

Characterization and Extraction of Rare-Earth Elements from East Kootenay Coalfield Samples, Southeastern British Columbia

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Kuppusamy, V.K., and Holuszko, M.E. (2019): Characterization and extraction of rare-earth elements from East Kootenay coalfield samples, southeastern British Columbia; *in* Geoscience BC Summary of Activities 2018: Minerals and Mining, Geoscience BC, Report 2019-1, p. 33–44.

Introduction

Rare-earth elements (REE) make up a group of 17 elements in the periodic table, including 15 lanthanides and 2 chemically similar transition metals, Sc and Y. Using the atomic number, REE are classified as heavy REE and light REE, with elements from Tb to Lu and Y belonging to the former group and La to Gd and Sc belonging to the latter group (Moldoveanu and Papangelakis, 2013; Zhang et al., 2015). With the emergence of new clean energy and defense-related technologies, consumption of rare-earth elements (REE) has increased rapidly (Tse, 2011). For example, it is projected that the demand for Dy, one of the REE, is expected to increase as much as 2600% by 2025 (Standing Committee on Natural Resources, 2014). In addition, traditional rare-earth ore deposits are quickly being depleted and are expected to meet the demand for only the next 15-20 years (Seredin and Dai, 2012). Based on supply and demand, REE are classified as critical elements by the United States and the European Union due to their importance in clean energy and defense applications (United States Department of Energy, 2010; European Commission, 2017). The National Energy Technology Laboratory in the United States has conducted a prospective analysis of coal deposits as a source of REE using the United States Geological Survey coal database, which contains the concentrations of rare-earth elements across United States coalfields (Bryan et al., 2015). Further, the United States Department of Energy has committed US\$10 million to several projects researching the techno-economic feasibility of domestic separation technologies of REE from coal and/or its byproducts containing a minimum of 300 ppm total REE and concentrating the REE to a level greater than or equal to 2% (by weight) in processed streams (United States Department of Energy, 2016).

There is an indication of the presence of REE in some Canadian coal deposits, especially in British Columbia (BC) coalfields (Goodarzi, 1988; Birk and White, 1991; Goodarzi et al., 2009); however, there is no proper quantification, characterization and extraction analysis currently available for coal deposits in BC or in other coal deposits across Canada. The purpose of this ongoing research is to characterize and quantify the REE and their mode of occurrence in BC coal deposits and in different processing products of coal for possible extraction study. Some of the initial results of the study were reported previously (Kumar et al., 2018). This paper shows the preliminary characterization and extraction results conducted on a sample received from a southeastern BC coal mine.

Materials and Methods

Sample

Samples and the method used in this study is part of the continuing research published in *Summary of Activities Report 2017: Minerals and Mining* (Kumar et al., 2018). As described by Kumar et al. (2018), a run-of-mine (ROM) coal sample of approximately 300 kg was collected from a mine in the East Kootenay coalfield, southeastern BC, and shipped in barrels to the Coal and Mineral Processing Laboratory at The University of British Columbia. Representative samples were obtained for further testing using the standard procedure for coal sample preparation, ASTM D2013/D2013M-12 (2012). Proximate analysis was conducted on the representative samples in duplicates using the standard methods ASTM D3174-12 (2012), ASTM D3172-13 (2013), ASTM D3173/D3173M-17a (2017) and ASTM D3175-17 (2017).

Tree Release Analysis

A tree release analysis procedure (Figure 1) can be used to generate an optimum separation performance curve for the flotation process, which is used for cleaning fine coal (Mohanty et al., 1998). A representative feed sample of 300 g was crushed to -500μ m. Flotation tests were con-

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ducted on the 2 L Denver laboratory cell with an initial slurry content of 14.3%. The coal sample was wetted for 5 minutes with an impeller speed of 1300 rpm. A flotation reagent emulsion of 25 g/t of methyl isobutyl carbinol (MIBC) and 50 g/t of kerosene in 200 mL of water was made using a high-speed laboratory blender and added to the flotation cell. The coal slurry with reagents was further conditioned for 10 minutes, followed by another 25 g/t of MIBC addition and conditioning for 1 minute. Using an aeration rate of 4-5 L/min, the concentrate was collected until the froth was barren. The initial products were each successively refloated twice to obtain eight products, as shown in Figure 1. The final products from the tree release analysis were dried at 65°C, weighed to obtain the yield data and analyzed for ash content. For the feed sample, the tree release analysis was conducted in duplicate. Based on the ash content of the products, three products were selected with low (<10%), middlings (10-60%) and tailings (>60%) to find REE concentration in the flotation products.

