BR07 - Recovery of Scandium, Iron, and Aluminum from Bauxite Residue by Carbothermic Smelting Followed by Acid Baking – Water Leaching

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Abstract

The management of bauxite residue produced in the Bayer process is a major economic and environmental cost associated with the production of alumina from bauxite ore. Conventional disposal of this highly alkaline and mineralogically complex solid byproduct in large dry heaps or tailing ponds is expensive and carries environmental risks. On a positive side, bauxite residue is rich in several valuable metals such as scandium, iron, and aluminum, which can be valorized, bringing value to this byproduct. In this study, an integrated multistep process is developed to extract these valuable metals from a Canadian bauxite residue by employing carbothermic smelting to separate crude metallic iron, followed by acid baking – water leaching to extract scandium and aluminum. To avoid the production of acidic solid waste, the calcium- and silicon-rich leaching residues are thermally desulfated to produce regenerated smelting flux for the smelting step of the process. The use of residue desulfation can potentially enable the recovery of the consumed sulfuric acid by the acid baking step of the process, reducing the overall reagent consumption and increasing process efficiency. This waste valorization process was developed and optimized using design of experiment and response surface methodology techniques alongside morphological and mineralogical characterizations to elucidate the underlying physicochemical mechanisms of the smelting and acid baking – water leaching processes and to demonstrate the feasibility of fully valorizing bauxite residue at the laboratory scale. The results of this study are the first step in utilizing bauxite residue as a low cost, readily available resource for valuable materials, which can help reduce the environmental impact and economic costs of the Bayer process.

Keywords: Bauxite residue, carbothermic smelting, acid baking – water leaching, near-zero-waste process.

1. Introduction

Around the world, there are increasing numbers of studies investigating techniques for the efficient utilization of bauxite residue, which is the solid byproduct of the Bayer process for alumina production from bauxite ore. Because of its high production rate (0.7–2 tonnes per tonne of Al_2O_3 produced, resulting in 150–165 million tonnes produced per year) and large global stockpiles (3–4 billion tonnes), [1–3] as well as problematic physical and chemical properties, in particular high alkalinity and complex mineralogy, the management of bauxite residue presents considerable economic and environmental costs. Currently, less than 3% of the global production of bauxite residue is utilized productively, primarily as an additive for construction materials, with the balance stored in dry heaps and tailing ponds, which can present considerable maintenance costs and potential environmental impacts. [2,4]

On the positive side, bauxite residue holds significant promise as a potential secondary resource of valuable metals because it is enriched in iron, scandium, aluminum, titanium, among others.[5] Bauxite residue is a particularly attractive resource because of its low cost, readily availability globally, and production in facilities that already have industrial infrastructure. Despite several attempts to valorize bauxite residue, typically focussing on direct leaching using mineral acids (hydrochloric, sulfuric, or nitric acid), no large scale approaches have been globally adopted due to the practical challenges of large consumption of concentrated acid solutions, long leaching durations, production of large amounts of acidic wastewater and solid residues, and the difficult separation of co-extracted elements, in particular iron and scandium. [6–9]

Acid baking – water leaching is a process that has previously been developed to address some of the challenges associated with direct acid leaching for the recovery of scandium.[5] In this process, solid bauxite residue is mixed with concentrated (98 wt%) sulfuric acid to neutralize its alkalinity, break down its mineral phases and convert the insoluble minerals to soluble sulfates. The acid–bauxite residue mixture is then baked at 200–400 °C to induce changes in the crystal structures to make them more soluble and achieve a higher extent of sulfate conversion. The resulting acid-baked residue is then leached in water at ambient temperature and pressure to dissolve the soluble sulfate products. This processing approach, which is similar to processes used in the rare earth industry,[10] offers the advantages of reduced acid and water consumption, shorter leaching durations, and reduced waste production.

