COLUMN FLOTATION STUDIES IN COPPER FLOTATION CLEANING CIRCUIT AT ETI BAKIR A.S. KURE CORPORATION

ETİ BAKIR A.Ş. KÜRE İŞLETMESİ BAKIR FLOTASYONU TEMİZLEME DEVRESİNDE KOLON FLOTASYONU ÇALIŞMALARI

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ABSTRACT

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The aim of this thesis is to investigate the possibility of increasing final concentrate grade by using column flotation at Eti Bakır A.Ş. Küre flotation plant. Since pyrite is the main gangue mineral in the concentrate, removing it from the copper concentrate also increases cobalt recovery into pyrite concentrate. For this purpose, experimental studies were performed by using a pilot scale flotation column. The studies were carried out at the first and second cleaning circuits and sampling surveys were performed in these circuits on the same day for the comparison the results with column flotation.

The studies started with the determination of the optimum parameters and the effect of airflow rate, type of bubble generator, feed rate, froth height and wash water rate were investigated. Final tests were conducted in the second cleaning circuit at optimum conditions. Higher concentrate grade was obtained by column flotation than the final copper grade obtained after four stages of cleaning at the process plant. This resulted in a decrease in the amount of cobalt reporting into the copper concentrate.

Keywords: Flotation, column flotation, flotation test works, flotation parameters

ÖZET

ETİ BAKIR A.Ş. KÜRE İŞLETMESİ
BAKIR FLOTASYONU TEMİZLEME DEVRESİNDE
KOLON FLOTASYONU ÇALIŞMALARI

FIRAT TUNCER

Yüksek Lisans, Maden Mühendisliği

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Ocak 2023, 68 sayfa

Bu tezin amacı, Eti Bakır A.Ş.'de kolon flotasyonu kullanılarak nihai konsantre tenörünün yükseltilme olasılığının araştırılmasıdır. Konsantredeki ana gang minerali pirit olduğu için, bakır konsantresinden çıkarılması kobaltın pirit konsantresine geri kazanımını da artırır. Bu amaçla pilot ölçekli flotasyon kolonu kullanılarak deneysel çalışmalar yapılmıştır. Çalışmalar 1. ve 2. temizleme devrelerinde yapılmış ve sonuçların mevcut sistemde ile karşılaştırılması için bu devrelerde aynı gün örnekleme sörveyleri yapılmıştır.

Optimum parametrelerin belirlenmesi ile başlayan çalışmalarda hava debisi, kabarcık oluşturucu tipi, besleme hızı, köpük yüksekliği ve yıkama suyu oranının etkisi incelenmiştir. İkinci temizleme devresinde optimum koşullarda son testler yapılmıştır. Proses tesisinde dört aşamalı temizlikten sonra elde edilen nihai bakır derecesinden daha yüksek konsantre tenoru kolon flotasyonu ile elde edilmiştir. Ayni zamanda bakır konsantresine kazanılan kobalt miktarında da azalma sağlanmıştır.

Anahtar Kelimeler: Flotasyon, kolon flotasyon, flotasyon test çalışmaları, flotasyon parametreleri

TEŞEKKÜR

Öncelikle tez konumun belirlenmesinde ve çalışmalarım esnasında yapıcı ve yönlendirici katkılarını her zaman sağlayan, değerli hocam Prof. Dr. Ş. Levent Ergün'e;

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Yüksek lisans için beni cesaretlendiren, desteğini bir an olsun eksik etmeyen, her zaman daha iyisini yapmam için beni motive eden hayat arkadaşım, canım eşim İdil Mutlu Tuncer'e;

Hayatımın her anında bana destek veren, bugünlere gelmemi sağlayan, yardımlarını asla esirgemeyen, var olma sebeplerim babam Tuncay Tuncer ve annem Fetiye Tuncer'e, canım kardeşlerim Murat Tuncer ve Meriç Tuncer'e;

Minnet, sevgi ve teşekkürlerimi sunarım.

