



Contents lists available at ScienceDirect

International Journal of Mining Science and Technology

journal homepage: www.elsevier.com/locate/ijmst

Improved recovery of a low-grade refractory gold ore using flotation–preoxidation–cyanidation methods

Faraz Soltani^a, Hossna Darabi^a, Rezgar Badri^{b,*}, Piroz Zamankhan^c

^a Mineral Processing Department, Faculty of Engineering, Tarbiat Modares University, Tehran, Iran

^b Mineral Processing Department, Faculty of Engineering, Tehran University, Tehran, Iran

^c School of Engineering, University of Kwazulu-Natal, Durban, South Africa

ARTICLE INFO

Article history:

Received 9 September 2013

Received in revised form 10 November 2013

Accepted 1 February 2014

Available online xxx

Keywords:

Low-grade refractory gold ore

Pre-treating

Flotation

Pre-oxidation

ABSTRACT

In this work, different flotation–preoxidation–cyanidation methods are considered for treating a low-grade refractory gold ore. On the one hand, the results of selective flotation show that 22% and 31.1% of total Sb and As, respectively, remained in the final tailings and only about 28% of the total Au remained for further cyanidation processes. On the other hand, in bulk method of flotation the maximum Au recovery of 90.6% achieved after 60 min of flotation at the grind size with K80 of 146 micron. In addition, the bulk flotation method resulted in the concentrate with low concentrations of Sb and As elements. To improve the recovery of low-grade refractory gold ores, flotation should be followed by roasting, biological, or pressure oxidation processes so that the gold could be liberated prior to cyanidation processes. It is also found that the pressure oxidation pre-treatment of the concentrates prior to cyanidation may yield high gold recoveries of over than 83%. In these processes, recoveries are controlled by the temperature and the oxygen partial pressure in the solvent. However, by utilizing the bio-oxidation technique, the oxidation of sulfur to sulfate cannot be completed and, consequently, the gold recovery may be limited to only 72.2%.

© 2014 Published by Elsevier B.V. on behalf of China University of Mining & Technology.

1. Introduction

In sulfidic refractory gold ores, fine gold particles may be highly disseminated and locked up in sulfide minerals such as pyrite and arsenopyrite. All the largely unoxidized ores exhibit very low gold recoveries (typically <20%) by direct cyanidation. This suggests that the primary ore, as a whole, is highly refractory in nature and will therefore have to be processed using an appropriate refractory method. Often, the gold bearing ores are refractory due to the gold grains and concentration by flotation is necessary, either followed by roasting, bacterial leaching, or pressure leaching in a way that the gold is liberated prior to cyanidation. Two basic initial approaches are available, either treating the whole ore or processing some form of concentrate into which a high proportion of the gold has been recovered. A moderately low grade of the bulk ore and the additional processing costs involved in whole ore treatment processes are most certainly not economically viable and therefore not being considered [1].

A significant proportion of the sulfidic refractory gold appears to be hosted by the arsenical pyrite, arsenopyrite and stibnite components of the ore. These minerals are amenable to recovery by flotation. Adequate recovery of these fine-grained components might be difficult to achieve, even if the grind size is relatively fine (e.g. $d_{80} = 38 \mu\text{m}$). Orpiment (As_2S_3) and realgar (AsS) are both naturally hydrophobic and will float readily, and will thereby contaminate any sulfide concentrate with excessive amounts of arsenic. A high proportion of these minerals is, however, present in the form of relatively coarse grained vein fillings. It is also believed that these minerals do not host significant amounts of refractory gold. Disposal of any arsenic-rich product of this nature might, however, cause significant problems. Stibnite (Sb_2S_3) is sporadically present in the ore and, at least in some cases, may also adhere some of the refractory gold components [1].

Monteth et al. examined hydrogen peroxide in the selective flotation of gold from pyrite with potassium amyl xanthate (PAX) which showed a high selectivity for gold from pyrite at pH 10 and higher, but with a low increase of recovery [2]. In the ores which contain free gold, it may be beneficial to involve selective flotation of the free gold to improve the overall gold recovery [3]. Bulatovic showed that the results obtained with bulk flotation were superior to those obtained by selective flotation [4].

