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Research and application of enrichment methods for gold in refractory form found in pyrite and arsenopyrite minerals

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Abstract: Gold occurs in diverse mineralogical associations, which significantly influence the choice of beneficiation methods. While cyanidation remains the primary technique for gold extraction, it is generally ineffective for refractory gold, where gold is finely disseminated within sulfide minerals such as pyrite and arsenopyrite. In such cases, flotation is widely employed as a pre-concentration method, followed by pyrometallurgical or hydrometallurgical processes to improve overall recovery. Refractory gold ores are often found in plants designed for conventional leaching, where they result in low recovery despite high gold grades. Process flowsheet modifications are therefore essential to recover gold inaccessible through standard cyanidation. This study focuses on the beneficiation of refractory gold ore from the Prevanly Region, part of ETI Bakir Co.'s Tereksay operation. Optical and scanning electron microscope analyses revealed no presence of native gold, indicating that gold occurs in sulfide phases. The gangue matrix primarily comprises quartz, muscovite, and dolomite. Initial bottle-roll leaching tests yielded low recoveries (20–25%), confirming the refractory nature of the ore. Subsequent flotation tests significantly improved recovery up to 95%, validating flotation as a suitable pre-treatment. To enhance the concentrate grade, a Knelson concentrator was integrated into the circuit, effectively removing fine clay and silicate gangue. This addition contributed to producing a higher-grade gold concentrate by reducing impurities.

Keywords: refractory gold, pyrite, arsenopyrite, flotation, hydrometallurgy, gravity

1. Introduction

Process selection entails identifying the most suitable metal extraction technique for a given feed material. In gold recovery, the primary objectives are to achieve economic efficiency and to implement a method that meets the specific technical and environmental requirements of the project. Preliminary tests are conducted to evaluate the characteristics of gold mineralization and to design an extraction approach that aligns with environmental regulations. Cyanidation assays are commonly employed to determine gold recoverability. However, suboptimal gold extraction may result from the presence of refractory components or other limiting factors. Sulfide and silicate minerals can contribute to gold refractoriness by encapsulating gold particles, while carbonaceous materials may adsorb dissolved gold (the "preg-robbing" effect), thereby reducing recovery. Additionally, reactive sulfide minerals can lead to excessive cyanide consumption, increasing operational costs. In some cases, even under favorable conditions, coarse gold may not be fully recovered due to its particle size and distribution. Therefore, careful process selection and the establishment of optimal processing parameters are critical to maximizing gold recovery and minimizing losses (McClelland et al., 2002).

The initial industrial applications of gold flotation began in the early 1930s with the introduction of xanthates and dithiophosphates as collectors (Weinig et al., 1937). Large-scale gold flotation operations have since been established in countries such as Canada, Australia, and Korea. However, these early plants predominantly utilized amine-type collectors (Richart, 1912; Hoover, 1912; Taggart, 1927). Over time, the number of flotation plants treating sulfide-hosted gold ores has steadily increased, driven by rising global demand for gold. As a result, significant advancements have also been made in oxidation

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processes via hydrometallurgical methods, as well as in pyrometallurgical recovery techniques such as roasting and smelting (Dunne, 1991). Both pyrite and arsenopyrite exhibit favorable floatability under acidic and alkaline conditions. Xanthates and thiocarbamates are commonly used as primary collectors under acidic pH conditions, while dithiophosphates typically serve as secondary collectors (Broekman et al., 1987). Since the introduction of the Knelson concentrator in the 1980s, gravity separation techniques have gained renewed attention (Ghaffari et al., 2017; Kökkılıç et al., 2015). The Multi-Gravity Separator (MGS) operates at relatively low centrifugal forces, whereas Falcon concentrators generate much higher forces—typically between 300 and 600 G (Aslan, 2007; Das et al., 2018; Farajzadeh et al., 2022; Majumder et al., 2006; Nayak et al., 2021). Knelson concentrators, capable of producing centrifugal forces up to 200 G, are widely used for the recovery of gold and platinum group metals and can operate in either continuous or semi-batch modes (FLSmidth, 2023; Chen et al., 2020). Their ability to recover particles as fine as 15–20 microns significantly enhances separation efficiency (Kökkılıç et al., 2015). In comparative studies, gravity separation tends to yield lower gold recoveries than flotation; however, the concentrate grades achieved by gravity methods are typically higher. In contrast, flotation processes generally produce concentrates with lower grades (Erkan et al., 2022).

Since the 1990s, hydrometallurgical techniques have gained significant prominence in the processing of refractory gold ores. At present, approximately one-third of global gold production is achieved through pre-oxidation followed by leaching. Common pre-oxidation methods include pressure oxidation (POX), bio-oxidation, and the Albion Process™ (Gorain et al., 2016).

2. Materials and methods

Samples were collected from the Prevanly Region of ETI Bakir Co.'s Tereksay Plant in Kyrgyzstan. The samples were crushed using laboratory-scale crushers, and all material was reduced to a particle size of less than 3.35 mm. The crushed samples were then homogenized and divided into representative portions.