One of the primary challenges associated with the direct acid baking – water leaching of bauxite residue for the recovery of scandium is the co-extraction of iron during leaching. In bauxite residue, scandium is closely associated with the iron-bearing phases. [5,11] Thus, any technique with high extraction of scandium results in high co-extraction of iron, which requires costly separation steps.[12] One approach for iron separation from bauxite residue is the preemptive removal of iron by carbothermic smelting.[6]

This pyrometallurgical route, adapted from iron and steel production, is based on the high temperature (1400 – 1600 °C) reduction of iron(III) of the bauxite residue, present as Fe_2O_3 and FeO(OH) into crude metallic iron, using carbon typically added as coal, according to reaction 1:

$$Fe_2O_{3(s\ or\ l)} + 1.5\ C_{(s)} \rightarrow 2\ Fe_{(l)} + CO_{2(g)} \tag{1}$$

This smelting approach is promising because liquid metallic iron is immiscible with the molten slag containing other components of the bauxite residue. In this process, fluxing agents are added to control the melt viscosity and to facilitate the phase separation. As a result, the crude iron can be separated as a value-added product for use as a feed for steelmaking, [13,14] while the slag can be treated to recover scandium, aluminum, and other valuable metals.

Previous studies on the recovery of valuable materials from bauxite residue slag have typically utilized direct acid leaching for the extraction step;[13,15] however, similar to the processing of unsmelted bauxite residue, acid baking – water leaching has the potential to achieve more efficient extraction compared with direct acid leaching. Another common shortfall of previous studies on bauxite residue valorization is the production of large amounts of acidic leaching residue, which present an additional environmental challenge and considerable barrier to the adoption of this process.

In this work, an integrated process consisting of carbothermic smelting followed by acid baking – water leaching was developed to extract metallic iron, scandium, and aluminum from a Canadian bauxite residue. The acid baking – water leaching step of this process was optimized to determine the optimum operating conditions for scandium and aluminum extraction, and to gain insight into the physicochemical mechanisms of this process. Moreover, the potential for recycling of leaching residues produced after the water leaching step was investigated by using thermal desulfation to regenerate smelting flux. This investigation integrates with a larger venture to develop an environmentally and economically sustainable process for the utilization of bauxite residue as a resource for valuable metals, such as scandium, iron, and aluminum with minimal waste generation. If this clean valorization of bauxite residue is enabled, it can help reduce the environmental burdens of managing bauxite residue stockpiles while securing a secondary resource for valuable metals that are in high demand.

2. Materials and Methods

2.1 Starting Material

The bauxite residue utilized in this work was obtained from the Rio Tinto Vaudreuil alumina plant located in Jonquière, Quebec, Canada. The compositional, mineralogical, and microstructural characterization of this material has been reported elsewhere.[5] This bauxite residue contains 31.1 mg_{Sc}/kg scandium and its major components are iron (24.3 wt%), aluminum (11.8 wt%), titanium (3.8 wt%), calcium (2.3 wt%), sodium (5.2 wt%), and silicon (6.1 wt%).

2.2 Carbothermic Smelting and Slag Processing

Carbothermic smelting experiments were performed in a box Furnace (Carbolite Gero) equipped with a fume extraction canopy, under 1.4 L/min argon flow (99.998 vol.%, Linde). The bauxite residue was mixed with lignite coal (< 125 µm, VWR), calcium oxide (99.5 wt%, Materion), and silicon (IV) oxide (99.5 wt%, Alfa Aesar) as fluxing agent. This mixture was pressed into a pellet under 240 MPa pressure for 3 min to enhance the contact between bauxite residue, lignite coal, and the fluxing agent. The pellets were placed in graphite crucibles and smelted at 1600 °C for 1 h.

After smelting, two distinct phases were obtained in the crucible. The iron-depleted slag product was manually separated from the strongly magnetic iron-rich metallic phase. To remove any remaining small droplets of iron from the slag phase, the slag was pulverized in a puck mill and magnetically separated in a Davis tube wet magnetic separator (2 passes at 200 mT). The remaining non-magnetic slag fraction was dried, sampled for analysis, and used for acid baking – water leaching trials.

