Journal of Environment and Sustainable Mining Vol.1 No.3 (2025) 69-82

University of Sistan and Baluchestan

Journal of
Environment and Sustainable Mining

Iranian Society of Mining Engineering
(IRSME)

Technical and Economical Evaluation of Concentration, Oxidation and Gold Extraction from Sulfide Ore of Chah-Zard

Hassan Hosseini Bohluli[1], Mohammad Reza Khalesi[1*], Nazanin Deyhimi[2] and Zahra Sadat Abdollahi Dooraki[3]

1 Department of Mining Engineering, Tarbiat Modares University, Tehran, Iran.
2 Day Yazd Industrial and Mining Company, Yazd, Iran.
3 Talavaran Company, Tehran, Iran.

ARTICLE INFO

Article type:
Case Study

Article history:
Received: 2025-07-25
Received in revised form:
2025-08-25
Accepted: 2024-09-22
Available online: 2025-09-29

Keywords:
Chah-Zard sulfide ore,
Diagnostic leaching,
Flotation,
Oxidation,
Economic evaluation.

ABSTRACT

Chah-Zard is a refractory gold ore mine that yields less than 20% recovery in direct cyanidation, due to the encapsulation of a significant portion (80%) of gold within pyrite. This study aims to identify an economic solution for the processing of this ore. Initially, the flotation process was performed to obtain a high-grade concentrate. The optimization of the flotation process, considering four factors, was examined through 28 experiments. Four alkaline oxidation experiments, three acidic oxidation tests, and three roasting tests were conducted. Flotation experiments achieved up to 80% gold recovery, aligning with the diagnostic leaching results. This recovery was achieved with a 137.5 g/t collector, 525 g/t activator, and 250 g/t dispersant at a particle size of 75 microns (d80). Gold dissolution recovery in the cyanidation of roasted concentrate (650°C and 60 minutes) reached 85%. Optimal operating conditions in the alkaline oxidation process were 70°C, 24 hours, and 35 kg/t sodium hydroxide, yielding 80% gold dissolution recovery on whole ore. In the acidic oxidation process on flotation concentrate with 200 kg/t nitric acid at 60°C and a 5-hour residence time, 68% gold dissolution recovery was achieved. Economic evaluation of the processes based on operational costs indicated that the break-even point for gold grade of the feed for the flotation-roasting process with 68% total gold recovery is 1.2 g/t, for the alkaline oxidation process with 80% total gold recovery is 2.2 g/t, and for the flotation-acidic oxidation process with 54% total gold recovery is 4.2 g/t.

Cite this article: Hosseini Bohluli, H., Khalesi, M. R., Deyhimi, N. and Abdollahi Douraki, Z. S. (2025). Technical and Economical Evaluation of Concentration, Oxidation and Gold Extraction from Sulfide Ore of Chah-Zard. *Journal of Environment and Sustainable Mining*, 1(3), 69-82. https//doi.org/10.22111/jesm.2025.49593.1018

Publisher: University of Sistan and Baluchestan.
DOI: https//doi.org/10.22111/jesm.2025.49593.1018

---

[*] Corresponding author: Mohammad Reza Khalesi
E-mail address: mrkhalesi@modares.ac.ir

1 .Introduction

The depletion of oxide gold reserves, the increase in gold prices, and the rising demand for this precious metal have led to attention towards refractory gold ores that were previously underutilized due to their low grade and structural complexity [1, 2]. A typical example of refractory gold ores, characterized by less than 80% recovery in direct cyanidation, is the association of gold with various sulfide minerals [3, 4]. Selecting an optimal processing scheme for sulfide gold ore, greatly depends on the mineralogy of the gold ore being processed [5]. Accurate characterization of ore is crucial for identifying the modes of gold occurrence and associated gold-bearing minerals, as well as for assessing the refractory behavior of gold during cyanidation. This information enables the determination of the ore's compatibility with various processing options [6]. Various analytical techniques, such as Optical Microscopy, Scanning Electron Microscopy (SEM), Electron Microprobe Analysis (EMPA), Proton-Induced X-ray Emission, and Dynamic Secondary Ion Mass Spectrometry (D-SIMS), are employed for characterizing and identifying gold ores [7]. Diagnostic Leaching is also used as an analytical tool to determine the distribution of gold in different minerals of an ore, before selecting a suitable process. This method involves sequential acid leaching and selective destruction of specific minerals to release gold, which is then followed by cyanidation to recover the liberated gold [8]. After identifying the gold-bearing minerals, concentrating these minerals and producing a gold concentrate (for instance by flotation) for subsequent pre-processing steps (such as oxidation) is one of the conventional options, especially for refractory sulfide gold ores.

Industrial-scale flotation for gold concentration started in the early 1930s with the introduction of water-soluble collectors, particularly xanthate and dithiophosphate collectors [9]. When gold is very fine (less than 10 microns) and closely associated with sulfide minerals, cyanidation performance tends to be poor. Flotation may be used as a pre-concentration step to allow more expensive processing operations on a smaller portion of the material. In this scenario, the goal of the flotation circuit is typically to maximize the recovery of all gold-bearing minerals [10] and the operational condition favorable to float the sulfide mineral is favorable to gold recovery [11]. The most significant challenges in the flotation of gold ores are related to the flotation of free and natural gold. The flotation of free gold is greatly affected by physical factors such as gold particle shape and size, as well as foam stability [9]. More recently, the flotation of disturbing elements and matters such as carbonaceous matters prior to gold extraction has been investigated [12].

For sulfide ores, oxidation may be necessary to dissolve some or all sulfide components to expose the gold or deactivate the surfaces of sulfide minerals. Available oxidation methods are divided into two main categories: hydrometallurgical and pyrometallurgical. Hydrometallurgical methods, ranging from simple pre-oxidation to complex processes, have been developed over the past decades and are suitable alternatives to roasting for many refractory ores and concentrates [7]. Commercially, roasting [13-15], pressure oxidation [16-18], chemical oxidation [19], and biological oxidation [20-23] are among the processes used for sulfide oxidation.

