An Experimental Study of New Depressants to Upgrade Fluorite Ore for Hydrofluoric Acid Production

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Abstract. Mongolia possesses substantial fluorspar (CaF_2) resources, with over 224 licensed deposits covering more than 61,000 hectares. The country holds strong potential to expand its export capacity by producing value-added products such as hydrofluoric acid (HF) and its downstream derivatives. This study explores the potential to process fluorite feed ore from one of Mongolia's major mining regions into a high-purity concentrate suitable for HF production. Laboratory-scale flotation tests were conducted to establish effective processing conditions. The sample was composed primarily of quartz, fluorite, and calcite, minerals with similar surface properties that pose challenges for selective separation. To improve flotation performance, the study introduced newly tested depressants, used on this ore type for the first time, to increase selectivity and enhance concentrate purity. The flotation process was optimized for grinding fineness, pH, and reagent combinations. The best results were obtained at 60% passing 74 µm, with flotation at pH 10 using fatty acid-based collectors and depressants dosed at 100-250 g/t. Open-circuit flotation produced a concentrate with 96.1% CaF_2 and 58% recovery. A closed-circuit configuration with multiple cleaning stages improved the final product to 96.03% CaF_2 and 68.8% recovery. XRD analysis confirmed a final CaF_2 purity of 97.2%, meeting the specifications for HF production. These results demonstrate that high-grade fluorite concentrate can be produced from Mongolian ore using optimized flotation conditions and new reagent strategies, supporting the country's ambitions to grow its chemical manufacturing capabilities through resource-based industrial development.

Keywords: Fluorite flotation, High-purity CaF_2, Hydrofluoric acid, Depressant optimization.

1 Introduction

Fluorite (CaF_2), one of the most important non-metallic minerals, is the primary raw material for industrial fluorine products and is widely used in industrial fields such as metallurgy, chemical engineering, optics and semiconductor manufacturing [1-3]. According to the U.S. Geological Survey [4], global fluorite reserves were estimated at 260 million tons in 2022, with over 73% concentrated in Mexico, China, South Africa, Mongolia, and Spain. Global fluorite production was reported at 8.3 million

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tons the same year [4], suggesting a static resource life of just over 30 years. With growing consumption driven by advanced technologies, the strategic importance of fluorite continues to rise [5].

Mongolia holds one of the largest fluorite reserves in Asia, contributing significantly to the country's mineral export economy. The nation is estimated to possess over 30 million tons of fluorite resources, with major deposits found in provinces such as Dornogovi, Tuv, and Sukhbaatar. Annual production in recent years has ranged from 400,000 to 600,000 tons [6], making Mongolia a key supplier in the Asia-Pacific market. Mongolian fluorite ores are typically associated with quartz and carbonate gangue minerals [6], and are processed mainly via flotation techniques [7-9].

However, the beneficiation of fluorite ores remains technically challenging due to the presence of calcium-bearing gangue minerals, especially calcite and dolomite, which exhibit similar physicochemical surface properties to fluorite, such as solubility, zeta potential, and hydrophobicity [8-10]. This similarity complicates selective flotation, often necessitating multi-stage flotation circuits and careful reagent selection to achieve high-grade concentrates suitable for hydrofluoric acid production.

Extensive research has therefore been conducted worldwide to improve fluorite beneficiation by developing selective collectors and depressants. In particular, the selective flotation of fluorite from calcite has attracted significant attention. Various collectors, such as organic acids, combined collectors, and functionalized surfactants, have been tested based on the difference in Ca surface activity and lattice match [5,10-12]. Likewise, both inorganic (e.g., sodium silicate, phosphates, sulfonated lignite) and organic (e.g., tannic acid, dextrin, polyaspartate) depressants have been applied to enhance the selectivity of flotation [9,10,13-15].

Despite decades of progress, achieving efficient separation of fluorite and calcite remains a persistent challenge. This study explores the use of new depressants for improving flotation selectivity in Mongolian fluorite ore and producing concentrates that meet the quality standards for hydrofluoric acid production.

2 Materials and Methods

2.1 Sample Collection and Preparation

A 50 kg sample of fluorite ore was obtained from a major fluorspar deposit in Mongolia. The bulk sample was crushed to below 2.0 mm using a laboratory-scale roll crusher. After homogenization using cone and quartering, representative sub-samples were prepared for chemical analysis and flotation experiments. All laboratory work was carried out under controlled conditions at the Mongolian-German Institute for Resources and Technology.