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ABBREVIATIONS AND SYMBOLS

C	Carrying capacity
Df	Bubble diameter in the froth
β	Particle packing factor on the bubble surface
ρ	Specific gravity of particle
Qg	Flowrate of the gas
D_p	Diameter of particle in the froth
ΔP	Difference of pressure between two points
ΔL	Vertical distance of two points
ρ_{sl}	Density of the slurry
V_s	Slurry volume
V_p	Probe Volume
k_o	Slurry conductivity with bubbles
k_s	Slurry conductivity without bubbles
S_b	Bubble surface area flux
J_g	Superficial gas velocity
d_b	Bubble diameter
J_g	Superficial gas velocity
Q	Volumetric air flow rate
A	Flotation cell cross sectional area
P_o	Atmospheric pressure
P_L	Pressure changing depending on the level at the bottom of the pipe
ρ_b	Aerated pulp density
J_b	Bias
J_c	Superficial concentrate rate
J_t	Superficial tailings rate
k_f	Feed conductivity
k_c	Concentrate conductivity
k_t	Tailings conductivity
k_w	Wash water conductivity
BU4	Copper flotation first cleaner circuit
BU7	Copper flotation second cleaner circuit

1. INTRODUCTION

The selectivity of flotation columns is higher than mechanical cells and provides higher concentration grade [1]. In the copper circuit, 18-20% and 0.34-0.37% grade concentrate is obtained with 85-88% recovery from the ore containing 2-2.5% copper and 0.32-0.40% cobalt. The flotation feed particle size P80 is around 45 microns. The problem of selectivity is the main issue in the plant, which is ground to a very fine particle size. In addition to this situation, pyrite and cobalt also come to the concentrate and losses are seen in the pyrite circuit. Although operational studies were carried out on existing mechanical cells, this problem could not be overcome, and this situation brought along new searches. For this reason, flotation columns, which are known to be more effective in terms of selectivity, have been tested on a pilot scale at the process plant and the results have been compared with the existing circuit.

The exploitation of resources has led to a gradual decrease in the grades of ore deposits. As a result, today's industry is faced with more complex ore types, selection problems in flotation, and ore types released in finer particle sizes. The need for innovative ideas to overcome these problems encountered in flotation has led to new equipment designs. Thus, flotation columns have found a prominent place in the sector [2].

The first industrial test workers of flotation columns, patented by Boutin and Tremblay in 1960, were made in 1963 at the Iron Ore Company of Canada and Opemiska Mines in Canada. The first commercial installations of the flotation columns, which continued to be developed in the following years, were completed in 1980 in the molybdenum cleaning circuit of Noranda's Gaspe Mine. In 1984, a three-stage copper cleaning column flotation circuit was established in the Gibraltar Mine and they were first designed as a circuit and took place in the industry. In 1988, a circuit consisting entirely of flotation columns was designed and commissioned in Pocatello, a phosphate mine [3].

As in conventional flotation, column flotation is also affected by interdependent or independent factors such as bubble size, feed density, pH and Eh values, reagent types and amounts, froth size, particle size, and mineral concentration. The decrease in the liberation size and complexity of the ore causes selectivity problems in the flotation circuit [3].

Within the scope of this thesis, the pilot column flotation studies were carried out in the first and second copper cleaning circuits within the Eti Bakir A.S. Kure Corporation Copper-Pyrite Flotation Plant. The purpose of the study is to increase the final copper concentrate and reduce the amount of pyrite and cobalt in the concentrate.

2. GENERAL INFORMATION

2.1. Flotation

The flotation process, the development of which can be examined in 3 main periods in the industry, emerged between 1860 and 1900 to float the valuable minerals in the ores, agglomerate, or to recover the fine-particleed concentrates from the tailings. These studies were relatively small-scale interventions. However, after 1900, when the necessity of economically concentrating fine-particleed sulphurous ores especially for copper minerals arose, major studies were conducted in Broken Hill in Australia until 1915 and in the western United States until 1925. In these periods, most of the copper used in the wires that provide the transmission of electricity was obtained by the flotation process. After 1960, as a result of modern developments, new flotation equipment was developed with the development of X-ray and on-flow radioisotope analysis systems, which provide faster and more accurate analysis results and enable accurate process control [4].

Flotation is a physicochemical process that is carried out by taking advantage of the hydrophobic and hydrophilic properties of minerals based on their surface chemistry. The system has a complex structure that includes physical and chemical interactions in addition to solid, liquid, and gas phases. Variables such as particle size, ore content, degree of liberation related to the ore, and variables such as bubble size and air rate related to the machine constitute physical factors. The reagents and surface chemistry that affect or provide the ore to be hydrophobic and hydrophilic constitute the chemical factors [3]. Although the flotation process was developed to process sulphurous minerals of copper, lead, and zinc, nowadays it is also used to process sulphite minerals containing metals such as gold, nickel, and platinum, oxides compounds such as hematite, cassiterite, and ores such as fluorite, potash, coal, and bitumen [1]. Generally, the valuable minerals are taken into the froth phase and it is called true flotation, on the other hand, if the separation is carried out by taking the tailings in the froth phase, it is called reverse flotation. So, either way, the flotation process is possible and it's all about the particles being hydrophobic or hydrophilic.

