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Optimization Of Flotation Parameters for Copper Recovery from Low-Grade Copper Ore: A Case Study in Pakistan

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Abstract

Low-grade copper–gold porphyry ores present significant processing challenges due to poor mineral dissemination and high gangue content. In this study, a representative ore sample from the Saindak Copper–Gold Project, Pakistan, with an average copper grade of approximately 0.56 wt%, was investigated to optimize flotation parameters for improved copper recovery. Mineralogical characterization using XRD, XRF, SEM–EDS, and ICP-OES confirmed chalcopyrite as the primary copper-bearing mineral, finely disseminated within silicate gangue and iron sulfides. A series of batch flotation experiments were conducted by systematically varying particle size, slurry pH, collector dosage, and flotation time. The results showed that optimal conditions (particle size <75 µm, pH 12, and 3-minute flotation time) enabled a rougher copper recovery of approximately 97% at a collector dosage of only 5 g/t, with the final concentrate grading 23 wt% Cu after cleaning. Flotation kinetics analysis indicated that the first-order kinetic model provided the best fit to the experimental data. The findings demonstrate that efficient copper upgrading from low-grade Saindak ore can be achieved with minimal collector consumption, offering a potentially cost-effective and sustainable processing strategy for similar porphyry copper deposits.

1. Introduction

Copper is one of the most important industrial metals due to its excellent electrical and thermal conductivity, corrosion resistance, and recyclability [1]. It plays a critical role in modern infrastructure, renewable energy technologies, and electric vehicles [2], [3]. With the continuous growth in global copper demand and the gradual depletion of high-grade ore deposits, the mining industry is increasingly compelled to exploit low-grade and complex copper ores [4], [5]. This shift has led to higher energy consumption, increased processing costs, and greater environmental challenges, making the development of efficient and sustainable beneficiation strategies essential [2], [6], [7].

Porphyry copper deposits constitute the largest source of global copper production [8]. These deposits are typically characterized by low metal grades, fine-grained mineralization, and complex gangue associations. Chalcopyrite ($CuFeS_2$) is the most abundant copper-bearing mineral in porphyry systems and is commonly associated with iron sulfides such as pyrite and magnetite, as well as silicate gangue minerals [2], [9], [10]. Due to the fine dissemination of valuable minerals and incomplete liberation at coarse particle sizes, beneficiation of porphyry copper ores remains technically challenging [11], [12]. Froth flotation is the most widely used technique for concentrating chalcopyrite from such ores; however, flotation performance is highly sensitive to operating conditions and reagent regimes [13], [14].

The efficiency of chalcopyrite flotation is governed by several interrelated parameters,

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including particle size, slurry pH, collector dosage, frother type and concentration, and flotation time [15], [16], [17], [18]. Particle size influences mineral liberation, collision probability, and attachment efficiency between mineral particles and air bubbles [19]. Excessively coarse particles suffer from poor liberation, while ultrafine particles may lead to slime coatings and reduced flotation selectivity. Slurry pH plays a crucial role in controlling mineral surface chemistry, reagent adsorption, and the selective depression of gangue and iron sulfide minerals. Similarly, collector type and dosage significantly affect hydrophobicity development on chalcopyrite surfaces, while flotation time determines the extent of mineral recovery and concentrate grade through flotation kinetics [20].

Extensive research has been conducted on chalcopyrite flotation using xanthate-based collectors under controlled laboratory conditions [21], [22], [23]. Most studies report improved copper recovery with increasing collector dosage; however, excessive collector addition often results in reduced selectivity and increased gangue recovery. Moreover, recent studies have indicated that chalcopyrite may exhibit a degree of natural hydrophobicity under alkaline conditions, particularly in the presence of suitable frother, raising the possibility of reduced or even collector-free flotation under optimized conditions [24], [25], [26], [27]. Despite these findings, the mechanisms and practical applicability of such flotation behavior remain insufficiently explored, especially for specific low-grade porphyry copper ores.

In Pakistan, the Saindak Copper–Gold Project represents a strategically important porphyry copper deposit with significant mineral resources but relatively low copper grades [28]. Efficient utilization of this resource requires the development of an optimized beneficiation process tailored to the mineralogical and physicochemical characteristics of the Saindak ore. To date, limited published work has addressed the flotation behavior of this ore, and systematic studies on the combined effects of particle size, pH, collector dosage, and flotation time are lacking. In particular, the potential for achieving high copper recovery with reduced reagent consumption has not been adequately investigated.

The present study aims to address these gaps by conducting a comprehensive flotation optimization study on low-grade Saindak copper ore. Detailed mineralogical and chemical characterization was first carried out using XRD, XRF, SEM–EDS, and ICP-OES techniques to identify the copper-bearing phases and gangue mineral associations. Batch flotation experiments were then performed to evaluate the effects of particle size, slurry pH, potassium amyl xanthate (PAX) dosage, and flotation time on copper grade and recovery. In addition, flotation kinetics were analyzed using different kinetic models to better understand the recovery behavior under optimized conditions. The outcomes of this study are expected to provide practical insights into reagent optimization and process design for low-grade porphyry copper ores, with potential implications for reducing operating costs and improving the sustainability of copper beneficiation operations.

