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THE EVALUATION OF A COPPER FLOTATION PLANT PERFORMANCE BY PLANT SURVEY AND LABORATORY TESTS

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In this study the performance evaluation study of Georgia Madneuli Copper Ore flotation plant was performed using the data obtained from the detailed survey at the flotation circuit. The solids contents of the streams around the circuit were measured and the particle size distribution of the samples taken from the plant was determined. Mass balancing studies were performed using JKSimmet software. Mass balance results provided the flow rates and the chemical assays around the circuit. Therefore using the flow rates around each stream, the recovery of every unit in the circuit was calculated. The laboratory flotation tests were performed to obtain the optimum flotation conditions. For this purpose the flotation rate constants were determined for the different ore types processed at the circuit. Using the data obtained from plant and laboratory, a new circuit design was proposed.

Key words: flotation, flotation rate, mass balance, flotation test

INTRODUCTION

Madneuli-Georgia Copper Concentrator processes mainly three different ore types (named as sulphide ore, complex ore and oxidized ore). The plant consists of three different grinding and flotation circuits. Flotation circuit consists of two stages of rougher, two stages of scavenger and two stages of cleaner circuits. In the circuit studied, the plant capacity was found as 85.4 t/h. According to the design capacity, 1.600.000. t/y, the circuits must be operated at the capacity of 100 t/h. Therefore it is revealed that, the plant is performing with 85 % of the design capacity.

The flotation circuit consists of rougher, scavenger and cleaning circuits. The volumetric size of the cells in the rougher, scavenger and cleaner circuits are 6.3 m^3,

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3.25 m^3, and 3.25 m^3 respectively. The flotation feed comprises 33-36 % solids with 1% Cu content. The concentration method is the selective flotation of the chalcopyrite. The pH value is kept around 11.5 for pyrite depression. Potassium isobutyl xanthate is used as the collector with the dosage of 25-32 g/t.

During the sampling survey, the average Cu grade of the flotation feed was 0.89 % and the concentrate grade was approximately 21 %. But the average Cu concentrate grade was approximately 18%.

In this study, the performance evaluation of Georgia Madneuli Copper Ore flotation plant was performed using the plant data and laboratory test results. For this purpose, pulp samples were taken from the flotation circuit. All samples were dried, weighed and sieved. Size distribution of each sample was determined down to 6 μm. Each size fraction was assayed for Cu, Fe and insoluble contents in the flotation circuit. Mass balancing of both grinding and flotation circuits was performed to calculate the flow rates in each stream.

Laboratory flotation tests were also applied to the ore samples taken from feed of the primary grinding mill to determine the best flotation conditions and to compare the laboratory results with existing conditions. This study provided the required information to enhance the existing circuit and operation conditions.

SAMPLING AND ORE CHARACTERIZATION STUDIES

To evaluate the performance of flotation circuit, pulp samples were taken from the points marked in Fig. 1.

The samples taken from flotation circuit were dried, weighed and sieved. Size distribution of each sample was determined. Flotation feed fineness was determined as d80=300 μm in the existing circuit. Each size fraction was assayed for Cu, Fe and insoluble contents.

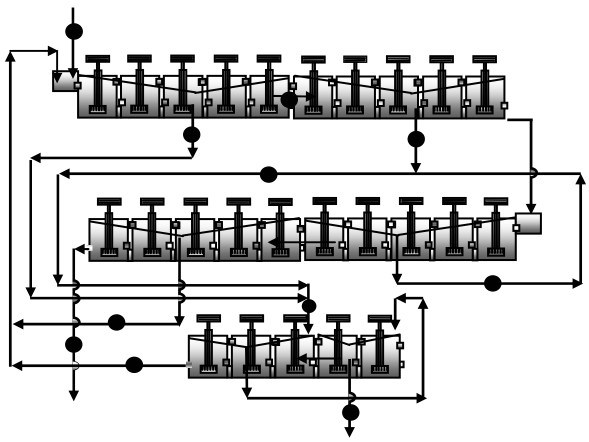

Fig. 1. Flotation circuits sampling points

In the plant mainly three different ore types (named as sulphide ore, complex ore and oxidized ore) are processed. Mineralogical compositions of the samples were determined by microscopic examination of the polished sections for sulphide minerals and X-Ray diffraction for non-sulphide minerals. The ore consists of mainly pyrite (FeS_2), chalcopyrite ($CuFeS_2$), traces of chalcocite (Cu_2S), covellite (CuS) and bornite (Cu_5FeS_4) as sulphide minerals and, quartz and clay minerals as non-sulphides. General chemical compositions of the ores are given in Table 1.

