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Selective extraction process of scandium from nickel laterite waste through hydrometallurgy

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Scandium (Sc) extraction from iron and aluminum waste is a promising technique for the recycling and valorization of laterite nickel ore waste. Iron and aluminum waste is one source of scandium during preparation of nickel and cobalt hydroxide by wet smelting of laterite nickel ore. The content of Sc is notably higher than that of the raw materials, as the element is enriched in the iron and aluminum waste. As the main element in the waste samples, Al exhibits significant interference in the selective separation of Sc. In this work, taking the production line for laterite nickel ore wet leaching as the research object, the trend and distribution of trace Sc were systematically analyzed. A series of leaching experiments was conducted under varied conditions of leaching pH (2.6–3.5), leaching time (0.5–2.0 h), volume of evaporated concentrated solution (0.2–1.0 L), mass of waste added (200–800 g), reaction temperature (240–260 °C), reaction time (0.5–1.5 h), H_2SO_4 addition amount (50–300 mL L^{-1}) and $(NH_4)_2SO_4$ addition amount (150–350 kg m^{-3}). Experimental results revealed that the optimum leaching conditions were observed at a pH value of 3.0 and a leaching duration of 1 h. The optimum condition for evaporation and concentration was at 0.35 L, while the optimum conditions for impurity removal were at a waste mass of 300 g, a reaction temperature of 255 °C and a reaction time of 1 h. Under these optimum conditions, scandium content was approximately 14.67%. The product was identified as a complex salt of Sc through XRD, which could be converted to high-purity Sc_2O_3 after roasting.

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1. Introduction

Rare earth element scandium (Sc) was first discovered in 1879.[1] Owing to the excellent properties of scandium and its compounds, scandium is primarily used in aerospace materials, electronic information, nuclear technology and other industries.[2,3] It is widely used in aluminum-scandium alloys, solid oxide fuel cells, scandium sodium halogen lamps, ultralight shape memory magnesium-scandium alloys and fast storage applications.[4,5] However, there is no single scandium ore in the world. Since scandium is extremely dispersed in the Earth's crust and often occurs in association with other minerals, it is considered one of the rarest and most precious rare earth elements.[6] The price of its halides and metals is even higher than that of gold, which has significantly hindered the widespread application of scandium in high-end technology fields.[7]

At present, the world's annual consumption of scandium is mainly met by recovering it from low-grade associated ores and secondary resources, which generally include red mud from high-pressure leaching of laterite nickel ore, rare earth tailings and waste acid from titanium dioxide production using ilmenite.[8–10] Among them, laterite nickel ore is an important secondary mineral resource containing Sc, Y and other rare metals, which is often used as a raw material for Sc recovery.[11] Hydrometallurgical processes such as acid leaching, solvent extraction and ion exchange adsorption are the most widely used methods for this purpose.[12] Onghena *et al.*[13] found that scandium can be recovered from Greek bauxite residue using a combination of sulfation-roasting-leaching and solvent extraction with the hydrophobic ionic liquid betainium bis(trifluoromethylsulfonyl)imide, [Hbet][Tf2N]. The preferred leaching technique is to dissolve scandium from bauxite residue due to its low acid consumption, good selectivity towards scandium and low co-dissolution of major elements. Thus, the technology of Sc extraction from red mud has gradually improved, but there are still shortcomings in improving product purity. Ariane *et al.*[14] studied extraction of Sc from nickel processing residue using Cyanex 923, and Cyanex 923 was shown to be more selective for Sc than Fe. In addition to red mud, rare earth tailings are also often used as scandium extraction materials. In the rare earth deposit of Bayan Obo in the Inner Mongolia province of China, the grade of Sc in the treated tailings is about 0.04%, which has a high recovery value.[15] Zhang *et al.*[16] found that the Sc_2O_3 and REO of Bayan Obo tailings were enriched in hydrochloric acid under the optimal roasting and leaching conditions, and the recovery efficiency of Sc_2O_3 and REO are

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97.60% and 97.18%, respectively. L. A. Pasechnik *et al.*[17] found that H_2SO_4 and $(NH_4)_2SO_4$ were shown to be favorable for precipitation of $NH_4Sc(SO_4)_2$, providing an effective separation of scandium from impurities of Al, Fe, Zr, Ti and Ca. Sulfuric acid and ammonium sulfate, which are the cheapest products of the chemical industry, are applied in the scandium crystallization technology. The acid leaching process with low energy consumption and carbon emissions can simultaneously dissolve aluminum, iron, sodium, titanium and rare earth elements in red mud, but how to efficiently separate various metal ions from the leaching solution is still a huge challenge.[18] The production of Sc from primary ores is a well-known process and is described in detail in previous studies, and the process of recovering Sc from low-grade associated ore and secondary resources can satisfy the production. Indeed, the current technologies have shown that the hydrometallurgical processing may be the main route for scandium extraction focusing on sustainability approach.[19]

However, the recovery of Sc from end-of-life products and metallurgical slag should be investigated in detail to gain knowledge of Sc recovery. In recent years, the rapid rise of new energy vehicles has made the demand for metal Ni rapidly increase.[20,21] Indonesia is one of the countries with the richest laterite nickel deposits in the world. Laterite nickel ore is an important mineral resource for extracting metal nickel. Compared to the preparation of nickel–iron by fire, the preparation of nickel–cobalt hydroxide (MHP) by wet smelting laterite nickel ore has the significant advantages of low energy consumption, low cost and high recovery rate.[22,23] It has promoted the rapid development of the new energy field. Iron and aluminum waste is the by-product produced in the alumina refinery from laterite nickel ore. In general, the associated Sc content in the brown iron laterite nickel ore is about 0.005%.[24] It is difficult to be used as a scandium extraction material, and the economic benefit is low. Because of the similar precipitation properties of Sc and Al, in the production line of laterite nickel ore wet smelting, Sc will be significantly enriched in the waste phase of the precipitation removal of iron and aluminum. It leads to the recovery of Sc in iron and aluminum becoming possible. Zela T. *et al.*[25] demonstrated that the simple, atmospheric leaching process with an organic acid is feasible to extract scandium from nickel laterite residues. Later, Wataru and Masahiro[26] found that PC-88A and TOPO were selected as the primary parts of the extractants for the ternary systems in the laterite nickel ore leaching solution, and the high-purity scandium oxide was prepared by multistage counter current extraction and purification. Thus, it is possible to process complex by-production with this method. In our study, the extraction of rare earth elements from iron and aluminum waste, which is the by-product produced in the alumina refinery from laterite nickel ore is focused, and the hydrometallurgy process route *i.e.*, dissolution, extraction, precipitation and roasting has been investigated.

