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FLOWSHEET DEVELOPMENT STUDIES FOR GOLD AND CLAY CONTAINING SULPHIDE ORES

ALTIN VE KİL İÇEREN SÜLFÜRLÜ MİNERALLER İÇİN AKIM ŞEMASI GELİŞTİRME ÇALIŞMALARI

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ABSTRACT

In this study, flowsheet development and economic approach studies have been carried out for a sulphide ore containing gold and clay. For this purpose, previous beneficiation test results were used. First, clay separation was performed from the ore. In the following step, direct leaching, gravity separation and flotation methods were tested and compared to obtain a gold pre-concentrates at different grinding sizes. According to the test results, 34% of the ore by mass was separated as clay concentrate, which was suitable for the ceramic industry. Different pre-concentrates were obtained from gravity and bulk sulphide flotation tests. The possible flowsheet alternatives were compared and discussed. Product specifications were discussed in terms of economic and environmental aspects. As a result, producing Au containing flotation concentrate without leaching was decided to be more economical. In this method, 34% of the ore could be obtained as a by-product. 77.77 g/t gold containing sulphide pre-concentrates can be obtained with a 70.05 % gold recovery.

Anahtar Sözcükler:
Altın,
Kil,
Flotasyon,
Yerçekimiyle zenginleştirme,
Liç.

ÖZ

Bu çalışmada, altın ve kil içeren sülfürlü bir cevher için akım şeması geliştirme ve ekonomik yaklaşım çalışmaları gerçekleştirilmiştir. Bu amaç için daha önce yürütülen zenginleştirme çalışmalarının sonuçları kullanılmıştır. İlk olarak cevherden kil ayırma çalışmaları yürütülmüştür. Bir sonraki adımda ise, altın içeren bir ön konsantre elde edilmesi için farklı öğütme boylarında doğrudan liç, yerçekimiyle zenginleştirme ve flotasyon yöntemleri test edilmiş ve karşılaştırılmıştır. Test sonuçlarına göre, cevherin ağırlıkça %34'ü seramik endüstrisinde kullanıma uygun şekilde kil konsantresi olarak ayrılmıştır. Yerçekimiyle zenginleştirme ve flotasyon testlerinden farklı ön konsantreler elde edilmiştir. Olası akım şeması alternatifleri karşılaştırılmış ve tartışılmıştır. Ürün özellikleri ekonomik ve çevresel etkiler açısından tartışılmıştır. Sonuç olarak, liç uygulanmadan flotasyon ile elde edilen Au konsantresinin daha ekonomik olabileceği kararlaştırılmıştır. Bu yöntemde cevherin %34'ü yan ürün olarak üretilebilmektedir. 77,77 g/t Au tenörlü bir ön konsantre %70,05 altın verimi ile elde edilebilmektedir.

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INTRODUCTION

The gold industry currently processes most ores via cyanide leaching and carbon adsorption. However, complex ores (i.e. ores containing sulphides, high copper & poly-metallic ores, and clay ores) may not be readily amenable to cyanidation either for economic, environmental or technical reasons. In this type of ores, it is sensible to recover optimum amount of free gold earlier in the process in order to prevent large recirculating loads of free gold or incomplete leaching. Besides, some ore types allow production of by product concentrates such as copper, zinc or even clay from the same deposit (Laplante and Gray, 2005).

The use of centrifugal and flotation operations in a closed circuit milling may be an option for reducing the circulating load and processing complex ores containing both liberated (‘‘free’’) gold as well as gold in a sulphide matrix (unliberated or refractory). In such cases, modern gravity circuits can be used to recover the larger particles of free gold (+100 µm), while a flotation circuit will produce a sulphide concentrate including finer free gold particles (-100 µm) and gold containing sulphides. Nowadays, coarse-sized free gold particles are usually recovered in batch or semi continuous gravity units, such as Knelson or Falcon brand batch centrifugal concentrators (BCCs). Recent works by various authors have suggested that metallurgical performance of these units are similar, in terms of gold beneficiation (Laplante, 1993; Ancia et al., 1997). Gravity separation is often used in combination with flotation and/or cyanidation methods.

