Physicochem. Probl. Miner. Process., 61(6), 2025, 216139

http://www.journalssystem.com/ppmp

Physicochemical Problems of Mineral Processing

ISSN 1643-1049

Enhanced recovery of unevenly disseminated copper ore dominated by chalcocite via a combined process incorporating flash flotation and split flotation

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Abstract: The flotation of unevenly disseminated minerals presents a persistent challenge in mineral processing. In this study, a combined flash flotation and split flotation approach was developed based on mineralogical analyses aimed at enhancing the recovery of unevenly disseminated copper sulfide minerals. Mineralogical analysis indicated that, under a feed particle size of P80 at 200 µm, 31.65% of copper sulfide minerals were fully liberated, with 28.40% falling within the 20-147 µm size range, which is optimal for flash flotation. The flash flotation yielded a concentrate grading 54.96% Cu at a Cu recovery of 31.03%. This, together with a subsequent process incorporating split flotation, has led to a final concentrate grading 48.85% Cu and with an overall Cu recovery of 95.67% being obtained. The recovery represents about 6.5% improvement when compared to the conventional flotation process that is currently employed on site. The improvement is also accompanied with a reduction in collector and frother consumptions of 23% and 28%, respectively. The results demonstrate the effectiveness of the combined process incorporating flash flotation and split flotation for handling unevenly disseminated Cu ores dominated by chalcocite.

***Keywords*:** flash flotation, split flotation, chalcocite, copper sulfide minerals, unevenly disseminated ores

1. Introduction

Flotation is a widely applied separation process for the beneficiation of valuable minerals, from which nearly 90% non-ferrous metal concentrate and 50% ferrous metal concentrate were produced (Wang and Li, 2025). Generally, the optimum size fraction for an efficient flotation process is 10 – 150 µm. Particles finer than 10 µm exhibit low bubble-particle collision efficiency, while particles coarser than 150 µm are subject to detaching from bubbles, leading to a poor recovery (Amini et al., 2017; Liu et al., 2023; Sutherland, 1989; Tao, 2005). Mineral liberation through grinding is essential for efficient flotation, it is ideal if the valuable minerals in ore bodies exhibit uniform size fraction. However, in most ore bodies, minerals are generally occur as a mixture of coarse, medium, and very fine grains, the distribution is unevenly disseminated (Qi et al., 2024).

Unevenly disseminated poses a great challenge for the comprehensive utilization of mineral resources. Firstly, coarse liberated particles are prone to overgrinding, reducing flotation efficiency (Jameson, 2010; Janishar Anzoom et al., 2024; Kromah et al., 2022). Secondly, finely grained valuable minerals are often locked in gangue minerals after primary grinding and it would likely be lost into the tailings if they are not effectively and efficiently captured to the regrinding circuit for further liberation. The locked particles are generally in the coarse size fraction while clays and excessive fines are generally found in the fine size fractions (Farrokhpay et al., 2021; Liu et al., 2025; Miettinen et al., 2010; Subrahmanyam and Forssberg, 1990). In short, fines and coarse particles generally have their respective issues, which would generally require different solutions to handle. It is not surprised that the solution

DOI: 10.37190/ppmp/216139

used to effectively tackle fines issues could negatively affect coarse flotation, and vice versa. It is thus reasonably speculated that the process might be more efficient in some cases if fines and coarse particles are separated for respective conditioning and flotation.

Flash flotation, a pivotal advancement in modern mineral processing, provides a feasible and economic way to recover liberated coarse mineral particles within grinding circuit (Newcombe et al., 2012b; Yan et al., 2005). Flash flotation receives cyclone underflow or mill discharge as the feed, typically of high slurry density and short residence time. With flash flotation, the valuable mineral particles could be recovered as early as possible so as not to become overground (Mackinnon et al., 2003; Newcombe et al., 2013a). The process can recover particles larger than 250 μm, which are otherwise not deemed recoverable through conventional flotation cells (Mankosa et al., 2016; McGrath et al., 2013). For example, at the Kanowna Belle Plant, the recovery of both gold and sulfur decreased dramatically when the flash flotation operation was closed, most likely attributed to issues related to overgrinding (Newcombe et al., 2013b). The simulation results claimed that the gold and sulfur recoveries in flash flotation concentrate could be about 17% higher than that of the concentrate generated through a gravity separation process (Erkan et al., 2022). In an industrial scale, the gold recovery in flash flotation concentrate reached 48.3%, and the overall gold recovery reached 90.6%, along with the silver recovery improvement (Arellano-Piña et al., 2023).

Split flotation is another flotation strategy where the feed slurry is firstly classified into coarse and fines fractions via a hydrocyclone cluster, and then each fraction is subjected to a separate flotation process optimized for its particle size and the respective issues. With the split flotation process, the coarse flotation could be almost free of clays, slimes and with minimal entrainment, and offers an effective way to capture the locked valuable minerals into a regrind circuit for further liberation while the fines flotation could tightly focus on dealing with mineral flotation kinetics, clays and entrainment (Farrokhpay et al., 2021; Ghasemi Flavarjani et al., 2025; Janishar Anzoom et al., 2024; Miettinen et al., 2010; Trahar, 1981). In the flotation of coal, compared to the conventional flotation, the split flotation (classification size at 150 μm) greatly improved the flotation efficiency, and the ash content decreased from 23.3% to 20.8% (Hoşten and Muratoğlu, 1996). Dey et al., studied the flotation behavior of coal before and after split flotation, and his results indicated that under the conventional flotation process, the ash content in the concentrate could not meet the requirement of the minimum standard, with the implementation of split flotation, several products with varied ash contents of 10.0%, 17.0%, 23.6% and 61.8% were obtained, which provides flexibility for products utilized in different situations and realizes a comprehensive utilization of coal resources (Dey and Bhattacharyya, 2007).

Chalcocite is the major copper sulfide mineral of interest in this study. Of particular interest is its density being high at 5.5 – 5.8 g/cm^3 and its Mohs hardness being low at less than 3.0, which makes it susceptible to accumulated in the cyclone underflow and then being back to the mill for unwanted further size reduction which eventually leads to overgrinding and generation of ultra-fine chalcocite particles. On the other hand, the ore also contains substantial, though varied, amounts of clay minerals. Without a particular attention, Cu recovery would suffer together with excessive reagents consumption.

This study aims at tackling the above mentioned issues effectively with a combination of flash flotation and subsequent split flotation.

2. Materials and methods

2.1 Minerals and reagents

The ore, used for the testwork, was collected from Kamoa site in DRC (Democratic Republic of the Congo). It was crushed to -2 mm, well blended and then split and bagged into approximately 430 g each using a rotary splitter. These bagged samples were stored in a freezer to prevent oxidation. The reagents used in the study including collector sodium isobutyl xanthate (SIBX), frother SF522 and dispersant/depressant sodium silicate. They were all of industrial purity.

