THE ARGHASH GOLD ORE SAMPLE TREATMENT[*]

A. MAHMOODI, M. NOAPARAST,[**] S. ASLANI AND A. GHORBANI

School of Mining Engineering, University of Tehran, I. R. of Iran
Email: noparast@ut.ac.ir

Abstract– The Arghash gold deposit includes two oxidized and sulfide zones. In this work, separate samples from mineralogical zones and drilling cores of both oxidized and sulfide zones were taken, in which the gold head grades were 2.5ppm and 0.8ppm, respectively. Microscopic studies of polished and thin sections of the samples showed that the Arghash mineralization was simple and composed of pyrite, iron oxide and hydroxide. To process the Arghash oxidized sample, the gravity separation treatment with a jig and shaking table were employed and different experiments were performed. The results obtained from these experiments were not acceptable as the gold particles were disseminated in fine sizes. The grade of the gold in the jig concentrate using feed size fractions of -2360+500 microns was 3.4ppm and in the shaking table using two feed size fractions of -500+75 and -300+75 microns was 3.8ppm and 2.4ppm respectively. The gold recovery in the jig test was 34% and for the shaking table test were 20.7% and 25.88% for the two size fractions. Because of the very low recovery obtained from gravity concentrations, cyanidation tests were carried out using an oxidized sample. Based on these experiments, the optimum value for size (d_{80}), cyanide concentration, pH and cyanidation time were determined as 75 microns, 1000ppm, 10.5 and 24 hours, respectively, which yielded 90% recovery.

The sulfide sample was used in the cyanidation process. The recovery of direct cyanidation on the sulfide sample was relatively low (about 66%); therefore, the sulfide zone of Arghash belongs to the "refractory gold" type. Thus as pretreatment, the sulfide part of the sample was separated by flotation. The optimum parameters for separation including the feed size (80% passing), collector dosage, frother dosage, pulp density and conditioning time were defined as 74 microns, 100g/t, 40g/t, 30% and 5 minutes, respectively. Roasting and ultrafine grinding procedures were also used for the sample pretreatment. The cyanidation test was then done using a flotation tail and roasted and very fine ground flotation concentrate. The maximum gold recovery (approximately 90%) was obtained by cyanidation of the roasted flotation concentrate.

Keywords– Arghash, gold, cyanidation, refractory gold, flotation, roasting, ultrafine grinding

1. INTRODUCTION

The Arghash gold deposit is located 45km southwest of Neyshabour city, Iran. The Arghash deposit has been explored in two different stages in 1984 and 1987 and 5 anomalies, accordingly, were explored [1]. The results of the exploration stages showed that the Arghash gold deposit has two significant zones: i) oxidized, and ii) sulfide. The limit of the oxidized zone is 35-40m depth and the sulfide zone continues to a depth of 162m which, of course, was the maximum depth of drilling [1].

The aim of the present work was to achieve the best methods to extract gold from the Arghash oxidized and sulfide samples taken from the mineralization zones, and then to optimize the values of the various processing parameters. The mineralogical study on the oxidized sample indicated that the gold was very fine and disseminated in the ore. Therefore, it was accordingly postulated that the gravity method

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would not be a convenient approach for the oxidized sample. Thus, the cyanidation method was employed for this sample treatment.

Because of the relatively low gold recovery in the initial tests by direct cyanidation, it was postulated that the Arghash sulfide sample belongs to the "refractory gold" type. Therefore, flotation was employed to recover sulfide minerals, and also the optimum values of the effective parameters on flotation were determined. Then cyanidation tests were conducted on roasted flotation concentrate, ultrafine ground flotation concentrate and the flotation tail. However, it is important to note that due to the gold malleability property, its grinding could cause difficulties in producing free fine gold particles [2, 3].

Cyanidation is a popular method for gold extraction, as it is inexpensive, readily available and very effective[4, 5]. The solubility of gold in an aqueous solution of cyanide was known from the beginning of the 19th century, but has been used economically since 1890 [6-8]. It was initially used for the extraction of gold from the residue solid of the amalgamation method, and due to its high efficiency was implemented for recovery of all the gold in the ore. The base of cyanidation is that a dilute sodium and potassium cyanide solution has a preferential solvent effect on the gold and silver particles in gold ore [9]. Although gold is inert to oxidation, in the presence of a suitable complex agent such as cyanide (CN⁻), gold forms a cyano complex [Au(CN)2]⁻ when it is oxidized[10, 11]. The dissolution of gold in alkaline cyanide solution is a heterogeneous reaction occurring at solid-liquid interface as Eq. (1) [7, 8, 12, 13, 14]:

$$2Au + 4KCN + O_2 + 2H_2O \Leftrightarrow 2AuKCN_2 + 2KOH + H_2O_2 \tag{1}$$
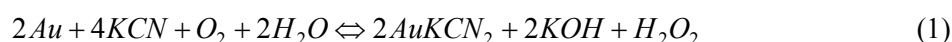
The cyanidation process is affected by a number of parameters such as cyanide concentration, pH/Eh, temperature, oxygen concentration, gold composites, gold particle surface and size distribution, some of which were investigated in this work.