This research presents the characterization and preliminary technical and economic feasibility studies for processing the sulfide ore of the Chah-Zard gold mine. Located 90 km southwest of Yazd City, Iran, a 300-ton-per-day oxide gold cyanidation plant is operational at the mine. However, due to the small volume and low grade of the oxide part, extraction and processing of gold from the sulfide section of the ore has been considered. Given the ore body shape the need for significant waste removal or to use of underground mining and the high extraction costs, the economic viability of gold extraction from the sulfide part is highly questionable. This study aims to evaluate viable industrial options, optimize them using experimental design and statistical methods, calculate the highest possible recovery, and then perform preliminary economic calculations to identify and select the most suitable option.

2. Materials and Methods

2.1. Sample Identification

A representative sample was collected from exploratory drilling boreholes. The sample was crushed to 3/2 mm using a jaw crusher and then ground to the desired particle size using a ball mill for flotation and oxidation tests. To determine the gold grade, the representative test sample underwent fire-assay analysis, and the solution sample was analyzed using atomic absorption spectroscopy (AAS). XRD and XRF analyses were used for identifying gold-bearing phases, and ICP-MS analysis was performed to determine element contents (Supplementary Information Table S1-S3).

2.2. Flotation

Flotation tests were performed using a Denver laboratory flotation cell. The pulp volume for flotation was 1.5 liters, and 500 grams of feed were used per test. The stirring speed was set at 1250 rpm, the pulp temperature was maintained at room temperature (25°C), and the solid percentage was subsequently adjusted to 30%. The pulp pH was neutral (6-7) at the beginning of the flotation reagent addition and was adjusted using lime. Potassium amyl xanthate was used as the collector, copper sulfate as the activator for sulfide particles containing gold, and sodium silicate as the dispersant according to the levels introduced in Table 1. pH, preparation time, and the type and concentration of frother (50 g/t MIBC) were kept constant for all tests. Minimum levels for the factors were determined in preliminary tests, and maximum levels were based on initial economic evaluation.

Table 1. Factors and Levels for Flotation of Sulfide Gold Ore.

Factor	Abbreviation	Level Low	Level High
Collector (g/t): Potassium Amyl Xanthate	A	100	400
Activator (g/t): Copper Sulfate	B	0	700
Dispersant (g/t): Sodium Silicate	C	0	1000
Particle Size (microns) d80	D	50	150

To optimize flotation conditions, experimental design was performed using Design-Expert software with the response surface methodology (RSM) and central composite design (CCD) at five levels with four replicates at the central point of the factors. According to Table 2, after 28 experiments, the optimal point was found, and 2 replicates were performed under the optimal conditions obtained and compared with the software's suggested optimal point. The responses of the experimental design included gold recovery in the concentrate and weight recovery of the concentrate, as defined in Equations 1 and 2.

$$\text{Weight Recovery (\%)} = \frac{\text{Concentrate Mass (g)}}{\text{Feed Mass (g)}} \times 100 \tag{1}$$

$$\text{Gold Recovery (\%)} = \frac{\text{Concentrate Mass (g)} \times \text{Gold Grade in Concentrate (g/t)}}{\text{Feed Mass (g)} \times \text{Gold Grade in Feed (g/t)}} \times 100 \tag{2}$$

Due to the high value of gold, gold recovery is much more important than weight recovery. Therefore, the experiment that achieved the highest gold recovery was considered the optimal result. This experiment was selected to prepare the concentrate for roasting tests.

Table 1. Conditions of Roasting Tests.

Test No.	1	2	3
Temperature (°C)	600	650	800
Time (minutes)	60	60	60
Aeration	None	None	None
Sample Weight before Treatment (gr)	210	120	160

2.3. Roasting

The conditions of Table 2 were used for roasting tests. The roasted concentrate was then subjected to cyanidation tests under the conditions provided in sections 2-6. Gold dissolution recovery and weight reduction of the concentrate after roasting were the responses considered to evaluate the roasting method. The roasting tests were conducted in an alumina refractory container.

2.4. Alkaline Oxidation

Alkaline oxidation was performed at atmospheric pressure by industrial sodium hydroxide. According to studies on alkaline oxidation, temperature and pH are important factors in this process [24-27]. Single-factor experiments led to the experimental conditions in Table 3 to achieve the highest possible recovery. However, due to the high consumption of sodium hydroxide, replacing it with another reagent like lime to reduce operating costs was also examined. Studies have shown that using lime in sulfide oxidation physically encapsulates pyrite grains with a coating layer, resulting in low oxidation and reduced gold recovery [28]. Therefore, lead nitrate was used to overcome this phenomenon [29] (Table 4). All tests were conducted at 70°C, 33% solids, 100% passing 45 microns, 24 hours oxidation time, and with aeration.

Table 2. Conditions of Initial Alkaline Oxidation.

Particle Size d_{100} (microns)	Oxidation Time (hours)	Oxidation Temperature (°C)	Solid Percentage	Sodium Hydroxide Consumption (98% purity, kg/ton)
45	24	70	33	35

Table 3. Conditions of Alkaline Oxidation for Reducing Sodium Hydroxide Consumption.

Oxidation Conditions	Test 1	Test 2	Test 3	Test 4
Sodium Hydroxide Concentration (98% purity, g/ton)	10000	5000	10000	5000
Lime Concentration (g/ton)	25000	30000	25000	30000
Hydrogen Peroxide Concentration (g/ton)	0	0	1000	1000
Lead Nitrate Concentration (g/ton)	0	0	1000	1000

2.5. Acidic Oxidation

Although an initial oxidation test with 4 M nitric acid (775.5 kg of 65% acid per ton of ore) at 70°C showed that achieving 95.6% cyanidation recovery was possible, three tests were conducted with 40, 100, and 200 kg of 65% nitric acid per ton of ore to reduce acid consumption and make it more feasible. The oxidation time was 5 hours, the particle size (d100) was 45 microns, the temperature was 60-70°C, and the solids percentage was 33.

2.6. Cyanidation

After performing concentration and oxidation operations using various methods, the samples underwent cyanidation. All cyanidation experiments were conducted for 24 hours, with a solid percentage of 33 and a dosage of 3000 grams per ton of sodium cyanide.

Table 4. Flotation Test Plan for Sulfide Sample.