This work will would provide a reference for the extraction of rare earth elements from iron and aluminum waste. The influence factors, such as the pH value, leaching time, waste mass and reaction temperature on scandium extraction were studied, and the mineralogy of the samples during the roasting process was investigated. The results could provide a reference for other companies with the same processing technology.

2. Experimental

2.1 Materials

2.1.1 Sample preparation and characterization. The iron and aluminum waste used in this study was received from a refinery in the Qingmeibang New Energy Co., Ltd of Indonesia. Based on the annual treatment of 3 million tons of dry ore, the accumulated content of Sc metal in raw ore was 150 tons. Combined with the production test status, the changes of Sc on the laterite-nickel ore are displayed in Fig. 1.

Based on the results of the trend and distribution of trace Sc in the wet-leaching production lines, the iron and aluminum waste were used as the raw in this study. The waste sample is grayish-brown in color, and the main components of iron and aluminum waste are shown in Table 1. The main components of the iron and aluminum waste sample are Ni, Fe, Al and Ca contents of 0.8%, 1.59%, 10.88% and 11.41%, respectively. Especially, the Sc content was 0.36%. The X-ray diffraction (XRD) analysis is shown in Fig. 2, which identified calcium carbonate $CaCO_3$ as the most dominant mineral. After sulfuric acid leaching, lime should be added to adjust the pH of the precipitate to remove impurities in the leaching solution of iron and aluminum. Because of the slow dissolution of lime, there will be a lot of calcium carbonate that does not react. In addition, the cost of calcium carbonate is low in industrial production, and mainly for cost and operability reasons. Lime is used to regulate pH. Thus, the main content of iron and aluminum waste samples was carbonate.

2.1.2 Experimental procedure. The detailed process flowsheet for scandium extraction from the iron and aluminum waste is showed Fig. 3. The operational parameters for the test are as follows, which consist of four stages. In this work, the leaching parameters studied are pH (2.5–3.5) and the leaching time (0.5–2.0 h). Immediately after, the leaching solution was evaporated and concentrated, and the volume was varied from 1.0 to 0.2. Subsequently, the solution was heated to remove impurities in the high pressure. In addition, the iron and aluminum waste was added to the solution to promote the forward progress of hydrolysis, and the amount of waste was changed from 200 g to 800 g. In the precipitation process, the precipitation recovery of elements was studied by complex precipitation, using two acid reagents (H_2SO_4 and $(NH_4)_2SO_4$).

For each test, 5 kg of iron and aluminum waste and 5 kg of tap water ($m_{\text{waste}} : m_{\text{water}} = 1:1$) were added subsequently mixing adequately, and the H_2SO_4 solution was added until the pH was 2–3 at the normal temperature. After leaching, the mixed reactants were separated by filtration. The filter residue was used in the pre-neutralization stage of the production line, and the filter liquor was used in evaporation and concentration. In the evaporation and concentration step, the filter liquor was poured into the high-pressure reactor and the excess liquid was evaporated by heating. To increase the concentration of Sc in the solution, the volume of the solution is usually evaporated 4–

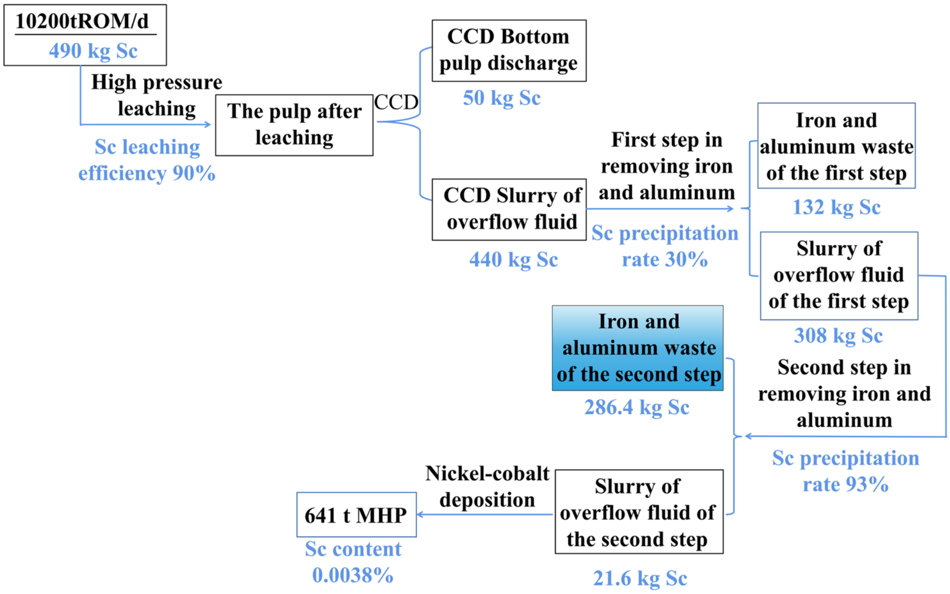

Fig. 1 Changes in scandium (Sc) content in laterite nickel ore .

Table 1 Main components of iron and aluminum waste in the second stage

Element	Ni	Co	Mn	Fe	Al	Ca	Mg	Sc
Content (%)	0.8	0.03	0.01	1.59	10.88	11.41	0.09	0.36

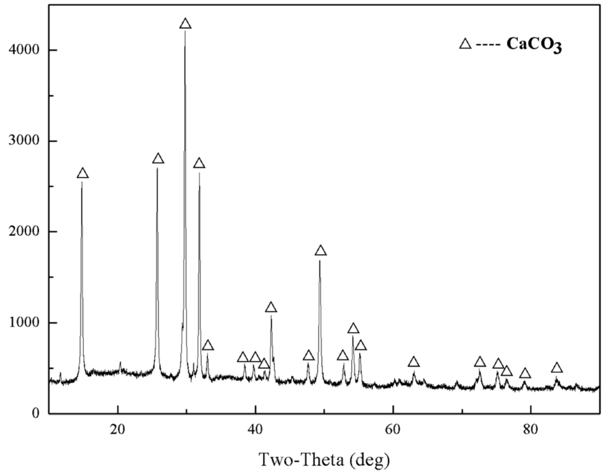

Fig. 2 XRD diffractogram of the iron and aluminum waste sample.

