Recovery of Scandium and Iron from Bauxite Residue by Carbothermic Smelting, Acid Baking – Water Leaching, and Solid–Liquid Extraction

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Abstract

Bauxite residue produced during the Bayer process represents a major cost in alumina production, as the conventional disposal of this complex alkaline byproduct by landfilling is expensive, requires large amounts of land and carries environmental risks. However, this residue contains considerable amounts of iron, aluminum, and critical materials such as scandium. In this work, a closed-loop valorization process is developed to sustainably recover these valuable materials. This process employs carbothermic smelting at 1600 °C to recover 99% of the iron content from the residue as crude metallic iron for steelmaking, combined with acid baking – water leaching with sulfuric acid to efficiently extract 99% of the scandium in the resulting slag into solution. This extraction process employs thermal desulfation of the acidic leaching residue to regenerate the sulfuric acid used in the acid baking step and the calcium silicate flux used in the smelting step; thus, eliminating the production of solid waste and reducing the consumption of reagents. To purify and recover scandium in the leachate, solid–liquid extraction, employing reusable silicon micropillars impregnated with bis(2,4,4-trimethylpentyl) phosphinic acid is employed. The engineered surface efficiently extracts 93% of scandium with high selectivity (140× over aluminum) and a high enrichment ratio (443× higher scandium concentration in the final solution over the initial leachate) without the environmental risks and process complexity associated with conventional liquid–liquid extraction. This solid–liquid extraction technology enables high-efficiency scandium recovery with minimal reagent consumption, energy use, and process complexity. The overall process enables the extraction of critical materials from bauxite residue with minimal waste production and supports efforts to achieve full productive utilization of bauxite residue and to reduce the environmental footprint of alumina production.

Keywords: Bauxite Residue Valorization, Carbothermic Smelting, Acid Baking – Water Leaching, Solid–Liquid Extraction, Extractant Impregnated Surfaces

1. Introduction

Current worldwide bauxite residue stockpiles are estimated at approximately 3–4 billion tonnes, and continues to increase by 150–165 million tonnes annually, a rate which is accelerating, since the primary aluminum production is expected to further increase by up to 38% by 2040 [1–4]. The conventional landfill management of the Bayer process bauxite residue represents a major environmental, land-use, and opportunity cost, since this side product can be considered an abundant, readily-available, and low-cost resource for scandium and iron.

Bauxite residue's high iron content and scandium-enriched nature have made it the subject of several previous studies which have sought to utilize it as a resource for critical materials; [5–9] however, widespread adoption of these valorization processes has been impeded by the technical challenges associated with the alkaline and mineralogically complex characteristics of bauxite residue. Most previous studies seeking to extract the critical minor elements within bauxite residue using direct leaching usually employ hydrochloric, sulfuric, or nitric acid solutions and they typically suffer from large rate of reagent consumption, long residence time requirement, the production of large volumes of hazardous wastewater and solid residue, and the non-specific coextraction of other elements. [5,6,10,11]

Some studies have attempted to address some of these challenges by employing multistep extraction processes which combine the dry (no added water) mixing of concentrated acids with bauxite residue, heat treatment steps at various temperatures, and leaching in water at ambient conditions.12 Acid baking – water leaching in particular, was shown to enable high extraction efficiencies, while allowing for rapid extraction kinetics, reduced acid and water use, and relatively mild operating conditions [12]. However, one common remaining challenge is that these multistep processes typically do not enable the productive recovery of iron, which accounts for 24 wt% of the dry bauxite residue mass.

One solution that has been proposed to enable the productive recovery of iron from bauxite residue, while enabling the extraction of critical materials is the carbothermic smelting of bauxite residue.13 In this process, bauxite residue is reacted with a reducing agent, in this case carbon, at high temperature (1400–1600 °C) to reduce the iron(III) content, primarily as Fe2O3 and FeO(OH), into crude metallic iron(0) which is immiscible with the slag formed by the remaining non-reduced elements, and can be readily separated to be used in steelmaking.14 Multiple studies have explored the extraction of critical materials from the slag remaining after bauxite residue smelting by acid leaching; however, these processes face many of the same challenges as direct leaching, notably, the production of acidic leaching residue [13,15,16].