Test No.	Collector (g/t)	Activator (g/t)	Depressant (g/t)	d80 (microns)	Gold Recovery (%)	Weight Recovery (%)
1	225	350	500	50	72.6	26.7
2	50	350	500	100	56.31	14.2
3	225	350	0	100	59.5	16.1
4	225	350	500	100	63.01	17.4
5	137.5	175	250	75	54.53	14.6
6	225	700	500	100	65.25	16.4
7	225	350	500	150	51.18	16.1
8	312.5	525	750	125	66.66	19.9
9	312.5	175	750	75	64.4	19.1
10	137.5	525	750	125	62.58	19
11	225	350	1000	100	61.09	16.4
12	312.5	525	750	75	58.44	13.3
13	137.5	175	750	125	56.25	13.6
14	225	0	500	100	64.83	16
15	137.5	525	250	125	68.24	19.4
16	225	350	500	100	63.55	18
17	312.5	175	750	125	68.98	16.9
18	137.5	175	750	75	65.71	20.6
19	312.5	175	250	75	65.78	20.5
20	312.5	175	250	125	69.41	21.3
21	137.5	175	250	125	61.27	20.7
22	225	350	500	100	78.8	26.2
23	137.5	525	250	75	81.93	26.3
24	137.5	525	750	75	65.77	24.6
25	312.5	525	250	125	69.37	21.5
26	400	350	500	100	70.71	22.4
27	225	350	500	100	71.12	23.7
28	312.5	525	250	75	73.22	28.5

2.7. Economic Evaluation

To perform the preliminary economic evaluation, the operating costs of each pre-processing method for this refractory sulfide ore were examined separately and combined with the associated crushing and cyanidation costs. In methods where flotation is included, the crushing cost covers the total input tonnage, while oxidation and cyanidation costs apply to the concentrate (about 30% of the input load). Revenue is based on the multiplication of the recoveries obtained by different process stages and the price of gold (e.g., flotation, roasting, and leaching). Recovery in the adsorption, elution, and electrowinning stages is considered 100%. Given the existing 300-ton-per-day oxide cyanidation plant at the mine, cyanidation costs were calculated based on the current actual costs, and new concentration and oxidation section costs were estimated based on laboratory consumption and estimated energy and labor costs. The break-even point was presented as zero profit for the feed of each proposed process.

3. Results and Discussion

3.1. Analytical Results

The gold grade in the studied ore was estimated at 1.5 g/t. The main phases included quartz, feldspar, albite, and muscovite, and pyrite was identified as a minor phase. Diagnostic leaching with four acids (sequentially with hydrochloric acid, sulfuric acid, nitric acid, and hydrofluoric acid) revealed that 12.51% of the gold was free, 1.05% was encapsulated in copper, zinc, and unstable pyrite sulfides, 80.14% encapsulated in pyrite, and 6.3% in silicates. Thus, the ore is classified as a refractory gold ore of the sulfide type. Elemental analysis showed that interfering elements like copper, antimony, and arsenic were negligible.

3.2. Flotation Test Results

The results of 28 conducted tests are presented in Table 5. The analysis of variance (ANOVA) for the gold recovery response indicates that besides the main factors, interaction effects also significantly impact gold recovery. Weight recovery in the tests was less than 30%, and due to its lesser importance, it is not analyzed here (Supplementary Information Table S4). The regression equations (real and not coded) for gold recovery and weight recovery responses are provided in Equation 3 and Equation 4, respectively. A, B, C, and D are introduced in Table 1.

$$\text{Gold Recovery (\%)} = 74.22 - 0.017 \times A + 0.067 \times B - 0.354 \times D - 0.0002 \times AB + 0.0011 \times AD - 0.00002 \times BC \quad (3)$$

$$\text{Weight Recovery (\%)} = 20.05 + 0.0188 \times A + 0.0156 \times B - 0.0956 \times D \quad (4)$$

Table 5. ANOVA for Gold Recovery Response.

Source	Sum of Squares	Degrees of liberation	Mean Squares	F-value	p-value	Significance
Model	680.09	6	113.35	4.68	0.0040	Significant
A	155.33	1	155.33	6.41	0.0198	Significant
B	118.21	1	118.21	4.88	0.0390	Significant
D	160.90	1	160.90	6.64	0.0180	Significant
AB	177.89	1	177.89	7.34	0.0135	Significant
AD	122.78	1	122.78	5.07	0.0358	Significant
BC	190.98	1	190.98	7.88	0.0109	Significant
Error	484.49	20	24.22	-	-	
Lack of Fit	406.99	17	23.94	0.93	0.6156	Not significant

The analysis of normal probability plots of residuals and the predicted versus observed gold recovery values (Supplementary Information Figure S1), along with the ANOVA results, validate the proposed model (Table 6). The factors of collector and activator concentration directly affect gold recovery. Increasing the concentration of these factors enhances the hydrophobicity of the gold-bearing sulfide mineral and thus improves flotation recovery. Additionally, reducing particle size and liberating more gold-bearing sulfide particles increases flotation recovery in the concentrate (Supplementary Information Fig. S2).

The two-factor interaction of particle size with the collector is a significant factor in the model. At 75 microns, gold recovery does not significantly vary with different collector concentrations. However, at 125 microns, higher collector concentrations are needed to increase gold recovery, likely due to incomplete liberation of the sulfide particles, requiring more hydrophobic species formation on the particle surface (which constitutes a small portion of the total surface area). Similarly, the two-factor interaction of the collector and activator is significant. Higher activator concentrations enhance the redox potential, creating an oxidizing environment for thiol collectors, thus improving flotation performance. This effect is more pronounced with lower collector concentrations when more copper sulfate is used (Supplementary Information Fig. S3-b) [30]. The interaction of the activator and dispersant is also significant. Increasing activator concentration requires precise control of dispersant (sodium silicate) consumption and pH, as sodium silicate, by alkalizing the flotation environment, reduces copper adsorption on sulfide surfaces (Supplementary information Fig. S3-c) [30].

The maximum gold recovery based on the regression equation (Equation 3) is achieved under the operational conditions of Table 7. Two replicate tests under these conditions yielded an average gold recovery of 80.2% and an average weight recovery of 29.73%, which, at a 95% confidence level, do not significantly differ from the predicted recovery, demonstrating the model's accuracy in identifying the optimal point. The conditions of Table 7 were used to prepare samples for oxidation tests. Sulfur grade results for the feed, flotation concentrate, and flotation tailings were 1.89%, 3.88%, and 0.56% respectively. Sulfur analysis indicates that 76.37% of the feed sulfur entered the flotation concentrate.

Table 6. Conditions and Results of Optimal Flotation Test.