Gold ores can be broadly categorized as "free milling" and "refractory" depending on their response to cyanide leaching. A gold recovery of >90% can be readily achieved with a conventional cyanide leaching of free milling ores. However, refractory gold ores are often characterized by the low gold extractions (<80%) in cyanide leaching [15, 16]. With the depletion of the oxidized free-milling gold reserves close to the earth's surface, most of the important new deposits being mined today do not respond to direct cyanidation. It has been found that gold is very finely disseminated and encapsulated in host matrices that are inert and/or impermeable to the cyanide solution. In this case, it is necessary to first break down the host matrices and then it is possible to leach the liberated gold. In many cases, the host matrices are sulfide minerals, which exhibit a strong association with finely disseminated gold particles in many ore bodies. Of these sulfides, pyrite is one of the most important minerals [17, 18].

A suitable pre-treatment process is often required to overcome the refractoriness and render the gold accessible to the lixiviate action of cyanide and oxygen. Ultrafine grinding, modified cyanidation, roasting, pressure oxidation and bacterial oxidation are the pretreatment methods currently practiced for refractory gold ores and concentrates [15].

2. MATERIALS AND METHOD

a) Sample preparation

According to the results of the exploration, about 500kg samples were taken from Anomaly 1 to 4 outcrops, trenches and the existing stockpile as the oxidized sample. In addition to the mineral processing study, some samples were prepared for mineralogical study. About 60kg of sulfide sample were collected from different seams of anomaly 3 drilling cores.

b) Mineralogical studies

The mineralogical study was carried out to determine the gold liberation size and the samples mineralization phases. The results of this study showed that the mineralization of Arghash ore was simple, and consisted of pyrite, iron oxide and iron hydroxide and some gold particle. Some different generations of pyrite were determined, and it seemed that gold mineralization was related to one of these generations. Table 1 shows the average size of some minerals and gold in the sample. As shown, the gold particles are very fine, about 20 microns. The total gold content in the sample was 2.5ppm.

Table 1. The minerals and gold average particle size

Minerals	Size (micron)
Pyrite	65-95
Goethite	115-130
Quartz	155-235
Gold	21-27

3. PROCESSING STUDIES (Oxidized Sample)

a) Gravity methods

The Jig and Shaking table were first used for the treatment of the oxidized sample. The Jig test was performed using a sample in the -2360+ 500 microns size fraction, and its results are presented in Table 2.

Shaking table experiments with a 4° slope were done on two samples with -500+75 and -300+75 microns size fractions. The results obtained from these tests are presented in Table 3.

As shown in Table 2, Au grade increased from 2.5ppm in the feed to 3.4ppm in the Jig concentrate with 34% recovery. The Au grades and recoveries in two Shaking table tests concentrates were 3.8ppm, 20.7%, and 2.4ppm, 25.88% respectively (Table 3). The poor results of the gravity tests could stem from the very fine gold particles size, and its dissemination in all size fractions. Thus, the gravity method was not suitable for the oxidized sample.

Table 2. The Jig concentration test results

Size fraction (micron)	Product	Gold grade (ppm)	Recovery (%)
-2360+500	Conc.	3.4	34.0
	Tail	2.2	64.0

Table 3. The Shaking table tests conditions and results

Size fraction (micron)	Product	Gold grade (ppm)	Recovery (%)
-500+75	Conc.	3.8	
	Middle	2.8	20.7
	Tail	2.5	
-300+75	Conc.	2.4	
	Middle	2.7	25.88
	Tail	2.4	

b) Cyanidation test

Due to the very low recovery of gravity concentration, cyanidation tests were carried out using the oxidized sample. In these tests the optimum value of size reduction, cyanide concentration and pH were determined.

