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An investigation into the pretreating of a low-grade bauxite ore using magnetic separation and flotation processes

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Abstract

A bauxite reserve with an Al_2O_3 grade less than 50% is classified as low-grade bauxite due to the presence of impurities such as silica and iron oxide, thus pretreatment before concentration (Bayer process) is required to reduce SiO_2 and Fe_2O_3. Hence, this study investigates the feasibility of removing SiO_2 and Fe_2O_3 from a low-grade bauxite ore using magnetic separation and flotation processes. Different magnetic methods were utilized to eliminate iron impurities, and the results indicated that the high-gradient magnetic separation (HGMS) with an intensity of 18000 Gauss had the best output in reducing Fe_2O_3 with 18.03% Fe_2O_3 content and a 31.15% removal rate. In the next step, reverse and direct flotation techniques were employed to remove silica from the sample. The findings showed that the highest removal efficiency of SiO_2 was achieved using the reverse flotation process at 750 g/t Armac C collector, 1200 g/t starch, 40 g/t MIBC and 11 pulp pH. Ultimately, the HGMS followed by reverse flotation with an Armac C collector (750 g/t) was implemented on the bauxite sample, and the content of Fe_2O_3, and SiO_2, silica removal efficiency, and Al_2O_3/SiO_2 mass ratio was found to be 17.76, and 22.11%, 45.22% and 1.63, respectively, indicating that HGMS followed by flotation is not effective enough in reducing iron oxide and silica, and therefore requires further pretreatment and/or the study of a suitable alternative method. The behaviour observed may be because iron oxide and silica minerals were fine-grained diseminated in the bauxite sample.

Keywords:
low-grade bauxite, silica, removal efficiency, magnetic separation, reverse flotation

1. Introduction

Aluminum is one of the most significant and plentiful non-ferrous metals in the Earth's crust and is widely applied in various industrial fields owing to the unique properties and its strategic nature, such as the high electrical and thermal conductivity attributes, high strength and hardness, good corrosion resistance, lightweight, and mechanical particularities (**Ma et al., 2009; Krishnan et al., 2019; Khodadadi Bordboland et al., 2024**). Bauxite ore is the primary resource for producing alumina/aluminum, and nearly 90% of the world's aluminum is obtained from bauxite. Bauxite includes gibbsite, boehmite and diaspore minerals, which are mixed with two iron oxides, goethite and hematite, kaolinite clay mineral and small amounts of anatase and ilmenite (**Verma et al., 2022; Kar et al., 2023; Banerjee et al., 2023; Khodadadi Bordboland et al., 2024**). The global market demand for this metal as well as the increase of aluminum consumption in various industrial fields, especially high-tech industries, has led to the shortage of high-quality bauxite resources and subsequently providing alumina from the low-grade bauxites in recent years. Bauxite with Al_2O_3 grade less than 50% is classified as low-grade bauxite due to the presence of impurities such as silica, iron oxide and calcium (**Pattanaik et al., 2020**). It is also estimated that a silica content higher than 8 wt% and an Al_2O_3/SiO_2 weight ratio less than 6.25 is called a low-grade bauxite ore (**Azof et al., 2020**). The most common and main hydrometallurgical method for producing pure alumina from the bauxite is the Bayer process, which is based on pressure alkaline leaching of the mined ore with a concentrated solution of sodium hydroxide (**Khodadadi Bordboland et al., 2024**). On the other hand, this technique does not respond well to low-grade resource, so it is urgently needed to pre-treat the low-grade bauxites and reduce the important impurities in bauxite such as silica and iron oxide for increasing Al_2O_3/SiO_2 (modulus) ratio and eliminating impurity problems.

In recent years, the diverse research works have been performed to efficiently extract alumina from the low-

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grade bauxites, which mostly focused on the pre-processing of low-grade resource to remove impurities and increase the modulus of sodium aluminate solution. For instance, **Bhagat et al. (2006)** applied the dry magnetic separation method for the bauxite ore from Durgamanwadi mines, and reported that this technique was able to reduce the content of Fe_2O_3 and TiO_2 to h1.52 and 5.16% in concentrate and 3.31 and 7.31% in the feed. **Massola et al. (2009)** investigated the separation process of silica from bauxite via reverse cationic flotation in pilot and bench scales and reported that the commercial ether-amine as the quartz collector, and starch as the gibbsite depressant had the best performance at pH 10. **Yu et al. (2016)** applied the reverse flotation process to separate silicate minerals from a low-grade (diaspore) bauxite with TAS101 cationic collector and starch depressant and obtained alumina recovery of 83.34% at pH 11. According to their results, Al_2O_3/SiO_2 ratio increased from 6.1 to 9.58. **Han et al. (2018)** conducted an experimental study on the desilication of low-grade bauxite by flotation method and reported that Al_2O_3/SiO_2 ratio and Al_2O_3 content in the concentrate promoted from 3.27 to 7.97 and from 58.91 to 65.05% under optimum operating conditions, respectively. **Zhang et al. (2019)** used the semi-smelting (reduction process) followed by a magnetic separation to recover iron and alumina from iron rich bauxite, and as a result, 96.84% of iron was removed from the studied sample. Their proposed optimal condition was CaO/SiO_2 ratio of 1.25, temperature of 1425°C, time of 15 min, anthracite ratio of 11.92 wt%, CaF_2 ratio of 1.96 wt%, and magnetic intensity of 4000 Gauss. **Yuan et al. (2020)** addressed removing iron minerals from the high-iron bauxite ore using an advanced roasting technology and reached to 80.64% alumina recovery and 85.32% removal rate of iron. **Birinci and Gök (2021)** investigated the flotation of a low-grade boehmitic bauxite before the Bayer processing and reported that a concentrate with high Al_2O_3/SiO_2 ratio (8.54) and low Al_2O_3 recovery (42.87%) could be achieved utilizing reverse cationic flotation. **Wang et al. (2022)** utilized suspension magnetization roasting–magnetic separation technology for high-efficiency separation of iron minerals from an iron rich bauxite. According to their findings the optimum separation condition was found to be 650°C roasting temperature, 20 min roasting time, 20% CO concentration, and particle size lower than 37 µm. **Li et al. (2022)** studied the pretreatment of low-grade diasporic bauxite using flotation process with oxalic acid. Their results indicated that this technique could reduce SiO_2 recovery from 53.26 to 47.30% and enhance Al_2O_3 recovery from 75.39 to 81.88%.

