



Recovery of gold from the Mouteh Gold Mine tailings dam

by A. Dehghani*, M. Ostad-rahimi*, S.H. Mojtahedzadeh*, and Kh.K. Gharibi*

Synopsis

It is estimated that the tailings dam of the Mouteh Gold Mine, in Iran, contains more than 770 kg of gold. Microscopic analysis reveals that sulphide minerals and active carbon are the main sources of the gold in the tailings residue.

Flotation and cyanidation tests on the samples of tailings were conducted. Flotation concentrates, containing around 87.79% of the gold, which were associated with sulphide minerals, were recovered. By regrinding, roasting, and cyanidation of the flotation concentrate, the gold dissolution recovery was in the range of 87.8 to 98.4%. The carbon-in-column method was used to recover 98% of the gold from the solution.

Keywords: gold ore, pyrite, flotation, roasting, cyanidation

Introduction

The Mouteh Gold Mine, located 270 km to the south-west of Tehran, and 10 km north-west of Mouteh village, is the main gold producer in Iran, with an annual production of 300 kg Au. The deposit is estimated to have 1 222 000 tons of ore reserve at an average grade of 2.5 g/t of gold.

Mineralogical analysis of the ore shows that the ore is refractory and contains free gold and gold telluride associated with pyrite (and chalcopyrite) (Figure 1). Gold grains are very small, mostly 5 μm , and are seldom larger than 15 μm in diameter.

The Mouteh processing plant is fed with sulphide and oxidized ore from nearby mines (Sinjeh and Chah Khatun). The nominal plant capacity is 600 tpd. The run-of-mine ore is crushed and ground to 80% finer than 75 μm , before being sent to the conditioning and cyanidation tanks. In the primary leaching stages, lime and sodium cyanide are added to the pulp (BHP Engineering 1991).

Conventional carbon-in-leach (CIL) cyanide leaching is used to recover the contained gold. The carbon is recovered from the leaching tanks by using cylindrical screens with 0.8 mm round aperture. The loaded carbon is stripped and the resultant solution is sent to the gold room, which includes the

electrowinning and the smelting units. The annual production of this unit is about 300 kg of gold and 36 kg of silver. The residue from the leaching circuit is pumped to the tailings dam (BHP Engineering 1991).

Studies show that the gold content of the tailings residue is more than the expected level (0.1 g/t). Gold in pyrite tends to be slow leaching and due to the short residence time, the recovery of the pyritic gold component is low (Muir *et al.*, 2005).

Using the metallurgical data from the Mouteh processing plant, it is estimated that the existing tailings dam contains around 778.5 kg of gold (0.5 g/t), while the average assay of gold in the feed to the plant is 2.5 g/t.

The residue tailings have been shown to be a source of 'cheap gold', since the mining and comminution costs have already been incurred. Advances in processing technology could provide extra gold and additional benefits, such as reducing environmental liabilities, may also follow from this exercise (Muir *et al.*, 2005).

In this work the retreatment of the gold residue tailings from the Mouteh Gold Mine, a mixture of sulphide and non sulphide minerals, was investigated. The test work included flotation, roasting and leaching, for the effective recovery of gold from the Mouteh tailings.

Representative samples from the Mouteh tailings dam were obtained. The objectives of the sampling were to quantify the potential of the deposit and to characterize the site and the resource material. After an initial sampling, 33 sampling holes, of 1 metre depth, were dug in the tailings dam. Samples from different depth were taken. The minimum weight of each sample was around 3.35 kg. The total weight

* Department of Mining and Metallurgical Engineering, Yazd University, Yazd, Iran.