1. Size Effect: The size of gold ores usually reduced until the gold particles were liberated and/or exposed to cyanide. If the ore is not ground enough, the possibility of cyanide penetration, gold solution and recovery would, accordingly, be decreased. On the other hand, more grinding needs more energy and cost consumption. Therefore, optimum grinding must be determined based on the techno-economical aspects. To distinguish the optimum grinding size, the gold recovery in different sizes was contemplated and the results are shown in Fig. 1. The recovery had a downward trend with a size increase. The recovery values for 65 and 150 microns feed size (80% passing) were 89.5% and 83.6% respectively. The maximum recovery was obtained in 65 and 75 microns of feed sizes (89.5%) and for a lower grinding cost, the larger feed size was preferred.
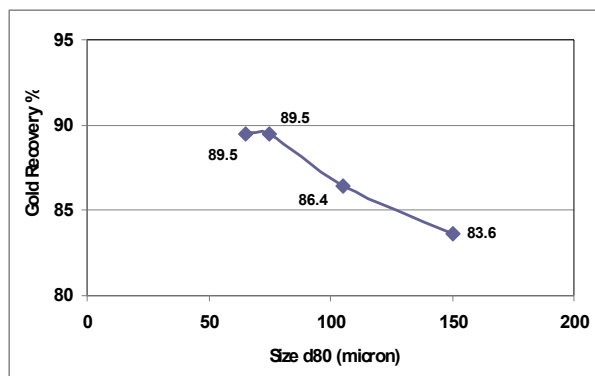
Fig. 1. Effect of size on gold recovery

2. Cyanide concentration effect: As Eq. (1) shows, oxygen is vital for the dissolution of gold in cyanide solution. At atmospheric pressure the cyanidation is not dependent on the cyanide concentration and the rate of gold dissolution is controlled by the constant oxygen concentration in the solution [5, 11]. In an industrial scale, the optimum value of the cyanide concentration is about 0.05% and/or 1lb per ton of solution [5]. Greater cyanide concentration has no effect on the gold extraction rate, but chemical and mechanical loss and therefore operation cost increases. Therefore this parameter must be maintained at a range of optimum value.

The effect of cyanide concentration on gold recovery was examined at different concentrations (1000, 2000, 3000 and 8000ppm). The results are presented in Fig. 2, which shows that increasing the cyanide concentration from 1000 to 3000ppm had no effect on gold extraction and increasing it to 8000ppm only caused about a 0.3% increase in gold recovery. To decrease cyanide consumption, 1000ppm was therefore considered as the optimum value.
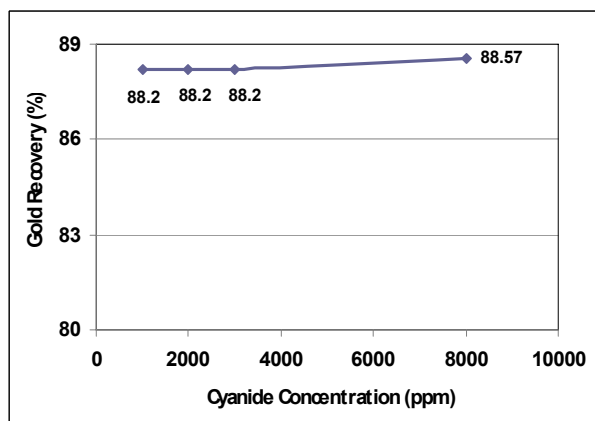

Fig. 2. Effect of cyanide concentration on gold recovery

3. pH Effect: The pH of the cyanide leaching solution should be carefully optimized to prevent HCN formation and achieve a high gold-leaching rate. As a rule, the optimum pH range in practice is 11 to 12 [5]. Thus, in this work the effect of pH on the cyanidation process was examined at 5 levels: 9.5, 10, 10.5,11 and 12. The results are illustrated in Fig. 3. The gold recovery increased from 88.24 to 91.84% by increasing the pH from 9.5 to 10, and then stabilized in the 10-11 range and finally decreased by increasing the pH from 11 to 12. In addition, the lime and cyanide consumption versus pH are illustrated in Figs. 4 and 5 respectively. The cyanide consumption decreased slowly with increasing the pH from 10.5 to 12. Therefore, the optimum value of pH was selected as 10.5.
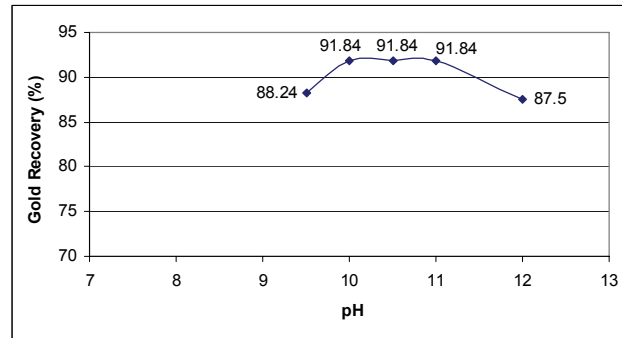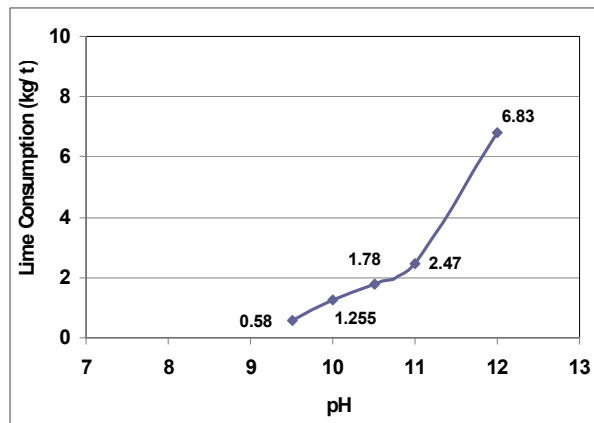