The proposed process aims to extract scandium from the smelting slag with high efficiency by employing acid baking – water leaching, and the elimination of acidic waste production by employing thermal desulfation to convert the residue back to a recycled flux for smelting, while regenerating acid for the acid baking step. Acid baking – water leaching + thermal desulfation offers high (99%) extraction efficiency with reduced water and acid consumption, and rapid leaching kinetics at ambient conditions, without the production of hazardous acidic waste, and allowing the recapture of any scandium that was not extracted in the single pass process.

The product of the acid baking – water leaching step of the process is an aqueous solution containing scandium, at a relatively low concentration (~ 3.0 mg/L), with comparatively higher concentrations of aluminum (~ 6000 mg/L), iron (~ 100 mg/L), titanium (~ 85 mg/L), magnesium (~ 50 mg/L), calcium (~ 600 mg/L), and silicon (~ 250 mg/L). To separate scandium from the mixture, liquid–liquid extraction (LLE), also known as solvent extraction, is the most commonly used method which separates compounds based on their different solubility in two immiscible liquids, usually water and an organic solvent. Although widely adopted, LLE comes with the following operational disadvantages: (1) the large volumes of organic solvents required can lead to environmental and safety issues; (2) the regeneration and reusability of extractant is poor due to solvent lost during phase separation procedure [17], which is less ecologically and economically efficient; (3) these processes are generally not efficient for the recovery of trace elements in highly concentrated solutions, such as those in the slag leachate; and (4) high intensity agitation of the solvent and aqueous phases is critical for high extraction efficiency, which requires high energy input for continuous and vigorous stirring [17–19].

In comparison, solid–liquid extraction (SLE) is a greener approach for extraction of elements, in part because it drastically reduces the consumption of the solvent and product of pollutants. The SLE category includes supported-liquid extraction and solid-phase extraction methods, both utilize the affinity of a flowing liquid containing the dissolved analytes with a solid, leading to the separation of the mixture into extracted and unextracted parts of the components, as the target components bind preferentially to the solid extractant [20].

In the supported-liquid extraction methods, the most commonly used supports consist of insoluble polymer matrix impregnated with a liquid extractant (e.g., cross-linked polystyrene); chromatographic-based resins are examples of commercialized products [20,21]. The solid-phase extraction technique is based on the adsorption of desired species onto the surface of a solid sorbent. Fabrication of the sorbent usually involves chemically anchoring an extracting agent on the supported solid, for instance, grafting functional ligands in porous carrier for extracting scandium [22–24]. However, the fabrication of such grafted sorbent usually requires multi-stage synthesis with careful control of temperature (sometimes also pressure) [24,25]. Hence, such solid-phase extraction approach can be time-costly and energy-intensive. Considering the increasing interest towards SLE processes compared with conventional LLE, various promising solutions have been developed in laboratory-scale studies, including pure mesoporous silica materials [24] modified porous materials [26], ionic liquid-functionalized solvent-impregnated resins [27], supported ionic liquid phase [28], and advanced magnetic nanosorbents [29].

In this study, a novel SLE system, named solvent-impregnated surface, was developed for selectively extracting scandium from bauxite residue slag leachate. Compared with other SLE methods [24,30], the proposed SLE system requires less complex extractant immobilization procedures which are more time-efficient and easily adaptable. Scandium recovery tests were performed using such SLE system and results showed 93% of scandium is extracted from bauxite residue slag leachate, and the selectivity of scandium is more than 140-fold higher than that of aluminum.

A block flow diagram for the overall process is shown in Figure 1. Carbothermic smelting is first used to extract and separate the crude metallic iron from the bauxite residue, while acid baking – water leaching is used on the slag to extract the scandium into solution. Once in solution, the scandium is separated and purified by solid–liquid extraction; thus, allowing the eventual recycling of the stripped water for leaching. Overall, this process enables the extraction of the valuable components of bauxite residue with minimal waste production.