8 times to stop heating. In order to further improve the purity of Sc, the iron and aluminum waste of the second step (containing 40–50% sulfuric acid), was added to the leaching solution after evaporation and concentration. Different mass of iron and aluminum waste liquid were added to the filtrate of 800 mL. When the reactants are fully mixed and poured into the high-pressure reactor at the temperature range of 250–260 °C and the pressure range of 4.1–4.5 MPa for 60 min. After the reaction was completed, the reactants were filtered. The effect of the amount of H_2SO_4 and $(NH_4)_2SO_4$ was investigated to complex precipitation, and it was added into the filter liquor for 1 h at the temperature range of 70–80 °C. In the H_2SO_4 case, the amount of $(NH_4)_2SO_4$ added was fixed at 250 kg m^{-3}. In the $(NH_4)_2SO_4$ case, the amount of H_2SO_4 added was fixed at 250 mL L^{-1}. The extraction and filtration of precipitated solids were dried and analyzed. Finally, the precipitation was roasted at 800 °C for purification. In the total process, ICP-OES was used to measure the elements in the leachate. The same leachate was measured three times in parallel, and the experimental error was within ±4%.

3. Results and discussion

3.1 Leaching at normal temperature

3.1.1 Effect of the pH on the leaching efficiency. In this test, the samples were fully leached at normal temperature (26 °C) for 2 h, the stirring speed was controlled at 200 rpm, and the solid mass, solid concentration and H_2SO_4 dosage were variable. The pH value of the endpoint of the reaction was adjusted by changing the amount of sulfuric acid. Table 2 presents the date of extracted Sc and co-extracted Ni, Co, Mn and Al from the leaching experiment. The results of the experiments showed that the leaching efficiency of Sc and Ni, Co, Mn, and Al increased with the decrease of pH in the range of 2.60–3.47 at the endpoint of the reaction. The extracted Sc was in the range of 93.00–97.10%. The sulfuric acid concentration determines the leaching agent's availability to react with the minerals and the stability of the dissolved metal ions. Sulfuric acid was added to dissolve it to ensure that the reaction proceeded completely and to maintain its solubility. When the leaching endpoint pH was about 3.00, the leaching efficiency of Sc was 96.90%. The leaching efficiency of Sc was slightly increased by further decreasing the leaching pH. In this study, the leaching time was selected to be 2.0 h to save energy.

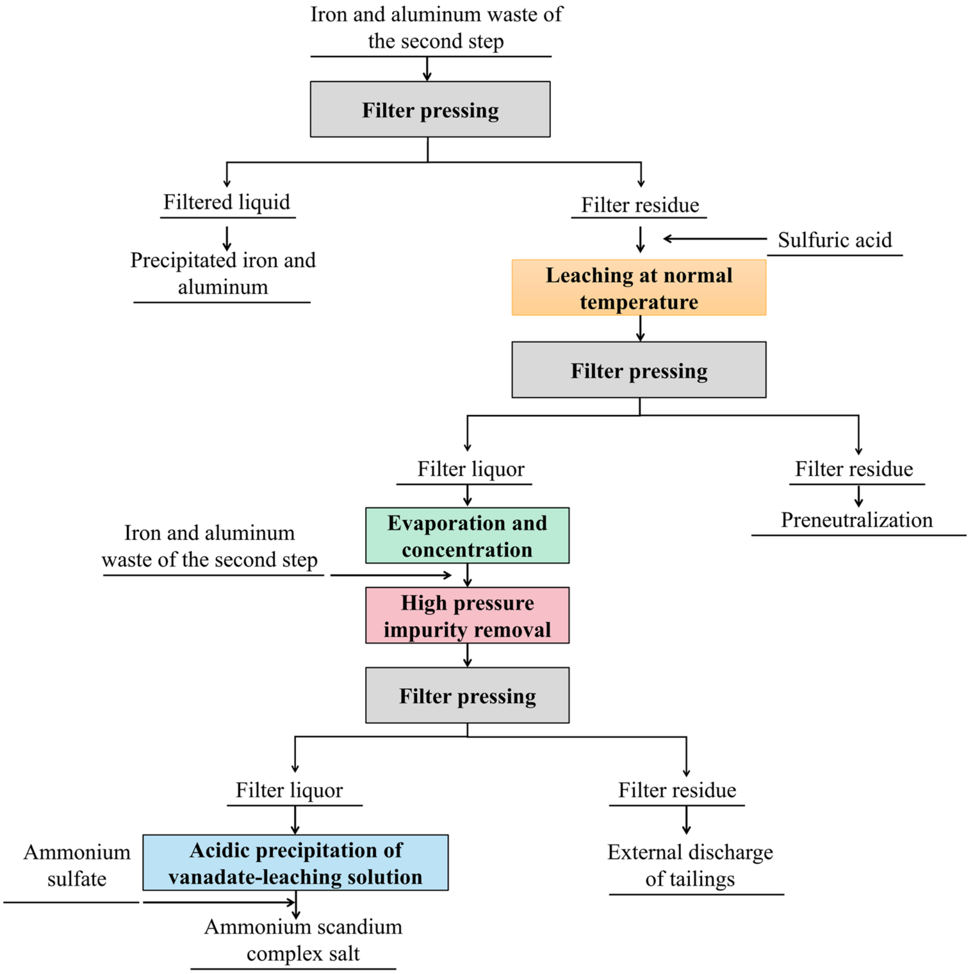

Fig. 3 Procedure diagram of the scandium extraction process .

3.1.2 Effect of leaching time on the leaching efficiency. Furthermore, the leaching time also affected the extractions. The effect of leaching time on the extraction of Sc was investigated at 0.5, 1.0, 1.5 and 2.0 h, the other experimental parameters were set as follows: the sample mass of 5000 g, the reaction pH was approximately 3.0 and the leaching temperature of 26 ° C. The results are shown in Table 3, and the extraction efficiency of Sc was above 95% when the leaching time was varied from 1.0 to 2.0 h. With the longer leaching time, the extraction efficiency of scandium slowly increased. The co-extracted Ni, Co, Mn and Al were also slowly increased with the leaching time range of 1.0 to 2.0. Thus, the appropriate leaching time was selected as 1.0 h.