Table 1. Chemical compositions of the ores

Contents	Flotation feed	Oxidized ore	Sulphide ore	Complex ore
Cu	% 0.89	% 0.7	% 1.3	% 0.5
Fe	% 6.28	% 3	% 5.2	% 3.6
Zn	% 0.02	% 0.01	% 0.02	% 0.02
Pb	<50 ppm	<50 ppm	<25 ppm	<50 ppm
SiO_2	% 80.58	% 76.38	% 79.89	% 87.18
Al_2O_3	% 3.78	% 15.12	% 5.67	% 3.78

MASS BALANCING

Flow rates of each stream were calculated and balanced by using Cu and Fe assays of the head samples. Fig. 2 represents the balanced flow rates and Cu grade of the every sampling point around the flotation circuit.

The results showed that, during the survey, Cu recovery of the circuit was 89%. But Cu recovery of the circuit is between 63 and 82 % in regular measurements during the operation.

Flotation behavior of size fractions in the circuit was also investigated using the following kinetic model. The flotation rate of the size fractions was investigated in the rougher flotation bank. The relationship between flotation rate parameters and particle size is given in Fig. 3.

$$k\lambda = R/(1-R) \qquad (1)$$

where; k is the flotation rate parameter, λ is the retention time, R is the recovery.

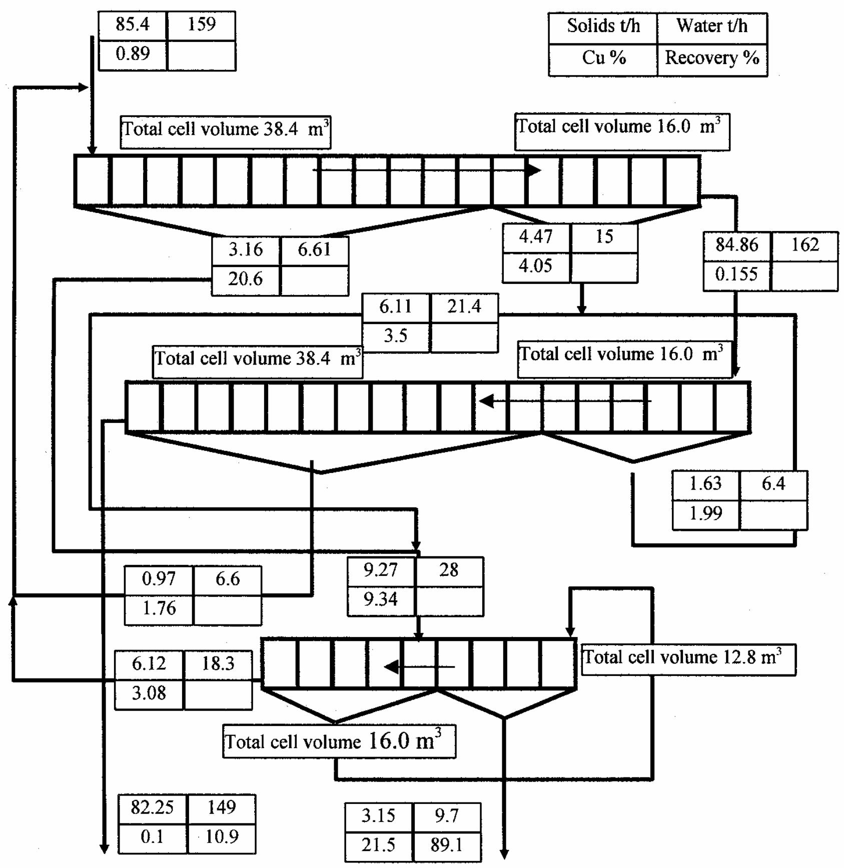

Fig. 2. Balanced flow rates, grades around the flotation circuit at actual operating condition