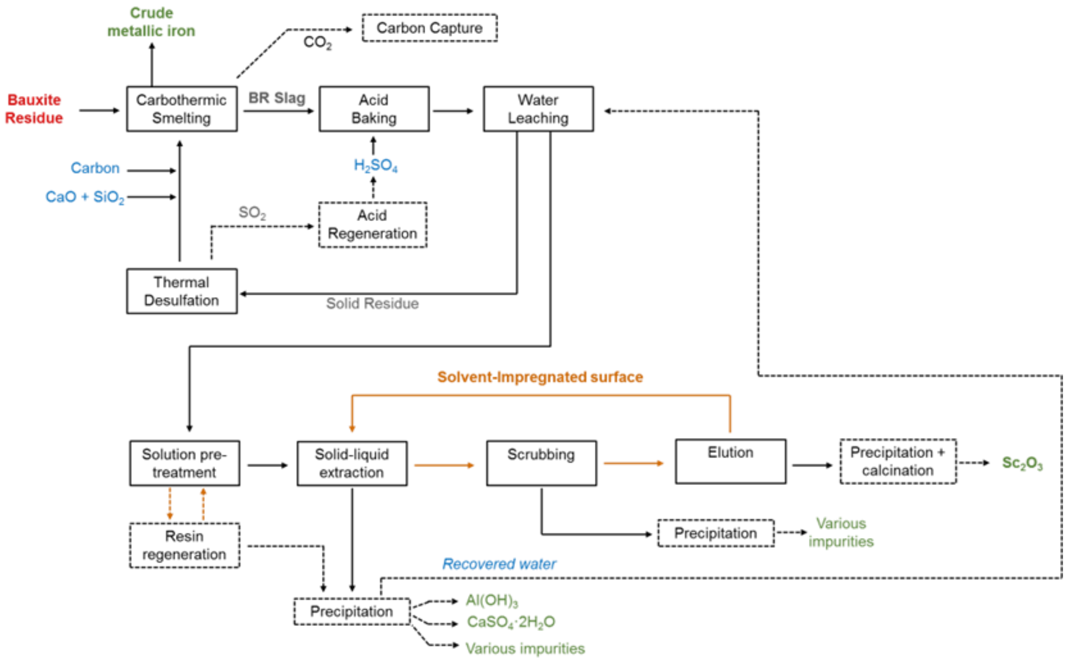

Figure 1. Block flow diagram of the proposed bauxite residue valorization process. Solid boxes and lines represent process steps that have been experimentally validated in this work and the dashed boxes/lines represent proposed steps to be validated in future work.

2. Extraction of Iron and Scandium from Bauxite Residue by Smelting – Acid Baking – Water Leaching – Thermal Desulfation

2.1 Iron Recovery from Bauxite Residue by Carbothermic Smelting

The initial step in the proposed bauxite residue valorization process is carbothermic smelting for the recovery and separation of crude metallic iron. This technique is based on the high reducibility of iron relative to all the other primary components of bauxite residue (aluminum, silicon, calcium, titanium, and others) and scandium, and the immiscibility of iron with these non-reduced oxide components, which form a recoverable slag phase.

Bauxite residue smelting was performed at the laboratory scale in a box furnace. Dried and crushed bauxite residue, alongside 0.070 g/gBR carbon (lignite coal) and a flux composed of 0.075 g/gBR CaO and 0.025 g/gBR SiO_2, was pelletized, then smelted in a graphite crucible at 1600 °C for 1 h under a 1.4 L/min flow of argon gas, to prevent oxidation of the graphite crucible. The smelting conditions, i.e., the temperature, the amount of flux added, the amount of carbon added, and the composition of the flux, was selected on the basis of previous optimization trials. Once solidified and cooled to room temperature, the resulting magnetic metallic iron nugget was physically separated from the non-magnetic slag (additional separation was achieved by crushing the slag, then performing wet magnetic separation). The slag was then characterized, as described elsewhere [31], then used as a feed for acid baking – water leaching.