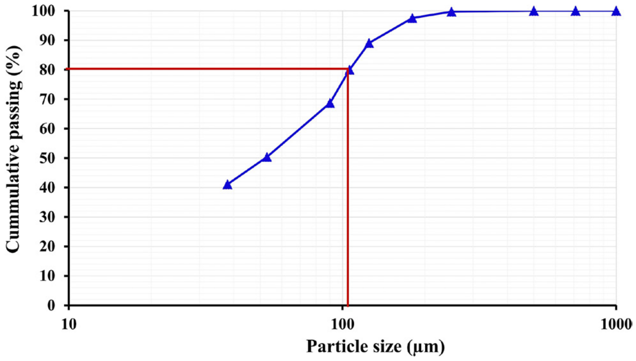

Figure 1. Particle size distribution curve of the bauxite sample studied

Table 1. XRF analysis of the main oxides in the bauxite sample studied

Composition	Fe_2O_3	TiO_2	CaO	SiO_2	Al_2O_3	MgO	L.O.I	ETC
Content (Wt.%)	i21.2	i3.39	i0.54	i26.5	i36.11	i0.24	i6.25	I5.77

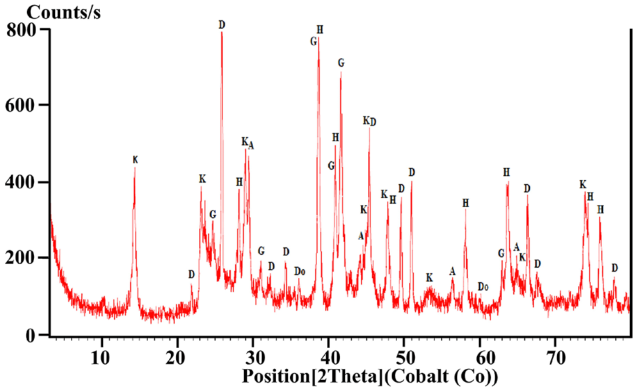

Figure 2. Mineralogical diffractograms of bauxite sample studied

Literature indicates that there are the various methods for pretreating the bauxite ores such as roasting-low intensity magnetic separator, calcination-magnetic separator, roasting-acid leaching, high intensity magnetic separator, flotation, flocculation, bioflocculation, reduction operation with salt and use of bacteria. Among these techniques, it is found that using the combined method of high-intensity magnetic separation and flotation can be appropriate and cost-effective to increase the modulus of bauxite. Therefore, this study was focused on the

Table 2. % liberation of iron minerals (goethite, hematite) and quartz in different fractions

Size fractions (µm)	i250	i150	i106	i75	i53	i38	< 38
% liberation for iron (goethite-hematite) (%)	i60.40	i77.96	i85.29	i88.82	i89.20	i92.61	i93.56
% liberation for quartz (%)	i57.14	i82.35	i95.65	i100	i100	i100	i100

Table 3. Experimental conditions of magnetic tests conducted on the Jajarm bauxite representative sample

Run	Sample code	Type of method	Size (d_{80}) (µm)	Magnetic field intensity (Gauss)
1	ASH-1	High intensity dry magnetic separation at room temperature.	2000	11000
2	ASH-2	High intensity dry magnetic separation at room temperature.	106	11000
3	ASH-3	High intensity wet magnetic separation at room temperature.	106	11000
4	A-SH-1	High intensity wet magnetic separation at room temperature.	2000	14000
5	B-B-M	High intensity dry magnetic separation at room temperature.	2000	14000
6	D1	High intensity wet magnetic separation at room temperature.	106	14000
7	D2	High intensity wet magnetic separation with the addition of coking coal at a rate of 5% of feed weight at room temperature.	106	11000
8	D3	High intensity wet magnetic separation combined with heating the sample for 30 min at 950°C.	106	11000
9	B-H-M	High intensity wet magnetic separation combined with heating the sample for 30 min at 800°C.	2000	14000
10	M-N-1	High intensity wet magnetic separation with the addition of coking coal at the rate of 5% of feed weight and heating for 30 min at 950°C.	106	11000
11	UT1	High gradient magnetic separation-wet method	106	18000
12	UT2	Low intensity magnetic separation (two-stage) with the addition of coking coal (100g) and heating for 150 min at 850°C.	106	1000 and 800

Table 4. The experimental conditions of reverse flotation tests with various cationic collectors

Run	Solid (%)	Pulp pH	Frother (g/t)	Depressant (g/t)	Collector type	Collector dose (g/t)	Conditioning time (min)	Collecting time of froth (min)
1	30	7.8	-	-	Armac T	250	7	2
2	24	7.8	40	-	Armac T	500	7	2
3	30	7.8	-	-	Armac T	750	7	2
4	15	10	40	1000	Armac C	750	12	2
5	30	11	40	1200	Armac C	750	12	2
6	15	11	40	1200	Armac T	750	12	2
7	15	11	80	1200	Armac T	750	12	3
8	30	11	-	1200	CTAB	750	11	3
9	15	10	-	1200	CTAB	750	11	3
10	30	9.5	-	1200	CTAB	750	11	3
11	20	11.5	-	-	Dodecylamine	250	7	2
12	20	11.5	-	-	Dodecylamine	1000	7	2
13	20	9.5	-	-	Dodecylamine	1000	7	2

pre-processing of a low-grade bauxite to reduce silica and iron using the magnetic and flotation processes.