The flotation method is a technique which is widely used for the recovery of fine gold from gold-containing copper ores, base metal ores, copper nickel ores, platinum group ores and many other ores where the other processes are not applicable. Flotation is also used for the removal of interfering impurities before hydrometallurgical treatment (i.e. carbon prefloat) for upgrading the low-sulphide and refractory ores for further treatment. Flotation is considered as the most cost-effective method for concentrating fine gold (Bulatovic ve Wyslouzil, 1996). Many of the gold ores around the world contain large amount of clay minerals and it is well known that clays have a major impact on mineral processing in various ways. They may affect froth stability as the overall flotation and the gravity separation performances (Farrokhpay, 2011; Farrokhpay and Bradshaw, 2012). In some cases, desliming of clay minerals by using multi stage hydrocycloning followed by various concentration methods are suitable for obtaining clay by-product. Attrition scrubbing of clay minerals, followed by classification with a multi-stage-hydrocycloning, is successful in removing much of the associated quartz and calcite to underflow. The hydrocyclone overflow, however, can be separated as slime, which may either be a pre concentrate or a final concentrate depending on the quality of the clay. In addition to desliming process, various physical and chemical processes like screening, magnetic separation, selective flocculation etc., can be used for clay production (Saikia et al., 2003). There have also been many studies conducted to upgrade the clays by air separation and/or by using hydrocyclone (Oats et al., 2010; Boylu et al., 2010; Dulaney and Theobold, 1974)

Gold ores can be classified as free milling, complex or refractory gold ores. Free milling ores can reach over 90% gold recovery with conventional 20-30 hours cyanide leaching processes. The ores, which are characterized due to high cyanide or oxygen consumption, are termed as complex ores whereas the ores that do not provide economic gold recovery with conventional cyanide leaching are classified as refractory ores (La Brooy et al., 1994). Arsenopyrite, pyrite and chalcopyrite are the principal primary sulphide minerals that are locking gold in refractory ores (Linge, 1992). Generally, the most economical approach for the sulphide ores has been to produce a gold-copper flotation concentrate for metal recovery by smelting. Gold pre concentrates can generally be produced by the stepwise gravity separation and flotation techniques (Gul et al., 2012).

In this paper, previous test results of different separation methods were discussed to obtain a gold containing sulphide pre concentrate and a clay by product. According to the previous test results, flowsheet development studies were performed to obtain an optimum process flowsheet of high-clay-containing sulphide ore.

1. EXPERIMENTAL RESULTS

Detailed mineralogical and characterization studies were performed. Various concentration methods were applied to the ore. Raw results were previously presented by the authors in the International Mineral Processing Symposium 2014 (Gucbilmez et al., 2014). In this part a short summary of the experimental works are summarized.

Approximately 250 kg of drill core samples were collected from an ore deposit in Akoluk-Ordu region in Turkey.

Table 1. Particle size distribution of the feed and the metal content of the fractions

Particle Size (µm)	Mass (%)	Au (g/t)	Ag (g/t)	S (%)	Al (%)	Cu (g/t)	Pb (g/t)	Zn (g/t)
-4650+3350	3.12	2.36	76	1.60	3.52	276	1948	5160
-3350+1180	18.38	2.43	70	1.51	3.94	430	3198	4806
-1180+600	7.38	2.95	95	1.46	4.61	890	5300	7897
-600+300	6.12	2.56	117	1.60	4.58	1294	5674	7294
-300+150	5.34	2.40	123	1.22	5.53	1272	4617	6408
-150+75	5.34	3.32	87	1.39	6.05	931	3420	5383
-75+38	5.40	3.29	64	1.32	5.94	601	2562	4712
-38	48.92	1.21	21.34	1.19	13.70	134	719	1205
Feed	100.00	1.97	54.52	1.33	9.10	448	2306	3548

The as-received samples were mixed homogeneously to prepare a representative composite feed sample, which was homogenous mixture of drill core samples. Particle size distribution of the feed was determined by wet sieving. The overall and the size-by-size chemical analyses of the feed were determined by XRF method. The particle size distribution and the size-by-size chemical analyses results are presented in Table 1. Fractional gold grade and gold recovery of the feed sample are given in Figure 1.