2.2 Chemical and Mineralogical Analysis

The chemical composition of the fluorite ore sample was determined using complexometric titration techniques standardized in Mongolia. Prior to analysis, representative subsamples were pulverized and digested using standard acid treatment protocols for multielement analysis.

Mineralogical analysis was conducted using X-ray diffraction (XRD) with a Rigaku Miniflex 300/600 diffractometer, equipped with a D/teX Ultra2 detector and a Cu-$K_{\alpha 1}$ radiation source ($\lambda = 1.5406$ Å). The X-ray generator operated at 40 kV and 15 mA. The acquired diffraction data were processed and interpreted using Match! 5.0 software, with reference to standard mineral databases for phase identification.

2.3 Particle Size Analysis

A representative ore sample from the deposit area was subjected to dry sieving using standard laboratory mesh sizes to separate it into the following fractions: +2 mm, −2+1 mm, −1+0.5 mm, −0.5+0.25 mm, −0.25+0.125 mm, −0.125+0.106 mm, −0.106+0.074 mm, and −0.074 mm. The weight percentage and CaF_2 content of each fraction were determined through chemical analysis.

2.4 Flotation Setup and Reagents

Flotation experiments were conducted to determine optimal processing parameters for upgrading the fluorite ore to high-grade concentrate. The study aimed to identify the most effective grinding conditions, optimize pulp pH, and evaluate suitable types and dosages of collectors and depressants for selective separation.

Optimization and selection of grinding conditions

Preliminary grinding tests were carried out using a SEPOR laboratory rod mill to determine the particle size required for effective liberation of fluorite from associated gangue minerals, as the degree of mineral liberation can vary depending on particle size. The experimental steps followed in this process are illustrated schematically in **Fig. 1.** Ore samples previously crushed to below 2.0 mm were divided into 1.0 kg test batches and ground for durations of 15, 20, and 25 minutes. The solid–liquid–rod (S:L:R) ratio was maintained at 1.0:0.65:9.00. After each grinding interval, the products were wet-sieved using a 0.074 mm screen, filtered, dried, and weighed to determine the percentage passing the target size.

To identify the most suitable grinding fineness for flotation, a series of flotation tests was conducted using ore samples ground to P80-74 μm, P70-74 μm, and P60-74 μm, respectively. All flotation experiments were carried out under identical conditions to isolate the effect of particle size on flotation performance. The flotation reagent scheme consisted of 400 g/t of TOFA collector, 900 g/t of sodium silicate, and 0.6 g/t of aluminum sulfate. The pulp pH was adjusted to 11 using sodium carbonate. Each flotation test was conducted with 2.2 liters of pulp, an impeller speed of 1200 rpm, and an air flow rate of 300 L/h. Froth was removed every 6 seconds over a total duration of 5 minutes. These standardized flotation conditions enabled a direct comparison of the flotation response across different grinding fineness levels, thereby allowing the optimal particle size for subsequent closed-circuit flotation tests to be selected.

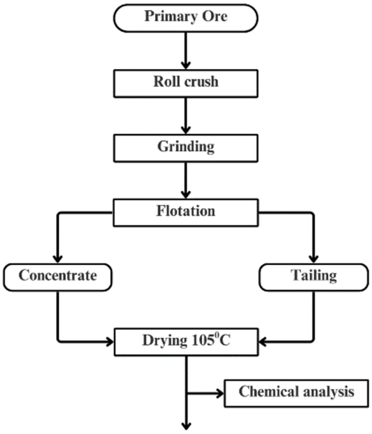

Fig. 1. Schematic illustration of the flotation experiment for optimizing grinding time.

pH and reagents optimization

Following the selection of optimal grinding fineness, a series of flotation experiments was carried out to optimize reagent combinations for improving fluorite recovery and concentrate purity. The flotation tests were performed using a Metso D12 laboratory flotation machine. Different collectors, including Berol, tall oil fatty acids (TOFA), and a mixture of the two, were applied to promote fluorite surface hydrophobicity. Methyl isobutyl carbinol (MIBC) was used as the frother. To improve selectivity against calcite, three depressants, Benefloat (a newly introduced reagent for this ore type), F100, and lignosulfonate, were tested. In all tests, a 5.0% aqueous solution of sodium silicate (Na_2SiO_3) and aluminum sulfate ($Al_2(SO_4)_3$) was also added to strengthen the depressing effect. The depressants were evaluated individually and at varying dosages to assess their influence on flotation performance. The influence of pH on flotation efficiency was also investigated by adjusting the pulp conditions with lime. Tests were conducted across a pH range of 6 to 11, with pH 10 found to provide the best balance between recovery and selectivity. Both open-circuit and closed-circuit flotation flowsheets were tested, including roughing, scavenging, and multiple cleaning stages. All flotation products, concentrates, tailings, and intermediate streams, were collected, dried, weighed, and analyzed for CaF_2 content to assess process performance under different reagent regimes.

