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# Modofication of the Sarcheshmeh copper complex flotation circuit in response to a reduction in feed grade

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**Abstract**
The new copper processing plant of the Sarcheshmeh copper complex consists of two parallel circuits. After a primary crushing, the ore is sent to a SAG mill, and the product is further ground in a ball mill. The overflow of the hydrocyclones is fed to a flotation circuit that contains 8 rougher tank cells (RCS130), 3 cleaner cells (RCS50), 5 scavenger cells (RCS50), and a flotation column (as recleaner). The circuit was initially designed to process a feed containing 0.8% Cu but due to a change in the ore type, the feed grade decreased to 0.6% Cu. This resulted in a reduction in the final concentrate grade and the recovery from 28% and 85.5% to 24% and 84.4%, respectively. Based on the original design, the copper and silica recovery in the cleaner cells should be 69% and 55%, respectively, but these values increased to 85% and 75% due to a higher retention time. The rather high silica recovery was found to be the main source of the lower final concentrate grade. In order to reduce the retention time of particles in the cleaner cell from 13.7 to 6.9 min, the rougher concentrates of two parallel circuits were fed to only one cleaner-scavenger and regrind circuit. This modification increased the cleaner and final concentrate grade from 15.1% and 24.5% to 17% and 26%, respectively. The overall outcome of the circuit modification was evaluated to be a 10% reduction in the energy consumption without any loss in the overall copper recovery.

**Keywords:** *Flotation Circuit, Residence Time, Cleaning Stage, Flotation Circuit Modification.*

## 1. Introduction

The Sarcheshmeh ore body contains 1 billion tonnes averaging 0.65% copper and 0.035% molybdenum. After one stage of crushing in a gyratory crusher, the ore is fed to two significantly different processing plants. Plant No. 1 started in 1979 with a traditional three-stage crushing followed by ball milling; whereas plant No. 2, which was commissioned in 2005, uses a SAG mill and a ball mill. In plant No. 1, after two stages of crushing, the ore with a $F_{80}$ (80% passing size) of 12.7 mm is sent to eight parallel ball mills working in closed circuit with hydrocyclones to produce a product 70% finer than 75 microns. Plant No. 2 consists of two parallel lines, where two 10 m diameter by 4 m length SAG mills and two overflow balls mills (6.7 m diameter by 9.9 m length) working in closed circuit with 15 hydrocyclones (66 cm diameter and 12.7 cm apex). The overflow of hydrocyclones (28% solids, w/w) is fed to the flotation circuit and the underflow is recycled to the ball mill as the circulating load [1] (Figure 1).