2.2. Flotation tests

430 g minerals together with 280 ml water was mixed and added to the mill to obtain the pre-determined P_{80}, the slurry was then transferred to the flotation cell for flotation tests. LFM01 flotation machine

(BGRIMM Technology Group, Beijing, China) and XFD flotation machine (0.5L, 0.75L and 1L, Jilin Exploration Machinery Plant, Changchun, China) were adopted for flash flotation and conventional flotation (coarse and fines flotation). The simplified flotation flowsheet is shown in Fig. 1. The slurry was conditioned with dispersant (if required), collector and frother in sequence, and the conditioning time for each reagent was 3 minutes. In flash flotation tests, the total residence time was 2 minutes and the concentrate was collected every 30 seconds. For the split flotation (coarse and fines flotation), the total residence time was 40 minutes, and five separate concentrate samples were collected at 2, 5, 13, 23, and 40 minutes of flotation, respectively. The flotation tests were conducted thrice, the average value and standard deviation were reported as final value and error bar.

The locked-cycle test was conducted for a total of 6 cycles. The criterion for the system achieving equilibrium was defined as the mass balance errors of the products (including the wet mass of concentrate, tailings and middlings) from the final three cycles (cycles 4-6) both being less than 5%. The final metallurgical indicators were calculated using the weighted average of these three cycles.

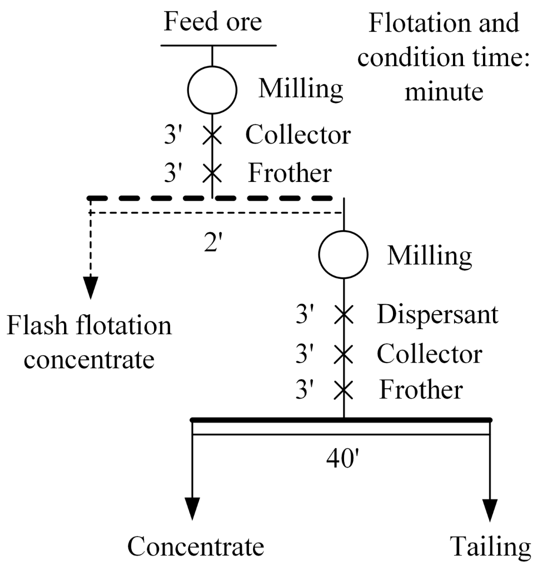

Fig. 1. Lab simplified flowsheet of flotation tests

2.3. Mineralogical analysis

The ore sample was homogenized and split to obtain a representative sample. The sample was then sieved into different size fractions: +147 μm, -147+74 μm, -74+38 μm, -38+25 μm, -25 μm, etc. Each representative fraction was mounted in epoxy resin and prepared as polished sections. The sections were examined under a Leica DM2700p polarizing microscope (Leica Microsystems GmbH, Germany) to assess particle uniformity and representativeness. After carbon coating, the sections were analyzed using a Tescan Vega3 scanning electron microscope (TESCAN, Czech Republic) under a vacuum of less than 2.0×10^{-3} Pa. Mineral composition and content, as well as the particle size distribution and liberation characteristics of copper sulphide minerals, were determined using an X Flash 7 energy-dispersive spectrometer (EDS) coupled with a BPMA process mineralogy analysis system (Bruker Nano GmbH, Germany).

3. Results and discussions

3.1. Mineralogical analyses

Mineralogical analyses were performed on the feed sample of interest. The mineral composition of the sample is detailed in Table 1. As seen in Table 1, chalcocite was the dominant copper sulphide minerals contained in the feed sample, e.g. 5.65% of 6.36% copper minerals. Also, copper oxide minerals, such as malachite, cuprite and chrysocolla, was found negligible. The dominate gangue minerals were silicate minerals including quartz and feldspar. Worth noting are clay minerals (e.g. muscovite, chlorite and kaolinite etc.), which accounted for more than 13% of the feed.

Table 1. Mineral composition of the sample

Mineral	Content/wt.%	Mineral	Content/wt.%	Mineral	Content/wt.%
Chalcocite	5.65	Hematite	2.22	Dolomite	2.34
Bornite	0.59	Llmenite	0.39	Albite	2.15
Chalcopyrite	0.06	Limonite	0.38	Diopside	1.68
Covellite	0.02	Rutile	0.17	Calcite	0.91
Native copper	0.02	Feldspar	40.03	Hornblende	1.29
Malachite	0.01	Quartz	23.81	Phlogopite	0.45
Cuprite	0.01	Muscovite	6.01	Biotite	0.40
Pseudomalachite	0.01	Chlorite	3.88	Apatite	0.17
Chrysocolla	0.01	Kaolinite	3.36	Other	0.12
Pyrite	0.44	Zoisite	3.33	Total	100.00

Results of liberation of copper sulfide minerals at P_{80} of 53 μm (the particle size of secondary milling cyclone overflow, feeding the rougher bank) are given in Table 2. Liberation classifications are determined based on the proportion of particle area occupied by the target mineral. "Fully liberated" refers to particles composed of 100% target mineral, "partially liberated" refers to particles containing 50% to <100% target mineral, and "locked" refers to particles containing less than 50% target mineral. Key points to note: 1) locked Cu remains high at 13.48%, and most of locked Cu is found to be in -74 μm size fraction.; 2) the proportion of Cu sulphide minerals in ultrafine size fraction (e.g. 0 – 10 μm) was high at 23.40%. The proportion of liberated Cu sulphide minerals was relatively high in the size fraction of 0 – 10 μm, strong evidence that chalcocite is prone to overground at a P_{80} of 53 μm. This is due to the facts that the SG (specific gravity: 5.5 – 5.8 t/m^3 for chalcocite) of chalcocite is significantly higher than that of major gangue minerals (e.g. quartz and feldspar, SG at about 2.6 – 2.7 t/m^3), and the Mohs hardness of chalcocite is low at about 2.5 – 3.0, close to clay minerals. The locked Cu concentrated in the coarse size fraction together with presence of clay minerals motivates selection of split flotation to deal with issues of coarse and fines separately so as to minimize the interaction of separate issues and thus improve flotation efficiency for both coarse and fines particles.

At the grind size P_{80} of 200 μm, which is the size similar to that of the site secondary milling hydrocyclone underflow, it was shown that the fully liberated Cu sulphide minerals accounted for about 31.65% of the total, supporting the employment of flash flotation to deal with coarse grain chalcocite, and to minimize chalcocite overgrinding and the generation of ultra-fine chalcocite.

Table 2. Distribution and liberation results of copper sulfide minerals with P_{80} of 53 μm.