Flotation circuit configuration

To evaluate the best process configuration for upgrading fluorite concentrate, both open- and closed-circuit flotation tests were conducted under the previously optimized conditions. In the open-circuit flotation test, a 1.0 kg sample was used for each test. The primary flotation was followed by four-stage cleaning and a single scavenger stage. The aim was to reduce fluorite losses in tailings and improve concentrate purity. Reagent additions and flotation parameters were consistent with the common conditions described above. A schematic flowsheet is presented in Fig. 2.

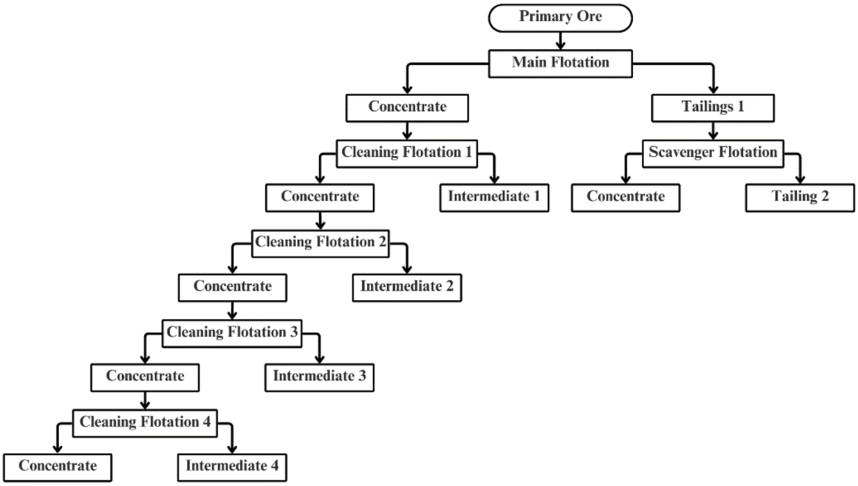

Fig. 2. Schematic flowsheet of the open-circuit flotation test setup.

In the closed-circuit flotation test, two separate 1.0 kg batches of fluorite rougher concentrate were processed using a complete recycling flowsheet based on the conditions established in the open-circuit tests detailed in Table 1. The flotation sequence included a rougher stage, four-stage cleaning, and a scavenger stage, with all intermediate streams (cleaner tailings and scavenger concentrate) returned to their respective upstream units to simulate an industrial continuous circuit as shown in Fig. 3. Reagent dosages applied at each stage of the closed-circuit process are summarized in Table 1.

Table 1. Reagent dosages for closed-circuit flotation stages.

Stage	Reagent dosage, g/t						
	TOFA	Berol	Benefloat	Na_2CO_3	Na_2SiO_3	$Al_2(SO_4)_3$	MIBC
Total (all stage)	1100	1100	650	2700	1900	1500	300
Rougher flotation	400	400	250	2700	900	600	50
1st cleaning	200	200	100	-	400	300	50
2nd cleaning	100	100	100	-	200	200	50
3rd cleaning	100	100	100	-	200	200	50
4th cleaning	100	100	100	-	200	200	50
Scavenger flotation	200	200	50	-	-	-	-