Concentration of collector (g/t)	Concentration of activator (g/t)	Concentration of depressant (g/t)	Particle size (microns)	Gold recovery (%)	Weight recovery(%)
137.5	525	250	75	80.2	29.73

3.3. Roasting Test Results

After roasting and removal of 85.57% of sulfur and cyanidation of the roasted sample, the gold dissolution recovery at the three tested temperatures (600, 700, and 800 ^{0}C) did not significantly differ, with values consistently around 82 ± 1%. The inability to dissolve about 20% of the gold in the roasted concentrate during cyanidation roughly corresponds with the 25% sulfur remaining after roasting.

3.4. Alkaline Oxidation Results

Gold dissolution recovery in the initial test with 35 kg/t sodium hydroxide reached 80%, but subsequent tests aimed at reducing sodium hydroxide consumption achieved a maximum recovery of 46% (43.5%, 46%, 41%, and 38% recovery for Tests 1 to 4 in Table 5, respectively). pH monitoring during oxidation in Test 2 and the 80% recovery test (initial test) (Fig. 1) showed that during the first 18 hours of oxidation in the initial test, the pH remained at 13. In Test 2, the pH was above 12 for only the first 9 hours. pH is a crucial factor in alkaline oxidation, and maintaining it higher than specific values requires a specific amount of sodium hydroxide. Therefore, lower sodium hydroxide consumption significantly reduces oxidation and subsequent gold dissolution recovery in cyanidation [31].

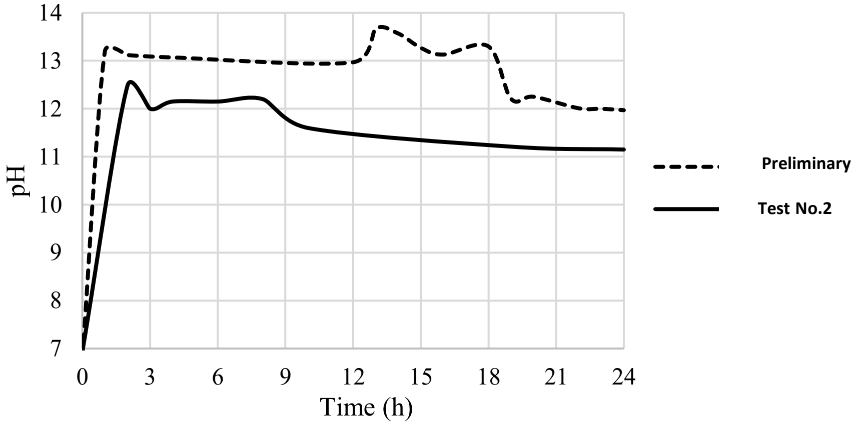


Fig. 1. Changes in pH during the oxidation process over time.

4. Oxidation Results

The results of these tests are presented in Table 8. The results show that at a concentration of 100 g/L (200 kg of 65% nitric acid per ton of concentrate), a maximum gold dissolution recovery of 68% was achieved, which is low relative to the costs incurred. Lower nitric acid consumption significantly reduced recovery, with recovery dropping by approximately 3.5 times when acid consumption was halved.

Table 8. Acidic Oxidation Results.

Test No.	Acid Concentration (65%, kg/ton)	Gold Dissolution Recovery (%)
1	40	12.9
2	100	19.35
3	200	68.75

5. Economic Evaluation Results

5.1. Flotation-Roasting-Cyanidation Process

Flotation tests achieved the maximum gold recovery was 80%. In roasting and cyanidation tests, up to 85% of the gold in the flotation concentrate was dissolved, resulting in a total gold recovery of 68%. The prices of flotation chemical reagents for collector, activator, depressant, and frother were assumed as $4, $3.9, $0.9, and $3 per kilogram, respectively. Table 10 lists the operating costs related to the cyanidation process, and Table 9 lists the operating costs related to the flotation-roasting process per ton of ore.

Table 8. Costs of the Gold Cyanidation Plant per Ton of Ore.

Row	Description	Unit	Quantity	Unit Cost (USD)	Cost per Ton of Ore (USD)
1	Cyanide consumption (purity of 40%)	kg	2	1.6	3.2
2	Lime consumption	kg	3	0.02	0.058
3	Electricity consumption	set	1	1440	0.16
4	Water consumption	set	1	4800	0.53
5	Employee salaries, insurance, and benefits	person	80	320	2.84
6	Repair and Maintenance	set	1	38400	4.27
7	Depreciation	set	1	25600	2.84
8	Financial expenses	set	1	51200	5.69
9	Extraction costs	set	1	76985.6	8.55
10	Crushing costs	set	1	29161.6	3.24
	Total operating costs (per ton)	-	-	31.39	

Table 9. Costs for the Flotation-Roasting Process.

Row	Description	Unit	Quantity	Cost per Ton of Ore (USD)
1	Consumption of Frother, Collector, and Co-Collector	g	1000	7.36
2	Sulfuric Acid Consumption	g	500	0.38
3	Lime Consumption	kg	5	0.09
4	Water Consumption	m³	2	0.9
5	Filter Cloth Consumption	unit	0.01	0.19
6	Repair and Maintenance Costs	set	1	2.24
7	Costs of Feed, Accommodation, Transportation, etc.	set	1	0.64
8	Costs of Contract Management and Consultation	set	1	0.19
9	Diesel Consumption in Drying and Roasting Furnaces	liter	3	0.1
10	Energy Consumption Costs	set	1	0.64
11	Employee Salaries and benefits	set	1	3.2
	Total Operating Costs (per ton)			15.94

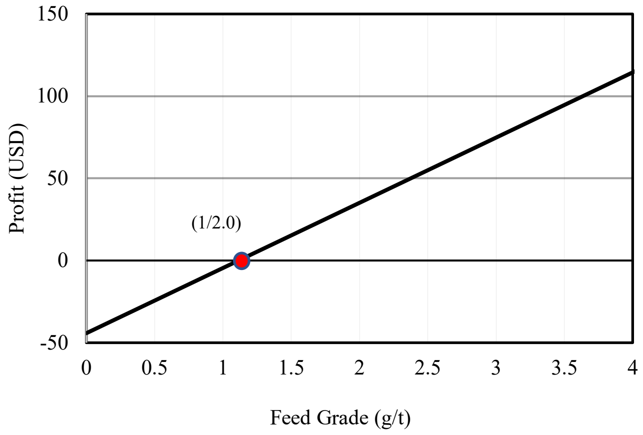


Fig. 2. Chart of Profit per Ton of Feed versus Grade in the Flotation-Roasting Process.