2.3 Acid Baking – Water Leaching and Residue Desulfation Trials

The acid baking – water leaching of the slag after smelting was tested and optimized by response surface methodology to determine the effect of four parameters, i.e., baking temperature (X_1), acid to slag ratio (X_2), water to acid-baked slag ratio (X_3), and leaching time (X_4) on the extraction efficiency of scandium and aluminum. The quadratic response surface for this system was assessed using a Roquemore Hybrid design of experiments, which combines the benefits of near-minimum point experimental designs and central composite designs.[16] Table 1 present a summary of the trials performed. The experimental data was fitted to quadratic response surface models of the form in equation 1, and simplified with 95% confidence, as described elsewhere.[5] The testing parameter levels were selected on the basis of our previous work on the acid baking – water leaching of unsmelted bauxite residue.

$$\hat{y} = \hat{\beta}_0 + \hat{\beta}_1 X_1 + \hat{\beta}_2 X_2 + \hat{\beta}_3 X_3 + \hat{\beta}_4 X_4 + \hat{\beta}_{12} X_1 X_2 + \hat{\beta}_{13} X_1 X_3 + \hat{\beta}_{14} X_1 X_4 + \hat{\beta}_{23} X_2 X_3 + \hat{\beta}_{24} X_2 X_4 + \hat{\beta}_{34} X_3 X_4 + \hat{\beta}_{11} X_1^2 + \hat{\beta}_{22} X_2^2 + \hat{\beta}_{33} X_3^2 + \hat{\beta}_{44} X_4^2 \quad (2)$$

Table 1. Summary of response surface trials performed for acid baking – water leaching optimization. Each trial produced 3 data points, corresponding to the three sampling time points.

Test ID	X_1: Baking Temperature	X_2: Acid to Slag Ratio	X_3: Water to acid-baked slag Ratio	X_4: Leaching Time
	°C	g_{H2SO4}/g_{SBR}	mL_{H2O}/g_{ABSBR}	min
1	300	0.78	16.1	30, 120, 210
2	300	0.78	3.9	30, 120, 210
3	262	1.25	12.5	30, 120, 210
4	405	0.94	12.5	30, 120, 210
5	338	0.30	12.5	30, 120, 210
6	195	0.61	12.5	30, 120, 210
7	338	1.25	7.5	30, 120, 210
8	405	0.61	7.5	30, 120, 210
9	262	0.30	7.5	30, 120, 210
10	195	0.94	7.5	30, 120, 210
11	300	0.78	10.0	30, 120, 210
12	300	0.78	10.0	30, 120, 210
13	300	0.78	10.0	30, 120, 210

In the acid baking step, the magnetically separated bauxite residue slag (SBR) was mixed in a heat resistant crucible with concentrated (98 wt%) sulfuric acid at acid to slag ratio of 0.30 to 1.25 g_{H2SO4}/g_{SBR}, forming a wet slurry. On the contrary to the observed behavior in unsmelted bauxite residue,[5] the slag did not immediately and violently react with acid upon mixing to form a clay-like paste, instead it formed a slurry. The slurry was baked in a box furnace (Degussa-Ney Vulcan™) installed in a ventilated fume hood at 195–405 °C for 2 h. The acid-baked slag (ABSBR) which was in a dry solid form was manually crushed to break apart large aggregates of particles.

In the water leaching step, the acid-baked slag was leached with deionized water at 25 °C in glass reaction vessel equipped with magnetic stirring (400 rpm) at a water to acid-baked slag ratio of 3.9 –16.1 mL_{H2O}/g_{ABSBR}. The leaching solution was periodically sampled (30, 120, 210 min) and filtered with 0.45 μm nylon syringe filters to halt the leaching reaction. The filtered samples were immediately diluted with 5 wt% HNO_3 solution for analysis. The leaching residue solids remaining after the water leaching trials were collected, mixed together and dried for use in residue desulfation trials.