Percent liberated	Distribution /%	Association/%			Size distribution/μm %				
		Sulfide minerals	Oxide minerals	Gangue minerals	0-10	10-20	20-38	38-74	74-147
Fully liberated	55.88	–	–	–	19.64	13.96	15.60	6.13	0.55
Partially liberated	30.63	0.14	3.47	27.02	3.15	7.56	12.46	6.6	0.86
Locked	13.48	0.01	0.93	12.54	0.61	2.73	4.04	6.10	–
Total	100.00	0.15	4.4	39.56	23.40	24.25	32.1	18.83	1.42

Table 3. Distribution and liberation results of copper sulfide minerals with P_{80} of 200 μm.

Percent liberated	Distribu-tion/%	Association/%			Size distribution/μm %						
		Sulfide minerals	Oxide minerals	Gangue minerals	0-10	10-20	20-38	38-74	74-147	147-295	>295
Fully liberated	31.65	–	–	–	0.25	1.78	8.16	13.59	6.65	1.07	0.16
Partially liberated	35.86	0.01	4.55	31.29	0.24	1.21	5.61	10.51	12.41	4.15	1.72
Locked	32.50	0.01	2.54	29.95	5.91	5.38	6.88	7.42	4.35	2.35	0.21
Total	100.00	0.03	7.09	61.23	6.40	8.36	20.66	31.52	23.41	7.56	2.09

3.2. Flash flotation

Flash flotation was simulated using a stage grinding and stage flotation approach in the laboratory tests. Specifically, the feed ore was firstly ground for pre-determined residence time to achieve P_{80} equivalent to that of the secondary milling hydrocyclone underflow. Flash flotation was then performed with this mill discharge slurry, and the tailings from the flash flotation were subject to further grinding to obtain P_{80} of 53 μm, followed by conventional flotation. Although the de-sliming effect of the hydrocyclone was not incorporated into this simulated process, the test results are considered indicative of the potential performance of flash flotation (Newcombe et al., 2012a).

To understand how the flash flotation would perform, some important parameters, e.g. particle size, Jg and collector dosage, were investigated in details.

3.2.1. Influence of grinding size

The grinding P_{80} plays a crucial role in determining flotation efficiency, especially for flash flotation. In order to demonstrate how the grinding P_{80} affects the Cu grade – recovery curve for flash flotation, the grind size P_{80} of 405 μm, 355 μm, 280 μm and 200 μm, were tested. The results are shown in Fig. 2. As indicated by Fig. 2a & 2b, the optimal Cu grade – recovery occurred at a grind size of P_{80} of 200 μm. At this grind size, a typical grind size for the site SM (secondary milling) hydrocyclone underflow, it was shown that a Cu recovery of about 30% could be attainable for a concentrate grading about 50% Cu. The implication from this result is that a significant amount of liberated or largely liberated chalcocite (e.g. proportion of about 30%) has been recovered by flash flotation.

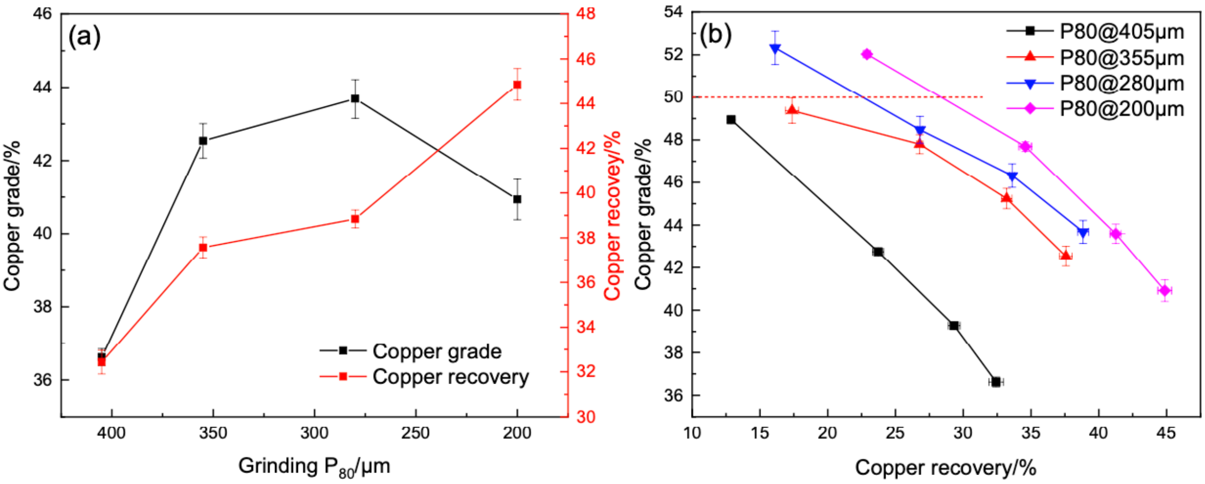

Fig. 2. Flash flotation results of varied grinding P_{80} (a: cumulative results, b: kinetics results)

3.2.2. Influence of Jg value

At the grind size P_{80} of 200 μm, the superficial gas velocity (Jg) was tested as shown in Fig. 3, as the Jg value increased from 1.0 cm/s to 1.5 cm/s, both the copper grade and recovery improved. When the Jg value was 2.0 cm/s, it was observed that the bubble stability is poor, making it difficult to form a stable froth layer, leading to the decrease of copper grade and recovery. Additionally, the flotation kinetics results also indicated Jg of 1.5 cm/s was preferable for flash flotation.

3.2.3. Influence of collector dosage

The flash flotation behavior under varied collector dosage was also studied and the results are displayed in Fig. 4. Based on the Fig. 4a, the copper recovery improved with the increasing collector dosage, whereas the copper grade exhibited an inverse tendency. According to flotation kinetics data in Fig. 4b, although there was minimal difference in flotation performance between 25 g/t and 30 g/t SIBX over the entire studied flotation period, a noticeable distinction was observed in the initial flotation stage. Specifically, when the copper grade was approximately 50%, copper recovery was higher at a dosage of 25 g/t. Therefore, the optimized dosage of SIBX was determined to be 25 g/t.

