontas No. 3 Coal Seam Preparation Plant Circuit Samples REE Concentrations versus Ash Content

For completeness in data reporting, concentration versus ash plots were also constructed for each of the individual elements monitored in the plant audit. The plots are provided in the following series of graphs.

As should be expected, these plots also indicate that two groupings of data exist for essentially all of the elements under consideration. The one notable exception to this trend was observed for scandium *(Sc)*, which primarily shows a linear relationship between ash and elemental concentration when reported on a whole-sample basis. This finding suggests that scandium may have a slightly different mineralogical association than the other elements. Several of the elements, such as dysprosium *(Dy)*, gadolinium *(Gd)*, holmium *(Ho)* and lutetium *(Lu)*, also show significantly more scatter in the data than those with higher elemental concentrations such as lanthanum *(La)* or cerium *(Ce)*. One final noteworthy observation is that the linear trend lines observed in the data do not appear to pass through a zero concentration for a zero ash content for the vast majority of the elements. The presence of a y-axis intercept for these types of plots has historically been used to suggest an organic association for some elements. However, additional detailed analyses of the data are recommended prior to drawing such a conclusion.

Figure 3-9: Elemental Concentration versus Ash Content Plots for Sc, Y, La, Eu, Gd and Tb

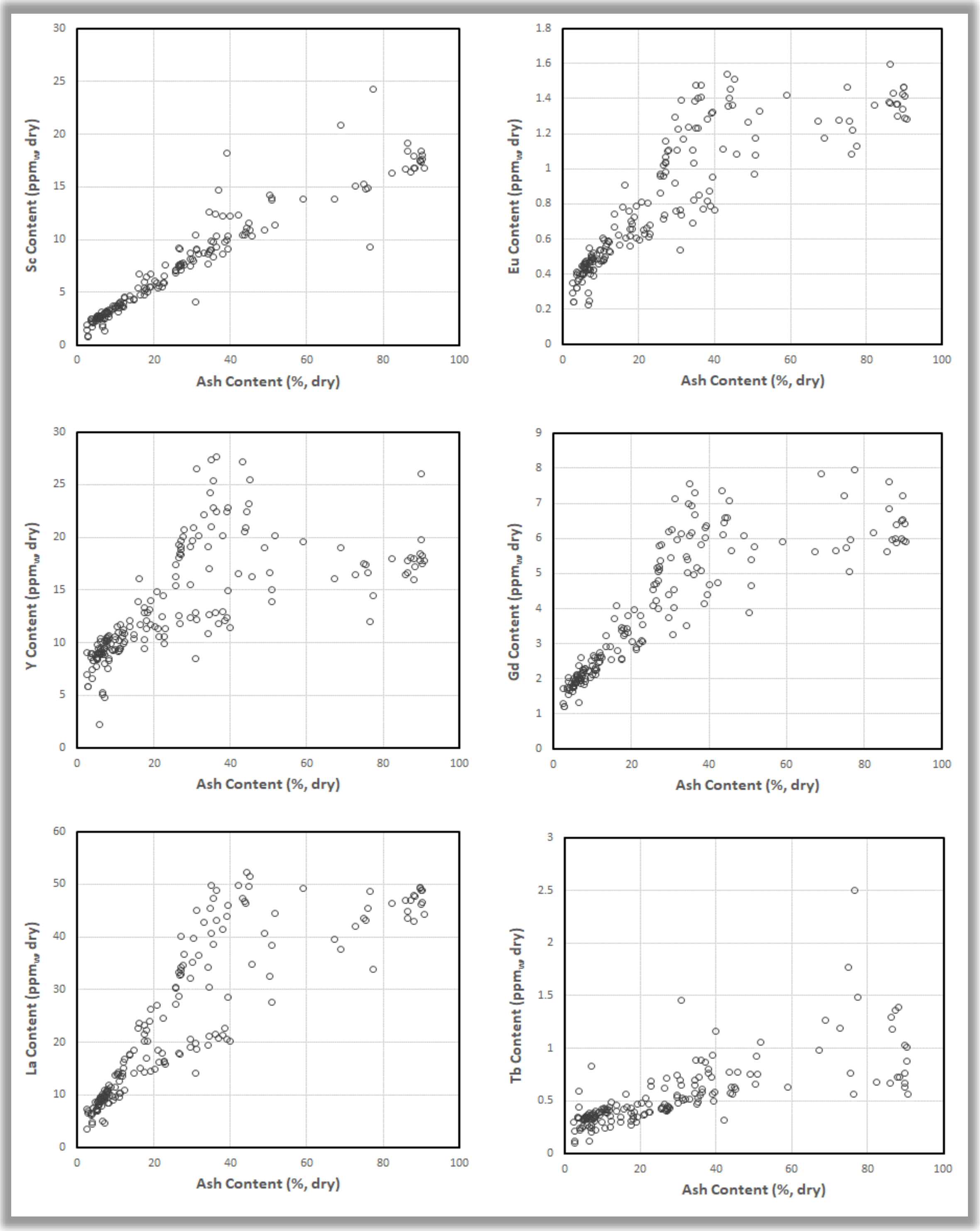


Figure 3-10: Elemental Concentration versus Ash Content Plots for Ce, Pr, Nd, Dy, Ho and Eu

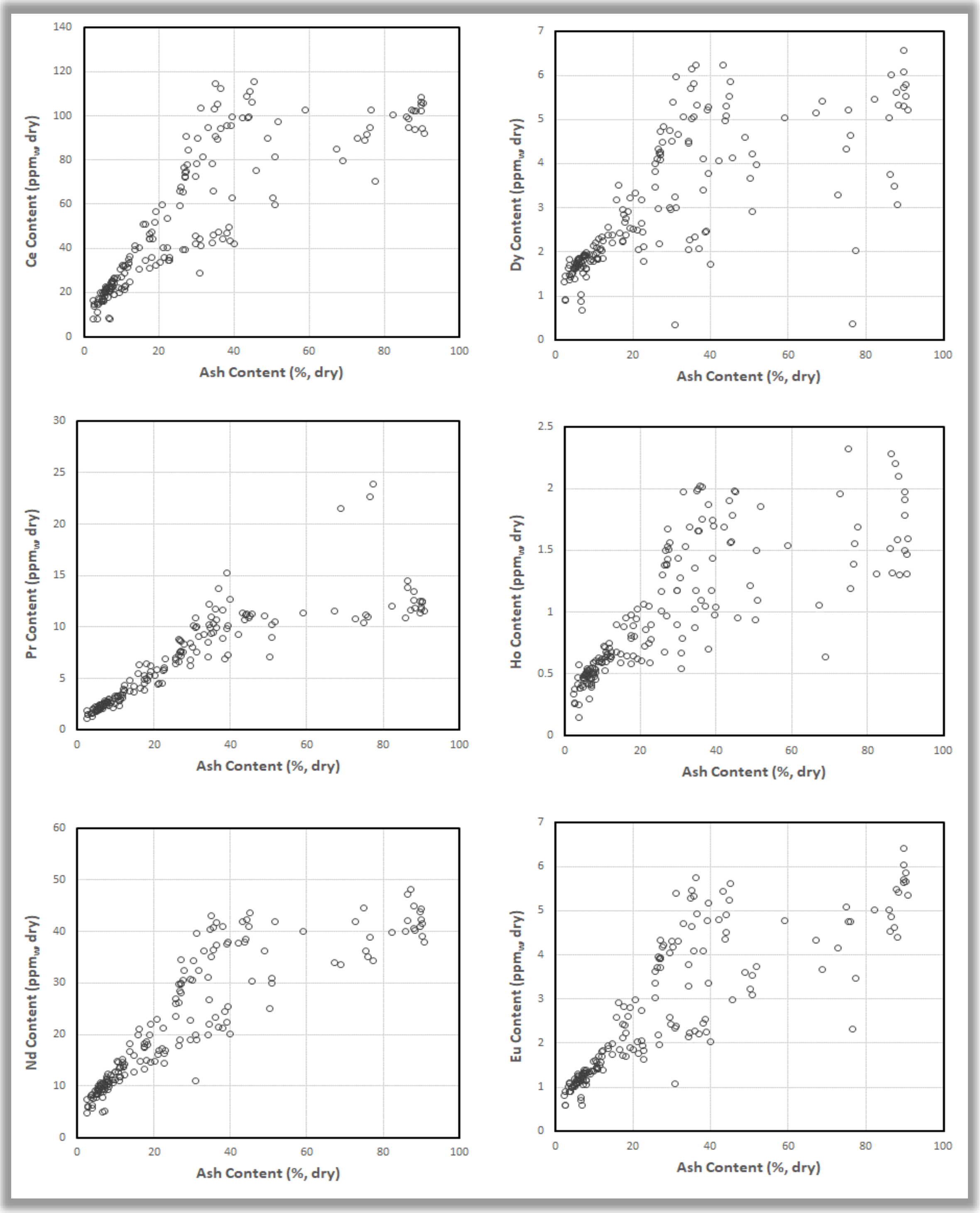
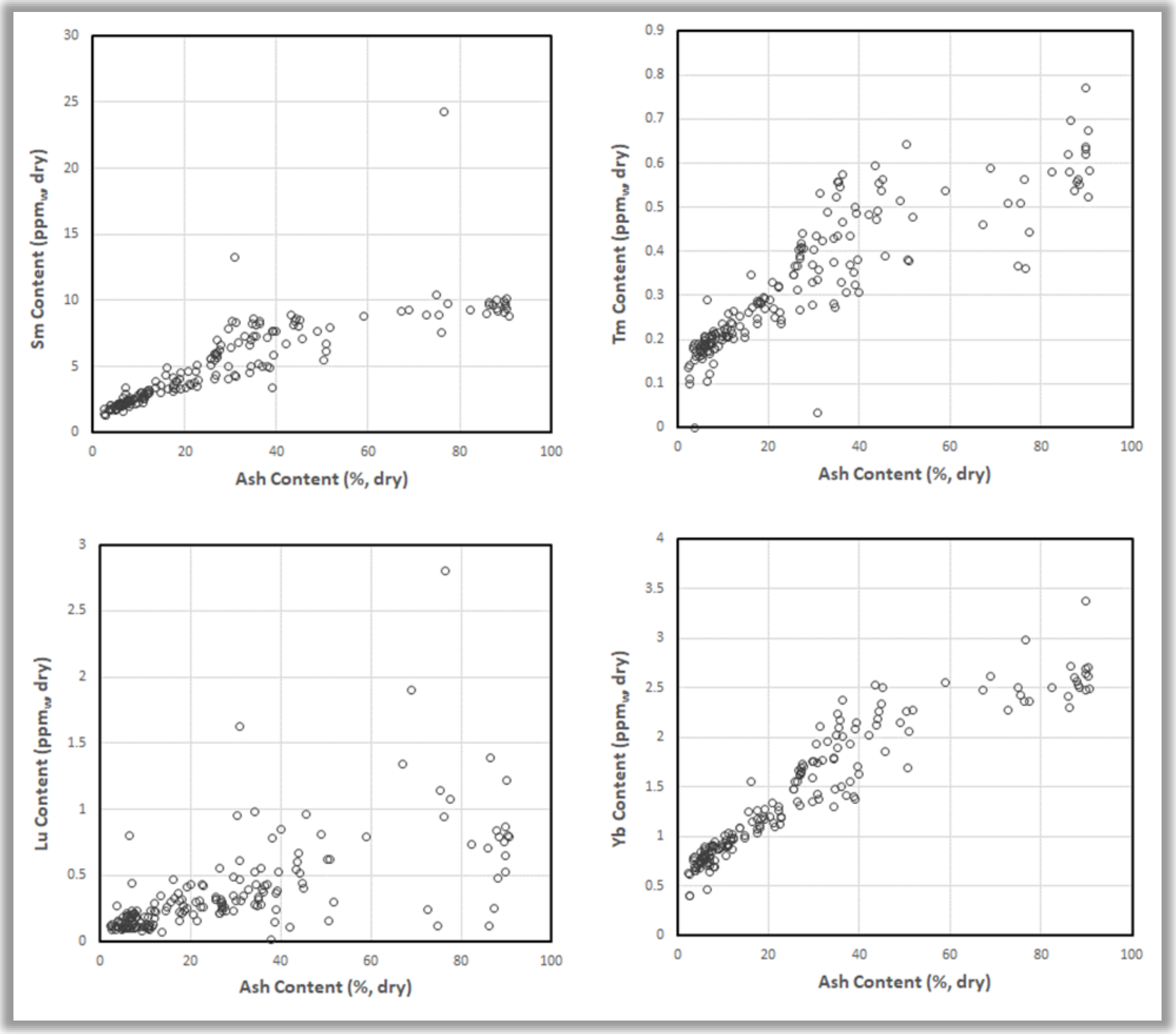
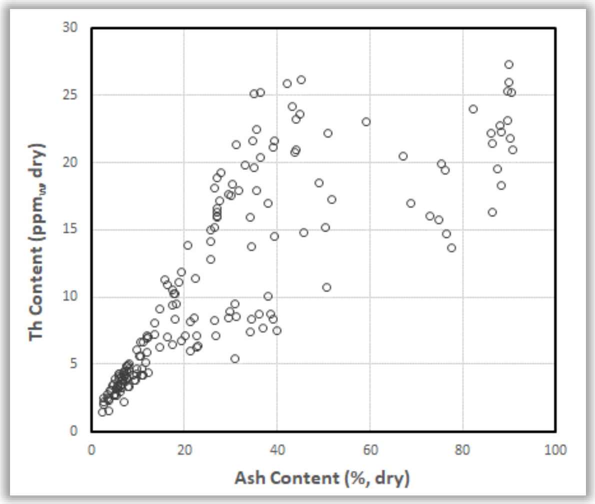


Figure 3-11: Elemental Concentration versus Ash Content Plots for Sm, Lu, Tm and Yb



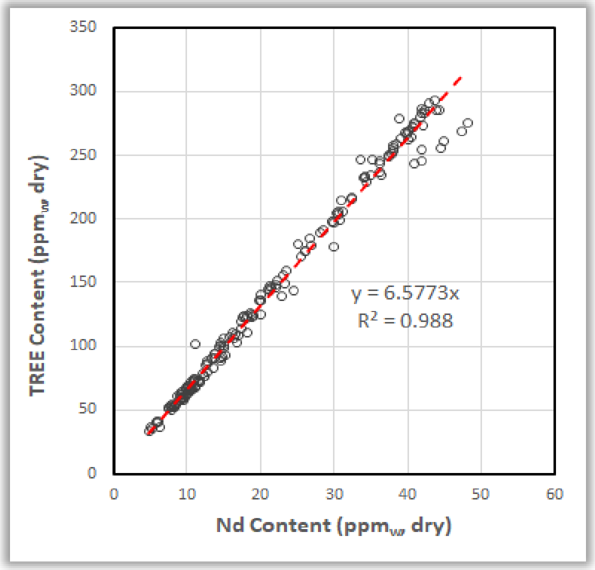
For environmental reasons, one additional element of interest in this project is thorium, which is commonly considered to be the major radioactive element in rare earth products. As such, a plot of the thorium concentration versus ash content was constructed as shown in the following graph. While considerable data scatter was noted, this element appears to also occur within two distinct groupings of data, i.e., a steep slope of elevated concentrations and a shallow slope of lesser concentrations. Although not shown, the most elevated values when reported on an ash-basis occurred over values in the 35-40 percent ash range. This trend suggests that thorium has a mineralogical association with the inorganic solids in this particular coal that is similar to that for the rare earth elements of interest. As such, these feedstocks may require process circuitry to ensure that undesirable radioactive components present in the middling feedstocks are not concentrated in the final saleable products.

Figure 3-12: Concentration of Thorium as a Function of Ash Content for the Pocahontas No. 3 Coal Seam Preparation Plant Samples



The final noteworthy observation derived from the plant data set is the very strong association between the TREE concentration and the element neodymium *(Nd)*. As shown the following graph, a nearly perfect linear correlation exists between the Nd and TREE concentrations for this particular coal.

Figure 3-13: Correlation between TREE and Nd Concentrations for the Pocahontas No. 3 Coal Seam Preparation Plant Samples

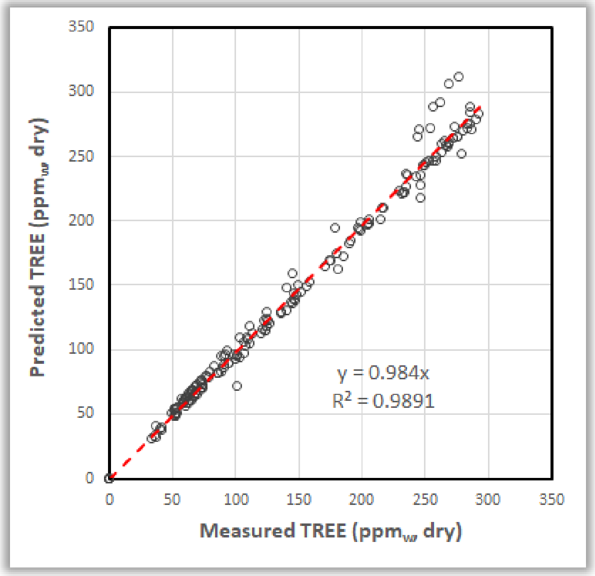


In fact, the correlation coefficient for this relationship was found to be R^2=0.988. As such, an empirical equation relating the Nd and TREE concentrations can be derived as:

TREE (ppmw) = 6.577 x Nd (ppmw) ……………………………………………………………[3.1]

Using this relationship, a plot of measured and predicted TREE concentrations was generated as shown in the following figure. This plot indicates that Eq. [3.1] can be used to estimate TREE concentrations within about +10 ppmw. Similar plots using other elements, such as La or Ce, did not provide estimates as good as that obtained using neodymium. The only points not falling closely on the regression line were the half dozen or so data points in the 40-50 ppmw concentration range for Nd. After reviewing the data, it was found that these points corresponded to fine coal samples collected around the flotation bank. An explanation for the good correlation between Nd and TREE and why some data points for the flotation bank did not follow this trend has not yet been established.

Figure 3-14: Measured versus Predicted TREE Concentrations Obtained Using Eq. [3.1]



3.1.3.6 Pocahontas No. 3 Seam Preparation Plant Circuit Sample Character versus Coal Mining Methods, Plant Low-Ash Middlings and Plant High-Ash Middlings Sample Test Results

Two samples were collected from Pocahontas No. 3 Seam preparation plant. The first of which was a low ash middling sample, henceforth referred to as low ash middling *(LAM)*, and the second was a coarse rejects sample from which a middling sample was produced, henceforth referred to as high ash middling *(HAM)*.

For the LAM sample, plant audit data reports high REE content for the 25-45 percent ash range. This middling product is currently produced from a secondary dense media cyclone circuit with roughly 28-30 percent feed ash. Similarly, plant audit data shows that ash rejects in the range of 50-90 percent ash is also high in REE content.

The HAM sample contains feed ash samples ranging from 49-95 percent.

The HAM and LAM samples were initially screened using 1 millimeter *(mm)* and 0.15 mm sieves. The material coarser than 1 mm was density fractionated using a medium developed by the addition of ultrafine magnetite in an amount that resulted in a medium specific gravity (*SG*) of 2.2. The 2.2 SG sink and float fractions were screened at 9.5 mm.

The test matrix for both the HAM and LAM samples is provided in the following table. In general, this test work will serve to help determine if the low-value middling product can be converted to high-value product while simultaneously producing an REE feedstock from the coal tailings.

Table 3-4: Pocahontas No. 3 Coal Seam Preparation Plant HAM and LAM Testing Matrix

Sample			Coal HHS			REE HHS			Flotation		
ID	Particle Size (mm)	SG	80 x 0	325 x 0	(10 x 0	80 x 0	325 x 0	10 x 0	80 x 0	325 x 0	10 x 0
High Ash Middling (HAM)											
1	+9.5	-2.2 SG									
2	+9.5	+2.2 SG									
3	9.5 x 1	-2.2 SG									
4	9.5 x 1	+2.2 SG									
5	1 x 0.15	--									
Low Ash Middling (LAM)											
6	50 x 12.5	--									

3.1.3.6.1 Pocahontas No. 3 Coal Seam Preparation Plant Circuit High Ash and Low Ash Middlings Sample Treatment Protocols

Sample preparation was completed on the six samples using a series of crushing and grinding techniques to achieve three different size fractions, i.e., -80-mesh, -325-mesh and -10 microns. Each of the three size fractions was treated to recover high quality coal and rare earth minerals in staged separation tests using the HHS process[3]. A conventional froth flotation test was also conducted on each sample and particle size fraction for comparison.

To achieve a top size of 80-mesh, the samples were sequentially crushed using a jaw and cone crusher and then ground using a laboratory disk grinder. Several passes were made through the disk grinder before the samples passed an 80-mesh screen. For each new pass, the samples were screened using a laboratory Ro-Tap at 80-mesh and the oversize fed again through the grinder. The cone crusher and the disk grinder used in his process are shown in the photographic images that follow.

---

[3] HHS is a fine particle separation process that utilizes a hydrophobic liquid to selectively collect fine (hydrophobic) particles while simultaneously displacing surface moisture.

Figure 3-15: (Left) Laboratory Cone Crusher, (Right) Laboratory Disk Grinder

After achieving a top size of 80-mesh, ball and attrition mill grinding was used to achieve the 325-mesh and 10 micron passing sizes. The 80-mesh sample was placed in a laboratory ball mill for wet grinding, using approximately 300 grams of sample. For the grinding media, iron balls were used ranging from 6 to 26 mm with a relatively even distribution. The sample was ground for 20 minutes in the ball mill and then screened and cleaned to achieve a slurry containing (-)325-mesh solids. To achieve the top size of 10 microns for liberation purposes, the sample was ground considerably longer in a laboratory attrition mill. For this grinding step, the media used was small steel balls ranging from 3 to 6 mm in diameter. The samples were subjected to wet grinding in the attrition mill for 1 hour, and then screened and cleaned to achieve a slurry containing 10 micron size particles or smaller. The laboratory ball and attrition mills used for this sample preparation are shown in the following figure.

Figure 3-16: (Left) Laboratory Ball Mill, (Right) Laboratory Attrition Mill

3.1.3.6.2 Pocahontas No. 3 Coal Seam Preparation Plant High Ash and Low Ash Middlings Samples Decarbonization Results

For decarbonization, the ground products were tested using two different methods: (1) HHS and (2) conventional flotation. Both methods were completed using a laboratory Denver two-liter flotation cell. For convenience, the test matrix has been updated to include the best ash and yield results for each of the six samples at the three different size fractions. As shown in the decarbonization summary in the following table, the +9.5-mm sink fraction was not performed and does not have any associated ash or yield values. This is due to the high feed ash in the sample. At roughly 95 percent ash by weight, there was little to no recoverable coal in this feed. However, it might prove to work well in terms of REE recovery. From the summary, it is easy to see that the best product ash (%) results relative to yield (%) results occur with the application of the HHS process to the LAM (50 x 2.5 mm feed) and the HAM (1 x 0.15 mm feed) samples when both are comminuted to a top-size below 10 microns (See the bold bordered cells in the following summary table). This is where the single digit (or close to single digit) % ash content values were achievable for this particular set of coal samples.

Table 3-5: Pocahontas No. 3 Coal Seam Preparation Plant Decarbonization Test Summary

Sample	Size (mm)	Specific Gravity	HHS Product			Flotation Product		
			Ash (%)/Yield (%)			Ash (%)/Yield (%)		
			-80 M	-325 M	-10 µm	-80 M	-325 M	-10 µm
HAM	9.5	Flt 2.2	TBD	TBD	TBD	TBD	TBD	TBD
	9.5	Sink 2.2	High Feed Ash	High Feed Ash	High Feed Ash	High Feed Ash	High Feed Ash	High Feed Ash
	9.5 x 1	Flt 2.2	20.9/50.7	18.2/53.1	15.0/51.4	44.3/88.3	38.3/78.2	26.2/57.6
	9.5 x 1	Sink 2.2	10.6/2.0	15.7/2.8	11.8/2.9	75.9/20.3	76.7/19.0	78.7/12.7
	1 x 0.15	--	19.2/6.7	13.8/9.2	11.2/9.5	62.0/41.7	61.4/39.4	59.2/32.5
LAM	50 x 12.5	1.4 x 1.7	20.9/75.3	16.5/77.2	7.2/66.3	22.1/81.0	27.7/92.2	22.6/77.1

The detailed results for the samples that were tested for decarbonization (Samples 3-6 in the test matrix) are shown in in the following tables.

Table 3-6: Pocahontas No. 3 Coal Seam Preparation Plant LAM Decarbonization Results

Grind Size	Feed Ash (%)	Ash (%)	Moist. (%)	Rejects Ash (%)	Mass Yield (%)	Recovery (%)	
						Comb.	Org.
HHS Product							
-80 M	31.4	20.9	1.1	58.2	75.3	85.2	86.4
-325 M	32.2	16.5	1.1	85.5	77.2	95.1	97.3
		19.5	4.0	85.7	80.8	95.9	97.8
-10 µm	30.1	7.2	2.7	79.0	66.3	89.7	92.5
		8.3	3.4	78.6	67.1	89.7	92.5
		8.2	3.8	79.6	67.5	90.3	93.1
Flotation Product							
-80 M	31.4	22.1	--	71.1	81.0	92.0	93.3
-325 M	32.2	27.7	--	85.1	92.2	98.3	99.0
-10 µm	30.1	22.6	--	55.4	77.1	85.4	86.4

Table 3-7: Pocahontas No. 3 Coal Seam Preparation Plant HAM 9.5 x 1 mm Float Sample Decarbonization Results

Grind Size	Feed Ash (%)	Ash (%)	Moist. (%)	Rejects Ash (%)	Mass Yield (%)	Recovery (%)	
						Comb.	Org.
HHS Product							
-80 M	48.9	30.7	1.2	77.2	60.9	82.5	86.2
		20.9	1.3	77.7	50.7	78.5	83.2
-325 M	48.8	18.2	2.3	83.5	53.1	84.9	90.3
		22.9	1.9	83.7	57.4	86.4	91.3
-10 µm	48.3	15.0	2.4	84.4	52.0	85.5	91.1
		16.0	5.6	84.7	52.3	85.0	90.4
Flotation Product							
-80 M	48.9	44.3	--	83.7	88.3	96.3	97.6
-325 M	48.8	38.3	--	86.5	78.2	94.3	97.0
-10 µm	48.3	26.3	--	78.2	57.6	82.1	86.2

Table 3-8: Pocahontas No. 3 Coal Seam Preparation Plant HAM 9.5 x 1 mm Sink Sample Decarbonization Results

Grind Size	Feed Ash (%)	Ash (%)	Moist. (%)	Rejects Ash (%)	Mass Yield (%)	Recovery (%)	
						Comb.	Org.
HHS Product							
-80 M	88.5	10.6	4.0	85.1	2.0	15.7	40.3
-325 M	89.5	15.7	2.8	91.6	2.8	22.2	68.8
		17.7	4.3	91.8	3.1	24.3	75.2
-10 µm	89.7	11.8	3.7	92.0	2.9	24.6	80.1
		12.4	4.1	91.5	2.3	19.4	63.1
Flotation Product							
-80 M	88.5	75.9	--	91.7	20.3	42.4	82.6
-325 M	89.5	76.7	--	92.5	19.0	42.1	97.6
-10 µm	89.7	78.7	--	91.3	12.7	26.3	70.0

Table 3-9: Pocahontas No. 3 Coal Seam Preparation Plant HAM 1.0 x 0.15 mm Sample Decarbonization Results

Grind Size	Feed Ash (%)	Ash (%)	Moist. (%)	Rejects Ash (%)	Mass Yield (%)	Recovery (%)	
						Comb.	Org.
HHS Product							
-80 M	79.6	20.1	1.4	87.6	11.8	46.4	66.1
		19.2	1.6	83.9	6.7	26.3	37.5
-325 M	79.7	18.3	1.7	88.7	12.8	51.5	73.7
		13.8	1.2	86.4	9.2	39.2	56.4
-10 µm	81.0	11.2	3.9	88.3	9.5	44.3	66.5
		11.4	6.8	88.5	9.7	45.4	68.1
Flotation Product							
-80 M	79.6	62.0	--	92.2	41.7	77.7	98.2
-325 M	79.7	61.4	--	91.6	39.4	74.9	95.3
-10 µm	81.0	59.2	--	91.5	32.5	69.8	93.6

In general, the best results are seen at the highest degree of liberation, i.e., 10 micron top size. For the LAM sample, this size fraction resulted in product ash values around 7 or 8 percent with yields between 66 and 67 percent. Combustible and organic recovery for these tests ranged from 85 to 98 percent and 86 to 99 percent, respectively. Two of the HAM samples also responded relatively well to HHS in the finest size fraction tested.

The 9.5 x 1 mm sink material recorded product ash values of 11.8 and 12.4 percent, with 80.1 and 63.1 percent organic recovery. However, yield and combustible recovery were extremely low for this sample due to the high feed ash of the material resulting in a low amount of recoverable coal.

The 1 x 0.15 mm sample also achieved better results at the 10 micron top size. Product ash values were 11.2 percent and 11.4 percent, with 66.5 and 68.1 percent organic recovery. Yield and combustible recovery were again relatively low at 9.5-9.7 percent and 44.3- 45.4 percent respectively.

The 80-mesh and 325-mesh fractions did not respond well with the lowest product ash reported being 10.6 percent with only 2 percent yield. This lower product ash is likely due to the recovery of only ultrafine coal during the process, while the coarser particles of higher ash sink and report to the tailings.

Interestingly, the 9.5 x 1 mm float material that was tested, at best, only produced a 15 percent ash product, but with significantly higher yield and recovery, when compared to the other HAM samples.

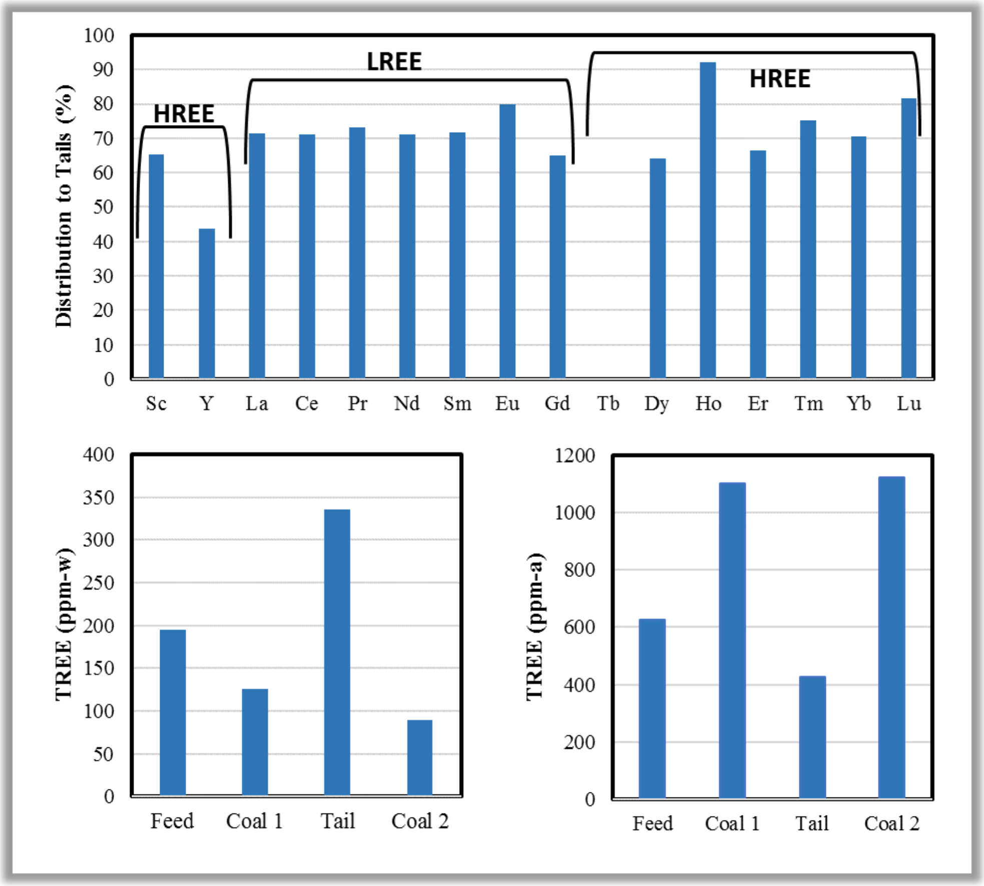
Flotation test results were poor across all size fractions for the samples tested. No product ash values below 20 percent were achieved.

From this data, it would support that HHS is the best method for decarbonizing the feed prior to REE recovery. While the samples did not respond well until after micronizing, they may respond positively with regards to REE HHS testing.

3.1.3.7 Pocahontas No. 3 Coal Seam Preparation Plant Circuits Low Ash Middlings HHS Coal and Tailings REE Results

The REE results obtained for the tests performed on the LAM 10 micron sample (8 percent ash) are provided in the following figure. The results include the TREE concentration values for each stream calculated as ppm on a whole-sample basis (ppmw) and on an ash basis (ppma), as well as, the distribution to tails.

Figure 3-17: Pocahontas No. 3 Coal Seam Preparation Plant Low Ash Middlings (LAM) HHS Coal TREE Analysis[4]



---

[4] Delineation of Light and Heavy REEs in accordance to USGS.

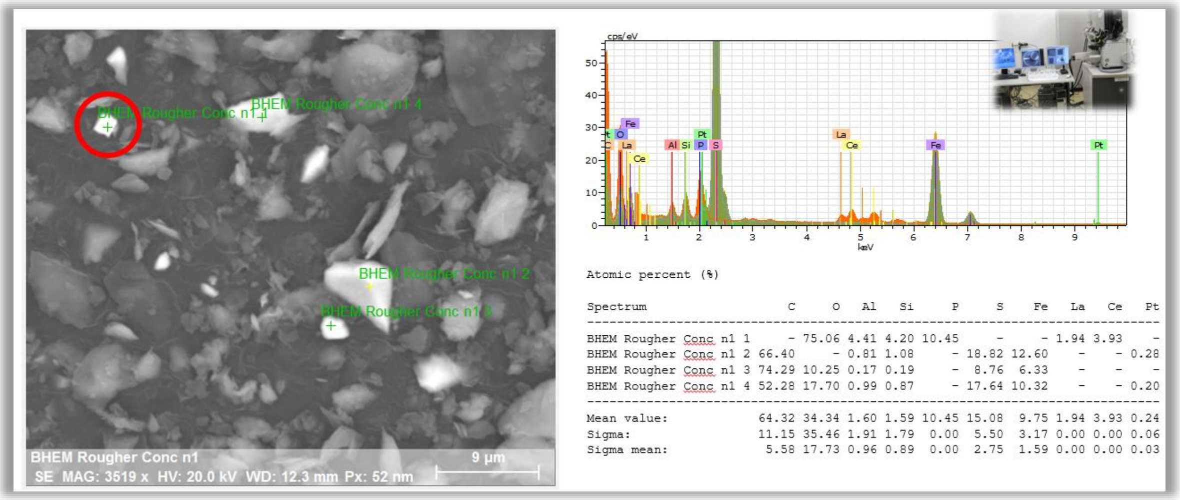
The LAM sample seems to have some peaks associated with Ho, Lu, and Eu (Holmium, Lutetium, and Europium) with each reaching around 80 percent. La, Ce, Pr, Nd, Sm, and Tm (Lanthanum, Cerium, Praseodymium, Neodymium, Samarium, and Thulium) all reported around 70 percent.

The tailings stream of the LAM sample resulted in a TREE (whole sample) of 335 ppm with a contained value of $75/ton and a TREE ash of 426 ppm with a contained value of $95/ton. The coal product, in contrast, resulted in a TREE whole of 89.8 ppm with a contained value of $26/ton, and a TREE ash of 1,123 ppm with a contained value of $336/ton.

3.1.3.8 Pocahontas No. 3 Coal Seam Preparation Plant Circuit High Ash Middlings HHS Coal and Tailings REE Results

An HHS test was conducted on the HAM sample using sodium oleate (C18) as the collector to recover RE minerals. The test used HHS as the decarbonization method for the 1 x 0.15 mm sample and the tailings were further ground to a 10 micron top size. From the test, four different streams were analyzed (coal, rougher concentrate, scavenger concentrate, and scavenger tails). Samples from the four resultant streams were analyzed by SEM-EDX. [5] The results of the SEM-EDX analysis of the HAM sample are shown in the following figures.

Figure 3-18: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Rougher Concentrate (A)

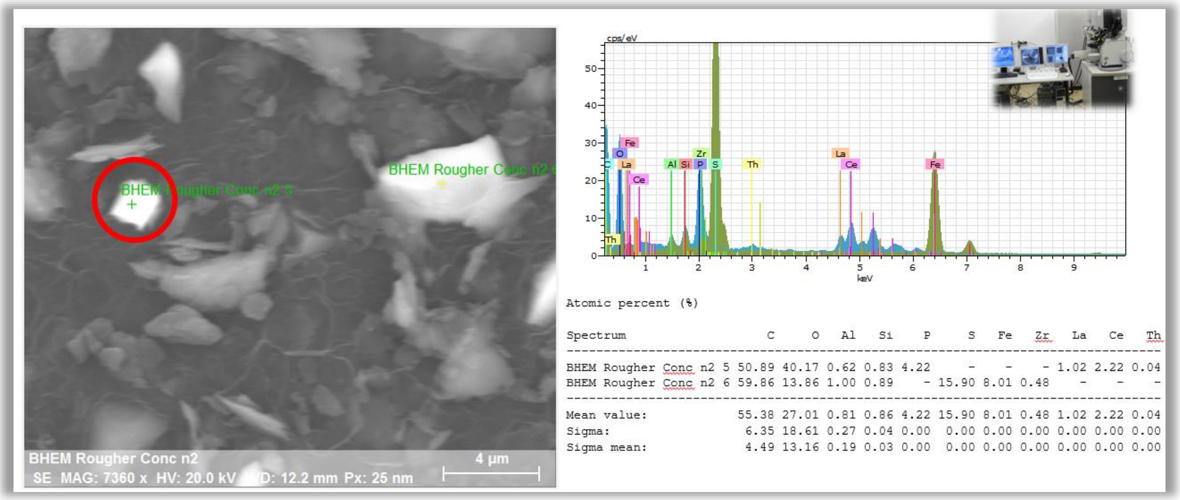


```
Atomic percent (%)

Spectrum                     C     O   Al   Si     P     S    Fe   La   Ce   Pt
---------------------------------------------------------------------------------
BHEM Rougher Conc n1 1       - 75.06 4.41 4.20 10.45     -     - 1.94 3.93    -
BHEM Rougher Conc n1 2 66.40     - 0.81 1.08     - 18.82 12.60    -    - 0.28
BHEM Rougher Conc n1 3 74.29 10.25 0.17 0.19     -  8.76  6.33    -    -    -
BHEM Rougher Conc n1 4 52.28 17.70 0.99 0.87     - 17.64 10.32    -    - 0.20
---------------------------------------------------------------------------------
Mean value:            64.32 34.34 1.60 1.59 10.45 15.08  9.75 1.94 3.93 0.24
Sigma:                 11.15 35.46 1.91 1.79  0.00  5.50  3.17 0.00 0.00 0.06
Sigma mean:             5.58 17.73 0.96 0.89  0.00  2.75  1.59 0.00 0.00 0.03
```

---

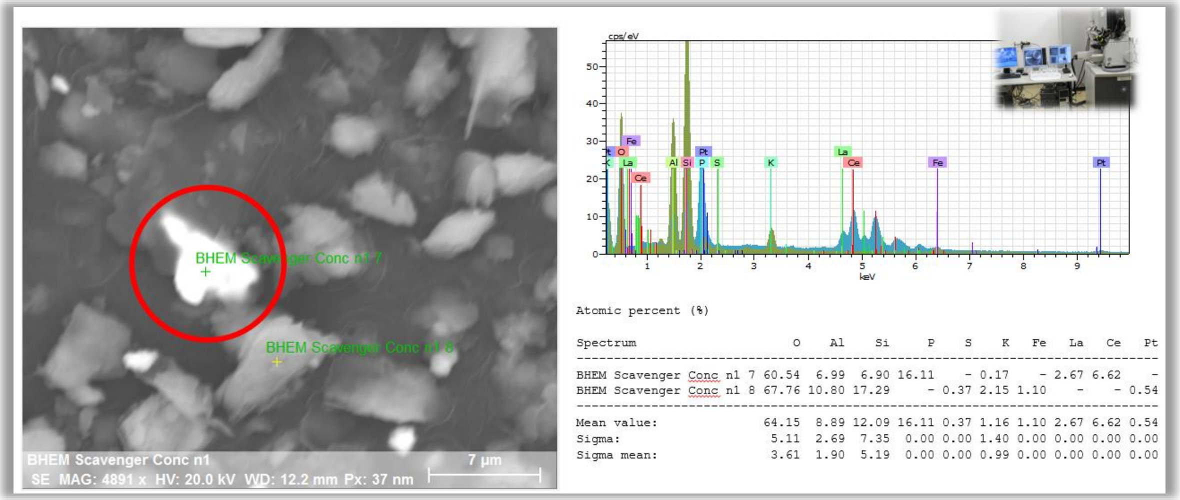
[5] SEM provides detailed high resolution images of the sample by rastering a focussed electron beam across the surface and detecting secondary or backscattered electron signal. An Energy Dispersive X-Ray Analyzer (EDX or EDA) is also used to provide elemental identification and quantitative compositional information; *https://www.lucideon.com › testing-characterization › techniques › sem-edx*

Figure 3-19: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Rougher Concentrate (B)



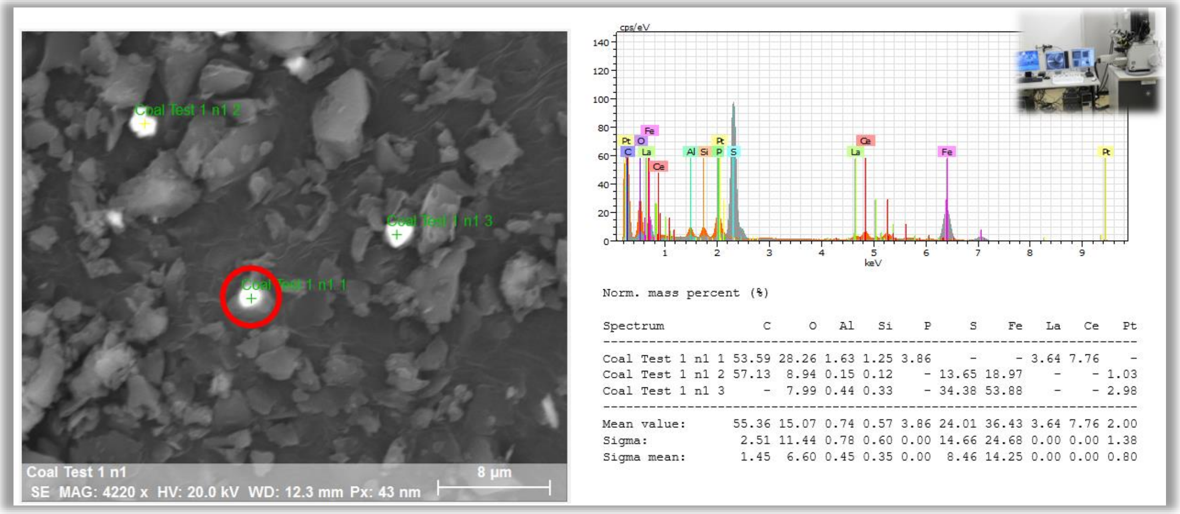
```
Atomic percent (%)

Spectrum                     C     O   Al   Si    P     S   Fe   Zr   La   Ce   Th
-----------------------------------------------------------------------------------
BHEM Rougher Conc n2 5 50.89 40.17 0.62 0.83 4.22     -    -    - 1.02 2.22 0.04
BHEM Rougher Conc n2 6 59.86 13.86 1.00 0.89    - 15.90 8.01 0.48    -    -    -
-----------------------------------------------------------------------------------
Mean value:            55.38 27.01 0.81 0.86 4.22 15.90 8.01 0.48 1.02 2.22 0.04
Sigma:                  6.35 18.61 0.27 0.04 0.00  0.00 0.00 0.00 0.00 0.00 0.00
Sigma mean:             4.49 13.16 0.19 0.03 0.00  0.00 0.00 0.00 0.00 0.00 0.00
```

Figure 3-20: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Scavenger Concentrate



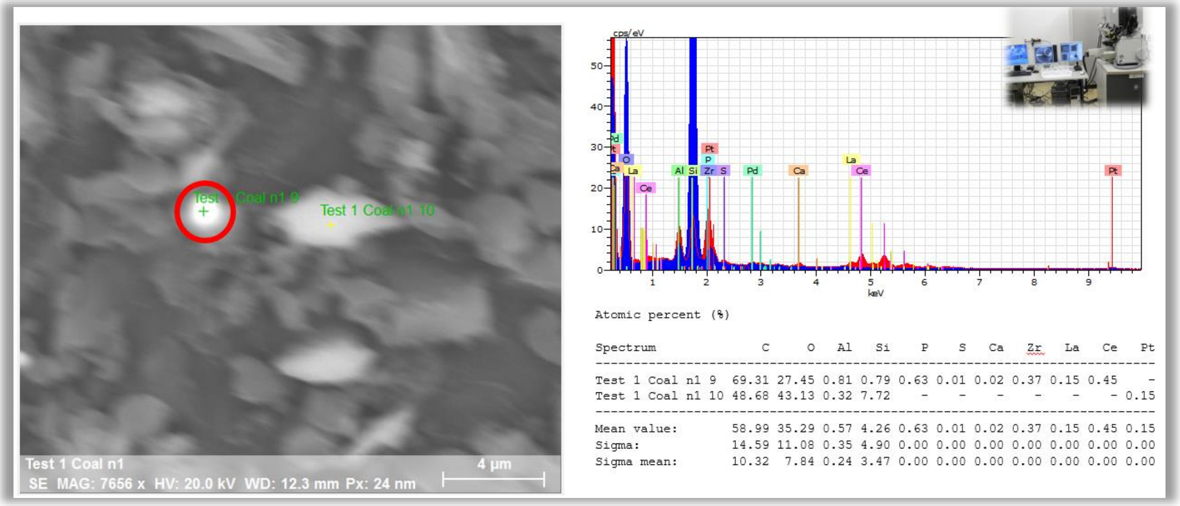
```
Atomic percent (%)

Spectrum                     O    Al    Si     P    S    K   Fe   La   Ce   Pt
-------------------------------------------------------------------------------
BHEM Scavenger Conc n1 7 60.54  6.99  6.90 16.11    - 0.17    - 2.67 6.62    -
BHEM Scavenger Conc n1 8 67.76 10.80 17.29     - 0.37 2.15 1.10    -    - 0.54
-------------------------------------------------------------------------------
Mean value:              64.15  8.89 12.09 16.11 0.37 1.16 1.10 2.67 6.62 0.54
Sigma:                    5.11  2.69  7.35  0.00 0.00 1.40 0.00 0.00 0.00 0.00
Sigma mean:               3.61  1.90  5.19  0.00 0.00 0.99 0.00 0.00 0.00 0.00
```

Figure 3-21: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Coal Product (A)



```
Norm. mass percent (%)

Spectrum              C     O   Al   Si    P     S    Fe   La   Ce   Pt
--------------------------------------------------------------------------
Coal Test 1 n1 1 53.59 28.26 1.63 1.25 3.86     -     - 3.64 7.76    -
Coal Test 1 n1 2 57.13  8.94 0.15 0.12    - 13.65 18.97    -    - 1.03
Coal Test 1 n1 3     -  7.99 0.44 0.33    - 34.38 53.88    -    - 2.98
--------------------------------------------------------------------------
Mean value:      55.36 15.07 0.74 0.57 3.86 24.01 36.43 3.64 7.76 2.00
Sigma:            2.51 11.44 0.78 0.60 0.00 14.66 24.68 0.00 0.00 1.38
Sigma mean:       1.45  6.60 0.45 0.35 0.00  8.46 14.25 0.00 0.00 0.80
```

Figure 3-22: Pocahontas No. 3 Coal Seam Preparation Plant HAM, SEM-EDX Analysis Coal Product (B)



```
Atomic percent (%)

Spectrum               C     O   Al   Si    P    S   Ca   Zr   La   Ce   Pt
------------------------------------------------------------------------------
Test 1 Coal n1 9   69.31 27.45 0.81 0.79 0.63 0.01 0.02 0.37 0.15 0.45    -
Test 1 Coal n1 10  48.68 43.13 0.32 7.72    -    -    -    -    -    - 0.15
------------------------------------------------------------------------------
Mean value:        58.99 35.29 0.57 4.26 0.63 0.01 0.02 0.37 0.15 0.45 0.15
Sigma:             14.59 11.08 0.35 4.90 0.00 0.00 0.00 0.00 0.00 0.00 0.00
Sigma mean:        10.32  7.84 0.24 3.47 0.00 0.00 0.00 0.00 0.00 0.00 0.00
```

All visible REE grains had a phosphate peak. Overall, the sample contains a lot of pyrite and very few distinct REE particles are visible. In summary, the following results were obtained after 6.5 hours of scanning:

- For the scavenger tails, 0 REE particles, 53 pyrite particles, 8 Zr particles and a few Fe particles were found.
- For the scavenger concentrate, 1 REE particle, 60 pyrite particles, 19 Zr particles and 1 CuFeSx was found.
- For the Rougher Concentrate, 2 REE particles, 24 pyrite particles, 3 Zr particles and one each of Fe, ZnS, and CuFeSx were found.
- Finally, for the coal sample, 1 REE particle was found, along with 77 pyrite particles, 3 iron particles, and one CuFeSx.

In terms of REEs of interest:

- A REE particle in the rougher concentrate contained a mean value of 1.94 percent Lanthanum, 3.93 percent Cerium, and 0.04 percent Thorium.
- A particle in the scavenger concentrate contained a mean value of 6.62 percent Cerium and 2.67 percent Lanthanum.
- A REE particle in the coal product contained a mean value of 7.76 percent Cerium and 3.64 percent Lanthanum.

3.1.3.9 Pocahontas No. 3 Coal Seam Rejects REE Concentration by Optical Sorting

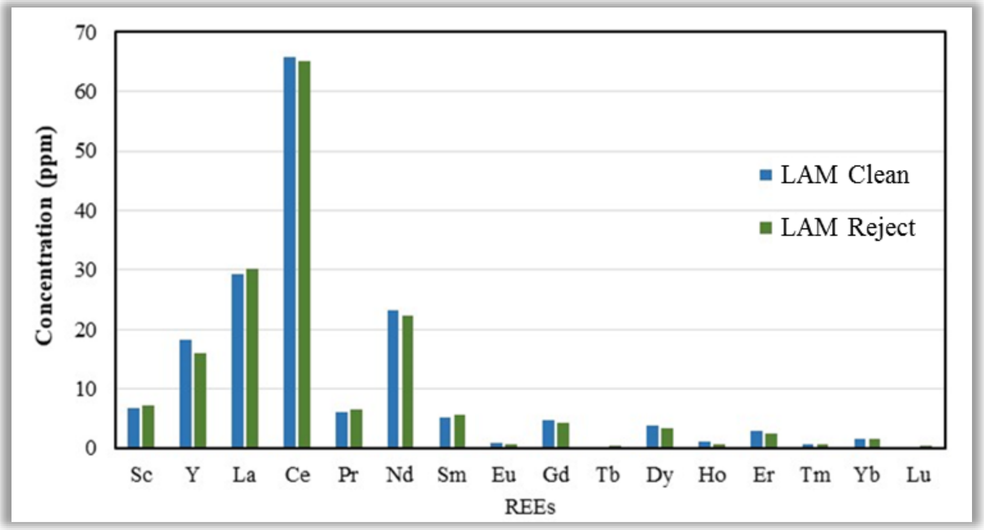
3.1.3.9.1 Dual X-Ray Transmission Sorting (DXRT)

The DriJet™ X-Ray sorter made by Mineral Separation Technologies was used to sort the LAM sample into a clean and a tailing product. The DriJet™ technology utilizes an x-ray source and an x-ray detector to measure the atomic weights of individual particles in order to separate them into two fractions using small precision air jets. The test results given in the following table showed that the sorter was able to produce a marginally cleaner product with 25.3 percent ash content, whereas the tailing contained 29.3 percent ash. However, the Total REE analysis of the products showed that the sorter was not able to make a significant separation of REEs to either of the products. As can be seen in the table and its associated figure, REEs were split almost equally into the clean and tailings products. This finding is not consistent with data obtained from testing of other coal samples and suggests that the mineral matter contained within this feedstock is finely disseminated and not separable. This observation is consistent with the HHS separation test data that showed good separations of organic and inorganic matter only occurred when the feed was micronized to below 10 microns.

Table 3-10: Pocahontas No. 3 Coal Seam Preparation Plant DriJet™ X-Ray Sorter Test Results

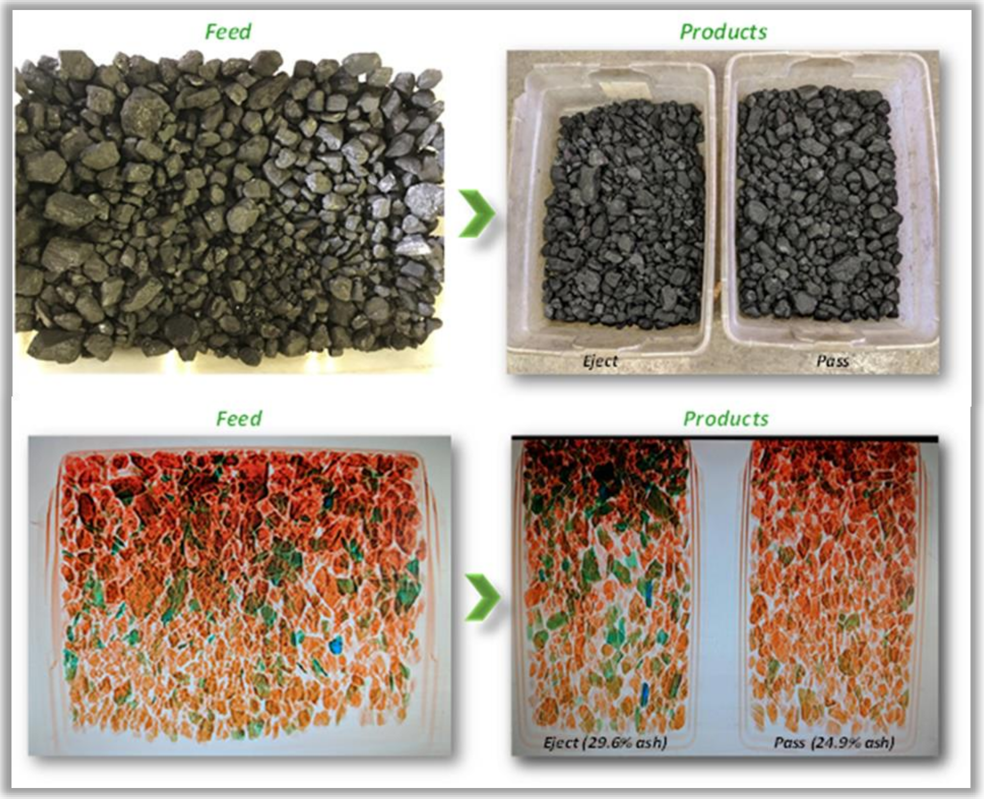
Sample	Yield (%)	Ash (%)	TREE (Whole Basis) (ppm)	TREE (Ash Basis) (ppm)
Clean	57.16	25.31	170.80	674.84
Reject	42.84	29.30	167.81	572.73

Figure 3-23: Pocahontas No. 3 Coal Seam Preparation Plant Individual REE Concentrations in DriJet™ X-Ray Sorter Products



The feed and products from the sorter tests were also scanned in an X-Ray scanner similar to the ones used at airports for scanning luggage. The resultant images are presented below. Orange-red color in these pictures represent the organic matter. The green color represents shale type minerals and the blue represents denser minerals, such as pyrite. These pictures indicate that most of the pure ash material went to the tailing product. However, since LAM sample is a middling product that is not liberated well, the ash contents of the two products were not very different.

Figure 3-24: Pocahontas No. 3 Coal Seam Preparation Plant
X-Ray scans of LAM DriJet™ Sorter Testing



To achieve a better understanding of what the different colors represent in X-Ray scans, a portion of the LAM samples were scanned and orange, blue and green particles were handpicked for further analysis. The results (presented in the following figure and table) show that the orange particles had the richest Total REE content of all with a grade of 901 ppm ash basis.

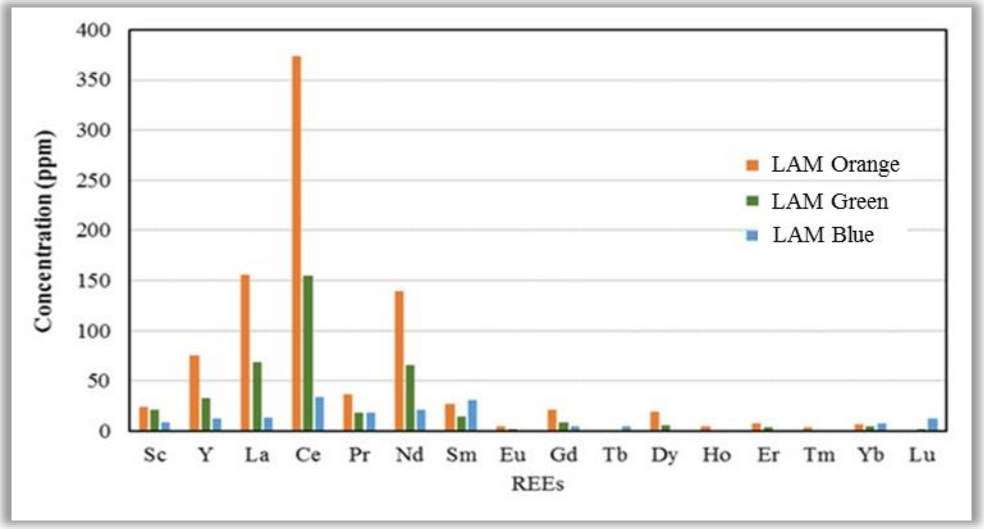
Figure 3-25: Pocahontas No. 3 Coal Seam Preparation Plant
Handpicked Particles from LAM Sample

Table 3-11: Pocahontas No. 3 Coal Seam Preparation Plant Analysis of LAM Particles with Different Colors from X-Ray Scans

Sample	Ash (%)	TREE (Whole Basis) (ppm)	TREE (Ash Basis) (ppm)
Orange	28.25	254.58	901.16
Green	58.16	236.35	406.38
Blue	37.34	63.06	168.87

The detailed analysis results given in the following figure indicates that the Cerium had the highest grade of any REEs with 374 ppm in orange particles, followed by Lanthanum, Neodymium and Yttrium with 156 ppm, 139 ppm and 76 ppm, respectively. Concentrations of these four elements were also higher than the rest of the REEs in green particles, but not as high as in orange. REE concentrations in blue particles were considerably lower with a total REE concentration of only 168.87 ppm ash basis.

Figure 3-26: Pocahontas No. 3 Coal Seam Preparation Plant Individual Analysis of LAM Particles with Different Colors from X-Ray Scans



3.1.4 Ferroglobe Preparation Plant Circuit Sampling Program

3.1.4.1 Ferroglobe High Ash (Jellico) and Low Ash (Blue Gem) Preparation Plant Circuit Samples

Ten (10) samples were collected from a coal processing facility that treats high-quality coal feedstocks from the Blue Gem (low ash content) and Jellico (mid-ash content) coal seams in eastern Kentucky[6]. The test work was largely focused on determining whether these particular feedstocks could be used to provide a high purity, high value coal product from an HHS decarbonization step, which would improve the overall economic feasibility of an REE processing facility.

---

[6] Gatliff Tipple, MSHA ID 15-09938, operated by Alden Resources LLC a subsidiary of Ferroglobe., Ibid.

Two (2) of the samples consisted of spiral concentrates. One from a low ash (Blue Gem Seam) production run and the other from a mid-ash (Jellico Seam) production run.

The remaining eight (8) samples consisted of flotation feed, flotation concentrate, flotation tails, and screenbowl effluent samples, from both a low ash (1 percent ash clean coal) and a mid-ash 3 percent ash clean coal) production run.

Testing was completed on each as-received sample before and after grinding to determine the efficiency and degree of upgrading of the HHS process in producing saleable high-purity clean coal product. Sizing data as well as ash and recovery results were produced from each of these tests.

3.1.4.1.1 Ferroglobe High Ash and Low Ash Sample Flotation Feed Testing and Screenbowl Effluent

No sample preparation was required for the flotation feed, flotation concentrate, flotation tails, and screenbowl effluent tests of the high ash (3 percent) and low ash (1 percent) product process steam circuits. In other words, "as-received" samples were used. Hence, in general, the best results were found for the flotation product samples with very low ash, high yield, and high recovery. For convenience, the best results obtained with each test are summarized in the test matrix provided in the following table.

Table 3-12: Low (Blue Gem) and High Ash (Jellico)
Test Matrix of Ferroglobe Flotation Circuit and Screenbowl Sampling

Sample	Run	% Passing 100-Mesh	% Solids (weight)	HHS Product Ash (%)/Yield (%)
Flotation Feed	3%	97	3.4	2.1 / 35.3
	1%	97	2.1	1.3 / 50.2
Flotation Product	3%	99	9.1	1.9 / 85.9
	1%	98	17.9	1.2 / 83.3
Flotation Tailings	3%	97	2.9	2.2 / 35.6
	1%	95	2.4	1.4 / 49.1
Screenbowl Effluent	3%	98	0.9	N/A
	1%	99	1.0	N/A

The sizing information for the flotation feed indicated a large percentage of material below 100-mesh (approximately 97 percent) is provided in the following table. As such, the material was ideal for the HHS process. The sample also contained a relatively high feed ash of approximately 57 percent.

Table 3-13: Particle Size Analysis of Flotation Feed
for Three (3) Percent Ash (Jellico) Run

Particle Size Fraction (mesh)	Individual (%)		Cumulative (%)	
	Weight	Ash	Weight	Ash
+35	0.2	8.1	0.2	8.1
-35+70	1.4	8.1	1.6	8.1
-70+100	1.0	2.8	2.6	6.0
-100+325	16.3	13.1	18.9	12.1
-325	81.1	67.2	100.0	56.8

The separation results for HHS processing of the flotation feed is provided in the next table for the three (3) percent ash (Jellico) production run. The sample responded very well to the HHS process as indicated by the product ash and moisture content values around two (2) percent as well as recovery values in the 80 percent and 90 percent range for combustible material and organic matter, respectively. However, the clean coal yield was relatively low at only 35 percent due to the relatively high feed ash content[7].

Table 3-14: HHS Test Results on Flotation Feed (As-Received) Material Targeting Three (3) Percent Product Ash (Jellico) Content

Feed (%wt.)		HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
Ash	%Solids	Ash	Moisture			Combustible	Organic
57.7	3.44	2.1	2.1	87.8	35.1	81.2	91.0
		2.1	2.7	88.0	35.3	81.6	91.5

A flotation feed sample was collected during the 1 percent ash production run from the plant. The feed sizing of this sample is summarized in the following table. The particle size analysis showed similar results to the 3 percent ash run, with approximately 97 percent of the material passing a 100-mesh screen. The feed ash of this sample was also lower, as expected, at 42 percent ash.

Table 3-15: Particle Size Analysis of the Flotation Feed Collected during the 1 Percent Ash (Blue Gem) Plant Run

Particle Size Fraction (mesh)	Individual (%)		Cumulative (%)	
	Weight	Ash	Weight	Ash
+35	0.2	5.5	0.2	5.5
-35+70	1.8	1.5	2.0	1.9
-70+100	1.5	1.8	3.5	1.9
-100+325	26.1	14.5	29.6	13.0
-325	70.4	56.3	100.0	43.5

The product from the HHS separation process of the forgoing sample is shown in the following table. That process attained the target ash of less than 1.5 percent and moisture values that were very low in the 2-3 percent range. The sample tested in the HHS process also provided very high recovery values, i.e., 86 percent for combustible recovery and 91 percent organic recovery. When compared to the 3 percent ash production run sample, the clean coal yield was significantly higher with values around 50 percent.

Table 3-16: HHS Separation Performance Achieved on Flotation Feed (As-Received) Collected from the 1 percent Ash Run Results

Feed (%wt.)		HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
Ash	%Solids	Ash	Moisture			Combustible	Organic
42.4	2.1	1.4	2.6	84.0	50.3	86.2	91.5
		1.3	3.4	83.8	50.2	86.0	91.3

---

[7] A small amount of sorbitan monooleate (SMO/span 80) was added to improve the kinetics of the selective agglomeration stage used in the HHS process. The flotation feed sample was the only sample tested where this addition was required. The results were not material and are not reported here. All other samples performed very well as-received.

In addition to the flotation feed, samples of flotation product from the plant were also tested using the HHS process. The particle size analysis data for the flotation product collected during the high-ash (3 percent ash) production run are shown in the following table. This concentrate product contained material that was approximately 99 percent passing 100-mesh which was ideal feed for the HHS process.

Table 3-17: Particle Size Analysis of the Flotation Product Collected during the 3 Percent Ash (Jellico) Plant Run

Particle Size Fraction (mesh)	Individual (%)		Cumulative (%)	
	Weight	Ash	Weight	Ash
+35	0.2	24.1	0.2	24.1
-35+70	0.1	24.1	0.3	24.1
-70+100	0.6	24.1	0.9	24.1
-100+325	12.0	2.0	12.9	3.5
-325	87.1	13.1	100	11.9

The performance data from the HHS experiments on the high ash (Jellico) process stream are summarized in the next table. As shown, the flotation product sample also responded very well to the HHS process. Feed ash for this sample was very low at 13 percent and was reduced to around 2 percent ash in the HHS product, which met the target ash of less than 3 percent ash. Yield and recoveries were also very good, i.e., 86 percent and 98 percent, respectively.

Table 3-18: HHS Performance when Treating Flotation Product (As-Received) Collected during a One (1) Percent Ash Plant Run

Feed (%wt.)		HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
Ash	%Solids	Ash	Moisture			Combustible	Organic
13.4	9.1	1.9	4.5	83.1	85.9	97.2	98.3
		2.1	2.9	82.7	86.0	97.2	98.2
		2.2	7.3	83.0	86.1	97.3	98.3

A sample of flotation product from a low ash (1 percent ash, Blue Gem) production run at the plant site was also tested using the HHS process. The size analysis data for this feedstock is summarized in the following table. Approximately 30.5 percent of the material passed 100-mesh.

Table 3-19: Size Analysis of Flotation Product for One (1) Percent Ash (Blue Gem) Run

Particle Size Fraction (mesh)	Individual (%)		Cumulative (%)	
	Weight	Ash	Weight	Ash
+35	0.1	6.9	0.1	6.9
-35+70	1.0	1.6	1.1	2.1
-70+100	1.2	1.4	2.3	1.7
-100+325	28.2	2.3	30.5	2.2
-325	69.5	20.0	100.0	14.6

The detailed results of the HHS performance on this sample are shown in *Table 3-20*. Again, the sample responded very well to selective agglomeration and recovery from the vibrating mixer it was encouraging. Ash reduction was very good for this sample, reducing from a feed ash of 14.5 percent to product ash values of 1.2 percent and 1.5 percent, which met the target of less than 1.5 percent ash. The yield was again very high, ranging from 83.3 to 84.6 percent, and recoveries ranged from 95.1 to 96.2 percent and 96.2 to 97.3 percent for combustibles and organics, respectively.

Table 3-20: HHS Performance when Treating Flotation Product (As-Received) Collected during a One (1) percent Ash (Blue Gem) Plant Run

Feed (%wt.)		HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
Ash	%Solids	Ash	Moisture			Combustible	Organic
14.5	17.9	1.2	1.9	74.4	83.3	95.1	96.2
		1.2	5.5	73.6	83.2	94.9	96.0
		1.5	3.6	78.7	84.6	96.2	97.3

The flotation tailings samples appeared to very closely resemble the flotation feed samples received. On a feed ash basis, they were nearly identical, and on a sizing basis, there was really no difference with the exception of a minor difference in ash contents of the coarsest material.

Table 3-21: Particle Size Analysis of the Flotation Tailings Sample Collected during the Three (3) Percent Ash (Jellico Seam) Plant Run

Particle Size Fraction (mesh)	Individual (%)		Cumulative (%)	
	Weight	Ash	Weight	Ash
+35	0.2	2.2	0.2	2.2
-35+70	0.7	2.2	0.9	2.2
-70+100	2.1	2.2	3.0	2.2
-100+325	15.8	6.3	18.8	5.6
-325	81.2	68.6	100.0	56.8

For this particular feedstock, the HHS results were again very encouraging, even without the use of SMO surfactant (sorbitan monooleate). The results for the 3 percent ash run are shown in the following table. The HHS process was able to successfully reduce the ash content from 56.9 percent to 2.2 percent, which met the target ash of less than 3 percent. The process recovered more than 80 percent of the combustible material in the feed and greater than 90 percent of the total organics. However, mass yield to the clean coal stream was relatively low, ranging only from 35.4 to 35.6 percent.

Table 3-22: HHS Performance when Treating Flotation Product (As-Received)

Collected during a Three (3) Percent Ash (Jellico Seam) Plant Run

Feed (%wt.)		HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
Ash	%Solids	Ash	Moisture			Combustible	Organic
56.9	2.92	2.2	2.5	87.0	35.4	80.5	89.8
		2.2	2.9	87.2	35.6	80.9	90.2

The performance achieved on the flotation tailings material collected during the 1 percent plant ash content run was similar to the 3 percent run. Again, there was a very close resemblance to the flotation feed sample, although the kinetics for the selective agglomeration step in the HHS process was somewhat faster. The particle size analysis for this feedstock is summarized in the following table. Size analysis was similar with more than 95 percent of the material falling below 100-mesh.

Table 3-23: Particle Size Analysis of the Flotation Tailings Sample Collected during the One (1) Percent Ash Plant Run

Particle Size Fraction (mesh)	Individual (%)		Cumulative (%)	
	Weight	Ash	Weight	Ash
+35	0.36	1.8	0.36	1.8
-35+70	2.38	1.8	2.74	1.8
-70+100	2.77	1.3	5.5	1.6
-100+325	24.8	6.4	30.3	5.5
-325	69.7	57.7	100.0	41.9

The HHS separation results of the forgoing sample are summarized in the following table. As shown, the test data obtained with this sample also reached a target product ash of less than 1.5 percent (i.e., product ash of 1.4 percent). Recoveries were also high at 84.9 percent for combustibles and 90.3 percent for organic matter. The clean coal yield was about 50 percent. This yield is better than previous tests with similar feed characteristics, but still relatively low compared to previous HHS results obtained with other sources of lower ash coal feedstocks.

Table 3-24: HHS Performance when Treating Flotation Product (As-Received) Collected during a One (1) Percent Ash Plant Run

Feed (%wt.)		HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
Ash	%Solids	Ash	Moisture			Combustible	Organic
43.0	2.4	1.5	1.8	83.5	49.4	85.3	90.7
		1.4	1.6	83.1	49.1	84.9	90.3
		1.6	1.2	84.4	50.0	86.3	91.7

Of the various flotation samples tested in the current project, the flotation product stream responded the best to the HHS process. However, the HHS process was able to attain the target ash content values for all sample sources while achieving high levels of combustible recovery. As such, the proposed HHS process is capable of producing low ash, low moisture saleable products from these particular feedstocks.

3.1.4.1.2 Ferroglobe Screenbowl Centrifuge Effluent Sample Testing

Screenbowl centrifuge effluents were also collected from the plant site. However, the samples from the low- and high-ash production runs were both very low, i.e., less than 1 percent solids. Due to the low solids content of the sample, there was insufficient material available for HHS testing and only sizing data was recorded. The size analyses for the two samples are shown in the following two tables. Both samples appear to be appropriate for HHS testing on a size basis with most of the material falling below 325-mesh, which is typical for screenbowl effluent samples.

Table 3-25: Particle Size Analysis of the Screenbowl Effluent Sample Collected during the Three (3) Percent Ash Plant Run

Particle Size Fraction (mesh)	Individual (%)		Cumulative (%)	
	Weight	Ash	Weight	Ash
+35	1.0	74.2	1.0	74.2
-35+70	0.3	8.3	1.3	59.0
-70+100	0.6	8.3	1.9	43.0
-100+325	1.8	8.3	3.7	26.1
-325	96.3	51.6	100.0	50.7

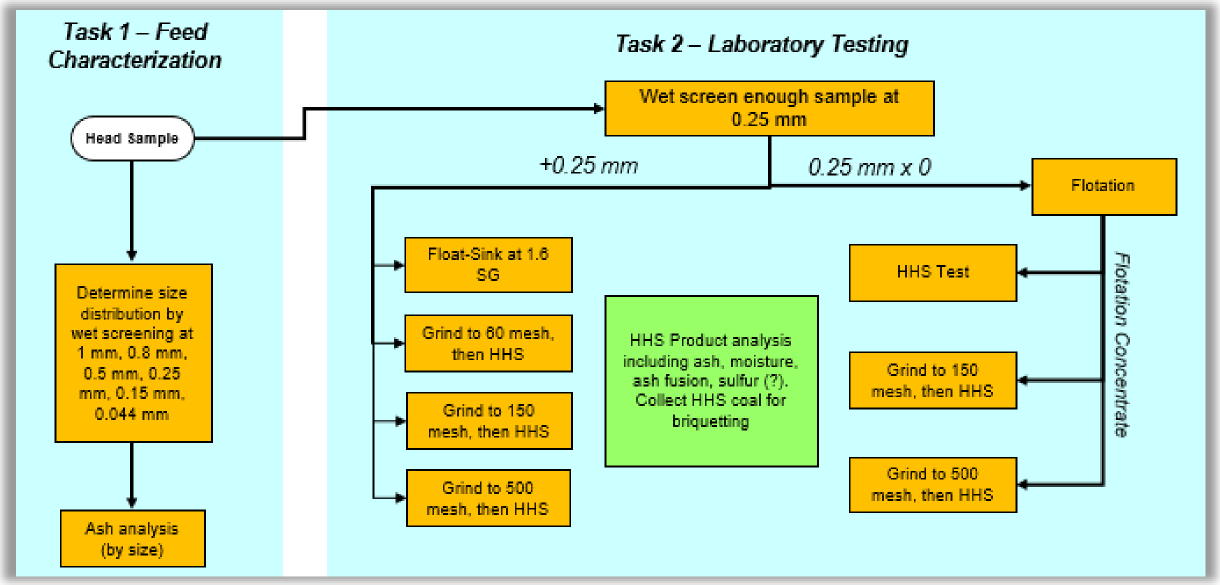
Table 3-26: Particle Size Analysis of the Screenbowl Effluent Sample Collected during the One (1) Percent Ash Plant Run

Particle Size Fraction (mesh)	Individual (%)		Cumulative (%)	
	Weight	Ash	Weight	Ash
+35	0	0	0	0
-35+70	0.21	13.0	0.81	13.0
-70+100	0.34	13.0	1.15	13.0
-100+325	0.25	13.0	1.8	13.0
-325	99.2	39.3	100.0	39.1

3.1.4.1.3 Ferroglobe Spiral Stream Testing

Another sample collected from the Ferroglobe processing plant while that plant was processing a Blue Gem Seam coal feedstock was from the clean coal sieve overflow in the spiral circuit. This material was coarser by definition and not wholly suitable for HHS testing as an "as-received" sample. Therefore, sample preparation was required for this feedstock. A flowchart was developed to define a proper testing procedure which is shown in the following figure. For laboratory testing, the samples were screened at 0.25 mm. The undersize was then pre-concentrated using conventional froth flotation and subsequently processed through the HHS process after grinding to finer than 150-mesh (100 microns) for one test series and grinding to pass 500-mesh (25 microns) for a second test series. The oversize was split to save a portion for float-sink analysis and then three splits were ground to obtain material with three different top sizes (i.e., 60-mesh, 150-mesh, and 500-mesh). For all laboratory testing, a product ash content less than 1.5 percent was targeted. Wet grinding was performed using a laboratory ball mill charged with approximately 300 grams of sample. For grinding media, a mix of iron balls were used ranging from 6 to 26 mm in diameter with a relatively even distribution.

Figure 3-27: Testing Procedure for Cleaned Coal Spiral Concentrate Samples



Samples collected during the low-ash content production run when processing Blue Gem Seam coal was evaluated first. The detailed results from these tests including particle size analysis are shown in *Table 3-27* through *Table 3-33*. A large portion of the as-received material (55 percent) existed in the -0.84+0.25 mm size fraction and contained a relatively low amount of ash (3.1 percent). The greater challenges with this sample was the fine fractions (i.e., -0.15 + 0.045 mm and -0.045 mm) due to elevated ash contents of 26.8 percent and 63.2 percent, respectively. This is typically the size fraction targeted for the HHS process, which is why there was more difficulty in producing the target ash of less than 1.5 percent for the undersize material than the oversize material. Several results were obtained for the minus 0.25 mm pre-concentrated run. The highlighted tests were run using a dispersant during the agglomeration stage. Without dispersant, the tests achieved low ash content but failed to meet the target of lower than 1.5 percent ash. Product ash values were between 1.5 percent and 1.7 percent with high yield and recovery. However, by using a small amount of sodium metaphosphate as a dispersant, it was possible to achieve ash values of 1.1 percent and 1.2 percent. This improvement was likely due to the presence of clay slime coatings on the coal particles, which were removed after adding the dispersant. Combustible recovery value for this series of tests were around 97 percent and the clean coal yields were around 80 percent. Once the undersize material was ground to 80 percent passing 95 microns, dispersant addition was still required to achieve the target ash. Product ash values ranged from 0.9 percent to 1.1 percent after adding the dispersant. The undersize was also ground to 80 percent passing 24 microns and the results were similar. The dispersant helped to achieve target ash values between 1.3 percent and 1.5 percent. These findings all indicate that, for this particular sample, there was no significant advantage to grinding to improve liberation since this action did not equate to lower ash products. On the other hand, the addition of dispersant allowed the production of concentrates with low ash and moisture values while achieving high recovery.

In the oversize material (+0.25 mm), the opposite phenomenon was observed from the separation tests. Grinding for liberation purposes provided very little difference to the product ash results, yield and recovery values. However, the addition of dispersant did not help achieve any lower ash values. This supports the theory that there was a relatively large amount of ultrafine clay material in the undersize fraction, which was not present in the oversize. Interestingly, all tests on the oversize material, even at the coarsest size, met the target ash value. When ground to 80 percent passing 270 microns, ash values were around 1 percent. When ground to 80 percent passing 80 microns, ash values were between 1.0 percent and 1.1 percent. When ground to 80 percent passing 35 microns, ash values were also around 1.0 percent. For all sizes, clean coal yield values were extremely high (around 97 percent) and recovery values were around 99 percent for both combustibles and organic materials. All moisture values were very low and typically less than 5 percent total moisture.

Table 3-27: Particle Size Analysis of Low Ash (Blue Gem) Spiral Concentrate

Particle Size Fraction (mesh)	Individual (%)		Cumulative (%)	
	Weight	Ash	Weight	Ash
+1.18	6.1	7.4	6.1	7.4
-1.18+0.84	9.7	2.9	15.7	4.6
-0.84+0.50	22.5	2.0	38.2	3.1
-0.50+0.25	33.5	2.2	71.8	2.7
-0.25+0.15	15.1	4.2	86.9	2.9
-0.15+0.045	10.5	26.8	97.4	5.5
-0.045	2.6	63.2	100.0	7.0

Table 3-28: HHS Results for the Low-Ash Blue Gem Undersize Material (-0.25 mm)

Head Feed Ash %	Flotation Ash% (Feed to HHS)	HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
		Ash	Moisture			Combustible	Organic
18.1	10.0	1.52	1.9	75.6	77.7	93.5	95.1
		1.48	1.7	73.0	76.6	92.1	93.7
		1.72	2.9	79.4	78.7	94.5	96.0
	9.7[1]	1.23	1.4	82.2	80.7	96.9	98.6
		1.15	3.3	82.4	80.8	97.0	98.7

[1] *One-stage re-cleaning*

Table 3-29: HHS Results for the Low-Ash Blue Gem Undersize Material Ground to D80=95 µm

Head Feed Ash %	Ground Feed Ash% to HHS	HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
		Ash	Moisture			Combustible	Organic
18.1	8.8	1.58	2.5	71.6	77.0	92.5	94.0
		1.52	5.4	71.4	76.9	92.5	94.0
	8.8[1]	0.97	5.3	74.7	77.8	94.1	95.7
		1.13	4.5	75.5	78.0	94.2	95.8
		1.11	3.1	69.2	75.7	91.4	92.9
		1.05	5.0	77.0	78.3	94.6	96.2

[1] *One-stage re-cleaning*

Table 3-30: HHS Results for the Low-Ash Blue Gem Undersize Material Ground to D80=24 µm

Head Feed Ash%	Ground Feed to HHS, ash (%wt.)	HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
		Ash	Moisture			Combustible	Organic
18.1	9.5	1.43	5.5	76.7	78.5	94.5	96.1
		1.40	7.0	76.5	78.5	94.5	96.1
	9.5[1]	1.43	4.5	78.0	78.9	94.9	96.6
		1.55	6.2	78.8	79.1	95.0	96.6
	9.1[2]	1.30	4.3	79.6	78.9	94.6	96.2
		1.34	4.2	79.7	78.9	94.6	96.2

[1] *One-stage re-cleaning;* [2]*Two-stage re-cleaning;*

Table 3-31: HHS Results for the Low-Ash Blue Gem Oversize Material Ground to D80=270 µm

Ground Feed Ash% to HHS	HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
	Ash	Moisture			Combustible	Organic
2.6	1.06	3.2	58.2	97.3	98.8	99.0
	1.06	2.2	59.0	97.4	98.9	99.0
2.6[1]	1.07	2.7	59.5	97.4	98.9	99.0
	1.11	2.2	59.1	97.4	98.9	99.0

[1] *One-stage re-cleaning*

Table 3-32: HHS Results for Low-Ash Blue Gem

Oversize Ground to D80=80 µm

Ground Feed Ash% to HHS	HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
	Ash	Moisture			Combustible	Organic
2.7	1.10	5.4	74.4	98.0	99.4	99.6
	1.07	2.6	74.5	98.0	99.5	99.6
2.7[1]	1.15	3.3	73.7	97.9	99.4	99.5
	1.11	4.3	73.9	97.8	99.4	99.5
	1.14	2.6	74.8	97.9	99.5	99.6

[1] *One-stage re-cleaning*

Table 3-33: HHS Results for Low Ash Blue Gem
Oversize Ground to D80=35 µm

Ground Feed Ash% to HHS	HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
	Ash	Moisture			Combustible	Organic
2.6	1.09	4.2	69.1	97.8	99.3	99.4
	1.04	3.7	69.0	97.7	99.3	99.4

In order to produce extremely low ash product (target of <1 percent ash), the test procedure was modified in the following manner. The as-received sieve overflow sample (Blue Gem Seam) was screened at 0.25 mm. The coarse size fraction (plus 0.25 mm), which previously achieved the best results with HHS, was ground in a laboratory ball mill (*Figure 3-16*) to a D80 size of 0.25 mm. The ground sample was then subjected to an initial "roughing" stage of HHS cleaning involving only selective agglomeration. The resulting clean product was then further ground in a laboratory attrition mill, also shown in *Figure 3-16* to a D80 size of 4 microns. The ground product was then subjected to a second "cleaning" stage of HHS processing that included selective agglomeration, coal dispersion and hydrocarbon stripping.

The results obtained using the new test procedure for the Ferroglobe sieve overflow sample described above are shown in the following table. As shown, extremely low ash (<1 percent) "ultraclean" products were achieved using this procedure. Clean coal ash values of 0.58-0.73 percent were achieved from feeds containing 2.4 percent ash. Reject ash values were also very good with all ash values falling between 75-77 percent ash. Recovery was extremely high (>99 percent) for both combustible matter and organic matter. The process of achieving extremely low ash (<1 percent) products from this particular feedstock may be further simplified by eliminating the initial "roughing" step before attrition mill grinding. With a low feed ash of 2.4 percent after attrition mill grinding, it may be possible to simply screen the plus 0.25 mm oversize material and directly grind in the attrition mill to a d_{80} size of 4 microns.

Table 3-34: Results Obtained from HHS Processing of Ferroglobe's Blue Gem Seam Coal (Plus 0.25 mm Sieve Oversize Ground to a Feed D80 Size of 4 Microns)

Feed Ash (%)	Product Ash (%)	Rejects Ash (%)	Yield (%)	Recovery (%)	
				Combustible	Organic
2.4	0.62	77.5	97.7	99.5	99.6
2.4	0.68	77.1	97.7	99.5	99.6
2.4	0.73	75.7	97.8	99.4	99.6
2.4	0.58	76.1	97.6	99.4	99.6

3.1.4.2 Ferroglobe - Blue Gem Seam Coal Coarse Rejects

Two buckets of coal coarse rejects were collected from a coal preparation plant[8] which treats "Blue Gem" coal. The Blue Gem Seam extends through much of Whitley and Knox counties in eastern Kentucky. The sample was air-dried, then density fractionated on the dry sample using a heavy media bath which was prepared by mixing a certain amount of magnetite with tap water. Three density fractions, i.e., 1.8 SG float, 1.8-2.2 SG, and 2.2 SG sink, were obtained. Fine magnetite attached onto the particle surfaces was removed by wet-screening at 200-mesh. The three samples were finally air-dried and crushed to minus 80-mesh.

Ash contents of the three samples were 38.24 percent, 83.50 percent, and 92.28 percent, respectively. When reported on a whole sample basis, the three samples contained 144 ppm, 255 ppm, and 252 ppm of REEs, respectively as shown in the following table. Heavy-to-light REE content ratios of the three samples were nearly identical (0.23-0.24) indicating that no preferential enrichment of HREEs occurred in the samples. The material finer than 80-mesh was used as feed for calcination and acid leaching tests.

Table 3-35: Blue Gem Seam - TREE, LREE, HREE, and Individual REE Contents on a Whole Sample Basis as well as Heavy to Light REE Content Ratio in the Float-Sink Products

Sample	TREE	LREE	HREE	H/L	Sc	Y	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu
1.8 SG float	144	116	28	0.24	9.76	14.47	22.61	44.63	6.98	25.53	6.39	1.09	4.82	0.85	2.12	0.04	1.37	0.29	2.16	1.04
1.8-2.2 SG	255	207	47	0.23	14.72	22.46	43.44	78.83	12.34	46.65	11.15	1.78	8.49	0.92	4.96	0.85	3.40	0.50	3.07	1.00
2.2 SG sink	252	203	49	0.24	14.84	23.05	44.53	69.93	12.78	49.14	12.07	1.70	8.96	0.95	4.73	0.61	3.66	0.44	3.28	1.27

3.1.4.3 Ferroglobe - Jellico Seam Coal Clean Coal Sieve Overflow Sample Testing

As previously stated, the Ferroglobe processing plant also treats coal from the Jellico coal seam. A representative sample was collected from the clean coal sieve overflow in the spiral circuit and processed using the same procedure described in *Figure 3-27*.

Similar to the Blue Gem spiral sample, a large portion of material (50 percent) was in the -0.84+0.25 mm particle size fraction with a relatively low ash of just 3.6 percent. Again the -0.15+0.045mm and -0.045mm sizes contained high ash values at 33.8 percent and 79.3 percent, respectively.

Table 3-36: Size Analysis of Mid-Ash Jellico Spiral Concentrate

Particle Size Fraction (mesh)	Individual (%)		Cumulative (%)	
	Weight	Ash	Weight	Ash
+1.18	5.1	5.1	5.1	5.1
-1.18+0.84	15.7	3.3	20.8	3.7
-0.84+0.50	20.1	3.2	40.9	3.4
-0.50+0.25	30.0	3.7	70.9	3.6
-0.25+0.15	13.5	6.0	84.4	3.9
-0.15+0.045	11.6	33.8	96.0	7.6
-0.045	4.0	79.3	100.0	10.4

---

[8] Gatliff Tipple, MSHA ID 15-09938, operated by Alden Resources LLC a subsidiary of Ferroglobe, Route 904, Gatliff, Whitley County, Kentucky.

Because testing the Blue Gem sample revealed that a dispersant would help achieve a target ash value of -1.5 percent, it was decided to run tests of the Jellico samples with a dispersant as well.

Interestingly enough, it was determined for these Jellico samples that not only did increased grinding not help reduce ash values significantly but neither did the addition of a dispersant. For the undersize material product, ash values remained around 1.7 percent regardless of grinding conditions, which is a little higher than the target ash of -1.5 percent but still a significant reduction from the feed ash of 27.1 percent. Yield and recovery for these tests were around 70 percent and 90 percent respectively across all grinding tests. For the oversize material, there was a slight reduction in ash from 1.5 percent to 1.3 percent when moving from 80 percent passing 230 microns to 80 percent passing 28 microns. The oversize, like the Blue Gem sample, performed much better overall meeting the target ash as well as achieving high yield and recovery.

In general, for these two samples, the oversize material with slight grinding can be a very good feed stream for the HHS process, while the undersize material will require the use of a dispersant to be effective.

The results of the tests of the Jellico samples are shown in the following series of tables.

Table 3-37: HHS Results for Mid-Ash Jellico Undersize

Head Feed Ash%	Flotation Ash% (Feed to HHS)	HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
		Ash	Moisture			Combustible	Organic
27.1	10.2	1.62	6.0	76.3	65.9	88.9	91.5
		1.55	3.5	76.6	66.0	89.1	91.7
	10.2[1]	1.52	2.1	77.4	66.3	89.6	92.2
		1.58	2.7	76.9	66.1	89.3	91.9

[1] *One-stage re-cleaning*

Table 3-38: HHS Results Mid-Ash Jellico Undersize Ground to D80=75 µm

Head Feed Ash%	Ground Feed Ash% to HHS	HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
		Ash	Moisture			Combustible	Organic
27.1	10.2	1.77	3.1	85.5	69.7	94.0	96.7
		1.95	3.6	84.9	69.7	93.7	96.4
	10.2[1]	1.69	3.4	79.9	67.5	91.0	93.7
		1.75	4.1	80.0	67.6	91.1	93.8

[1] *One-stage re-cleaning*

Table 3-39: HHS Results for Mid-Ash Jellico Undersize Ground to D80=30 µm

Head Feed Ash%	Ground Feed Ash% to HHS	HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
		Ash	Moisture			Combustible	Organic
27.1	10.4	1.92	5.0	80.8	68.1	91.6	94.2
		2.10	6.7	80.7	68.2	91.6	94.2
	10.4[1]	1.73	4.0	79.3	67.3	90.7	93.3
		1.82	4.9	79.6	67.5	90.9	93.5

[1] *One-stage re-cleaning*

Table 3-40: HHS Results for Mid-Ash Jellico Oversize Ground to D80=230 µm

Ground Feed Ash% to HHS	HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
	Ash	Moisture			Combustible	Organic
3.2	1.58	2.4	44.3	96.2	97.8	98.0
	1.60	2.5	44.5	96.3	97.9	98.0
3.2	1.46	2.7	50.9	96.5	98.2	98.4
	1.64	5.0	48.7	96.7	98.3	98.4

Table 3-41: HHS Results for Mid-Ash Jellico Oversize Ground to D80=75 µm

Ground Feed Ash% to HHS	HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
	Ash	Moisture			Combustible	Organic
3.3	1.35	3.1	45.5	95.6	97.5	97.7
	1.42	2.6	45.4	95.7	97.6	97.7
3.3[1]	1.41	2.6	42.6	95.4	97.3	97.4
	1.47	2.9	43.1	95.6	97.4	97.6

[1] *One-stage re-cleaning*

Table 3-42: HHS Results for Mid-Ash Jellico Oversize Ground to D80=28 µm

Ground Feed Ash% to HHS	HHS Product (%wt.)		Rejects Ash (%wt.)	Mass Yield (%wt.)	Recovery (%wt.)	
	Ash	Moisture			Combustible	Organic
3.2	1.34	8.2	60.0	96.7	98.6	98.8
	1.32	7.5	59.8	96.7	98.6	98.8
3.2[1]	1.43	5.4	57.3	96.2	98.3	98.5
	1.36	9.2	58.0	96.6	98.5	98.7

[1] *One-stage re-cleaning*

3.1.4.4 Ferroglobe - Summary Observations of the HHS of the Blue Gem and Jellico Coal Seam Rejects Samples

Altogether, of the tested samples collected from the Ferroglobe plant, the flotation circuit streams, and particularly those from the flotation product stream, tested very well in the HHS process. All tests generated a low-ash, low-moisture product that met the target market specifications. The oversize material from both the Blue Gem and Jellico spiral concentrate samples also performed extremely well. Ultra-low-ash, high-value products were produced from these coal processing rejects streams with little to no grinding necessary. The performance on the -0.25 mm undersize material was promising as well, and capable of producing extremely low-ash coal products; however, this material requires the addition of a dispersant to aid in the selective agglomeration step.

Overall, the Blue Gem and Jellico feedstocks collected during this test program were considered ideal for upgrading ultrafine coal using the HHS process.

3.1.5 Arq Corbin LLC Plant Material – HHS Prospect

Another coal feedstock with potential to produce a high purity carbon product was provided by the **Arq Corbin, LLC *(Arq)*** group which is operating at a location near Corbin, Kentucky.[9] The sample consisted of a mixed source of pond fines from an abandoned coal processing plant waste slurry impoundment.

3.1.5.1 Background to the Arq Investigation

Arq is a private technology-led energy group, headquartered in London, UK, with operations in Lexington and Corbin, Kentucky; Truro, UK; and Queensland, Australia. Among other things, specializes in converting coal fines from coal waste dumps into particles small enough to be blended with fuel-oil products (see Arq Technology™). Through its USA subsidiary, Arq Corbin, LLC, Arq is in the process of installing a processing/conversion facility in the city of Corbin, Kentucky, adjacent to large coal waste embankments and attendant lagoons adjacent to the site of a former coal handling and preparation plant (locally known as the "U.S. Steel Property". The Arq Technology™ process of extracting and milling coal from coal waste dumps is expected to generate a highly decarbonized waste stream that could be beneficial for REE concentration purposes.

3.1.5.2 Arq Project – HHS Test Results of an "As-Received" Sample of Coal Rejects Impoundment

In these experiments, a random sample from the abandoned impoundment was initially tested on an "as-received" basis using the HHS process without any additional grinding or micronizing. The feed was found to have a D80 size of 67 microns. The testing program included the use of a chemical dispersant and several stages of agglomerate re-cleaning in an attempt to achieve an ultraclean (<1 percent ash) product. Unfortunately, as shown in the following table, the tests conducted using the "as-received" samples produced clean coal products containing 3.4-3.9 percent ash, which did not meet the ultraclean target specifications. Combustibles recovery values of these tests were also lower than desired with values ranging between 80-84 percent.

Table 3-43: Results Obtained from HHS Processing of Arq As-Received Pond Fines (d80=67 um)

Test #	Feed Ash (%)	HHS Product (%)		Rejects Ash (%)	Yield (%)	Combustible Recovery (%)
		Moisture	Ash			
1	46.5	1.6	3.9	84.1	46.9	84.2
2	46.5	2.2	3.9	84.3	47.0	84.4
3	46.5	1.8	3.4	80.9	44.3	80.2
4	46.5	5.6	3.9	84.0	46.8	84.1

3.1.5.2.1 Arq Project – HHS Test Results of Milled Sample of Coal Rejects Impoundment

After failing to achieve the target ash on an as-received basis, it was decided to attempt to improve liberation by grinding the sample in a ball mill to D80 passing size of 12 microns. For these tests, steel media was used in the ball mill with ball diameters ranging from 6 to 26 mm.

---

[9] Corbin Project, LLC, operated by Arq Corbin, LLC, MSHA ID 15-02134, Corbin, Knox County, Kentucky. Corporate Address: 301 East Main Street, Suite 1100, Lexington, Kentucky, 40507.

Unfortunately, as shown in the following table, the grinding did little to improve the ash reduction with clean coal product ashes ranging between 3.2 percent and 4.9 percent (test runs 5 through 8). However, in Test 9, a slight improvement to 2.6 percent ash was achieved by subjecting the ground feed to two stages of HHS processing (i.e., an initial roughing stage followed by reprocessing of agglomerates in a cleaning stage) to release additional mineral contained in water droplets trapped within the agglomerates.

Table 3-44: Results Obtained from HHS Processing of Arq Micronized Pond Fines (D80=12 um)

Test #	Feed Ash (%)	HHS Product (%)		Rejects Ash (%)	Yield (%)	Combustible Recovery (%)
		Moisture	Ash			
5	46.5	5.4	3.8	84.1	46.9	84.2
6	46.5	3.5	4.9	89.9	51.1	90.8
7	46.5	5.5	3.2	89.5	49.9	90.2
8	46.5	2.5	3.9	88.6	49.7	89.2
9 (2 Stage)	46.5	1.9	2.6	88.4	49.0	89.0

3.1.5.2.2 Arq Project – Sample Ground to D80 Passing 8 Microns

After realizing an improvement in ash reduction with agglomerate reprocessing, the Arq sample was ground to a D80 passing size of 8 microns in a laboratory ball mill and then subjected to various stages of cleaning. One to four sequential stages of re-cleaning were evaluated to determine whether any improvement could be made in product purity using this approach. The test data obtained from these experiments are summarized in the following table. As shown, the clean coal product ash values for these tests ranged between 1.9 percent and 2.9 percent, which were slightly improved over those obtained in single stage tests. However, the qualities still fell short of the target specifications of <1 percent ash. The test results do, however, indicate that additional stages of cleaning can be used to improve the overall quality of the clean coal products generated by the HHS process. The best results obtained in these tests were associated with three or four stages of re-cleaning. It is also possible that the slightly smaller grind size (8 versus 12 micron) used in these tests allowed a slightly better ash level to be achieved.

Table 3-45: Results Obtained from HHS Processing of Arq Micronized Pond Fines (d80=8 um) with Multiple Sequential Stages of Cleaning and Recleaning

Test #	Feed Ash (%)	HHS Product (%)		Rejects Ash (%)	Yield (%)	Combustible Recovery (%)
		Moisture	Ash			
10[1]	46.3	2.8	2.9	83.3	45.8	83.1
11[2]	46.3	4.7	2.5	84.2	46.1	84.0
12[3]	46.3	2.2	2.3	84.2	46.0	84.0
13[4]	46.3	1.7	2.0	84.0	45.8	83.8
	46.3	1.1	2.1			
14[3]	46.3	1.5	1.9	86.2	47.1	86.4
	46.3	2.3	1.9			

[1] *One-stage re-cleaning;* [2]*Two-stage re-cleaning;*
[3]*Three-stage re-cleaning;* [4]*Four-stage re-cleaning.*

3.1.5.2.3 Arq Project – HHS Test Results of Milled Sample of Coal Rejects Impoundment with Dispersant Additive

After quantifying the benefits of multistage cleaning, further improvements in ash reduction were attempted using sodium silicate as a dispersant during the agglomeration step. Unfortunately, as shown in the following table, the test results with the dispersant did not differ much from the previous results, with product ash values ranging from 2.4 percent to 3.0 percent ash. The use of sodium silicate actually produced slightly higher clean coal ash levels. This was surprising since the feed used in these tests was ground in an attrition mill to a very small D80 passing size of 5 microns using steel media.

Table 3-46: Results Obtained from HHS Processing of Arq Pond Fines (D80=5 microns) with Multiple Stages of Cleaning and Sodium Silicate Dispersant (for Selected Tests)

Test #	Feed Ash (%)	Sodium Silicate Added?	HHS Product (%)		Rejects Ash (%)	Yield (%)	Combustible Recovery (%)
			Moisture	Ash			
15	52.2	Yes	3.1	3.0	91.9	44.7	90.7
16[1]	52.2	Yes	2.9	2.7	91.8	44.2	89.9
17[2]	52.2	Yes	1.7	2.7	91.6	44.4	90.2
18[3]	52.2	Yes	1.9	2.6	91.5	44.3	90.1
19[1]	52.2	No	2.3	2.5	92.2	44.6	91.0
20[2]	52.2	No	3.7	2.5	91.9	44.4	90.6
21[3]	52.2	No	1.8	2.4	91.6	44.3	90.3

[1] One-stage cleaning; [2] Two-stage cleaning; [3] Three-stage cleaning.

3.1.5.3 Arq Project – HHS Test Results of Attrition Grinding of Sample of Coal Rejects

It was decided to attrition grind the agglomerates (much like the Ferroglobe procedure) down to an even smaller D80 passing size of 4 microns. The micronizing was also performed using ceramic media instead of iron media to minimize any increase in feed ash due to media wear and corrosion. As shown the table below, this new test procedure generated product ash values between 1.0-1.2 percent ash. These purity levels were very close to the <1 percent ash specification targeted for this particular feedstock. In fact, one test (run 24) produced a 0.96 percent ash product after two-stages of agglomerate cleaning. These results suggest that regrinding of an agglomerated product is the most effective way of generating very low ash products with the HHS process.

Table 3-47: Results Obtained from HHS Processing of Micronized (D80=4 um) Clean Coal Agglomerates from an Initial Stage of HHS Processing on As-Received Arq Pond Fines

Test #	Feed Ash (%)	HHS Product (%)		Rejects Ash (%)	Yield (%)	Combustible Recovery (%)
		Moisture	Ash			
22	46.5	0.8	1.21	86.3	46.8	86.4
23[1]	46.5	1.3	1.03	86.3	46.6	86.2
24[2]	46.5	0.9	0.96	86.1	46.5	86.1
25	46.5	2.1	1.14	85.7	46.4	85.7
26[1]	46.5	2.3	1.07	85.7	46.3	85.7
27[2]	46.5	1.7	1.02	85.6	46.3	85.6

Agglomeration with as-received feed, followed by grinding of agglomerates in a stirred ball mill for 1 hour using ceramic (alumina) media. [1]One-stage cleaning; [2]Two-stages of cleaning.

3.1.6 Blue Diamond Coal Company: Fire Clay Coal Seam (Hazard 4) Seam Rejects

3.1.6.1 Fire Clay Seam (Hazard 4) Coal Reject Feedstock

The investigative team was aware of prior investigations[10,11] of the presence of REE in the rejects from coal handling and preparation plants that were processing the Fire Clay Seam coal (Hazard 4) in Kentucky. The team collaborated with one of the principal coal mine operators in the Fire Clay Seam, **Blue Diamond Coal Company**, a.k.a. **Blackhawk Mining LLC *(Blackhawk or Leatherwood)*** and that operator's lessor, **Kentucky River Properties LLC *(KRP)***, to undertake an in depth study of the recovery of REE oxides (*REO*) from the preparation plant rejects generated from this seam.

The Blue Diamond, KRP, and MMA 29956 collaboration culminated in the detailed REO system design reported in this chapter. The reader should note that the details of the REE resource base in the Hazard 4 Seam are recited in Chapter 2.0. The reader should review Chapter 2.0 in concert with Chapter 3.0. Furthermore, the critical techno-economics aspects of harvesting REO from the Hazard 4 Seam rejects are discussed in Chapter 4.0. And finally, the mine operating aspects of Blue Diamond's Hazard 4 Seam mines that affect the REO recovery rates are recited in portions of Chapter 5.0.

The following discussions will step the reader through the systematic research efforts associated with this particular REE feedstock.

3.1.6.2 Hazard 4 Thickener Underflow Froth Flotation Test Results

Thickener underflow samples from Blue Diamond's No. 76 Plant (locally known as the Leatherwood coal processing facility) were collected and evaluated to determine the ability of producing a clean coal product while also producing a middlings material as a feedstock for a REE recovery

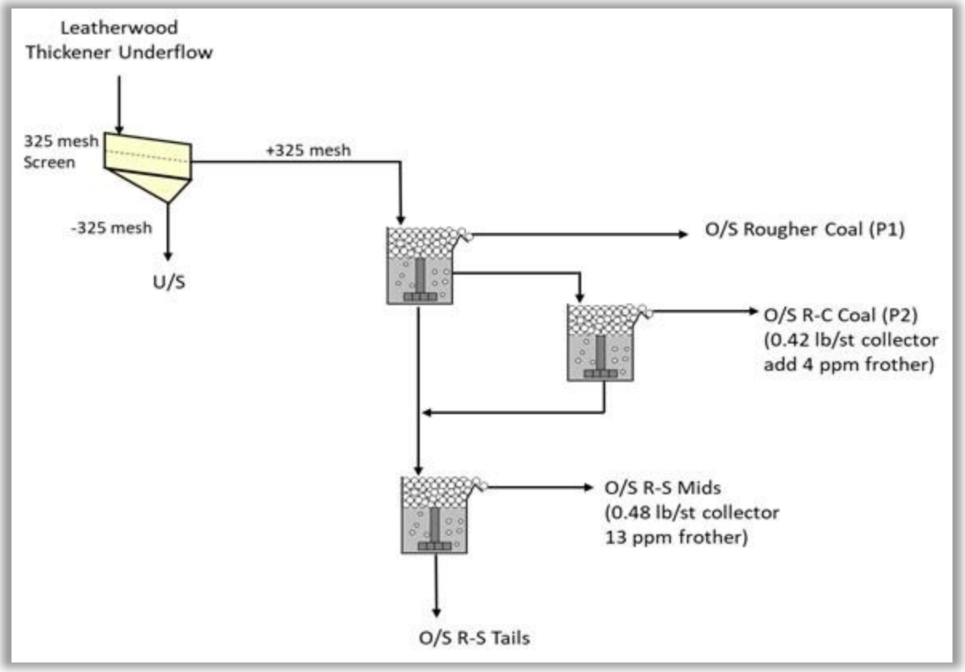
---

[10] "Lanthanide, yttrium, and zirconium anomalies in the Fire Clay coal bed, Eastern Kentucky", James C. Hower, Leslie F. Ruppert , Cortland F. Eble, Center for Applied Energy Research, Lexington, Kentucky 40511 USA, U.S. Geological SurÍey, Reston, VA, USA, Kentucky Geological Survey, Lexington, KY 40506, USA, 1999.

[11] "Association characteristic study and preliminary recovery investigation of rare earth elements from Fire Clay Seam coal middlings", Wencai Zhang, Xinbo Yang, Rick Q. Honaker, University of Kentucky, Lexington, Kentucky 40506-0107, USA, November 28, 2017.

Approximately 10 gallons of sample were initially screened using a 325-mesh (45 µm) sieve to deslime the feed to froth flotation, which is a sizing step commercially achieved using 6-in diameter classifying cyclones. The material coarser than 325-mesh was treated in a froth flotation cell using a 'grab-and-run' approach in two steps to obtain low-ash clean coal concentrates. The tailings of the first stage of flotation was then re-floated using more enriched conditions (higher collector and frother dosages and increased aeration rate) to float the higher-ash particles that represent a middlings product. The flotation procedure can be described as a typical rougher-scavenger flotation circuit. The process is depicted in the following figure.

Figure 3-28: Procedure Used to Process the Leatherwood Thickener Underflow Sample to Assess REE Recovery Potential Using a Rougher-Scavenger Froth Flotation Circuit



The processing strategy was successful in producing a clean coal product containing 4.6 percent ash and a middlings product consisting of 18.3 percent ash and a total REE content of 716 ppm on an ash basis and 131 ppm on a whole sample basis (*see the following table*). Similar to previous findings, the highest ash-based REE values were associated with the lowest ash content fractions. The highest whole-sample REE value was obtained from the +325-mesh flotation tailings with a value of nearly 300 ppm. However, 86 percent by weight of the REEs in the thickener underflow material existed in the -325-mesh size fraction which was not treated by flotation. The middlings product accounted for approximately 2 percent of the total REEs and 3.2 percent of the total weight.

Table 3-48: Flotation Test Results for the Leatherwood Thickener Underflow

Sample		Weight (%)	Ash (%)	Total REE Content (ppm)	
				Ash-based	Whole-Sample
+325-mesh	Rougher Ultra Clean Coal (P1)	0.1	4.6	1336	61
	Rougher-Cleaner Product (P2)	6.4	4.6	1440	66
	Rougher Scavenger Product (M)	3.2	18.3	716	131
	Rougher Scavenger Tails (T)	7.0	87.2	344	299
-325-mesh	Undersize	83.4	63.4	347	220
Total		**100.0**	**59.8**	**429**	**256**

3.1.6.3 HHS of the Hazard 4 Seam Thickener Underflow of the Leatherwood Plant

Tests were also conducted to evaluate the potential of using the HHS process to recover a REE concentrate stream. The tests evaluated the ability of the process to recover rare earth minerals using sodium oleate and hydroxamate as collectors. For convenience, REE concentrations are reported on both an ash-basis and on a whole-basis. Each test produced a rougher concentrate, scavenger concentrate and tailings sample.

Per the testing procedure, the sample was decarbonized using oil agglomeration and then ground to a d_{80} passing size of 6 microns using a stirred ball mill. The sample was then conditioned under low-shear mixing conditions with sodium silicate (*SS*). After conditioning, the sample was subjected to high-shear mixing conditions with sodium oleate (*NaOL*) for Tests 1 and 3 and hydroxamate (*KOHX*) for Tests 2 and 4. The procedure for each test is schematically shown in the next four (4) figures.

Figure 3-29: Procedure for Leatherwood Plant Thickener Underflow HHS REE Recovery, Hazard 4 Seam (Test 1 – NaOL Collector)

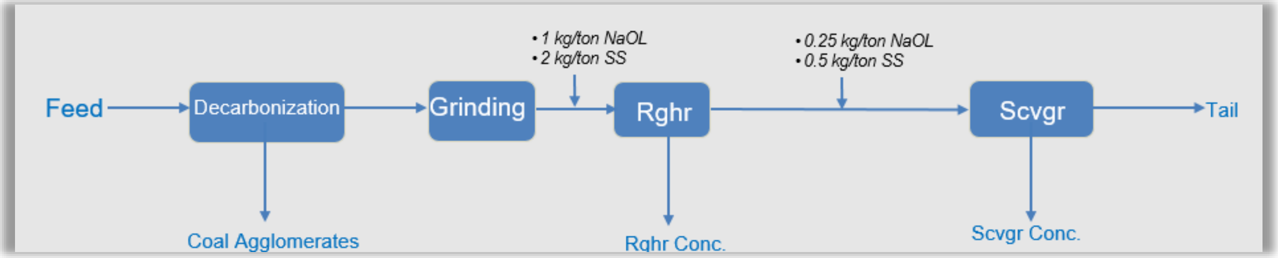


Figure 3-30: Procedure for Leatherwood Plant HHS REE Recovery, Hazard 4 Seam (Test 2 – KOHX Collector).

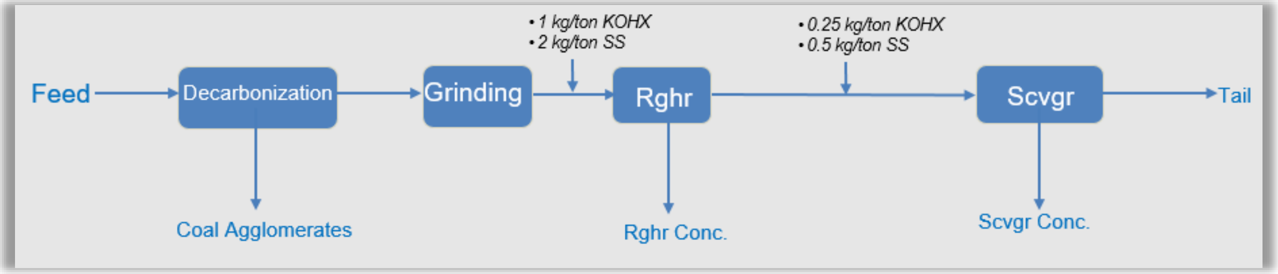


Figure 3-31: Procedure for Leatherwood Plant Thickener Underflow HHS REE Recovery, Hazard 4 Seam (Test 3 – NaOL Collector)

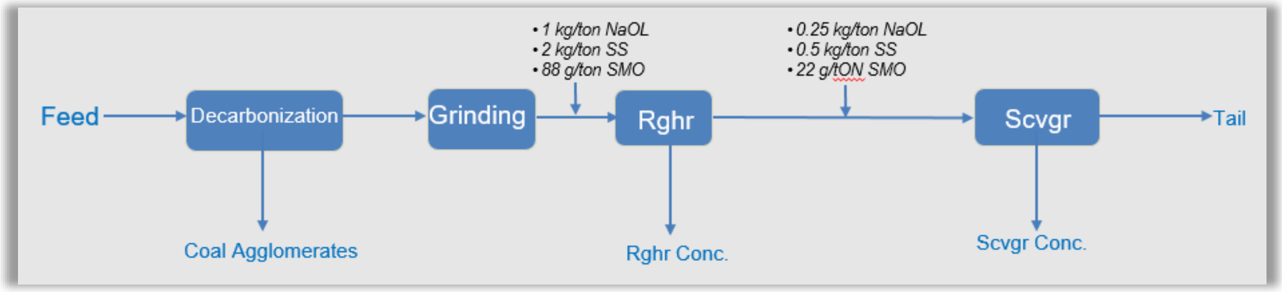
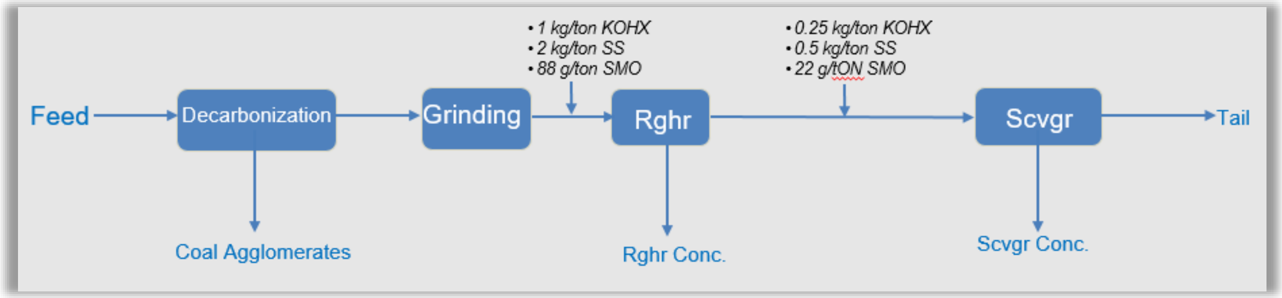


Figure 3-32: Procedure for Leatherwood Plant Thickener Underflow HHS REE Recovery, Hazard 4 Seam (Test 4 – KOHX Collector)



<u>3.1.6.3.1</u> <u>*Results of Standard HHS Testing Procedures of the Leatherwood Plant, Hazard 4, Thickener Underflow*</u>

Standard HHS testing procedures were followed after the conditioning step where oil-in-water emulsions (o/w) were formed and then cleaned and dewatered to produce rougher and scavenger products.