The mixed leaching residue was thermally desulfated at 1400 °C for 3 h, under 1.4 L/min argon flow. For this process, the leaching residue was pelletized at 240 MPa for 3 min, then heat-treated in a graphite crucible. The resulting desulfated residue was collected and pulverized for characterization.

2.4 Compositional and Mineralogical Characterization

For compositional analysis, solids of interests were digested by alkali fusion (9 min at 1050 °C) using 49.75 wt% $Li_2B_4O_7$: 49.75 wt% $LiBO_2$: 0.50 wt% LiBr flux (Claisse) followed by dissolution in 5 wt% HNO_3. The digested samples were characterized by inductively coupled plasma optical emission spectrometry (ICP-OES – Perkin Elmer Optima 8000). Mineral phase identification was performed by powder X-ray diffraction (XRD – 2θ: 5-70°, scan rate: 1.25°/min, Rigaku MiniFlex 600 Diffractometer).

3. Results and Discussion

3.1. Bauxite Residue Smelting

Previous work has demonstrated that carbothermic smelting of bauxite residue results in separation of iron as a distinct phase. This separation enables recovery of scandium and aluminum from bauxite residue without the need for the downstream separation after the acid baking and water leaching step. [17,18] In this study, experimental trials were focused on optimizing three operating parameters of the smelting step, including smelting temperature, amount and composition of the flux, and carbon amount. Based on the results, optimum operating conditions that result in maximum scandium and aluminum content in the slag product were determined to be as follows:

1. Smelting temperature = 1600 °C
2. Smelting flux = 75 wt% CaO–25 wt% SiO_2 mixture
3. Flux amount = 0.1 g_{flux}/g_{BR}
4. Carbon amount = 0.07 g_{carbon}/g_{BR}

As shown in Table 2, the slag product of the smelting step had very low iron content compared with the starting bauxite residue, decreasing from 24.3 wt% to 0.28 wt%, while the concentrations of scandium and aluminum were almost doubled.

Table 2. Composition of dried bauxite residue feed and the slag product after smelting.
The slag compositions represent the average of 3 replicate characterizations.

Element	**Sc**	**Fe**	**Al**	**Ti**	**Ca**	**Na**	**Si**	**S**
	mg/kg	**wt%**	**wt%**	**wt%**	**wt%**	**wt%**	**wt%**	**wt%**
Starting BR	31.1	24.3	11.8	3.8	2.3	5.2	6.1	NA
Standard deviation	*(± 0.9)*	*(± 1.0)*	*(± 0.4)*	*(± 0.1)*	*(± 0.1)*	*(± 0.2)*		
Smelted BR slag	54.9	0.28	22.4	4.7	14.7	2.2	11.2	0.54
Standard deviation	*(± 0.44)*	*(± 0.02)*	*(± 0.27)*	*(± 0.05)*	*(± 0.20)*	*(± 0.12)*	*(± 0.12)*	*(± 0.02)*

The starting bauxite residue is a complex mixture of several mineral phases, including Fe_2O_3, $FeO(OH)$, $AlO(OH)$, $Al(OH)_3$, $CaTiOSiO_4$, TiO_2, SiO_2, and $(Na_6(Al_6(Si_{5.5}(H_4)_{0.5})O_{24})(NaCl)_{0.4}(NaOH)_{0.7})$. During the carbothermic smelting process, the mixture of bauxite residue, flux (SiO_2 and CaO), and lignite coal melts and undergoes several physical processes and chemical reactions which contribute to iron separation and scandium and aluminum concentration in the slag phase.