* Corresponding author. Tel.: +98 2692108361.

E-mail address: rezgarbadri08@gmail.com (B. Rezgar).

Among pre-oxidation methods, acidic pressure oxidation is considered the most effective method. Typical operating conditions include temperatures from 170 to 225 °C and oxygen partial pressures from 350 to 700 kPa over retention times from 1 to 3 h [5]. An overview of the role of pressure oxidation in treating refractory gold ores was given and subsequent developments have been highlighted. Examples of pressure oxidation plants commissioned since the mid-1980's and a brief description of their operation can be found in the article by Marsden and House [2].

Bacterial oxidation technology can be applied to refractory sulfide ores and concentrates to break down the mineral matrix and release the precious and base metals [6]. During bio-oxidation refractory gold ores processes, mineral decomposition is believed to be mostly due to chemical attack by ferric iron, with the main role of the microorganisms being to re-oxidize the resultant ferrous iron back to ferric iron [7]. If the Fe, As and S of the arsenopyrite are effectively oxidized by pretreating with bio-oxidation, gold will be naked and its recovery will be increased [8].

It will be necessary to undertake a series of metallurgical scoping tests to determine the nature, composition, and gold content of potential concentrates that might be produced from the ore. Therefore, it is necessary to evaluate a series of initial bench-scale metallurgical scoping tests to determine if it is possible to recover adequate properties of the refractory gold by flotation. The concentrate (bulk or sequential) from flotation stage should be further processed using pre-oxidation methods to liberate the locked gold and recover it by cyanidation. Thus, one of the aims of this study was to evaluate selective and bulk flotation pre-concentration methods to find out which of these methods could produce better concentrates for further processing stages like cyanidation. Also, two pre-oxidation methods (pressure oxidation-bio-oxidation) in acidic media were compared and subsequent cyanide leaching of the pretreated ores was investigated.

2. Materials and methods

2.1. Sample characterization

A sulfidic refractory gold ore containing 2.7×10^{-6} Au was obtained and used for all the experiments. Optical mineralogy by using the polished and thin sections, semi-quantitative X-ray diffraction (SQXRD) and ICP techniques are applied for ore characterization. Results of ICP and SQXRD are shown in Table 1 and Fig. 1 [1].

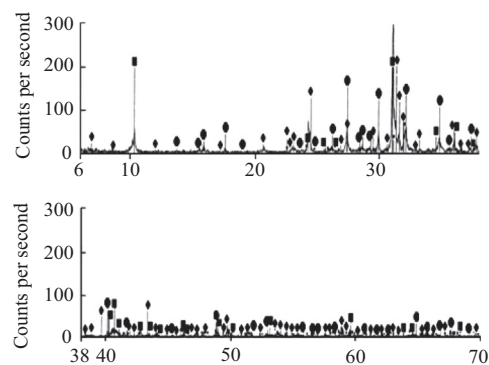


Fig. 1. XRD analysis of the ore sample.

A portion of the sample was submitted for mineralogical characterization. As indicated in Fig. 1 and Table 1, the sample was dominated by silicate minerals including quartz, potassium feldspar, and mica. Sulfide minerals comprised 8% of the sample with pyrite accounting for most of the sulfide content. Other sulfide minerals identified as very minor components were arsenopyrite, galena, orpiment, realgar, sphalerite, chalcocite, Cu sulfat, and stibnite. The arsenic content of the pyrite determined the analysis to be in the range of 0–36% (by weight) by electron microprobe.

The mineralogical sub-samples were washed and wet-screened to recover the 106–500 µm size. Afterwards, these size fractions were subjected to heavy liquid separation using a Na-polytungstate solution with a density of 2.89 g/cm³. The resultant heavy mineral concentrates were molded into epoxy resin and polished before examination using reflected light microscopy and SEM based techniques. Heavy liquid separation of the feed sample concentrated almost half of the gold in 1.9% (by weight), which was predominantly pyrite. The calculated gold grade was 94 g/t, but no visible gold was identified. Gold was also not identified in the overall head sample. The results suggest that the majority of the gold was present as a solid solution component in pyrite [1].