The compositions of the initial bauxite residue and the smelted slag are given in Table 1. Iron, which initially comprised 24.3 wt% of the starting residue was reduced to 0.3 wt% in the slag, representing a metal recovery of 99.5%. Meanwhile, the scandium became concentrated in the slag phase alongside the non-reduced elements. One note is that the metal phase had slag adhered to its surface that was not removed during the physical separation of the solidified phases. As a result, the measured amount of slag phase recovered was lower than it would be if a full phase separation had been achieved; thus, the estimate of aluminum, titanium, calcium, silicon, and scandium recovery in the slag (73.8% for scandium) was somewhat underestimated. At the industrial scale, this separation would be done at high temperature in the liquid phase, where adhesion of slag to the metal would not be an issue; thus, higher slag recoveries of the non-reduced elements would be achieved.

Overall, these results demonstrate that the smelting step allows iron to be separated from the bauxite residue as a value-added product, while transferring scandium to a slag that can be subjected to further hydrometallurgical processing.

2.2 Scandium Extraction from Bauxite Residue Slag by Acid Baking – Water Leaching

Following the smelting of the bauxite residue, the resulting slag contained the scandium content, alongside all the non-reduced oxides, primarily forming a variety of $Ca_xAl_ySi_zO_{(x+3/2\ y+2z)}$ phases. The scandium contained within this slag was extracted by acid baking – water leaching, which provides a rapid and efficient hydrometallurgical approach to bringing the scandium into aqueous solution. In this technique, the slag is mixed with concentrated (95–98 wt%) H_2SO_4 at a ratio of 1.25 g H_2SO_4/g slag, then baked at 150 °C in a box furnace for 2 h. During the acid baking step, the calcium and aluminum in the slag become sulfated, according to Equation 1, resulting in the break-down of the slag phases; thus, allowing the sulfation of scandium (Equation 2). The acid baked slag (ABSBR) was then leached in water at a ratio of 7.5 mL H2O/g ABSBR, resulting in the dissolution of the soluble scandium sulfate into the aqueous solution. The conditions for acid baking were selected on the basis of previous optimization studies [31].

$$Ca_xAl_ySi_zO_{(x+3/2\ y+2z)(s)} + (x+2y)H2SO_{4(l)} = xCaSO_{4(s)} + y(H_3O)Al(SO_4)_{2(s)} + zSiO_{2(s)} + (x+11/2y)H_2O(g) \quad (1)$$

$$Sc_2O_{3(s)} + 2H2SO_{4(l)} = Sc_2(SO_4)_{3(s)} + 3H_2O(l) \quad (2)$$

The results of the acid baking–water leaching extraction are shown in Table 1. As can be observed, the initial extraction is rapid, with over 50% of the total extraction occurring within the first 5 min of leaching, and 99.3% extraction of scandium achieved after 3.5 h. These high extents of extraction indicate that the acid baking conditions were sufficient to fully sulfate the scandium content of the slag, while the amount of water used was sufficient to fully solubilize the sulfated scandium. These extraction results demonstrate the advantage of using acid baking–water leaching relative to direct acid leaching. Previous studies have investigated the extraction of scandium from bauxite residue slag, but when direct acid leaching was used, comparable extractions were only achieved when high concentrations of acid solution ($\geq$ 3 N HCl, H_2SO_4, or HNO3) were used, often at high liquid/solid ratios (10–50 mL H_2O/g slag) and high temperatures (70–100 °C) [13,15,16]. On the contrary, at the optimum acid baking – water leaching conditions (1.25 g H_2SO_4/g slag, 7.5 mLH_2O/g ABSBR), the direct leaching equivalent concentration was 0.76 M H_2SO_4 (i.e., 1.5 NH_2SO_4), at an equivalent L/S of 16.0 mL H_2O/g slag at ambient temperature.

Table 1. Summary of the carbothermic smelting and acid baking – water leaching trial results. Measurements were performed by alkali fusion digestion of the solid products, and inductively coupled plasma – optical emission spectroscopy (ICP-OES), as described elsewhere [31].

