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Increasing final concentrate grade of the Sarcheshmaeh Copper Complex floatation circuit by flowsheet modification

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ABSTRACT

The Sarcheshmeh copper complex flotation circuit of plant No.1 consists of two identical north and south sections where each includes four rougher (each bank consists of 14 cells), two cleaners (each bank consists of 8 cells), two recleaners (each bank consists of 2 cells), and two scavenger banks (each bank consists of 10 cells). The reduction of feed grade along with a change in the mineralogical composition is the main reason for the current lower concentrate grade (24%) compared with the design concentrate grade (32%). Because of a lower feed grade, the amount of rougher concentrate has decreased which in turn has significantly reduced the feed rate to the cleaner, recleaner, and scavenger banks. This has increased the mean residence time of the material in the cleaner section resulting in a lower final concentrate grade. A laboratory study showed that the final concentrate grade can be increased by 2-4% if one cleaner stage is added to the flotation circuit. In this research, based on the laboratory results, one cleaning stage was added to the current flotation circuit. In order to make this modification instrumentally possible in the plant, the final concentrate was gravity transported to the Mo-Cu thickeners. This released two pumps and associated tanks which made the addition of one cleaning stage practicable. Finally, a part of the cleaner and scavenger cells was used as the third cleaning stage. This decreased the residence time in the cleaner and scavenger banks. After implementation of one cleaning stage in the south section of the plant, the performance of the circuit compared with the identical north section. It was found that at the same recovery, the concentrate grade of the south section increased by 2.5%. The promising results led to the implementation of adding one cleaning stage in all sections of the flotation circuit.

Keywords: *Flotation, Gravity transport, Residence time, Third cleaning*

1. Introduction

In general, a circuit will give better separation than a single-stage operation, especially for non-sharp separations. Froth flotation is an example of a non-sharp separation system [1]. Froth flotation is a process used for the selective separation of hydrophobic materials from hydrophilic. Over the years, various criteria have been introduced for evaluating flotation circuits. Technically, grade and recovery are the most common indicators for the evaluation of a flotation circuit performance [2]. Separation of valuables from gangue in flotation circuits is often accomplished in several stages along with circuiting loads to obtain a concentrate with the desired quality [3]. The rougher inputs significant entrained fine gangue, which the combination of cleaner stages rejects. In cleaner stages, pulps may be diluted to take advantage of increased selectivity against entrained material. Cleaners do not fully eliminate entrainment; even after several stages of flotation, there is evidence that operational changes can still be effective in reducing its impact [1].

The arrangement of cells in flotation circuits is of prime importance which is often determined using empirical rules. Figure 1 shows two possible three-stage circuit configurations for coal flotation; two stages in three-cell units and one stage in units of single-cell are carried out. If a feed with certain characteristics is entered into the circuit (a), kinetics-based modeling of this circuit indicates that the concentrate ash content and yield would be 11% and 64.7%, respectively. If the same feed enters circuit (b) which has an equal number of stages but with a different configuration, the concentrate ash content will be within an acceptable level (i.e., 11.2%) and yield will increase to 65.8% [3].

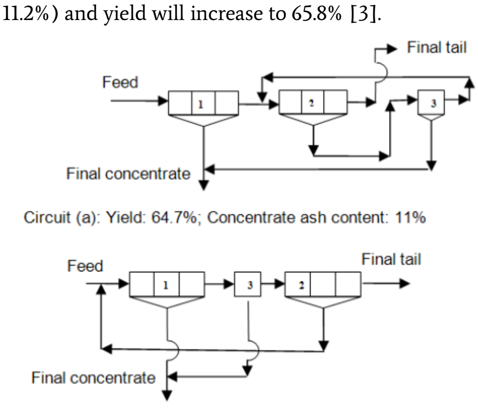

Fig. 1- Effect of flotation circuit configuration on efficiency [3]