The bulk of the stibnite appears to be developed within transgressive late-stage veinlets where it is commonly intergrown with orpiment and/or quartz. The bulk of the stibnite is therefore also relatively coarse grained compared to the arsenical pyrite and arsenopyrite. The bulk of the stibnite should therefore be clearly liberated during comminution and, if necessary, it should be possible to separate it from the arsenical pyrite–arsenopyrite–pyrite-rich concentrate by differential flotation.

Table 1
The results of ICP scan of the gold ore sample [1].

Element	ICP scan		Element (g/t)	Sample	Element (g/t)	Sample
	Sample	Element (g/t)				
Au (g/t)	2.700	Al	76,000	Mo	<20	
Ag (g/t)	1.600	Ba	1500	Na	2000	
S (%)	2.170	Be	2	Ni	23	
S ²⁻ (%)	1.720	Bi	<20	P	740	
SO ₄ ²⁻ (%)	<0.400	Ca	2100	Pb	420	
As (%)	0.400	Cd	<5	Se	<100	
Sb (%)	0.044	Co	12	Sn	<100	
Fe (%)	3.310	Cr	16	Sr	100	
Tl (%)	0.230	Cu	22	Ti	160	
C(t) (%)	0.300	K	34,000	V	52	
CO ₃ (%)	1.360	Li	<10	Y	9	
C(g) (%)	0.010	Mg	3800	Zn	120	
TO (%) (Total organic)	<0.050	Mn	750	Tl	<100	

2.2. Flotation tests

1000 g of mill product (65% solids) was used for the flotation tests. A Denver flotation machine, a 3 L flotation cell and tap water were used for the batch flotation tests. Selective and bulk flotation experiments were carried out at about 30% solids. The impeller speed was set at 1800 r/min. In the first and second stages the pH was adjusted using lime and a pH meter (Orion 420).

Selective flotation tests were conducted to selectively prefloat a realgar-orpiment concentrate to reduce the arsenic in the concentrate for further gold recovery treatment. Firstly a stibnite concentrate and secondly a bulk sulfide (pyrite) concentrate were recovered. In the realgar-orpiment pre-float only frother was added. Subsequently, lead nitrate was added to activate the stibnite and small additions of Cytec's dithiophosphate, 3477, were applied to recover a stibnite concentrate. Finally, portions of potassium amyl xanthate (PAX) were added to recover all remaining sulfides (primarily pyrite). Table 2 shows the conditions of the selective flotation procedure. The d80 of the selective flotation feed was about 117 micron after 30 min grinding.

Conditions of the bulk flotation as a pre-floatation method is shown in Table 3. PAX was applied as the primary collector in all tests. All batch flotation tests were conducted at pH 7.

Fig. 2 illustrates the methods used for pre-treating a sulfidic refractory gold ore.

2.3. Pressure/bio-oxidation methods

The concentrate bulk samples (20 kg) from the batch flotation tests were used for POX (pressure oxidation) and BIOX (bio-oxidation) pre-oxidation methods. Screen analysis of the bulk flotation concentrate has shown that 80% of the samples were finer than 159 μm . The results of elemental analysis of the bulk flotation concentrate are reported in Table 4. Furthermore, the sample was dominated by silicate minerals, pyrite, arsenopyrite and stibnite. Other sulfide minerals identified, as very minor components, were galena, orpiment and realgar. The grade of the iron, arsenic and sulfur is 9.6%, 1.44% and 9.2%, respectively.

2.3.1. Pressure oxidation/cyanidation testwork

The bulk flotation sample (500 g) was pulped to ~33% solids with water. The pH of the pulp was adjusted to 2.5 with

Table 2
Conditions of the selective flotation method [1].