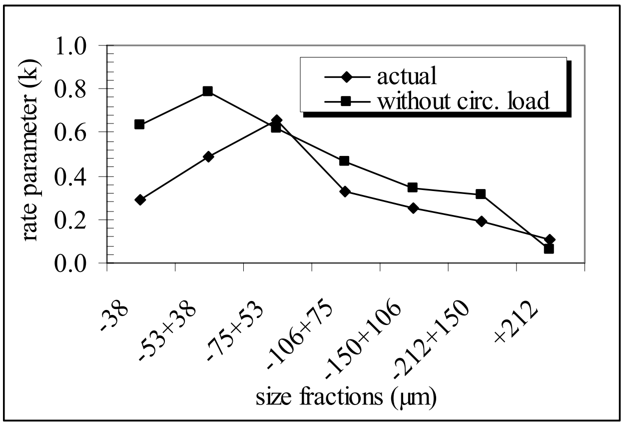

Fig. 3. Rate parameters obtained from plant data

Cu recoveries and flotation rate parameters of each size fraction were also calculated by using flow rates and Cu assays of the size fractions. It was observed that especially Cu recoveries of the fine size fractions were greater than that of the coarse sizes.

LABORATORY STUDIES

To obtain the best flotation behavior of the ore a series of laboratory flotation tests were carried out following the procedures given in Fig. 4.

In the laboratory Cu recovery higher than 94 % could be obtained in the rougher stage and the grade of this concentrate was increased to 26 % Cu by 3 stage cleaning. Although locked cycle tests were not performed, the Cu recovery was approx. 80% after cleaning.

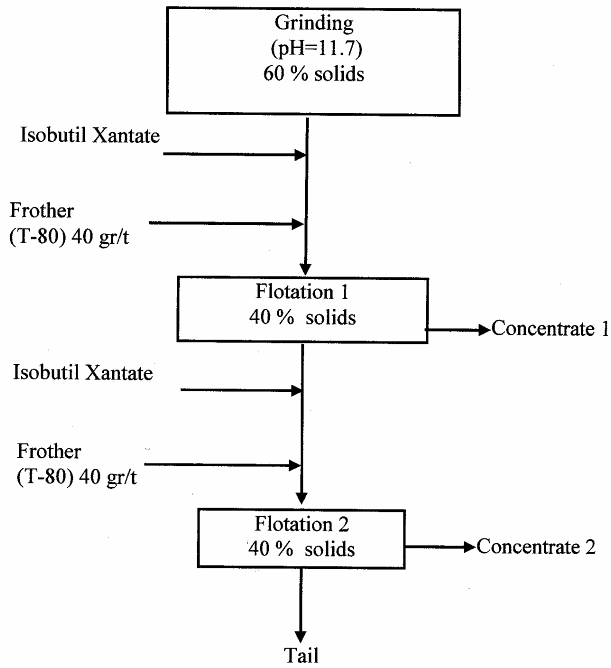

Fig. 4. Laboratory flotation test procedure

Finally, the laboratory studies showed that it is possible to obtain higher Cu recovery and grade by finer grinding.

In the laboratory, depending on the kinetic test results, retention times for each stage on the plant were estimated. By using the retention time values and flow rates, the required cell volumes and the number of cells were calculated. The laboratory revealed that different ore types showed different flotation behavior. The flotation behavior of different ores is illustrated in Fig. 5.