Beyond parameter optimization, this study addresses a chalcopyrite–pyrite separation regime that is under-represented in the literature. The Saindak ore contains approximately 9.3 wt% Fe against only 0.56 wt% Cu, with a pyrite-to-chalcopyrite mass ratio of roughly 12:1, whereas most selectivity studies use synthetic mixtures or ores with far lower pyrite contents. Within this regime, we show that copper recovery is essentially independent of collector dosage between 5 and 20 g/t at pH 12, indicating near-collectorless flotation, and we exploit this along with a short-residence rougher flotation and concentrate regrind to achieve high-grade copper concentrates with minimal reagent consumption.

2. Materials and Methods

2.1. Materials and reagents

2.1.1. Materials

A representative bulk sample of low-grade copper ore was collected from the Saindak Copper–Gold Project, Pakistan. The sample was crushed, homogenized, and riffle-split to obtain representative subsamples for characterization and flotation testing. Chemical analysis was conducted using XRF and ICP-OES to determine the elemental composition, while mineralogical analysis was performed using XRD and SEM–EDS to identify copper-bearing phases and gangue mineral associations. The chemical composition of the copper ore sample is given in Table 1.

2.1.2. Reagents

All reagents used were of chemical grade and obtained from Wako Chemicals Japan. PAX (Potassium amyl xanthate, C5H11OCSSK) and MIBC (Methyl Isobutyl Carbinol, C6H14O) were used as the standard collector and frother, respectively.

Table 1. Chemical composition of the Saindak ore sample (wt%)

Mg	Al	SiO_2	Zn	Mn	Fe	Cu
1.50	12.30	68.20	0.10	0.05	9.30	0.56

2.2. Methods

2.2.1. Sample preparation

The ore sample was ground in a laboratory ball mill to produce different particle size fractions (+150, −150+75, −75+38, and −38 µm). Grinding time was adjusted to achieve the target particle sizes, and size classification was performed using standard sieve analysis.

2.2.2. Flotation

Batch flotation experiments were conducted in a 500 mL mineral separation (MS) machine comprising two sections (mixing and frothing sections) and a tube connected to a generator for air supply. A 50 g sample was charged into the cell with distilled water to achieve a pulp density of 10%. The slurry was tested at its natural pH of 8.5 or adjusted to the target pH using sodium hydroxide (NaOH) or sulfuric acid (H_2SO_4). The collector used was potassium amyl xanthate (PAX, $C_5H_{11}OCSSK$), while the frother was methyl isobutyl carbinol (MIBC, $C_6H_{14}O$). In all flotation tests, MIBC addition was fixed at 400 g/t. The air flow rate and impeller speed were maintained constant across all tests. All flotation experiments were performed in duplicate on separate representative subsamples obtained by riffle splitting. Reported values are means, and the error bars and ± values represent the spread of the replicates. Head grades were verified by comparing the direct assay with the value back-calculated from the concentrate and tailings; agreement was within acceptable limits for all tests. The overall experimental procedure, from sampling through characterization and the two-stage flotation circuit, is summarized in Figure 1.

3. Results and Discussion

3.1. Effect of particle size on chalcopyrite flotation

The results indicate a clear increase in copper recovery with decreasing particle size. Coarse particles (+150 µm) exhibited comparatively lower copper recovery, which can be attributed to incomplete liberation of chalcopyrite from the gangue matrix. In this size range, chalcopyrite remains partially locked within silicate and iron-bearing minerals, limiting effective interaction between the mineral surface and the collector. In contrast, finer particle size fractions showed significantly higher copper recovery for the −75 µm fraction. Improved flotation performance at finer sizes is primarily associated with enhanced liberation of chalcopyrite, which exposes fresh mineral surfaces and promotes effective collector adsorption, thereby increasing hydrophobicity and flotation efficiency. However, excessive grinding to ultrafine sizes may adversely affect selectivity due to slime coating and increased entrainment of gangue minerals. Beyond liberation, particle size also governs bubble–particle collision, detachment, and the entrainment of fine gangue. The non-monotonic concentrate grade in Table 2—the −150+75 µm fraction assaying higher in Cu than the −75+38 µm fraction—reflects this balance: recovery improves as size decreases through better liberation, while concentrate grade is eroded at the finest sizes by increased entrainment of fine silicate gangue.

The differences in mineral liberation between coarse and fine size fractions are further supported by SEM–EDS analyses, as shown in Figures 2 and 3, which clearly demonstrate improved chalcopyrite liberation at finer particle sizes.

Table 2. Effect of particle size on the flotation recovery of copper.

Sieve, Fraction µm	Concentrate			
	Grade, wt %		Recovery, %	
	Cu	Fe	Cu	Fe
+150	2.48 ± 0.40	28.61 ± 1.56	84.88 ± 2.78	45.62 ± 3.34
−150+75	5.72 ± 0.35	31.22 ± 1.04	91.68 ± 1.90	37.55 ± 3.19
−75+38	4.91 ± 0.06	20.32 ± 0.40	95.54 ± 1.87	61.68 ± 0.93
−38	5.39 ± 0.03	17.63 ± 0.45	96.90 ± 1.58	54.12 ± 0.62