2.2. Flotation Chemicals

2.2.1. Collectors

The purpose of the use of collectors, which are organic compounds, is to bind to the surface of minerals that are not naturally hydrophobic and make them hydrophobic. The collectors consist of polar and non-polar ends. The polar ends are absorbed into the mineral surface and the non-polar part takes a place towards the slurry, so that the mineral surfaces gain hydrophobic properties through the non-polar part. Collectors are divided into 2 groups ionizable or non-ionizable. While ionizable collectors dissociate into their ions in water and adhere to mineral surfaces, non-ionizable collectors cover the mineral surfaces as a thin film layer [5]. Figure 1 shows the classification of collectors.

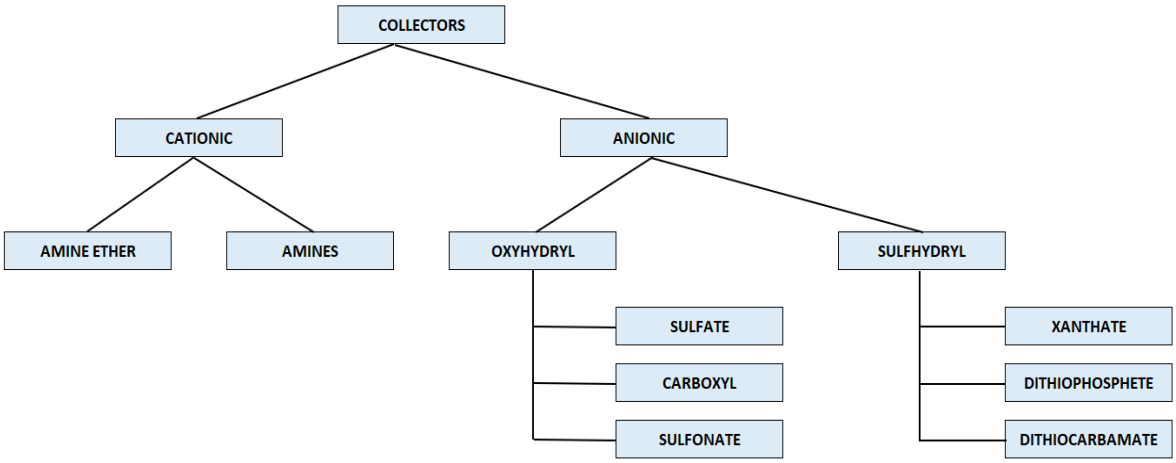


Figure 1. Classification of collectors [5]

2.2.2. Frothers

Frothers, which are surface-active compounds, are absorbed in the water-air interface and contain a hydrocarbon radical and a polar group. The non-polar (hydrophobic) parts tend towards the gas phase and the polar parts are oriented to the water phase. While frothers are providing the stability to air bubbles created by air bubble generating systems such as rotor-stator or sparger-cavitation systems, they also reduce the surface tension of water. The frothers place in the part between water and air, thus preventing the bubbles from encountering each other [6].

2.2.3. Modifiers

Modifiers are classified as pH adjusting reagents, activators, depressants, and dispersants reagents. They are used when the surface of a mineral is not suitable for flotation. Activators are chemicals used for molecules that cannot be attached to the collector but need to be floated. It is generally formed by chemically adsorbing metal ions to the mineral surface. For example, CuSO4 (Copper Sulphate) is used to regenerate ZnS (Zinc Sulphate). Due to the reaction between copper sulphate and zinc sulphate, a structure like the covellite mineral forms on the sphalerite [7]. Depressants are used when a collector is not desired to be absorbed by a mineral. For example, ZnSO4 (Zinc Sulphate) can be given as an example to suppress sphalerite in sulphur flotation, and SMBS (sodium metabisulfite) to suppress galena [8]. pH regulators are used to provide optimum flotation conditions. Some of the chemicals used in general are lime, sulfuric acid, and soda ash. In addition, dispersants are also used for desired to distribute the clay content, for instance.