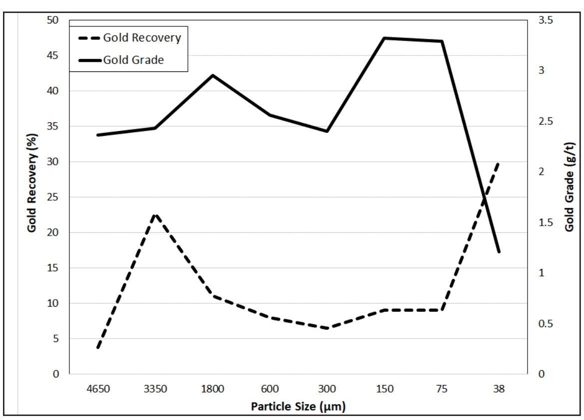

Figure 1. Gold grade and recovery of the composite feed

Figure 1 illustrates that the finest size fraction has the lowest gold grade. However, this size fraction has the highest fractional gold recovery because of fractional mass. As a result of mineralization studies, illite was defined as the main wall rock in the metallic mineralization (Yaylali-Abanuz and Tuysuz, 2010; MTA, 2011). According to Table 1, fractional Al grade (13.70%) and the recovery (73.68%) increased significantly in the -38 µm size fraction. Higher Al grade of that size fraction was thought to be a sign of clay. This estimation was supported by XRD analysis. The product of three-stage-cyclone-classification (cyclone overflow) was analyzed by XRD method to determine the mineral phases of the overflow. According to XRD pattern, the cyclone overflow consisted of 25% illite, 25% mica (mostly muscovite, little amount of biotite), 22% kaolinite, 8% chlorite, 6% quartz + feldspar and 8% amorphous material.

1.1. Clay Separation Test Results

A simplified flowsheet of clay separation process is given in Figure 2.

Table 2. Mineral phases of the final clay concentrate and a commercial grade clay (Kogel et al., 2006)

Mineral Composition	Al_2O_3 (%)	CaO (%)	Fe_2O_3 (%)	K_2O (%)	MgO (%)	Na_2O (%)	SiO_2 (%)	TiO_2 (%)	LOI (%)
Clay Concentrate	31.8	0.4	1.0	6.1	0.9	<0.1	51.3	0.2	6.35
Commercial Grade	24.3 min.	0.4 max.	2.0 max.	7.8 max.	2.5 max.	low	49.3 min.	0.6 max.	8.0

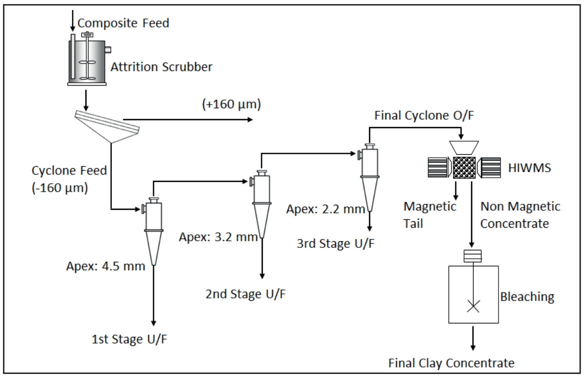

Figure 2. Simplified flowsheet of clay separation process

It can be observed from the clay removal studies that 34% of the feed material was classified as cyclone overflow. The overflow stream was concentrated up to 13.73% Al. Au, Ag and S contents of cyclone overflow were reduced to 0.53 g/t, 9 g/t and 0.60%, respectively. Au and Ag losses in cyclone overflow were 9.16% and 5.62%, respectively (Gucbilmez et al., 2014).

At the final stage of clay removal, the non-magnetic material was bleached. The mineralogical phases of the bleached material are presented in Table 2 by comparing the commercial grades of clay (Kogel et al., 2006).

1.2. Gravity Concentration Test Results

Hydrocyclone underflow and screen oversize material were mixed and ground in order to perform gravity concentration and flotation tests. That sample was called either as gravity feed or as flotation feed. Gravity concentration tests were performed by using a L40 model Falcon concentrator with the test materials having P_{80} 90 µm and P_{80} 38 µm, respectively. Panning was applied to the Falcon concentrates to determine the possibility of increasing the gold grade.

Gravity feed was concentrated up to 17.50 g/t Au and 521.00 g/t Ag at P_{80} 90 µm size. Composite feed was concentrated up to 97.31 g/t Au and 1858.05 g/t Ag at P_{80} 38 µm size by the combination of Falcon concentration and panning (Gucbilmez et al., 2014).