2.1. Characteristics of experimental sample

2.1.1. Chemical analysis results

Chemical analysis revealed that the sample contained significant amounts of clay and silicate minerals. Gold content, determined using the cupellation method, was measured at 2.99 g/t. The results of the analysis are presented graphically in Table 1.

Table 1. Chemical analysis results of Prevanly feeding sample (mass fraction, %)

Element	Au*	Ag*	Fe	Cu	Zn	As	SiO_2	S
Grade	2.99	<3.00	3.00	0.008	0.036	0.419	63.88	1.20
Method	FA*			ARD*, ICP-OES			GM*	FR*

* The units of Au and Ag are g/t, *ARD: Aqua regia digestion, *FA: Fire assay (cupellation), * GM: Gravimetric dissolution method, *FR: Tube furnace roasting method

2.1.2. Mineralogical analysis results

Mineral images of the samples were obtained using an ore microscope for petrographic analysis. Samples were prepared by casting 2 grams of each fraction into molds with a diameter of 30 mm. After sanding, the samples were polished using a diamond suspension. The prepared specimens were then analyzed and photographed using a Leica DM2700 top-illuminated microscope. The microscopic images, presented in Fig. 1, show that pyrite and arsenopyrite minerals are liberated at coarse sizes. Although pyrite and arsenopyrite were clearly observed, no visible gold particles were detected during the microscopic analysis.

The surface of the prepared bakelite was covered with carbon, and mineralogical studies were performed using a scanning electron microscope (SEM). Mineralogical examinations were performed using a Thermo Fisher Scientific Apreo 2C scanning electron microscope (SEM) due to the inability to find gold in the ore during microscopic investigations. The Mineral Liberation Analysis (MLA) software

in the pertinent microscope was utilized to ascertain the degrees of mineral liberation. The liberation degrees of pyrite and arsenopyrite in the feed sample, categorized by size fractions, are presented in Table 2 and Table 3.

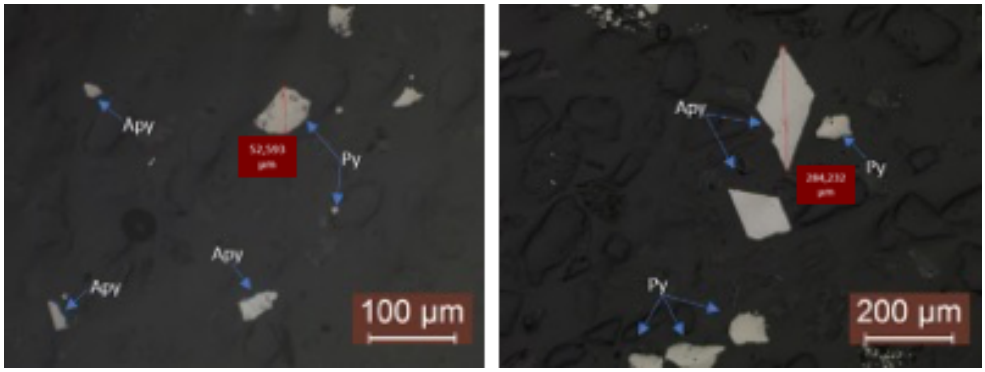

Fig. 1. Microscope images of sample (Py: Pyrite, Apy: Arsenopyrite)

Table 2. Pyrite liberation degrees according to size fractions

Size Fraction, µm	Total Liberated, %	Total Binary, %	Total Ternary, %	Total Complex, %
-106+75	68.63	14.70	8.09	8.78
-75+53	63.64	20.54	7.86	7.96
-53+38	71.01	17.00	6.33	5.65
-38+20	70.62	17.07	6.98	5.33

Table 3. Arsenopyrite liberation degrees according to size fractions

Size Fraction, µm	Total Liberated, %	Total Binary, %	Total Ternary, %	Total Complex, %
-106+75	75.02	13.33	6.27	5.38
-75+53	71.78	18.30	5.23	4.69
-53+38	72.54	18.74	4.90	3.82
-38+20	65.44	25.52	5.50	3.54