Sequential Extraction

Sequential extraction was first developed to analyze particulate trace metals (Tessier et al., 1979). This procedure was later modified for coal (Finkelman et al., 1990; Finkelman, 1994) and successfully applied to REE in coal by Dai et al. (2002). A similar extraction method has also been applied to other trace elements in coal such as arsenic, mercury and selenium (Dai et al., 2004; Zhang et al., 2007; Zheng et al., 2008; Bai et al., 2011; Zhou et al., 2016). Based on these

studies, a simplified three-step procedure was adopted and used in this study, as shown in Table 1.

Alkali-Acid Leach Test

Background

To improve the utilization properties and to mitigate the environmental impacts, the ROM coal is processed to remove undesirable impurities such as mineral matter and sulphur to meet the market quality requirements depending on the final use. Coarse coal particles (>0.5 mm) are processed using gravity-based separation techniques. Fine coal particles (<0.5 mm) of metallurgical coal are generally treated using flotation (Laskowski, 2001). Due to the limited effectiveness of physical beneficiation methods to remove fine dispersed mineral matter in coal, alternative techniques such as chemical leaching processes are used to produce clean coal. Among several alternative processes, alkali-acid leaching has been studied in detail (Meshram et al., 2012). Waugh and Bowling (1984) showed that for Australian coals, alkali-acid leaching can be used to reject more than 90% of ash-forming minerals with a 10% sodium hydroxide (NaOH) solution at 200–300°C under vapour pressure followed by a dilute acid treatment. Wang et al. (1986) achieved 90% reduction in mineral matter for Australian coals using milder temperature conditions at 125-185°C with a 4-80% NaOH solution. Sriamoju et al. (2017) explained the mechanism of the coal chemical leaching process and proposed an acid regeneration method using the pilot scale test.



Figure 1. Schematic diagram of tree release analysis procedure. In this test, the coal sample (feed) was first separated using flotation into float F1 and tailings T1 (level 1). In the level 2, both float F1 and tailings T1 were refloated separately to obtain a total of four products: float F2 and tailings T2, and float F3 and tailings T3, respectively. In level 3, the four products from the second stage were again floated to obtain final products, P1 to P8. The yellow bubble and grey portion in the figure represent froth and tailings obtained during flotation, respectively.



There is some similarity between the alkali-acid leach process of coal and processing rare-earth concentrates. Traditional rare-earth concentrates such as monazite and xenotime (rare-earth phosphate minerals) are generally processed using sulphuric acid or sodium hydroxide. In the caustic process, finely ground monazite (<45 μ m) is leached at 140–150°C using 60–70% NaOH for 3–4 hours to form insoluble hydroxides of rare-earth elements. These hydroxides are dissolved in acid for subsequent processes (Habashi, 2013; Krishnamurthy and Gupta, 2016). The reaction mechanism is given below:

$$\mathsf{REPO}_4 + 3\mathsf{NaOH} \rightarrow \mathsf{RE}(\mathsf{OH})_3 + \mathsf{Na}_3\mathsf{PO}_4 \tag{1}$$

There is no previous study on using an alkali-acid leach for the simultaneous extraction of REE and clean coal from the coal waste stream. In the present work, alkali-acid leaching was conducted on coal tailings obtained from the ROM coal treated by flotation.