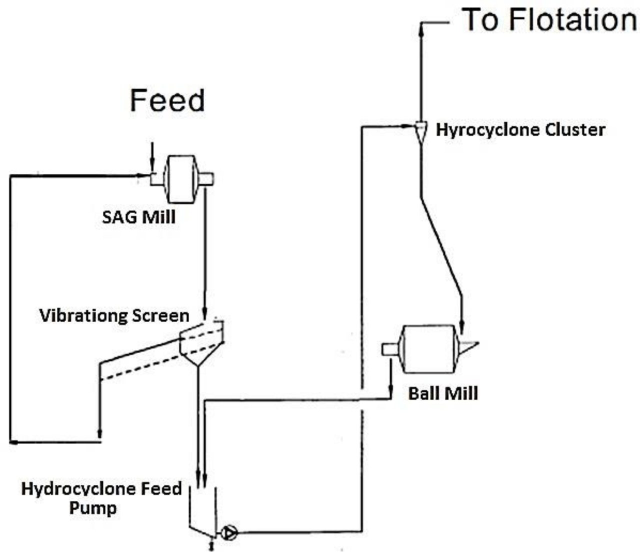


**Figure 1. Grinding circuit of plant No. 2 of the Sarcheshmeh copper complex [1].**

### 1.1. The Sarcheshmeh concentration plant flotation circuit

The flotation circuit consists of the roughing, regrinding, cleaning, recleaning, and scavenging stages. Regrind circuit includes a ball mill operating in a closed circuit with hydrocyclones (Figure 2). The rougher stage in each line uses 8 tank cells, each with a capacity of 130 $m^3$ (RCS 130). The rougher tailing constitutes most of the final tailing, and its concentrate is combined with the scavenging concentrate and is fed to the regrind circuit. The main function of the regrind section has traditionally been to provide the size for the final stage of flotation to reach the desired concentrate grade [2]. At the Sarcheshmeh concentration plant, it was found that particles in the range of 20-40 μm had the highest flotation rate constant (Figure 3) [3]. It is, therefore, logical to assume that the higher the amount of particles above 70 μm, the lower the copper recovery (Figure 3). The feed to the regrind circuit is first classified by 10 hydrocyclones (38.1 cm in diameter), where the underflows are sent to a ball mill (of 3.9 m diameter and 5.8 m length). The hydrocyclone overflows are fed to 3 flotation cells, each with a capacity of 50 $m^3$ (RCS 50) known as cleaners. The cleaner concentrate is transferred to a flotation column (of 4 m diameter and 12 m height), where the final concentrate is obtained. The tailing of the column cell is then combined with the cleaners' feed. The tailing of cleaner cells is gravity-driven to 5 tank cells, each with a capacity of 50 $m^3$ (RCS 50) to fulfill the scavenging task. The concentrate of this stage is combined with the rougher concentrate and is sent to the regrind circuit. The tailing along with the rougher tailing constitutes the final tailings.

The special arrangement of equipment in the plant has provided a good degree of connection flexibility to test various circuit configurations (Figure 4). For example, the overflow of two parallel primary hydrocyclone clusters could be combined and then distributed between two rougher units. Similarly, in the case of the regrind circuit feed, it could also be fed to the cleaning and scavenging sections.

At the concentrator No. 2 (Phase 1 and Phase 2), each line has been designed to treat 900 t/h of ore with a grade of 0.81% Cu to achieve 22 t/h of concentrate assaying 28% Cu. It has been observed that the mineralogy of ore changes as the mine depth increases. As a result of the ore type change, the feed grade decreased to 0.61% 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 15% coarser rougher feed compared to the original design. All these changes led to a reduction in the final concentrate from 28 to 24.5% Cu and also a decrease in the overall copper recovery from 85.5 to 84.4%. The objective of this work was to propose and implement a solution to increase the final copper concentrate grade such that it fulfills the minimum requirements for the smelting.