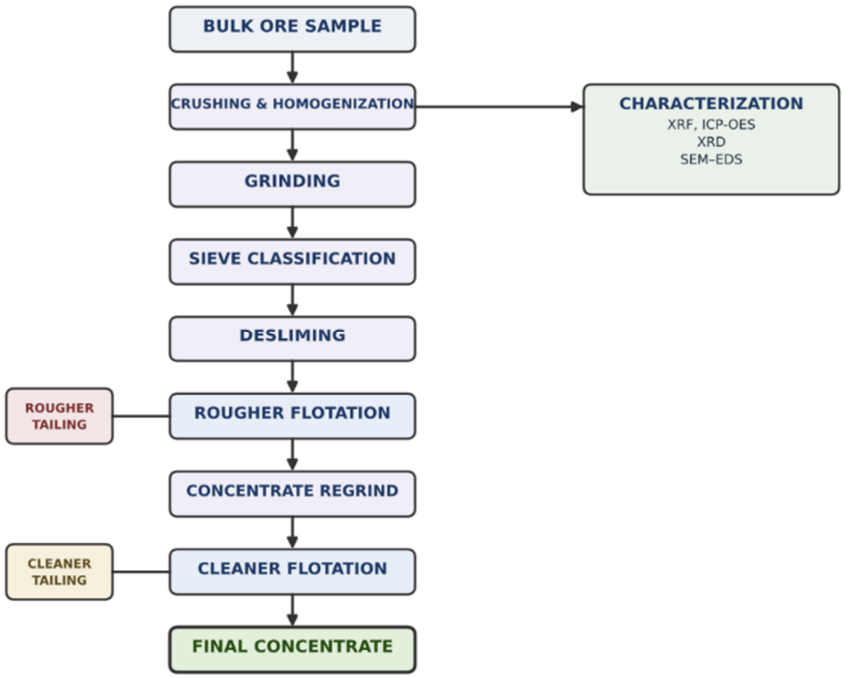

Figure 1. Experimental procedure for flotation of Saindak low-grade copper ore

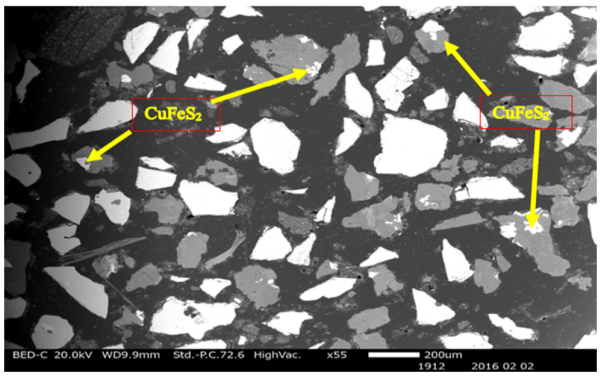

Figure 2. SEM-EDS analysis image for +150 µm particle size.

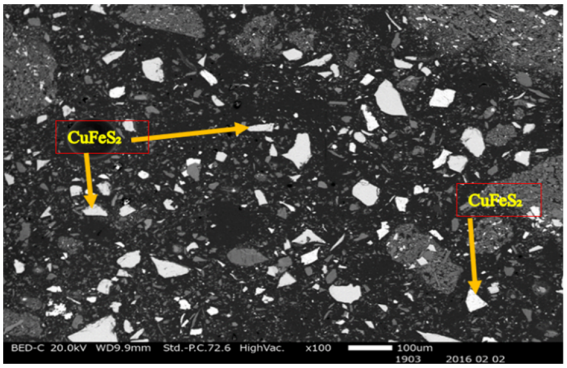

Figure 3. SEM-EDS analysis image for – 75 µm particle size.

The XRD patterns of concentrates corresponding to the four particle size fractions are presented in Figure 4. Distinct diffraction peaks associated with chalcopyrite are clearly observed in the finer particle size ranges, indicating improved liberation and concentration of the copper-bearing mineral. In contrast, chalcopyrite peaks are less pronounced in the coarser size fractions, likely due to incomplete liberation and peak overlap with gangue minerals.

Based on the combined flotation results and mineralogical evidence, particle sizes finer than 75 µm were selected for subsequent flotation experiments, as this size range provided improved chalcopyrite liberation and superior flotation performance.

3.2. Effect of Slurry pH on chalcopyrite flotation

After selecting the optimal particle size, the effect of slurry pH on chalcopyrite flotation was investigated by varying the pH from acidic to strongly alkaline conditions (pH 4–12). All other flotation parameters were kept constant, including particle size (−75 µm), pulp density (10%), potassium amyl xanthate (PAX) dosage (25 g/t), methyl isobutyl carbinol (MIBC) dosage (400 g/t), and flotation time (10 min). Sodium hydroxide (NaOH) and sulfuric acid (H_2SO_4) were used to adjust the slurry pH. The flotation results obtained under different pH conditions are presented in Figure 5.

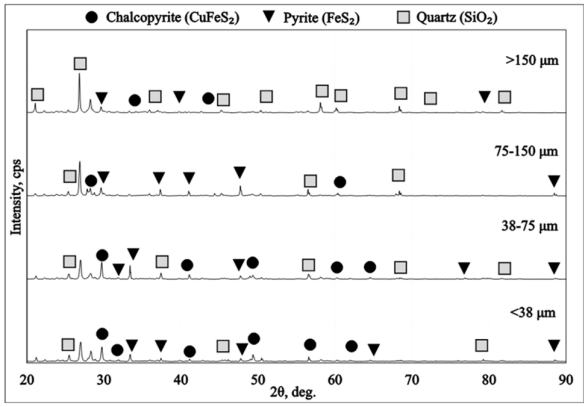

Figure 4. XRD pattern of the concentrate for the particle size ranges.