2. Experimental section

2.1. Materials

To conduct the tests, firstly, the target bauxite sample (about 100 kg) was provided by the Jajarm bauxite mine belonging to the Iran Alumina Company. The samples were subjected to one-stage comminution operations using a roller crusher so that all the particles passed through the 10 mesh (2 mm) control sieve. Thereafter, the samples were mixed and homogenized several times using the cone-and-quarter method and riffling, and then placed into sealed polyethylene bags. Generally, after homogenization, the samples were divided into two parts, one of which was archived as a backup sample, and the other part of the samples was used for the tests. After that, the samples weighing i100 g for magnetic tests and 600 g for flotation tests were prepared using riffle and different homogenizations. Part of the prepared representative samples was pulverized and transferred for various analyses, such as sieve analysis, XRF, and XRD. Sieve analysis was performed on the representative sample to determine the particle size distribution and d_{80} of the samples in the fractions of 250, 180, 125, 90, 53, 38, and less than 38 microns. **Figure 1** displays particle size distribution curve of the bauxite sample studied after the roller crushing process. As can be seen, d_{80} is equal to 106 µm. The chemical and mineralurgical compositions of representative sample were also determined by X-ray Fluorescence (XRF 1800, Shimadzu, Japan) and X-ray diffraction (XRD, ASENWARE, AW-XDM300, China), respectively, as shown in **Table 1** and **Figure 2**. It can be observed from the XRF results presented in **Table 1** that the amounts of Fe_2O_3, SiO_2 and Al_2O_3 oxides in the bauxite sample studied were 21.2, 26.5 and 36.11%, respectively. The XRD analysis (see **Figure 2**) indicates that the major components identified in the bauxite sample investigated were kaolinite, diaspore, hematite, goethite, and anatase.

Additionally, a 3 kg sample was selected to provide the polished and thin sections for determining % liberation of metallic (ferrous compounds) and non-metallic (silica) impurities, and these results are reported in **Table 2**.

2.2. *Methods*

2.2.1. Magnetic separation

To investigate the possibility of separating iron-containing minerals from the Jajarm bauxite low-grade ore, different magnetic methods (such as high-intensity, high-gradient and low-intensity) under various operating conditions were established on the representative samples. Two dimensional fractions of -2 mm and -106 µm were selected to conduct the experiments. It is also noteworthy that regarding the type of iron in the studied sample (hematite) the higher field intensities were further considered in the experiments. The high-intensity magnetic tests were conducted in wet and dry forms. In tests, 100 grams of the bauxite representative sample was used as input into the separator. Additionally, some tests were performed by adding coking coal to the bauxite sample (magnetic separator feed) and mixing and heating the samples by an electric furnace. In addition to the high-intensity experiments, a high-gradient magnetic separation test with a field intensity of 18000 Gauss was conducted on a 1900 (1870) g-bauxite sample with d_{80}= 106 µm. Moreover, the low-intensity magnetic separation test was carried out on a 1000 g representative sample with d_{80}= 1106 µm, which was mixed with 100 g of coking coal for 150 minutes at 850°C (30 min to reach a temperature of 850°C and then 120 minutes at a constant temperature of 850°C) in an electric furnace. In the next step, the sample derived from the furnace was subjected to a two-stage magnetic separation operation with a field intensity of 1000 and 800 Gauss. It is notable that in this test, the weight of the sample obtained from the furnace was about 911 g. The operating conditions of magnetic tests (high-intensity, high-gradient, and low-intensity) are given in **Table 3**.

Table 5. Taguchi L9 design matrix used for reverse flotation tests with Armac C

Factor	Levels			
	Low (-1)	Center (0)	High (+1)	
(A) Collector concentration (g/t)	250	500	750	
(B) Depressant concentration (g/t)	600	900	1200	
(C) Frother concentration (g/t)	20	30	40	
(D) Pulp pH	9	10	11	
Run	A	B	C	D
1	L500	L900	L40	L9
2	L750	L1200	L30	L9
3	L500	L600	L30	L11
4	L500	L1200	L20	L10
5	L750	L600	L40	L10
6	L750	L900	L20	L11
7	L250	L1200	L40	L11
8	L250	L600	L20	L9
9	L250	L1200	L30	L10

2.2.2. Flotation experiments

The flotation tests were conducted on the representative samples using a Denver flotation machine in a 2 L cell at a certain pulp density and pH with the impeller speed of 850 rpm. The pulp pH was adjusted using sodium hydroxide and sulfuric acid at the desired amount, and then starch was added into the cell as the depressant of iron minerals. After 3-4 min and resetting the pH, the collector was added to the cell and after 3-5 min, MIBC (Methyl Isobutyl Carbinol) frother was added into the cell and, then after 1 min, air was opened and flowed inside the cell, and the froth was collected for 3 min with a scraping interval of every 10 s. Thereafter, the float and

Table 6. The experimental conditions of direct flotation tests with a sodium oleate collector

Run	Solid (%)	Pulp pH	Frother (g/t)	Depressant (g/t)	Collector dose (g/t)	Conditioning time (min)	Collecting time of froth (min)
1	10	11	-	400	300	11	2
2	25	9.5	40	300	250	12	2
3	25	9.5	40	-	250	9	2
4	20	7.8	20	300	250	12	2
5	25	6	20	500	300	12	2
6	25	4	20	500	300	12	2

sink fractions were filtered, dried and assayed. Reverse flotation experiments were performed using various cationic collectors such as Armac C, Armac T, cetyltrimethylammonium bromide (CTAB) and direct flotation tests were conducted with sodium oleate collector.

Reverse flotation tests were aimed in two phases. In the first phase, the experiments were conducted with different collectors under various experimental conditions, as displayed in **Table 4**. In the second phase, after selecting the appropriate collector, the experiments were designed and performed according to the design of experiments (DOE) technique. For this goal, the Taguchi approach (L9 design) was employed to design the experiments, as shown in **Table 5**. In addition to reverse flotation tests, some direct flotation experiments were conducted, and these operating conditions are reported in **Table 6**.