The experimental results obtained from the HHS testing of the Hazard 4 coal thickener underflow are summarized in the next four (4) tables. The data indicate that the HHS tests carried out with sodium oleate performed slightly better than those conducting with hydroxamate. The sodium oleate collector produced rougher and scavenger concentrates of 1439-1690 ppm and 660-684 ppm, respectively, on an ash-basis and 1063-1153 ppm and 609-626 ppm, respectively, on a whole-basis. In comparison, the HHS tests conducted with hydroxamate produced slightly worse results, with rougher and scavenger concentrates of 924-1248 ppm and 648-1118 ppm, respectively, on an ash-basis and 782-961 ppm and 605-1005 ppm, respectively, on a coal-basis.

Table 3-49: HHS REE Recovery Test Results for Leatherwood Plant Thickener Underflow, Hazard 4 Seam (Test 1 - NaOL)

Products	Mass (%)	Ash (%)	REE (ppm)				Recovery (%)	
			Ash-Basis		Whole-Basis			
			Ind.	Cum.	Ind.	Cum.	Ind.	Cum.
Coal	50.0	36.9	462.7	462.7	170.7	170.7	33.0	33.0
Rghr Conc	1.0	68.2	1,690.5	506.8	1,153.6	190.1	4.5	37.5
Scvgr Conc	3.1	91.5	684.6	529.8	626.4	215.2	7.5	45.0
Tail	45.9	94.1	329.3	396.9	309.8	258.6	55.0	100.0

Feed	100.0	65.2	396.9	-	258.6	-	100.0	-

Table 3-50: HHS REE Recovery Test Result for Leatherwood Thickener Plant Underflow, Hazard 4 Seam (Test 2 - KOHX)

Products	Mass (%)	Ash (%)	REE (ppm)				Recovery (%)	
			Ash-Basis		Coal-Basis			
			Ind.	Cum.	Ind.	Cum.	Ind.	Cum.
Coal	50.0	36.9	462.7	462.7	170.7	170.7	32.5	32.5
Rghr Conc	2.8	84.6	924.1	514.5	782.2	202.6	8.2	40.7
Scvgr Conc	4.0	93.3	648.5	534.8	605.1	230.9	9.2	49.9
Tail	43.3	93.8	324.2	403.4	304.2	262.6	50.1	100.0
Feed	100.0	65.1	403.4	-	262.6	-	100.0	-

Table 3-51: HHS Test Results for Leatherwood Thickener Underflow (Test 3 – NaOL)

Products	Mass (%)	Ash (%)	REE (ppm)				Recovery (%)	
			Ash-Basis		Coal-Basis			
			Ind.	Cum.	Ind.	Cum.	Ind.	Cum.
Coal	43.8	29.6	507.1	507.1	150.3	150.3	25.2	25.2
Rghr Conc	1.8	73.9	1,439.5	594.9	1,063.5	186.8	7.4	32.6
Scvgr Conc	4.5	92.3	660.1	609.5	609.2	224.6	10.5	43.0
Tail	49.9	94.1	317.0	399.5	298.3	261.4	57.0	100.0
Feed	100.0	65.4	399.5	-	261.4	-	100.0	-

Table 3-52: HHS REE Recovery Test Results for Leatherwood Plant Thickener Underflow, Hazard 4 Seam (Test 4 – KOHX)

Products	Mass (%)	Ash (%)	REE (ppm)				Recovery (%)	
			Ash-Basis		Coal-Basis			
			Ind.	Cum.	Ind.	Cum.	Ind.	Cum.
Coal	43.8	29.6	507.1	507.1	150.3	150.3	25.1	25.1
Rghr Conc	1.7	77.0	1,248.7	573.8	961.9	180.0	6.1	31.2
Scvgr Conc	2.5	89.9	1,118.8	647.0	1,005.5	222.4	9.4	40.6
Tail	52.1	93.8	319.7	402.3	299.8	262.7	59.4	100.0
Feed	100.0	65.3	402.3	-	262.7	-	100.0	-

As a side note, an experiment was also performed to evaluate the efficiency of the HHS process in recovering a high-quality coal product from the thickener underflow samples from the Leatherwood Plant. As shown in the following table, the HHS process performed well for this particular feedstock, reducing the ash from 24.8 percent in the feed down to 3.9 percent in the cleaned product with a combustible recovery of better than 94 percent.

Table 3-53: Leatherwood Plant Thickener Underflow Sample HHS Decarbonization Results (Hazard 4 Seam)

Feed Ash (%)	Product Ash (%)	Rejects Ash (%)	Yield (%)	Recovery (%)		Ash Rejection (%)
				Combust.	Organic	
24.8	3.9	83.2	73.7	94.1	96.4	88.5

<u>3.1.6.3.2 Supplemental Treatment of Leatherwood Plant Thickener Underflow Prior to HHS to Improve REM Recovery</u>

Additional tests were completed on the Leatherwood thickener underflow sample to improve the performance of the HHS process for REE minerals (*REM*) recovery. From previous test work, it was determined that the REM population (i.e., hydrophobic particle population) and liberation are the main issues hindering the HHS process. In this series of test work, steps were taken to address these issues via increasing the hydrophobic particle population and attempting a new method for liberating REM.

3.1.6.3.2.1 HHS Feedstock Preparation by Blunging

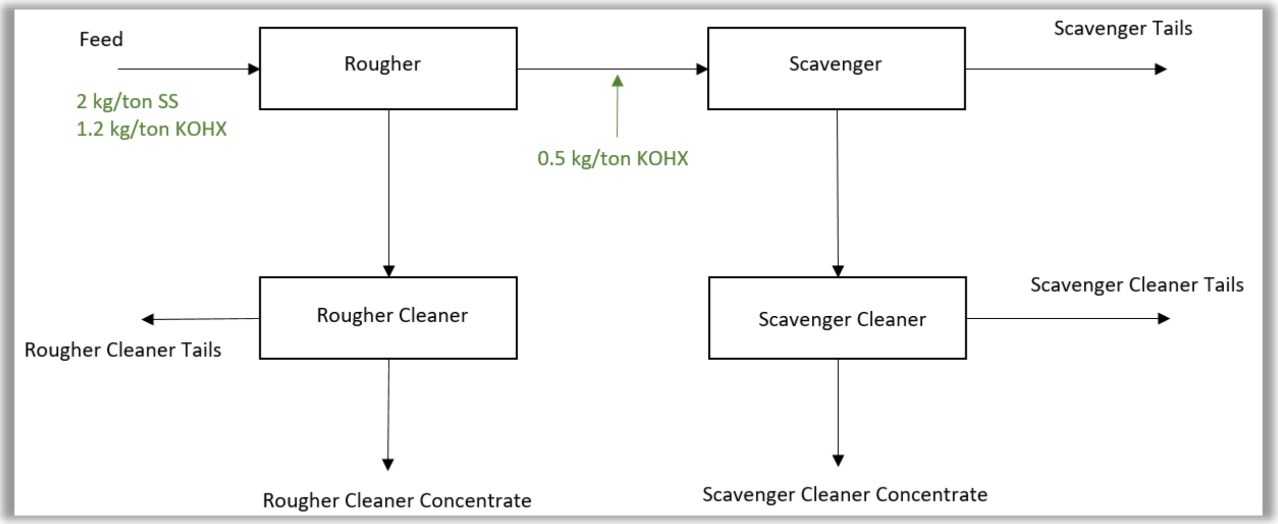
The first issue addressed was liberation. Blunging is a common technique employed in the clay industry for dispersing ultrafine clay. In the case of the Leatherwood thickener underflow sample, SEM-TEM data indicated that monazite particles bearing REEs have a particle size ranging from 1 µm to 10 µm and appear to be already well-liberated in the thickener underflow sample. Therefore, in the following test, blunging was introduced as a method of dispersing and removing fine clay particles that could create a slime effect and reduce REM recovery by the HHS process. In this test, an octagonal vessel was used with a blunging impeller, as shown in the following figure, to achieve the high shear agitation needed to disperse ultrafine clay particles.

Figure 3-33: (Left) Blunging Impeller, (Right) Blunging Mixing Vessel

Roughly 500 grams of decarbonized Leatherwood thickener underflow sample in slurry at 50 percent solids was mixed in the vessel at 1,000 revolutions per minute *(rpm)*, with the addition of 10lb/ton of sodium silicate dispersant. The mixture was allowed to condition with the sodium silicate for 30 minutes before being transferred to a settling cylinder. Settling distance was determined using 8 µm monazite settling velocity. After a predetermined settling time and distance, the overflow was collected for standard HHS testing.

After blunging, the HHS process component of the test was completed following the procedure depicted in the following figure.

Figure 3-34: Blunged Sample in the HHS Procedure



One hundred grams of sample after blunging was mixed with 350 ml of water at roughly 400 rpm. The slurry was allowed to condition with 2kg/ton of sodium silicate for 15 minutes. This was to help with the remaining clay in the sample and depress non-rare earth bearing minerals preventing collector adsorption. After 15 minutes, 1.2kg/ton of hydroxamate collector was added to hydrophobized the REE-bearing minerals and conditioned for an additional 15 minutes. Two hundred milliliters of heptane oil, which is the preferred hydrophobic liquid for HHS REE testing, was added and mixed under high shear conditions to form oil-in-water emulsions. The sample would then be poured into a separatory funnel phase separated to remove tailings. Hand shaking was then employed with the addition of more oil in an attempt to induce phase inversion and release hydrophobic REM to the oil phase. A rougher, scavenger, and cleaner circuit was used to further clean the sample and attempt to increase REE recovery. Both cleaner circuits required no additional reagent consumption; only water washing was used to break and reform emulsions releasing entrained tailings.

Blunging the Leatherwood thickener underflow sample prior to HHS processing does not generate an improvement over conventional HHS testing results. These results are presented in the following table.

Table 3-54: HHS REE Results of Blunged Leatherwood Thickener Underflow

	Mass			REE (ppm)		
Product	Grams	%wt.	Ash (%wt.)	Ash Basis	Whole Basis	Recovery (%wt.)
Feed	104.1	100.0	92.9	318.0	295.3	100.0
Rougher Cleaner Conc.	0.1	0.1	52.4	622.5	326.0	0.1
Rougher Cleaner Tail	1.2	1.1	92.2	287.4	265.0	1.0
Scavenger Cleaner Conc.	5.1	4.9	53.6	538.4	288.8	4.8
Scavenger Cleaner Tail	2.7	2.6	92.0	197.1	181.3	1.6
Scavenger Tail	95.0	91.3	93.0	316.9	294.6	91.1

As can be observed from the preceding table, on a whole ash basis, there is little separation seen, with REE ppm values reaching only 326 ppm in the rougher cleaner concentrate product and 288 ppm in the scavenger cleaner concentrate product. This is increased only slightly from the feed grade of 295 ppm for the rougher cleaner concentrate and decreased in the scavenger cleaner concentrate. On an ash basis, the recovery is slightly better with 622 ppm in the rougher cleaner concentrate, 538 ppm in the scavenger cleaner concentrate and 318 ppm in the feed. Additionally, only 0.1 grams were recovered in the rougher cleaner concentrate and 5.1 in the scavenger cleaner concentrate, out of 104 grams feed material. Recovery was low for the two concentrate products at 0.1 percent and 4.8 percent for rougher and scavenger, respectively. This separation is poor compared to the separation seen with traditional grinding of the sample and HHS testing. A graphical representation of the results is shown in the following two figures.

Figure 3-35: TREE Whole Sample Basis Leatherwood Plant Thickener Underflow; Blunged HHS Feedstock Testing Results (Hazard 4 Seam)

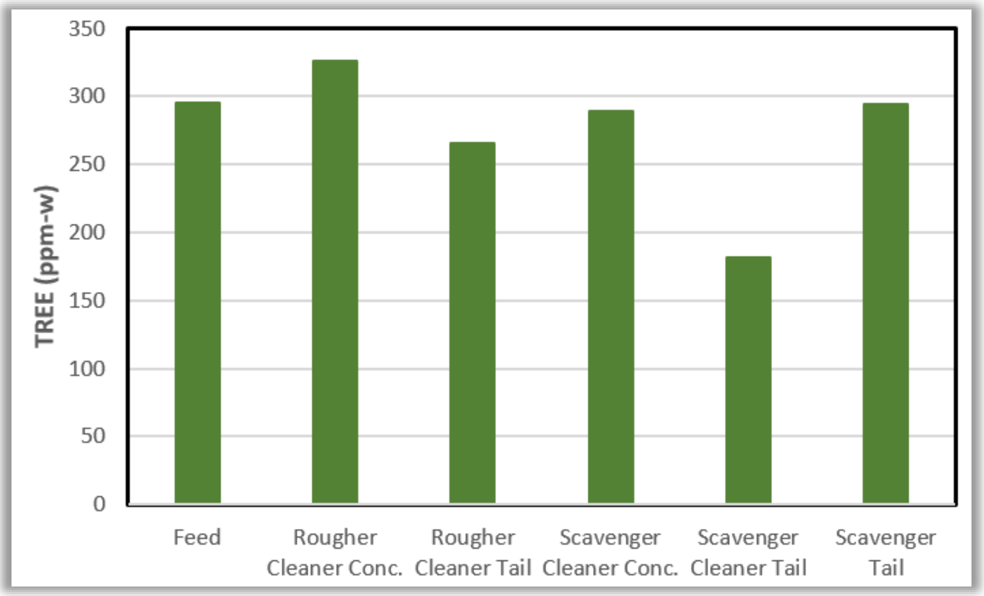
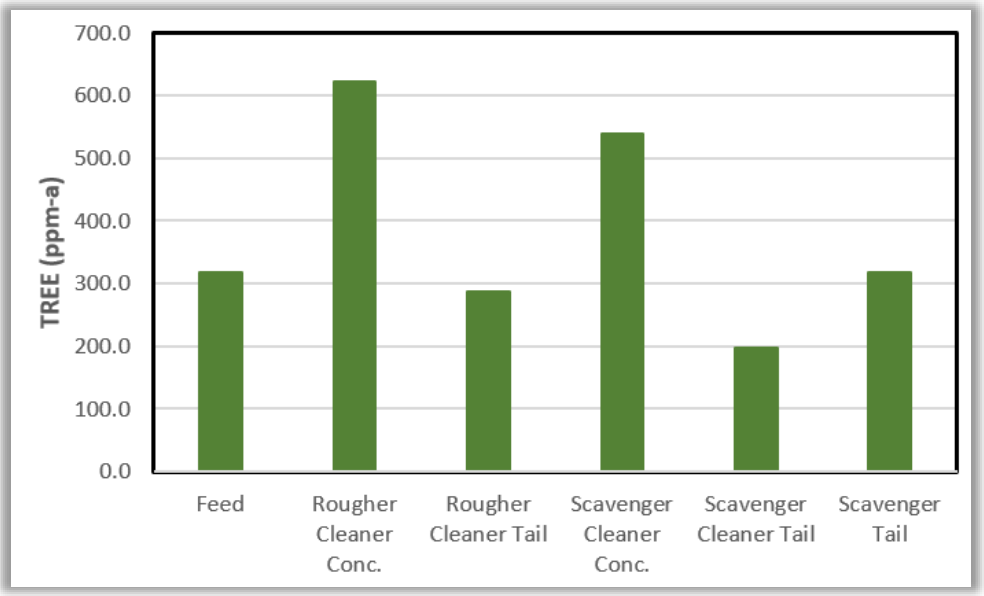


Figure 3-36: TREE Ash Basis Leatherwood Plant Thickener Underflow; Blunged HHS Feedstock Testing Results (Hazard 4 Seam)



3.1.6.3.2.2 Oil Agglomeration and removal of Coal Particles from the HHS Feedstock

The second parameter that was investigated in the test program was increasing the population of hydrophobic particles to improve emulsion formation and collection of REM into the oil phase. To accomplish this goal, a sample of Leatherwood thickener underflow was decarbonized using oil-agglomeration. The resultant coal agglomerates were removed and saved for later use. The tailings from oil-agglomeration were vacuum filtered and then wet-ground in a laboratory attrition mill for one (1) hour to a D80 size of 4 µm.

Three tests were then run on this sample, two with coal added back to the HHS reactor, and one as a traditional HHS REE test for comparison.

<u>3.1.6.3.2.2.1 HHS Process Applied to D80 Size of 4 µm Feedstock with Coal Added Back</u>

In the first test (test “a”), six grams of coal were added to the mixing tank with approximately 72 grams of dry solids thickener underflow which was slurried. The mixture was conditioned with 1 kg/ton of hydroxamate for 30 minutes, after which a small amount of heptane oil was added and mixed under high shear conditions to form agglomerates rather than traditional emulsions. After 30 minutes of high shear mixing, small micro-agglomerates were formed and screened off from the tails. The agglomerates were transferred to the laboratory Morganizer used during decarbonization studies for HHS coal testing. The laboratory Morganizer is a glass column with two vibrating mesh screens. The column is separated into a water phase and an oil phase. The vibrating screens provide a mechanical force to de-stabilize and de-agglomerate the agglomerates in the oil phase. This action releases entrained gangue minerals and water droplets which are allowed to coalesce and fall to the water phase. The free hydrophobic particles move into the oil phase and out through the overflow port.

REE feed grade for this sample was 290 ppm. It appears that this sample had an extremely good separation with an REE concentrate grade of 1,789 ppm and a corresponding recovery of 16.2 percent. However, the original sample was lost during analysis due to over-pressure and had to be re-run with the little sample that remained, causing a high degree of possible error. It is possible that this high recovery is also due to a nugget effect skewing the results. Neither of the other two tests runs on this thickener underflow sample performed nearly as well. The results are presented in the following table and related figures

Table 3-55: Leatherwood Thickener Underflow Coal Addition HHS REE Test "a" Results

Product	Mass		Ash (%wt.)	REE (ppm)		Recovery (%wt.)
	Grams	%wt.		Ash Basis	Whole Basis	
Feed	77.9	100.0	94.2	308.1	290.3	100.0
Concentrate	2.1	2.6	4.0	44,282.2	1,789.0	16.2
Agglomerates	2.8	3.5	6.8	1,463.3	99.8	1.2
Morganizer Tails	0.7	0.9	19.0	773.4	146.6	0.5
Tails	75.2	96.5	92.2	283.2	261.1	86.8

Figure 3-37: TREE Whole Sample Basis Leatherwood Thickener Underflow Coal Addition Test "a"

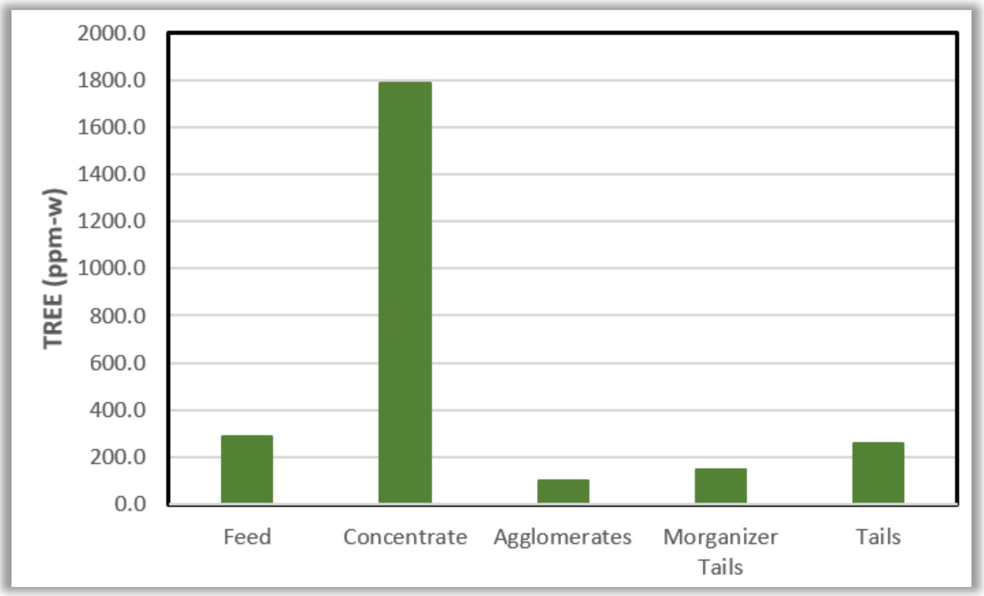
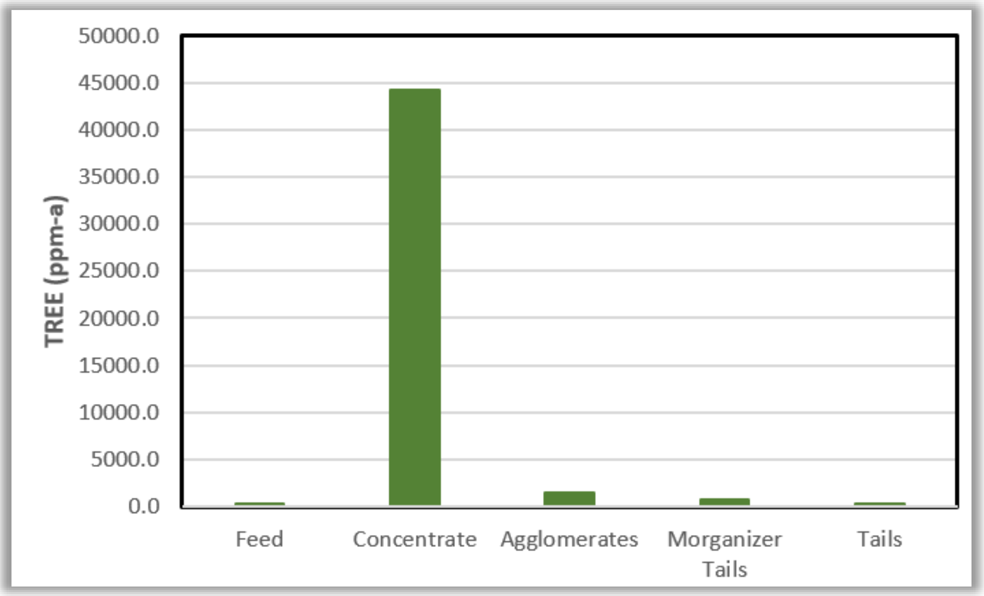


Figure 3-38: TREE Ash Basis Leatherwood Thickener Underflow Coal Addition Test "a"



3.1.6.3.2.2.2 Traditional Two-Liquid Flotation Method HHS Process Applied to D80 Size of 4 µm Feedstock with Coal Added Back

In the second test ("b"), the procedure used in test one was modified by using the more traditional two-liquid flotation method of HHS testing with coal addition. For this test oil-in-water emulsions were formed and separated in a separatory funnel. A 1 kg/ton of hydroxamate was used to condition the slurry and 6 grams of coal were added at this time. A larger amount of heptane oil was added to form emulsion after 15 minutes of conditioning and the emulsions were transferred to a separatory funnel. After removing the water phase, which held gangue minerals, additional oil was added to the system and hand-shaken to break the emulsions releasing hydrophobic particles to the oil phase and entrapped gangue to the water phase. The oil phase was collected as a concentrate sample while the remaining emulsions that did not break were collected as a middling product.

This test was unsuccessful in increasing REE concentration by increasing the hydrophobic particle population. This result is tabulated below and depicted in the attendant figures. As such, there was essentially no separation as both the concentrates and middling products produced grades of 101 ppm and 121 ppm, respectively compared to a feed grade of 290 ppm. In essence, only coal was recovered during this test.

Table 3-56: Leatherwood Thickener Underflow Coal Addition HHS REE Test "b" Results

Product	Mass		Ash (%wt.)	REE (ppm)		Recovery (%wt.)
	Grams	%wt.		Ash Basis	Whole Basis	
Feed	91.6	100.0	94.2	308.1	290.3	100.0
Concentrate	2.9	3.1	3.1	1,228.0	101.3	1.1
Tails	87.5	95.5	95.5	324.4	303.0	99.7
Middling	1.3	1.4	1.4	1,298.5	121.0	0.6

Figure 3-39: TREE Whole Sample Basis Leatherwood Thickener Underflow Coal Addition Test "b"

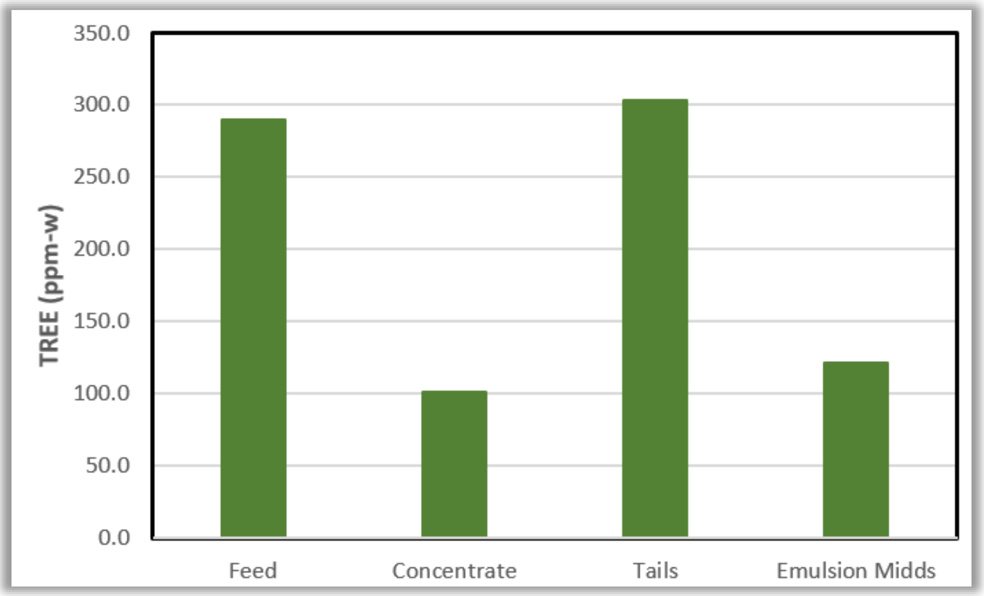
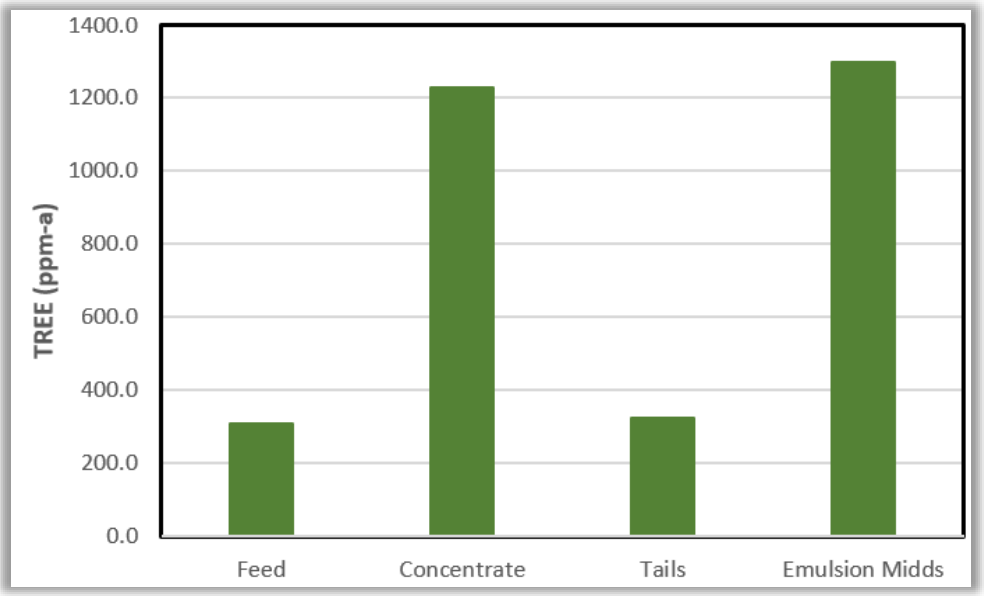


Figure 3-40: TREE Ash Leatherwood Thickener Underflow Coal Addition Test "b"



<u>3.1.6.3.2.2.3 HHS Process Applied to D80 Size of 4 µm Feedstock without Coal Added Back</u>

The final test "c" was a control test without any addition of coal for comparison purposes to indicate the impact, if any, that coal addition had on REE concentration. The test was performed in the same way as Test 2 but without the addition of coal. Additionally, two stages of water washing were employed in an attempt to further remove entrapped gangue minerals and induce phase inversion to allow for emulsion breakage and release of particles to the oil phase.

This test performed much better than the previous two tests, assuming Test 1 is in error. The cleaner concentrate product had a grade of 583 ppm whole basis and 1,197 ppm ash basis, compared to 290 ppm whole and 308 ppm ash observed in the feed. Interestingly, the middling sample recorded the highest-grade product at 641 ppm whole basis. Unfortunately, recovery values were low at 1.8 percent and 0.9 percent for the cleaner concentrate and middling products, respectively.

Table 3-57: Leatherwood Thickener Underflow Coal Addition HHS REE Test "c" Results

Product	Mass		Ash (%wt.)	REE (ppm)		Recovery (%wt.)
	Grams	%wt.		Ash Basis	Whole Basis	
Feed	83.1	100.0	94.2	308.1	290.3	100.0
Cleaner Concentrate	0.7	0.9	48.8	1,197.2	583.6	1.8
First Cleaner Tails	1.4	1.7	92.2	286.0	263.6	1.5
Final Cleaner Tails	1.7	2.0	94.0	327.9	308.1	2.1
Middling	0.3	0.4	90.4	710.0	641.9	0.9
Rougher Tails	79.0	95.0	94.3	286.8	270.5	88.6

Figure 3-41: TREE Whole Basis Leatherwood Thickener Underflow Coal Addition Test "c"

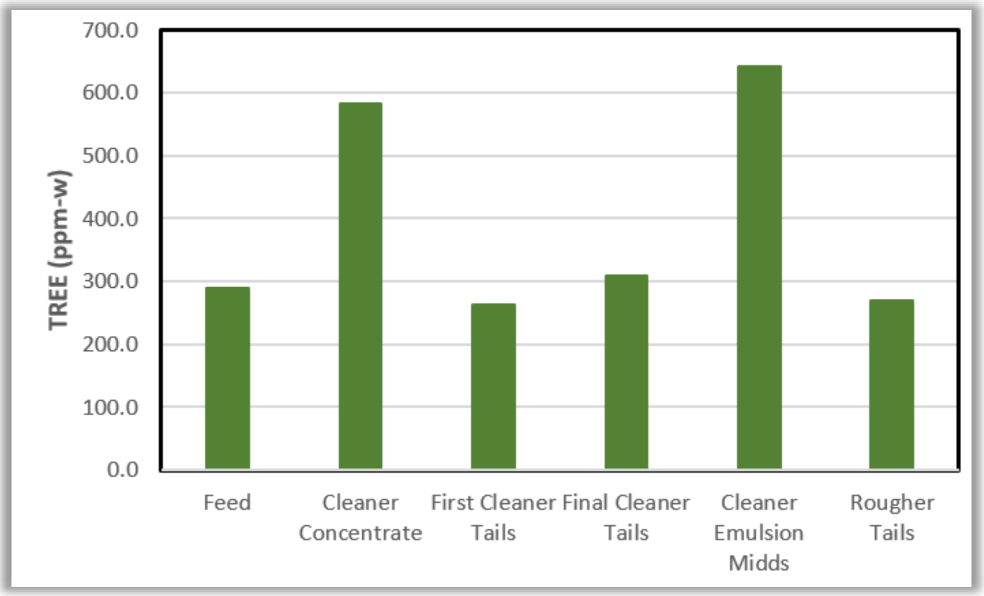
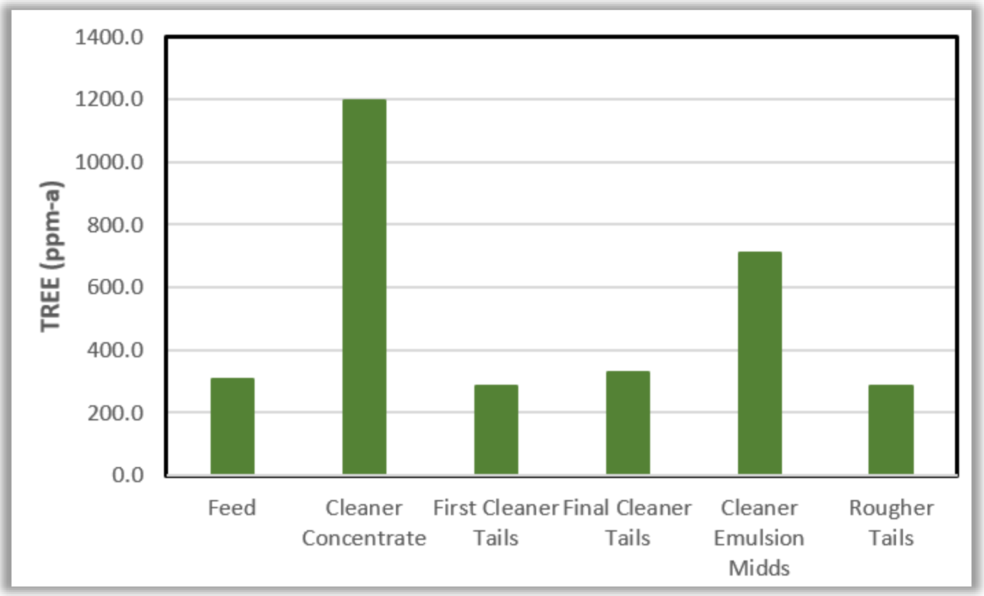


Figure 3-42: TREE Ash Basis Leatherwood Thickener Underflow Coal Addition Test "c"



<u>3.1.6.3.2.2.4 Conclusions related to Optimization Attempts "a", "b" and "c"</u>

Overall, coal addition does not appear to improve the HHS REE process in terms of improving REE concentration based on the few tests run. It appears in most cases the HHS process preferentially recovers coal in these cases without collecting REE minerals, as coal is naturally hydrophobic and well-suited for HHS. Additional tests should be undertaken to validate or invalidate the results of Test 1, and further investigate solutions to liberation and population which still appear to hinder the HHS process.

3.2 Roasting and Acid Leaching Experiments

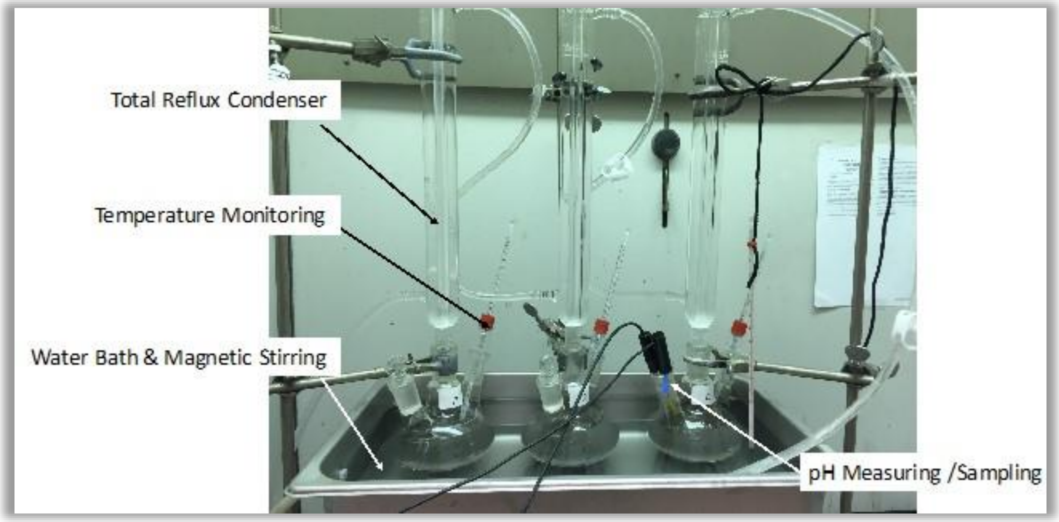
3.2.1 Roasting

Roasting was performed using an oven manufactured by Thermo Fisher at different temperatures (500℃, 600℃, and 750℃). The roasting temperatures were reached by elevating from 25℃ at a rate of 10℃/min. The samples were roasted at temperature for two hours, after which, the system was cooled down automatically to 25℃. Acid leaching tests were performed on the roasted samples together with non-roasted samples for comparison purposes.

3.2.2 Acid Leaching

The leachability of the REEs and contaminant elements were assessed using the test apparatus shown in the following figure. This apparatus allows control of leaching temperature and stirring speed. Evaporation during the leaching process were negligible due to the condensers.

Figure 3-43: Apparatus used for the Acid Leaching Tests



The standard leaching conditions included: leaching temperature (75℃), magnetic stirring speed (500 rpm), solid concentration (1 percent, 10 grams in a total slurry of 1 liter), and acid concentration (1.2 M H_2SO_4). For each test, a series of representative samples (15 mL) were collected at 5, 15, 30, 45, 60, 90, 120, 180, 240, and 300 minutes *(min)* of the leaching process. Weights of the samples were recorded for mass balance evaluation. The samples were centrifuged at 4,000 rpm for 10 min to collect the supernatants which were filtered using a 0.45-µm PVDF membrane filter. The leaching tests were stopped at 300 min and the residual slurry was filtered using a 5-µm pore size filter paper. The filtrates were cooled to room temperature and their volumes were recorded. The filter cakes were dried in an oven at 60℃ for 12 hours and the solid residual dry weights were recorded. REE contents (REE_f, ppm) in the feed solids were back-calculated using the following equation:

$$REE_f = 100 \times (C_s \times m_s + C_l \times V_l) \times m_i/(m_i - m_m), \qquad [3.2]$$

where C_s and C_l represent REE concentrations (ppm) in the solid residual and final leachate; m_s (kg) the weight of the leaching solid residual; V_l (liter) the volume of the final leachate; m_i (kg) the weight of the leaching slurry; m_m (kg) the total weight of the representative samples collected during the leaching process. The difference between the back-calculated REE contents and the values measured directly using ICP was within ±5 percent, thereby indicating minor experimental errors. Leaching recovery (R, percent) were calculated using the following expression:

$$R = 100\% \times C_i/(REE_f \times 0.01), \qquad [3.3]$$

where C_i (ppm) represents REE concentration in the representative samples collected during the leaching process and REE_f (ppm) is the back-calculated REE contents in the feed solids. It is noted that all REE contents in the solids are reported on a dry whole sample basis.

3.2.3 Leachability Tests of the Rejects from the Pocahontas No. 3 Coal Seam

3.2.3.1 Unroasted Pocahontas No. 3 Coal Seam Channel Sample Segments (Roof, Coal, and Floor)

As part of the study and characterization of the Pocahontas No. 3coal deposit at the West Virginia coal mining and preparation plant operation, strata segments were obtained from an entry-high channel sample of a mined opening. Leaching tests performed on a carefully selected grouping of these stratigraphic segments (*see the following table*).

Table 3-58: Pocahontas No. 3 Coal Seam Channel Sample C-1

Petrographic Unit Thickness (ft.)	Geologist's Unit Description	Leaching Sample ID	Leaching Sample Thickness (ft.)
0.25	Shale, dk gray, hard; ~30' from C-1	C-1-roof	0.25
0.20	Bone with coal streaks	C-1-1	0.20
0.37	Clarain with vitrain layers	C-1-2	0.37
0.06	Coal with thick pyrite layer	C-1-3	0.27
0.21	Vitrain with fusain streaks		
0.20	Bone with coal streaks	C-1-4	0.80
0.15	Vitrain		
0.25	Bone		
0.20	Vitrain with bone streaks		
0.40	Bone with coal streaks	C-1-5	1.93
0.60	Clarain with vitrain streaks		
0.93	Vitrain with fusain streaks		
0.21	Bone	C-1-6	0.21
0.78	Vitrain	C-1-7	0.78
1.50	Siltstone, very hard; sampled only uppermost part	C-1-floor	1.50
		Total	**6.31**

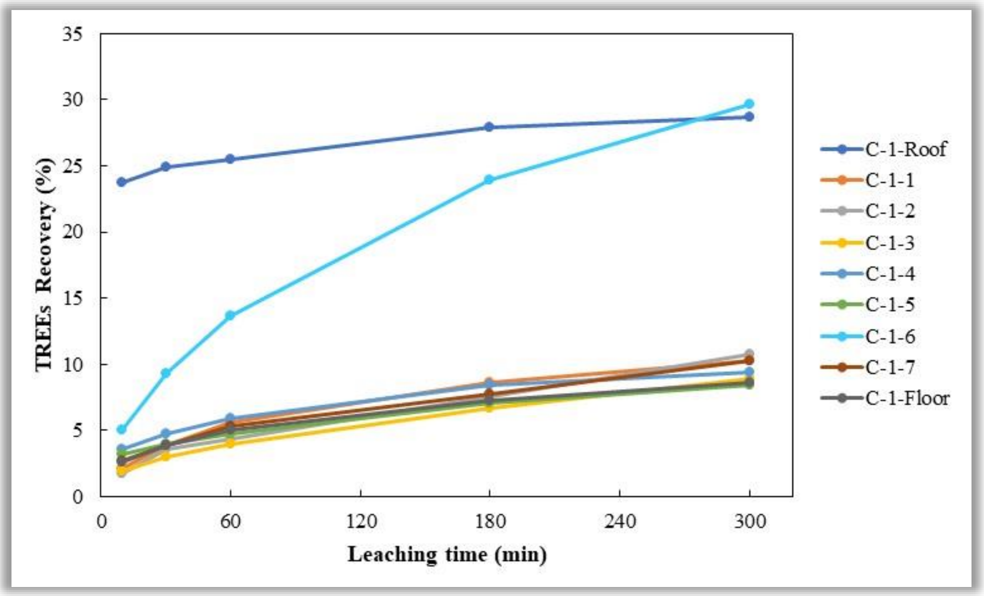
These nine groups of stratigraphic units were selected to gain insights into the leaching recovery of REEs from the Pocahontas No. 3 Seam prior to preparation plant activities. This is in contrast to leaching tests performed on the coarse rejects of the Pocahontas No. 3 Seam as those rejects-based results reflect an admixture.

The leachability tests of Pocahontas No. 3 Seam were conducted with 10 g/L solid in 1.2 mol/L sulfuric acid at 75 ℃ for 5 hours. There were 15 mL slurry samples collected at 10, 30, 60, 180, and 300 minutes, respectively. The leachate was then separated from the slurry by centrifugation and filtration. The 5-hour leaching solid residue was separated by filtration, washed with de-ionized water and then dried at 55 ℃.

As shown in the following figure the nine stratigraphic segments of the channel sample can be grouped into three different leaching responses.

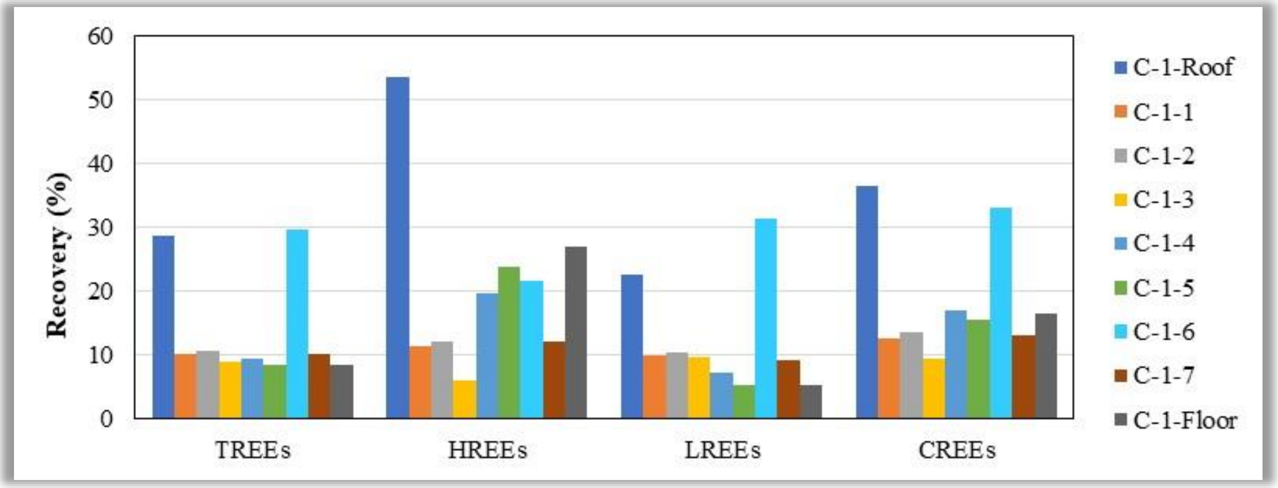
1. The first type is the leaching behavior of C-1-Roof, whose TREEs leaching recovery was 23.69 percent only after 10 min. Afterwards, leach recovery stabilized and only increased to 28.65 percent even after leaching for 5 hours.
2. The second type is represented by the leaching characteristics of C-1-6, whose TREE leaching recovery was only 5.03 percent after leaching for 10 minutes. However, TREE recovery significantly increased to 13.62 percent after leaching for 1 hour and to 23.95 percent after 3 hours. The 5-hour TREE leaching recovery of C-1-6 was 29.61 percent.
3. The remaining channel samples provided the third type of leaching behavior. Their TREE leaching recovery values were only about 2.0 percent to 3.5 percent after 10 min of leaching, while their 5 hours TREE leaching recovery values were also relatively low ranging from about 8.5 percent to 10.8 percent.

Figure 3-44: TREEs Leaching Recovery versus Leaching Time of Unroasted Pocahontas No. 3 Coal Seam Channel Sample Stratigraphic Segments



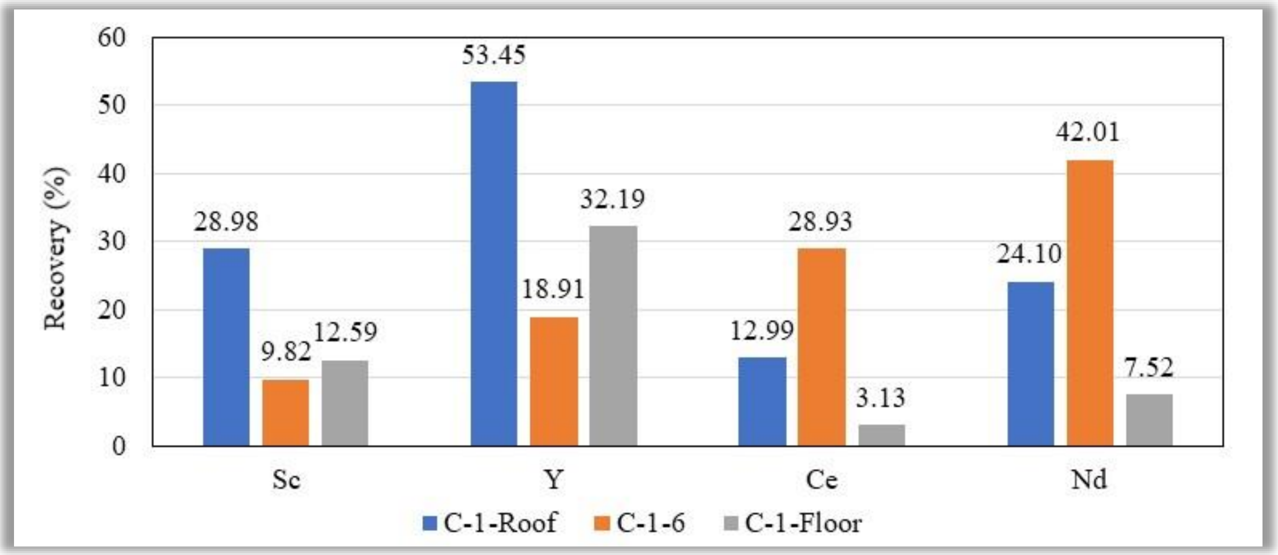
The leaching recovery of the TREEs, HREEs, LREEs and critical REEs *(CREEs)* recovery are important to evaluate the economic efficiency of leaching processing. The following figure provides the TREEs, HREEs, LREEs and CREEs recovery results of five (5) hours standard acid leaching on Pocahontas No. 3 Seam channel samples.

Figure 3-45: REEs Leaching Recovery of the West Virginia Pocahontas No. 3 Coal Seam Channel Samples after Standard Acid Leaching for Five (5) Hours



C-1-Roof has the highest heavy REEs leaching recovery, which was 53.49 percent. It also exhibits the highest critical REEs recovery of 36.51 percent. However, C-1-6 showed the highest TREEs recovery, which is due to LREEs recovery. In addition, C-1-Floor also showed high recoveries of HREEs and CREEs. Therefore, Sc, Y, Ce and Nd were selected as representative RE elements for analyzing the REEs leachability in C-1-Roof, C-1-6 and C-1-Floor, as showed in the following figure.

Figure 3-46: Typical REE recovery after Five (5) hours Acid Leaching of Selected West Virginia Pocahontas No 3 Coal Seam Channel Samples

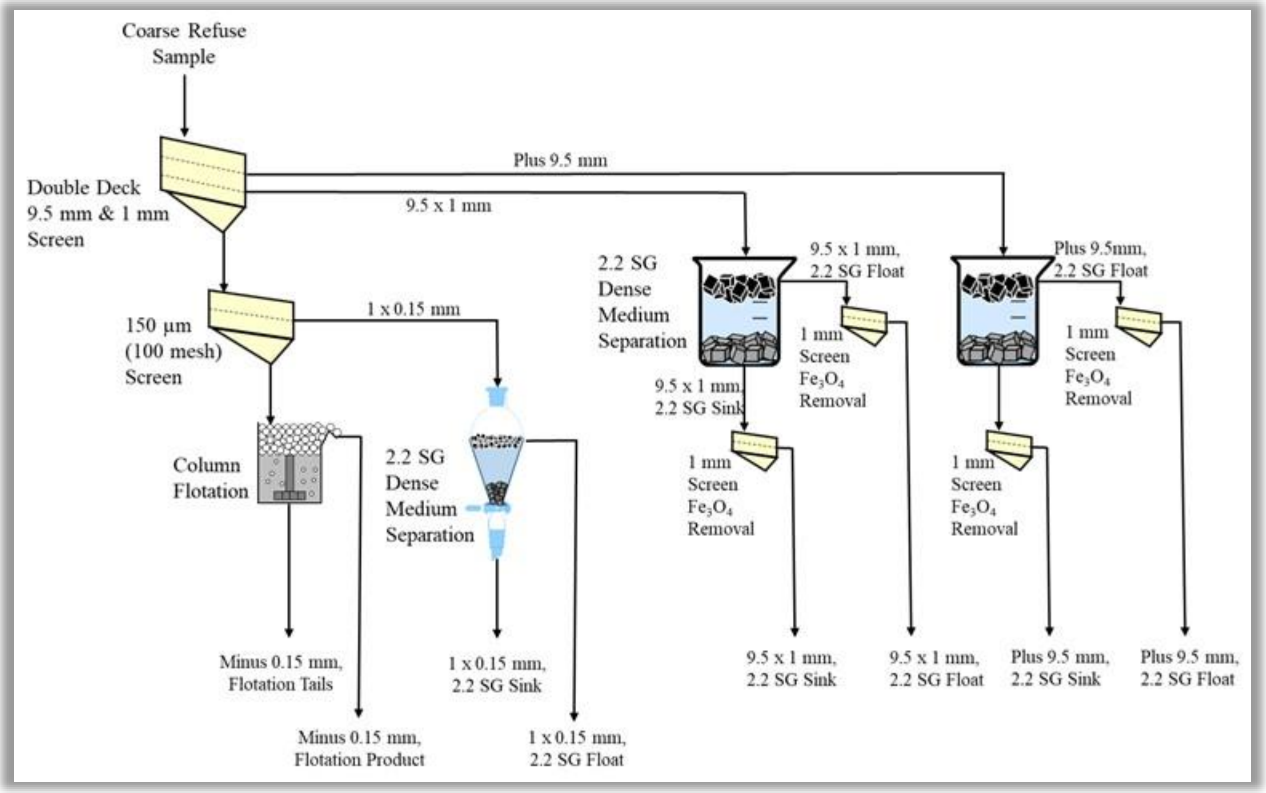


C-1-Roof had the highest leaching recoveries of Sc and Y, which was 29.98 percent and 53.45 percent, respectively. C-1-6 had the highest leaching recoveries of Ce and Nd, which were 28.93 percent and 42.01 percent, respectively. C-1-Roof contained relatively high Sc and Y concentrations while C-1-6 contained the highest Ce and Nd concentration. These results show favorable Sc, CREE and HREE extraction in the C-1-Roof material while C-1-6 has favorable CREE leaching characteristics.

3.2.3.2 Unroasted Pocahontas No. 3 Coal Seam Plant Coarse Rejects

Two 55-gallon drums of coarse rejects were collected from the sweep-belt sampler at the Pocahontas No. 3 Seam preparation plant. Upon delivery to the laboratory, the material was air dried and screened into size fractions of coarser than 9.5 mm, 9.5 x 1 mm, 1 x 0.15 mm and finer than 0.15 mm). The material coarser than 0.15 mm was density fractionated using a dense medium developed from a mixture of ultrafine magnetite (Fe_3O_4) and water. The medium was chosen over traditional organic and salt-based liquids to avoid any chemical effects on the solid material. The density fractions were washed to remove magnetite from the surfaces, dried and subsequently screened into different size fractions. Each particle density and size fraction were weighed, analyzed for ash and REE contents, and evaluated for leachability characteristics. A flowsheet of the sample preparation process is provided in the following figure.

Figure 3-47: Sample Characterization Flowsheet for Coarse Rejects Samples from the Pocahontas No. 3 Coal Seam Preparation Plant

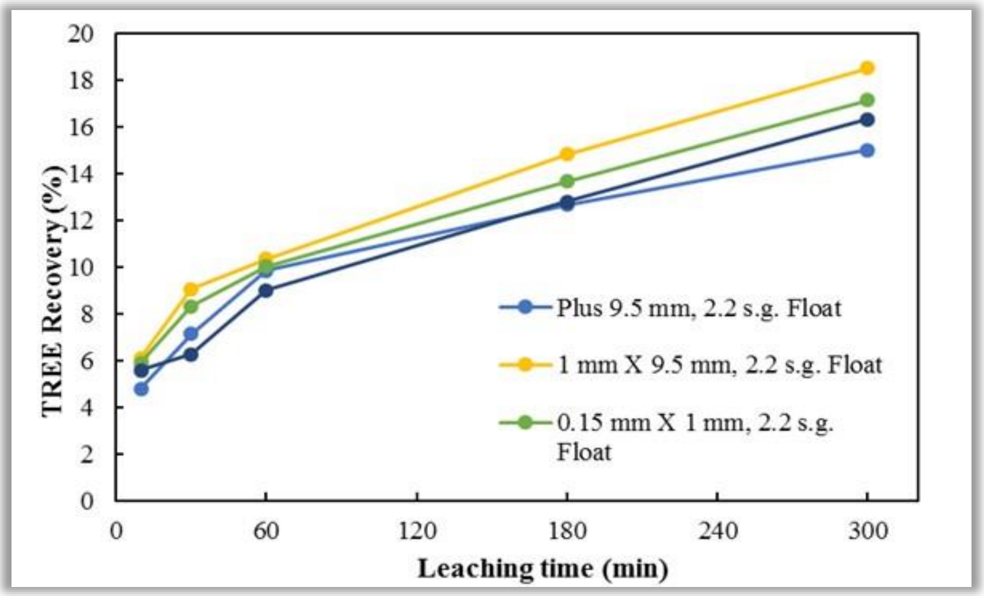


Leachability tests were used to determine the suitability of the material for REE recovery. The leaching tests were performed with 10 g/L solids in 1.2 mol/L sulfuric acid at 75℃ for 5 hours. The float-and-sink fractions were prepared according to the preceding figure. Each fraction was subjected to a leach test during which leachate samples were taken after 10, 30, 60, 180 and 300 min. Samples were 15 ml in volume. The leachate was then separated from the solids by centrifugation and filtration. The solid residue remaining after five (5) hours was separated by filtration, washing with water and then dried at 55℃.

The leaching kinetics of the Pocahontas No. 3 Coal Seam coarse rejects are shown the following figure for the 2.2 specific gravity *(SG)* float. The results show that TREE recovery increased sharply during the first 10 minutes of leaching followed by a near linear increase after the first hour. The 9.5 x 1mm S.G. 2.2 float fraction showed the best leachability with a TREE recovery of 18.50 percent, while the material finer than

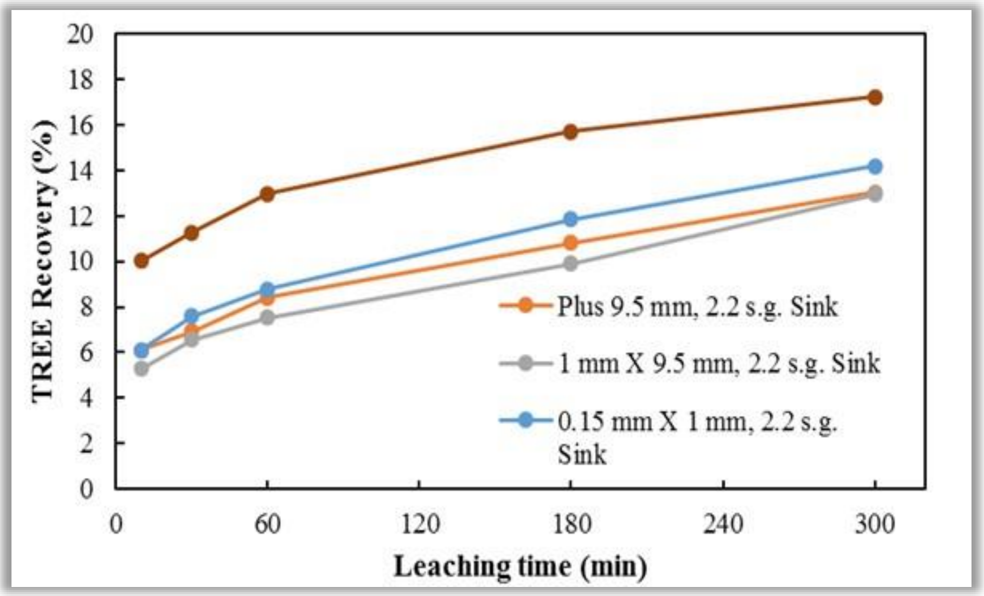
0.15 mm in the flotation tailings showed a relative high leaching recovery among sink products with a TREE recovery of 17.21 percent. All the various float sizes shared a similar recovery trend.

Figure 3-48: TREE Leaching Recovery versus Leaching Time of Float Products (Pocahontas No. 3 Coal Seam)



The results for the 2.2 SG sink fraction are shown in the next figure. This graph of the data shows somewhat different leaching characteristics from the float fraction. The 10-minute recover results were similar to the results obtained from the float fractions, but the incremental recovery gain with time was lower indicating slower leaching kinetics. The recovery results for the various sink sizes are similar except for the smallest particle size fraction which shows a significant difference of about 2 percent from the other size fractions.

Figure 3-49: TREE Leaching Recovery versus Leaching Time of Sink Products (Pocahontas No. 3 Coal Seam)



<u>3.2.3.2.1</u> *<u>Summary Observations of the Leaching Tests of the Unroasted Pocahontas No. 3 Coal Seam Coarse Rejects</u>*

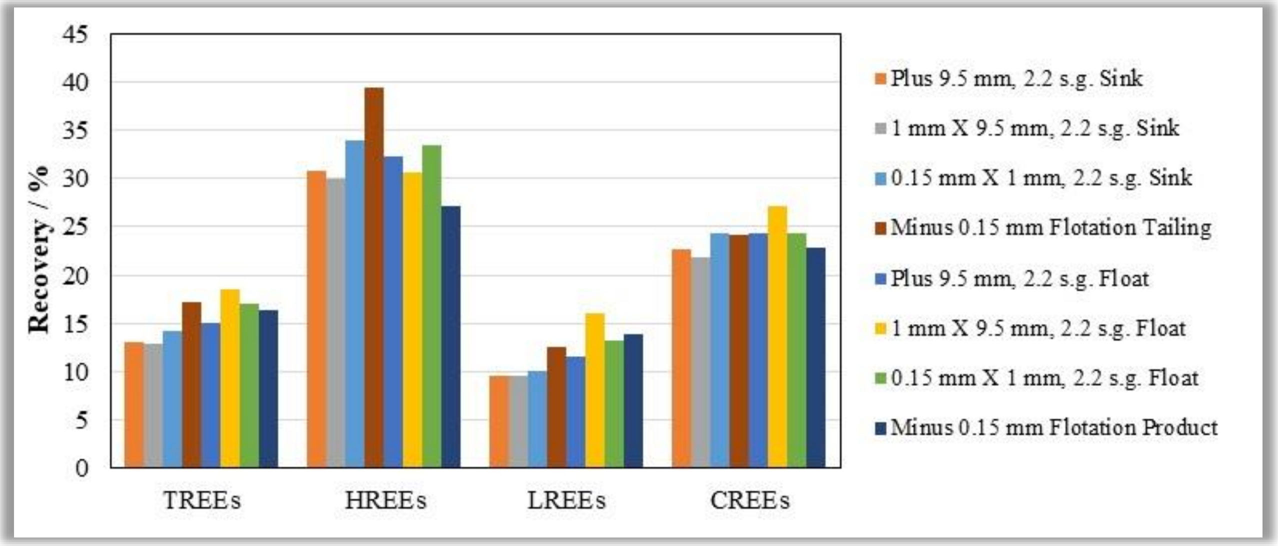
The leaching of the Pocahontas No. 3 Coal Seam preparation plant coarse rejects is characterized by three distinct stages, i.e.,

1. fast leaching kinetics within the first 10 minutes;
2. slower, more modest rates between 10 minutes and 1 hour; followed by
3. slow leaching rates after 1 hour.

The overall leaching performances on all fractions of the coarse rejects is likely below economically extractable limits.

To assist with understanding the underlying economics of the leachability of the unroasted Pocahontas No. 3 Coal Seam preparation plant coarse rejects samples, the recovery of TREEs, HREEs, LREEs and CREEs were calculated and presented in the next graph. This figure shows the overall leaching recovery values of the preceding two graphs after five (5) hours on the 8 float-sink products of the unroasted Pocahontas No. 3 Coal Seam preparation plant coarse rejects material. The results show selective recovery of the HREEs over the LREEs. The highest recovery of HREEs was found in the minus 0.15 mm flotation at 39.46 percent (*see the following figure for product stream references*). However, this same stream showed a low recovery of light REEs recovery of around 12.54 percent.

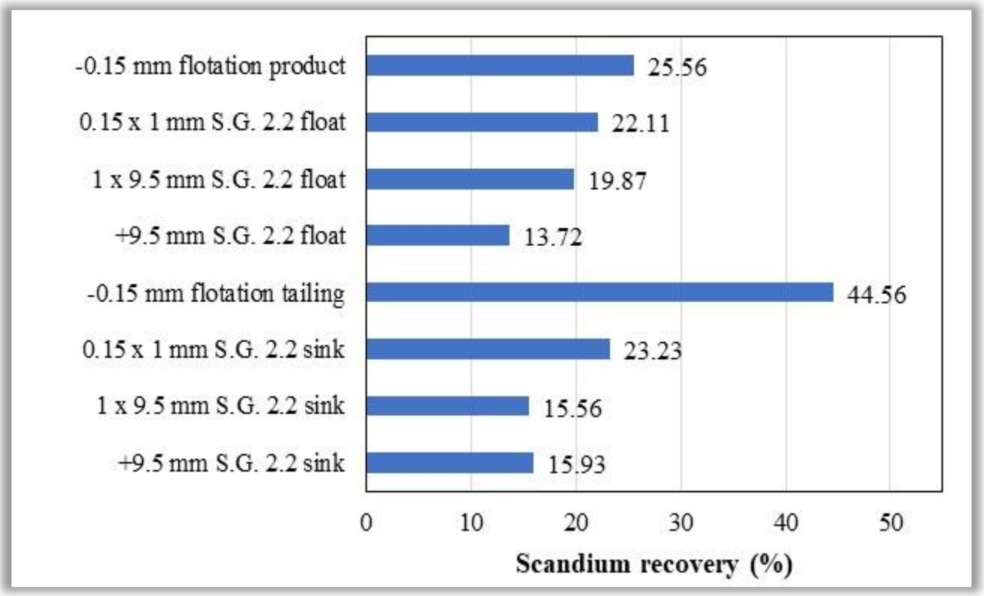
Figure 3-50: REEs Leaching Recovery of Sink-float Products after Standard Acid Leaching for Five (5) Hours (Pocahontas No. 3 Coal Seam)



To provide further insights into the leachability of the unroasted Pocahontas No. 3 Coal Seam preparation plant coarse rejects samples, the investigative team compiled the following figures to show the recovery of individual REEs (Sc, Y, Ce and Nd) after five (5) hours of acid leaching.

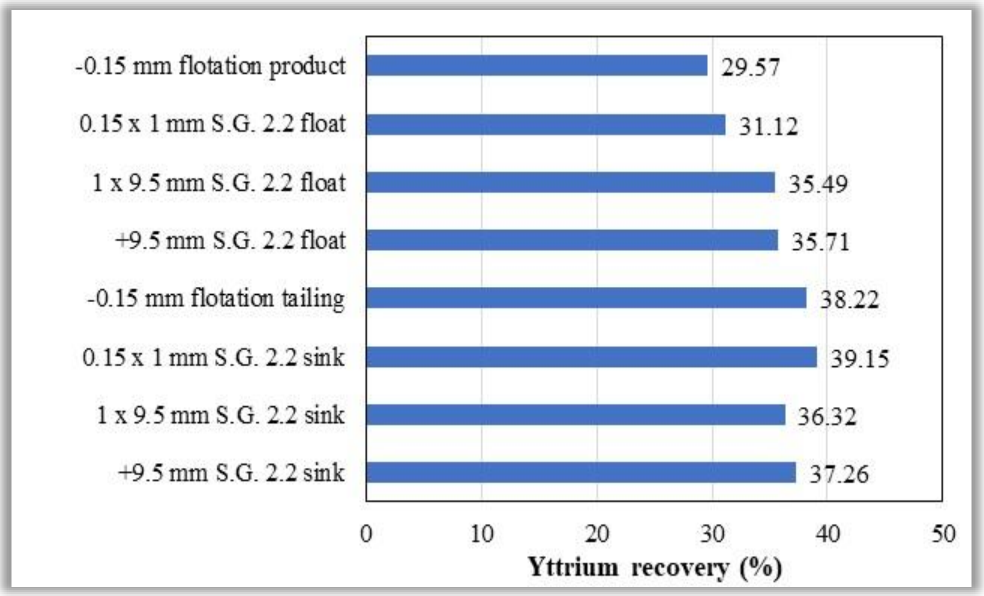
Due to the high market price of Sc, recovery is a significant economic driver. The figure below presents the Sc recovery after 5 hours of acid leaching. The overall trend shows improving recovery with decreasing particle size with the float fractions with slightly higher recovery values. For the float fraction, Sc recovery decreased from 25.56 percent to 13.72 percent with increasing size with the highest recovery value being obtained from the smallest froth flotation fraction. The sink products showed a similar trend. The highest Sc recovery (44.6 percent) was obtained from the material finer than 0.15 mm in the flotation tailings.

Figure 3-51: Sc Recovery after Five (5) hours Acid Leaching in Sink-float Products (Pocahontas No. 3 Coal Seam)



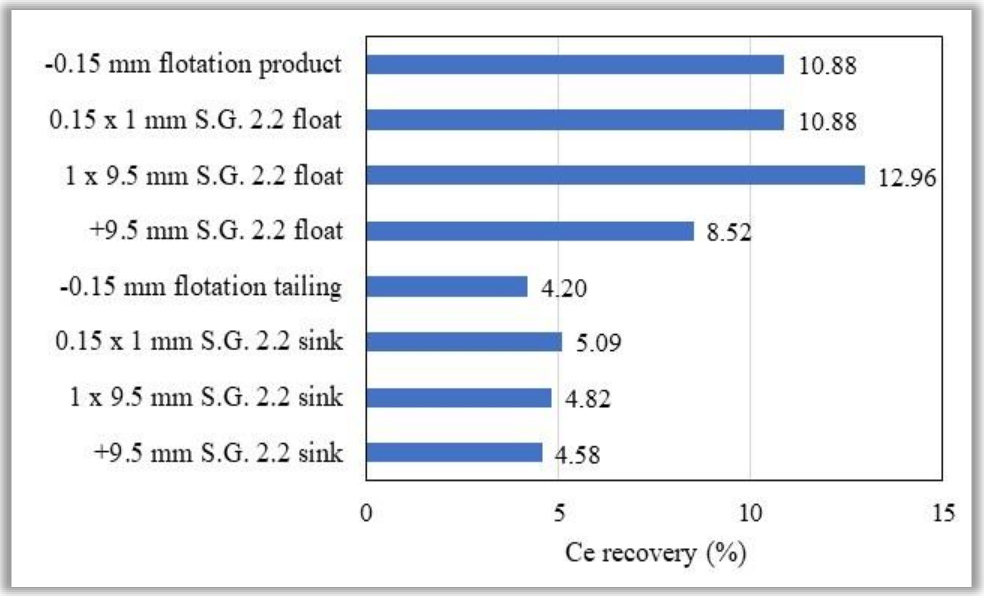
Yttrium *(Y)* recovery values are shown in the next figure. The Y leach recovery values appear similar to the overall HREEs recoveries. Whereas, Sc recovery of float products show an increase in recovery with a decreasing particle size, Y tends to increase recovery with increasing size for the float fractions or remain consistent with sink fractions. The recovery of Y tended to be between 30 and 40 percent overall.

Figure 3-52: Y Recovery after Five (5) Hours Acid Leaching in Sink-float Products (Pocahontas No. 3 Coal Seam)



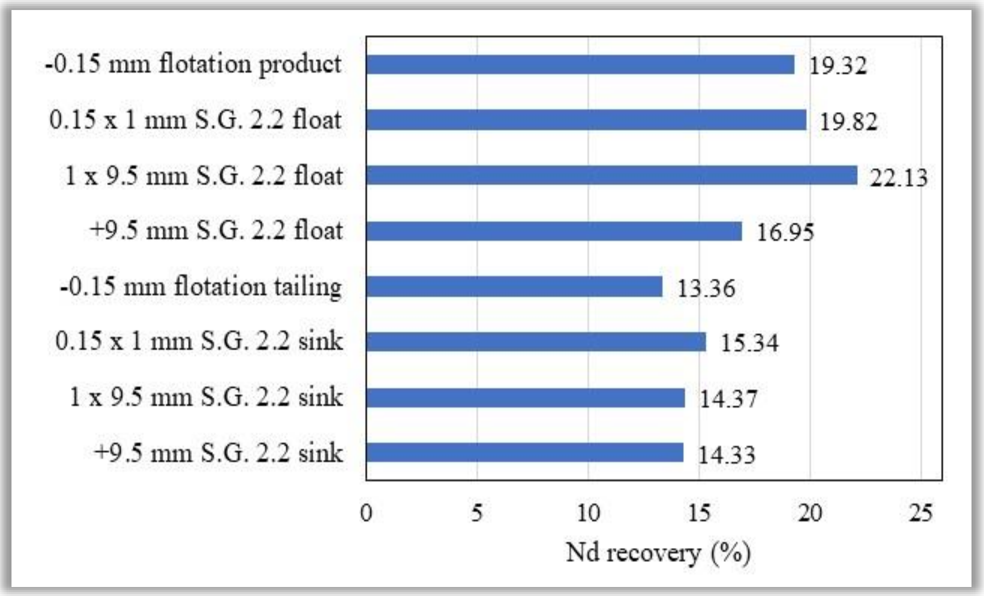
Cerium *(Ce)* was selected as one of the typical LREEs. The next figure provides the Ce recovery after five (5) hours acid leaching in the float-sink samples. An interesting finding is that the recovery values obtained from leaching the float products are almost two times than that of sink products. This result suggests differences in the mode of occurrence for Ce between the float and sink fractions with the float products leaching better.

Figure 3-53: Ce Recovery after 5 hours Acid Leaching in Sink-float Products (Pocahontas No. 3 Coal Seam)



Neodymium *(Nd)* was identified as a CREE and, as such, was included for individual reporting as shown in the following figure. Nd is often associated with Ce bearing minerals. As such, it was not surprising that Nd recovery showed similar trends to that of Ce (*see preceding figure*).

Figure 3-54: Nd Recovery after Five (5) hours Acid Leaching in Sink-float Products (Pocahontas No. 3 Coal Seam)



In summary, the overall recovery values trended between 5 percent and 40 percent depending on the sample and the element for the 5-hour leaching tests. These values were not encouraging based on the decarbonization work of the previous section. If the recovery of REEs through HHS or a similar process is assumed to be 70 percent, the overall recovery through leaching would fall to 3.5 percent and 28 percent after leaching.

3.2.3.3 Acid Leaching Response to Roasting Decarbonized Pocahontas No. 3 Coal Seam Preparation Plant Middlings Material

Preliminary leaching tests were performed on the Pocahontas No. 3 Coal Seam preparation plant middlings sample to assess the benefits of roasting on the leachability characteristics of that sample. The middlings product was selected to provide an additional data point linking the REE occurrence described in the channel samples to the products of the various coal preparation plant circuits. As discussed in previous sections, leaching unroasted samples resulted in low REE recovery values, with recovery values typical less than 50 percent. As a result, roasting of the samples prior to leaching was undertaken to improve REE recovery. In fact, this roasting process has been widely used to alter rare earth minerals crystals (e.g., monazite and xenotime) via thermal decomposition, to improve recovery.

3.2.3.3.1 Step 1: Comparison of REE Recovery from Unroasted and Roasted Decarbonized Middlings Samples Prior to Acid Leaching

In this step, REE recovery values from the Pocahontas No. 3 Seam middling are reported based on both roasted and non-roasted samples for comparison purposes. The samples were initially decarbonized using froth flotation. The samples were subsequently roasted by first mixing and then roasting in crucibles at 750°C for two hours in a muffle furnace. This process further decarbonized each sample and may have altered the rare earth mineral crystals. The consequence of this roasting process is revealed in the following two (2) tables. Those tables record the ash content and REEs composition on minus 0.18 mm and 10 micron Pocahontas No. 3 Seam middling decarbonize products. The tables also offer a comparison of the compositions of the original feed to the final roasted product. The values were mass-balanced, and cross checked back to the feed to check accuracy relative to experimental error. As expected, the roasting process increased the concentration of REEs, presumably as the result of removing carbon.

Table 3-59: Ash Content and TREE Content Results of Unroasted and Roasting Products of Minus (-)0.18 mm Pocahontas No. 3 Coal Seam Middling Decarbonized Flotation Tailings

Sample	Ash Content (%)	Yield (%)	TREE (ppm, whole basis)
Feed	86.73	-	358.22
Minus (-)0.18 mm roasted	99.72	86.10	413.63
Mass Balance Cross Check to Compared to Feed	85.86	-	356.14

Table 3-60: Ash Content and REEs Composition Results of Roasting Products from Minus (-)10 micron Pocahontas No. 3 Coal Seam Middling Decarbonized Flotation Tailings

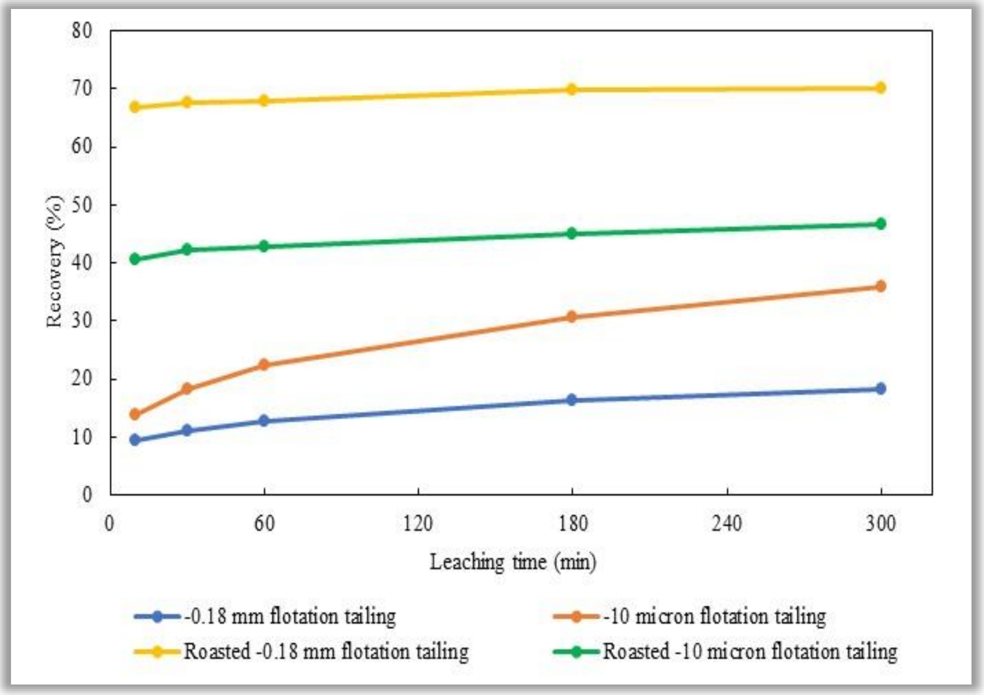
Sample	Ash Content (%)	Yield (%)	TREE (ppm, whole basis)
Feed	91.01	-	432.98
minus (-)10 micron roasted	99.82	90.03	480.87
Mass Balance Cross Check to Compared to Feed	89.87	-	432.93

3.2.3.3.2 Step 2: Comparison of TREE Recovery from Acid Leaching Unroasted and Roasted Pocahontas No. 3 Seam Middlings Samples

Leachability tests were conducted on different Pocahontas No. 3 Coal Seam decarbonized flotation tailings before and after roasting with 10 g/L of solids in 1.2 mol/L sulfuric acid at 75 °C for 5 hours. The 15 mL slurry samples were taken at 10, 30, 60, 180 and 300 min, respectively. The leachate was then separated from the slurry by centrifugation and filtration. The solid residue of the five (5) hours leaching process was separated by filtration, washed with water and then dried at 55°C.

As shown in the following figure, the four samples (i.e., two particle size fractions, roasted and unroasted) displayed different TREEs leaching kinetics. (Further insights into the mechanism of roasting will be covered in later sections.)

Figure 3-55: TREE Leaching Recovery versus Leaching Time of the Pocahontas No. 3 Coal Seam Unroasted and Roasted Decarbonize Flotation Tailings Samples

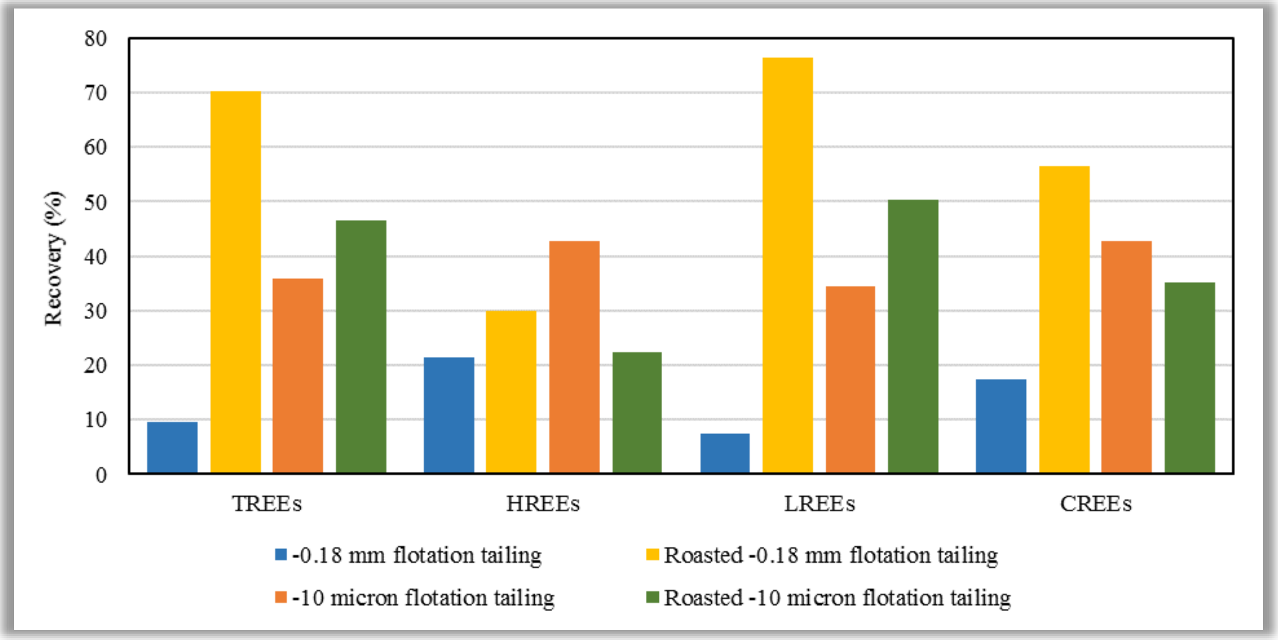


- \> This figure depicts a dramatic improvement in TREE recovery from 18.18 percent to 66.83 percent after the roasted minus (-)0.18 mm flotation tailing sample was leached for five (5) hours.
- \> The improvement in TREE recovery from the minus (-)10 microns material, is also significant, increasing from 35.78 percent (unroasted sample) to 46.52 percent (roasted sample) after five (5) hours of leaching.

3.2.3.4 Distribution of TREE, HREE, LREE and CREE Content in Acid Leachates from Unroasted and Roasted Decarbonized Samples of Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit

The following figure was used to evaluate the impact of leaching times on the types of REEs in acid leachates. This figure presents the recovery results of TREEs, HREEs, LREEs and CREEs after five hours of leaching. The feed samples comprised decarbonize flotation tailings and roasted samples of the middlings circuit of the Pocahontas No. 3 Coal Seam preparation plant.

Figure 3-56: REEs Leaching Recovery of Pocahontas No. 3 Coal Seam Decarbonized Flotation Tailings and Roasted Samples after Standard Acid Leaching for five (5) Hours



3.2.3.4.1 Results: Effects on TREE, HREE, LREE and CREE

The roasted minus 0.18 mm flotation tailing had the highest total, light and critical REEs leaching recoveries, which were 70.20 percent, 76.53 and 56.54 percent, respectively. However, the minus10 micron flotation tailing had the highest heavy REEs leaching recovery, which was 42.69 percent.

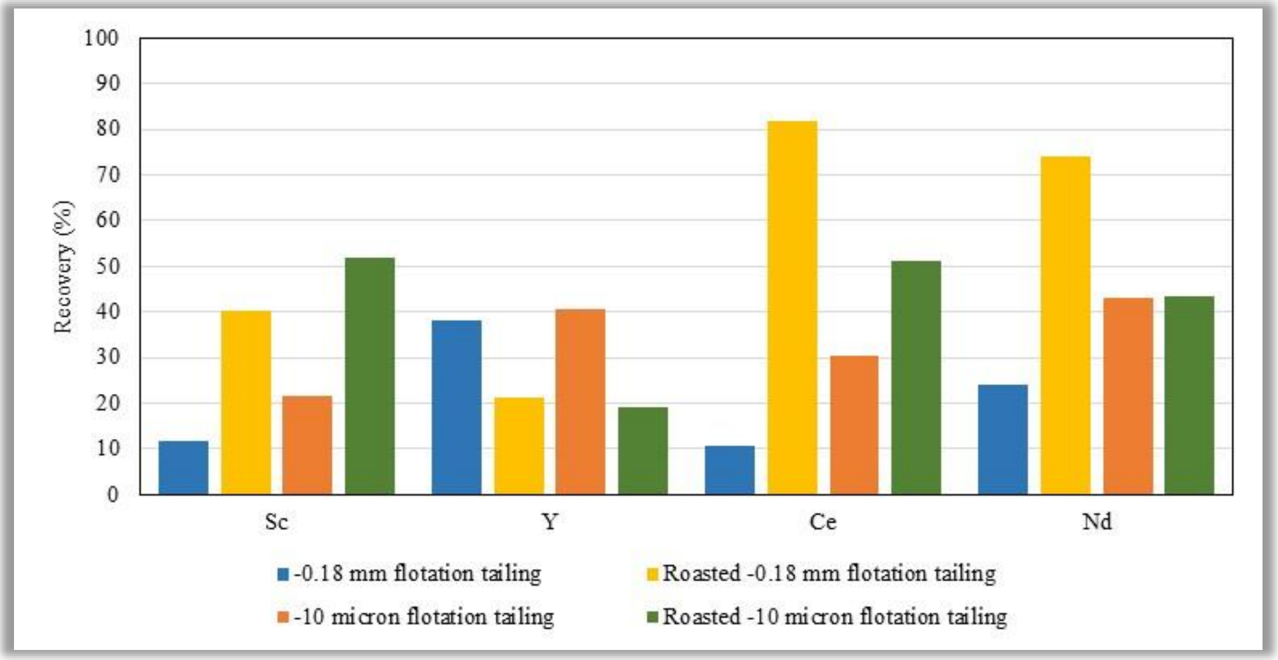
The heavy REE leaching recovery of the roasted minus (-)10 micron flotation tailing was reduced to 22.43 percent. In addition, the light REEs leaching recovery of minus (-)10 micron flotation tailing increased from 34.53 percent to 50.37 percent after roasting.

There results indicated that roasting preferentially improved the recovery of the light REEs and actually negatively impacted the recovery of heavy REEs. It is hypothesized that most of the light REEs existed in mineral form, which roasting modified to improve recovery. Further, it is suspected that heavy REEs have a greater tendency to adsorb on clays. This hypothesis is supported by the suppression of leach recovery after roasting since the roasting temperatures are known to initiate pozzolanic behavior with the clay particles. This behavior closes the internal pores and restricts access to the inner pore surfaces where heavy REEs may exist.

3.2.3.4.2 Results: Discussion on the Recovery of CREE

The following figure shows the leachability of flotation tailings size with and without roasting for critical groups of elements (*CREE*). Of note is the leaching behavior of Sc. The leaching recovery of Sc increased from 21.73 percent to 51.87 percent after roasting, which implied half of the Sc was associated with substances that can be improved by roasting. It is interesting to observe elements which improved with roasting (Sc, Ce) versus elements that did not (Y). Nd, as observed, either improved or showed no change depending on the particle size fraction.

Figure 3-57: Typical REE Recovery after Five (5) hours Acid Leaching in Selected Pocahontas No. 3 Coal Seam Decarbonized Flotation Tailings and Roasted Samples



3.2.3.5 Acid Leaching of Unroasted and Roasted Coarse Rejects and Middlings from the Pocahontas No. 3 Coal Seam Preparation Plant

A major purpose of this aspect of the project was to evaluate the improvement of REE recovery by thermal activation. Kinetic leaching tests were performed on both unroasted samples and samples that had been roasted at 400, 500, 600, and 750°C. Subsequent leaching tests were conducted under uniform conditions, i.e., 1.2 M acid solution, 1 percent solid concentration, and 75°C. The leaching behavior of the other major and trace elements were also investigated to obtain an improved understanding of the REE leaching mechanism and associated mineralogy.

3.2.3.5.1 Impact of Roasting on Proximate Analyses of the Coarse Rejects Sample Prior to Acid Leaching

Proximate analyses of the feed and roasted products are provided in the following table. Fixed carbon contents in the roasted samples decreased with an increase in roasting temperature and almost all the fixed carbon was removed after roasting at 500°C for two hours. Volatile contents were increased at 400°C, which was due to the preferential removal of fixed carbon.

Table 3-61: Dry volatile, Fixed Carbon and Ash Contents of the Unroasted and Roasted Pocahontas No 3 Coal Seam Coarse Rejects Samples (Two hours of Roasting)

Sample	Roasting	Volatile (%)	Fixed Carbon (%)	Ash (%)
Coarse Rejects	No	11.09	32.86	56.05
	400°C	14.63	14.28	71.09
	500°C	4.02	0.44	95.54

	600℃	1.41	0.30	98.29
	750℃	0	0.10	99.90

3.2.3.5.2 Acid Leaching Kinetics: Effects of Roasting Coarse Rejects and Middlings Circuit Samples from the Pocahontas No. 3 Coal Seam Preparation Plant

3.1.6.3.2.3 Effects of Roasting Temperature

Roasting at elevated temperatures causes complicated changes in solid materials. For coal-based samples, roasting will cause the loss of moisture, volatile, and fixed carbon, which makes the material become porous. Roasting also changes the mineralogy of the REEs which likely impacts leachability. In addition, mineralogy of the other dominant elements such as Al, Fe, Ca and Mg may also change significantly after roasting, which could negatively impact REE leaching. As such, a detailed experimental study was performed to evaluate and explain the roasting effect.

Kinetic leaching tests were performed on samples without and with roasting for comparison purposes. Four temperature values, i.e., 400, 500, 600 and 750℃, were used for roasting over a period of two hours.

1. Acid Leaching Kinetics: Effects of Roasting Coarse Rejects

Leaching recovery values of total REEs, LREEs and HREEs as a function of reaction time are shown in *Figure 3-58*. Several important trends can be observed:

a) Roasting enhanced REE recovery significantly. Total REE recovery were increased to 74 percent (1.2 M H_2SO_4)[12] and 81 percent (1.2 M HCl) by roasting at 600℃.

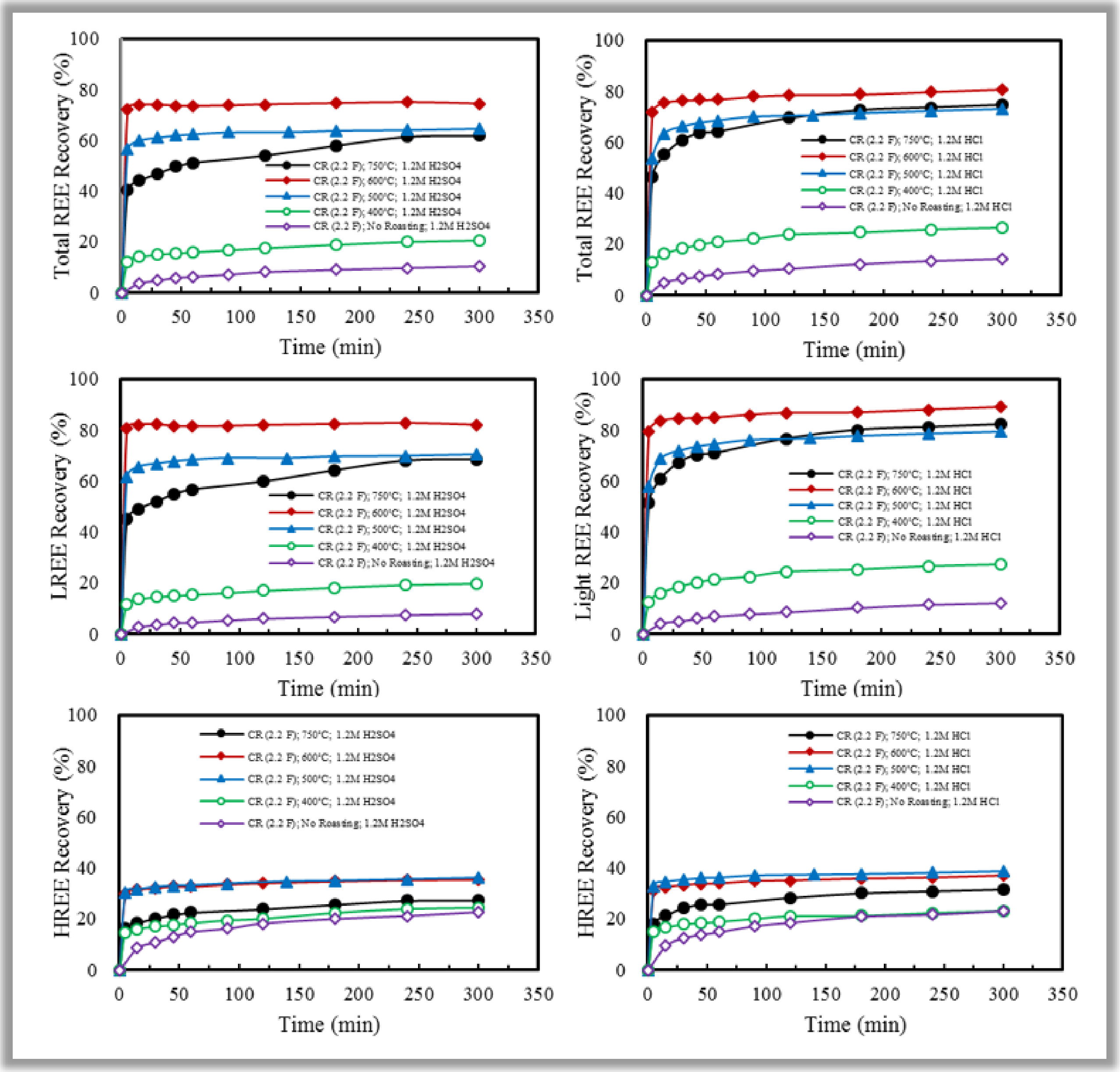
b) For LREEs, maximum recovery occurred at 600℃. The leaching characteristics of the material roasted at 750℃ was significantly different than material roasted at 500℃ and 600℃. During the first 30 minutes, the LREEs recovery increased significantly and stabilized for the 500℃ and 600℃ roasted material. For the 750℃ roasted material, a gradual increase in recovery was observed using either sulfuric acid or hydrochloric acid.

c) HREE recovery values at 500℃ and 600℃ were similar. After 300 min of interaction, 36 percent and 39 percent of HREEs were recovered using 1.2 M H_2SO_4 and 1.2 M HCl, respectively. Unlike LREEs, roasting at 400℃ did not improve HREEs recovery compared with the non-roasted material, while the leaching rate on the 400℃ roasted material was larger than the non-roasted material.

---

[12] Molarity (**M**) is mols/liter.

Figure 3-58: Effects of Roasting Temperature on REE Recovery from the Pocahontas No. 3 Coal Seam Preparation Plant Coarse Rejects (2.2 Float, (-) 80-mesh)



Two factors may explain the increase in recovery at 500°C and 600°C:

1. The liberation of REE host particles due to the removal of organic matter and,
2. REE minerals were transformed into more leachable forms during roasting.

The recovery increase between 400℃ to 500℃ is more likely due to the liberation of REEs from the organic matrix, while REEs minerals were transformed into more leachable forms between 500℃ and 600℃.

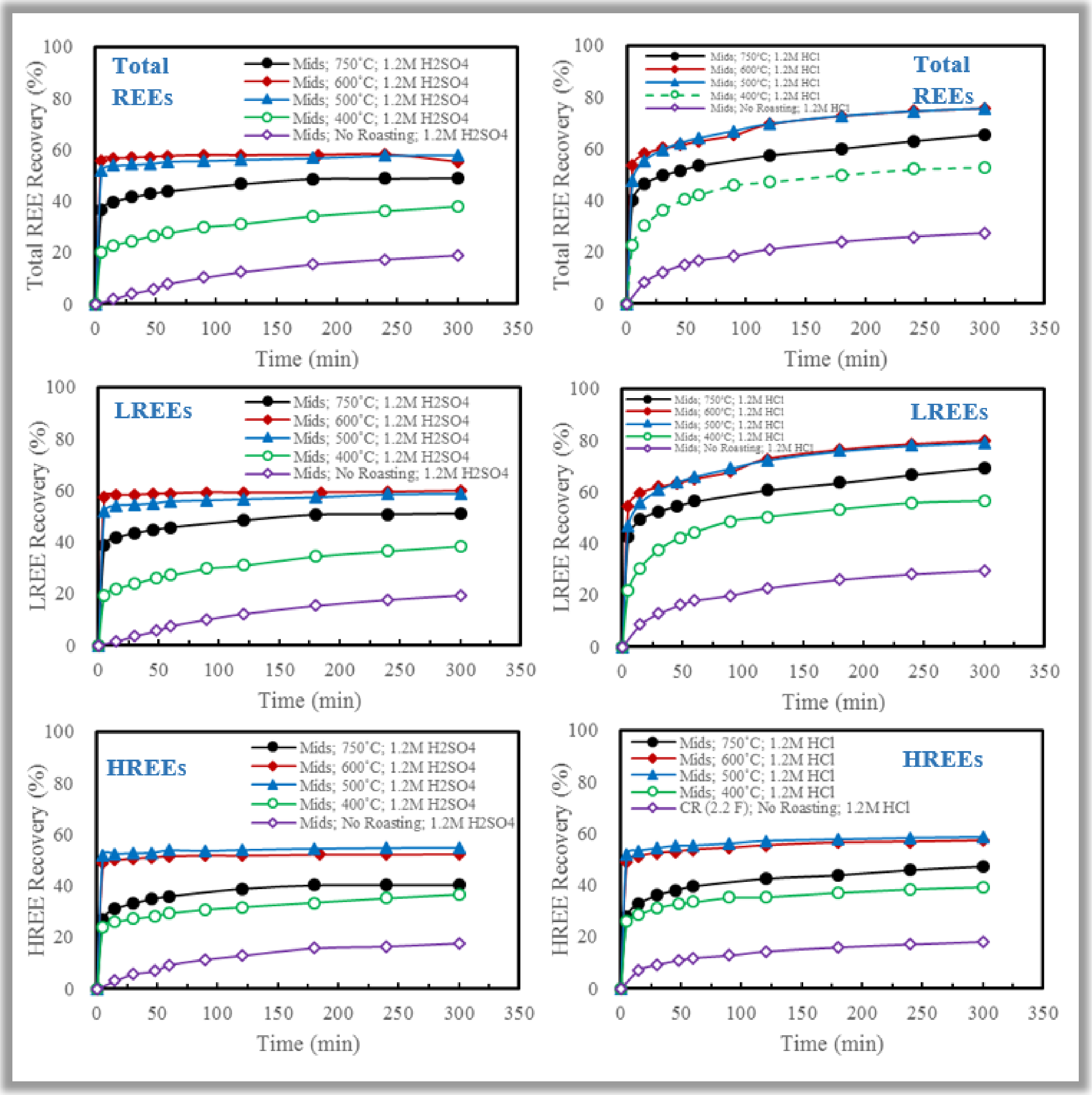
Significant mineralogical changes likely occurred at 750℃. For example, materials with a pozzolanic characteristics may be formed from clays at this high temperature, which may lock some REE particles and slow leaching kinetics.

It has to be noted that the HREEs recovery at 400℃ were similar to the non-roasted material, indicating that leachability of HREEs of the ash material liberated at 400℃ was similar to that of the original coal material.

3. Acid Leaching Kinetics: Effects of Roasting Middlings Circuit Product

The leaching behavior of both LREEs and HREEs in the roasted samples of the Pocahontas No. 3 Coal Seam preparation plant middlings were different from the coarse rejects. Instead of achieving maximum recovery at 600℃, the recovery of REEs from the middlings roasted at 500℃ were similar to the results obtained at 600℃ (*Figure 3-59*). This finding indicated that thermal activation effects did not play a significant role in the leaching process. In other words, liberation of the ash materials from the middlings circuit control the leaching behaviors of REEs. Unlike the coarse rejects materials, roasting at 400℃ significantly increased HREEs recovery which was most likely due to the release of a portion of the REEs after combustion of the organic matter.

Figure 3-59: Effects of Roasting Temperature on REE Recovery from the Pocahontas No. 3 Coal Seam Preparation Plant Middlings Product



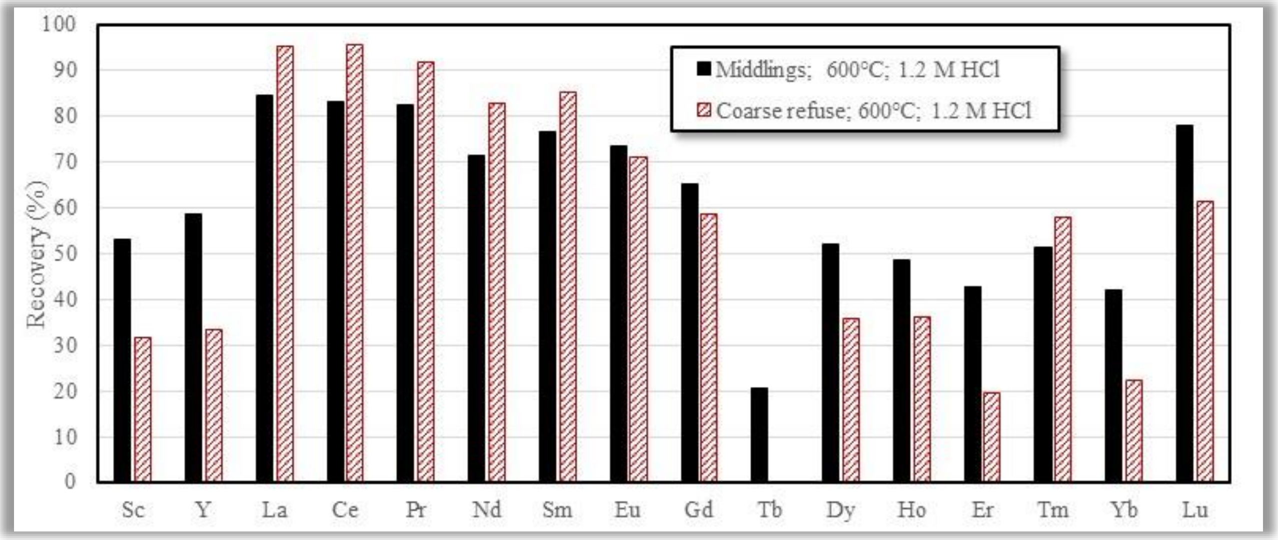
<u>3.2.3.5.3</u> *<u>Effects of Inorganic Acid Type on REE Recovery from the Roasted Pocahontas No. 3 Coal Seam Preparation Plant Coarse Rejects and Middlings Circuit Product Samples</u>*

REE recovery from the middlings samples also depended heavily on which type of inorganic acids was utilized, which also differed from the findings obtained from the coarse reject tests. Recovery increases of 20 and 5 absolute percentage points were obtained when using 1.2 M HCl compared with 1.2 M H_2SO_4 for the 500°C and 600°C roasted samples.

3.2.3.5.4 REE Recovery from Coarse Rejects and Middlings Circuit Samples at Optimum Temperature and Acid Type Conditions

The recovery of individual REEs from the coarse rejects and middlings samples under optimum roasting conditions (600℃) and inorganic acid type is shown in the next figure.

Figure 3-60: Recovery of Individual REEs from the Coarse Rejects and Middlings Circuit Samples Which Were roasted at 600℃ for Two (2) Hours



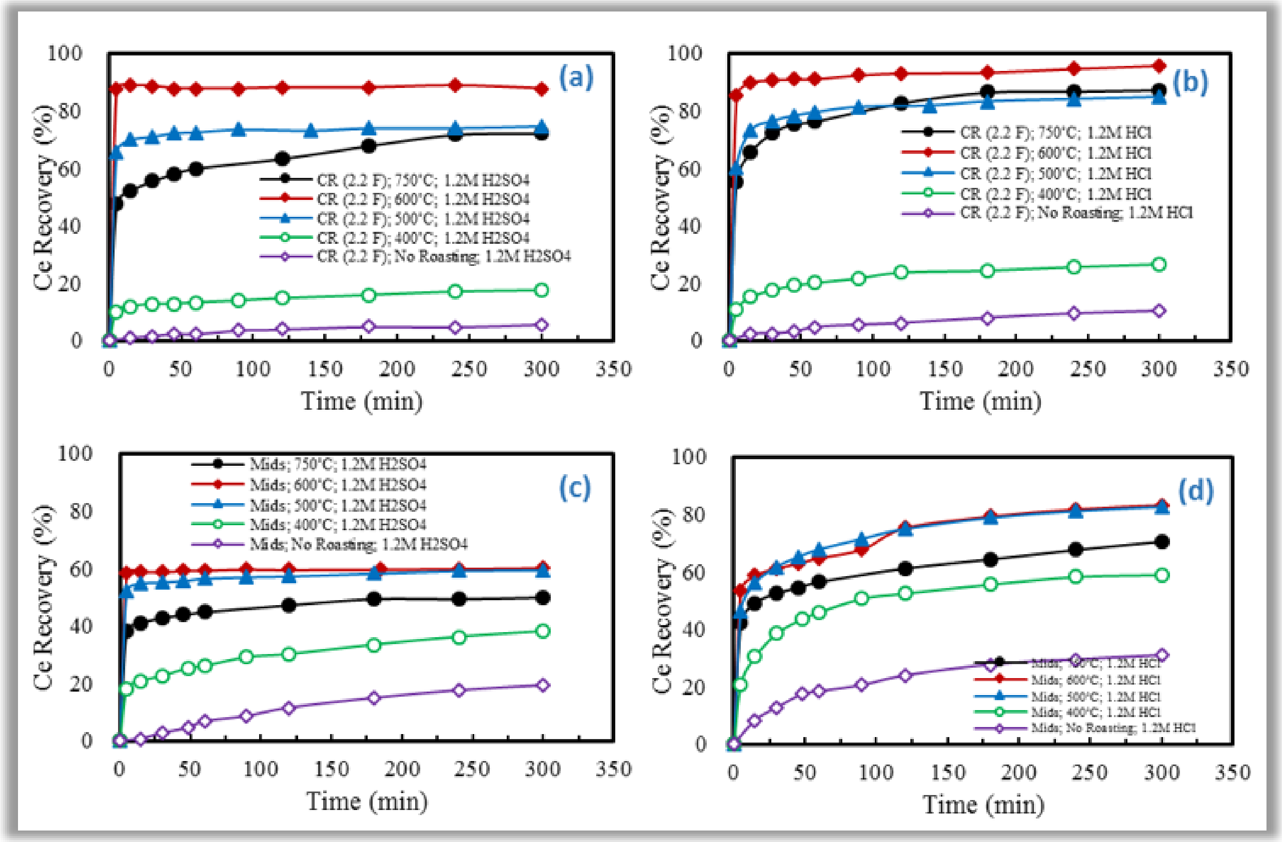
Given those results, the middlings material appears to be a more attractive feed material for REE recovery due to its enhanced HREE leachability (i.e., Sc, Y, Eu, Gb, Dy, Ho, Er, Yb, Lu).

3.2.3.5.5 Effect of Roasting Temperature, Acid Type, and Leach Tank Temperature on Ce, Y, Dy and Sc Recovery from the Pocahontas No. 3 Coal Seam Preparation Plant Middlings and the 2.2 Specific Gravity (SG) Float Fraction of its Coarse Rejects

The leaching behavior of four individual REEs, i.e., Ce, Y, Dy and Sc were evaluated individually to assist with a systematic analysis of the REE recovery methods.

As shown in *Figure 3-61*, more than 95 percent of the Ce was recovered from the 600℃ roasted sample of the coarse rejects, while 83 percent was recovered from the middlings roasted sample under the same condition.

Figure 3-61: Effect of Roasting Temperature and Acid-Type on Ce Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Float of Coarse Rejects, (a) and (b) at 75℃; and the Middlings (c) and (d) at 25℃



The leaching behaviors of Y and Dy are shown in *Figure 3-62* and *Figure 3-63*, respectively. Roasting at 400℃ provided a significant positive effect on HREEs recovery for the middlings sample while the effect on the coarse rejects was minor.

Figure 3-62: Effect of Roasting Temperature and Acid-Type on Y Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Float of Coarse Rejects, (a) and (b) at 75℃; and the Middlings (c) and (d) at 25℃

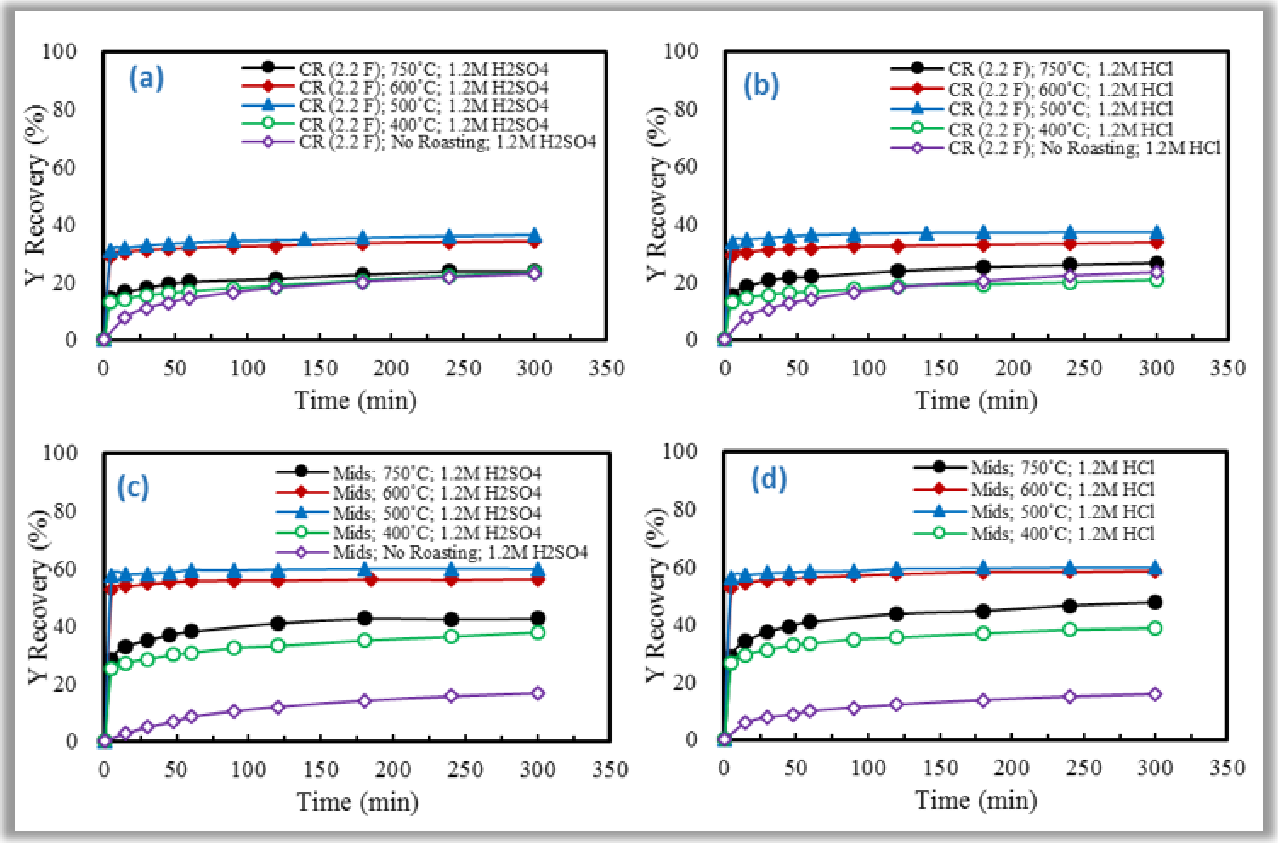
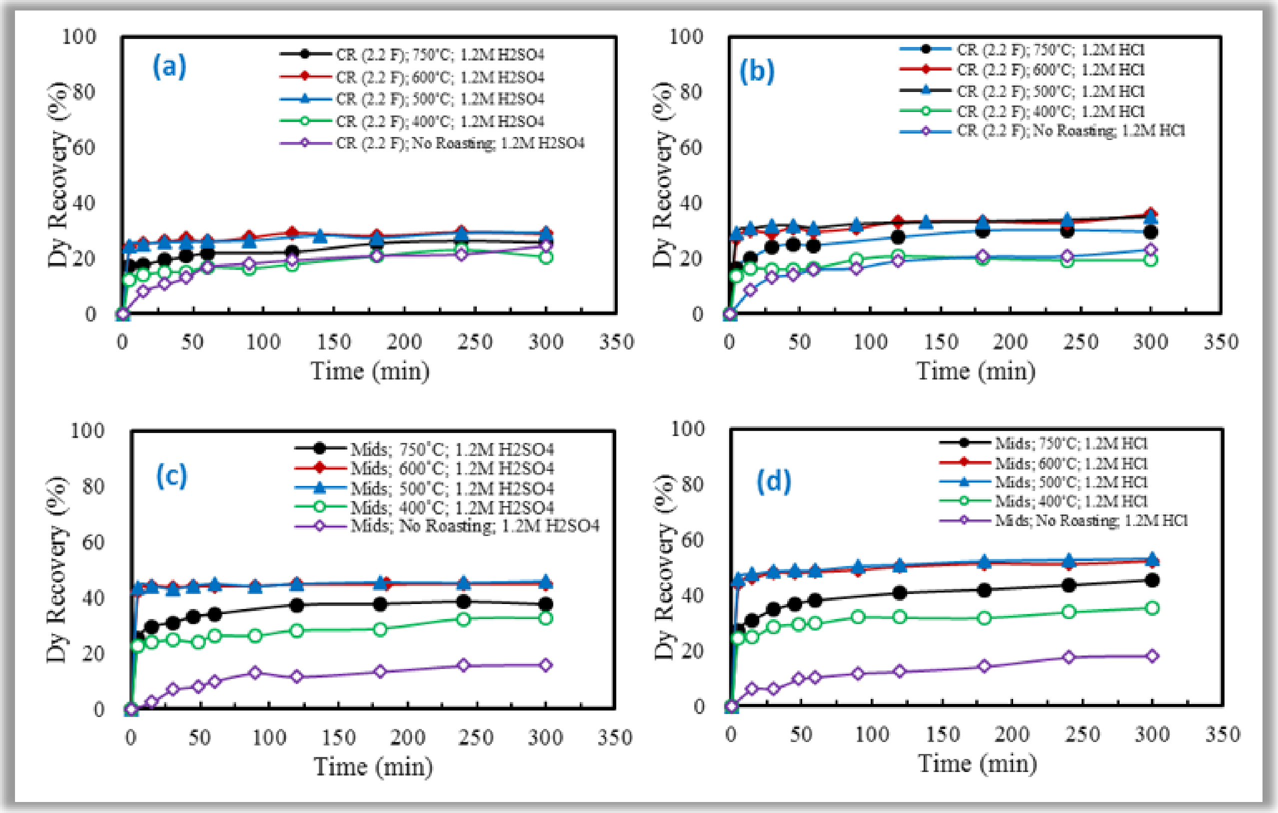
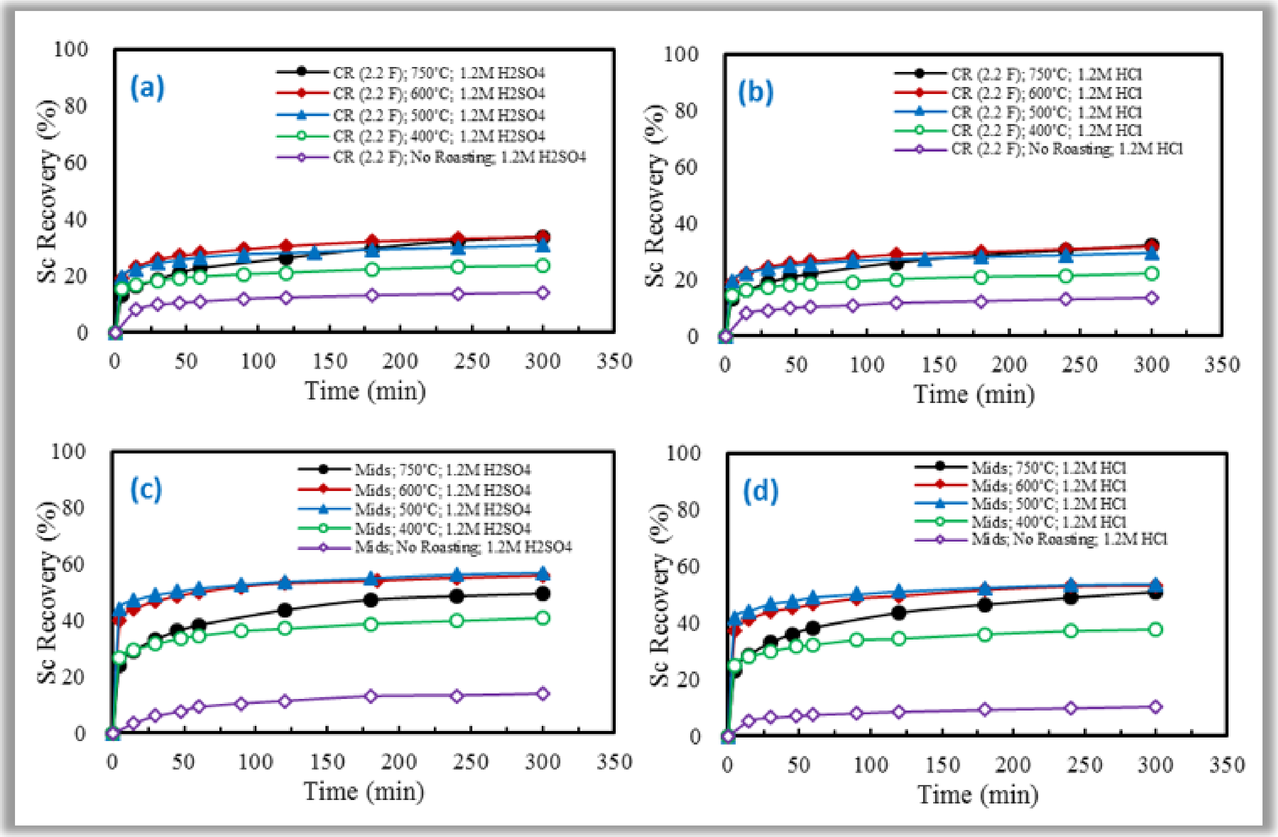


Figure 3-63: Effect of Roasting Temperature and Acid-Type on Dy Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Float of Coarse Rejects, (a) and (b) at 75℃; and the Middlings (c) and (d) at 25℃



Scandium *(Sc)* showed different leaching behavior relative to other REEs (*see the next figure*). Roasting at 400℃ increased its recovery for both the coarse rejects and middlings samples, indicating that the ash materials liberated from the organic matrix have higher Sc recovery than the original coal samples. Furthermore, thermal activation has minor impacts on Sc recovery for both the coarse rejects and middlings samples since the leaching recovery at 500℃ and 600℃ are nearly identical. Finally, using HCl did not provide enhanced Sc recovery of the coarse rejects sample or middlings samples.