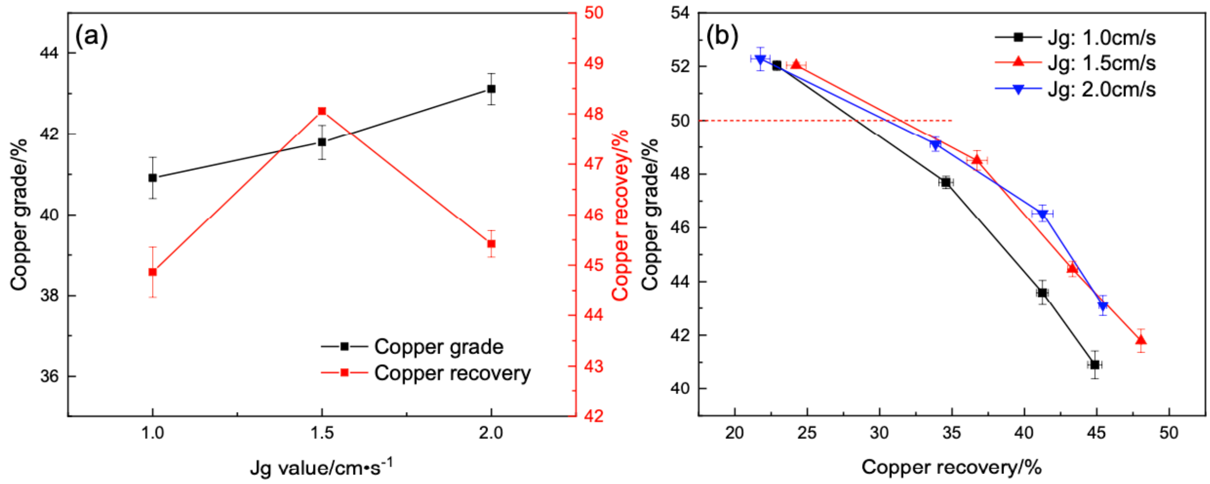

Fig. 3. Flash flotation results of varied Jg value (a: cumulative results, b: kinetics results)

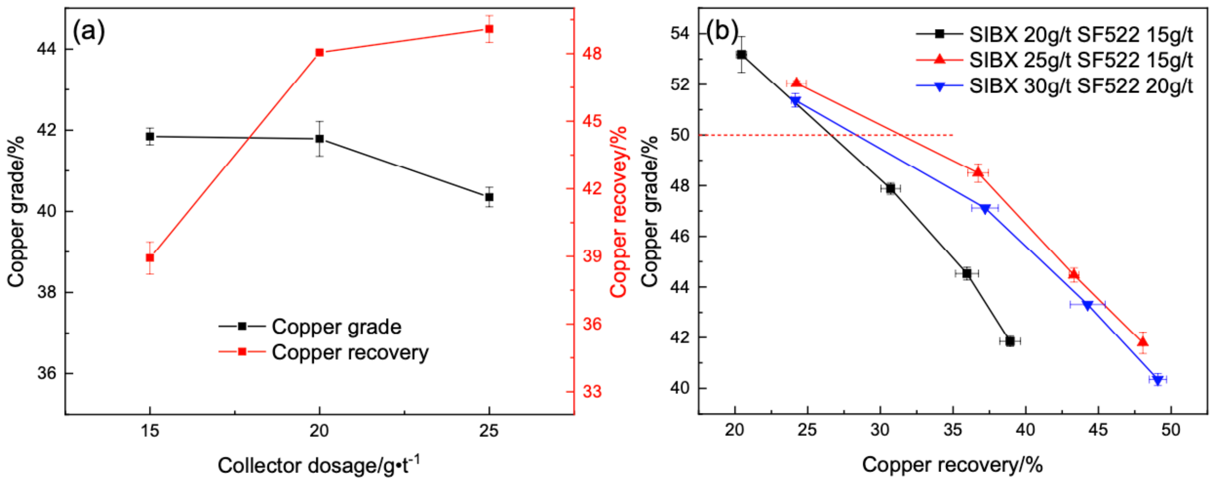

Fig. 4. Flash flotation results of varied collector dosage (a: cumulative results, b: kinetics results)

3.2.4. Open-circuit comparison between routine flotation and flash flotation

With results obtained from the tests described above, the impact of flash flotation on the overall flotation process remains unclear. To disclose it, the flash flotation with a laboratory simulated process as explained earlier in the text was compared to the conventional flotation (e.g. conventional flotation performed with the ore directly ground to give a P_{80} of 53 µm). The results, given in Table 4, indicate that with flash flotation the overall Cu rougher recovery reached 94.67% for a rougher concentrate grade of 11.65% as compared to an overall Cu rougher recovery of 93.02% for a rougher concentrate grade of 11.31% that was achieved with the conventional flotation, representing a Cu recovery improvement by 1.65% with flash flotation. This is generally deemed to be due to a reduction of generation of ultrafine chalcocite as a result of application of flash flotation minimizing occurrence of overgrinding.

Table 4. Comparison tests of routine flotation and flash flotation

Flowsheet	Product	Mass pull/%	Grade/%	Recovery/%
Conventional flotation	Rougher Concentrate	41.30	11.31	93.02
	Tailings	58.70	0.60	6.98
	Feed ore	100.00	5.02	100.00
Flash flotation followed by conventional flotation	Flash Concentrate	2.45	51.25	25.05
	Flash and Rough Concentrate	40.85	11.65	94.67
	Tailings	59.15	0.45	5.33
	Feed ore	100.00	5.04	100.00

3.3. Split flotation

3.3.1. Cut size selection

The cut size was evaluated for the cyclone split. For this purpose, three sizes, e.g. 38 µm, 25 µm and 20 µm, were assessed. The results are given in Table 5. As shown, the copper losses in the combined tailings (e.g. coarse tailings plus fines tailings) were 8.02%, 7.50% and 7.49% for the cut sizes of 38 µm, 25 µm and 20 µm, respectively. The optimal cut size was found to be 25 µm. The increase in Cu loss at a coarse cut size (e.g. 38 µm) is deemed to be due to an increased negative interaction between coarse and fines particles in the fine size fraction.

Table 5. Copper losses in tailings under varied classification size

Classification size/µm	Copper losses in tailings/%		
	Coarse	Fines	Total
38	4.35	3.67	8.02
25	4.94	2.56	7.50
20	4.99	2.50	7.49

3.3.2. Coarse flotation (+25 µm)

Firstly, the effect of pulp density was tested for coarse flotation. Results are shown in Fig. 5. As shown in flotation results, when the pulp density increased from 33.64% to 38.49%, the rougher copper recovery increased from 88.51% to 89.16%, and the copper grade was slightly decreased from 20.62% to 20.00%, further increasing pulp density to 41.12% leading to the decrease of copper recovery. Therefore, pulp density of 38.49% was chosen for the following tests.