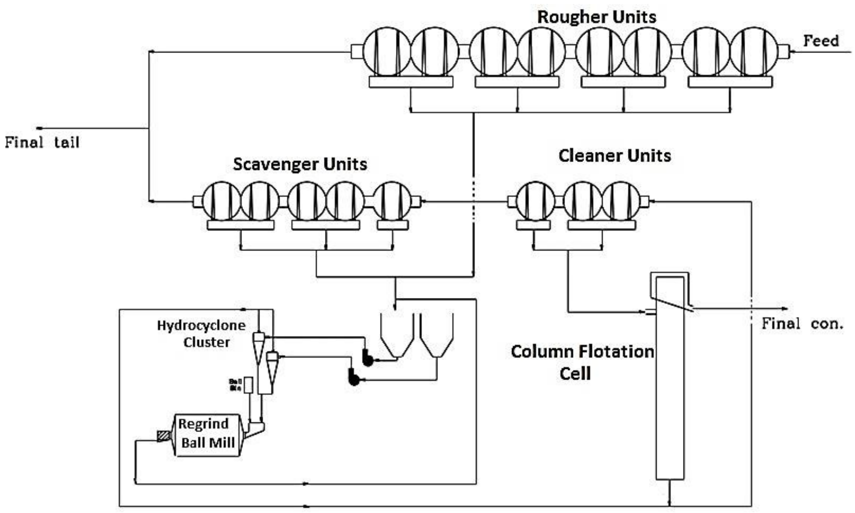


**Figure 2. Flotation circuit of the Sarcheshmeh concentrator No. 2 [1].**

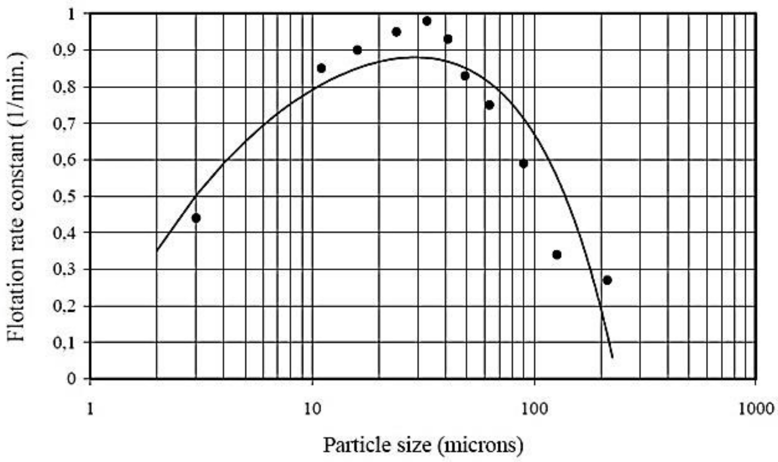


**Figure 3. Variation in flotation rate constant with particle size [3].**

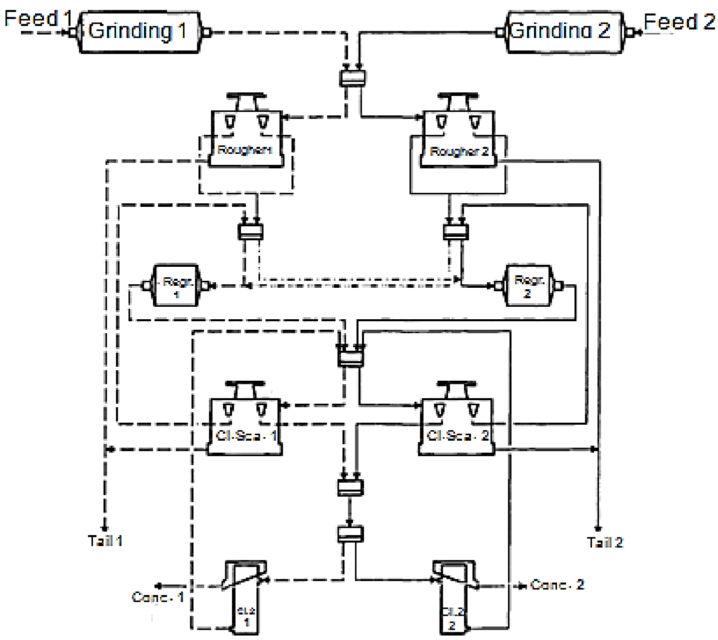


**Figure 4. Equipment layout of the Sarcheshmeh concentrator No. 2.**

## 2. Methodology

To evaluate the performance of the flotation circuit, the circuit was sampled at various operating conditions and the data were mass balanced using a mass balancing software (Movazen 2.1) [4]. The recovery of each stage along with the grade and flowrate of various streams were also calculated and compared with the design values. To evaluate the performance of the flotation circuit after using only one line of cleaning and scavenging stages, samples were also taken from various streams, and the new data were mass balanced and compared with the pervious data.

## 3. Results and discussion

### 3.1. Comparison of the original and proposed circuit configurations

A comparison of the present assays and flowrates of the circuit with the design data are shown in Table 1. Inspection of the data shows that except for the feed rate, the difference between the design and current values is significant. For example, the mass flowrate of the rougher concentrate decreased from 129 t/h (design) to 104 t/h (present), and the assay also followed a decline trend from 5 to 4.5%. The substantial differences between assays were observed for the cleaning section where a 9% reduction in the cleaner concentrate grade was realized. The enrichment ratio (i.e. concentrate grade divided by the feed grade) increased in all states except for the cleaning stage compared with the design data (Table 2). In the cleaning stage, the enrichment ratio decreased from 4.1 to 3.4 as a result of the operators' effort to achieve the desired overall recovery. As a result, it was decided to initiate a detailed study on the cleaning section.

The current overall recovery of the circuit is 84.4%, which is 1.1% lower than the design value. It was observed that the rougher recovery decreased by 2.5%, while in the cleaning section, the recovery increased by 17.8% (Table 3). The low concentrate grade and enrichment ratio of the cleaning stage (Tables 1 and 2) make the task of achieving the desired final concentrate grade very difficult. In order to increase the final concentrate, the cleaner concentrate grade should normally be increased, which results in a reduction in the stage recovery.

To increase the grade of the cleaning stage, a practical approach is to reduce the retention time in this stage. The mean retention time of the cleaning stage increased from 7.9 to 13.7 min due to a decrease in the feed mass flowrate. On account of a reduction in the feed rate, the scavenger stage retention time also increased from 14.6 to 30.2 min (Table 4). To decrease the retention time in the cleaning and scavenging stages, two parallel lines were combined and fed to one processing line. In other words, in this configuration, only one line of cleaning and scavenging section was used.