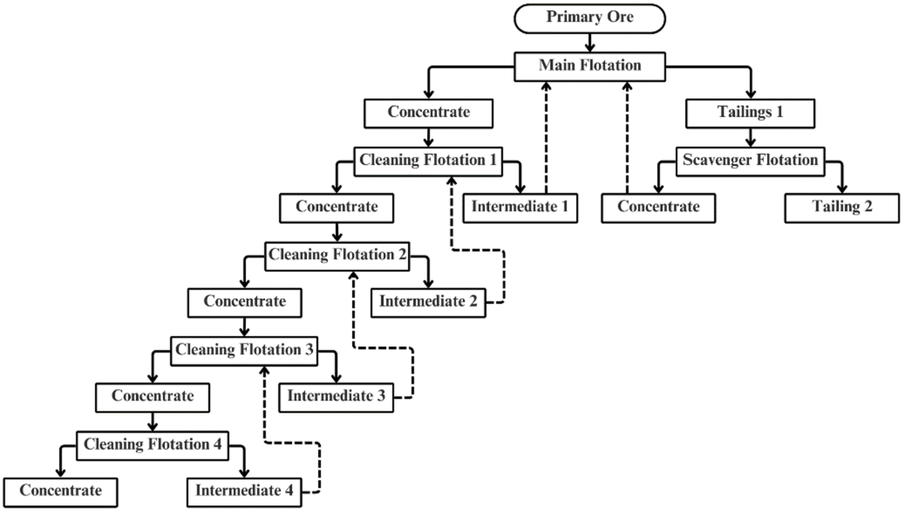

Fig. 3. Flotation flowsheet for cleaning impure fluorite concentrate.

3 Results and Discussion

3.1 Chemical and Mineralogical Composition

The fluorite content of the bulk ore sample was measured in the range of 35% to 41%, indicating the potential for beneficiation. Chemical analysis of the run-of-mine sample revealed a composition of 34.65% CaF_2, 4.21% $CaCO_3$, and 56.2% SiO_2, with negligible sulfur and phosphorus contents (both below 0.05%). XRD analysis supported the chemical findings and identified the principal mineral phases as quartz (SiO_2 – 52.3%), fluorite (CaF_2 – 39.0%), and calcite ($CaCO_3$ – 8.7%). The XRD pattern, included in Fig. 4, displayed distinct fluorite peaks at 2θ values of 28.3°, 47.0°, and 55.1°, and calcite peaks at 29.4°, 39.4°, and 46.1°. Minor quartz reflections were also present. The high abundance of quartz confirms it as the dominant gangue mineral, while the calcite content, although lower, requires selective depression due to its similar surface chemistry to fluorite. These results clearly demonstrate the critical role of reagent selection and process control in producing high-grade fluorite concentrates by flotation.

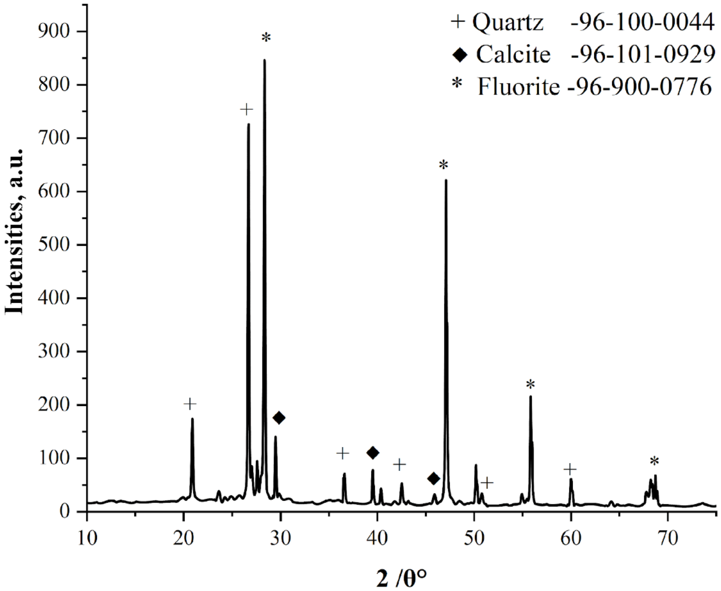

Fig. 4. XRD spectrum of the ore sample.

3.2 Particle Size Distribution

Sieve analysis was conducted to assess the distribution of CaF_2 and $CaCO_3$ across different size fractions. The results, presented in Table 2, show that CaF_2 content remains relatively consistent as particle size decreases, with slightly higher concentrations observed in finer fractions below 0.5 mm. In contrast, $CaCO_3$ content varies more significantly between fractions. These trends suggest that finer grinding promotes better liberation of fluorite from gangue minerals, particularly calcite, thereby enhancing its potential recovery during flotation. The insights gained from this distribution were crucial for selecting optimal grinding fineness in subsequent flotation optimization tests.

Table 2. Sieve analysis result.