Stage	Reagents added (g/t)						Time (min)			pH
	Diesel	3477	PAX	208	Pb(NO ₃) ₂	DF250	Grind	Cond.	Froth	
Grind							30			6.8
Rougher 1						15		1	5	
Rougher 2						5		1	4	
Rougher 3	5					5		2	3.5	
Conditioning					500			7		
Rougher 4		2.5				5		1	2	
Rougher 5		2.5						1	2	
Rougher 6			20					1	3	
Rougher 7			20					1	3	
Rougher 8			20	10				1	3	

Table 3
Conditions of the bulk flotation method [1].

Test No.	BF1	BF2	BF3	BF4	BF5	BF6
K80 (μm)	60	83	114	146	83	114
Collectors (time of flotation, min)	225 g/t PAX (40)	225 g/t PAX (40)	225 g/t PAX (40) 325 g/t PAX (60)	225 g/t PAX (40) 325 g/t PAX (60)	400 g/t PAX (40) 200 g/t3418A	225 g/t PAX (40) 325 g/t PAX (60) 500 g/t CuSO ₄

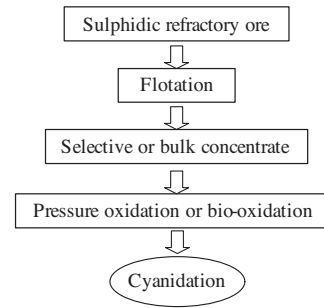


Fig. 2. Schematic presentation of pre-treating method used in this study.

Table 4
Bulk flotation concentrate analysis.

Element	Content	Element (g/t)	Content	Element (g/t)	Content
Au (g/t)	12.60	Ca	1800	Mo	<20
Ag (g/t)	4.00	Cd	5.2	Na	1700
As (%)	1.44	Co	44	Ni	49
Sb (%)	0.16	Cr	41	Pb	1200
Fe (%)	9.60	Cu	180	Sr	83
Al (g/t)	65,000	K	37,000	Ti	180
Ba (g/t)	270	Li	<10	Tl	<100
Be (g/t)	2	Mg	2500	V	56
Bi (g/t)	<20	Mn	600	Zn	380

concentrated sulfuric acid and maintained for 30 min. The stirred 2-liter acid resistance autoclave vessel was then sealed and heat up began. Once the sample was at desired temperature, desired O₂ over pressure was applied. At the end of the test the autoclave was closed and pressure relieved. The pulp was neutralized to pH 10.5–11 and forwarded to cyanidation. The pressure oxidation product (400 g) was pulped with water to ~33% solids in a 2.5 L bottle and placed on rolls. The pH was adjusted to 11 and maintained for 4 h with lime. NaCN was then added at a concentration of 1 g/L. Throughout the leach period, the pH and NaCN concentration were monitored and maintained. At the end of the test (24 h) the pulp was filtered. The filtercake was washed several times with water. The pregnant solution and wash solutions were combined for assay. The conditions and results are given in Tables 5 and 6.

Table 5
Summary of pressure oxidation tests.

Test No.	Feed size (μm)	Temp ($^{\circ}\text{C}$)	O ₂ overpressure (kPa)	POX time (min)	Emf (mV)	Free acid (g/L)	Fe (t) (mg/L)	Fe ²⁺ (mg/L)	As (mg/L)	Oxidation (%)
POX1	159	225	690	60	620	39	9380	64	2060	99.9
POX2	43	225	690	60	640	42	10,800	60	2300	99.8
POX3	159	225	690	45	620	43	12,800	59	2540	99.9
POX4	159	225	690	30	575	44	19,700	759	3310	99.7
POX5	159	225	345	30	470	43	14,900	7250	1710	69.6
POX6	159	200	345	30	470	30	7740	4220	719	48.4
POX7	159	200	690	60	552	50	15,600	1790	1870	97.2
POX8	159	200	690	60	520	49	16,400	2046	1790	93.8
POX9	159	190	690	60	522	45	14,900	3670	1280	80.7
POX10	159	210	690	60	525	47	17,400	2590	2210	94.6
POX11	159	210	690	60	555	46	10,300	619	2330	99.6

Table 6
Summary of cyanidation results.