	Al	Fe	Ti	Ca	Si	Sc
Smelting						
Bauxite residue composition (wt%)	11.8	24.3	3.8	2.3	6.1	31.1 mg/kg
Slag composition (wt%)	22.4	0.3	4.7	14.7	11.2	54.9 mg/kg
Estimated slag recovery (%)*	79.3	0.5	51.2	80.4	64.4	73.8
Estimated metal recovery (%)*	20.7	99.5	48.8	19.6	35.6	26.2
Acid baking - water leaching						
Extraction - 5 min (%)	24.6	27.5	6.1	14.1	0.5	50.1
Extraction - 30 min (%)	36.8	47.4	7.7	13.7	1.9	72.9
Extraction - 120 min (%)	44.9	65.9	8.5	9.9	4.6	93.8
Extraction - 210 min (%)	47.3	72.6	8.7	8.9	6.4	99.3

*The slag recovery estimated was based on the mass of the slag solids recovered after physical separation. Upon separation, the metallic phase had slag adhered to the surface which was not removed when the phases were physically separated, resulting in a decrease in the estimated slag-phase recovery of Al, Ti, Ca, Si, and Sc, and an increase in the estimated metal-phase recovery of these elements.

2.3 Regeneration of Smelting Flux by Thermal Desulfation

One of the major concerns of any acid leaching process is the production of acidic residue after extraction, due to the environmental cost of disposal. The residue from the acid baked slag was primarily composed of insoluble SiO_2, $CaSO_4{\cdot}2H_2O$, and AlO(OH). In addition to the environmental challenges of managing this acidic residue, its production also represents a loss of sulfates from the H_2SO_4 added during acid baking, SiO_2, and CaO which were added during smelting, and aluminum which is a potential valuable byproduct of the extraction process, as its recovery represents an overall increase in the Bayer process efficiency.

Thermal desulfation of the leaching residue was employed as an approach to both reduce the production of solid waste and regenerate several of the reagents consumed by the extraction process. The feasibility of residue desulfation and recycling was demonstrated experimentally by treating the dried and pelletized leaching residue at 1400 °C for 3 h in a carbon crucible to enable the conversion of calcium sulfate to CaO, SO_2, and O_2, according to Equations 3 and 4, which are known to occur at above 1375 °C [32]. The CaO produced in these reactions can then be recycled as flux, substituting fresh CaO that would otherwise be added during smelting. Furthermore, SiO_2, which is also present in the residue, can also be recycled. The AlO(OH) content can also be recycled, since AlO(OH) is a component of bauxite residue, and its recycling would represent a general increase in the overall productive extraction of aluminum during the process. Finally, the SO_2 produced during desulfation can also be converted back to H_2SO_4; thus, reducing the acid demand of the process.

$$CaSO_4{\cdot}2H_2O_{(s)} = CaSO_{4(s)} + 2H_2O(g) \tag{3}$$

$$CaSO_{4(s\ or\ l)} = CaO_{(s\ or\ l)} + SO_{2(g)} + 1/2O_{2(g)} \tag{4}$$

The feasibility of operating the extraction process as a closed loop was demonstrated experimentally by conducting a smelting trial using desulfated leaching residue as a substitute for CaO and SiO_2 in the flux and conducting acid baking – water leaching on this recycled slag. The leaching residue utilized was collected from multiple previous optimization trials, several of which had low scandium extraction, which resulted in a residue that contained unextracted scandium. As shown in Table 2, the thermal desulfation step reduced of the residue sulfur content from 3.1 wt% to 0.64 wt% (84% removal), resulting in a regenerated smelting flux that contains all the scandium that was not extracted in the initial acid baking – water leaching. Furthermore, this regenerated flux was successfully used for smelting with new bauxite residue (added at ratio of 0.1 g flux/g BR), enabling iron removal (slag iron concentration of 0.7 wt%), and the concentration of scandium in the slag (71.8 mg/kg in the slag, relative to 31.1 mg/kg in untreated bauxite residue). When this recycled slag was subjected to acid baking – water leaching (1.25 g H_2SO_4/g slag, 2 h at 200 °C, 7.5 mL H_2O/g ABSBR), near-complete scandium extraction was achieved within 2 h of leaching.

Table 2. Summary of the leaching residue recycling trials.