№	Size fraction, mm	Content, %	
		$CaCO_3$	CaF_2
1	+2	3.8	28.4
2	–2+1	6.0	36.9
3	–1+0.5	2.5	49.8
4	–0.5+0.25	3.7	51.4

5	–0.25+0.125	4.0	45.1
6	–0.125+0.106	4.4	43.2
7	–0.106+0.074	2.1	43.6
8	–0.074+0	6.1	47.5

3.3 Optimization of Flotation Test Parameters

Grinding fineness optimization

To establish the grinding time required to achieve specific particle size distributions, a series of preliminary grinding tests was conducted using a laboratory rod mill. Ore samples were ground for 900, 1200, and 1500 seconds, corresponding to cumulative passing percentages of 60%, 70%, and 80% through a 74 µm screen, respectively. The particle size distributions for each grinding interval are shown in Table 3 and Fig. 5.

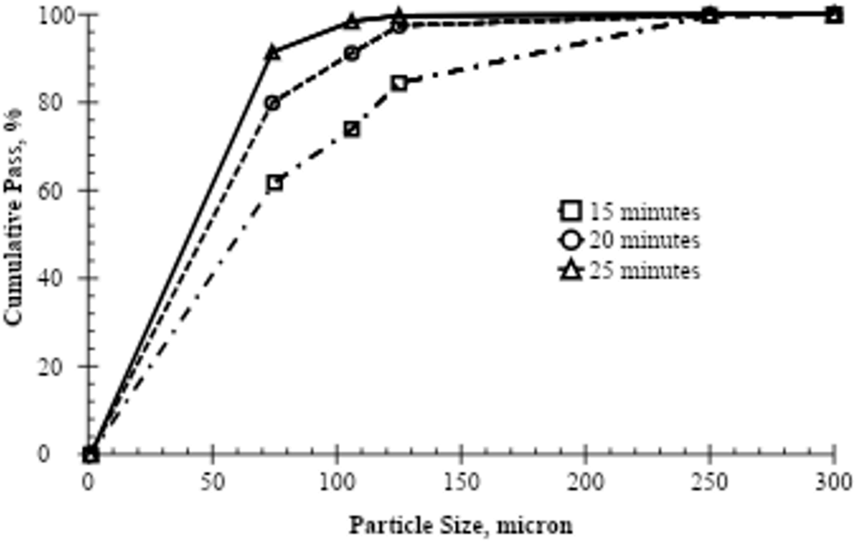

Fig. 5. Particle size distribution curves of ground ore products at different grinding times.

Using these optimized grinding times, flotation tests were subsequently conducted on samples representing P60–74 µm, P70–74 µm, and P80–74 µm size distributions. All flotation tests were carried out under identical reagent conditions to evaluate the impact of grinding fineness on flotation performance.

Table 3. Result of grinding condition test for determining optimal particle size distribution.

Grinding time, s	Cum. Pass. through, g					Cum. pass. through 0.074 mm, %	Grinding time, s
	+0.5 mm	+0.25 mm	+0.125 mm	+0.075 mm	–0.075 mm		
1500	0.0	0.2	2.6	83.2	914.0	80	1245.4
1200	0.0	0.6	25.1	175.0	799.3	70	1045.7
900	0.0	2.2	154.2	225.1	618.5	60	846

The flotation test results, shown in Table 4, demonstrate the influence of grinding fineness on both fluorite recovery and concentrate grade. At the finest grind with P80–74 μm, the concentrate achieved the highest CaF_2 grade of 89.5%, but the fluorite recovery was moderate at 62.3%. P70–74 μm resulted in a lower grade of 71.2% and 60.0% recovery, indicating suboptimal liberation or increased entrainment of gangue minerals. Interestingly, the coarsest grind, P60–74 μm, yielded the highest recovery of 76.3% with an acceptable concentrate grade of 78.6% CaF_2. These results suggest that, although finer grinding improves product purity, it may reduce recovery due to excessive slime generation and poorer froth handling. Based on these results, P60–74 μm was selected as the optimal grinding fineness for subsequent flotation experiments.

Table 4. Flotation results for optimizing grinding conditions.

Particle size distribution	Fraction	Yield, %	CaF_2 content, %	Fluorite recovery, %
P80–74 μm	Concentrate	28.55	89.5	62.3
	Tailings	71.45	21.8	37.7
	Feed ore	100.00	41.1	100.0
P70–74 μm	Concentrate	34.58	71.2	60.0
	Tailings	65.42	22.9	40.0
	Feed ore	100.00	39.6	100.0
P60–74 μm	Concentrate	39.81	78.6	76.3
	Tailings	60.19	14.4	23.7
	Feed ore	100.00	40.0	100.0

pH optimization

To systematically improve flotation performance, optimization experiments were conducted in a stepwise manner. Each key parameter, pulp pH, collector type, and depressant type and dosage, was varied independently, with the best-performing condition adopted for the next phase.