Figure 3-64: Effect of Roasting Temperature and Acid-Type on Sc Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Float of Coarse Rejects, (a) and (b) at 75°C; and the Middlings (c) and (d) at 25°C



<u>*3.2.3.5.6 Summary Observations of Acid Leaching Behavior of Unroasted and Roasted Samples*</u>

It is difficult to understand the leaching behaviors of HREEs based on the test data obtained to-date. Roasting certainly increased the HREEs recovery. However, HREE recovery was relatively lower (i.e., about 40 percent for the coarse rejects and 60 percent for the middlings when roasted at 600℃ for 2 hours). The recovery curves were similar to LREEs, i.e., a quick release followed by a gradual decline in the reach rate as a constant recovery value is obtained with time. The recovery of other major elements gradually increased with an increase in reaction time most likely due to gradual dissolution of the solids. This trend implies the possibility that HREEs occurred as dispersed elements in the samples (e.g., ion-substitution in clays).

3.2.3.6 Acid Leaching Behaviors of other Major Elements and Trace Elements in Unroasted and Roasted Samples of the Pocahontas No. 3 Coal Seam Preparation Plant

Acid leaching behaviors of the other major elements and trace elements, including Al, Fe, Ca, Mg, Co, Cu, Zn, and V, in the original and roasted coal samples were also investigated for a better understanding of REEs leaching characteristics and mineralogy (*see the following table and associated nine [9] figures*).

Table 3-62: Contents of Al, Fe, Ca, Mg (%) and Trace Elements (ppm) in the Pocahontas No. 3 Coal Seam Preparation Plant Samples Roasted at 600℃ for Two (2) Hours

Sample	Al	Fe	Ca	Mg	Cu	Zn	Co	V	P
Roasted Middlings	13.5	2.5	0.31	0.35	173	76	33	194	398
Roasted Coarse Rejects	14.3	2.8	0.28	0.72	113	91	26	190	368

Figure 3-65: Effect of Roasting Temperature and Acid-Type on Al Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product

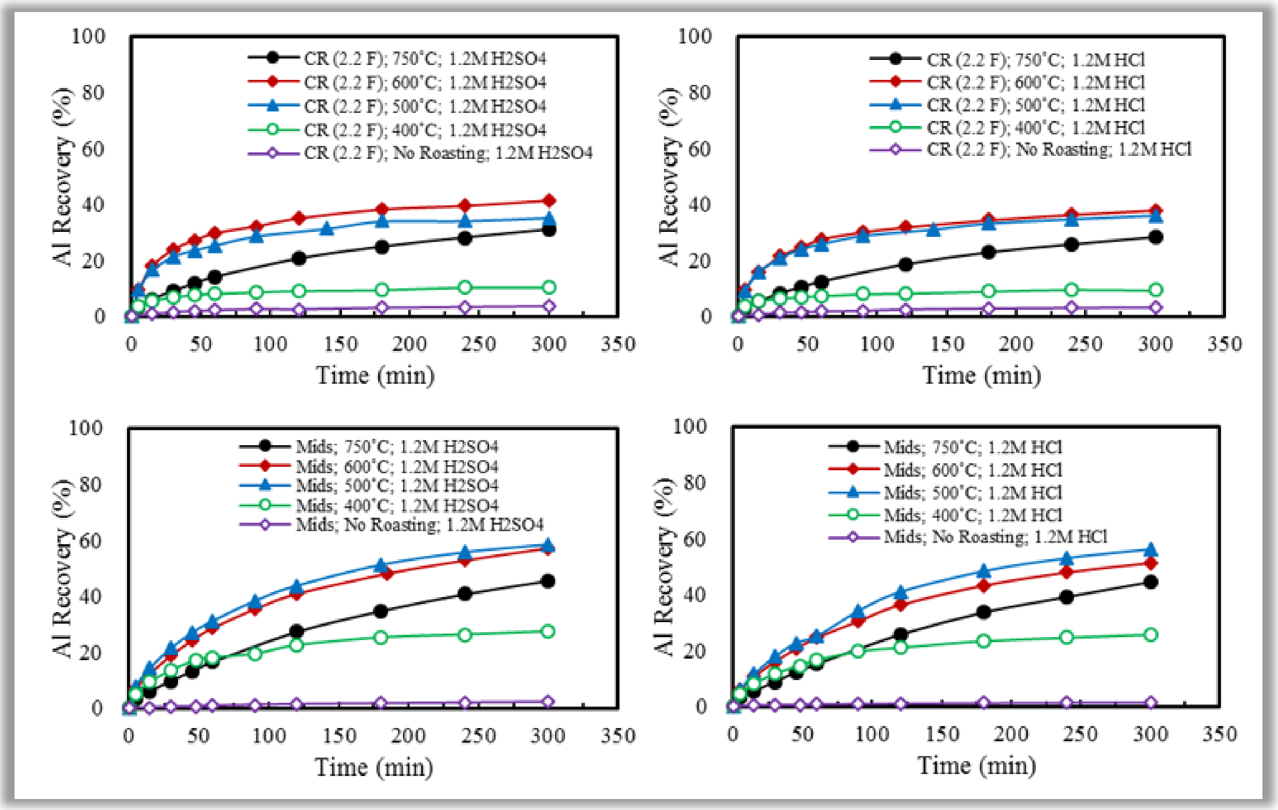


Figure 3-66: Effect of Roasting Temperature and Acid-Type on Fe Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product

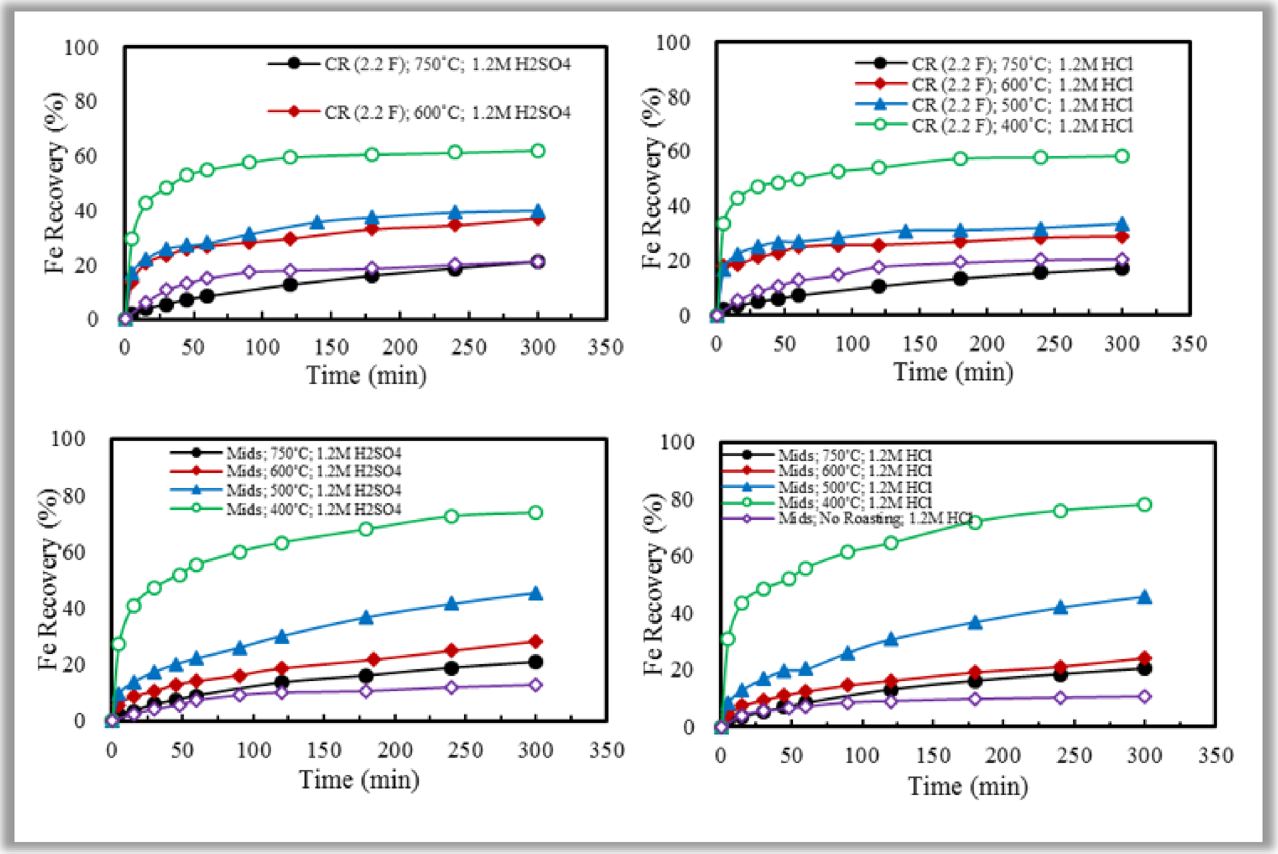


Figure 3-67: Effect of Roasting Temperature and Acid-Type on Ca Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product

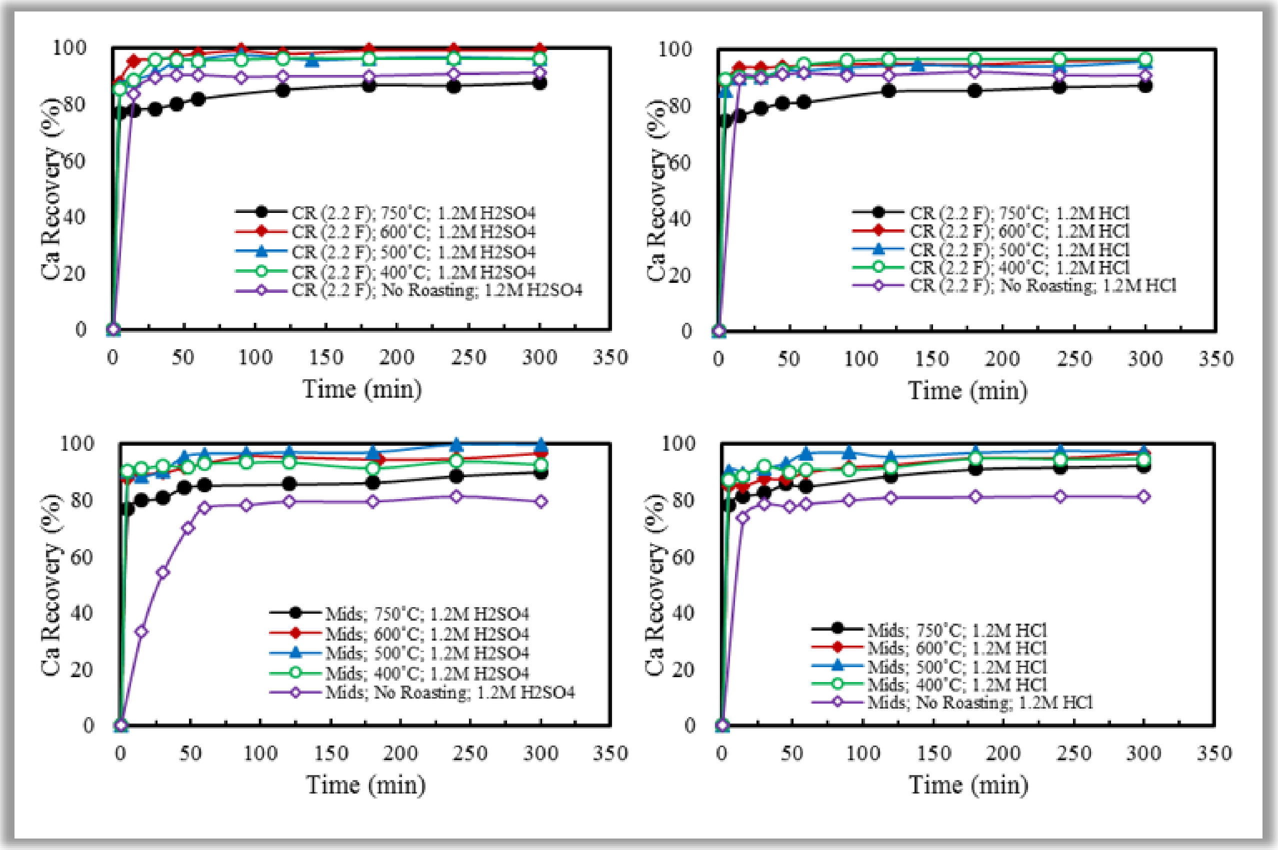


Figure 3-68: Effect of Roasting Temperature and Acid-Type on Mg Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product

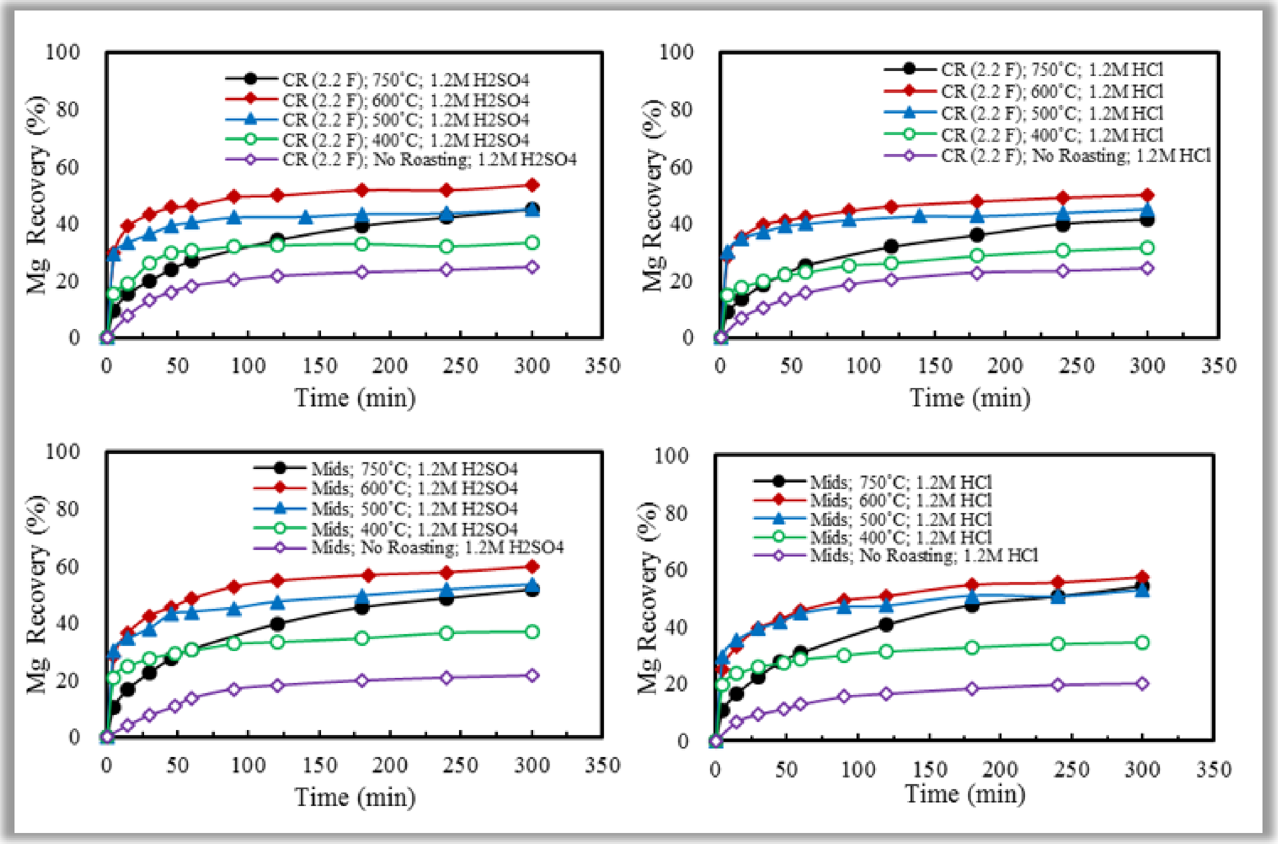


Figure 3-69: Effect of Roasting Temperature and Acid-Type on Cu Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG. Coarse Rejects Float Fraction and the Middlings Product

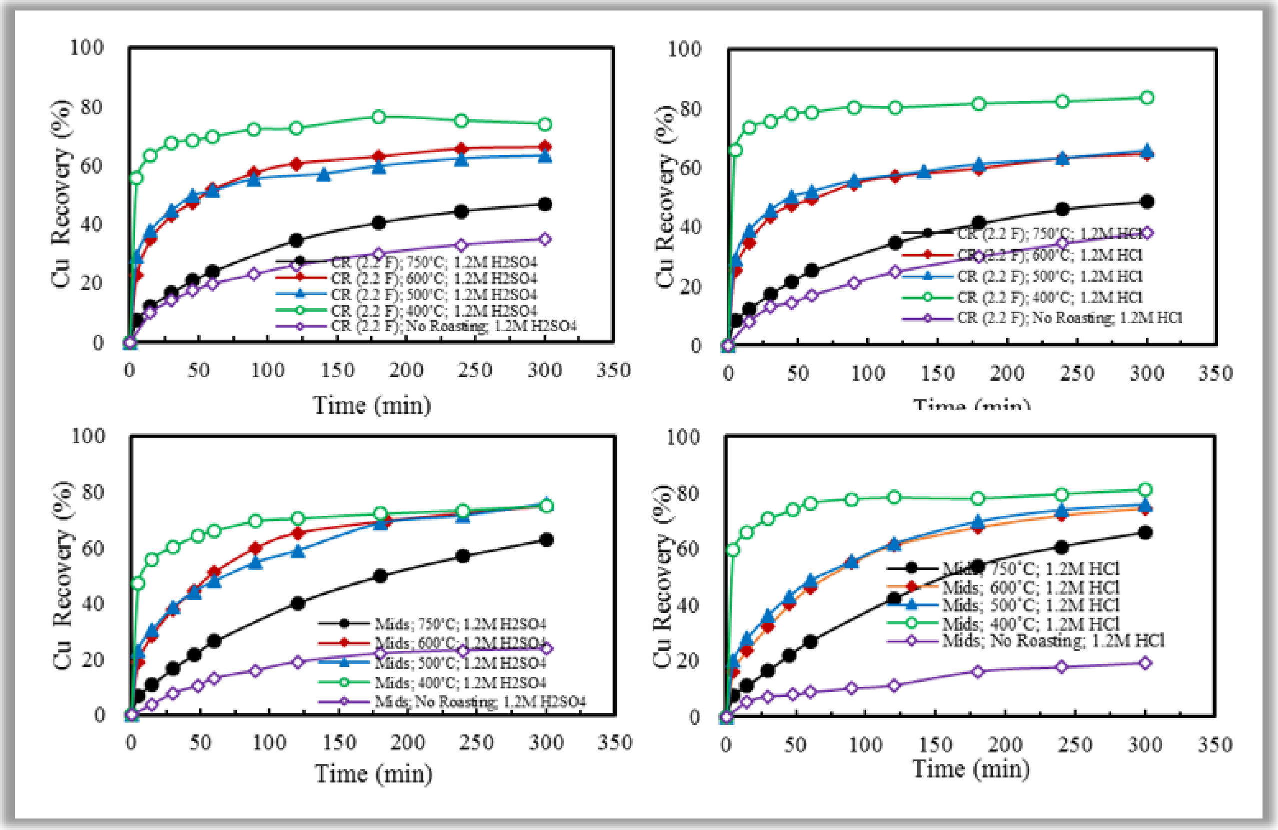


Figure 3-70: Effect of Roasting Temperature and Acid-Type on Co Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product

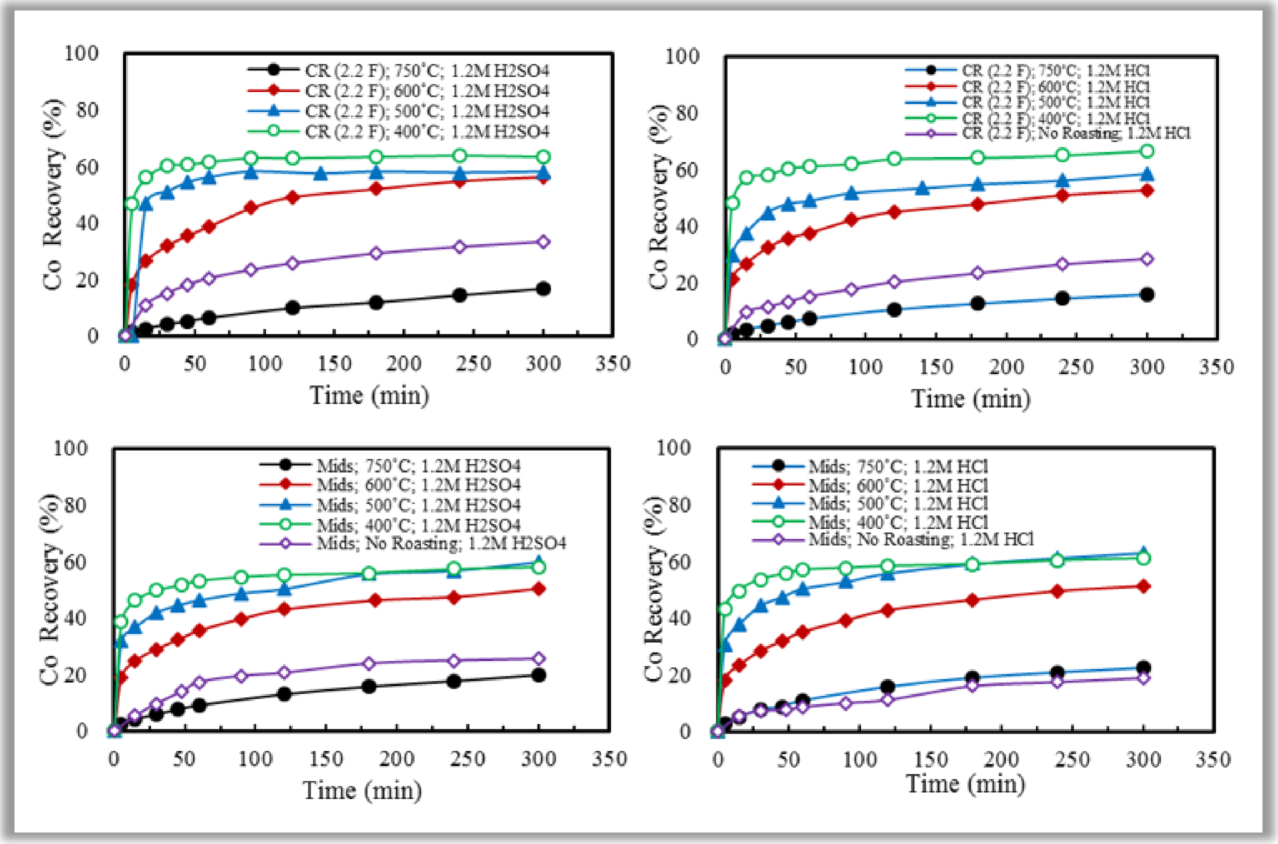


Figure 3-71: Effect of Roasting Temperature and Acid-Type on Zn Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product

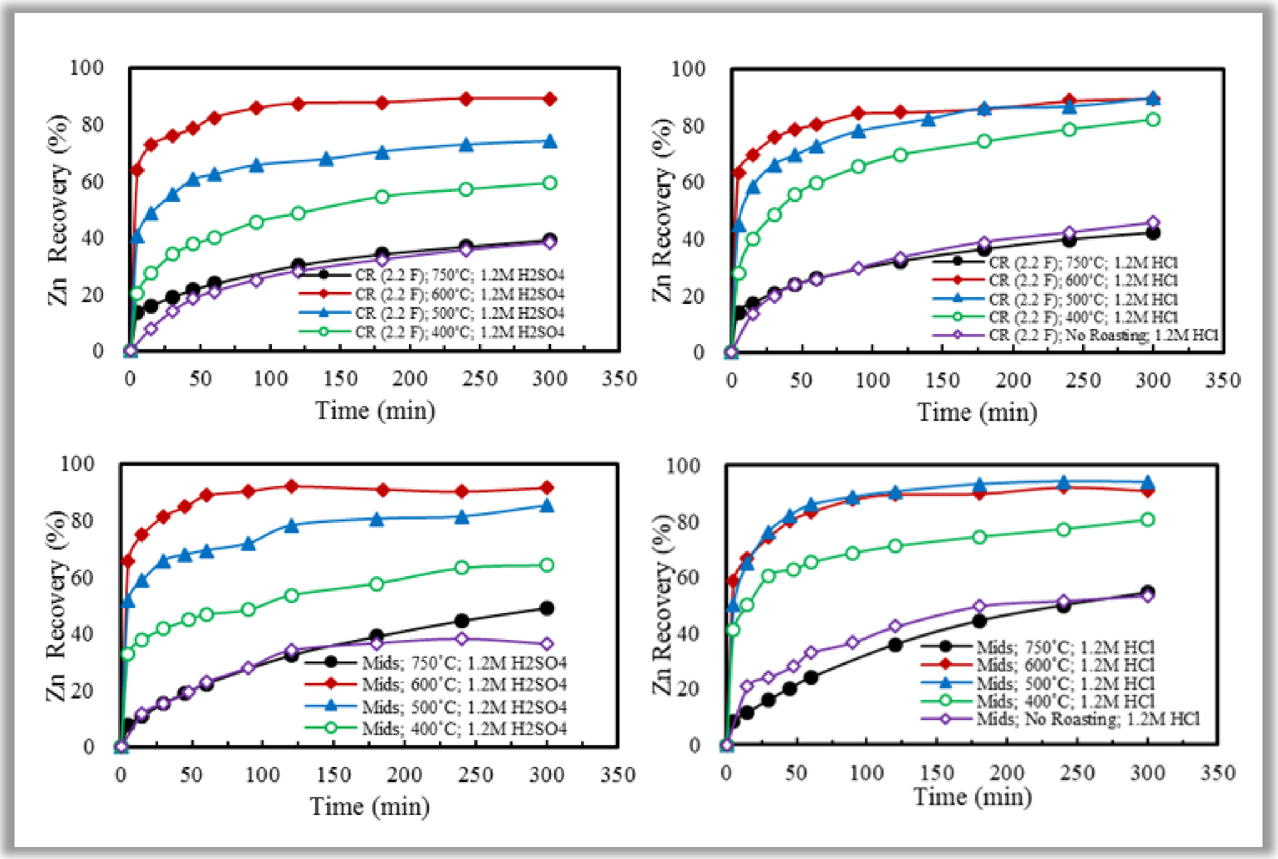


Figure 3-72: Effect of Roasting Temperature and Acid-Type on V Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product

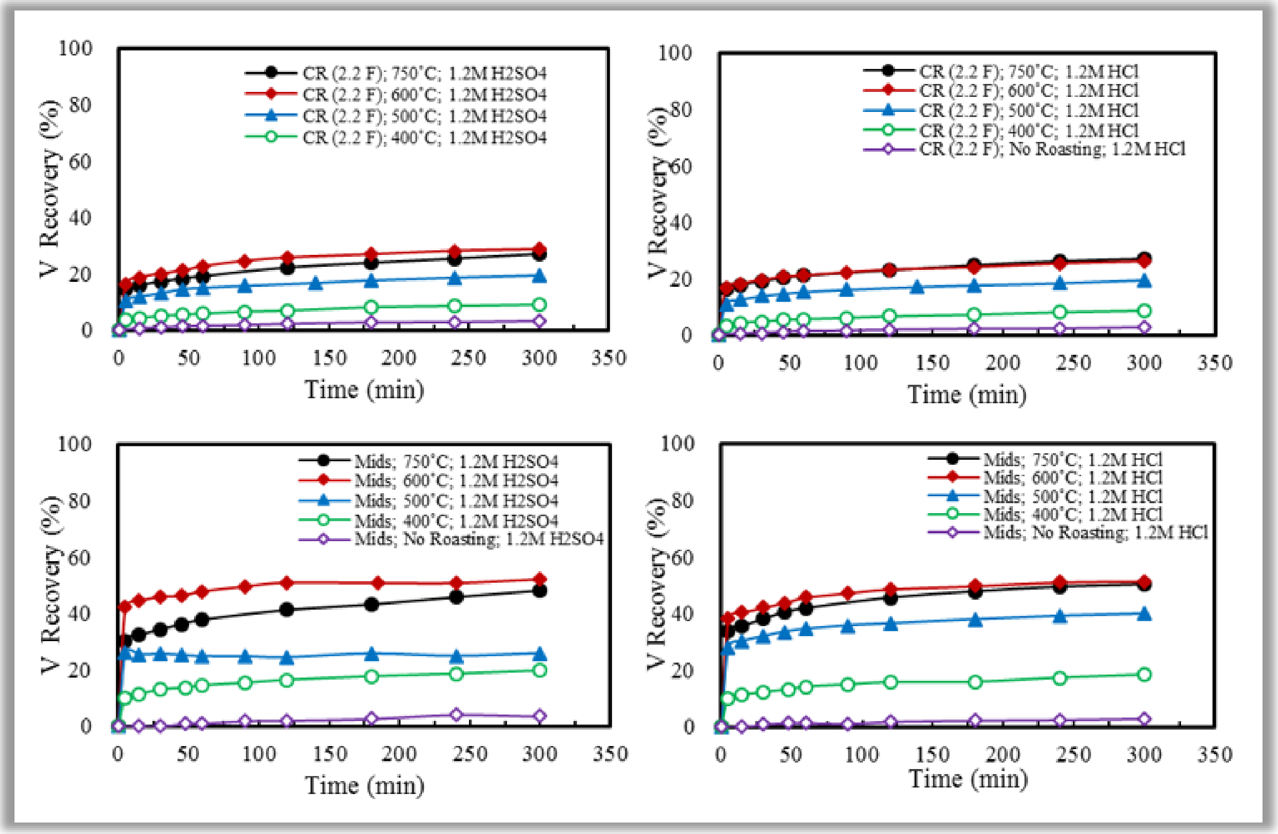
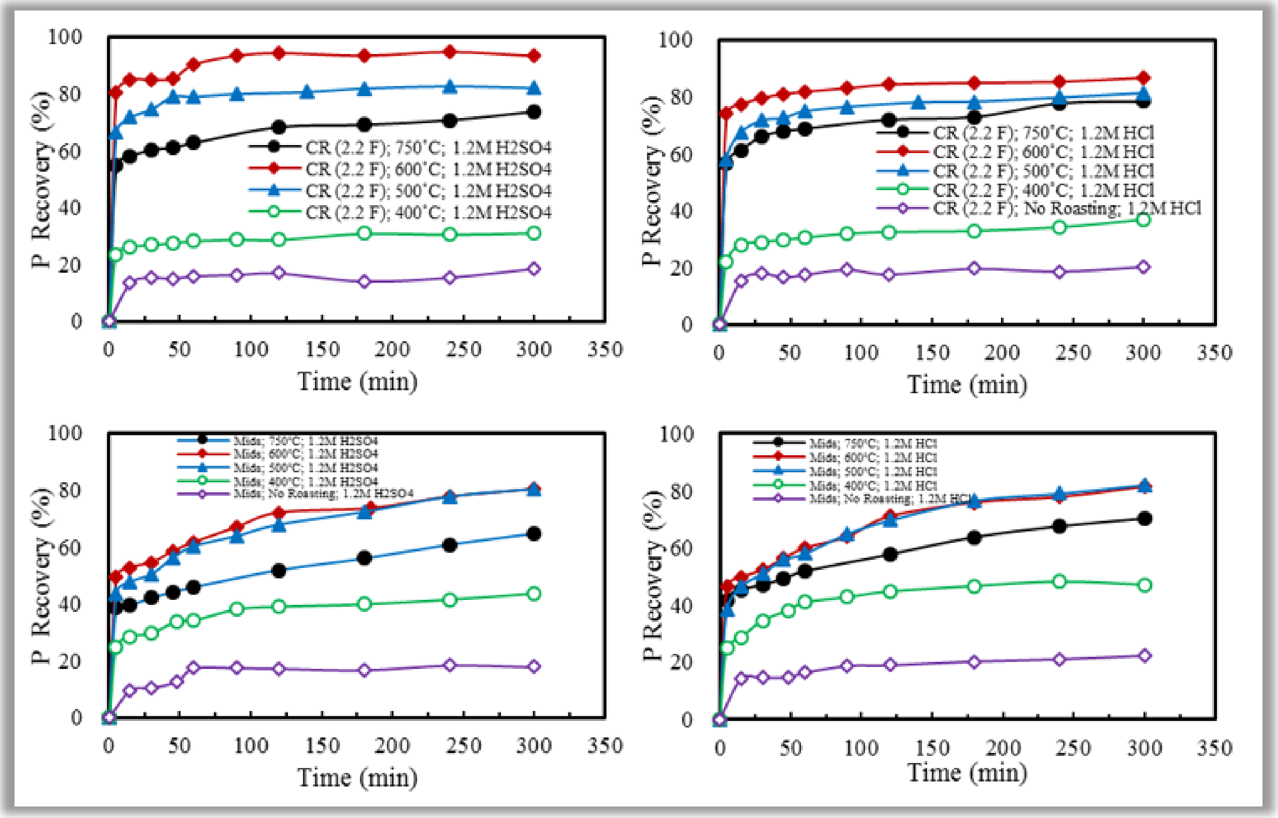


Figure 3-73: Effect of Roasting Temperature and Acid-Type on P Recovery from Leaching the Pocahontas No. 3 Coal Seam Preparation Plant 2.2 SG Coarse Rejects Float Fraction and the Middlings Product



3.2.3.6.1 Summary Observations of the Acid Leaching of other Major Elements and Trace Elements from Unroasted and Roasted Samples of the Pocahontas No. 3 Coal Seam Preparation Plant

Based on the roasting temperature where maximum recovery values were achieved, the elements can be categorized into four groups:

1. Fe, Cu, and Co (maximum recovery occurred at 400℃);
2. Al, Mg, Zn, and V (maximum recovery occurred at 600℃ or 500℃);
3. Ca (similar recoveries were obtained between 400℃ or 600℃).
4. P (recoveries in large part similar to those of REE, subject to the selection of acid type.)

A quick release of Fe, Cu, and Co occurred at 400℃ while roasting at the higher temperatures reduced the leaching kinetics and final recovery as shown in *Table 3-62*. Some volatile and fixed carbon remained in the samples roasted at 400℃, which means that the ash materials were not totally liberated. However, more than 60 percent and 70 percent of iron leached from the coarse rejects and middlings roasted samples, respectively. This may be due to an increase in porosity after roasting. Furthermore, roasting at 400℃ provided an optimum oxidization condition for the iron minerals. Elevation in roasting temperature may generate new mineral phases of iron which have low solubility. REEs are not expected to be associated with the host minerals of these three elements due to the fact that maximum recovery of REEs occurred at higher temperatures (i.e., 600℃).

Recovery of Al increased with an increase in temperature and reached maximum in the ranges of 500℃ and 600℃. The recovery curves of Al, however, are completely different from that associated with REEs. Based on the aforementioned results, more than 70 percent of the REEs in the 600℃ roasted coarse rejects and 50 percent of the REEs in the 600℃ roasted middlings, (accounting for almost all of the extractable REEs in those samples) were recovered in the first five (5) min when using sulfuric acid. The REE recovery of both samples levelled off at a relatively constant value after five (5) min. However, only a minor fraction of Al was leached under those conditions. The Al recovery curves more likely represent the gradual dissolution of Al bearing minerals (e.g., clays). As such, REEs are not expected to be associated with Al in forms of ion-substitution. Complete locking of REE mineral particles by Al-bearing particles is also not possible.

Calcium primarily occurs as calcite in the Pocahontas No. 3 Coal Seam samples, which explains the high leaching recovery of that element even without roasting. For the middlings sample, most ash materials were dispersed in the organic matrix and the soluble mineral forms were leachable under acidic conditions. As such, the significant increase in REE recovery after roasting at 500℃ and 600℃ cannot be effectively explained by the increase in liberation. It is more likely due to the changes in mineralogy during the roasting process.

Leaching behaviors of P were similar to that of LREEs (*Figure 3-58* and *Figure 3-59*). For the coarse rejects roasted samples, P was released quickly in the first five (5) min and then maintained constant, which is similar to LREEs. For the roasted middlings samples, about 40 percent of P was recovered in the first five (5) min followed by a relatively slow increase in recovery (*Figure 3-73)*. This finding is consistent with the leaching behavior of LREEs. The results indicated that LREEs were associated with P, possibly in forms of rare earth phosphates. However, when using 1.2 M H_2SO_4 to leach the roasted middlings, the LREEs recovery did not follow the same pattern as P, i.e., the recovery of REEs maintained nearly constant after the first five (5) min while P recovery kept increasing. The increase in P recovery indicated that the phosphate minerals in the roasted samples may have been dissolved. The constant REE recovery after five (5) min may be due to the in-situ precipitation of REE double sulfates.

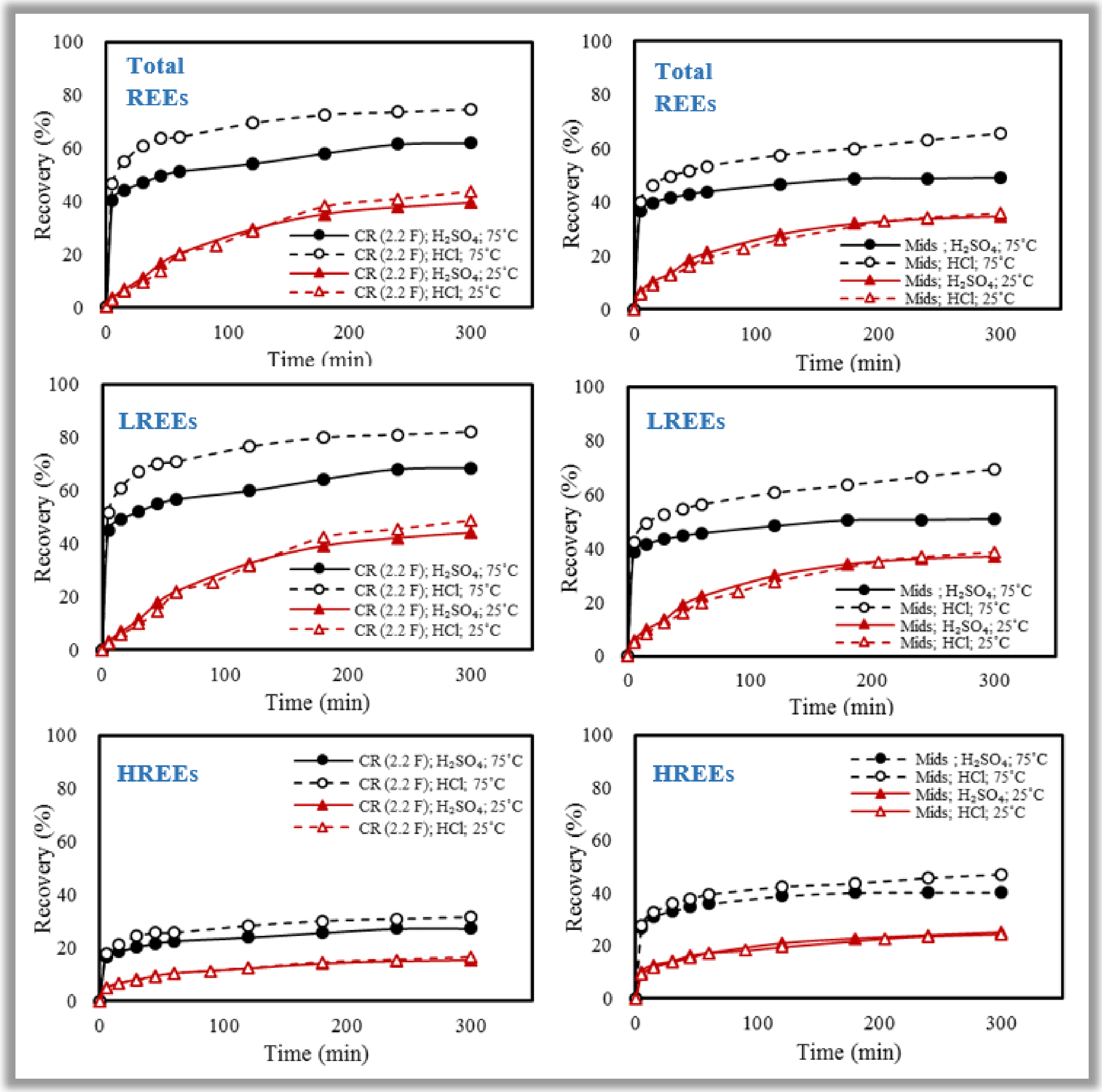
3.2.3.7 Effects of Acid Leaching Tank Temperature and Acid Type on the Recovery of REE from Roasted Samples from the Pocahontas No. 3 Coal Seam Preparation Plant

3.2.3.7.1 Effects of Leaching Temperature

Enhancement in leach recovery at higher solution temperatures was usually observed in previous studies. Increases in temperature accelerate the displacement of ions from bulk minerals to minerals surfaces. The diffusion rates of lixiviant from bulk solution to particle surface and dissolved species from particle surface to bulk solution are also increased at elevated temperatures.

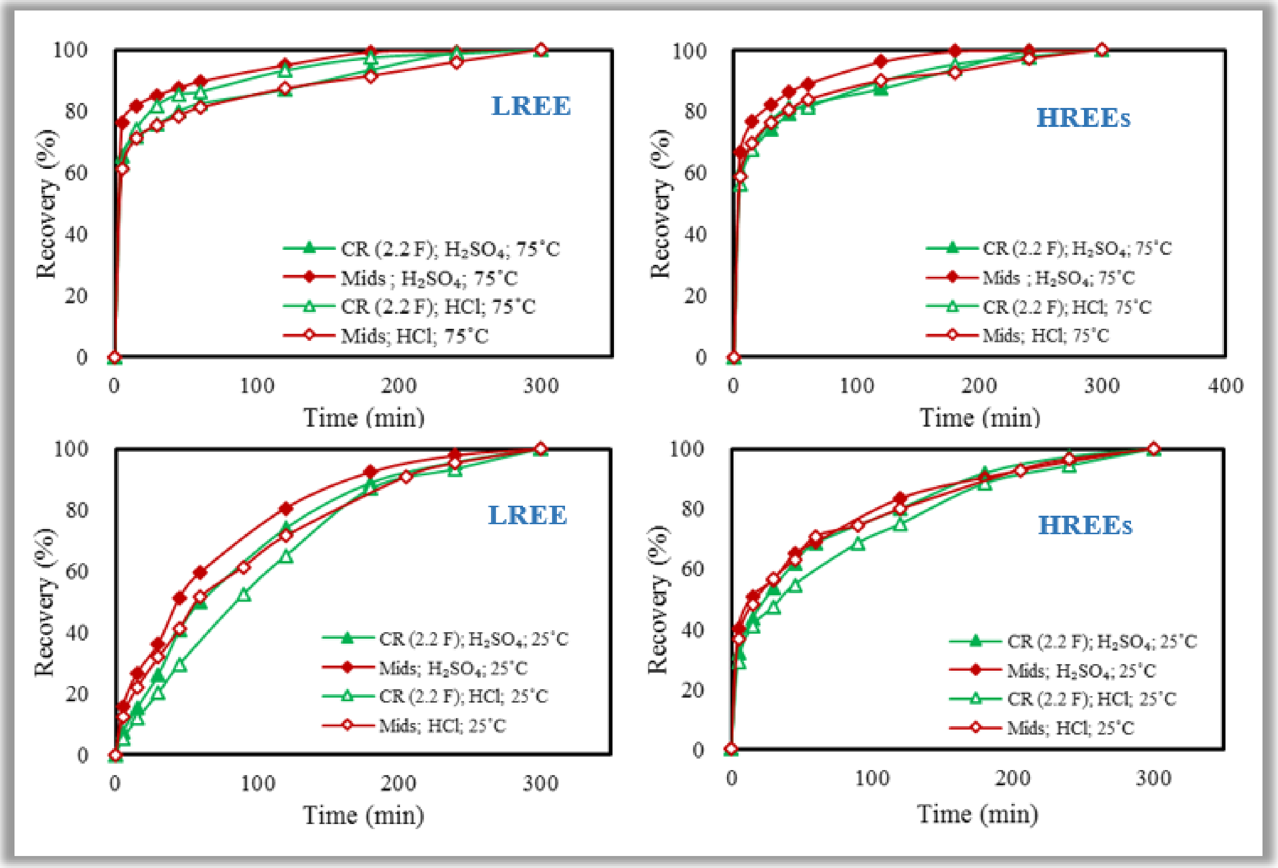
As shown in the following figure, a decrease in leachate solution temperature from 75℃ to 25℃ imposed similar effects on the coarse rejects and middlings, i.e., REE recoveries were decreased significantly. Leaching kinetics of LREEs at 75℃ are similar to HREEs based on the fact that the majority of elements are recovered in the first five (5) min and the curves gradually levelled off with an increase in reaction time, indicating a similar reaction mechanism controlling the leaching process.

Figure 3-74: REE Recovery as a Function of Time for both Pocahontas No. 3 Coal Seam Preparation Plant Coarse Rejects and Middlings Roasted Samples (750℃ for 2 hours) at 75℃ and 25℃



The next figure depicts the recovery of the “extractable REEs” as a function of reaction time. A similar trend was observed for the “extractable” LREEs and HREEs at 75℃, which also proves the previous conclusion.

Figure 3-75: Recovery of "Extractable REEs" as a Function of Time for both Pocahontas No. 3 Coal Seam Preparation Plant Coarse Rejects and Middlings Roasted Samples (750℃ for 2 hours) at 75℃ and 25℃



However, at 25℃, about 30-40 percent of "extractable HREEs" were recovered in the first five (5) min, while less than 15 percent for the LREEs. These results indicate that HREEs can be selectively leached using lower solution temperatures.

Finally, the test data suggests that LREEs in the roasted samples (750℃ for 2 hours) occurred in forms that are more sensitive to the leaching solution temperature.

3.2.3.8 Effects of Acid Leaching Solution Temperature and Acid Type on the Recovery of other Major Elements Associated with REEs in the Roasted Samples from Pocahontas No. 3 Coal Seam Preparation Plant

The association characteristics of other major elements to REEs in coal-derived samples are complicated.

Rare earth minerals such as monazite, xenotime and fluorapatite have been detected using scanning electron microscope equipped with energy dispersive X-ray spectroscopy (SEM-EDX). Based on the physical association of rare earth mineral particles with the other mineral particles, three types of REE mineral particles can be classified, i.e.,

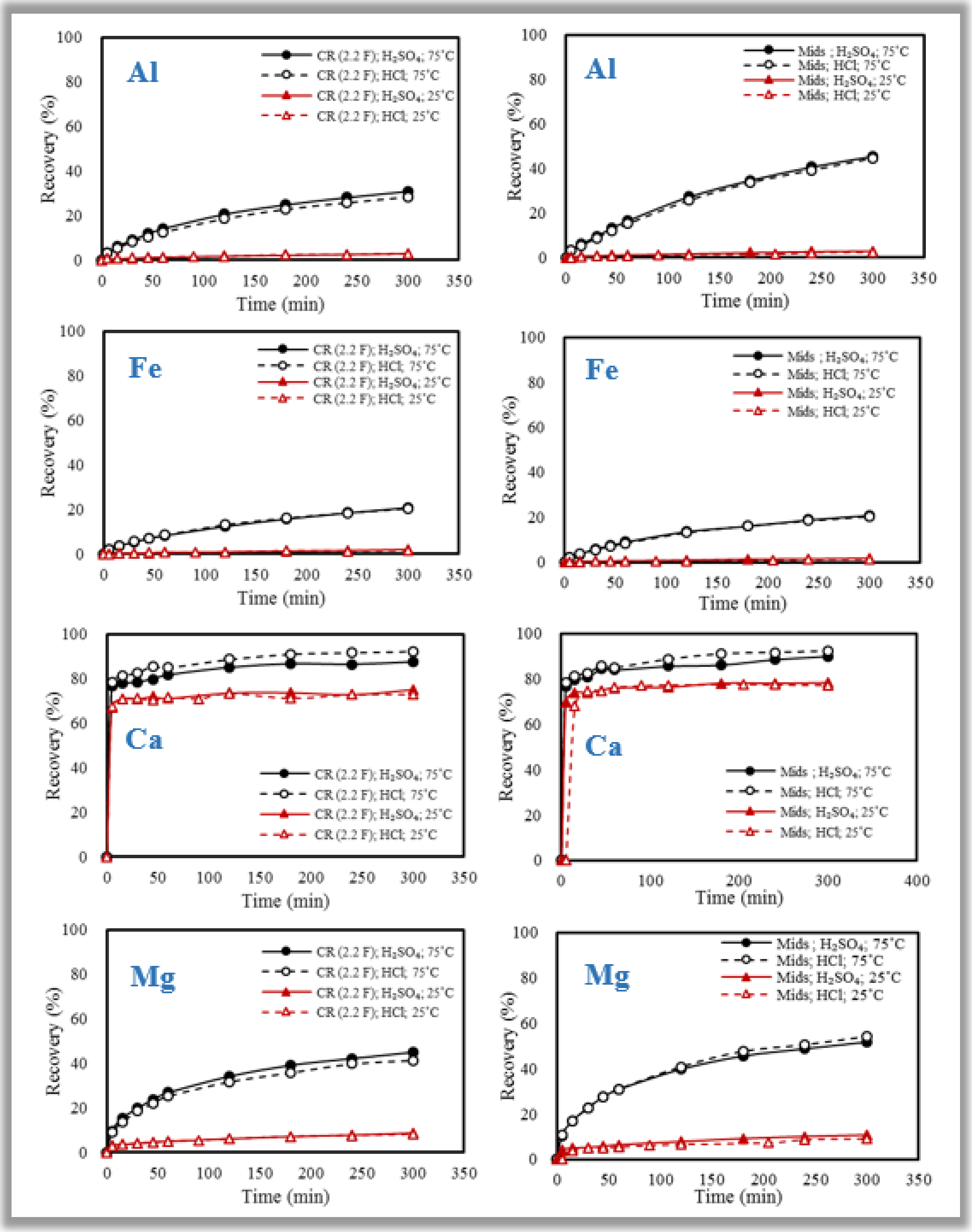
1. completely liberated,
2. partially liberated, and
3. totally unliberated (encapsulated in other mineral particles).

REEs can also occur as trace elements in the other minerals such as ion-adsorption and/or ion-substitution in clays. Furthermore, for coals of low rank, some REEs are also associated with the organic matter.

A study of the leaching characteristics of major elements was performed on the roasted samples to provide insights into the association characteristics of REEs.

The following figure depicts the leaching characteristics of Al, Fe, Ca, and Mg associated with the Pocahontas No. 3 Coal Seam preparation plant roasted samples (750℃ for 2 hours) as a function of reaction time.

Figure 3-76: Recovery of Major Elements Associated with REEs in the Roasted Coarse Rejects and Roasted Middlings Samples of the Pocahontas No. 3 Coal Seam Preparation Plant (750℃ for 2 hours)



Minor amounts (<3 percent) of Al and Fe were dissolved at 25℃ using either 1.2 M H_2SO_4 or 1.2 M HCl, indicating that Al and Fe minerals (e.g., kaolinite and pyrite) were not dissolved under this condition. However, after 300 min of reaction, a relatively significant amount of REEs (about 45 percent LREEs and 16 percent HREEs for the coarse rejects; 40 percent LREEs and 25 percent HREEs for the middlings) were leached from the roasted samples. As such, these leachable REEs more likely occurred as partially liberated particles and/or ion-adsorbed forms in the roasted samples.

More than 70 percent of Ca was leached at 25℃ within the first 15 min of reaction and the leaching recovery remained constant afterward, indicating that this fraction of calcium was associated with calcite. At a higher solution temperature (75℃), more than 80 percent of Ca was leached within the first 15 min. The difference in calcium recovery between 75℃ and 25℃ may be due to the dissolution of other calcium-based minerals that have a lower solubility when compared to calcite. The REEs are less likely associated with calcium minerals due to the fact that significant differences in REE recovery were observed at the two different temperatures, especially within the first 15 min.

3.2.3.9 Effects of Leachate Solution pH and Acid Type on the Recovery of REEs and Associated Elements from the Roasted Coarse Rejects Sample of the Pocahontas No. 3 Coal Seam Preparation Plant

3.2.3.9.1 Effects of Solution pH

Efficient acid leaching was normally achieved by intensive reaction between protons and target mineral particles. As such, the activity of protons in solution affects leaching recovery. For minerals with high solubility, complete dissolution may be realized using relatively high solution pH values, under which condition smaller amount of contaminants report to the leachate thereby resulting in lower operation costs (e.g., acid consumption).

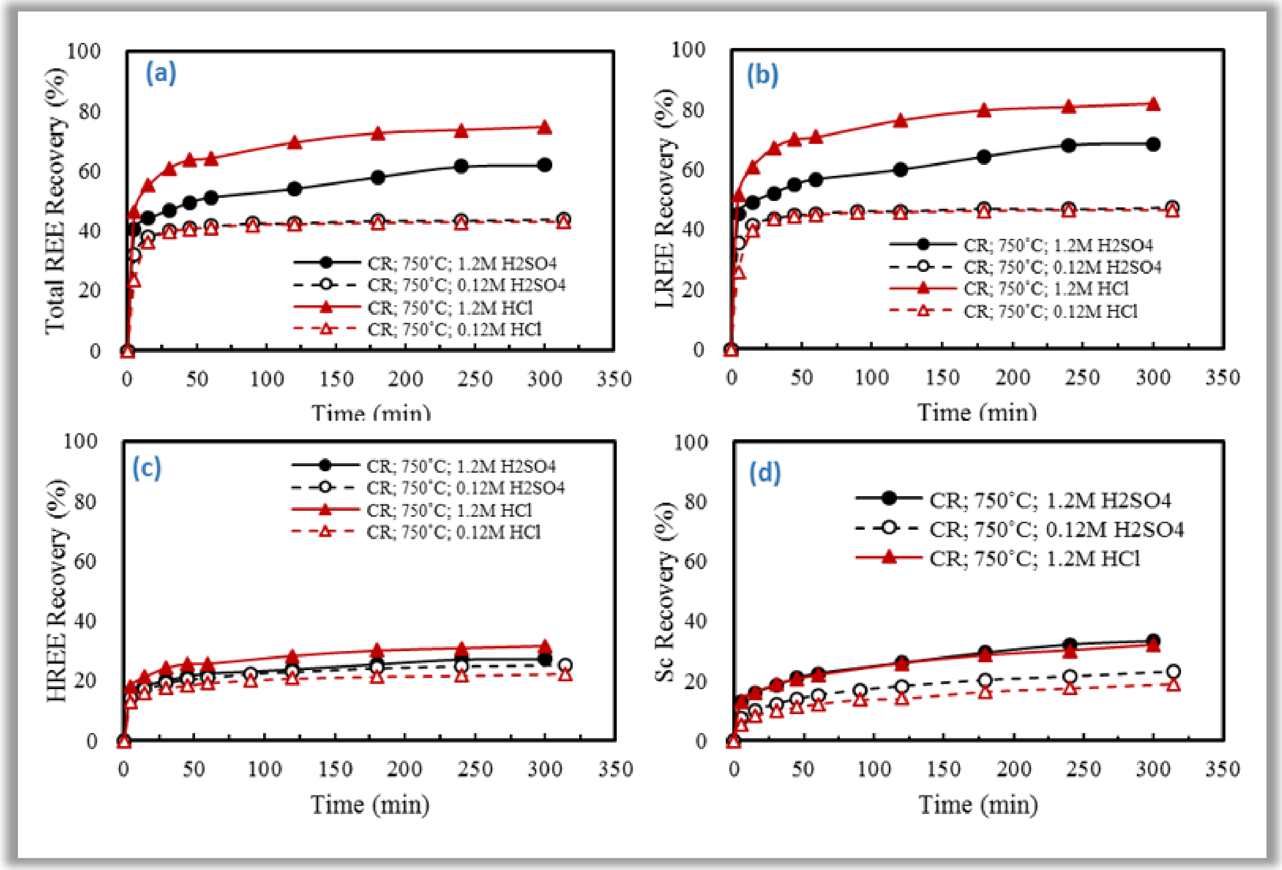
The leaching behaviors of individual REEs and the other elements under relatively mild acid conditions were investigated for the Pocahontas No. 3 Coal Seam preparation plant coarse rejects 2.2 SG float sample after roasting at 750℃ for 2 hours.

3.2.3.9.2 Effects of Acid Concentration on the Leaching Recovery of LREEs and HREEs

Recovery of LREEs using less concentrated acid solutions (i.e., 0.12 M H_2SO_4 and 0.12 M HCl) were lower compared to the values obtained using 1.2 M H_2SO_4 and 1.2 M HCl (*Figure 3-77* (a) and(b)). The LREEs recovery values under both conditions increased significantly after the first 15 min, while with less concentrated acid solutions, the recovery remained constant (about 46 percent) with leaching time. The LREEs recovery using more concentrated acid solutions maintained an upward trend with leaching time. The results indicated that, after roasting at 750℃, a part of the LREEs became very leachable and recoverable using moderate acid concentrations or solution pH values around 1. However, a portion of the LREEs are nevertheless relatively difficult to recover by acid leaching.

A similar trend was observed for HREEs under both conditions, i.e., a quick release occurred at the beginning followed by a gradual increase in recovery (*Figure 3-77*(c)). The HREE recovery values achieved under both conditions were less than 10 percent, indicating that a minor portion of the HREEs were very leachable while the majority remained very difficult to recover by acid leaching even after roasting the feedstock. The leaching behavior of Sc was also included in *Figure 3-77*(d) due to its significant economic value. As shown, Sc followed a similar trend to the HREEs, indicating similar associated characteristics in the roasted sample.

Figure 3-77: Effects of Acid Concentration on the Leaching Recovery of REEs from Roasted Coarse Rejects from the Pocahontas No. 3 Coal Seam Preparation Plant



<u>*3.2.3.9.3 Effects of Acid Concentration on the Leaching Recovery of other Elements Associated with REEs*</u>

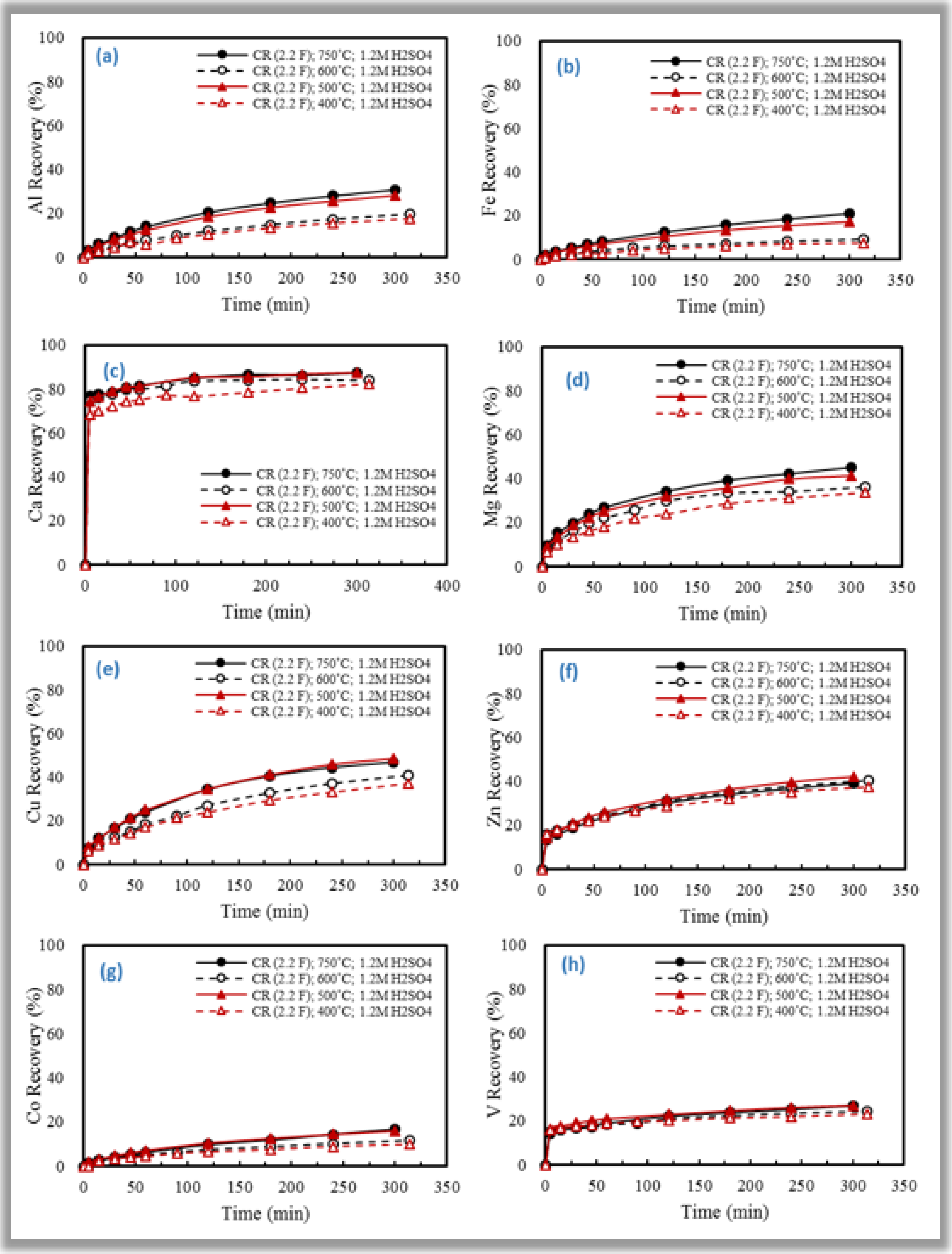
The leaching characteristics of the other elements are shown in *Figure 3-78*. Three different types of kinetic leaching behaviors can be observed:

1. A gradual increase in recovery with time (Fe, Al, Mg, Cu and Co);
2. a quick leaching rate at the beginning of the test followed by a period of slow leaching (Zn and V); and
3. a fast leaching rate at the beginning of the test followed by a near complete stop in the leaching reaction as indicated by a constant recovery value (Ca).

It is important to note that REEs belong to the second kinetic leaching behaviour classification enumerated above. This information provides some insights about elemental mineralogy. The elements of Classification 2 occur in two types of mineral phases with extremely different solubilities. Another possibile explanation for the slow increase in recovery for the second type of elements is an association with mineral particles which are locked by the other mineral particles and/or association with other mineral particles in an ion-subsitution form. (However, futher study is needed to clarify this issue.) In contrast, the

Elements of Classification 1 and 3 are more likely occur in a single mineral phase or multiple mineral phases with similar solubilities.

Figure 3-78: Effects of Acid Concentration on the Leaching Recovery of other Elements Associated with REEs in the Roasted Coarse Rejects Samples of the Pocahontas No. 3 Coal Seam Preparation Plant



3.2.3.10 Summary Observations of the Effects of Roasting Temperature, Leachate Solution pH and Acid Type on the Recovery of REEs and Associated Elements from the Roasted Coarse Rejects Sample of the Pocahontas No. 3 Coal Seam Preparation Plant

The major findings from the roasting and leaching studies performed on the coarse rejects and middlings material collected from the Pocahontas No. 3 coal seam preparation plant are enumerated as follows.

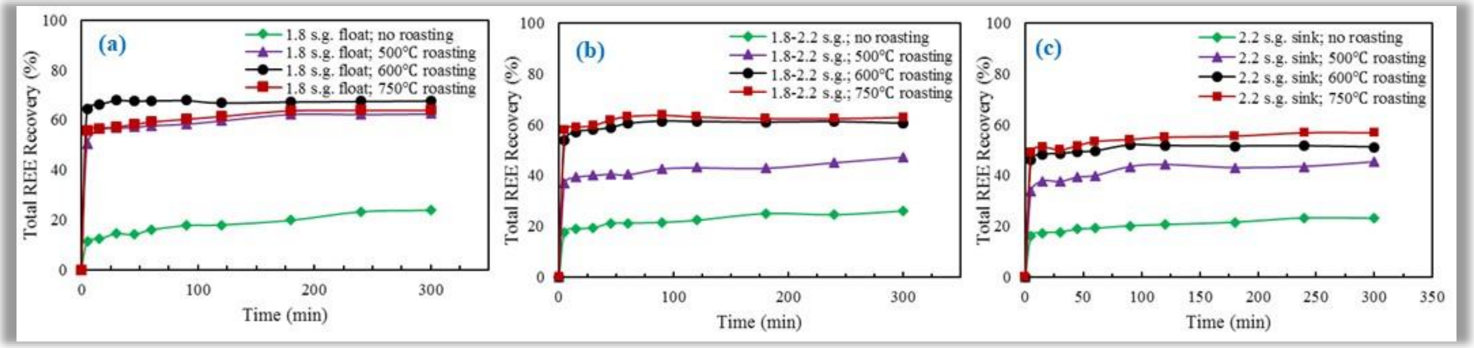
1. Total REE recovery values were increased significantly by roasting at 600℃ for two hours, i.e., about 80 percent and 75 percent of the total REEs were leached from the coarse rejects and middlings, respectively, when using 1.2 M HCl and 75℃ leaching temperature.
2. For the coarse rejects samples, H_2SO_4 performed as efficiently as HCl while, for the middlings material, recovery decreased by about 15 absolute percentage points when using HCl.
3. Roasting at 500℃ and 600℃ was more efficient for LREEs compared to HREEs. For example, a Ce recovery of 96 percent was realized from treating the roasted coarse rejects samples using 1.2 M HCl while only about 35 percent of Y and Dy were leached.
4. Under the same roasting and leaching conditions, LREEs recovery from the coarse rejects samples were higher than the middlings, while HREEs were relatively easier to be leached from the middlings. About 60 percent of the HREEs were recovered from the 600℃ roasted middlings.
5. Based on the effects of roasting on leaching kinetics, three groups of elements were studied, i.e.: Fe, Cu and Co (maximum recovery occurred at 400℃); Ca (maximum recovery occurred between 400℃ and 600℃, more than 80 percent was leached for the non-roasted samples); and REEs, Al, Mg, Zn and V (maximum recovery occurred at 500℃ or 600℃).
6. Two factors apparently contributed significantly to the improved REE recovery by roasting, i.e., liberation of ash materials and REE mineralogy transformation.
7. REE mineral particles may be completely and/or partially locked by the other mineral particles such as clays. However, this observation alone may not be a dominant factor in limiting REEs acid leachability.
8. For the samples roasted at 750℃, the REEs recovery were sensitive to acid leaching solution temperature as well as solution pH. Significant decreases in recovery occurred under any given solution pH when the acid leaching solution temperature was decreased from 75℃ to 25℃.

3.2.4 Leachability Tests of the Rejects from the Blue Gem Seam Coal

3.2.4.1 Effects of Roasting on TREE Recovery from Three Float-sink Products of Rejects from Ferroglobe's Gatliff Plant (Blue Gem Seam)

The effects of roasting on TREE recovery from three float-sink products (i.e., 1.8 SG float, 1.8-2.2 SG, and 2.2 SG sink) were assessed and compared to the performance on unroasted samples as shown in the following figure.

Figure 3-79: Effects of Roasting on the TREE Recovery from (a) 1.8 SG Float, (b) 1.8-2.2 SG, and (c) 2.2 SG Sink Fraction of the Blue Gem Seam Coarse Reject Samples



3.2.4.1.1 Summary Observations of the Effects Roasting on Total REE Recovery from Three Float-Sink Products of Rejects from Ferrogloble's Gatliff Plant

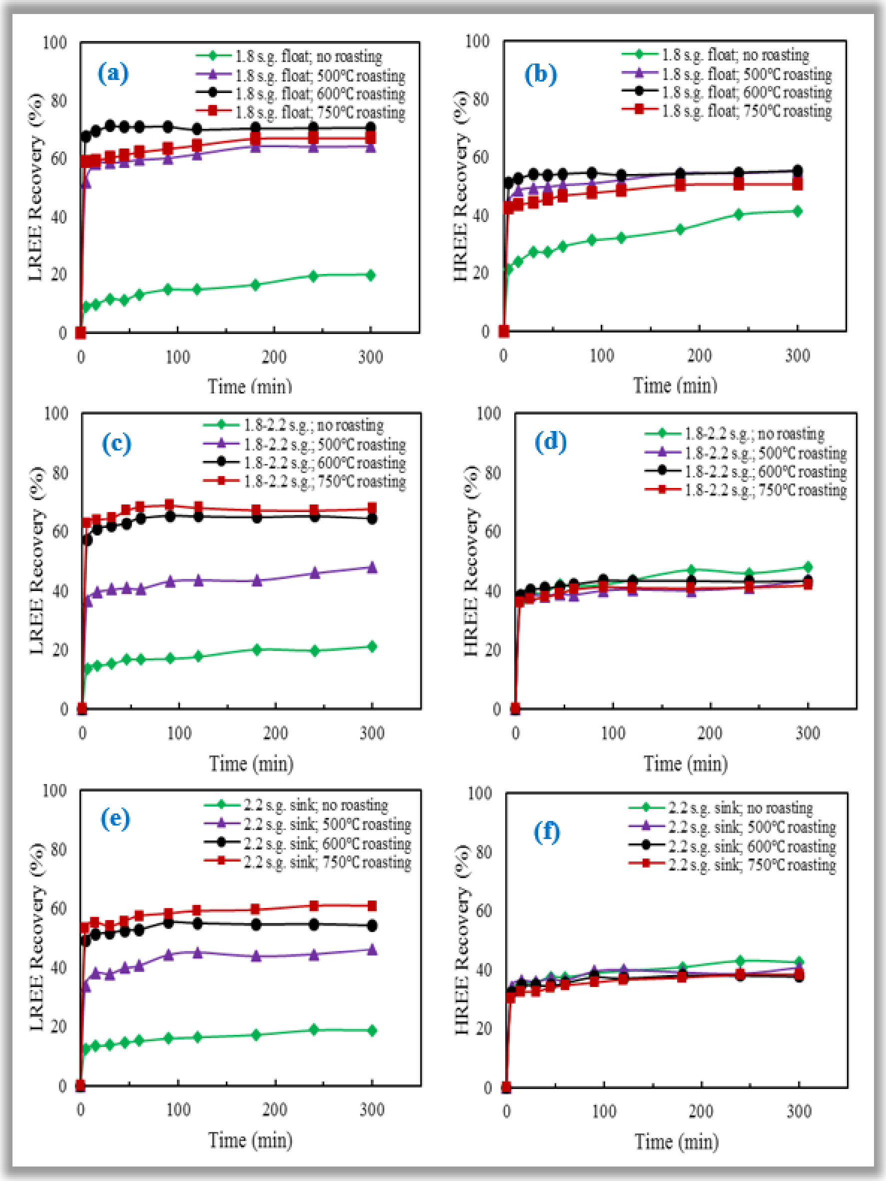
1. For the non-roasted samples, REE recovery between the density fractions were within 25 percent of each other.
2. Roasting significantly improved recovery with the highest (55-65 percent of REEs) occurring between 600°C and 750°C.
3. It should be noted that the 1.8-2.2 SG and 2.2 SG sink samples contained very high ash contents (83.50 percent and 92.28 percent). Accordingly, it is believed that the improvement in REE recovery is not correlated with the organic matter removal and micro-dispersed ash material in organic matrix. Rather, the increase in REE recovery is more likely explained by the thermal decomposition of rare earth minerals (mainly phosphates) at high temperatures.
4. The results shown in *Figure 3-79* (b) and (c) indicate recovery from the 500°C roasted samples was less than that of the 600°C and 750°C samples. It is therefore apparent that the decomposition of rare earth minerals for improved leach recovery in the Blue Gem coarse rejects is favorable in temperatures above 600°C.
5. A comparison between Figure 3-79 (a) and (b) shows that REE recovery of the 500°C roasted 1.8 SG float material was higher than that of the 1.8-2.2 SG (63 percent versus 47 percent).
6. The difference in recovery between 500°C and 600°C roasted samples was minimal for the 1.8 SG float sample.
7. More than 90 percent of the organic matter in the 1.8 SG float sample was removed by roasting at 500°C for two hours. The improved recovery by roasting at 500°C is partially due to the release of REEs associated with organic matter and/or micro-dispersed ash material in the organic matrix. Previous studies have reported that REEs in the finely dispersed ash material in coal have higher leachability.
8. For the 1.8 SG roasted sample, the recovery values achieved with the 600°C roasted material was a little higher than the 750°C, while the 1.8-2.2 SG and 2.2 SG sink samples show an opposite trend. This finding may be due to the different mineralogy between 1.8 SG float and the other two fractions.

9. The test results also indicate that there is a temperature limit where leach recovery values begin to decrease with increasing temperature. This indicates that morphology changes are temperature-dependent. Roasting at 750℃ and above may begin to sinter the ash material in the 1.8 SG float fraction which is known to reduce leach recovery.

3.2.4.2 Effects of Roasting on HREE and LREE Leach Recovery from Three Float-sink Products of Rejects from Ferroglobe’s Gatliff Plant (Blue Gem Seam)

Effects of roasting on HREE and LREE leach recovery were assessed separately. The next figure shows the leaching kinetic data for the samples roasted at different temperatures.

Figure 3-80: Effects of Roasting on the LREE and HREE Recovery from Blue Gem Seam Rejects: (a, b) 1.8 SG Float; (c, d) 1.8-2.2 SG, and (e, f) 2.2 SG Sink Fractions



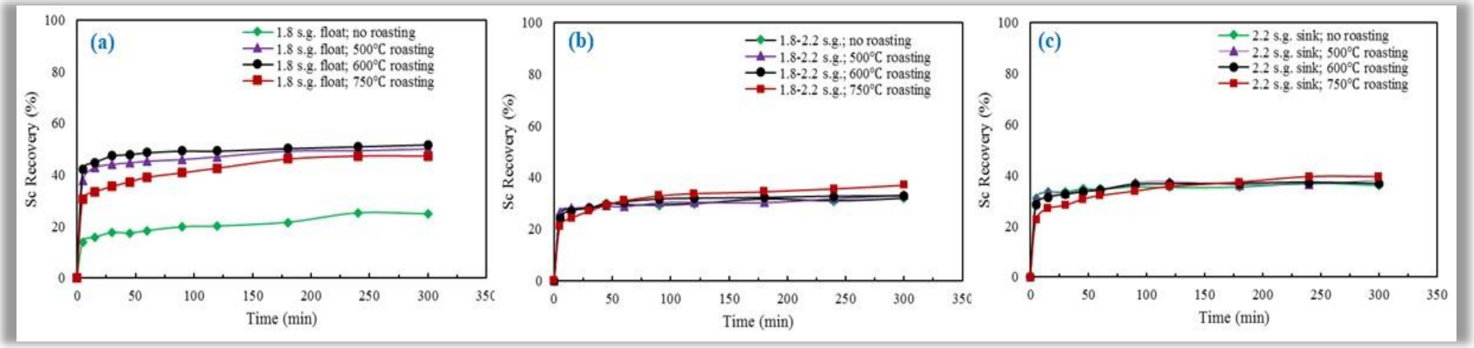
3.2.4.2.1 Summary Observations of the Effects of Roasting on HREE and LREE Leach Recovery from Three Float-Sink Products of Rejects from Ferrogloble's Gatliff Plant

1. Recovery and leaching kinetic rates of LREEs (Figure 3-80 (a), (c), and (e)were similar to that of TREEs (Figure 3-79).
2. However, HREEs responded differently.
    a) For the non-roasted control samples, HREE recovery was higher than LREEs. For example, 41 percent of HREEs were leached from the non-roasted 1.8 SG float sample after 300 min of reaction, while only 20 percent of LREEs were recovered (Figure 3-80 (a) versus (b)). The increased recovery of HREEs may be due to a portion of HREEs occurring in more soluble forms such as carbonates, ion-substitutions, etc.
    b) For the 1.8 SG float sample, HREE recovery increased after roasting, which was likely due to the release of some organic associated and/or micro-dispersed HREEs.
    c) However, for the 1.8-2.2 S.G. and 2.2 S.G. sink samples, HREE recovery remained essentially unchanged. This may be due to insoluble HREE minerals requiring higher decomposition temperatures. For example, it was previously stated that iron recovery decreased by roasting at temperatures higher than 400℃ from the transformation from pyrite to hematite.

3.2.4.3 The Effects of Roasting on the Leaching of Sc from (a) 1.8 SG Float, (b) 1.8-2.2 SG, and (c) 2.2 SG Sink Fractions of the Blue Gem Seam Coarse Reject Samples

The effects of roasting on the leaching of Sc from 1.8 SG Float, 1.8-2.2 SG, and (c) 2.2 SG Sink Fractions of the Blue Gem Seam Coarse Reject Samples are provided in *Figure 3-81*(a)-(c).

Figure 3-81: Effects of Roasting on Sc Recovery from (a) 1.8 SG Float, (b) 1.8-2.2 SG, and (c) 2.2 SG Sink Fractions of Blue Gem Coal Coarse Rejects



3.2.4.3.1 Summary Observations of the Effects of Roasting on Sc Recovery from Three Float-Sink Products of Rejects from Ferrogloble's Gatliff Plant

1. Maximum recovery values of 52 percent, 37 percent, and 39 percent were achieved from the leaching of the 1.8 SG float, 1.8-2.2 SG, and 2.2 SG sink fractions, respectively.
2. Roasting did not significantly impact recovery from the two highest density fractions (1.8-2.2 SG and 2.2 SG sink).
3. Sc recovery from the 1.8 SG float fraction increased from 24 percent to 52 percent after roasting, which may be attributed to the removal of organic matter and liberation of Sc-bearing minerals.

3.2.4.4 Summary of Methodology and Observations of the Leachability Tests of the Rejects from the Blue Gem Seam Coal

1. Methodology for evaluating the roasting as a feed material pretreatment was studied on REE leaching recovery from the Blue Gem coal coarse rejects.
    a) The sample was density fractionized into three portions, i.e., 1.8 SG float, 1.8-2.2 SG, and 2.2 SG sink.
    b) The three samples were roasted at 500℃, 600℃, and 700℃.
    c) Acid leaching tests were performed on both the roasted and non-roasted samples.
2. Summary Observations
    a) LREE recovery was elevated from about 25 percent to 55-65 percent by roasting at 600℃ and 700℃ .
    b) No significant increase in recovery was observed for HREEs.
    c) About 40 percent of HREEs were leached from the 1.8-2.2 SG and 2.2 SG sink roasted samples.
    d) For the non-roasted samples, HREE recovery was higher than LREEs.
    e) Roasting provided Sc recovery improvement for the 1.8 SG float fraction only which may be due to mineral liberation or simple release from the organic matter after combustion.

3.2.5 Leachability Tests of the Rejects from the Fire Clay (Hazard No. 4) Seam Coal

3.2.5.1 Fire Clay (Hazard No.4) Coal, the Host Land Company, and the Host Coal Mining and Processing Company.

The experimental work performed on material collected from Fire Clay (Hazard No. 4) Seam sources involved the collection of samples of from the rejects (coarse and thickener underflow) and middlings circuits of Blue Diamond Coal Co's No. 76 Plant[13] (a.k.a. Blackhawk Mining's Leatherwood preparation plant) located near the community of Slemp, Perry County, Kentucky.

The plant produces a low-ash coal product from a primary dense medium vessel and a medium-ash content product from a secondary dense medium vessel, dense medium cyclone circuit and spiral concentrator

---

[13] Blue Diamond Coal Company, No. 76 Plant, (a.k.a. Leatherwood Plant) MSHA ID 15-16520, controlled by Blackhawk Mining LLC, 48 Beech Fork Road, Slemp, Perry County, Kentucky, 41763.

circuit. Reject streams include the coarse reject belt material (plus (+)150 µm) and fine reject thickener underflow slurry (minus (-)150 µm).

The plant is primarily supplied run-of-mine *(ROM)* coal feedstock from "company operated" mines in the Fire Clay Seam (Hazard No. 4) but it also takes coals from captive from independent commercial mine operators. The plant operator will often selectively process the Fire Clay (Hazard No. 4) Seam for sale to a higher value pulverized coal injection *(PCI)* metallurgical coal market. Other cleaned coal products and high-ash products of the Fire Clay (Hazard No. 4) are typically sold into the steam coal markets.

The discussions that follow are based on samples from the rejects and middlings circuits of the No. 76 Plant while cleaning the Fire Clay (Hazard No. 4) Seam feedstock. As addressed in other portions of this report, the Fire Clay (Hazard No. 4) Seam (to include its coal benches and its attendant partings, "riders", roof and floor lithologies) present a relatively well-known supply of rare earth minerals. Most of those rare earth bearing minerals are concentrated in the rejects and middlings circuits of the No. 76 Plant.

3.2.5.2 Effects of Roasting on the Mineralogy of the Fire Clay (Hazard No. 4) Rejects Material

To conduct the study, a large bulk sample of crushed and sized No 76 Plant (Leatherwood) coarse rejects was sent to a commercial facility[14] to subject the sample to roasting prior to leaching. To determine if the roasting treatment was effective, representative samples of roasted and unroasted material was analyzed by x-ray diffraction *(XRD)* to determine if changes in the crystallinity and mineralogy occurred. A summary of the XRD results are found in the following table.

Table 3-63: Results of XRD Examination of the Roasted Fireclay (Hazard No. 4) Rejects Crystalline Structure

Constituent	Quartz (wt.%)	Kaolinite (wt.%)	Muscovite (wt.%)	Illite (wt.%)
Roasted	82.00	0.00	12.50	5.55
Unroasted	55.70	25.40	9.60	9.40

The results of the study show an overall shift from kaolinite to other forms of mineralization. A brief literature search yielded a paper titled "Influence of Thermal Treatment on Kankara Kaolinite" (Edomwonyi-Out, 2013), which reported that kaolinite undergoes dehydroxylation to form metakaolin at 528.86°C. In reference to these results, the change in crystalline structure coupled with dihydroxylation and oxidation yields both a structure and form more readily penetrable and leachable by acid forms.

As further discussion on this subject note, roasting the rejects material prior to submitting it to acid leaching significantly improved REE recovery, increased leaching kinetics (with most of the leaching reactions occurring within five minutes from the start of the process), and significantly decreased acid consumption.

3.2.5.3 No. 76 Plant (Leatherwood) Fire Clay Seam (Hazard No. 4) Thickener Underflow Acid Leaching Testing Methodology

To assess the recoverability of the REEs from the Leatherwood thickener underflow material, leaching studies were conducted on the flotation middling and tailing products as well as the material finer than 45 µm (-325-mesh).

---

[14] Nex-Gen Industries, 610 Trus Joist Lane, Chavies, Kentucky, 41727.

Standard leaching conditions were used in the tests which included the use of a 1.2 M sulfuric acid solution at a temperature of 75°C.

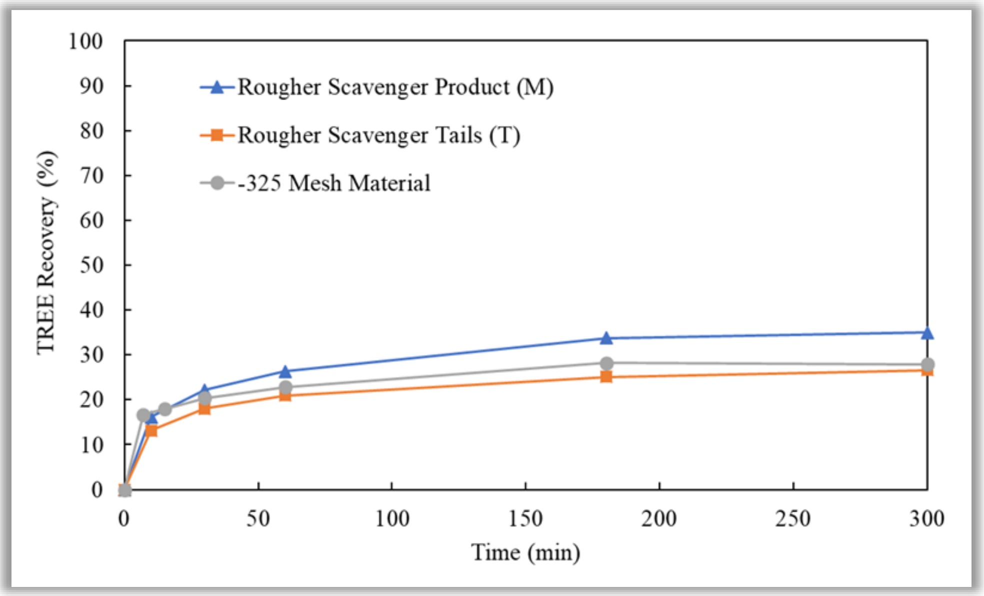
Leaching experiments were conducted on roasted and unroasted samples to assess the effect on recovery.

Roasting was used as a pretreatment step prior to leaching at a temperature of 600°C for a period of two hours.

3.2.5.3.1 Acid Leaching Test Results of Unroasted Fire Clay Seam (Hazard No. 4) Flotation Middling Material from the No. 76 Plant (Leatherwood) Thickener Underflow

The TREE recovery achieved after leaching the unroasted flotation middling material for five hours was 35 percent as shown in the following figure.

Figure 3-82: Total REE Recovery Achieved by Leaching the Unroasted Products Generated of the No. 76 Plant (Leatherwood) Thickener Underflow Material Samples



Approximately one-half of the recovered REEs was extracted within the first 10 minutes.

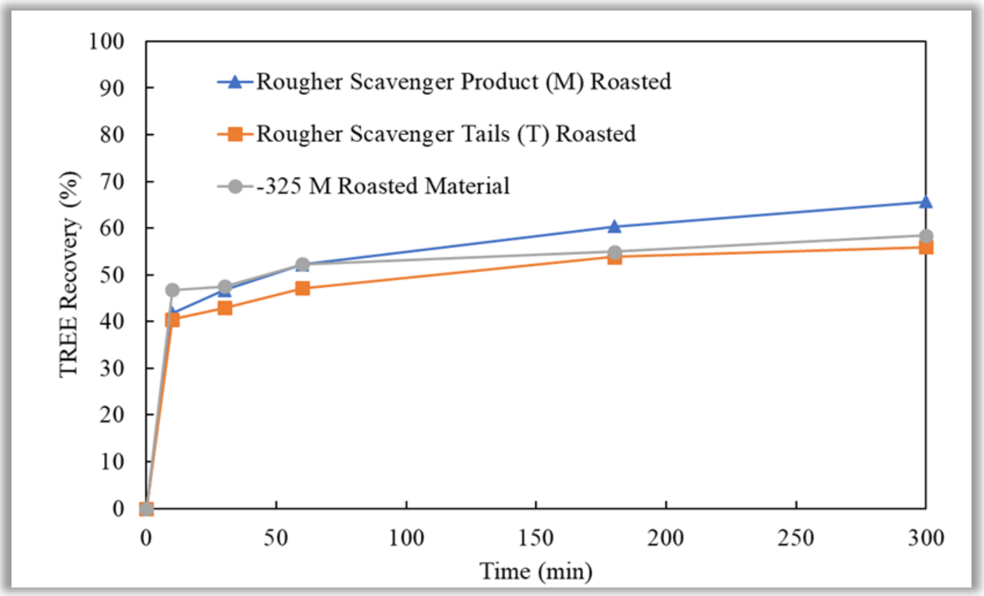
The REE recovery rates and final values for the flotation tailings and -45µm fraction were lower which agrees with previously reported findings.

After 300 minutes of leaching, recovery values obtained from the treatment of the tailings and -45µm size fraction reached 28 percent.

3.2.5.3.2 Acid Leaching Test Results of Roasted Fire Clay Seam (Hazard No. 4) Flotation Middling Material from the No. 76 Plant (Leatherwood) Thickener Underflow

After roasting, leaching rates and overall recovery improved significantly for all three samples as shown in the following figure.

Figure 3-83: TREE Recovery Achieved by Leaching the Roasted Products Generated from the Processing of the Leatherwood Thickener Underflow Material



TREE recovery values greater than 40 percent were achieved within only 10 minutes of leaching for each of the three roasted materials which signifies a substantial increase in the release rate of the REEs as compared to the test results on the unroasted samples. Given that the roasting process was conducted under 600^{O}C which is below the ash fusion point, the significantly enhanced leach rates and improved REE recovery values was likely due to the decomposition of the clay structure and phosphate minerals from which highly soluble rare earth minerals were released. After 10 minutes of leaching, recovery values gradually increased and reached levels of 58 percent to 65 percent after five hours of leaching.

3.2.5.4 No. 76 Plant (Leatherwood) Fire Clay Seam (Hazard No. 4) Coarse Rejects Acid Leaching Testing Methodology

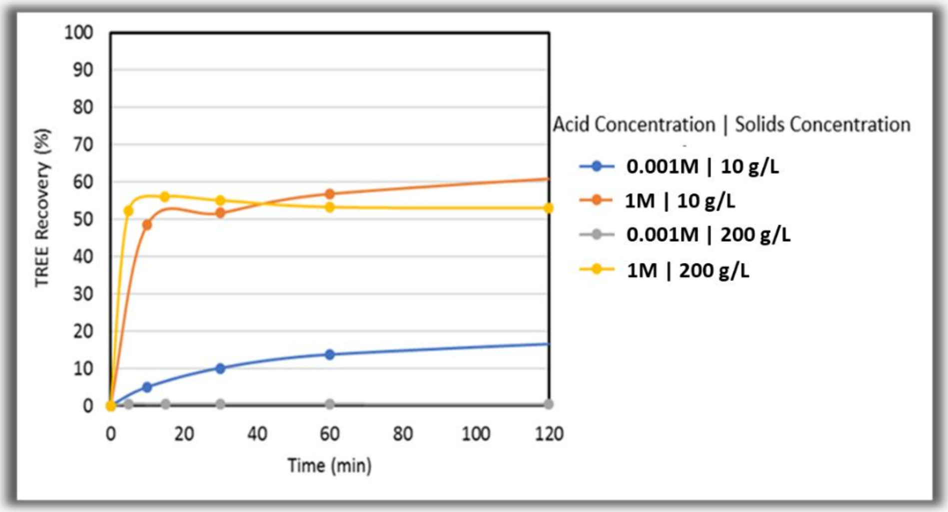
The 1.8 x 2.2 SG density fraction in the coarse rejects material coarser than 9mm (3/8-in) were roasted at 600°C for two hours and subjected to parametric leaching studies to determine the effects of percent solids and acid concentration on leaching recovery.

The standard test conditions were performed using 1M H_2SO_4 at a solids concentration of 1 percent or 10 g/L of slurry.

3.2.5.4.1 Effect of Solids and Acid Concentration on Leaching Recovery and Kinetics for the REEs Associated with the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material)

The REE recovery data presented in the next figure shows that leaching recovery and kinetics achieved from the treatment of roasted 1.8 x 2.2 SG material in the Fire Clay (Hazard No. 4) coarse rejects.

Figure 3-84: Effect of Solids and Acid Concentration on Leaching Recovery and Kinetics for the REEs Associated with the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material



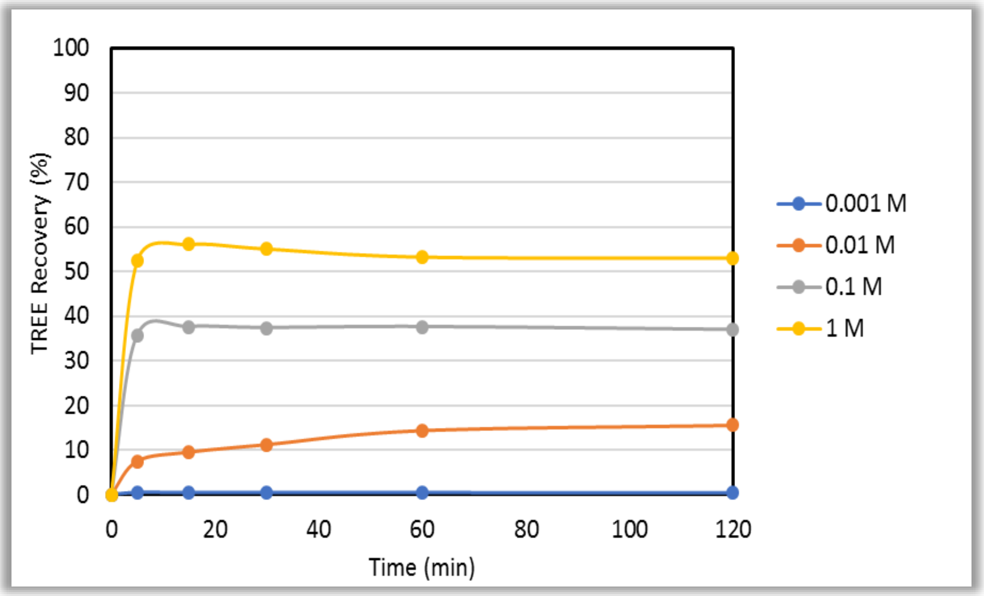
The test was minimally impacted by an increase in the solids concentration of the leachate from 1 percent (10 gm/L) to 20 percent (200 gm/L) by weight when using 1M acid concentration. However, it is noted that REE recovery maximized after 15 minutes of leaching and slightly declined by three percentage points thereafter possibly due to localized precipitation and mineralization due to the dissolution of calcite and other alkaline earth minerals.

The effect of solids concentration was very prevalent when using a very low acid concentration of 0.001M. Under this condition, the dissolution of calcite in a high solids content slurry consumes the small amount of acid in the system thereby making the leaching process ineffective for REE recovery.

3.2.5.4.2 Effect of Acid Concentration on Leaching Recovery and Kinetics for the REEs Associated with the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No 4) Coarse Rejects Material

The acid concentration effects on REE leachability from roasted Fire Clay coarse rejects is shown in the following figure.

Figure 3-85: Effect of Acid Concentration on Leaching Recovery and Kinetics for the REEs Associated with the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay Coarse Rejects Material (Solids Concentration = 20 percent by Weight)



Maximum REE recovery for both the 1M and 0.1M acid concentrations occurred within approximately five minutes from the start of the tests. The 1M concentration provided REE recovery values that were about 15 absolute percentage points higher.

Acid concentrations of 0.01M and 0.001M resulted in low REE recovery values with the latter condition recovering less than a couple percentage points in recovery.

3.2.5.4.3 Effect of Acid Concentration on Leaching Recovery and Kinetics of other Elements Associated with REE Mineral in the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material

An advantage of using a lower acid concentration is a reduction in the leaching of contaminate ions such as Fe, Al and Ca that impact the downstream concentration processes associated with REE recovery.

The leaching data for the major contaminant ions are shown in the following figure.

Figure 3-86: Effect of Acid Concentration on Leaching Recovery and Kinetics of Fe for the Roasted 1.8 x 2.2 SG fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material (Solids Concentration = 200 gm/L)

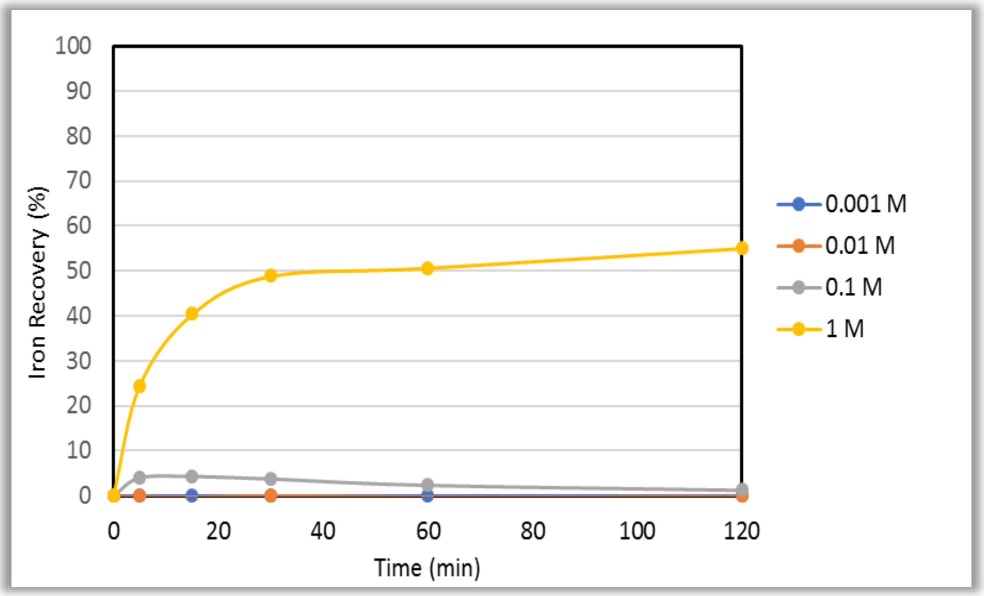


Figure 3-87: Effect of Acid Concentration on Leaching Recovery and Kinetics of Al for the roasted 1.8 x 2.2 SG fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material; Solids Concentration = 200 gm/L

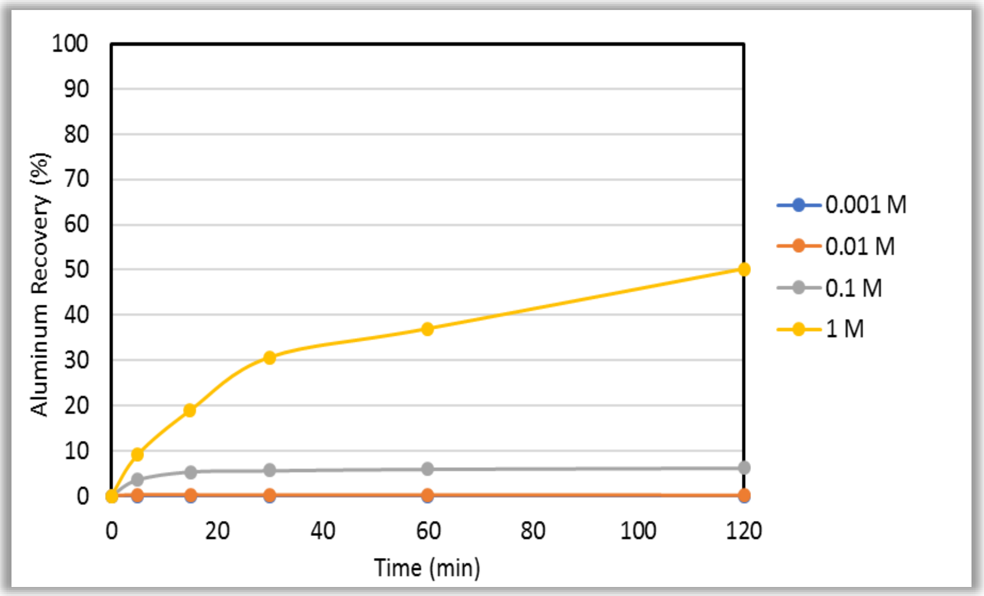
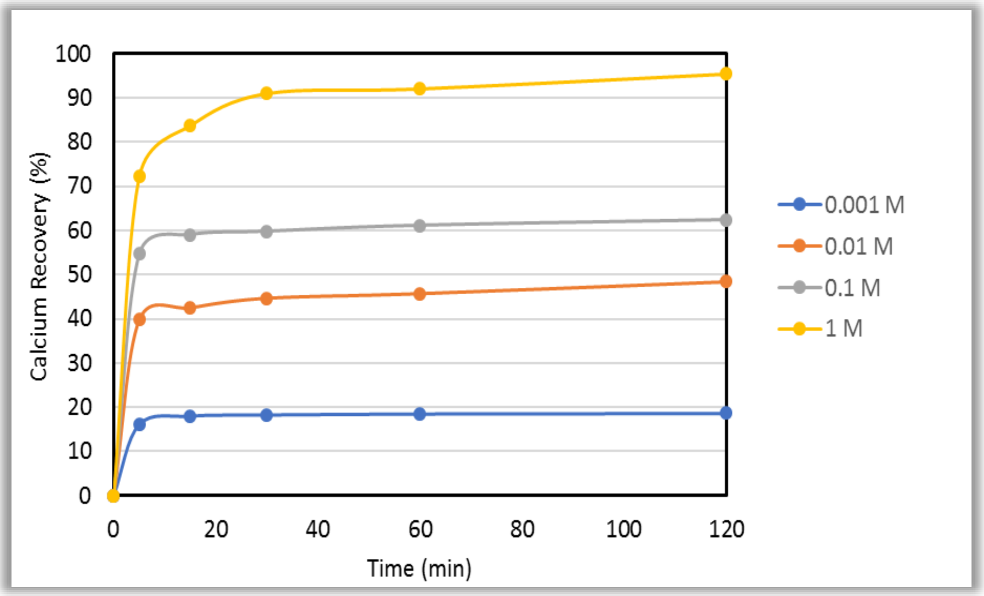


Figure 3-88: Effect of Acid Concentration on Leaching Recovery and Kinetics of Ca for the Roasted 1.8 x 2.2 SG Fraction of the Fire Clay (Hazard No. 4) Coarse Rejects Material (Solids Concentration = 200 gm/L)

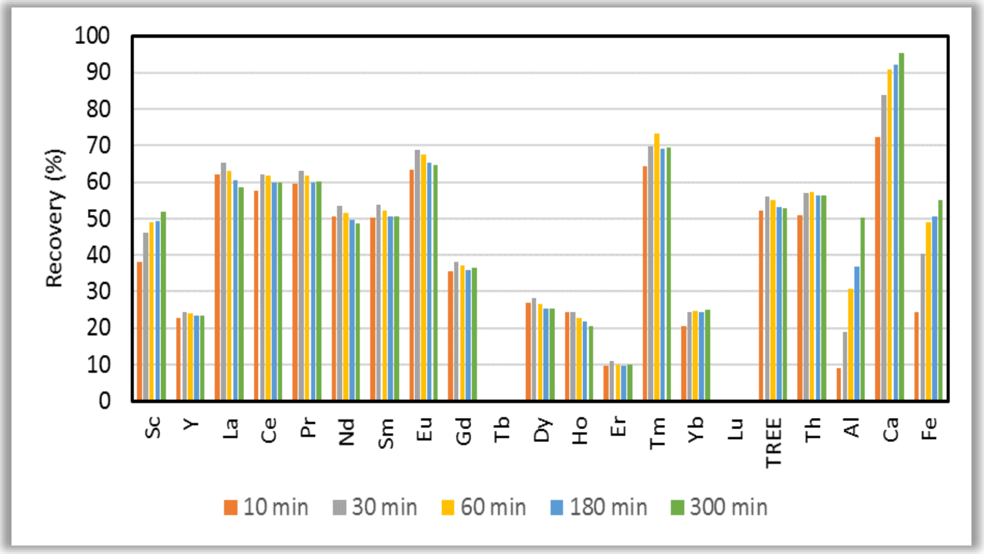


Using an acid concentration of 0.1M reduces the leaching of Fe and Al to less than five (5) percent while also decreasing Ca release by about 25 percent. This benefit along with the cost savings needs to be balanced with the decrease in REE recovery.

3.2.5.4.4 Effects of Acid Leaching Time on the Recovery of REE and Associated Contaminant Elements

For additional clarity, the high acid/high solids data was plotted in the next figure as a function of leach time to show the overall change in recovery beyond 10 minutes.

Figure 3-89: REE and Contaminant Element Recovery as a Function of Leach Time When Using an Acid Concentration of 1M H_2SO_4 and a Solids Concentration of 200 Grams per Liter of Solution

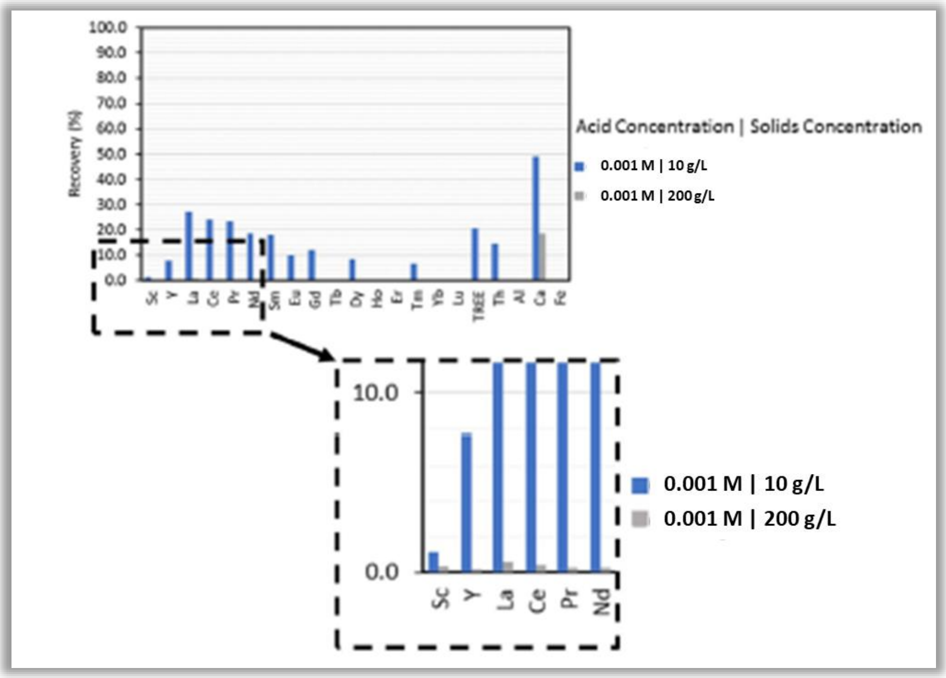


This foregoing graph shows that roasting provides relatively quick leaching kinetics but with limited benefits in exceeding 10 minutes of leaching. The exception to this observation is Sc. In general, by maintaining the leaching time to 10 minutes, the recovery of the contaminants Al, Ca, and Fe is maintained at low values. This aspect is critical for decreasing downstream REE recovery processing costs.

3.2.5.4.5 Effects of Percent Solids in the Leaching Tank Contents on Recoveries of REE and Other Associated Elements from the Fire Clay (Hazard No. 4) Seam Rejects when using Very Low Acid Concentrations

The next figure depicts the effects of percent solids on recoveries when using a very low acid concentration.

Figure 3-90: Effects of Percent Solids on Weak Acid Leaching of Fire Clay (Hazard No. 4) Material



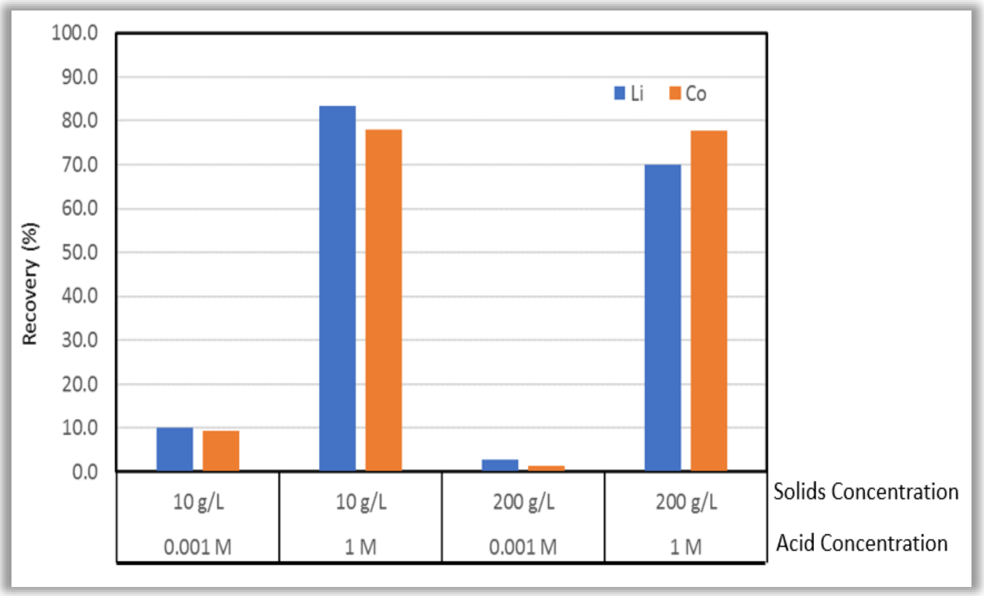
Previous data obtained at higher acid concentrations revealed that solids concentration had little to no effect on REE recovery. However, the above figure clearly shows that the presence of acid consuming minerals such as calcite will determine the critical amount of acid needed to allow leaching of the REEs. Essentially, the graph reflects the fact that the required amount of acid needed to drive the leaching reaction is based on grams of acid used per kg of solid treated. At low acid concentrations, there is only enough acid to drive the leaching reaction for a solution containing a low concentration of solids. When the calcium-containing solids concentration is high, the acid concentration is too low to drive the desired REE leaching reactions as the calcium will preferentially consume most (if not all) of the acid. What acid remains generates only very low REE recovery values.