3.2 Evaporation and concentration

Under suitable atmospheric pressure leaching conditions, the extracted Sc, Al, Mn, Co and Ni were almost completely dissolved. Since the content of Sc in the raw sample was much lower than that of co-extracted metals, the concentration of Sc in the leaching solution was low, as well as it was difficult to directly remove impurities. The leaching reaction is the liquid–solid reaction, and the volume of the leaching solution could significantly affect the extraction efficiency. Thus, the concentration of Sc in the leaching solution was increased by evaporation and concentration.

Table 2 The influence of experimental conditions on pH and their effect on leaching efficiency

				Leaching efficiency (%)				
Mass (g)	Solid concentration (%)	H_2SO_4 dosage (g)	pH	Ni	Co	Mn	Al	Sc
7240	35	271.3	2.60	88.32	93.21	97.46	95.58	97.10
7240	35	285.2	3.00	86.51	91.01	97.39	95.07	96.90
5000	41.5	262.6	3.26	62.51	78.68	91.63	85.65	94.43
5000	45.4	284.7	3.47	61.06	78.34	93.79	90.49	93.00

Table 3 The influence of leaching time on leaching efficiency

pH	Time (h)	Leaching efficiency (%) Ni	Co	Mn	Al	Sc
2.94	0.5	79.24	85.33	90.67	90.36	89.64
2.90	1.0	85.16	90.67	95.34	93.15	95.67
3.04	1.5	85.67	89.65	95.94	93.44	95.74
3.07	2.0	86.47	90.69	97.24	93.58	97.06

In the test, the filter liquor was poured into the high-pressure reactor at the temperature of 120 °C, then the exhaust valve was opened for evaporation of the water vapor. The experimental results are displayed in Table 4. The element content of Sc was 0.73 g L^{-1}, but the main impurity Al was 22.12 g L^{-1} in the solution after atmospheric pressure leaching when the solution was not evaporated and concentrated. As the volume decreased, the concentrations of Sc and Al both increased. When the evaporation volume was increased to 80%, the element contents of Sc and Al were 3.65 g L^{-1} and 110.60 g L^{-1}, respectively. The concentration of Al was increased by about five times. The purpose of evaporation was to increase the concentration of Sc and facilitate the removal of impurity aluminum in the solution. Thus, considering the subsequent impurity removal and the energy consumption of evaporation and concentration, the concentration of the leaching solution could be 3–4 times.

3.3 High-pressure impurity removal

3.3.1 Effect of waste mass on the impurity removal. In order to remove impurities such as Fe and Al, the filter liquor necessitates high-pressure decontamination. In the test, an 800 mL concentrated liquid sample was added to varied amounts of leaching slag, the autoclave temperature was fixed at 255 °C and the leaching reaction time was set as 1 h. The influence of the amount of leaching waste on the removal rate of impurities was investigated. Under the conditions of a high temperature of 255 °C and a pressure of 4.4 MPa, Al^{3+} and Fe^{3+} are hydrolyzed and precipitated, and H^{+} is subsequently released.[27,28] The release of H^{+} can be consumed by adding the iron and aluminum waste of the second step, thus promoting the forward progress of hydrolysis. Table 5 shows the effect of waste mass on the removal efficiency of elements, and as the amount of waste added increased, the impurity removal rates of Fe and Al decreased gradually. In addition, Sc and Ni loss rates were also increasing. Especially, when the additional amount of waste was increased from 300 g to 400 g, the loss rates of Sc increased from 11.64% to 17.68%. In addition, when the additional amount of waste was 300 g, the liquid–solid ratio was 0.375.

Table 5 The effect of waste mass on the removal efficiency of elements

Waste mass (g)	Removal efficiency of elements (%) Ni	Co	Mn	Fe	Al	Ca	Mg	Sc
200	0.96	0.00	13.83	91.36	91.21	0.22	0.00	13.76
300	1.53	0.44	16.51	90.27	81.23	1.76	0.35	11.64
400	0.94	1.37	13.55	87.65	81.46	0.75	0.00	17.68
600	2.70	0.59	15.36	85.67	82.67	0.95	0.24	18.07
800	3.60	0.00	14.00	77.57	88.23	1.06	2.68	18.53

3.3.2 Effect of reaction temperature on the impurity removal. In the high-pressure reactor at a pressure of 4.4 MPa, the leaching time was fixed at 1 h, and the reaction temperature could significantly affect the reaction pressure, then the removal efficiency of Al^{3+} was further affected in the liquid phase. The effect of the reaction temperature on the removal efficiency of elements is shown in Table 6. With the increase in the reaction temperature, the removal rate of Al^{3+} in the leaching solution gradually increased. When the reaction temperature was 255 °C, the removal efficiency of Al^{3+} was 89.20%, and it was close to 90% by this time. At this temperature, the removal efficiency of Sc was 19.43%. Based on the effect of impurity removal and energy consumption, the optimum temperature of the reaction is 255 °C.

3.3.3 Effect of reaction time on the impurity removal. In the high-pressure reactor, reaction time was shown to have the most decisive influence on the extraction percentage of scandium and co-extraction percentage of aluminum. Table 7 displays the effect of reaction temperature on the removal efficiency of elements. The extracted scandium increased from 16.39% to 19.43% when the reaction time was increased from 0.8 h to 1 h, which may be associated with the entrapment or adsorption of the scandium ion onto the iron precipitates. In addition, the removal efficiency of Al^{3+} increased from 89.20% to 90.31% at this reaction time range. Therefore, the reaction time of high-pressure leaching was controlled for 1 h.