To determine the most favorable pulp condition for selective flotation of fluorite, experiments were conducted across four pH levels: 6, 8, 10, and 11. The performance at each pH was evaluated in terms of concentrate grade, yield, and fluorite recovery as detailed in Table 5.

Table 5. Flotation results for optimizing pulp pH, with P60–74 µm.

pH of pulp	Fraction	Yield, %	CaF_2 content, %	Fluorite recovery, %
11	Concentrate	28.55	89.5	62.3
	Tailings	71.45	21.8	37.7
	Feed ore	100.00	41.1	100.0
10	Concentrate	37.20	82.9	75.2
	Tailings	62.80	14.0	24.8
	Feed ore	100.00	39.6	100.0
8	Concentrate	31.23	73.1	55.7
	Tailings	68.77	19.1	44.3
	Feed ore	100	35.9	100
6	Concentrate	11.83	64.6	18.6
	Tailings	88.17	38.1	81.4
	Feed ore	100.00	41.3	100.0

As shown in this table, fluorite recovery improved markedly with increasing pH, peaking at pH 10 with a recovery of 75.2%. This level offered the best balance between selectivity and recovery, outperforming both lower and higher pH conditions. At pH 6, the recovery dropped drastically to 18.6%, likely due to insufficient surface charge differences between fluorite and calcite, limiting selective separation. Although flotation at pH 11 also produced a relatively high-grade concentrate (89.5% CaF_2), the associated recovery (62.3%) was still inferior to that at pH 10. Based on these results, pH 10 was selected as the optimal condition for subsequent reagent optimization tests.

Collector optimization

Following pH optimization, a series of tests was conducted to determine the optimal collector type and dosage for enhancing fluorite recovery. Collector dosage and type had a significant influence on both fluorite recovery and concentrate quality as detailed in Table 6.

Table 6. Comparative flotation results using different collector reagents at pH 10 and P60–74 µm particle size.

Collector	Fraction	Yield, %	CaF_2 content, %	Fluorite recovery, %
TOFA 1000g/t	Concentrate	50.28	58.7	72.0
	Tailings	49.72	26.8	28.0
	Feed ore	100.00	42.9	100.0
TOFA 800g/t	Concentrate	44.59	69.2	75.3
	Tailings	55.41	21.8	24.7
	Feed ore	100.00	42.9	100.0

TOFA 400g/t	Concentrate	28.55	89.5	62.3
	Tailings	71.45	21.8	37.7
	Feed ore	100	41.1	100
TOFA 400g/t + Berol 400 g/t	Concentrate	42.58	**73.5**	**76.4**
	Tailings	57.42	18.7	23.6
	Feed ore	100.00	42.0	100.0

As shown in Table 6, increasing the TOFA dosage from 400 g/t to 1000 g/t progressively improved recovery but reduced the CaF_2 grade of the concentrate. At 1000 g/t TOFA, the recovery reached 72.0%, but the concentrate grade dropped to 58.7%. The best concentrate quality was achieved using 400 g/t TOFA, which yielded a CaF_2 content of 89.5%, though with a moderate recovery of 62.3%. A mixed collector system combining 400 g/t TOFA with 400 g/t Berol provided a balanced result, delivering a recovery of 76.4% and a concentrate grade of 73.5%. These results indicate that while higher collector dosages enhance recovery, they tend to compromise selectivity. Therefore, the mixed collector strategy offered the most favorable compromise between recovery and grade under the tested conditions (pH 10, P60–74 µm).

Depressant type and dosage optimization

The effect of different depressants on calcite suppression was evaluated at pH 10 and a grind size of P60–74 µm, as summarized in Table 7. Among the tested reagents, Benefloat showed the best performance in reducing calcite content in the concentrate. While lignosulfonate and F100 achieved moderate calcite depression, their effectiveness was comparatively lower, as indicated by higher $CaCO_3$ levels in the concentrate.

Table 7. Flotation results using various depressants for P60-74 µm particles at pH 10.