Fig. 3. Effect of pH on gold recovery
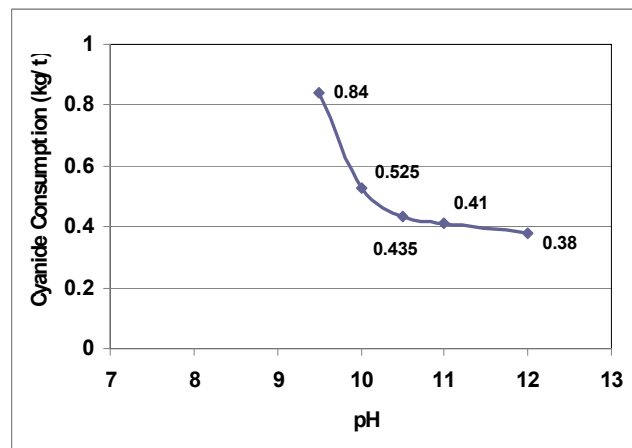

Fig. 4. Lime consumption versus pH

Fig. 5. The cyanide consumption versus pH

4. Cyanidation kinetics: The longer the time required for cyanidation to achieve a desired recovery from a gold ore, the larger the capacity required for the leaching tank, and hence the larger capital cost of a plant. In practice, a wide range of residence time (10 to 72 hours) is encountered in a gold plant [5].

To determine the optimum time for cyanidation, two experiments were carried out in parallel. For each test, 1333.3g samples ground to 80% passing 74 microns were placed into a glass reactor at room temperature. Water was added to bring a pulp with 40% solid content. The pH was adjusted to 10.5 with lime addition and cyanide concentration at 1000ppm, and then the test was started. Test total time was 48 hours and after 2, 4, 8, 12, 24, 36, 48 hours, some samples were taken for gold analysis and cyanide titration. When the test finished, the pulp was filtered and the residue gold in the solid determined. Then, based on the results, the recovery of gold as a function of time was plotted. This is presented in Fig. 6. This figure shows that the gold recovery has no significantly upward trend with increasing time after 24 hours, so this time was selected as the cyanidation time.
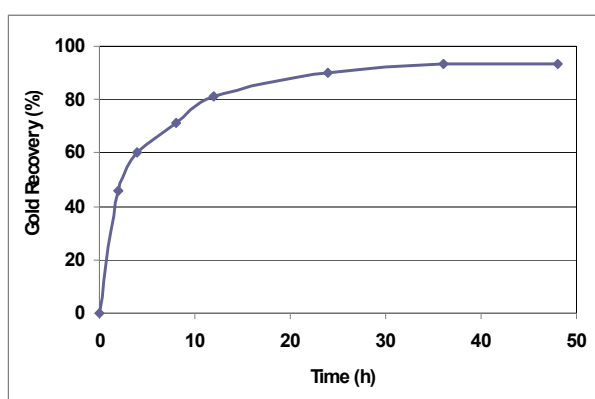

Fig. 6. Gold recovery versus time

4. PROCESSING STUDIES (Sulfide Sample)

a) Direct cyanidation

This test was done to determine the gold recovery of direct cyanidation using the sulfide sample with conditions: 40% pulp solid content in weight, d_{80}=74 microns. Natural pH was measured and by the addition of lime was adjusted to 10.5. Cyanide concentration was regulated at 1000ppm. Test duration was 24 hours, and the test pulp was then filtered and washed. The gold content in the solid and liquid phases was analyzed and the results shown in Table 1. Results indicated that the recovery of direct cyanidation of the sulfide sample, even after 24 hours, was low, about 66%.