	Al	**Fe**	**Ti**	**Ca**	**Si**	**Sc**	**S**
Desulfation + Smelting							
Leaching residue composition (wt%)	15.6	0.1	4.3	12.5	10.8	28.9 mg/kg	**3.08**
Desulfated residue composition (wt%)	20.7	0.2	5.8	16.5	14.5	43.4 mg/kg	**0.64**
Recycled slag composition (wt%)	28.4	0.7	5.8	8.9	11.3	71.8 mg/kg	**0.29**
Acid baking - water leaching							
Extraction - 5 min (%)	12.1	28.3	15.9	19.4	0.2	48.2	
Extraction - 30 min (%)	20.4	46.3	28.4	25.9	1.0	73.2	
Extraction - 120 min (%)	31.8	67.1	36.7	18.4	1.7	100	
Extraction - 210 min (%)	35.3	73.8	38.2	17.4	1.8	100	

Overall, these proof-of-concept tests demonstrate that the extraction section of the proposed process, incorporating carbothermic smelting, acid baking–water leaching, and thermal desulfation, can be operated in a closed loop to separate crude metallic iron from bauxite residue, and efficiently extract over 99% of the scandium content without producing acidic leaching residue, while regenerating some of the primary consumed reagents (CaO, SiO_2, H_2SO_4).

3. Purification of Aqueous Scandium by Solid – Liquid Extraction

3.1 Design and Fabrication of the Extractant-Impregnated Surface

As the name solvent-impregnated surface implies, this SLE system comprises a solid substrate that is specifically designed to be impregnated with a thin layer of extractant solvent on its surface. The cross-section illustration of the solvent-impregnated surface is shown in Figure 2a. First, the silicon substrate with micro-scale posts was fabricated, then a modifier layer was coated on the pillar-patterned surface to decrease the surface energy. hotolithography, a technique widely used in microfabrication to pattern parts on a substrate, was utilized to fabricate the micro-post features on a smooth silicon surface (Figure 2b), followed by solution deposition of an octadecyl trichlorosilane (OTS) modifier layer. The last step was to impregnate the micro-post silicon substrate with extractant using dip-coating. Bis(2,4,4-trimethylpentyl) phosphinic acid (HBTMPP) is a commonly used extractant in industrial LLE systems to extract scandium and was selected for this study as the impregnated extractant and was added as a 45 vol.% solution with kerosene. The dimensions of micro-posts, and the post-to-post spacing were specifically designed such that a thin layer of the extractant is trapped stably inside the low surface energy micro-posts substrate [33,34].

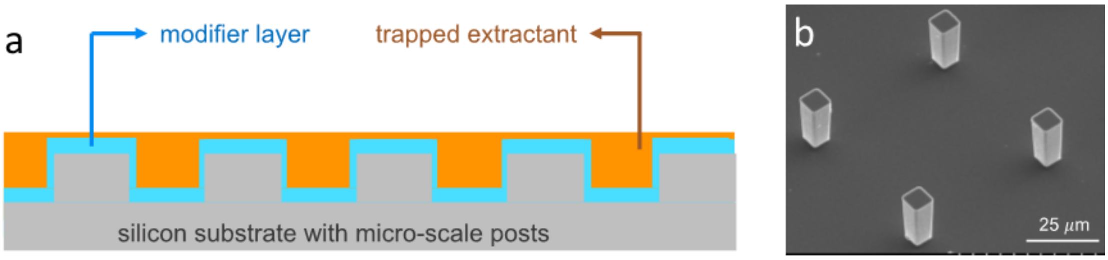

Figure 2. Sketch showing cross-section of solvent-impregnated surface (a) and SEM image of fabricated micro-scale post-featured silicon substrate (b).

3.2 Scandium Recovery Performance

To evaluate scandium recovery performance using solvent-impregnated surfaces, extraction tests were performed using the bauxite residue slag leachate produced by carbothermic smelting followed by acid baking – water leaching. The performance was measured as the scandium extraction efficiency, the selectivity of scandium over aluminum, and the scandium elution efficiency. The extraction performance is summarized in Table 3. The procedures for scandium recovery using solvent-impregnated surfaces, including the extraction, scrubbing, and elution steps are presented in Figure 3.

Table 3. The elemental composition and efficiency at different processing stages.