3. FLOTATION MACHINES

A flotation machine has a bubble-forming system so that the particles can be attached and concentrated. This interface, which is created to ensure maximum contact of particles and bubbles, is provided by the air fed into the system [8]. Diverse types of flotation machines have been developed in the past. Although some of these machines cannot protect their existence today, some designs are used in large masses and scales today by being developed. Generally, flotation machines can be divided into 3 main groups according to air supply types: mechanical flotation machines, pneumatic flotation machines (separator-reactor machines), and flotation columns [5]. In addition to these 3 groups, there are also vacuum and dissolved air flotation, which is based on the precipitation of dissolved air on particles, and electroflotation, which is based on the principle of hydrolysis of water to form fine hydrogen and oxygen bubbles, but they could not find a place for themselves as much as the first 3 groups [8].

3.1. Mechanical Flotation Machines

This type of machine consists of 3 main zones. Firstly, there is a high turbulent zone to increase the probability of collision of the bubbles with the particles and this turbulence is created by a stator and rotor (or impeller) and the particles are kept in a suspension. Secondly, there is a quiescent zone with relatively less turbulence where the bubbles holding the particles can rise easily and the froth zone where the concentrate is transported. Subsequently, either mechanical pallets are used to remove the transported bubbles from the system, or the bubbles are overflowed unaided [9]. 3 main zones are shown in Figure 2.

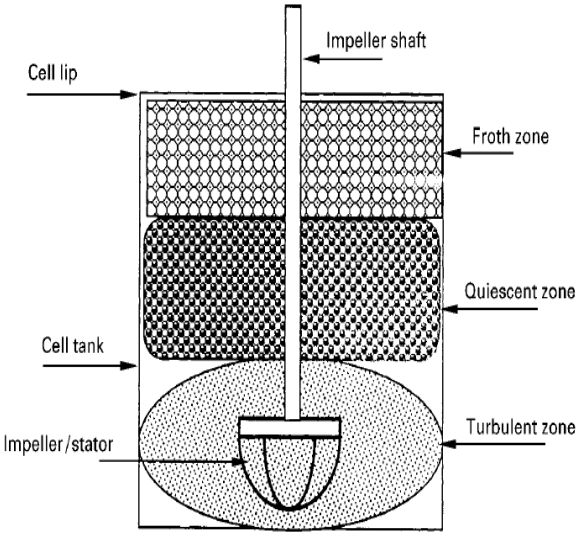


Figure 2. Hydrodynamic zones in a mechanical flotation cell [9]

Dispersion of gas into fine bubbles is one of the most essential hydrodynamic characteristics in a mechanical flotation cell. In a mechanical cell, the bubble generating mechanism is a two-stage process. First, a low-pressure zone is occurred by creating air gaps at the rear of the wings. Later, when this air gap is separated from the wings with the help of the stator, bubbles are formed. Three features of air dispersion into bubbles may be identified: bubble size, gas hold-up, and superficial gas velocity. In industrial mechanical cells, the mean bubble size ranges from 0.5 to 2 mm, gas hold-up ranges from 5 to 15%, and superficial gas velocity ranges from 0.6 to 1.5 cm s^{-1}, depending on cell operating conditions (impeller speeds and air rates) related with the cell duty in plant operations such as roughing, scavenging, cleaning. There are two designs in which the air enters the cell. The first of these designs is the air system fed through the rotor shaft with the help of a blower, and the second is the air system that is vacuumed into the system with the vortex effect [10].

3.1.1. Denver Cells

The Denver "Sub-A," which was commonly employed in small plants and multistage cleaning circuits, was possibly the most well-known cell-to-cell machine. Although the machine name and basic design looked to have derived from an older machine made by Minerals Separation Corporation, one of the pioneering businesses in flotation cell design and manufacture, Arthur W. Fahrenwald patented the Sub-A design in 1922 [11]. An adjustable weir separates the Sub-A impeller mechanism from the next square cell. A feed

pipe transports pulp from the previous cell's weir to the next cell's impeller, with the impeller's suction assisting flow. The impeller's suction draws air down the hollow standpipe that surrounds the shaft and into the pulp. As the air and pulp are intimately combined by the revolving impeller, the air stream is sheared into fine bubbles typically with the help of a frother. A stationary cowl sits directly above the impeller, preventing cell-to-cell"sanding-up" of the impeller when the machine is turned off. Four baffles are attached to the hood and reach almost to the cell's corners. The baffles reduce pulp agitation above the impeller, resulting in a quiet zone where mineralized bubbles can climb without being exposed to high turbulence, which could cause particle detachment. The crowding force of subsequent bubbles carries the bubbles higher to the overflow as they cross from the pulp zone to the froth zone. In some circumstances, froth is removed by revolving froth paddles that help with overflow. Particles that are too heavy to flow over the tailings weir are diverted through sand relief ports, which prevent coarse material from accumulating [12]. The Denver cell-to-cell design is shown in Figure 3.