**Table 1. Comparison of design mass flowrates and copper assays with the current values.**

| Stream | Mass Flowrate (t/h) | | Assay Cu (%) | |
|---|---|---|---|---|
| | Design | Current | Design | Current |
| Rougher Feed | 900 | 900 | 0.81 | 0.61 |
| Rougher Concentrate | 129 | 104 | 5 | 4.5 |
| Rougher Tailings | 771 | 796 | 0.11 | 0.09 |
| Rougher+Scavenger Concentrate | 161 | 119.6 | 5.8 | 4.48 |
| Cleaner Concentrate (Recleaner Feed) | 29 | 32 | 24 | 15.1 |
| Cleaner Tailings (Scavenger Feed) | 139 | 101 | 2.22 | 0.79 |
| Scavenger Concentrate | 32 | 15.6 | 9 | 4.35 |
| Scavenger Tailings | 107 | 85.4 | 0.17 | 0.14 |
| Final Tailings | 878 | 881.4 | 0.12 | 0.10 |
| Final Concentrate | 22 | 18.6 | 28 | 24.5 |

**Table 2. Comparison of the design enrichment ratios and current values.**

| Flotation stage | Current Enrichment Ratio | Design Enrichment Ratio | Relative Changes (%) |
|---|---|---|---|
| Rougher | 7.1 | 6.2 | **15.7** |
| Cleaner | 3.4 | 4.1 | **–18.5** |
| Recleaner | 5.5 | 4.1 | **35.8** |
| Scavenger | 1.6 | 1.2 | **39.1** |

**Table 3. Design and current assay and recovery of various flotation stages.**

| Stage | Concentrate Grade Cu (%) | | Recovery (%) | |
|---|---|---|---|---|
| | Design | Current | Design | **Current** |
| Rougher | 5 | 4.5 | 88.4 | **85.9** |
| Cleaner | 24 | 15.1 | 69.1 | **86.9** |
| Recleaner | 28 | 24.5 | 90.8 | **88.1** |
| Scavenger | 9 | 4.35 | 94.1 | **85.0** |
| Cleaner-Recleaner-Scavenger | - | - | 96.8 | **97.4** |
| Cleaner-Recleaner | - | - | 67.0 | **85.1** |
| Overall | 28 | 24.5 | 85.5 | **84.4** |

**Table 4. Design and current volume and retention time of various flotation stages.**

| Stage | Total Volume ($m^3$) | Mean Residence Time (min) | |
|---|---|---|---|
| | | Design | Current |
| Rougher | 1040 | 23.7 | 23.7 |
| Cleaner | 150 | 7.9 | 13.7 |
| Scavenger | 250 | 14.6 | 30.2 |

### 3.2. Metallurgical changes after circuit modification

The flotation rate constant could be used to quantify the effects of numerous variables on the flotation process [5]. It was experimentally shown that the kinetics rate constant of copper and silica bearing minerals in the cleaner stage was 0.117 and 0.085 1/min, respectively [6]. Taking these values into consideration and knowing that the residence time in the cleaners is 13.7 min, the current recovery of 87% and 75% for copper and silica bearing minerals, respectively, is plausible. It should be mentioned that the respective recoveries were 69% and 55% in the original design. The rather high silica recovery (i.e. 75%) has been found to be the main cause of the lower final concentrate grade. Another issue with the flotation circuit is the high sensitivity of the final concentrate to the feed silica content. This caused a significant difficulty in increasing the recovery of the roughers due to the direct impact of its silica content on the final concentrate grade. In practice, this imposed a serious obstacle over the recovery increase because of a limitation on the silica content of the final concentrate.

By combining the feed of two cleaner stages and transferring it to one line, the slurry mean retention time in the cleaning and scavenging stages were estimated to become 6.8 min and 15.1 min, respectively. Based on these retention times it could well be estimated that the respective copper and silica recovery decrease to 65% and 50%. These changes could increase the cleaner and final concentrate grades to 17% and 27%, respectively. Because of a high retention time in the scavenger cells, the overall recovery is thought to be unaffected by these changes. To prevent a possible reduction in the recovery, the froth depth in the scavenger cells was reduced.

### 3.3. Circuit modification and new opportunities for improvement

The objective of increasing the final concentrate grade without an adverse effect on the recovery was fulfilled by using only one line of cleaning, scavenging, and regrinding. These changes saved 10% of energy consumed in the processing plant, and increased the final concentrate assay by 1.5%. A slight increase of 0.02% in the scavenger stage tailings grade caused a small decrease in the recovery of cleaner-recleaner-scavenger section from 97.4 to 97.0%. The overall recovery decrease was 0.3%, which was not found to be significant in comparison to an increase of 1.5% in the final concentrate grade. With the modified circuit configuration, the cleaning stage recovery became very close to the original design, and in order to increase the overall circuit recovery, it was decided to explore ways of increasing the recovery of the roughers.