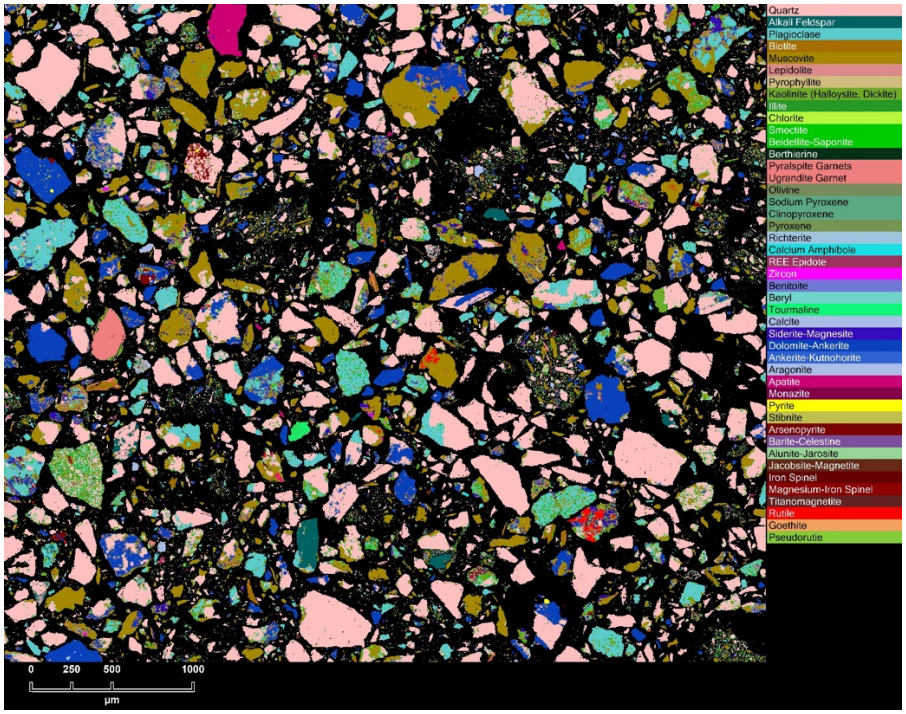

Fig. 2. SEM map image of the sample containing 57.21 ppm gold

SEM-MLA studies were conducted on the flotation concentrate to quantitatively and qualitatively assess the occurrence and morphology of gold within the sample. The results of the modal mineralogy analysis, including mineral abundance and liberation characteristics, as well as elemental assays, are summarized in Table 4. Fig. 2 displays scanning electron microscope (SEM) images accompanied by detailed annotations highlighting key mineral phases. The sample analyzed via the cupellation method contained 57.21 ppm gold; however, despite this significant gold content, no discrete gold particles were detected in the SEM-MLA analysis.

Table 4. Modal mineralogy and elemental assay table to sample of containing 57,21 ppm gold.

Mineral	Weight, %	Element	Grade, %
Pyrite	55.18	S	38.29
Arsenopyrite	21.57	Fe	32.61
Quartz	5.31	O	9.67
Muscovite	4.42	As	8.07
Dolomite-Ankerite	2.48	Si	4.91
Unclassified	1.74	Al	1.49
Illite	1.37	Zn	1.02
Sphalerite	1.34	Ca	0.82
Sodium Pyroxene	1.12	Mg	0.54
Plagioclase	0.98	Cu	0.04
Others	4.50	Others	2.53

2.2. Grinding parameters

A 178 x 356 mm Unal brand rod mill was used for the grinding tests. Except for the grinding time, all other parameters were maintained at the levels specified in Table 5. Samples ground for different durations were subjected to sieve analysis to determine their particle size distribution.

Table 5. Grinding parameters in the experiments

Feed Size (d_{100}), mm	Solid, g	Water, mL	Solid Ratio, %	Mill Speed, rpm	Grinding Media
3.35	1000	500	67	65	Rod

To evaluate the time-dependent grinding behavior of the samples, a grinding kinetics test was performed. Grinding durations were set at 5, 10, 20, and 40 minutes. In the experiments, the feed particle size distribution was determined based on the exponential equation x=y, as shown in the graph. The results of the grinding tests are illustrated in Fig. 3.

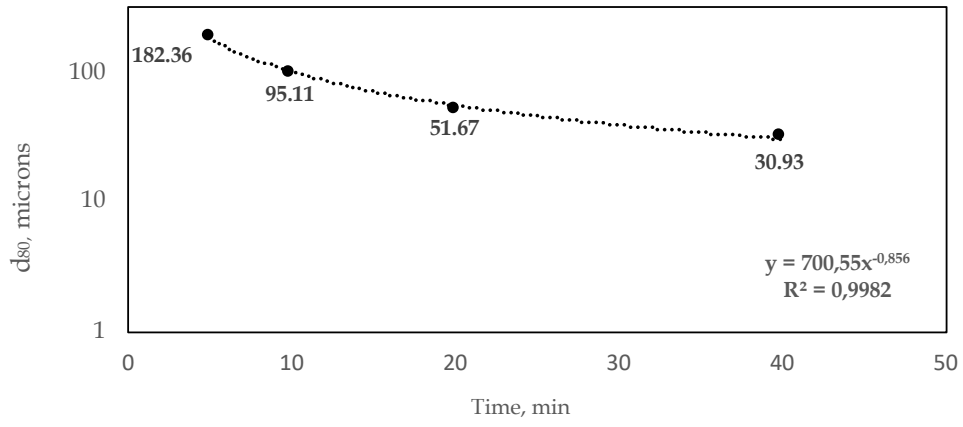

Fig. 3. Log-normal graph of time dependent d_{80} particle size of the samples

2.3. Flotation, gravity and bottle-roll tests

Unal brand Denver-type flotation cells were used in the experiments. Xanthate (SBX) was employed as the collector, MIBC as the frother, and Aero MaxGold 900 was added to enhance gold recovery. All tests were conducted at a natural pH range of 8.40 to 8.60.