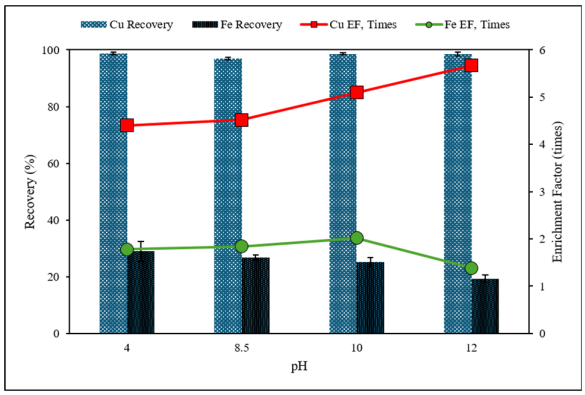

Figure 5. Effect of pH on the flotation recovery of copper.

The enrichment ratio, defined as the ratio of concentrate grade to feed grade, was used to evaluate the selectivity of the flotation process. As shown in Figure 5, copper recovery under acidic conditions (pH 4) was relatively high; however, the enrichment ratio was comparatively low, accompanied by a high recovery of iron. Since iron-bearing minerals act as impurities during copper concentration, such conditions are undesirable for selective chalcopyrite flotation.

With increasing pH, a significant improvement in flotation selectivity was observed. At pH 12, under the conditions of the pH test (25 g/t PAX, 10 min), copper recovery reached approximately 98%, while iron recovery decreased to its lowest level. This behavior can be attributed to the selective depression of iron sulfide minerals under alkaline conditions. This improved selectivity arises from several complementary mechanisms of iron-sulfide depression under alkaline conditions. First, xanthate confers hydrophobicity on pyrite mainly through its oxidation to dixanthogen, which requires the pulp potential to exceed the reversible potential of the xanthate/dixanthogen couple; at pH 12, this threshold is not met, so dixanthogen does not form and pyrite remains hydrophilic [29]. Second, hydroxyl ions promote hydrophilic iron oxyhydroxide films on the pyrite surface, evidenced by the FeOOH detected in the pH-12 tailing. Third, OH^- competes with xanthate for surface sites. Given the high pyrite content of this ore, galvanic interaction between pyrite and chalcopyrite further influences surface oxidation. Chalcopyrite escapes depression because its oxidation produces a copper-deficient, sulfur-rich surface layer that is itself hydrophobic [30]; this same self-induced hydrophobicity accounts for the near-collectorless flotation observed at 5 g/t PAX.

In addition to improved selectivity, flotation is generally preferred under alkaline conditions because most sulfide collectors, including xanthates, exhibit greater chemical stability at high pH, and equipment corrosion is minimized. It should be noted, however, that the laboratory optimum was established using NaOH, whereas industrial operation would rely on lime; Ca^{2+} species adsorb on pyrite and contribute to depression independently of hydroxyl concentration, so comparable selectivity may be attainable at a somewhat lower pH in a lime circuit, while high lime demand and gypsum scaling represent the principal operating-cost constraints. pH 12 is therefore reported here as a laboratory optimum requiring economic validation rather than a recommended plant setpoint.

Further mineralogical evidence supporting iron mineral depression at high pH is provided by the XRD pattern of tailings obtained at pH 12, shown in Figure 6. The presence of FeOOH phases confirms the formation of iron hydroxide species under strongly alkaline conditions, consistent with the observed suppression of iron flotation.

Based on the combined flotation performance, enrichment behavior, and mineralogical evidence, a slurry pH of 12 was selected as the optimal condition for subsequent flotation experiments.

3.3. Effect of PAX dosage on the flotation of chalcopyrite

A series of experiments were performed by varying the PAX dosage from 5 to 20 g/t to study its effect on chalcopyrite flotation. All other conditions were kept constant, including particle size (−75 µm), pulp density (10%), slurry pH (12), methyl isobutyl carbinol (MIBC) dosage (400 g/t), and flotation time (10 min). The flotation results at different collector dosages are presented in Figure 7.

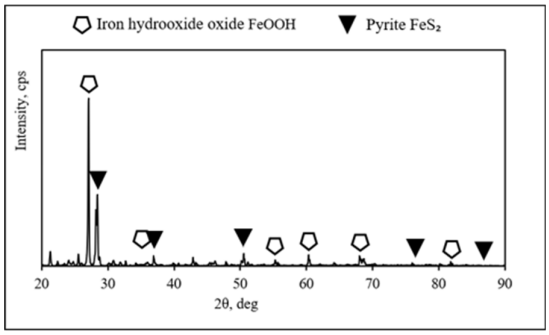

Figure 6. XRD pattern of pH 12 tail confirming the presence of FeOOH.

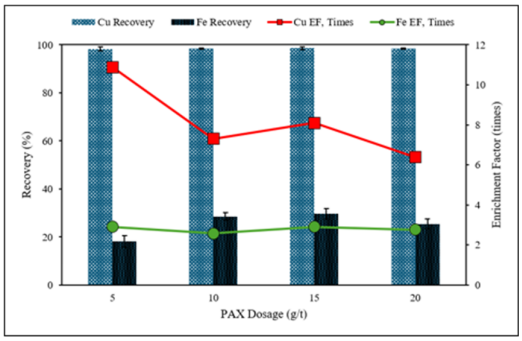

Figure 7. Effect of PAX dosage on the flotation recovery of copper.

As shown in Figure 7, copper recovery remained relatively constant across the investigated PAX dosage range, indicating that effective chalcopyrite flotation can be achieved even at low collector dosages under the selected alkaline conditions. However, a noticeable variation in the enrichment factor was observed with increasing PAX dosage. The highest enrichment factor was obtained at the lowest collector dosage of 5 g/t, while higher PAX additions resulted in a gradual decrease in the enrichment factor.