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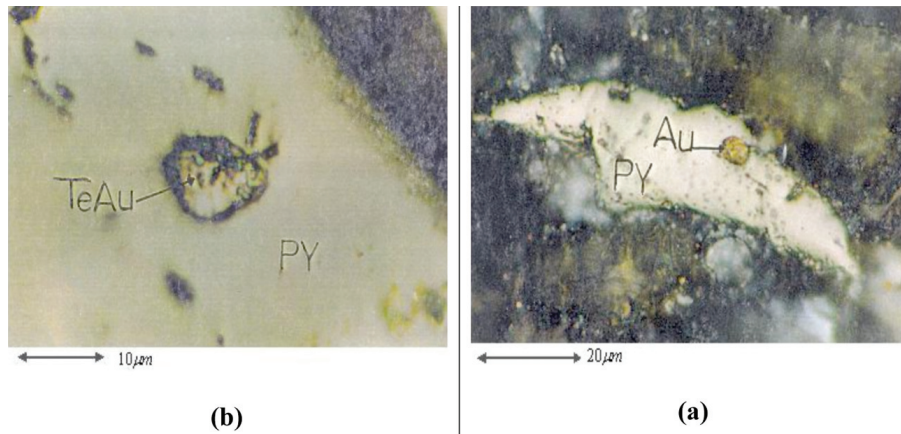


Figure 1—Electronic images of polished sections of the samples: (a) gold grain locked in a pyrite particle; (b) gold telluride in a pyrite particle

of samples taken was around 200 kg (5–7 kg from each hole). The gold assay of the samples ranged from 0.6 to 2.3 g/t, with an average of 0.976 g/t (1 g/t). It was observed that the gold assay increases from the bottom to the top of the holes, indicating that the plant used to perform better at the beginning of its life.

The mineralogy of the tailings indicates that froth flotation is the best method of removing the pyrite for separate treatment. However, ultra fine grinding may be used when liberation of gold particles from the surrounding mineral matrix is the primary refractory characteristic of the ore. Roasting can also be used to oxidize both sulphur and organic carbon at high temperatures using air and/or oxygen (Muir *et al.*, 2005, Deschenes *et al.*, 2005).

Experimental

Materials

The samples used in this work were leaching residues taken from different places on the Mouteh tailings dam. In order to characterize the size and chemical nature of the gold particles found in the tailings, polished sections of the samples were prepared. Mineralogical determination was performed using optical microscopy and SEM-EDS, which revealed that the ore is refractory and contains fine gold particles (smaller than 10 μm in diameter) as free gold or as gold compounds with tellurium associated with pyrite (Figure 1).

Further analysis of the samples by electron microprobe also confirmed that the gold is associated with pyrite and to a lesser extent with chalcopyrite.

A size analysis of the samples was performed (Table I and Figure 2) to determine the size distribution of the tailings samples and the gold content of each size fraction.

The d_{80} of the samples was 140 μm (compared to the designed plant regime of 75 μm). Material larger than 75 μm contained more than 63% of the gold in the tailings. The gold assays in the upper size fractions are slightly higher than the lower fractions. The -53 μm fraction contained around 28% of the gold in the tailings.

The gold in the +212 μm fraction, which was higher than the other fractions, was due to the gold in the tailings associated with active carbon. The active carbon used in the

carbon-in-leach circuit of the plant, was washed away because it was significantly finer than the carbon recovery screens, typically used to recover activated carbon.

Experimental procedure

The following experiments were conducted to study the possibility of recovering gold from the Mouteh tailings dam samples:

Table I

Size analysis of the tailings samples

Sieve aperture (μm)	Weight retained %	Gold assay (ppm)	Gold distribution %
212	8.1	3.0	28.62
150	9.8	1.0	11.61
106	11.1	0.9	11.75
75	11.0	0.8	10.35
63	6.4	0.7	5.25
53	6.1	0.6	4.33
- 53	47.6	0.5	28.09
Total	100	1.07	100