In summary, the amount of acid required is a function of the concentration of acid consuming solids in the leach tank slurry and the critical amount would be measured in grams of acid per kg of material being treated.

3.2.5.4.6 Evaluation of the Co-Extraction of Lithium (Li) and Cobalt (Co) with REEs from the Fire Clay (Hazard No. 4) Seam Coarse Rejects of the No. 76 Plant (Leatherwood)

The co-extraction of Li and Co with REEs was also discovered as part of the research program. The data from *Figure 3-91* suggests that the higher acid conditions favorably recover both elements. This observation raises the prospects of two byproducts to the proposed REE concentration activities that could be beneficial to other industries.

Figure 3-91: Effect of Acid Concentration and Solids Concentration in Leachate Solution on Li and Co Extraction



3.2.5.5 Evaluation of the Effectiveness of Commercial Roasting Services with Respect to Acid Leaching of REEs from Roasted Fire Clay (Hazard No. 4) Seam Coarse Rejects

Samples of material from a 23-ton crushed bulk sample of Fire Clay (Hazard No. 4) Seam rejects that had been processed through a rotary roaster of a commercial entity were studied in laboratory experiments to assess their leaching characteristics.

3.2.5.5.1 Description of Commercial Roasting Trials Provided by Nex-Gen Industries

Commercial roasting was provided by Nex-Gen Industries which occupies portions of the industrial facility owned by KRP at the Coal Fields Industrial Park near Chavies, Kentucky. Roasting was achieved using the rotary unit depicted in *Figure 3-92*.

Figure 3-92: Roasting Equipment Operated by Nex-Gen Industries

Based on measurements by a hand-held infrared gun, the temperature inside the drum varied between 550°C to 650°C with the first half of the kiln heated. The residence time was calculated to be around 15 to 30 minutes.

During inspection of the roasted material, it was noted that the commercial roasting produced 'gray' and a 'brown' colored material as shown in *Figure 3-93*.

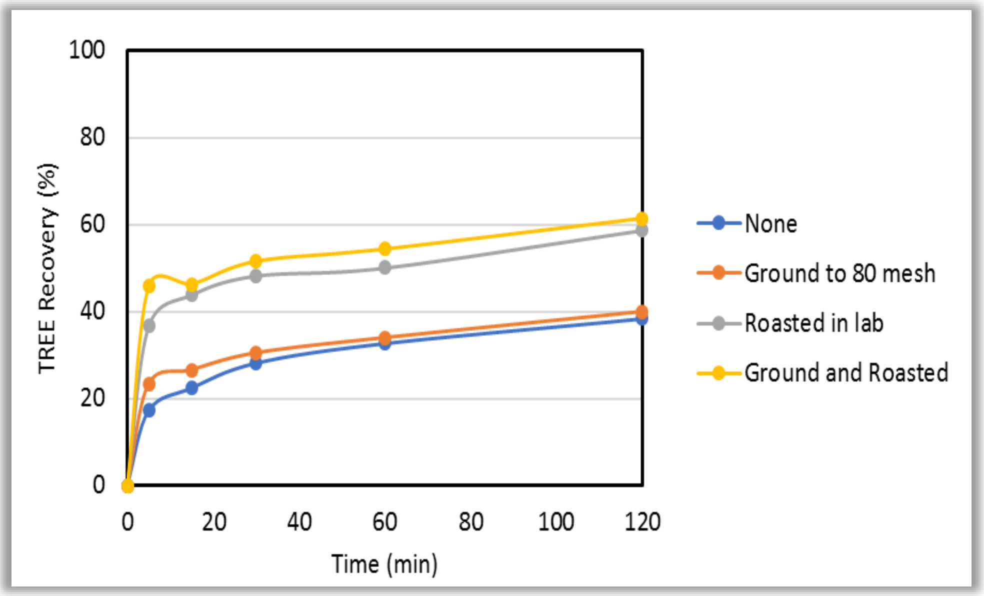
Figure 3-93: Roasted Products Generated by the Commercial Roasting of Fire Clay (Hazard No. 4) Seam Crushed Coarse Rejects Material

3.2.5.5.2 Comparison Laboratory Testing of the Leaching Characteristics of the Commercially Roasted Material

Laboratory leaching tests were conducted on both material types with and without additional roasting at 600^{O}C in a muffle furnace for two hours. The results were compared with the findings from treating un-roasted material obtained from the same bulk sample. All materials were initially finer than 1 mm and the additional testing involved a series of tests that investigated the benefit of reducing the top particle size to 212 microns (80-mesh).

Figure 3-94 shows the leaching characteristics obtained on the comparison Fire Clay (Hazard No. 4) Seam coarse rejects sample that was not treated in the commercial roaster. The results show that roasting the material in the laboratory at 600^{O}C provided improved leaching characteristics with REE recovery values approximately 20 absolute percentage points higher relative to the results obtained from the unroasted material. Reducing the particle size from 1 mm to 0.212 mm provided a minimal improvement.

Figure 3-94: Leaching Characteristics of Fire Clay (Hazard No. 4) Seam Coarse Rejects that was not treated by the Commercial Roaster (Acid Concentration = 1M H_2SO_4)



The leaching characteristics of the ‘gray’ and ‘brown’ commercially roasted materials are provided in *Figure 3-95* and *Figure 3-96*. REE recovery values of around 40 percent were obtained. Grinding the commercially roasted sample to a smaller particle size and submitting that sample to additional roasting had no effect on REE leachability.

Figure 3-95: Leaching Characteristics of Roasted 'Gray' Fire Clay (Hazard No. 4) Seam Crushed Coarse Rejects Treated by the Commercial Roaster (Acid Concentration = 1M H_2SO_4)

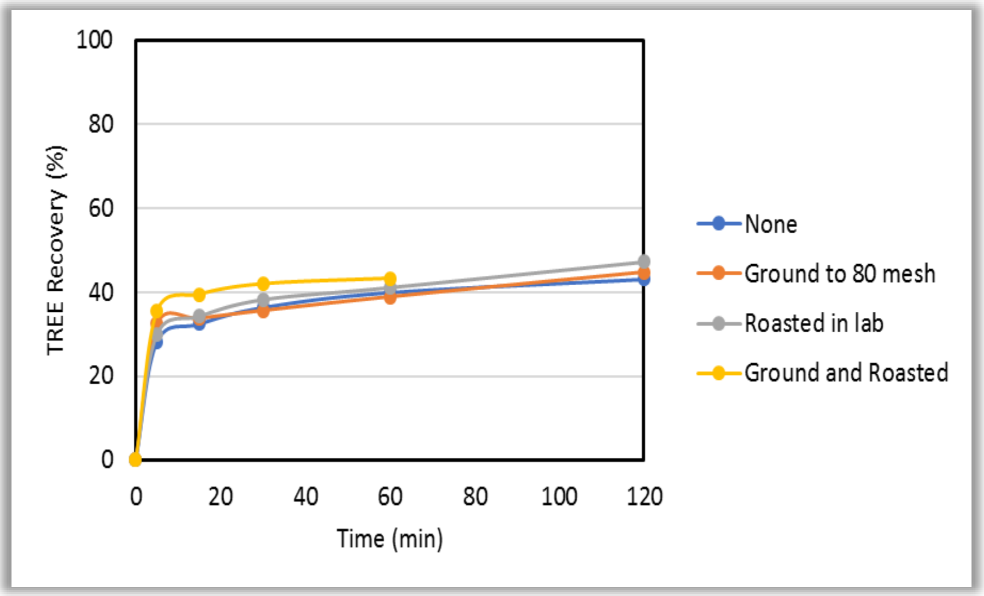
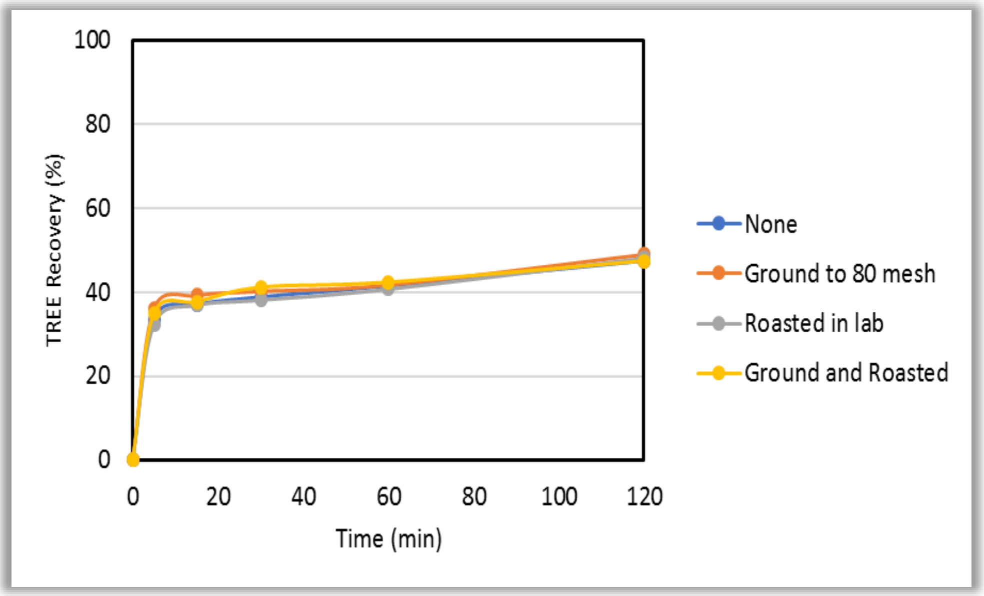


Figure 3-96: Leaching Characteristics of Roasted 'Brown' Fire Clay (Hazard No. 4) Seam Crushed Coarse Rejects Treated by the Commercial Roaster (Acid Concentration = 1M H_2SO_4)



3.2.5.5.3 Preliminary Conclusion of the Effects of Commercial Roasting Services by Nex-Gen Industries

Based on examination of the comparison testing results, it was preliminarily concluded that the commercial roasting process generated irreversible effects on the material which limited REE recovery values to levels 20 percentage points lower than that achieved from a comparison sample. That causative chemical and physical processes were not established. Further work with the Nex-Gen Industries roaster was deferred until a better understanding could be collected. All work on MMA 29956 proceeded thereafter with the use of a relatively large capacity laboratory-style roaster.

3.3 Solvent Extraction

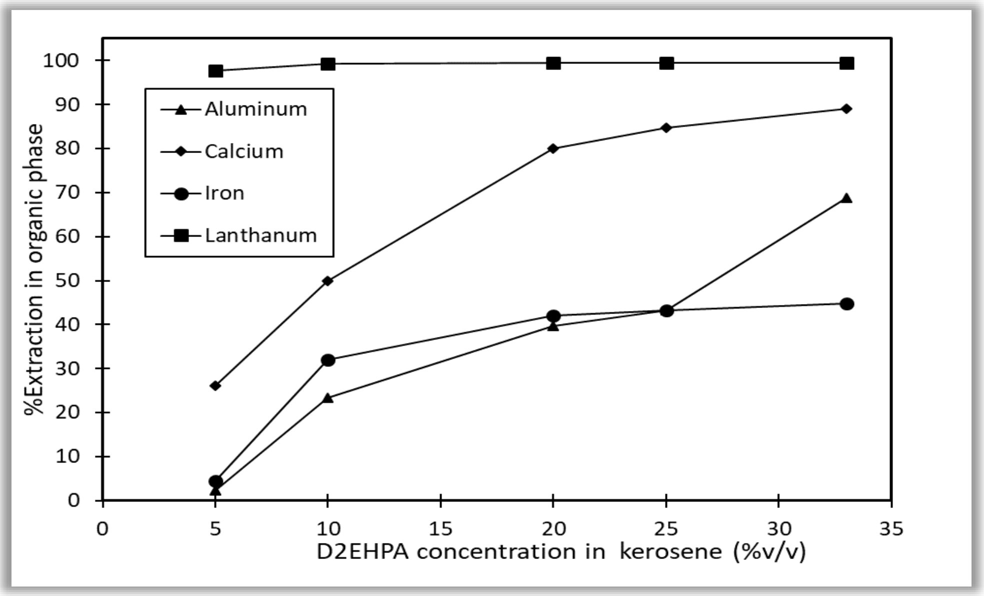
3.3.1 Solvent Extraction Rougher/Cleaner Testing

3.3.1.1 Extractant Concentration Effect on Solvent Extraction

D2EHPA in kerosene adds significant cost to the solvent extraction process. To better understand the effect of D2EHPA concentration on the extraction efficiency of rare earth metals and the contaminant metals, single stage extraction tests with the concentration of DEHPA varying from 5 percent v/v to 33 percent v/v were performed. For this experiment the contact of time of 15 minutes was chosen to allow the reaction to reach equilibrium values and the pH was selected as 2.0.

As shown in the *Figure 3-97*, it was observed that the extraction of rare earths reaches the maximum at 5 percent concentration at which point the extraction of aluminum and iron is not observed and extraction of calcium is less than 20 percent. From the result, it can be deduced that once all the rare earths are extracted in the organic phase, any additional extraction in the organic phase only contributes to extraction of contaminants. The ideal concentration of DEHPA in kerosene for extraction can be calculated using stoichiometry in the extraction reaction based on the concentration of rare earth in the leachates which is different from source to source and condition to condition.

Figure 3-97: Impact of D2EHPA Concentration on the Metal Ion Extraction at pH 2 using Solvent Extraction of an Artificial Mixture

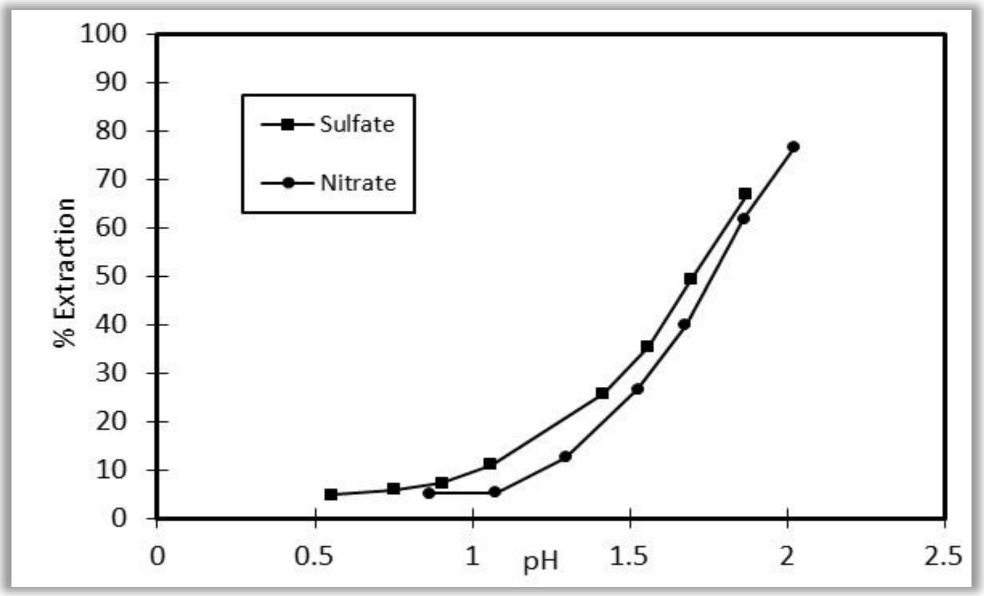


3.3.1.2 Effect of Different Anions on Solvent Extraction

Another aspect requiring study is the use of different acids for utilization with solvent extraction. Tests were performed to determine if there were extreme or adverse performance issues associated with different acid use in the solvent extraction circuit.

The results from one experiment in which the extraction behavior of calcium in two different anionic media was compared are shown in the following figure.

Figure 3-98: Effect of Anionic Species on the Extraction of Metal from an Artificial Mixture



It was observed from the forgoing figure that the extraction curve for the metal shifts to the left when the anionic media is changed from nitrate to sulphate. The extraction of metals in the organic phase is dependent on the anionic species present in the leachate. As the extraction reaction is an equilibrium reaction, the final concentration of the species is determined by the activities of the species in the system. Different anionic species will have different activities and consequently the extraction properties of a metal are affected by the anionic species. This means that the acid used for leaching the solids will have a significant effect on the extraction behavior of the metal. This is also true of scrubbing and extraction steps.

3.3.1.3 Bench-Scale Testing of the Solvent Extraction Proposed for this Project

To assist in the design and feasibility of the hydrometallurgical flowsheet solvent extraction testing was undertaken to determine key parameters and performance of the solvent extraction system. The tested flow sheet consisted of two stage SX with a rougher and a cleaner stage.

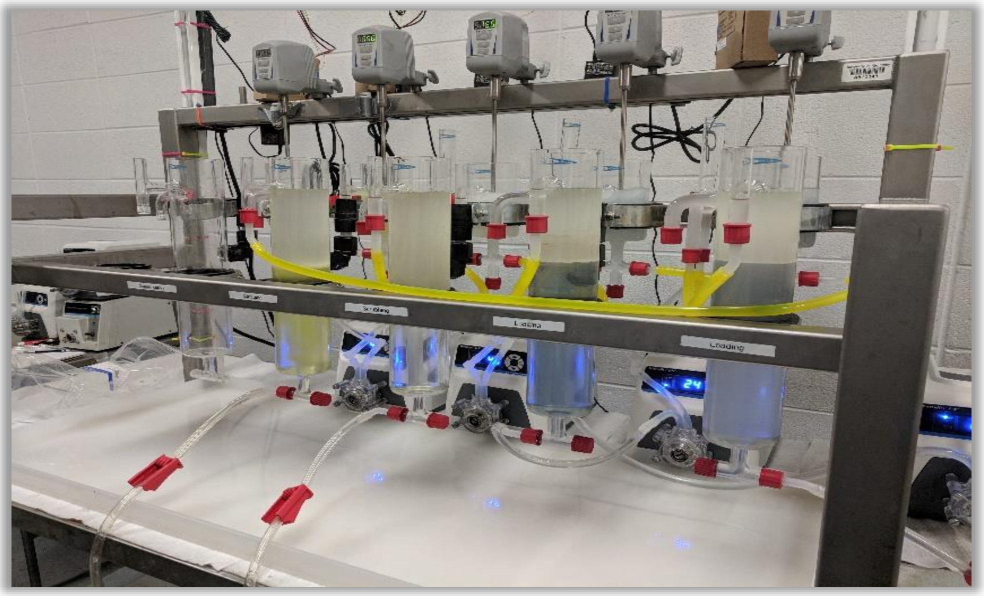
3.3.1.3.1 Laboratory Set-up of Rougher Loading, Scrubbing and Stripping

To perform the SX bench-scale test a setup comprising of six sets of mixer settlers was used to test different operating parameters of the designed flowsheet.

For the roughing stage four sets of mixer-settlers were utilized (two for loading step, one for scrubbing step and one for stripping step) to conduct the experiment on a continuous basis. Cole-Parmer Model 07528-10 peristaltic pumps with variable speed were used to control the flow of the aqueous phase streams whereas the organic phase was driven from one stage to another by gravity and recirculated using a pump. Variable speed agitators (Caframo Model BDC 250) were used to mix the aqueous and organic phase in the mixers. The interface of organic and aqueous phase was maintained by adjusting the pump flow rate and the drain flowrate. Sampling points were provided at each aqueous stream for sample collection for ICP analysis.

The following figure displays the experimental setup with four active mixer-settler for continuous roughing operation.

Figure 3-99: Lab Scale Solvent Extraction Setup for Conducting the Continuous SX Tests at the Hydrometallurgical Laboratory at the University of Kentucky



A 5 percent (v/v) solution of Di-(2-ethylhexyl)phosphoric acid (DEHPA) in SX Orform^{TM} as a diluent was used as an organic extractant for the continuous test. Analytical grade ascorbic acid was used to reduce the iron from Fe^{3+} to Fe^{2+} and a 10-M NaOH (sodium hydroxide) solution was used to regulate the pH of the SX feed to the desired value. A 0.1-M solution of HCl was used for scrubbing calcium from the loaded organic phase and a 6-M solution HCl was used to strip the metals from loaded organic phase to aqueous phase.

3.3.1.3.2 Continuous Flow SX Testing using Pocahontas No. 3 Coal Seam Preparation Plant Reject Samples

For the SX testing, the middling fraction of the Pocahontas No. 3 Coal Seam preparation plant rejects was ground in a batch attrition mill. About 400g of 80-mesh coal middlings were ground in the mill for 30 min at 10 percent solids by weight. The ground coal was decarbonized in a column floatation cell using fuel oil #2 as a collector and MIBC as a frother.

The following figure depicts the experimental column flotation setup utilized for sample decarbonization.

Figure 3-100: Column Flotation Setup used to Decarbonize the Various Coal Rejects Prior to Leaching

The tailings from the column flotation were filtered using a pressure filter system and air dried. Subsequently, the dried solids were leached in a leaching reactor (*see the following figure*) with 2.0 M HCl solution at 75°C for 2 hours at a slurry concentration of 20 percent solids.

Figure 3-101: Leaching Reactor used in the Hydrometallurgy Lab to Leach the Solids from Various Coal Reject Sources

The temperature of the leaching vessel was controlled by circulating a heated solution of ethylene glycol in a coil through the solution.

The leached solids were dewatered using overnight settling followed by vacuum filtration.

The leachate was processed through the rougher stage using the bench-scale SX setup comprising of four sets of mixer settlers. *Figure 3-102* shows the experimental schematic with four mixer-settlers with two being used for loading of REEs, one for scrubbing of contaminants and one for stripping of REEs.

Figure 3-102: Laboratory Schematic for the Rougher Circuit as Implemented on Bench-Scale SX-Setup for Various Coal Rejects Sources

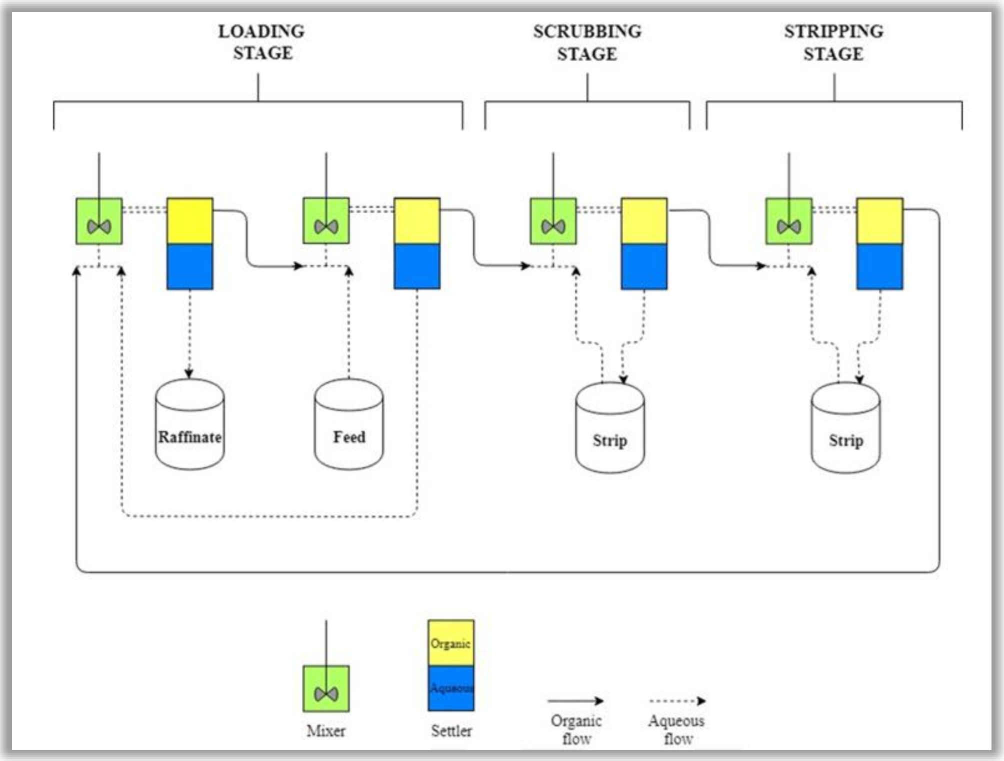


Figure 3-103 shows the bench-scale implementation of the flow sheet shown in *Figure 3-102*. Given the 2.0 M HCl solution used during the leaching process, the feed solution to the solvent extraction was highly acidic with a pH value of less than zero. As such, a 10 M NaOH solution was used to regulate the pH to 1.90, which was the optimum pH value for loading the REEs into the organic phase.

Figure 3-103: Bench-Scale Setup for Implementing the Rougher and the Cleaner Circuit of Solvent Extraction of Various Coal Rejects Sources

3.3.1.4 Evaluation of Extraction Characteristics of the REEs and Contaminant Elements

The extraction characteristics of the REEs and the contaminant elements was ascertained by conducting bench-scale shakeout tests.

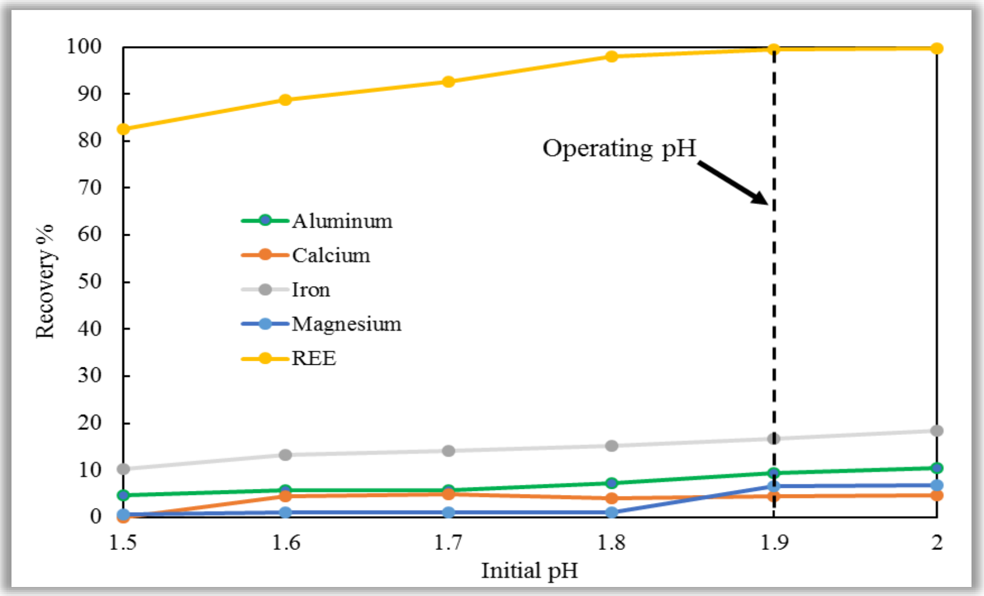
3.3.1.4.1 Rougher Stage Activities

The leach liquor was contacted with five (5) percent (v/v) solution of DEHPA in Orform for 15 minutes for a range of initial pH values. The metal content in the raffinate was analyzed using ICP-OES and the amount of corresponding metal recovery was calculated by mass balance.

For the rougher stage emphasis was given on maximizing the recovery of REEs as the co-extracted contaminants were removed in the cleaning circuit.

The next figure displays the results of the recovery of REEs and co-extraction of contaminants from the Pocahontas No. 3 Coal Seam preparation plant middlings circuit rejects as a function of feed solution pH.

Figure 3-104: Recovery Values of Metal Ions in the Organic Phase for the Leach Liquor with Five (5) Percent DEHPA Solution (Pocahontas No. 3 Coal Seam Preparation Plant Reject Material)

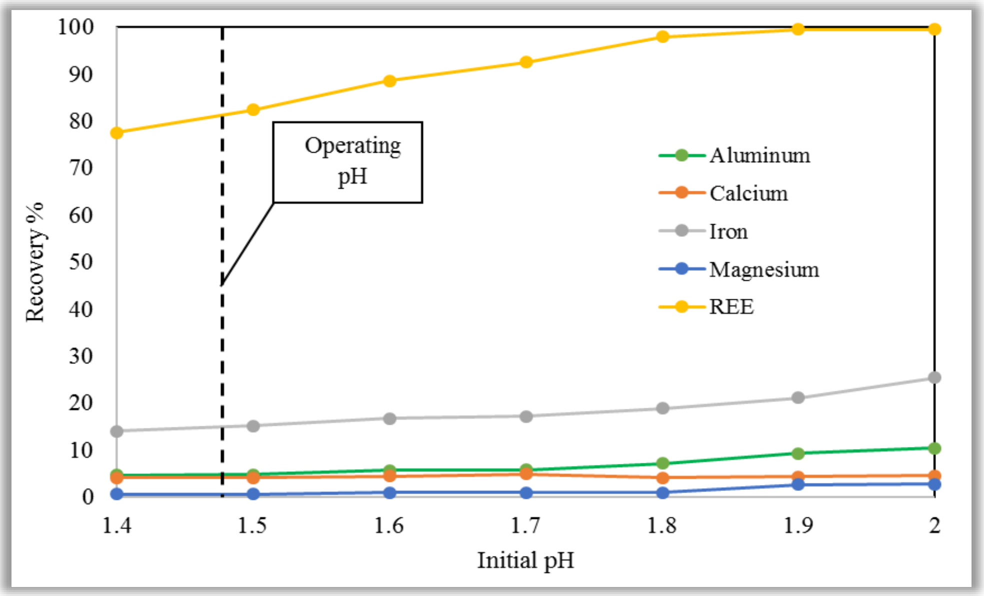


The preceding figure shows a small increase in contaminate as the pH raises to a near 100 percent extraction of REEs. Based on the extraction curves the ideal initial pH for the rougher stage to operate was selected 1.90 for maximum recovery of REEs. The REE recovery obtained at 1.9 pH was 99.56 percent and the recoveries for the contaminants were 9.38 percent for aluminum, 4.45 percent for calcium, 18.46 percent for iron and 6.69 percent for magnesium.

<u>3.3.1.4.2 *Extraction of Metal Ions in the Stripped Solution from the Rougher Stage*</u>

From the roughing stage a stripped solution containing REEs and contaminants was obtained and analyzed for metal content using ICP-OES. Bench shakeout tests were repeated for the stripped solution for a range of initial pH values to select the appropriate feed pH. The emphasis for the cleaner stage was not on the maximizing of recovery but on maximizing the rejection of the contaminants present in the solution. The following figure reports the results of these tests.

Figure 3-105: Extraction Curves of Metal Ions in the Stripped Solution Obtained from the Rougher Stage of the Extraction Process (Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit Rejects)



Based on the extraction curves shown in *Figure 3-105*, the initial pH was selected to be 1.48. This value showed an appropriate value of the recovery of REEs with the rejection of contaminants.

The REE recovery obtained at 1.48 pH was 82.46 percent and the recoveries for the contaminants were 4.7 percent for aluminum, 4.24 percent for calcium, 15.46 percent for iron and 1.06 percent for magnesium.

The 17.64 percent of TREEs not recovered at this stage will not be lost as the entire raffinate stream will be recirculated to the leaching circuit allowing the recovery of those TREEs.

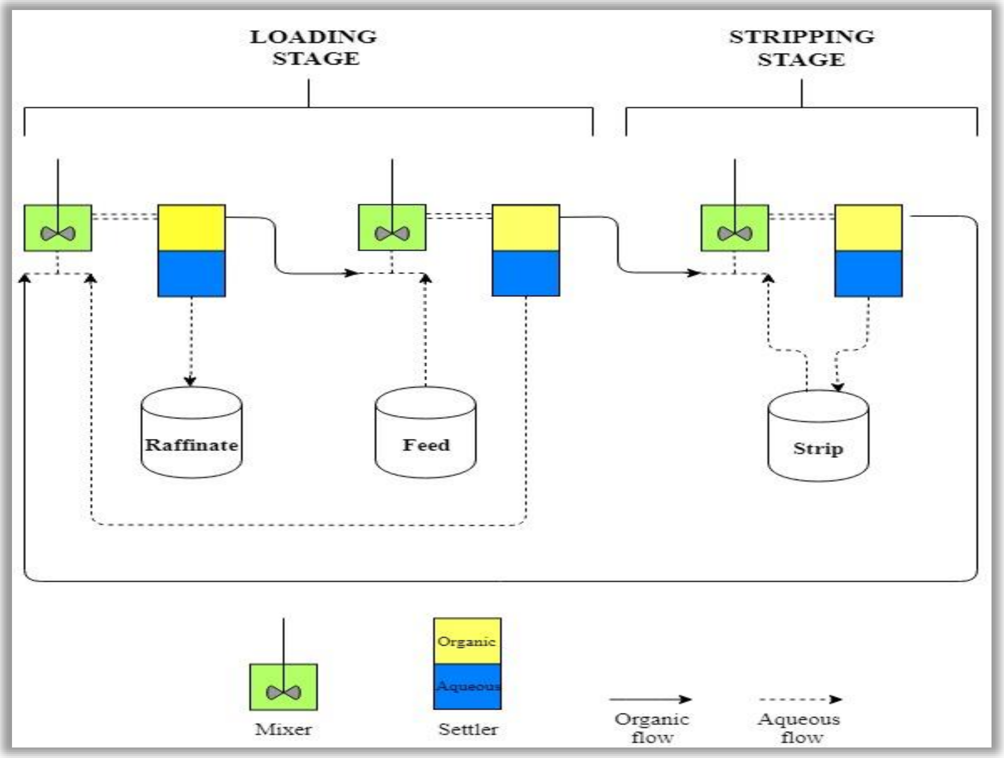
3.3.1.4.3 Cleaner Circuit Simulations

The SX setup was re-plumbed to run the stripped solution from the rougher circuit through the cleaner circuit.

In the re-plumbed circuit, only three sets of mixer settlers were operated, two for loading and one for stripping. The scrubbing stage was eliminated from the cleaner circuit as most of the calcium was eliminated during the scrubbing in the rougher stage.

The stripped solution was neutralized to 1.48 before feeding to the cleaner circuit. The pH was selected to keep the co-extraction of the contaminants to a minimum. The lab schematic of the cleaner circuit is shown in the following figure.

Figure 3-106: Laboratory Schematic for the Cleaner Circuit as Implemented on Bench-Scale of the Pocahontas No. 3 Rejects Samples



The stripped solution from the cleaner circuit was analyzed for metal content using ICP-OES. The relative concentration of TREE with respect to contaminants increased from 0.002 (*see Figure 3-107*) in the leach solution to 0.171 in the rougher stripped solution *(see Figure 3-108*) and to 0.531 in the cleaner stage stripped solution (*see Figure 3-109*). Only one cleaner stage was employed in the flowsheet as the composition of the final stripped solution has low enough concentration of contaminants to generate a high purity REO using oxalate precipitation.

Figure 3-107: Concentration of Contaminants and TREE in the SX Feed for the Rougher Stage of Extraction (Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit Rejects)

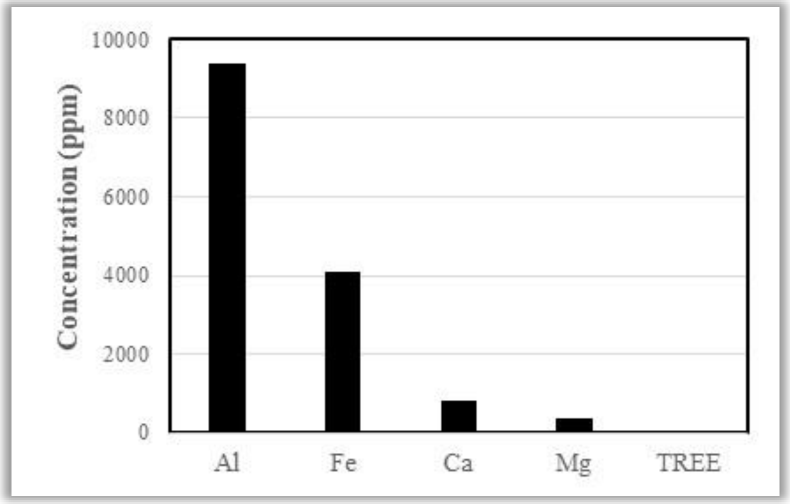


Figure 3-108: Concentration of contaminants and TREE in the Stripped Solution Obtained from the Rougher Stage of Extraction (Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit Rejects)

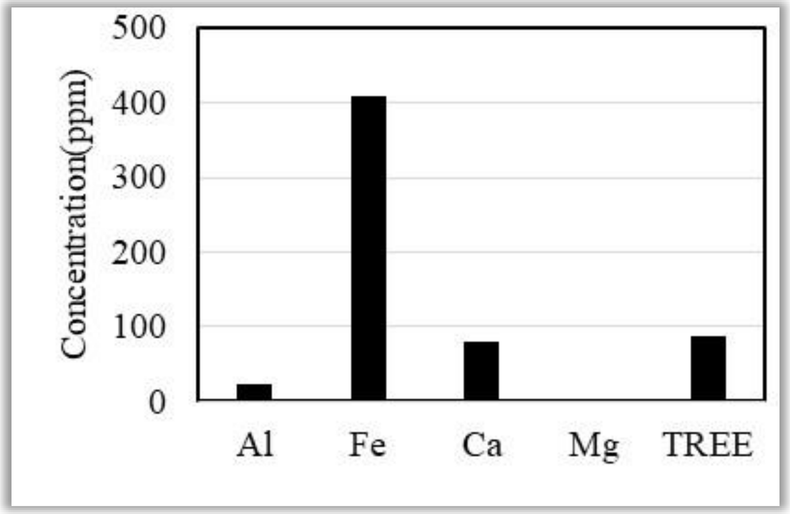
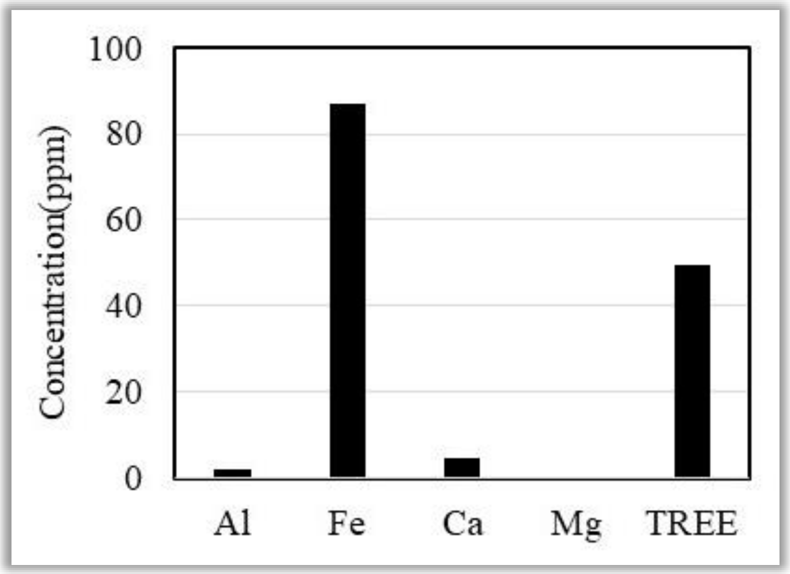


Figure 3-109: Concentration of Contaminants and TREE in the stripped Solution Obtained from the Cleaner Stage of Extraction (Pocahontas No. 3 Coal Seam Preparation Plant Middlings Circuit Rejects)



3.3.1.4.4 Precipitation of REO From Stripped Solution by use of Oxalic Acid

The stripped solution from the cleaner stage extraction was then subjected to oxalate precipitation. Since the REEs were stripped from the organic phase using 6 M HCl, the pH of the stripped solution was adjusted to 1.070 to optimize precipitation performance by adding a solution of 10 M NaOH dropwise. Subsequently, a 1.77 M solution of oxalic acid was added dropwise until the REEs precipitated as oxalates. The oxalate precipitate produced was roasted at 750℃ for 120 minutes to produce rare earth oxide as the final product.

3.3.1.4.5 Analysis of the Final Rare Earth Oxide Product

The rare earth oxide was analyzed for TREE content. The oxide contained 74.92 percent TREEs by weight which corresponds to 89.69 percent TREO by weight.

Of the TREEs in the product, 78.81 percent were LREE while the HREEs contributed 21.19 percent. The corresponding H to L ratio of the final product was 0.27 which was similar to 0.24, that of the leachate, the feed for the SX systems.

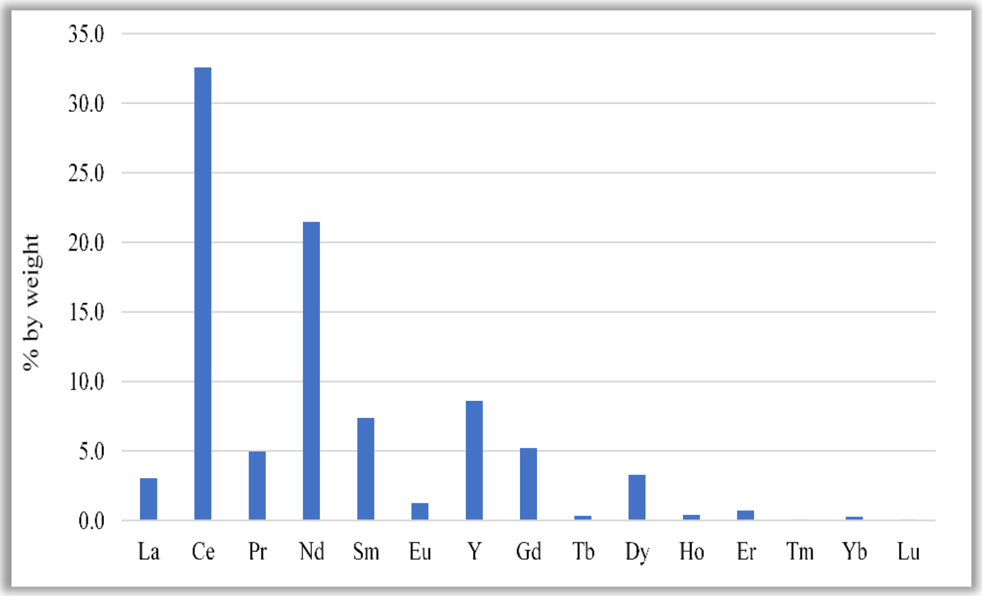
The following table (*Table 3-64*) contains the elemental results of the final REO product with *Figure 3-110* showing the REO weight percent by element. This figure is interesting in that is reveals the relative distribution of elements in the REO produced.

It is important to note the Sc is missing from both *Table 3-64* and *Figure 3-110*. This is because Sc must be recovered in separate processing steps and will be discussed in subsequent sections.

Table 3-64: Elemental Analysis of the Final REO Generated from Pocahontas No. 3 Coal Seam Preparation Plant Middlings Reject Material

Element	REE in Solution (ppm)	REO (ppm)
Lanthanum	26,040	30,519
Cerium	265,200	325,745
Praseodymium	41,176	49,741
Neodymium	184,080	214,637
Samarium	63,536	73,676
Europium	10,896	12,616
Yttrium	67,736	8,6018
Gadolinium	45,152	52,042
Terbium	3,110	3,468
Dysprosium	28,744	32,989
Holmium	3,466	3,970
Erbium	6,274	7,173
Thulium	712.8	814
Ytterbium	2,494	2,838
Lutetium	658	749
Total	**749,274**	**89,6995**

Figure 3-110: Elemental Distribution of REEs in the Final REO Generated from the Pocahontas No. 3 Coal Seam Preparation Plant Middlings Reject Material



3.3.1.4.6 *Ascorbic Acid Optimization to Control Iron (Fe) Contamination of the Leach Liquor from the Pocahontas No. 3 Coal Rejects*

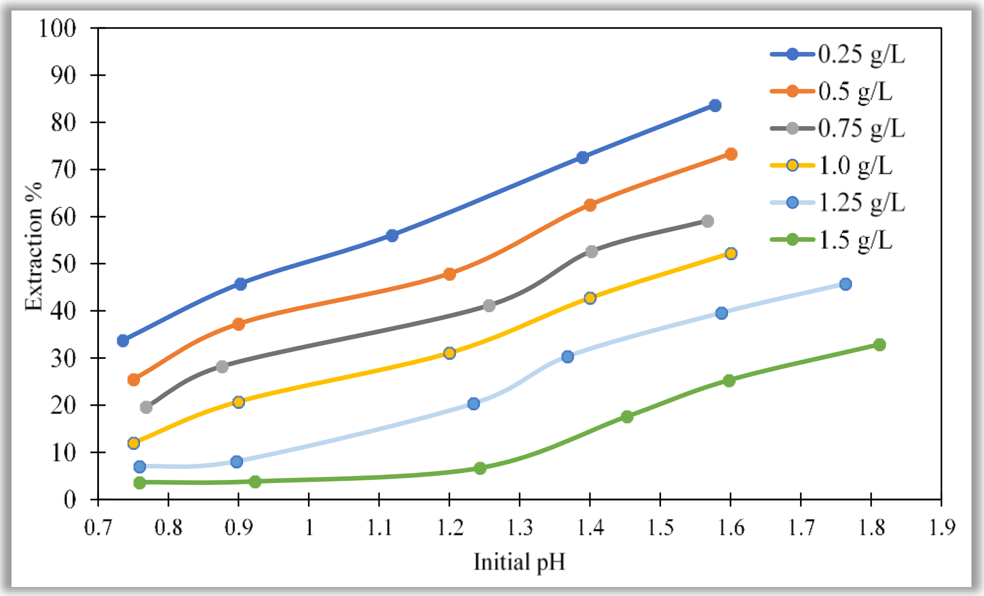
The investigators noted that their SX feed, i.e., leach liquor, often contained a significant concentration of iron in solution especially when compared to the concentration of REE ions. The investigators concluded that leaching the feed solids at an elevated temperature in concentrated sulfuric acid created a strong oxidizing environment. Furthermore, when iron inherent to the solids was extracted and solubilized, it occurred predominantly in a trivalent state i.e., Fe^{3+} instead of Fe^{2+}. The organic extractant DEHPA has a stronger affinity for trivalent ions as compared to divalent version in solution due to higher charge density. As the concentration of Fe^{3+} was orders of magnitude larger than that of REE ions in solution, there was coextraction of iron in the organic phase leading to contamination of the final REO product. To avoid this, it was necessary to reduce the iron in solution to its divalent state Fe^{2+} using a strong reducing agent, i.e., Ascorbic acid. The Ascorbic acid reduces the iron from its Fe^{3+} state to the Fe^{2+} state by the following mechanism.

$$2Fe^{3+} + H_2A = 2Fe^{2+} + A + 2H^+$$

In this project, test work was carried out to optimize the dosage of the reducing agent to minimize the co-extraction of the iron in its organic phase without overdosing. It was critical to prevent the overdosing of Ascorbic acid as it was one of the largest cost components of the SX process.

To simplify the test, an artificial solution of 1.2 g/L Fe^{3+} ion using $FeSO_4$ was prepared in de-ionized *(DI)* water. Extraction curves of Fe with initial pH were generated for different dosages of reducing agent. A concentrated solution of Ascorbic acid was prepared in DI water by dissolving 10g Ascorbic acid in 50 mL water. The solution was added to the $FeSO_4$ solution incrementally by pipette. As the dosage of Ascorbic acid was increased, the extraction curve of the Fe shifted to a higher pH. The optimal dosage was determined at the point after which the curve stopped shifting, indicating a complete reduction of Fe^{3+} to Fe^{2+}. Based on the test results the optimal dosing for Ascorbic acid was determined to be 1.5 g/L. The dosing which was used for the previous economic analysis was 25 g/L and based on the optimization test, the Ascorbic acid cost was reduced by an order of magnitude. These results are shown in the following figure.

Figure 3-111: Impact of Different Dosing of Ascorbic Acid on Extraction Curves of Iron with Five (5) Percent DEHPA and 10 percent TBP (Artificial Solution)



3.3.1.4.7 Scandium (Sc) Recovery from Pocahontas No. 3 Seam Middlings Rejects

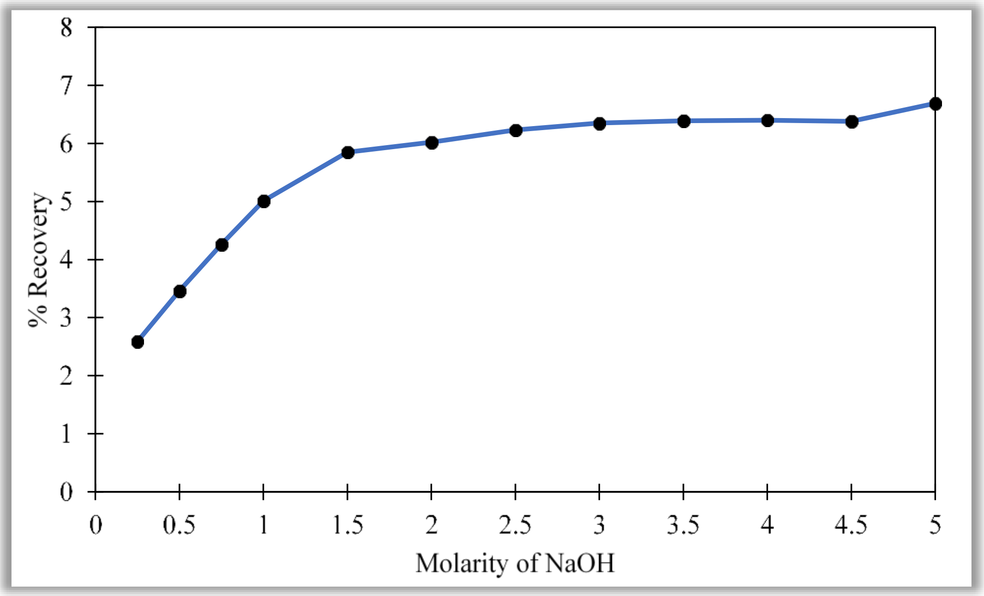
Scandium is one of the most valuable REEs because of its unique chemical and physical properties. Though not a lanthanide, it is grouped together with REEs because of its similar chemical properties and occurrence. Scandium, however, has a relatively small ionic radius and therefore has a higher charge density than the other REEs. Because of this characteristic scandium exhibits different extraction behavior as compared to other REEs. The resulting chemical complex formed with scandium and DEHPA is more stable and harder to strip.

Scandium is extracted at a much lower pH relative to REEs and it does not report to the aqueous phase when stripped with a strong acid. Consequently, the scandium builds up in the organic phase. To recover the scandium from the organic phase, the organic phase should be saponified using a strong NaOH solution. The Na ion overloads the organic phase and releases the scandium ions back into solution.

One of the hurdles of using a saponification stage is a third phase formation due to high loading of sodium ions in the organic phase. The third phase restricts phase separation and leads to disturbed emulsion stability. To rectify the third phase formation, a phase modifier like Tri-n-Butyl Phosphate *(TBP)* is used. The phase modifier forms a hydrophobic complex with the metal ions in the organic phase preventing the third phase. The phase modifier TBP, also demonstrated an effect on the extraction behavior of REEs. Based on the test results it was observed that the extraction curves of individual rare earth elements tended to spread out, i.e., the extraction curves of lighter REE shifted to higher pH with no perceptible difference in the extraction behavior of heavier REEs. Therefore, the phase modifier can be used to facilitate the individual separation of REEs.

Test work was carried out to determine the strength of the saponification solution required to recover scandium from the solution. The leach liquor obtained from the Pocahontas No. 3 Coal Seam preparation plant middlings rejects was used as a test feed stock. The organic phase comprising of 5 percent DEHPA and 10 percent TBP by volume in SX Orfom was loaded up with REEs using a contact time of 15 minutes. The REEs (except Sc) were stripped out using a 6M HCl solution. The organic solution loaded with scandium was saponified with different strength solutions of NaOH. The saponified solution was then analyzed by ICP. From the test result, it was determined that 2M NaOH is an optimum concentration for the scandium recovery as the scandium recovery peaked at that concentration. The scandium recovery for 2M NaOH was 6 percent by mass present in the organic phase. However, the scandium recovery in a continuous setup is expected to be much higher as its concentration in the organic phase will be built up as the test is continued for longer durations. The graphic results of this test are shown in the following figure.

Figure 3-112: Effect of Molarity of NaOH used for Saponification for the Recovery of Scandium (Sc) (Pocahontas No. 3 Coal Seam Middlings Rejects)

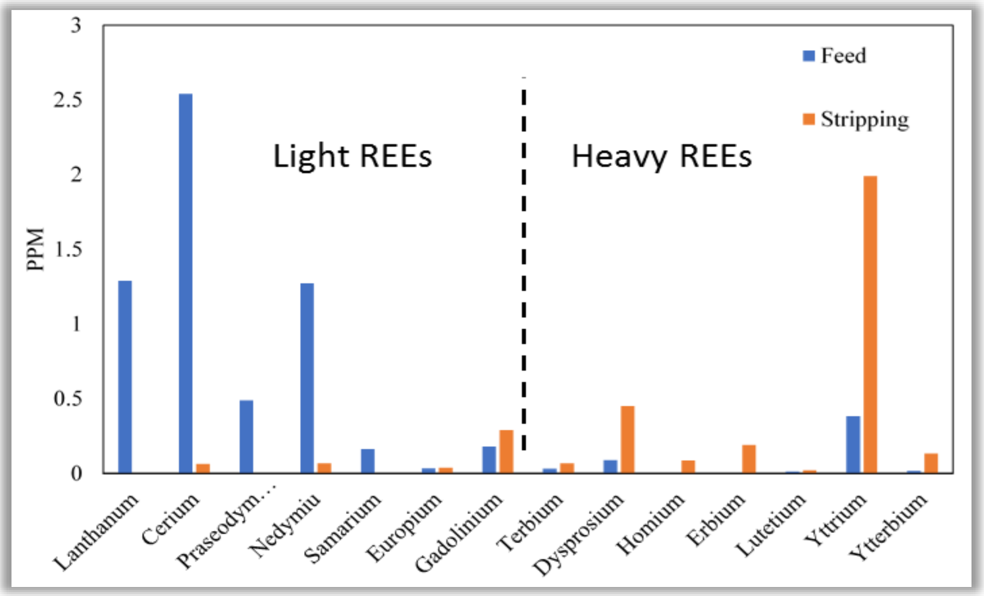


3.3.1.5 Selective Loading of REEs in Solvent from Ferroglobe's Blue Gem Seam Tailings Materials

The tailings produced from the HHS process (courtesy of VT) from Ferroglobe's Blue Gem Seam were used as a feed stock for the SX circuit. The tailings were roasted at 750°C in a muffle furnace to improve the leaching kinetics. The solids were leached with 1.6 M H_2SO_4 at 75°C for 15 minutes. The leachate generated was processed by the bench-scale rougher SX circuit. The unit was plumbed to run the predesigned rougher flowsheet with two stages for loading the REEs, one stage for scrubbing the contaminants and one stage for stripping the REEs (*see the previously discussed Figure 3-102*).

The operational parameters of the process, i.e., the composition of the organic phase and initial pH were adjusted based on the test results. An organic solvent of 5 percent DEHPA, 10 percent TBP was used in SX Orfom to run the test. The initial pH for this test was 2.1 instead of 1.9 due to the changed extraction behavior. The stripped solution was analyzed using ICP-OES. Based on the results it was observed that an effective separation between heavy REE and light REE was made. The H-to-L ratio was 0.122 in the SX Feed and was 19.305 in the stripped solution. There was 95.07 percent of TREE in the stripped solution were HREE and only 4.93 percent were LREE. The HREE were selectively loaded in the organic phase while the LREE were separated. Two separate final products composed primarily of heavy and light rare earth elements, respectively, were generated.

Figure 3-113: Distribution of REEs in Ferroglobe's Gatliff Plant (Blue Gem Seam) Feed versus Stripping Showing Selective Loading of Heavy REEs in the Organic Phase[15]



The saponification step was implemented on a batch scale due to equipment limitations. A recovery of around 6 percent of scandium was achieved in the batch scale. Due to the low concentration of scandium in the feed stock, a final product could not be generated. To generate a high-grade scandium product a larger feed stock is required. Testing of Sc recovery in a pilot plant will be necessary to produce enough Sc to determine final purity.

3.3.1.6 Characteristics of Individual REEs with Extractants

3.3.1.6.1 *Effects of Extractants and Parameters on Individual REEs Collected from W. Ky No. 13 (a.k.a., Baker) Coal Seam Acid Mine Drainage (AMD)*

As the purpose of this project is to purify at least three rare earth elements to a purity of greater than 90 percent, the correct design of the solvent extraction parameters is critical. The UK has been working to develop chemistries that enhance the separability of REEs. To this end, a single extractant mixture was tested to evaluate the potential for REE separations.

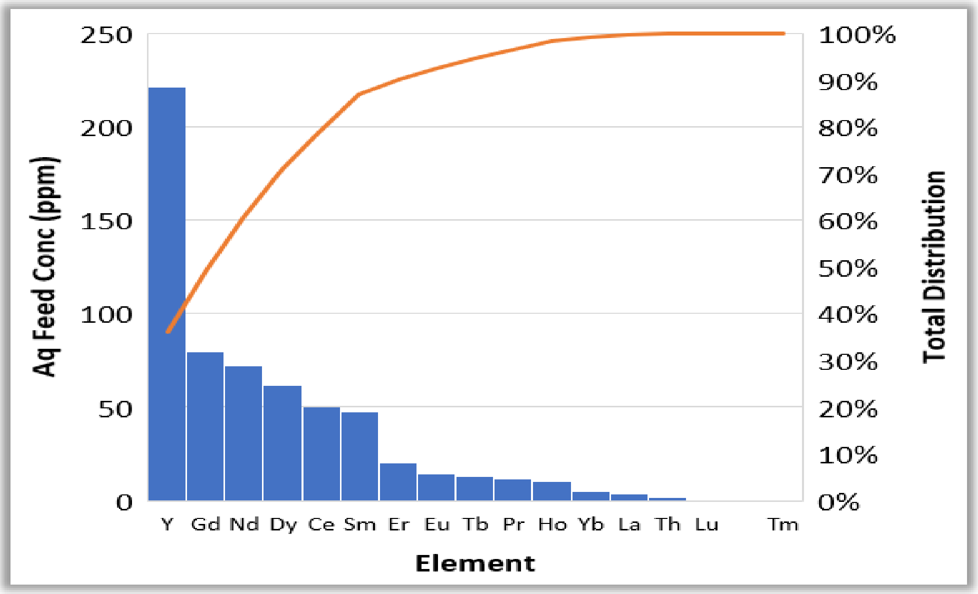
To perform this test, a solution was created from an inventory of REOs which had been collected over time from a number of laboratory tests of acid mine drainage (*AMD*) from mine workings and reject embankments associated with the W. Ky No. 13 coal seam (Baker)[16]. To create this solution, 1 g of this W. Ky. No. 13 based REO concentrate was added to a water-HCl mixture for re-dissolution. This prepared solution (as shown in the next figure) is depicted as a Pareto graph of the REE distribution in the test solution.

---

[15] Delineation of Light and Heavy REEs in accordance to USGS

[16] Personal Communication with Dr. Honaker, Ph.D., P.E., This work was performed using an REO sample of high percent purity generated from W. Ky. No. 13 coal seam AMD at a 0.25 ton per hour proof-of-concept pilot plant in West Kentucky operated by the University of Kentucky.

Figure 3-114: Aqueous Rare Earth Element Concentration for Test Solutions (Using REO from W. Ky. No. 13 Coal Seam AMD



Solvent extraction shakeout tests were then performed by varying the pH of the solution and observing the remaining elements in the aqueous solution.

Figure 3-115 shows the results of the experimentation with reasonable separation curves between elements.

In *Figure 3-116*, the greatest six elements in terms of concentration are presented separately to display the differences in separation. Particularly good are the separations of Y from Dy and Gd.

Further for reference, *Figure 3-117* is included to show the differences between the University of Kentucky extractant blend and the manufactures data for Cyanex 572.

It is important to note that the respective concentrations as part of the testing was different between the two data sets. However, the comparison is useful in demonstrating the enhanced separation between high concentration elements. This type of differentiation is critical to reducing the number of stages and therefore costs associated with purification of REEs.

Figure 3-115: Extraction versus pH for Blended Extraction with Prepared Solution (Using REO from W. Ky. No. 13 Coal Seam AMD)

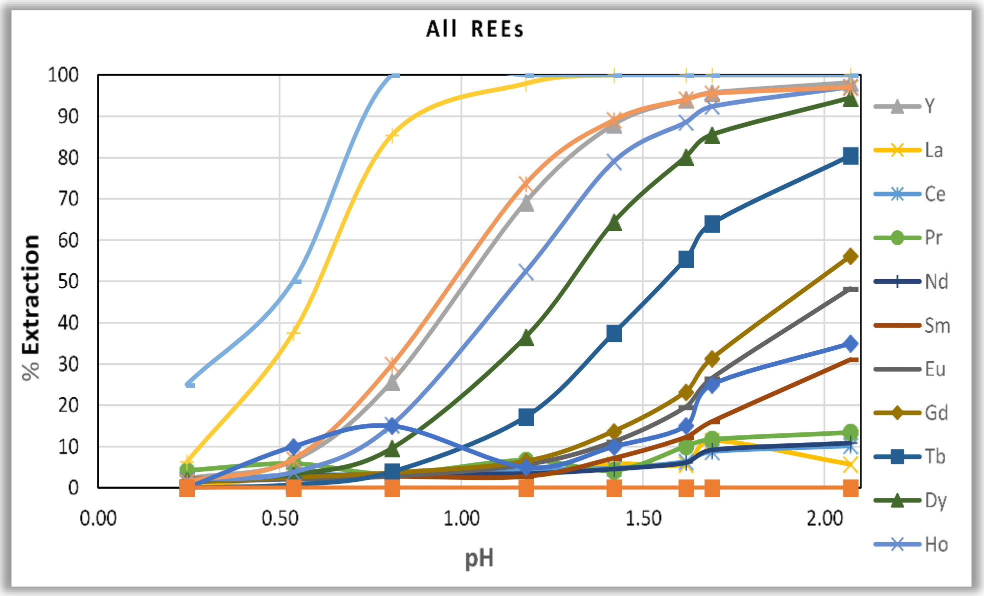


Figure 3-116: Extraction versus pH for Blended Extraction with Prepared Solution for High Concentration Rare Earth Elements (using REO from W. Ky. No. 13 Coal Seam AMD)

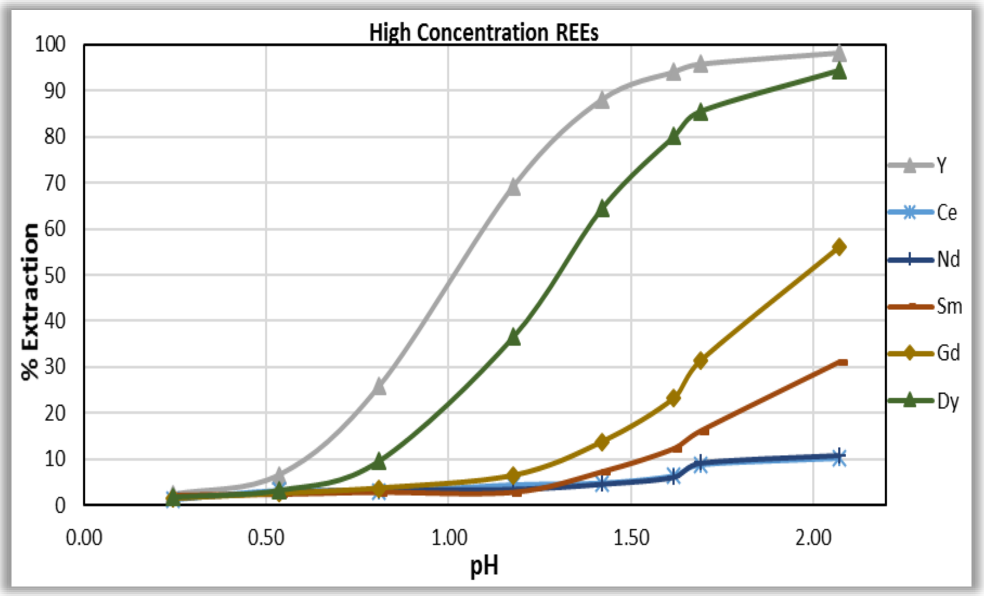
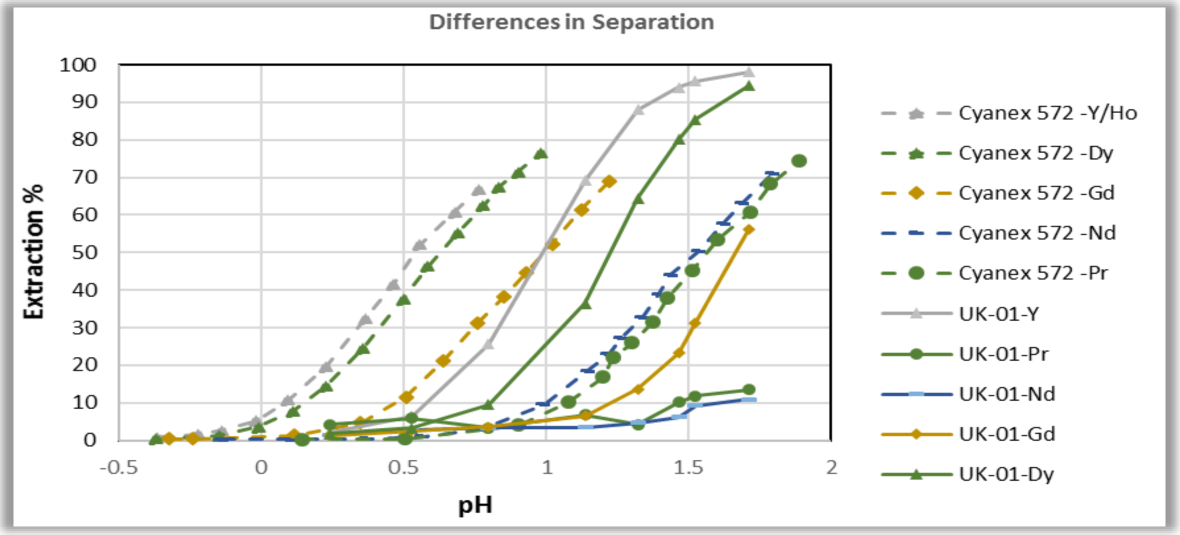


Figure 3-117: Comparison of the UK extractant Blend to Cyanex 572 Manufacturer's Data (using REO from W. Ky. No. 13 Coal Seam AMD)



3.3.1.7 Exploration of the Separation of Individual REEs (Y, Dy, and Nd from W. Ky. No. 13 Coal Seam AMD)

Individual separation of REEs from each other is one of the most challenging tasks faced by the team. The prime reason for the difficult separation behavior of REEs is the similar chemical properties shown by this group of elements. A primary property which is responsible for the difficult separation is the valency of the REEs. All REEs except cerium (*Ce*) and europium (*Eu*) show trivalent nature. Ce (IV and Eu (II) are commonly found in tetravalent and divalent states, respectively (Eyring, Gschneidner, & Lander, 2002). As the valency determines the oxidation states and reaction chemistry, the existence of all the REEs in the same oxidation state makes it difficult to separate them individually.

Irrespective of the associated challenges, numerous processes such as ion-exchange, fractional precipitation and solvent extraction (*SX*) have been developed to separate them into individual concentrates. Amongst all the processes, solvent extraction is found to be the most promising for the production of high purity rare earths (Brown & Sherrington, 1979). The SX process exploits the difference in basicity of REEs to make the separation. Another reason for the popularity of SX is its ability to treat large volumes of liquid economically (Xie, Zhang, Dreisinger, & Doyle, 2014), and is easy to control.

Exploratory work for the separation of REEs using SX was initiated for this project. Three rare earths Yttrium (*Y*), Dysprosium (*Dy*) and Neodymium (*Nd*) were identified as selected elements for the separation. Laboratory test work was performed to understand the extraction and separation behaviors of the identified elements.

All the test work was performed using the REO sample of 74 percent purity generated at a proof of concept pilot plant in West Kentucky operated by the University of Kentucky. The REO sample was generated by the concentration of pregnant leachate solution (*PLS*) using SX followed by oxalic acid precipitation and roasting. The plan to produce the Y, Dy and Nd of >99 percent purity was proposed and was initiated by experimental work. The ultimate objective of the research was to develop a separation process for Y, Dy and Nd. This report summarizes the experimental plan developed to achieve the objective and the progress achieved to-date.

This work was divided into three stages with each serving as a foundation for the next. The first two stages included the experimental work which identified the important operational variables of the flowsheet. The third stage involved the modeling of the SX process integrated with experimental data to produce a flowsheet for the separation of Y, Dy and Nd.

3.3.1.7.1 Stage One

The first step in separation of an individual rare earth is to identify the extraction behavior of the associated rare earths as a function of pH. As mentioned previously, a change in pH (basicity) affects the extraction of individual REEs differently and is exploited in the separation process. The second variable which is less widely used is the change in the extractant concentration. The change in the extractant concentration has a significant effect on the selectivity of rare earths. HREEs have more affinity towards the extractant and they are extracted first during the extraction process. Thus, having a low concentration of extractant can be utilized in the separation of LREEs from HREEs. The final variable is the comparison of different extractant types and the component mixture of the extractant. The reason for testing different extractants is because coordination and equilibrium chemistry changes according to the extractant being used. For example, it has been reported that strong extractants which have more affinity toward the metal ion are more difficult to strip requiring stronger stripping reagents which may be uneconomical for separation processes. If the use of different extractant aids the separation of individual rare earths, it is preferable to match extractants to each element (Tong, Wang, Liao, & Li, 2013). Hence the synergistic effect of different extractants needs to be studied to promote selectivity and reduce stripping costs.

The pH condition, extractant dosage and extractant combination planned for this study are summarized in the next table. The selection was made based on previous experience of operation of the SX circuit at UK's proof of concept pilot plant and literature review.

Table 3-65: Extraction Parameters
(Based on REO Collected from W. Ky. No. 13 Coal Seam AMD)

Variable	Range
pH	0-2.5 (max)
Extractant Concentration (v/v)	1% 2% 5% 10%
Extractant Name	D2EHPA Cyanex 572 D2EHPA and TBP mixture

Data obtained from the above experiments were used in finalizing:

1. Order of the separation of elements (Y, Dy and Nd);
2. Type of extractants or extractant mixture to be used and the dosage required;
3. Separation pH or cut-off pH for the respective elements; and
4. Conceptual flowsheet for the mentioned elements.

3.3.1.7.2 Stage Two

After the preliminary flowsheet was developed, information on the loading and stripping characteristics along with the number of stages required was determined. This required the development of the distribution isotherm for both loading and stripping characteristics of the respective element. The information thus obtained was used in determining the operating variables such as:

1. Volumetric flow rate of the organic and aqueous phases;
2. Number of stages required for separation;
3. Concentration of the element after every stage.

3.3.1.7.3 Stage Three

The final aspects of the research modeled the continuous extraction process of the proposed flowsheet. The conceptual flowsheet proposed was simulated under various operating conditions to predict the percent extraction and overall recovery of the process. The modeling task were performed in MATLAB®[17] and/or ASPEN[18]. The subroutine for the equilibrium extraction process was programmed in the software which was coupled with an optimization scheme to maximize recovery and minimize cost. The models were used in determining the equilibrium concentration of the rare earths based on extraction curve and distribution isotherm. Models were solved iteratively and predicted the product concentration after every stage and recovery under different sets of operating feed conditions.

3.3.1.7.4 Procedure: Extraction Experiment, Stage One

Laboratory scale SE experiments were carried out at different pH and different extractant concentrations.

The aqueous sample used for the analysis was prepared by dissolving 1 gm mixed REO (74 percent w/w) in 1M HCl acid to obtain a solution concentration of 1 gm/L of REO. The REO used was obtained from oxalic acid precipitation of stripped SX cleaner circuit solution from UK's pilot plant obtained from mine water. The pH of the aqueous solution was adjusted to the required value by adding 12.5 M NaOH in small amounts.

The organic solution was prepared by mixing the required volume of extractant in diluents to get a desired v/v percent. For each experiment, 50 mL of aqueous and organic solution were taken in a conical flask and were mixed using a magnetic stirrer at room temperature and pressure for 30 minutes. After mixing the solution it was transferred to a separatory funnel where the organic and the aqueous phases could disengage (*see the next figure*). The phase separation was rapid, with 10 minutes needed to ensure effective phase separation.

The pH of the aqueous solution after extraction was measured. The concentration of the rare earth in feed and raffinate after the experiment were measured by ICP-OES (SPECTRO ARCOS FHX22). The concentration in the organic was calculated by mass balance. No change in volume of the organic and

---

[17] MATLAB is a multi-paradigm numerical computing environment and proprietary programming language developed by MathWorks.

[18] ASPEN is a process simulation software package widely used in industry today. Given a process design and an appropriate selection of thermodynamic models, ASPEN uses mathematical models to predict the performance of the process. This information can then be used in an iterative fashion to optimize the design. ASPEN can handle very complex processes, including multiple-column separation systems, chemical reactors, distillation of chemically reactive compounds, and even electrolyte solutions like mineral acids and sodium hydroxide solutions.

aqueous was observed. The same procedure was repeated at a different pH ranging from 0-2.5, extractant concentrations and extractant types.

Figure 3-118: Mixing (Left) and Phase Separation (Right).

3.3.1.7.5 Procedure: Distribution Isotherm, Stage Two

A preliminary attempt to construct the McCabe Thiele diagram for the Y was made using feed stock prepared from REOs.

The pH of the aqueous feed was adjusted to 1 and the SX experiment was performed using DEHPA and TBP mixture (2 percent and 10 percent v/v respectively). The aqueous to organic ratio was kept to 1:1. The loaded organic was then stripped with 6M HCl to extract the loaded rare earth to aqueous phase. The purpose of this exercise was to isolate maximum Y (III) from the other REEs. The aqueous sample obtained will be rich in Y. The sample was then used to construct the distribution isotherm. The SX experiment at an aqueous-organic ratio of 1:1 to 1:5 (DEHPA and TBP mixture) was performed keeping the aqueous volume constant to 10 ml. The loaded organic was stripped at an aqueous to organic ratio of 1:1. The raffinate, feed and the stripped solution were then analyzed for the individual rare earth content by ICP-OES.

3.3.1.7.6 Extraction Experiment, Stage One

The extraction of the rare earths particularly Y, Dy and Nd were studied in the aqueous chloride media using D2EHPA, Cyanex 572 and a mixture of D2EHPA and TBP.

The effect of the aqueous solution pH (H^+ ion concentration) and extractant concentration were investigated. The pH of the system was varied from 0 to 2.5 for all three extractants. The extractant dosage used for different extractant schemes are summarized in the following table.

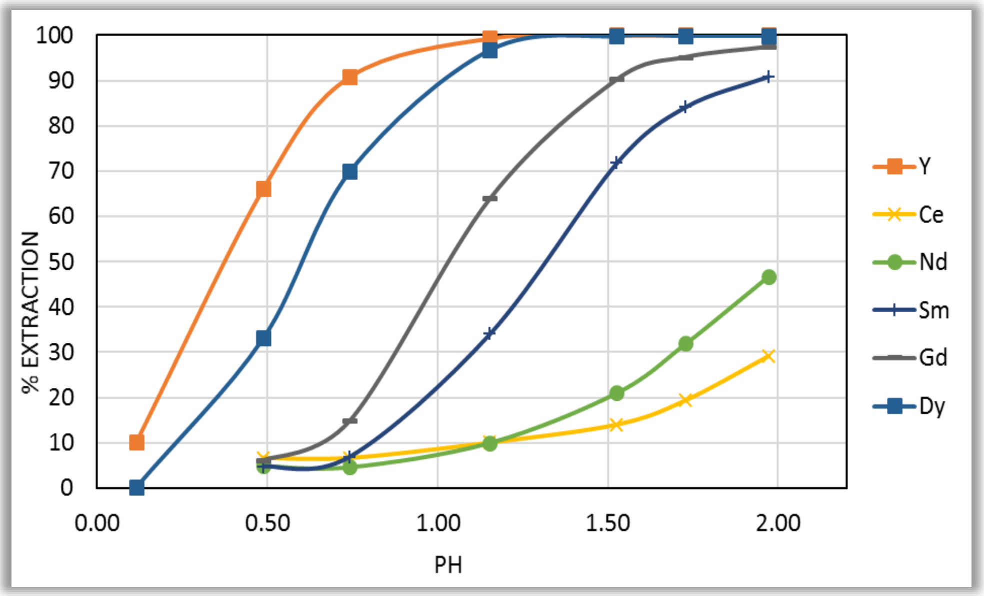
Table 3-66: Extractant Dosage Utilized

Extractant	v/v percent	Molar Concentration (mol/L)	Remarks
D2EHPA	2	0.0303	-
	5	0.151	
	10	0.303	
Cyanex 572	2	NA	Mixture of phosphonic and phosphinic acid
	5		
	10		
D2EHPA + TBP	2% and 10%	0.151 and 0.37	

3.3.1.7.7 Effect of Extractant pH

The following figure displays the percentage extraction obtained for the pure extractant D2EHPA when used 2 percent by volume. This figure demonstrates that the percent extractant value for all the rare earth increases with the increase in pH. Also, the order of extraction for the rare earths was Y > Dy > Gd > Sm > Nd and Ce, which confirms the HREEs are extracted first at a lower pH compared to LREEs.

Figure 3-119: Extraction Curve Using Two (2) Percent DEHPA (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)



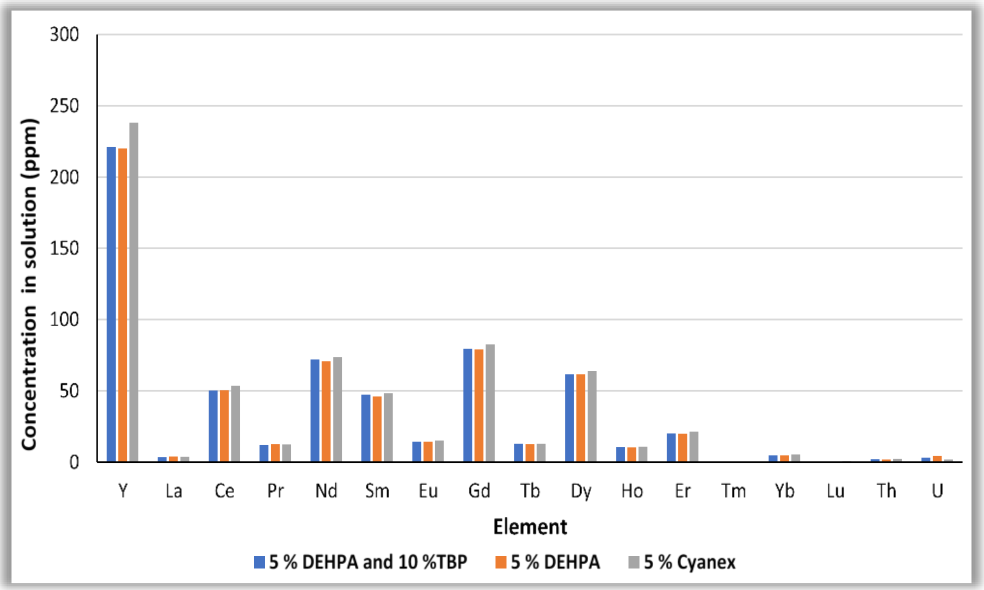
It should be noted that the above analysis was conducted on solution prepared from a mixture of REO as described previously. The oxide contained the suite of rare earths in varying proportions. The solution concentration of the individual rare earth in ppm (µg/L) is shown in the following figure. It is important to note that this distribution is dependent on the feed concentration which was derived from Dotiki's[19]

---

[19] **Webster County Coal, LLC** operates Dotiki, which is an underground mine located near the city of Providence in Webster County, Kentucky. Dotiki utilizes continuous mining units employing room-and-pillar mining techniques to produce high-sulfur coal. Dotiki's preparation plant has a throughput capacity of 1,800 tons of raw

processing plant rejects heap leachate[20]. Also, when analyzing the extraction data REEs with concentration only, REEs higher than 50 ppm in the feed were considered and the remainder were neglected because of low concentration. Note that that a major portion (> 90 percent) of Y/Dy are extracted around a pH of 1 which can be utilized in their separation from Nd.

Figure 3-120: Concentration of Individual REE in Feed Samples used for the Test (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)



Similar extraction curves were also obtained using a mixture of D2EHPA and TBP (2 percent and 10 percent v/v) *Figure 3-121*. It was observed that the order of extraction was the same except for Nd and Ce which was hard to distinguish. Also, Sm, Nd and Ce were not extracted in large quantities with the combination of extractant even at a pH of 2.0. This indicates TBP not only suppresses the extraction of REEs at lower pH and the effect is even more significant on LREEs at a higher pH. However, this behavior of TBP can be used to an advantage in separation of Y from Dy which was not possible when using D2EHPA alone.

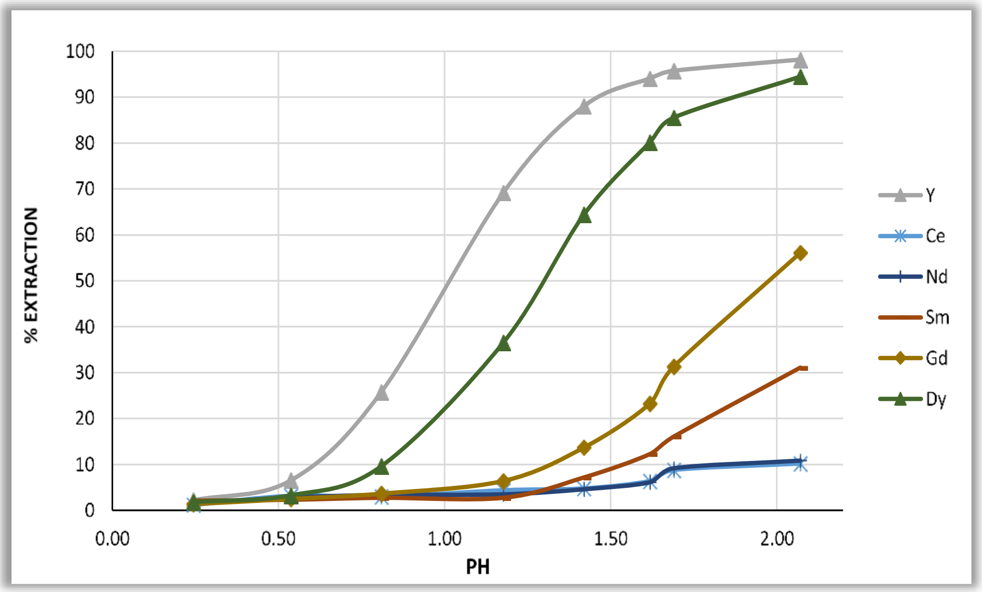
The following figure demonstrates that if a separation is made at a pH of 1.00, 50 percent of Y and 22 percent of Dy are extracted. Thus, this mixture can be utilized in the separation of Y from Dy utilizing multiple stages and re-cleaning at a lower pH to improve the quality of products.

---

coal per hour. Dotiki staff constructed a coal preparation plant rejects heap leaching site for use by the University of Kentucky.

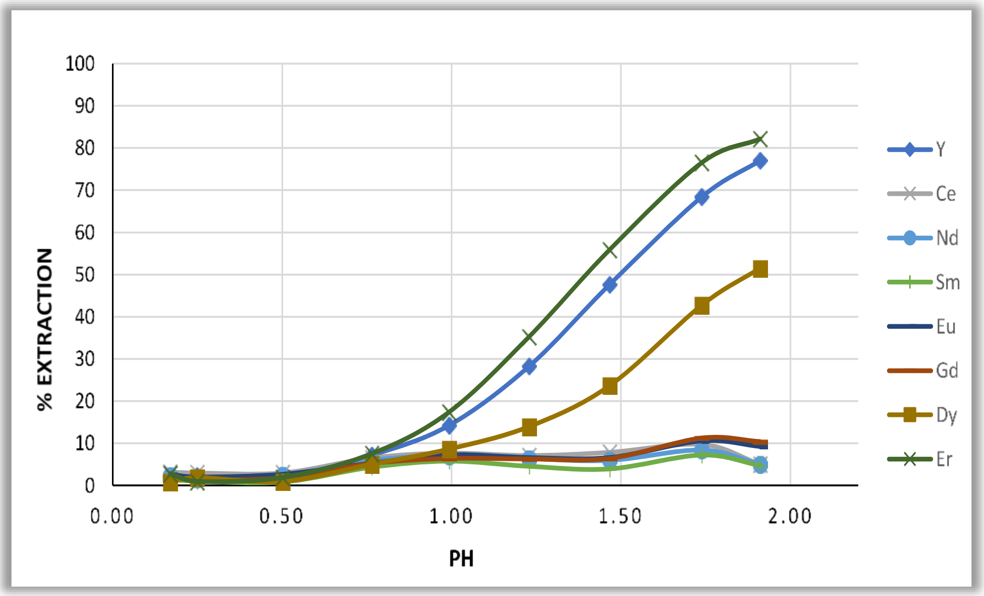
[20] A similar processing plant rejects heap leachate source does not exist at the Blue Diamond No. 76 Plant (a.k.a., "Leatherwood") in Perry County, Kentucky, near the community of Slemp.

Figure 3-121: Extraction Curve using 2 Percent DEHPA and 10 Percent TBP (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)



The third category of extractant tested was Cyanex 572 which is a mixture of phosphonic and phosphinic acid and is now being considered for rare earth separation by many researchers (Quinn, Soldenhoff, & Stevens, 2014; Wang, Huang, Li, Dong, & Sun, 2017). *Figure 3-122* displays the extraction curve developed using 2 percent Cyanex 572. It was found that the extraction by Cyanex 572 was less effective compared to DEEHPA and DEHPA+TBP mixture at a low pH. The order of extraction was the same as the DEHPA systems. The higher extractant percent for Y/Dy was observed when pH was increased to more than 1.5. However, unlike DEHPA and TBP mixture, Cyanex 572 was not able to separate Y from Dy. More tests at a higher pH is required to determine the effect of pH on LREEs.

Figure 3-122: Extraction Curve Using 2 Percent Cyanex
(Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)

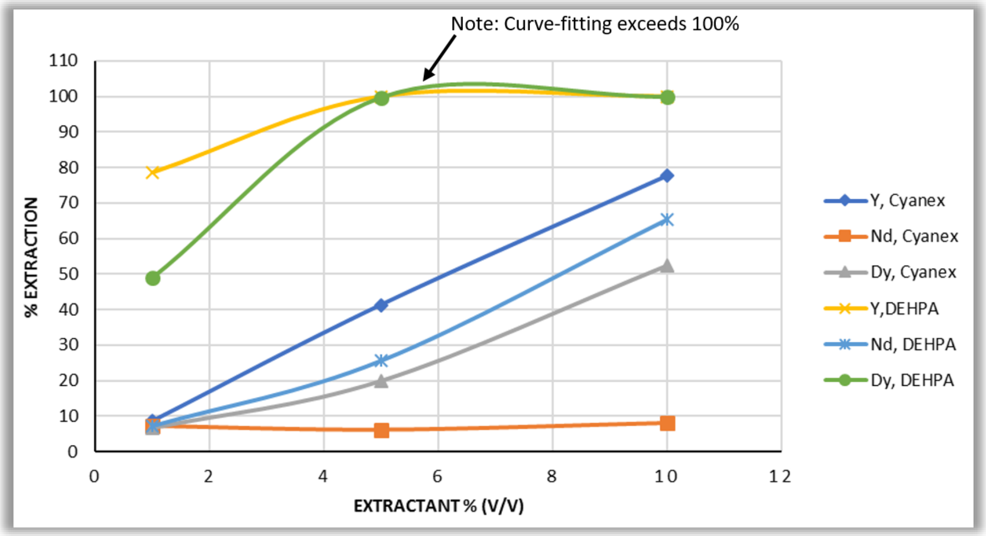


3.3.1.7.8 Effect of Extractant Concentration of the Recovery of Dy and Y using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD

The effect of extractant concentration was studied by performing sets of tests varying the extractant and maintaining constant pH. The initial extractant concentration had a significant effect on the extraction of the individual rare earths. Three scenarios are presented (at specific pH) where the extractant dosage was varied.

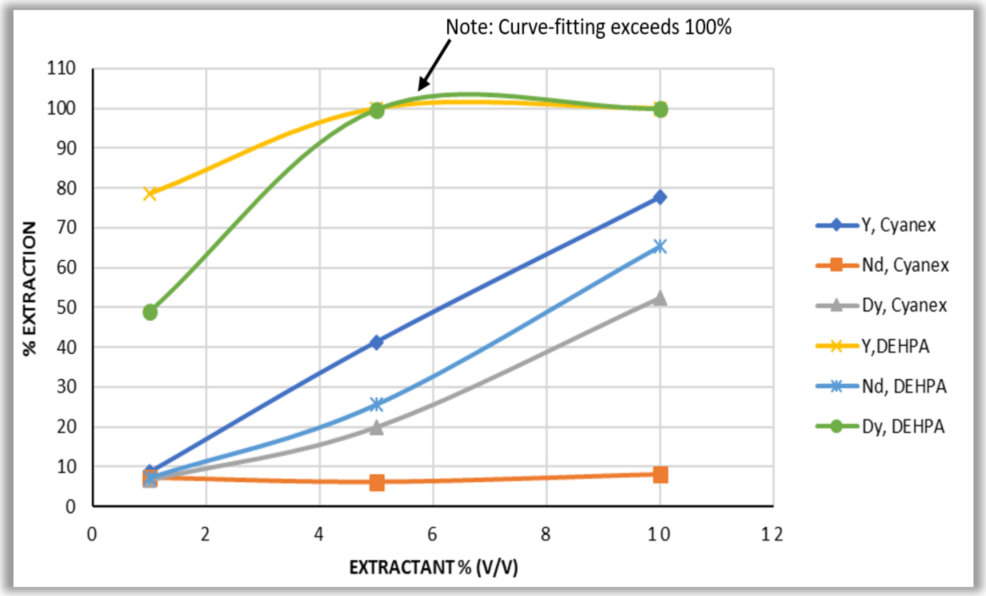
1. *pH 0.5* (The following figure) When the pH of aqueous feed solution (extraction) was 0.5 and the extractant dosage was varied, the selectivity of Y and Dy were higher for DEHPA both at 5 percent and 10 percent v/v. This again confirms the suitability of DEHPA for separation of Y/Dy at low pH. The extractant efficiency with Cyanex was very low irrespective of extractant dosage.

Figure 3-123: Effect of Extractant Dosage on Percent Extraction for Cyanex (at pH 0.5) and DEHPA (at pH 0.47) (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)



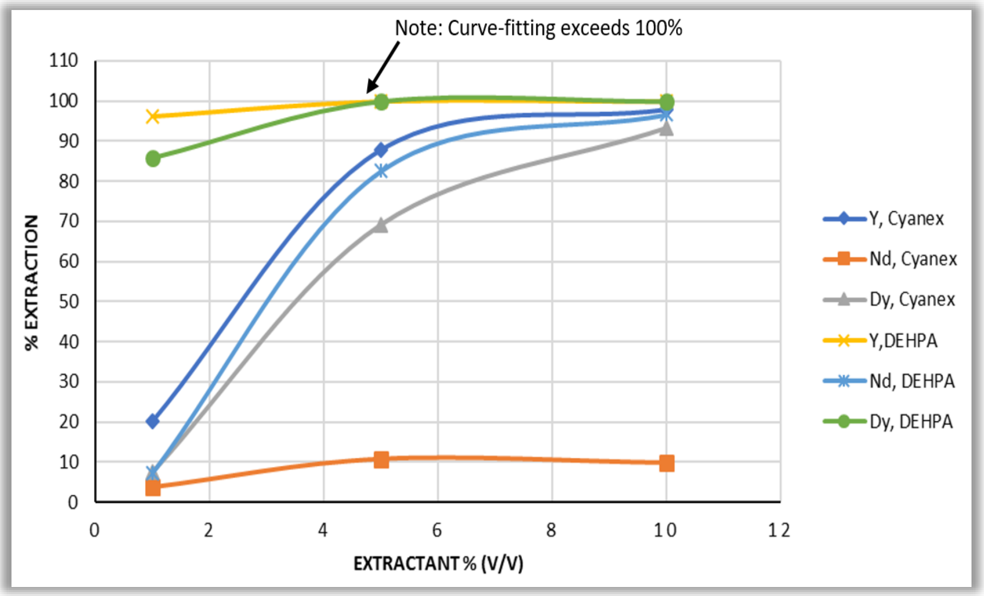
2. *pH 1.0* (The following figure) At pH 1, when the extractant dosage was varied, the percent extraction for Y and Dy was higher at all concentrations of DEPHA. However, the extraction at 1 percent DEHPA dosage for Y was high compared to Dy (30 percent extraction difference). This difference in selectivity of Y/Dy can be exploited to separate Y from Dy. Cyanex 572 did not show any improvement in the selectivty nor in the percentage extaction. Hence, Cyanex 572 is not a suitable extractant at low pH.

Figure 3-124: Effect of Extractant Dosage on Percent Extraction for Cyanex (at pH 0.99) and DEHPA (at pH 1.001) (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)



3. *pH 1.5* (The following figure) Shows a plot when pH of the extraction was 1.5 and the extractant dosage was varied. It was observed that separation of Y/DY from Nd was better at a lower dosage of DEHPA (1 percent v/v). Whereas separation of Y/Dy with Cyanex 572 was better at higher dosage (10 percent v/v).

Figure 3-125: Effect of Extractant Dosage on Percent Extraction for Cyanex (at pH 1.47) and DEHPA (at pH 1.542) (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)

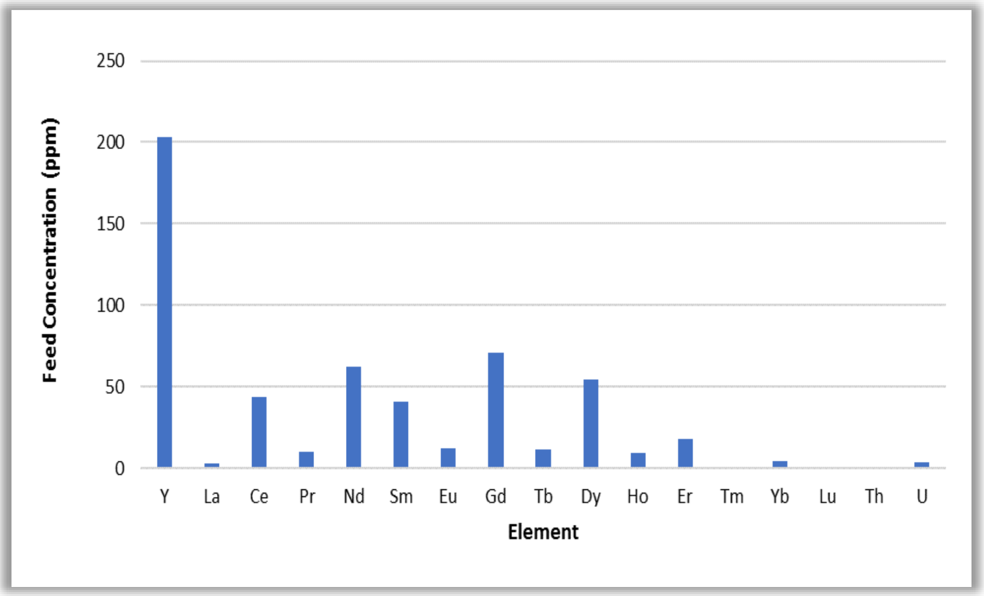


The above analysis indicates that both pH and extractant dosage can provide an effective separation of the Y, Nd and Dy from each other.

3.3.1.7.9 Distribution Isotherm, Stage Two

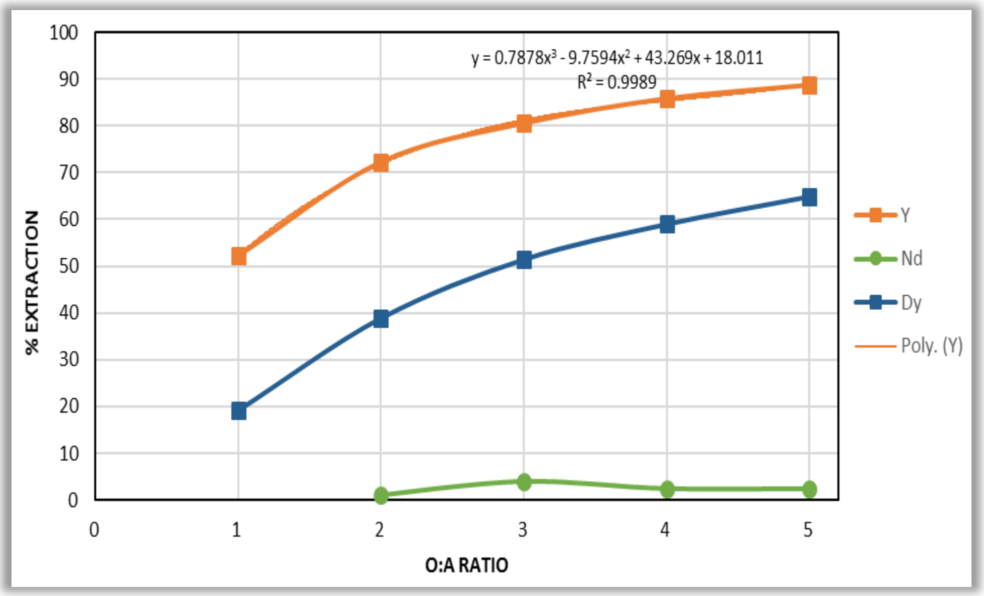
The distribution isotherm data was obtained by conducting a series of SX experiments on a pre-treated sample at different organic-to-aqueous (O:A) volumetric ratios. The individual rare earth composition of the feed solution used is shown in *Figure 3-126*. From the analysis it was found that minimum extraction occurred when the O:A ratio was 1:1. The percent extraction at this ratio for Y and Dy was found to be 52 percent and 19 percent (*Figure 3-127*) which was in accordance with the extraction curve from *Figure 3-121* at a pH of 1. The percent extraction increased by increasing the O:A ratio.

Figure 3-126: Feed Composition of Sample Used for Distribution Isotherm (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)



The application of distribution isotherms assists in identifying the number of stages of extraction to achieve the desired concentration. In addition, they define the maximum thermodynamic limit of the extractant of the system. Thus, from the plot in *Figure 3-127* it can be seen that the maximum extraction that can be achieved at the given pH (1.00) for Y is 90 percent after which increasing the organic has diminishing returns, irrespective of the number of stages employed. Similarly, for Dy the maximum percent extraction that can be achieved is 65 percent. Hence, different pH should be sought for extraction of Dy. However, the distribution isotherm shown can be generated for the higher O:A ratio which can lead to higher percent extraction for Y. However, this approach is less preferred as increasing the organic volume can cause flow issues within the circuit. Thus, distribution isotherms at different pH should be generated. Also, for future work (time allowing) the distribution isotherm will be generated using the salt of Y, Dy and Nd to get more detailed data on individual rare earth.

Figure 3-127: Distribution Isotherm for Loading of Y and Dy Mixture (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)



3.3.1.7.10 Distribution Coefficients

Based on the work listed above the distribution coefficients were calculated for each of the systems described, namely, 5 percent Depha-10 percent TBP, 2 percent Cyanex, 5 percent Cyanex, 10 percent Cyanex, 1 percent Depha, 2 percent Depha, 5 percent Depha, and 10 percent Depha. The distribution coefficient is defined as:

$$D = \frac{Y}{X} \qquad [3.4]$$

where D is the distribution coefficient, Y is the concentration of the element of interest in the organic phase and X is the concentration of the element of interest in the aqueous phase. As an example, *Table 3-67* gives the distribution coefficients for the 5 percent Depha-10 percent TBP system. Distribution coefficients were calculated for all systems listed previously. Next the separation coefficients were determined based on the order of extraction for the next nearest element. The separation coefficient is given by the ratio of the distribution coefficients of the elements of interest. It is a quantitative measure of selectivity. Each of the systems were compared for selectivity with the 5 percent Depha 10 percent TBP and the 5 percent Depha systems exhibiting the greatest number of highest selectivity element pairs. *Table 3-68* shows the separation coefficients associated with the 5 percent Depha 10 percent TBP system. The easiest and most efficient separations appear to be more likely to occur between Yb and Er as indicated by the high coefficient values. The darker red separations in *Table 3-68* represent easier separations and the lighter red shading are more difficult.

The data is converted into a useful form by the following derivation. The first being the equilibrium reaction between the aqueous metal ion and the organic loaded with hydrogen ions as follows.

$$M^{3+}_{aq} + 3\,HA_{org} \leftrightarrow M_{org} + H^{+}_{aq} \qquad [3.5]$$

The equilibrium expression is given as follows:

$$K_{ex} = \frac{[M_{org}][H^{+}_{aq}]^{3}}{[M^{3+}_{aq}][HA_{org}]^{3}} \qquad [3.6]$$

Which is then simplified to utilize the distribution coefficient D:

$$K_{ex} = D \times \frac{[H^{+}_{aq}]^{3}}{[HA_{org}]^{3}} \qquad [3.7]$$

Solving for D provides:

$$D = K_{ex}[HA_{org}]^{3} \times [H^{+}]^{-3} \qquad [3.8]$$

And determining which variables are in fact constants allows for the simplification:

$$D = C[H^{+}]^{Q} \qquad [3.9]$$

Recalling that the definition of pH is:

$$pH = -\log(aH^{+})\,;aH^{+} = 10^{-pH} \qquad [3.10]$$

The following substitution and simplification can be made:

$$\log D = -Q\log[H^{+}] + log\ c \qquad [3.11]$$

Taken as the slope of a straight line the above equation in the form of y = m*x + B, where y = logD; m = -Q; B = logC. The linearization is applied to the 5 percent Depha-10 percent TBP system is shown in *Table 3-69*. The data shows in general a good agreement with the linear fit. This data is used extensively for the modeling of the refining circuit giving the model the ability to determine the distribution of REEs in the aqueous and organic phases as a function of pH.

Table 3-67: Log of the Distribution Coefficients of 5 Percent Depha 10 Percent TBP System (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)

Log(D)																		
pH of Feed	**pH after SX**	**Y**	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Th**	**U**
0.244	0.241	-1.64		-1.92	-1.36	-1.81	-1.67	-1.85	-1.85		-1.74	-1.71	-1.82		-1.18	-0.48		
0.537	0.527	-1.15	-1.53	-1.48	-1.20	-1.54	-1.62	-1.67	-1.59	-2.10	-1.47	-1.40	-1.13		-0.22	0.00	-0.95	
0.809	0.796	-0.46	-1.53	-1.51	-1.46	-1.46	-1.55	-1.54	-1.42	-1.39	-0.98	-0.75	-0.37		0.77		-0.75	
1.177	1.137	0.35	-1.53	-1.34	-1.14	-1.44	-1.55	-1.23	-1.16	-0.68	-0.24	0.04	0.45		1.67		-1.28	
1.42	1.321	0.87	-1.22	-1.30	-1.36	-1.32	-1.11	-0.90	-0.80	-0.22	0.26	0.58	0.91				-0.95	
1.618	1.466	1.20	-1.22	-1.17	-0.95	-1.19	-0.85	-0.61	-0.52	0.10	0.61	0.89	1.20				-0.75	
1.69	1.524	1.35	-0.89	-1.02	-0.88	-1.00	-0.72	-0.44	-0.34	0.25	0.77	1.08	1.33				-0.48	

Table 3-68: Separation Coefficients of 5 Percent Depha 10 Percent TBP System (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)

pH after SX	Lu/Yb	Yb/Er	Er/Y	Y/Ho	Ho/Dy	Dy/Tb	Tb/Gd	Gd/Th	Th/Sm	Sm/Pr	Pr/Nd	Nd/Ce	Ce/La	Gd/Eu	Eu/Sm	Sm/La	La/Nd	Nd/Pr	Pr/Ce
0.241	5.00	4.40	0.67	1.17	1.07		0.00		0.00	0.49	2.82	1.28		0.99	0.66		0.00	0.35	3.62
0.527		8.01	1.07	1.77	1.18	4.26	0.31	0.23	4.67	0.38	2.18	0.87	1.12	1.20	0.90	0.81	1.03	0.46	1.89
0.796		13.76	1.23	1.93	1.70	2.61	1.07	0.21	6.24	0.81	1.01	1.12	1.05	1.32	1.02	0.96	0.85	0.99	1.13
1.137		16.83	1.24	2.04	1.91	2.77	3.03	1.30	1.86	0.39	2.00	0.78	1.56	1.16	2.10	0.96	0.82	0.50	1.57
1.321			1.10	1.95	2.08	3.02	3.78	1.43	1.43	1.77	0.91	0.96	0.83	1.26	1.63	1.28	1.26	1.10	0.87
1.466			0.99	2.05	1.91	3.25	4.11	1.72	1.26	1.25	1.72	0.96	1.13	1.25	1.74	2.31	0.93	0.58	1.64
1.524			0.95	1.86	2.05	3.32	3.91	1.37	1.74	1.44	1.32	1.05	0.75	1.26	1.89	1.48	1.28	0.76	1.38

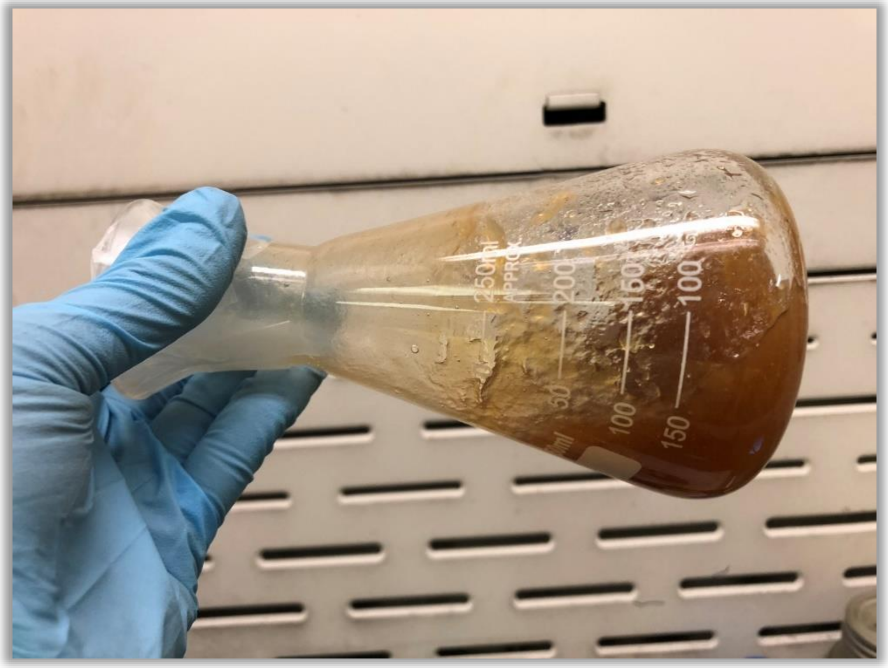
Table 3-69: Linearization of the Distribution Coefficients of Table 3-67, Including R^2 fit (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)

		Y	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Th	U
Slope	M	2.39	0.54	0.55	0.29	0.50	0.70	1.06	1.12	2.32	2.03	2.26	2.47		3.21	1.67	0.24	
Intercept	B	-2.32	-1.93	-1.94	-1.48	-1.89	-2.00	-2.24	-2.22	-3.30	-2.44	-2.45	-2.39		-1.91	-0.88	-1.13	
R Squared	R^2	0.997	0.665	0.887	0.418	0.877	0.765	0.932	0.946	0.999	0.982	0.989	0.999		0.995	1.000	0.124	

3.3.1.7.11 Saponification Refinements

Due to the difficulty in obtaining enough leachate and operating a lab scale solvent extraction unit, opportunities were utilized when available to test specific aspects required for this project from process outputs and reagents utilized for other rare earth projects. One such example was the work that was conducted for saponification. Due to the similarities of the process flow streams and characteristics of the solvent extraction processes, the organic from the quarter (¼) ton per hour pilot plant was tested for lab scale evaluations in this project. One of the key observations was the poor performance of the saponification circuit in the quarter (¼) ton project in operation. As part of this work additional lab scale testing was needed for this project to determine the cause of poor performance and for process flow sheet design. This failure can be observed in the next figure as a formation of a gel during saponification. To perform this test, a sample of the organic that had been in operation for several months was subjected to a typical saponification solution. The team returned to the source literature to review the conditions of operation versus previously published results. It was hypothesized that Fe^{3+} was loaded into the organic and not removed in the subsequent scrubbing and stripping stages. The residual iron was causing the gelling by the formation of an iron precipitate.

Figure 3-128: Failure of Solvent Extraction in the Saponification Stage of a Mixed Coal-Related Source Organic from a Pilot Scale Circuit Showing the Formation of a Gel



To test this hypothesis a test was performed which subjected the organic to two stages of sulfuric acid scrubbing at 5M prior to saponification with NaOH at 2.5M. The results of this test are depicted in the following figure. This shows that proper phase disengagement was achieved by the scrubbing of iron prior to saponification.

Figure 3-129: Post Organic and Solution in a Separatory Funnel after Shakeout

The results of this test also confirmed a mystery that had hitherto been unknown to the team. Over the course of the work with solvent extraction, the results for U and Th, had been lower than expected. Upon analysis of the aqueous strip and saponification aqueous solutions if was found that Sc, Th, and U had apparently not been stripping from the organic, creating the potential for poisoning. As shown in *Figure 3-130* and *Figure 3-131* significant iron and Th were removed in the sulfuric acid washing stages. U, Sc and Iron were recovered in the saponification stages. In this manner it appeared that the U and Th were accumulating in the organic phase and not stripping, as these elements were not reporting to the REE concentrate in anticipated levels. This result was significant because prior to this work a separate Sc recovery circuit was planned prior to the extraction of rare earth elements. This design change resulted in the savings of an anticipated $1.2 million by the elimination of a large-scale Sc circuit.

Figure 3-130: Distribution of REEs and U+Th in the Various Saponification Stages (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)

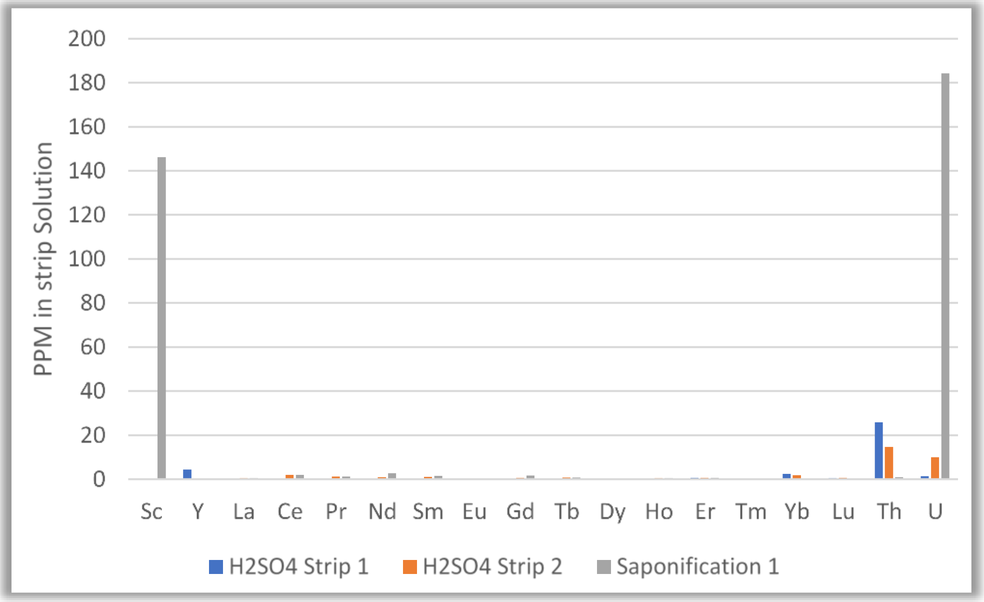
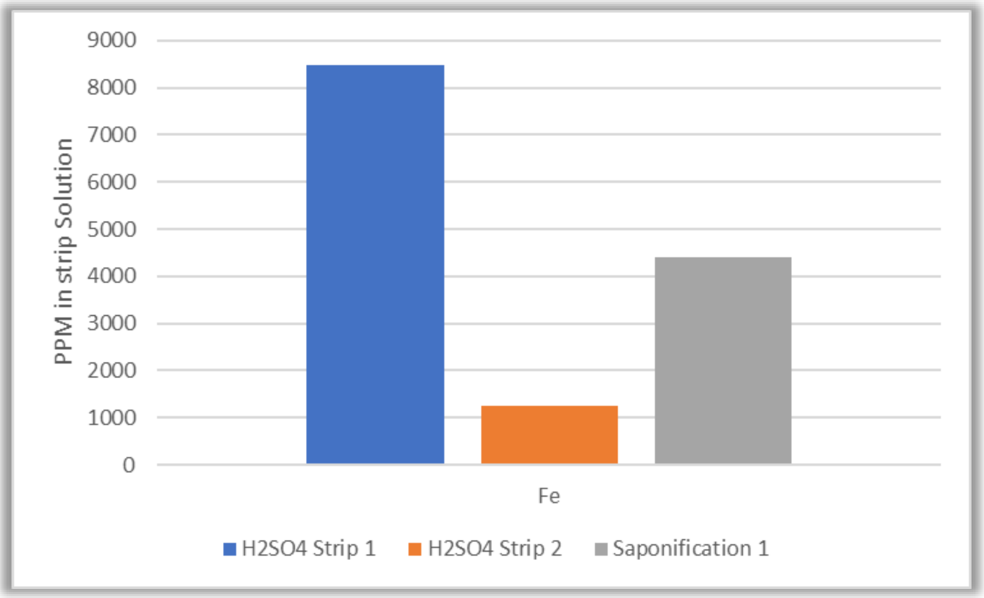


Figure 3-131: Distribution of Iron in the Various Saponification Stages (Using a Solution based on REO Collected from W. Ky. No. 13 Coal Seam AMD)



3.4 Detailed Flow Diagram

3.4.1 Flow Diagram Description

The data collected from the flowsheet development activities were used to construct a process flow diagram for REE extraction. For engineering purposes, the flow diagram was subdivided into five functional circuits, i.e.:

> Circuit 1 – Feed Preparation;
> Circuit 2 – Acid Leaching and Waste Treatment;
> Circuit 3 – Rougher Solvent Extraction;
> Circuit 4 – Cleaner Solvent Extraction; and
> Circuit 5 – Scandium Solvent Extraction.