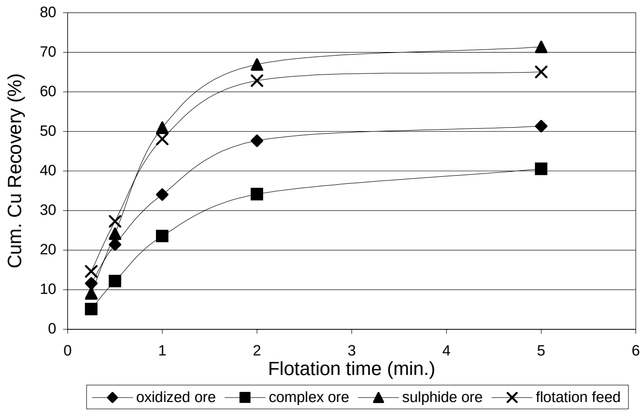

Fig. 5. Flotation behavior of the different ore types

The recovery and flotation rate of complex and oxidized ores were substantially lower than the sulphide ore. This was attributed to excessive oxidation of the sulphide minerals in these ore types. Therefore the flotation feed was prepared as a blend containing mainly sulphide ore and small amounts of oxidized and complex ores. Special flotation procedures and reagent schemes should be proposed for affective flotation of these problematic ore types.

The flotation behavior of the size fractions was also investigated. The relationship between flotation rate parameters and particle size is given in Fig. 6.

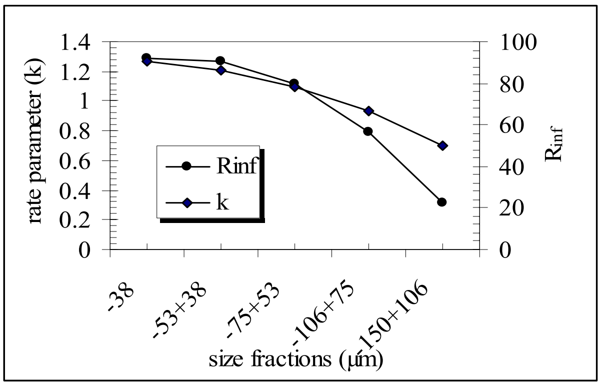

Fig. 6. Rate parameters obtained in laboratory study

To investigate the effect of feed fines on the flotation performance, two different samples (d80= +312 μm and d80= -312 μm) were prepared and used in laboratory flotation tests. In these tests, the copper recovery was increased from 89.79% to 94.56 for coarse (d80= +312 μm) and fine (d80= -312 μm) feed respectively. The concentrate Cu grade was reached to 26.07% after 3 stages cleaning for fine feed. This indicated that it is possible to increase the Cu grades by cleaning easily. Therefore the recovery must be as high as possible.

Consequently, the laboratory tests and plant scale studies showed that it is possible to obtain a concentrate with higher Cu recovery and grade by finer grinding.

Based on the results of the kinetic tests, retention time of the pulp was estimated for each stage in the plant. By using the retention time values and flow rates, required cell volumes and the number of cells were calculated using the following equation.

$$V_t = (Q \cdot \lambda \cdot S)/(60 \cdot a) \quad (2)$$

where Q is pulp volume m^3/h, λ is retention time, S is scale up factor (1.7 for lab. batch test) a is aeration factor (0.85).

Using the actual plant data and the laboratory test results a new circuit layout was developed in order to increase the performance of the circuit. The proposed circuit is given in Fig. 7. Therefore the number of cells in the circuit was increased from 18 to 24 for rougher, from 18 to 24 for scavenger and from 11 to 13 for cleaning circuits. In addition to that, the number of cleaning stages was increased from 2 to 3 to increase the Cu grade of the concentrate.