Table 4. Sulfide sample cyanidation results

Sample	Cyanide conc. (ppm)	Free cyanide (ppm)	Cyanide consumption (ppm)	Cyanide consumption (kg/t)	Solid gold (ppm)	Solved gold (ppm)	Gold recovery (%)
Sulfide ore	1500	1000	500	1.5	0.1	0.45	66.67

Due to the relatively low gold recovery in direct cyanidation, it was postulated that this sample belongs to the "refractory gold" type, therefore, roasting and very fine grinding were examined to increase the recovery of gold. To separate sulfide minerals from ore, a flotation approach was used. Because of the negative effects of the flotation reagent on the downstream cyanidation process, the weakest reagent and the lowest dosages were preferred during the flotation tests. For example, sodium ethyl xanthate (SEX), the weakest member of xanthates, was used as the collector.

b) Flotation test

Flotation tests were performed in order to determine the optimum values of feed size, collector and frother dosage, pulp density and time. The purpose of these tests was to float pyrite, and the sulfur recovery was therefore considered as the efficiency comparison parameter. The tests were carried out and the obtained results are presented in the following.

1. Feed size: To determine the optimum grinding size, the sulfur recovery in different sizes was contemplated and the results are shown in Fig. 7. Three samples with d_{80} of 65, 75 and 95 microns were used in these tests with sodium ethyl xanthate (SEX) as the collector (100g/t) and methyl isobutyl carbonyl (MIBC) as the frother (40g/t). The flotation pulp solid content (in weight) was 30%. The conditioning and flotation times were 4 and 3 minutes, respectively.

As shown in Fig. 7, the maximum sulfur recovery was obtained for a sample of d_{80}=75 microns and recovery decreased with decreasing the size to 65 microns. Thus, 75 microns was considered as the optimum size.
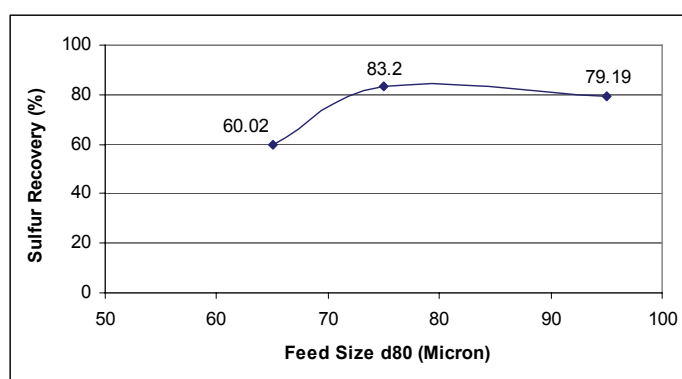

Fig. 7. Effect of feed size on sulfur recovery

2. Collector dosage: A series of tests were carried out to determine the collector dosage with the following conditions: sample size (80% passing) 75 microns, MIBC 40g/t as the frother and SEX as the collector with 30, 50, 100 and 150g/t dosages. The flotation pulp solid content in weight was 30%. The conditioning and flotation times were 4 and 3 minutes, respectively.

The result is shown in Fig. 8. This figure reveals that increasing the collector dosage to 100g/t has a significant effect on sulfur recovery, but this argument is not correct for greater increases to 150g/t, and the optimum value for the collector dosage was accordingly selected as 100g/t.
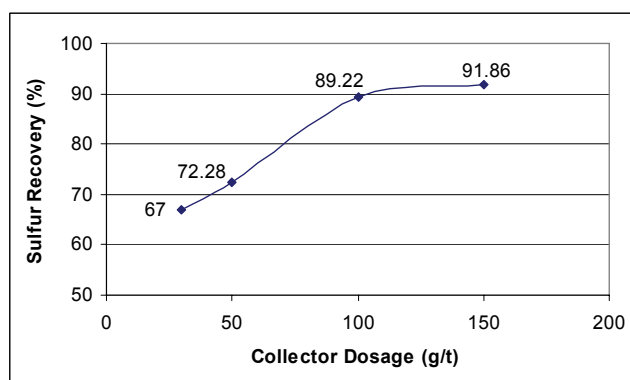