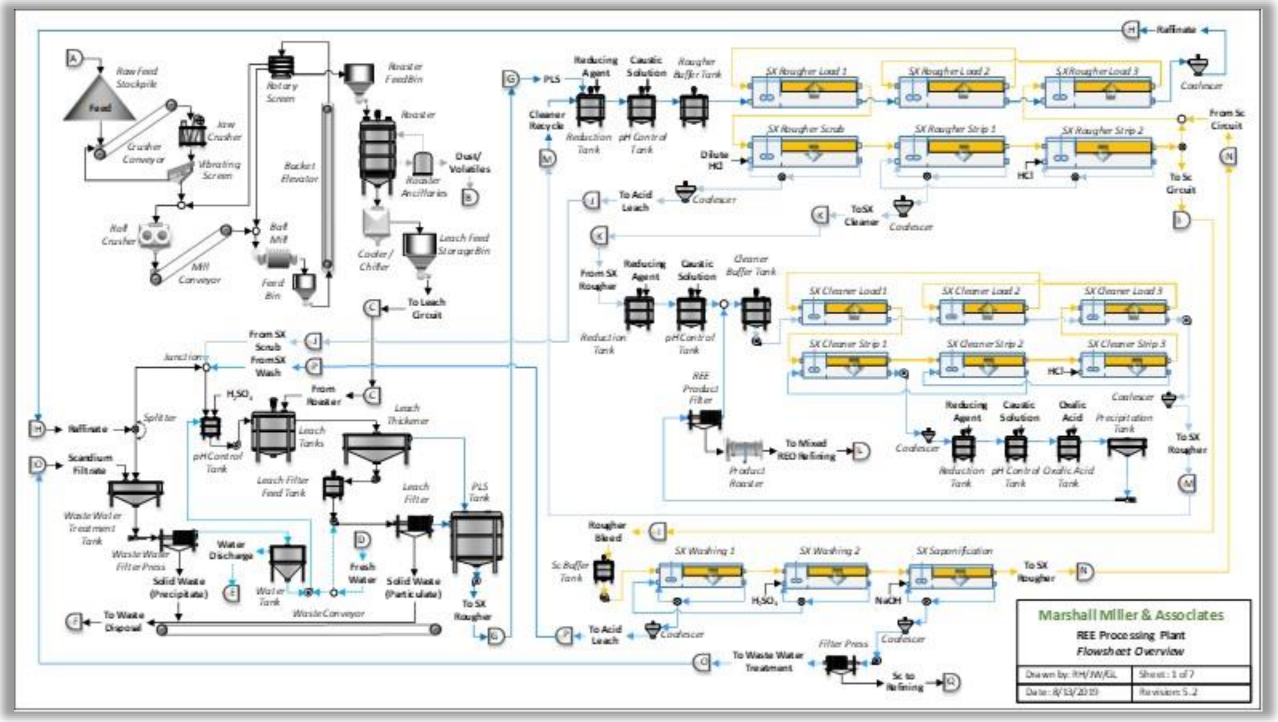
For each circuit, experimental data was collected and evaluated for use in the flow diagram development. Economic factors were also considered in the flow diagram development, as described in later sections of this document. The technical work focused on detailed engineering activities including:

1. Calculations of mass flow rates for solid, liquid and component (e.g., ash, REEs, elemental impurities) species entering and exiting each circuit,
2. preparation of a general listing of required processing equipment (type, size, capacity, power and consumables) for each circuit, and
3. preparation of a generalized process flowsheet (flow diagram) showing the unit-to-unit arrangement of all unit operations and connecting/recirculated streams for each circuit.

Numerical values for the performance indicators used in the flowsheet development work (i.e., mass splits, liquid splits, component assays, etc.) were obtained from the aforementioned battery of experimental characterization studies and separation/extraction tests described elsewhere in this report. [Note: Full-size copies of the flow diagrams are provided in the *Appendix* of this chapter**.**]

The following figure depicts the final version of the flow diagram identified by the process engineering team as the most viable configuration for REE extraction.

Figure 3-132: Process Flow Diagram for REE Extraction from Coal-based Sources

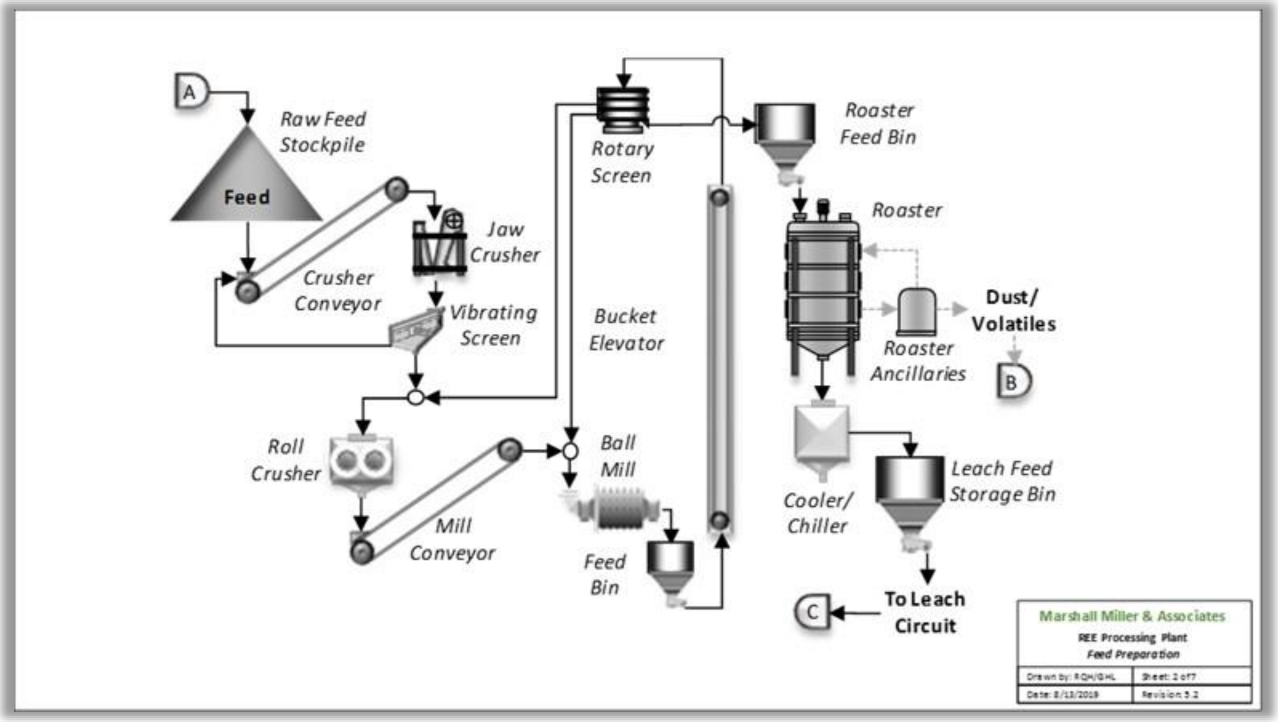


It is important to note that several flowsheets were considered by the team that incorporated other physical separations such as ore sorting, froth flotation and selective agglomeration. Unfortunately, as described in other sections of this report, none of the physical separation options were able to attain a sufficient level of upgrading to warrant inclusion into the final process flow diagram. On the other hand, the characterization and laboratory testing data compiled by the project team indicated that low-temperature roasting was an essential and necessary processing step to ensure that high extraction recoveries of REE were achieved and that acid consumption levels were kept to a minimum. No viable circuit was identified that did not include the roasting step. As such, water-based preconcentration options, such as froth flotation or selective agglomeration, were excluded from consideration since the resultant products would have to be subjected to costly dewatering and drying operations prior to roasting. Some success was achieved in producing an enriched concentrate of rare earth minerals using these physico-chemical separation methods. However, the resultant concentrate did not respond well to acid leaching. As such, the flowsheet would have required the addition of a high pressure, high temperature cracking unit, which could not be justified by the team in light of potential safety concerns.

3.4.1.1 Circuit 1 – Feed Preparation

Unit operations included in this part of the process flow diagram included equipment necessary to crush, size, grind and roast the dry coal-based REE feedstocks. In this circuit, feed material is transported via a feeder/conveyor belt to a jaw crusher for primary breakage. The crusher undersize is passed over a vibrating screen configured to retain oversize that circulates back to the crusher feed. The undersize fraction from the screen passes on to a secondary stage roll crusher. The pulverized product from the roll crusher is then transported via a second conveyor belt to a tertiary ball mill. The ground product from the ball mill is lifted by a bucket elevator to a three-product rotary screen. The coarsest oversize fraction from the screen is circulated back to the roll crusher, while the next coarsest intermediate fraction is passed back to the ball mill. The finest size fraction from the rotary screen is passed to a storage bin designed to hold feed for the roasting operation. Based on laboratory testing, the ground product size was targeted to be 80 percent passing 80-mesh. A vibrating feeder is used to regulate the mass feed rate of dry solids to the roaster. After roasting, the converted feed is passed through a cooler/chiller to reduce the temperature before being passed to a final storage bin. This bin, which is also equipped with a regulated vibratory feeder, serves as the feed system for introducing solids into the leaching circuit. As such, the only products exiting the feed preparation circuitry are (i) the dry solids feed for acid leaching and (ii) a very small amount of lost mass due to fugitive dust and emitted volatiles. This circuit is shown in the following figure.

Figure 3-133: Detailed Process Flow Diagram for the Feedstock Preparation Circuit

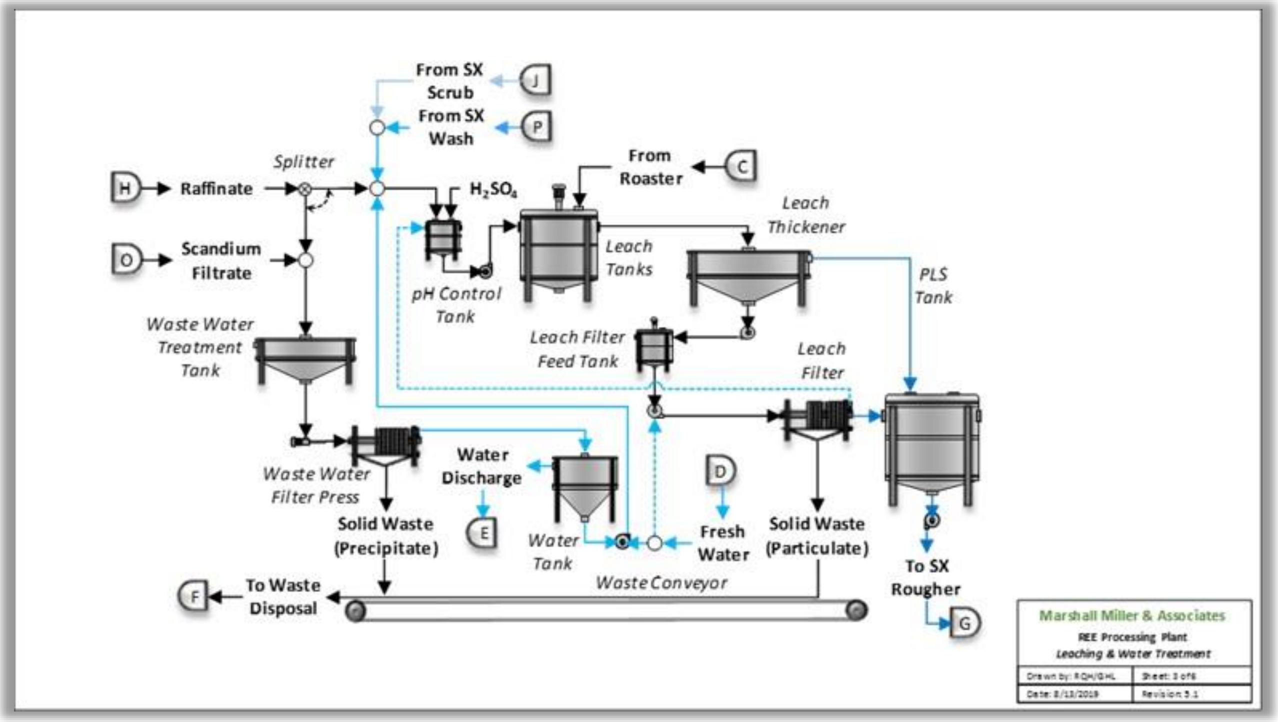


It should be noted that one variation of the feed preparation circuitry that may warrant further evaluations is dry upgrading. Preliminary data suggests that unit operations such as electronic sorters, air tables/jigs, or other dry processes for feedstock segregation may offer significant benefits in terms of increasing the grade of dry material fed to the roasting operation. Unfortunately, test results obtained by the project team at the time of final report preparation were inconclusive and, as such, did not warrant the inclusion of such equipment into the process flow diagram at this time.

3.4.1.2 Circuit 2 – Acid Leaching and Waste Treatment

This portion of the circuitry is used to (i) solubilize and store the targeted REE constituents as PLS and (ii) handle the solid and liquid wastes generated by the overall extraction facility. Unit operations in this circuitry include various tanks for slurry mixing/acid contacting, holding tanks for process water streams, thickeners for leach slurry densification and water clarification/treatment, pressure filters for solids dewatering, and centrifugal/moyno pumps for slurry/fluid transfer. The acid leaching circuit receives dry roasted solids from the feed preparation circuit. The dry powder is introduced directly into the first of a series of leach tanks. Although not shown in the flow diagram, a total of three sequential tanks are utilized in series to ensure that short-circuiting of the feed solids is minimized. All three tanks are also included in the process equipment and costing lists. To achieve the desired percent solids during leaching, the first leach tank is fed a leach solution from a pH control tank. The resultant leach slurry passes through the first mixed leach tank and sequentially overflows into two downstream tanks. The overflow from the last leach tank flows by gravity to a leach thickener. The thickener provides a densified underflow of leached/spent particulate solids and a clarified overflow that is essentially free of solids. The leach thickener underflow is dewatered using a pressure filter. The filter cake is discharged as a particulate waste after being purged with make-up water. The filter effluent (and water purge) and leach thickener overflow are passed to a large tank that serves as storage for the PLS. The following figure depicts the circuitry for acid leaching and waste treatment.

Figure 3-134: Detailed Process Flow Diagram for the Acid Leaching and Waste Treatment Circuit



One of the key components in the acid leach circuit is the pH control tank. This tank is used to monitor and automatically adjust the leach solution for optimal leaching performance, chemical demand and water usage. Streams recycled back to the pH control tank consist of:

1. A portion of the raffinate from the third-stage loading step in the rougher solvent extraction circuit,
2. the scrubber aqueous stream from the scrubbing step in the rougher solvent extraction circuit, and
3. the first-stage scandium washing aqueous stream from the scandium solvent extraction circuit.

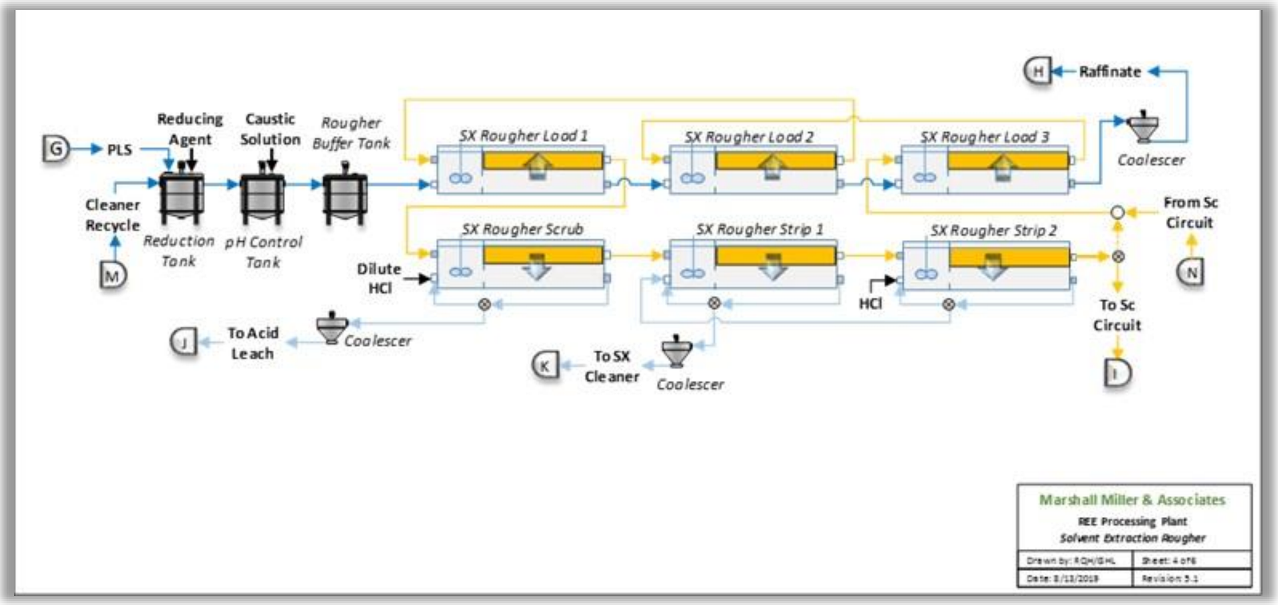
Raffinate from the rougher circuit that is not passed to the pH control tank is diverted into a water treatment tank. The filtrate from the scandium saponification step is also circulated back to the water treatment circuit to avoid contamination of the leach solution by reagents from the saponification step. The precipitates formed in the water treatment tank pass through a pressure filter to provide a clarified effluent and filter cake of waste precipitates. The clarified effluent is recycled back into the acid leach circuit as process water, while the filter cake is discharged as a solid waste. Both the precipitate filter cake from water treatment and the particulate filter cake from the leach thickener/filter are neutralized prior to being discharged as a solid waste. The solid waste is transferred to an appropriate disposal area by overland conveyor. As such, the final products generated by the acid leaching and water treatment circuit include:

1. The pregnant leach solution (PLS),
2. spent solid waste (particulates) from acid leaching, and
3. solid waste (precipitates) from water treatment.

3.4.1.3 Circuit 3 – Rougher Solvent Extraction

The last three circuits in the process flow diagram are hydrometallurgical operations that enrich and purify the PLS from the acid leaching circuit. The primary unit operations used for this purpose are SX reactors that separate species in solution based on their relative solubilities in two immiscible liquids (i.e., water and organic solvent). The equipment used for solvent extraction consists primary of simple mixer-settler reactors. Mixer-settler reactors include a first stage that thoroughly mixes the immiscible phases together followed by a quiescent settling stage that allows the phases to separate by gravity. Solvent extraction is typically performed in successive sequential stages with circulating/recycled aqueous and organic solvent streams to achieve high levels of separation efficiency. As such, the process flow diagram for REE extraction includes solvent extraction units for "Rougher Solvent Extraction," "Cleaner Solvent Extraction" and "Scandium Solvent Extraction," as illustrated previously in *Figure 3-132*. The next figure shows the enlarged process flow diagram for the rougher solvent extraction circuit.

Figure 3-135: Detailed Process Flow Diagram for the Rougher Solvent Extraction Circuit



The primary function of this portion of the circuit is to upgrade PLS from acid leaching into a bulk aqueous concentrate that can be further enriched into a final produce in a cleaner solvent extraction circuit.

In terms of processing sequence, solution stored in the PLS tank is fed by centrifugal pump into the rougher reduction tank where a chemical reducing agent is added. The overflow from this tank then passes into the rougher pH control tank with caustic solution is added to reduce the pH level. This tank then overflows into a rougher buffer tank, which regulates/stabilizes fluctuations in the volumetric flow and ensures that reagent reactions have been completed. The treated solution from the rougher buffer tank is passed directly into the first stage of three sequential SX "loading" mixer-settlers. The chemical conditions in the loading stage promotes the selective transfer of desirable dissolved species from the aqueous phase into the organic phase. The aqueous solution from the third and last rougher loading stage is typically referred to as raffinate. This stream is recycled/circulated back to the leaching circuit to maximize elemental recoveries and to minimize acid consumption. However, to avoid the build-up of unwanted contamination, some portion of the rougher raffinate is intentionally bled off into the water treatment circuit (i.e., wastewater treatment tank in the acid leaching circuit).

The three stages of rougher loading utilize a counter-current flow of organic solvent to improve extraction transfer/kinetics. The fully loaded organic solvent from the first stage of rougher loading is passed to a rougher "scrubbing" stage. Dilute acid (HCl) is added into the mixer-settler at this point to reverse the transfer of dissolved species from organic solution back into an aqueous solution. For REE processing, the scrubber stage is essential to reduce contamination levels of unwanted elements (e.g., Al, Si, Fe, etc.) in the organic solution.

After scrubbing, the organic solution continues through two stages of rougher "stripping." The first stage of rougher stripping produces an aqueous solution that represents the bulk REE concentrate from the rougher SX circuit. In the next stage of stripping, a stronger acid (HCl) solution is added to further drive species from the circulating organic solvent back into aqueous solution. The aqueous solution from the second rougher stripping stage is recycled back to the feed of the first stripper stage to maximize the recovery of desirable species.

Since the chemical conditions used in the rougher circuit do not efficiently recovery scandium, the organic solution from the second stage of rougher stripping is passed to a separate scandium recovery circuit, which is described below. After scandium extraction, the entirety of the organic solvent is returned back to the rougher solvent extraction circuit as a depleted organic solvent.

The return organic flow enters the rougher circuit through the third loading mixer-settler. The three aqueous product streams generated by the rougher SX circuit are the primary rougher raffinate, rougher scrubbing recycle and rougher stripping concentrate.

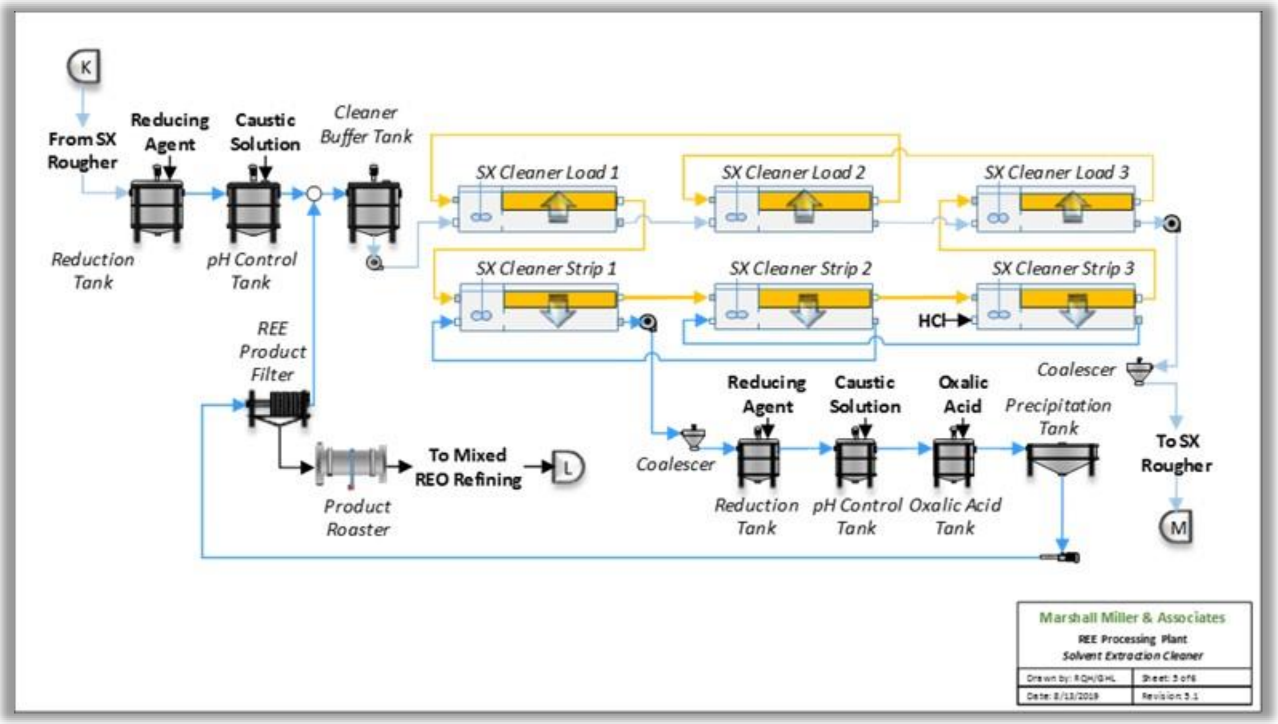
Each of these three streams pass through coalescers to remove any finely dispersed organic droplets that are too small to be separated by gravity. The coalescer equipment feature three compartments that consist of:

1. A feed inlet compartment that acts as an after-settler for primary removal of entrainment,
2. a coalescing bed compartment for removal of the finely dispersed droplets, and
3. a distribution compartment for the final separation and recovery of the organic and aqueous phases.

3.4.1.4 Circuit 4 – Cleaner Solvent Extraction

The process flow diagram for the "Cleaner Solvent Extraction" Circuit is shown in the following figure. This portion of the hydrometallurgical circuitry is designed to produce a final mixed REO product that can be marketed to REE refining groups.

Figure 3-136: Detailed Process Flow Diagram for the Cleaner Solevent Extraction Circuit

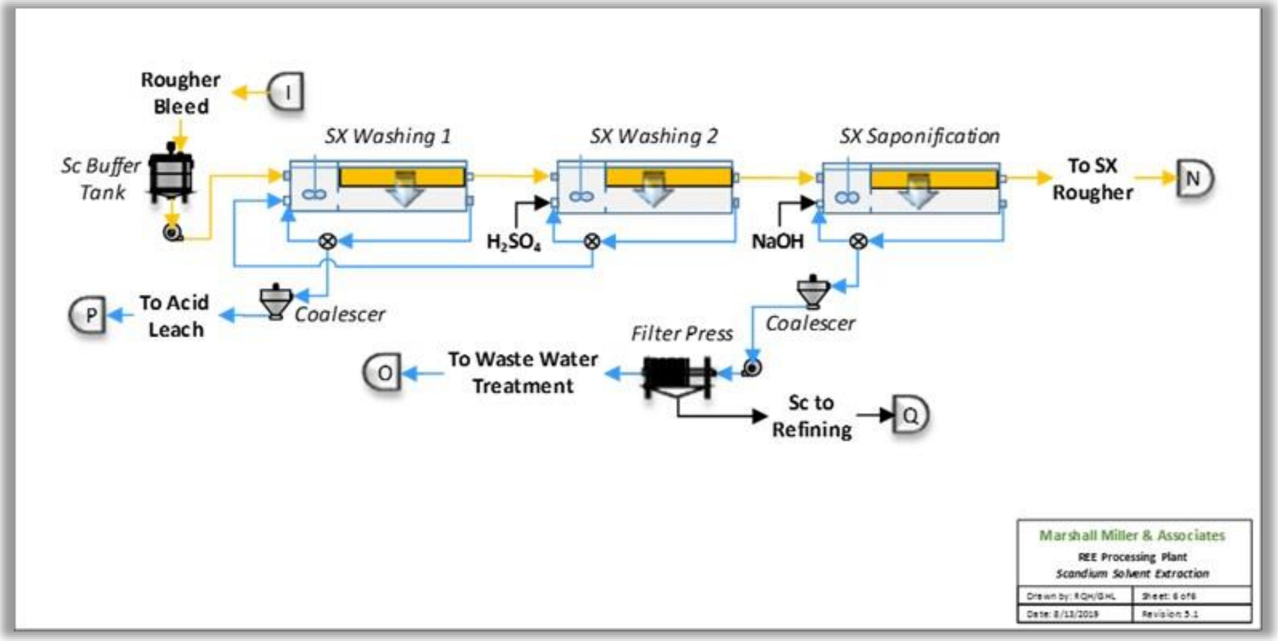


The cleaner SX circuit receives an aqueous feed as concentrate from the rougher SX circuit. The feed stream passes through two stages of mixed tanks where a chemical reducing agent and caustic are sequentially added. The conditioned overflow from the second tank (i.e., cleaner pH control tank) enters the cleaner buffer tank along with the effluent stream from the REE product filter. The aqueous solution then passes through three in-series stages of solvent extraction loading. This portion of the circuit is designed to transfer dissolved species in aqueous solution into the organic phase. The depleted aqueous solution from the third cleaner loading stage is recycled/circulated back to feed of the rougher reduction tank to ensure that losses of targeted species are minimized. The loaded organic solution that is circulated in counter-current fashion back through the three stages of cleaner loading is then passed to three stages of cleaner stripping. In these mixer settlers, the loaded organic solution is treated in three sequential stages of cleaner stripping to transfer targeted species back into aqueous solution. To promote this transfer, acid (HCl) is added to the third/final cleaner stripper and circulated back through the first two stripping stages. The newly loaded aqueous solution from the stripping unit is then passed from the first stage cleaner stripping unit to a final product reduction tank, product pH control tank and product oxalic acid tank. The chemical additions of reducing agent, caustic solution and oxalic acid create a chemical environment that promotes the selective precipitation of targeted REE species. The REE-bearing precipitates are collected in a precipitation tank and then dewatered in a pressure filter. The filter effluent is circulated back to the cleaner buffer tank, while the filter cake is subjected to a product roasting step. The final product roaster converts the hydrated REE precipitates into a powdered REO product that can be directly sold to REE marketing and/or refining groups.

3.4.1.5 Circuit 5 – Scandium Solvent Extraction

The following figure is an enlarged portion of the process flow diagram from the scandium solvent extraction circuit. This additional circuitry is required since the element scandium, which is highly valued, does not tend to be effectively recovered in the rougher REE concentrate. Instead, this particular element tends to build-up and accumulate in the circulated rougher organic stream.

Figure 3-137: Detailed Process Flow Diagram for the Scandium Solvent Extraction Circuit



To recover scandium, the organic solvent (or some portion of the solvent) from the last stage of rougher stripping is intentionally bled off to a separate scandium recovery circuit, which consists of a buffer tank followed by two sequential stages of washing (i.e., scandium stripping) of the organic solvent. The aqueous product from the second stage of washing is circulated back to the first washing stage, while the aqueous product from the first stage of washing is circulated back to the acid leaching circuit to minimize losses. Acid (H_2SO_4) is added to the second washing stage to reduce pH and to drive the species transfer in the two stages of washing. A coalescer is used to minimize the accidental and unwanted losses of organic solvent to the acid leaching circuit. The organic stream from the second stage of washing overflows into a final mixer-settler where a base (NaOH) is added to induce saponification.

The saponification step drops out scandium that is recovered using a pressure filter. The filtered scandium product is then pass to an additional refining step (discussed elsewhere in this report) prior to being offered to customer markets.

The effluent from the scandium filter is circulated back to the water treatment units in the acid leaching circuit. None of this effluent is circulated back into the acid leaching operation until after passing through water treatment operations that neutralize the chemical additives used in scandium washing and saponification.

To close the scandium circuit, the organic solvent from the saponification step is passed back to the rougher solvent extraction circuit as organic feed to the third rougher loading mixer-settler.

3.5 Flowsheet Simulation to Evaluate Process Circuitry for REE Concentration from Coal-based Sources

3.5.1 Flowsheet Development Software Selection

In order to streamline the flowsheet development work, the project team utilized a process flowsheet simulation tool.

3.5.1.1 Evaluation of LIMN

The simulation initially made use of the LIMN flowsheet processor to create, model and analyze different flowsheet configurations. However, as work progressed, it became apparent that the use of this particular tool was limited due to a lack of hydrometallurgical models and constraints associated with software licensing and distribution.

3.5.1.2 Evaluation of REESim

To avoid the LIMN software use and application issues, the engineering team switched to a spreadsheet-based simulation package that was being developed under another DOE sponsored project entitled "*Pilot-Scale Testing of an Integrated Circuit for the Extraction of Rare Earth Minerals and Elements from Coal and Coal Byproducts Using Advanced Separation Technologies* (DE-FE0027035). One of the key tasks to be completed under this on-going project involved the development of a flowsheet simulation tool that can be applied to develop, design and evaluate process circuitry for REE concentration from coal-based sources. This software tool, which is currently referred to as REESim, was ideally suited for use in the current project.

The user-manual for the latest working version of the REESim software (Ver. 190802.1) is attached to this chapter as an *Appendix* item.

3.5.1.2.1 REESim Software Description

Since the software was developed specifically for REE extraction from coal-based sources, the REESim tool was ideally configured for the flowsheet development work performed in this project

The REESim simulation tool makes use of the Microsoft Excel spreadsheet software. This open-source programming platform provides the following benefits:

1. REESim avoids licensing fees commonly required by other software alternatives,
2. provides a user-friendly interface that can be customized without formal training in advanced programming languages, and
3. makes it possible to take advantage of open-source built-in features such as an iteration engine, form controls, and VBA (Visual Basic for Applications) coding.

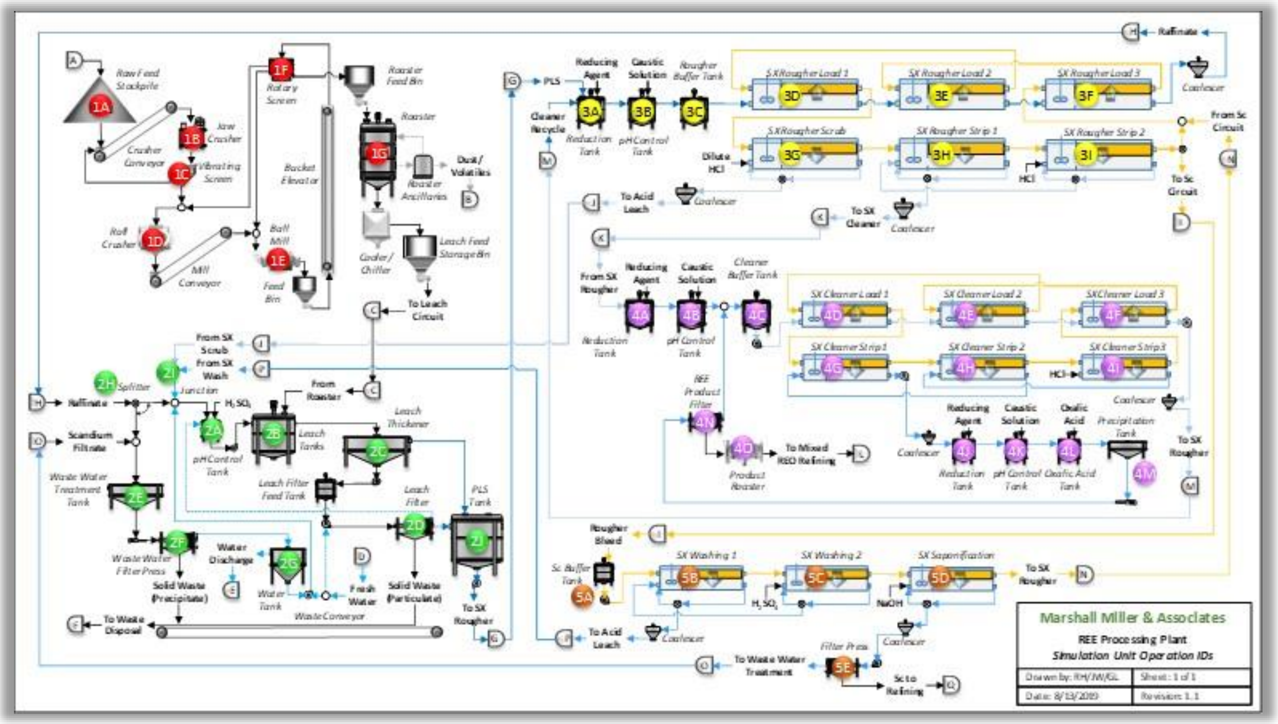
The REESim simulation package utilizes a separate worksheet "Tab" for each unit operation included within a process flowsheet. The interconnection of streams between the various unit operations (spreadsheet tabs) are handled using built-in dropdown menus that specify which streams within the circuit are fed to a particular unit operation. As such, the tool makes it very easy to add/delete unit operations or to reconfigure streams entering/exiting a particular process.

To facilitate transfer of process stream data, a standard input/output matrix is integrated into each unit operation/worksheet. The input/output matrix provides an interface for entering and reporting values for dry solids mass rate, volumetric slurry flow rate, chemical/reagent addition rates and species/component assay values.

3.5.2 Components of REESim Circuit Simulation Workbook for Project MMA 29956

In order to streamline the simulation work, identification (*ID*) numbers were assigned to each of the process units included in the process flow diagram. The IDs consisted of a two-digit code where the First Digit represented the process circuit number (i.e., 1–feed preparation, 2–acid leaching and waste treatment, 3–rougher solvent extraction, 4–cleaner solvent extraction, 5–scandium solvent extraction) and the Second Digit was a letter representing the alphabetical order of units present in the circuit. The ID numbers are shown in the following flow diagram.

Figure 3-138: Identification Numbers Assigned to Facilitate the Flow Diagram Simulation Work



Descriptions of each of the 51 worksheets used in the REESim simulation are summarized in the following table. *Note that the two-digit ID code was used to start the name each of the 46 worksheet tabs (unit operations) incorporated into the REESim simulation of the process flow diagram.*

Table 3-70: Summary of ID Numbers, Worksheet/tab Names and Unit Operation Descriptions

Worksheet	Worksheet Name	Worksheet/Tab Description
1	Overview	Standard “overview” tab used by REESim.
2	Flowsheet	Standard “flowsheet” tab used by REESim.
3	Nodes	Standard “node” tab used by REESim.
4	Reagents	Standard “reagents” tab used by REESim.
5	Blank	Standard “blank” tab used by REESim.
6	1A_Feed	Worksheet for entering feed characterization data.
7	1B_JawCrush	Worksheet to simulate jaw crusher.
8	1C_VScreen	Worksheet to simulate vibrating screen.
9	1D_RollCrush	Worksheet to simulate roll crusher.
10	1E_Mill	Worksheet to simulate grind mill.
11	1F_RScreen	Worksheet to simulate rotary screen.
12	1G_Feed_Roaster	Worksheet to simulate feed roaster.
13	2A_pHTank	Worksheet to simulate acid leach pH control tank.
14	2B_LeachTanks	Worksheet to simulate acid leach tanks (3 tanks).
15	2C_LeachThickener	Worksheet to simulate acid leach thickener.
16	2D_LeachFilter	Worksheet to simulate thickener underflow filter.
17	2E_WaterTreat	Worksheet to simulate wastewater treatment tank.
18	2F_WaterFilter	Worksheet to simulate waste sludge filter.
19	2G_WaterTank	Worksheet to simulate process water storage.
20	2H_Splitter	Worksheet to simulate splitting of rougher raffinate.
21	2I_Junction	Worksheet to simulate blending of recycle streams.
22	2J_PLSTank	Worksheet to simulate PLS storage and blending.
23	3A_REhTank	Worksheet to simulate rougher SX Eh control tank.
24	3B_RpHTank	Worksheet to simulate rougher SX pH control tank.
25	3C_RBufferTank	Worksheet to simulate rougher SX buffer blend tank.
26	3D_SXRLoad1	Worksheet to simulate rougher SX stage 1 loading.
27	3E_SXRLoad2	Worksheet to simulate rougher SX stage 2 loading.
28	3F_SXRLoad3	Worksheet to simulate rougher SX stage 3 loading.
29	3G_SXRScrub	Worksheet to simulate rougher SX scrubbing.
30	3H_SXRStrip1	Worksheet to simulate rougher SX stage 1 stripping.
31	3I_SXRStrip2	Worksheet to simulate rougher SX stage 2 stripping.
32	4A_CEhTank	Worksheet to simulate cleaner SX Eh tank.
33	4B_CpHTank	Worksheet to simulate cleaner SX pH tank.
34	4C_CBufferTank	Worksheet to simulate cleaner SX buffer blend tank.
35	4D_SXCLoad1	Worksheet to simulate cleaner SX stage 1 loading
36	4E_SXCLoad2	Worksheet to simulate cleaner SX stage 2 loading.
37	4F_SXCLoad3	Worksheet to simulate cleaner SS stage 3 loading.
38	4G_SXCStrip1	Worksheet to simulate cleaner SX stage 1 stripping.
39	4H_SXCStrip2	Worksheet to simulate cleaner SX stage 2 stripping.
40	4I_SXCStrip3	Worksheet to simulate cleaner SX stage 3 stripping.
41	4J_PEhTank	Worksheet to simulate product REE Eh tank.
42	4K_PpHTank	Worksheet to simulate product REE pH tank.
43	4L_POATank	Worksheet to simulate product REE oxalic acid tank.
44	4M_ProdPrecip	Worksheet to simulate product REE precipitation tank.
45	4N_ProdFilter	Worksheet to simulate product REE filter.
46	4O_ProdRoast	Worksheet to simulate product REE roaster.
47	5A_ScBufferTank	Worksheet to simulate scandium buffer blend tank.
48	5B_SXWash1	Worksheet to simulate scandium SX stage 1 wash.
49	5C_SXWash2	Worksheet to simulate scandium SX stage 2 wash.
50	5D_SXSaponify	Worksheet to simulate scandium SX saponification.
51	5E_ScFilter	Worksheet to simulate scandium product filter.

3.5.3 Component Flow Rates Generated by REESim for Project MMA 29956

The complete printout of all of the pages from the REESim package for the current simulation is provided in the *Appendix*. The package tracked the mass rates of coal (organic matter), rock (inorganic matter) and elemental constituents (REEs and other contaminants) that moved between the various unit operations in the process flow diagram.

The mass rate partitioning was based on partition factors that were either 1) manually entered as known values from experimental data or engineering analyses or 2) automatically calculated by the software based on modelling routines. These partition values, which are mathematically equivalent to elemental recoveries, extraction efficiencies, etc., are presented and discussed in the first section of this report chapter. The numerical partition values are clearly listed in the *Appendix* of this chapter as part of each of the 46 process worksheet printouts.

Due to the large amount of information contained within these simulation output files; summary tables were created for the mass flow rates of components in the primary streams entering and exiting each of the five circuits in the process flow diagram. The summary tables include:

- Mass Rate Summary for the Overall Plant (*Table 3-71*).
- Mass Rate Summary for the Feed Preparation Circuit (*Table 3-72*).
- Mass Rate Summary for the Acid Leaching and Waste Treatment Circuit *(Table 3-73*).
- Mass Rate Summary for the Rougher Solvent Extraction Circuit (*Table 3-74*).
- Mass Rate Summary for the Cleaner Solvent Extraction Circuit (*Table 3-75*).
- Mass Rate Summary for the Scandium Solvent Extraction Circuit (*Table 3-76*).

The mass flow rates listed in these summary tables (and in the individual REESim worksheets files) were used to estimate the size of processing units required for the economic analysis section of this report.

The mass rate summary for the overall plant shows that the facility would be expected to treat 1,000 kg/hr of feed (dry solids) and generate two product output streams, i.e.:

- Mixed Rare Earth Oxide Final Product
- Scandium Filter Cake Final Product

In addition, the circuitry would be expected to also generate four waste streams, which include:

- Feed Roaster Dust/Volatiles
- Acid Leach Filter Cake (Neutralized Waste)
- Water Treatment Filter Cake (Neutralized Waste)
- REO Roaster Dust/Volatiles

Table 3-71: Mass Rate Summary for the Overall Plant

1-5 - TOTAL PLANT										
	IN	**OUT**						**TOTALS**		
	Raw Feed	**Roast Dust/Vol.**	**AL Filter Cake**	**WT Filter Cake**	**REO Product**	**REO Dust**	**Sc Filter Cake**	**Mass In**	**Mass Out**	**Mass Delta**
Mass (kg/hr.)										
Total Mass	1000.00	10.00	928.45	61.48	0.06	0.00	0.00	1000.00	1000.00	0.00
Combustible	188.76	1.89	186.88	0.00	0.00	0.00	0.00	188.76	188.76	0.00
Ash	811.24	8.11	741.57	61.48	0.06	0.00	0.00	811.24	811.24	0.00
Ash Constituents - Primary (kg/hr.)										
Unknown	407.78	4.08	403.70	0.00	0.00	0.00	0.00	407.78	407.78	0.00
Al	118.36	1.18	100.29	16.89	0.00	0.00	0.00	118.36	118.36	0.00
Ca	13.39	0.13	1.02	12.23	0.00	0.00	0.00	13.39	13.39	0.00
Fe	73.82	0.74	40.73	32.35	0.00	0.00	0.00	73.82	73.82	0.00
Si	197.54	1.98	195.56	0.00	0.00	0.00	0.00	197.54	197.54	0.00
Ash Constituents - Trace (gm/hr.)										
Sc	16.251	0.163	13.428	0.244	0.083	0.004	2.329	16.251	16.251	0.000
Y	33.450	0.334	29.181	0.035	3.704	0.195	0.000	33.450	33.450	0.000
La	62.369	0.624	44.298	6.672	10.236	0.539	0.000	62.369	62.369	0.000
Ce	132.506	1.325	95.259	8.006	26.519	1.396	0.000	132.506	132.506	0.000
Pr	15.359	0.154	11.017	1.193	2.845	0.150	0.000	15.359	15.359	0.000
Nd	59.191	0.592	44.917	0.123	12.880	0.678	0.000	59.191	59.191	0.000
Pm	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Sm	10.287	0.103	8.306	0.017	1.768	0.093	0.000	10.287	10.287	0.000
Eu	1.631	0.016	1.455	0.001	0.150	0.008	0.000	1.631	1.631	0.000
Gd	8.437	0.084	7.163	0.289	0.856	0.045	0.000	8.437	8.437	0.000
Tb	1.196	0.012	0.441	0.121	0.591	0.031	0.000	1.196	1.196	0.000
Dy	6.838	0.068	6.050	0.026	0.659	0.035	0.000	6.838	6.838	0.000
Ho	1.261	0.013	1.248	0.000	0.000	0.000	0.000	1.261	1.261	0.000
Er	3.610	0.036	3.574	0.000	0.000	0.000	0.000	3.610	3.610	0.000
Tm	0.508	0.005	0.362	0.037	0.099	0.005	0.000	0.508	0.508	0.000
Yb	3.275	0.033	2.920	0.023	0.284	0.015	0.000	3.275	3.275	0.000
Lu	0.477	0.005	0.358	0.030	0.080	0.004	0.000	0.477	0.477	0.000
Th	0.001	0.000	0.001	0.000	0.000	0.000	0.000	0.001	0.001	0.000
U	0.001	0.000	0.001	0.000	0.000	0.000	0.000	0.001	0.001	0.000

Note: U and Th not determined or tracked. Placeholder values of 0.001 used for raw feed.

Table 3-72: Mass Rate Summary for the Feed Preparation Circuit

1 - FEED PREPARATION CIRCUIT						
	IN	**OUT**		**TOTALS**		
	Raw Feed	**AL Filter Cake**	**WT Filter Cake**	**Mass In**	**Mass Out**	**Mass Delta**
Mass (kg/hr.)						
Total Mass	1000.00	10.00	990.00	1000.00	1000.00	0.00
Combustible	188.76	1.89	186.88	188.76	188.76	0.00
Ash	811.24	8.11	803.12	811.24	811.24	0.00
Ash Constituents - Primary (kg/hr.)						
Unknown	407.78	4.08	403.70	407.78	407.78	0.00
Al	118.36	1.18	117.18	118.36	118.36	0.00
Ca	13.39	0.13	13.25	13.39	13.39	0.00
Fe	73.82	0.74	73.08	73.82	73.82	0.00
Si	197.54	1.98	195.56	197.54	197.54	0.00
Ash Constituents - Trace (gm/hr.)						
Sc	16.251	0.163	16.088	16.251	16.251	0.000
Y	33.450	0.334	33.115	33.450	33.450	0.000
La	62.369	0.624	61.745	62.369	62.369	0.000
Ce	132.506	1.325	131.181	132.506	132.506	0.000
Pr	15.359	0.154	15.205	15.359	15.359	0.000
Nd	59.191	0.592	58.599	59.191	59.191	0.000
Pm	0.000	0.000	0.000	0.000	0.000	0.000
Sm	10.287	0.103	10.184	10.287	10.287	0.000
Eu	1.631	0.016	1.615	1.631	1.631	0.000
Gd	8.437	0.084	8.352	8.437	8.437	0.000
Tb	1.196	0.012	1.184	1.196	1.196	0.000
Dy	6.838	0.068	6.770	6.838	6.838	0.000
Ho	1.261	0.013	1.248	1.261	1.261	0.000
Er	3.610	0.036	3.574	3.610	3.610	0.000
Tm	0.508	0.005	0.503	0.508	0.508	0.000
Yb	3.275	0.033	3.243	3.275	3.275	0.000
Lu	0.477	0.005	0.472	0.477	0.477	0.000
Th	0.001	0.000	0.001	0.001	0.001	0.000
U	0.001	0.000	0.001	0.001	0.001	0.000

Table 3-73: Mass Rate Summary for the Acid Leaching and Waste Treatment Circuit

2 - ACID LEACHING AND WASTE TREATMENT											
	IN					**OUT**			**TOTALS**		
	Roasted Product	**R-Load3 Raffinate**	**Sc-Filter Effluent**	**R-Scrub Aqueous**	**Sc-Wash1 Aqueous**	**AL Filter Cake**	**WT Filter Cake**	**Product PLS**	**Mass In**	**Mass Out**	**Mass Delta**
Mass (kg/hr.)											
Total Mass	990.00	68.99	0.00	26.50	0.00	928.45	61.48	95.55	1085.48	1085.48	0.00
Combustible	186.88	0.00	0.00	0.00	0.00	186.88	0.00	0.00	186.88	186.88	0.00
Ash	803.12	68.99	0.00	26.50	0.00	741.57	61.48	95.55	898.60	898.60	0.00
Ash Constituents - Primary (kg/hr.)											
Unknown	403.70	0.00	0.00	0.00	0.00	403.70	0.00	0.00	403.70	403.70	0.00
Al	117.18	18.95	0.00	3.14	0.00	100.29	16.89	22.09	139.26	139.26	0.00
Ca	13.25	13.72	0.00	4.65	0.00	1.02	12.23	18.37	31.62	31.62	0.00
Fe	73.08	36.30	0.00	18.70	0.00	40.73	32.35	55.00	128.09	128.09	0.00
Si	195.56	0.00	0.00	0.00	0.00	195.56	0.00	0.00	195.56	195.56	0.00
Ash Constituents - Trace (gm/hr.)											
Sc	16.088	0.018	0.230	0.000	0.278	13.428	0.244	2.942	16.614	16.614	0.000
Y	33.115	0.039	0.000	0.000	0.000	29.181	0.035	3.938	33.155	33.155	0.000
La	61.745	7.485	0.000	0.018	0.000	44.298	6.672	18.278	69.248	69.248	0.000
Ce	131.181	8.980	0.000	0.015	0.001	95.259	8.006	36.911	140.177	140.177	0.000
Pr	15.205	1.338	0.000	0.001	0.000	11.017	1.193	4.334	16.544	16.544	0.000
Nd	58.599	0.137	0.000	0.009	0.000	44.917	0.123	13.704	58.745	58.745	0.000
Pm	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Sm	10.184	0.019	0.000	0.000	0.000	8.306	0.017	1.880	10.203	10.203	0.000
Eu	1.615	0.002	0.000	0.000	0.000	1.455	0.001	0.160	1.616	1.616	0.000
Gd	8.352	0.324	0.000	0.000	0.000	7.163	0.289	1.225	8.677	8.677	0.000
Tb	1.184	0.135	0.000	0.000	0.000	0.441	0.121	0.758	1.319	1.319	0.000
Dy	6.770	0.029	0.000	0.000	0.000	6.050	0.026	0.723	6.799	6.799	0.000
Ho	1.248	0.000	0.000	0.000	0.000	1.248	0.000	0.000	1.248	1.248	0.000
Er	3.574	0.000	0.000	0.000	0.000	3.574	0.000	0.000	3.574	3.574	0.000
Tm	0.503	0.041	0.000	0.000	0.000	0.362	0.037	0.146	0.544	0.544	0.000
Yb	3.243	0.026	0.000	0.000	0.000	2.920	0.023	0.325	3.268	3.268	0.000
Lu	0.472	0.033	0.000	0.000	0.000	0.358	0.030	0.117	0.505	0.505	0.000
Th	0.001	0.000	0.000	0.000	0.000	0.001	0.000	0.000	0.001	0.001	0.000
U	0.001	0.000	0.000	0.000	0.000	0.001	0.000	0.000	0.001	0.001	0.000

Table 3-74: Mass Rate Summary for the Rougher Solvent Extraction Circuit

3 - ROUGHER SOLVENT EXTRACTION										
	IN			OUT				TOTALS		
	Feed PLS	Cleaner Recycle	Sapon Organic	R-Load3 Raffinate	R-Scrub Aqueous	R-Strip1 Aqueous	R-Strip2 Organic	Mass In	Mass Out	Mass Delta
Mass (kg/hr.)										
Total Mass	95.55	0.03	0.00	68.99	26.50	0.10	0.00	95.58	95.58	0.00
Combustible	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ash	95.55	0.03	0.00	68.99	26.50	0.10	0.00	95.58	95.58	0.00
Ash Constituents - Primary (kg/hr.)										
Unknown	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Al	22.09	0.00	0.00	18.95	3.14	0.00	0.00	22.09	22.09	0.00
Ca	18.37	0.02	0.00	13.72	4.65	0.03	0.00	18.40	18.40	0.00
Fe	55.00	0.00	0.00	36.30	18.70	0.00	0.00	55.00	55.00	0.00
Si	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ash Constituents - Trace (gm/hr.)										
Sc	2.942	0.001	0.013	0.018	0.000	0.087	2.850	2.955	2.955	0.000
Y	3.938	0.039	0.000	0.039	0.000	3.938	0.000	3.977	3.977	0.000
La	18.278	0.317	0.000	7.485	0.018	11.092	0.000	18.595	18.595	0.000
Ce	36.911	0.284	0.000	8.980	0.015	28.199	0.001	37.195	37.195	0.000
Pr	4.334	0.030	0.000	1.338	0.001	3.025	0.000	4.365	4.365	0.000
Nd	13.704	0.137	0.000	0.137	0.009	13.695	0.000	13.841	13.841	0.000
Pm	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Sm	1.880	0.019	0.000	0.019	0.000	1.880	0.000	1.899	1.899	0.000
Eu	0.160	0.002	0.000	0.002	0.000	0.160	0.000	0.162	0.162	0.000
Gd	1.225	0.009	0.000	0.324	0.000	0.910	0.000	1.234	1.234	0.000
Tb	0.758	0.006	0.000	0.135	0.000	0.629	0.000	0.764	0.764	0.000
Dy	0.723	0.007	0.000	0.029	0.000	0.700	0.000	0.730	0.730	0.000
Ho	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Er	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Tm	0.146	0.001	0.000	0.041	0.000	0.106	0.000	0.147	0.147	0.000
Yb	0.325	0.005	0.000	0.026	0.000	0.305	0.000	0.330	0.330	0.000
Lu	0.117	0.001	0.000	0.033	0.000	0.085	0.000	0.118	0.118	0.000
Th	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
U	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000

Table 3-75: Mass Rate Summary for the Cleaner Solvent Extraction Circuit

4 - CLEANER SOLVENT EXTRACTION							
	IN	**OUT**			**TOTALS**		
	R-Strip1 Aqueous	**Cleaner Recycle**	**REO Product**	**REO Dust**	**Mass In**	**Mass Out**	**Mass Delta**
Mass (kg/hr.)							
Total Mass	0.096	0.030	0.062	0.003	0.096	0.096	0.000
Combustible	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ash	0.096	0.030	0.062	0.003	0.096	0.096	0.000
Ash Constituents - Primary (kg/hr.)							
Unknown	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Al	0.004	0.004	0.000	0.000	0.004	0.004	0.000
Ca	0.026	0.024	0.002	0.000	0.026	0.026	0.000
Fe	0.001	0.001	0.000	0.000	0.001	0.001	0.000
Si	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ash Constituents - Trace (gm/hr.)							
Sc	0.087	0.001	0.083	0.004	0.087	0.087	0.000
Y	3.938	0.039	3.704	0.195	3.938	3.938	0.000
La	11.092	0.317	10.236	0.539	11.092	11.092	0.000
Ce	28.199	0.284	26.519	1.396	28.199	28.199	0.000
Pr	3.025	0.030	2.845	0.150	3.025	3.025	0.000
Nd	13.695	0.137	12.880	0.678	13.695	13.695	0.000
Pm	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Sm	1.880	0.019	1.768	0.093	1.880	1.880	0.000
Eu	0.160	0.002	0.150	0.008	0.160	0.160	0.000
Gd	0.910	0.009	0.856	0.045	0.910	0.910	0.000
Tb	0.629	0.006	0.591	0.031	0.629	0.629	0.000
Dy	0.700	0.007	0.659	0.035	0.700	0.700	0.000
Ho	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Er	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Tm	0.106	0.001	0.099	0.005	0.106	0.106	0.000
Yb	0.305	0.005	0.284	0.015	0.305	0.305	0.000
Lu	0.085	0.001	0.080	0.004	0.085	0.085	0.000
Th	0.000	0.000	0.000	0.000	0.000	0.000	0.000
U	0.000	0.000	0.000	0.000	0.000	0.000	0.000

Table 3-76: Mass Rate Summary for the Scandium Solvent Extraction Circuit

5 - SCANDIUM SOLVENT EXTRACTION								
	IN	**OUT**				**TOTALS**		
	R-Strip2 Organic	**Sc-Wash1 Aqueous**	**Sapon Organic**	**Sc Filter Cake**	**Sc Filter Effluent**	**Mass In**	**Mass Out**	**Mass Delta**
Mass (kg/hr.)								
Total Mass	0.003	0.000	0.000	0.002	0.000	0.003	0.003	0.000
Combustible	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ash	0.003	0.000	0.000	0.002	0.000	0.003	0.003	0.000
Ash Constituents - Primary (kg/hr.)								
Unknown	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Al	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ca	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Fe	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Si	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ash Constituents - Trace (gm/hr.)								
Sc	2.850	0.278	0.013	2.329	0.230	2.850	2.850	0.000
Y	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
La	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ce	0.001	0.001	0.000	0.000	0.000	0.001	0.001	0.000
Pr	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Nd	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Pm	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Sm	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Eu	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Gd	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Tb	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Dy	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ho	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Er	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Tm	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Yb	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Lu	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Th	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
U	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000

3.5.4 Circuit-by-Circuit Performance Indicators

Process performance indicators, such as mass yield, component assays, and product recoveries, were calculated for the overall plant and for each of the five individual process circuits included in the process flow diagram. Using the mass flow rates listed in *Table 3-71* through *Table 3-76*, the following tabular summaries of performance indicates were established:

- Overall Plant (*Table 3-77* = Assays; *Table 3-78* = Distributions)
- Feed Preparation Circuit (*Table 3-79* = Assays; *Table 3-80* = Distributions)
- Acid Leaching and Waste Treatment Circuit (*Table 3-81* = Assays; *Table 3-82* = Distributions)
- Rougher Solvent Extraction Circuit (*Table 3-83* = Assays; *Table 3-84* = Distributions)
- Cleaner Solvent Extraction Circuit (*Table 3-85* = Assays; *Table 3-86* = Distributions)
- Scandium Solvent Extraction Circuit (*Table 3-87* = Assays; *Table 3-88* = Distributions)

3.5.4.1 Summary Discussion of the Simulation Results for Project MMA 29956

In general, the simulation results can be used to derive several key observations related to the final process flow diagram. These insights include the following.

- In the feed preparation circuit, the roasting operation is an essential step in generating a dry solid feedstock that responses well to REE leaching. This operation also ensures that essentially no water enters with the dry feed, which is very beneficial in establishing proper water balances for the facility.
- Extraction efficiencies for acid leaching achieved an average value of only 23.96 percent for the REEs of interest in this project.
    - The lowest recovery of 9.81 percent was obtained for europium, while a high of 63.35 percent was realized for terbium.
    - Scandium, which has the highest market value of the elements under consideration, is associated with an extraction recovery of only 18.1 percent during acid leaching.
- The simulation data indicated that the circuits in the proposed flow process diagram should be able to produce an REO product with a purity level of 97.4 percent or higher. The same simulation indicates a scandium purity of 99.999 percent.
    - *Note: Incorporation of actual experience with low roasting conversion and poor acid leaching efficiencies into REESim reveals a predicted plant recovery of total rare earth elements of only 17.03 percent. Predicted Scandium recovery can be even poorer, at 0.65 percent.*
- The recycling of raffinate back to the primary pH control tank for acid leaching was found to be necessary to maintain proper flow and acid balances in the process facility.
    - A recirculation rate of approximately 85-90 percent was established as a reasonable value for balancing flow/acid demands against a build-up of unwanted impurities in the PLS.

Table 3-77: REESim Predicted Assay Values for the Overall Plant Operations

1-5 - TOTAL PLANT										
	IN	**OUT**								
	Raw Feed	**Roast Dust/Vol.**	**AL Filter Cake**	**WT Filter Cake**	**REO Product**	**REO Dust**	**Sc Filter Cake**	**Feed In**	**Product Out**	**Delta --**
Circuit Performance Indicators										
Yield (%)	100.00	1.00	92.84	6.15	0.01	0.00	0.00	100.00	100.00	--
Ash (%)	81.12	81.12	79.87	100.00	100.00	100.00	100.00	81.12	81.12	--
Impurity Assays (%, dry)										
Unknown %	40.78	40.78	43.48	0.00	0.00	0.00	0.00	40.78	40.78	0.00
Al %	11.84	11.84	10.80	27.46	0.02	0.02	0.00	11.84	11.84	0.00
Ca %	1.34	1.34	0.11	19.89	2.52	2.52	0.00	1.34	1.34	0.00
Fe %	7.38	7.38	4.39	52.62	0.03	0.03	0.00	7.38	7.38	0.00
Si %	19.75	19.75	21.06	0.00	0.00	0.00	0.00	19.75	19.75	0.00
Trace Assays (%, dry)										
Sc %	0.0016	0.0016	0.0014	0.0004	0.1324	0.1324	99.9993	0.0016	0.0016	0.0000
Y %	0.0033	0.0033	0.0031	0.0001	5.9397	5.9397	0.0000	0.0033	0.0033	0.0000
La %	0.0062	0.0062	0.0048	0.0109	16.4162	16.4162	0.0001	0.0062	0.0062	0.0000
Ce %	0.0133	0.0133	0.0103	0.0130	42.5305	42.5305	0.0003	0.0133	0.0133	0.0000
Pr %	0.0015	0.0015	0.0012	0.0019	4.5631	4.5631	0.0000	0.0015	0.0015	0.0000
Nd %	0.0059	0.0059	0.0048	0.0002	20.6566	20.6566	0.0001	0.0059	0.0059	0.0000
Pm %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Sm %	0.0010	0.0010	0.0009	0.0000	2.8354	2.8354	0.0000	0.0010	0.0010	0.0000
Eu %	0.0002	0.0002	0.0002	0.0000	0.2412	0.2412	0.0000	0.0002	0.0002	0.0000
Gd %	0.0008	0.0008	0.0008	0.0005	1.3721	1.3721	0.0000	0.0008	0.0008	0.0000
Tb %	0.0001	0.0001	0.0000	0.0002	0.9480	0.9480	0.0000	0.0001	0.0001	0.0000
Dy %	0.0007	0.0007	0.0007	0.0000	1.0564	1.0564	0.0000	0.0007	0.0007	0.0000
Ho %	0.0001	0.0001	0.0001	0.0000	0.0000	0.0000	0.0000	0.0001	0.0001	0.0000
Er %	0.0004	0.0004	0.0004	0.0000	0.0000	0.0000	0.0000	0.0004	0.0004	0.0000
Tm %	0.0001	0.0001	0.0000	0.0001	0.1593	0.1593	0.0000	0.0001	0.0001	0.0000
Yb %	0.0003	0.0003	0.0003	0.0000	0.4560	0.4560	0.0000	0.0003	0.0003	0.0000
Lu %	0.0000	0.0000	0.0000	0.0000	0.1279	0.1279	0.0000	0.0000	0.0000	0.0000
Th %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
U %	0.0000	0.0000	0.0000	0.0000	0.0001	0.0001	0.0000	0.0000	0.0000	0.0000
TREE%	0.0357	0.0357	0.0291	0.0274	97.4349	97.4349	99.9999	0.0357	0.0357	--

Note: The raw feed parameters are based on a coarse rejects sample collected from the Leatherwood Plant on 11-16-2018. The analytical information from this sample (as well as others Collected from this plant) is located on the RESEARCHER DATABASE.

Table 3-78: REESim Distribution Values for the Overall Plant Adjusted for Pilot Plant Experience

1-5 - TOTAL PLANT										
	IN	**OUT**								
	Raw Feed	**Roast Dust/Vol.**	**AL Filter Cake**	**WT Filter Cake**	**REO Product**	**REO Dust**	**Sc Filter Cake**	**Feed In**	**Product Out**	**Delta --**
Mass Distributions										
Total	100.00	1.00	92.84	6.15	0.01	0.00	0.00	100.00	100.00	--
Ash	100.00	1.00	99.00	0.00	0.00	0.00	0.00	100.00	100.00	--
Impurity Distributions										
Unknown	100.00	1.00	82.63	1.50	0.51	0.03	14.33	100.00	100.00	0.00
Al	100.00	1.00	87.24	0.11	11.07	0.58	0.00	100.00	100.00	0.00
Ca	100.00	1.00	71.03	10.70	16.41	0.86	0.00	100.00	100.00	0.00
Fe	100.00	1.00	71.89	6.04	20.01	1.05	0.00	100.00	100.00	0.00
Si	100.00	1.00	71.73	7.77	18.53	0.98	0.00	100.00	100.00	0.00
Trace Element Distributions										
Sc	100.00	1.00	82.63	1.50	0.51	0.03	14.33	100.00	100.00	0.00
Y	100.00	1.00	87.24	0.11	11.07	0.58	0.00	100.00	100.00	0.00
La	100.00	1.00	71.03	10.70	16.41	0.86	0.00	100.00	100.00	0.00
Ce	100.00	1.00	71.89	6.04	20.01	1.05	0.00	100.00	100.00	0.00
Pr	100.00	1.00	71.73	7.77	18.53	0.98	0.00	100.00	100.00	0.00
Nd	100.00	1.00	75.89	0.21	21.76	1.15	0.00	100.00	100.00	0.00
Pm	100.00	1.00	99.00	0.00	0.00	0.00	0.00	100.00	100.00	0.00
Sm	100.00	1.00	80.74	0.16	17.19	0.90	0.00	100.00	100.00	0.00
Eu	100.00	1.00	89.20	0.09	9.22	0.49	0.00	100.00	100.00	0.00
Gd	100.00	1.00	84.90	3.43	10.14	0.53	0.00	100.00	100.00	0.00
Tb	100.00	1.00	36.90	10.08	49.42	2.60	0.00	100.00	100.00	0.00
Dy	100.00	1.00	88.48	0.38	9.63	0.51	0.00	100.00	100.00	0.00
Ho	100.00	1.00	99.00	0.00	0.00	0.00	0.00	100.00	100.00	0.00
Er	100.00	1.00	99.00	0.00	0.00	0.00	0.00	100.00	100.00	0.00
Tm	100.00	1.00	71.22	7.21	19.54	1.03	0.00	100.00	100.00	0.00
Yb	100.00	1.00	89.16	0.70	8.68	0.46	0.00	100.00	100.00	0.00
Lu	100.00	1.00	75.16	6.24	16.72	0.88	0.00	100.00	100.00	0.00
Th	100.00	1.00	93.70	5.23	0.07	0.00	0.00	100.00	100.00	0.00
U	100.00	1.00	90.05	2.59	6.04	0.32	0.00	100.00	100.00	0.00
	100.00	1.00	75.70	4.72	17.03	0.90	0.65	100.00	100.00	0.00

Table 3-79: Assay Values of the Feed Preparation Circuit

1 - FEED PREPARATION CIRCUIT						
	IN	**OUT**				
	Raw	**Roast**	**Roasted**	**Feed**	**Product**	**Delta**
	Feed	**Dust/Vol.**	**Product**	**In**	**Out**	**--**
Circuit Performance Indicators						
Yield (%)	100.00	1.00	99.00	100.00	100.00	--
Ash (%)	81.12	81.12	81.12	81.12	81.12	--
Impurity Assays (%, dry)						
Unknown %	40.78	40.78	40.78	40.78	40.78	0.00
Al %	11.84	11.84	11.84	11.84	11.84	0.00
Ca %	1.34	1.34	1.34	1.34	1.34	0.00
Fe %	7.38	7.38	7.38	7.38	7.38	0.00
Si %	19.75	19.75	19.75	19.75	19.75	0.00
Trace Assays (%, dry)						
Sc %	0.0016	0.0016	0.0016	0.0016	0.0016	0.0000
Y %	0.0033	0.0033	0.0033	0.0033	0.0033	0.0000
La %	0.0062	0.0062	0.0062	0.0062	0.0062	0.0000
Ce %	0.0133	0.0133	0.0133	0.0133	0.0133	0.0000
Pr %	0.0015	0.0015	0.0015	0.0015	0.0015	0.0000
Nd %	0.0059	0.0059	0.0059	0.0059	0.0059	0.0000
Pm %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Sm %	0.0010	0.0010	0.0010	0.0010	0.0010	0.0000
Eu %	0.0002	0.0002	0.0002	0.0002	0.0002	0.0000
Gd %	0.0008	0.0008	0.0008	0.0008	0.0008	0.0000
Tb %	0.0001	0.0001	0.0001	0.0001	0.0001	0.0000
Dy %	0.0007	0.0007	0.0007	0.0007	0.0007	0.0000
Ho %	0.0001	0.0001	0.0001	0.0001	0.0001	0.0000
Er %	0.0004	0.0004	0.0004	0.0004	0.0004	0.0000
Tm %	0.0001	0.0001	0.0001	0.0001	0.0001	0.0000
Yb %	0.0003	0.0003	0.0003	0.0003	0.0003	0.0000
Lu %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Th %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
U %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
TREE%	0.0357	0.0357	0.0357	0.0357	0.0357	--

Note: The raw feed parameters are based on a coarse rejects sample collected from the Leatherwood Plant on 11-16-2018. The analytical information from this sample (as well as others Collected from this plant) is located on the RESEARCHER DATABASE.

Table 3-80: Distribution Values to the Feed Preparation Circuit

1 - FEED PREPARATION CIRCUIT						
	IN	**OUT**				
	Raw	**Roast**	**Roasted**	**Feed**	**Product**	**Delta**
	Feed	**Dust/Vol.**	**Product**	**In**	**Out**	**--**
Mass Distributions						
Total	100.00	1.00	99.00	100.00	100.00	--
Ash	100.00	1.00	99.00	100.00	100.00	--
Impurity Distributions						
Unknown	100.00	1.00	99.00	100.00	100.00	0.00
Al	100.00	1.00	99.00	100.00	100.00	0.00
Ca	100.00	1.00	99.00	100.00	100.00	0.00
Fe	100.00	1.00	99.00	100.00	100.00	0.00
Si	100.00	1.00	99.00	100.00	100.00	0.00
Trace Element Distributions						
Sc	100.00	1.00	99.00	100.00	100.00	0.00
Y	100.00	1.00	99.00	100.00	100.00	0.00
La	100.00	1.00	99.00	100.00	100.00	0.00
Ce	100.00	1.00	99.00	100.00	100.00	0.00
Pr	100.00	1.00	99.00	100.00	100.00	0.00
Nd	100.00	1.00	99.00	100.00	100.00	0.00
Pm	100.00	1.00	99.00	100.00	100.00	0.00
Sm	100.00	1.00	99.00	100.00	100.00	0.00
Eu	100.00	1.00	99.00	100.00	100.00	0.00
Gd	100.00	1.00	99.00	100.00	100.00	0.00
Tb	100.00	1.00	99.00	100.00	100.00	0.00
Dy	100.00	1.00	99.00	100.00	100.00	0.00
Ho	100.00	1.00	99.00	100.00	100.00	0.00
Er	100.00	1.00	99.00	100.00	100.00	0.00
Tm	100.00	1.00	99.00	100.00	100.00	0.00
Yb	100.00	1.00	99.00	100.00	100.00	0.00
Lu	100.00	1.00	99.00	100.00	100.00	0.00
Th	100.00	1.00	99.00	100.00	100.00	0.00
U	100.00	1.00	99.00	100.00	100.00	0.00
	100.00	1.00	99.00	100.00	100.00	0.00

Table 3-81: Assay Values for the Acid Leaching and Waste Treatment Circuit

2 - ACID LEACHING AND WASTE TREATMENT

	IN					**OUT**					
	Roasted Product	**R-Load3 Raffinate**	**Sc-Filter Effluent**	**R-Scrub Aqueous**	**Sc-Wash1 Aqueous**	**AL Filter Cake**	**WT Filter Cake**	**Product PLS**	**Feed In**	**Product Out**	**Delta --**
Circuit Performance Indicators											
Yield (%)	99.00	6.90	0.00	2.65	0.00	92.84	6.15	9.55	108.55	108.55	--
Ash (%)	81.12	100.00	100.00	100.00	100.00	79.87	100.00	100.00	82.78	82.78	--
Impurity Assays (%, dry)											
Unknown %	40.78	0.00	0.00	0.00	0.00	43.48	0.00	0.00	37.19	37.19	0.00
Al %	11.84	27.46	0.00	11.86	0.03	10.80	27.46	23.12	12.83	12.83	0.00
Ca %	1.34	19.89	0.00	17.56	0.23	0.11	19.89	19.23	2.91	2.91	0.00
Fe %	7.38	52.62	0.00	70.58	0.01	4.39	52.62	57.56	11.80	11.80	0.00
Si %	19.75	0.00	0.00	0.00	0.00	21.06	0.00	0.00	18.02	18.02	0.00
Trace Assays (%, dry)											
Sc %	0.0016	0.0000	99.9985	0.0000	99.1543	0.0014	0.0004	0.0031	0.0015	0.0015	0.0000
Y %	0.0033	0.0001	0.0000	0.0000	0.0348	0.0031	0.0001	0.0041	0.0031	0.0031	0.0000
La %	0.0062	0.0108	0.0001	0.0001	0.0980	0.0048	0.0109	0.0191	0.0064	0.0064	0.0000
Ce %	0.0133	0.0130	0.0003	0.0001	0.2491	0.0103	0.0130	0.0386	0.0129	0.0129	0.0000
Pr %	0.0015	0.0019	0.0000	0.0000	0.0267	0.0012	0.0019	0.0045	0.0015	0.0015	0.0000
Nd %	0.0059	0.0002	0.0001	0.0000	0.1210	0.0048	0.0002	0.0143	0.0054	0.0054	0.0000
Pm %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Sm %	0.0010	0.0000	0.0000	0.0000	0.0166	0.0009	0.0000	0.0020	0.0009	0.0009	0.0000
Eu %	0.0002	0.0000	0.0000	0.0000	0.0014	0.0002	0.0000	0.0002	0.0001	0.0001	0.0000
Gd %	0.0008	0.0005	0.0000	0.0000	0.0080	0.0008	0.0005	0.0013	0.0008	0.0008	0.0000
Tb %	0.0001	0.0002	0.0000	0.0000	0.0056	0.0000	0.0002	0.0008	0.0001	0.0001	0.0000
Dy %	0.0007	0.0000	0.0000	0.0000	0.0062	0.0007	0.0000	0.0008	0.0006	0.0006	0.0000
Ho %	0.0001	0.0000	0.0000	0.0000	0.0000	0.0001	0.0000	0.0000	0.0001	0.0001	0.0000
Er %	0.0004	0.0000	0.0000	0.0000	0.0000	0.0004	0.0000	0.0000	0.0003	0.0003	0.0000
Tm %	0.0001	0.0001	0.0000	0.0000	0.0009	0.0000	0.0001	0.0002	0.0001	0.0001	0.0000
Yb %	0.0003	0.0000	0.0000	0.0000	0.0027	0.0003	0.0000	0.0003	0.0003	0.0003	0.0000
Lu %	0.0000	0.0000	0.0000	0.0000	0.0007	0.0000	0.0000	0.0001	0.0000	0.0000	0.0000
Th %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
U %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
TREE%	0.0357	0.0270	99.9991	0.0002	99.7259	0.0291	0.0274	0.0894	0.0343	0.0343	--

Note: The raw feed parameters are based on a coarse rejects sample collected from the Leatherwood Plant on 11-16-2018. The analytical information from this sample (as well as others Collected from this plant) is located on the RESEARCHER DATABASE.

Table 3-82: Distribution Values for the Acid Leaching and Waste Treatment Circuit

2 - ACID LEACHING AND WASTE TREATMENT											
	IN					**OUT**					
	Roasted Product	**R-Load3 Raffinate**	**Sc-Filter Effluent**	**R-Scrub Aqueous**	**Sc-Wash1 Aqueous**	**AL Filter Cake**	**WT Filter Cake**	**Product PLS**	**Feed In**	**Product Out**	**Delta --**
Mass Distributions											
Total	99.00	6.90	0.00	2.65	0.00	92.84	6.15	9.55	108.55	108.55	--
Ash	99.00	0.00	0.00	0.00	0.00	99.00	0.00	0.00	99.00	99.00	--
Impurity Distributions											
Unknown	99.00	0.00	0.00	0.00	0.00	99.00	0.00	0.00	99.00	99.00	0.00
Al	99.00	16.01	0.00	2.65	0.00	84.73	14.27	18.66	117.66	117.66	0.00
Ca	99.00	102.50	0.00	34.76	0.00	7.63	91.36	137.27	236.26	236.26	0.00
Fe	99.00	49.17	0.00	25.33	0.00	55.17	43.82	74.50	173.50	173.50	0.00
Si	99.00	0.00	0.00	0.00	0.00	99.00	0.00	0.00	99.00	99.00	0.00
Trace Element Distributions											
Sc	99.00	0.11	1.42	0.00	1.71	82.63	1.50	18.10	102.24	102.24	0.00
Y	99.00	0.12	0.00	0.00	0.00	87.24	0.11	11.77	99.12	99.12	0.00
La	99.00	12.00	0.00	0.03	0.00	71.03	10.70	29.31	111.03	111.03	0.00
Ce	99.00	6.78	0.00	0.01	0.00	71.89	6.04	27.86	105.79	105.79	0.00
Pr	99.00	8.71	0.00	0.01	0.00	71.73	7.77	28.22	107.72	107.72	0.00
Nd	99.00	0.23	0.00	0.02	0.00	75.89	0.21	23.15	99.25	99.25	0.00
Pm	99.00	0.00	0.00	0.00	0.00	99.00	0.00	0.00	99.00	99.00	0.00
Sm	99.00	0.18	0.00	0.00	0.00	80.74	0.16	18.28	99.18	99.18	0.00
Eu	99.00	0.10	0.00	0.00	0.00	89.20	0.09	9.81	99.10	99.10	0.00
Gd	99.00	3.84	0.00	0.00	0.00	84.90	3.43	14.52	102.84	102.84	0.00
Tb	99.00	11.30	0.00	0.02	0.00	36.90	10.08	63.35	110.32	110.32	0.00
Dy	99.00	0.43	0.00	0.00	0.00	88.48	0.38	10.57	99.43	99.43	0.00
Ho	99.00	0.00	0.00	0.00	0.00	99.00	0.00	0.00	99.00	99.00	0.00
Er	99.00	0.00	0.00	0.00	0.00	99.00	0.00	0.00	99.00	99.00	0.00
Tm	99.00	8.09	0.00	0.00	0.00	71.22	7.21	28.66	107.09	107.09	0.00
Yb	99.00	0.78	0.00	0.00	0.00	89.16	0.70	9.92	99.78	99.78	0.00
Lu	99.00	7.00	0.00	0.00	0.00	75.16	6.24	24.60	106.00	106.00	0.00
Th	99.00	5.87	0.00	0.00	0.00	93.70	5.23	5.94	104.87	104.87	0.00
U	99.00	2.90	0.00	0.00	0.00	90.05	2.59	9.26	101.90	101.90	0.00
	99.00	5.22	0.06	0.01	0.08	75.70	4.72	23.96	104.37	104.37	0.00

Table 3-83: Assay Values for the Rougher Solvent Extraction Circuit

3 - ROUGHER SOLVENT EXTRACTION										
	IN			**OUT**						
	Feed	**Cleaner**	**Sapon**	**R-Load3**	**R-Scrub**	**R-Strip1**	**R-Strip2**	**Feed**	**Product**	**Delta**
	PLS	**Recycle**	**Organic**	**Raffinate**	**Aqueous**	**Aqueous**	**Organic**	**In**	**Out**	**--**
Circuit Performance Indicators										
Yield (%)	9.55	0.00	0.00	6.90	2.65	0.01	0.00	9.56	9.56	--
Ash (%)	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	--
Impurity Assays (%, dry)										
Unknown %	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Al %	23.12	12.54	0.00	27.46	11.86	3.95	0.00	23.11	23.11	0.00
Ca %	19.23	80.29	0.00	19.89	17.56	26.91	0.02	19.25	19.25	0.00
Fe %	57.56	4.31	0.00	52.62	70.58	1.37	0.00	57.55	57.55	0.00
Si %	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Trace Assays (%, dry)										
Sc %	0.0031	0.0019	99.9993	0.0000	0.0000	0.0915	99.9163	0.0031	0.0031	0.0000
Y %	0.0041	0.1312	0.0000	0.0001	0.0000	4.1182	0.0035	0.0042	0.0042	0.0000
La %	0.0191	1.0575	0.0001	0.0108	0.0001	11.5999	0.0097	0.0195	0.0195	0.0000
Ce %	0.0386	0.9459	0.0003	0.0130	0.0001	29.4900	0.0247	0.0389	0.0389	0.0000
Pr %	0.0045	0.1012	0.0000	0.0019	0.0000	3.1639	0.0027	0.0046	0.0046	0.0000
Nd %	0.0143	0.4564	0.0001	0.0002	0.0000	14.3221	0.0120	0.0145	0.0145	0.0000
Pm %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Sm %	0.0020	0.0627	0.0000	0.0000	0.0000	1.9659	0.0016	0.0020	0.0020	0.0000
Eu %	0.0002	0.0053	0.0000	0.0000	0.0000	0.1673	0.0001	0.0002	0.0002	0.0000
Gd %	0.0013	0.0303	0.0000	0.0005	0.0000	0.9513	0.0008	0.0013	0.0013	0.0000
Tb %	0.0008	0.0210	0.0000	0.0002	0.0000	0.6573	0.0006	0.0008	0.0008	0.0000
Dy %	0.0008	0.0234	0.0000	0.0000	0.0000	0.7325	0.0006	0.0008	0.0008	0.0000
Ho %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Er %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Tm %	0.0002	0.0035	0.0000	0.0001	0.0000	0.1104	0.0001	0.0002	0.0002	0.0000
Yb %	0.0003	0.0181	0.0000	0.0000	0.0000	0.3187	0.0003	0.0003	0.0003	0.0000
Lu %	0.0001	0.0028	0.0000	0.0000	0.0000	0.0887	0.0001	0.0001	0.0001	0.0000
Th %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
U %	0.0000	0.0000	0.0000	0.0000	0.0000	0.0001	0.0000	0.0000	0.0000	0.0000
TREE%	0.0894	2.8612	99.9999	0.0270	0.0002	67.7776	99.9730	0.0903	0.0903	--

Table 3-84: Distribution Values for the Rougher Solvent Extraction Circuit

3 - ROUGHER SOLVENT EXTRACTION	IN			OUT						
	Feed	**Cleaner**	**Sapon**	**R-Load3**	**R-Scrub**	**R-Strip1**	**R-Strip2**	**Feed**	**Product**	**Delta**
	PLS	**Recycle**	**Organic**	**Raffinate**	**Aqueous**	**Aqueous**	**Organic**	**In**	**Out**	**--**
Mass Distributions										
Total	9.55	0.00	0.00	6.90	2.65	0.01	0.00	9.56	9.56	--
Ash	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	--
Impurity Distributions										
Unknown	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Al	18.66	0.00	0.00	16.01	2.65	0.00	0.00	18.67	18.67	0.00
Ca	137.27	0.18	0.00	102.50	34.76	0.19	0.00	137.45	137.45	0.00
Fe	74.50	0.00	0.00	49.17	25.33	0.00	0.00	74.51	74.51	0.00
Si	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Trace Element Distributions										
Sc	18.10	0.00	0.08	0.11	0.00	0.54	17.54	18.19	18.19	0.00
Y	11.77	0.12	0.00	0.12	0.00	11.77	0.00	11.89	11.89	0.00
La	29.31	0.51	0.00	12.00	0.03	17.78	0.00	29.81	29.81	0.00
Ce	27.86	0.21	0.00	6.78	0.01	21.28	0.00	28.07	28.07	0.00
Pr	28.22	0.20	0.00	8.71	0.01	19.70	0.00	28.42	28.42	0.00
Nd	23.15	0.23	0.00	0.23	0.02	23.14	0.00	23.38	23.38	0.00
Pm	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Sm	18.28	0.18	0.00	0.18	0.00	18.27	0.00	18.46	18.46	0.00
Eu	9.81	0.10	0.00	0.10	0.00	9.81	0.00	9.90	9.90	0.00
Gd	14.52	0.11	0.00	3.84	0.00	10.78	0.00	14.63	14.63	0.00
Tb	63.35	0.53	0.00	11.30	0.02	52.55	0.00	63.87	63.87	0.00
Dy	10.57	0.10	0.00	0.43	0.00	10.24	0.00	10.67	10.67	0.00
Ho	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Er	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Tm	28.66	0.21	0.00	8.09	0.00	20.77	0.00	28.87	28.87	0.00
Yb	9.92	0.17	0.00	0.78	0.00	9.30	0.00	10.09	10.09	0.00
Lu	24.60	0.18	0.00	7.00	0.00	17.78	0.00	24.78	24.78	0.00
Th	5.94	0.91	0.00	5.87	0.00	0.98	0.00	6.85	6.85	0.00
U	9.26	0.06	0.00	2.90	0.00	6.42	0.00	9.33	9.33	0.00
	23.96	0.24	0.00	5.22	0.01	18.17	0.80	24.20	24.20	0.00

Table 3-85: Assay Values for the Cleaner Solvent Extraction Circuit

4 - CLEANER SOLVENT EXTRACTION							
	IN	**OUT**					
	R-Strip1 Aqueous	**Cleaner Recycle**	**REO Product**	**REO Dust**	**Feed In**	**Product Out**	**Delta --**
Circuit Performance Indicators							
Yield (%)	0.01	0.00	0.01	0.00	0.01	0.01	--
Ash (%)	100.00	100.00	100.00	100.00	100.00	100.00	--
Impurity Assays (%, dry)							
Unknown %	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Al %	3.95	12.54	0.02	0.02	3.95	3.95	0.00
Ca %	26.91	80.29	2.52	2.52	26.91	26.91	0.00
Fe %	1.37	4.31	0.03	0.03	1.37	1.37	0.00
Si %	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Trace Assays (%, dry)							
Sc %	0.091	0.002	0.132	0.132	0.091	0.091	0.000
Y %	4.118	0.131	5.940	5.940	4.118	4.118	0.000
La %	11.600	1.057	16.416	16.416	11.600	11.600	0.000
Ce %	29.490	0.946	42.530	42.530	29.490	29.490	0.000
Pr %	3.164	0.101	4.563	4.563	3.164	3.164	0.000
Nd %	14.322	0.456	20.657	20.657	14.322	14.322	0.000
Pm %	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Sm %	1.966	0.063	2.835	2.835	1.966	1.966	0.000
Eu %	0.167	0.005	0.241	0.241	0.167	0.167	0.000
Gd %	0.951	0.030	1.372	1.372	0.951	0.951	0.000
Tb %	0.657	0.021	0.948	0.948	0.657	0.657	0.000
Dy %	0.732	0.023	1.056	1.056	0.732	0.732	0.000
Ho %	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Er %	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Tm %	0.110	0.004	0.159	0.159	0.110	0.110	0.000
Yb %	0.319	0.018	0.456	0.456	0.319	0.319	0.000
Lu %	0.089	0.003	0.128	0.128	0.089	0.089	0.000
Th %	0.000	0.000	0.000	0.000	0.000	0.000	0.000
U %	0.000	0.000	0.000	0.000	0.000	0.000	0.000
TREE%	67.778	2.861	97.435	97.435	67.778	67.778	--

Table 3-86: Distribution Values for the Cleaner Solvent Extraction Circuit

4 - CLEANER SOLVENT EXTRACTION							
	IN	**OUT**					
	R-Strip1 Aqueous	**Cleaner Recycle**	**REO Product**	**REO Dust**	**Feed In**	**Product Out**	**Delta --**
Mass Distributions							
Total	0.01	0.00	0.01	0.00	0.01	0.01	--
Ash	0.00	0.00	0.00	0.00	0.00	0.00	--
Impurity Distributions							
Unknown	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Al	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ca	0.19	0.18	0.01	0.00	0.19	0.19	0.00
Fe	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Si	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Trace Element Distributions							
Sc	0.54	0.00	0.51	0.03	0.54	0.54	0.00
Y	11.77	0.12	11.07	0.58	11.77	11.77	0.00
La	17.78	0.51	16.41	0.86	17.78	17.78	0.00
Ce	21.28	0.21	20.01	1.05	21.28	21.28	0.00
Pr	19.70	0.20	18.53	0.98	19.70	19.70	0.00
Nd	23.14	0.23	21.76	1.15	23.14	23.14	0.00
Pm	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Sm	18.27	0.18	17.19	0.90	18.27	18.27	0.00
Eu	9.81	0.10	9.22	0.49	9.81	9.81	0.00
Gd	10.78	0.11	10.14	0.53	10.78	10.78	0.00
Tb	52.55	0.53	49.42	2.60	52.55	52.55	0.00
Dy	10.24	0.10	9.63	0.51	10.24	10.24	0.00
Ho	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Er	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Tm	20.77	0.21	19.54	1.03	20.77	20.77	0.00
Yb	9.30	0.17	8.68	0.46	9.30	9.30	0.00
Lu	17.78	0.18	16.72	0.88	17.78	17.78	0.00
Th	0.98	0.91	0.07	0.00	0.98	0.98	0.00
U	6.42	0.06	6.04	0.32	6.42	6.42	0.00
	18.17	0.24	17.03	0.90	18.17	18.17	0.00

Table 3-87: Assay Values for the Scandium Solvent Extraction Circuit

5 - SCANDIUM SOLVENT EXTRACTION								
	IN	**OUT**				**Circuit**	**Circuit**	**Circuit**
	R-Strip2	**Sc-Wash1**	**Sapon**	**Sc Filter**	**Sc Filter**	**Feed**	**Product**	**Delta**
	Organic	**Aqueous**	**Organic**	**Cake**	**Effluent**	**In**	**Out**	**--**
Circuit Performance Indicators								
Yield (%)	0.0003	0.0000	0.0000	0.0002	0.0000	0.0003	0.0003	--
Ash (%)	100.00	100.00	100.00	100.00	100.00	100.00	100.00	--
Impurity Assays (%, dry)								
Unknown %	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Al %	0.00	0.03	0.00	0.00	0.00	0.00	0.00	0.00
Ca %	0.02	0.23	0.00	0.00	0.00	0.02	0.02	0.00
Fe %	0.00	0.01	0.00	0.00	0.00	0.00	0.00	0.00
Si %	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Trace Assays (%, dry)								
Sc %	99.916	99.154	99.999	99.999	99.998	99.916	99.916	0.000
Y %	0.003	0.035	0.000	0.000	0.000	0.003	0.003	0.000
La %	0.010	0.098	0.000	0.000	0.000	0.010	0.010	0.000
Ce %	0.025	0.249	0.000	0.000	0.000	0.025	0.025	0.000
Pr %	0.003	0.027	0.000	0.000	0.000	0.003	0.003	0.000
Nd %	0.012	0.121	0.000	0.000	0.000	0.012	0.012	0.000
Pm %	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Sm %	0.002	0.017	0.000	0.000	0.000	0.002	0.002	0.000
Eu %	0.000	0.001	0.000	0.000	0.000	0.000	0.000	0.000
Gd %	0.001	0.008	0.000	0.000	0.000	0.001	0.001	0.000
Tb %	0.001	0.006	0.000	0.000	0.000	0.001	0.001	0.000
Dy %	0.001	0.006	0.000	0.000	0.000	0.001	0.001	0.000
Ho %	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Er %	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Tm %	0.000	0.001	0.000	0.000	0.000	0.000	0.000	0.000
Yb %	0.000	0.003	0.000	0.000	0.000	0.000	0.000	0.000
Lu %	0.000	0.001	0.000	0.000	0.000	0.000	0.000	0.000
Th %	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
U %	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
TREE%	99.973	99.726	100.000	100.000	99.999	99.973	99.973	--

Table 3-88: Distribution Values for the Scandium Solvent Extraction Circuit

5 - SCANDIUM SOLVENT EXTRACTION								
	IN	OUT				Circuit Feed In	Circuit Product Out	Circuit Delta
	R-Strip2 Organic	Sc-Wash1 Aqueous	Sapon Organic	Sc Filter Cake	Sc Filter Effluent			--
Mass Distributions								
Total	0.0003	0.0000	0.0000	0.0002	0.0000	0.0003	0.0003	--
Ash	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	--
Impurity Distributions								
Unknown	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Al	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Ca	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Fe	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Si	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Trace Element Distributions								
Sc	17.5373	1.7099	0.0791	14.3310	1.4173	17.5373	17.5373	0.0000
Y	0.0003	0.0003	0.0000	0.0000	0.0000	0.0003	0.0003	0.0000
La	0.0004	0.0004	0.0000	0.0000	0.0000	0.0004	0.0004	0.0000
Ce	0.0005	0.0005	0.0000	0.0000	0.0000	0.0005	0.0005	0.0000
Pr	0.0005	0.0005	0.0000	0.0000	0.0000	0.0005	0.0005	0.0000
Nd	0.0006	0.0006	0.0000	0.0000	0.0000	0.0006	0.0006	0.0000
Pm	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Sm	0.0005	0.0005	0.0000	0.0000	0.0000	0.0005	0.0005	0.0000
Eu	0.0002	0.0002	0.0000	0.0000	0.0000	0.0002	0.0002	0.0000
Gd	0.0003	0.0003	0.0000	0.0000	0.0000	0.0003	0.0003	0.0000
Tb	0.0013	0.0013	0.0000	0.0000	0.0000	0.0013	0.0013	0.0000
Dy	0.0003	0.0003	0.0000	0.0000	0.0000	0.0003	0.0003	0.0000
Ho	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Er	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Tm	0.0005	0.0005	0.0000	0.0000	0.0000	0.0005	0.0005	0.0000
Yb	0.0002	0.0002	0.0000	0.0000	0.0000	0.0002	0.0002	0.0000
Lu	0.0004	0.0004	0.0000	0.0000	0.0000	0.0004	0.0004	0.0000
Th	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
U	0.0002	0.0002	0.0000	0.0000	0.0000	0.0002	0.0002	0.0000
	0.7995	0.0784	0.0036	0.6530	0.0646	0.7995	0.7995	0.0000

3.5.5 Detailed Listing of Equipment Requirements of a 1 tph Feed Rate Pilot Plant

A detailed listing of process equipment required to complete the construction of the REE extraction facility was prepared by the engineering team. For management purposes, the equipment lists were subdivided into six major cost centers, i.e.:

01 – Crushing and Grinding (*Table 3-89* and *Table 3-96*)
02 - Roasting and Scrubbing (*Table 3-90* and *Table 3-97*)
03 – Acid Leaching (*Table 3-91* and *Table 3-98*)
04 – Solvent Extraction (*Table 3-92* and *Table 3-99*)
05 – Washing and Tailings (*Table 3-93* and *Table 3-100*)
06 – Water Treatment (*Table 3-94* and *Table 3-101*)
07 – Ancillary Operations (*Table 3-95* and *Table 3-102*)

The equipment lists include:

1. equipment descriptions,
2. reference identification numbers,
3. CAD drawing numbers,
4. air and/or water needs, and
5. generalized equipment specifications.

Additional details related to equipment size and power demand are also provided in tabular summarizes broken down by cost center. The tabular summaries provide additional specifications regarding:

1. equipment descriptions,
2. reference identification numbers,
3. horsepower demand,
4. total equipment weight,
5. equipment dimensions (height, width, diameter) and
6. equipment footprint length.

Table 3-89: Detailed Equipment List for Crushing and Grinding Operations (Cost Center 01)

Description	ID	Drawing Number	Air	Water	Specification
Storage Bin (B-1.1)	B-1.1	3166-EM-103	--	--	8'Ø x 8' hi with cone
Storage Bin (B-1.2) (Storage Bin for Finished Ground Ore)	B-1.2	3166-EM-101	--	--	10'Ø x 8" hi with cone
Vibrating Bin Activator (BA-1.1)	BA-1.1	3166-EM-103	--	--	8'Ø x 8' hi with cone
Vibrating Bin Activator (BA-1.2)	BA-1.2	3166-EM-103	--	--	8'Ø x 8' hi with cone
Jaw Crusher Disch Belt Conveyor (BC-1.1)	BC-1.1	3166-EM-102	--	--	16" width
Tail Pully Conv (BC-1.2)	BC-1.2	3166-EM-101	--	--	16" width
18' Belt Conveyor (BC-1.3)	BC-1.3	3166-EM-101/3166-EF-11	--	--	16" width
Bag House (BH-1.1)	BH-1.1	3166-EM-101	--	--	940 ft^2
Bucket Elevator (BL-1.1)	BL-1.1	3166-EM-103	--	--	4" x 6" buckets
Bucket Elevator (BL-1.2)	BL-1.2	3166-EM-103	--	--	4" x 6" buckets
(Conical) Ball Mill (BM-1.1)	BM-1.1	3166-EM-101	--	--	4-1/2' Ø
Exhaust Blower (BO-1.1)	BO-1.1	3166-EF-11	Yes	--	7, 000 cfm
(Dust) Cyclone (CY-1.1)	CY-1.1	3166-EM-103	--	--	30" Ø
(Rotary) Dryer (D-1.1)	D-1.1	3166-EM-102	Yes	--	5' Ø x 23'
Jaw Crusher Belt Conveyor Feeder (FD-1.1)	FD-1.1	3166-EF-11/3166-EM-102	--	--	12" wide x 5' long
Storage Bin Disch Weight Belt Conveyor (FD-1.2)	FD-1.2	3166-EF-11	--	--	12" wide
(Cyclone Discharge) Rotary Feeder (FD-1.3)	FD-1.3	3166-EM-103	--	--	6"
Baghouse Discharge Rotary Feeder (FD-1.4)	FD-1.4	3166-EF-11	--	--	6"
Jaw Crusher (JC-1.1)	JC-1.1	3166-EM-102	--	--	10" x 20"
Vibrating Conveyor (PC-1.1)	PC-1.1	3166-EM-101	--	--	12" wide
Roll Crusher Disch. Pan Conveyor (PC-1.1)	PC-1.1	3166-EF-11	--	--	12" wide
12" Vibrating Conveyor (Bin Discharge - Pan Conveyor) (PC-1.2)	PC-1.2	3166-EM-103	--	--	12" wide
Roll Crusher Smooth Face (RC-1.1)	RC-1.1	3166-EF-11	--	--	18" Ø x 12"
Roll Crusher (1.1)	RC-1.1	3166-EM-102	--	--	18" Ø x 12"
(Ball Mill) Screw Conveyor (SC-1.2)	SC-1.2	3166-EM-103	--	--	--
(Bin Disch) Screw Conveyor (SC-1.3)	SC-1.3	3166-EM-103	--	--	6" Ø
(Bin Disch) Screw Conveyor (SC-1.4)	SC-1.4	3166-EF-11	--	--	4" Ø
Bag House Disch Trough Type Screw Conveyor (SC-1.5)	SC-1.5	3166-EM-103	--	--	4" Ø
Cyclone Discharge Trough Type Screw Conveyor (SC-1.6)	SC-1.6	3166-EF-11	--	--	4" Ø
Vibrating Screen (SN-1.1)	SN-1.1	3166-EM-102	--	--	1-1/2' Ø
Vibrating Screen (Double Deck) (SN-1.2)	SN-1.2	3166-EM-104	--	--	60" Ø
Tails Slurry Hold Tank	TK 5.1	--	--	--	--

Table 3-90: Detailed Equipment List for Roasting and Scrubbing Operations (Cost Center 02)

Description	ID	Drawing Number	Air	Water	Specification
Quench Tank Agitator (A-2.1)	A-2.1	3166-EF-21	--	--	3-1/2' Ø
HCI Mix Tank Agitator (A-2.2)	A-2.2	3166-EF-21	--	--	9"Ø prop 350 rpm
First Stage H2O Scrubber Hold Tank Agitator (A-2.3)	A-2.3	3166-EF-22	--	--	--
Sulfur Bin (B-2.1)	B-2.1	3166-EF-21	--	--	1 ft^3
Bucket Elevator (BL-2.1)	BL-2.1	3166-EN-104	--	--	4" x 6" buckets
Scrubber Exhaust Blower (BO-2.1)	BO-2.1	3166-EF-22	Yes	--	500 scfm 6" sp
Dust Cyclone (CY-2.1)	CY-2.1	3166-EF-21	--	--	16" Ø
Exo-Gas Generator (EG-2.1)	EG-2.1	3166-EF-21	--	--	5,000 scfh
Sulfur Table Feeder (FD-2.1)	FD-2.1	3166-EF-21	--	--	0-10 lb./hr.
Roaster Cyclone Rotary Feeder (FD-2.2)	FD-2.2	3166-EF-21	--	--	6"
Rotary Valve Feeder (FD-2.3)	FD-2.3	3166-EF-21	--	--	--
Carbon Monoxide Heater	HX-2.001		--	--	--
HCL Metering Pump (P-2.1)	P-2.1	3166-EF-21	--	Yes	1/4 gpm
Quench Tank Slurry Pump (P-2.2)	P-2.2	3166-EF-21	--	Yes	3/4"-3 gpm
First Stage H2O Scrubber Pump 20 GPM (P-2.3)	P-2.3	3166-EF-22	--	Yes	--
HCL Drum Metering Pump (P-2.5)	P-2.5	3166-EF-21	--	Yes	1/4 gpm
Alkaline Scrubber Circ Pump (P-2.6)	P-2.6	3166-EF-22	--	Yes	--
Pug Mill (PM-2.1)	PM-2.1	3166-EF-21	--	--	4' L - 6" Ø
Roaster (R-2.1)	R-2.1	3166-EN-101	--	--	5' ID
Multiple Hearth Roaster w/ Central Col. Fan (R-2.1)	R-2.1	3166-EF-21	--	--	5' ID
First Stage H2O Scrubber Ventur (S-2.1)	S-2.1	3166-EF-22	--	Yes	--
Second Stage H2O Scrubber Packed Bed (S-2.2)	S-2.2	3166-EF-22	--	Yes	--
Alkaline Scrubber Venturi (S-2.3)	S-2.3	3166-EF-22	--	Yes	--
Screw Conveyor (SC-1.4)	SC-1.4	3166-EN-104	--	--	4" Ø
Cyclone Disch Screw Conveyor (SC-2.1)	SC-2.1	3166-EF-21	--	--	--
Cyclone Disch Screw Conveyor Water Cooled (SC-2.2)	SC-2.2	3166-EF-21	--	--	--
HCL Platform Scale (SL-2.1)	SL-2.1	3166-EN-104	--	--	200 lb. cap
HCL Platform Scale (SL-2.1)	SL-2.1	3166-EF-21	--	--	200 lb. cap
HCL Mix Tank (TK-2.1)	TK-2.1	3166-EN-104	--	Yes	3' Ø x 4'
Tank TK-2.1 HCK Make up Tank	TK-2.1	3166-EM-103	--	--	3' Ø x 4'
Quench Tank TK-2.2	TK-2.2	3166-EM-104	--	Yes	4' Ø x 5'
Agitated First Stage H2O Scrubber Hold Tank (TK-2.3)	TK-2.3	3166-EF-22	--	Yes	--
Alkaline Scrubber Hold Tank (TK-2.5)	TK-2.5	3166-EF-22	--	Yes	--
Rotary Cyclone Discharge Feeder	--	--	--	--	--
Sur-Lite Waste Gas Incinerator	--	--	--	--	--
Upper Hearth Burners	--	--	--	--	--

Table 3-91: Detailed Equipment List for Acid Leaching Operations (Cost Center 03)

Description	ID	Drawing Number	Air	Water	Specification
Super Agitator (A-3.1)	A-3.1	3166-EF-31	--	--	3' x 3
(A-3.2)	A-3.2	3166-EF-31	--	--	3' x 3'
(A-3.3)	A-3.3	3166-EF-31	--	--	3' x 3'
(A-3.4)	A-3.4	3166-EF-31	--	--	3' x 3'
(A-3.5)	A-3.5	3166-EF-31	--	--	3' x 3'
(A-3.6)	A-3.6	3166-EF-31	--	--	3' x 3'
Tank	A-3.7	--	--	--	9" Ø prop
Agitator Rubber Covered (A-4.7)	A-4.7	3166-EF-31	--	--	--
Leach Air Blower (BO-3.1)	BO-3.1	--	--	--	15 cfm - 7psi
Wet Cyclone Classifier (CY-3.1)	CY-3.1	3166-EF-21	--	--	--
Thickener Under/Flow Pump (P-3.1)	P-3.1	3166-EF-31	--	Yes	0-3 gpm
Calcine Grinding Circuit Pump (P-3.10)	P-3.10	--	--	--	--
Calcine Grinding Circuit Pump (P-3.11)	P-3.11	--	--	--	--
Thickener Under/Flow Pump (P-3.2)	P-3.2	3166-EF-31	--	Yes	0-3 gpm
Thickener Over/Flow Pump (P-3.3)	P-3.3	3166-EF-31	--	Yes	0-3 gpm
Leached Slurry Pump (P-3.4)	P-3.4	3166-EF-31	--	Yes	0-3 gpm
(P-3.5)	P-3.5	3166-EF-31	--	Yes	3/4"-2 gpm
Thickener Over/Flow Pump (P-3.6)	P-3.6	3166-EF-31	--	Yes	3/4"-2 gpm
Preg. Liquor Circ. Pump (P-3.7)	P-3.7	3166-EF-31	--	Yes	1/2" -5 gpm
Spare thickener Overflow Pump P-3.8	P-3.8	--	--	--	--
Preg. Liquor Circ. Pump (P-3.9)	P-3.9	3166-EF-31	--	Yes	5 gpm
Agitated Reslurry Tank (TK-10)	TK-10	3166-EF-31	--	Yes	--
Leach Liq. Storage Tank (TK-11)	TK-11	3166-EF-31	--	Yes	--
Agitated Leach Tank (TK-3.1)	TK-3.1	3166-EF-31	--	Yes	3' Ø x 3'
Agitated Tank	TK-3.10	--	--	--	27" Øx 27"
Agitated Tank	TK-3.11	--	--	--	6' Ø x 6'
Agitated Leach Tank (TK-3.2)	TK-3.2	3166-EF-31	--	Yes	3' Ø x 3'
Ball mill discharge Tank	TK-3.20	--	--	--	--
Cyclone Discharge Tank	TK-3.21	--	--	--	--
Agitated Leach Tank (TK-3.3)	TK-3.3	3166-EF-31	--	Yes	3' Ø x 3'
Agitated Leach Tank (TK-3.4)	TK-3.4	3166-EF-31	--	Yes	3' Ø x 3'
Agitated Leach Tank (TK-3.5)	TK-3.5	3166-EF-31	--	Yes	3' Ø x 3'
Agitated Leach Tank (TK-3.6)	TK-3.6	3166-EF-31	--	Yes	3' Ø x 3'
Thickener Tank (TK-3.7)	TK-3.7	3166-EF-31	--	Yes	6' Ø x 6' deep
Thickener Tank (TK-3.8)	TK-3.8	3166-EF-31	--	Yes	6' Ø x 6' deep
Thickener Tank (TK-3.9)	TK-3.9	3166-EF-31	--	Yes	6' Ø x 6' deep
Rake Mechanism (TM-3.1)	TM-3.1	3166-EF-31	--	--	--
(TM-3.2)	TM-3.2	3166-EF-31	--	--	--
(TM-3.3)	TM-3.3	3166-EF-31	--	--	--

Table 3-92: Detailed Equipment List for Solvent Extraction Operations (Cost Center 04)

Description	ID	Drawing Number	Air	Water	Specification
Reslurry Tank. Agitator (A-4.1)	A-4.1	3166-EF-41	--	--	9"Ø prop 350 rpm
Reslurry Tank Agitator (A-4.2)	A-4.2	3166-EF-41	--	--	9"Ø prop 350 rpm
Reslurry Tank Agitator (A-4.3)	A-4.3	3166-EF-41	--	--	9"Ø prop 350 rpm
Tails Hold Tank Agitator (A-4.4)	A-4.4	3166-EF-41	--	--	3-1/2'Ø
Thickener Under/Flow Pump (P-4.1)	P-4.1	3166-EF-41	--	Yes	0-3 gpm
Thickener Over/Flow Pump (P-4.10)	P-4.10	3166-EF-41	--	Yes	3/4"-2 gpm
Thickener Under/Flow Pump (P-4.2)	P-4.2	3166-EF-41	--	Yes	0-3 gpm
Thickener Under/Flow Pump (P-4.3)	P-4.3	3166-EF-41	--	Yes	0-3 gpm
Thickener Under/Flow Pump (P-4.4)	P-4.4	3166-EF-41	--	Yes	0-3 gpm
Reslurry Transfer Pump (P-4.5)	P-4.5	3166-EF-41	--	Yes	3/4"-3 gpm
Reslurry Transfer Pump (P-4.6)	P-4.6	3166-EF-41	--	Yes	3/4"-3 gpm
Reslurry Transfer Pump (P-4.7)	P-4.7	3166-EF-41	--	Yes	3/4"-3 gpm
Tails Slurry Pump (P-4.8)	P-4.8	3166-EF-41	--	Yes	3/4"-3 gpm
Thickener Over/Flow Recycle Liquor Pump (P-4.9)	P-4.9	3166-EF-41	--	Yes	3/4"-2 gpm
Thickener Tank (TK-4.1)	TK-4.1	3166-EF-41	--	Yes	6' Ø x 6' deep
Thickener Tank (TK-4.2)	TK-4.2	3166-EF-41	--	Yes	6' Ø x 6' deep
Thickener Tank (TK-4.3)	TK-4.3	3166-EF-41	--	Yes	6' Ø x 6' deep
Thickener Tank (TK-4.4)	TK-4.4	3166-EF-41	--	Yes	6' Ø x 6' deep
Agitated Reslurry Tank (TK-4.5)	TK-4.5	3166-EF-41	--	Yes	27" Øx 27"
Agitated Reslurry Tank (TK-4.6)	TK-4.6	3166-EF-41	--	Yes	27" Øx 27"
Agitated Reslurry Tank (TK-4.7)	TK-4.7	3166-EF-41	--	Yes	27" Øx 27"
Agitated Tails Hold and Preheat Tank (TK-4.8)	TK-4.8	3166-EF-41	--	Yes	4'Øx6' 8hr hold
Rake Mechanism (TM-4.1)	TM-4.1	3166-EF-41	--	--	--
Rake Mechanism (TM-4.2)	TM-4.2	3166-EF-41	--	--	--
Rake Mechanism (TM-4.3)	TM-4.3	3166-EF-41	--	--	--
Rake Mechanism (TM-4.4)	TM-4.4	3166-EF-41	--	--	--

Table 3-93: Detailed Equipment List for Washing and Tailings Operations (Cost Center 05)

Description	ID	Drawing Number	Air	Water	Specification
Tails Hold Tank Agitator (A-5.1)	A-5.1	--	--	--	3-1/2' Ø
CA Exhaust Blower BO-5.1	BI-5.1	--	--	--	--
Condenser Absorber (CA-5.1)	CA-5.1	--	--	--	10"Øx10' high
Tailings Slurry Pump P--5.1	P-5.1	--	--	--	20 gpm
CA Circ. Pump P-5.2	P-5.2	--	--	--	--
Leach Liquor Pump P-5.3	P-5.3	--	--	--	5 gpm
Waste Slurry Pump P-5.4	P-5.4	--	--	--	0-3 gpm
Condenser	PC-5.002	--	--	--	--
Precooler	PC-5.1	--	--	--	3/4" tuber 20' long
CA Precooler (PC-5.1)	PC-5.1	--	--	--	3/4" tuber 20' long
NH3 SCALE SL-5.1	SL-5.1	--	--	--	500 lb. cap
CO2 SCALE SL-5.2	SL-5.2	--	--	--	400 lb. cap
Tailings Stripper (NH3 stripper)	ST-5.1	--	--	--	10" Ø 13'
Pregnant Solution Ammonia Stripper	ST-5.2	--	--	--	--
Leach Solution Ammonia Stripper	ST-5.3	--	--	--	--
Leach Liq. Storage Tank (TK-5.2)	TK 5.2	--	--	--	--

Table 3-94: Detailed Equipment List for Water Treatment Operations (Cost Center 06)