Test No.	Reagent cons (kg/t)		Extr'n in cyn Au (%)	CN-residue		Head calc. Au, (g/t)	Overall Au recovery flot. + POX/cyn (%)
	NaCN	CaO		Au (g/t)	S ²⁻ (%)		
POX 1	0.82	29.4	95	0.60	0.01	11.5	83.8
POX 2	0.83	28.3	94.2	0.61	0.02	9.9	83.1
POX 3	1.43	49.0	91.8	0.89	0.01	10.6	81.0
POX 4	0.46	25.8	94.7	0.64	0.03	10.9	83.5
POX 5	0.37	13.3	86.1	1.76	2.77	11.6	75.9
POX 6	0.25	13.2	71.1	3.44	4.59	11.2	62.7
POX 7	0.25	7.4	94.2	0.80	0.26	11.5	83.1
POX 8	0.25	6.3	93.2	0.90	0.58	11.7	82.2
POX 9	0.30	5.9	91.4	1.10	1.78	11.9	80.6
POX 10A	3.32	233	92	0.68		12.0	81.1
POX 10B	0.28	4.8	94.2	0.80	0.50	12.4	83.1
POX 11A	3.04	248	94.3	0.45		12.1	83.2
POX11B	1.20	13.3	94.7	0.68	0.04	11.8	83.5

* Cyanidation on neutralized pulp.

2.3.2. Bio-oxidation/cyanidation testwork

A bio-oxidation test was performed on the bulk sulfide flotation concentrate. Samples were reground to K80 66 μm . The culture medium used during bio-oxidation is named 9 K culture medium, and the compositions are listed in Table 7. A 200 g sample was pulped to ~12% solids with 9 K solution (1500 mL) in a 2 L reaction kettel. The sample was heated in a waterbath to 40 $^{\circ}\text{C}$ and mixed with a mechanical mixer. The pH of the sample was adjusted to 1.5 and maintained for 6 h with concentrated sulfuric acid. Air was sparged through the pulp at a flow rate of 2 L/min. Once the pulp was at 40 $^{\circ}\text{C}$, with a stable pH of ~1.5, a volume of 100 mL of inoculum was added. A sample of the inoculum was filtered (20 mL) and the filtercake washed, dried and submitted for Au assay. The pH was maintained at 1.3–1.7 for the duration of the test using concentrated sulfuric acid. The EMF was monitored for the duration of the test with an Ag/AgCl electrode. After 7 days of leaching time, the air sparging system was changed and an additional 50 mL of inoculum was added. A sample of inoculum was filtered (25 mL) and the filtercake washed, dried and submitted for Au assay. Thief solution samples (4 mL) were taken at 1, 3, 7, 9 and 10 days and

Table 7
Compositions of culture medium.

Composition	Content (g/L)
(NH ₄) ₂ SO ₄	3.00
KCl	0.10
KH ₂ PO ₄	0.50
MgSO ₄ 7H ₂ O	0.50
Ca(NO ₃) ₂	0.01
FeSO ₄ 7H ₂ O	44.20

titrated for Fe²⁺. After 10 days, the pulp was filtered and the filtrate was collected and measured. The filtercake was displacement, washed 1 \times 200 mL with pH 1.5 H₂SO₄ acid solution and then 2 \times 300 mL with fresh water. The wash solutions were combined and discarded. The filtercake was forwarded to cyanidation. The residue from the BIOX (190 g) was advanced to cyanidation. The residue was pulped with water to 33% solids in a 2.5 L bottle. Lime and NaCN were added and the leaching was carried out over 24 h on rolls (400 mL volume solution). NaCN concentration and pH were maintained throughout the leaching period. At the end of the leaching period, the pulp was filtered and washed several times with water. The combined pregnant/wash solution and the residue were submitted for Au analysis. The results are presented in Table 8.

3. Results and discussion

3.1. Flotation as a pre-concentration method

3.1.1. Selective flotation

The aim of the selective flotation method is to minimize the amounts of As and Sb from tailings to further processing options. As it is discussed elsewhere, a significant portion of gold (24%) was recovered in the initial stage of flotation with the simple addition of a frother [1]. This suggests that fine free gold was present and therefore not possible to pre-float realgar and orpiment to remove it from downstream processing.