Table 4 Concentration of valuable metals in the evaporated concentrated solution

Percentage of evaporation volume (%)	Element content (g L^{-1}) Ni	Co	Mn	Fe	Al	Ca	Mg	Sc
0	0.64	0.03	0.03	0.32	22.12	0.58	0.17	0.73
57	1.12	0.05	0.05	0.56	38.81	1.02	0.30	1.28
59	1.56	0.07	0.07	0.78	53.95	1.41	0.41	1.78
65	1.83	0.09	0.09	0.91	63.20	1.66	0.49	2.09
76	2.67	0.13	0.13	1.33	92.17	2.42	0.71	3.04
80	3.20	0.15	0.15	1.60	110.60	2.90	0.85	3.65

Table 6 The effect of reaction temperature on the removal efficiency of elements

Reaction temperature (°C)	Removal efficiency of elements (%)							
	Ni	Co	Mn	Fe	Al	Ca	Mg	Sc
240	1.05	0.19	16.51	68.43	79.43	0.00	0.35	11.64
245	1.23	0.44	16.53	74.02	81.23	1.17	0.20	19.71
250	1.44	0.58	16.76	80.50	87.29	1.06	0.51	18.50
255	1.50	1.18	17.65	84.59	89.20	0.18	0.39	19.43
260	1.53	1.19	17.91	90.27	91.47	0.87	0.56	19.33

Table 7 The effect of reaction time on the removal efficiency of elements

Reaction time (h)	Removal efficiency of elements/%							
	Ni	Co	Mn	Fe	Al	Ca	Mg	Sc
0.5	0.97	0.00	12.87	58.37	68.37	0.00	0.27	10.57
0.8	0.85	0.38	15.39	79.32	80.69	0.14	0.33	16.39
1	1.50	1.18	17.65	84.59	89.20	0.18	0.39	19.43
1.5	0.98	0.87	17.68	86.38	90.31	0.19	0.48	19.25

3.4 Acidic precipitation of leaching solution

In this study, the precipitation recovery of elements was studied using H_2SO_4 and $(NH_4)_2SO_4$ reagents. In the H_2SO_4 case, the experiments were carried out under the single variations in H_2SO_4 concentration (50–300 mL L^{-1}), and $(NH_4)_2SO_4$ (250 kg m^{-3}), temperature (80 °C) and duration (1 h). The results are shown in Fig. 4. The elements such as Al, Ni, Co and Mn were hardly precipitated with the addition of H_2SO_4. With the increase of the additional amount of H_2SO_4, the precipitation recovery of Sc showed an upward trend. When the additional amount of H_2SO_4 was 250 mL L^{-1}, the precipitation recovery of Sc was above 95%. The increase in sulfate concentration is conducive to Sc^{3+} combining with Sc^{3+} to form scandium ammonium sulfate precipitation. In the $(NH_4)_2SO_4$ case, the experiments were carried out under the single variations in $(NH_4)_2SO_4$ concentration (150–350 kg m^{-3}), and H_2SO_4 (250 mL L^{-1}), temperature (80 °C) and duration (1 h). When $(NH_4)_2SO_4$ was added in this range, the precipitation recovery of Sc was approximately 90%, and the elements such as Al, Ni, Co and Mn were hardly precipitated. Thus, the additional amount of $(NH_4)_2SO_4$ was 250 kg m^{-3}, the precipitation recovery of Sc was approximately 95%, as well as other elements, were hardly precipitated.

XRD was used to analyze the residue sample after high-pressure leaching, and the results are shown in Fig. 5. It showed that the main components of the residue sample after high-pressure leaching are the hydrolyzed salt of aluminum ($Al_3(SO_4)_2$) and the original $CaSO_4$.

The content of valuable metals in the complex salt of scandium ammonium sulfate was analyzed to further reveal the extracted efficiency, and the analysis results are shown in Table 8. The content of Sc in the scandium ammonium sulfate is about 14.67%. This is because ammonium sulfate is a weak acid and it may preferentially react with Mn, Fe and Mg, and then with Ni, Co, Al and Ca during the roasting process.

In the high-pressure impurity removal step, the oxides and/or hydroxides of iron and aluminum are dissolved by the reaction with acid, and are hydrolyzed to oxides, releasing sulfuric acid. The dissolved oxides and/or hydroxides of iron and aluminum are as follows.[29,30]

$$Fe_2O_3 + 3H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 3H_2O$$

$$2\alpha - FeOOH + 3H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 4H_2O$$

$$Al_2O_3 + 3H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 3H_2O$$

$$2Al(OH)_3 + 3H_2SO_4 \rightarrow Al_2(SO_4)_3 + 6H_2O$$

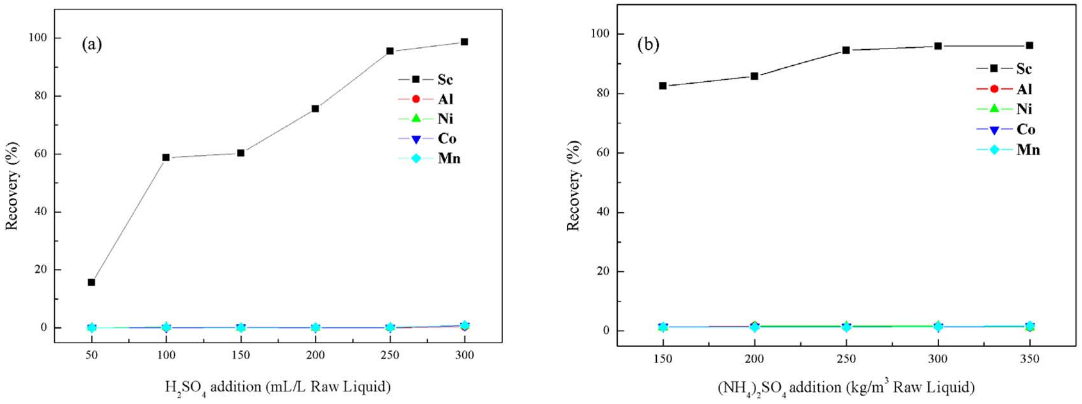

Fig. 4 The effect of addition amounts of H_2SO_4 (a) and $(NH_4)_2SO_4$ (b) on the recovery of elements.