As shown in Fig. 2, the break-even grade is estimated at 1.2 g/t. In other words, with a feed containing 1.2 g/t gold, the revenue equals the operating costs incurred, and with a feed gold grade above 1.2 g/t, the process becomes economical. It is important to note that this preliminary estimate is based solely on operating costs, and capital investment costs for constructing the flotation and roasting units are not considered. However, in gold processing, due to the high value of gold, payback periods are usually less than one year, and therefore capital costs do not change the economic evaluations very much.

5.2. Alkaline Oxidation-Cyanidation Process

In tests conducted to evaluate this method, gold recovery of 80% was achieved with 35 kg/t sodium hydroxide consumption (98% purity) at 70°C and 24 hours residence time (total gold recovery was 80%). The economic estimate is presented in Fig. 3 for ores with varying feed grades. According to Fig. 3, the break-even point is estimated at 2.2 g/t, and with a feed gold grade above 2.2 g/t, the process becomes economical. Given the 1.5 g/t grade of the representative mine sample, alkaline oxidation of the entire feed is not economical.

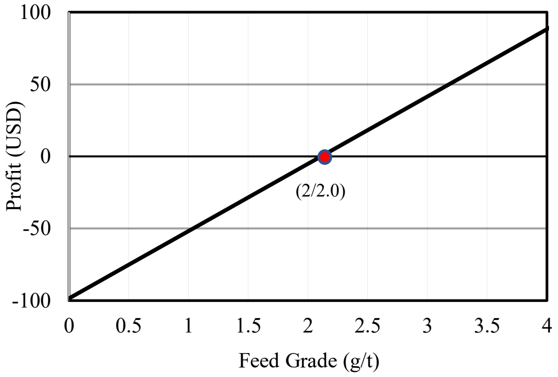


Fig. 3. Chart of Profit per Ton of Feed versus Grade in the Alkaline Oxidation Process.

5.3. Flotation-Acidic Oxidation-Cyanidation Process

The economic estimate of this method with a 100 g/L acid concentration and a 68% gold dissolution recovery is presented in Fig. 4 for ores with varying feed grades (total gold recovery was 54%). Acidic oxidation of the flotation concentrate with nitric acid is not economical even for high-grade feeds (4 g/t). Higher acid consumption increases gold recovery to 95%, but costs also rise significantly.

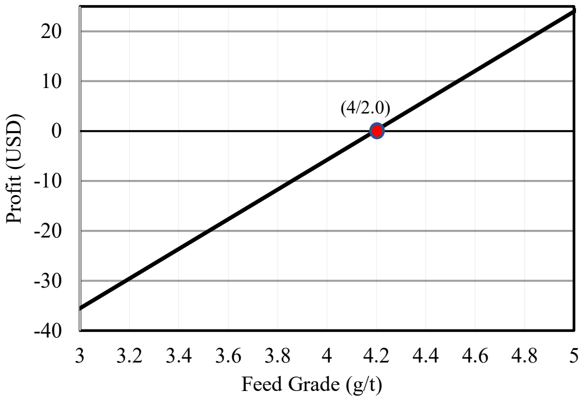


Figure 4. Chart of Profit per Ton of Feed versus Grade in the Flotation-Acidic Oxidation Process.

6. Conclusion

The sulfidic ore from Chah-Zard, with less than 20% dissolution recovery in direct cyanidation, is classified as highly refractory gold ore. Characterization of this ore indicates pyrite as the sulfide mineral carrying gold, with 80% of the gold in the sulfide phase. Based on the findings, pre-processing of the ore before cyanidation was necessary. The flotation method was evaluated for ore concentration to obtain a high-grade concentrate. With 137.5 g/t potassium amyl xanthate collector, 525 g/t copper sulfate activator, 250 g/t sodium silicate dispersant at 75 microns (80% passing), gold recovery reached 80%, and weight recovery was 29.73%. The high-grade flotation concentrate was subjected to roasting trials to assess gold dissolution in the subsequent cyanidation process. It was found that with a residence time of 60 minutes at 800°C, 75% of sulfur was removed, and gold dissolution recovery reached 80%.

The alkaline oxidation method was also evaluated for the entire ore. In the initial test, using 35 kilograms per ton of sodium hydroxide (with 98% purity), at a temperature of 70 degrees Celsius and a retention time of 24 hours, gold dissolution recovery reached 80%. However, replacing the alkaline reagent with another agent, such as lime, resulted in a significant reduction in gold dissolution in cyanidation, attributed to the low pH levels observed during the initial oxidation phase. Therefore, the use of sodium hydroxide is deemed unavoidable. Acidic oxidation proved to be inefficient for this ore, due to its high acid consumption for achieving sufficient gold recovery.

Economic evaluations were conducted for each method. All cyanidation costs were calculated based on the actual costs of the existing 300-ton-per-day oxide processing plant in Yazd, and the costs of each method were estimated based on laboratory consumption. The operating costs of the flotation-roasting process are lower compared to the hydrometallurgical alkaline or acidic oxidation processes, but the gold recovery in the flotation-roasting process is significantly lower. Roasting requires sulfur gas recovery due to sulfur emissions. It is noteworthy that the amount of gold entering the tailings in the flotation-roasting process is much higher than in the alkaline oxidation process. Therefore, if the feed grade is between 1.2 to 2.2 g/t, flotation concentration followed by roasting or alkaline oxidation is required. However, higher-grade feed can be processed directly by alkaline oxidation followed by cyanidation.

Acknowledgments

We sincerely thank the Day Yazd Industrial and Mining Company for their financial support, assistance with sample preparation, and provision of cost data. Additionally, we appreciate the financial and logistic support provided by Talavaran Pars Faravar Company.

Ethical Considerations

The authors avoided data fabrication, falsification, and plagiarism, and any form of misconduct.

Funding

This research did not receive any specific grant from funding agencies in the public, commercial, or not-for-profit sectors.

Conflict of Interest

The authors declare no conflict of interest.