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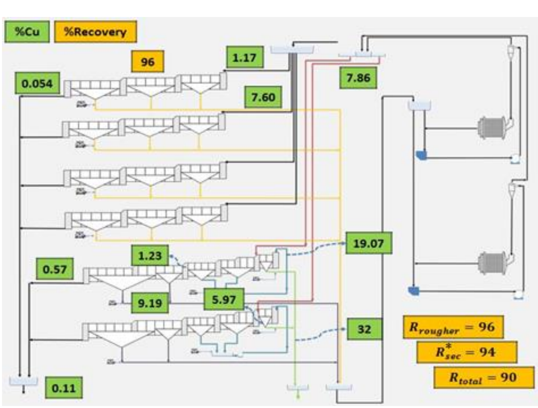
The final concentrate of any processing plant is one of the important indicators of plant efficiency; hence managers of the processing plants have always sought to find ways to achieve the desired grade. One of the problems over time at the Sarcheshmeh copper complex is the reduction of the final concentrate grade. The main reason for the decline is the feed grade reduction and a change in the ore mineralogy. At concentrator No. 1, the circuit has been designed for a feed grade of 1.17 % Cu to achieve concentrate assaying 32% Cu. It has been observed that the mineralogy of ore has changed from chalcocite to chalcopyrite as the mine depth increased. As a result of the ore type change, the feed grade decreased to 0.63% Cu. In order to achieve the target copper concentrate tonnage with the new ore type, the feed rate to the mills was increased. This, in turn, decreased the particles' residence time in the mills, leading to a 10% coarser rougher feed compared to the original design. All these changes led to a reduction in the final concentrate from 32 to about 24% Cu.

The lower concentrate grade makes the smelting a challenging task and increases the production costs. In the past, a number of researches have been performed to increase the final concentrate grade. The use of froth washing in the final cleaner cells [4, 5], changing the reagent distribution [6], and decreasing the residence time by reducing the number of cells reduction in the cleaner banks [7] are examples of these efforts. In these researches, the main efforts were focused to increase the final concentrate grade by modifications of the initial flowsheet design. But these measures did not completely resolve the problem of the low concentrate grade. In continuation of efforts to overcome this issue, a study was performed on a laboratory scale which suggested that the addition of a cleaner stage to the flotation circuit can increase the final concentrate grade by 2-4% [8].

2. Flotation circuit of the Sarcheshmeh copper complex

In the Sarcheshmeh copper processing plant after three stages of crushing, the ore with an F_{80} (80% passing size) of 12.7 mm is fed to eight parallel ball mills in a closed circuit with cyclones to produce a product 70% finer than 75 µm as flotation circuit feed.

The flotation circuit consists of two identical north and south sections. In each section of this circuit 4 parallel primary rougher banks (each bank consist of 3 units, a unit with 4 cells and two units with 5 cells), two parallel scavenger banks (each bank consist of 10 cells in 2 units), two parallel cleaner banks (each bank consist of 3 units, two units with 2 cells and a unit with 4 cells), and two parallel recleaner banks (each bank consist of 2 cells in a unit) are used (Figure 2). The rougher and scavenger concentrate of each section is grounded in 2 parallel regrind ball mills in a closed circuit with cyclones to produce a product 86% finer than 44 µm as cleaner feed. The final circuit tail is the combination of the rougher and scavenger tails and the final circuit concentrate is the concentrate of the recleaner stage that its copper grade based on the initial design must be 32%, but it currently is about 24%.



*:Total copper recovery in a bank of recleaner, cleaner, and scavenger

Fig. 2- One section of the flotation circuit of the Sarcheshmeh copper complex (with design data).

At the Sarcheshmeh copper complex, the feed grade reduction decreased the recovery of the rougher stage. Consequently, the feed rate of the cleaner and scavenger banks reduced which resulted in an increase in the residence time in these banks. On the other hand, it is well documented that an increase in the resistance time in a flotation circuit decreases the grade and increases the recovery [1]. It is then expected that the copper grade of the Sarcheshmeh copper complex flotation circuit decreases. The previous researches [4-6, 8] resulted in a rather small improvement in the copper grade, but in this research due to a change in the circuit layout and adding one cleaner stage to the circuit, the final copper assay significantly increased. There have also been successful experiences of changing the circuit layout to increase the final concentrate grade [9, 10].