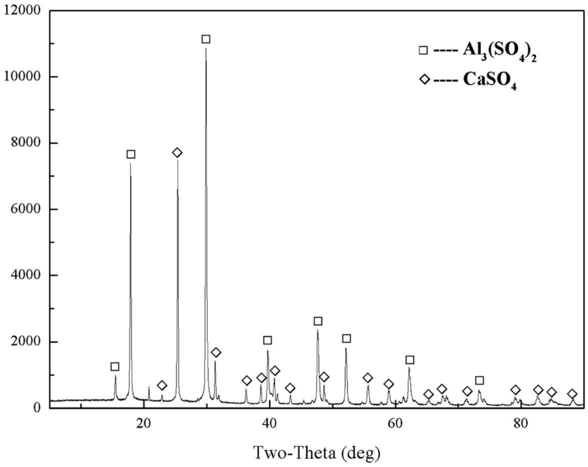

Fig. 5 XRD results of the residue sample after high-pressure leaching.

Table 8 The content of valuable metals in the complex salt of scandium ammonium sulfate (w.t.%)

Valuable metals	Ni	Co	Mn	Fe	Al	Ca	Mg	SiO_2	Sc
Content (w.t.%)	0.15	0.02	0.84	0.22	0.06	0.15	0.23	0.58	14.67

Scandium hydroxide is dissolved in sulfuric acid, and iron and aluminum slag comes from the precipitation produced by the pH adjustment of lime, and is precipitated by hydroxide. Thus, the scandium hydroxide is dissolved in sulfuric acid.

$$2Sc(OH)_3 + 3H_2SO_4 \rightarrow Sc_2(SO_4)_3 + 6H_2O$$

In addition, the dissolved iron tends to be hydrolyzed at a high temperature in the acidic solution to form hematite or hydronium jarosite according to the following reactions.[31]

$$Fe_2(SO_4)_{3(aq)} + 3H_2O_{(l)} \rightarrow Fe_2O_{3(s)}\downarrow + 3H_2SO_{4(l)}$$

$$3Fe_2(SO_4)_{3(aq)} + 14H_2O_{(l)} \rightarrow 2[H_3OFe_3SO_{42}(OH)_6]_{(s)} + 5H_2SO_{4(aq)}$$

Meanwhile, aluminum dissolved and then precipitated through the following hydrolysis reaction.[32,33] Thus, pressure is important to the hydrolysis.

$$3Al_2(SO_4)_{3(aq)} + 14H_2O_{(l)} \rightarrow 2[H_3OAl_3SO_{42}(OH)_6]_{(s)} + 5H_2SO_{4(aq)}$$

When $(NH_4)_2SO_4$ was added, it was assumed that the following equilibrium are reached in the saturated solutions between the solid substance and the ions forming this substance to determine the solubility.[34]

$$Sc^{3+} + NH_4^+ + 2SO_4^{2-} \leftrightarrow NH_4Sc(SO_4)_2$$

$$Sc^{3+} + 3NH_4^+ + 3SO_4^{2-} \leftrightarrow (NH_4)_3Sc(SO_4)_3$$

The above-mentioned results indicated that in the process of solution, extraction, precipitation and roasting are optional, which could extract the scandium from iron and aluminum waste and it was received from an alumina refinery including co-extracted elements Ni, Co, Mn and Al.

When ammonium sulfate was added to the residue as an additive, the possible reactions during the roasting process were as follows.[35–37] In the complex precipitation process, the individual compounds $NH_4Sc(SO_4)_2$ and $(NH_4)_3Sc(SO_4)_3$ were preliminarily synthesized and certified as described earlier.[34] After the roasting, the thermal decomposition of $NH_4Sc(SO_4)_2$ through $Sc_2(SO_4)_3$ into Sc_2O_3.[38]

$$4NH_4Sc(SO_4)_2 \rightarrow 2Sc_2(SO_4)_3 + 4NH_3\uparrow + 2SO_2\uparrow + O_2\uparrow + 2H_2O\uparrow$$

$$2Sc_2(SO_4)_3 \rightarrow 2Sc_2O_3 + 6SO_2\uparrow + 3O_2\uparrow$$

Fig. 6 presents the XRD patterns of the scandium ammonium sulfate and roasted residues. The main mineral phase in the scandium ammonium sulfate is $Sc_2(SO_4)_3$, whereas the scandium sulfate remained stable, and the results are shown in Fig. 6(a). After the roasting process, the Sc_2O_3 is detected and it is depicted in Fig. 6(b). Thus, the process system is characterized by the production of value-added Sc_2O_3.

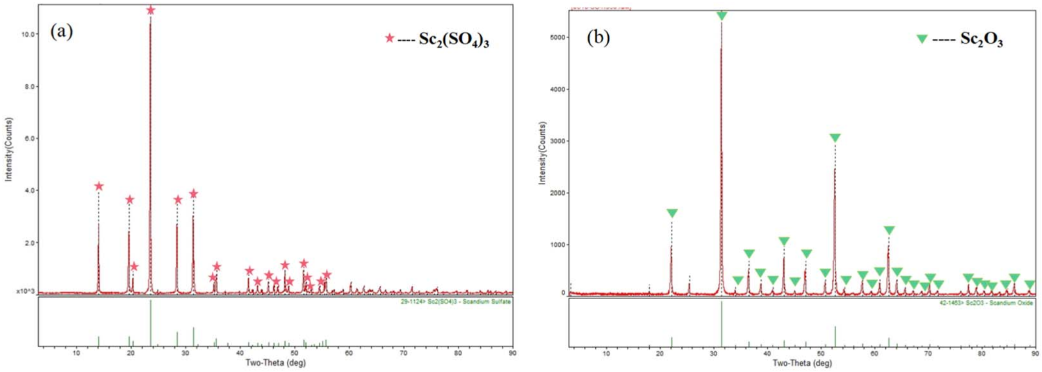

Fig. 6 XRD results of the complex salt of scandium ammonium sulfate (a) and the composite salt precipitate after roasting at 800 °C (b).

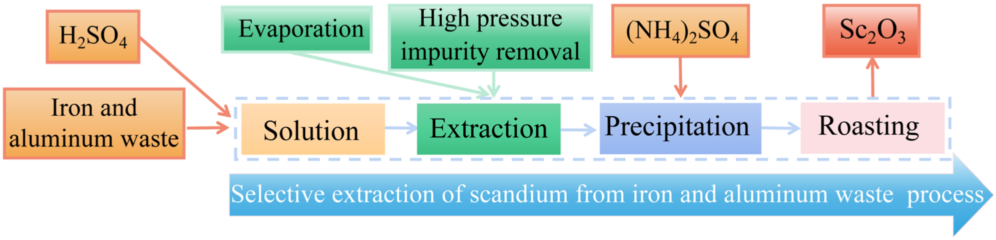

Fig. 7 Schematic of the extraction process.