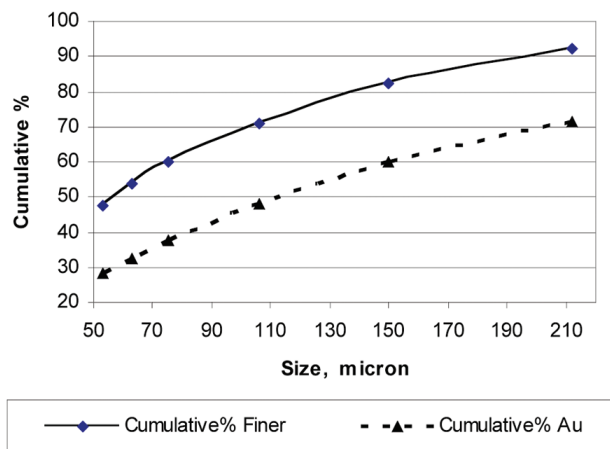


Figure 2—Particle size distribution and gold content of the Mouteh tailings samples

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Direct cyanidation of tailings

Initial cyanidation tests were run on the tailings samples with a high concentration of cyanide, with and without further grinding. The objective of these tests was to determine the effect of cyanide concentration, retention time, and grinding on the gold extraction from the tailings.

For the direct cyanidation experiments, the tailings samples were mixed with demineralized water at 25% solid concentration, air was sparged into the pulp, and the pH was adjusted to 10.5, using calcium carbonate. After 10 minutes conditioning, the sodium cyanide (2 000 g/t) was added. The duration of these experiments was 72 hours, during which time, pulp samples were taken at intervals of about 24, 48 and 72 hours for metal assay. The liquid phase and the remaining solid phase (washed and dried), were analysed for gold (Table II). The free cyanide ion was also checked. Gold in solution was combined with gold in the solids to give the total gold grade.

Flotation

As the main phases containing gold were identified as active carbon and pyrite, flotation tests were conducted to separate these phases from the tailings, in order that they could be treated separately for recovering the gold. Active carbon was separated using crude oil as the collector and Aerofloat 39 as the frother. The pulp was then conditioned with potassium ethyl xanthate as collector, Sacsol 95 as co-collector, and Aerofloat 39 as frother, for recovering pyrite. The effect of various operating variables, including pH, conditioning time, collector and frother additions on pyrite flotation performance were investigated. A Taguchi Design for four variables at two levels was employed.

Regrinding, roasting and cyanidation of pyrite

The results of cyanidation, roasting, and fine grinding on the flotation concentrate (pyrite) were compared. First, direct cyanidation and then fine grinding followed by cyanidation tests were conducted. Fine grinding of concentrate, at d_{80} of 53 μm , was carried out in a ball mill (200 mm dia \times 200 mm).

The samples were roasted at 620°C for one hour. Cyanidation tests on roasted samples, and ground and roasted samples, were also conducted.

Comparing CIL and CIC

The carbon-in-leach (CIL) and carbon-in-column (CIC) methods, for recovering the gold from the solution, were compared.

Results and discussion

Direct cyanidation

The results of direct cyanidation tests on the tailings samples, using a sodium cyanide concentration of 2 000 g/t, (a higher concentration compared to the actual plant leaching regime), solid concentration of 25%, pH 10.5, and test duration of 72 hours, is shown in Table II. The results revealed that around 40 per cent of the gold can be recovered from the tailings samples, but this method is not efficient enough to extract all the gold.

Size reduction (-53 μm) and cyanidation of the tailings samples also did not show any improvement in gold extraction (Table II).

The results of Table II revealed that extending the duration of the leaching did not improve gold extraction. By increasing the time, it is possible that other elements are also dissolved, reducing the assay of the gold in the solution. Furthermore, the size reduction produced fine particles of active carbon, which adsorbed the gold, therefore reduced the gold extraction.

Size reduction of the whole tailings samples did not increase the gold recovery, and active carbon needed to be removed before size reduction. The amount of free cyanide ions present in the solution for the reduced sized samples was low, indicating that the cyanide is consumed by other ions that are produced during the size reduction (i.e. iron, copper), resulting in an increased consumption of cyanide. Consideration was therefore given to introducing a flotation process.

Flotation of sulphide minerals

The objective of the flotation tests was to separate the active carbon and recover the sulphide minerals (pyrite) from the tailings. The flotation of active carbon was carried out using crude oil as collector and a frother. The pyrite was recovered by using ethyl xanthate as collector, Sacsol 95 as co-collector and Aeroflot 39 as frother.