3. Results and discussion

3.1. Magnetic separation

Magnetic separation is known as an effective physical method for recovering iron from ore, and its simple operation, low investment cost, and minimal pollution are its features (**Long et al., 2020; Aslam et al., 2024**), so it was applied as a pre-treating process. It is also noteworthy that the removal efficiency was measured utilizing equations 1 and 2. Additionally, it should be noted that the feedstock for the subsequent stages is the non-magnetic fractions obtained from the magnetic separation process.

$$Ff = Cc + Tt \tag{1}$$

$$\%R = 100 \times (1 - \frac{Tt}{Ff}) \tag{2}$$

In the above equations, R is the removal efficiency (%), F implies the weight of feed, and C, and T are the weights of magnetic and non-magnetic fractions, respectively. Also f is the grade of feed, and c, and t are the grades of magnetic and non-magnetic fractions, individually.

The findings obtained from the magnetic tests are illustrated in **Table 7**. As can be from **Table 7**, the high gradient magnetic separation (HGMS) with a field intensity of 18000 Gauss (Run 11), the high intensity wet magnetic separation combined with heating the sample for 30 min at 950°C, and the high intensity magnetic test performed with wet method at a magnetic field of 14000 Gauss (Run 8) at room temperature have better performance in comparison to the other tested methods. It was found to be 31.15, for the high gradient magnetic separation with a field intensity of 18000 Gauss. It was also obvious that the content of Fe_2O_3 reduced from 21.2% to ~18%. The higher separation by the HGMS is due to the fact that HGMS separators use a magnetic matrix that creates a high magnetic field gradient to attract particles with high magnetic force to them (**Baik et al., 2010**). The HGMS separators have been widely utilized to separate and concentrate fine weakly magnetic minerals in

Table 7. The XRF results of non-magnetic samples obtained from the magnetic tests

Row	Sample code	Al_2O_3	SiO_2	CaO	TiO_2	Fe_2O_3	MgO	LOI & ETC	Weight percentage (%)	Removal efficiency (%)
K1	ASH-1	K37.93	K27.53	K0.58	K3.47	K20.24	< DL	K10.25	75.5	27.92
K2	ASH-2	K37.12	K26.05	K0.66	K3.36	K19.14	< DL	K13.67	78.8	28.86
K3	ASH-3	K38.06	K26.93	K0.72	K3.37	K18.38	< DL	K12.54	80.18	30.48
K4	A-SH-1	K36.15	K25.45	K0.7	K3.21	K20.15	< DL	K14.33	78.4	25.48
K5	B-B-M	K37.83	K27.2	K0.54	K3.47	K21.04	< DL	K9.92	76.6	23.98
K6	D1	K37.96	K26.98	K0.69	K3.29	K18.33	< DL	K12.77	80.03	30.80
K7	D2	K33.55	K23.82	K0.97	K3.24	K19.56	< DL	K18.86	77.5	28.5
K8	D3	K39.43	K27.24	K0.81	K3.52	K18.05	< DL	K10.95	81.95	30.23
K9	B-H-M	K41.50	K24.76	K0.56	K3.54	K21.37	< DL	K8.27	69.8	29.64
K10	M-N-1	K39.68	K26.32	K0.72	K3.36	K19.76	< DL	K10.16	76.7	28.51
11	UT1	37.99	26.14	0.63	3.43	18.03	0.23	13.55	80.95	31.15

Table 8. The XRF results of samples obtained from the magnetic tests with adding coking coal (100 g) and heating for 150 min at 850°C in two stages

Sample type	Sample code	Al_2O_3	SiO_2	CaO	TiO_2	Fe_2O_3	MgO	LOI & ETC
Non-magnetic fraction	UT2t	K34.23	K33.73	K0.97	K3.96	K19.86	K0.35	K6.9
Magnetic fraction	UT2c	K27.71	K25.19	K0.45	K3.22	K41.66	K0.25	K1.52

the mineral processing industry (**Zheng et al., 2015; Ge et al., 2017**). **Hu et al. (2023)** reported that the HGMS separation is a useful method for selective enrichment of fine weakly magnetic particles, and the capture phenomenon of particles is a two-step process including transport and attachment. Also, the main reason for effective selectivity of HGMS compared to other magnetic methods is owing to the retention of non-magnetic minerals as the feed passes through the magnetized matrix stack. However, in general, the removal efficiency of more than 31% was not observed for iron separation, which may be because iron (hematite, goethite) and silica minerals are usually fine-grained and distributed throughout a bauxite sample (**Cardenia et al., 2019**). Consequently, fine particle size and severe agglomeration lead to low separation efficiency and concentrate quality (low-grade iron) (**Aslam et al. 2024**). **Li et al. (2011)** utilized a high-gradient magnetic separation for the recovery of iron from red mud, and reported a product with Fe grade of 35.5% and a recovery rate of 10%.

In addition, the low-intensity magnetic test (two-step mode) was conducted on the bauxite sample (see **Table 8**) after the addition of coking coal and heating for 150 min at 850°C. Coking coal, by producing reducing gases such as CO and CO_2 at high temperatures, can facilitate the conversion of iron oxides into magnetic compounds. This transformation enables iron to be easily attracted to a magnetic field and separated from other materials. According to **Liu et al. (2017)**, in the reduction step, the phase transition of iron bearing and aluminium minerals can be shown as reactions (3), and (4), respectively.

$$FeOOH \rightarrow Fe_2O_3 \rightarrow FeO.Al_2O_3 \rightarrow Fe \qquad (3)$$

$$Al(OH)_3 \,/\, AlOOH \,/\, n(Al_2O_3).SiO_2.nH_2O \qquad (4)$$

$$\rightarrow Al_2O_3 \rightarrow FeAl_2O_4 \rightarrow Al_2O_3 \,/\, Al_6Si_2O_{13} \qquad (5)$$

The findings indicated that the magnetic tests with low intensity even in the presence of coke and at temperatures around 850°C was not efficient for the bauxite sample investigated.