Coal Tailings Sample Preparation

The coal tailings samples for the alkali-acid leach tests were obtained using flotation of the representative ROM sample. For the flotation test, a representative feed sample of approximately 4.5 kg was crushed to obtain a size of -500 µm. Approximately 1 kg of feed coal was used in batches to conduct flotation tests in a 10 L Denver laboratory cell. The coal sample was initially conditioned for 5 minutes at an impeller speed of 1300 rpm. A flotation reagent emulsion of 25 g/t of methyl isobutyl carbinol (MIBC) and 240 g/t of kerosene in 200 mL of water was prepared using a high-speed laboratory blender and added to the flotation cell. Further, the coal slurry with reagents was conditioned for 15 minutes followed by another 25 g/t of frother (MIBC) addition and conditioning for 1 minute. Using an aeration rate of 4.5 L/min, the concentrate was collected until the froth was barren. Tailings were pressure filtered and both the filtered tailings as well as the concentrate were dried at 65°C. After drying, the tailings and concentrate collected from different flotation batches were homogenized and representative samples were split to conduct ash analysis. Tailings from these flotation tests were used as the feed sample for alkali-acid leach tests.

Alkali-Acid Leach Test Procedure

A 15 g representative sample of the coal tailings was mixed with 60 g of solution of different concentrations of NaOH, at 20% solid content. Alkali leaching was performed using a Parr® 4590 micro stirred reactor with 100 mL reactor volume and carried out at appropriate temperatures with vapour pressure for different leaching times. Alkali-leached coal slurry was filtered to obtain the NaOH leach solution and the solid residue was both washed and filtered twice to remove any excess NaOH solution and water-soluble components. The filtered alkali leach residue was repulped and 7.5 wt. % of concentrated hydrochloric acid (HCl) was added to the slurry in a beaker. The beaker was placed in a water bath maintained at 50°C for 30 minutes and the entire set-up was mounted on a magnetic stirrer to agitate the slurry during the acid leaching process. Finally, the acid-leached coal slurry was filtered to obtain the HCl solution and the clean coal residue that was later washed and filtered until the wash water reached a neutral pH. The clean coal (representing residue obtained after alkali-acid leaching) was dried at 65°C and analyzed for ash content. To obtain statistically reliable data at the preliminary stage of the study, experimental runs for leaching tests were obtained by adopting an experimental design using JMP® software with four levels for NaOH concentration (5–30 wt. %), three levels for temperature (130–190°C) and three levels for time (30-120 minutes).

Analytical Methods

For total REE quantification, feed and test product samples (0.2 g) were added to lithium metaborate and lithium tetraborate flux and mixed thoroughly. The samples were then fused in a furnace at 1025°C. Finally, the resulting melts were cooled and digested in an acid mixture containing nitric, hydrochloric and hydrofluoric acids. The digested solutions were then analyzed by inductively coupled plasma–mass spectrometry (ICP-MS). The analytical results were corrected for inter-elemental spectral interference. This analysis was conducted by the MS Analytical laboratory (Langley, BC). In this study, REE in coal are expressed as follows: whole coal basis (REE concentration in the coal sample) and ash basis (REE concentration in the ash of the coal sample).

Table 1. Sequential extraction procedure used in this study. Abbreviation: ICP-MS, inductively coupled plasma-mass spectrometry.

Step	Mode of occurrence	Procedure
1	Ion-exchangeable phase	A 15 g sample was transferred into a 150 mL beaker and 90 mL of 1 N ammonium acetate was added to the beaker. The slurry was magnetically stirred for 24 hours at room temperature and filtered using Whatman grade 42 filter paper. The liquid was labelled and stored for ICP-MS analysis and the solid residue was dried at 65°C.
2	Organic bound	The dried residue from step 1 was mixed with chloroform (1.47 g/cm ³) and was magnetically stirred for 24 hours at room temperature. Then, the mixture was centrifuged at 3500 rpm for 20 minutes to get float and sink fraction. The float fraction (<1.47 g/cm ³) was air dried in the fume hood and stored for ICP-MS analysis.
3	Inorganic bound	The sink fraction (>1.47 g/cm ³) from step 2 was air dried in the fume hood and stored for ICP-MS analysis.