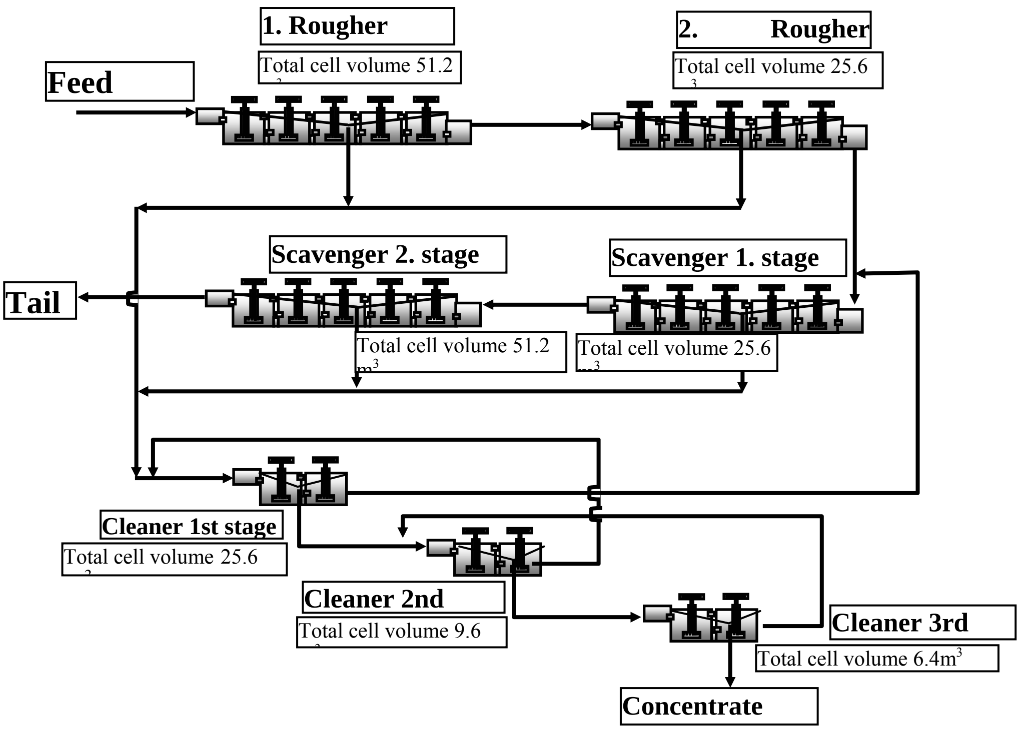

Fig. 7. Proposed flowsheet

CONCLUSIONS

The performance evaluation studies of the flotation circuit indicated that it is possible to obtain 21 % Cu copper concentrate with 90 % recovery. During the sampling survey flow rates of the flotation feed and the cooper concentrate were measured as 85.4 t/h and 3.15 t/h respectively. According to these results annually 1.610.000 tons of ores would make 60.000 tons of concentrate.

Performance evaluation and laboratory test results showed that an improvement can be obtained by finer grinding of the ore and some changes in the circuit design. In the plant, the ore could not be ground to even design values, and retention time is not enough to handle the capacities greater than 85 t/h. Therefore, the required total number of cells was increased to handle the longer retention time after fine grinding. In addition to this, instead of 2 stages of cleaning as in the existing circuit, 3 stages cleaning were proposed to obtained higher Cu grade. Flotation studies revealed that the grinding circuit must be optimized to produce finer flotation feed.

Ergün L., Ekmekçi Z., Gülsoy O., H. Benzer H., *Poprawa pracy zakładu flotacji na podstawie opróbowania i testów laboratoryjnych,* Physicochemical Problems of Mineral Processing, 38, (2004) 95-102 (w jęz. ang.).

W pracy przedstawiono wyniki badań mające na celu poprawę pracy instalacji flotacyjnej w kopalni miedzi Madneuli w Gruzji. Węzeł flotacji został opróbowany, określone zostały przepływy, skład ziarnowy i zawartość części stałych. Badania bilansu masowego węzła flotacji przeprowadzono z wykorzystaniem programu komputerowego JKSimmet. Otrzymane wyniki pozwoliły określić właściwe prędkości przepływów oraz wychody. Przeprowadzone zostały także laboratoryjne badania flotacyjne w celu określenia optymalnych warunków flotacji rudy miedzi. Określono na podstawie testów laboratoryjnych stałe flotacji dla poszczególnych rodzajów rud. Otrzymane dane z optymalizacji zakładu i z testów laboratoryjnych pozwoliły na zaproponowanie nowej instalacji do prowadzenia procesu flotacji.