The reduction in the enrichment factor at elevated collector dosages can be attributed to the non-selective adsorption of xanthate on gangue minerals, which promotes their flotation alongside chalcopyrite. Excessive collector addition increases overall hydrophobicity in the system, leading to increased gangue entrainment and a corresponding decrease in copper selectivity.

The results suggest that the studied ore exhibits sufficient natural hydrophobicity under strongly alkaline conditions, allowing efficient chalcopyrite flotation at a relatively low collector dosage. Based on the combination of high copper recovery and maximum enrichment factor, a PAX dosage of 5 g/t was selected as the optimal collector concentration for subsequent flotation experiments. While the absolute reagent savings from operating at 5 g/t are modest, the primary benefits include improved concentrate grade—reducing downstream freight and smelter penalties—and decreased xanthate residue in tailings water. Operating near the minimum effective dosage, however, leaves little margin for ore variability; therefore, a modest safety factor would be applied in practice.

3.4. Effect of varying time of flotation on chalcopyrite flotation

Flotation time is a critical operating parameter that directly influences both mineral recovery and concentrate quality. To evaluate its effect on chalcopyrite flotation, batch flotation experiments were conducted by varying the flotation time from 1 to 10 minutes, while keeping all other parameters constant, including particle size (−75 µm), slurry pH (12), and PAX dosage (5 g/t). The copper recovery and enrichment factor determined by ICP-OES analysis under different flotation times are presented in Figure 8.

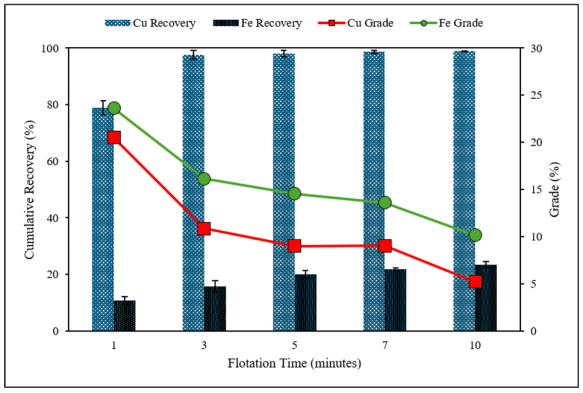

Figure 1. Effect of flotation time on the flotation recovery of copper.

As shown in Figure 8, both copper recovery and enrichment factor exhibited a strong dependence on flotation time. The highest copper enrichment factor was achieved within the first minute of flotation, corresponding to a copper recovery of approximately 79%. This initial stage represents the rapid flotation of well-liberated and highly hydrophobic chalcopyrite particles, which are preferentially recovered early in the process. With increasing flotation time, copper recovery continued to increase; however, this improvement was accompanied by a progressive decrease in enrichment factor and concentrate grade. This behavior can be attributed to the delayed flotation of less hydrophobic particles and the increasing entrainment of gangue minerals into the froth

phase. As illustrated schematically in Figure 9, chalcopyrite particles are predominantly recovered during the early stages of flotation, whereas prolonged flotation promotes the recovery of gangue minerals, thereby reducing concentrate selectivity.

These results indicate a trade-off between copper recovery and concentrate quality with increasing flotation time. Although extended flotation enhances overall recovery, it adversely affects selectivity. Therefore, shorter flotation times are preferred when higher concentrate grades are desired, while longer flotation times may be employed when maximizing recovery is the primary objective.

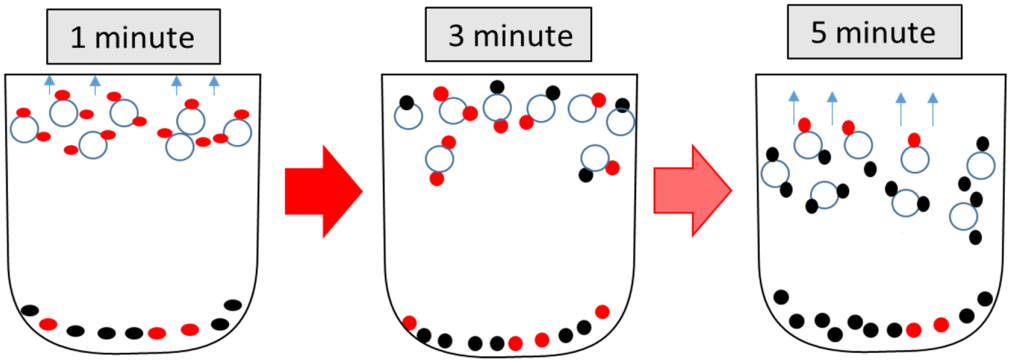

Figure 2. Illustration explaining the effect of time on chalcopyrite flotation.

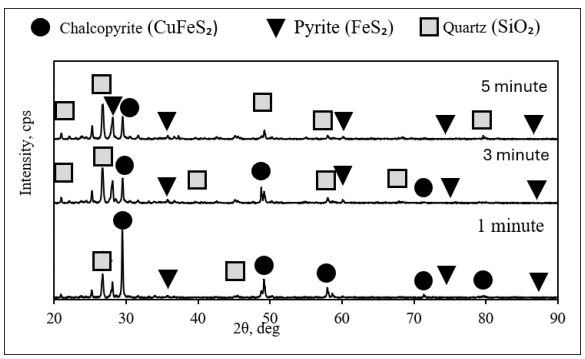

Figure 3. XRD pattern of froth for the effect of time on flotation.