Since antimony causes problems in the cyanidation process, its separation from the other sulfides is desirable [9]. 22% of antimony remains in the final tailings of the selective flotation process. Also, the amount of antimony in this sample was too low to produce a

Table 8

Summary of bio-oxidation testwork.

Test No.	Biox time (day)	S ²⁻ removal (%)	S Oxid'n* (%)	Reagent cons (kg/t)		Extr'n in cyn Au (%)	CN residue			Head calc. Au (g/t)	Overall Au recovery flot. + BIOX (%)
				NaCN	CaO		Au (g/t)	S ⁰ (%)	S ²⁻ (%)		
Biox1	10	51.3	48.7	2.19	5.69	59.8	4.36	0.24	4.48	10.8	52.7
Biox2	14	85.8	64.2	2.14	3.24	78.7	2.48	1.98	1.31	11.6	69.4
Biox3	22	98.9	77.7	0.84	3.9	81.9	2.20	1.95	0.1	12.1	72.2
Biox4	26	98.9	76.5	2.94	6.52	79.0	2.40	2.05	0.11	10.9	69.7

* Based on S²⁻ and S⁰ analyses of the residue.

stibnite concentrate and thus further test work to produce a stibnite concentrate was not guaranteed.

Results showed that using selective flotation method, only about 28% of the total Au remained for further processing (especially cyanidation). The presence of 31.1% As in the tailing showed that the removal of As by selective flotation method is not warranted. The presence of about 22.7% of Sb in tailing is the third reason for failure of the selective flotation method in pre-treating the sulfidic gold ore which was aimed at As and Sb removal to the downstream processes.

3.1.2. Bulk flotation

As shown in the work done by authors, Bulk flotation tests were directed at maximizing gold recovery in a bulk sulfide concentrate [1]. The maximum Au recovery was obtained after 40–60 min of flotation. The fineness of grind had little impact on the recovery of gold and gold flotation kinetics. In fact, the kinetics improved slightly as the fineness of grind decreased. The best results were achieved at the constant grind size with a K80 of 114 micron.

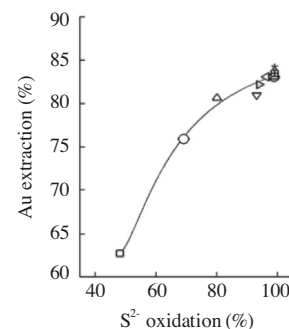
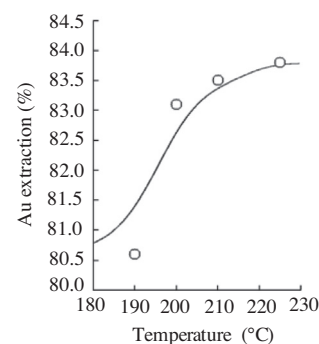
In all bulk testworks, recovery was between 77.3% and 90.6% and grade was between 10.4 and 15.9 × 10⁻⁶. The maximum recovery of 90.6% was obtained in conditions in which K80 and flotation time were 146 micron and 60 min. It may be concluded from the results that the bulk flotation method as a pre-treating method resulted in the production of a feed with less Arsenic amounts for further processing stages such as cyanidation, compared to the selective flotation method [1].

3.2. Comparison between pressure/bio-oxidation methods for bulk flotation concentrate oxidation

3.2.1. Pressure oxidation/cyanidation testwork

Pressure oxidation tests were performed on the bulk flotation concentrate. The effects of temperature, oxygen overpressure, oxidation time and regrinding were examined. The control factors and their levels used in the pre-oxidation experiments are given in Table 5. When the refractory head sample was pre-oxidized under different conditions, the sulfide matrices were decomposed by various degrees. The degree of sulfide breakdown can be determined by the arsenic, ferric, and sulfur content of the oxidized residues. The lower the arsenic, ferric, and sulfur content were, the more thoroughly the sulfide matrices were broken down. The relationship between ore sample oxidation and Au extraction is presented in Fig. 3. As can be seen from Table 5, Regrinding was not required to achieve good oxidation.