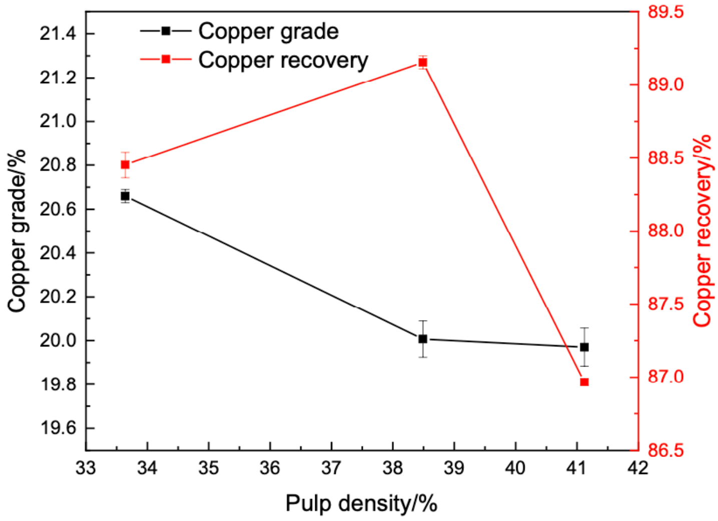

Fig. 5. Coarse flotation results of varied pulp density

Secondly, the collector dosage was tested, and results are shown in Fig. 6. With the increased collector dosage, the copper recovery was improved from 87.15% to around 89.62%, but the copper grade was decreased by about 1%. At the collector dosage of 70 g/t for coarse flotation, both of the recovery and grade was satisfactory.

The last but not least, the flotation kinetics was investigated for coarse flotation. Results are shown in Fig. 7 and Table 6. It is not a surprise that the coarse flotation gave a reasonably high Cu flotation kinetics as the slurry in coarse size fraction is almost free of clay minerals. For the coarse flotation, the residence time could be either 5 minutes or 13 minutes. This might not be so critical as the coarse tails is directed to the regrind circuit for liberating the locked Cu, which means there is no real loss of Cu even the coarse flotation is concluded at a shorter residence time. A short residence time has an implication of a small number of flotation cells required for the coarse flotation. But for the testwork, 13 minutes was chosen as the coarse flotation residence time to continue.

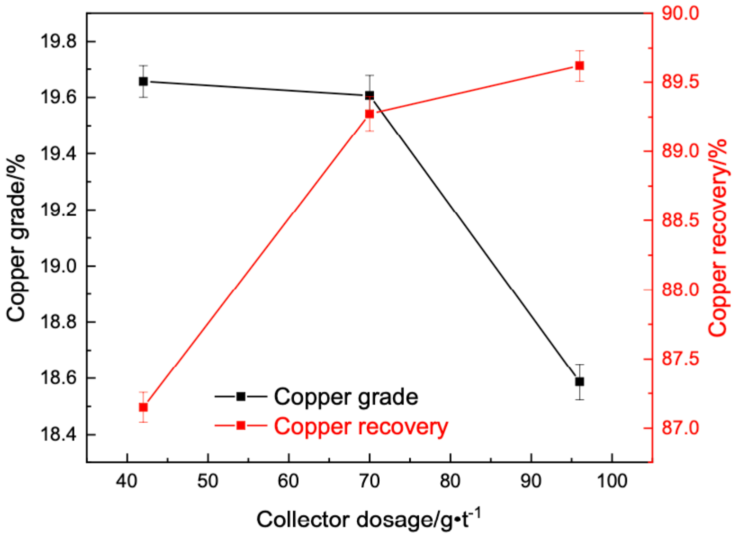

Fig. 6. Coarse flotation results of varied collector dosage

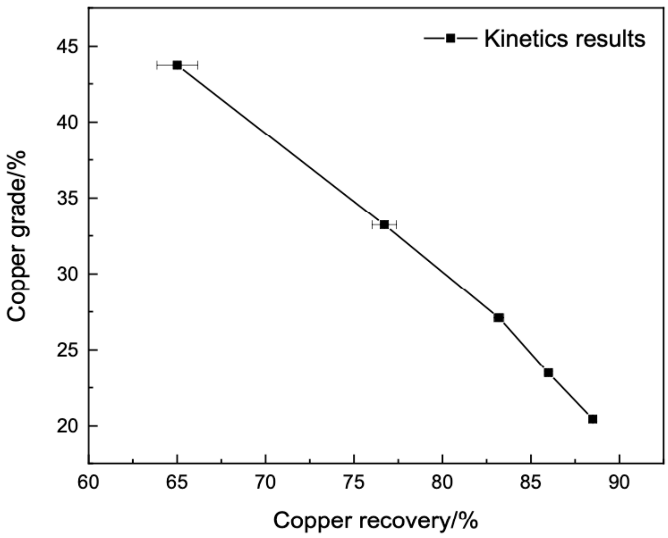

Fig. 7. Kinetics results of coarse flotation

Table 6. Kinetics results of coarse flotation

Product	Mass pull/%	Grade/%	Recovery/%
Concentrate 1	7.56	43.71	65.01
Concentrate 1-2	11.74	33.22	76.71
Concentrate 1-3	15.61	27.10	83.19
Concentrate 1-4	18.64	23.46	85.99
Concentrate 1-5	22.02	20.44	88.50
Tailings	79.98	0.75	11.50
Feed ore	100.00	5.09	100.00

3.3.3. Fines flotation (-25 μm)

The ore contains clay minerals, and the general understanding is that it would largely report to the fines. To manage the clay minerals and minimize its impact onto Cu recovery, the depressant sodium silicate was tested with different dosages. Results are shown in Fig. 8. Both Cu recovery and rougher concentrate grade firstly increased with the sodium silicate dosage up to about 1000 – 1500 g/t and then decreased with a further increase in the dosage. This testwork chose 1000 g/t to continue.

The Jg was also tested for fines flotation. Results are shown in Fig. 9. As indicated, the copper recovery and grade showed opposite tendency, with Jg of 0.90 cm/s, the concentrate with copper grade of 14.99% and copper recovery of 95.38% was obtained.