Quantitative X-ray diffraction data were collected on selected samples to identify the mineral phases. The X-ray powder-diffraction data were collected over a range $3-80^{\circ}$ of 2θ with CoK α radiation on a Bruker D8 Advance Bragg-Brentano diffractometer equipped with an Fe monochromator foil, $0.6 \text{ mm} (0.3^{\circ})$ divergence slit, incident- and diffracted-beam Soller slits, and a LYNXEYE XE detector. The long fine-focus Co X-ray tube was operated at 35 kV and 40 mA using a take-off angle of 6° . The X-ray diffractogram was analyzed using the International Centre for Diffraction Data[®] PDF-4+ 2019 database using the search/match module by Bruker. X-ray powder-diffraction data of the sample were refined with Rietveld program TOPAS 4.2 by Bruker.

Scanning electron microscope–energy X-ray dispersive spectrometry (SEM-EDX) of selected samples was carried out to examine finely disseminated rare-earth minerals and to determine their mineral associations and size using a FEI QuantaTM 650 scanning electron microscope (SEM) operated at 20 kV at a working distance of 10 mm.

Results and Discussion

Proximate Analysis

The result of the proximate analysis of ROM coal and tailings coal generated for leaching tests is listed in Table 2. According to the ASTM D388-17 (2017) standard, feed coal is classified as medium-volatile bituminous coal.

Flotation Behaviour of the Feed Sample and REE Concentration

The tree release analysis provided a separation performance curve achievable under a predetermined set of conditions for flotation of the feed sample. The shape of the performance curve indicates the ability to clean the coal using flotation such as ideal separation, easy cleanability, difficult cleanability and impossible separation (Laskowski, 2001). As shown in Figure 2, at 10% ash, 83% yield of the clean coal can be achieved for feed coal. Figure 2 also shows that the sample exhibits easy cleanability using flotation and its combustible recovery exceeded 99%.

The feed sample and selected three tree release analysis products containing low-ash (<10%), middlings (10–60%) and high-ash (>60%) products were analyzed for rare-earth element concentrations, the results of which are presented in Table 3. The highest concentration of REE on an ash basis was reported in the low-ash flotation product (550 ppm; Table 3). On whole coal basis, however, the concentration of REE in the low-ash flotation product (25 ppm; Table 3) was six times lower than tailings (173 ppm). It shows that the concentration of REE in the tailings enriched during the flotation process, which indicates the association of REE with mineral matter in the feed coal. Further, REE distribution calculated using the mass balance on the low-ash products (<10% ash), middlings (10–60% ash) and tailings (>60% ash) as obtained from the tree release analysis re-

 Table 2. Proximate analysis of feed (run-of-mine

 [ROM] coal) and tailings coal samples.

Samplo	ROM coal	Tailings		
Sample	(as-received)	(dried)		
Moisture (%)	1.33	1.46		
Volatile matter (%)	18.22	15.71		
Ash (%)	24.82	46.21		
Fixed carbon (%)	55.63	36.61		



Figure 2. Flotation performance curve obtained from tree release analysis of the feed sample. Test was performed in duplicates as represented by test 1 and test 2 data.



vealed that most of the REE by weight percent was reported to the middlings plus tailings streams, as shown in Figure 3.

Sequential Extraction Results

The results of the sequential extraction of the feed sample is shown in Figure 4. Total recovery of approximately 88% of the REE was reported, whereas the loss is reported as an error in Figure 4, which might be due to cumulative sampling error, analytical error and weight loss during the experiment. The ion exchangeable REE in the sample is negligible and most of the REE were below the detectable limit. The REE associated with the organic and inorganic portions of the coal was found to be approximately 20% and approximately 68%, respectively. As shown in Figure 3, most of the REE is reported in middlings plus tailings, which further validates the result of the sequential extraction.

Dai et al. (2002), using a similar sequential extraction procedure as in this study, showed that coal samples from the Shitanjing, Shizuishan and Fenfeng coalfields contain approximately 90% of REE associated with aluminum silicate minerals. The organically associated REE were reported previously, especially in low-ash and low-rank coals (refers to subbituminous and lignite; Eskenazy, 1987, 1999; Dai and Finkelman, 2018). Because coal in this study belongs to high-ranking coal (bituminous), it can be inferred that not all rare-earth elements in the studied samples of clean coal have an organic association. Further, the density-based sequential extraction process does not account for the fine-grained minerals dispersed within the coal matrix and thus erroneously assigns the organic-associated REE percentage. Direct measurement using SEM and other advanced characterization techniques should, therefore, be incorporated to understand the exact nature of organic-associated REE in the sample.