3.2.1.1. Effect of temperature. The most significant parameter with regard to gold recovery was found to be temperature as shown in Fig. 4. The highest ore oxidation was found as 99.9% at 225 °C and oxygen overpressure of 690 kPa. When the oxygen overpressure was maintained at 690 kPa, the oxidation of sulfides remained high (>95%) as the temperature decreased to 200 °C. At 190 °C, the oxidation of sulfide decreased to 81% resulting in a decrease in gold

**Fig. 3.** S²⁻ oxidation vs Au extraction.**Fig. 4.** Temperature vs Au extraction.

extraction from 94% to 91%. Under mildly oxidizing conditions (100–160 °C), elemental sulfur may be formed as the preferred or intermediate product of the oxidation of arsenopyrite, pyrrhotite and base metal sulfides. Molten sulfur at temperatures below 120 °C is an effective collector of various sulfides and is able to collect gold [10]. This seems to explain the very low gold recovery observed at low temperature in Fig. 4.

3.2.1.2. Effect of oxygen overpressure. Results (Table 5) showed an approximately linear relationship between recovery and oxygen overpressure. Reducing the oxygen overpressure from 690 to 345 kPa did result in a decrease in gold recovery from 95% to 86% as the sulfide oxidation decreased from 99% to 70% at 225 °C.

3.2.1.3. Effect of oxidation time. As can be seen from Fig. 5, at 225 °C, the oxidation of sulfides and recoveries of gold were similar after 30 and 60 min.

3.2.1.4. Effect of filtration of autoclave discharge. As can be seen from Table 5, in test 10 and 11, the POX discharge was split into two similar parts, on half of the pulp was neutralized directly (no filtration) and leached with cyanide, and the second part was filtered,

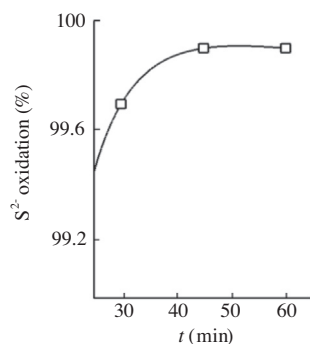


Fig. 5. Effect of pre-oxidation time on sample oxidation (temperature = 225 °C, O₂ overpressure = 690 kPa).

Table 9
Additional analysis of the leach residue from CN3 and CN4 (%).

Element	POX CN3 residue	POX CN4 residue
S	0.01	0.03
As	1.05	0.95
Sb	0.16	0.17
Hg	3.60	36.00
Ti	47.00	51.00

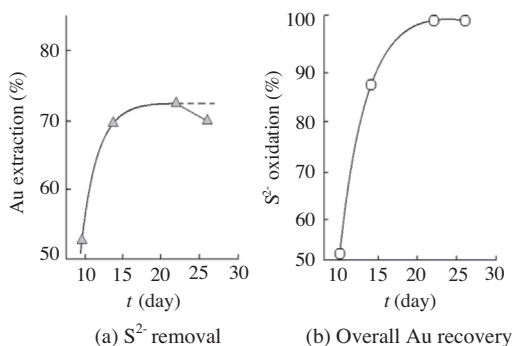


Fig. 6. BIOX time vs S²⁻ removal and overall Au recovery.

washed and repulped with fresh water for cyanidation as in most of the tests. The results of the comparative tests showed a significant increase in cyanide consumption when filtration of the autoclave discharge was omitted (3–10×). In test POX10, the gold extraction was slightly reduced without filtration, but this was not confirmed in the duplicate test, POX11.

The cyanidation residue from POX3 was examined mineralogically. Several occurrences of fine sulfide inclusions in silicate gangue were noted. Some liberated pyrite grains were also identified with irregular grain margins. Arsenic-rich margins was analyzed on one pyrite grain by scanning electron microscope. Several jarosite occurrences were identified including a jarosite-coated pyrite grain. The presence of jarosite may account for the high lime consumption. Some additional analyses were performed on the leach residues from POX3 and POX4 (Table 9).