As a result of the above analysis, a schematic diagram of the process and mechanism for the extraction of Sc from iron and aluminum waste by the solution–extraction–precipitation–roasting was designed, as shown in Fig. 7. The waste can be basically dissolved at normal temperature with the reaction duration of 1 h at the end of 3.0 pH. After reaction at 255 °C and liquid–solid ratio of 0.375 for reaction time 1 h, the removal rate of Al in the leaching solution exceeded 90% and the loss rate of Sc was between 15–20%. After evaporation and concentration, the leaching solution was precipitated using 250 mL L^{-1} of sulfuric acid, 250 kg m^{-3} of ammonium sulfate, and 80 °C in a water bath for 1 h, and could be worthy of a complex salt of Sc and ammonium sulfate with a higher content of Sc of 14.67% and a lower content of other valuable metals. It can be used as the crude product of Sc and can be further prepared with high-purity Sc_2O_3 through roasting.

4. Conclusions

The high-pressure leaching wet smelting process of laterite nickel ore has an enrichment effect on dispersed metal Sc in the second iron and aluminum removal section, and the iron and aluminum waste can be used as raw material to extract and recover Sc. The leaching experiments showed that the elements such as Ni, Co, Mn, Fe and Al in the iron and aluminum waste can be basically dissolved at normal temperature with the reaction duration of 1 h at the end of 3.0 pH. The leaching solution was concentrated and then added to autoclaves to remove impurities after normal pressure leaching. The removal rate of Al in the leaching solution exceeded 90% and the loss rate of Sc was between 15–20% with the reaction at 255 °C and the liquid–solid ratio of 0.375 for a reaction time of 1 h. After the impurity removal, the leaching solution was precipitated and could be worthy of a complex salt of Sc and ammonium sulfate with a higher content of Sc of 14.67% and a lower content of other valuable metals. It can be used as the crude product of Sc and can be further prepared with high-purity Sc_2O_3 through the process of dissolution, extraction, precipitation and roasting. This work will provide a reference for the extraction of rare earth elements from iron and aluminum waste, which is the by-product produced in the alumina refinery from laterite nickel ore. It is a promising technique for the recycling and valorization of Sc from iron and aluminum waste.

Data availability

The authors confirm that the data supporting the findings of this study are available within the article.

Author contributions

Yumeng Chen: investigation, data curation, writing-original draft, funding acquisition; Huiming Wu: data curation, funding acquisition; Biaogang Xu: funding acquisition; Wenjie Zhang: investigation, data curation, writing-review & editing.

Conflicts of interest

There are no known competing financial interests or personal relationships that could influence the publishing of this manuscript.

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References

1 D. Zou, Y. Deng, J. Chen and D. Li, A review on solvent extraction of scandium, *J. Rare Earths*, 2022, **40**(10), 1499–1508.

2 W. W. Wang and C. Y. Cheng, Separation and purification of scandium by solvent extraction and related technologies: a review, *J. Chem. Technol. Biotechnol.*, 2011, **86**(10), 1237–1246.

3 X. Su, H. P. Liu and G. Tian, A green extraction process for the selective recovery of Sc(III) based on hydrophobic betaine derivative ionic liquids, *RSC Adv.*, 2024, **14**(7), 4853–4860.

4 G. Li, Q. Ye, B. Deng, J. Luo, M. Rao, Z. Peng and T. Jiang, Extraction of scandium from scandium-rich material derived from bauxite ore residues, *Hydrometallurgy*, 2018, **176**, 62–68.

5 A. Belosludtsev, Y. Yakimov, R. Mroczynski, S. Stanionyte, M. Skapas, D. Buinovskis and N. Kyzas, Effect of Annealing on Optical, Mechanical, Electrical Properties and Structure of Scandium Oxide Films, *Phys. Status Solidi*, 2019, **216**(18), 1900122.1–1900122.7.

6 A. Rollat, D. Guyonnet, M. Planchon and J. Tuduri, Prospective analysis of the flows of certain rare earths in

Europe at the 2020 horizon, *Waste Manage.*, 2016, **49**, 427–436.

7 F. Meng, X. Li, L. Shi, Y. Li and Y. Wei, Selective extraction of scandium from bauxite residue using ammonium sulfate roasting and leaching process, *Miner. Eng.*, 2020, **157**, 106561.

8 F. Li, X. Li, L. Shi, X. Li, D. Liang, Y. Wei and T. Fujita, Solvent extraction of scandium from leaching solution of red mud roasted with ammonium sulfate using D2EHPA/TBP☆, *J. Rare Earths*, 2024, **42**(10), 1943–1949.

9 H. Habibi, M. Mokmeli, S. Shakibania, D. Pirouzan and Z. Pourkarimi, Separation and recovery of titanium and scandium from the red mud, *Sep. Purif. Technol.*, 2023, **317**, 123882.

10 X. Dai, N. Thi Hong Nhung, M. F. Hamza, Y. Guo, L. Chen, C. He and T. Fujita, Selective adsorption and recovery of scandium from red mud leachate by using phosphoric acid pre-treated pitaya peel biochar, *Sep. Purif. Technol.*, 2022, **292**, 121043.

11 D. Wei, X. Jun-Hui, P. Yang, S. Si-Yue, C. Tao, Z. Kai and W. Zhen, Extraction of Scandium and Iron from Red Mud, *Miner. Process. Extr. Metall. Rev.*, 2022, **43**(1), 61–68.

12 N. Zhang, H. X. Li and X. M. Liu, Recovery of scandium from bauxite residue-red mud: a review, *Rare Met.*, 2016, **35**(12), 887–900.

13 B. Onghena, C. R. Borra, T. Van Gerven and K. Binnemans, Recovery of scandium from sulfation-roasted leachates of bauxite residue by solvent extraction with the ionic liquid betainium bis(trifluoromethylsulfonyl)imide, *Sep. Purif. Technol.*, 2017, **176**, 208–219.

14 A. G. O. Souza, P. Aliprandini, D. C. R. Espinosa and J. A. S. Tenório, Scandium Extraction from Nickel Processing Waste Using Cyanex 923 in Sulfuric Medium, *JOM*, 2019, 101081.