Fig. 8. Effect of collector dosage on sulfur recovery

3. Frother dosage: In order to find out the effect of the frother dosage on sulfur recovery, three different frother dosages were tested (20, 40 and 100g/t). Operational data: collector (SEX) dosage: 100g/t, conditioning time: 5 minutes, flotation time: 3 minutes and 30% of solid content. Figure 9 shows the effect of frother dosage on sulfur recovery, which indicates that by increasing the frother dosage from 40 to 100g/t, no significant effect on sulfur recovery occurred. Therefore, the optimum value of the frother dosage was considered as 40g/t.
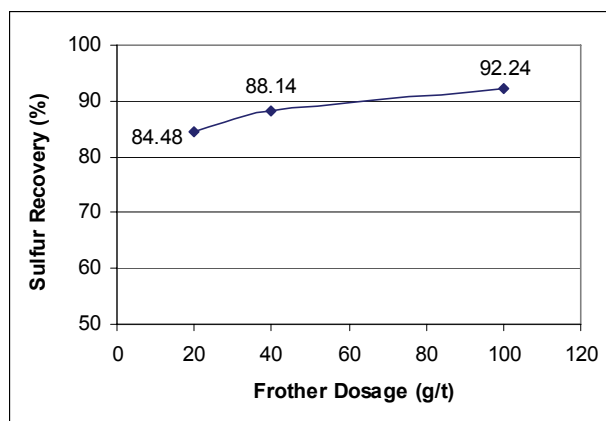

Fig. 9. Effect of frother dosage on sulfur recovery

4. Conditioning time: Three conditioning times (3, 5 and 7 minutes) were examined to determine the effect of this parameter on sulfur recovery and the results are presented in Fig. 4. Collector (SEX) dosage was considered as 100g/t, frother (MIBC) dosage as 40g/t, flotation time as 3 minutes and 30% of solid content. Figure 10 shows that increasing the conditioning time from 5 to 7 minutes had no significant effect on sulfur recovery; therefore, 5 minutes was selected as the optimum time.
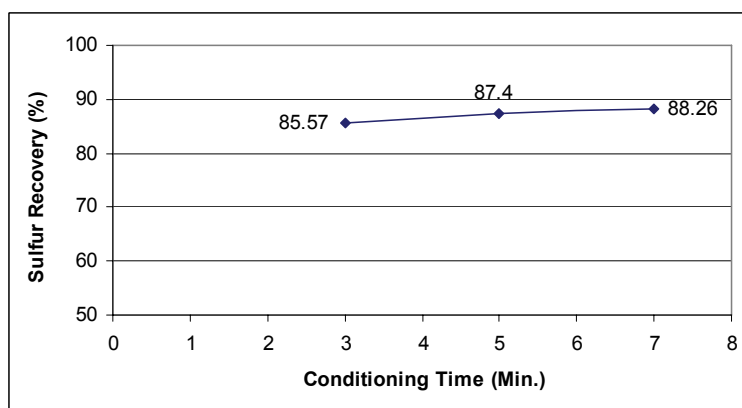

Fig. 10. Effect of conditioning time on sulfur recovery

5. Pulp density: To investigate the effect of pulp density (on sulfur recovery), three tests were carried out with conditions: collector (SEX) dosage: 100g/t, frother (MIBC) dosage: 40g/t, conditioning time: 5 minutes, flotation time: 3 minutes and solid content: 25, 30 and 35%. The results are shown in Fig. 11 and increasing the solid content from 25 to 30% had no effect on sulfur recovery, but a higher increase to 35% caused a recovery decrease. Therefore, 30% was selected as the optimum value for the solid content parameter.
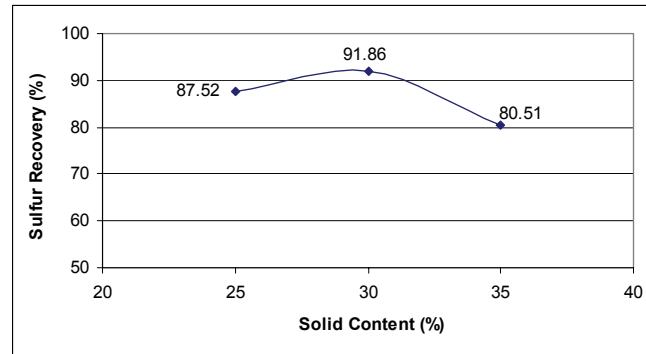
Fig. 11. Effect of pulp density on sulfur recovery

6. Flotation kinetics: Flotation kinetics test is the investigation of particle flotation rate versus time. This object is very important for flotation cell design and to determine the effect of different parameters on flotation performance. The scale up of laboratory test results to industrial scale is practically impossible without considering this issue, and/or causing a high error. The flotation kinetic equation of first-order is as follows [19]:

$$y = y_{in}\left(1 - e^{-kt}\right)$$

where:

y_{in}= the recovery in theoretically infinite time (%)

k= kinetic constant or flotation rate (sec.$^{-1}$)

Flotation test should be done in a batch process and the concentrate is collected within the flotation time steps and the relevant recovery would then be plotted as a function of time, in which the kinetic constant (k) and infinite recovery (y_{in}) is accordingly specified. The most important use of the kinetic graph is to figure out a suitable residence time. This test was implemented, and during the experiment and at different times the required samples were taken. The time steps in this test were 10, 10, 10, 15, 15, 30, 30, 30 and 60 seconds. Thus 9 samples were accordingly taken and analyzed. The obtained results are presented in Tables 5 and 6, and Fig. 12.