1.3. Flotation Test Results

According to the flotation test results, 1.88% of the feed by mass was concentrated up to 65.00 g/t Au and 2470.00 g/t Ag at P_{80} 90 µm size. 65.43% of the feed was taken as final tail with 1.41 g/t Au and 34.14 g/t Ag grade. Total recoveries of Au and Ag in the final concentrate were 45.18% and 59.64% at P_{80} 90 µm. 2.44% of the feed by mass was concentrated up to 77.77 g/t Au and 2479.00 g/t Ag at P_{80} 38 µm. 44.67% of the feed was taken as final tail with 0.53 g/t Au and 21.96 g/t Ag grade. Total recoveries of Au and Ag in the final concentrate were 70.05% and 77.57% at P_{80} 38 µm (Gucbilmez et al., 2014).

The effect of the grinding size on recovery was also apparent for each element. The recoveries of Au, Ag and S were higher at finer grinding size (P_{80} 38 µm). It was clearly seen that Au, Ag and S grades increased in cleaner stages. There was no significant decrease in the Au and Ag recoveries at P_{80} 38 µm grinding size. However, S recovery decreased sharply from 70% to 30% from rougher stage to cleaner stages. It can be concluded that S tended to report to tailings. This result also showed that Au and Ag were mainly presented as liberated particles instead of locked in sulphide minerals.

1.4. Gold Extraction by Cyanide Leaching

Leaching tests were performed to composite feed and flotation concentrate separately. Leach test was performed to the composite feed to determine the leaching kinetics and cyanide consumption at 10 g/l CN concentration. Kinetic leaching test result of the composite feed is illustrated in Figure 3.

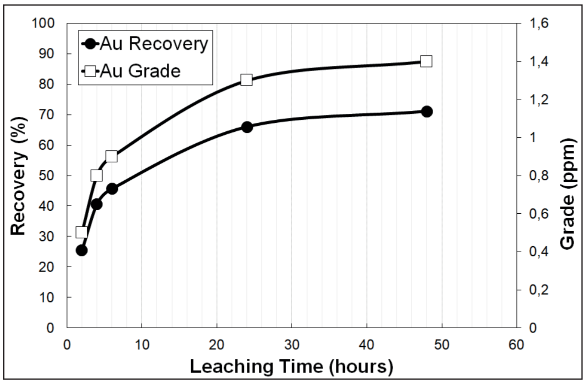

Figure 3. Kinetic leaching test result of the composite feed

The test results revealed that the leaching kinetics of the composite feed was slow, because the Au recovery increased only to 71.07% after 48 hours. Under same conditions, the leaching test was also performed to the flotation concentrate at P_{80} 38μm. According to the leaching results of the flotation concentrate, 23.80 g/t Au remained in the undissolved tailing. Leaching results revealed that 69.40% of the Au could be obtained as a leaching concentrate and 30.60% of the Au was reported to tailings. According to the classification of ore refractoriness, the composite feed sample could be classified as moderately refractory gold ore (50-80% recovery) (La Brooy et al., 1994). It can be interpreted that, Au which could not be beneficiated was probably in the form of enclave and because of that, it could not be in the reaction of cyanide. To determine the exact reasons for this problem, it is suggested to apply detailed mineralogical analyses and microprobe tests to the tailings of the leaching process. As a result of the tests, kinetic test result of the composite feed was found quite compatible with the flotation concentrate.

1.5. Comparison of the Test Results

The effect of the separation methods and grinding sizes on Au grade and recovery were presented in Table 3. Au grade of gravity and flotation feed was 2.71 g/t, respectively. According to the results, Au grade and recovery of gravity concentration were quite low at coarser grinding size. Approximately 60% of the Au was lost in the final tailing in that test.

Flotation results were, however, better than gravity concentration results at coarser size. The concentrate with 65 g/t Au could be obtained. Au loss, on the other hand, could be decreased by flotation method significantly.

Increase in grinding size increased the grades and recoveries in both methods. Better liberation of sulphide minerals and Au particles at finer particle size, improves the concentration properties of Au particles and their responses to centrifugal and flotation processes. The Au grade of gravity concentrate was higher than flotation at P_{80} 38 μm. However, Au recovery was significantly lower in gravity concentration than flotation.