3.2. Flotation

In the second phase, the flotation process was aimed to investigate the possibility of reducing the amount of silica from the studied bauxite sample. In this regard, firstly a series of initial reverse flotation tests was performed under different conditions with the Armac T collector, and the results are shown in **Figure 3**. It can be seen from **Figure 1** that the removal efficiency of silica increases with a gentle slope as the pH increases from 8 to 11 and the collector dose increases from 250 to 750 g/t. In general, the sorption density of a cationic collector on the silica surface increases with an increase in the collector dose (**Wu et al., 2023**), and stronger flotation of silica occurs in alkaline pH due to the fact that the surface charge becomes more negative as pH enhances, and amine molecules begin to form in alkaline pH (**Ma et al., 2009**). It is also observed that an increase in the amount of depressant (starch) has a significant effect on the flotation performance and the removal efficiency increases drastically with the increase in starch dosage. According to **Gibson et al. (2017)** and **Yu et al. (2016)**, starch acts as a strong depressant on aluminum minerals, especially diaspore at alkaline pH, and an increase in starch concentration is led to render further depression. Indeed, the chemical adsorption of starch occurs on the available Al-O sites on the surface of diaspore particles. **Yu et al. (2016)** reported a pH of 11 and starch dosage of 1200 g/t for better separation of the silicate compounds from the diasporic bauxite sample by reverse flotation. Meanwhile, it is found that the use of 40 g/t MIBC frother is led to enhance the removal efficiency of silica in the investigated system, and increasing the amount of frother from this amount caused a sharp decrease in silica removal efficiency. In addition, another set of reverse flotation experiments was performed with several of cation collectors, and these results are presented in **Table 9**. As can be seen in **Table 9**, reverse flotation of the bauxite sample using 750 g/t Armac C collector (Run 5) at 30% solid present, 11 pulp pH, 40 g/t MIBC, 1200 g/t starch, and 12 min conditioning time had a higher removal efficiency compared to the other modes. In these conditions, the amount of SiO_2 in the feed decreased from 25.5 to 21.17%, and the removal efficiency of silica was determined to be 52.95%. Thus, regarding the better removal yield of Armac C, this collector was selected to continue the study. A series of reverse flotation tests was designed and established based on the Taguchi approach (L9 design) to evaluate and optimize parameters affecting the removal efficiency of silica, and these results are illustrated in **Table 10** and **Figure 4**. It can be observed from **Figure 2** that the removal efficiency generally improves with an increase in the values of operating parameters in the investigated range. However, according to **Figure 4**, the collector and frother depicted a similar behaviour, so that first, the removal efficiency increased slightly with the increment of the collector dosage from 250 (level 1) to 500 g/t (level 2) and frother concentration from 10 (level 1) to 30 g/t (level 2), and then increased significantly as the consumption of these two chemicals enhanced to level 3 (750 g/t for collector and 40 g/t for frother).

Table 10 demonstrates that T5 has the higher silica removal efficiency compared to other runs. Additionally, a relatively high amount of Fe_2O_3 is still observed, which means that slightly more hematite and goethite remain in the concentrate owing to the depressing impact of starch (**Birinci and Gök, 2021**). The operating conditions for T5 were found to be a pH of 10, collector concentration of 750 g/t, depressant (starch) dosage of 600 g/t, and frother concentration of 40 g/t. Moreover, comparing the results of **Tables 9** and **10** for Armac C collector indicate

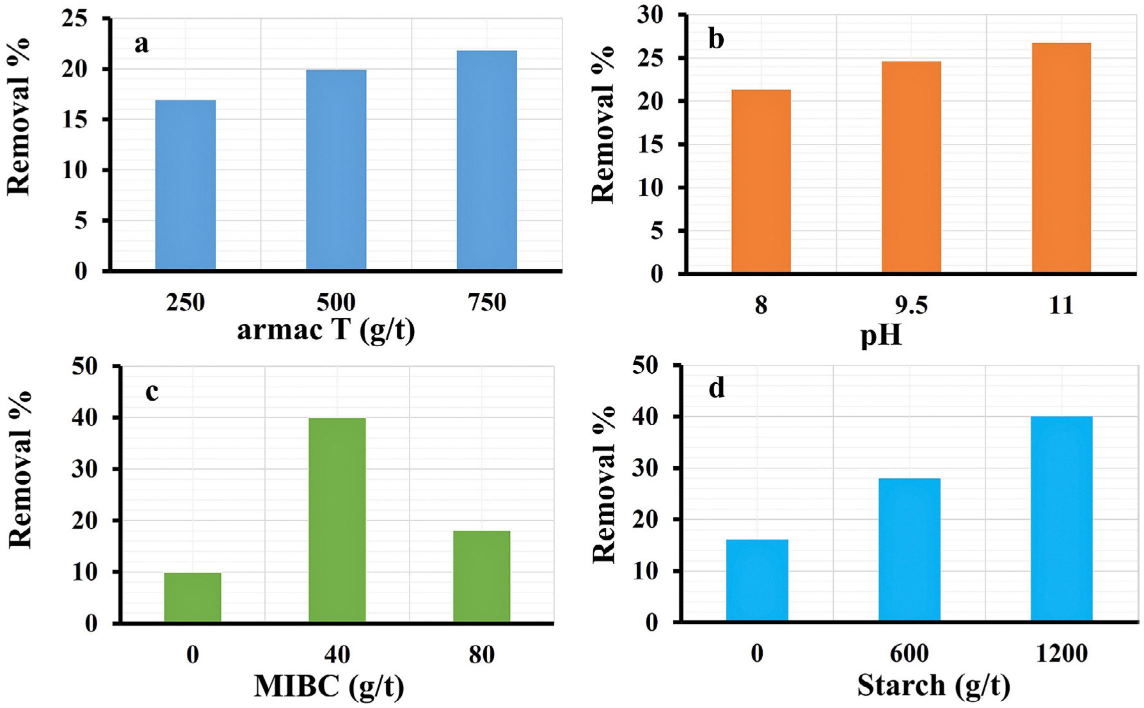

Figure 3. Effect of different operating parameters on silica removal efficiency from bauxite sample in reverse flotation with Armac T collector (**a**: effect of collector at pH 7.8, frother 40 g/t, solid percentage 30, and conditioning time 12; **b**: effect of pH at Armac T g/t 750, frother 40 g/t, solid percentage 30, and conditioning time 12; **c**: effect of frother at Armac T 750 g/t, pH 11, depressant 1200 g/ton, solid percentage 30, and conditioning time 12; **d**: effect of depressant at Armac T g/t 750, pH 11, frother g/t 40, solid percentage 30, and conditioning time 12).