3. Methodology

Increasing the performance of flotation circuits has been the subject of many studies [11-22]. In this study, in order to study in detail, the circuit performance and compare it with initial design values samples were collected from the circuit, and then the results were mass balanced using the Movazan software [23].

The addition of one cleaner stage to the circuit requires installing a bank of cells, a sump box, and a pump. The high residence time in the cleaner and scavenger cells allowed us to use part of these cells as a new cleaning stage (3rd cleaner). With this strategy besides adding a new cleaning stage and saving the costs of new cells, the residence time in the cleaner and scavenger stages decreases which is beneficial. Moreover, installing a new pump and a tank in the plant because of the limitation of floor space in the plant was not possible. On the other hand, the height difference between the recleaner cells and concentrate thickeners made the direct and gravity transportation of the final concentrate to the thickeners possible. In order to implement adding one cleaner stage, it was decided to transfer the final flotation concentrate by gravity to the Cu-Mo thickeners through a channel (Figures 3&4).

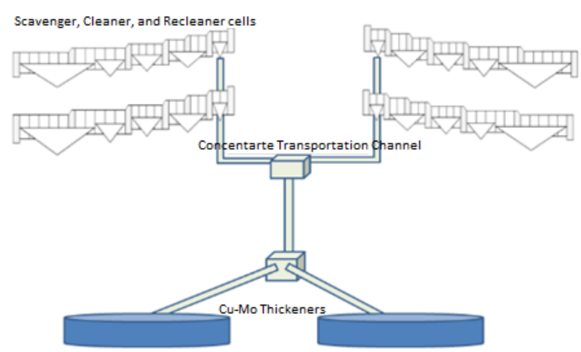


Fig. 3- Schematic design of the gravity transportation of the final concentrate.

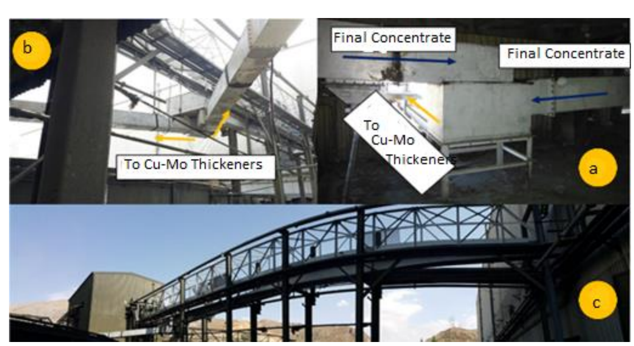


Fig. 4- Combining final concentrate of two sections (a), dividing the concentrate between two Cu-Mo thickeners (b), overall view of the gravity transportation channel (c).

4. Results and discussion

The samples were assayed and mass balanced using the Movazen software (Figure 5). The results of mass balancing showed that the residence time in the cleaner and scavenger stages increased about 70% because of a 40% reduction in the feed flow rate. It is expected to have a lower grade and higher recovery because of an increase in residence time.

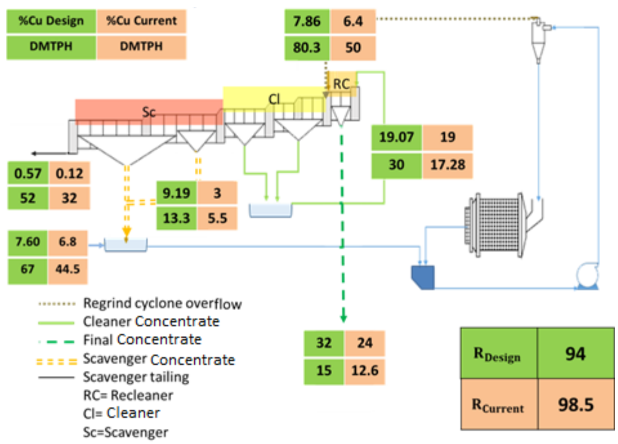


Fig. 5- Mass balanced data for the cleaner, recleaner, and scavenger stages.