Table 5. Time steps considered for kinetic test

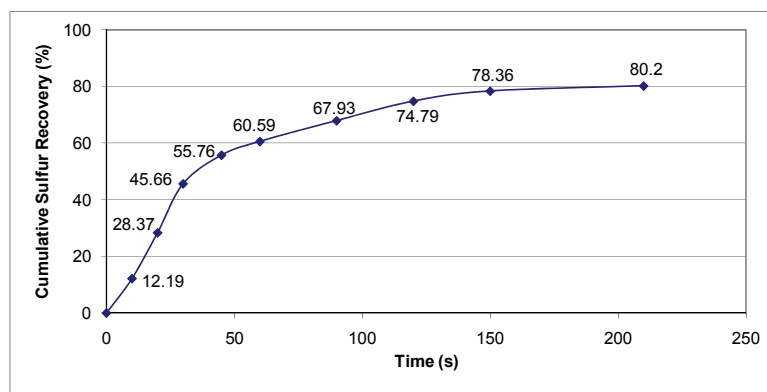| Sample No.	1	2	3	4	5	6	7	8	9
Time steps (sec.)	10	10	10	15	15	30	30	30	60

Fig. 12. Kinetic flotation test result

Table 6. Kinetic flotation test result

Sample Numbers	Time steps (sec.)	Conc. Weight (g)	Conc. Weight (%)	Conc. Grade (%)	Sulfur Rec. (%)	Cumulative Rec. (%)
AK1	10	25.3	12.77	23.2	12.19	12.19
AK2	10	36.4	14.37	21.4	16.18	28.37
AK3	10	33.3	16.81	25	17.29	45.66
AK4	15	26.3	13.28	18.5	10.1	55.76
AK5	15	12.3	6.21	18.9	4.83	60.59
AK6	30	21.3	10.75	16.6	7.34	67.93
AK7	30	22.3	11.26	14.8	6.86	74.79
AK8	30	12.3	6.21	14	3.58	78.36
AK9	60	8.6	4.34	10.3	1.84	80.2

As shown in Fig. 12, sulfur recovery increased insignificantly after 3 minutes of flotation time. So, 3 minutes was suitable as the laboratory scale residency time. To determine the value of k and y_{in}, sulfur recovery was plotted versus time as a first order graph (Fig. 13). The equation of the plotted graph is as follows:

$$y = 81.5\left(1 - e^{-0.027t}\right)$$

Therefore, the kinetic constant of this sample achieved equal to 0.027 sec.$^{-1}$ and the maximum sulfur recovery was 81.5%.
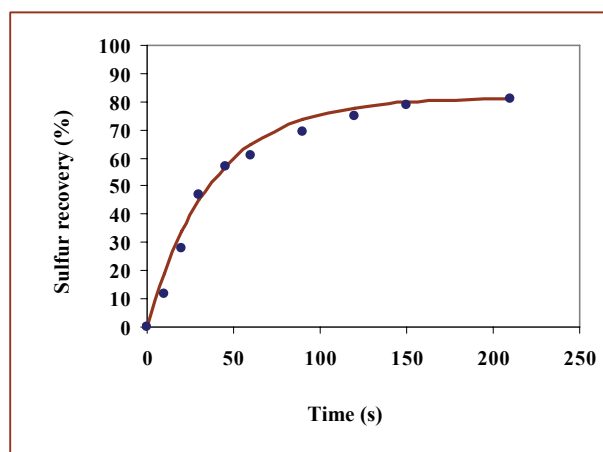

Fig. 13. Graph plotted to kinetic flotation result

c) Cyanidation test on roasted flotation concentrate

To prepare the required sample for this test, the flotation concentrate was roasted in a laboratory kiln at 550 °C. The kiln had a grate to ventilate air during roasting. Cyanidation test was then carried out on the roasted sample with these conditions: pulp density: 40%, pH: 10.5, cyanide concentration: 1500ppm. After 24 hours, pulp agitation was stopped and the test pulp was filtered. The gold was then analyzed in solid and solution. The result of this test is presented in Table 7, which was an indication of the gold recovery increase (to 90.7%) with roasting pretreatment.

d) Cyanidation test, using flotation tail

As the flotation tail was the oxidized part of the sample and included about 90% of feed, direct cyanidation test was performed using the tail sample. The conditions of this test were: pulp density: 40%, pH: 10.5, cyanide concentration: 1000ppm. The result is given in Table 8 and gold recovery was 85%.