Above the liquidus temperature of the mixture, various components dissociate into a liquid phase containing ionic oxygen species (O^0, O^-, O^{2-}), metal cations (Fe^{2+}, Fe^{3+}, Al^{3+}, Na^+, Ca^{2+}, Ti^{4+}, Sc^{3+}), and ionic complexes (SiO_4^{4-}, AlO_4^{5-}, FeO_4^{5-}), whose speciation is dictated by acid-base and polymerization reactions.[19,20] Similar to the reactions in a blast furnace during ironmaking, the oxidized iron(III) within the bauxite residue (either in the solid pellet or the molten slag) is reduced to metallic iron(0) by coal and/or carbon monoxide gas (CO) produced by the partial oxidation of coal. [21,22] At 1600 °C, the carbothermic reduction of iron is spontaneous, but scandium and aluminum remain in the slag in their oxidized forms.[13,20] Because the metallic iron and the ionic slag are immiscible, they physically separate into two distinct phases that can be manually separated upon cooling.

Sodium is another major component of bauxite residue that is removed during the smelting step. At 1600 °C, sodium evaporates from the slag phase, which results in mass reduction of slag; hence, concentrating scandium and aluminum further. The sodium content of bauxite residue can be considered another valuable product, since it corresponds to NaOH entrained from the Bayer process in the bauxite residue. This lost sodium can potentially be recovered from the smelting fumes and converted to NaOH to be recycled back to the Bayer process.

Upon cooling, the slag components recrystallize into several calcium-aluminum-silicate phases ($Ca_xAl_ySi_zO_{(x+3/2y+2z)}$), such as $Ca_2Al_2SiO_7$, $CaAl_2Si_2O_8$, $CaO(Al_2O_3)_6$, CaO, and SiO_2. This slag is enriched in scandium which is in agreement with previous studies on blast furnace slag which showed scandium can substitute within the slag structure.[23] This scandium- and aluminum-enriched slag was used for the acid baking – water leaching trials.

3.2. Acid Baking – Water Leaching of Carbothermic Smelting Slag

Our previous study showed that acid baking – water leaching results in higher extraction of valuable metals (scandium and aluminum) from bauxite residue with less reagent consumption compared with direct acid leaching.[5,17,18] The principle behind acid baking – water leaching is to physically separate the acid digestion and conversion of insoluble minerals to soluble sulfate species from the dissolution of these soluble sulfate species in water, which allows optimization of processing conditions of these two processes independently, rather than compromising between the two, as is done during direct acid leaching.

Given that the composition of the smelting slag was considerably different from unsmelted bauxite residue, the response to the tested operating parameters, namely acid to slag ratio, baking temperature, water to acid-baked slag ratio, and leaching duration, were expected to be different from the optimum levels obtained for the unsmelted bauxite residue. Therefore, a series of response surface methodology trials were conducted to determine the optimal operating parameters that result in maximum scandium and aluminum extraction.

Empirical response surface models of the form given in equation 1 were constructed from the extraction results for these trials. The calculated factor effects, corresponding to the relative impact of changing each tested parameter are shown in Figure 1.