Gravity separation experiments were carried out to recover metals and remove silicates from the flotation cleaning circuit tailings. These tests utilized a KC-MD3 model Knelson concentrator manufactured by FLSmidth. Open-circuit flotation tests on the cleaning circuit tailings were performed at centrifugal forces of 90 G and 120 G, with operating parameters detailed in Table 6. Based on the results of the open-circuit tests, the Knelson concentrator was integrated into the locked-cycle test-3.

To recover gold from the tailings of flotation tests, the tailings from locked-cycle Test-1 and Test-2 were subjected to bottle-roll tests as detailed in Table 7.

Table 6. Parameters of Knelson concentrator tests

Sample (g)	Sample Feed Type	Water (L/min)	G Force
800	Dry Feeding	3.50	90, 120

Table 7. Parameters of bottle-roll tests

Solid (gr)	Solid Vol. (cm^3)	Water (mL)	SG* (gr/cm^3)	SR* (%)
750.00	272.73	1125.00	2.75	40.00
Total Volume (cm^3)	Feed NaCN Quantity (g/l)	Air Supply (l/min)	pH	Leaching Time (h)
1397.73	1.00	1.00	11-11.5	48.96

* SG Avg.: Specific gravity of solid, *SR: Solid ratio in the pulp

2.3.1. Open-circuit flotation tests

A decrease in feed particle size was observed to correspond with an increase in gold recovery, accompanied by a reduction in the gold grade of the concentrate. Detailed experimental parameters and results are presented in Table 8. The tests were conducted at particle size cut-offs of 106 and 75 microns. Copper sulfate is used as an activator in the flotation of sulfide gold ores (O'Rourke, 2000). In the flotation experiments, copper sulfate was applied to enhance gold recovery. Five tests were conducted to determine the optimal dosage. Details and results of these experiments are presented in Table 9.

Table 8. Details of flotation tests to analyze the effect of particle size

Particle size (d_{80})	Reagents added, g/t			Time, minutes		Au Grade, g/t	Au Recovery %
	SBX	MIBC	MaxGold 900	Grinding	Ind.		
General Parameters	175	60	30		22		
200 µm				4.2		23.69	83.42
150 µm				6		20.71	85.54
106 µm				9		14.62	90.29
75 µm				13.5		10.77	91.85

Table 9. Parameters and results of flotation tests to analyze the effect of copper sulphate

Tests	Reagents added, g/t			Time, minutes		Au Grade, g/t	Au Rec., %
	SBX	MIBC	$CuSO_4$	Grinding	Ind.		
Test - 1	200	50	0	9	15	12.68	74.39
Test - 2	200	50	400	9	15	13.86	77.06
Test - 3	200	50	600	9	15	16.42	69.55
Test – 4	200	50	800	9	15	16.41	66.44
Test – 5	200	50	1000	9	15	15.55	64.75

Based on the results obtained, 400 g/t of copper sulfate was used in the experiments. A rougher kinetic flotation test was conducted to determine the maximum gold recovery, with the parameters provided in Table 10.

To produce a concentrate with a high gold grade, two stages of cleaning were applied to the rougher concentrate. The experimental parameters are presented in Table 11.

Table 10. Parameters of rougher kinetic test

Rougher flotation test	Reagents added, g/t			Time, min			d_{80}, µm
	SBX	MIBC	MaxGold 900	Grinding	Cond.	Ind.	
Grinding				13.5			75
Conc 1	100	50			5	5	
Conc 2	50	10			2	7	
Conc 3	50	10	25		3	9	
Conc 4	50	10	5		3	9	
Total	250	80	30		13	30	

Table 11. Parameters of two stage cleaner tests

Cleaner flotation tests	Reagents added, g/t			Time, min			d_{80}, µm
	SBX	MIBC	MaxGold 900	Grinding	Cond.	Ind.	
Grinding				9			106
Rougher	75	50	20		5	12	
Scavenger	50	10	10		2	10	
Cleaner 1					3	10	
Cleaner 2						5	
Total	125	60	30		10		

2.3.2. Locked – cycle flotation test 1 and 2

Following the determination of all parameters in open-circuit experiments, two locked-cycle flotation tests were conducted. These locked-cycle tests aimed to obtain values that more accurately represent the plant's operational conditions. Both tests were carried out over six cycles. In the first test, MaxGold900 was used alongside SBX, with the parameters provided in Table 12. In contrast, only xanthate was used as the collector in Test 2.

The parameters for Locked-cycle test 2 are given in Table 13. In this test, unlike the first, only MaxGold 900 was not used.