Description	ID	Drawing Number	Air	Water	Specification
C20 Mix Tank Agitator (A-6.1)	A-6.1	3166-EF-61	--	--	0
DC Agitator (A-6.2)	A-6.2	--	--	--	9" Ø prop
DC Agitator (A-6.3)	A-6.3	--	--	--	9" Ø prop
Decomposer Agitator (DC-3)	DC-3	3166-EF-61	--	--	--
Decomposer Tank (DC-6.1)	DC-6.1	3166-EF-61	--	--	0
In-Line Polishing Filter Precoat Type (F-6.1)	F-6.1	3166-EF-61	--	--	5 gpm
Horiz. Belt Filter. (F-6.2)	F-6.2	3166-EF-62	--	--	1' x 8'
In-Line Polishing Filter Precoat Type-Rubber Lined (F-6.3)	F-6.3	3166-EF-62	--	--	5 gpm
Preg. Liquor Heat Exchanger (HX-6.1)	HX-6.1	3166-EF-61	--	--	3/4" & 2" pipe 6' long
Stripper Preheat Heat Exchange (HX-6.2)	HX-6.2	3166-EF-61	--	--	3/4" & 2" pipe 8' long
Filter Feed Pump. (P-6.1)	P-6.1	3166-EF-61	--	Yes	1.5 gpm
Drain Pump 6 GPM (P-6.11)	P-6.11	3166-EF-62	--	Yes	--
Stripper Slurry Pump (P-6.2)	P-6.2	3166-EF-61	--	Yes	1L2-1 gpm
Thickener Over/Flow Pump (P-6.3)	P-6.3	3166-EF-62	--	Yes	3/4" -2 gpm
Thickener Under/Flow Pump (P-6.4)	P-6.4	3166-EF-62	--	Yes	0-3 gpm
Belt Filter Vacuum Pump (P-6.5)	P-6.5	3166-EF-62	--	Yes	--
Filtrate Pump (P-6.6) X 2	P-6.6	3166-EF-62	--	Yes	60 gpm
$(NH_4)_2S$ Solution Metering Pump (P-6.8)	P-6.8	3166-EF-61	--	Yes	1/4 gpm
Var. Volume C2O Makeup Pump (P-6.9)	P-6.9	3166-EF-61	--	Yes	1/4 gpm
Platform Scales (SL-6.1)	SL-6.1	3166-EF-62	--	--	800 lb.
Steam Stripper (ST-6.1)	ST-6.1	3166-EF-61	--	--	10" Øx 13' long
Stripper Slurry Thickener Tank (TK-6.1)	TK-6.1	3166-EF-62	--	Yes	6' Øx 6' deep
Filtrate Receiver Tank. (TK-6.3)	TK-6.3	3166-EF-62	--	Yes	--
Precoat Tank (TK-6.4)	TK-6.4	3166-EF-62	--	Yes	1' ØX 2'
Preg. Liquor Aeration Tank (TK-6.5)	TK-6.5	3166-EF-61	--	Yes	3' ØX 4'
Filter Precoat Mix Tank (TK-6.6)	TK-6.6	3166-EF-61	--	Yes	1' Ø X 2'
Water Trap Silencer-Separator (TK-6.7)	TK-6.7	3166-EF-62	--	Yes	--
H20 C20 Mix Tank (TK-6.8)	T-K6.8	3166-EF-61	--	Yes	--
Rake Mechanism (TM-6.1)	TM-6.1	3166-EF-62	--	--	--

Table 3-95: Detailed Equipment List for Ancillary Operations (Cost Center 07)

Description	ID	Drawing Number	Air	Water	Specification
C20 Mix Tank Agitator (A-6.1)	A-6.1	3166-EF-61	--	--	0
DC Agitator (A-6.2)	A-6.2	--	--	--	9" Ø prop
DC Agitator (A-6.3)	A-6.3	--	--	--	9" Ø prop
Decomposer Agitator (DC-3)	DC-3	3166-EF-61	--	--	--
Decomposer Tank (DC-6.1)	DC-6.1	3166-EF-61	--	--	0
In-Line Polishing Filter Precoat Type (F-6.1)	F-6.1	3166-EF-61	--	--	5 gpm
Horiz. Belt Filter. (F-6.2)	F-6.2	3166-EF-62	--	--	1' x 8'
In-Line Polishing Filter Precoat Type-Rubber Lined (F-6.3)	F-6.3	3166-EF-62	--	--	5 gpm
Preg. Liquor Heat Exchanger (HX-6.1)	HX-6.1	3166-EF-61	--	--	3/4" & 2" pipe 6' long
Stripper Preheat Heat Exchange (HX-6.2)	HX-6.2	3166-EF-61	--	--	3/4" & 2" pipe 8' long
Filter Feed Pump. (P-6.1)	P-6.1	3166-EF-61	--	Yes	1.5 gpm
Drain Pump 6 GPM (P-6.11)	P-6.11	3166-EF-62	--	Yes	--
Stripper Slurry Pump (P-6.2)	P-6.2	3166-EF-61	--	Yes	1L2-1 gpm
Thickener Over/Flow Pump (P-6.3)	P-6.3	3166-EF-62	--	Yes	3/4" -2 gpm
Thickener Under/Flow Pump (P-6.4)	P-6.4	3166-EF-62	--	Yes	0-3 gpm
Belt Filter Vacuum Pump (P-6.5)	P-6.5	3166-EF-62	--	Yes	--
Filtrate Pump (P-6.6) X 2	P-6.6	3166-EF-62	--	Yes	60 gpm
$(NH_4)_2S$ Solution Metering Pump (P-6.8)	P-6.8	3166-EF-61	--	Yes	1/4 gpm
Var. Volume C2O Makeup Pump (P-6.9)	P-6.9	3166-EF-61	--	Yes	1/4 gpm
Platform Scales (SL-6.1)	SL-6.1	3166-EF-62	--	--	800 lb.
Steam Stripper (ST-6.1)	ST-6.1	3166-EF-61	--	--	10" Øx 13' long
Stripper Slurry Thickener Tank (TK-6.1)	TK-6.1	3166-EF-62	--	Yes	6' Øx 6' deep
Filtrate Receiver Tank. (TK-6.3)	TK-6.3	3166-EF-62	--	Yes	--
Precoat Tank (TK-6.4)	TK-6.4	3166-EF-62	--	Yes	1' ØX 2'
Preg. Liquor Aeration Tank (TK-6.5)	TK-6.5	3166-EF-61	--	Yes	3' ØX 4'
Filter Precoat Mix Tank (TK-6.6)	TK-6.6	3166-EF-61	--	Yes	1' Ø X 2'
Water Trap Silencer-Separator (TK-6.7)	TK-6.7	3166-EF-62	--	Yes	--
H20 C20 Mix Tank (TK-6.8)	T-K6.8	3166-EF-61	--	Yes	--
Rake Mechanism (TM-6.1)	TM-6.1	3166-EF-62	--	--	--
Air Compressor (C-8.1)	C-8.1	3166-EF-82	Yes	--	--
Cooling Tower (CT 8.1)	CT-8.1	3166-EF-81	--	Yes	--
Air Dryer (D-8.2)	D-8.2	3166-EF-82	Yes	--	--
Well Pump 56 GPM (P-8.1)	P-8.1	3166-EF-81	--	Yes	--
Raw Water Supply Pump 50 GPM (P-8.3)	P-8.3	3166-EF-81	--	Yes	--
Raw Water Supply Pump 50 GPM (P-8.3A)	P-8.3A	3166-EF-81	--	Yes	--
CWS Pump 200 GPM (P-8.4)	P-8.4	3166-EF-81	--	Yes	--
Sump Pump (P-8.5)	P-8.5	3166-EF-81	--	Yes	--
Feed Water Chemical Metering Pump 3 GPM (P-8.7)	P-8.7	3166-EF-81	--	Yes	--
Fuel Oil Transfer Pump 0.5 GPM (P-8.7)	P-8.7	3166-EF-82	--	Yes	--
Steam Generator (SG-8.1)	SG-8.1	3166-EF-81	--	Yes	--
Raw Water Storage (TK-8.1)	TK-8.1	3166-EF-81	--	Yes	--
Feed Water Chemical Tank (TK-8.2)	TK-8.2	3166-EF-81	--	Yes	--
Water Softener/Water Treater with Brine Tank (TK-8.3)	TK-8.3	3166-EF-81	--	Yes	--
Raw Water Pressure Vessel (TK-8.4)	TK-8.4	3166-EF-81	--	Yes	--
Fuel Oil Storage (TK-8.5)	TK-8.5	3166-EF-82	--	--	--
Compressed Air Receiver (V-8.1)	V-8.1	3166-EF-82	Yes	--	--
480V Motor Control Center	--	--	--	--	--
Tailings Storage Tank	TK-9.001	--	--	--	--
Carbon Monoxide Generator	--	--	--	--	--
Chromite Hydrocyclone	CY-9.001	--	--	--	--
Spare Parts	P-5.2	--	--	--	--
Various filter components and spares T-76	T-76	--	--	--	--
Font end Loader	--	--	--	--	--

Table 3-96: Equipment Specifications for Crushing and Grinding (Cost Center 01)

Description	ID	HP	Wt. (lbs.)	Height	Width	Diameter	Length
Storage Bin (B-1.1)	B-1.1	0	5500	8' - 0"	---	8' - 0"	---
Storage Bin (B-1.2) (Storage Bin for Finished Ground Ore)	B-1.2	0	7600	8' - 0"	---	10' - 0"	---
Vibrating Bin Activator (BA-1.1)	BA-1.1	1/2	875	2'-4"	0	6'-1"	---
Vibrating Bin Activator (BA-1.2)	BA-1.2	1/2	1000	3'-0"	0	7'-8"	---
Jaw Crusher Disch Belt Conveyor (BC-1.1)	BC-1.1	3	4310	---	---	---	60'-0"
Tail Pully Conv (BC-1.2)	BC-1.2	3	4310	---	---	---	62'-0"
18' Belt Conveyor (BC-1.3)	BC-1.3	3	4310	---	---	---	58'-0"
Bag House (BH-1.1)	BH-1.1	0	4900	15'-6"	9'-0"	---	6'-0"
Bucket Elevator (BL-1.1)	BL-1.1	1 1/2	2750	40'-0"	16"	---	4'-0"
Bucket Elevator (BL-1.2)	BL-1.2	1 1/2	2750	40'-0"	16"	---	4'-0"
(Conical) Ball Mill (BM-1.1)	BM-1.1	20	14,000	5'	4'-6"	---	8'-4"
Exhaust Blower (BO-1.1)	BO-1.1	30	250	--	--	Included	0
(Dust) Cyclone (CY-1.1)	CY-1.1	--	340	3'-0" / 2'-0"	3'-6" / 2'-0"	---	9'-4" / 3'-0"
(Rotary) Dryer (D-1.1)	D-1.1	7 1/2	25,400	7'-0"	10'-0"	5'-0"	8'-0" / 33'-0"
Jaw Crusher Belt Conveyor Feeder (FD-1.1)	FD-1.1	1/3	1900	3'-10"	2'-6"	---	5'-6"
Storage Bin Disch Weight Belt Conveyor (FD-1.2)	FD-1.2	1/3	400	1'-6"	1'-8"	---	4'-6"
(Cyclone Discharge) Rotary Feeder (FD-1.3)	FD-1.3	1/3	120	0	6 x 4	400	400
Baghouse Discharge Rotary Feeder (FD-1.4)	FD-1.4	1/3	120	0	6 x 4	120	120
Jaw Crusher (JC-1.1)	JC-1.1	20	6200	4'-7"	2'-10"	---	4'-2"
Vibrating Conveyor (PC-1.1)	PC-1.1	1	600	2'-0"	1'-6"	---	12'-0"
Roll Crusher Disch. Pan Conveyor (PC-1.1)	PC-1.1	1	600	2'-0"	1'-6"	---	12'-0"
12" Vibrating Conveyor (Bin Discharge - Pan Conveyor) (PC-1.2)	PC-1.2	2	600	2'-0"	1'-6"	---	12'-0"
Roll Crusher Smooth Face (RC-1.1)	RC-1.1	15	4500	3'-0"	6'-4"	---	11'-8"
Roll Crusher (1.1)	RC-1.1	15	4500	3'-0"	6'-4"	---	11'-8"
(Ball Mill) Screw Conveyor (SC-1.2)	SC-1.2	1/3	500	--	--	--	--
(Bin Disch) Screw Conveyor (SC-1.3)	SC-1.3	1/3	300	0	0	est. 4" ID	est. 4'-0"
(Bin Disch) Screw Conveyor (SC-1.4)	SC-1.4	1/3	300	0	0	est. 4" ID	est. 6'-0"
Bag House Disch Trough Type Screw Conveyor (SC-1.5)	SC-1.5	1/3	330	0	0	4" ID	12'-0"
Cyclone Discharge Trough Type Screw Conveyor (SC-1.6)	SC-1.6	1/3	250	0	0	4" ID	9'-0"
Vibrating Screen (SN-1.1)	SN-1.1	1	675	4'-6"	3'-0"	0	4'-6"
Vibrating Screen (Double Deck) (SN-1.2)	SN-1.2	1	1125	4'-7"	0	3'-4"	--
Tails Slurry Hold Tank	TK 5.1	--	500	--	--	--	--

Table 3-97: Equipment Specifications for Roasting and Scrubbing (Cost Center 2)

Description	ID	HP	Wt. (lbs.)	Height	Width	Diameter	Length
Quench Tank Agitator (A-2.1)	A-2.1	3	750	4'-0"	2'-0"	--	5'-0"
HCL Mix Tank Agitator (A-2.2)	A-2.2	1/4	500	3'-0"	1'-6"	--	4'-0"
First Stage H20 Scrubber Hold Tank Agitator (A-2.3)	A-2.3	0	75	--	--	--	--
Sulfur Bin (B-2.1)	B-2.1	0	100	0	Included	55	55
Bucket Elevator (BL-2.1)	BL-2.1	1-1/2	2750	40'-0"	16"	0	4'-0"
Scrubber Exhaust Blower (BO-2.1)	BO-2.1	7-1/2	520	1'-0" / 1'-8"	1'-6" / 1'-8"	0	1'-0" / 2'-0"
Dust Cyclone (CY-2.1)	CY-2.1	0	55	---	--	1'6"	2'-10"
Exo-Gas Generator (EG-2.1)	EG-2.1	0	6,200	4'-9"	8'-0"	0	13'-6"
Sulfur Table Feeder (FD-2.1)	FD-2.1	1/4	300	1'-10"	1'-3"	0	1'-10"
Roaster Cyclone Rotary Feeder (FD-2.2)	FD-2.2	1/3	120	6x4	6200	6200	6200
Rotary Valve Feeder (FD-2.3)	FD-2.3	1/3	300	--	--	--	--
Carbon Monoxide Heater	HX-2.001	0	150	--	--	--	--
HCL Metering Pump (P-2.1)	P-2.1	1/6	120	1'-10"	1'-8"	0	2'-0"
Quench Tank Slurry Pump (P-2.2)	P-2.2	1/2	270	1'-4" / 1'-0"	2'-0" / 1'-2"	0	1'-6" / 10"
First Stage H2O Scrubber Pump 20 GPM (P-2.3)	P-2.3	1/3	250	1'-0"	1'-4"	0	3'-0"
HCL Drum Metering Pump (P-2.5)	P-2.5	1/6	75	0	Included	250	250
Alkaline Scrubber Circ Pump (P-2.6)	P-2.6	1/3	250	1'-0"	1'-4"	0	3'-0"
Pug Mill (PM-2.1)	PM-2.1	5	500	1'-2"	1'-4"	0	6'-3"
Roaster (R-2.1)	R-2.1	2	60,300	0	0	0	4pcs
Multiple Hearth Roaster w/ Central Col. Fan (R-2.1)	R-2.1	2	250	0	0	0	4pcs
First Stage H2O Scrubber Ventur (S-2.1)	S-2.1	0	610	--	--	--	--
Second Stage H2O Scrubber Packed Bed (S-2.2)	S-2.2	0	540	--	--	--	--
Alkaline Scrubber Venturi (S-2.3)	S-2.3	0	400	--	--	--	--
Screw Conveyor (SC-1.4)	SC-1.4	1/3	400	0	0	est. 4" ID	est. 6'-0"
Cyclone Disch Screw Conveyor (SC-2.1)	SC-2.1	1/3	175	--	--	--	--
Cyclone Disch Screw Conveyor Water Cooled (SC-2.2)	SC-2.2	1/3	600	--	--	--	--
HCL Platform Scale (SL-2.1)	SL-2.1	0	222	4'-9"	1'-9"	0	3'-1"
HCL Platform Scale (SL-2.1)	SL-2.1	0	--	4'-9"	1'-9"	0	3'-1"
HCL Mix Tank (TK-2.1)	TK-2.1	0	--	4'-6"	0	3'-6"	---
Tank TK-2.1 HCK Make up Tank	TK-2.1	0	150	4'-6"	0	3'-6"	---
Quench Tank TK-2.2	TK-2.2	0	1290	6'-7"	0	4'-5"	---
Agitated First Stage H2O Scrubber Hold Tank (TK-2.3)	TK-2.3	0	420	4'-6"	4'-6"	0	5'-0"
Alkaline Scrubber Hold Tank (TK-2.5)	TK-2.5	0	420	4'-6"	4'-6"	0	5'-0"
ROTARY CYCLONE DISCHARGE FEEDER	--	0	150	--	--	--	--
Sur-Lite Waste Gas Incinerator	--	1/2	500	--	--	--	--
Upper Hearth Burners	--	0	50	--	--	--	--

Table 3-98: Equipment Specifications for Acid Leaching (Cost Center 3)

Description	ID	HP	Wt. (lbs.)	Height	Width	Diameter	Length
Super Agitator (A-3.1)	A-3.1	1 1/2	500	0	Included	470	470
(A-3.2)	A-3.2	1 1/2	--	0	Included	470	470
(A-3.3)	A-3.3	1 1/2	--	0	Included	470	470
(A-3.4)	A-3.4	1 1/2	--	0	Included	470	470
(A-3.5)	A-3.5	1 1/2	--	0	Included	470	470
(A-3.6)	A-3.6	1 1/2	--	0	Included	470	470
Tank	A-3.7	1/4	110	1'-0"	1'-0"	0	3'-0"
Agitator Rubber Covered (A-4.7)	A-4.7	1 1/2	25	--	--	--	--
Leach Air Blower (BO-3.1)	BO-3.1	2	250	1'-2"	2'-0"	0	3'-4"
Wet Cyclone Classifier (CY-3.1)	CY-3.1	0	--	--	--	--	--
Thickener Under/Flow Pump (P-3.1)	P-3.1	1/2	250	1'-6"	9"	0	3'-2"
Calcine Grinding Circuit Pump (P-3.10)	P-3.10	1/2	250	--	--	--	--
Calcine Grinding Circuit Pump (P-3.11)	P-3.11	1/2	250	--	--	--	--
Thickener Under/Flow Pump (P-3.2)	P-3.2	1/2	250	1'-6"	9"	0	3'-2"
Thickener Over/Flow Pump (P-3.3)	P-3.3	1/2	250	1'-6"	9"	0	3'-2"
Leached Slurry Pump (P-3.4)	P-3.4	1/2	250	1'-6"	9"	0	3'-2"
(P-3.5)	P-3.5	1/4	156	2'-0"	2'-2"	0	2'-9"
Thickener Over/Flow Pump (P-3.6)	P-3.6	1/4	156	2'-0"	2'-2"	0	2'-9"
Preg. Liquor Circ. Pump (P-3.7)	P-3.7	1/4	30	10"	10"	0	1'-3"
Spare thickener Overflow Pump P-3.8	P-3.8	1/2	--	0	Included	156	156
Preg. Liquor Circ. Pump (P-3.9)	P-3.9	1/4	30	10"	10"	0	1'-3"
Agitated Reslurry Tank (TK-10)	TK-10	0	--	--	--	--	--
Leach Liq. Storage Tank (TK-11)	TK-11	0	--	--	--	--	--
Agitated Leach Tank (TK-3.1)	TK-3.1	0	710	3'-0"	3'-0"	0	4'6"
Agitated Tank	TK-3.10	0	210	3'-0"	3'-0"	0	3'-0"
Agitated Tank	TK-3.11	0	470	6'-6"	0	6'-3"	---
Agitated Leach Tank (TK-3.2)	TK-3.2	0	710	3'-0"	3'-0"	0	4'6"
Ball mill discharge Tank	TK-3.20	0	500	--	--	--	--
Cyclone Discharge Tank	TK-3.21	0	500	--	--	--	--
Agitated Leach Tank (TK-3.3)	TK-3.3	0	710	3'-0"	3'-0"	0	4'6"
Agitated Leach Tank (TK-3.4)	TK-3.4	0	710	3'-0"	3'-0"	0	4'6"
Agitated Leach Tank (TK-3.5)	TK-3.5	0	710	3'-0"	3'-0"	0	4'6"
Agitated Leach Tank (TK-3.6)	TK-3.6	0	710	3'-0"	3'-0"	0	4'6"
Thickener Tank (TK-3.7)	TK-3.7	1/2	6,000	12'-6"	0	7'-6"	---
Thickener Tank (TK-3.8)	TK-3.8	1/3	6,000	12'-6"	0	7'-6"	---
Thickener Tank (TK-3.9)	TK-3.9	1/4	6,000	12'-6"	0	7'-6"	---
Rake Mechanism (TM-3.1)	TM-3.1	0	0	0	Included	110	110
(TM-3.2)	TM-3.2	0	0	0	Included	110	110
(TM-3.3)	TM-3.3	0	0	0	Included	110	110

Table 3-99: Equipment Specifications for Solvent Extraction (Cost Center 04)

Description	ID	HP	Wt. (lbs.)	Height	Width	Diameter	Length
Reslurry Tank. Agitator (A-4.1)	A-4.1	1/4	110	1'-0"	1'-0"	0	3'-0"
Reslurry Tank Agitator (A-4.2)	A-4.2	1/4	110	1'-0"	1'-0"	0	3'-0"
Reslurry Tank Agitator (A-4.3)	A-4.3	1/4	110	1'-0"	1'-0"	0	3'-0"
Tails Hold Tank Agitator (A-4.4)	A-4.4	3	400	1'-8" Max	1'-8" Max	0	6'-6" Max
Thickener Under/Flow Pump (P-4.1)	P-4.1	1/2	250	1'-6"	9"	0	3'-2"
Thickener Over/Flow Pump (P-4.10)	P-4.10	1/2	156	2'-0"	2'-2"	0	2'-9"
Thickener Under/Flow Pump (P-4.2)	P-4.2	1/2	250	1'-6"	9"	0	3'-2"
Thickener Under/Flow Pump (P-4.3)	P-4.3	1/2	250	1'-6"	9"	0	3'-2"
Thickener Under/Flow Pump (P-4.4)	P-4.4	1/2	250	1'-6"	9"	0	3'-2"
Reslurry Transfer Pump (P-4.5)	P-4.5	1/2	240	1'-6"	9"	0	3'-2"
Reslurry Transfer Pump (P-4.6)	P-4.6	1/2	240	1'-6"	9"	0	3'-2"
Reslurry Transfer Pump (P-4.7)	P-4.7	1/2	240	1'-6"	9"	0	3'-2"
Tails Slurry Pump (P-4.8)	P-4.8	1/2	375	1'-0" / 1'-0"	1'-0" / 10"	0	1'-4" / 1'-6"
Thickener Over/Flow Recycle Liquor Pump (P-4.9)	P-4.9	1/2	156	2'-0"	2'-2"	0	2'-9"
Thickener Tank (TK-4.1)	TK-4.1	1/2	10,200	12'-6"	--	1'-0" 7'-6"	12'-0"
Thickener Tank (TK-4.2)	TK-4.2	1/2	6,600	1'-8" / 12'-6"	2'-0"	7'-6"	1'-6"
Thickener Tank (TK-4.3)	TK-4.3	1/2	7,250	2'-0" / 12'-6"	2'-6"	7'-6"	1'-6"
Thickener Tank (TK-4.4)	TK-4.4	1/2	6,000	12'-6"	--	7'-6"	---
Agitated Reslurry Tank (TK-4.5)	TK-4.5	0	210	3'-0"	3'-0"	0	3'-0"
Agitated Reslurry Tank (TK-4.6)	TK-4.6	0	210	3'-0"	3'-0"	0	3'-0"
Agitated Reslurry Tank (TK-4.7)	TK-4.7	0	210	3'-0"	3'-0"	0	3'-0"
Agitated Tails Hold and Preheat Tank (TK-4.8)	TK-4.8	0	1,250	4'-6"	4'-6"	0	6'-6"
Rake Mechanism (TM-4.1)	TM-4.1	0	0	0	Included	3	3
Rake Mechanism (TM-4.2)	TM-4.2	0	0	0	Included	3	3
Rake Mechanism (TM-4.3)	TM-4.3	0	0	0	Included	3	3
Rake Mechanism (TM-4.4)	TM-4.4	0	0	0	Included	3	3

Table 3-100: Equipment Specifications for Washing and Tailings (Cost Center 05)

Description	ID	HP	Wt. (lbs.)	Height	Width	Diameter	Length
Tails Hold Tank Agitator (A-5.1)	A-5.1	3	750	6'-0"	2'-0"	0	6'-0"
CA Exhaust Blower BO-5.1	BI-5.1	2	60	--	--	--	--
Condenser Absorber (CA-5.1)	CA-5.1	0	1,417	22'-0"	0	2'-0"	---
Tailings Slurry Pump P--5.1	P-5.1	3	520	2'-0"	1'-4"	0	2'-8"
CA Circ. Pump P-5.2	P-5.2	1/2	50	10"	1'-8"	0	10"
Leach Liquor Pump P-5.3	P-5.3	1/2	30	10"	1'-3"	0	10"
Waste Slurry Pump P-5.4	P-5.4	1/2	375	1'-0" / 1'-0"	1'-0" / 10"	0	1'-4" / 1'-6"
Condenser	PC-5.002	0	800	--	--	--	--
Precooler	PC-5.1	0	620	1'-0"	1'-2"	0	20'-0"
CA Precooler (PC-5.1)	PC-5.1	0	--	1'-0"	1'-2"	0	20'-0"
NH3 SCALE SL-5.1	SL-5.1	0	222	4'-9"	1'-9"	0	3'-1"
CO2 SCALE SL-5.2	SL-5.2	0	222	4'-9"	1'-9"	0	3'-1"
Tailings Stripper (NH3 stripper)	ST-5.1	0	200	14'-0"	0	1'-4"	---
Pregnant Solution Ammonia Stripper	ST-5.2	0	300	--	--	--	--
Leach Solution Ammonia Stripper	ST-5.3	0	300	--	--	--	--
Leach Liq. Storage Tank (TK-5.2)	TK 5.2	0	400	--	--	--	--

Table 3-101: Equipment Specifications for Water Treatment (Cost Center 06)

Description	ID	HP	Wt. (lbs.)	Height	Width	Diameter	Length
C20 Mix Tank Agitator (A-6.1)	A-6.1	1/20	80	1'-4"	1'-0"	0	3'-6"
DC Agitator (A-6.2)	A-6.2	1/4	28	1'-0"	1'-0"	0	3'-6"
DC Agitator (A-6.3)	A-6.3	1/4	28	1'-0"	1'-0"	0	3'-6"
Decomposer Agitator (DC-3)	DC-3	1/2	28	--	--	--	--
Decomposer Tank (DC-6.1)	DC-6.1	1/2	220	2'-0"	2'-6"	0	5'-0"
In-Line Polishing Filter Precoat Type (F-6.1)	F-6.1	0	8	11"	0	6"	---
Horiz. Belt Filter. (F-6.2)	F-6.2	10	4,000	7'-1"	5'-10"	0	13'-10"
In-Line Polishing Filter Precoat Type-Rubber Lined (F-6.3)	F-6.3	0	8	11"	0	6"	---
Preg. Liquor Heat Exchanger (HX-6.1)	HX-6.1	0	300	1'-8"	1'-4"	0	3'-6"
Stripper Preheat Heat Exchange (HX-6.2)	HX-6.2	0	300	1'-8"	1'-4"	0	3'-6"
Filter Feed Pump. (P-6.1)	P-6.1	1/2	50	1'-0"	1'-3"	0	1'-7"
Drain Pump 6 GPM (P-6.11)	P-6.11	1/2	37	0	0	0	0
Stripper Slurry Pump (P-6.2)	P-6.2	1/2	200	9"	1'-0"	0	2'-11"
Thickener Over/Flow Pump (P-6.3)	P-6.3	1/2	156	2'-0"	2'-2"	0	2'-9"
Thickener Under/Flow Pump (P-6.4)	P-6.4	1/2	250	1'-6"	9"	0	3'-2"
Belt Filter Vacuum Pump (P-6.5)	P-6.5	1	--	Included	0	0	0
Filtrate Pump (P-6.6) X 2	P-6.6	1/2	--	Included	0	0	0
(NH4)2S Solution Metering Pump (P-6.8)	P-6.8	1/6	120	1'-8"	2'-0"	0	1'-10"
Var. Volume C20 Makeup Pump (P-6.9)	P-6.9	1/6	120	1'-8"	2'-0"	0	1'-10"
Platform Scales (SL-6.1)	SL-6.1	0	222	4'-9"	1'-9"	0	3'-1"
Steam Stripper (ST-6.1)	ST-6.1	0	850	14'-0"	0	1'-4"	---
Stripper Slurry Thickener Tank (TK-6.1)	TK-6.1	1/2	7,100	12'-0"	0	8'-0"	---
Filtrate Receiver Tank. (TK-6.3)	TK-6.3	0	150	Extra	0	0	0
Precoat Tank (TK-6.4)	TK-6.4	0	50	24"	0	12"	---
Preg. Liquor Aeration Tank (TK-6.5)	TK-6.5	0	395	4'	0	3'	---
Filter Precoat Mix Tank (TK-6.6)	TK-6.6	0	50	24"	0	12"	---
Water Trap Silencer-Separator (TK-6.7)	TK-6.7	0	--	Included	0	0	0
H20 C20 Mix Tank (TK-6.8)	T-K6.8	0	50	--	--	--	--
Rake Mechanism (TM-6.1)	TM-6.1	0	--	Included	0	0	0

Table 3-102: Equipment Specifications for Ancillary Operations (Cost Center 07)

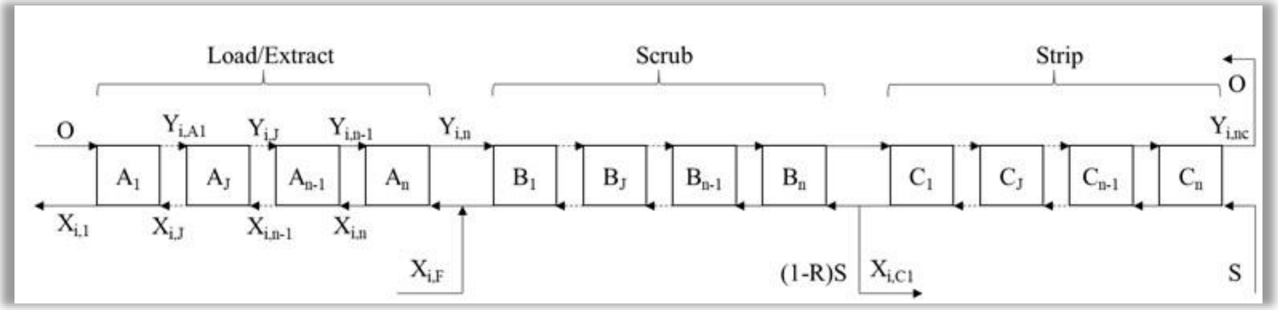
Description	ID	HP	Wt. (lbs.)	Height	Width	Diameter	Length
Cobalt Shot Column	CM-7.1	0	150	--	--	--	--
pH Controller	CP-7.1	0	50	--	--	--	--
Component Equipment (EC-7.1)	EC-7.1	0	250	--	--	--	--
Air Compressor (C-8.1)	C-8.1	10	--	--	--	--	--
Cooling Tower (CT 8.1)	CT-8.1	10	--	--	--	--	--
Air Dryer (D-8.2)	D-8.2	5	--	--	--	--	--
Well Pump 56 GPM (P-8.1)	P-8.1	1	--	--	--	--	--
Raw Water Supply Pump 50 GPM (P-8.3)	P-8.3	1	--	--	--	--	--
Raw Water Supply Pump 50 GPM (P-8.3A)	P-8.3A	1	--	--	--	--	--
CWS Pump 200 GPM (P-8.4)	P-8.4	2	--	--	--	--	--
Sump Pump (P-8.5)	P-8.5	1	--	--	--	--	--
Feed Water Chemical Metering Pump 3 GPM (P-8.7)	P-8.7	1/2	--	--	--	--	--
Fuel Oil Transfer Pump 0.5 GPM (P-8.7)	P-8.7	1/2	--	--	--	--	--
Steam Generator (SG-8.1)	SG-8.1	5	--	--	--	--	--
Raw Water Storage (TK-8.1)	TK-8.1	0	--	--	--	--	--
Feed Water Chemical Tank (TK-8.2)	TK-8.2	0	--	--	--	--	--
Water Softener/Water Treater with Brine Tank (TK-8.3)	TK-8.3	0	--	--	--	--	--
Raw Water Pressure Vessel (TK-8.4)	TK-8.4	0	--	--	--	--	--
Fuel Oil Storage (TK-8.5)	TK-8.5	0	--	--	--	--	--
Compressed Air Receiver (V-8.1)	V-8.1	0	--	--	--	--	--
480V Motor Control Center	--	--	10,000	--	--	--	--
Tailings Storage Tank	TK-9.001	--	--	--	--	--	--
Carbon Monoxide Generator	--	--	--	--	--	--	--
Chromite Hydrocyclone	CY-9.001	--	--	--	--	--	--
Spare Parts	P-5.2	--	50	10"	1'-8"	0	10"
Various filter components and spares T-76	T-76	--	--	--	--	--	--
Font end Loader	--	--	--	--	--	--	--

3.5.6 MATLAB App for SX Design

The separation and the purification of the rare earth elements using solvent extraction process is influenced by numerous factors. The design of a SX circuit consists of many parts such as choosing appropriate circuit configuration, number of stages for loading, scrubbing and stripping, recirculation, pH, O:A ratio, etc. Because of such a large number of variables in a multi-species system like rare earths it becomes very difficult and tedious to select an appropriate configuration and design parameters. In order to simplify the design process, modeling of SX process is often required. These models are developed based on the fundamental approach of mass balance and equilibrium chemistry. The developed models are used to predict the concentration of elements after every stage and assess the purity of products under various design conditions. This section describes an approach which was adopted to model a multi-species rare earth system for determining the number of stages based on required purity of the products.

A typical solvent extraction system consists of three processes, loading, scrubbing and stripping as shown in *Figure 3-139*. The purpose of loading is to extract the desired element into the organic phase. Scrubbing is done to remove the entrained impurities which is followed by stripping to extract the loaded metal back to aqueous phase. All three processes are done in stages. In *Figure 3-139* the loading, scrubbing and stripping are represented by A, B and C respectively and “j” indicates the stage. The aqueous feed consisting of dissolved metal ion and impurities is introduced at stage “n” and organic is introduced at Stage 1 of the loading process. The liquids flow in counter-current direction and proceed to the next process. As all the processes are of similar nature, the analysis of one can be generalized to other two processes with slight modifications. For stripping, the fresh acid is introduced at stage "n" of stripping process and a bleed is taken out of Stage 1. The fraction of bleed taken out is referred to as reflux ratio which is defined by the fraction of the stripping solution diverted to the scrubbing section.

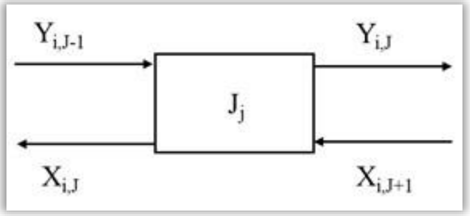
Figure 3-139: A Solvent Extraction System Diagram Showing Loading, Scrubbing and Stripping Arranged so that part of the Scrubbing Stream is Refluxed into the Scrubbing Stages



Modeling of the process can be approached by utilizing the fundamental approach of mass balance on any stage "j" and species "I" (*Figure 3-140*) which in general form is given by:

$$Mass\ IN - Mass\ OUT\ = ACCUMULATION \qquad [3.12]$$

Figure 3-140: Representation of a Single Stage



Thus, for the system shown above at a steady state when accumulation goes to zero the equation results to the following form:

$$Y_{i,J_A-1}O + X_{i,J_A+1}(SR + F) = Y_{i,J_A}O + X_{i,J_A}(SR + F) \qquad [3.13]$$

Where $Y_{i,j}$ represents the concentration of species i in the organic phase, $X_{i,j}$ represents the concentration of species i in the aqueous phase, O is the organic flowrate, S is strip flow rate, R is the reflux ratio and subscript A indicates the loading stage.

The above equation represents a solution for the loading stage. The above equation has four unknown concentrations which can be simplified following the approach of equilibrium chemistry. The general form of chemical reaction which takes place during the loading stage is given by:

$$\mathrm{M^{q+}} + \mathrm{q(RH)_{org}} \leftrightarrow \mathrm{(MR_q)_{org}} + \mathrm{qH^+} \qquad [3.14]$$

Where M^{+q} represents trivalent rare earth metal ion, $(RH)_{org}$ is the organic extractant, $(MRq)_{org}$ is the rare earth complex formed during the extraction phase and H^+ is the hydrogen ion released in the process. The subscript "org" in the chemical reaction represents the species that is in the organic phase.

The relation of equilibrium constant with concentration of species is given by:

$$K_{eq} = \frac{[\mathrm{MR}_q]_{\mathrm{org}}[\mathrm{H}^+]^q}{[\mathrm{M^{q+}}][\mathrm{RH}]_{\mathrm{org}}^q} \qquad [3.15]$$

The ratio of the metal concentration in organic and aqueous phase is represented by the D which is called a distribution coefficient. Thus, distribution coefficient for any species at a given stage can be represented as:

$$D_{i,n} = \frac{Y_{i,n}}{X_{i,n}};\ X_{i,n} = \frac{Y_{i,n}}{K_{i,n}} \qquad [3.16]$$

Utilizing above relation, substituting $X_{i,jA}$ in equation 3.13 and rearranging terms results in following form:

$$Y_{i,J_A-1}O - Y_{i,J_A}\left(O + \frac{SR+F}{D_{i,A}}\right) + Y_{i,J_A+1}\frac{(SR+F)}{D_{i,A}} = 0 \qquad [3.17]$$

The above equation is the general and simplified form of mass balance for the system shown in *Figure 3-140* which can be expanded and applied to entire SX system shown in *Figure 3-139*. Thus, for a three-stage loading, scrubbing and stripping a system of linear equations can be formulated. This system of equation can be solved using matrix approach presented in in the following table. Note that this matrix is modified to include the boundary conditions of inlets and outlets. Further, it defines the associated flow rates found in each stage of the circuit.

Table 3-103: Matrix Formulation for a Load, Scrub, Strip System each with 3 Stages as Shown in Figure 3-139

$Y_{i,1_A}$	$Y_{i,2_A}$	$Y_{i,3_A}$	$Y_{i,1_B}$	$Y_{i,2_B}$	$Y_{i,3_B}$	$Y_{i,1_C}$	$Y_{i,2_C}$	$Y_{i,3_C}$	Stage	In/Out
$-\left(O+\frac{SR+F}{D_{iA}}\right)$	$\frac{(SR+F)}{D_{i,A}}$	0	0	0	0	0	0	○	1 A	0
○	↘	↘	0	0	0	0	0	0	2 A	0
0	○	↘	$\frac{SR}{D_{i,B}}$	0	0	0	0	0	3 A	$-X_{i,F}F$
0	0	○	$-\left(O+\frac{RS}{D_{i,B}}\right)$	$\frac{SR}{D_{i,B}}$	0	0	0	0	1 B	0
0	0	0	○	↘	↘	0	0	0	2 B	0
0	0	0	0	○	↘	$\frac{SR}{D_{i,C}}$	0	0	3 B	0
0	0	0	0	0	○	$-\left(O+\frac{S}{D_{i,C}}\right)$	$\frac{S}{D_{i,C}}$	0	1 C	0
0	0	0	0	0	0	○	↘	↘	2 C	0
0	0	0	0	0	0	0	○	↘	3 C	$X_{i,S}S$

The information on Di in the system of equation is derived from experimental test. The extraction curve generated previously using different reagent scheme are used for this purpose. Substituting D in Eq. 3.16 and taking log transformation results in following form from which Di can be easily evaluated:

$$\log D = q * pH + log\, K_{ex}\left[HA_{org}\right]^{q} \qquad [3.18]$$

The relationship is most important as it integrates the experimental results to the final model form described by Eq. 3.18. The approach can be extended to any number of stages of loading, scrubbing and stripping and hence can be used to evaluation concentration and number of stages.

The derivations described above were coded into a MATLAB “App” as shown in the following figure. This App shows the input parameters of the number of loading, stripping and scrubbing stages for a solvent extraction unit.

Further, a table is included to enter in the desired elements in the feed stream, the linear parameters of the distribution coefficient, and the concentration of elements in the feed stream. The user may also specify the organic and stripping flow rates and the reflux ratio. The feed pH and the stripping pH are also specified by the user. The App allows settings to be read to and from MS Excel spreadsheets for ease of use.

The App outputs the raffinate concentrations, the percentage of extracted elements. Furthermore, the concentrations of elements in the strip solution and an output for mass balance are determined.

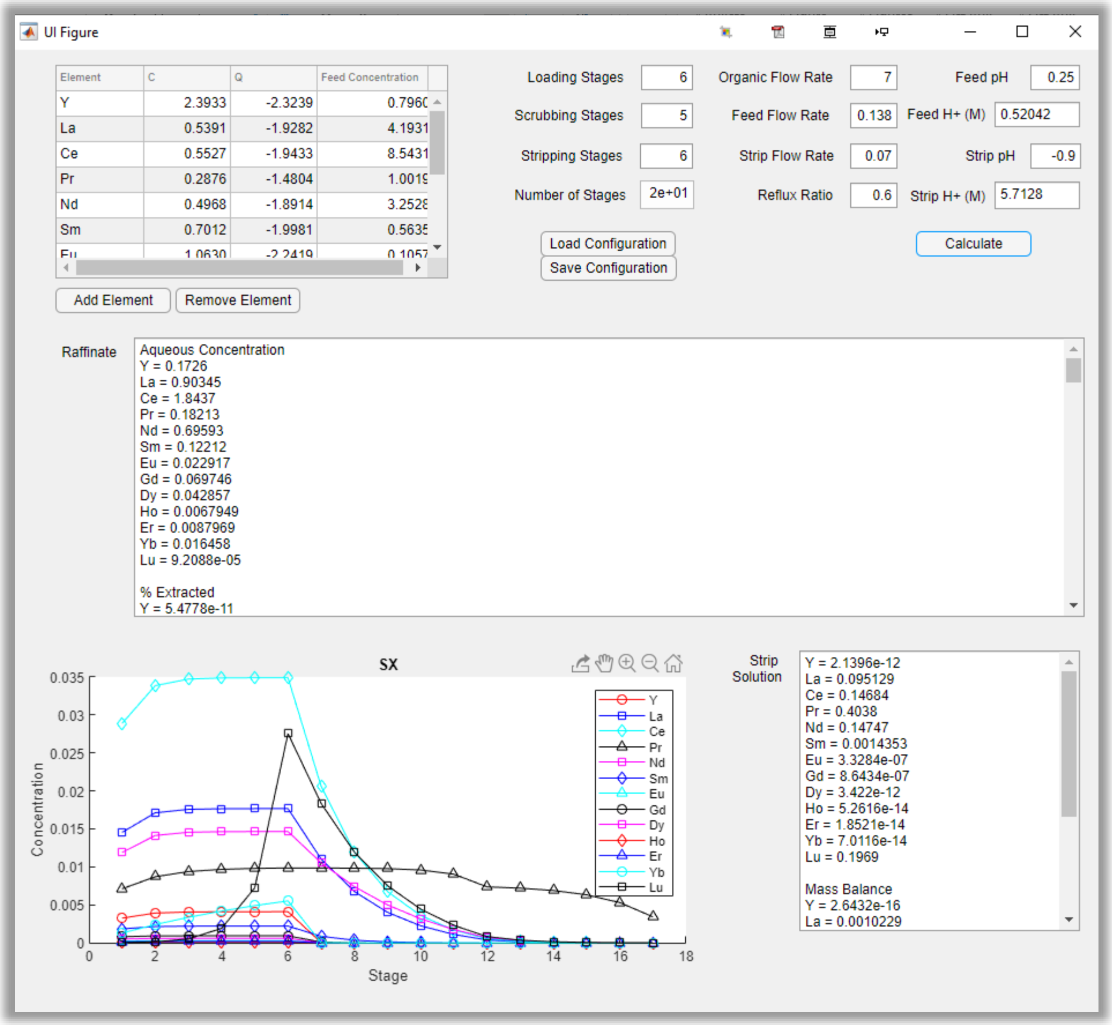
The model in its current form also accounts for internal dilution where the scrub solution is neutralized to the pH of the feed solution. This allows for model validation. The graphical output shows the concentration of each element in the organic phase to assist the engineer in the design of the SX circuit.

By utilizing this App, various stages of the proposed refining circuit were designed through an iterative process. Although care was utilized in the development of this application the following items for verification and testing are required:

1. Due to what are thought to be rounding errors, the model does not have a perfect mass balance. This is exaggerated for elements that are highly refined. The imbalances are significant and compound for refined elements.
2. The data set utilized for the distribution coefficients did not contain stripping data. Thus, the stripping simulation may not be as accurate as needed.
3. Further testing is highly recommended to validate the circuit in operation in a small scale to validate the model.

We believe that the SX modeling is a good tool for initial approximations, but additional verifications and validations are required for further use.

Figure 3-141: MATLAB App Showing the Calculation of SX Parameters



Element	C	Q	Feed Concentration
Y	2.3933	-2.3239	0.796
La	0.5391	-1.9282	4.1931
Ce	0.5527	-1.9433	8.5431
Pr	0.2876	-1.4804	1.001
Nd	0.4968	-1.8914	3.252
Sm	0.7012	-1.9981	0.563
Eu	1.0630	-2.2419	0.1057

Loading Stages 6
Scrubbing Stages 5
Stripping Stages 6
Number of Stages 2e+01
Organic Flow Rate 7
Feed Flow Rate 0.138
Strip Flow Rate 0.07
Reflux Ratio 0.6
Feed pH 0.25
Feed H+ (M) 0.52042
Strip pH -0.9
Strip H+ (M) 5.7128

Raffinate

```
Aqueous Concentration
Y = 0.1726
La = 0.90345
Ce = 1.8437
Pr = 0.18213
Nd = 0.69593
Sm = 0.12212
Eu = 0.022917
Gd = 0.069746
Dy = 0.042857
Ho = 0.0067949
Er = 0.0087969
Yb = 0.016458
Lu = 9.2088e-05

% Extracted
Y = 5.4778e-11
```

Strip Solution

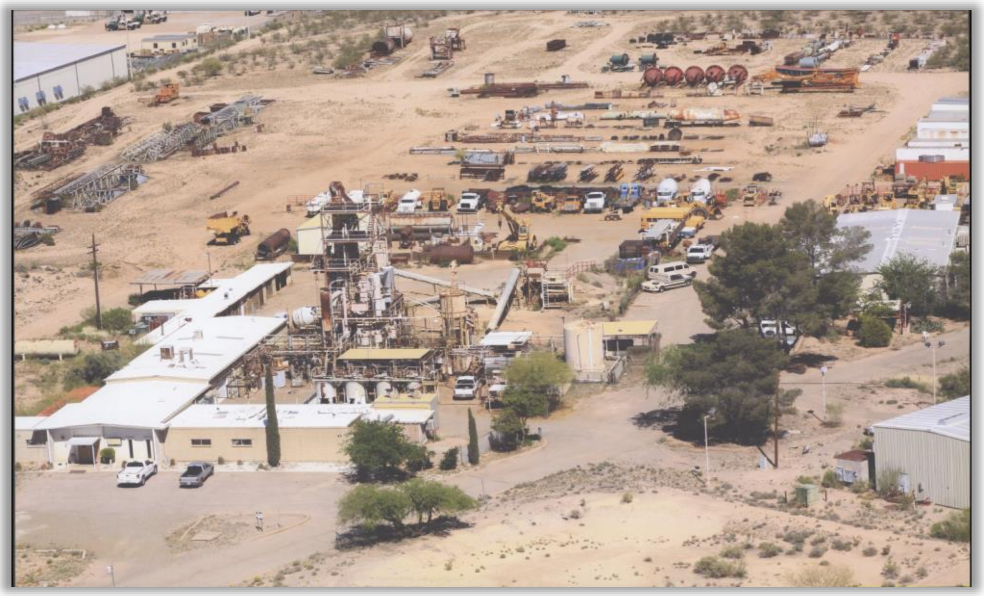
```
Y = 2.1396e-12
La = 0.095129
Ce = 0.14684
Pr = 0.4038
Nd = 0.14747
Sm = 0.0014353
Eu = 3.3284e-07
Gd = 8.6434e-07
Dy = 3.422e-12
Ho = 5.2616e-14
Er = 1.8521e-14
Yb = 7.0116e-14
Lu = 0.1969

Mass Balance
Y = 2.6432e-16
La = 0.0010229
```

3.6 Flow Sheet of 1 tph Feed Pilot Plant

3.6.1 Engineering Drawings Needed for Construction/Fabrication and Installation of System

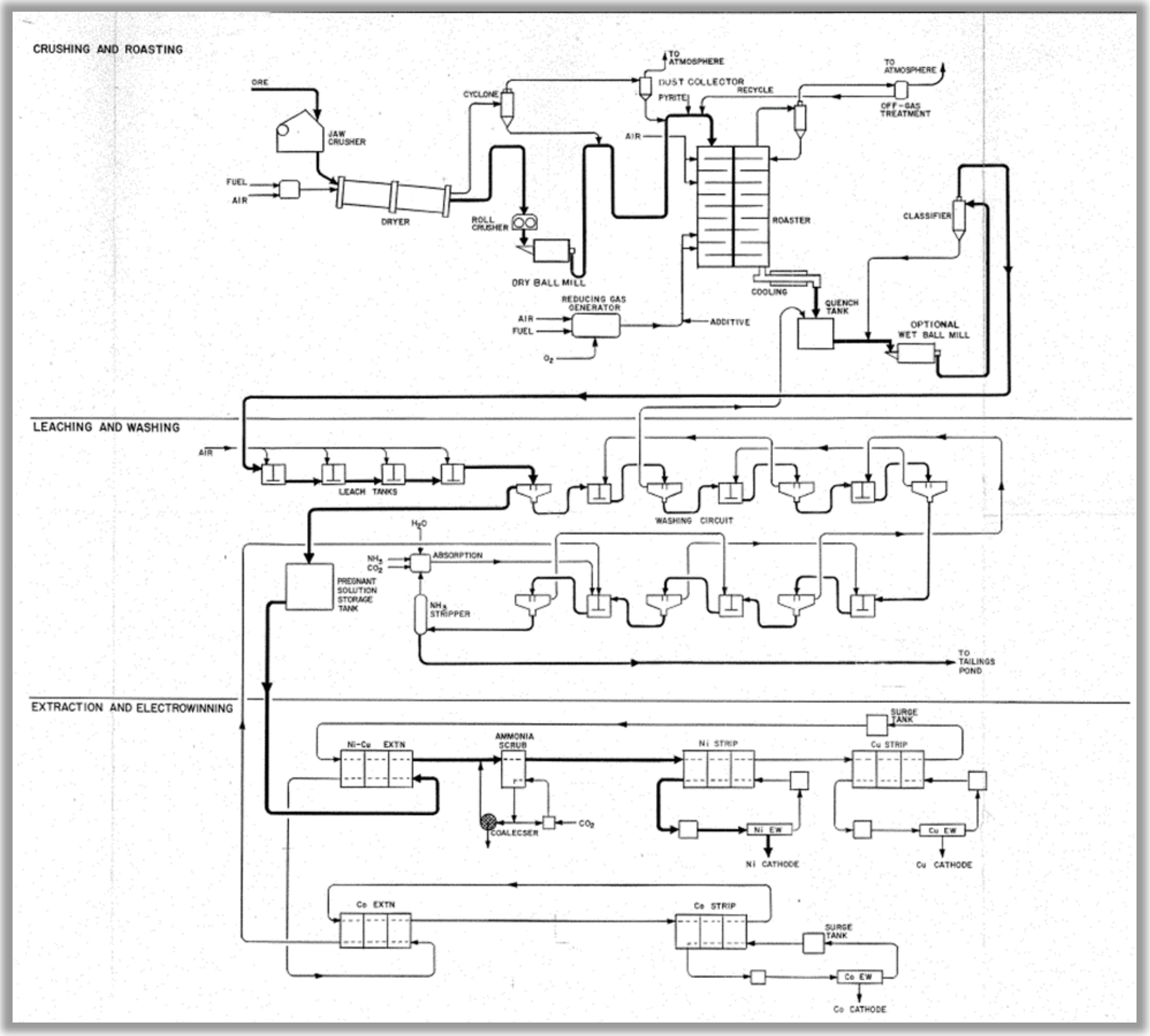
Due to the short time frame allotted to the Project in Phase 2, the procurement and fabrication of equipment for the project was the primary concern. From the knowledge and network of the design team, a potential pilot plant was located in the Western United States that may be suitable for re-engineering and restoration. This could represent a significant time and labor savings for the project to decrease risks. The following figure shows an aerial photograph of the plant. As part of the due diligence, critical process drawings were obtained to better understand the intended function of the plant and assess the suitability for our process.

Figure 3-142: Previously Operated Pilot Plant Potentially Suited for Reutilization



Based upon the documentation received to-date, it appears that this Pilot Plant was intended to process Ni Laterite ores. The following figure shows the process flow sheet as originally configured. It is unique in terms of the processing equipment which is generally not found in this configuration.

Figure 3-143: Pilot Plant Original Flow Sheet

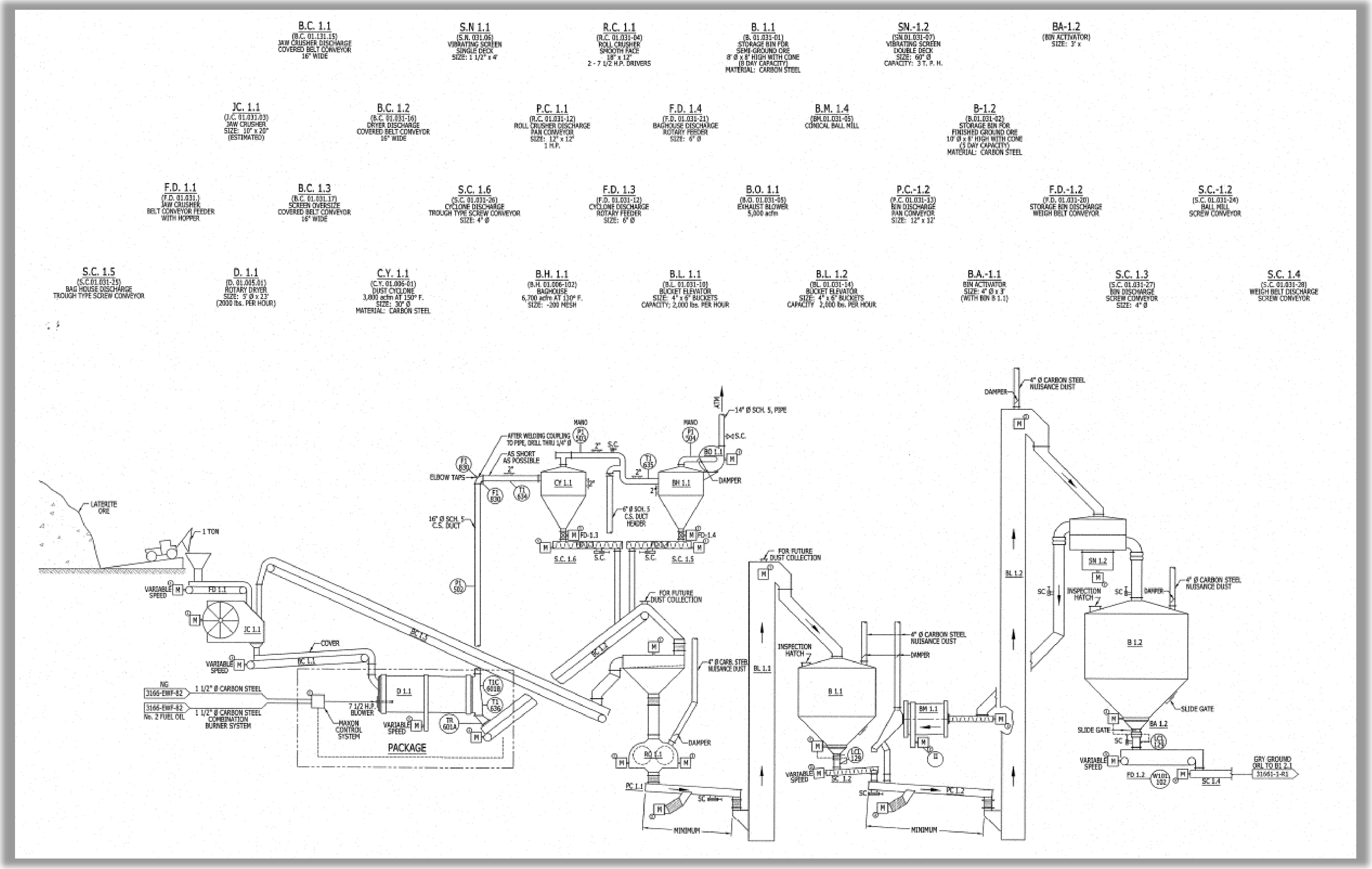


The plant consists of a comminution circuit, a roaster and a hydrometallurgical circuit which included solvent extraction. The plant was designed to take coarse and wet ore through a jaw crusher into a dryer. At this point, a dry ball mill with a dust collection circuit feeds a roasting circuit which includes an emissions scrubber to remove sulfur. This is significant for our project due to the pyrite content in coals. This arrangement is ideal and would be what we are proposing in light of the significant improvements provided by roasting. It appears from this figure that additional grinding is possible after roasting, which may be beneficial for our material. What is supremely unique about this flowsheet is the 7-Stage counter-current leaching circuit with 10 leaching vessels. Four leaching vessels are arranged to receive the roasted feed in a way that will approach a plug-flow reactor performance feeding a thickener to remove the solids from the PLS. The residual solids are further processed in six leach tanks with each paired with a thickener to allow counter-current flow. From the flow sheet, it appears that ammonia was used as a lixiviant. One key issue will be to reconnoiter the material of construction to determine suitability for acid leaching. The flowsheet appears to send thickener underflow directly to tailings. If this plant proves suitable, additional design will be required to neutralize and dewater the tailings.

The SX portion of the circuit is interesting and very amenable to our process. As designed, it has dual loading for two different SX streams. We intend to have three as per the preceding figure. Due to the nature of using ammonia as a lixiviant, it appears that ammonia was soluble to some extent in the organic phase of the Ni-Cu SX circuit. As shown in that preceding figure, an ammonia removal step is implemented between loading and scrubbing/stripping. Co-extraction was accomplished via a second load/strip SX phase. The end result was three streams fed to respective electrowinning processes.

The following figure shows the piping and instrumentation diagram (P&ID) of the crushing plant which gives some clue to the capacity of the plant. BL 1.1 and 1.2 are the bucket elevators which are sized to 2,000 lb. per hour. Assuming that the plant capacity matches the bucket elevator, this plant would require 300 ppm REE feed material at a minimum to produce the requisite 10 lb. per day of REE with 24 per day operation assuming 70 percent recovery. Conversely, with a 600-ppm feed, only 12 hours are needed. Thus, this plant is sized well for the proposed operation.

Figure 3-144: P&ID Diagram of the Crushing Segment of the Previously Used Pilot Plant



3.6.1.1 Utilization of the Flow Sheets from an Existing Pilot Plant

As detailed in Chapter 8, a Pilot Plant has been located which included a set of drawings. As part of this task, the drawings were evaluated and are briefly presented here as part of the flow sheet development effort.

The following drawings will be directly applicable to utilization as part of the flow sheet. These *Figure 3-145* which shows the crushing and grinding circuit, *Figure 3-146* which shows the roasting circuit, and which shows the roaster gas scrubbing system. As demonstrated from these diagrams, pumps, pipe sizes, valves, and instrumentation are listed. These drawings are extremely valuable to the project and result in substantial labor savings.

The other portions of the plant which include the leaching sections, water, and tailings are still currently available.

Figure 3-145: Previously Used Pilot Plant Crushing and Grinding P&ID Diagram (Repeated Diagram)

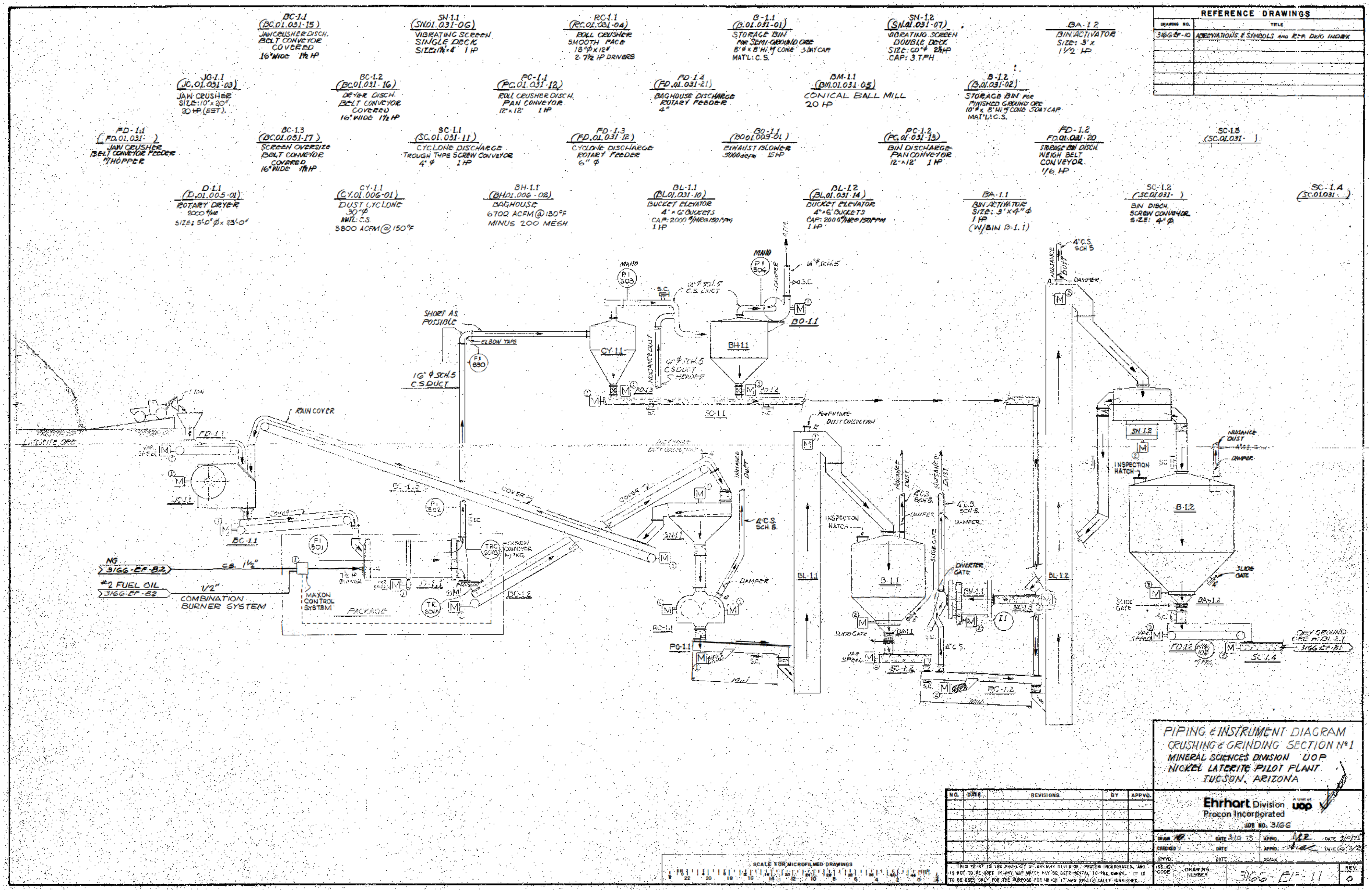


Figure 3-146: Previously Used Pilot Plant Roasting Circuit P&ID

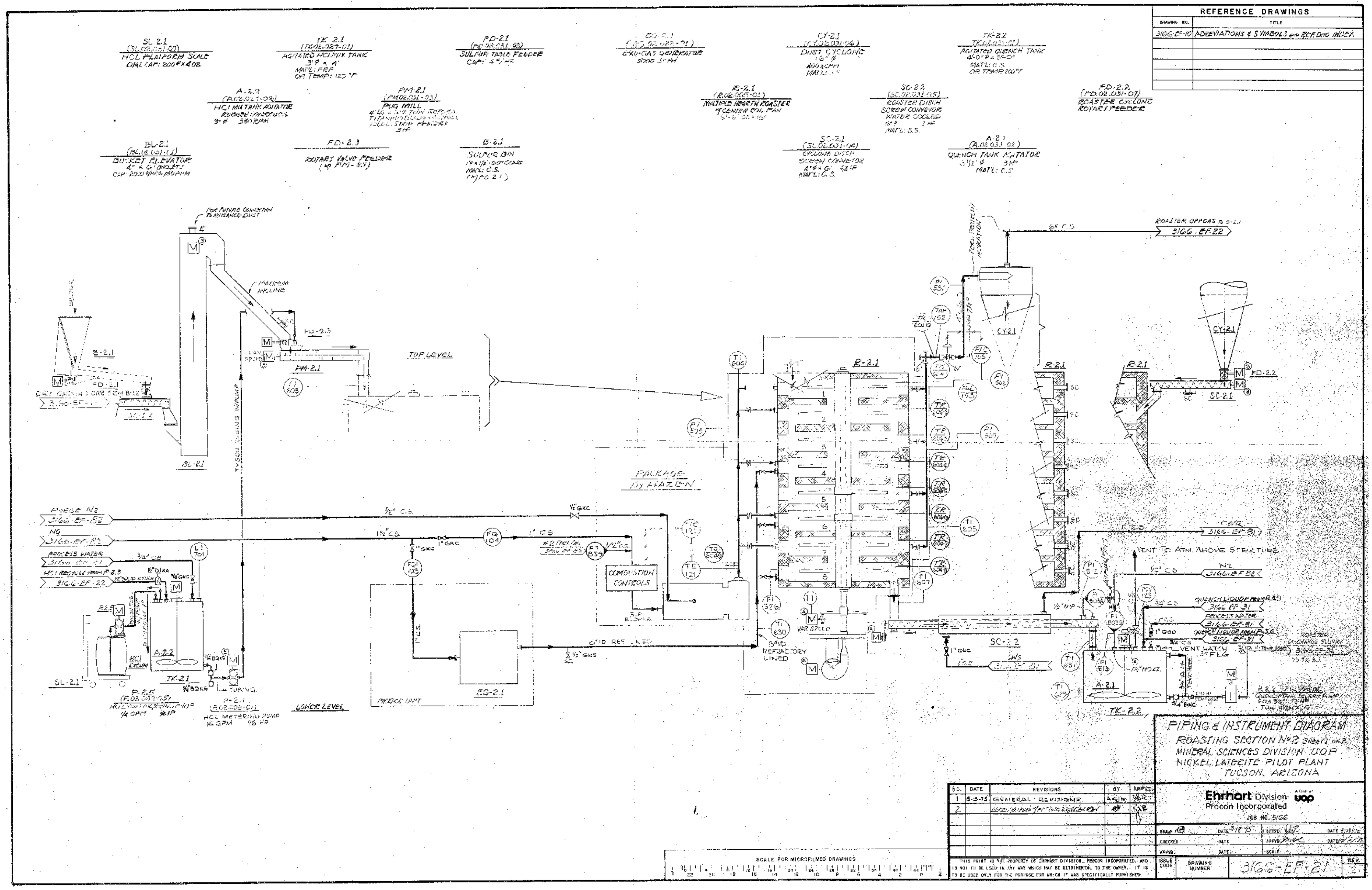
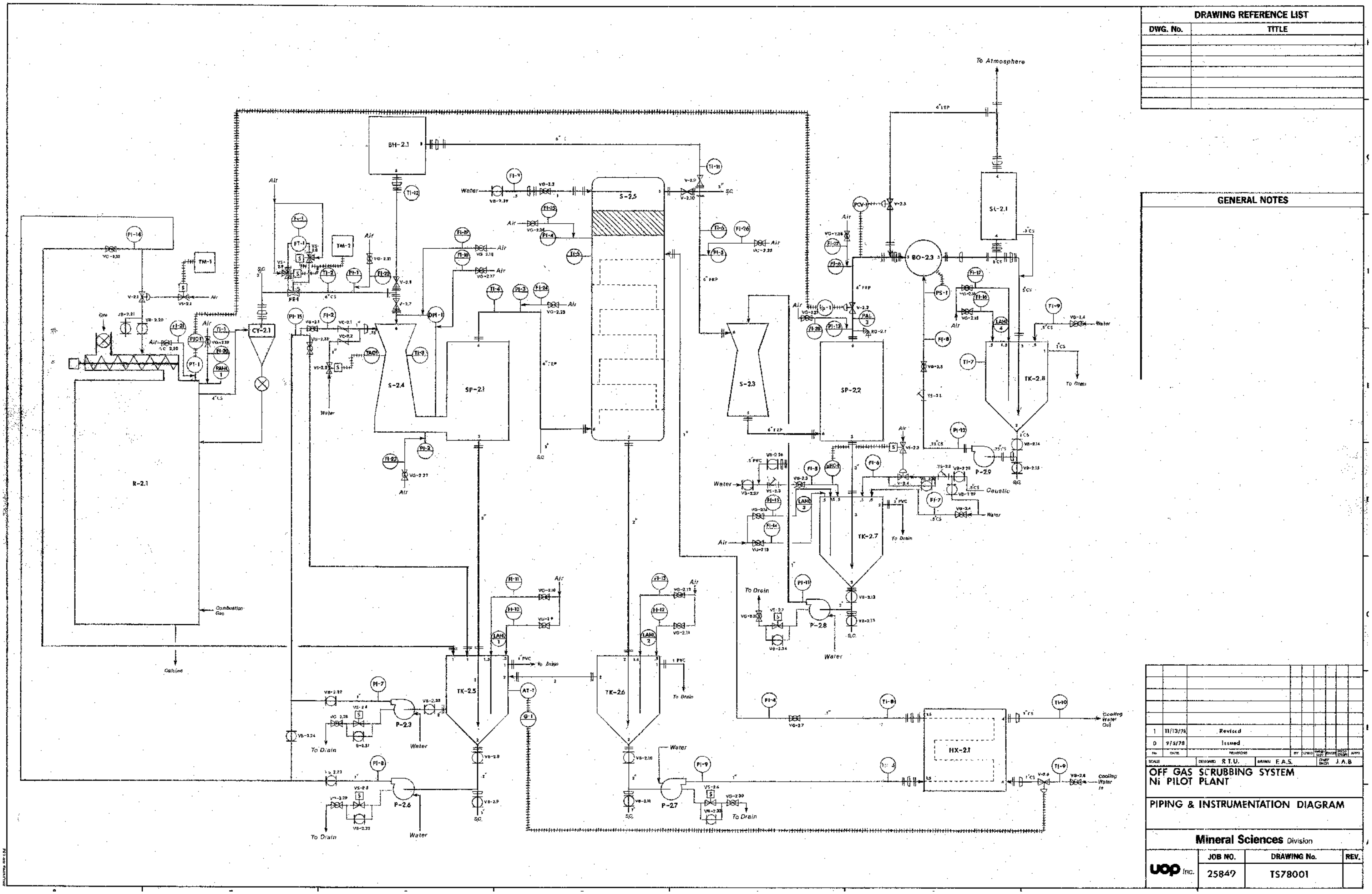


Figure 3-147: Previously Used Pilot Plant Roasting Gas Scrubbing P&ID



3.6.1.2 Previously Used Pilot Plant Construction Drawings Showing Spatial Layout of Equipment/Piping/Plumbing/Electrical/Mechanical

3.6.1.2.1 Crushing and Grinding Equipment and Layout

To accomplish the objectives of the project, a detailed engineering design of the flowsheet has been performed during the quarterly reporting periods. As stated in those prior reports, the team was able to acquire most of the original equipment drawings for the pilot plant. As such, the equipment was redrawn utilizing SOLIDWORKS®[21]. The level of detail selected was to provide accurate layout and placement positioning. Care was given to the foundations of the proposed facility. Equipment was drawn from provided equipment drawings, general arrangements and where insufficient data existed, pictures. This methodology was utilized to provide the requisite level of detail for this study. The drawings contained herein are preliminary and subject to change and additional formatting. They are included to show the engineering progress and design to this point.

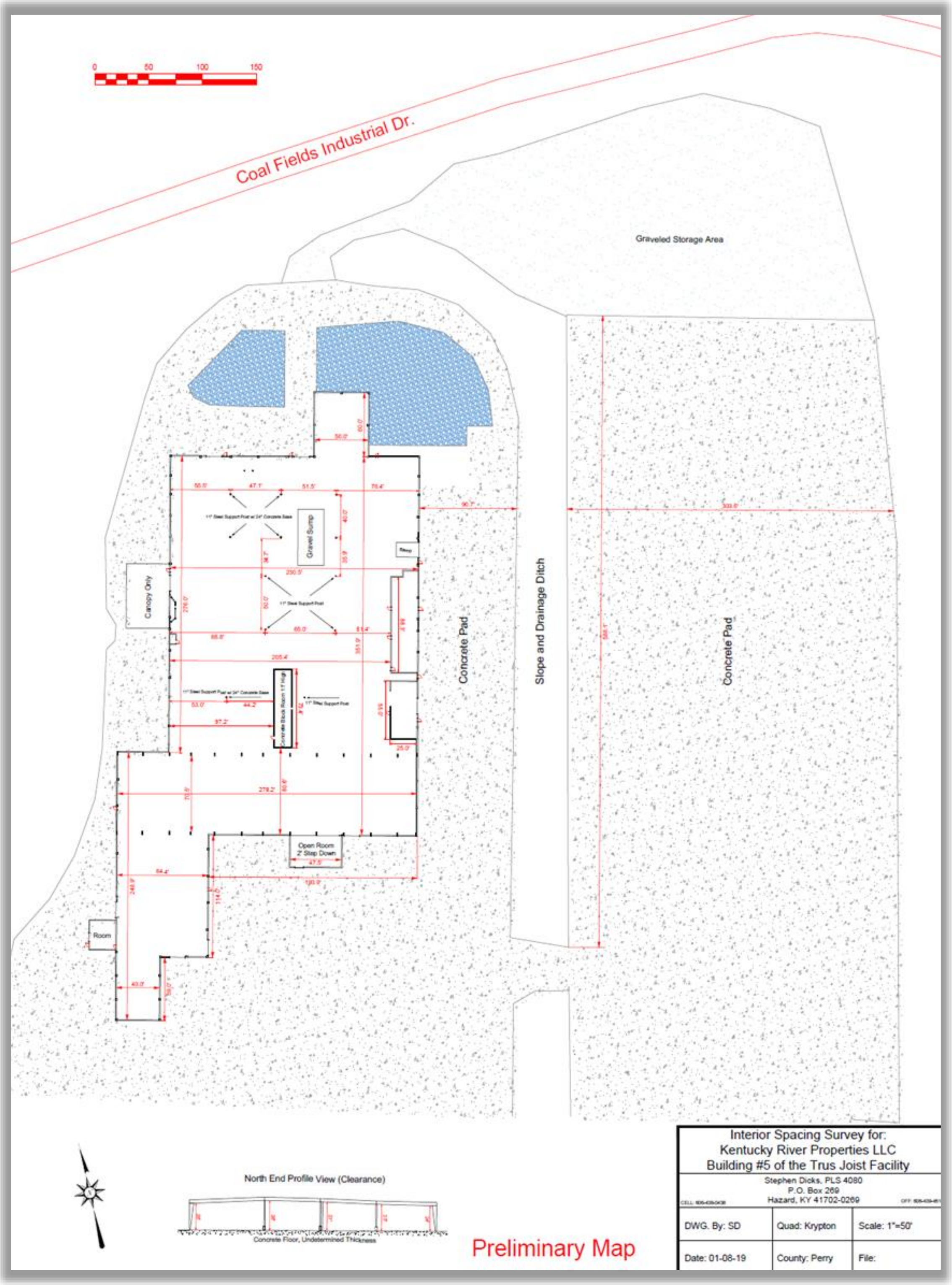
A survey of the Trus Joist facility[22] was conducted previously to determine an accurate representation of the interior of the facility and to assist the team designing the Pilot Plant layout. The site and building map of the surveyed Trus Joist facility is shown in the following figure.

---

[21] SOLIDWORKS® is a solid modeling computer-aided design (CAD) and computer-aided engineering (CAE) computer program that runs on Microsoft Windows. SolidWorks is published by Dassault Systèmes.

[22] Trus Joist is a former wood laminate beam fabrication facility located on 610 Trus Joist Lane in the Coal Fields Industrial Park, near Chavies, Kentucky. The site is now owned by a subsidiary of KRP and is the proposed site for a 1 tph Pilot Plant.

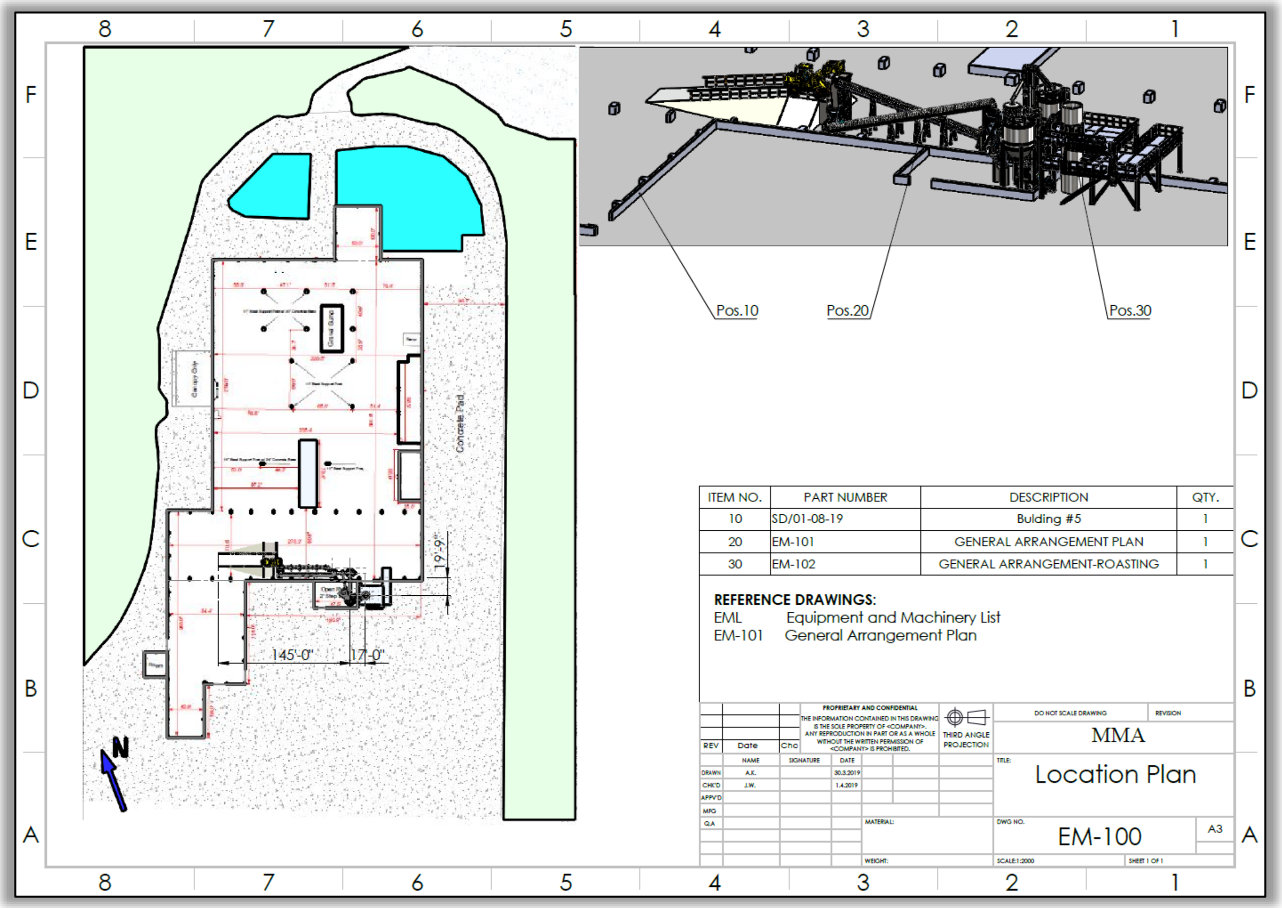
Figure 3-148: Surveyed Map of the Truss Joist Facility



The layout in the Truss Joist facility was determined as shown in the following figure. The crushing, grinding and roasting circuit was determined to best fit along the south side of the building. The crushing and grinding circuit was determined to best be located indoors. The primary reason for this decision was for dust control. Locating the circuit indoors would allow the stockpiling of materials out of the wind. It will also prevent run-off by rainwater. This is important to prevent environmental contamination from the coal rejects feed stock. Location of the crushing and grinding equipment indoors will also preserve the equipment from degradation from the elements and enhance operator comfort.

An outdoor location was identified as the best option for the roasting circuit. This decision was determined to be the most desirable due to the heat generated by the roaster and the potential for combustion gas generation. If located indoors, extensive ventilation would be needed to ensure proper safety and an acceptable working environment. In this configuration additional roofing will be needed, but the roasting portion of the plant will be largely exposed allowing for heat rejection, ease of access to the roaster and firefighting access in the unlikely event of a fire.

Figure 3-149: Location of Crushing and Roasting Equipment in Relation to the Truss Joist Facility



ITEM NO.	PART NUMBER	DESCRIPTION	QTY.
10	SD/01-08-19	Bulding #5	1
20	EM-101	GENERAL ARRANGEMENT PLAN	1
30	EM-102	GENERAL ARRANGEMENT-ROASTING	1

REFERENCE DRAWINGS:
EML Equipment and Machinery List
EM-101 General Arrangement Plan

PROPRIETARY AND CONFIDENTIAL
THE INFORMATION CONTAINED IN THIS DRAWING IS THE SOLE PROPERTY OF <COMPANY>. ANY REPRODUCTION IN PART OR AS A WHOLE WITHOUT THE WRITTEN PERMISSION OF <COMPANY> IS PROHIBITED.

THIRD ANGLE PROJECTION

DO NOT SCALE DRAWING
REVISION
MMA

	NAME	SIGNATURE	DATE
DRAWN	A.K.		30.3.2019
CHK'D	J.W.		1.4.2019
APPV'D			
MFG			
Q.A			

REV Date Chc

MATERIAL:
WEIGHT:
TITLE: Location Plan
DWG NO. EM-100
A3
SCALE:1:2000
SHEET 1 OF 1

To show specific details of the crushing and grinding circuit, the following two figures are provided showing the general arrangement of the equipment.

Figure 3-150: 3D View of the Proposed 1 tph Pilot Plant's Crushing and Grinding Circuit showing Jaw Crusher, Conveyors, Roller Crusher, Screens Ball Mill and Storage Bins

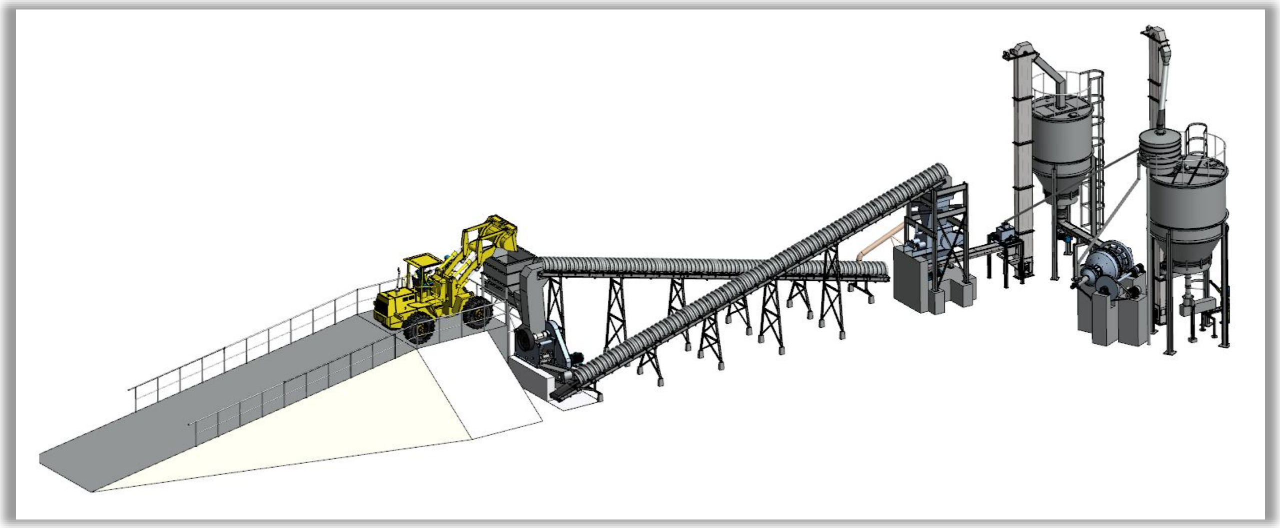


Figure 3-151: Alternate View of the Proposed 1 tph Pilot Plant's Crushing and Grinding Circuit

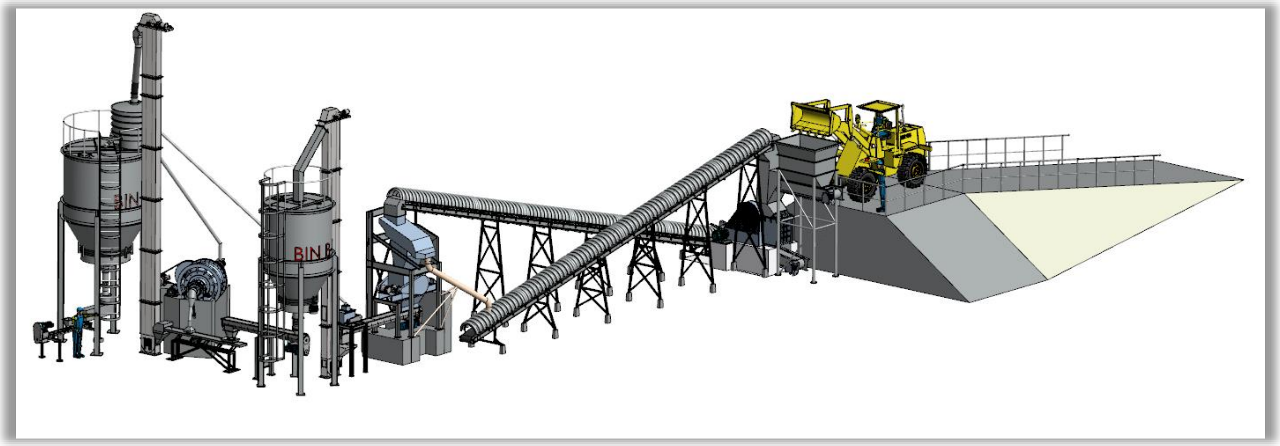
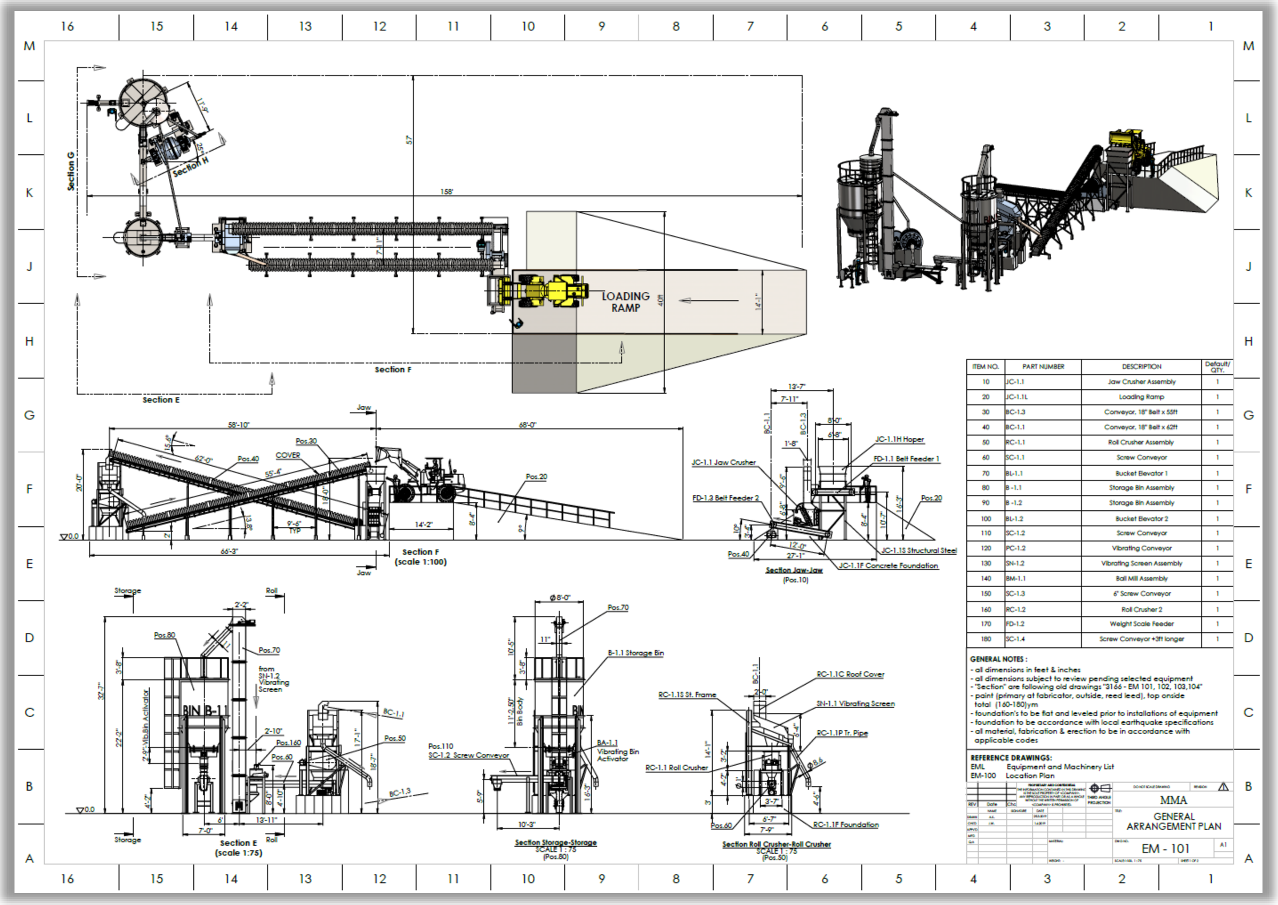


Figure 3-152 and *Figure 3-153* show the general arrangement of the crushing and grinding circuit and associated dimensions. The Pilot Plant will in general preserve the layout of the original location with the notable exception of the dryer. In the original Pilot Plant, a dryer was included so that the ore could be effectively sized and screened. The team feels with the coarse sizing of the material indicated by the laboratory testing, the drying step will not be needed. However, due to the layout of the plant and original design, provisions are available to add the dryer if needed. The current layout modifications consist of conveyor re-arrangement to allow for the oversize return from jaw crusher screen to feed back to the jaw crusher. The table that follows the two figures is included to list the equipment described in those figures.

Figure 3-152: General Arrangement of the Proposed Relocated 1 tph Pilot Plant Crushing and Grinding Circuit



ITEM NO.	PART NUMBER	DESCRIPTION	Default/ QTY.
10	JC-1.1	Jaw Crusher Assembly	1
20	JC-1.1L	Loading Ramp	1
30	BC-1.3	Conveyor, 18" Belt x 55ft	1
40	BC-1.1	Conveyor, 18" Belt x 62ft	1
50	RC-1.1	Roll Crusher Assembly	1
60	SC-1.1	Screw Conveyor	1
70	BL-1.1	Bucket Elevator 1	1
80	B -1.1	Storage Bin Assembly	1
90	B -1.2	Storage Bin Assembly	1
100	BL-1.2	Bucket Elevator 2	1
110	SC-1.2	Screw Conveyor	1
120	PC-1.2	Vibrating Conveyor	1
130	SN-1.2	Vibrating Screen Assembly	1
140	BM-1.1	Ball Mill Assembly	1
150	SC-1.3	6" Screw Conveyor	1
160	RC-1.2	Roll Crusher 2	1
170	FD-1.2	Weight Scale Feeder	1
180	SC-1.4	Screw Conveyor +3ft longer	1

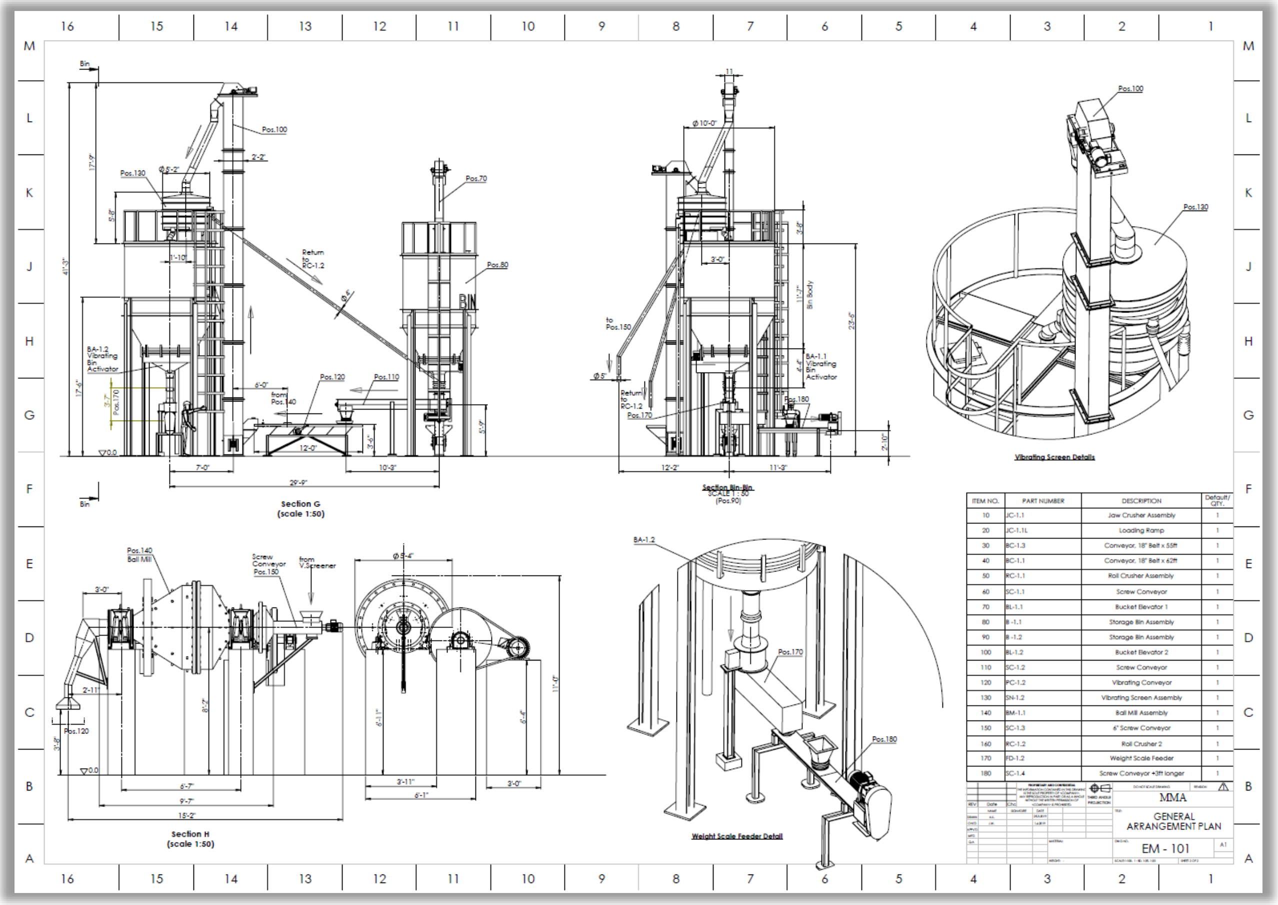
GENERAL NOTES :

- all dimensions in feet & inches
- all dimensions subject to review pending selected equipment
- "Section" are following old drawings "3166 - EM 101, 102, 103,104"
- paint (primary at fabricator, outside, reed leed), top onside total (160-180)ym
- foundation's to be flat and leveled prior to installations of equipment
- foundation to be accordance with local earthquake specifications
- all material, fabrication & erection to be in accordance with applicable codes

REFERENCE DRAWINGS:

EML Equipment and Machinery List
EM-100 Location Plan

Figure 3-153: Additional Equipment Details of a Proposed Relocated 1 tph Pilot Plant Crushing and Grinding Circuit



ITEM NO.	PART NUMBER	DESCRIPTION	Default/ QTY.
10	JC-1.1	Jaw Crusher Assembly	1
20	JC-1.1L	Loading Ramp	1
30	BC-1.3	Conveyor, 18" Belt x 55ft	1
40	BC-1.1	Conveyor, 18" Belt x 62ft	1
50	RC-1.1	Roll Crusher Assembly	1
60	SC-1.1	Screw Conveyor	1
70	BL-1.1	Bucket Elevator 1	1
80	B -1.1	Storage Bin Assembly	1
90	B -1.2	Storage Bin Assembly	1
100	BL-1.2	Bucket Elevator 2	1
110	SC-1.2	Screw Conveyor	1
120	PC-1.2	Vibrating Conveyor	1
130	SN-1.2	Vibrating Screen Assembly	1
140	BM-1.1	Ball Mill Assembly	1
150	SC-1.3	6" Screw Conveyor	1
160	RC-1.2	Roll Crusher 2	1
170	FD-1.2	Weight Scale Feeder	1
180	SC-1.4	Screw Conveyor +3ft longer	1

Table 3-104: Descriptions of the Proposed Pilot Plant Equipment

Item #	Part #	Description	Quantity
10	JC-1.1	Jaw Crusher Assembly	1
20	JC-1.1L	Loading Ramp	1
30	BC-1.3	Conveyor, 18" Belt x 55ft	1
40	BC-1.1	Conveyor, 18" Belt x 62ft	1
50	RC-1.1	Roll Crusher Assembly	1
60	SC-1.1	Screw Conveyor	1
70	BL-1.1	Bucket Elevator 1	1
80	B-1.1	Storage Bin Assembly	1
90	B-1.2	Storage Bin Assembly	1
100	BL-1.2	Bucket Elevator 2	1
110	SC-1.2	Screw Conveyor	1
120	PC-1.2	Vibrating Conveyor	1
130	SN-1.2	Vibrating Screen Assembly	1
140	BM-1.1	Ball Mill Assembly	1
150	SC-1.3	6" Screw Conveyor	1
160	RC-1.2	Roll Crusher 2	1
170	FD-1.2	Weight Scale Feeder	1
180	SC-1.4	Screw Conveyor +3ft longer	1

3.6.2 Design Sufficient for CapEx Estimates

Based on the work recited in this Chapter, the design efforts were sufficient to have the following items estimated:

1. Plant disassembly and loadout (Cimetta Engineering).
2. Plant transport, refurbishment, improvement and reassembly.
3. Solvent extraction equipment needed for the roughing/and cleaning circuit.
4. Feeds and sizes and stages needed for refining.

3.7 Project Risk and Fatal Flaw Analysis

As part of the scope of work a fatal flaw analysis was performed. In this case, the analysis identifies what the team feels are the most likely and most severe failures that may occur in the technical components of the project. Several important safety considerations are also included.

3.7.1 Risk and Fatal Flaw Analysis Methodology

The technique utilized was a failure mode and effects analysis *(FMEA)*. A FMEA is a matrix of the analysis of potential failures, their severity, what causes them and what may be done to prevent and detect failures. The FMEA was populated with the following number scheme in order of increasing severity. These are 1,3,7, and 10. The highest ranking, 10 is reserved for significant safety items. The ratings for failure severity, occurrence and detection are multiplied together to determine the risk priority number *(RPN)*. The higher the RPN the greater the risk to the project.

3.7.2 Limits of the Analysis Procedure

The analysis was performed over the main portions of the proposed process flow diagram which included feed, crushing and grinding, roasting, leaching, various modes of solvent extraction including refining, waste materials and regulatory compliance items which include environmental protection. This analysis was not meant to be all-inclusive but is expected to identify the potentially fatal flaws of the process design or project implementation from a technical perspective.

3.7.2.1 Risk and Fatal Flaw Analysis Matrix

There are a number of risks identified as part of the analysis. Please consult the following FEMA-based tables for an awareness of the perceived technical risks.

Table 3-105: Risk Analysis Related to Feedstock Material

Process Number	Process Name/ Description	Potential Failure Mode	Potential Effect(s) of Failure	Severity	Potential Cause of Failures	Occurrence	Current Process Controls Prevention	Current Process Controls Detection	Detection	R.P.N.
0	Feed material	Variation	Low REEs	7	Deposit characteristics	3	Geological Survey	Equipment inspection	5	105
		Insufficient REEs	Project Failure	7	Inadequate exploration and resource analysis	3	See resource and reserve estimation section	Exploration and analysis	5	105

Table 3-106: Risk Analysis Related to Crushing and Grinding Processes

Process Number	Process Name/ Description	Potential Failure Mode	Potential Effect(s) of Failure	Severity	Potential Cause of Failures	Occurrence	Current Process Controls Prevention	Current Process Controls Detection	Detection	R.P.N.
1	Crushing and Grinding	Refurbishment	Cost overruns	7	Equipment requires more refurbishment that anticipated	3	Equipment inspection	Equipment inspection	5	105
		Equipment not refurbishable	Time Delay	7	Equipment requires more refurbishment that anticipated	3	Equipment inspection	Equipment inspection	5	105
		Equipment capacity too little	Not able to meet production quota	7	Characteristics of the ore	3	Ore testing	Engineering experience of the team	5	105
		Ore too wet	Plugging of screens	3	Insufficient drying	3	Determining moisture content of ore	Visual	3	27
		Ore not crushed to correct size	Poor Leaching Performance	5	Ore not characterized	3	Lab testing	Lab testing	3	45
		Friction point	Conveyor Fire	10	Jammed idler	3	Maintenance	SOPs	3	90

Table 3-107: Risk Analysis Related to the Ore Roasting Process

Process Number	Process Name/ Description	Potential Failure Mode	Potential Effect(s) of Failure	Severity	Potential Cause of Failures	Occurrence	Current Process Controls Prevention	Current Process Controls Detection	Detection	R.P.N.
2	Roasting	Refurbishment of roaster	Roaster cannot be rebuilt	7	Shell in poor condition	5	Equipment inspection	Equipment inspection	5	175
		Refurbishment of roaster	Roaster cannot be rebuilt	7	Drawings insufficient or reverse engineering not possible	3	Consultation with fabricator	Equipment inspection	5	105
		Refurbishment of gas handling not possible	Cost overruns	7	Equipment requires more refurbishment that anticipated	5	Consultation with OEM	Equipment inspection	5	175
		Reconfiguration to oxidizing roasting	Design problems leading to cost overruns	7	Engineering Experts not consulted	5	Consultation with experts in detailed engineering phase to cost	Detailed engineering	3	105
		Reducing atmosphere	Poor leaching performance	5	Insufficient oxygen	5	Conversion of roaster	Detailed engineering	3	75
		Temperature too low	Pool leaching performance	5	Roaster operated incorrectly	3	PLC and SOPs	Thermo-couples	1	15
		Temperature to high	Pool leaching performance	5	Roaster operated incorrectly	3	PLC and SOPs	Thermo-couples	1	15
		Fine Coal Dust in Oxidizing Environment	Coal dust explosion	10	Finely dispersed coal dust	5	Consultation with experts	Detailed engineering	5	250
		Explosive gas mixture	Gas explosion	10	Improper startup and roaster operation	3	PLC and SOPs	Gas monitoring in detailed engineering	3	90
		Roaster gas handling not sufficient	Carbon Monoxide Poisoning	10	Positive pressure in the roaster off gas system	3	Engineering controls, designed to operate at negative pressure	Carbon Monoxide monitors/ roaster located outside	3	90

Table 3-108: Risk Analysis Related to the Leaching Process

Process Number	Process Name/ Description	Potential Failure Mode	Potential Effect(s) of Failure	Severity	Potential Cause of Failures	Occurrence	Current Process Controls Prevention	Current Process Controls Detection	Detection	R.P.N.
3	Leaching	Poor Leaching Performance	Project Failure	7	Low acid concentration	3	PLC and SOPs	pH Prob	1	21
		Poor Leaching Performance	Project Failure		Temperature too high in roasting	3	See Roasting Section	See Roasting Section	NA	NA
		Excessive Acid Consumption	Costs	5	Excessive acid consumers	3	Stability of the ore body	Geological testing	5	75
		Acid Spills	Health issues	10	Overflowed tanks	3	PLC	Tank level sensors	1	30
		Acid Spills	Health issues	10	Make up tank left filling unattended	3	SOPs and training	Reliance upon operators	7	210
		Vapors	Health issues	10	Ventilation System	3	Engineering design	Reliance upon operators	3	90
		Feed not cooled sufficiently	Fire	10	Heat exchanging screw feeder operated improperly	3	PLC and SOPs	Startup testing	3	90

Table 3-109: Risk Analysis Related to the Roughing SX Process

Process Number	Process Name/ Description	Potential Failure Mode	Potential Effect(s) of Failure	Severity	Potential Cause of Failures	Occurrence	Current Process Controls Prevention	Current Process Controls Detection	Detection	R.P.N.
4	Roughing SX	Poisoning of organic	Loss of REEs	7	Fe, Th, U, Sc not removed from organic	5	Correct operation of the saponification circuit	Assay	3	105
		Dragout	Loss of Organic	7	Coalescer not working as designed	3	Coalescer	Organic level monitoring	5	105
		Dragout	Loss of REEs	7	Coalescer not working as designed	3	Coalescer	Assay	3	63
		Dragout	Excessive reagent consumption	7	Coalescer not working as designed	3	Coalescer	Regent monitoring	5	105
		No Loading	Loss of REEs	7	Loading pH Incorrect	3	PLC and SOPs	pH Prob	1	21
		Concentration of Radionuclides	Health issues	10	Saponification not effectively removing	3	Correct operation of the saponification circuit	Assay	3	90
		Poor performance	Project Failure	7	System not characterized well	7	Additional testing in detailed engineering phase 2	Test work	5	245

Table 3-110: Risk Analysis Related to the Cleaning and Saponification SX Processes

Process Number	Process Name/ Description	Potential Failure Mode	Potential Effect(s) of Failure	Severity	Potential Cause of Failures	Occurrence	Current Process Controls Prevention	Current Process Controls Detection	Detection	R.P.N.
5	Cleaning SX	Contamination	Not able to meet specifications	7	System not characterized well	7	Additional testing in detailed engineering phase 2	Test work	5	245
		Poor performance	Project Failure	7	System not characterized well	7	Additional testing in detailed engineering phase 2	Test work	5	245
6	Saponi-fication SX	Insufficient removal of elements	Poisoning	7	Saponification not effectively removing	3	Correct operation of the saponification circuit	Assay	3	63
		Poor performance	Project Failure	7	System not characterized well	7	Additional testing in detailed engineering phase 2	Test work	5	245

Table 3-111: Risk Analysis Related to the Refining SX Process

Process Number	Process Name/ Description	Potential Failure Mode	Potential Effect(s) of Failure	Severity	Potential Cause of Failures	Occurrence	Current Process Controls Prevention	Current Process Controls Detection	Detection	R.P.N.
7	Refining SX	Excessive Reagent Consumption	Project Failure	7	Poor design	7	Additional testing in detailed engineering phase 2	Test work	5	245
		Poor Design	Project Failure	7	System not characterized well	7	Additional testing in detailed engineering phase 2	Test work	5	245
		Poor performance	Project Failure	7	System not characterized well	7	Additional testing in detailed engineering phase 2	Test work	5	245

Table 3-112: Risk Analysis Related to the Process Waste Handling

Process Number	Process Name/ Description	Potential Failure Mode	Potential Effect(s) of Failure	Severity	Potential Cause of Failures	Occurrence	Current Process Controls Prevention	Current Process Controls Detection	Detection	R.P.N.
8	Waste	Solid waste continues to leach	Health	10	Unreacted constituents	5	Characterization work	TCLP test	3	150
		Wastewater contaminated	Environment	10	Water not treated properly	3	Wastewater monitoring	Assay/ 3rd party verification	3	90

Table 3-113: Risk Analysis Related to the Regulatory Matters

Process Number	Process Name/ Description	Potential Failure Mode	Potential Effect(s) of Failure	Severity	Potential Cause of Failures	Occurrence	Current Process Controls Prevention	Current Process Controls Detection	Detection	R.P.N.
9	Regulatory	Air emission do not meet specification	Monetary fines and/or project shut down	7	Design of air handling not sufficient	5	Operational testing	Emissions testing	5	175
		Surface and storm water runoff do not meet discharge and/or water quality standards	Monetary fines and/or project shut down	5	Design not meeting regulatory specification	5	Detailed run off water engineering	Water monitoring, treatment, and discharge management	7	175
		Treated process water discharges do not meet standards	Monetary fines and/or project shut down	7	Design not meeting regulatory specification	5	Detailed wastewater engineering	Further testing	7	245
		Solid tailings do not meet standards	Monetary fines and/or project shut down	7	Not permitted to dispose of tailings	5	Permitting process	Work with regulators	7	245
		Permits are not secured or renewed in a timely manner	Project delay	7	Failure to start on time	7	Permitting process	Work with regulators	8	392

3.7.3 Summary of Significant Concerns

The following items of particular concern appear to be present, given the resultant RPN numbers generated by the risk and fatal flaw analysis.

3.7.3.1 Timing of Regulatory Permits, Agreements or Contracts

The most significant risk facing the project was determined to be the rate at which the permits could be filed, negotiated and granted. In that regard securing the appropriate environmental protection permits is expected to require extensive base-line surveys, planning, document processing, and regulatory agency(ies) evaluation(s), as discussed in other chapters of this report. The critical nature of these permits cannot be overstated as consultation with the appropriate regulators reveals that no phase of the project construction will be allowed prior to appropriate regulatory approvals.

3.7.3.2 Performance of the Proposed Solvent Extraction Process

The next most severe potential fatal flaw relates to the risks associated with the limited testing of the proposed SX process. This abbreviated testing of the SX process was an artifact of the protracted and evaluation of various sources of coal preparation plant related feedstocks during the preliminary phases of this project. The remaining schedule did not provide the project team sufficient time to adequately test the roughing, cleaning, saponification and recycling circuits of the proposed REE pilot plant. In many instances preliminary or best-available data was utilized for the design reported herein. These designs are preliminary and as expressed in the analysis will require further testing and validation to remove risk.

3.7.3.3 Roaster Performance with a Coal Preparation Plant Rejects Feedstocks

The other significant risk identified was the operation of the roaster. The roaster is a multiple stage rotary hearth design. This means that the internal rakes will cause material to cascade from one stage to the next. In a heated, oxidizing environment the risk of creating a coal dust explosion is not readily known. For this reason, additional study and consultation with experts is required prior to construction to lower the risk.

3.8 Acknowledgments

The following individuals and / or entities are acknowledged for their openness and willingness to share information that proved valuable to this undertaking:

Dr. Rick Q. Honaker Ph.D., Professor of Mining Engineering, University of Kentucky, College of Engineering, Department of Mining Engineering, Lexington, Kentucky, who took the lead in the research and development of the processing concepts addressed in this chapter. Dr. Honaker contributed to the narrative of this report and generally coordinated its assembly.

Joshua Werner, PhD, P.E., Mechanical Engineer, Assistant Professor, University of Kentucky, Department of Mining Engineering, Lexington, Kentucky who drafted major portions of the narrative to this chapter and who contributed his engineering expertise in the design as well as layout of the proposed REE processing facility.

Dr. Gerald H. Luttrell Ph.D., Massey Professor, Virginia Polytechnic Institute and State University, Department of Mining & Minerals Engineering, Blacksburg, Virginia, who contributed significantly to the critique of the system design, assisted in drafting the narrative to this chapter, and generated many of the schematics depicted in this chapter.

Aaron Noble, Ph.D., Associate Professor, Virginia Tech, Department of Mining and Minerals Engineering, Blacksburg, Virginia, who contributed his knowledge of process controls, hydro-met chemistry, and financial aspects of an REE concentration facility.

Steven A. Keim, Ph.D., P.E., Senior Mining Engineer and Senior Vice President of Marshall Miller & Associates, Inc., Blacksburg, Virginia, Office, principal investigator of project MMA 29956, who contributed his project management expertise in guiding the composition of this chapter and scheduled support staff to assure its professional standing.

Hans E. Naumann, Jr., P.E., Principal Engineer Marshall Miller & Associates, Inc., Lexington, Kentucky, Office, who managed the overall coordination of the tasks associated with the reporting of the results of MMA 29956 and who contributed his editorial and formatting expertise to this chapter.

3.9 References

Brown, C. G., & Sherrington, L. G. (1979). Solvent extraction used in industrial separation of rare earths. Journal of Chemical Technology and Biotechnology, 29(4), 193-209.

Eyring, L., Gschneidner, K. A., & Lander, G. H. (2002). Handbook on the physics and chemistry of rare earths (Vol. 32): Elsevier.

Quinn, J., Soldenhoff, K., & Stevens, G. (2014). Separation of rare earth elements using Cyanex 572. Paper presented at the International solvent extraction conference ISEC.