Figure 5 shows SEM-EDX images of clean coal and tailings. Spot analyses have shown that REE is associated with aluminosilicate minerals (likely clays) in both clean coal and tailings, thus it can be inferred that the actual amount of organic-associated REE in the coal is less than the reported value in the results of the sequential extraction. It can be further concluded that inorganic associations are the dominant mode of REE occurrence in the studied coal samples. REE associated with clays were previously reported in the Jungar coalfield, China; the Huaibei coalfield, China; the

Table 3. Rare-earth element (REE) concentration in feed sample and tree release analysis products (in ppm). Low ash denotes product with <10% ash, middlings denote product with 10–60% ash and tailings denote product with >60% ash.

Sample	Sample name	Ash (%)	Sc	Ce	Dy	Er	Eu	Gd	Но	La	Lu	Nd	Pr	Sm	Tb	Tm	Y	Yb	Total REE (ash basis)	Total REE (whole coal basis)
S1	Feed coal	24.82	12.1	89.5	7.5	4.6	2.2	8.5	1.7	48.0	0.6	41.5	10.6	8.1	1.3	0.8	48.0	4.9	289.8	71.9
S2	Low ash product	4.5	N/A	177.7	14.7	9.6	4.2	15.8	3.6	97.7	1.3	82.2	21.1	15.3	2.4	1.3	93.3	9.3	549.4	24.7
S3	Middlings	35.92	11.1	76.8	6.7	4.3	1.7	7.1	1.5	41.2	0.6	34.0	9.0	7.0	1.2	0.7	42.9	4.3	250.0	89.8
S4	Tailings	85.55	7.01	64.9	4.9	3.1	1.4	5.6	1.1	35.0	0.5	30.2	7.9	5.9	0.9	0.5	30.9	2.9	202.3	173.1





Figure 3. Distribution of rare-earth elements (REE) by weight percent in the tree release analysis flotation products calculated using data shown in Table 3. Low ash denotes product with <10% ash, middlings denote product with 10–60% ash and tailings denote product with >60% ash.

Figure 4. Sequential extraction results of feed sample showing the distribution of rare-earth elements (REE) in various modes of occurrences. Ion-exchangeable REE was below the detectable limit. Error denotes the cumulative sampling error, analytical error and weight loss during the experiment.



Antaibao mining district, China; Far East deposits, Russia; and Fire clay coal, United States (Seredin, 1996; Dai et al., 2012; Zheng et al., 2007; Wang et al., 2008; Hower et al., 2018).

Leaching Test Results

Because it was found that most of the REE by weight percent reported in middlings and tailings, the coal tailings were used for the extraction study. Flotation of the feed coal, as described in section about coal tailings sample preparation, resulted in 10.12% and 46.21% ash with corresponding yields of approximately 58% and approximately 42% in the concentrate and tailings, respectively. Because coal tailings were used for subsequent alkali-acid leaching tests, particle size distribution was performed using the sieve analysis and the P_{80} of the sample was found to be 250 µm. To capture the interaction effect between the factors in the initial stage of the experiment, partial factorial design was adopted for this study. Figure 6 depicts the product ash obtained after the acid leaching stage for 12 experiments. Maximum ash reduction of approximately 70% was observed for test 10 (30% NaOH at 190°C for 30 minutes) with a corresponding product ash content of 14%. Further, fitting the data using the standard least squares model showed that significant factors affecting the product ash were NaOH concentration and temperature, with p-values of 0.00176 and 0.00786, respectively.

Effect of NaOH Concentration and Temperature

Sodium hydroxide concentration has a significant effect on the product ash, as shown in Figure 7. Irrespective of the temperature, increases in sodium hydroxide concentration proportionately decrease product ash content. The general mechanism for alkali-acid leaching of the ash content can be explained in two stages: the alkali stage and the acid stage. During the alkali-acid leaching stage, the mineral matter—such as quartz and clays—reacts with sodium hydroxide to form soluble sodium silicate and sodium aluminate (Waugh and Bowling, 1984; Mukherjee and Borthakur, 2001; Dash et al., 2013; Behera et al., 2017; Sriramoju et al., 2017).