Table 12. Parameters of locked-cycle test-1

Locked-Cycle Test-1	Reagents added (per cycle), g/t				Time, min			d_{80}, µm
	SBX	MIBC	MaxGold 900	$CuSO_4$	Grinding	Cond.	Ind.	
Grinding					9			106
Rougher	150	50	30			5	12	Amount
Cleaner 1		10				3	10	of cycle
Cleaner 2		10					5	6

Table 13. Parameters of locked-cycle test-2

Locked-Cycle Test-2	Reagents added (per cycle), g/t				Time, min			d_{80}, µm
	SBX	MIBC	MaxGold 900	$CuSO_4$	Grinding	Cond.	Ind.	
Grinding					9			106
Rougher	150	50				5	12	Amount
Cleaner 1		10				3	10	of cycle
Cleaner 2		10					5	6

The objective was to minimize the concentrate volume in order to improve its grade. Therefore, a second locked-cycle flotation test was conducted to achieve this goal. The experimental results showed that MaxGold 900 enhanced recovery but reduced selectivity. Consequently, the second test used only the SBX collector. Without MaxGold 900, flotation kinetics slowed down, leading to a longer rougher flotation time. The experimental flowsheet is shown in Fig. 4.

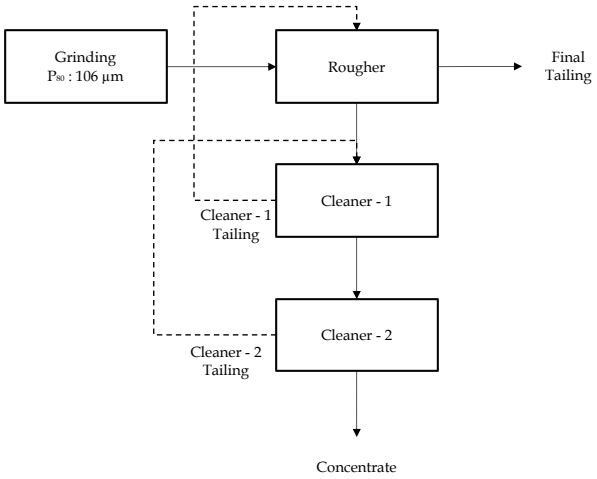

Fig. 4. Flowsheet of locked – cycle tests 1 and 2

2.3.3. Knelson concentrator integrated locked-cycle test (Test – 3)

Two locked-cycle experiments demonstrated that clay and silicate minerals elevated the circuit load, consequently diminishing the concentrate grade. To achieve high-quality concentrate, a 'middling' circuit was integrated into the flow sheet. The objective of the middling circuit was to sustain a low solid ratio to produce a concentrate with elevated gold content. The waste from the intermediate circuit was processed through a Knelson Concentrator, which improved gold recovery. Knelson's concentrate was subsequently reintroduced into the middling circuit, and a locked-cycle test-3 experiment was performed. The flowsheet for the third locked-cycle test was shown in Fig. 5 and parameters was presented in Table 14.

Table 14. Parameters of locked-cycle test-3

Locked-Cycle	Reagents added (per cycle), g/t				Time, min			d_{80}, µm
Test-3	SBX	MIBC	MaxGold 900	$CuSO_4$	Grinding	Cond.	Ind.	
Grinding					13,5			75
Rougher	125	50				5	15	Amount
Scavenger	50	20	10			3	10	of cycle
Cleaner							2	6
Middling	30	10					10	
Middling Cleaner		10					6	

3. Results and discussion

3.1. Open–circuit rougher and cleaner tests

As a result of the rougher kinetic experiment, 95.26% gold recovery was achieved in 30 minutes. Based on the results obtained, rougher time was determined as 12 minutes and scavenger time as 10 minutes in the experiments. The results of the experiment are illustrated in Fig. 6.

Flotation experiments incorporating a two-stage cleaning circuit were conducted following the kinetic tests. A concentrate with a gold content of 45.65 ppm was obtained; however, the mass of the "Cleaner 1 Tail" waste was significantly high. The circulating load was primarily composed of clay and silicate minerals. These gangue minerals increased the circulating load in locked-cycle tests 1 and 2,

while simultaneously reducing the gold grade in the final concentrate. The experimental results are presented in Table 15. To address this issue, optimizations were implemented in the flotation circuit, aiming to enhance metal recovery by removing gangue minerals through the use of a Knelson concentrator.

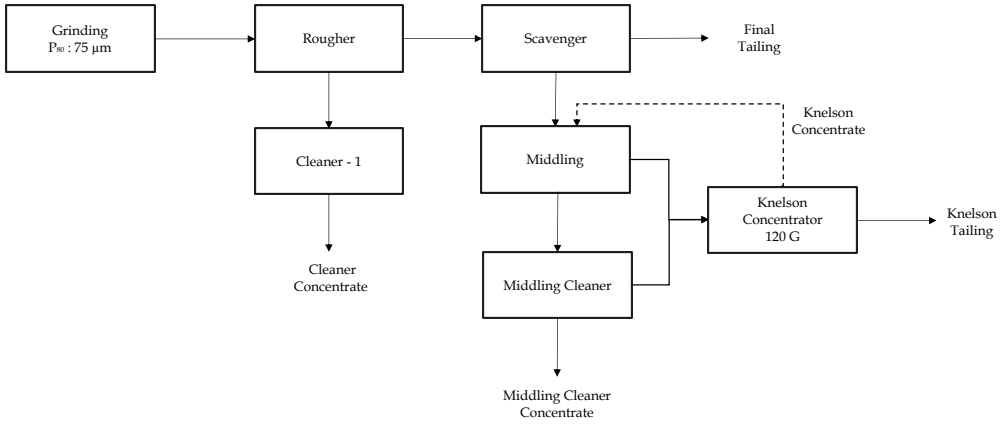

Fig. 5. Flowsheet of locked-cycle test 3 with the inclusion of a Knelson concentrator