When the gravity transportation of the final concentrate was successfully made operational, in addition to saving energy as a result of not using transfer pumps the recurrent problem of the overflow of sump boxes was eliminated. This prevented not only a loss of concentration but also saved 350,000 m³ of water annually used for washing the overflows and plant floor, and 740,000 kWh electrical energy used for pumps. After implementation of the gravity transportation of the final concentrate, the project of adding one cleaning stage using the existing cells, sumps, and pumps was started (Figures 6&7).

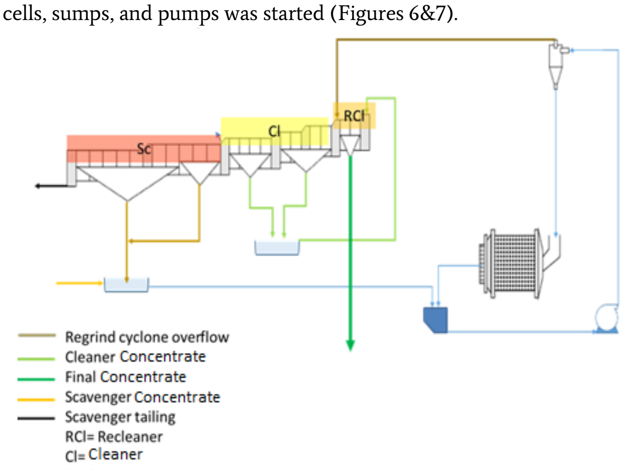


Fig. 6- Part of the layout of the flotation circuit (initial plant design)

At the first stage, one cleaner stage was added to only the north section of the circuit and the results of this section were compared with the identical south section. The results of performance evaluation for a period of 5 working days showed that the average copper grade increased in this section by 2.5% compared to the south section because of adding the new cleaning stage to the north section, (Fig. 8). It was then concluded that adding one cleaner stage to the circuit was more effective than previous modifications [4-7] to improve the grade.

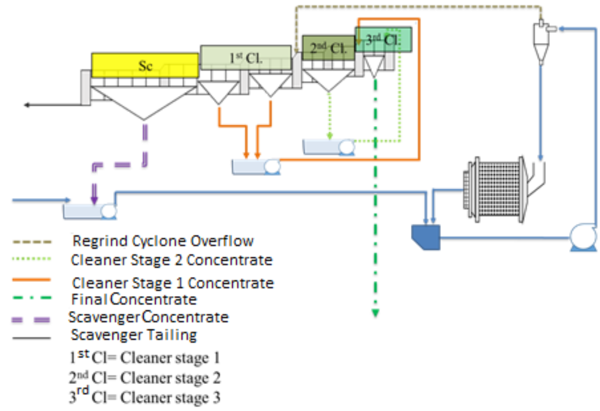


Fig. 7- Part of the layout of the flotation (after adding a cleaner stage).

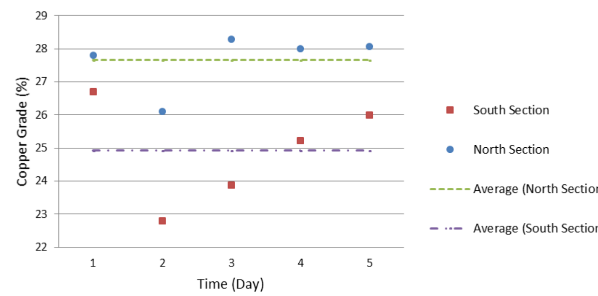


Fig. 8- Final copper concentrate grade in the north (new layout) and south sections (old layout).

On account of promising results of adding one cleanser stage in the north section, it also was implemented in the south section. The circuit modification increased the final copper concentrate grade by 2%. Figure 9 shows the final copper concentrate grade of the circuit for a period of 30 days before and 30 days after the modifications.