Table 9. XRF results of bauxite sample after reverse flotation experiments and the measured values of silica removal efficiency

Run	Sample code	Al_2O_3	SiO_2	CaO	TiO_2	Fe_2O_3	MgO	LOI&ETC	Weight percentage (%)	Removal efficiency (%)
K1	T1	K35.22	K25.59	K0.55	K3.36	K20.55	< DL	K13.72	95.3	K7.97
K2	T2	K36.66	K25.43	K0.51	K3.36	K20.51	< DL	K14.51	80.80	K22.46
K3	T3	K36.54	K25.45	K0.50	K3.39	K20.61	< DL	K13.48	84.09	k19.24
K4	T4	K36.38	K22.95	K0.53	K3.32	K20.64	k0.23	K15.94	62.35	k46
K5	T5	K35.36	K21.17	K0.46	K3.16	K19.33	k0.22	K20.28	58.89	k52.95
K6	T6	K38.25	K25.51	K0.51	K3.28	K20.92	k0.26	K11.24	64.57	k37.84
K7	T7	K38.20	K26.18	K0.59	K3.46	K21.23	k0.27	K10.06	92.87	k8.25
K8	T8	K32.01	K24.05	K0.56	K3.24	K20.01	K0.26	K19.87	94.41	k14.32
K9	T9	K33.04	K23.64	K0.57	K3.27	K20.37	k0.26	K18.84	94.71	k15.51
K10	T10	K36.45	K23.61	K0.51	K3.34	K20.85	k0.26	K14.96	92.29	k17.78
K11	T11	K33.91	K24.41	K0.55	K3.21	K19.69	k0.27	K17.93	94.08	k13.34
K12	T12	K34.76	K24.44	K0.58	K3.26	K19.98	K0.27	K16.69	87.74	K19.08
K13	T13	K37.85	K24.91	K0.62	K3.31	K21.15	K0.28	K11.40	56.96	K46.46

that when the values of pH, collector (Armac C), depressant (starch), and frother (MIBC) are adjusted to 11, 750 g/t, 1200 g/t, and 40 g/t, they can have a better performance in reducing silica. Therefore, this test was selected as the optimal condition for decreasing silica from the studied bauxite sample. **Wulandari et al. (2018)** applied reverse flotation to improve the quality of a bauxite sample containing 36.80 wt% of Al_2O_3, and 24.05 wt% of SiO_2 (almost similar to the present work) and showed that this process could not effectively separate silica from the investigated bauxite sample owing to the complexity of silica-alumina minerals.

In addition to reverse flotation with cationic collectors, the desilication process of low-grade bauxite sample was also investigated by direct flotation method using a number of experiments, and the XRF results of the samples are given in **Table 11**. However, the difficulty in dewatering the concentrate and its great reagent con-

Table 10. XRF results of reverse flotation experiments based on L9 plan and the measured values of silica removal efficiency

Run	Sample code	Al_2O_3	SiO_2	CaO	TiO_2	Fe_2O_3	MgO	LOI&ETC	Silica removal efficiency (%)
K1	L1	K34.22	K23.01	K0.56	K3.20	K19.58	K0.27	K19.16	K46.24
K2	L2	K34.75	K23.20	K0.55	K3.22	K19.98	K0.27	K18.03	K47.54
K3	L3	K34.43	K23.09	K0.54	K3.22	K19.80	K0.25	K18.67	K45.79
K4	L4	K34.79	K23.54	K0.52	K3.24	K20	k0.26	K17.65	k46.07
K5	L5	K34.59	K23.29	K0.53	K3.21	K19.84	k0.25	K18.29	k52.44
K6	L6	K34.49	K22.56	K0.47	K3.22	K19.83	k0.24	K19.19	K50.79
K7	L7	K34.20	K22.56	K0.49	K3.17	K19.72	k0.25	K19.61	K51.79
K8	L8	K34.37	K22.66	K0.50	K3.20	K19.69	K0.25	K19.33	K38.92
K9	L9	K34.47	K22.88	K0.54	K3.22	K19.43	k0.26	K19.20	K44.13