In the modified circuit, the reasonable recovery of the cleaning stage decreased the circuit sensitivity to the feed and rougher concentrate grades. This made possible to increase the recovery of roughers by increasing the dosage and addition points of reagents, in particular, collectors. A detailed study in the plant where various combinations of stage addition were tested, revealed that 75-25-0% (i.e. percentages of total reagents added to the first, second, and third unit, respectively) addition regime could increase the total copper recovery by 1.3% [7-8]. The modified circuit configuration put one line of cleaner, scavenger, and regrind section on standby, and provided an extra flexibility for the maintenance, which, in turn significantly increased the circuit availability.

## 4. Conclusions

- A decrease in the feed grade due to the ore type change at the Sarcheshmeh copper complex plant reduced the final concentrate grade from 28% to 24.5% Cu.
- Increasing the mean retention time in the cleaning stage from 7.9 min (design) to 13.7 min decreased the enrichment ratio from 4.1 to 3.4, and increased the recovery from 69.1% to 86%.
- By transferring the cleaning stage feed to only one line of cleaner-scavenger units instead of the usual two lines, the mean retention time decreased from 13.7 to 6.9 min.
- On account of the modification of the circuit configuration, the final concentrate grade increased from 24.5% to 26%, while the scavenger tailings increased by 0.02%, which led to a decrease of only 0.3% in the overall recovery.
- It was found that the recovery of the cleaner-scavenger-recleaner section even after the circuit modification was 0.2% higher, while the rougher recovery was 2.5% lower than the design values. This made the recovery increase in the rougher cells the only viable option to increase the circuit overall recovery.
- By using only one line of cleaner, scavenger, and regrinding section, 10% saving in the energy was realized, and an extra flexibility for maintenance significantly increased the circuit availability.

**Acknowledgments**

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# اصلاح مدار فلوتاسیون کارخانه تغلیظ ۲ مجتمع مس سرچشمه با توجه به کاهش عیار خوراک

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**چکیده:**

مدار فرآوری کارخانه تغلیظ ۲ مجتمع مس سرچشمه از دو خط موازی تشکیل شده است. خردایش در هر خط توسط یک آسیای نیمه خودشکن و یک آسیای گلوله‌ای که در مدار بسته با یک خوشه هیدروسیکلون است، انجام می‌گیرد. مدار فلوتاسیون هر خط از واحدهای پرعیارکنی اولیه (۸ سلول RCS130)، شستشو (۳ سلول RCS50)، رمق‌گیر (۵ سلول RCS50) و شستشوی نهایی (یک ستون ۱۲×۴ متر) و یک آسیای گلوله‌ای برای خردایش مجدد کنسانتره پرعیارکنی اولیه و رمق‌گیری تشکیل شده است. این مدار برای سنگ معدنی با عیار ۰/۸ درصد مس طراحی شده است ولی با تغییر مینرالوژی سنگ معدن، عیار مس خوراک به حدود ۰/۶ درصد کاهش یافته است، در نتیجه عیار کنسانتره نهایی از ۲۸ درصد به ۲۴/۵ درصد و بازیابی کلی مدار نیز از ۸۵/۵ به ۸۴/۴ درصد کاهش یافته است. با توجه به طرح اولیه کارخانه، بازیابی مس و سیلیس در واحد شستشو به ترتیب باید برابر با ۶۹ و ۵۵ درصد باشند. اما به دلیل افزایش زمان ماند این اعداد به ترتیب برابر با ۸۵ و ۷۵ درصد می‌باشند. افزایش بازیابی سیلیس، عامل اصلی کاهش عیار مس نهایی است. از این‌رو، در این تحقیق برای کاهش زمان ماند ذرات در واحد شستشو از ۱۳/۷ به ۶/۹ دقیقه، کنسانتره سلول‌های پرعیارکنی اولیه هر دو خط، فقط به یک ردیف سلول شستشو-رمق گیر- خردایش مجدد خوراک‌دهی شدند. در نتیجه تغییر مدار، عیار واحد شستشو از ۱۵/۱ درصد به حدود ۱۷ درصد و عیار کنسانتره نهایی از ۲۴/۵ درصد به حدود ۲۶ درصد افزایش یافت. همچنین به واسطه تغییرات اعمال شده ۱۰ درصد کاهش در مصرف انرژی، بدون هیچ تغییری در بازیابی کلی مس مشاهده شد.

**کلمات کلیدی:** مدار فلوتاسیون، زمان ماند، مرحله شستشو، اصلاحات مدار فلوتاسیون.