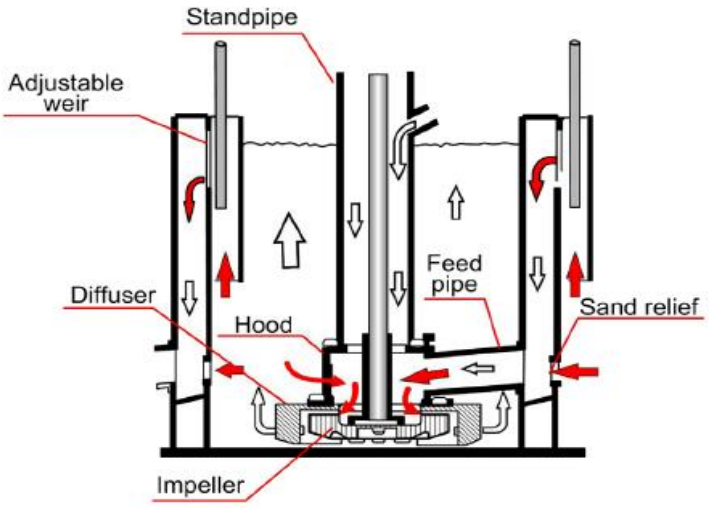


Figure 3. Denver cell to cell design [13]

The Denver D-R machine, which came in sizes ranging from 2.8 to 36.1 m^3 and was designed in response to the requirement for a machine that could handle greater tonnages in bulk flotation circuits, required supercharging. The lack of intermediate partitions and weirs between cells distinguished these units. Individual cell feed lines have been removed, allowing the pulp to flow freely throughout the machine. A single tailings weir at the trough's end could automatically manage the pulp level. The operation of the bank is relatively

straightforward, and the requirement for operator attention is minimal [12]. The Denver D-R open circuit design is shown in Figure 4.

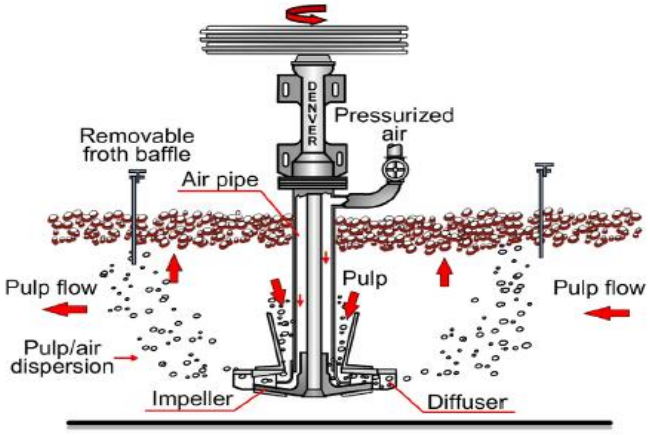


Figure 4. Denver D-R open circuit design [12]

3.1.2. Galigher Agitair

The Galigher Agitair is one of the most well-known forced-air machines. A gravity head produces a straight-line flow of pulp through a row of cells in this system. A separate impeller rotates in a stationary baffle system in each compartment, which may have a volume of up to 42.5 m3. The pulp is blown into fine bubbles by air forced into the pulp through the hollow standpipe encircling the impeller shaft, with the volume of air adjusted separately for each compartment. The depth of pulp in each cell can be regulated by adjusting the number and size of froth weir bars provided for each cell, while the depth of froth in each cell can be controlled by varying the number and size of froth weir bars provided for each cell [13]. The general view of the Agitair cell is as in Figure 5.