	Al	**Fe**	**Ti**	**Mg**	**Ca**	**Si**	**Sc**
Initial leachate composition (mg/L)	6050.6	101.6	85.0	51.4	591.7	245.4	3.0
IX resin adsorption efficiency (%)	43.6	48.0	4.7	39.3	69.0	18.7	11.9
$C_{i,0}$ **(mg/L)**	3481.0	52.8	81.0	31.2	183.7	199.5	2.6
$C_{i,f}$ **(mg/L)**	3187.0	46.8	76.9	29.2	158.9	179.4	0.2
Extraction efficiency (%)	8.6	11.4	5.0	6.4	13.5	10.1	92.8
C_{scrub} **(mg/L)**	101.9	4.5	0.3	0.9	6.5	7.9	0.2
Scrub efficiency (%)	45.5	75.0	7.4	46.6	26.2	39.2	5.7
Elution solution composition (mg/L)	2.2	0.4	0.5	0.1	0	0.6	2.0
Elution efficiency (%)	1.8	26.7	12.5	12.2	0	4.7	81.3

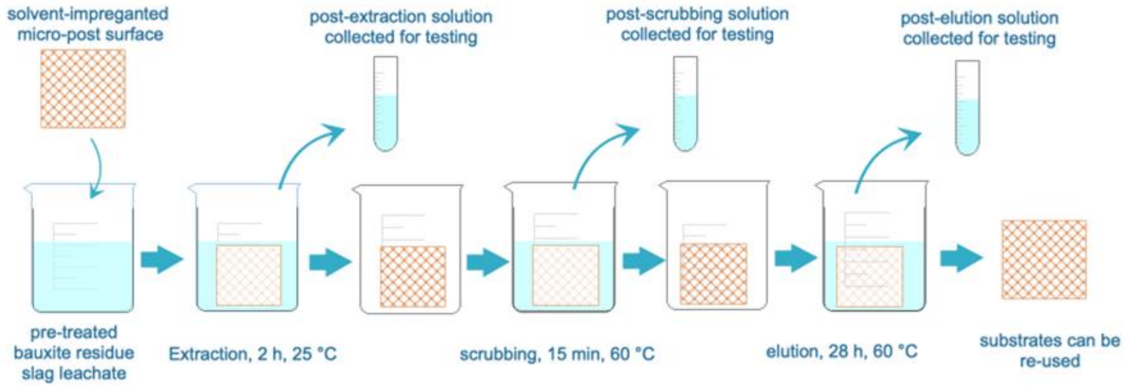

Figure 3. Scandium recovery using solvent-impregnated surfaces.

Before solid–liquid extraction, ABWL slag leachate was pretreated with cation-exchange resins (Dowex G-26, hydrogen form) followed by pH adjustment. The ion exchange (IX) resins were used to remove a large proportion of aluminum, iron, magnesium, and calcium from the solution to improve the selectivity of the solvent-impregnated surface. The pH adjustment was performed because at lower pH, H+ ions in the reaction medium compete with the scandium ions for the adsorption reaction, and at higher pH (pH>4), scandium ions tend to form hydroxide precipitation with OH- ions. Hence, the pH value of resin-treated bauxite residue slag leachate was adjusted to 3 for extraction trials which was found to be the optimal condition. Specifically, 1.5 g of resin was stirred with 15 mL of bauxite residue leachate (pH adjusted to 3) for 5 min, then the solution was collected for further pH adjustment. The resin can be regenerated by simply soaking in acid solution.

Following pre-treatment, extraction was conducted by dipping the solvent-impregnated surface into the solution. After 2 h of contact time, the ion concentrations in the post-extraction solution were characterized using ICP-OES. The scandium extraction efficiency Eextraction (%) was defined as Equation 5.

$$E_{extraction}(\%) = \frac{C_{i,0}-C_{i,f}}{C_{i,0}} \times 100 \tag{5}$$

where Ci,0 represents concentrations of ion i (mg/L) in pre-treated bauxite residue slag leachate, and Ci,f are the concentrations of ion i (mg/L) after extraction, respectively.