15 X. J. Bao, Z. J. Wang, L. J. Xi, J. B. Liu, Z. F. Su, T. Yang and D. B. Zhou, Separation of Sc2O3 from Bayan Obo tailings through an innovative roasting method, *Rare Met.*, 2022, **41**(3), 1071–1076.

16 B. Zhang, X. X. Xue and H. Yang, A novel process for recovery of scandium, rare earth and niobium from Bayan Obo tailings: NaCl-Ca$(OH)_2$-coal roasting and acid leaching, *Miner. Eng.*, 2022, **178**, 107401.

17 L. A. Pasechnik, V. M. Skachkov, A. Y. Chufarov, A. Y. Suntsov and S. P. Yatsenko, High purity scandium extraction from red mud by novel simple technology, *Hydrometallurgy*, 2021, **202**(1), 105597.

18 X. L. Pan, H. F. Wu, Z. Y. Lv, H. Y. Yu and G. F. Tu, Recovery of valuable metals from red mud: A comprehensive review, *Sci. Total Environ.*, 2023, **904**, 166686.

19 A. B. Botelho Junior, D. C. R. Espinosa, J. Vaughan and J. A. S. Tenório, Recovery of scandium from various sources: A critical review of the state of the art and future prospects, *Miner. Eng.*, 2021, **172**, 107148.

20 H. Ibrahim, M., D. Batstone, J. Vaughan and K. Steel, Electrochemical separation of sulfuric acid from magnesium sulfate solutions: Application for nickel laterite processing, *Sep. Purif. Technol.*, 2024, **336**, 126291.

21 W. Astuti, F. Nurjaman, F. Rofiek Mufakhir, S. Sumardi, D. Avista, K. Cleary Wanta and H. Tri, Bayu Murti Petrus, A novel method: Nickel and cobalt extraction from citric acid leaching solution of nickel laterite ores using oxalate precipitation, *Miner. Eng.*, 2023, **191**, 107982.

22 W. Wang and C. Y. Cheng, Separation and purification of scandium by solvent extraction and related technologies: a review, *J. Chem. Technol. Biotechnol.*, 2011, **86**(10), 1237–1246.

23 X. Peng, L. Li, M. Zhang, Y. Cui, X. Jiang and G. Sun, Preparation of ultra-high pure scandium oxide with crude product from titanium white waste acid, *J. Rare Earths*, 2023, **41**(5), 764–770.

24 K. Shibayama, T. Yokogawa, H. Sato, M. Enomoto, O. Nakai, T. Ito and Y. Hattori, Taganito HPAL Plant Project, *Miner. Eng.*, 2016, **88**, 61–65.

25 Z. T. Ichlas, S. A. Jones, D. C. Ibana, G.-G. Lee and R. D. Alorro, Technospheric mining of scandium from hydrometallurgical tailings of nickel laterite processing: Selection of lixiviant and optimisation of leaching variables, *Miner. Eng.*, 2022, **179**, 107436.

26 W. Yoshida and M. Goto, Ternary extractant system consisting of PC-88A, TOPO, and Versatic 10 for recovery of scandium(III) from nickel laterite processing liquors, *Hydrometallurgy*, 2023, **217**, 106024.

27 Y. Zhang, H. Zhao, M. Sun, Y. Zhang, X. Meng, L. Zhang and G. Qiu, Scandium extraction from silicates by hydrometallurgical process at normal pressure and temperature, *J. Mater. Res. Technol.*, 2020, **9**(1), 709–717.

28 S. Aydoğan, A. Aras, G. Uçar and M. Erdemoğlu, Dissolution kinetics of galena in acetic acid solutions with hydrogen peroxide, *Hydrometallurgy*, 2007, **89**(3), 189–195.

29 M. A. R. Önal and Y. A. Topkaya, Pressure acid leaching of Caldag lateritic nickel ore: An alternative to heap leaching, *Hydrometallurgy*, 2014, **142**, 98–107.

30 J. Anawati, *Metallurgical Processes for Extraction of Strategic Materials from Bauxite Residue and Ionic Clays*, University of Toronto: Canada, Ontario, CA. 2022. vol. 272.

31 B. I. Whittington, R. G. McDonald, J. A. Johnson and D. M. Muir, Pressure acid leaching of arid-region nickel laterite ore: Part I: effect of water quality, *Hydrometallurgy*, 2003, **70**(1), 31–46.

32 S. Sangita, N. Nayak and C. R. Panda, Extraction of aluminium as aluminium sulphate from thermal power plant fly ashes, *Trans. Nonferrous Met. Soc. China*, 2017, **27**(9), 2082–2089.

33 B. Onghena and K. Binnemans, Recovery of Scandium(III) from Aqueous Solutions by Solvent Extraction with the Functionalized Ionic Liquid Betainium Bis(trifluoromethylsulfonyl)imide, *Ind. Eng. Chem. Res.*, 2015, **54**(6), 1887–1898.

34 L. A. Pasechnik, I. S. Medyankina and S. P. Yatsenko, Scandium extraction from multicomponent systems by crystallization of complex sulfates, *IOP Conf. Ser.:Mater. Sci. Eng.*, 2020, **848**, 012064.

35 F. Meng, X. Li, L. Shi, Y. Li, F. Gao and Y. Wei, Selective extraction of scandium from bauxite residue using

ammonium sulfate roasting and leaching process, *Miner. Eng.*, 2020, **157**, 106561.

36 X. Song, J. Zhao, Y. Li, Z. Sun and J. Yu, Thermal decomposition mechanism of ammonium sulfate catalyzed by ferric oxide, *Front. Chem. Sci. Eng.*, 2013, **7**(2), 210–217.

37 D. Tian, X. Y. Shen, Y. C. Zhai, P. Xiao and P. Webley, Extraction of iron and aluminum from high-iron bauxite by ammonium sulfate roasting and water leaching, *J. Iron Steel Res. Int.*, 2019, **26**(6), 578–584.

38 L. A. Pasechnik, A. P. Tyutyunnik, A. N. Enyashin, I. V. Baklanova and V. M. Skachkov, Synthesis and crystal structure of 3R and 1T' polytypes of $NH_4Sc(SO_4)_2$, *J. Solid State Chem.*, 2017, **255**, 50–60.