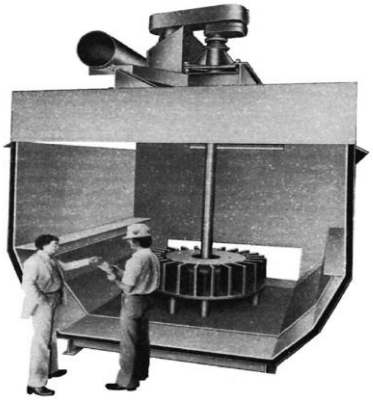


Figure 5. Agitair Cell [13]

3.1.3. Wemco Fagergen Cells

The earliest Fagergren flotation machines were self-aerated and used a horizontal impeller mechanism, similar to other flotation machines of the time. At the beginning of the second quarter of the 1900s, the model developed into a vertical rotator with a draft-tube and false bottom for sludge recirculation, and Western Machinery Co. (WEMCO) started offering the machines, which became prominent under the name WEMCO Fagergren 1+1 cells at the 1970s which means of the 1+1 is equality of the length of the rotor to its' diameter [12].

Wemco SmartCells was designed on a circular tank, which is a significant shift from Eimco's usual precedent of only offering rectangular cells for normal flotation operations. Thus, the dead zones formed in the corners are eliminated. Despite this drastic change, they did not give up on the aeration system design of Wemco 1+1 cells, which has proven their reliability. Instead of the cylindrical-shaped draft tube used only in old designs, they have developed the design of the conical-shaped draft tube, which increases the chance of contact of particles and bubbles by creating a more stable mixing environment [14]. Wemco Fagerfgen Cell design can be seen in Figure 6.

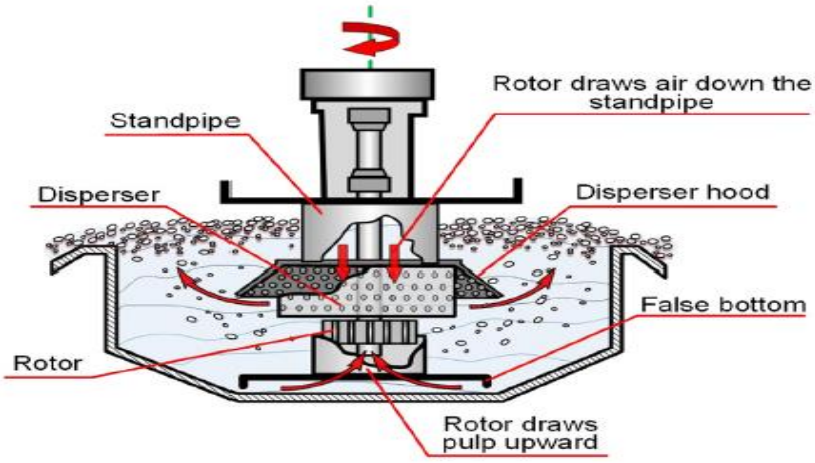


Figure 6. Wemco Fagergen Cell [12]

3.1.4. Dorr-Oliver Cells

At the beginning of the fourth quarter of the 1900s, Dorr-Oliver created a flotation cell in the meet to the requisition for larger cells. Through an aperture in the impeller top plate, air flows through the standpipe surrounding the impeller shaft and into the gas room. The cells are available in rectangular, U-shaped bottom, and cylindrical configurations of varied sizes [12]. Figure 7 shows the Dorr-Oliver Cell design.