Compared to other minerals, the specific gravity of Au is significantly high. A small amount of Au contained in a particle may vary the specific gravity, so these particles can report to Falcon concentrator. The particle size also affects the performance of the gravity separation. The performance can decrease at finer particle size fractions. As a result, high-grade concentrate can be obtained by using centrifugal separation in low recovery values. Furthermore, quartz particles

Table 3. Comparison of sulphide concentration methods at different grinding sizes

Grinding size	Method	Concentrate		Tail	
		Au (g/t)	Recovery (%)	Au (g/t)	Recovery (%)
P_{80} 90 μm	Gravity	17.50	19.49	1.50	61.08
	Flotation	65.00	45.18	1.41	33.96
P_{80} 38 μm	Gravity	97.31	36.03	1.16	38.02
	Flotation	77.77	70.05	0.53	8.74

containing little amount of gold may report to the tailings owing to the low specific gravity of quartz.

On the other hand, flotation is a physicochemical process, therefore, all sulphide minerals as well as the free gold can be collected into the froth phase as concentrate by using a specific collector. However, since all sulphide minerals would be collected, particles without gold may also be present in the concentrate phase. Therefore, in the experimental tests, comparatively higher recoveries were obtained with low grades by flotation at P_{80} 38 µm size fraction. Some losses in the flotation could be attributed to the quartz particles containing the locked gold that reports to the tailings.

2. FLOWSHEET DEVELOPMENT STUDIES

The ultimate purpose of the flowsheet development and the process planning are to devise a strategy that will optimize the project economics within the physical constraints of the deposit characteristics. Geology of the orebody, mine life, cut-off grade of the ore, location of the infrastructure/mining site and investments are some of the important technical, economic and environmental concepts, which should be considered (Pohl, 2011).

In this section, the possible flowsheet designs were discussed according to the experimental studies (Gucbilmez et al., 2014).

In the first stage of the flowsheet, clay separation was performed. After clay separation by multi-stage hydrocyclone, flotation method was preferred to obtain an Au containing pre-concentrate. A flowsheet was developed to obtain a bulk sulphide pre-concentrate and clay by-product. The flowsheet consisted of closed-circuit grinding, multi-stage hydrocyclone classification, magnetic separation and flotation. Flotation circuit consisted of a rougher and three-stage cleaner steps (Figure 4).

3. ECONOMICAL APPROACHES

Flowsheet options or different concentration methods ensure different grade and quality products. Production of a gold concentrate should be considered in terms of economic aspects. The commodity prices of the common clays and gold are 20 $/ton and 1290 $/ounce, respectively (MTA, 2015; Anon, 2017).

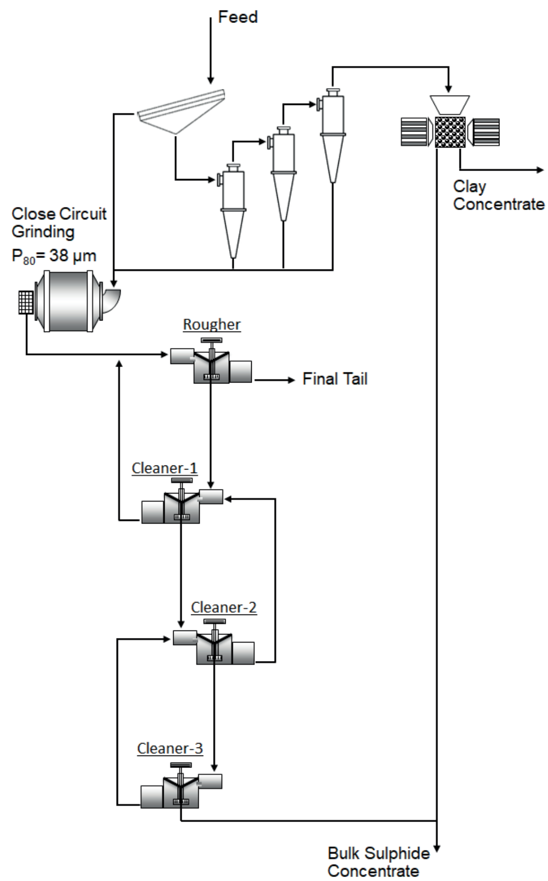

Figure 4. Recommended flowsheet of gold and clay containing sulphide ore

Approximately 70 g/t Au containing sulphide concentrate price was estimated as 800 $/ounce (this price may change according to downstream processes).