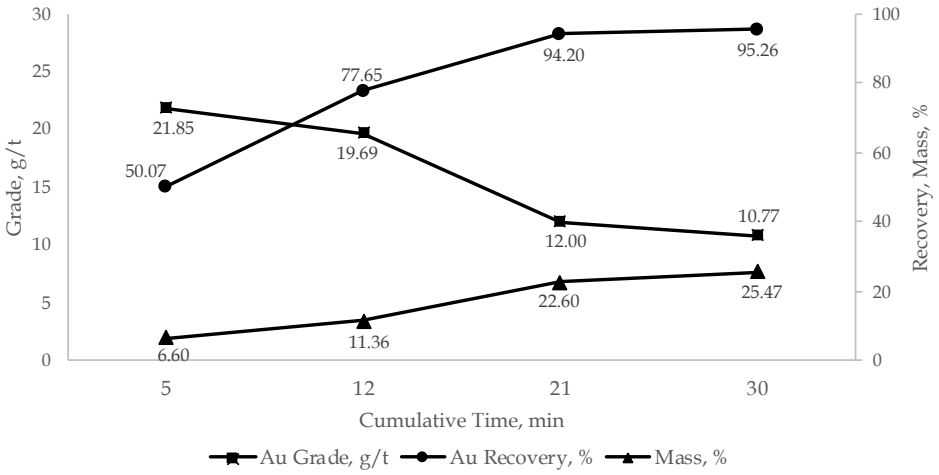

Fig. 6. Results of rougher kinetic flotation test

Table 15. Results of two stage cleaner flotation test

Product	Mass, %	Au, g/t	∑ Au, g/t	Au Rec., %	∑ Au Rec., %
Cleaner 2 Con.	2.18	45.652	45.65	35.21	35.21
Cleaner 2 Tail	2.88	22.158	32.28	22.58	57.79
Cleaner 1 Tail	12.2	6.944	14.35	30.09	87.88
Scavenger Con.	1.4	4.746	13.63	2.36	90.25
Final Tail	81.3	0.339	2.83	9.75	100.00
Total	100.0	2.83*		100.00	

* Calculated

3.2. Knelson concentrator tests

To address the accumulation of gangue minerals within the flotation circuit, Knelson concentrator tests were implemented. Recirculation of cleaning circuit tailings—rich in clay and silicate phases—into the flotation feed was found to significantly increase the circulating load and adversely impact the gold grade of the final concentrate. Analytical evaluations of the cleaning circuit tailings confirmed the

presence of a high proportion of gangue minerals and demonstrated the potential for their effective removal. Experimental trials conducted at centrifugal forces of 90 G and 120 G indicated that metal recovery improved proportionally with increased G-force, as detailed in Table 16.

Table 16. Knelson test results at 90 G and 120 G

G Force: 90 G	Mass, %	Grade						Recovery, %					
		Au, ppm	S, %	Al, %	Na, %	K, %	Ca, %	Au, %	S, %	Al, %	Na, %	K, %	Ca, %
Conc.	21.64	5.37	1.58	0.94	0.07	0.54	3.08	47.83	42.49	20.94	26.41	20.79	22.96
Tail	78.36	1.62	0.59	0.98	0.05	0.57	2.85	52.17	57.51	79.06	73.59	79.21	77.04
Total	100.00	2.43	0.80	0.97	0.05	0.56	2.90	100.00	100.00	100.00	100.00	100.00	100.00
G Force: 120 G	Mass, %	Grade						Recovery, %					
		Au, ppm	S, %	Al, %	Na, %	K, %	Ca, %	Au, %	S, %	Al, %	Na, %	K, %	Ca, %
Conc.	29.89	5.03	1.28	0.80	0.04	0.45	3.13	56.46	64.91	30.02	36.88	21.36	32.18
Tail	70.11	1.66	0.30	0.79	0.03	0.71	2.81	43.54	35.09	69.98	63.12	78.64	67.82
Total	100.00	2.66	0.59	0.80	0.03	0.63	2.91	100.00	100.00	100.00	100.00	100.00	100.00

Note: Knelson concentrator tests were conducted on samples with a target particle size of d_{80} = 75 μm. Sulfur quantification was performed via the tube furnace roasting technique, while the concentrations of other elements were determined using Inductively Coupled Plasma Optical Emission Spectrometry (ICP-OES). Reported total elemental contents represent stoichiometrically derived values based on analytical data.

3.3. Locked–cycle flotation tests

As presented in Table 17, the first experiment yielded a concentrate with a gold grade of 27.45 ppm, corresponding to a recovery rate of 81.90%. These results demonstrate effective gold enrichment under the applied test conditions.