As the main focus was on the pyritic ore in the tailings, an experiment design was employed to find the optimum selected levels of the factors affecting the recovery of pyrite in the concentrate (Table III).

As can be seen from the table, at an optimum level, the flotation concentrate contains 87.8 per cent of gold in the tailings feed. (This does not include the gold contained in the active carbon.)

Table II

The results of cyanidation and grinding tests on the Mouteh tailings samples

Time (hours)	Direct cyanidation			Size reduction and cyanidation		
	Gold assay (ppm)		Gold recovery (%)	Gold assay (ppm)		Gold recovery (%)
	solid	solution		solid	solution	
24	0.59	0.13	39.8	0.60	0.12	37.6
48	0.62	0.13	38.5	0.57	0.11	36.6
72	0.58	0.12	38.2	0.61	0.12	37.0

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Table III

Optimum conditions for flotation of sulphide minerals

Collector g/ton	Frother g/ton	pH	Conditioning time (min.)	Gold assay gr/ton	Gold weight %	Gold recovery %
150	200	8	4.5	8.38	6.27	87.79
150	300	8	6	7.70	6.77	72.45

Table IV

Gold assay and recovery for cyanidation, roasting and grinding tests on flotation concentrates from Mouteh tailings samples

Samples	Time, hours	Gold assay of the products		Gold recovery (%)
		Solution (ppm)	Solid residue (ppm)	
Concentrate leaching	24	0.57	3.4	40.2
	48	0.69	2.71	50.5
Grinding (- 53 μ m) and leaching of concentrate	24	1.1	1.39	76.0
	48	1.24	0.69	87.8
Roasting and leaching of concentrate	24	1.39	0.67	89.3
	48	1.50	0.28	95.7
Grinding, roasting and leaching of concentrate	24	1.44	0.52	91.8
	48	1.54	0.10	98.4

Cyanidation of flotation concentrate

The results of cyanidation tests on the flotation concentrate (pyrite) showed that the gold recovery is low (Table IV, Figure 3). This was an indication of refractory behaviour caused by either very fine-grained native gold inclusions in the pyrite, or the presence of gold-bearing tellurides.

In order to improve leaching and recover the gold from the sulphide concentrate, further size reduction and roasting of the flotation concentrate was considered.

Size reduction and roasting of flotation concentrate

In the light of the previous results, the flotation concentrate was ground to less than 53 μ m and subjected to cyanidation. Size reduction reduced the retention time and increased the gold extraction from 50% to 87.8%. Although cyanidation of the ground concentrate increased the gold recovery, it was still not satisfactory (Table IV, Figure 3).

Roasting of flotation concentrate without and with size reduction was also examined (Table IV, Figure 3). Roasting usually involves oxidation by air at 650°C, producing metallic gold and TeO₂. Grinding of the concentrate, at d₈₀ of 53 μ m, was conducted with a ball mill (200 mm dia \times 200 mm). The results of cyanidation on the roasted ground samples, under the same conditions as the previous sample, showed a gold extraction of 98.4%.

Recovering gold from the solution

Gold can be recovered from the solution by adsorption onto activated carbon. The standard methods are carbon-in-leach (CIL) and carbon-in-column (CIC). Both methods were examined using the solution obtained from the leaching of

the ground and roasted flotation concentrates. In the CIL method 18 g/l activated carbon and sodium cyanide (2000 g/t) were used for 48 hours. In the CIC the filtered solution was mixed with 18 g/l activated carbon. The recovery of gold in the CIC was slightly better than the CIL method (Table V). Using the CIC method, 98% of the gold can be recovered from the flotation concentrate.

Around 87.79% of the gold present in the Mouteh tailings dam that was contained in the sulphide minerals was recovered in the flotation concentrate. Therefore, by separating the sulphide minerals from the tailings, grinding and roasting them, it is possible to recover more than 86% of the gold from the tailings samples, using the CIC method (Figure 4).