The calculations were made by the assumption that an ore of 1.97 g/t Au is concentrated by flotation method and the clay concentrate is obtained as a by-product. Assumptions were made due to sale prices.

It was accepted that, 71.07% of the Au would be leached from the ore and 69.40% of the Au would be leached from the flotation concentrate directly.

In case of using the sample flowsheet, 34% of the ore was concentrated as a saleable clay material. 2.44% of the ore could be concentrated up to

77 g/t Au and 2400 g/t Ag with total recoveries of 70% and 77%, respectively. By using those assumptions and the prices, economical approaches were discussed in different production methods (Table 4).

According to Table 4, leaching of the ore or the flotation concentrate had the highest economic value. The amount of material for leaching the flotation concentrate was very low compared to the direct leaching of the ore. From the environmental point of view, cyanide free plant is always advantageous. A plant with flotation and leaching processes may need more investment and causes extra costs and operational difficulties for this particular circuit.

Direct leaching of the ore can be easier in terms of operation and requires less amount of capital and operational investment. However, direct leaching of the ore may cause problems due to high clay presence in the ore. Clays have a major impact on various unit operations. For instance, the ore may not be heap leachable since agglomeration reduces the effect in very heavy clay ores (Conelly, 2011). Besides that, leaching the huge amount of ore causes transportation problems as well as environmental concerns. Higher Au grade of ore does not mean that the recovery of Au will be higher in a leaching process. The presence of arsenic sulphides, organic carbon and gold locked within the sulphide matrix may cause consuming large quantities of cyanide with less Au recovery (Alp et al., 2003).

CONCLUSION

Detailed material characterization, laboratory scale tests and analyses were performed to develop an optimum flowsheet for gold and clay containing sulphide ore. Effects of different concentration methods and particle size on gold concentration were discussed. In the first stage of the study, a salable clay concentrate was obtained successfully by using effective attrition scrubbing and three-stage hydrocycloning. A clay by-product had two benefits for a possible flowsheet. First, a salable by-product was obtained and the total amount of plant tail was decreased significantly. Second, the negative effects of clay minerals on the performance of milling and downstream processes such as flotation and gravity concentration were eliminated. 0.53 g/t Au and 9 g/t Ag losses occurred in the cyclone overflow. These grades may be economic for a regular type of gold deposit. But the high clay content of this material may not be suitable for physical separation methods and/or leaching. A detailed characterization and concentration tests should be performed to clay concentrate.

Sulphide concentration studies were performed to obtain an Au containing pre-concentrate at different grinding sizes. In addition to this, leaching tests were performed to determine the leaching kinetics and cyanide consumption. Leaching test results revealed that the ore could be classified as moderately refractory gold ore (50-80% recovery). Sulphide concentration studies revealed that an Au containing pre-concentrate could be obtained by performing centrifugal gravity separation and flotation, separately.

Decreasing the particle size from P_{80} 90 µm to P_{80} 38 µm increased the grade and the recovery values of Au and Ag for both methods. A high grade concentrate could be obtained by using centrifugal gravity separation in lower recoveries. Therefore, comparatively higher recoveries were obtained in lower grades by flotation at P_{80} 38 µm size.

Table 4. Economical approaches for different production methods

METHOD	Mass to handle (tph)	Au in the mass (g)	Leach Recovery (%)	Total Au (g)	Gold return ($)	Clay Price ($)	Total ($)
Direct leaching of ore	100.00	197.00	71.07	140	6371	-	6371
Bulk Sulphide Concentrate	2.44	187.88	-	188	5302	680	5982
Leaching of Concentrate	2.44	187.88	69.40	130	5933	680	6613

According to the economical and the environmental considerations, producing an Au containing pre concentrate by using flotation could be more economical. In this method, 34% of the ore could be obtained as a by-product and the total amount of the tail could be decreased significantly. In addition to that, Ag content of the concentrate could increase the price of the final concentrate.

In order to design a plant, total cost of any method should be considered. Leaching of the flotation concentrate could be an alternative for producing a final Au concentrate. Direct leaching of the ore might cause problems due to the presence of high amount clay in the ore.

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