As shown in Figure 10, chalcopyrite grains are clearly identifiable and more prominent in the concentrate obtained during the first minute of flotation. In contrast, with increasing flotation time, chalcopyrite becomes less distinguishable due to increased overlap with gangue minerals, predominantly silica. This observation supports the flotation results, indicating that chalcopyrite is preferentially recovered during the initial stages of flotation, whereas prolonged flotation leads to increased gangue entrainment in the concentrate.

Based on the above findings, a flotation time of 3 minutes was selected as optimal for achieving high concentrate selectivity as well as recovery in the rougher stage. However, despite improved enrichment at short flotation times, a portion of copper remained unrecovered in the tailings. To address this limitation and enhance overall copper recovery, multistage flotation experiments were subsequently conducted, incorporating a desliming step prior to flotation.

3.5. Flotation Recovery Model Fitting

The flotation recovery behavior of copper was evaluated using four commonly applied kinetic models, as summarized in Table 3. It should be noted that the fully mixed, modified gas–solid adsorption, and second-order models are mathematically equivalent, each reducing to the rectangular-hyperbola form $R = R\infty \cdot t/(\tau + t)$ under reparameterization of the rate constant; they are therefore reported here as a single model and yielded identical fits ($R\infty$ = 104.7%, R^2 = 0.998, RMSE = 1.71).

The classical first-order model (Table 4) gave the best description of the data, with the highest coefficient of determination (R^2 = 0.9999) and the lowest error (RMSE = 0.27), and — importantly — a physically admissible ultimate recovery ($R\infty$ = 99.4%). In contrast, the hyperbolic and rectangular-distribution forms returned $R\infty$ values of 104–107%, which exceed the theoretical maximum of 100% and are therefore physically meaningless. This over-prediction is a known limitation of fitting two-parameter asymptotic models to data collected over a finite time window in which recovery has not fully plateaued: the hyperbolic forms approach their asymptote more slowly than the exponential and inflate $R\infty$ to accommodate the steep early rise in recovery. The classical first-order model avoids this because its exponential form reaches its asymptote quickly, consistent with the fast flotation observed here.

Table 1. Flotation Kinetic Models Applied [31]

No.	Model	Formula
1	Classical first-order model	$R = R_{\infty,1}(1 - e^{-K_1 t})$
2	First-order with a rectangular distribution	$R = R_{\infty,2}[1 - \frac{1 - e^{-K_2 t}}{K_2 t}]$
3	Fully mixed factor model	$R = R_{\infty,3}[1 - \frac{1}{1 + t/K_3}]$
	Improved gas/solid adsorption model	$R = \frac{R_{\infty,4} K_4 t}{1 + K_4 t}$
	Second-order model	$R = \frac{R_{\infty,5}^2 K_5 t}{1 + R_{\infty,5} K_5 t}$
4	Second-order with a rectangular distribution	$R = R_{\infty,6}\{1 - \left[\frac{1}{K_6 t}\ln(1 + K_6 t)\right]\}$

*Models 3–5 are algebraically equivalent (all reduce to R = R∞·t/(τ + t)); the four numbered entries represent four distinct model forms.

The fitted first-order rate constant, k = 1.57 min^{-1}, corresponds to a characteristic flotation time of 1/k ≈ 0.64 min, indicating that roughly 63% of the floatable copper is recovered within the first ~40 s and over 95% within 2 min. This rapid flotation kinetics reflects the strongly hydrophobic, well-liberated chalcopyrite floating early in the process and provides the kinetic basis for the short rougher residence time adopted in this study. The predominance of first-order behavior indicates that recovery is controlled by the rate of particle–bubble attachment and the availability of floatable surface, consistent with previous work on sulfide flotation [19]. It is noted, however, that the perfectly mixed reactor model has been reported as the best fit for Sarcheshmeh porphyry copper ore [18]; the classical first-order fit obtained here likely reflects the narrow floatability distribution of the liberated chalcopyrite in the −75 µm feed.

Table 2. The calculated values of parameters for Kinetic Models.

Model	Classical First-order Model	First-order with Rectangular Distribution	Hyperbolic / fully-mixed (2nd-order)*	Second-order with Rectangular Distribution
R∞	99.36	103.70	104.68	107.15
k	1.57	4.21	-	9.36
R^2	0.9999	0.9985	0.9978	0.9967
RMSE	0.27	1.39	1.71	2.06
Physically admissible R∞?	Yes	No	No	No

*Single form representing the three equivalent models

3.6. Second-stage flotation

The concentrate obtained from the initial flotation stage was subjected to a second-stage flotation process while maintained in slurry form. This approach minimized exposure to atmospheric oxygen and reduced the risk of surface oxidation, which can adversely affect sulfide flotation performance. The operating conditions optimized during the first-stage flotation were retained for the second stage to ensure process consistency. The feed was deslimed prior to rougher flotation to remove fine slimes; because the particle size had already been optimized to −75 µm, no additional grinding was applied at this stage. The rougher concentrate was subsequently reground to P80 ≈ 20 µm before cleaner flotation to liberate residual composite particles and improve cleaning selectivity.

As shown in Figure 11, multistage flotation significantly improved both copper recovery and concentrate grade. Following the first flotation stage, copper recovery exceeded 97%, with a concentrate grade of approximately 9 wt%, while iron recovery to the rougher concentrate was limited to about 17%. Subjecting the first-stage concentrate to a second flotation stage resulted in a copper recovery of approximately 93% with a substantially higher concentrate grade of 23 wt%. The overall copper recovery achieved through the multistage flotation process was approximately 90%.