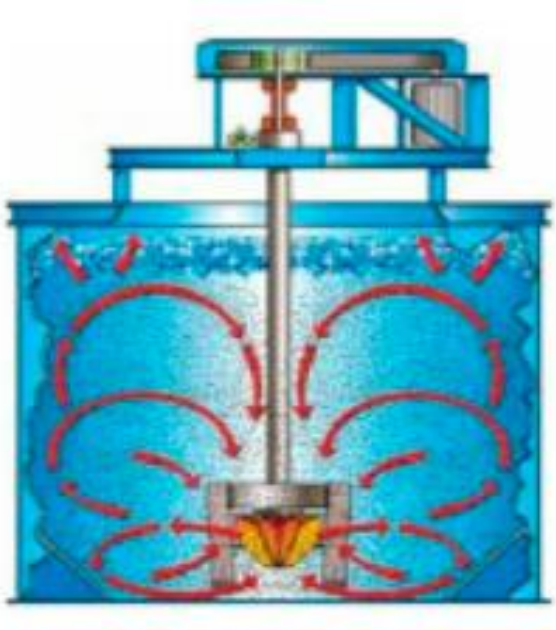


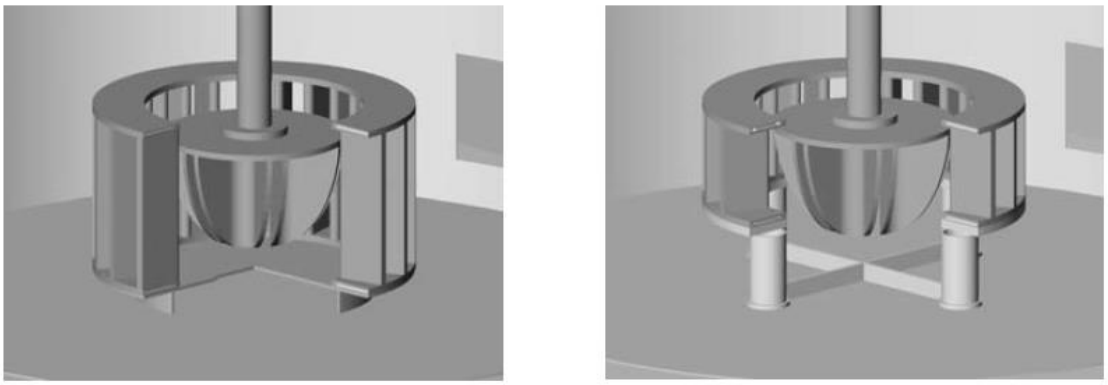
Figure 7. Dorr-Oliver Cell [15]

3.1.5. Outokumpu OK Cells (Tank Cells) and RCS System

3.1.5.1. Tank Cell Designs

Outokumpu flotation cell research began in the last part of the 1960s, principally to meet the company's own needs in processing complicated sulphide ores at its mines in Finland better than the available current machines. The OK rotor/stator design was the first significant innovation. In 1970, a research group of experienced metallurgists from Outokumpu's primary concentrators, as well as specialists in physics and hydrodynamics, continued to create a patented, large-volume flotation machine for internally operate as well as export. The first commercial tank cell, the TC-16, was installed in East Malartic (Now Barrick Gold) in 1991 [16].

In addition to the tank cells of the R-series, which have a rectangular shape, and the U-series in the traditional form, there is two types of Outokumpu impeller designs MM and FF. The MM (Multi mix) design is for the widespread use and provides enhanced buoyancy in fine particles. The FF (free flow) design is designed for coarse particles and provides great size. Feeding and discharging are provided by connections placed in rectangular boxes and these boxes are semicircular. In general, tank cell designs have been developed without the need for baffles, but they can be added when the extra suspension is desired [16]. MM and FF rotor-stator designs of the tank cells are shown in Figure 8.



(a) **(b)**

Figure 8. Tank Cells MM rotor-stator design (a) and FF rotor-stator design (b) [16]

3.1.5.2. Skimair Cells

Skimair cells are special flotation cells applied to the coarse particles of the material coming from the grinding circuit after the classifier. With this flash flotation, over-grinding of the material is avoided, and a larger concentration is obtained. Although it is generally produced for soft and high specific weight signs such as gold, it is also widely used in the flotation of copper, nickel, and lead [16]. Figure 9 shows the Skimair Cell design.

Figure 9. Skimair Cell [16]

3.1.5.3. RCS Systems

The Denver DR flotation mechanism was the most common Denver design used in open-type mineral flotation machines at the time. The profiled impeller vane is a key component of the RCS impeller. A flange connects the impeller to the shaft, and air flows straight to the rear air cavities via the air ports beneath the air shelf. The air shelf ensures that air flows directly from the hollow shaft to the air cavities, maximizing the aeration rate and air dispersion into the slurry. The RCS mechanism necessitates the use of an independent low-pressure air feed, which has the advantage of allowing the aeration ratio to be precisely regulated to meet the specific flotation task. To ensure that the slurry exits the mechanism in a proper radial direction, the revolutionary RCS impeller is encircled by a static diffuser. The diffuser vane spacing, and height must be precise to ensure that the impeller outlet flow and lower region return flow routes are in the vertical direction; this minimizes bulk sludge movement in the circular tank cell and eliminates the need for inner vertical baffles. The top of the RCS mechanism is open to offer a supplementary slurry flow path into the high-energy region within the impeller–diffuser zone, promoting extra ultrafine particle capture. This is a distinctive characteristic of the RCS patent that no other flotation machine supplier offers. The slurry is swept across the cell toward the mechanism by the upper vortex, while the lower vortex goes down the cell wall and across the cell base to enter the impeller axially. These flow regimes encourage particle–bubble collection by impact in the cell body and a high-energy process inside the process, as well as reducing short-circuiting and bottom sanding [16]. RCS design can be seen in Figure 10 and Figure 11 shows the several types of rotor-stator designs for mechanical flotation machines.