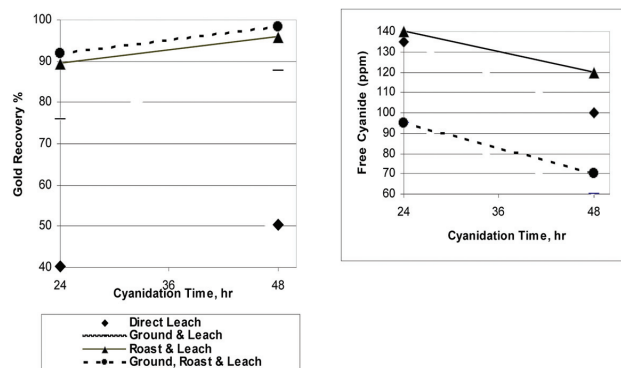


Figure 3—Gold recovery and free cyanide for leaching tests on the flotation concentrate of the Mouteh tailings

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Table V

Comparison of the results of CIC and CIL tests on flotation concentrate leaching

Carbon-in-column (CIC)			Carbon-in-leach (CIL)			
Gold assay (ppm)		Gold recovery (%)	Gold assay (ppm)			Gold recovery (%)
Active carbon	Solution		Active carbon	Solution	Tailings	
81.67	0.03	98.00	80.00	0.04	0.30	94.50

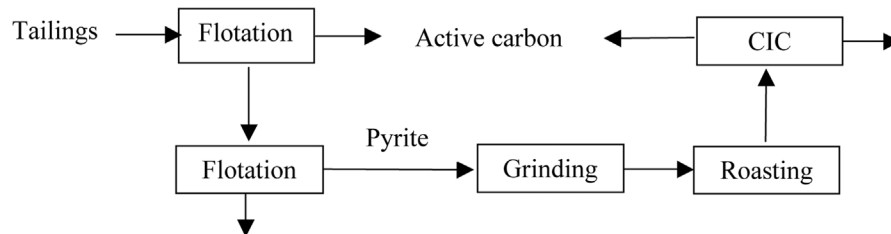


Figure 4—A proposed flow diagram for extraction of the gold from the Mouteh tailings

Conclusions

The tailings dam of the Mouteh gold mine contains a considerable amount of gold. Two sources have been identified for the gold lost in the leaching residue. They are gold associated with active carbon misplaced into the tailings (due to malfunctioning of screens), and gold that is trapped in the pyrite minerals (which has not been freed after size reduction at the design level).

Size reduction and cyanidation of the tailings samples did not produce promising results, probably due to the presence of active carbon in the samples.

Two stage flotation, firstly for removing the active carbon and secondly for recovering the sulphide minerals, which are the main sources of gold in the tailings, were proposed. It was shown that at the optimum levels of the operational parameters, in the range investigated, a concentrate of sulphide minerals (pyrite), containing 87.79% of the gold, could be obtained. Different methods of recovering gold from the flotation concentrates were compared. It was shown that by regrinding (to - 53 μm) and roasting the concentrate, around 98.4 per cent of the gold can be recovered into the leaching solution. Roasting of the pyrite, after grinding, will result in more gold recovery. Although fine grinding of the flotation concentrate enhances the gold recovery, it increases the cyanide consumption. Therefore an optimum size reduction that can provide high gold recovery at an acceptable level of cyanide consumption needs to be found.

By using a combined leach and CIC method, recovery was 98% of the contained gold. An economical evaluation is needed for each case.

Separation, size reduction, and roasting of the pyrite contained in the tailings dam of the Mouteh gold mine, showed a significant potential to increase overall gold extraction from the tailings.

The reground pyrite can be fed to the existing processing facilities of the Mouteh mine, to make the process more efficient. Regrinding of the plant feed size to -53 μm should also be considered.

The benefits from milling, flotation, pyrite treatment (leach, roast and leach calcine) and the type of gold recovery circuit (CIL) have been shown. In the light of the test work results and based on the unit processes required, a flowsheet can be developed.

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