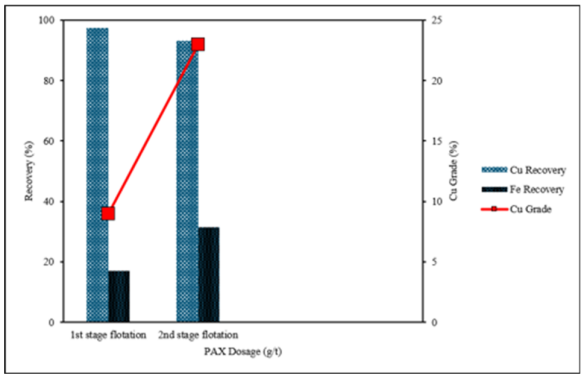

Figure 11. Second-stage flotation results.

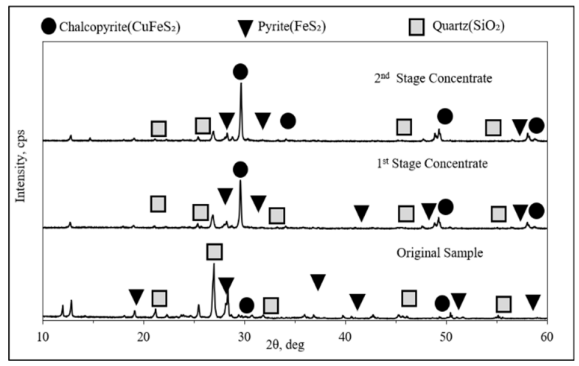

Figure *12*. XRD pattern for three stages of ore sample.

Mineralogical changes associated with multistage flotation are illustrated in Figure 12, which presents the XRD patterns of the original ore, first-stage concentrate, and second-stage concentrate. In the original ore, quartz and pyrite phases were dominant, and chalcopyrite peaks were poorly resolved due to overlapping with gangue minerals. In contrast, chalcopyrite diffraction peaks became clearly distinguishable in both the first- and second-stage flotation concentrates, confirming effective upgrading of the copper-bearing mineral and progressive gangue rejection through multistage flotation.

3.7. Mass Balance

A general process flow chart for the flotation process, developed based on all experimental results and the preceding discussion, is presented in Figure 13. The flow sheet represents a two-stage flotation circuit. The first stage was designed to establish optimized conditions for the flotation of chalcopyrite from the low-grade copper deposit, while the second stage served to further upgrade the copper grade and recovery and to reject gangue minerals such as pyrite from the final concentrate. Starting from a feed grade of 0.56 wt.% Cu, the multistage circuit produced a final concentrate assaying 23 wt.% Cu, corresponding to an enrichment ratio of approximately 41-fold. The second (cleaner) stage alone recovered about 93% of the copper reporting to the rougher concentrate, and the overall copper recovery across both stages was approximately 90% (Table 5), confirming the effectiveness of the proposed flow sheet for treating this low-grade copper ore.

Table 3. Mass balance for the two-stage flotation circuit (basis: feed = 100).

Stream	Yield, %	Cu grade, %	Cu distribution, %	Fe grade, %	Fe distribution, %
Feed	100.0	0.56	100.0	9.30	100.0
Rougher concentrate	6.04	9.00	97.0	26.20	17.0
Cleaner concentrate (final)	2.20	23.00	90.2	22.50 ‡	5.3
Cleaner tailing	3.84	0.99	6.8	28.31	11.7
Rougher tailing	93.96	0.018	3.0	8.22	83.0