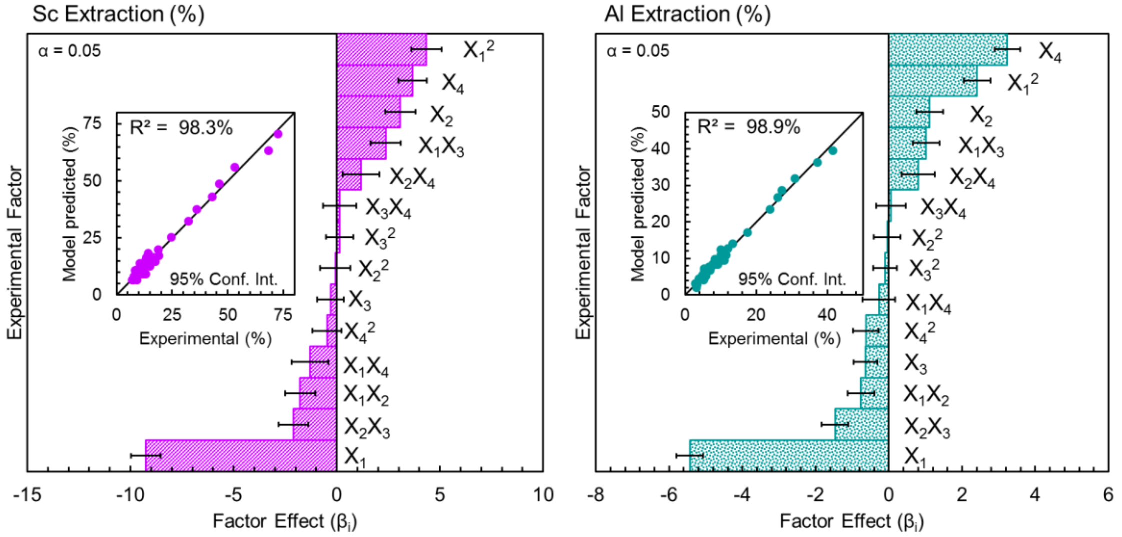

Figure 1. Ordered charts of factor effect coefficients for the empirical extraction models for scandium and aluminum. The inset residual plots show the correlation between the experimental and model predicted extractions. The empirical models were constructed from trials 1-13. The models were constructed at 95% confidence (α = 0.05), and the variability was estimated from repeat runs 11-13 (n = 3).

As shown in Figure 1, baking temperature (X_1) had the strongest effect on scandium and aluminum extraction, with a large negative effect and a concave quadratic inflection (X_1^2), which indicates that the highest extraction can be achieved at the lowest baking temperature (195 °C). On the contrary to unsmelted bauxite residue, in which higher extractions were observed at high baking temperature (400 °C), the smelted slag did not rapidly react with the acid upon mixing, which suggests that the reactions between the slag and acid are considerably slower than those of bauxite residue-acid reactions. Sulfuric acid is known to evaporate and decompose at above 300 °C; thus, given that the acid was not immobilized upon mixing with the slag, it is likely that considerable portions of the acid evaporated before having a chance to react; thus, reducing the extent of reaction and potential extraction.

Acid to slag ratio (X_2) had a positive impact on scandium and aluminum extraction, suggesting that the acid is the limiting reagent for the reactions. Furthermore, water leaching time also had a positive impact on extraction, indicating that the dissolution of the soluble phases was relatively slow.

Based on the empirical extraction models, optimum operating parameters within the tested range were predicted to be as follows:

1. X_1: Baking temperature = 195 °C
2. X_2: Acid to slag ratio = 1.25 g_{H2SO4}/g_{SBR}
3. X_3: Water to acid-baked slag ratio = 7.5 mL_{H2O}/g_{ABSBR}
4. X_4: Leaching duration = 210 min

The acid baking – water leaching process was performed using these parameter levels, and the measured extractions were compared with the model prediction results:

a) Scandium: Predicted extraction = 84.1 % Experimental extraction = 86.4 %
b) Aluminum: Predicted extraction = 46.3 % Experimental extraction = 50.5 %

As shown in the XRD diffractogram in Figure 2, the three primary phases produced during the acid baking process were $CaSO_4$, $(H_3O)Al(SO_4)_2$, and SiO_2. Given that the primary phases in the starting slag can be expressed in the general form $Ca_xAl_ySi_zO_{(x+3/2y+2z)}$, the reactions occurring during acid baking can be written as follows:

$$Ca_xAl_ySi_zO_{\left(x+\frac{3}{2}y+2z\right)_{(s)}} + (x+2y)H_2SO_{4(l)} \rightarrow x\,CaSO_{4(s)} + y\,(H_3O)Al(SO_4)_{2(s)} + z\,SiO_{2(s)} + \left(x+\frac{1}{2}y\right)H_2O_{(g)} \quad (3)$$