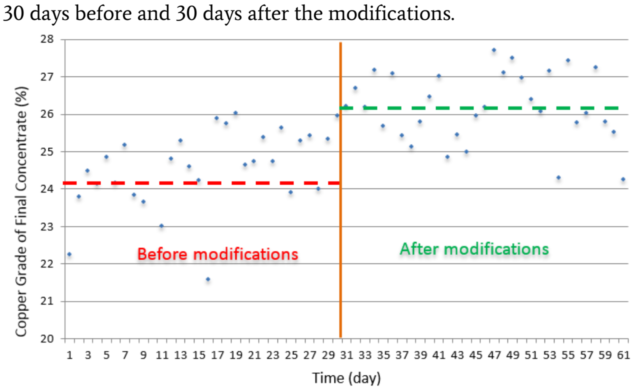


Fig. 9- Final copper concentrate before and after modifications.

5. Conclusions

- The reduction of copper content of the feed forced the solids flow rate of rougher concentrate to be reduced by 40% to maintain the desired stage grade.
- A reduction of 40% in the solids flowrate of the rougher concentrate increased the residence time in the subsequent flotation next stages by 70%.
- The increase in the residence time in the cleaner and scavenger stages decreased the copper grade of final concentrate from 32 to 24%.
- Gravity transportation of the final concentrate of the flotation circuit to the Cu-Mo thickeners in addition to annual saving 350,000 m³ of water and 740,000 kWh of the electrical energy made it possible to add one cleaner stage to the circuit.

- Adding one cleaner stage to the north section of the circuit increased the concentrate grade by 2.5% compared to the identical south section circuit.
- Monitoring of the circuit performance for a period of 30 days showed that the implementation of 3 stages of cleaning in the flotation circuit increased the final copper concentrate of the flotation circuit by 2%.
- It was concluded that adding one cleaner stage to the circuit was more effective than other modifications made to improve the grade.

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REFERENCES

[1] Wills B. A. and Finch J. A., "Chapter 12 - Froth Flotation," in *Wills' Mineral Processing Technology (Eighth Edition)*, B. A. Wills and J. A. Finch Eds. Boston: Butterworth-Heinemann, 2016, pp. 265-380.

[2] Radmehr Vahid, Shafaei Sied Z, Noaparast Mohammad, and Abdollahi Hadi, "Optimizing Flotation Circuit Recovery by Effective Stage Arrangements: A Case Study," *Minerals,* no. 8(10), pp. 1-14, 2018, DOI: https://doi.org/10.3390/min8100417.

[3] Ghobadi P., Yahyaei M., and Banisi S., "Optimization of the performance of flotation circuits using a genetic algorithm oriented by process-based rules," *International Journal of Mineral Processing,* vol. 98, no. 3, pp. 174-181, 2011/03/09/ 2011, DOI: https://doi.org/10.1016/j.minpro.2010.11.009.

[4] Banisi S., Kargar A., Pourkani M., Sarvi M., and Hamidi D., "Recent changes at the Sarcheshmeh copper mine flotation circuit," presented at the 33 rd Annual Canadian Mineral Processors Operators Conference, Ottawa, Canada, January 23-25, 2001.

[5] Banisi S., Sarvi M., Hamidi D., and Fazeli A., "Flotation circuit improvements at the Sarcheshmeh copper mine," *Mineral Processing and Extractive Metallurgy,* vol. 112, pp. 198-206, 2003, DOI: 10.1179/037195503225003681.

[6] Hamidi D., "Optimization of reagent addition to rougher flotation cells of the Sarcheshmeh concentration plant," M.Eng. Thesis, Mining Engineering Department, Shahid Bahonar University of Kerman, Kerman, 2001.

[7] Rajabi M.J., "Process auditing of the flotation circuit of the Sarcheshmeh copper complex," M.Eng. Thesis, Mining Engineering Department, Shahid Bahonar University of Kerman, Kerman, 2016.