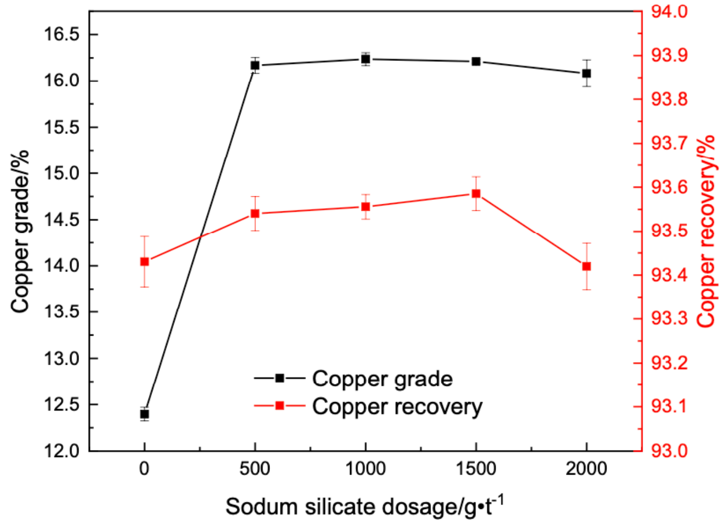

Fig. 8. Influence of sodium silicate dosage on fines flotation

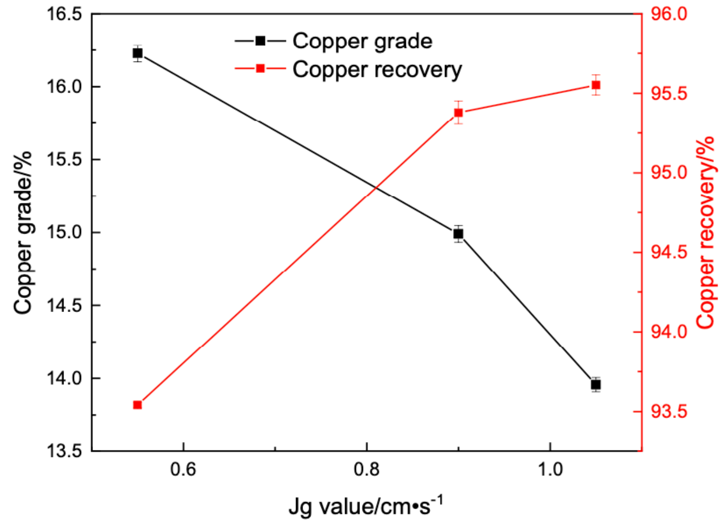

Fig. 9. Influence of Jg value on fines flotation

Also for the purpose of alleviating the impact of clay minerals, the pulp density was tested. Results are shown in Fig. 10. An optimal performance was achieved with a pulp density of 29% where Cu recovery reached 95.38% for a concentrate grading about 14.99% Cu. At a lower pulp density, e.g. % solids at 15% to 25%, Cu recovery was low, which is deemed to be due to insufficient collision between bubbles and mineral particles. At a higher pulp density, e.g. % solids at 34% to 37%, the concentrate grade tended to drop as the impact of clay minerals started. The testwork chose the pulp density of 29% to continue.

The effect of collector dosage on fines flotation are shown in Fig. 11. As indicated by flotation results, with the increased collector dosage from 140 g/t to 160 g/t, the rougher copper recovery was slightly increased. When the collector dosage increased from 160 g/t to 185 g/t, the rougher copper recovery remained unchanged, but the rougher copper grade decreased from 14.99% to 14.59%, therefore, the suitable collector dosage for fines flotation was 160 g/t.

The fines flotation kinetics was also tested. Results are shown in Fig.12 and Table 7. At the initial 5 minutes of fines flotation, Cu rougher recovery reached 87.80% for a concentrate grading 32% Cu. Extending the residence time to 13 minutes resulted in only a marginal increase of 4.14% in copper recovery that was accompanied by a sharp decrease in the concentrate grade to 24.05% Cu. Therefore, the residence time for the fines rougher flotation is selected as 5 minutes.

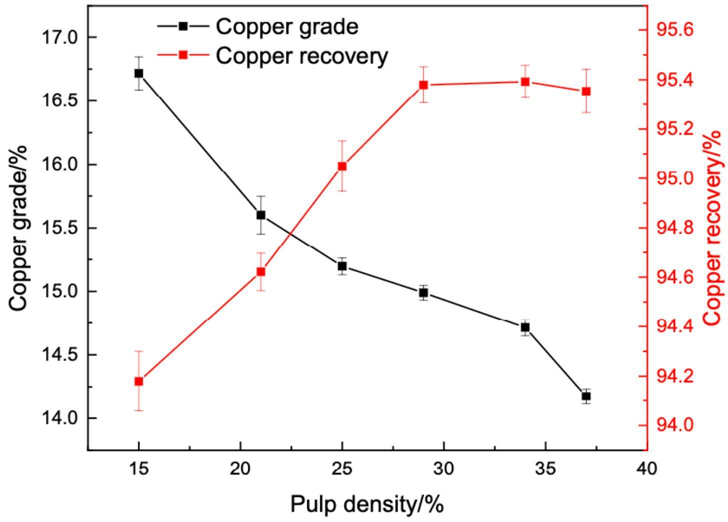

Fig. 10. Influence of pulp density on fines flotation

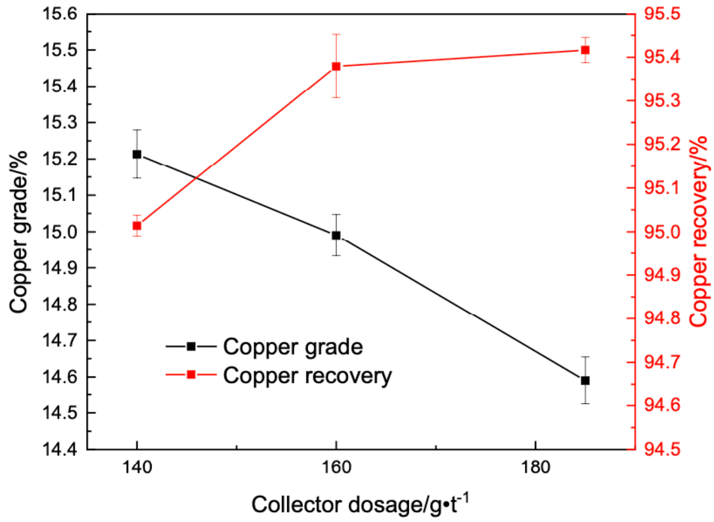

Fig. 11. Influence of collector dosage on fines flotation

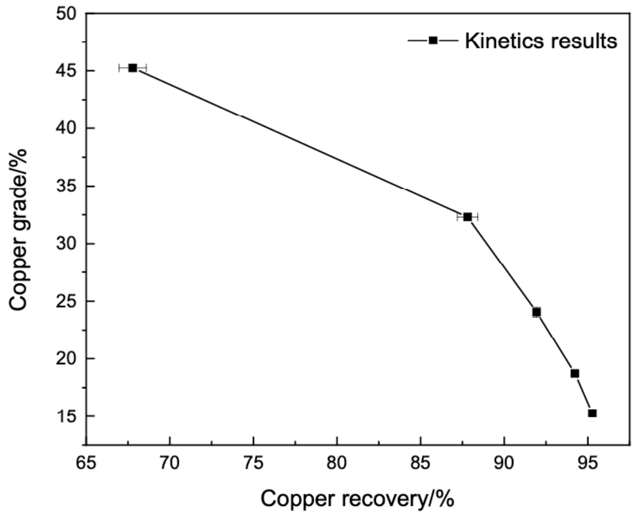

Fig. 12. Kinetics results of fines flotation

Table 7. Kinetics results of fines flotation

Product	Mass pull/%	Grade/%	Recovery/%
Conc 1	7.29	45.25	67.77
Conc 1-2	13.23	32.30	87.80
Conc 1-3	18.60	24.05	91.94
Conc 1-4	24.49	18.72	94.21
Conc 1-5	30.37	15.27	95.28
Tailings	69.63	0.33	4.72
Feed ore	100.00	4.87	100.00