Among these products, $(H_3O)Al(SO_4)_2$ is water-soluble, $CaSO_4$ is sparingly soluble, and SiO_2 is insoluble. This allows extraction of aluminum by leaching in water. In this reaction, the monolithic calcium-aluminum-silicate phases of the slag break down and are transformed into new phases. As such, any scandium that is trapped within the crystal structure of the $Ca_xAl_ySi_zO_{(x+3/2y+2z)}$ or found in minor phases encapsulated within the primary slag phases would be released and exposed to H_2SO_4, which results in the formation of highly soluble $Sc_2(SO_4)_3$.[5]

Upon leaching with water, most of the scandium content and a large proportion of the aluminum content is leached, leaving behind an acidic solid residue comprising primarily $CaSO_4 \bullet 2H_2O$, $CaSO_4$, SiO_2, and AlO(OH). The dissolved scandium and aluminum can be further purified and recovered by solvent extraction, selective precipitation, or ion exchange. While it represents a considerably lower mass of solid waste than the initial bauxite residue, the production of acidic solids during the water leaching needs to be managed.

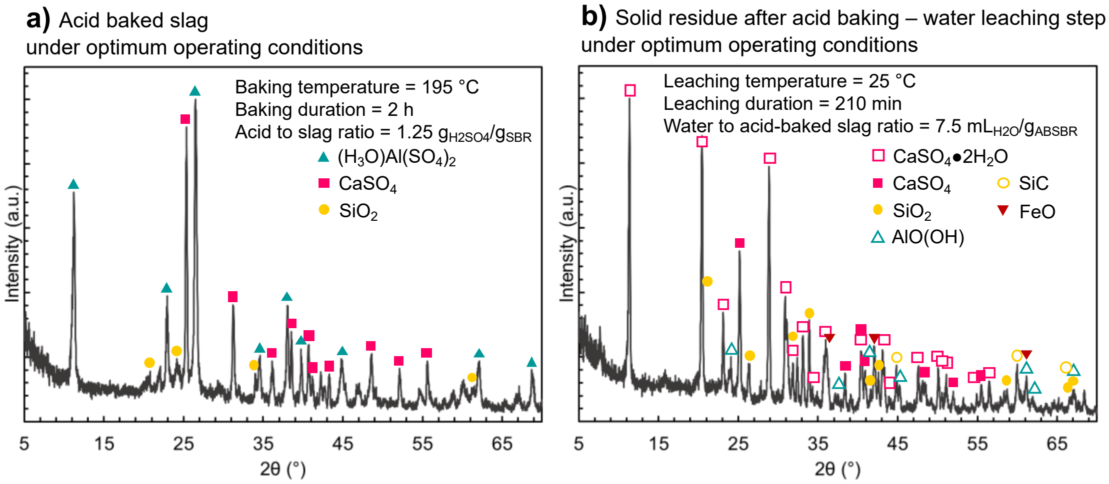

Figure 2. XRD diffractogram of a) acid baked slag under optimum operating conditions, and b) solid residue after acid baking – water leaching under optimum operating conditions.

3.3. Desulfation of Leaching Residue

To address the potential challenge of producing hazardous solid waste, the possibility of recycling of the leaching residues was investigated. Specifically, the residue was desulfated to produce a regenerated smelting flux. The $CaSO_4{\cdot}2H_2O$ and $CaSO_4$ are known to thermally decompose to CaO at temperatures above 1375 °C, according to reactions 4 and 5.[24]

$$CaSO_4 \cdot 2H_2O_{(s)} \rightarrow CaSO_{4(s)} + 2\,H_2O_{(g)} \tag{4}$$

$$CaSO_{4(s\ or\ l)} \rightarrow CaO_{(s\ or\ l)} + SO_{2(g)} + \frac{1}{2}O_{2(g)} \tag{5}$$

Aside from eliminating the production of solid waste, flux regeneration can reduce the consumption of reagents throughout the process because the regenerated flux can substitute the CaO and SiO_2 flux in the smelting step. Furthermore, the SO_2 gas produced by the desulfation reaction can be re-oxidized and condensed into regenerated H_2SO_4 for acid baking through conventional sulfuric acid production techniques.[25] A further advantage of this process is that the unrecovered scandium and aluminum can be recycled to the beginning of the process, allowing complete recovery in the eventual closed-loop operation.