Scandium extraction efficiency was measured to be 92.8%, whereas extraction efficiencies of aluminum and iron were 8.6% and 11.4%, respectively. Results of ions concentrations and their extraction efficiencies are summarized in Table 1. Besides extraction efficiency, the extraction selectivity of scandium over other ions is also critical. Since the initial concentration of scandium in the leachate was considerably lower than the competing ions (aluminum ion concentration was typically over 2000× higher than that of scandium), the extraction technique applied must present a strong tendency (i.e., high selectivity) to adsorb scandium rather than other ions in the medium to achieve separation. The selectivity of different ions can be evaluated by distribution coefficient Kd (mL/cm2), given by Equation 6. The Kd values for scandium and other competitive ions are presented in Figure 4, which indicate promising selectivity for scandium over other ions.

$$K_d = \frac{V}{A} \times \frac{C_{i,0} - C_{i,f}}{C_{i,f}} \tag{6}$$

Where V and A are the volume of bauxite residue leachate (mL) and the area of the extractant-impregnated substrate (cm2), respectively.

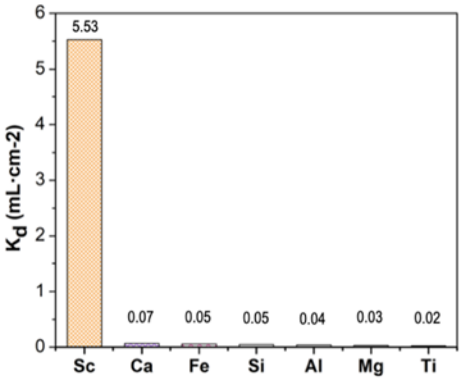

Figure 4. Distribution coefficient Kd for different elements.

The performance of elution (also referred as stripping) of scandium from micro-posts substrates was evaluated by elution efficiency Eelution (%), defined as Equation 7.

$$E_{elution}(\%) = \frac{C_{elution}}{(C_{\mathrm{i},0} - C_{i,f}) - C_{scrub}} \times 100 \tag{7}$$

where Celution and Cscrub are the ion concentrations (mg/L) in the elution and scrubbing solutions, respectively.

The results showed that scandium elution efficiency was 81.3%, which was significantly higher than that of aluminum (1.8%). Elution data for other ions are listed in Table 3. The higher elution efficiency of scandium indicates that, with such extractant-loaded micro-posts substrates, scandium is easier to be stripped out of the substrates than other co-extracted ions, which is preferred in the selective recovery of scandium from bauxite residual slag leachate. A scandium-enriched solution is expected after repeating the extraction, scrubbing and elution procedures discussed above, and scandium oxide can be recovered accordingly by adding oxalic acid.

4. Conclusions

By employing a multi-step extraction process combining carbothermic smelting, acid baking – water leaching, and solid–liquid extraction, valuable iron and scandium can be extracted and separated from bauxite residue efficiently, while producing minimal waste products. The use of carbothermic smelting allows 99% of the iron content in bauxite residue to be recovered as crude metallic iron, which can be used for steelmaking, while transferring scandium, aluminum, and other non-reduced components to an immiscible slag. The slag can then be subjected to acid baking – water leaching, which allows over 99% of the scandium in the slag to be extracted into solution at milder conditions than would be required for direct acid leaching. The acidic leaching residue can then be thermally desulfated to regenerate reagents for smelting and acid baking, recapture any scandium that was not extracted during leaching, and eliminate the production of acidic solid waste. Scandium can then be separated and purified from the aqueous solution by employing solid–liquid extraction using solvent-impregnated surfaces. The proposed solvent-impregnated micro-posts substrate design presents promising performance for selectively recovering scandium from bauxite residue slag leachate, without the typical challenges associated with liquid–liquid extraction systems. The high scandium extraction efficiency (~93%) and selectivity (140 times larger than aluminum) make this solid–liquid extraction system appealing for the recovery of dilute concentrations of scandium with the presence of more abundant competing ions. Overall, the proposed integrated bauxite residue valorization process enables the efficient extraction, separation, and recovery of iron and scandium from bauxite residue with minimal waste production and reagent consumption.

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