References

[1] Adams, M. (2016). Overview of the gold mining industry and major gold deposits. In: Adams, M. D. (Ed), Gold Ore Processing, (Second Edition), Elsevier, p. 25-30. https://doi.org/10.1016/B978-0-444-63658-4.00002-5

[2] Asamoah, R.K. (2021). Specific refractory gold flotation and bio-oxidation products: Research overview. *Minerals*. 11(1): 93. https://doi.org/10.3390/min11010093

[3] Fraser, K.S. Walton, R.H. and Wells, J.A. (1991). Processing of refractory gold ores. *Minerals Engineering*. 4(7): 1029-1041.

[4] Cilek, E.C., Tuzci, G. (2021). Flotation behavior of native gold and gold-bearing sulfide minerals in a polymetallic gold ore. *Particulate Science and Technology*. 40(5): 558-566. https://doi.org/10.1080/02726351.2021.1975334

[5] Lee, S., Gibson, C.E., Ghahreman, A. (2021). Flotation of Carbonaceous Matter from a Double Refractory Gold Ore: The Effect of MIBC on Flotation Performance and Kinetics. *Minerals*. 11(9): 1021. https://doi.org/10.3390/min11091021

[6] Goodall, W.R. and Scales, R.J. (1994). The flotation of gold-bearing ores - a review. *Minerals Engineering*. 7(7): 839-849. https://doi.org/10.1016/0892-6875(94)90128-7

[7] Ngovene L.R. (2019). Refractory gold ores: characterization and implications for gold processing. Doctoral dissertation, Curtin University.

[8] Olin, E. J. (2017). Diagnostic leaching for refractory gold ores. *SRK News Issue Metallurgy & Mineral Processing*.

[9] Dunne, R. (2016). Flotation of gold and gold-bearing ores, in Gold Ore Processing., Elsevier. p. 315-338. https://doi.org/10.1016/B978-0-444-63658-4.00020-7

[10] Marsden, J. and House, I. (2006). The chemistry of gold extraction. SME. p. 238-259.

[11] John, J., Evans, C., & Johnson, N. W. (2020). The influence of lime and sodium hydroxide conditioning on sulfide sulfur behaviour in pyrite flotation. *Minerals Engineering*, 151: 106304. https://doi.org/10.1016/j.mineng.2020.106304

[12] Lee, S., Gibson, C. E., & Ghahreman, A. (2021). Flotation of carbonaceous matter from a double refractory gold ore: the effect of MIBC on flotation performance and kinetics. *Minerals*, 11(9): 1021. https://doi.org/10.3390/min11091021

[13] Qin, H., Guo, X., Tian, Q., Yu, D., & Zhang, L. (2021). Recovery of gold from sulfide refractory gold ore: Oxidation roasting pretreatment and gold extraction. *Minerals Engineering*, 164: 106822. https://doi.org/10.1016/j.mineng.2021.106822

[14] Wu, T., Shen, Z., Shi, Z., Wang, J., Qiu, Y., & Mao, S. (2025). Enhancement of gold extraction by citric acid-assisted microwave roasting of Carlin-type gold ores and sulfide/thiosulfate leaching. *Hydrometallurgy*, 231: 106410. https://doi.org/10.1016/j.hydromet.2024.106410

[15] Hapid, A., Zullaikah, S., Mahfud, M., Kawigraha, A., Azmi, M. U., Haryanto, I., & Assabumrungrat, S. (2024). Optimization of microwave-assisted roasting: Box-behnken design for oxidation of sulfide minerals and control of atmospheric sulfur in refractory gold ore pretreatment. *Case Studies in Chemical and Environmental Engineering*, 10: 100826. https://doi.org/10.1016/j.cscee.2024.100826

[16] Mystrioti, C., Kousta, K., Papassiopi, N., Adam, K., Taxiarchou, M., & Paspaliaris, I. (2024). Gold recovery from pressure oxide residues using thiosulfate. *Technical Annals*, 1(5). https://doi.org/10.12681/ta.36954

[17] Ng, W. S., Liu, Y., Wang, Q., & Chen, M. (2023). The fate of the arsenic species in the pressure oxidation of refractory gold ores: Practical and modelling aspects. *Mineral Processing and Extractive Metallurgy Review*, 44(2): 155-187. https://doi.org/10.1080/08827508.2022.2030326

[18] Zhang, L., Guo, X. Y., Tian, Q. H., Zhong, S. P., & Qin, H. (2022). Extraction of gold from typical Carlin gold concentrate by pressure oxidation pretreatment-Sodium jarosite decomposition and polysulfide leaching. *Hydrometallurgy*, 208(3): 105743. https://doi.org/10.1016/j.hydromet.2021.105743

[19] Deblonde, G. J. P., Chagnes, A., & Cote, G. (2023). Recent advances in the chemistry of hydrometallurgical methods. *Separation & Purification Reviews*, 52(3): 221-241. https://doi.org/10.1080/15422119.2022.2088389

[20] Saim, A., Akuffo, G. O., & Amankwah, R. K. (2022). Oxidation behaviour and bio-oxidation of gold-bearing sulphide ores: Oxygen capabilities and challenges. *Ghana Mining Journal*, 22(2):15-25. https://doi.org/10.4314/gm.v22i2.3

[21] McNeice, J., Marzoughi, O., Kim, R., & Ghahreman, A. (2021). Gold extraction from refractory sulfide gold concentrates: a comparison of bio-oxidation and neutral atmospheric pre-treatment and economic implications. *Journal of Sustainable Metallurgy*, 7(3): 1354-1367. https://doi.org/10.1007/s40831-021-00427-2

[22] Zhang, S., Yang, H., Ma, P., Luan, Z., Tong, L., Jin, Z., & Sand, W. (2022). Column bio-oxidation of low-grade refractory gold ore containing high-arsenic and high-sulfur: Insight on change in microbial community structure and sulfide surface corrosion. *Minerals Engineering*, 175: 107201. https://doi.org/10.1016/j.mineng.2021.107201

[23] Yang, T., Dai, M., Tang, G., Guo, Z., Yang, Y., & Jiao, H. (2024). Effect of an electro-assisted biochemical cycle reactor on bio-oxidation of gold ore. *Minerals Engineering*, 209: 108630. https://doi.org/10.1016/j.mineng.2024.108630