Tong, H., Wang, Y., Liao, W., & Li, D. (2013). Synergistic extraction of Ce (IV) and Th (IV) with mixtures of Cyanex 923 and organophosphorus acids in sulfuric acid media. Separation and Purification Technology, 118, 487-491.

Wang, Y., Huang, C., Li, F., Dong, Y., & Sun, X. (2017). Process for the separation of thorium and rare earth elements from radioactive waste residues using Cyanex® 572 as a new extractant. Hydrometallurgy, 169, 158-164.

Xie, F., Zhang, T. A., Dreisinger, D., & Doyle, F. (2014). A critical review on solvent extraction of rare earths from aqueous solutions. Minerals Engineering, 56, 10-28.

Chapter 4.0

TECHNO-ECONOMIC ANALYSIS

Chapter 4.0
Techno-Economic Analysis

TABLE OF CONTENTS

4.1 General Approach to the Techno-Economic Analysis 1

4.2 Technical Review 3

4.2.1 Summary of All Major Experimental Data, Engineering Analysis, Computations and Test Results (Fire Clay Seam (Hazard No. 4) Rejects) 3

4.2.2 Synopsis of Individual and Combined Capabilities of Unit Operations Regarding Yield, Purity, Throughput 6

4.2.3 Review of Scale-Up Procedures and Reliability of Estimation Procedures 6

4.2.4 Listing of All Process Assumptions, Design Deficiencies and Deleterious Elements that Could Have Impact on Production 8

4.2.4.1 Technical Assumptions Regarding Process Design 8

4.2.4.2 Target Marketable Oxides of Sc, Dy, Gd, and Mixed Rare Earths 8

4.2.4.3 Economic Assumptions...8

4.2.5 Fatal Flaw Analysis... 10

4.3 Economic Evaluation 10

4.3.1 Tabular Summary of +10 percent to 15 percent CapEx for Pilot-Scale and Full-Scale Commercial Installation of Circuitry/Ancillary Operations................ 10

4.3.1.1 Pilot-Scale Installation...10

4.3.1.2 Commercial-Scale Installation...23

4.3.2 Tabular Listing of all Expected Operational and Maintenance Costs Including Labor, Electric, Reagents, and Other Consumables 26

4.3.2.1 Labor Costs 27

4.3.2.2 Miscellaneous Fixed Costs...27

4.3.2.3 Electric Power Costs...28

4.3.2.4 Reagent Costs..28

4.3.2.5 Other Direct Costs 29

4.3.2.6 Overhead Costs...29

4.3.2.7 Operating Cost Summary 29

4.3.3 Cost/Benefit Analysis that Highlights Annual Production Projections, Cash Flow Forecasts and Other Key Economic Performance Indicators (IRR, NPV, Payback).. 30

4.3.4 Sensitivity Analysis on Pricing, Feedstock Quality, Product Purity, CapEx, Open, Transportation Costs, etc.. 32

4.3.4.1 Analysis of Feed Rate 32

4.3.4.2 Analysis of Feed Grade and REE Price 33

4.3.4.3 Analysis of Technical Performance Measures...34

4.3.4.4 Potential Pathway to Profitability 35
4.4 Key Findings 36
4.5 Acknowledgments 37
4.6 Special Note and Addendum to the Techno-Economic Analysis 38
4.6.1 Justification of the 357 ppmw Feed Grade of the Economic Model 38
4.6.2 Influence of Optimal Roasting Conditions 39

TABLES (IN THE TEXT)

Table 4-1: Worksheets Descriptions for REE-Econ Software 2
Table 4-2: Listing of Process Modules Used in Techno-Economic Analysis 2
Table 4-3: Listing of Accounting Codes Used in Techno-Economic Analysis 3
Table 4-4: Feedstock REE Concentration (Nominal) Used in Techno-Economic Analysis 4
Table 4-5: REE to REO Conversion Factors Used in Techno-Economic Analysis 5
Table 4-6: Overall Element-by-Element Recovery Values Used in Techno-Economic Analysis 5
Table 4-7: Recovery, Purity, and Throughput Data for Individual Unit Operations 6
Table 4-8: Overall Process Yield to Final Product Streams Used in Techno-Economic Analysis 6
Table 4-9: Scale-Up Criteria used for Commercial-Scale Cost Analysis 7
Table 4-10: Used Equipment Costs for Equipment Components in Process Section 1: Crushing and Grinding 11
Table 4-11: Shipping and Refurbishment Costs for Process Section 1: Crushing and Grinding 12
Table 4-12: Used Equipment Costs for Equipment Components in Process Section 2: Roasting 13
Table 4-13: Shipping and Refurbishment Costs for Process Section 2: Roasting 14
Table 4-14: Used Equipment Costs for Equipment Components in Process Section 3: Leaching 15
Table 4-15: Shipping and Refurbishment Costs for Process Section 3: Leaching 16
Table 4-16: Used Equipment Costs for Equipment Components in Process Section 4: Wastewater Treatment 17
Table 4-17: Shipping and Refurbishment Costs for Process Section 4: Wastewater Treatment 18
Table 4-18: Used Equipment Costs for Equipment Components in Process Section 5: Leaching Ancillary 19
Table 4-19: Shipping and Refurbishment Costs for Process Section 5: Leaching Ancillary 19
Table 4-20: Used Equipment Costs for Components in Process Section 6: SX Ancillary 20
Table 4-21: Shipping and Refurbishment Costs for Process Section 6: SX Ancillary 21
Table 4-22: Combined Equipment Costs for all Process Sections 21
Table 4-23: Combined Shipping and Refurbishment Costs for all Process Sections 22
Table 4-24: Quoted Construction Costs Associated with Disassembly of Existing 22
Table 4-25: Total Capital Cost Estimate for 2,000 lb./hr. Pilot-Scale Plant 23
Table 4-26: Capital Cost Model Parameters Used in Techno-Economic Analysis 24
Table 4-27: Lang Factor Estimation for Commercial-Scale Installation Costs 24
Table 4-28: Itemized List of Capital Costs for Commercial-Scale Plant (500 st/hr.) 25
Table 4-29: Module-Level Capital Costs for Commercial-Scale Plant (500 st/hr.) 26
Table 4-30: Labor Cost Estimate for Commercial-Scale Plant (500 st/hr.) 27
Table 4-31: Cost Basis for Miscellaneous Fixed Costs 27
Table 4-32: Cost Summary for Miscellaneous Fixed Costs at Commercial-Scale (500 st/hr.) 27
Table 4-33: Electric Power Cost Summary for Commercial-Scale Plant (500 st/hr.) 28

Table 4-34: Unit Reagent Costs 28
Table 4-35: Reagent Cost Summary for Commercial-Scale Plant (500 st/hr.) 29
Table 4-36: Waste Disposal Cost Summary for Commercial-Scale Plant (500 st/hr.) 29
Table 4-37: Operating Cost Summary by Process Module for Commercial-Scale Plant (500 st/hr.) 30
Table 4-38: Operating Cost Summary by Accounting Code for Commercial-Scale Plant (500 st/hr.) 30
Table 4-39: REE Price Deck Used for Commercial-Scale Economic Analysis 31
Table 4-40: Summary of Economic Indicators for Commercial-Scale Plant (500 st/hr.) 32
Table 4-41: Economic Indicators for Commercial Plant under Improved Input Conditions 36
Table 4-42: REE (ppmw) Data Source Inventory (Fire Clay Seam) 38
Table 4-43: Project Economic Summary for Material Processed under Differing Roasting and Acid Application Conditions 40

FIGURES (IN THE TEXT)

Figure 4-1: Sensitivity of Plant Capital Cost with Respect to Plant Feed Rate 33
Figure 4-2: Sensitivity of Plant Feed Grade and REE Price Multiple with Respect to Project NPV 33
Figure 4-3: Tornado Analysis Showing the Sensitivity of Plant Operating Cost with respect to Several Technical Performance Measures 34
Figure 4-4: Sensitivity of Reagent Consumption with Respect to this Project NPV 35

4.1 General Approach to the Techno-Economic Analysis

To assess the feasibility of recovering REEs from coal rejects, a detailed technical review and economic evaluation were conducted. The primary input data for this analysis consisted of experimental testing conducted at both the bench-scale and small pilot-scale. This experimental data has been extensively described in earlier chapters of this report. In addition to the experimental data, other inputs to the techno-economic analysis were derived from detailed flowsheet simulations and engineering designs, which have also been reported in earlier sections of this report. The actual cost analysis was conducted at both a large pilot-scale and a commercial-scale. The large pilot-scale utilizes a nominal capacity of 2,000 lb./hr. 1-short ton [st/hr.] or 0.91[metric-ton] mt/hr.), and these results were used as the basis for the engineering work documented throughout earlier chapters of this report. To assess the future commercial viability of the process technology, the analysis was also conducted at a commercial-scale with a nominal capacity of 500 st/hr. Results from both analyses are presented in this chapter.

To facilitate the technical and economic review, the engineering team used a spreadsheet-based economic assessment tool that has been developed under another **Department of Energy** ***(DOE)*** sponsored project entitled "*Pilot-Scale Testing of an Integrated Circuit for the Extraction of Rare Earth Minerals and Elements from Coal and Coal Byproducts Using Advanced Separation Technologies* (DE-FE0027035). One of the key tasks to be completed under this ongoing project involved the development of an economic evaluation tool that can be applied to process circuitry for rare earth element *(REE)* concentration from coal-based sources. This software, which is currently referred to as REE-Econ, was ideally suited for use in the current project. The latest working version of the REE-Econ software (Ver. 3.9), which includes integrated user instructions, is attached in the *Appendix* of this report.

The REE-Econ software integrates both technical factors (e.g., process feed rates, unit recovery values, reagent consumption rates, scale-up criteria, etc.) and economic factors (e.g., process capital costs, reagent costs, finance and tax structure, etc.), while outputting key economic indicators, such as project net present value *(NPV)*, internal rate of return *(IRR*), and capital payback period. Per DOE guidelines, the REE-Econ tool makes use of the Microsoft Excel Spreadsheet software. The software contains no locked or hidden cells; however, it does extensively use "named ranges" to carry values between sheets. Nevertheless, the "open" nature of the software ensures that all of the calculation steps can be readily traced by an experienced Excel user.

The software workflow is driven by various worksheet "tabs." For typical use, the flow is left to right across the worksheets and top to bottom within the worksheets. A listing and description of each worksheet tab is shown in the following table. Within each worksheet, the spreadsheet cells are delineated between a printable area, highlighted with a white background, and a nonprintable area denoted by a grey background. The printable area contains key process information and economic indicators, while the nonprintable area contains secondary calculations, explanatory text, and other tangential information.

Table 4-1: Worksheets Descriptions for REE-Econ Software

Worksheet Name	Brief Description
Cover Page	Contains model run metadata.
Economic	Specifies global economic assumptions.
Technical	Specifies process recovery and feed rates.
CapEx	Compiles total capital costs for process modules.
OpEx	Compiles total operating costs for process modules.
Revenue	Estimates revenue from various product streams.
Cash Flow	Shows pro forma cash flow based on revenue and costs.
PMx	Contains design and costing data for each process module.
Global Constant	Contains important model parameters.

In the REE-Econ software, the costs are itemized according to "process modules." In this respect, a process module is a group of similar units that can be described by a common feed rate and an overall process objective. Each process module has a unique worksheet tab (denoted as PM1, PM2, PM3, etc.) that contains all of the pertinent design and costing data. A complete listing of the process modules used for this project is shown in the following table. Please note that the delineation used in this techno-economic analysis contains finer resolution than other circuit module classifications used in Chapter 3. As an example, for the techno-economic analysis presented here, crushing and grinding are denoted by two separate process modules (PM1 and PM2), while they are described as a single process circuit in Chapter 3 (i.e., Circuit 1, Feed Preparation). This slight discrepancy is justified as the higher number of process modules employed in the techno-economic analysis allows a more nuanced analysis of process cost drivers.

Table 4-2: Listing of Process Modules Used in Techno-Economic Analysis

Module ID	Process	Feed Type	Terminator
PM1	Crushing and Screening	Solid, Dilute	Input
PM2	Dry Grinding	Solid, Dilute	--
PM3	Roasting	Solid, Dilute	--
PM4	Leaching	Solid, Dilute	--
PM5	Rougher SX	Aqueous	--
PM6	Cleaner SX	Aqueous	--
PM7	SX Wash/Saponification	Aqueous	Output
PM8	REE Precipitation	Aqueous	--
PM9	Water Treatment	Aqueous	--
PM10	REO Refining	Solid, Concentrate	Output

The REE-Econ software traces costs according to both process module and a functional accounting code. A list of accounting codes is shown in the following table.

Table 4-3: Listing of Accounting Codes Used in Techno-Economic Analysis

Code	Description
OL	Operating Labor
TL	Technical Labor
PR	Power/Utilities
LS	Tolling/Lease Agreements
CR	Consumables/Reagents
WD	Waste Disposal
OH	Plant Overhead
GSA	General Sales, and Administration

4.2 Technical Review

4.2.1 Summary of All Major Experimental Data, Engineering Analysis, Computations and Test Results (Fire Clay Seam (Hazard No. 4) Rejects)

The following economic analysis is based on tests performed on roasted[1] Fire Clay Seam (Hazard No. 4) rejects at a 0.25 ton per hour proof-of-concept pilot plant located in West Kentucky and operated by the University of Kentucky. Prior to conducting the techno-economic analysis, the experimental data was reviewed and analyzed by the engineering team. This technical review ensured that all model inputs were based on reliable and reproducible experimental results. The results of this technical analysis were integrated into the flowsheet simulations conducted using the REESim™ software. This data has been extensively reported in Chapter 3. The results pertinent to the techno-economic analysis, including REE feed concentrations, overall REE recovery values and element to oxide conversion factors, are shown in the following tables.

---

[1] Personal Communication with Dr. Honaker, Ph.D. The Nex-Gen roasting facilities located in the Coal Fields Industrial Park near Chavies, Kentucky, was used in this process. At that stage of the project, the required control of roasting temperature and roasting atmosphere had not yet been determined. Evidence now shows that this sample had been overheated. Leaching results from this roasted sample were subsequently significantly less than optimal.

Table 4-4: Feedstock REE Concentration (Nominal) Used in Techno-Economic Analysis[2]

REE	Feed Conc. (ppm)	Distribution (%)
Scandium (Sc)	16.25	5.40
Yttrium (Y)	33.45	9.31
Lanthanum (La)	62.37	16.26
Cerium (Ce)	132.50	34.56
Praseodymium (Pr)	15.36	6.50
Neodymium (Nd)	59.19	15.33
Samarium (Sm)	10.29	3.53
Europium (Eu)	1.63	0.46
Gadolinium (Gd)	8.44	2.84
Terbium (Tb)	1.20	0.25
Dysprosium (Dy)	6.84	1.73
Holmium (Ho)	1.26	0.50
Erbium (Er)	3.61	1.55
Thulium (Tm)	0.51	0.35
Ytterbium (Yb)	3.28	1.13
Lutetium (Lu)	0.48	0.30
Total:	**356.64**	**100.00**

Note: Based on coarse preparation plant rejects sample KRP_RE10-18 collected on 11-16-2018

---

[2] The raw feed parameters are based on a coarse rejects sample collected from the Leatherwood Plant on 11-16-2018 (Sample ID: KRP_RE10-18). The analytical information from this sample (as well as others collected form this plant) is located in the RESEARCHER DATABASE.

Table 4-5: REE to REO Conversion Factors Used in Techno-Economic Analysis

Compound	% REE in Pure REO	Source
Sc_2O_3	65.2%	[1]
Y_2O_3	78.7%	[2]
La_2O_3	85.3%	[2]
CeO_2	81.4%	[2]
Pr_6O_{11}	82.8%	[2]
Nd_2O_3	85.7%	[2]
Sm_2O_3	86.2%	[2]
Eu_2O_3	86.4%	[2]
Gd_2O_3	86.8%	[2]
Tb_4O_7	85.0%	[2]
Dy_2O_3	87.1%	[2]
Ho_2O_3	87.3%	[1]
Er_2O_3	87.5%	[2]
Tm_2O_3	87.6%	[1]
Yb_2O_3	87.8%	[2]
Lu_2O_3	87.9%	[1]

Sources:

1) *https://www.sigmaaldrich.com*
2) *https://www.bluelinecorp.com/re-calculator*

Table 4-6: Overall Element-by-Element Recovery Values Used in Techno-Economic Analysis

REE	Recovery to Various Product Streams (%)				
	Sc_2O_3	Dy_2O_3	Gd_2O_3	Mixed REO	Losses
Scandium (Sc)	14.3%	0.0%	0.0%	0.5%	85.2%
Yttrium (Y)	0.0%	0.0%	0.0%	11.1%	88.9%
Lanthanum (La)	0.0%	0.0%	0.0%	16.4%	83.6%
Cerium (Ce)	0.0%	0.0%	0.0%	20.0%	80.0%
Praseodymium (Pr)	0.0%	0.0%	0.0%	18.5%	81.5%
Neodymium (Nd)	0.0%	0.0%	0.0%	21.8%	78.2%
Samarium (Sm)	0.0%	0.0%	0.0%	17.2%	82.8%
Europium (Eu)	0.0%	0.0%	0.0%	9.2%	90.8%
Gadolinium (Gd)	0.0%	0.0%	10.1%	0.0%	89.9%
Terbium (Tb)	0.0%	0.0%	0.0%	49.4%	50.6%
Dysprosium (Dy)	0.0%	9.6%	0.0%	0.0%	90.4%
Holmium (Ho)	0.0%	0.0%	0.0%	0.0%	100.0%
Erbium (Er)	0.0%	0.0%	0.0%	0.0%	100.0%
Thulium (Tm)	0.0%	0.0%	0.0%	19.5%	80.5%
Ytterbium (Yb)	0.0%	0.0%	0.0%	8.7%	91.3%
Lutetium (Lu)	0.0%	0.0%	0.0%	16.7%	83.3%
Product Purity (%)					
Oxide Basis	95.0%	99.0%	99.0%	--	--
REE Basis	61.9%	86.2%	85.9%	81.0%	--

4.2.2 Synopsis of Individual and Combined Capabilities of Unit Operations Regarding Yield, Purity, Throughput

Using the design data described in Chapter 3, the individual capabilities of each unit operation were determined with respect to REE recovery, incremental and final product purity, and pilot-scale throughput. These values are summarized for the primary unit operations in the following table. In addition, the overall yield to various final waste and product streams is also shown in a separate table.

Table 4-7: Recovery, Purity, and Throughput Data for Individual Unit Operations

Unit Operation	Nominal REE Recovery (%)	Product Purity		Pilot-Scale Throughput	Commercial-Scale Throughput	
Crushing Circuit	100%	357	ppm	1,000 kg/hr.	500	TPH
Grinding Circuit	100%	357	ppm	1,000 kg/hr.	500	TPH
Roaster	100%	ppm	ppm	1,000 kg/hr.	500	TPH
Leaching Circuit	24%	37	mg/L	990 kg/hr.	495	TPH
SX Rougher	75%	694	mg/L	1,800 L/hr.	23,131	LPM
SX Cleaner	95%	2,211	mg/L	4.4 L/hr.	925	LPM
REO Precipitation	99%	81	% REE	2.7 L/hr.	231	LPM
REE Refining	100%	> 95	% REO	0.04 kg/hr.	37.44	kg/hr.
Wastewater Treatment	NA	NA		2,530 L/hr.	20,818	LPM

Table 4-8: Overall Process Yield to Final Product Streams Used in Techno-Economic Analysis

Output Stream	Overall Yield (%)
Non-Hazardous Solid Waste	92.8%
Non-Hazardous Precipitate	6.1%
Dust/Volatiles	1.0%
REE Products	<1%

Mass yield values are a ratio of the product mass and the total feed mass entering the system. Since the REE content in the coal sources is significantly less than 1% of the total mass, the mass yield to the product stream would also be less than 1% if 100% of the REEs were recovered and a high purity product produced.

4.2.3 Review of Scale-Up Procedures and Reliability of Estimation Procedures

As described in Chapter 3, the capacity of the unit operations in the pilot-scale extraction facility have been determined by a detailed engineering analysis, which includes laboratory and small pilot data collection, as well as comprehensive flowsheet simulations. Given the level of detail in the data and the engineering design, the capital cost estimates for this operational scale have been determined by direct quotes[3] for many of the equipment items, including: (1) base equipment price for new and used components; (2) disassembly, transport, and reassembly of used equipment; (3) rehabilitation and refurbishment of used equipment; and

---

[3] The cost of constructing the refining circuit was not directly quoted, but instead based on estimates from prior experience and references.

(4) other construction costs as needed. As such, this capital cost estimate represents a Class 2 estimate (by AACE guidelines) with an expected accuracy of +10 percent to 15 percent.

To extend this cost analysis to the commercial-scale, a scale-factor analysis was conducted using standard scale-up criteria on a unit-by-unit basis. Since this analysis lacked the detailed engineering design used in the pilot-scale study, this capital cost estimate represents a Class 4 estimate with an expected accuracy of +50 percent to 30 percent.

For the commercial-scale factor analysis, the unit capacity for each unit operation was first determined by a fundamental assessment of the processing needs. Not all unit operation costs scale by simple feed throughput, so a detailed unit-by-unit appraisal was required. For example, the capital cost of roasting operations tends to scale exponentially with respect to heat input in terms of million BTUs/hour (MBtu/hr.), while the capital cost of grinding mills scale is related to power (expressed as horsepower *(HP)* draw). The engineering team consulted several standard cost engineering reference guides, including the Mine and Mill Estimator's Guide, Matches.com, Gentry and O'Neil (1986), and Garett, (1989), to determine the relevant scaling criteria for each equipment type. Next, a standard scaling rule was used to determine the value of the capacity parameter for each unit operation. For example, the power draw for ball mills was determined by using a constant Bond Work Index with a known feed and product size. These scaling parameters were then used to calculate the power draw required at the commercial feed rate. Once the power draw was determined, an exponential capital cost model was used to determine the final cost. A similar procedure was used for each major equipment item. Details on the scale-up criteria and unit capacity measures are shown in the following table. Further details on the specific capital cost models are included later in this chapter under the economic analysis.

Table 4-9: Scale-Up Criteria used for Commercial-Scale Cost Analysis

Process Operation	Cost Model Input	Scale-Up Criteria
Loaders	Number of units	Constant TPH per unit (unit size fixed)
Crushers	Power Draw [HP]	Constant HP per TPH
Screens	Screen area [ft^2]	Constant TPH/ft^2 screen area for a given screen opening size
Conveyors	Throughput [TPH]	Simple throughput
Storage Bins	Storage capacity [tons]	Percent of Hourly production surge capacity
Mills	Power Draw [HP]	Constant Bond Work Index
Storage Tanks	Storage capacity [gallons]	Constant Storage time/retention time
Mixers	Power Draw [HP]	Constant Power Intensity (HP/1000 gal.)
Pumps	Feed rate [gpm]	Simple throughput
Cyclones	Feed rate [gpm]	Simple throughput
Thickeners	Thickener area [ft^2]	Constant unit capacity (ft^2/t/day)
Filter Presses	Filter volume [ft^3]	Constant cycle time
Dryers	Water evaporation rate [lb./hr. H_2O]	Constant moisture reduction
Roasters	Heat Input [MBtu/hr.]	Constant heat requirement per unit feed
Scrubbers	Gas rate [cfm]	Constant airflow per TPH
Spray Quenchers	Gas rate [cfm]	Constant airflow per TPH
Chillers	Heat Input [MBtu/hr.]	Constant heat requirement
Leach Tanks	Volume [gal]	Constant Retention Time
Solution Heaters	Heat Input [MBtu/hr.]	Constant heat requirement
Mixer Settlers	Volume [gal]	Constant Retention Time

4.2.4 Listing of All Process Assumptions, Design Deficiencies and Deleterious Elements that Could Have Impact on Production

4.2.4.1 Technical Assumptions Regarding Process Design

Several assumptions were required in order to develop the various flowsheets shown in the earlier chapters of this report. The experimental input data used to create each set of simulations was obtained directly from various project participants and represents a combination of both laboratory and pilot-scale test results. As such, no attempt was made to evaluate the consistency of the data sets in terms of unit-to-unit flows of masses (solid or dissolved) or volumes. Therefore, the personnel working to complete the simulation runs could not validate the reliability of all experimental data sets used as input data.

Another key assumption is that the elemental mass recovery values for each element of interest remained unchanged when recycle streams were iteratively recycled back through the various separating units. While the experimental test program did include some pilot-scale continuous flow runs, some unit operations were only evaluated at the laboratory scale. In these cases, continuous steady-state flow data could not be obtained. This assumption simplified the simulation runs so that each unit operation is defined by a simple partition factor representing the recovery of a particular component to a particular product steam. The use of partition factors for flowsheet simulation, which is common, has the advantage of ensuring that a consistent set of mass balanced data is available when completed.

Finally, it should be noted that many of the design decisions were impacted by the scope and funding constraints imposed by DOE and the other participants. These additional limitations influenced the scale of operation, selection of final equipment, ultimate layout of the pilot-scale facility and the projected duration of the test program. As such, these particular factors were outside the control of the design team used to prepare the design outlined in this report.

4.2.4.2 Target Marketable Oxides of Sc, Dy, Gd, and Mixed Rare Earths

In this analysis, the commercial plant was assumed to produce four REE products:

- Scandium oxide (Sc_2O_3)
- Dysprosium oxide (Dy_2O_3)
- Gadolinium oxide (Gd_2O_3)
- Mixed Rare Earth Oxides (MREO)

As indicated in this list, other products were assumed to be sold as a bulk mixed rare earth oxide (*MREO*).

4.2.4.3 Economic Assumptions

Several economic assumptions were also applied to facilitate the commercial-scale techno-economic analysis. These assumptions are listed below. Many of the items, including those regarding the financing structure, escalation rates, tax calculations, and operating period have been supplied by **National Energy Technology Laboratory *(NETL)*** in the *Guidance for Development of Techno-Economic Analyses* document.

- All amounts are in US dollars.
- The total operational period for the plant is 20 years.
- The plant feed rate is fixed at 500 short TPH (st/hr.), with a feedstock concentration of 357 parts per million (ppm) REE on a whole sample basis.
- Inflation has been applied to sales revenue and operating costs using a fixed rate of 3 percent.

> Capital costs are spread over a period not to exceed three years, and the allocation between those three years is 10 percent, 60 percent, and 30 percent for years one through three, respectively. Thus, the total analysis period (capital purchase plus operating) is not to exceed 23 years.

> During the capital expenditure period, capital costs escalate at a constant rate of 3.6 percent.

> The project is debt financed for 50 percent of the total overnight[4] capital requirement. The remaining 50 percent is financed by equity.

> The debt repayment terms include: 6 percent interest rate, 10-year loan period, and no grace period on debt repayment. The repayment uses a standard amortization schedule with constant payments throughout the payoff period.

> Working capital is not included in this estimate and will instead be borne by the operating entity at no cost to the project.

> The combined federal and local tax rate is fixed at 26 percent. This value is lower than the one required in the NETL guidance document; however, it reflects a recent reduction in corporate tax rates.

> All capital is depreciable, using a 150 percent declining balance depreciation schedule over 20 years. The depreciation method was NOT changed to straight line when conditions favored the switch.

> The mineral depletion rate for REEs is 14 percent. Depletion is charged at the appropriate rate times the net sales revenue after deducting royalties and any severance tax, provided that the total amount calculated by depletion rates does not exceed 50 percent of the taxable income before depletion.

> The plant is part of a larger entity with sufficient revenue to offset negative taxable income. Thus, losses are not carried forward and are instead calculated as a “negative tax” that indicates the reduction in tax burden required for overall entity.

> The land and mineral rights are leased, and royalties will be paid to the landowner.

> Royalties are charged at standard rate of 6.5 percent of total sales revenue.

> All process modules are located at the same facility with no significant transportation cost between modules.

> The plant operation schedule is fixed at eight (8) hours per shift, three (3) shifts per day, 336 operational days per year.

> Equipment costs were determined by scale-factor analysis. Installation costs were determined by an overall Lang[5] factor.

> Reagents, consumables, power, and labor were determined by itemized cost analysis.

---

[4] The overnight market is the component of the money market involving the shortest term loan. Lenders agree to lend borrowers funds only "overnight" i.e. the borrower must repay the borrowed funds plus interest at the start of business the next day. Given the short period of the loan, the interest rate charged in the overnight market, known as the overnight rate is, generally speaking, the lowest rate at which banks lend money.

[5] The Lang Factor is an estimated ratio of the total cost of creating a process within a plant, to the cost of all major technical components. It is widely used in industrial engineering to calculate the capital and operating costs of a plant. The factors were introduced by H. J. Lang and Dr Michael Bird in *Chemical Engineering* magazine in 1947 as a method for estimating the total installation cost for plants and equipment.

- Other operating costs, including maintenance and supply, QA/QC, sales, admin, property tax, insurance, and plant overhead were all calculated using proportional factors.
- The rare earth price deck is primarily based on a standard price deck provided by NETL. Elements not included in this price deck were derived from Asian Metal, Inc.[6]

The REE sales price was based on a 100 percent price realization for the purified oxide products highlighted in this project:

- Scandium oxide (Sc_2O_3)
- Dysprosium oxide (Dy_2O_3)
- Gadolinium oxide (Gd_2O_3)

All other products were assumed to be sold as a bulk MREO at a 65 percent price realization[7].

4.2.5 Fatal Flaw Analysis

Chapter 3 – "System Design Package Preparation" addresses this subject, and the reader is directed to that chapter for a description of the analysis process, its results and the investigative teams ensuing recommendations.

4.3 Economic Evaluation

4.3.1 Tabular Summary of +10 percent to 15 percent CapEx for Pilot-Scale and Full-Scale Commercial Installation of Circuitry/Ancillary Operations

4.3.1.1 Pilot-Scale Installation

A detailed engineering design was conducted to determine the equipment items needed for the pilot-scale system installation. To minimize the capital investment needed for this operation, used equipment was considered for many of the operations. The following tables contain detailed costing records for the various used equipment components as well as estimated costs for shipping, unloading and refurbishment. The component numbers in this section represent the original plant IDs for the used equipment. These loosely correlate to the process modules described above; however, not all equipment items can be conveniently classified into a single process module. As such, the original section numbering is retained.

---

[6] Asian Metal, Inc., No. 6 Jianguomenwai Avenue, Suite 16B, Tower C, SK Plaza, Beijing, 100022, China, http://www.asianmetal.com/RareEarthsPrice/RareEarths.html

[7] Actual results may differ substantially from this estimate.

Table 4-10: Used Equipment Costs for Equipment Components in Process Section 1: Crushing and Grinding

Section Number	Name	Cost (1975)	Cost (2019)	Weight (lbs.)
1	Storage Bin (B-1.1)	$ 3,976.00	$ 12,446.70	5,500
1	Storage Bin (B-1.2) (Storage Bin for Finished Ground Ore)	$ 5,706.00	$ 17,862.39	7,600
1	Vibrating Bin Activator (BA-1.1)	$ 3,187.00	$ 9,976.77	875
1	Vibrating Bin Activator (BA-1.2)	$ 4,882.00	$ 15,282.89	1,000
1	Jaw Crusher Discharge Belt Conveyor (BC-1.1)	$ 5,885.00	$ 18,422.74	4,310
1	Tail Pulley Conveyor (BC-1.2)	$ 6,339.00	$ 19,843.97	4,310
1	18' Belt Conveyor (BC-1.3)	$ 5,849.00	$ 18,310.04	4,310
1	Bag House (BH-1.1)	$ 9,335.00	$ 29,222.82	4,900
1	Bucket Elevator (BL-1.1)	$ 4,787.00	$ 14,985.50	2,750
1	Bucket Elevator (BL-1.2)	$ 4,787.00	$ 14,985.50	2,750
1	(Conical) Ball Mill (BM-1.1)	$ 11,500.00	$ 36,000.26	14,000
1	(Dust) Cyclone (CY-1.1)	$ 1,150.00	$ 3,600.03	340
1	(Rotary) Dryer (D-1.1)	$ 29,500.00	$ 92,348.48	25,400
1	Jaw Crusher Belt Conveyor Feeder (FD-1.1)	$ 4,482.00	$ 14,030.71	1,900
1	Storage Bin Discharge Weight Belt Conveyor (FD-1.2)	$ 6,114.00	$ 19,139.61	400
1	(Cyclone Discharge) Rotary Feeder (FD-1.3)	$ 1,296.00	$ 4,057.07	120
1	Baghouse Discharge Rotary Feeder (FD-1.4)	$ 1,296.00	$ 4,057.07	120
1	Jaw Crusher (JC-1.1)	$ 3,400.00	$ 10,643.55	6,200
1	Vibrating Conveyor (PC-1.1)	$ 1,903.00	$ 5,957.26	600
1	Roll Crusher Discharge. Pan Conveyor (PC-1.1)	$ 1,902.00	$ 5,954.13	600
1	12" Vibrating Conveyor (Bin Discharge - Pan Conveyor) (PC-1.2) *		$ -	600
1	Roll Crusher Smooth Face (RC-1.1)	$ 7,700.00	$ 24,104.52	4,500
1	Roll Crusher (1.1) *		$ -	4,500
1	(Ball Mill) Screw Conveyor (SC-1.2)	$ 1,830.00	$ 5,728.74	500
1	(Bin Discharge) Screw Conveyor (SC-1.3)	$ 1,497.00	$ 4,686.29	300
1	(Bin Discharge) Screw Conveyor (SC-1.4) *		$ -	300
1	Bag House Discharge Trough Type Screw Conveyor (SC-1.5)	$ 1,513.00	$ 4,736.38	330
1	Cyclone Discharge Trough Type Screw Conveyor (SC-1.6)	$ 1,547.00	$ 4,842.82	250
1	Vibrating Screen (SN-1.1)	$ 2,134.00	$ 6,680.40	675
1	Vibrating Screen (Double Deck) (SN-1.2)	$ 5,692.00	$ 17,818.56	1,125
1	Tails Slurry Hold Tank	$ 1,998.00	$ 6,254.65	500
	TOTAL:	**$141,187.00**	**$441,979.85**	**101,815**

Note:

* Blank cells in the "cost" column denote items that are an assumed to be a component of an "assembly" that were in fact priced as an "assembly" in the source documents collected by the project investigators. However, the individual cost of each component was not available, and the investigators were not provided the means to extract those costs. On the other hand, although individual costs could not be determined, the weights of the components were listed separately. Thus, the weights could be reported, and these are shown in the table for greater accuracy of this report.

Table 4-11: Shipping and Refurbishment Costs for Process Section 1: Crushing and Grinding

Name	Freight Cost @ $0.32/lb.	Cost of Labor @ $0.015/lb.	Unloading Crane Cost @ $0.015/lb.	Unloading Forklift Cost @ $0.01/lb.	Unloading Truck Cost @ $0.01/lb.	Equipment Refurbishment
Storage Bin (B-1.1)	$ 1,760.00	$ 82.50	$ 82.50	$ 55.00	$ 55.00	$ 7,952.00
Storage Bin (B-1.2) (Storage Bin for Finished Ground Ore)	$ 2,432.00	$ 114.00	$ 114.00	$ 76.00	$ 76.00	$ 11,412.00
Vibrating Bin Activator (BA-1.1)	$ 280.00	$ 13.13	$ 13.13	$ 8.75	$ 8.75	$ 6,374.00
Vibrating Bin Activator (BA-1.2)	$ 320.00	$ 15.00	$ 15.00	$ 10.00	$ 10.00	$ 9,764.00
Jaw Crusher Discharge Belt Conveyor (BC-1.1)	$ 1,379.20	$ 64.65	$ 64.65	$ 43.10	$ 43.10	$ 11,770.00
Tail Pulley Conveyor (BC-1.2)	$ 1,379.20	$ 64.65	$ 64.65	$ 43.10	$ 43.10	$ 12,678.00
18' Belt Conveyor (BC-1.3)	$ 1,379.20	$ 64.65	$ 64.65	$ 43.10	$ 43.10	$ 11,698.00
Bag House (BH-1.1)	$ 1,568.00	$ 73.50	$ 73.50	$ 49.00	$ 49.00	$ 18,670.00
Bucket Elevator (BL-1.1)	$ 880.00	$ 41.25	$ 41.25	$ 27.50	$ 27.50	$ 9,574.00
Bucket Elevator (BL-1.2)	$ 880.00	$ 41.25	$ 41.25	$ 27.50	$ 27.50	$ 9,574.00
(Conical) Ball Mill (BM-1.1)	$ 4,480.00	$ 210.00	$ 210.00	$ 140.00	$ 140.00	$ 23,000.00
(Dust) Cyclone (CY-1.1)	$ 108.80	$ 5.10	$ 5.10	$ 3.40	$ 3.40	$ 2,300.00
(Rotary) Dryer (D-1.1)	$ 8,128.00	$ 381.00	$ 381.00	$ 254.00	$ 254.00	$ 59,000.00
Jaw Crusher Belt Conveyor Feeder (FD-1.1)	$ 608.00	$ 28.50	$ 28.50	$ 19.00	$ 19.00	$ 8,964.00
Storage Bin Discharge Weight Belt Conveyor (FD-1.2)	$ 128.00	$ 6.00	$ 6.00	$ 4.00	$ 4.00	$ 12,228.00
(Cyclone Discharge) Rotary Feeder (FD-1.3)	$ 38.40	$ 1.80	$ 1.80	$ 1.20	$ 1.20	$ 2,592.00
Baghouse Discharge Rotary Feeder (FD-1.4)	$ 38.40	$ 1.80	$ 1.80	$ 1.20	$ 1.20	$ 2,592.00
Jaw Crusher (JC-1.1)	$ 1,984.00	$ 93.00	$ 93.00	$ 62.00	$ 62.00	$ 6,800.00
Vibrating Conveyor (PC-1.1)	$ 192.00	$ 9.00	$ 9.00	$ 6.00	$ 6.00	$ 3,806.00
Roll Crusher Discharge Pan Conveyor (PC-1.1)	$ 192.00	$ 9.00	$ 9.00	$ 6.00	$ 6.00	$ 3,804.00
12" Vibrating Conveyor (Bin Discharge - Pan Conveyor) (PC-1.2) *	$ 192.00	$ 9.00	$ 9.00	$ 6.00	$ 6.00	$ -
Roll Crusher Smooth Face (RC-1.1)	$ 1,440.00	$ 67.50	$ 67.50	$ 45.00	$ 45.00	$ 15,400.00
Roll Crusher (1.1)	$ 1,440.00	$ 67.50	$ 67.50	$ 45.00	$ 45.00	$ -
(Ball Mill) Screw Conveyor (SC-1.2)	$ 160.00	$ 7.50	$ 7.50	$ 5.00	$ 5.00	$ 3,660.00
(Bin Discharge) Screw Conveyor (SC-1.3)	$ 96.00	$ 4.50	$ 4.50	$ 3.00	$ 3.00	$ 2,994.00
(Bin Discharge) Screw Conveyor (SC-1.4)	$ 96.00	$ 4.50	$ 4.50	$ 3.00	$ 3.00	$ -
Bag House Discharge Trough Type Screw Conveyor (SC-1.5)	$ 105.60	$ 4.95	$ 4.95	$ 3.30	$ 3.30	$ 3,026.00
Cyclone Discharge Trough Type Screw Conveyor (SC-1.6)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 3,094.00
Vibrating Screen (SN-1.1)	$ 216.00	$ 10.13	$ 10.13	$ 6.75	$ 6.75	$ 4,268.00
Vibrating Screen (Double Deck) (SN-1.2)	$ 360.00	$ 16.88	$ 16.88	$ 11.25	$ 11.25	$ 11,384.00
Tails Slurry Hold Tank	$ 160.00	$ 7.50	$ 7.50	$ 5.00	$ 5.00	$ 3,996.00
TOTAL	**$32,580.80**	**$1,527.23**	**$1,527.23**	**$1,018.15**	**$1,018.15**	**$ 282,374.00**

Note:
* The source documents suggest that this component is part of a larger assembly and that its refurbishment cost will, thus, be accounted for elsewhere.

Table 4-12: Used Equipment Costs for Equipment Components in Process Section 2: Roasting

Section Number	Name	Cost (1975)	Cost (2019)	Weight (lbs.)
2	Quench Tank Agitator (A-2.1)	$ 1,559.00	$ 4,880.38	750
2	HCI Mix Tank Agitator (A-2.2)	$ 800.00	$ 2,504.37	500
2	First Stage H_2O Scrubber Hold Tank Agitator (A-2.3) *	$ -	$ -	75
2	Sulfur Bin (B-2.1)	$ 200.00	$ 626.09	100
2	Bucket Elevator (BL-2.1)	$ 4,787.00	$ 14,985.50	2,750
2	Scrubber Exhaust Blower (BO-2.1) *	$ -	$ -	520
2	Dust Cyclone (CY-2.1)	$ 476.00	$ 1,490.10	55
2	Exo-Gas Generator (EG-2.1)	$ 15,085.00	$ 47,222.95	6,200
2	Sulfur Table Feeder (FD-2.1)	$ 401.00	$ 1,255.31	300
2	Roaster Cyclone Rotary Feeder (FD-2.2) *	$ -	$ -	120
2	Rotary Valve Feeder (FD-2.3) *	$ -	$ -	300
2	Carbon Monoxide Heater	$ 8,635.00	$ 27,031.50	150
2	HCL Metering Pump (P-2.1)	$ 258.50	$ 809.22	120
2	Quench Tank Slurry Pump (P-2.2)	$ 1,795.00	$ 5,619.17	270
2	First Stage H_2O Scrubber Pump 20 GPM (P-2.3)	$ 2,160.00	$ 6,761.79	250
2	HCL Drum Metering Pump (P-2.5)	$ 234.50	$ 734.09	75
2	Alkaline Scrubber Circ Pump (P-2.6)	$ 2,160.00	$ 6,761.79	250
2	Pug Mill (PM-2.1)	$ 14,000.00	$ 43,826.40	500
2	Roaster (R-2.1)	$ 120,000.00	$ 375,654.85	60,300
2	Multiple Hearth Roaster w/ Central Col. Fan (R-2.1) *	$ -	$ -	250
2	First Stage H_2O Scrubber Venturi (S-2.1)	$ 5,563.00	$ 17,414.73	610
2	Second Stage H_2O Scrubber Packed Bed (S-2.2)	$ 3,680.00	$ 11,520.08	540
2	Alkaline Scrubber Venturi (S-2.3)	$ 2,610.00	$ 8,170.49	400
2	Screw Conveyor (SC-1.4)	$ 1,497.00	$ 4,686.29	400
2	Cyclone Discharge Screw Conveyor (SC-2.1)	$ 1,697.00	$ 5,312.39	175
2	Cyclone Discharge Screw Conveyor Water Cooled (SC-2.2)	$ 2,200.00	$ 6,887.01	600
2	HCL Platform Scale (SL-2.1)	$ 648.00	$ 2,028.54	222
2	Tank TK-2.1 HCK Make up Tank	$ 850.24	$ 2,661.64	150
2	Quench Tank TK-2.2	$ 1,578.00	$ 4,939.86	1,290
2	Agitated First Stage H_2O Scrubber Hold Tank (TK-2.3)	$ 5,563.00	$ 17,414.73	420
2	Alkaline Scrubber Hold Tank (TK-2.5) *	$ -	$ -	420
2	Rotary Cyclone Discharge Feeder	$ 1,296.00	$ 4,057.07	150
2	Sur-Lite Waste Gas Incinerator	$ 19,502.00	$ 61,050.17	500
2	Upper Hearth Burners	$ 3,315.00	$ 10,377.47	50
	TOTAL:	**$ 222,550.24**	**$ 696,683.98**	**79,762**

Note:

* Blank cells in the “cost” column denote items that are an assumed to be a component of an “assembly” that were in fact priced as an “assembly” in the source documents collected by the project investigators. However, the individual cost of each component was not available, and the investigators were not provided the means to extract those costs. On the other hand, although individual costs could not be determined, the weights of the components were listed separately. Thus, the weights could be reported, and these are shown in the table for greater accuracy of this report.

Table 4-13: Shipping and Refurbishment Costs for Process Section 2: Roasting

Name	Freight Cost @ $0.32/lb.	Cost of Labor @ $0.015/lb.	Unloading Crane Cost @ $0.015/lb.	Unloading Forklift Cost @ $0.01/lb.	Unloading Truck Cost @ $0.01/lb.	Equipment Refurbishment
Quench Tank Agitator (A-2.1)	$ 240.00	$ 11.25	$ 11.25	$ 7.50	$ 7.50	$ 3,118.00
HCl Mix Tank Agitator (A-2.2)	$ 160.00	$ 7.50	$ 7.50	$ 5.00	$ 5.00	$ 1,600.00
First Stage H_2O Scrubber Hold Tank Agitator (A-2.3) *	$ 24.00	$ 1.13	$ 1.13	$ 0.75	$ 0.75	$ -
Sulfur Bin (B-2.1)	$ 32.00	$ 1.50	$ 1.50	$ 1.00	$ 1.00	$ 400.00
Bucket Elevator (BL-2.1)	$ 880.00	$ 41.25	$ 41.25	$ 27.50	$ 27.50	$ 9,574.00
Scrubber Exhaust Blower (BO-2.1) *	$ 166.40	$ 7.80	$ 7.80	$ 5.20	$ 5.20	$ -
Dust Cyclone (CY-2.1)	$ 17.60	$ 0.83	$ 0.83	$ 0.55	$ 0.55	$ 952.00
Exo-Gas Generator (EG-2.1)	$1,984.00	$ 93.00	$ 93.00	$ 62.00	$ 62.00	$ 30,170.00
Sulfur Table Feeder (FD-2.1)	$ 96.00	$ 4.50	$ 4.50	$ 3.00	$ 3.00	$ 802.00
Roaster Cyclone Rotary Feeder (FD-2.2) *	$ 38.40	$ 1.80	$ 1.80	$ 1.20	$ 1.20	$ -
Rotary Valve Feeder (FD-2.3) *	$ 96.00	$ 4.50	$ 4.50	$ 3.00	$ 3.00	$ -
Carbon Monoxide Heater	$ 48.00	$ 2.25	$ 2.25	$ 1.50	$ 1.50	$ 17,270.00
HCL Metering Pump (P-2.1)	$ 38.40	$ 1.80	$ 1.80	$ 1.20	$ 1.20	$ 517.00
Quench Tank Slurry Pump (P-2.2)	$ 86.40	$ 4.05	$ 4.05	$ 2.70	$ 2.70	$ 3,590.00
First Stage H_2O Scrubber Pump 20 GPM (P-2.3)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 4,320.00
HCL Drum Metering Pump (P-2.5)	$ 24.00	$ 1.13	$ 1.13	$ 0.75	$ 0.75	$ 469.00
Alkaline Scrubber Circ Pump (P-2.6)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 4,320.00
Pug Mill (PM-2.1)	$ 160.00	$ 7.50	$ 7.50	$ 5.00	$ 5.00	$ 28,000.00
Roaster (R-2.1)	$19,296.00	$ 904.50	$ 904.50	$603.00	$603.00	$ 240,000.00
Multiple Hearth Roaster w/ Central Col. Fan (R-2.1) *	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ -
First Stage H_2O Scrubber Venturi (S-2.1)	$ 195.20	$ 9.15	$ 9.15	$ 6.10	$ 6.10	$ 11,126.00
Second Stage H_2O Scrubber Packed Bed (S-2.2)	$ 172.80	$ 8.10	$ 8.10	$ 5.40	$ 5.40	$ 7,360.00
Alkaline Scrubber Venturi (S-2.3)	$ 128.00	$ 6.00	$ 6.00	$ 4.00	$ 4.00	$ 5,220.00
Screw Conveyor (SC-1.4)	$ 128.00	$ 6.00	$ 6.00	$ 4.00	$ 4.00	$ 2,994.00
Cyclone Discharge Screw Conveyor (SC-2.1)	$ 56.00	$ 2.63	$ 2.63	$ 1.75	$ 1.75	$ 3,394.00
Cyclone Discharge Screw Conveyor Water Cooled (SC-2.2)	$ 192.00	$ 9.00	$ 9.00	$ 6.00	$ 6.00	$ 4,400.00
HCL Platform Scale (SL-2.1)	$ 71.04	$ 3.33	$ 3.33	$ 2.22	$ 2.22	$ 1,296.00
Tank TK-2.1 HCK Make up Tank	$ 48.00	$ 2.25	$ 2.25	$ 1.50	$ 1.50	$ 1,700.48
Quench Tank TK-2.2	$ 412.80	$ 19.35	$ 19.35	$ 12.90	$ 12.90	$ 3,156.00
Agitated First Stage H_2O Scrubber Hold Tank (TK-2.3)	$ 134.40	$ 6.30	$ 6.30	$ 4.20	$ 4.20	$ 11,126.00
Alkaline Scrubber Hold Tank (TK-2.5) *	$ 134.40	$ 6.30	$ 6.30	$ 4.20	$ 4.20	$ -
Rotary Cyclone Discharge Feeder	$ 48.00	$ 2.25	$ 2.25	$ 1.50	$ 1.50	$ 2,592.00
Sur-Lite Waste Gas Incinerator	$ 160.00	$ 7.50	$ 7.50	$ 5.00	$ 5.00	$ 39,004.00
Upper Hearth Burners	$ 16.00	$ 0.75	$ 0.75	$ 0.50	$ 0.50	$ 6,630.00
TOTAL:	**$25,523.84**	**$1,196.43**	**$1,196.43**	**$797.62**	**$797.62**	**$ 445,100.48**

Note:
* The source documents suggest that this component is part of a larger assembly and that its refurbishment cost will, thus, be accounted for elsewhere.

Table 4-14: Used Equipment Costs for Equipment Components in Process Section 3: Leaching

Section Number	Name	Cost (1975)	Cost (2019)	Weight (lbs.)
3	Super Agitator (A-3.1) *	$ -	$ -	500
3	Unknown ***	$ -	$ -	110
3	Agitator Rubber Covered (A-4.7) *	$ -	$ -	25
3	Leach Air Blower (BO-3.1)	$ 863.20	$ 2,702.21	250
3	Wet Cyclone Classifier (CY-3.1) **	$ 577.00	$ 1,806.27	-
3	Thickener Under/Flow Pump (P-3.1)	$ 1,398.00	$ 4,376.38	250
3	Calcine Grinding Circuit Pump (P-3.10)	$ 1,379.00	$ 4,316.90	250
3	Calcine Grinding Circuit Pump (P-3.11)	$ 1,379.00	$ 4,316.90	250
3	Thickener Under/Flow Pump (P-3.2)	$ 1,398.00	$ 4,376.38	250
3	Thickener Over/Flow Pump (P-3.3)	$ 1,398.00	$ 4,376.38	250
3	Leached Slurry Pump (P-3.4)	$ 1,268.00	$ 3,969.42	250
3	(P-3.5)	$ 418.00	$ 1,308.53	156
3	Thickener Over/Flow Pump (P-3.6)	$ 418.00	$ 1,308.53	156
3	Pregnant. Liquor Circuit Pump (P-3.7)	$ 200.00	$ 626.09	30
3	Spare thickener Overflow Pump P-3.8 **	$ 418.00	$ 1,308.53	-
3	Pregnant Liquor Circuit Pump (P-3.9)	$ 200.00	$ 626.09	30
3	Agitated Reslurry Tank (TK-10) **	$ 466.00	$ 1,458.79	-
3	Leach Liquor Storage Tank (TK-11) **	$ 1,535.39	$ 4,806.47	-
3	Agitated Leach Tank (TK-3.1)	$ 1,741.00	$ 5,450.13	710
3	Unknown ***	$ -	$ -	210
3	Unknown ***	$ -	$ -	470
3	Agitated Leach Tank (TK-3.2)	$ 1,741.00	$ 5,450.13	710
3	Ball mill Discharge Tank	$ 725.00	$ 2,269.58	500
3	Cyclone Discharge Tank	$ 725.00	$ 2,269.58	500
3	Agitated Leach Tank (TK-3.3)	$ 1,741.00	$ 5,450.13	710
3	Agitated Leach Tank (TK-3.4)	$ 1,741.00	$ 5,450.13	710
3	Agitated Leach Tank (TK-3.5)	$ 1,741.00	$ 5,450.13	710
3	Agitated Leach Tank (TK-3.6)	$ 1,741.00	$ 5,450.13	710
3	Thickener Tank (TK-3.7)	$ 6,359.00	$ 19,906.58	6,000
3	Thickener Tank (TK-3.8)	$ 6,359.00	$ 19,906.58	6,000
3	Thickener Tank (TK-3.9)	$ 659.00	$ 2,062.97	6,000
	TOTAL	$ 38,588.59	$ 120,799.93	26,697

Notes:

* Blank cells in the "cost" column denote items that are an assumed to be a component of an "assembly" that were in fact priced as an "assembly" in the source documents collected by the project investigators. However, the individual cost of each component was not available, and the investigators were not provided the means to extract those costs. On the other hand, although individual costs could not be determined, the weights of the components were listed separately. Thus, the weights could be reported, and these are shown in the table for greater accuracy of this report.

** Blank cells in the "Weight" column denote items for which no weight information was provided.

*** "Unknown" items were attributed a weight in the source documents, but no description was provided.

Table 4-15: Shipping and Refurbishment Costs for Process Section 3: Leaching

Name	Freight Cost @ $0.32/lb.	Cost of Labor @ $0.015/lb.	Unloading Crane Cost @ $0.015/lb.	Unloading Forklift Cost @ $0.01/lb.	Unloading Truck Cost @ $0.01/lb.	Equipment Refurbishment Cost
Super Agitator (A-3.1) *	$ 160.00	$ 7.50	$ 7.50	$ 5.00	$ 5.00	$ -
Unknown ***	$ 35.20	$ 1.65	$ 1.65	$ 1.10	$ 1.10	$ -
Agitator Rubber Covered (A-4.7) *	$ 8.00	$ 0.38	$ 0.38	$ 0.25	$ 0.25	$ -
Leach Air Blower (BO-3.1)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 1,726.40
Wet Cyclone Classifier (CY-3.1) **	$ -	$ -	$ -	$ -	$ -	$ 1,154.00
Thickener Under/Flow Pump (P-3.1)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 2,796.00
Calcine Grinding Circuit Pump (P-3.10)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 2,758.00
Calcine Grinding Circuit Pump (P-3.11)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 2,758.00
Thickener Under/Flow Pump (P-3.2)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 2,796.00
Thickener Over/Flow Pump (P-3.3)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 2,796.00
Leached Slurry Pump (P-3.4)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 2,536.00
(P-3.5)	$ 49.92	$ 2.34	$ 2.34	$ 1.56	$ 1.56	$ 836.00
Thickener Over/Flow Pump (P-3.6)	$ 49.92	$ 2.34	$ 2.34	$ 1.56	$ 1.56	$ 836.00
Pregnant Liquor Circ. Pump (P-3.7)	$ 9.60	$ 0.45	$ 0.45	$ 0.30	$ 0.30	$ 400.00
Spare thickener Overflow Pump P-3.8 **	$ -	$ -	$ -	$ -	$ -	$ 836.00
Pregnant Liquor Circ. Pump (P-3.9)	$ 9.60	$ 0.45	$ 0.45	$ 0.30	$ 0.30	$ 400.00
Agitated Reslurry Tank (TK-10) **	$ -	$ -	$ -	$ -	$ -	$ 932.00
Leach Liq. Storage Tank (TK-11) **	$ -	$ -	$ -	$ -	$ -	$ 3,070.78
Agitated Leach Tank (TK-3.1)	$ 227.20	$ 10.65	$ 10.65	$ 7.10	$ 7.10	$ 3,482.00
Unknown ***	$ 67.20	$ 3.15	$ 3.15	$ 2.10	$ 2.10	$ -
Unknown ***	$ 150.40	$ 7.05	$ 7.05	$ 4.70	$ 4.70	$ -
Agitated Leach Tank (TK-3.2)	$ 227.20	$ 10.65	$ 10.65	$ 7.10	$ 7.10	$ 3,482.00
Ball mill Discharge Tank	$ 160.00	$ 7.50	$ 7.50	$ 5.00	$ 5.00	$ 1,450.00
Cyclone Discharge Tank	$ 160.00	$ 7.50	$ 7.50	$ 5.00	$ 5.00	$ 1,450.00
Agitated Leach Tank (TK-3.3)	$ 227.20	$ 10.65	$ 10.65	$ 7.10	$ 7.10	$ 3,482.00
Agitated Leach Tank (TK-3.4)	$ 227.20	$ 10.65	$ 10.65	$ 7.10	$ 7.10	$ 3,482.00
Agitated Leach Tank (TK-3.5)	$ 227.20	$ 10.65	$ 10.65	$ 7.10	$ 7.10	$ 3,482.00
Agitated Leach Tank (TK-3.6)	$ 227.20	$ 10.65	$ 10.65	$ 7.10	$ 7.10	$ 3,482.00
Thickener Tank (TK-3.7)	$1,920.00	$ 90.00	$ 90.00	$ 60.00	$ 60.00	$ 12,718.00
Thickener Tank (TK-3.8)	$1,920.00	$ 90.00	$ 90.00	$ 60.00	$ 60.00	$ 12,718.00
Thickener Tank (TK-3.9)	$1,920.00	$ 90.00	$ 90.00	$ 60.00	$ 60.00	$ 1,318.00
TOTAL:	**$8,543.04**	**$ 400.46**	**$ 400.46**	**$266.97**	**$266.97**	**$ 77,177.18**

Notes:

* The source documents suggest that this component is part of a larger assembly and that its refurbishment cost will, thus, be accounted for elsewhere.

** The source documents did not provide weights, but the source document description of the item allowed a general estimate of refurbishment cost.

*** An “Unknown” item was attributed a weight in the source documents and thus handling costs on a weight basis could be estimated. However, no description of the item was provided and, thus, no refurbishment cost could be estimated.

Table 4-16: Used Equipment Costs for Equipment Components in Process Section 4: Wastewater Treatment

Section Number	Name	Cost (1975)	Cost (2019)	Weight (lbs.)
4	Reslurry Tank. Agitator (A-4.1)	$ 819.00	$ 2,563.84	110
4	Reslurry Tank Agitator (A-4.2)	$ 819.00	$ 2,563.84	110
4	Reslurry Tank Agitator (A-4.3)	$ 819.00	$ 2,563.84	110
4	Tails Hold Tank Agitator (A-4.4)	$ 1,784.00	$ 5,584.74	400
4	Thickener Under/Flow Pump (P-4.1)	$ 1,398.00	$ 4,376.38	250
4	Thickener Over/Flow Pump (P-4.10)	$ 418.00	$ 1,308.53	156
4	Thickener Under/Flow Pump (P-4.2)	$ 1,398.00	$ 4,376.38	250
4	Thickener Under/Flow Pump (P-4.3)	$ 1,398.00	$ 4,376.38	250
4	Thickener Under/Flow Pump (P-4.4)	$ 1,398.00	$ 4,376.38	250
4	Reslurry Transfer Pump (P-4.5)	$ 1,268.00	$ 3,969.42	240
4	Reslurry Transfer Pump (P-4.6)	$ 1,268.00	$ 3,969.42	240
4	Reslurry Transfer Pump (P-4.7)	$ 1,268.00	$ 3,969.42	240
4	Tails Slurry Pump (P-4.8)	$ 1,268.00	$ 3,969.42	375
4	Thickener Over/Flow Recycle Liquor Pump (P-4.9)	$ 418.00	$ 1,308.53	156
4	Thickener Tank (TK-4.1)	$ 6,359.00	$ 19,906.58	10,200
4	Thickener Tank (TK-4.2)	$ 6,359.00	$ 19,906.58	6,600
4	Thickener Tank (TK-4.3)	$ 6,359.00	$ 19,906.58	7,250
4	Thickener Tank (TK-4.4)	$ 6,359.00	$ 19,906.58	6,000
4	Agitated Reslurry Tank (TK-4.5)	$ 466.00	$ 1,458.79	210
4	Agitated Reslurry Tank (TK-4.6)	$ 466.00	$ 1,458.79	210
4	Agitated Reslurry Tank (TK-4.7)	$ 466.00	$ 1,458.79	210
4	Agitated Tails Hold and Preheat Tank (TK-4.8)	$ 1,798.00	$ 5,628.56	1,250
	TOTAL:	**$ 44,373.00**	**$ 138,907.77**	**35,067**

Table 4-17: Shipping and Refurbishment Costs for Process Section 4: Wastewater Treatment

Name	Freight Cost @ $0.32/lb.	Cost of Labor @ $0.015/lb.	Unloading Crane Cost @ $0.015/lb.	Unloading Forklift Cost @ $0.01/lb.	Unloading Truck Cost @ $0.01/lb.	Equipment Refurbishment
Reslurry Tank. Agitator (A-4.1)	$ 35.20	$ 1.65	$ 1.65	$ 1.10	$ 1.10	$ 1,638.00
Reslurry Tank Agitator (A-4.2)	$ 35.20	$ 1.65	$ 1.65	$ 1.10	$ 1.10	$ 1,638.00
Reslurry Tank Agitator (A-4.3)	$ 35.20	$ 1.65	$ 1.65	$ 1.10	$ 1.10	$ 1,638.00
Tails Hold Tank Agitator (A-4.4)	$ 128.00	$ 6.00	$ 6.00	$ 4.00	$ 4.00	$ 3,568.00
Thickener Under/Flow Pump (P-4.1)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 2,796.00
Thickener Over/Flow Pump (P-4.10)	$ 49.92	$ 2.34	$ 2.34	$ 1.56	$ 1.56	$ 836.00
Thickener Under/Flow Pump (P-4.2)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 2,796.00
Thickener Under/Flow Pump (P-4.3)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 2,796.00
Thickener Under/Flow Pump (P-4.4)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 2,796.00
Reslurry Transfer Pump (P-4.5)	$ 76.80	$ 3.60	$ 3.60	$ 2.40	$ 2.40	$ 2,536.00
Reslurry Transfer Pump (P-4.6)	$ 76.80	$ 3.60	$ 3.60	$ 2.40	$ 2.40	$ 2,536.00
Reslurry Transfer Pump (P-4.7)	$ 76.80	$ 3.60	$ 3.60	$ 2.40	$ 2.40	$ 2,536.00
Tails Slurry Pump (P-4.8)	$ 120.00	$ 5.63	$ 5.63	$ 3.75	$ 3.75	$ 2,536.00
Thickener Over/Flow Recycle Liquor Pump (P-4.9)	$ 49.92	$ 2.34	$ 2.34	$ 1.56	$ 1.56	$ 836.00
Thickener Tank (TK-4.1)	$3,264.00	$ 153.00	$ 153.00	$102.00	$102.00	$ 12,718.00
Thickener Tank (TK-4.2)	$2,112.00	$ 99.00	$ 99.00	$ 66.00	$ 66.00	$ 12,718.00
Thickener Tank (TK-4.3)	$2,320.00	$ 108.75	$ 108.75	$ 72.50	$ 72.50	$ 12,718.00
Thickener Tank (TK-4.4)	$1,920.00	$ 90.00	$ 90.00	$ 60.00	$ 60.00	$ 12,718.00
Agitated Reslurry Tank (TK-4.5)	$ 67.20	$ 3.15	$ 3.15	$ 2.10	$ 2.10	$ 932.00
Agitated Reslurry Tank (TK-4.6)	$ 67.20	$ 3.15	$ 3.15	$ 2.10	$ 2.10	$ 932.00
Agitated Reslurry Tank (TK-4.7)	$ 67.20	$ 3.15	$ 3.15	$ 2.10	$ 2.10	$ 932.00
Agitated Tails Hold and Preheat Tank (TK-4.8)	$ 400.00	$ 18.75	$ 18.75	$ 12.50	$ 12.50	$ 3,596.00
TOTAL:	**$11,221.44**	**$ 526.01**	**$ 526.01**	**$350.67**	**$350.67**	**$ 88,746.00**

Table 4-18: Used Equipment Costs for Equipment Components in Process Section 5: Leaching Ancillary

Section Number	Name	Cost (1975)	Cost (2019)	Weight (lbs.)
5	Tails Hold Tank Agitator (A-5.1)	$ 1,811.00	$ 5,669.26	750
5	CA Exhaust Blower BO-5.1	$ 35,106.00	$ 109,897.83	60
5	Condenser Absorber (CA-5.1)	$ 5,721.00	$ 17,909.35	1,417
5	Tailings Slurry Pump P--5.1	$ 1,594.00	$ 4,989.95	520
5	CA Circuit Pump P-5.2	$ 372.00	$ 1,164.53	50
5	Leach Liquor Pump P-5.3	$ 200.00	$ 626.09	30
5	Waste Slurry Pump P-5.4	$ 1,759.00	$ 5,506.47	375
5	Condenser	$ 7,111.00	$ 22,260.68	800
5	Precooler *	$ -	$ -	620
5	CA Precooler (PC-5.1) **	$ 985.00	$ 3,083.50	-
5	NH_3 Scale SL-5.1	$ 648.00	$ 2,028.54	222
5	CO_2 Scale SL-5.2	$ 648.00	$ 2,028.54	222
5	Tailings Stripper (NH3 Stripper)	$ 3,309.00	$ 10,358.68	200
5	Pregnant Solution Ammonia Stripper	$ 3,131.00	$ 9,801.46	300
5	Leach Solution Ammonia Stripper	$ 3,131.00	$ 9,801.46	300
5	Leach Liquor Storage Tank (TK-5.2)	$ 1,619.51	$ 5,069.81	400
	TOTAL:	**$ 67,145.51**	**$ 210,196.14**	**6,266**

Notes:

* Blank cells in the "cost" column denote items that are an assumed to be a component of an "assembly" that were in fact priced as an "assembly" in the source documents collected by the project investigators. However, the individual cost of each component was not available, and the investigators were not provided the means to extract those costs. On the other hand, although individual costs could not be determined, the weights of the components were listed separately. Thus, the weights could be reported, and these are shown in the table for greater accuracy of this report.

** Blank cells in the "Weight" column denote items for which no weight information was provided.

Table 4-19: Shipping and Refurbishment Costs for Process Section 5: Leaching Ancillary

Name	Freight Cost @ $0.32/lb.	Cost of Labor @ $0.015/lb.	Unloading Crane Cost @ $0.015/lb.	Unloading Forklift Cost @ $0.01/lb.	Unloading Truck Cost @ $0.01/lb.	Equipment Refurbishment
Tails Hold Tank Agitator (A-5.1)	$ 240.00	$ 11.25	$ 11.25	$ 7.50	$ 7.50	$ 3,622.00
CA Exhaust Blower BO-5.1	$ 19.20	$ 0.90	$ 0.90	$ 0.60	$ 0.60	$ 70,212.00
Condenser Absorber (CA-5.1)	$ 453.44	$ 21.26	$ 21.26	$ 14.17	$ 14.17	$ 11,442.00
Tailings Slurry Pump P--5.1	$ 166.40	$ 7.80	$ 7.80	$ 5.20	$ 5.20	$ 3,188.00
CA Circuit Pump P-5.2	$ 16.00	$ 0.75	$ 0.75	$ 0.50	$ 0.50	$ 744.00
Leach Liquor Pump P-5.3	$ 9.60	$ 0.45	$ 0.45	$ 0.30	$ 0.30	$ 400.00
Waste Slurry Pump P-5.4	$ 120.00	$ 5.63	$ 5.63	$ 3.75	$ 3.75	$ 3,518.00
Condenser	$ 256.00	$ 12.00	$ 12.00	$ 8.00	$ 8.00	$ 14,222.00
Precooler *	$ 198.40	$ 9.30	$ 9.30	$ 6.20	$ 6.20	$ -
CA Precooler (PC-5.1) **	$ -	$ -	$ -	$ -	$ -	$ 1,970.00
NH_3 Scale SL-5.1	$ 71.04	$ 3.33	$ 3.33	$ 2.22	$ 2.22	$ 1,296.00
CO_2 Scale SL-5.2	$ 71.04	$ 3.33	$ 3.33	$ 2.22	$ 2.22	$ 1,296.00
Tailings Stripper (NH_3 stripper)	$ 64.00	$ 3.00	$ 3.00	$ 2.00	$ 2.00	$ 6,618.00
Pregnant Solution Ammonia Stripper	$ 96.00	$ 4.50	$ 4.50	$ 3.00	$ 3.00	$ 6,262.00
Leach Solution Ammonia Stripper	$ 96.00	$ 4.50	$ 4.50	$ 3.00	$ 3.00	$ 6,262.00
Leach Liquor Storage Tank (TK-5.2)	$ 128.00	$ 6.00	$ 6.00	$ 4.00	$ 4.00	$ 3,239.02
TOTAL:	**$2,005.12**	**$ 93.99**	**$ 93.99**	**$ 62.66**	**$ 62.66**	**$ 134,291.02**

Notes:

* The source documents suggest that this component is part of a larger assembly and that its refurbishment cost will, thus, be accounted for elsewhere.

** The source documents did not provide weights, but the source document description of the item allowed a general estimate of refurbishment cost.

Table 4-20: Used Equipment Costs for Components in Process Section 6: SX Ancillary

Section Number	Name	Cost (1975)	Cost (2019)	Weight (lbs.)
6	C20 Mix Tank Agitator (A-6.1)	$ 331.00	$ 1,036.18	80
6	DC Agitator (A-6.2)	$ 836.00	$ 2,617.06	28
6	DC Agitator (A-6.3)	$ 836.00	$ 2,617.06	28
6	Decomposer Agitator (DC-3) *	$ -	$ -	28
6	Decomposer Tank (DC-6.1)	$ 1,295.00	$ 4,053.94	220
6	In-Line Polishing Filter Precoat Type (F-6.1)	$ 116.00	$ 363.13	8
6	Horizontal Belt Filter. (F-6.2)	$ 23,000.00	$ 72,000.51	4,000
6	In-Line Polishing Filter Precoat Type-Rubber Lined (F-6.3)	$ 116.00	$ 363.13	8
6	Pregnant Liquor Heat Exchanger (HX-6.1) *	$ -	$ -	300
6	Stripper Preheat Heat Exchange (HX-6.2)	$ 754.00	$ 2,360.36	300
6	Filter Feed Pump. (P-6.1)	$ 382.00	$ 1,195.83	50
6	Drain Pump 6 GPM (P-6.11)	$ 263.00	$ 823.31	37
6	Stripper Slurry Pump (P-6.2)	$ 1,584.00	$ 4,958.64	200
6	Thickener Over/Flow Pump (P-6.3)	$ 418.00	$ 1,308.53	156
6	Thickener Under/Flow Pump (P-6.4)	$ 1,398.00	$ 4,376.38	250
6	$(NH_4)_2S$ Solution Metering Pump (P-6.8)	$ 219.00	$ 685.57	120
6	Var. Volume C20 Makeup Pump (P-6.9) *	$ -	$ -	120
6	Platform Scales (SL-6.1)	$ 648.00	$ 2,028.54	222
6	Steam Stripper (ST-6.1)	$ 3,634.00	$ 11,376.08	850
6	Stripper Slurry Thickener Tank (TK-6.1)	$ 7,731.00	$ 24,201.56	7,100
6	Filtrate Receiver Tank. (TK-6.3)	$ 685.24	$ 2,145.11	150
6	Precoat Tank (TK-6.4)	$ 74.42	$ 232.97	50
6	Pregnant Liquor Aeration Tank (TK-6.5)	$ 697.00	$ 2,181.93	395
6	Filter Precoat Mix Tank (TK-6.6)	$ 74.42	$ 232.97	50
6	H20 C20 Mix Tank (TK-6.8)	$ 91.00	$ 284.87	50
	TOTAL:	**$ 45,183.08**	**$ 141,443.69**	**14,800**

Note:
* Blank cells in the "cost" column denote items that are an assumed to be a component of an "assembly" that were in fact priced as an "assembly" in the source documents collected by the project investigators. However, the individual cost of each component was not available, and the investigators were not provided the means to extract those costs. On the other hand, although individual costs could not be determined, the weights of the components were listed separately. Thus, the weights could be reported, and these are shown in the table for greater accuracy of this report.

Table 4-21: Shipping and Refurbishment Costs for Process Section 6: SX Ancillary

Name	Freight Cost @ $0.32/lb.	Cost of Labor @ $0.015/lb.	Unloading Crane Cost @ $0.015/lb.	Unloading Forklift Cost @ $0.01/lb.	Unloading Truck Cost @ $0.01/lb.	Equipment Refurbishment
C20 Mix Tank Agitator (A-6.1)	$ 25.60	$ 1.20	$ 1.20	$ 0.80	$ 0.80	$ 662.00
DC Agitator (A-6.2)	$ 8.96	$ 0.42	$ 0.42	$ 0.28	$ 0.28	$ 1,672.00
DC Agitator (A-6.3)	$ 8.96	$ 0.42	$ 0.42	$ 0.28	$ 0.28	$ 1,672.00
Decomposer Agitator (DC-3) *	$ 8.96	$ 0.42	$ 0.42	$ 0.28	$ 0.28	$ -
Decomposer Tank (DC-6.1)	$ 70.40	$ 3.30	$ 3.30	$ 2.20	$ 2.20	$ 2,590.00
In-Line Polishing Filter Precoat Type (F-6.1)	$ 2.56	$ 0.12	$ 0.12	$ 0.08	$ 0.08	$ 232.00
Horizontal Belt Filter. (F-6.2)	$1,280.00	$ 60.00	$ 60.00	$ 40.00	$ 40.00	$ 46,000.00
In-Line Polishing Filter Precoat Type-Rubber Lined (F-6.3)	$ 2.56	$ 0.12	$ 0.12	$ 0.08	$ 0.08	$ 232.00
Pregnant Liquor Heat Exchanger (HX-6.1) *	$ 96.00	$ 4.50	$ 4.50	$ 3.00	$ 3.00	$ -
Stripper Preheat Heat Exchange (HX-6.2)	$ 96.00	$ 4.50	$ 4.50	$ 3.00	$ 3.00	$ 1,508.00
Filter Feed Pump. (P-6.1)	$ 16.00	$ 0.75	$ 0.75	$ 0.50	$ 0.50	$ 764.00
Drain Pump 6 GPM (P-6.11)	$ 11.84	$ 0.56	$ 0.56	$ 0.37	$ 0.37	$ 526.00
Stripper Slurry Pump (P-6.2)	$ 64.00	$ 3.00	$ 3.00	$ 2.00	$ 2.00	$ 3,168.00
Thickener Over/Flow Pump (P-6.3)	$ 49.92	$ 2.34	$ 2.34	$ 1.56	$ 1.56	$ 836.00
Thickener Under/Flow Pump (P-6.4)	$ 80.00	$ 3.75	$ 3.75	$ 2.50	$ 2.50	$ 2,796.00
$(NH_4)_2S$ Solution Metering Pump (P-6.8)	$ 38.40	$ 1.80	$ 1.80	$ 1.20	$ 1.20	$ 438.00
Variable Volume C20 Makeup Pump (P-6.9) *	$ 38.40	$ 1.80	$ 1.80	$ 1.20	$ 1.20	$ -
Platform Scales (SL-6.1)	$ 71.04	$ 3.33	$ 3.33	$ 2.22	$ 2.22	$ 1,296.00
Steam Stripper (ST-6.1)	$ 272.00	$ 12.75	$ 12.75	$ 8.50	$ 8.50	$ 7,268.00
Stripper Slurry Thickener Tank (TK-6.1)	$2,272.00	$ 106.50	$ 106.50	$ 71.00	$ 71.00	$ 15,462.00
Filtrate Receiver Tank. (TK-6.3)	$ 48.00	$ 2.25	$ 2.25	$ 1.50	$ 1.50	$ 1,370.48
Precoat Tank (TK-6.4)	$ 16.00	$ 0.75	$ 0.75	$ 0.50	$ 0.50	$ 148.84
Pregnant Liquor Aeration Tank (TK-6.5)	$ 126.40	$ 5.93	$ 5.93	$ 3.95	$ 3.95	$ 1,394.00
Filter Precoat Mix Tank (TK-6.6)	$ 16.00	$ 0.75	$ 0.75	$ 0.50	$ 0.50	$ 148.84
H20 C20 Mix Tank (TK-6.8)	$ 16.00	$ 0.75	$ 0.75	$ 0.50	$ 0.50	$ 182.00
TOTAL:	**$4,736.00**	**$ 222.00**	**$ 222.00**	**$ 148.00**	**$ 148.00**	**$ 90,366.16**

Note:
* The source documents suggest that this component is part of a larger assembly and that its refurbishment cost will, thus, be accounted for elsewhere.

Table 4-22: Combined Equipment Costs for all Process Sections

Section Number	Name	Cost (1975)	Cost (2019)	Weight (lbs.)
1	Crushing and Grinding	$ 141,187	$ 441,980	101,815
2	Roasting	$ 222,550	$ 696,684	79,762
3	Leaching	$ 38,589	$ 120,800	26,697
4	Waste Water Treatment	$ 44,373	$ 138,908	35,067
5	Leaching Ancillary	$ 67,146	$ 210,196	6,266
6	SX Ancillary	$ 45,183	$ 141,444	14,800
	TOTAL:	**$ 559,027**	**$ 1,750,011**	**264,407**

Table 4-23: Combined Shipping and Refurbishment Costs for all Process Sections

Section Number	Name	Freight Cost @ $0.32/lb.	Cost of Labor @ $0.015/lb.	Unloading Crane Cost @ $0.015/lb.	Unloading Forklift Cost @ $0.01/lb.	Unloading Truck Cost @ $0.01/lb.	Equipment Refurbishment
1	Crushing and Grinding	$ 32,581	$ 1,527	$ 1,527	$ 1,018	$ 1,018	$ 282,374
2	Roasting	$ 25,524	$ 1,196	$ 1,196	$ 798	$ 798	$ 445,100
3	Leaching	$ 8,543	$ 400	$ 400	$ 267	$ 267	$ 77,177
4	Waste Water Treatment	$ 11,221	$ 526	$ 526	$ 351	$ 351	$ 88,746
5	Leaching Ancillary	$ 2,005	$ 94	$ 94	$ 63	$ 63	$ 134,291
6	SX Ancillary	$ 4,736	$ 222	$ 222	$ 148	$ 148	$ 90,366
	TOTAL:	**$ 84,610**	**$ 3,966**	**$ 3,966**	**$ 2,644**	**$ 2,644**	**$ 1,118,055**

In addition to these used equipment costs, the engineering team was provided a quote for the disassembly and loading of equipment at the current host location as well as an additional quote for the remediation of the shell of an existing industrial building at the host site. These costs are summarized in the following table.

Table 4-24: Quoted Construction Costs Associated with Disassembly of Existing Plant Equipment and Reassembly at New Host Site

Item Description	Cost
Used Equipment Disassembly	
Structural and equipment marking	$ 7,000
Dismantling plant	$ 218,400
Equipment rental	$ 40,000
Load plant onto trucks	$ 10,500
Construction superintendent	$ 23,520
Sub-Total:	***$ 299,420***
New Plant Construction	
Site Preparation	$ 18,900
Concrete	$ 267,772
Structural Steel	$ 263,040
Equipment Assembly	$ 597,560
Electrical	$ 480,000
Sub-Total:	***$1,627,272***

With regard to new equipment, the engineering team received a budgetary quote for a new 12-stage solvent extraction system with the appropriate capacity for the rougher, cleaner, and saponification circuits in the 2,000 lb./hr. pilot-scale installation. In addition, the team used this baseline costing data as well as process modeling data to estimate the capital requirements of the refining circuit. In order to generate three high-purity refined rare earth oxide *(REO)* products, the team estimates that the refining circuit will require 118 mixer settlers with an average mixing volume of 204 liters. A scale factor analysis was applied to estimate the overall cost of this refining plant.

The following table summarizes the overall capital cost of the 2,000 lb./hr. pilot-scale plant, including 2019 values for the used equipment, estimated costs for refurbishment of used equipment, quoted costs for the new equipment, estimates for freight and unloading of equipment, and quoted costs for dismantling and loading of used equipment.

Table 4-25: Total Capital Cost Estimate for 2,000 lb./hr. Pilot-Scale Plant

Item Description	Cost
Used Equipment Value (2019)	$ 1,750,011
Disassembly	$ 299,420
Freight	$ 97,831
Refurbishment	$ 1,118,055
New Equipment (Rougher, Cleaner, Saponification SX)	$ 1,462,145
New Equipment (Refining SX)	$ 7,042,449
Plant Construction	$ 1,627,272
Total:	**$ 13,397,183**

4.3.1.2 Commercial-Scale Installation

A detailed bottom-up assessment was performed to determine the total expected capital expenditures for a hypothetical commercial-scale facility. For estimation and analysis purposes, all capital costs were determined assuming new equipment, free-on-board *(FOB)* shipping point. All costs were initially estimated using a 2016 index year, which was then escalated to a 2019 construction year using the Chemical Engineering Plant Cost Index *(CEPCI)*. Using the scale-up factors and design capacities described in *Section 4.1.3*, commercial-scale equipment requirements were determined based on a nominal plant feed rate of 500 st/hr. Capital cost estimates for individual units were then determined using scale factor estimates dictated by a standard exponential cost model:

$$CAPEX = \left[\frac{CI_C}{CI_H}\right][K][aX^b]$$

where CI_C and CI_H are the current and historic cost indices, K is the equipment cost ratio (or Lang) factor, X is the unit capacity, and “a” and “b” are fitting constants unique to each equipment item (Gentry and O’Neil, 1986). The following table shows the capital cost model fitting parameters (a and b) for the various equipment categories used in the cost model. The data used to generate these models were gathered from several standard equipment costing resources (MAMEC, Gentry and O’Neil, Garett, 1989).

Table 4-26: Capital Cost Model Parameters Used in Techno-Economic Analysis

Process Operation	Coefficient, a	Exponent, b	Capacity Unit, X	Reference
Jaw Crusher	651	1.25	HP	MAMEC
Roll Crusher	1,120	0.85	HP	MAMEC
Screen	1,000	0.91	ft2	MAMEC
Bin	4,440	0.62	Tons	MAMEC
Dry Ball Mill	35,000	0.56	HP	MAMEC
PE Tank	1	0.95	gal	MAMEC
Tank Mixer	10,640	0.56	HP	MAMEC
Slurry Pump	2,150	0.38	gpm	MAMEC
Thickener	280	0.8	ft^2	MAMEC
Filter Press	6,070	0.72	ft^3	MAMEC
Conveyor	2,090	0.55	TPH	MAMEC
Roaster	390,000	0.48	MBTU/hr.	Garett, 1989
Scrubber	7	0.94	cfm	MAMEC
Chiller	97,600	0.6	MBTU/hr.	Matches
Solution Heater	25,900	0.95	MBTU/hr.	MAMEC
Mixer Settler	9,182	0.45	gal	Matches

After the base equipment prices were established, installed capital costs were determined by applying Lang factors to the direct equipment costs. Typical ranges for Lang factors are specified by Gentry and O'Neil (1984) and are shown in the following table. The Lang factors used in the current analysis reflect "brownfield" conditions, where existing infrastructure, including buildings, utilities, and plant services, are already in place (i.e., the case in this instance). Each individual Lang Factor is categorized as an ancillary cost; a buildings/infrastructure cost; or an engineering, procurement, construction, and management *(EPCM)* cost. An overall contingency was also applied to the final base equipment price.

Table 4-27: Lang Factor Estimation for Commercial-Scale Installation Costs

Module	Type	% of Base	Typical
Piping, Materials and Labor	Ancillary	20%	7 to 25%
Electrical, Materials and Labor	Ancillary	20%	13 to 25%
Instrumentation	Ancillary	8%	3 to 12%
Plant Services	Ancillary	10%	7 to 15%
Process Buildings	Buildings	40%	33 to 50%
Auxiliary Buildings	Buildings	15%	7 to 15%
Site Improvements	Buildings	10%	3 to 18%
Equipment Installation	EPCM	17%	17 to 25%
Field Expenses	EPCM	12%	10 to 12%
Project Management and Construction	EPCM	30%	30 to 33%
Contingency	Contingency	15%	10 to 15%
Total Lang Factor (=1+sum % Base):		**297%**	
Alternatively, a multiplier of:		**2.97**	

Using this cost model and Lang Factor, the total capital costs for the 500 st/hr. plant were determined. The following tables show this value for each major unit operation as well as the cumulative costs for each process module. This data indicates that the roasting operation is the costliest component of the circuit, followed by rougher solvent extraction.

Table 4-28: Itemized List of Capital Costs for Commercial-Scale Plant (500 st/hr.)

Unit ID	Unit Operation	Installed Design		No. of Units	Total Cost
PM1.1	Front End Loader	500	TPH	5	$ 737,000
PM1.2	Jaw Crusher	589	HP	1	$ 1,922,101
PM1.3	Roll Crusher	430	HP	1	$ 191,979
PM1.4	Vibrating Screen	124	ft^2	1	$ 80,131
PM1.5	Conveyors	575	TPH	2	$ 137,058
				Crushing and Screening Total:	**$ 3,068,269**
PM2.1	Vibrating Screen	332	ft^2	1	$ 314,467
PM2.2	Storage Bins	100	Tons	1	$ 76,638
PM2.3	Air Swept Ball Mill	5,609	HP	1	$ 4,230,785
PM2.4	Bucket Elevator	625	TPH	1	$ 322,000
PM2.5	Elevator Motor	58	HP	1	$ 24,988
				Dry Grinding Total:	**$ 4,968,877**
PM3.1	Storage Bins	100	Tons	2	$ 153,275
PM3.2	Conveyors	575	TPH	1	$ 68,529
PM3.3	Roaster	737	MBTU/hr.	1	$ 9,275,229
PM3.4	Gas Scrubber	11,500	cfm	1	$ 43,908
PM3.5	Spray Chamber Quencher	11,500	cfm	3	$ 414,702
PM3.6	Chiller	131	MBTU/hr.	1	$ 1,816,959
				Roasting Total:	**$11,772,602**
PM4.1	Polyethylene Tanks	164,805	gal	3	$ 370,959
PM4.2	Tank Mixer	474	HP	3	$ 1,029,368
PM4.3	Process Pump	10,987	gpm	3	$ 448,874
PM4.4	Thickener	22,590	ft^2	1	$ 870,880
PM4.5	Solid Waste Filter Press	3,600	ft^3	1	$ 2,183,148
PM4.6	Solution Heater	2	MBTU/hr.	1	$ 59,829
				Leaching Total:	**$ 4,963,058**
PM5.1	Polyethylene Tanks	35,136	gal	6	$ 171,096
PM5.2	Tank Mixer	20	HP	2	$ 115,870
PM5.3	Process Pump	7,027	gpm	1	$ 126,176
PM5.4	Mixer Settler	61,107	gal	6	$ 7,712,058
				Rougher SX Total:	**$ 8,125,199**
PM6.1	Polyethylene Tanks	1,405	gal	5	$ 6,716
PM6.2	Tank Mixer	1	HP	2	$ 18,8732
PM6.3	Process Pump	281	gpm	3	$ 110,898
PM6.4	Mixer Settler	2,444	gal	6	$ 1,821,101
				Cleaner SX Total:	**$ 1,957,587**
PM7.1	Polyethylene Tanks	3,514	gal	3	$ 9,616
PM7.2	Tank Mixer	2	HP	1	$ 15,818
PM7.3	Process Pump	703	gpm	2	$ 104,859
PM7.4	Mixer Settler	18,332	gal	3	$ 2,247,406
PM7.5	Filter Press	0	ft^3	1	$ 2,297
				SX Wash/Saponification Total:	**$ 2,379,996**
PM8.1	Polyethylene Tanks	1,504	gal	4	$ 4,089
PM8.2	Tank Mixer	1	HP	3	$ 24,069
PM8.3	Process Pump	70	gpm	1	$ 21,786
PM8.4	Filter Press	0	ft^3	1	$ 3,171
PM8.5	Roaster	0	MBTU/hr.	1	$ 235,395
				REE Precipitation Total:	**$ 288,511**
PM9.1	Polyethylene Tanks	453,131	gal	2	$ 645,913
PM9.2	Process Pump	78,805	gpm	2	$ 634,437
PM9.3	Filter Press	469	ft^3	1	$ 506,968
PM9.4	Conveyors	569	TPH	1	$ 68,114
				Water Treatment Total:	**$ 1,855,432**

Note: The "Installed Design" represents the total capacity required for each respective unit operation. For operations requiring large volumes, this capacity will be split among several smaller parallel operations in the final design. Likewise, "No. of Units" represents the number of identical units needed in each parallel train.

Table 4-29: Module-Level Capital Costs for Commercial-Scale Plant (500 st/hr.)

Module ID	Process Module	Equipment Cost	Reagent Fills	Ancillary	Buildings	EPCM	Contingency	Total Plant Cost	
								Index Year	Constructed Year
PM1	Crushing and Screening	$ 3,068,269	$ -	$ 1,779,596	$ 1,994,375	$ 1,810,279	$ 460,240	$ 9,112,759	$ 10,253,242
PM2	Dry Grinding	$ 4,968,877	$ -	$ 2,881,949	$ 3,229,770	$ 2,931,638	$ 745,332	$ 14,757,566	$ 16,604,509
PM3	Roasting	$ 11,772,602	$ -	$ 6,828,109	$ 7,652,191	$ 6,945,835	$ 1,765,890	$ 34,964,628	$ 39,340,532
PM4	Leaching	$ 4,963,058	$ -	$ 2,878,574	$ 3,225,988	$ 2,928,204	$ 744,459	$ 14,740,282	$ 16,585,062
PM5	Rougher SX	$ 8,125,199	$ 1,218,073	$ 4,712,615	$ 5,281,379	$ 4,793,867	$ 1,218,780	$ 25,349,913	$ 28,522,513
PM6	Cleaner SX	$ 1,957,587	$ 48,723	$ 1,135,400	$ 1,272,431	$ 1,154,976	$ 293,638	$ 5,862,755	$ 6,596,492
PM7	SX Wash/Saponification	$ 2,379,996	$ 182,711	$ 1,380,397	$ 1,546,997	$ 1,404,197	$ 356,999	$ 7,251,298	$ 8,158,814
PM8	REE Precipitation	$ 288,511	$ -	$ 167,336	$ 187,532	$ 170,221	$ 43,277	$ 856,876	$ 964,116
PM9	Water Treatment	$ 1,855,432	$ -	$ 1,076,150	$ 1,206,031	$ 1,094,705	$ 278,315	$ 5,510,632	$ 6,200,300
	TOTAL:	**$ 39,379,529**	**$ 1,449,507**	**$ 22,840,127**	**$ 25,596,694**	**$ 23,233,922**	**$ 5,906,929**	**$118,406,709**	**$ 133,225,581**

4.3.2 Tabular Listing of all Expected Operational and Maintenance Costs Including Labor, Electric, Reagents, and Other Consumables

The operating expense *(OpEx)* estimate for the proposed facility was categorized according to (i) operating labor, (ii) technical labor, (iii) power/utilities, (iv) lease/toll agreements, (v) consumables/reagents, (vi) waste disposal, (vii) plant overhead, and (viii) general sales and administration. Like the capital cost estimate, a bottom-up approach was used to determine the expected operating costs from first principles. Direct operating costs were specified using designed labor requirements along with publicly available wage and burden rates. Direct operating costs were also specified via itemized equipment power requirements, reagent dosing requirements, waste disposal costs, and expected maintenance requirements. General sales, and administrative *(GSA)* costs were specified using standard overhead charge rates. For accounting and analysis purposes, costs were allocated based on cost type (fixed, variable or overhead), and each line item cost was also allocated to a process unit using the following codes:

- PM1 Crushing and Screening
- PM2 Dry Grinding
- PM3 Roasting
- PM4 Leaching
- PM5 Rougher SX
- PM6 Cleaner SX
- PM7 SX Wash/Saponification
- PM8 REE Precipitation
- PM9 Water Treatment
- PM10 REO Refining

4.3.2.1 Labor Costs

Median pay rates for each labor category for the commercial-scale facility were determined using data from the Bureau of Labor Statistics *(BLS).* A standard fringe burden rate of 25 percent was applied for all categories. The labor requirements per shift were determined by designed manpower requirements and prior experience in similarly-sized projects. A summary of the direct labor costs is shown in the following table.

Table 4-30: Labor Cost Estimate for Commercial-Scale Plant (500 st/hr.)

Labor Category	Code	Labor Rate ($/hr.)	Fringe Benefits	No. Per Shift	Shift Cost	Source/Notes
Skilled Operator	OL	$24.98	25%	4	$ 999.20	May 2018 KY BLS, Occupation Code: 51-9011
Operator	OL	$19.08	25%	9	$ 1,717.20	May 2018 KY BLS, Occupation Code: 51-9012
Foreman/System Operator	OL	$30.39	25%	2	$ 607.80	May 2018 KY BLS, Occupation Code: 47-1011
Maintenance	OL	$22.73	25%	2	$ 454.60	May 2018 KY BLS, Occupation Code: 49-9043
Lab Technician	TL	$21.97	25%	2	$ 439.40	May 2018 KY BLS, Occupation Code: 19-4031
Metallurgist/Chemical Engineer	TL	$45.85	25%	3	$ 1,375.50	May 2018 KY BLS, Occupation Code: 17-2041
			Total:	**22**	**$ 5,593.70**	

Labor rate data retrieved from: https://www.bls.gov/oes/current/oes_ky.htm

4.3.2.2 Miscellaneous Fixed Costs

Other fixed costs were charged at standard rates based on prior experience and standard process engineering assumptions. The following tables show both the cost basis and the costs for these items.

Table 4-31: Cost Basis for Miscellaneous Fixed Costs

Item	Code	Basis	Units	Base	Units
Maintenance and Supply Material	OH	2	% of Capital / Anum	$ 133,225,581	$ Total
Sample Analysis, QA/QC	OH	10	% of operating labor	$ 3,779	$ / shift
Sales, IP, and R&D Costs	GSA	1	% of Sales / Anum	$ 65,332,537	$ Total
Administrative & Support Labor	GSA	20	% of Direct Labor	$ 3,809,030	$ / Anum
Property Taxes & Insurance	GSA	1	% of Capital / Anum	$ 133,225,581	$ Total

Table 4-32: Cost Summary for Miscellaneous Fixed Costs at Commercial-Scale (500 st/hr.)

Item	Shift Cost	Daily Cost	Annual Cost	Cost Per Total Product kg	Cost Per Plant Feed Short-Ton
Maintenance and Supply Material	$ 2,643.36	$ 7,930	$ 2,664,512	$10.48	$0.66
Sample Analysis, QA/QC	$ 377.88	$ 1,134	$ 380,903	$ 1.50	$0.09
Sales, IP, and R&D Costs	$ 324.07	$ 972	$ 326,663	$ 1.28	$0.08
Administrative & Support Labor	$ 755.76	$ 2,267	$ 761,806	$ 3.00	$0.19
Property Taxes & Insurance	$ 1,321.68	$ 3,965	$ 1,332,256	$ 5.24	$0.33
Total:	**$ 5,422.76**	**$ 16,268**	**$ 5,466,139**	**$21.49**	**$1.36**

4.3.2.3 Electric Power Costs

System electric power requirements were estimated using an itemized analysis of the horsepower *(HP)* rating for each process equipment included in the major equipment list. Power was then charged at a constant rate of $0.07/kW-hr. (kilowatt-hour) assuming a fixed motor/distribution efficiency of 80 percent. The following table shows the total power costs broken down by process module.

Table 4-33: Electric Power Cost Summary for Commercial-Scale Plant (500 st/hr.)

Module ID	Process Module	Total HP	Shift Cost	Daily Cost	Annual Cost	Cost Per Total Product kg	Cost Per Plant Feed Short-Ton
PM1	Crushing and Screening	1,107	$ 577.74	$ 1,733	$ 582,357	$ 2.29	$0.14
PM2	Dry Grinding	5,715	$ 2,983.37	$ 8,950	$ 3,007,237	$ 11.82	$0.75
PM3	Roasting	265	$ 138.31	$ 415	$ 139,419	$ 0.55	$0.03
PM4	Leaching	2,858	$ 1,492.04	$ 4,476	$ 1,503,974	$ 5.91	$0.37
PM5	Rougher SX	583	$ 304.18	$ 913	$ 306,611	$ 1.21	$0.08
PM6	Cleaner SX	37	$19.50	$ 59	$ 19,659	$ 0.08	$0.00
PM7	SX Wash/Saponification	92	$ 48.13	$ 144	$ 48,510	$ 0.19	$0.01
PM8	REE Precipitation	4	$ 2.11	$ 6	$ 2,126	$ 0.01	$0.00
PM9	Water Treatment	4,054	$ 2,116.26	$ 6,349	$ 2,133,189	$ 8.39	$0.53
Total:		**14,716**	**$ 7,681.63**	**$ 23,045**	**$ 7,743,084**	**$ 30.45**	**$1.92**

4.3.2.4 Reagent Costs

Reagent costs were assigned using industry sources and publicly-available bulk prices, and consumption rates were estimated from dosing requirements found during parametric laboratory and pilot tests. The tables shown below include the unit costs for the reagents used in this study, as well as the total reagent costs itemized by process module.

Table 4-34: Unit Reagent Costs

Reagent	Unit	$/[unit]
H_2SO_4	mt	$200
NaOH	mt	$350
HCl	mt	$250
Ascorbic Acid	kg	$2
Oxalic Acid	kg	$1
Kerosene	mt	$400
D2EHPA	kg	$15
Natural Gas	MCF	$5.79

Table 4-35: Reagent Cost Summary for Commercial-Scale Plant (500 st/hr.)

Module ID	Process Module	Shift Cost	Daily Cost	Annual Cost	Cost Per Total Product kg	Cost Per Plant Feed Short-Ton
PM1	Crushing and Screening	$ 300.56	$ 902	$ 302,962	$ 1.19	$ 0.08
PM2	Dry Grinding	$ -	$ -	$ -	$ -	$ -
PM3	Roasting	$ 5,721.77	$ 17,165	$ 5,767,543	$ 22.68	$ 1.43
PM4	Leaching	$ 197,473.80	$ 592,421	$ 199,053,595	$ 782.68	$ 49.37
PM5	Rougher SX	$ 151,094.57	$ 453,284	$ 152,303,329	$ 598.86	$ 37.77
PM6	Cleaner SX	$ 43,355.17	$ 130,066	$ 43,702,016	$ 171.84	$ 10.84
PM7	SX Wash/Saponification	$ 7,149.97	$ 21,450	$ 7,207,168	$ 28.34	$ 1.79
PM8	REE Precipitation	$ 1,223.97	$ 3,672	$ 1,233,763	$ 4.85	$ 0.31
PM9	Water Treatment	$ 18,536.52	$ 55,610	$ 18,684,816	$ 73.47	$ 4.63
	Total:	**$ 424,856.34**	**$ 1,274,569**	**$ 428,255,194**	**$ 1,683.91**	**$ 106.21**

4.3.2.5 *Other Direct Costs*

Waste disposal costs were determined from the process mass balance and standard per ton charges based on published values and prior experience. The processes, and the associated costs needed to neutralize solid and liquid wastes have already been considered in the flowsheet design, and as a result, these costs only reflect those needed to transport and store the material in a long-term waste storage facility. The waste disposal costs are summarized in the following table.

Table 4-36: Waste Disposal Cost Summary for Commercial-Scale Plant (500 st/hr.)

Waste Stream	Daily Generation (st/day)	Disposal Cost ($/st)	Shift Cost	Daily Cost	Annual Cost	Cost Per Total Product kg	Cost Per Plant Feed Short-Ton
Non-Hazardous Solid Waste	11,136	$1.00	$3,712	$11,136	$3,741,696	$14.71	$0.93
Non-Hazardous Precipitate	732	$5.00	$1,220	$3,660	$1,229,760	$4.84	$0.31
Dust/Volatiles	120	$1.00	$40	$120	$40,320	$0.16	$0.01
		Total:	**$4,972**	**$14,916**	**$5,011,776**	**$19.71**	**$1.24**

Refining costs were estimated to be $5.00 per kilogram *(kg)* of REE processed. This cost accounts for the toll refining needed to convert the mixed rare earth product into individual REO. This cost can be predicted with better accuracy once actual sales/tolling agreements are in-place.

4.3.2.6 *Overhead Costs*

Lastly, overhead costs were charged at a standard rate of 20 percent of direct costs. This cost accounts for all other auxiliary facilities and functions needed to support the plant operations. Examples of overhead costs include payroll, accounting, legal, public relations, office supplies, marketing, etc.

4.3.2.7 *Operating Cost Summary*

Final summaries of total operating cost by cost type and cost center are shown in the following tables. The major cost centers for this operation are reagents and consumables, particular those in leaching and rougher solvent extraction.

Table 4-37: Operating Cost Summary by Process Module for Commercial-Scale Plant (500 st/hr.)

Module ID	Process Module	Shift Cost	Daily Cost	Annual Cost	Per kg Cost	Per Ton Cost
PM1	Crushing and Screening	$ 878	$ 2,635	$ 885,319	$ 3.48	$ 0.22
PM2	Dry Grinding	$ 2,983	$ 8,950	$ 3,007,237	$ 11.82	$ 0.75
PM3	Roasting	$ 5,860	$ 17,580	$ 5,906,963	$ 23.23	$ 1.47
PM4	Leaching	$ 198,966	$ 596,898	$ 200,557,569	$ 788.60	$ 49.74
PM5	Rougher SX	$ 151,399	$ 454,196	$ 152,609,941	$ 600.07	$ 37.85
PM6	Cleaner SX	$ 43,375	$ 130,124	$ 43,721,676	$ 171.91	$ 10.84
PM7	SX Wash/Saponification	$ 7,198	$ 21,594	$ 7,255,679	$ 28.53	$ 1.80
PM8	REE Precipitation	$ 1,226	$ 3,678	$ 1,235,889	$ 4.86	$ 0.31
PM9	Water Treatment	$ 20,653	$ 61,958	$ 20,818,006	$ 81.86	$ 5.16
PM10	REO Refining	$ 1,215	$ 3,645	$ 1,224,663	$ 4.82	$ 0.30
NA	Unallocated	$ 105,937	$ 317,810	$ 106,784,226	$ 419.88	$ 26.48
	Total:	**$ 539,690**	**$ 1,619,069**	**$ 544,007,167**	**$ 2,139.05**	**$ 134.92**

Table 4-38: Operating Cost Summary by Accounting Code for Commercial-Scale Plant (500 st/hr.)

Cost Center	Shift Cost	Daily Cost	Annual Cost	Cost Per Total Product kg	Cost Per Plant Feed Short-Ton
Operating Labor	$3,779	$11,336	$3,809,030	$14.98	$0.94
Technical Labor	$1,815	$5,445	$1,829,419	$7.19	$0.45
Power/Utilities	$7,682	$23,045	$7,743,084	$30.45	$1.92
Lease Agreements	$1,215	$3,645	$1,224,663	$4.82	$0.30
Consumables/Reagents	$424,856	$1,274,569	$428,255,194	$1,683.91	$106.21
Waste Disposal	$4,972	$14,916	$5,011,776	$19.71	$1.24
Plant Overhead	$92,970	$278,909	$93,713,276	$368.48	$23.24
General Sales, and Administration	$2,402	$7,205	$2,420,725	$9.52	$0.60
Total:	**$539,690**	**$1,619,069**	**$544,007,167**	**$2,139.05**	**$134.92**

4.3.3 Cost/Benefit Analysis that Highlights Annual Production Projections, Cash Flow Forecasts and Other Key Economic Performance Indicators (IRR, NPV, Payback)

Revenue from the commercial-scale facility was estimated by using market REE prices and the expected REE production for various products. In this analysis, the plant was assumed to produce four individual REE products:

- Scandium oxide (Sc_2O_3)
- Dysprosium oxide (Dy_2O_3)
- Gadolinium oxide (Gd_2O_3)
- Mixed Rare Earth Oxides (REOM)

These four products were valued at 100 percent of market price for revenue estimates. All other REEs were assumed to be produced in a MREO product, which was valued at 65 percent of the market price for the individual constituents. The market prices used in this study include those provided by DOE as well as additional prices determined from reports compiled by **Asian Metal, Inc.** ***(AMI)*** The overall price deck along with the feed based "basket price"[8] is shown in the following table.

Table 4-39: REE Price Deck Used for Commercial-Scale Economic Analysis

Element	REE Feed Dist. (%)	REE Oxide Price ($/kg)	REE Price (Elemental Basis) ($/kg)	Price Source Code
Scandium (Sc)	5.40	$ 4,200	$ 6,442	[1]
Yttrium (Y)	9.31	$ 6	$ 8	[1]
Lanthanum (La)	16.26	$ 2	$ 2	[1]
Cerium (Ce)	34.56	$ 2	$ 2	[1]
Praseodymium (Pr)	6.50	$ 52	$ 63	[1]
Neodymium (Nd)	15.33	$ 42	$ 49	[1]
Samarium (Sm)	3.53	$ 2	$ 2	[2]
Europium (Eu)	0.46	$ 150	$ 174	[1]
Gadolinium (Gd)	2.84	$ 32	$ 37	[1]
Terbium (Tb)	0.25	$ 400	$ 471	[1]
Dysprosium (Dy)	1.73	$ 230	$ 264	[1]
Holmium (Ho)	0.50	$ 53	$ 61	[2]
Erbium (Er)	1.55	$ 34	$ 39	[1]
Thulium (Tm)	0.35	Not Available	Not Available	[2]
Ytterbium (Yb)	1.13	$ 29	$ 33	[2]
Lutetium (Lu)	0.30	$ 797	$ 906	[2]
		REE Basket Price ($/kg):	$ 372.8	
		REE Contained Value ($/ plant feed short ton):	$ 93.2	

[1] NETL Standard Price Deck
[2] Asian Metal

An overall economic assessment for the commercial-scale plant was conducted by integrating the capital cost estimates, the operating cost estimates, the revenue estimates, and the global financial assumptions – all described in the prior sections of this chapter. The following table includes the key output parameters from this overall analysis. Unfortunately, these results show that the overall REE recovery process is not economically viable, as the process costs greatly exceed the processing revenue in each year. While many factors contribute to this outcome, the final unprofitable result can be principally attributed to the high chemical consumption values, the low REE feed grades, and the low REE recovery values, which all contribute to high operating costs on a $/kg basis. In addition, low market prices for several REE compounds also limits the economic feasibility of the venture. Lastly, the final project flowsheet analyzed in this chapter does not produce marketable byproducts other than the three refined REEs and the bulk MREO product. Modifications to the flowsheet or the feedstock selection may provide an opportunity to produce other byproducts such as recoverable coal or non-REE critical materials whose sales will offset the cost of the REE concentration process. If produced in a synergistic manner, these additional sources of revenue may greatly improve the economic viability of the REE recovery process.

---

[8] With respect to REEs, a deposit's "basket price" (expressed in $US/kg) is defined as the value ($) of one-unit mass (1 kg) of separated rare earth oxides *(SREOs)*, in which those SREOs are in the same proportion (i.e., distribution) as the deposit. Note, the term REO is often used instead of REE as this is the form that the final products are priced and sold in the global market, regardless of whether the specific companies' business plan is to produce separated oxides.

Table 4-40: Summary of Economic Indicators for Commercial-Scale Plant (500 st/hr.)

Parameter	Unit	Value
Technical Results		
Plant Feed Rate	sTPH*	500
Plant Feed Grade	ppm	357
Overall REE Production	kg/hr.	39.3
Sc_2O_3 Production	kg/hr.	1.9
Dy_2O_3 Production	kg/hr.	0.4
Gd_2O_3 Production	kg/hr.	0.5
MREO Production	kg/hr.	36.6
Economic Results		
Overnight Capital Cost	$	133,225,581
Constant Dollar OpEx	$/yr.	544,007,167
(unit conversion)	$/st feed	134.92
(unit conversion)	$/kg REE	2,139.05
Constant Dollar Revenue	$/yr.	65,332,537
Net Present Value (@10% discount)	$	(3,522,801,107)
Internal Rate of Return	%	N/A
Payback Period	Operating Years	N/A

**sTPH = short tons per hour*

4.3.4 Sensitivity Analysis on Pricing, Feedstock Quality, Product Purity, CapEx, Open, Transportation Costs, etc.

A sensitivity analysis was conducted to assesses and quantify the important technical and economic factors that influence project economic potential in terms of the NPV. This sensitivity analysis has been conducted to investigate potential pathways to commercial viability and provide priorities for future research and development activities.

4.3.4.1 Analysis of Feed Rate

The following figure shows the overall plant capital cost as a function plant feed rate. This curve follows a clear power law[9] trend, with an overall exponent of approximately 0.6. This result is expected given both the form of the individual cost models and general process engineering principles (i.e. the six-tenths rule[10]).

---

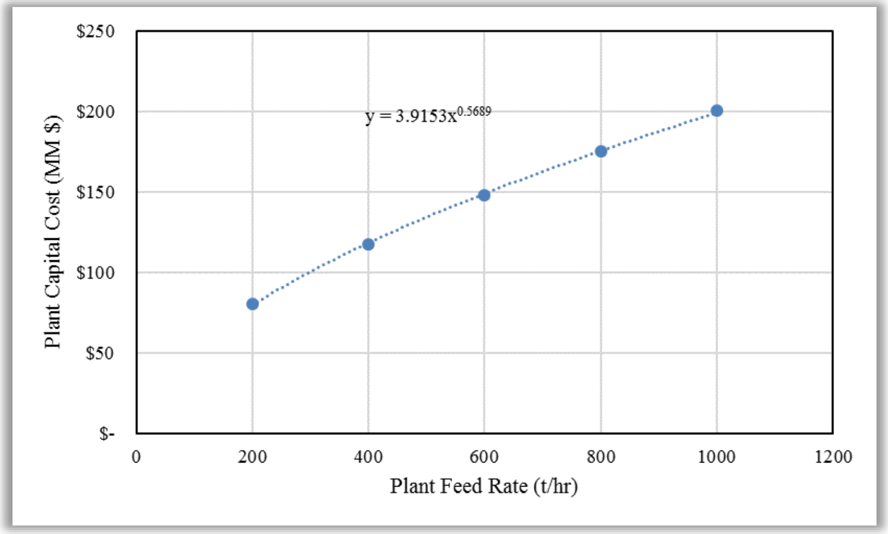
[9] A "power law" is a functional relationship between two quantities, where a relative change in one quantity results in a proportional relative change in the other quantity.

[10] Turton et al., and other authors in various texts, give this relationship between purchased cost and an attribute related to units of capacity as shown in the formula below.

$$\frac{C_a}{C_b} = \left(\frac{A_a}{A_b}\right)^n \qquad (eq. 3.1)$$

The value for "n" is often around 0.6. Using this common value for n is referred to as the "six-tenths rule".

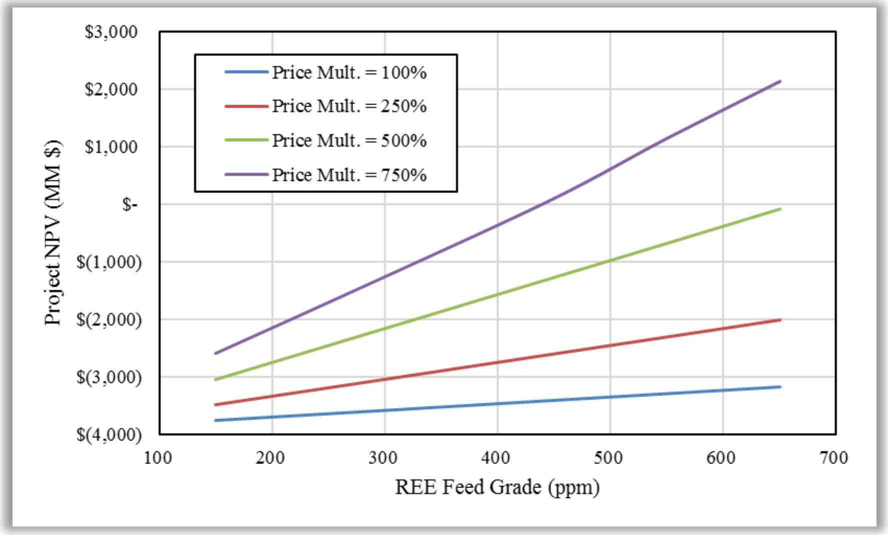
Figure 4-1: Sensitivity of Plant Capital Cost with Respect to Plant Feed Rate



4.3.4.2 *Analysis of Feed Grade and REE Price*

In most process operations, the plant feed grade and the overall sales price are two of the most influential parameters on overall profitability. The proposed REE extraction facility follows a similar trend, as these two values have a significant influence on final economic indicators. To illustrate this relationship, the following figure shows project NPV as a function of REE feed grade for four unique price scenarios. In this example, the price scenario is depicted as an overall multiple applied to the standard prices used in the baseline analysis. This plot shows the various "cut-off grade" needed to attain positive NPV at each price scenario. For the standard pricing scenario, the project will not be profitable, even at fairly high plant feed grades. At higher pricing scenarios (i.e. 500 percent and 750 percent), the project can be profitable at moderate plant feed grades (e.g., 650 ppm at a 500 percent price multiple and 440 ppm at a 750 percent price multiple. The data shows that for the simulated condition (i.e., 357 ppm feed grade), a price multiplier much greater than 750 percent would be needed to break even.