[8] Poorkani M., "Investigation the possibility of increasing the copper grade of final concentrate of the Sarcheshmeh copper complex by adding a cleaner stage," R&D Section, Sarcheshmeh Copper Complex, 2014.

[9] AGAR G.E., "Optimizing the design of flotation circuits," *CIM Bulletin,* vol. 73, 824, pp. 173-181, 1989.

[10] Sutherland D. N., "A study on the optimization of the arrangement of flotation circuits," *International Journal of Mineral Processing,* vol. 7, no. 4, pp. 319-346, 1981/01/01/ 1981, DOI: https://doi.org/10.1016/0301-7516(81)90027-2.

[11] Banerjee P. K., Gupta A. K., Mukherjee A. K., Das P., Singh N. P., and Singh R. S., "Optimization of Reagents Distribution Down a Coal Flotation Bank to Improve the Recovery of Coarser Particles," *Coal Preparation,* vol. 27, no. 1-3, pp. 39-56, 2007/06/06 2007, DOI: 10.1080/07349340701249711.

[12] Bulatovic S. M., "12 - Flotation of Copper Sulfide Ores," in *Handbook of Flotation Reagents*, S. M. Bulatovic Ed. Amsterdam: Elsevier, 2007, pp. 235-293.

[13] Runge K.C., "Particle Size Distribution Effects that Should be Considered when Performing Flotation Geometallurgical Testing," presented at the second ausimm international geometallurgy conference, Carlton, VIC, Australia, 30 September-2 October 2013.

[14] Sao José F. and Pereira C., "Evaluation of Reagents Dispersing for Sphalerite and Galena Particles System," presented at the IMPC2014, Santiago, Chile, 20-24 October 2014, C1316.

[15] Tamara M., Valentin C., Tatiana I., and Nadezhda G., "New Reagent Modes for Selective Flotation of Gold-Sulfide Minerals from Refractory Ores," presented at the IMPC2012, New Delhi, India, 23-28 September 2012, 344.

[16] Vianna S., "The Effect of Particle Size, Collector Coverage, and Liberation on the Floatability of Galena Particles in an Ore," Ph.D. Thesis., Juius Kruttschnitt Mineral Research Center, Brisbane, 2004.

[17] Bazin C. and Proulx M., "Distribution of reagents down a flotation bank to improve the recovery of coarse particles," *International Journal of Mineral Processing,* vol. 61, no. 1, pp. 1-12, 2001/01/01/ 2001, DOI: https://doi.org/10.1016/S0301-7516(00)00022-3.

[18] Fosu S., Pring A., Skinner W., and Zanin M., "Characterisation of coarse composite sphalerite particles with respect to flotation," *Minerals Engineering,* vol. 71, pp. 105-112, 2015/02/01/ 2015, DOI: https://doi.org/10.1016/j.mineng.2014.08.023.

[19] Matveeva T.N., Chanturiya V.A., Ivanova T.A., and Gromova N.K., "New Reagent Modes for Flotation Recovery of Gold from Refractory," presented at the IMPC 2016, Quebec City, Canada, 11-15 September 2016.

[20] Mular M.A. and Veloo C., "Circuit Modifications at Westmin Resources Myra Falls Operations," presented at the Proceedings 24th Annual Meeting of the CMP, Canada, 1992, 26.

[21] Shannon L.K. and Trahar W.J., "The Role of Collector in Sulfide Ore Flotation," in *Advances in mineral processing*. Colorado: SME, 1986, pp. 408-425.

[22] Venter C. P.E. and Van Loggerenberg C., "Modifications to the coal-preparation circuit at the Grootegeluk Coal Mine to improve its efficiency," *Journal of the Southern African Institute of Mining and Metallurgy,* vol. 92, pp. 53-61, Feb. 1992 1992.

[23] Yarahmadi M. and Banisi S., "Movazen: A mass balancing software," presented at the Chemical Engineering Congress of Iran, Tehran, 1998.