[24] Bidari, E, Aghazadeh, V. (2018). Alkaline leaching pretreatment and cyanidation of arsenical gold ore from the Carlin-type Zarshuran deposit. *Canadian Metallurgical Quarterly*. 57(3):283-93. https://doi.org/10.1080/00084433.2018.1444931

[25] Rego, A.J. (2018). Atmospheric alkaline pre-oxidation of refractory sulphide gold ores (Doctoral dissertation, University of British Columbia). https://doi.org/10.14288/1.0366302

[26] Mahmoodi, F., Azizi, A. (2018). Optimizing the alkaline oxidation pretreatment of a refractory gold ore using taguchi orthogonal array method. *Materials Research Express*, 5(12):126516. https://doi.org/10.1088/2053-1591/aae196

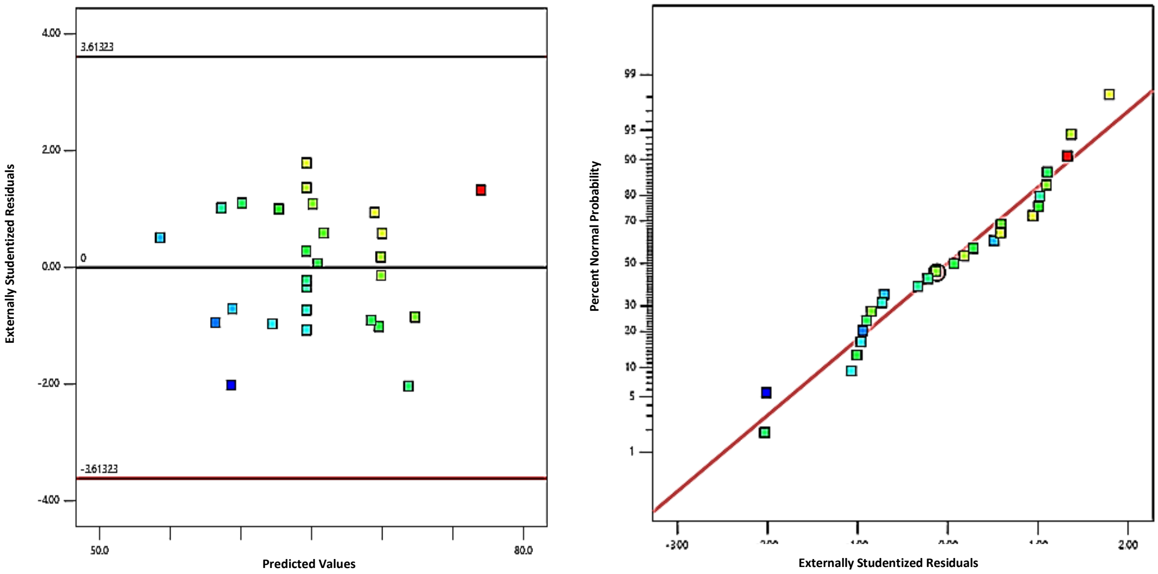
[27] Celep, O., Alp, İ., Paktunç, D., Thibault, Y. (2011). Implementation of sodium hydroxide pretreatment for refractory antimonial gold and silver ores. *Hydrometallurgy*, 108(1-2):109-14. https://doi.org/10.1016/j.hydromet.2011.03.005

[28] Li, J., Dabrowski, B., Miller, J. D., Acar, S., Dietrich, M., LeVier, K.M., Wan, R.Y. (2006). The influence of pyrite pre-oxidation on gold recovery by cyanidation. *Minerals Engineering*, 19(9):883-95. https://doi.org/10.1016/j.mineng.2005.09.052

[29] Deschenes, G., Lastra, R., Brown, J.R., Jin, S., May, O., Ghali, E. (2000). Effect of lead nitrate on cyanidation of gold ores: progress on the study of the mechanisms. *Minerals Engineering*, 13(12): 1263-1279. https://doi.org/10.1016/S0892-6875(00)00109-6

[30] Chandra, A. P., and Gerson, A. R. (2009) A review of the fundamental studies of the copper activation mechanisms for selective flotation of the sulfide minerals, sphalerite and pyrite. *Advances in colloid and interface science*, 145(1-2): 97-110. https://doi.org/10.1016/j.cis.2008.09.001

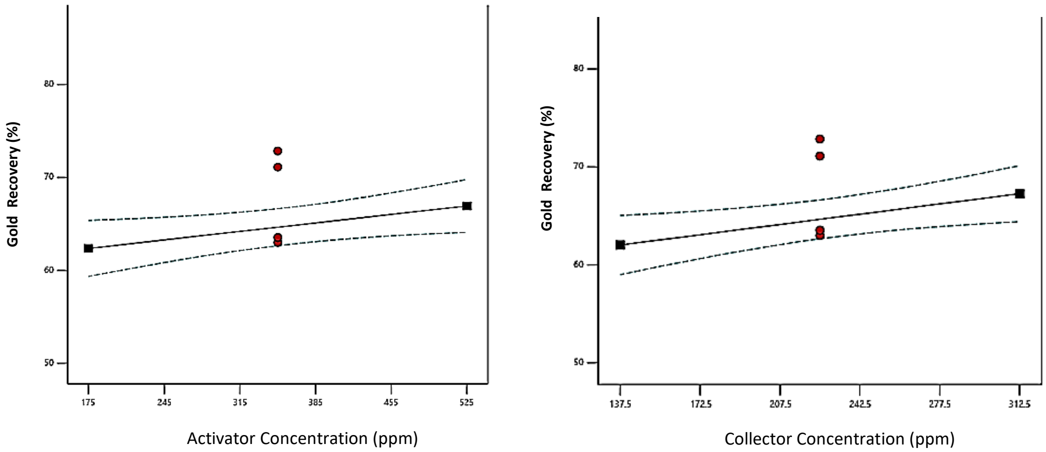
Supplementary File



B- plot of residuals versus predicted values.

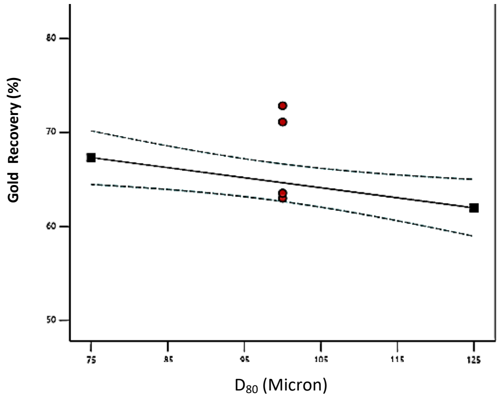
A- normal residuals plot.