To test the possibility of thermal desulfation for flux regeneration, the water leaching residues from tests 1-13 were collected and dried to produce a model mixed residue. The residue was then pelletized and heated to 1400 °C for 3 h to desulfate the $CaSO_4{\cdot}2H_2O$ and $CaSO_4$ phases. The composition of the residue before and after desulfation are shown in Table 3. One important note is that since several of the trials in tests 1-13 had relatively low scandium and aluminum extractions, the concentration of these elements in the mixed residue was relatively high.

Table 3. Elemental composition of the mixed solid residue from the water leaching step of trials 1–13 before and after thermal desulfation.

Parameter	Unit	Mixed residue	Desulfated residue
Mass	g	11.947	9.236
Sc content	mg/kg	28.9	43.4
Fe content	wt%	0.13	0.24
Al content	wt%	15.6	20.7
Ti content	wt%	4.3	5.8
Ca content	wt%	12.5	16.5
Na content	wt%	0.8	0.8
Si content	wt%	10.8	14.5
S content	**wt%**	**3.1**	**0.6**
Sulfur removal	**%**	**0**	**84**

The desulfation process at 1400 °C removed 84% of the sulfur contained in the water leaching residue, producing a flux that was primarily composed of calcium-aluminum-silicates, which should be equivalent to the CaO and SiO_2 used in the initial smelting, with some additional aluminum content. Future work is underway to test the useability of this desulfated residue as the fluxing agent.

From an industrial perspective, the addition of this desulfation step would not introduce considerable additional operational complexity or equipment costs, since the step could be performed in-situ during smelting. In previous studies investigating pilot-scale bauxite residue smelting, the fluxing agents were typically pre-heated in furnace ladles, with the bauxite residue added gradually once the target temperature was reached. As such, the desulfation could be performed during preheating by initially charging the furnace ladle with the water leaching residue and any additional make-up flux and collecting the SO_2-rich preheating off-gasses for acid recovery.

4. Conclusions

An integrated process was developed to extract scandium, aluminum, and iron from a Canadian bauxite residue. By combining bauxite residue smelting, acid baking – water leaching of the resulting slag product, and recycling of water leaching residues, value-added products can be efficiently recovered without producing hazardous waste byproducts.

In this work, the acid baking – water leaching step of the proposed process was optimized, resulting in 86% scandium and 51% aluminum extraction. During the acid baking step, the calcium-aluminum-silicates contained within the bauxite residue slag were broken down, and converted to soluble $(H_3O)Al(SO_4)_2$, sparingly soluble $CaSO_4$, and insoluble SiO_2. During water leaching at ambient temperature, the soluble sulfate species were dissolved, allowing extraction of scandium and aluminum, and leaving a residue composed primarily of $CaSO_4{\cdot}2H_2O$, $CaSO_4$, SiO_2, and AlO(OH). The solid residue was thermally desulfated to regenerate the flux for the smelting step of the process.

Future work will focus on the purification and recovery of extracted scandium and aluminum. Moreover, the process would be designed such that minimal solid, liquid or gaseous waste products are produced. This near-zero waste operation could be achieved by developing the separation and purification steps such that the wastewater is recycled to the water leaching step, implementing carbon capture to retain the CO_2 produced in the smelting step, and developing the sodium vapor capture and recovery during smelting.

Overall this work supports a larger project to develop an environmentally and economically sustainable process to fully utilize bauxite residue as a resource for several critical materials, while reducing reliance on environmentally-damaging primary mining and reducing the environmental footprint of the Bayer process.

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