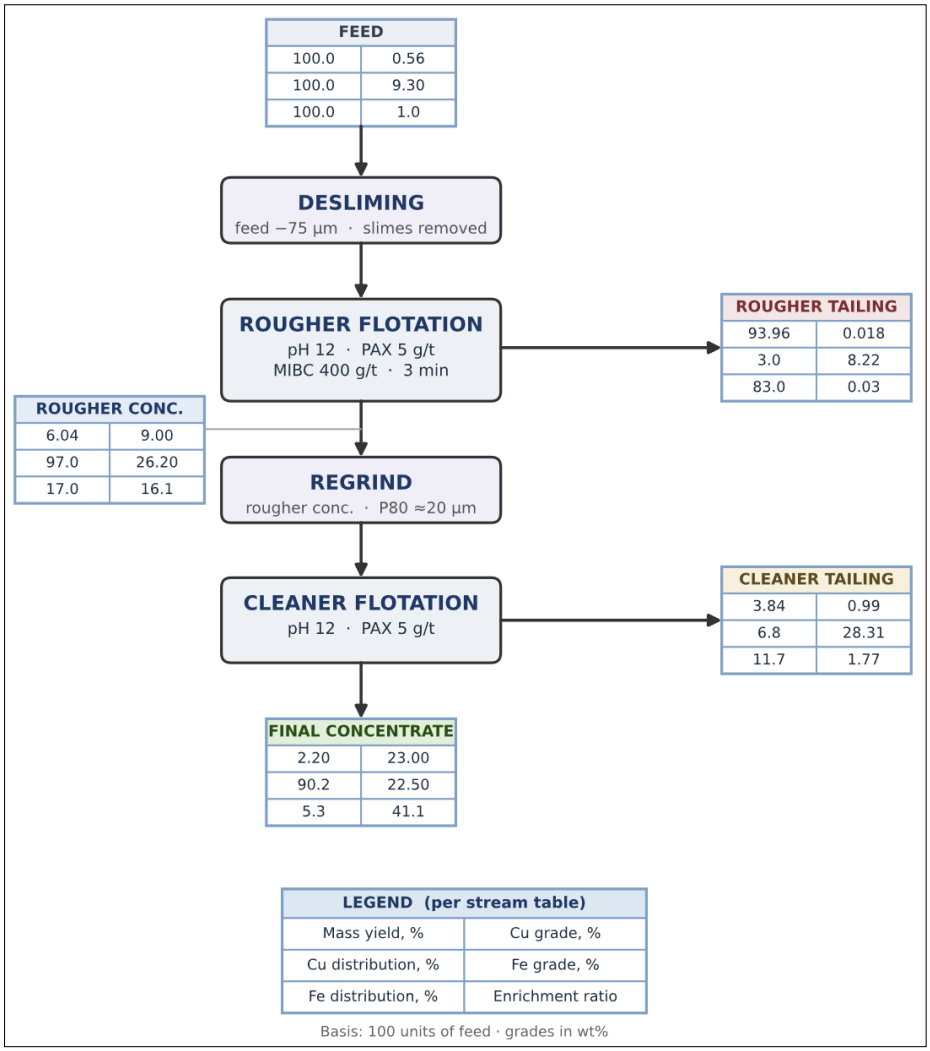

Figure 4. Two-stage flotation flow sheet with mass balance for the low-grade copper ore.

4. Conclusions

This study investigated the flotation behavior of low-grade copper ore from the Saindak Copper–Gold Project, Balochistan, Pakistan, with the aim of optimizing flotation parameters for efficient copper recovery and upgrading. The effects of particle size, slurry pH, collector dosage, and flotation time on chalcopyrite flotation performance were systematically evaluated to establish optimal operating conditions.

The results demonstrated that fine grinding (particle size <75 µm), strongly alkaline conditions (pH 12), a low potassium amyl xanthate (PAX) dosage of 5 g/t, and a short flotation time of 3 min provided the most favorable balance between copper recovery and concentrate selectivity. Application of these optimized conditions in a multistage flotation scheme significantly improved flotation performance, resulting in a copper recovery of approximately 90.2% with a concentrate grade of 23 wt%. Simultaneously, iron reporting to the final concentrate was reduced to about 5% of the feed iron, indicating strong rejection of iron-bearing gangue across the two stages.

The present optimization was conducted using a sequential (one-factor-at-a-time) approach and qualitative liberation assessment; quantitative liberation analysis by automated mineralogy and a statistical experimental design resolving parameter interactions are recommended to further strengthen and generalize these findings.

More broadly, the results indicate that near-collectorless, high-pH flotation is viable for pyrite-rich low-grade porphyry ores, with the pyrite-to-chalcopyrite ratio serving as a transferable criterion. Validation of interaction effects through statistical design, assessment of a lime-based

circuit, and pilot-scale confirmation remain the key next steps.

The findings of this study confirm that efficient upgrading of low-grade porphyry copper ore can be achieved through careful optimization of flotation parameters and the application of multistage flotation. The developed approach offers a practical and potentially cost-effective processing strategy for the Saindak project and may be applicable to similar low-grade copper deposits in Pakistan and other regions.

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بهینه‌سازی پارامترهای فلوتاسیون برای بازیابی مس از کانسنگ‌های کم‌عیار: مطالعه موردی در پاکستان

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کلمات کلیدی

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آنالیز اندازه

چکیده

سنگ‌های پورفیری مس-طلا با عیار پایین، به دلیل پراکندگی ضعیف مواد معدنی و محتوای بالای گانگ، چالش‌های فرآوری قابل توجهی را ایجاد می‌کنند. در این مطالعه، یک نمونه سنگ معدن از پروژه مس-طلای Saindak در پاکستان، با میانگین عیار مس تقریباً ۰.۵۶ درصد وزنی، برای بهینه‌سازی پارامترهای فلوتاسیون برای بهبود بازیابی مس مورد بررسی قرار گرفت. مشخصه‌یابی کانی‌شناسی با استفاده از XRD، XRF، SEM-EDS و ICP-OES، کالکوپیریت را به عنوان کانی اصلی حاوی مس، که به طور ریز در گانگ سیلیکات و سولفیدهای آهن پراکنده شده است، تأیید کرد. مجموعه‌ای از آزمایش‌های فلوتاسیون دسته‌ای با تغییر سیستماتیک اندازه ذرات، pH دوغاب، دوز کلکتور و زمان فلوتاسیون انجام شد. نتایج نشان داد که شرایط بهینه (اندازه ذرات <۷۵ میکرومتر، pH 12 و زمان فلوتاسیون ۳ دقیقه‌ای) امکان بازیابی مس خشن‌تر تقریباً ۹۷ درصد را با دوز کلکتور تنها ۵ گرم در تن فراهم می‌کند، و کنسانتره نهایی پس از تمیزکاری دارای عیار ۲۳ درصد وزنی مس است. تحلیل سینتیک فلوتاسیون نشان داد که مدل سینتیکی مرتبه اول بهترین برازش را با داده‌های تجربی ارائه می‌دهد. یافته‌ها نشان می‌دهند که ارتقای کارآمد مس از سنگ معدن کم‌عیار سیندک را می‌توان با حداقل مصرف کلکتور به دست آورد، که یک استراتژی فرآوری بالقوه مقرون‌به‌صرفه و پایدار برای ذخایر مس پورفیری مشابه ارائه می‌دهد.

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