Fig. S1. Charts of Gold Recovery Model Validation



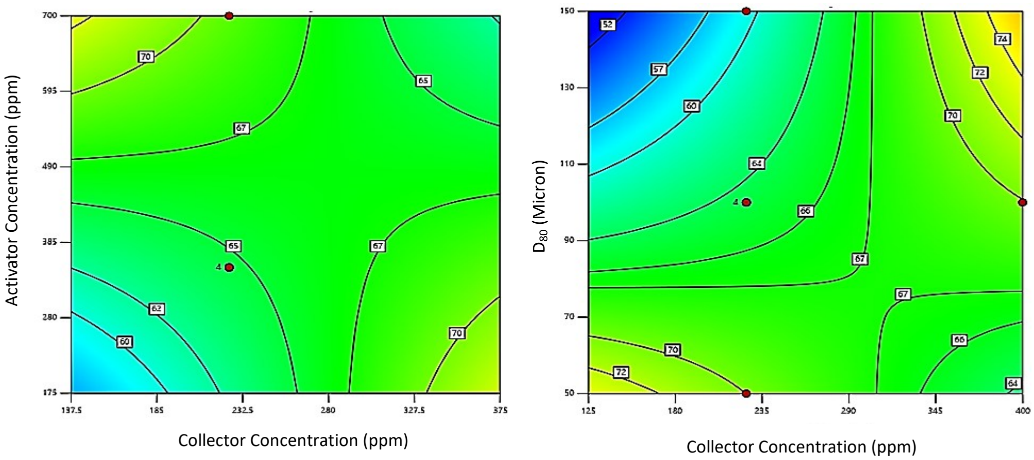
B- Plot of gold recovery changes versus activator concentration.

A- Plot of gold recovery changes versus collector concentration.



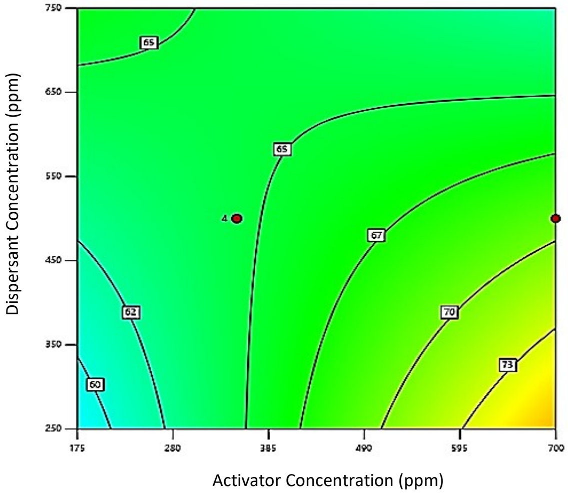
C- Plot of gold recovery changes versus particle size distribution.

Fig. S2. Variations in gold recovery relative to collector concentration, activator concentration, and particle size.



B. Gold recovery contour against collector concentration and activator concentration.

A. Gold recovery contour against collector concentration and particle size distribution.



C. Gold recovery contour against activator concentration and dispersant concentration.

Fig. S3. Contour related to the interactive effect of dual factors on gold recovery in the concentrate.

Table S1 - XRD Analysis Results.

Main Phases	Formula	Mineral Name	Minor Phases	Formula	Mineral Name	Very Minor Phases	Formula	Mineral Name
Quartz	SiO2	Quartz	Gypsum	CaSO4·2H2O	Gypsum	Chlorite	(Mg,Fe)6(Si,Al)4O10(OH)8	Chlorite
Albite	NaAlSi3O8	Albite	Pyrite	FeS2	Pyrite	-	-	-
Muscovite	KAl2Si3AlO10(OH)2	Muscovite	-	-	-	-	-	-
Potassium Feldspar	KAlSi3O8	Potassium Feldspar	-	-	-	-	-	-

Table S2 - XRF Analysis Results.

Main Oxides	SiO_2	Al_2O_3	BaO	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	P_2O_5	SO_3	TiO_2	LOI
Content (%)	60.93	14.58	0.12	1.62	3.97	6.49	0.99	0.36	1.03	0.13	0.4	0.4	4.26

Table S3 - ICP-MS Analysis Results.

Element	**Al**	**As**	**Ba**	**Be**	**Bi**	**Ca**	**Cd**	**Ce**	**Co**	**Cr**	**Cs**	**Cu**
Grade (g/t)	1951	1010	514	0.16	<0.1	830.8	0.55	9	9	76	9.7	33
Element	**Dy**	**Er**	**Eu**	**Fe**	**Gd**	**Hf**	**In**	**K**	**La**	**Li**	**Lu**	**Mg**
Grade (g/t)	1.4	1.7	1.8	7047	2.73	0.15	<0.1	38891	11	30	<0.1	4128
Element	**Mn**	**Mo**	**Na**	**Nd**	**Ni**	**P**	**Pb**	**Pr**	**Rb**	**S**	**Sb**	**Sc**
Grade (g/t)	1848	19	85331	41.8	48.1	55	0.2	6.78	14802	15.7	<0.1	5.16
Element	**Se**	**Si**	**Sr**	**Ta**	**Tb**	**Te**	**Th**	**Ti**	**Tl**	**Tm**	**U**	**V**
Grade (g/t)	<0.1	-	134	0.14	<0.1	0.14	6	1810	7.7	<0.1	4.5	26
Element	**W**	**Y**	**Yb**	**Zn**	**Zr**	-	-	-	-	-	-	-
Grade (g/t)	45	19.3	-	26	-	-	-	-	-	-	-	-

Table S4 - ANOVA for Weight Recovery Response of Concentrate.

Source	Sum of Squares	Degrees of liberation	Mean Squares	F-value	p-value	Significance
Model	238.45	3	79.48	14.93	0.0027	Significant
B	61.54	1	61.54	11.56	0.0366	
÷	49.72	1	49.72	9.34	0.0139	
D	117.4	1	117.4	22.05	0.0033	
Residual	101.17	19	5.32	-	-	
Lack of Fit	45.71	16	2.86	0.1546	0.8464	Not Significant