In the second locked-cycle test, the exclusion of MaxGold900, while reducing overall gold recovery efficiency, resulted in an improved concentrate grade. This outcome suggests that MaxGold900, despite enhancing recovery, may adversely affect selectivity. As detailed in Table 18, the test produced a concentrate with a gold content of 32.03 ppm and an associated recovery of 65.48%.

Following the initial two experiments, which did not attain the desired grade and recovery targets, the results of the third locked-cycle flotation test—conducted to enhance performance—are presented in Table 19.

After optimizing the flowsheet and incorporating the Knelson concentrator into the flotation circuit, a concentrate exhibiting a gold grade of 57.21 ppm was achieved, with a corresponding recovery of 76.48%. The comparative performance of the three locked-cycle tests is depicted in Fig. 7.

Table 17. Results of locked-cycle test-1

Product	Mass, %	Grade		Recovery, %	
		Au, g/t	S, %	Au	S
Concentrate	8.41	27.45	10.70	81.90	87.08
Tailing	91.54	0.56	0.15	18.12	12.94
Total	100.00	2.82*	1.03*	100.00	100.00

* Calculated

Table 18. Results of locked-cycle test-2

Product	Mass, %	Grade		Recovery, %	
		Au, g/t	S, %	Au	S
Concentrate	5.30	33.62	19.25	65.39	82.20
Tailing	94.70	0.95	0.23	34.61	17.80
Total	100.00	2.68*	1.24*	100.00	100.00

* Calculated

Table 19. Results of the Locked-Cycle Test with Integrated and Optimized Knelson Concentrator (Test – 3)

Product	Mass, %	Grade		Recovery, %	
		Au, g/t	S, %	Au	S
Concentrate	3.69	57.21	23.49	76.48	82.47
Tailing	96.27	0.68	0.19	23.52	17.53
Total	100.00	2.76*	1.05*	100.00	100.00

* Calculated

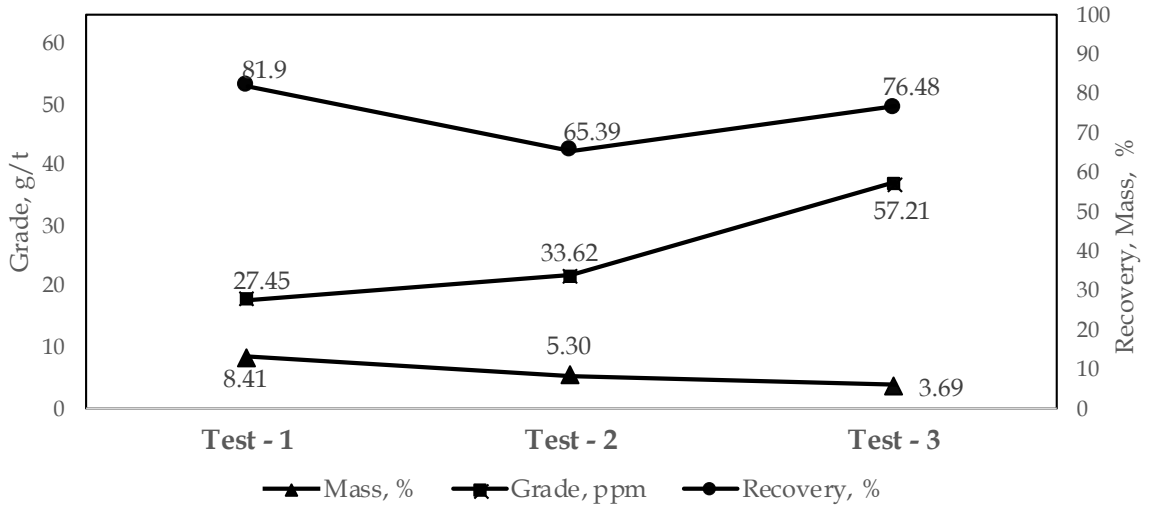

Fig. 7. Comparison table of locked-cycle tests results

3.4. Bottle–roll test results

The tailings from the first locked-cycle test contained 0.558 ppm gold. Bottle roll tests were conducted at three different particle size fractions (d_{80}: 106, 75, and 53 microns) to recover residual gold from the tailings. Each test utilized 750 grams of solids with a slurry density of 40%, and a cyanide dosage of 1 g/L was maintained. The leaching duration was 48 hours. Results presented in Fig. 8 demonstrate gold recoveries ranging from 10% to 13%, with recovery decreasing as particle size diminished.

Tailings from the second locked-cycle flotation test, with a gold content of 0.945 g/t, were subjected to bottle tests at d_{80} particle sizes of 106 and 45 microns. All other experimental parameters remained consistent with those of the previous tests. The results, illustrated in Fig. 9, show that gold recovery ranged from 10% to 10.5% after 96 hours of leaching. The absence of pre-oxidation treatment did not yield any significant improvement in gold recovery.