Table 7. Sulfide sample cyanidation result

Sample	Cyanide conc. (ppm)	Free cyanide (ppm)	Cyanide consumption (ppm)	Cyanide consumption (kg/t)	Solid gold (ppm)	Solved gold (ppm)	Gold recovery (%)
Roasted Flotation Conc.	3333.3	2600	733.3	0.22	0.65	2.15	90.7

Table 8. Sulfide sample cyanidation result

Sample	Cyanide conc. (ppm)	Free cyanide (ppm)	Cyanide consumption (ppm)	Cyanide consumption (kg/t)	Solid gold (ppm)	Solved gold (ppm)	Gold recovery (%)
Flotation Tail	3333.3	2600	733.3	0.45	0.02	0.07	85.71

e) Cyanidation test, using ultrafine ground concentrate

The flotation concentrate was finely ground by 5 minutes of pulverizing to prepare the required sample for this test. Then test pulp with 20% solid content, pH=10.5 and a cyanide concentration of 1000ppm was prepared. After 24 hours, the pulp was filtered and the solid and solution were then analyzed. The result is presented in Table 9. This result shows that the recovery of the cyanidation test on the ultrafine ground sample was about 82%.

Table 9. Sulfide sample cyanidation result

Sample	Cyanide conc. (ppm)	Free cyanide (ppm)	Cyanide consumption (ppm)	Cyanide consumption (kg/t)	Solid gold (ppm)	Solved gold (ppm)	Gold recovery (%)
Ultrafine grounded	3333.3	1950	1383.3	4.15	0.6	2.2	81.82

5. RESULTS AND DISCUSSION

- A mineral processing study was carried out on the Arghash gold oxidized and sulfide gold samples. Two separate samples were prepared from mineralogical zones and drilling cores in which the gold head grade was 2.5 ppm and 0.8 ppm for the oxidized and sulfide sample respectively.
- After sample preparation the microscopic investigation was carried out. Mineral processing experiments including Jig, Shaking Table and cyanidation tests were conducted to determine the optimum value of the effective parameters for the oxidized sample. Also, cyanidation and roasting and ultrafine grinding pretreatment were used to extract gold from the sulfide sample. The significant results obtained are as follows.
- According to the microscopic investigation consisting of mineralogical (polished section) and petrography (thin section), Arghash mineralization is simple and composed of pyrite, iron oxide and hydroxide.
- To process the Arghash oxide sample by gravity separation, methods such as Jig and Shaking Table were employed and different experiments were performed. The results from these experiments were not acceptable because the gold particles were disseminated in fine sizes. The grade of the gold in the jig concentrate was 3.4ppm and in the shaking table using two feed size fractions of -500+75 and -300+75 microns, was 3.8ppm and 2.4ppm respectively. The gold recovery in the jig test was 34% and for the shaking table tests were 20.7% and 25.88% for the two size fractions.
- Because of the very low recovery of gravity concentration, cyanidation tests were carried out, and the optimum values for size, cyanide concentration, pH and cyanidation time were determined as 75 microns,

1000ppm, 10.5 and 24 hours, respectively. The optimum value of gold recovery was 90%, under these conditions.

- The recovery of direct cyanidation on the Arghash sulfide gold sample was relatively low, so this sample belongs to the refractory ore type. Therefore, sulfide minerals of a sample must be separate and pretreatment should be employed before the cyanidation test. Flotation method was used to separate sulfide minerals and the optimum value of the affecting parameters including the feed size, collector (SEX) dosage, frother (MIBC) dosage, pulp density and conditioning time were 75 micron, 100gr/t, 40gr/t, 30% and 5 minutes, respectively. According to the flotation kinetics test, the suitable flotation time was determined to be 3 minutes in the laboratory scale.

- Roasting and ultrafine grinding approaches were implemented as a pretreatment of the flotation concentrate. Then, a cyanidation test was carried out on the flotation tail and roasted and ultrafine ground concentrate. The recovery of gold for the cyanidation of the flotation tail, roasted and ultrafine ground concentrate were about 85.71%, 90.7% and 80.82%, respectively. A techno- economical study must be carried out to select a suitable pretreatment method.

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