Figure 4-2: Sensitivity of Plant Feed Grade and REE Price Multiple with Respect to Project NPV

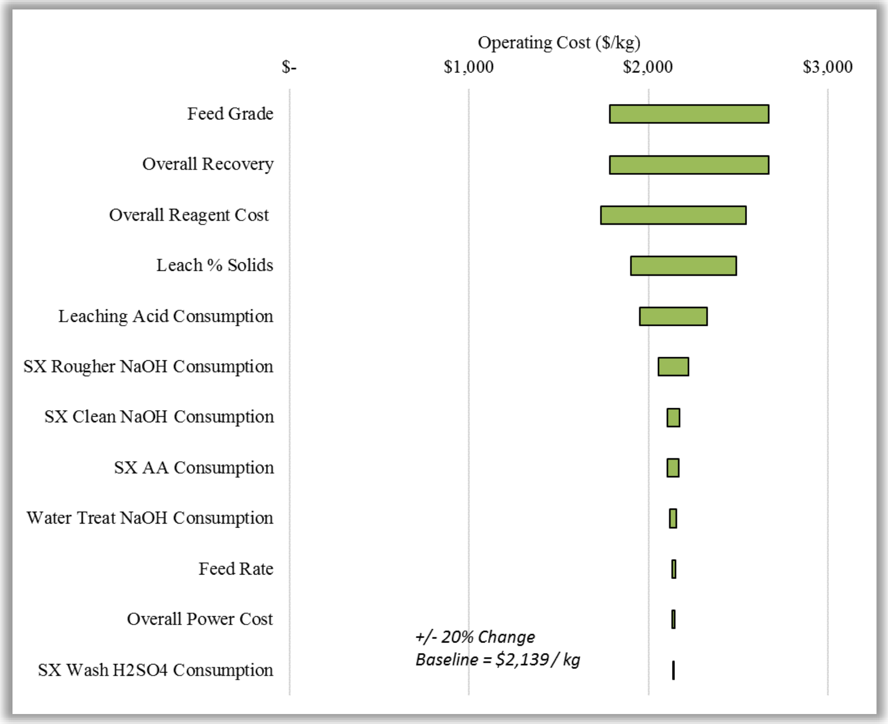


4.3.4.3 Analysis of Technical Performance Measures

To analyze the sensitivity of the project profitability to key technical variables, a tornado diagram[11] approach was used, whereby each input variable was adjusted one-at-a-time at a fixed percentage. For this analysis, each variable was adjusted +/-20 percent, and the operating cost ($/kg REE) was monitored as the key economic output. Input variables to this analysis include the overall process recovery variables, overall reagent and power costs, plant feed grade, plant feed rate, itemized reagent consumption, and leaching percent solids. The results are shown in the following figure and they indicate that process recovery, particularly in the leaching and rougher solvent extraction stages, are key sensitive parameters.

The tornado analysis confirms that feed grade, process recovery, and reagent cost (or reagent consumption, which would be mathematically identical) are the most significant variables that influences overall profitability. Interestingly, leaching percent solids were also an important technical parameter, and this analysis shows that this value must be kept relatively high to reduce the size and cost of the downstream solvent extraction units.

Figure 4-3: Tornado Analysis Showing the Sensitivity of Plant Operating Cost with respect to Several Technical Performance Measures

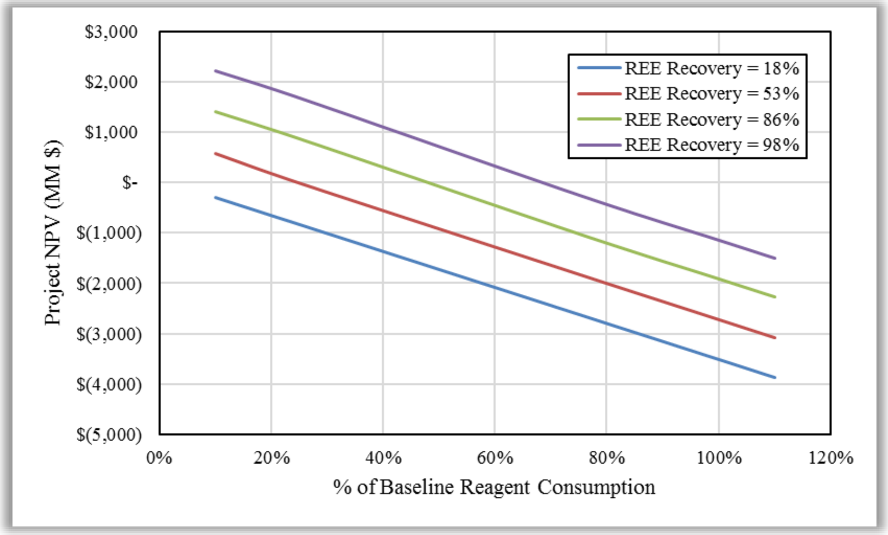


---

[11] A tornado diagram is a common tool used to depict the sensitivity of a result to changes in selected variables. It shows the effect on the output of varying each input variable at a time, keeping all the other input variables at their initial (nominal) values. Typically, an analyst will choose a "low" and a "high" value for each input. The result is then displayed as a special type of bar graph, with bars for each input variable displaying the variation from the nominal value. It is standard practice to plot the bars horizontally, sorted so that the widest bar is placed at the top. When drawn in this fashion, the diagram takes on the appearance of a tornado, hence its name.

Given the paramount significance of overall recovery and overall reagent cost, a separate analysis was conducted to determine the influence of these values on this project's NPV. The results of this analysis are shown graphically in the following figure and indicate that drastic improvements to both parameters is needed to break even with respect to NPV. For example, at an overall recovery of 86 percent (approximately five times the current recovery value), a 55 percent reduction in reagent cost (100 percent to 45 percent of baseline reagent consumption) would be needed to break even. It should be noted however, that this analysis was conducted while holding other critical values (such as REE feed grade and REE price) at their nominal baseline values.

Figure 4-4: Sensitivity of Reagent Consumption with Respect to this Project NPV



4.3.4.4 *Potential Pathway to Profitability*

Altogether, the foregoing sensitivity analysis indicates that a potential pathway to profitability must include:

1. improved control of the feedstock roasting temperature
2. increases to the plant feed grade to reflect more recent information;
3. reduction in reagent consumption; and/or
4. improvements to process recovery.

To illustrate the combined influence of these three parameters, an "ideal case" model scenario was evaluated. In this scenario:

1. the overall reagent consumption was reduced by 75 percent relative to baseline values,
2. the REE feed grade was increased to 450 ppm, and
3. the overall recovery was increased to 53 percent (three times baseline values).
4. All other input parameters and model assumptions remain unchanged from their baseline values described earlier in the chapter.

(Despite the relatively high influence of REE price, this factor was not considered in the ideal case analysis, since these prices are not within the control of plant operator [i.e., guaranteed price subsidies were not considered a viable option to attain profitability]).

The results of this analysis are shown in the following table and indicate that these changes greatly improve the economic outcomes for the project.

Table 4-41: Economic Indicators for Commercial Plant under Improved Input Conditions

Parameter	Unit	Value
Technical Results		
Plant Feed Rate	TPH	500
Plant Feed Grade	ppm	450
Overall REE Production	kg/hr.	148.5
Sc_2O_3 Production	kg/hr.	7.1
Dy_2O_3 Production	kg/hr.	1.7
Gd_2O_3 Production	kg/hr.	1.9
MREO Production	kg/hr.	137.8
Economic Results		
Overnight Capital Cost	$	132,813,905
Constant Dollar OPEX	$/yr.	189,394,298
(unit conversion)	$/stfeed	46.97
(unit conversion)	$/kg REE	197.51
Constant Dollar Revenue	$/yr.	242,876,222
Net Present Value (@10% discount)	$	153,265,686
Internal Rate of Return	%	26%
Payback Period	Operating Years	5.5

4.4 Key Findings

In general, the results from the techno-economic assessment can be used to derive several key findings related to the economic performance of the proposed extraction process. These key findings include:

1. The process equipment to be used in the pilot-scale facility represents a total equipment cost of over $10.3 million (including $1.8 million in used equipment and $8.5 million in new equipment), but the used equipment will require an additional investment of $1.5 million to disassemble, transport, rehabilitate, refurbish, reassemble, and restart. Construction of the new process buildings is expected to be $1.6 million, bringing the total plant value to $13.4 million.
2. Capital cost estimates at both the pilot-scale and commercial-scale show that the roasting operation is the most significant capital investment category, representing nearly one-third of the total capital expenditure.
3. A scale-factor economic analysis of a potential 500 st/hr. production facility shows that the proposed process circuitry is too costly to be economically viable without government subsidy or significant technical improvements.
4. Sensitivity analysis of the cost model shows that process recovery, plant feed grade, and reagent consumption are critical components influencing overall profitability.

5. As such, additional research and development should seek to identify higher-grade REE resources, as well as process alternatives that reduce acid and base consumption while improving recovery.
6. Additional revenue from other byproducts such as recoverable coal or critical materials should be considered.

4.5 Acknowledgments

The following individuals contributed significantly to the database, compilation, review and editing of this chapter:

Aaron Noble, Ph.D., Associate Professor, Virginia Tech, Department of Mining and Minerals Engineering, Blacksburg, Virginia, who took the lead in compiling this chapter and generated the workbooks, tables, and figures that are a part of the narrative.

Dr. Gerald H. Luttrell Ph.D., Massey Professor, Virginia Polytechnic Institute & State University, Department of Mining & Minerals Engineering, Blacksburg, Virginia, who contributed significantly to compilation of the Excel-based financial model that is the foundation of this chapter.

Joshua Werner, Ph.D, P.E., Mechanical Engineer, Assistant Professor, University of Kentucky, Department of Mining Engineering, Lexington, Kentucky who assembled, tabulated and reported the capital expenditure estimates shown in this chapter.

Dr. Rick Q. Honaker Ph.D., Professor of Mining Engineering, University of Kentucky, College of Engineering, Department of Mining Engineering, Lexington, Kentucky, who took the lead in the research and development of the processing concepts addressed in this chapter.

Mr. Antonio Cimetta, President and CEO of Cimetta Engineering & Construction Co, Inc., of Tucson, Arizona, who facilitated site visits to a Pilot Plant in Arizona.

Mr. Jared Garland, Project Engineer, Cumberland Mine Services, Inc., of Totz, Kentucky, subcontractor to Project MMA 29956 who provided advice on the transportation and re-assembly of major components of the Arizona Plant to a proposed industrial park near Chavies, Kentucky.

Mr. Roger A Daugherty, President, Darco Energy Management Corporation of Buffington, South Carolina, a certified machinery and equipment appraiser, who provided insights to the market value of the various components of the Arizona Plant.

Steven A. Keim, Ph.D., P.E., Senior Mining Engineer and Senior Vice President, Marshall Miller & Associates, Inc., Blacksburg, Virginia, Office, principal investigator of project MMA 29956, who contributed his project management expertise in guiding the composition of this chapter and scheduled support staff to assure its professional standing.

Hans E. Naumann, Jr., P.E., Principal Engineer Marshall Miller & Associates, Inc., Lexington, Kentucky, Office, who managed the overall coordination of the tasks associated with the reporting of the results of MMA 29956 and who contributed his editorial and formatting expertise to this chapter.

4.6 Special Note and Addendum to the Techno-Economic Analysis

4.6.1 Justification of the 357 ppmw Feed Grade of the Economic Model

Throughout the course of the project, the research team acquired and analyzed numerous coal and rock samples from the Blue Diamond/Blackhawk Leatherwood facility. These samples included plant rejects samples, channel samples, and core samples as well as various size and density fractions derived from the original materials. A summary of this sampling effort, including the type of sample, amount of sample collected, and the TREE value is included in the following table. Overall, these values range from approximately 150 to 375 ppmw. The value used in the techno-economic analysis described in this chapter of the report, 357 ppmw, represents a composite of a coarse rejects belt sample (RE10-18) collected on 11/16/2018. The REE analysis for this material was conducted on a size-by-density basis (±½ inch, ±2.00 SG), and the final value represents the mass weighted average of the subsamples.

Table 4-42: REE (ppmw) Data Source Inventory (Fire Clay Seam)

Sample Identification Number	Source	Sample Collection Date	Type of Sample	Quantity (lbs., ft, Inches, %)	Analysis By:	TREE in Refuse (PPM TREE+Y+Sc, Whole Sample Basis)
KYJRLW0003	"Leatherwood"	7/29/2014	Coarse Refuse From Plant Feed	726+/- lbs.	DOE Contractor	301.77
KYJRLW0004		7/29/2014	Fine Refuse from Plant Feed			227.99
RE01-18	Blue Diamond Mining Company No. 76 Plant (a.k.a. Blackhawk - Leatherwood)	8/2/2018	+1/4-inch Refuse	52.63%	Mineral Labs, Inc.	288.76
			1/4 x 100 mm Refuse	30.94%	"	229.74
			100mm x 0 Refuse	16.43%	"	232.63
			Composite	**100.00%**	- -	**261.28**
RE02-18	"	8/2/2018	+1/4-inch Refuse	45.28%	Mineral Labs, Inc.	209.83
			1/4 x 100 mm Refuse	33.15%	"	252.84
			100mm x 0 Refuse	21.57%	"	152.46
			Composite	**100.00%**	- -	**211.71**
RE03-18	"	8/2/2018	Coal (Low Ash)	8.65 lbs.	Mineral Labs, Inc.	51.61
RE05-18	"	8/2/2018	Coal (High Ash)	?	Mineral Labs, Inc.	197.70
RE06-18	"	8/8/2018	Fine Middling Product	2.23 kg	Mineral Labs, Inc.	88.26
RE08-18	"	8/24/2018	Coarse Refuse Float 2.00	5.83%	Mineral Labs, Inc.	26.45
			Coarse Refuse Sink 2.00	94.17%	"	243.38
			Composite	**100.00%**	- -	**230.73**
RE09-18	"	9/14/2018	Slurry	?	Mineral Labs, Inc.	272.81
RE10-18	"	11/16/2018	Coarse Refuse + 1/2, Float 2.00	19.51%	Mineral Labs, Inc.	370.59
			Coarse Refuse + 1/2, Sink 2.00	37.28%	"	349.04
			Coarse Refuse 1/2x 0, Float 2.00	2.15%	"	357.35
			Coarse Refuse 1/2x 0, Sink 2.00	41.06%	"	356.89
			Composite	**100.00%**	- -	**356.64**
DPP 5, 6, 7	"	11/16/2018	Coarse Refuse	46,000 +/- lbs.	University of KY	308 (average)
?	"	12/10/2018	Coarse Refuse	20000 +/- lbs.		?

Sample Identification Number	Source	Sample Collection Date	Type of Sample	Quantity (lbs., ft, Inches, %)	Analysis By:	TREE in Refuse (PPM TREE+Y+Sc, Whole Sample Basis)
RELW-18-01	"Study Area" on Cutshin Creek, Leslie County, KY	12/4/2018	Core (Rock)	2.62 ft	KY Geological Survey	341.60
RELW-18-02	"	12/7/2018	Core (Rock)	3.03 ft	"	376.09
RELW-18-03	"	12/14/2018	Core (Rock)	2.45 ft	"	348.71
RELW-19-01	"	1/25/2019	Core (Rock)	3.27 ft	"	357.02
RECH-A	Mine 89	1/13/2017	Channel Sample (Rock)	0.97 ft	KY Geological Survey	334.93
RECH-1	Mine 81	2/6/2019	Channel Sample (Rock)	1.67 ft	"	354.44
RECH-2	Mine 89	2/6/2019	Channel Sample (Rock)	3.10 ft	"	265.40

When compared to other refuse samples shown in the preceding table, the 357 ppmw samples represent one of the higher values in the distribution. Nevertheless, this value is justified for cost modeling purposes as it indirectly and unintentionally reflects how a change in mining practice may influence the grade of REEs extracted from a mine. In the days preceding the sampling event, the mine surveyor identified a substantial total quantity of “rock” at the mining face. A cursory review of the mine maps suggest that similar events may not have coincided with the other refuse samples, thus prompting lower REE assay values. Moreover, the resource study depicted in Chapter 2 of this report estimated an overall average REE assay of 341 ppmw for the entire study area. However, this analysis was based on a limited number of core and channel samples, and it presumed a constant 60-inch mine opening. Prior to any commercialization, significantly more core drilling tests will be undertaken, and the subsequent mine plan will likely deviate to the recovery of rock strata with increased REE concentration. The mine opening will likely exceed 60 inches in some places. Altogether, the 357 ppmw represents the possibility of what may be achieved, it is nevertheless grounded by real data from actual samples taken during this study.

4.6.2 Influence of Optimal Roasting Conditions

The techno-economic analysis presented in the preceding sections of this report (hereafter referenced as the “*Baseline Case”*) used test data from a bulk sample of a commercially roasted feedstock that was later determined to be significantly overheated. All of the model input parameters for this case were based on actual performance measures acquired from pilot-scale testing of this commercially roasted feedstock. Subsequent laboratory testing, however, revealed that the poorly roasted material led to both high chemical consumption and low REE recovery. These two negative outcomes prompted very high production costs and the poor economic outcomes reported in the prior sections of this chapter.

Despite these poor pilot-scale results, additional laboratory data (depicted in Chapter 3) show that superior outcomes can be achieved by carefully controlling the roasting temperature during leaching. When properly roasted, fair leaching recoveries can be achieved even at relatively low acid doses, indicating that there is an optimal leaching pH which balances REE recovery with respect to the costs of acid doses. To evaluate this tradeoff, a supplemental techno-economic analysis was conducted using leaching recovery and chemical consumption values derived from laboratory testing. In particular, a Weibull growth model, fit to the relevant experimental data, was used to define the relationship between leaching acid concentration and leaching recovery. The research team then analyzed both a high-recovery/high-acid concentration leaching scenario (0.5 M, 53 percent TREE recovery) as well as a low-recovery/low-acid concentration leaching scenario (0.05M, 24 percent TREE recovery) of properly roasted feedstock in relation to the poorly roasted feedstock of the base case. The results of these analyses are shown in the following table.

Table 4-43: Project Economic Summary for Material Processed under Differing Roasting and Acid Application Conditions

Parameter	Unit	Base Case[1]	Case A[2]	Case B[3]
Technical Results				
Plant Feed Rate	TPH	500.00	500.00	500.00
Plant Feed Grade	ppm	357.00	357.00	357.00
Overall REE Production	kg/hr.	39.30	65.39	39.3.
Sc_2O_3 Production	kg/hr.	1.90	4.90	1.90
Dy_2O_3 Production	kg/hr.	0.40	0.67	0.40
Gd_2O_3 Production	kg/hr.	0.50	0.34	0.50
MREO Production	kg/hr.	36.60	59.49	36.60
Economic Results				
Overnight Capital Cost	$	133,225,581.00	133,441,060.35	133,225,581
Constant Dollar OPEX	$/yr.	544,007,167.00	545,534,880.79	217,782,406
(unit conversion)	$/t feed	134.92	135.30	54.01
(unit conversion)	$ /kg REE	2,139.05	1,298.79	856.33
Constant Dollar Revenue	$/yr.	65,332,537.00	168,004,021.38	65,332,537
Net Present Value (@10% discount)	$	(3,522,801,107.00)	(2,863,280,245.21)	(1,244,501,311)
Internal Rate of Return	%	N/A	N/A	N/A
Payback Period	Operating Years	N/A	N/A	N/A

1. Base Case = Poor roasting performance, high acid concentration in leaching, low REE Recovery.

2. Case A = Optimal roasting, high acid concentration in leaching, high REE recovery.

3. Case B = Optimal roasting, low acid concentration in leaching, low REE recovery.

Although the results depicted by Case B appear to be counterintuitive, the low-recovery/low-acid concentration leaching scenario is economically favored because the lower acid use prompts a sequence of reduced chemical consumption rates throughout the leaching, SX, and water treatment circuits. Put another way, a simple cost-benefit test does not support an intensive high-acid leaching operating scenario That observation aside, both cases A and B represent a considerable improvement to processing costs and overall project profitability when compared to the Baseline Case. While the reductions in Cases A and B are not sufficient to prompt economic viability, the results do represent significant gains in processing knowledge and provide some input on the appropriate pathway to process optimization. .

Chapter 8.0

BENCHMARK PROCESS AUDITS

Chapter 8.0
Benchmark Process Audits

TABLE OF CONTENTS

8.1 Introduction **1**
8.2 Benchmark Visit Sites **4**
8.2.1 MP Materials, Mountain Pass, California 4
8.2.2 Previously Operated Pilot Plant (Trip 1) 10
8.2.2.1 Introduction **10**
8.2.2.2 Process Description **11**
8.2.2.3 Process and Construction Drawings **12**
8.2.2.4 Equipment and Condition **25**
8.2.3 Previously Operated Pilot Plant (Trip No. 2) 31
8.2.3.1 Appraisal of the Arizona Pilot Plant **31**
8.2.3.2 Salvage Equipment Power and Original Cost **31**
8.3 Acknowledgments **35**

TABLES (IN THE TEXT)

Table 8-1: Summary of Benchmark Process Audits 4
Table 8-2: List of Original P&ID Diagrams for the Previously Operated Plant 23
Table 8-3: Subsequent Drawings Indicating Modifications to the Roasting Gas Scrubbing and Burner Systems of the Previously Operated Pilot Plant 23
Table 8-4: General Layouts and Elevations of the Crushing/Grinding and Roasting Sections of the Previously Operated Pilot Plan 23
Table 8-5: Electrical and Control Drawings of the Previously Operated Pilot Plant 24
Table 8-6: Steel and Foundation Drawings of the Previously Operated Pilot Plant 24
Table 8-7: Roaster-Specific Drawings of the Previously Operated Pilot Plant 25
Table 8-8: Appraised Value, Effective February 12, 2019, of the Previously Operated Pilot Plant 31
Table 8-9: List of Proposed Previously Operated Pilot Plant Salvaged Equipment Summary Data showing Description, Equipment Identification., Referenced Drawing, Power, Original Cost and Weight 32

FIGURES (IN THE TEXT)

Figure 8-1: General Area Map Depicting the Disposition of the MP Materials' MP Mining Operations, LLC Site at Mountain Pass, California 2
Figure 8-2: General Location Map of the Previously Operated Nickel-laterite Ore Hydro-met Pilot Plant near Tucson, Arizona 3
Figure 8-3: Previously Operated Nickel-laterite Ore Hydro-met Pilot Plant near Tucson, Arizona 4
Figure 8-4: Photograph of Mountain Pass Ore 6
Figure 8-5: Ball Mill at Mountain Pass 7
Figure 8-6: Historic Solvent Extraction Train 8
Figure 8-7: Alternate Solvent Extraction Vessel Design 8
Figure 8-8: Newest Solvent Extraction Train 9

Figure 8-9: Overlook of Separations and Chemical Facility 9
Figure 8-10: Previously Operated Pilot Plant - High Level Flow Sheet 11
Figure 8-11: Previously Operated Pilot Plant Drawing 3166-EF-11. Crushing and Drying 14
Figure 8-12: Previously Operated Pilot Plant Drawing 3166-EF-21 - Roaster 15
Figure 8-13: Previously Operated Pilot Plant Drawing 3166-EF-22 - 2-Stage Roaster Scrubbing System 16
Figure 8-14: Previously Operated Pilot Plant Drawing 25801-PI-1 - Modified Dry Scrubbing System 17
Figure 8-15: Previously Operated Pilot Plant Drawing 3166-EF-31 - Leaching Circuit (Section 3) 18
Figure 8-16: Previously Operated Pilot Plant Drawing 3166-EF-41 - Counter Current Decantation (Section 4) 19
Figure 8-17: Previously Operated Pilot Plant Drawing 3166-EF-51 - Counter Current Decantation (Section 4) 20
Figure 8-18: Previously Operated Pilot Plant Drawing 3166-EF-61 - Solvent Extraction Pre-Treatment 21
Figure 8-19: Previously Operated Pilot Plant Drawing 3166-EF-71 - Solvent Extraction and Electrowinning of Ni 22
Figure 8-20: Aerial View of Jaw Crusher and Supporting Conveyors of the Previously Operated Pilot Plant 26
Figure 8-21: Rotary Kiln Dryer Showing Baghouse and Conveyors of the Previously Operated Pilot Plant 26
Figure 8-22: Screen and Roller Crusher with Supporting Conveyors of the Previously Operated Pilot Plant 27
Figure 8-23: Panorama Showing the Top of Roaster, Gas Scrubbing, and Ore Bins of the Previously Operated Pilot Plant 27
Figure 8-24: Top Shell of the Roaster of the Previously Operated Pilot Plant Showing Corrosion due to HCl 28
Figure 8-25: Alternate View of the Roaster Showing Shell Damage. 28
Figure 8-26: Interior View of Roaster of the Previously Operated Pilot Plant Showing Example of Serviceable Rakes 29
Figure 8-27: Lower Hearth Gear and Conditions of the Previously Operated Pilot Plant 29
Figure 8-28: Picture Showing Gas Scrubbing Equipment of the Previously Operated Pilot Plant 30
Figure 8-29: Aerial View of Gas Scrubbing Equipment and Portions of the Leaching Circuit of the Previously Operated Pilot Plant 31

APPENDIX

8-1 .... DARCO Appraisal

8.1 Introduction

The Statement of Project Objectives included the performance of benchmark process audits to provide:

1. an understanding of industrial "best practices" in similar plants and
2. the ability to test design assumptions against operational experience.

The benchmark audits were to include:

1. An operating rare earth elements' (REE) producer utilizing a similar proposed flowsheet;
2. Toll processor; and
3. Feedstock provider.

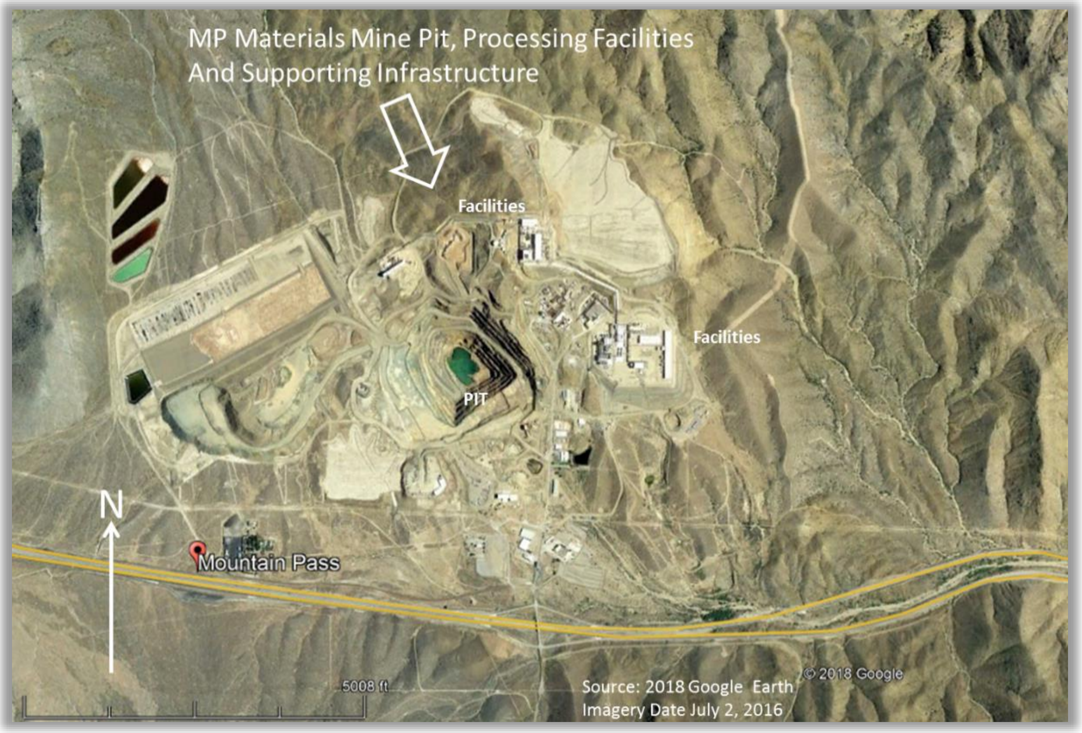
Members of this research team spent three days in February 2018 at the **Mountain Pass Rare Earth Mine *(Mountain Pass)*** located near the unincorporated community of Mountain Pass in San Bernardino County, California. The Mountain Pass operation is comprised of an open cast REE mine, physical concentration plant, REE recovery plant and a refining circuit that produces individual REE concentrates. It was been operated by a privately held **Molycorp Minerals LLC *(Molycorp)***, when the operation filed for Chapter 11 bankruptcy. Affiliates of two investment fund advisors and a Chinese entity took over the operation in July 2017 under the name of **MP Materials**[1]. The mine operating entity is listed as **MP Mining Operations, LLC (*MPMO*)**[2]. At the time of MM&A research team's visit in February 2018, MPMO was entering its start-up period. Notwithstanding these events, the visit satisfied the objectives of benchmark auditing both a REE producer and a feed stock provider.

---

[1] Sources: https://en.wikipedia.org/wiki/Mountain_Pass_rare_earth_mine and https://mpmaterials.com/

[2] Sources: Nuclear Regulatory Commission, "Order Approving Direct Transfer of Two Export Licenses", November 27, 2017, https://www.nrc.gov/docs/ML1732/ML17325B702.pdf and "Mountain Pass Sells for $20.5M" https://www.mining

Figure 8-1: General Area Map Depicting the Disposition of the MP Materials' MP Mining Operations, LLC Site at Mountain Pass, California



Source: 2018 Google Earth
Imagery Date July 2, 2016

A trip was not taken to a toll processor for several reasons primarily because of the lack of refining capacity in the United States. For instance, during the execution of Project MMA29956, the researchers became aware that only MP Materials, **Rare Earth Salts**[3], and **Ucore Rare Metals, Inc. (*Ucore*)**[4] were known to possess the capacity needed for processing REEs for the improvement of purity. In pursuit of that matter, a number of discussions were held with Ucore. However, Ucore would not allow research team members to visit its REE processing facility in Salt Lake City, Utah. Next, although it was originally proposed that the **Blue Line Corporation**[5] ***(Blue Line)*** would serve as the toll refiner for the project team it was discovered through conversations with Blue Line that the company did not possess the required in-house refining capability to sustain a tolling operation. Rare Earth Salts was not visited as their process differed from the circuit proposed by the team. (Due to process economics requiring vertical integration, the team decided to incorporate refining circuits as part of the project design effort, a concept not compatible with Rare Earth Salts.)

---

[3] Source: Rare Earth Salts is a privately held industrial and applications technology company focused on the separation and refining of all 16 REEs to high purity from various feedstocks; 5331 Element Ave. Beatrice, NE 68310, Tel: +1 402-806-4400, https://rareearthsalts.com/

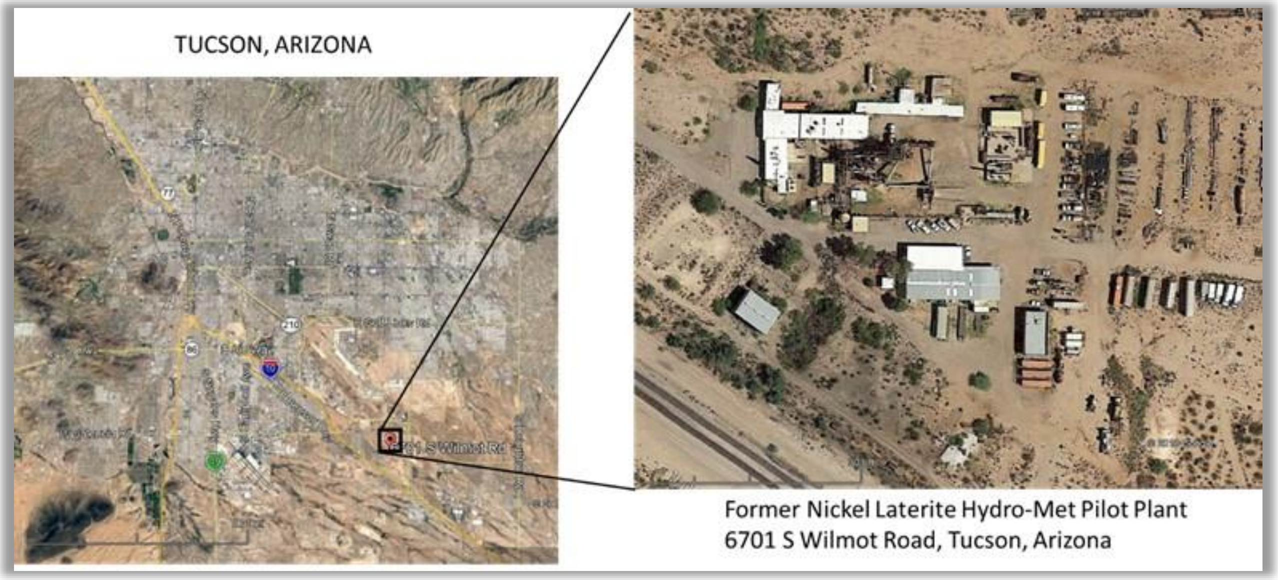
[4] Source: Ucore Rare Metals Inc. is a development-phase company focused on rare metals resources, extraction and beneficiation technologies; 210 Waterfront Drive, Suite 106 Bedford, Nova Scotia, Canada B4A 0H3 Tel: (902) 482-5214, www.ucore.com

[5] Blue Line Corp, 3443 East Commerce Street, San Antonio, Texas 78220-1322, Tel: +1 (210) 225-0400, http://www.bluelinecorp.com

Given the poor results in locating a cooperative toll processor, the researchers elected to undertake two additional visits to an idle pilot plant located near Tucson, Arizona ("Previously Operated Pilot Plant")[6]. The first trip to the Plant was to identify the equipment and process, determine plant suitability and to determine the availability of the plant for purchase. The second trip was to determine to the extent possible the detailed engineering parameters associated with the pilot plant and its appraised value. In aggregate, both visits, bolstered by the assistance of a very cooperative owner of the facility, provided surprisingly detailed knowledge of relevant process units and circuits as well as potential access to equipment that could be used in Phase 2 of the project. Specifically, the trips to the Arizona plant provided the following valuable contributions to the project:

1. Assistance with the development of the REE recovery flowsheet;
2. Technical knowledge of the flowsheet design and specification;
3. Critical plant operating knowledge;
4. Valuable knowledge associated with process failures and corrections;
5. Understanding of safety/environmental issues;
6. Detailed knowledge of the process control methodology; and
7. Understanding of critical cost sources.

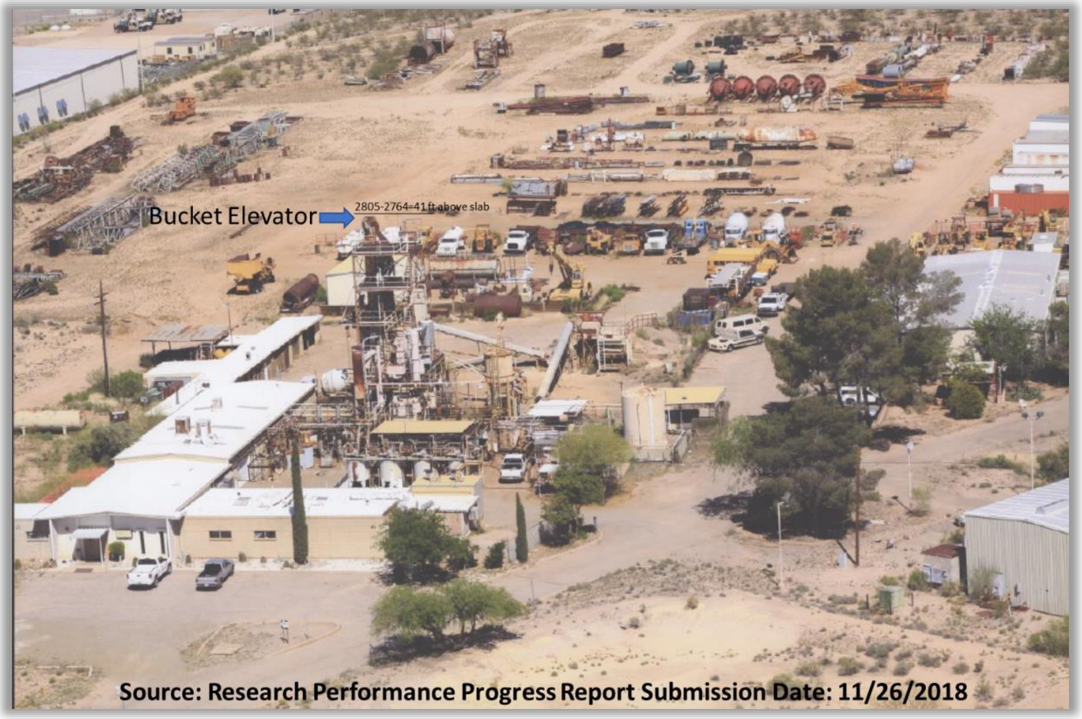
Figure 8-2: General Location Map of the Previously Operated Nickel-laterite Ore Hydro-met Pilot Plant near Tucson, Arizona



---

[6] Cimetta Engineering & Construction Co. Inc., 6701 S. Wilmot Road Tucson, Arizona 85756, Tel: +1 (520) 574-2930.

Figure 8-3: Previously Operated Nickel-laterite Ore Hydro-met Pilot Plant near Tucson, Arizona



Source: Research Performance Progress Report Submission Date: 11/26/2018

The following table summarizes the accomplishments of each trip associated with the benchmark process audit activities.

Table 8-1: Summary of Benchmark Process Audits

Objective	Mountain Pass Mine	Arizona Pilot Plant Trip I	Arizona Pilot Plant Trip II
Date	**2/11/2018 - 2/15/2018 3 Days at site**	**11/4/2018 - 11/7/2018 2 Days at site**	**2/11/2019 - 2/14/2019 2 Days at site**
1. Assist in the development of the REE recovery flowsheet	X	X	
2. Evaluate if the flowsheet meets design intent and specification	X	X	
3. Identify critical plant learnings	X		X
4. Learn from significant process failures and corrections	X		
5. Understand safety/ environmental issues	X		X
6. Evaluate process control methodology	X		X
7. Understand significant cost sources	X		X

8.2 Benchmark Visit Sites

8.2.1 MP Materials, Mountain Pass, California

The trip to the MP Materials mine was predicated on each of the visiting team members signing individual non-disclosure agreements. As such, the team was legally bound by the amount and type of information that can be reported. However, to meet project objectives, examples will be provided of the general overview of the visit.

The trip to the processing facilities of the MP Materials mine was essential to the team's understanding of the practical implications of feedstock leaching and subsequent recovery of REEs from the leachate using solvent extraction. Highlights and key learnings from the trip include:

- Several root causes of the financial failure of the predecessor Mountain Pass operation from an engineering/project management prospective;
- Examination and discussion of the flowsheet of the predecessor Mountain Pass mine;
- Feedstock run-of-mine ore beneficiation circuit at Mountain Pass;
- Technical leaching details of the ore at Mountain Pass;
- Detailed insights of the solvent extraction flowsheet as the facility was examined and the process-flow was audited;
    - Solvent extraction operating parameters;
    - Solvent extraction reagents;
    - Practical considerations associated with solvent extraction operation;
- Detailed process control methodology;
    - Critical process variables affecting performance;
- Detailed discussions about equipment, both of what worked well and what did not;
    - Names of equipment vendors suitable for the MMA29956 project;
- Context of the in-house REE SX matlab model showing the recovery and purity of REEs;
- Various analytical procedures for process evaluation; and
- Mountain Pass strategy for the treatment of radionuclides.

Although the team conducted and documented its learnings and discussions, they were legally bound not to release the details. Nevertheless, the trip proved to be both essential and very instructive. Significant insights collected from the Mountain Pass visit were applied to the flowsheet development of Project MMA29956.

The photographs that follow show various non-privileged areas of the Mountain Pass complex. These are included herein as evidence of the team's presence at Mountain Pass and to thus satisfy Project MMA29956 contract obligations. Those figures document the ore, ball mill and several of the solvent extraction "trains".

Figure 8-4: Photograph of Mountain Pass Ore

Figure 8-5: Ball Mill at Mountain Pass

Figure 8-6: Historic Solvent Extraction Train

Figure 8-7: Alternate Solvent Extraction Vessel Design

Figure 8-8: Newest Solvent Extraction Train

Figure 8-9: Overlook of Separations and Chemical Facility

8.2.2 Previously Operated Pilot Plant (Trip 1)

8.2.2.1 Introduction

The pilot plant located at 6701 South Wilmot Road, Tucson, Arizona 85756-2930 was found to be suitable for refit and revitalization for the extraction of REEs. The current owner of the plant is **Cimetta Engineering & Construction Co. Inc.** ***(Cimetta Engineering)***[7]. The history of the plant according to the current plant owner is summarized as follows:

UOP, Inc[8]. ***(UOP)*** constructed the facility in the late 70's to early 80's. The drawings of the facility are dated in the range of 1975-1981. From anecdotal information, UOP attempted to establish a nickel recovery technology center based on its proprietary hydrometallurgical technology for nickel ore concentration.

By the mid 1980's, UOP elected not to continue operating the pilot plant. At this point, UOP donated the property and equipment to the University of Arizona.

Mr. Cimetta, who was involved in the construction of additions to the pilot plant facility and thus has extensive knowledge of the plant and associated capabilities, began leasing the premises from the University of Arizona for metallurgical test work. This work primarily involved the comminution and hydrometallurgical portions of the circuit.

After a certain period, the property was sold to Mr. Cimetta by the University of Arizona. Mr. Cimetta has used the comminution and the hydrometallurgical portions of the plant for his engineering consulting business.

It is not clear how much material was processed through this plant, as there were no visible tailings or refuse accumulation at this site. It was further communicated that the Nickel *(Ni)* laterite ores that this facility was designed to process were not sourced locally.

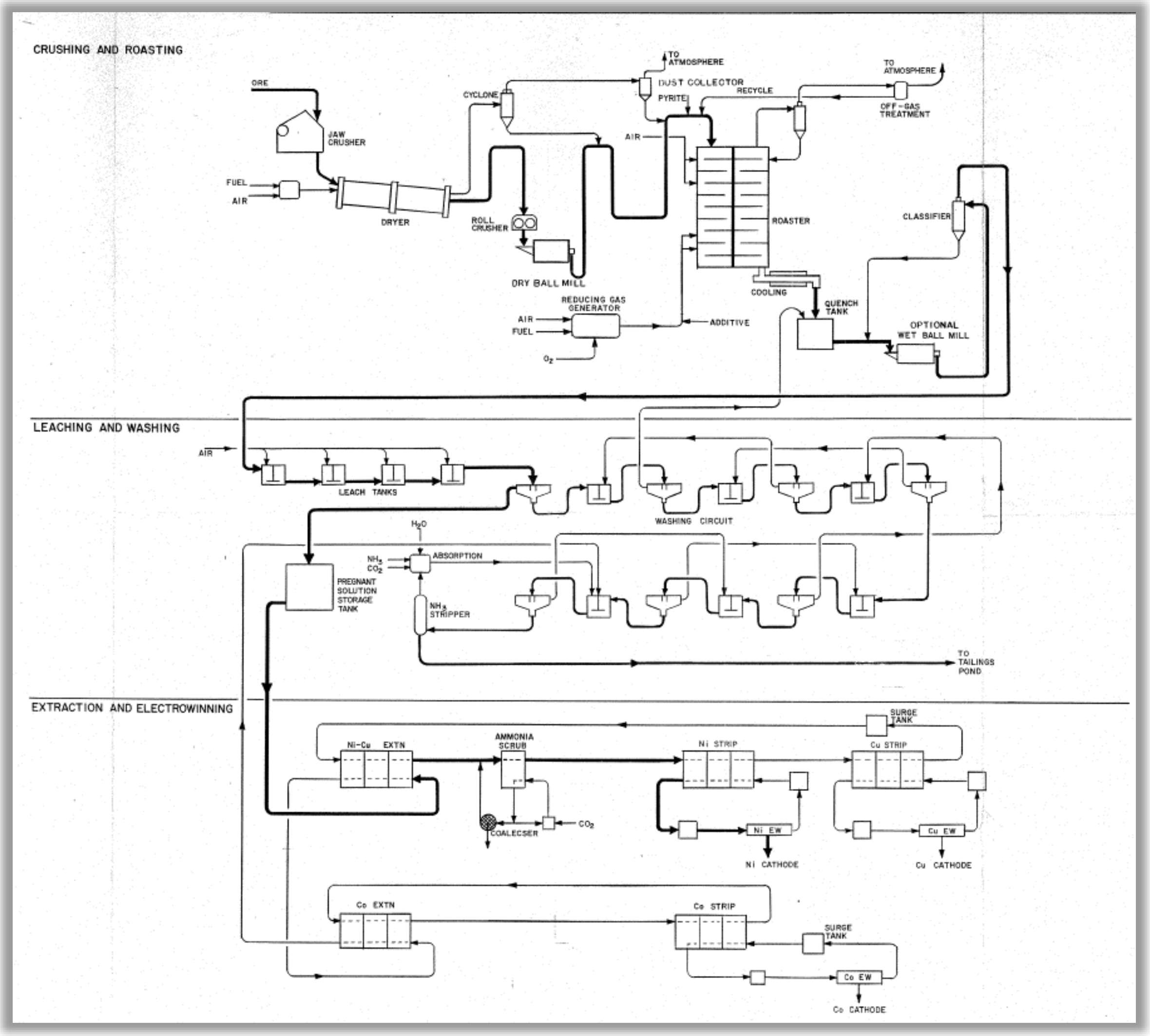
As no additional operational information has been acquired, part of this audit was to determine the operating conditions in which this plant was constructed. The following figure shows a high-level process flow chart associated with the Pilot Plant.

---

[7] Cimetta Engineering and Construction Co., Inc., is listed as a "general contractor" with an address at 6701 Wilmot Road, Tucson, AZ. *Source: Better Business Bureau.*

[8] UOP, Inc., (formerly known as **Universal Oil Products** ***(UOP)***) a division of **Allied Signal**, maintained a Tucson Technical Development Center during the 1970's to early 1980's. The technical center's staff took part in the construction and operation of the subject Pilot Plant, specifically to evaluate the viability of its nickel laterite reduction roasting, ammonia leaching and electrowing process. *Source: CV of Martin C. Kuhn, Ph.D., P.E., QP, former manager of the Technical Center and the operation of the Pilot Plant, 08/30/2013*.
On or about 1983-1985, Allied Signal closed or sold every business unit not related to oil and gas. During that reorganization, UOP dropped its ventures into mineral exploitation (including the subject Pilot Plant) and resumed its core business associated with oil products. *Source: https://www.uop.com/about-us/uop-history/a-friendly-acquisition/*

Figure 8-10: Previously Operated Pilot Plant - High Level Flow Sheet



8.2.2.2 Process Description

Further research showed that pilot plant utilized a modified version of the "Caron" process for the reduction and extraction of Ni. The process is included in the book Extractive Metallurgy of Nickel, Cobalt and Platinum Group Metals by Crundwell et. al. These process steps are summarized as follows:

Ore dying/grinding - Objectives: Remove 95% of mechanically entrained water from the ore and grind the dried product to ~75 µm, so that it flows steadily and evenly through the following hearth roasters. Equipment: Co-current rotary kilns using combustion gas at 1000°C to vaporize the water.

Ore reduction roast - Objective: Reduce Nickel and Cobalt minerals to metallic Ni-Co-[low Fe] alloy in preparation for subsequent dissolution of the Ni and Co by aqueous NH_3, CO_2, air leaching. Equipment: Counter-current hearth roasters (760°C) using oil/air combustion gas for reduction. Molar CO/CO_2 and H_2/H_2O ratios in reducing gas are ~1 to reduce Ni and Co in laterite to alloy and its Fe to magnetite. NiOOH reduces more easily than FeOOH (thermodynamically). Representative Reactions:

$$2NiOOH(s) + 3H_2(g) \rightarrow 2Ni(s) + 4H_2O(g)$$

$$2CoOOH(s) + 2CO(g) + H_2 \rightarrow 760°C \rightarrow 2Co(s) + 2CO_2(g) + 2H_2O(g)$$

$$6FEOO(s) + H_2 \rightarrow 760°C \rightarrow 2Fe_3O_4(s) + 4H_2O(g)$$

$$6FeOOH(s) + CO(g) \rightarrow 760°C \rightarrow 2Fe_3O_4(s) + CO_2(g) + 3H_2O(g)$$

Calcine Quenching in NH_3/CO_2 aqueous solution - Objective: Avoid re-oxidation of nickel and cobalt, begin leaching of Ni-Co alloy. Equipment: Ore cooling (to 150°C) tubes followed by quench tanks. Product quench slurry is ~100°C.

Calcine Leaching - Objective: Dissolve Ni and Co from alloy into aqueous NH_3 + CO_2 + O_2 (from air) solution in preparation for Ni carbonate precipitation. Equipment: Leach solid/liquid separation tanks. The representative reaction is:

$$Ni(s) + 6NH_3(g) + CO_2(g) + 0.5O_2 \rightarrow 10°C \rightarrow Ni(NH_3)_6^{2+} + CO_2^{3-}$$

Solid liquid separation - Objective: Separate unleached solids from nickel- and cobalt-rich solution. Equipment: Thickeners and filters.

Precipitation of basic Ni carbonate - ($3Ni(OH)_2 \bullet 2NiCO_3$) Objective: Produce high-purity solid ($3Ni(OH)_2 \bullet 2NiCO_3$) product. Method: Remove ammonia from solution by counter-current steam heating in cylindrical multi-tray stripping towers (Boldt & Queneau, 1967).

Representative Reaction:

$$5Ni(NH_3)^{6}_{2+}(aq) + 2CO_3^{2-}(aq) + 6OH^-(aq) \rightarrow steam\ heating$$
$$\rightarrow (3Ni(OH)_2 \bullet 2NiCO_3)(s) + 30NH_3(g)$$

From this description and additional literature, it was apparent that the system was set up for reductive roasting. It also appears that this process was further modified to include variations of reducing gasses. For example, the on-site configuration includes a coal gasifier to provide CO to the furnace, where an Exo gas generator (exothermic gas) produces reducing gasses. It is apparent that UOP constructed this system as a modification of the Caron process.

8.2.2.3 Process and Construction Drawings

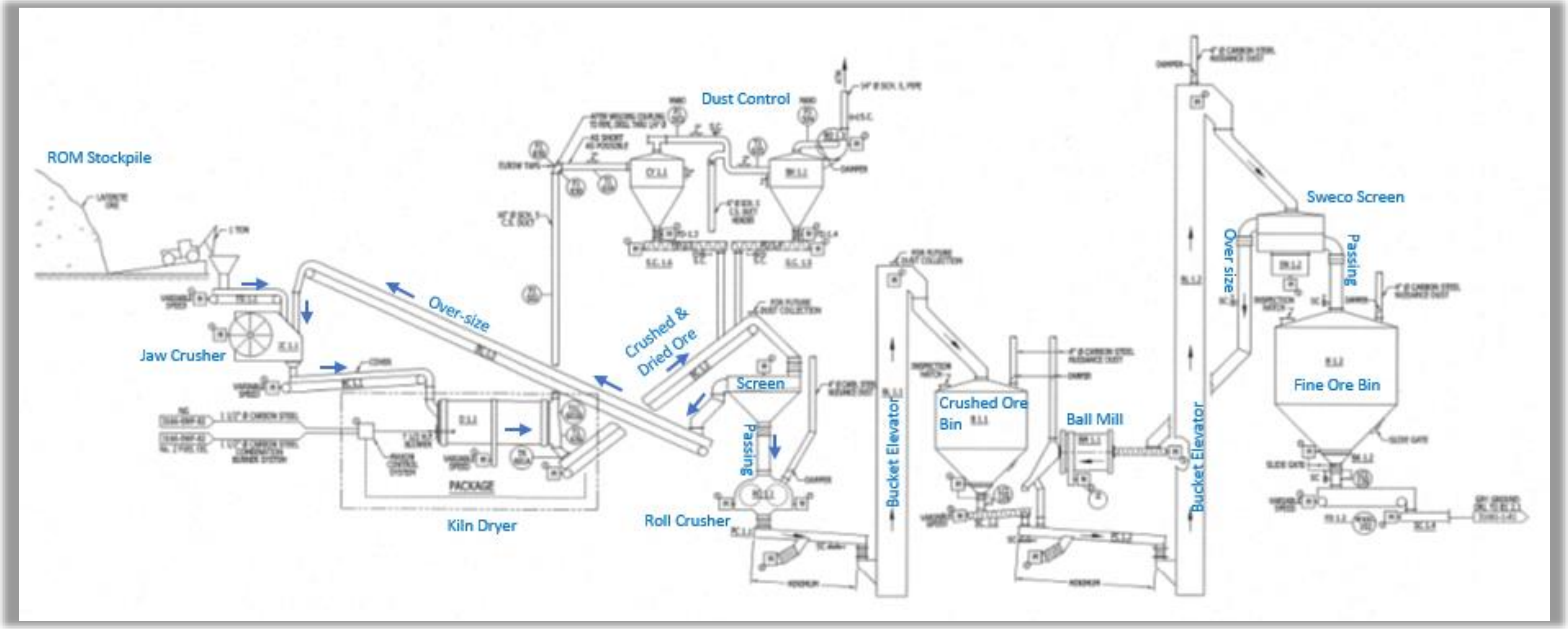
A significant appeal of this pilot plant was the existing particle sizing equipment, properly scaled equipment and the reduction in lead time needed for acquisition if the project proceeded to Phase 2. Further, in the weeks after the initial visit, the owner was able to locate a significant amount of engineering documentation. The tables included in this section indicate the quality and quantity of the engineering drawings obtained. The original plant consisted of eight sections, i.e.:

- Crushing and Grinding;
- Roasting;
- Leaching;
- Gas Scrubbing;
- NH_3CO_2 Recovery Product Recovery (Tails);
- Product Recovery;
- Solvent Extraction; and
- Utilities.

An apparently complete set of piping and instrumentation diagrams *(P&ID*) detailing the operation of the plant and design intent was obtained from the owner. A selected number of these drawings is discussed in the following narratives.

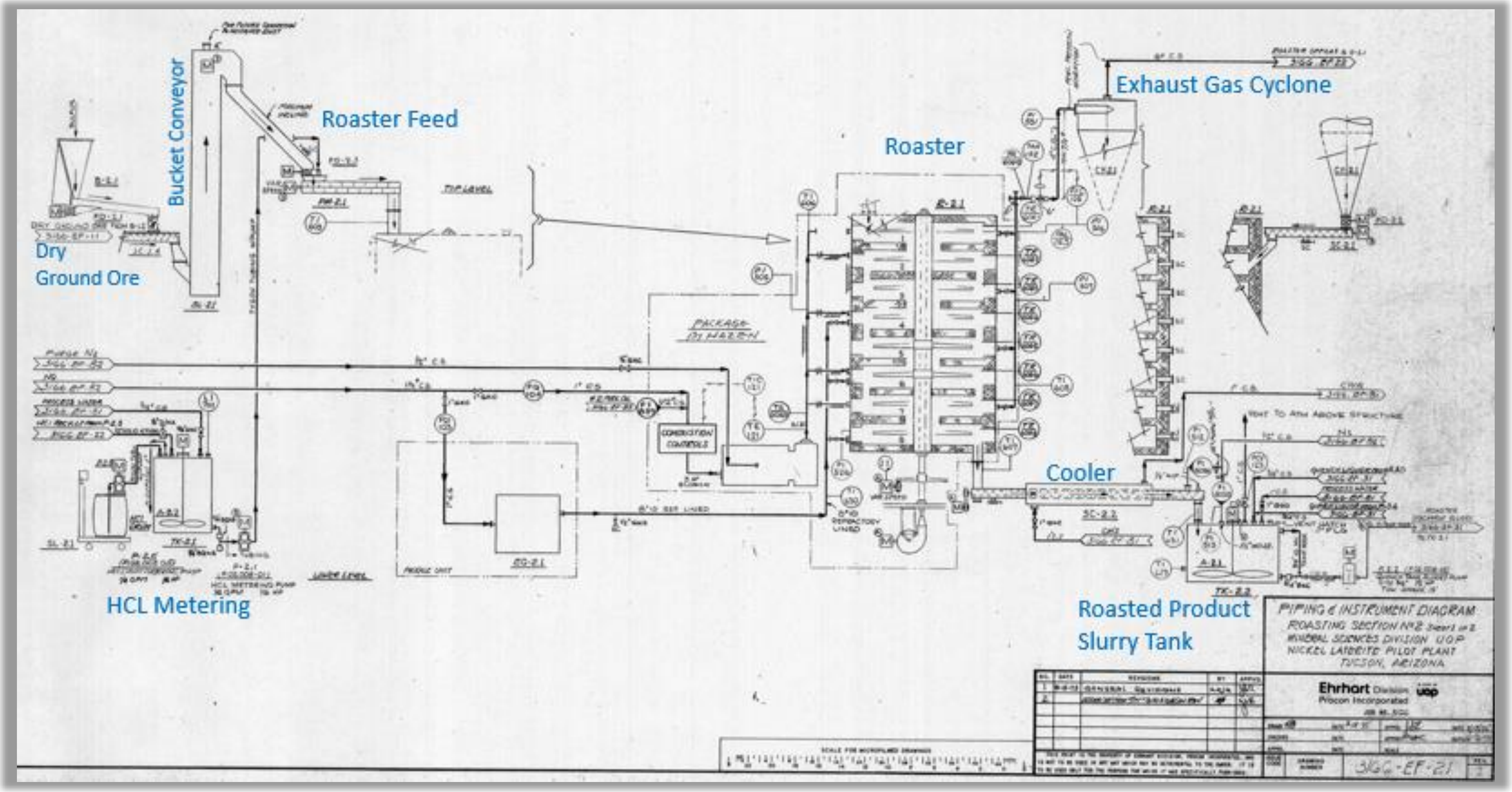
Drawing 3166-EF-11 detailed the crushing circuit which was designed to both dry and size laterite ore. Section 1 consists mainly of a jaw crusher feeding a rotary kiln dryer. The dryer assembly included a bag house for dust emissions control. Sizing was accomplished through initial screening, where the oversized material was returned via conveyer to the jaw crusher for re-crushing purposes. The undersized material was further reduced via a roll crusher. The output of the roll crusher fed a bucket elevator terminating in a coarse crushed/sized ore bin. The discharge of the dust collection system from the dryer also fed the output of the coarse ore bin. The material from the coarse crushed/screen ore bin was lifted via a bucket elevator to a Sweco screen. The oversized material of the Sweco screen was further reduced via a dry ball mill which discharged back to the bucket elevator to the Sweco screen. The undersized material from the Sweco screen was discharged to the fine ore bin for feed into the roaster.

Figure 8-11: Previously Operated Pilot Plant Drawing 3166-EF-11. Crushing and Drying



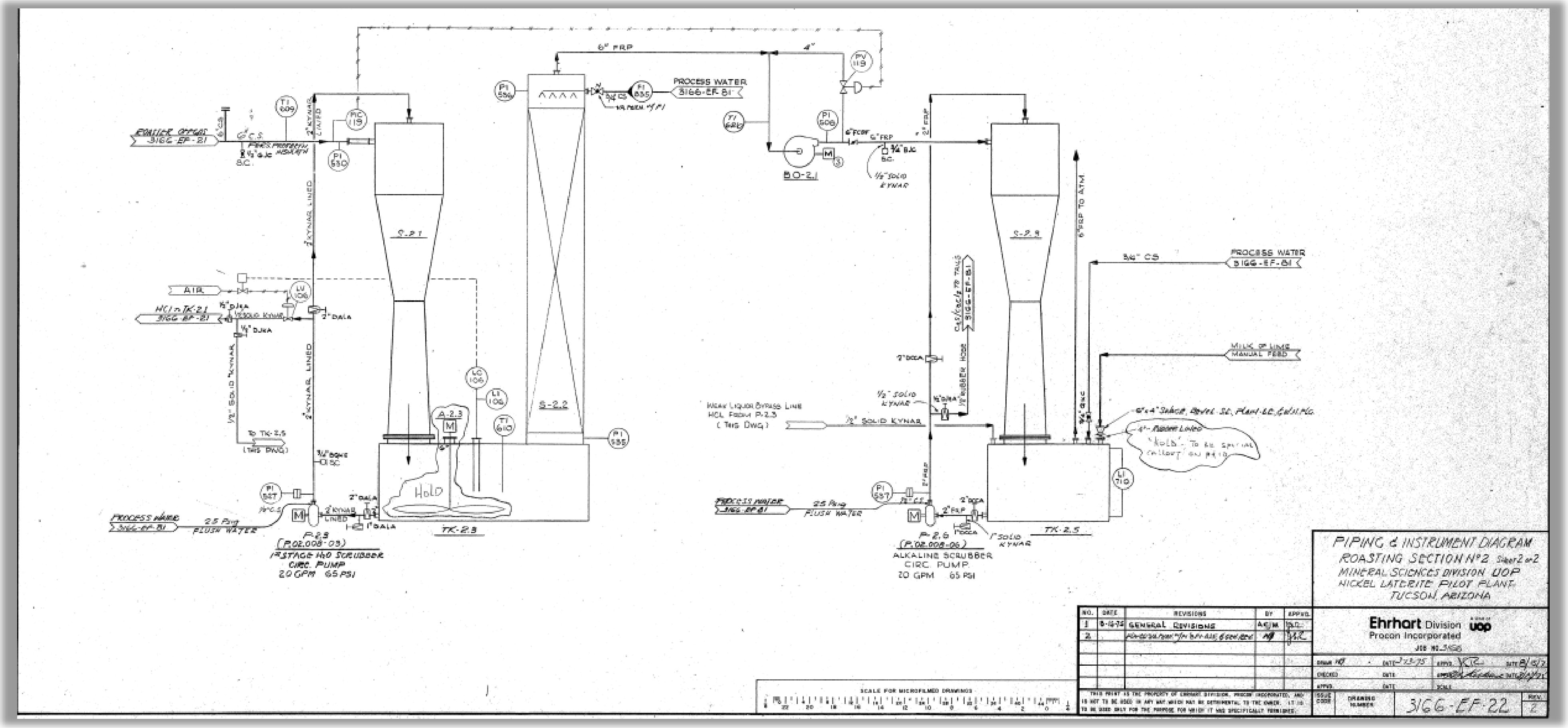
Drawing 3166-EF-21 details the feed and operation of the roaster. Upon discharge from the fine ore bin, a bucket elevator carried the material to the top of the roaster. A Hydrochloric acid *(HCl)* dosing system was included to improve the leachability of the ore post-roasting. HCl was metered into the roasting feed. The roaster was a skinner, 8-stage multiple-hearth furnace. The furnace flue included a cyclone to return dust to the upper portions of the furnace prior to scrubbing. The output of the furnace passed through a water-cooled discharge conveyor where the material was cooled prior to entry onto the roaster discharge slurry tank.

Figure 8-12: Previously Operated Pilot Plant Drawing 3166-EF-21 - Roaster



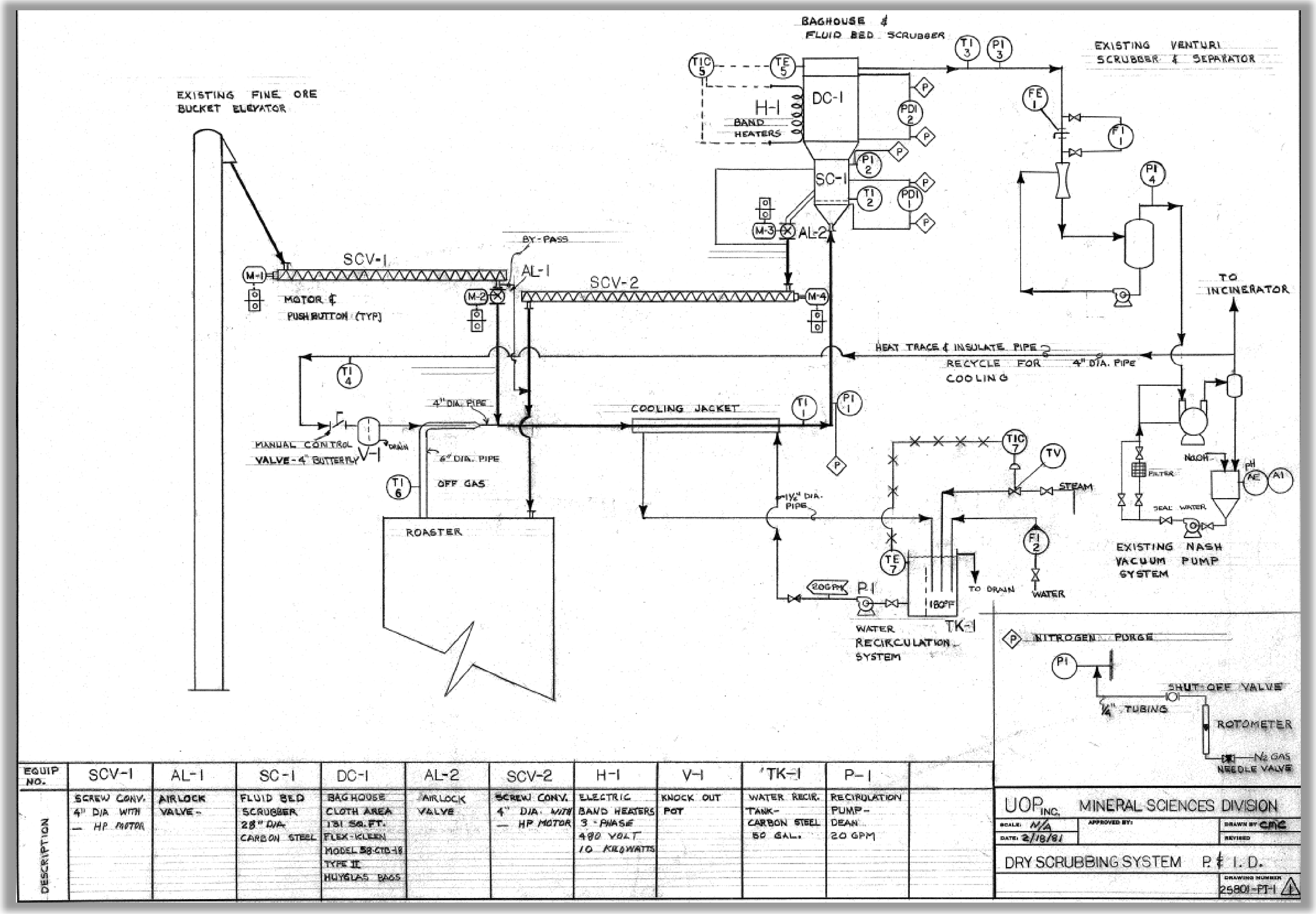
Drawing 3166-EF-22 was the second drawing in the roaster series and contained a 2-stage scrubber. The first stage was a wet venturi which captured the HCl and returned to the roaster HCl feed. The exhaust passed through a spray tower to further capture the HCl. The second wet venturi was assumed to assist in capturing the sulfur as the sump was flushed with lime as a neutralizing agent. Afterwards, the second scrubber air was vented to the atmosphere.

Figure 8-13: Previously Operated Pilot Plant Drawing 3166-EF-22 - 2-Stage Roaster Scrubbing System



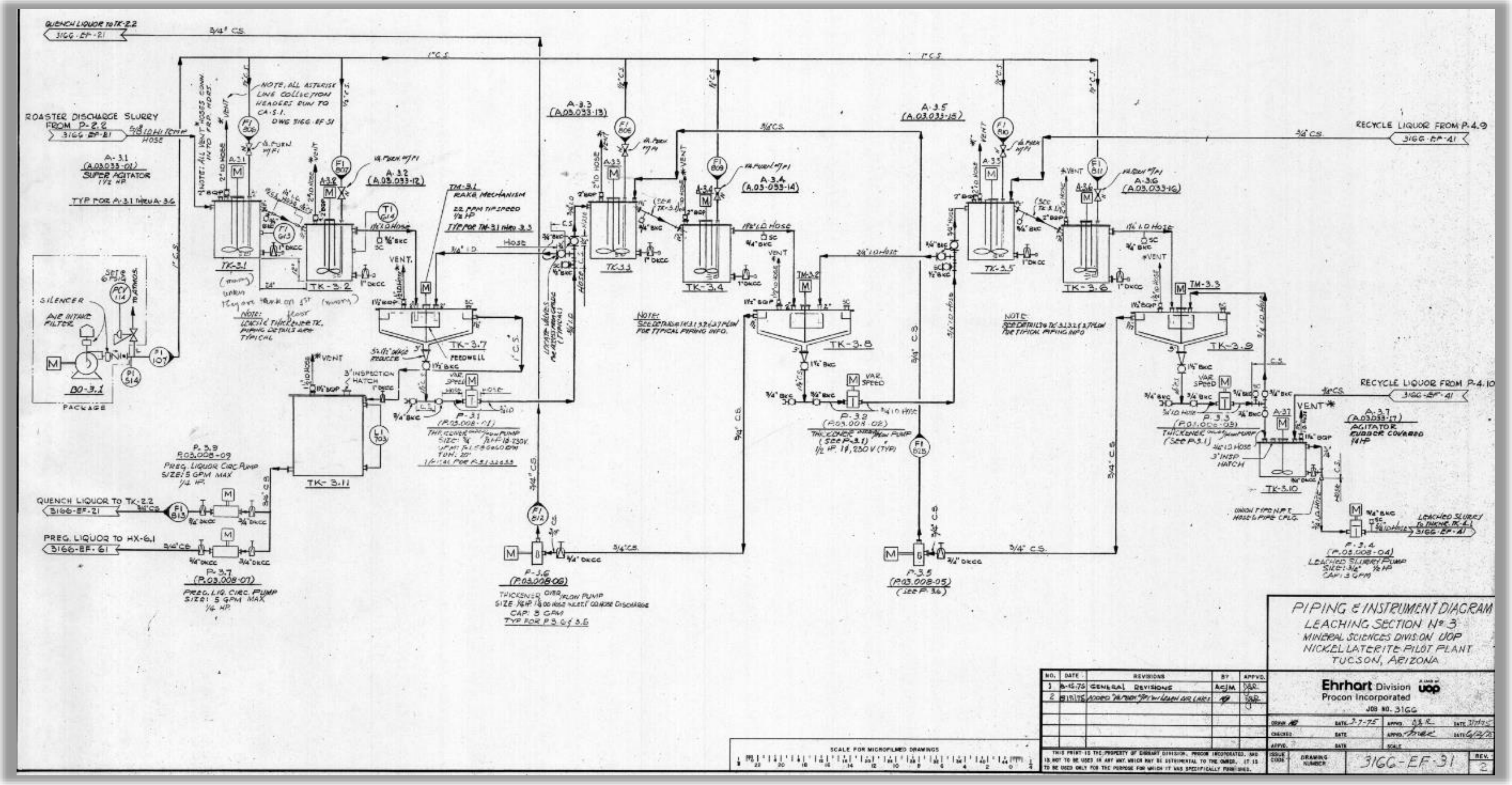
The roaster gas treatment system was modified over time. The details of this modification are shown in TS78001 and a dry scrubbing option is shown in 25801-PI-1.

Figure 8-14: Previously Operated Pilot Plant Drawing 25801-PI-1 - Modified Dry Scrubbing System



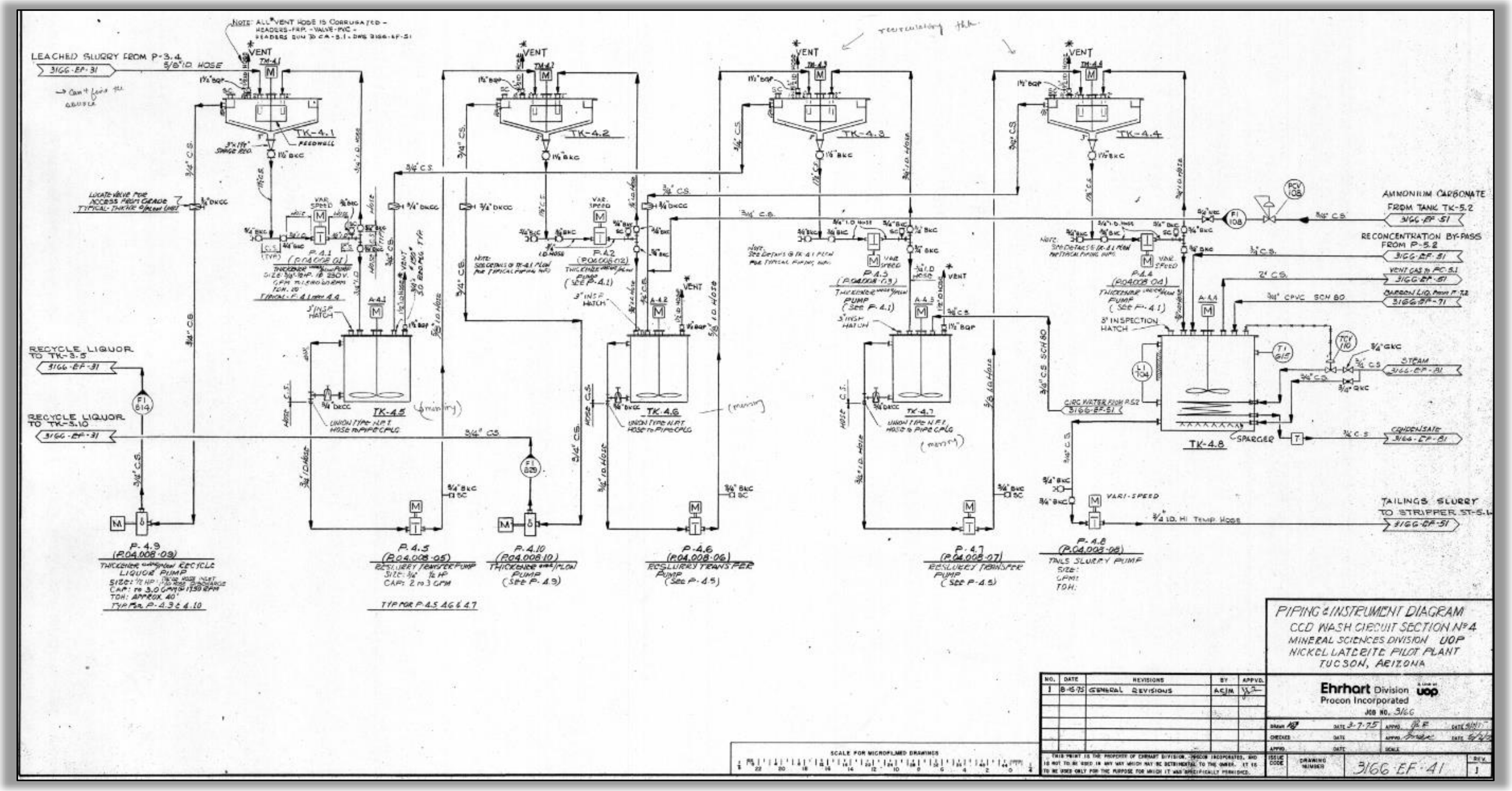
Drawing 3166-EF-31 is an upgrade from the original process flow sheet in that it shows three leach tanks prior to decantation in a thickener. The circuit utilized a typical counter-flow design, showing three leach/separation stages with two feed leach tanks each. An air blower was included to aerate the leach solution. In the first thickener, the overflow containing raffinate was split between the quench tank and feeding the SX circuit. The output of the third leach group is collected in a slurry tank prior to feeding into Section 4.

Figure 8-15: Previously Operated Pilot Plant Drawing 3166-EF-31 - Leaching Circuit (Section 3)



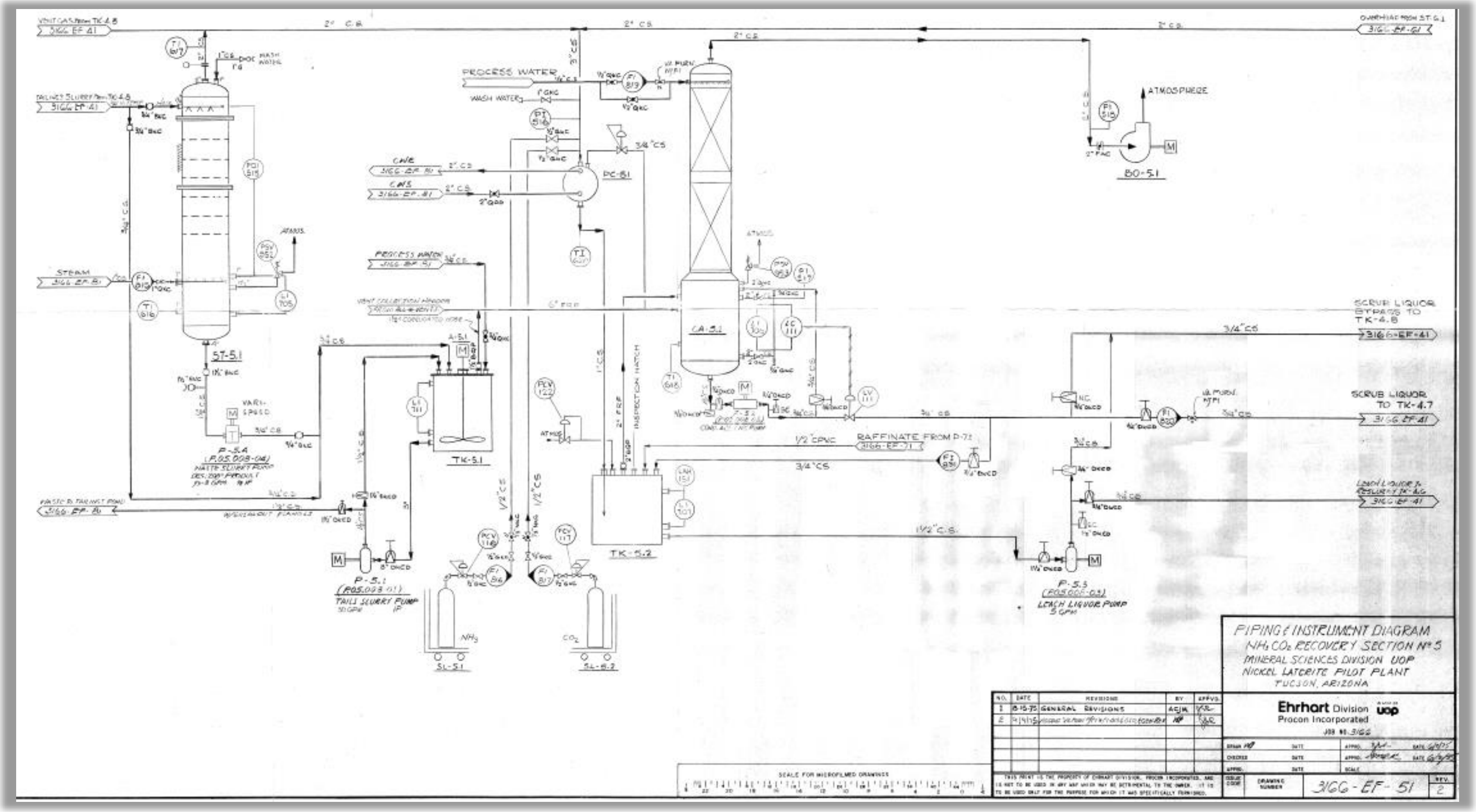
Drawing 3166-EF-41 showed a circuit identified as the "CCD" circuit which is believed to stand for counter-current decantation. The counter-current wash was accomplished in four stages. The leach reagent was fed in the middle and end of this circuit. Heating was accomplished through steam coil heating in the barren leach tank. The design does not include additional heating as apparently heat losses with the small volumes involved with this pilot program were not a significant concern.

Figure 8-16: Previously Operated Pilot Plant Drawing 3166-EF-41 - Counter Current Decantation (Section 4)



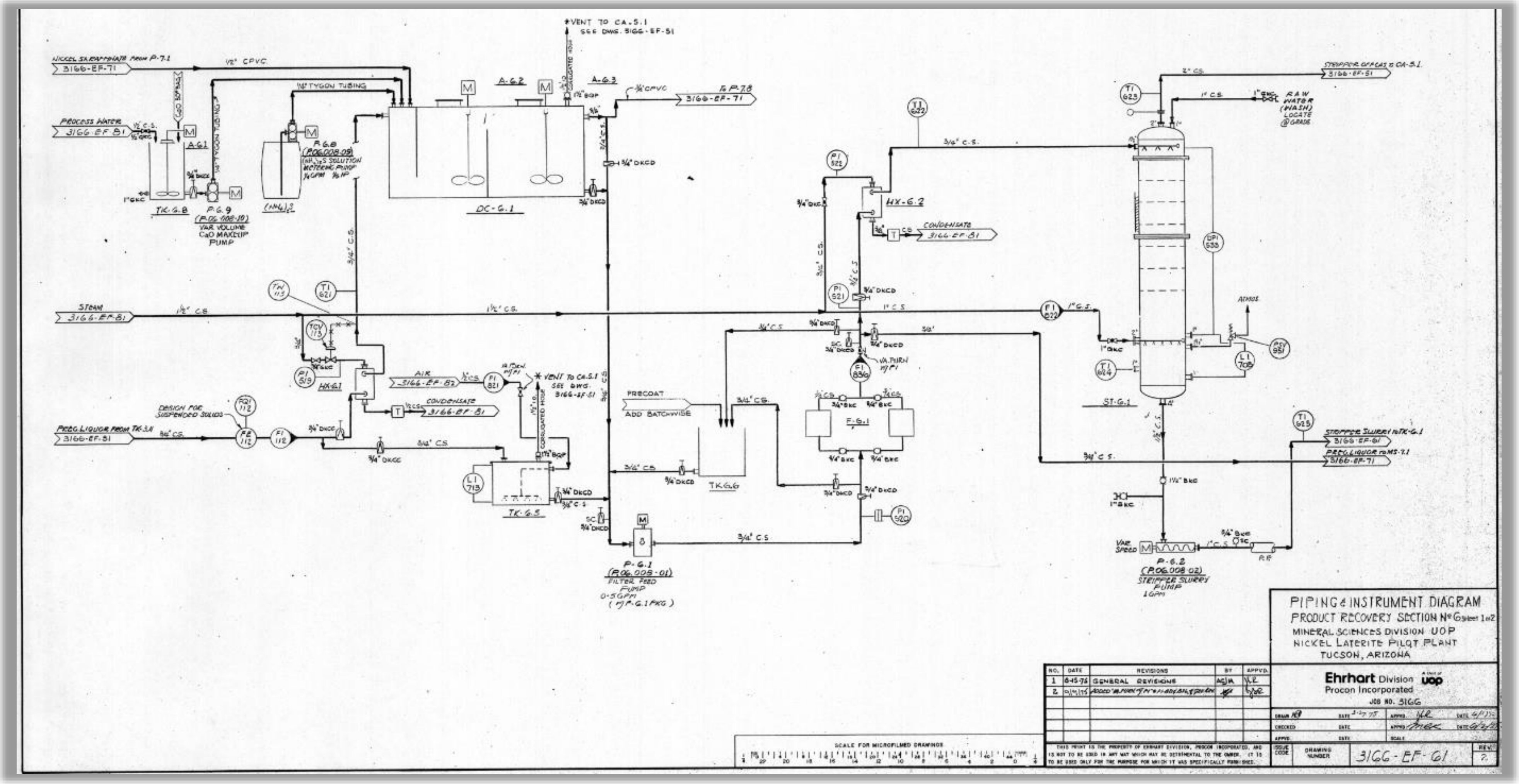
Drawing 3166-EF-51 shows the leachate recovery section, where the tails are treated, and the effluent was reconstituted via gaseous reagents. The tailings were removed at this point and shown moving to a tailings pond. Further, an Ni carbonate product was produced by precipitation and recovered via a belt filter.

Figure 8-17: Previously Operated Pilot Plant Drawing 3166-EF-51 - Counter Current Decantation (Section 4)



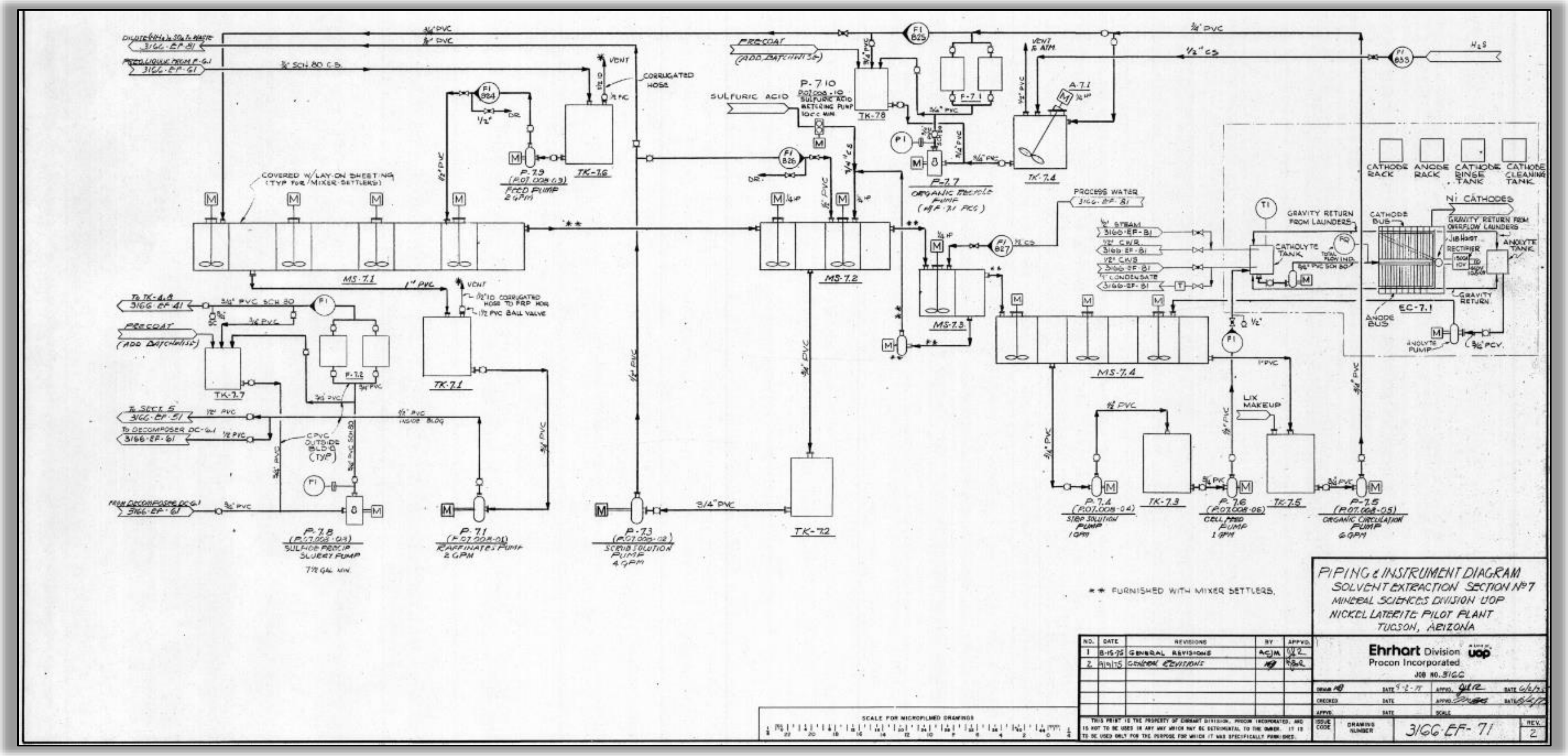
Drawing 3166-EF-61 shows the solvent extraction (SX) feed pre-treatment prior to SX.

Figure 8-18: Previously Operated Pilot Plant Drawing 3166-EF-61 - Solvent Extraction Pre-Treatment



Drawing 3166-EF-71 shows the proposed circuitry for SX and electrowinning recovery of Ni.

Figure 8-19: Previously Operated Pilot Plant Drawing 3166-EF-71 - Solvent Extraction and Electrowinning of Ni



The reader will note that the P&ID drawings are quite detailed, showing each valve, line size and composition, with pumps and equipment.

Table 8-2: List of Original P&ID Diagrams for the Previously Operated Plant

Section #	Drawing Description	Drawing Number	Sheet Number
--	UOP Nickle Process		
1	Piping and Instrument Diagram; Crushing & Grinding Section No. 1	3166-EF-11	NA
2	Piping and Instrument Diagram; Roasting Section No. 2	3166-EF-21	1 of 2
2	Piping and Instrument Diagram; Roasting Section No. 2	3166-EF-22	2 of 2
3	Piping and Instrument Diagram; Leaching Section No. 3	3166-EF-31	NA
4	Piping and Instrument Diagram; CCD Wash Circuit Section No. 4	3166-EF-41	NA
5	Piping and Instrument Diagram; Leachate Recovery Section No. 5	3166-EF-51	NA
6	Piping and Instrument Diagram; Product Recovery Section No. 6	3166-EF-61	1 of 2
6	Piping and Instrument Diagram; Product Recovery Section No. 6	3166-EF-62	2 of 2
7	Piping and Instrument Diagram; Solvent Extraction Section No. 7	3166-EF-71	NA
8	Piping and Instrument Diagram; Utilities Section No. 8	3166-EF-81	1 of 2
8	Piping and Instrument Diagram; Utilities Section No. 8	3166-EF-82	2 of 2

Further, it appears that some of the details relating to the modifications to the roasting gas scrubbing system and burner were captured and included in part of the drawing package that we have as indicated in the following table.

Table 8-3: Subsequent Drawings Indicating Modifications to the Roasting Gas Scrubbing and Burner Systems of the Previously Operated Pilot Plant

Section #	Drawing Description	Drawing Number	Sheet Number
2	Piping and Instrumentation Diagram; Off Gas Scrubbing System Ni Pilot Plant	TS78001	NA
2	Dry Scrubbing System P&ID	25801-PI-1	NA
2	Piping and Instrumentation; Hauk Oil Burner	TS78003	NA

General arrangements are provided for the first two sections of the plant, namely crushing and grinding, as shown in the below table.

Table 8-4: General Layouts and Elevations of the Crushing/Grinding and Roasting Sections of the Previously Operated Pilot Plan

Section #	Drawing Description	Drawing Number	Sheet Number
1	General Arrangement Plan.; Crushing and Grinding Section No. 1	3166-EM-101	NA
1	Piping and Instrument Diagram; Roasting Section No. 2	3166-EM-102	NA
1	Piping and Instrument Diagram; Roasting Section No. 2	3166-EM-103	NA
2	General Arrangement Plans and Elevations Roasting Section No. 2	3166-EM-104	NA

The following table shows the associated electrical and control drawings. Prints 3166-EE-102 and 103 reference the crushing circuit and roaster respectively. It appears that several electrical drawings are missing. This is of great concern as for the age of this plant; it is assumed that the electric circuits will be replaced and modernized. The remaining drawings are of little concern, as one is a control panel and the other is for a gas generator that will not be utilized.

Table 8-5: Electrical and Control Drawings of the Previously Operated Pilot Plant

Section #	Drawing Description	Drawing Number	Sheet Number
1	Electrical Schematics Diagram	3166-EE-102	NA
1	Electrical Schematics Diagram	3166-EE-103	NA
1	Exothermic Gas Generator Electric Diagram	TS79007	NA
2	Off Gas Scrubbing System; Panel - 3rd Deck Panel	TS79012	NA

The acquisition and construction drawings, as shown the following table are of great interest, as it appears that the entire set of both foundational and steel drawings are available. These drawings will be extremely useful for the reassembly of the plant and re-stamping the structural components in Kentucky. The general orientation and engineering of this pre-existing structure will readily assist the project in completion.

Table 8-6: Steel and Foundation Drawings of the Previously Operated Pilot Plant

Section #	Drawing Description	Drawing Number	Sheet Number
- -	Struct Steel Location Plan	3166-ES-110	NA
- -	Ore Storage Area	ES-101	NA
- -	Process Slabs & Footings	ES-102	NA
- -	Foundation Details I	ES-103	NA
- -	Foundation Details II	ES-104	NA
- -	Struct. Steel Framing Plan	ES-105	NA
- -	Structural Steel Sections	ES-106	NA
- -	Structural Steel Details	ES-107	NA
- -	Feed Hopper Ramp	ES-108	NA
- -	Foundation Details III	ES-109	NA
- -	Misc. Steele Structures	ES-111	NA
- -	Ladder & Cage Details	SC-14-12	NA
- -	Structural Railing Details	SC-14-5	NA
- -	Stair Details	SC-14-7	NA
- -	Concrete Inserts & General Notes	SC-14-21	NA
1	Foundation: Wet Grinding Ball Mill	TS78002	NA
9	Pilot Plant Gasifier Structural Framing Plans	Unknown	NA

Owing to the use of HCl in the roasting feed, it is not surprising that the roaster shell has been damaged by corrosion. It is convenient that a set of drawings has been provided that will allow the reconstruction of the damaged portions of the shell for reutilization. The following table shows the drawings received specific for the roaster and shell assemblies.

Table 8-7: Roaster-Specific Drawings of the Previously Operated Pilot Plant

Section #	Drawing Description	Drawing Number	Sheet Number
2	Skinner Furnace; 5'-0 Diameter; Bridge and Upper Seal Assy	101220	NA
2	Skinner Furnace; 5'-0 Diameter; Drive Assembly	101221	NA
2	Roaster Bridge & Upper Seal Assembly	101226	
2	Skinner Furnace; 5'-0 Diameter; Bottom Hearth Assembly; Movable Side	101254	NA
2	Skinner Furnace; 5'-0 Diameter; Bottom Hearth Assembly; Stationary Side	101258	NA
2	Roaster Bottom Hearth Assembly	101259	
2	Skinner Furnace; 5'-0 Dia X 8 Spilt Shell; "In" Hearth & Roof Ach Assy	101260	NA
2	Skinner Furnace; 5'-0 Dia X 8 Hearth Split Shell; "Out" Hearth Assembly	101261	NA

Notably, it appears from a first-hand examination of the idle facilities that there were changes incorporated which included switching from a natural gas fueled generator to a coal-based gasifier for the production of a reducing atmosphere of carbon oxide. This switch has no particular benefit to Project MMA29956, as the current rare earth compound concentration design proposes the use of an oxidizing atmosphere.

8.2.2.4 Equipment and Condition

As described previously, the crushing and certain portions of Cimetta's hydrometallurgical plant have been utilized within the last 10 years. These activities have not been well-documented, but they appear to have been limited in both quantity and duration. Observations of the site by research team members suggest that those activities have not materially affected the components of interest to Project MMA29956.

It should be noted that due to the nature of the original processes used by UOP, basic chemistry carbon steel is present throughout the existing facility for pipes, tanks, rakes, and other hydrometallurgical components. Given the acid-based process proposed by MMA29956 for rare earth compound concentration, all pilot plant components beyond the roasting section will be replaced with fiber-reinforced plastic tanks, PVC/PEX piping with as well as suitable acid resistant hardware/equipment and machinery.

The figures that follow are included for reference and to provide a general overview of the condition of the plant. As such, the pictures will document primarily Sections 1 and 2 of the original UOP pilot plant flow sheets.

The crushing and grinding circuit are of great utility to the project. The following figure shows the crusher feed hopper, jaw crusher and associated ramp used to load raw feedstock material into the process. The jaw crusher is a "Pacific Jaw Crusher 10x20" with a 25 HP motor.

Figure 8-20: Aerial View of Jaw Crusher and Supporting Conveyors of the Previously Operated Pilot Plant

After primary sizing with the jaw crusher, the material is dried in the rotary kiln as shown in the following figure. Also shown is the associated dust suppression system.

Figure 8-21: Rotary Kiln Dryer Showing Baghouse and Conveyors of the Previously Operated Pilot Plant

The roll crusher is a 7.5 HP Grundler 18x12 double roll which is fed by a 2’x3’ one-deck utility scalping screen as can be observed in the following figure.

Figure 8-22: Screen and Roller Crusher with Supporting Conveyors of the Previously Operated Pilot Plant

The following figure presents a panoramic overview overlooking the crushing and grinding circuits of the facility. Also shown is the gas scrubbing and a portion of the roaster.

Figure 8-23: Panorama Showing the Top of Roaster, Gas Scrubbing, and Ore Bins of the Previously Operated Pilot Plant

Upon examination, it appeared that the effects of adding HCl to the roaster feed in combination with the elevated temperatures of the roaster resulted in corrosion of the upper portions of the roaster as shown in the figures that follow. These figures also show missing portions of the roaster shell. As the outer shell is composed of sheet steel, these missing sections should be readily replaceable. Rebuilding the outer shell should be expedited given the roaster drawings provided by the pilot plant owner.

Figure 8-24: Top Shell of the Roaster of the Previously Operated Pilot Plant Showing Corrosion due to HCl

Figure 8-25: Alternate View of the Roaster Showing Shell Damage.

The roaster rakes appear to be in serviceable condition as shown in the following figure.

Figure 8-26: Interior View of Roaster of the Previously Operated Pilot Plant Showing Example of Serviceable Rakes

An important aspect of the roaster is that its condition improves as the bottom of the roaster is approached. This can be observed in the following figure. Note that the pinion gear is in good shape, and that the outer shell at the bottom of the roaster is in much better condition than that of the top. The favorable condition of the lower portion of the roaster should play a major positive role in rebuilding the roaster.

Figure 8-27: Lower Hearth Gear and Conditions of the Previously Operated Pilot Plant

It is also important to the project that the equipment incorporates the requisite components for emissions control. These controls, in the form of roaster flue gas scrubbing, are shown in part by the following figure. It is apparent from both drawings and the remaining equipment that an air scrubbing unit was integrated into the unit to account for the emissions produced by the roaster. However, we do not currently possess any data concerning the performance of the scrubbing unit.

Figure 8-28: Picture Showing Gas Scrubbing Equipment of the Previously Operated Pilot Plant

The following figure provides a top-down view of the assembly of roaster exhaust gas scrubbing equipment.

Figure 8-29: Aerial View of Gas Scrubbing Equipment and Portions of the Leaching Circuit of the Previously Operated Pilot Plant

8.2.3 Previously Operated Pilot Plant (Trip No. 2)

8.2.3.1 Appraisal of the Arizona Pilot Plant

As part of the second visit to the plant to continue the benchmark audit process, an appraisal was conducted by Roger A. Daugherty, President of **Darco Energy Management Corporation *(Darco)***. The estimate included the cost to replace the equipment and fair market value at removal. The definition of values was uniform with those used by the **American Society of Appraisers (*ASA*).** The assets appraised were the personal property and engineering drawing. The appraised values effective on February 12, 2019, are tabulated below[9].

Table 8-8: Appraised Value, Effective February 12, 2019, of the Previously Operated Pilot Plant

Asset Group	Replacement Cost	Fair Market Value-Removal
NCH Plant	S14,993,000	$1,097,000
Engineering and Drawings	$1,199,400	$1,049,500
Total	$16,192,400	$2,146,500

8.2.3.2 Salvage Equipment Power and Original Cost

During the second visit of the process audit performed on the pilot plant, the team was able to gain access to binders, index cards and drawings describing much of the original costs, specifications, and vendors of the equipment. This data was compiled as the basis for a detailed equipment selection. Although not an exhaustive list, the following table represents equipment most likely to be salvaged and moved[10] to a new

---

[9] NCH-Met Hydro-Met Plant Appraisal, Tucson, Arizona, Roger A Daugherty, Certified Senior Appraiser, EAANA, February 22, 2019.

[10] Significant assistance in this estimation process was provided by **Cumberland Mine Services, Inc.**, of Totz, Harlan County, Kentucky 40870-7118

location at the former Trus Joist facility[11] near Chavies, Kentucky. Since all the plant equipment was designed and sized for the correct capacity, the data is valuable to the team as a starting point for the plant equipment costing. It also provides the means for estimating shipping costs.

Table 8-9: List of Proposed Previously Operated Pilot Plant Salvaged Equipment Summary Data showing Description, Equipment Identification., Referenced Drawing, Power, Original Cost and Weight

Section Number	Name	Equipment ID Number	Drawing Number	HP	Cost (1975)	Weight (lbs.)
1	Storage Bin (B-1.1)	B-1.1	3166-EM-103 (also -102, -101, -11)	0	$3,976	5,500
1	Storage Bin (B-1.2) (Storage Bin for Finished Ground Ore)	B-1.2	3166-EM-101 (also -11)	0	$5,706	7,600
1	Vibrating Bin Activator (BA-1.1)	BA-1.1	3166-EM-103 (also -11)	1/2	$3,187	875
1	Vibrating Bin Activator (BA-1.2)	BA-1.2	3166-EM-103 (also -11)	1/2	$4,882	1,000
1	Jaw Crusher Disch Belt Conveyor (BC-1.1)	BC-1.1	3166-EM-102 (also -101, -11)	3	$5,885	4,310
1	Tail Pully Conv (BC-1.2)	BC-1.2	3166-EM-101 (also -102, -11)	3	$6,339	4,310
1	18' Belt Conveyor (BC-1.3)	BC-1.3	3166-EM-101 (also -102), 3166-EF-11	3	$5,849	4,310
1	Bag House (BH-1.1)	BH-1.1	3166-EM-101 (also -11)	0	$9,335	4,900
1	Bucket Elevator (BL-1.1)	BL-1.1	3166-EM-103 (also -102, -101, -11)	1 1/2	$4,787	2,750
1	Bucket Elevator (BL-1.2)	BL-1.2	3166-EM-103 (also -101, -EF-21, -11)	1 1/2	$4,787	2,750
1	(Conical) Ball Mill (BM-1.1)	BM-1.1	3166-EM-101 (also -11)	20	$11,500	14,000
1	Exhaust Blower (BO-1.1)	BO-1.1	3166-EF-11	30	$0	
1	(Dust) Cyclone (CY-1.1)	CY-1.1	3166-EM-103, -11)	0	$1,150	340
1	(Rotary) Dryer (D-1.1)	D-1.1	3166-EM-102 (also -101, -11))	7 1/2	$29,500	25,400
1	Jaw Crusher Belt Conveyor Feeder (FD-1.1)	FD-1.1	3166-EF-11, 3166-EM-102 (also -101)	1/3	$4,482	1,900
1	Storage Bin Disch Weight Belt Conveyor (FD-1.2)	FD-1.2	3166-EF-11	1/3	$6,114	400
1	(Cyclone Discharge) Rotary Feeder (FD-1.3)	FD-1.3	3166-EM-103 (also -11)	1/3	$1,296	120
1	Baghouse Discharge Rotary Feeder (FD-1.4)	FD-1.4	3166-EF-11	1/3	$1,296	120
1	Jaw Crusher (JC-1.1)	JC-1.1	3166-EM-102 (also -101, -11)	20	$3,400	6,200
1	Vibrating Conveyor (PC-1.1)	PC-1.1	3166-EM-101	1	$1,903	600
1	Roll Crusher Disch. Pan Conveyor (PC-1.1)	PC-1.1	3166-EF-11	1	$1,902	
1	12" Vibrating Conveyor (Bin Discharge - Pan Conveyor) (PC-1.2)	PC-1.2	3166-EM-103 (also -101, -11)	2		600
1	Roll Crusher Smooth Face (RC-1.1)	RC-1.1	3166-EF-11 (also -101)	15	$7,700	4,500
1	Roll Crusher (1.1)	RC-1.1	3166-EM-102	15		
1	(Ball Mill) Screw Conveyor (SC-1.2)	SC-1.2	3166-EM-103 (also -11)	1/3	$1,830	
1	(Bin Disch) Screw Conveyor (SC-1.3)	SC-1.3	3166-EM-103 (also -11)	1/3	$1,497	

---

[11] The former "Trus Joist" facility is a large steel building shell with adjacent lands located in the Coal Fields Industrial Park near Chavies, Kentucky. The building and land are now owned by **Kentucky River Properties, LLC *(KRP)***. KRP is a member of Project MMA29956 and has offered the use of a portion of that facility for Pilot Plant operations purposes.

Section Number	Name	Equipment ID Number	Drawing Number	HP	Cost (1975)	Weight (lbs.)
1	Bag House Disch Trough Type Screw Conveyor (SC-1.5)	SC-1.5	3166-EM-103 (also -11)	1/3	$1,513	330
1	Cyclone Discharge Trough Type Screw Conveyor (SC-1.6)	SC-1.6	3166-EF-11	1/3	$1,547	250
1	Vibrating Screen (SN-1.1)	SN-1.1	3166-EM-102 (also -101, -11)	1	$2,134	675
1	Vibrating Screen (Double Deck) (SN-1.2)	SN-1.2	3166-EM-104 (also -101, -11)	1	$5,692	1,125
1	Tails Slurry Hold Tank	TK 5.1			$1,998	
2	Quench Tank Agitator (A-2.1)	A-2.1	3166-EF-21	3	$1,559	750
2	First Stage H2O Scrubber Hold Tank Agitator (A-2.3)	A-2.3	3166-EF-22	0		
2	Bucket Elevator (BL-2.1)	BL-2.1	3166-EN-104	1-1/2	$4,787	2,750
2	Scrubber Exhaust Blower (BO-2.1)	BO-2.1	3166-EF-22	7-1/2	$0	520
2	Roaster Cyclone Rotary Feeder (FD-2.2)	FD-2.2	3166-EF-21	1/3		120
2	Rotary Valve Feeder (FD-2.3)	FD-2.3	3166-EF-21	1/3		
2	Carbon Monoxide Heater	HX-2.001		0	$8,635	
2	First Stage H2O Scrubber Pump 20 GPM (P-2.3)	P-2.3	3166-EF-22 (also -104)	1/3	$2,160	250
2	Alkaline Scrubber Circ Pump (P-2.6)	P-2.6	3166-EF-22	1/3	$2,160	250
2	Pug Mill (PM-2.1)	PM-2.1	3166-EF-21	5	$14,000	500
2	Roaster (R-2.1)	R-2.1	3166-EN-101 (also -104, 25801-PI-1	2	$120,000	60,300
2	Multiple Hearth Roaster w/ Central Col. Fan (R-2.1)	R-2.1	3166-EF-21	2		
2	First Stage H2O Scrubber Venturi (S-2.1)	S-2.1	3166-EF-22	0	$5,563	610
2	Second Stage H2O Scrubber Packed Bed (S-2.2)	S-2.2	3166-EF-22	0	$3,680	540
2	Alkaline Scrubber Venturi (S-2.3)	S-2.3	3166-EF-22	0	$2,610	400
2	Screw Conveyor (SC-1.4)	SC-1.4	3166-EN-104 (also -11)	1/3	$1,497	
2	Cyclone Disch Screw Conveyor (SC-2.1)	SC-2.1	3166-EF-21	1/3	$1,697	175
2	Cyclone Disch Screw Conveyor Water Cooled (SC-2.2)	SC-2.2	3166-EF-21	1/3	$2,200	600
2	Quench Tank TK-2.2	TK-2.2	3166-EM-104 (also, -21)	0	$1,578	1,290
2	Agitated First Stage H2O Scrubbr Hold Tank (TK-2.3)	TK-2.3	3166-EF-22	0	$5,563	420
2	Alkaline Scrubber Hold Tank (TK-2.5)	TK-2.5	3166-EF-22	0	$0	420
2	ROTARY CYCLONE DISCHARGE FEEDER			0	$1,296	
2	Sur-Lite Waste Gas Incinerator			1/2	$19,502	
2	Upper Hearth Burners				$3,315	
5	CA Exhaust Blower BO-5.1	BI-5.1		2	$35,106	60
5	Condenser	PC-5.002		0	$7,111	
5	Precooler	PC-5.1		0		620
5	CA Precooler (PC-5.1)	PC-5.1		0	$985	
5	Tailings Stripper (NH3 stripper)	ST-5.1		0	$3,309	

Section Number	Name	Equipment ID Number	Drawing Number	HP	Cost (1975)	Weight (lbs.)
5	Pregnant Solution Ammonia Stripper	ST-5.2		0	$3,131	
5	Leach Solution Ammonia Stripper	ST-5.3		0	$3,131	
5	Leach Liq. Storage Tank (TK-5.2)	TK 5.2		0	$1,620	
6	Horiz. Belt Filter. (F-6.2)	F-6.2	3166-EF-62	10	$23,000	4,000
6	In-Line Polishing Filter Precoat Type-Rubber Lined (F-6.3)	F-6.3	3166-EF-62	0	$116	8
6	Preg. Liquor Heat Exchanger (HX-6.1)	HX-6.1	3166-EF-61	0		300
6	Belt Filter Vacuum Pump (P-6.5)	P-6.5	3166-EF-62	1	Incl	
6	Filtrate Pump (P-6.6) X 2	P-6.6	3166-EF-62	1/2	incl	
6	Steam Stripper (ST-6.1)	ST-6.1	3166-EF-61	0	$3,634	850
6	Stripper Slurry Thickener Tank (TK-6.1)	TK-6.1	3166-EF-62	1/2	$7,731	7,100
6	Water Trap Silencer-Separator (TK-6.7)	TK-6.7	3166-EF-62	0	INCL. @ belt filter	
7	Two Electrowinning(?) Cells w/ Component Equipment (EC-7.1)	EC-7.1	3166-EF-71	0	$1,836	
7	Cobalt EW Cell	EC-7.1		0	$1,601	
7	Co Ew Cell	EC-7.2		0	$1,601	
7	Extraction Mixer-Settlers (MS-7.1)	MS-7.1	3166-EF-71	0	$460	
7	Organic Scrub Mixer-Settlers (MS-7.2)	MS-7.2	3166-EF-71	0	$378	
7	Organic Wash Mixer-Settlers (MS-7.3)	MS-7.3	3166-EF-71	0	$67	
7	Strip Mixer-Settler (MS-7.4)	MS-7.4	3166-EF-71	0	$67	
7	Raffinate Storage Tank (TK-7.1)	TK-7.1	3166-EF-71	0		
7	Co Pregnant Solution Tank	TK-7.11		0	$499	
7	Cobalt Surge Tank	TK-7.12		0	$1,787	
7	Co Raffinate Tank	TK-7.13		0	$499	
7	Spent Wash Solution	TK-7.16		0	$366	
7	Organic Scrub Surge Tank (TK-7.2)	TK-7.2	3166-EF-71	0	$279	
7	Catholyte Storage Tank (TK-7.3)	TK-7.3	3166-EF-71	0		
7	Sulfide Precip Tank (TK-7.4)	TK-7.4	3166-EF-71	0		
7	Organic Storage Tank (TK-7.5)	TK-7.5	3166-EF-71	0	$279	
7	Filtered Feed Storage Tank (TK-7.6)	TK-7.6	3166-EF-71	0	$279	
7	Precoat Tank (TK-7.7)	TK-7.7	3166-EF-71	0	$67	
7	Wash Solution Tank	TK-7.7		0	$279	
7	Precoat Tank (TK-7.8)	TK-7.8	3166-EF-71	0	$67	
7	Strong Acid Tank (Ni)	TK-7.8		0	$279	
7	Cu loaded organic	TK-7.8		0	$279	
7	XI-51 Strip Organic Tank	TK-7.8		0	$279	
7		Tk-7-14		0	$279	

Section Number	Name	Equipment ID Number	Drawing Number	HP	Cost (1975)	Weight (lbs.)
7	Cu Electrolyte Tank	TK-7-18		0	$279	
7	SX Mixer settlers			7	$19,525	
8	Air Compressor (C-8.1)	C-8.1	3166-EF-82	10	$4,008	
8	Cooling Tower (CT 8.1)	CT-8.1	3166-EF-81	10	$3,705	
8	Air Dryer (D-8.2)	D-8.2	3166-EF-82	5		
8	Raw Water Supply Pump 50 GPM (P-8.3)	P-8.3	3166-EF-81	1		
8	Raw Water Supply Pump 50 GPM (P-8.3A)	P-8.3A	3166-EF-81	1		
8	CWS Pump 200 GPM (P-8.4)	P-8.4	3166-EF-81	2	$855	
8	Fuel Oil Transfer Pump 0.5 GPM (P-8.7)	P-8.7	3166-EF-82	1/2	$323	
8	Raw Water Storage (TK-8.1)	TK-8.1	3166-EF-81	0		
8	Feed Water Chemical Tank (TK-8.2)	TK-8.2	3166-EF-81	0		
8	Raw Water Pressure Vessel (TK-8.4)	TK-8.4	3166-EF-81	0		
8	Fuel Oil Storage (TK-8.5)	TK-8.5	3166-EF-82	0	$6,086	
8	Compressed Air Receiver (V-8.1)	V-8.1	3166-EF-82	0		
8	480V Motor Control Center				$23,991	10,000

8.3 Acknowledgments

Special acknowledgment is due to the following individuals and/or entities whose openness and willingness to share information proved invaluable to this undertaking:

Dr. Steven A. Keim, Ph.D., P.E, Senior Mining Engineer, and Senior Vice President of Marshall Miller & Associates' Blacksburg, Virginia, office who participated in this chapter as the Principal Investigator of Project MMA29956.

Dr. Joshua Werner, Ph.D., P.E., Mechanical Engineer, Assistant Professor, University of Kentucky, Mining Engineering – Institute of Sustainable Manufacturing, who undertook the investigation of the process audit prospects and who drafted this chapter.

Dr. Rick Honaker, Ph.D., P.E., Professor of Mining Engineering, University of Kentucky, College of Engineering, Department of Mining Engineering who participated in the process audit with site visits and who followed up with the critiques as well as assessments of the merits of the Arizona Plant.

Chuck Mullins, P.E., L.S. Chief Engineer, Kentucky River Properties, LLC, a member of the MMA 29956 team who actively participated in the evaluation of the Trus Joist facility near Chavies, Kentucky as the future site of major components of the Arizona Plant.

Mr. Antonio Cimetta, President and CEO of Cimetta Engineering & Construction Co, Inc., of Tucson, Arizona, who facilitated site visits to the Pilot Plant in Arizona and who assisted in data collection and organization.

Mr. Jared Garland, Project Engineer, Cumberland Mine Services, Inc, of Totz, Kentucky, subcontractor to Project MMA29956 who contributed significantly to the estimation of the cost to tear-down, transportation and re-assembly of major components of the Arizona Plant to the former Trus Joist Facility near Chavies, Kentucky.

Mr. Roger A Daugherty, President, Darco Energy Management Corporation of Buffington, South Carolina, a certified machinery and equipment appraiser, who provided insights to the market value of the various components of the Arizona Plant.

Hans E. Naumann, Jr., P.E. Principal Engineer, Marshall Miller & Associates, Inc., who contributed in the organization of support groups on behalf of this chapter of the Phase 1 report of Project MMA29956 and has assisted in the compilation of the narrative and exhibits to this chapter.