Collector	Fraction	$CaCO_3$ content, %
F100 1000g/t	Concentrate	4.2
	Tailings	4.2
	Feed ore	4.2
Lignosulfonate 1000g/t	Concentrate	3.4
	Tailings	4.0
	Feed ore	3.7
Benefloat/t	Concentrate	2.0
	Tailings	4.4
	Feed ore	4.0

Subsequent tests using Benefloat at varying dosages (100, 250, and 1000 g/t) demonstrated that dosage had a marked effect on both concentrate grade and fluorite recovery (Table 8). At 1000 g/t, the concentrate had the highest CaF_2 purity with 89.9%, but with a significantly lower recovery of 33.5%. In contrast, a dosage of 100 g/t yielded the highest recovery of 78.8% with a still acceptable concentrate grade of

76.4% CaF_2. A dosage of 250 g/t provided a moderate balance but did not outperform the 100 g/t condition. These findings indicate that lower Benefloat dosages of 100 – 250 g/t are more effective for achieving a good compromise between selectivity and recovery, making it the optimal choice under the tested conditions.

Table 8. Results of optimized consumption of Benefloat calcite depressant reagent at pH10 abd particle size P60-74 μm.

Benefloat dosage, g/t	Fraction	Yield, %	CaF_2 content, %	Fluorite recovery, %
1000	Concentrate	15.28	89.9	33.5
	Tailings	84.72	35.0	66.5
	Feed ore	100.00	43.4	100.0
250	Concentrate	40.82	74.1	73.7
	Tailings	59.18	17.1	26.3
	Feed ore	100.00	40.4	100.0
100	Concentrate	42.26	76.4	78.8
	Tailings	57.74	21.8	21.2
	Feed ore	100.00	44.9	100.0

Comparative evaluation of open- and closed-circuit flotation

Flotation tests were performed using both open- and closed-circuit configurations under the same optimized conditions detailed in Table 9.

Table 9. Operating conditions applied in both open- and closed-circuit flotation tests.

Parameter	Value / Setting
Ore particle size	P60–74 μm
Pulp pH	10 (adjusted with Na_2CO_3)
Flotation volume	2.2 L (rougher & scavenger), 1.1 L (cleaner)
Impeller speed	1200 rpm (rougher & scavenger), 800 rpm (cleaner)
Air flow rate	300 L/h (rougher & scavenger), 200 L/h (cleaner)
Froth removal interval	6 s (rougher & scavenger), 10 s (cleaner)
Collectors	TOFA 400 g/t + Berol 400 g/t
Depressants	Benefloat 250 g/t, Na_2SiO_3 900 g/t, $Al_2(SO_4)_3$ 0.6 g/t
Frother	MIBC 50 g/t

The open-circuit test produced a fluorite concentrate with a CaF_2 grade of 96.1%, yield of 22.3%, and fluorite recovery of 58%, as detailed in Table 10. This was achieved through a flotation sequence comprising a rougher, one scavenger, and four cleaner stages. While the concentrate quality was high, some recovery losses occurred due to intermediate stream discard.

Table 10. Results of open-circuit flotation test using four-stage cleaning and one-stage scavenging under optimized conditions

Test stage	Initial mass, g	Initial grade CaF_2, %	Conc. Mass, g	Conc. Grade CaF_2, %	Tailings mass, g	Tailings grade CaF_2, %	Yield, %	Recovery, %
Rougher	1000.0	36.6	432.0	76.2	568.0	6.5	43.2	90
Scavenger	568.0	6.5	26.0	31.3	542.0	5.3	2.6	2
Cleaner 1	432.0	76.2	363.0	88.1	69.0	13.6	36.3	87
Cleaner 2	363.0	88.1	312.1	93.1	50.9	57.4	31.2	79
Cleaner 3	312.0	93.1	271.5	95.3	41.0	78.6	27.2	71
Cleaner 4	272.0	95.3	222.7	96.1	49.0	91.5	22.3	58

In comparison, the closed-circuit flotation test, which reprocessed all intermediate streams to simulate industrial continuous operation, resulted in a concentrate yield of 25.4%**,** a CaF_2 grade of 96% (more precisely 96.03%), and a significantly higher recovery of 68.8%**,** as summarized in Table 11. This demonstrates that integrating intermediate products back into the circuit enhances recovery without compromising product quality. The comparative results confirm that the closed-circuit configuration is more effective for maximizing fluorite recovery while maintaining HF-grade concentrate purity.