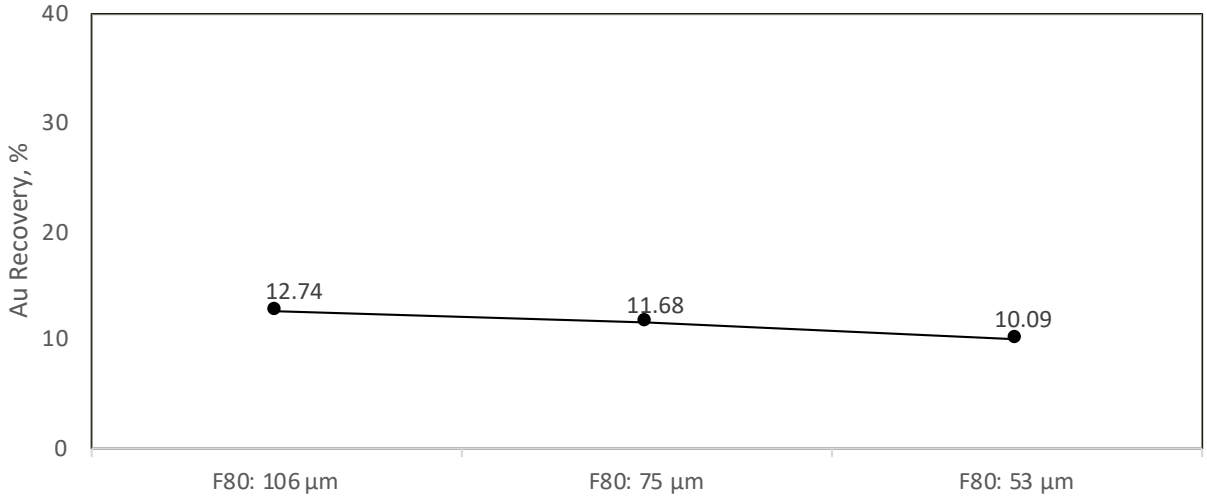

Fig. 8. First locked-cycle test's bottle-roll tests results in 48 hours

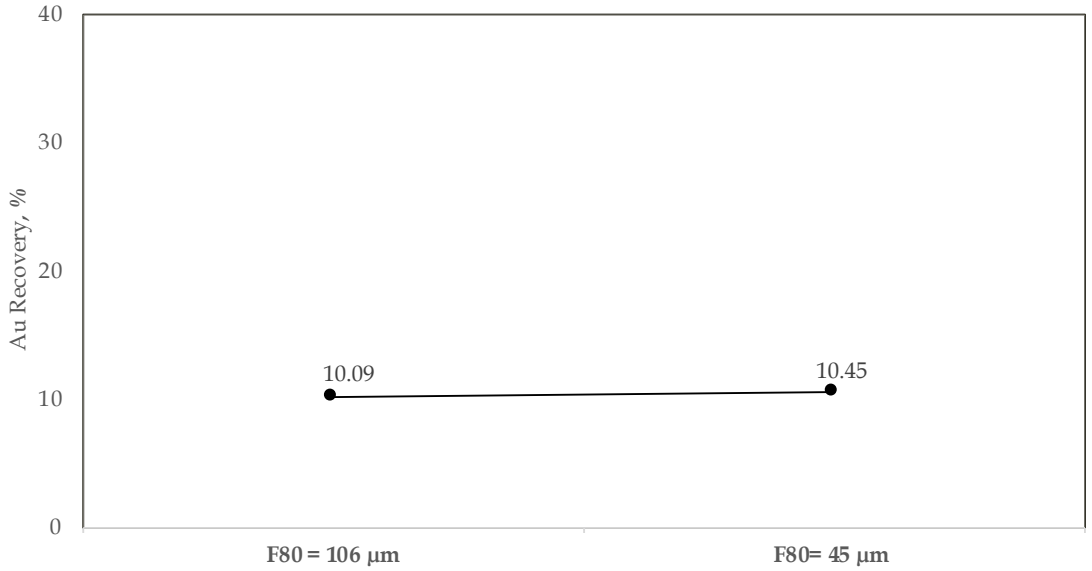

Fig. 9. Second locked-cycle test's bottle-roll tests results in 96 hours

4. Conclusions

Laboratory-scale flotation, Knelson concentrator, and bottle-roll tests were conducted to produce a concentrate with a high gold grade. Various parameters were evaluated in the flotation tests, and locked-cycle flotation tests were performed under optimal conditions. As a result, advanced methods such as the middling circuit and Knelson concentrator have been integrated into the flotation flowsheet. Experimental results indicate that clay and silicate minerals can be effectively removed using a Knelson concentrator, while metal recovery is maintained. Integrating the Knelson concentrator into the locked-cycle flotation circuit enables the production of a high-grade concentrate and reduces gold losses. The increase in gold grade, along with a reduction in concentrate mass, helps decrease the overall cost per ton of gold recovery via pyrometallurgical methods.

However, bottle test results on flotation tailings were unsatisfactory. Although flotation and cyanidation methods were applied to improve gold recovery, significant enhancement was not achieved without pre-oxidation of the sulfide minerals in the ore.

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