3.4. Determination of regrinding size and Jg value for cleaner flotation post regrinding

The primary purpose of regrinding is to improve the liberation of target Cu minerals (e.g. chalcocite), thereby improving the quality of the final concentrate. In this study, three middling streams were directed to the regrinding circuit. The middlings include coarse flotation tailings, fines scavenger flotation concentrate (5-40 minutes), and cleaner flotation tailings of the combined rougher copper concentrates of coarse and fines flotation. The lab flowsheet is shown in Fig. 13. With this flowsheet, the regrind size and Jg for cleaner flotation post regrinding were tested. Results are shown in Fig. 14. As seen from Fig. 14, the optimal flotation performance was achieved with a regrind P_{80} of 10 μm in combination with a Jg of 0.90 cm/s.

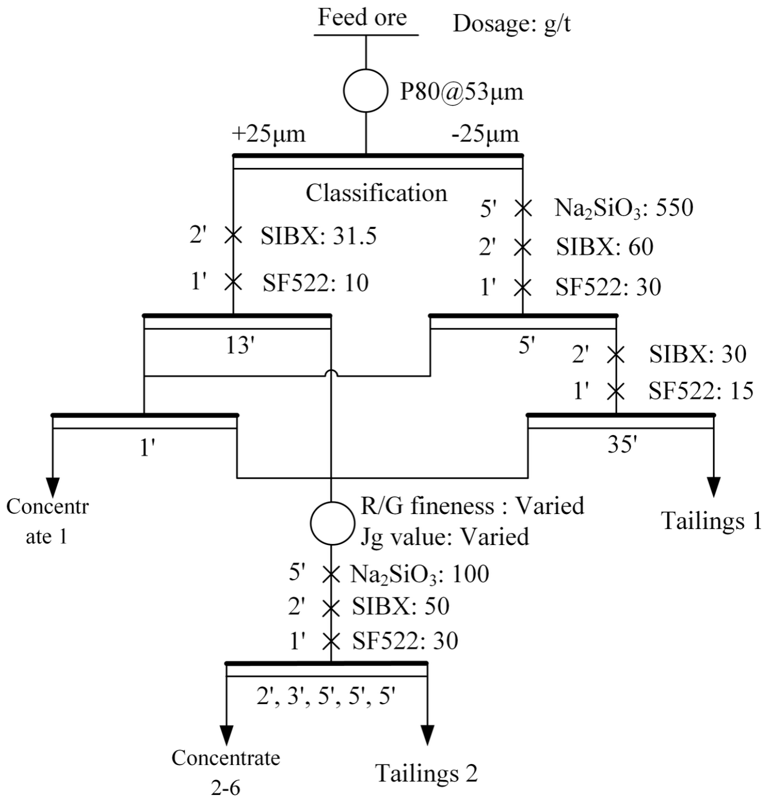

Fig. 13. Flowsheet of varied regrind size and Jg value

3.5. Locked-cycle tests

To evaluate the newly developed flowsheet incorporating flash flotation and split flotation (new flowsheet), locked-cycle tests were conducted and compared with the existing conventional flowsheet (conventional flowsheet). The conventional flowsheet is illustrated in Fig. 15. Results the locked cycle tests with the conventional flowsheet are given in Table 8. As seen from Table 8, the conventional flowsheet achieved a Cu recovery of 89.12% for a concentrate grading 53.47% Cu. The copper losses in tailings 1 and 2 were 6.96% and 3.93%, respectively.

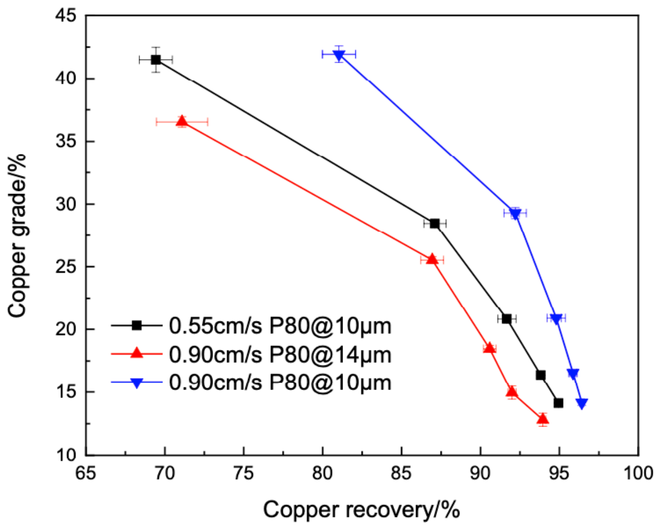

Fig. 14. Flotation results under varied regrind size and Jg value.

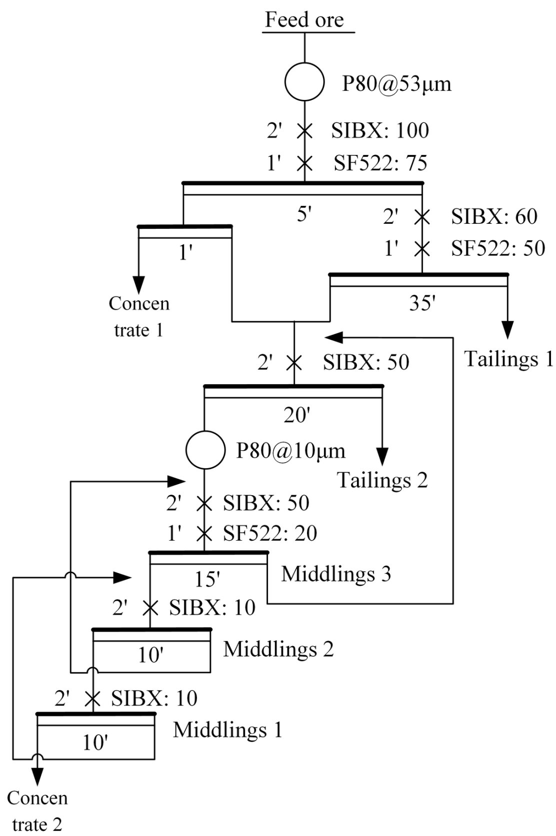

Fig. 15. Flowsheet of locked-cycle test with conventional flowsheet

Table 8. Results of locked-cycle test with conventional flowsheet

Product	Mass pull/%	Grade/%	Recovery/%
Concentrate 1	5.03	56.94	56.71
Concentrate 2	3.38	48.31	32.41
Tailings 1	62.65	0.56	6.96
Tailings 2	28.94	0.69	3.93
Final Concentrate	8.41	53.47	89.12
Final Tailings	91.59	0.60	10.90
Feed ore	100.00	5.05	100.00

As compared, the new flowsheet (shown in Fig. 16) gave a significantly improved Cu recovery (as given in Table 9), Cu recovery reached 95.67% for a concentrate grading 48.85% Cu. A couple of points to mentioned: 1) flash flotation provided a copper concentrate grading 54.96% Cu and a Cu recovery of 31.03%; 2) The Cu losses via tailings 1 and tailings 2 were significantly reduced down to 2.36% and 1.98%, respectively, as compared to the conventional flowsheet; 3) moreover, the collector and frother consumption for the new flowsheet was significantly reduced from 280 g/t to 216.5 g/t, and from 145 g/t to 105 g/t, respectively, as compared to the conventional flowsheet.