Table 11. Results of closed-circuit flotation test with full recycle of intermediate products under the same optimized conditions

Product name	Mass, g	Yield, %	CaF_2, %	$CaCO_3$, %	CaF_2 recovery, %
Concentrate 1	226.3	11.3	92.7	4.8	30.0
Concentrate 2	282.0	14.1	96.0	1.9	38.8
Total concentrate	508.3	25.4	94.5	3.2	68.8
Scavenger product	50.5	2.5	35.1	7.2	2.5
Intermediate 1	90.5	4.5	14.1	4.5	1.8
Intermediate 2	84.5	4.2	37.8	3.8	4.6
Intermediate 3	98.5	4.9	70.7	13.1	10.0
Intermediate 4	71.9	3.6	86.9	7.6	8.9
Total intermediate	345.4	17.3	51.2	7.4	25.3
Tailings 1	530.1	26.5	3.0	2.5	2.3
Tailings 2	565.7	28.3	1.4	2.2	1.1
Total tailings	1095.8	54.8	2.1	2.3	3.4
Feed ore	2000.0	100.0	34.9	3.6	100.0

3.4 XRD Characterization of Final Products

X-ray diffraction analysis was performed on the final flotation concentrate and tailings to evaluate the mineralogical composition and assess the effectiveness of the separation process. The XRD results of the concentrate sample confirmed a high purity of fluorite, with a composition of approximately 97.2% CaF_2 and minor calcite content (2.8% $CaCO_3$) (Fig. 6). The distinct fluorite peaks and minimal secondary mineral presence suggest that the produced concentrate is of sufficient quality for industrial applications, particularly in metallurgy and chemical production.

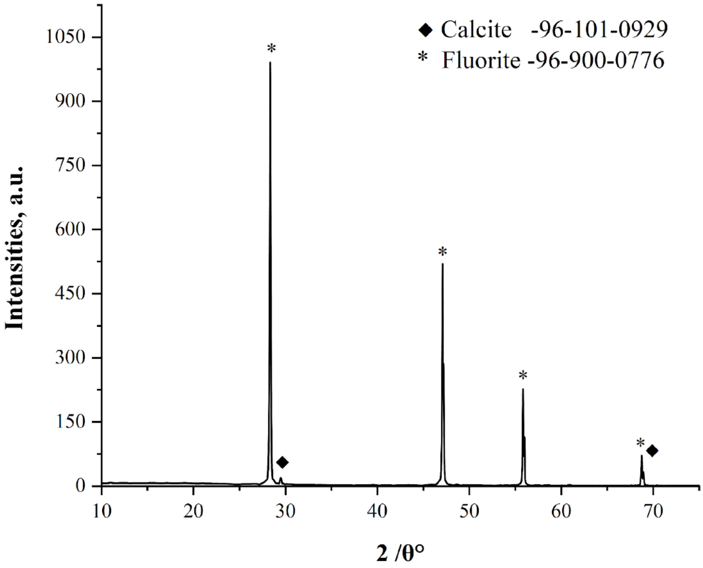

Fig. 6. XRD analysis of the final fluorite concentrate, indicating high CaF_2 purity with minor calcite presence.

In contrast, the XRD analysis of the tailings sample revealed that it predominantly consists of quartz (SiO_2 – 96.9%), with only trace amounts of fluorite (2.0% CaF_2) and calcite (1.1% $CaCO_3$). This indicates that the flotation process successfully separated the target minerals from the gangue, leaving behind mostly inert quartz in the tailings. The minimal presence of residual fluorite and calcite reflects the high selectivity and efficiency of the beneficiation process.

4 Conclusions

A comprehensive technological study was conducted on a fluorite ore sample to investigate its mineralogical composition, chemical characteristics, and flotation behavior. X-ray diffraction analysis showed that the ore contained approximately 39.3% calcium fluoride, 52.4% quartz, and 8.3% carbonate minerals. Chemical analysis supported these findings, revealing CaF_2 at 34.65%, SiO_2 at 56.2%, and $CaCO_3$ at 4.21%, with negligible levels of phosphorus and sulfur. A variety of flotation parameters, including pH, collectors, depressants, and their dosages, were systematically optimized. Closed-circuit flotation tests incorporating four cleaning stages and one scavenger stage yielded a fluorite concentrate with a CaF_2 grade of 96.03%, a recovery of 68.8%, and a yield of 25.4%. Among the depressants evaluated for improving calcite rejection, Benefloat demonstrated the most effective performance at dosages between 100–250 g/t, contributing to enhanced selectivity and concentrate quality. These findings confirm that, with proper reagent management and process design, high-quality fluorite concentrate can be efficiently obtained for use in chemical and metallurgical industries.

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