 $SiO_2(s) + 2NaOH(aq) \rightarrow Na_2SiO_3(aq) + H_2O(aq) \tag{2}$

$$\begin{array}{ll} \mbox{Al}_2\mbox{O}_3\mbox{2SiO}_2 \cdot \mbox{2H}_2\mbox{O}(s) + 6\mbox{NaOH}(aq) \rightarrow \\ \mbox{Na}_2\mbox{SiO}_3(aq) + 2\mbox{NaAIO}_2(aq) + 5\mbox{H}_2\mbox{O}(aq) \end{array} \tag{3}$$

With the increase in alkali concentration, temperature and leaching time, concentrations of silicate and aluminate ions increase in the leach solution. Beyond the solubility limit, sodium aluminosilicate precipitates out as gel, as shown in the reaction below (Buhl et al., 1997; Mukherjee and Borthakur, 2001; Rahman et al., 2017):

$$\begin{array}{l} \mathsf{Na}_2\mathsf{SiO}_3(\mathsf{aq}) + \mathsf{Na}\mathsf{AlO}_2(\mathsf{aq}) + \mathsf{H}_2\mathsf{O}(\mathsf{aq}) + \mathsf{Na}\mathsf{OH}(\mathsf{aq}) \rightarrow \\ & [\mathsf{Na}_x(\mathsf{AlO}_2)_y(\mathsf{SiO}_2)_z \cdot \mathsf{Na}\mathsf{OH} \cdot \mathsf{H}_2\mathsf{O}] \end{array} \tag{4}$$

$$\begin{array}{l} 3Al_2Si_2O_5(OH)_5(s) + 8NaOH(aq) \rightarrow \\ 2Na_4Si_3Al_3O_{12}(OH) \downarrow + 9H_2O \end{array} \tag{5}$$

During the acid leaching stage, sodalite formed in the alkali-acid leach stage is dissolved into sodium and aluminum chlorides and silicic acid, as shown below:

$$\begin{split} & [\text{Na}_8(\text{AIO}_2)_2(\text{SiO}_2)_3 \cdot (\text{OH})_6] + 26\text{HCI} \rightarrow \\ & 8\text{NaCI} + 2\text{AICI}_3 + 3\text{SiCI}_4 + 16\text{H}_2\text{O} \end{split}$$

From the X-ray diffraction (XRD) analysis, as shown in Figure 8, clays and quartz account for more than 90% of the crystalline mineral matter in the feed coal. The reduction in



Figure 5. Scanning electron microscope—energy dispersive X-ray spectrometry (SEM-EDX) images of a) clean coal; elements observed in highlighted grains: AI, Er and O and b) tailings; elements observed in highlighted grains: AI, Ba, Br, Ca, Ce, O, P and Si.





Figure 6. Effect of test conditions on product ash (light blue bar) compared with feed tailings ash (black line). Light orange bar represents percentage reduction of product ash compared with feed tailings ash after the alkali-acid leach test.

ash content of coal tailings treated by alkali-acid leaching, therefore, follows a similar mechanism, as explained in equations 4–8.

Ash reduction was more pronounced when the temperature was increased from 130 to 160°C, as shown in Figure 7. But when the temperature was further increased to 190°C, the ash reduction was comparatively less effective.

Recovery of Rare-Earth Elements

Because alkali-acid leaching of coal leaches clay minerals, and because of the association of REE with clay minerals, it was expected that there might be a relationship between the ash content in clean coal residue and REE recovery. To understand the relationship between ash reduction in the alkali-acid leach test and rare-earth recovery, clean coal residue samples with varying ash percent were selected for REE element analysis. REE recovery was calculated for individual elements using equation 7 and the results are shown in Figure 9. Promethium is extremely rare and generally considered not to exist in nature (Resende and Morais, 2010), and

scandium was not assayed, so neither of these elements were included in the calculation.