3.2.2. Bio-oxidation/cyanidation testwork

The bio-oxidation/cyanidation experiments were carried out according to Table 6. The sulfide–sulfur analysis decreased from 8.3% in the feed to 0.1% in tests BIOX3 and BIOX4. The oxidation of sulfur to sulfate was only 77% complete with these sulfide levels.

The incomplete oxidation is shown in the significant levels of elemental sulfur. As a result, the gold recovery was limited to 82% (test BIOX3).

As shown in Fig. 6, S²⁻ removal and Au recovery increased with an increase of bio-oxidation time.

The cyanidation residue from BIOX3 was submitted for a mineralogical examination. Minor amounts of pyrite were identified; both liberated (up to 100 × 100 μm) and associated with quartz. Minor jarosite (up to 50 × 100 μm) was also present. It is possible that elemental sulfur may have occluded gold but there is no mineralogical evidence of that in the preliminary study. The bio-oxidation experiments did not yield an acceptable value.

4. Conclusions

In selective flotation method, it would not be possible to pre-float realgar and orpiment to remove them from downstream processing, because of the presence of fine free gold particles. 22% of antimony (Sb) remained in the final tailings and produce a stibnite concentrate which was not warranted. Presence of 31.1% As in the tailing postulated that the removal of As by using selective flotation method is not warranted.

In bulk flotation method, the maximum Au recovery obtained 90.6% after 60 min of flotation with 60% As recovery. The best results of the bulk flotation method were achieved at the constant grind size with a K80 of 114 micron and the bulk flotation method as a pre-treating method resulted in the feed with less penalty elements like As and Sb.

Pressure oxidation pretreatment of the concentrates prior to cyanidation gave high gold recoveries of over than 83%. Recoveries have been shown to be very much dependent on the temperature and pressure of oxygen. In the bio-oxidation method, the oxidation of sulfur to sulfate was only 77% complete (72.2% Au overall recovery) with these sulfide levels. As a result, the gold recovery was limited to 72.2% and compared to pressure oxidation method, the bio-oxidation experiments did not yield an acceptable value.

According to the results obtained in this paper, for this type of refractory gold ore, the combination of bulk flotation–pressure oxidation route is suggested as a pre-concentration process followed by cyanidation.

References

- [1] Badri R, Zamankhan P. Sulphidic refractory gold ore pre-treatment by selective and bulk flotation methods. *Adv Powder Technol* 2013;24:512–9.
- [2] Marsden J, House I. The chemistry of gold extraction. Colorado: Society for Mining, Metallurgy and Exploration; 2006.
- [3] Forrest K, Yan D, Dunne R. Optimisation of gold recovery by selective gold flotation for copper-gold-pyrite ores. *Miner Eng* 2001;14:227–41.
- [4] Bulatovic SM. Flotation behaviour of gold during processing of porphyry copper-gold ores and refractory gold-bearing sulphides. *Miner Eng* 1997;10:895–908.
- [5] Long H, Dixon DG. Pressure oxidation kinetics of orpiment (As₂S₃) in sulfuric acid. *Hydrometallurgy* 2007;85:95–102.
- [6] Sun L, Zhang Z, Tan WS, Zhu ML. Effect of agitation intensity on the biooxidation process of refractory gold ores by *Acidithiobacillus ferrooxidans*. *Hydrometallurgy* 2012;99(103):127–8.
- [7] Qingcui L, Dengxin L, Fangjun L. Pre-oxidation of high-sulfur and high-arsenic refractory gold concentrate by ozone and ferric ions in acidic media. *Hydrometallurgy* 2009;97:61–6.
- [8] Long H, Dixon DG. Bio-oxidation of arsenopyrite. *Trans Nonferrous Met Soc China* 2008;18:1433–8.
- [9] O'Connor CT, Dunne DG. The flotation of gold bearing ores – a review. *Miner Eng* 1994;7:839–49.
- [10] Gudyanga FP, Mahlangum T, Roman RJ, Mungoshi J, Mbeve K. Acidic pressure oxidation pre-treatment of refractory gold concentrates from the KweKwe roasting plant. *Miner Eng* 1999;99: 074–6.