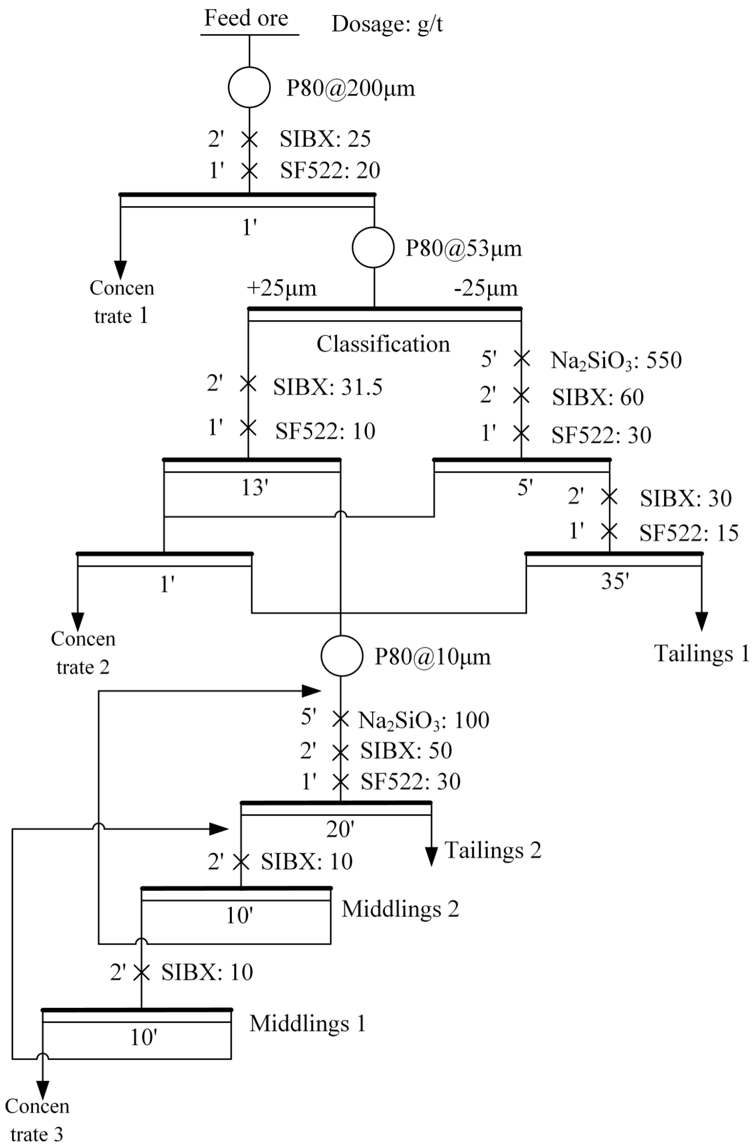

Fig. 16. Flowsheet of locked-cycle test with new flowsheet

Table 9. Results of locked-cycle test with new flowsheet

Product	Mass pull/%	Grade/%	Recovery/%
Concentrate 1	2.76	54.96	31.03
Concentrate 2	2.72	48.41	26.95
Concentrate 3	4.09	45.02	37.68
Tailings 1	39.08	0.30	2.36
Tailings 2	51.34	0.19	1.98
Final Concentrate	9.58	48.85	95.67
Final Tailings	90.42	0.23	4.33
Feed ore	100.00	4.89	100.00

The combined flash flotation and split flotation flowsheet yielded an overall improvement of approximately 6.5% copper recovery. However, we recognize that several operational challenges may arise during industrial implementation.

At a plant scale, the performance of hydrocyclones for precise 25 μm classification may be less accurate than in laboratory. This could lead to a proportion of mixed coarse and fine particles reporting to the wrong streams, potentially degrading flotation selectivity. To mitigate this risk and ensure control over the cut size, the implementation of small-diameter hydrocyclones operating is crucial. Second, the split flotation flowsheet significantly increases the throughput of middlings directed for regrinding. This elevated load may exceed the capacity of the current regrinding mill. To accommodate this change, additional regrinding equipment will be necessary to maintain the target P_{80} and ensure sufficient mineral liberation for subsequent flotation. Third, the adoption of split flotation (coarse and fines flotation) might initially appear to impose additional demands on the total flotation cell capacity. However, based on test results, the coarse fraction requires only a single rougher flotation at a relatively high pulp density. Upon assessment of the existing plant configuration, the current flotation capacity is sufficient, and no major expansion of flotation equipment is necessary for this stage.

4. Conclusions

This study evaluated the effectiveness of flash and split flotation for recovering copper from unevenly disseminated ores with copper minerals dominated by chalcocite. Mineralogical analysis revealed: 1) the ore contains coarse grain chalcocite and finely grain chalcocite; 2) quartz and feldspar are the major gangue minerals; 3) the ore also contains clay minerals. A combination of flash flotation and split flotation was found to effectively improve Cu flotation performance, in that: 1) flash flotation significantly reduces occurrence of overgrinding, and thus improve Cu flotation kinetics that may otherwise be compromised by presence of ultrafine chalcocite particles; 2) split flotation makes it very effectively in dealing with issues of coarse and fines separately, e.g. exclusively captures and sends the locked Cu in the coarse size fraction to the regrind circuit for liberation while only needing to deal with clay related issues during fines flotation. As a result, the overall copper recovery was improved by 6.5% as compared to the conventional flowsheet currently employed in the plant. In additional, the reagent consumption was reduced significantly, with collector cut by 23% less collector and the frother cut by 28%. This study has demonstrated that a combination of flash flotation and split flotation is the way to process unevenly disseminated copper sulfide ores that are dominated by chalcocite.

Acknowledgment

This work was supported by the Natural Science Foundation of Fujian Province, China (Grant No. 2024J01229).

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