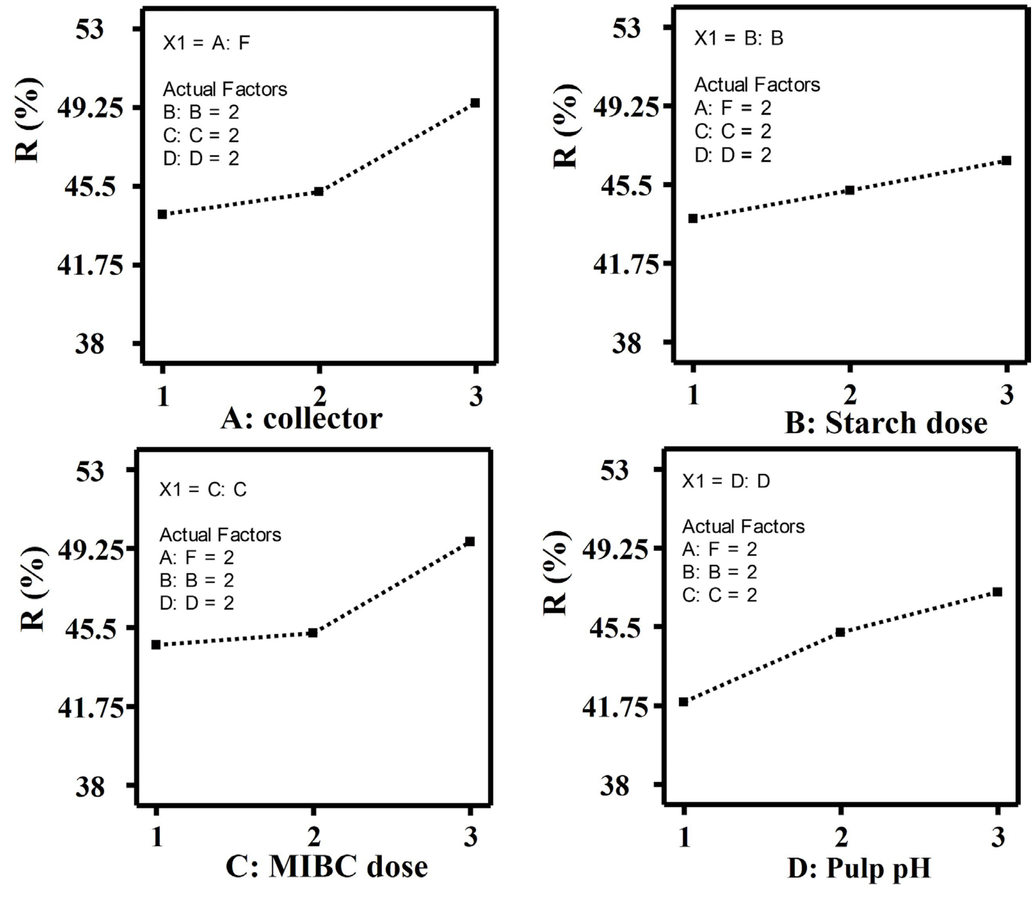

Figure 4. The effect of important operating parameters on the removal efficiency of silica with the Armac C collector

sumption have limited its widespread use in industry (**Yu et al. 2016**). XRF results showed that direct flotation gave very poor results compared to reverse flotation, so that the weight recovery and removal efficiency for direct flotation tests was less than K30%, so the use of direct flotation is not useful for the studied sample. In general, the use of cationic collectors and reverse flotation have some advantages relative to direct flotation, such as lower operating costs, and collector consumption, easier dewatering, and better reduction of impurities (**Gibson et al. 2017**).

Ultimately, regarding the better results of reverse flotation under conditions of 750 g/t Armac C collector, 30% solid present, 11 pulp pH, 40 g/t MIBC frother, 1200 g/t starch depressant, and 12 min conditioning time in decreasing SiO_2, as well as the better performance of

Table 11. XRF results of samples obtained from direct flotation tests

Run	Sample code	Al_2O_3	SiO_2	CaO	TiO_2	Fe_2O_3	MgO	LOI&ETC	Silica removal efficiency (%)
K1	DF1	K34.96	K27.21	K0.56	K3.35	K21.27	K0.31	K11.35	10.2
K2	DF2	K34.45	K27.75	K0.53	K3.38	K21.23	K0.29	K11.37	27.68
K3	DF3	K36.37	K27.32	K0.49	K3.36	K21.42	K0.28	K9.77	23.36
K4	DF4	K35.80	K27.21	K0.42	K3.38	K21.16	k0.31	K10.71	9.8
K5	DF5	K32.80	K31.81	K0.41	K3.43	K20.99	k0.24	K10.32	32.87
K6	DF6	K32.65	K31.27	K0.58	K3.53	K20.96	k0.27	K10.74	27.16

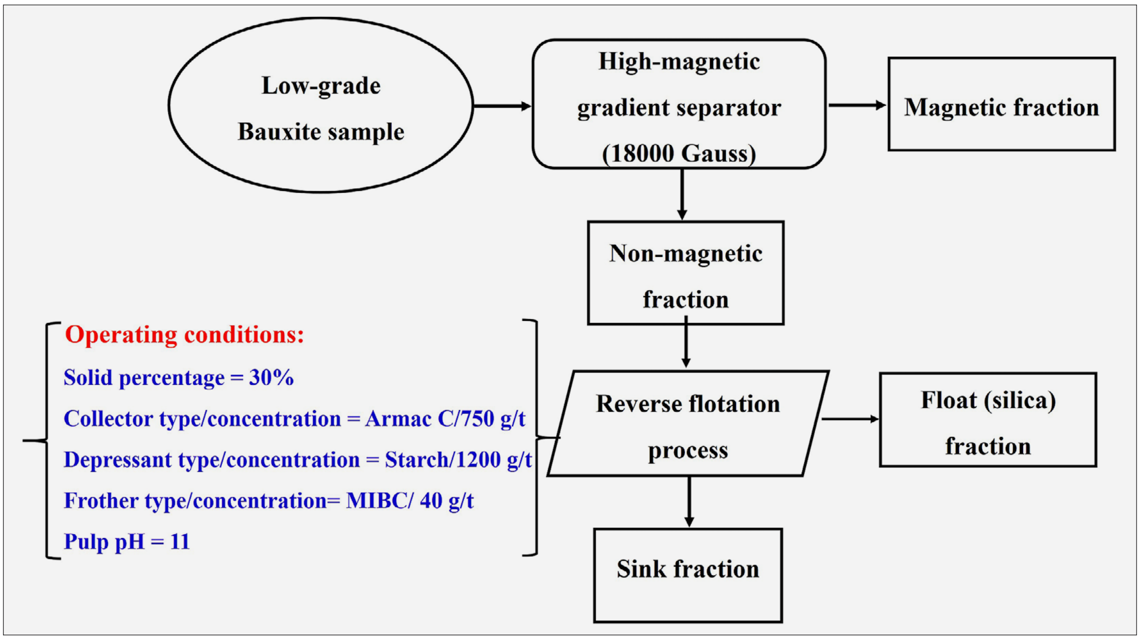

Figure 5. Flowchart of the final test performed (magnetic separation followed by reverse flotation process)

Table 12. XRF result of the final flotation test and the measured values of removal efficiency