REE recovery (%):
$$1 - \frac{REE \text{ in Residue}}{REE \text{ in Feed}} \times 100$$
 (7)

The highest recovery of REE was observed in test 1 and the corresponding ash content of the clean coal was 24.6%. Even though test 13 resulted in the lowest ash content for clean coal, the REE recovery was comparatively lower than in test 1. This observation implies that the recovery of REE was not increasing with the decreasing percent of ash in the clean coal; consequently, no correlation exists between the clean coal ash (as produced from alkali-acid leaching) and the REE recoveries. The REE extraction during the al-



Figure 7. Effect of sodium hydroxide concentration on product ash (%) for various temperatures: A (130°C), B (160°C), C (190°C).

kali-acid leaching of coal can be explained using traditional REE phosphate mineral-leaching mechanism. During the alkali leaching of coal, the REE phosphate is converted to an insoluble rare-earth hydroxide, as explained in equation 1. During the acid leaching stage, rare-earth hydroxide is leached into rare-earth chloride, as shown below:

$$RE(OH)_{3}(s) + 3HCI(aq) \rightarrow RECI_{3}(aq) + 3H_{2}O(I)$$
(8)

A previous study on monazite concentrate showed that phosphate dissolution increases with NaOH concentration, temperature and time (Panda et al., 2014). Figure 9 shows that the highest recovery of REE corresponds to 20 wt. % NaOH concentration, a temperature of 190°C and 60 min-



Figure 8. Quantitative X-ray diffraction (XRD) analysis of feed coal sample used for the alkali-acid leach test.





utes of leaching. The lowest recovery of REE corresponds to test 9 and test 8, which have lower concentrations of NaOH (5%) and temperature (130°C), respectively. Using ICP-MS analysis of the leach solution, the amount of REE in the HCl leach solution was found to be significantly higher than that in the NaOH leach solution for test 1. These data indicate that REE dissolution might follow the mechanism for traditional rare-earth phosphate leaching, as explained previously in equation 1. Further, it is observed that recoveries of light REE are higher than those of heavy REE for all tests.

Conclusions

Coal and its byproducts are viewed as a potential alternative for traditional rare-earth deposits. In this regard, several studies are being conducted for the characterization and extraction of rare-earth elements from coal sources. To understand the nature of REE association in coal as well as



to estimate the enrichment of rare-earth elements during coal beneficiation in East Kootenay coalfield samples, the feed sample was subjected to tree release analysis. Sequential extraction and mineralogical study using SEM-EDX identified REE associated with clays in both clean coal concentrate and tailings. The identification of REE associated with clays in the low-ash product reiterate the dominance of inorganic-associated REE in the samples.

Coal tailings generated using ROM samples from the East Kootenay coalfield in British Columbia were studied using an alkali-acid leaching process for the simultaneous extraction of clean coal and rare-earth elements. An approximate 70% reduction in the ash content of coal tailings was achieved to obtain clean coal. During the coal cleaning process, almost 97% of LREE and 76% of HREE were extracted simultaneously. The possible mechanisms of rare-earth elements extraction were also hypothesized.

There was no valid relationship found between ash reduction in coal (residue after leaching) and rare-earth-element extraction due to different target minerals available for leaching, such as silica and clay, for coal cleaning and rare-earth phosphates for REE extraction. Both processes depend on NaOH concentration, temperature and time. Subsequent study is required to optimize process parameters to maximize coal cleaning and rare-earth–element extraction simultaneously. The coal samples under consideration are of metallurgical quality and the production of clean coal and rare-earth concentrate from the coal tailings will enhance the economic viability of the process.

With the development of cost-effective extraction techniques for REE in coal, these elements will be extracted as byproducts of coal mining, thus strengthening brownfield operations by increasing profitability as well as potential green credits because these REE are used in clean energy technologies. In case of the greenfield operations, the possibility of extracting REE from coal will increase the competitiveness of deposits and more deposits may be feasible for mining and extraction.

Acknowledgments

The authors gratefully acknowledge the Geoscience BC scholarship. The authors also thank M. Mastalerz, Indiana University, and M. Mackay for their valuable comments and suggestions to improve this manuscript.

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