Chemical composition	Al_2O_3	SiO_2	CaO	TiO_2	Fe_2O_3	MgO	LOI&ETC	Silica removal efficiency (%)	A/S ratio
Content (% wt.)	K36.06	K22.11	K0.56	K3.26	K17.76	0.21	K11.35	45.24	K1.63

the high gradient magnetic separation method in reducing iron, these two conditions were implemented on the bauxite feed sample, as shown in **Figure 5**. The XRF results of the final product, which is the feed of the Bayer process, are reported in **Table 12**. According to the results presented in **Table 12**, the values of Fe_2O_3 and SiO_2 using high gradient magnetic separation followed by reverse flotation with Armac C collector reached to 17.76% and 22.11%, respectively. Also, the removal efficiency of silica was determined to be 45.24%. Thus, in general, this study demonstrates that the values of iron and silica in the feed of the Bayer process can be significantly reduced. However, it must be pointed out that in the treatment of bauxite ores using the Bayer process, which is the main and conventional technique for treating these deposits worldwide, high SiO_2 content (more than 8%) and low mass ratio of Al_2O_3 to SiO_2 (lower than 4) is not suitable. As can be seen from **Table 12**, the mass ratio Al_2O_3 to SiO_2 (A/S) is about 1.63, indicating that the magnetic separation and flotation alone, despite the effective reduction of iron and silica, cannot be efficient for decreasing iron and silica from the low-grade bauxite source studied, and accordingly a suitable and alternative method should be considered. Therefore, to use this feed in the Bayer process, further pre-processing is required, for example, performing additional tests such as completing the flotation circuit and conducting cleaner and re-cleaner tests, performing heavy media cyclone tests alone or before flotation tests. Also, since the main iron phases in bauxite were goethite or hematite, magnetic roasting could be a necessary process before HGMS separation with the aim of reducing iron oxide to magnetite.

4. Conclusions

The rapid development of the aluminum industry and global growth of aluminum demand and its compounds have led to increased attention to the processing of low-grade bauxite reserves. Thus, in this research, the removal of iron and silica impurities from a low-grade bauxite deposit was examined applying the magnetic separation to reduce iron and the flotation method to reduce silica. The observations indicated that the high gradient magnetic separation with a field intensity of 18000 Gauss had the highest removal efficiency of Fe_2O_3 (31.15%) among the magnetic techniques used. Meanwhile, according to XRF analysis, the value of Fe_2O_3 reduced from 22.1 to 18.03%. It was also found that low-intensity magnetic separation had no positive effect on iron separation from the studied bauxite sample. This could be due to the distribution of the very fine particles of iron and silica minerals present in the bauxite sample. Additionally, the low-grade bauxite sample was subjected to reverse and direct flotation tests for decreasing silica. The direct flotation tests using sodium oleate collector showed that this technique could not be a suitable option for reducing silica due to the low weight recovery rates. The findings derived from various reverse flotation tests demonstrated the highest amount of silica removal efficiency (52.95%) was obtained under optimal conditions including 750 g/t of Armac C collector, 1200 g/t of starch depressant, 40 g/t of MIBC frother and pulp pH of 11. Finally, the high magnetic gradient separation (18000 Gauss) followed by reverse flotation with Armac C (750 g/t) was carried out on the low-grade bauxite sample studied. The results showed that in this case, the silica removal efficiency was 45.22%, and the magnitude of Fe_2O_3 and SiO_2 reached to 17.76 and 22.11%, respectively. Also, the modules mass ratio of Al_2O_3 to SiO_2 from 1.36 in feed reached to 1.63, indicating that despite an improvement, gradient separation followed by flotation was not beneficial enough in removing bauxite impurities, and so requires more pre-processing.

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SAŽETAK

Istraživanje mogućnosti predobrade niskokvalitetne boksitne rude korištenjem magnetske separacije i flotacije

Rezerve boksita sa sadržajem Al_2O_3 manjim od 50 % klasificiraju se kao boksit niske kakvoće zbog prisutnosti nečistoća kao što su silicij i željezni oksid. Takvim rezervama potrebna je prethodna obrada prije koncentriranja (Bayerov proces) za redukciju SiO_2 i Fe_2O_3. Ova studija istražuje izvedivost uklanjanja SiO_2 i Fe_2O_3 iz niskokvalitetne boksitne rude pomoću magnetske separacije i flotacije. Različite magnetske metode korištene su za uklanjanje željeza, a rezultati su pokazali da je visokogradijentna magnetska separacija (*high-gradient magnetic separation*, HGMS) s intenzitetom od 18 000 Gaussa imala najbolji učinak u redukciji sadržaja Fe_2O_3 od 18,03 % i stopu uklanjanja od 31,15 %. U sljedećem koraku korištene su tehnike obrnute i izravne flotacije za uklanjanje silicijeva dioksida iz uzorka. Rezultati su pokazali da je najveća učinkovitost uklanjanja SiO_2 postignuta korištenjem procesa obrnute flotacije pri 750 g/t Armac C kolektora, 1200 g/t škroba, 40 g/t MIBC i 11 pH pulpe. U konačnici, HGMS praćen obrnutom flotacijom s Armac C kolektorom (750 g/t) primijenjen je na uzorku boksita i tada je sadržaj Fe_2O_3 bio 17,76 %, a sadržaj SiO_2 bio je 22,11 %, dok je učinkovitost uklanjanja silicijeva dioksida bila 45,22 % i omjer mase Al_2O_3/SiO_2 bio je 1,63. To pokazuje kako kombinacija HGMS-a i flotacije nije dovoljno učinkovita u redukciji željeznoga oksida i silicijeva dioksida te je potrebna daljnja predobrada i/ili daljnje istraživanje odgovarajuće alternativne metode. Dobiveni rezultat možda je uzrokovan time što su željezni oksid i minerali silicijeva dioksida bili fino zrnato raspršeni u uzorku boksita.

Ključne riječi:
niskokvalitetni boksit, silicij, učinkovitost uklanjanja, magnetska separacija, reverzna flotacija

Author’s contribution

Amirhossein Ashrafi (PhD student): conceptualization, data curation, formal analysis, investigation, methodology, software and writing – original draft. **Asghar Azizi** (Associate professor): conceptualization, methodology, formal analysis, project administration, supervision, validation, visualization, writing – review & editing. **Zahra Bahri** (Assistant professor): conceptualization, formal analysis, funding acquisition, visualization, writing – review & editing.
All authors have read and agreed to the published version of the manuscript.