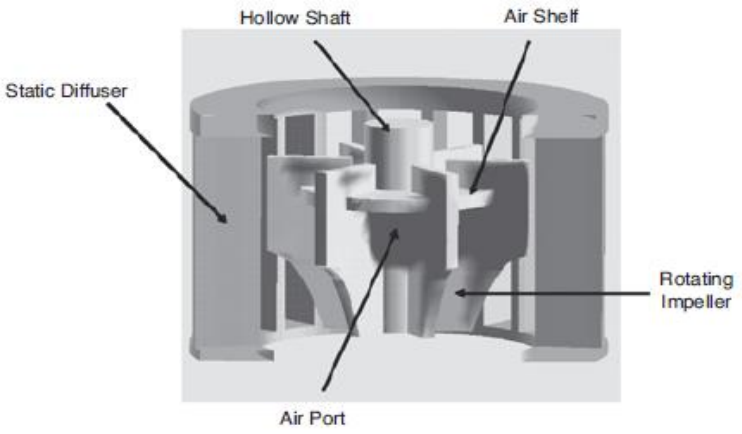


Figure 10. RCS mechanism [16]

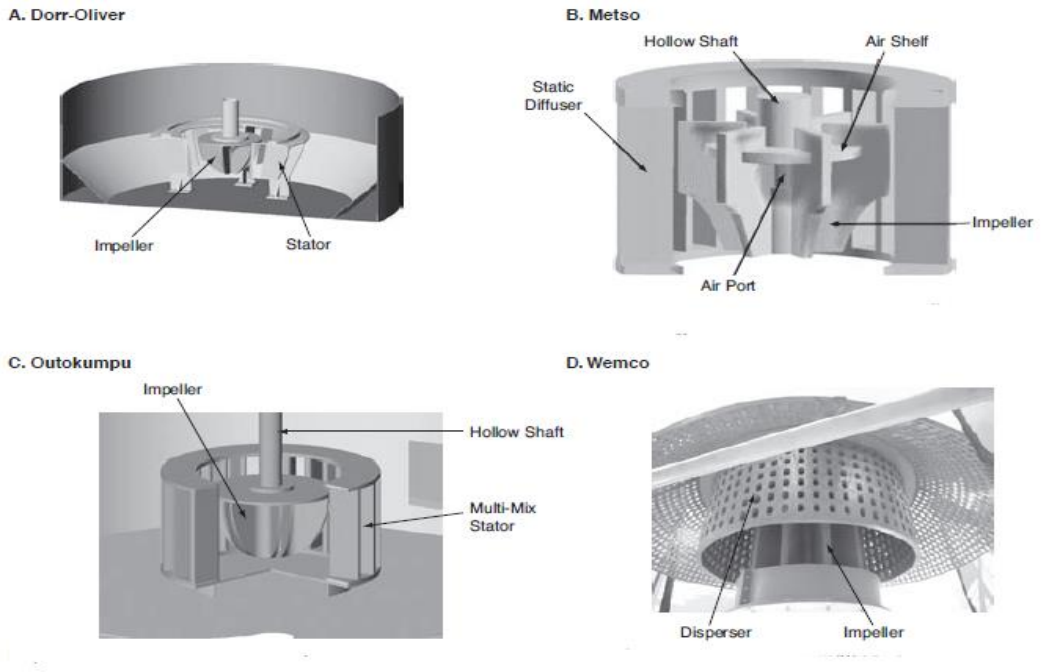


Figure 11. Diverse types of rotor-stator designs of the different machines [16]

3.2. Pneumatic Machines

Air entry to pneumatic machines is either by giving it into the cell with the slurry or by inducing air of the machine. However, in these machines, baffles, partitions, or a permeable layer are used to disperse the air. Pulp and air are fed into these cells using a nozzle to create close contact between the air and the particles. The air jet is utilized not only for aeration, but also to suspend particles and circulate the air [17].

3.2.1. Davcra Cells

Davcra machines are cell types in which slurry and air are fed to the tank using cyclone-type injectors. When the pressurized slurry enters the injector, a turbulent flow is formed and at this time, the air is supplied from the second channel of the injector. Thus, a shear effect is created, and a turbulent slurry flow and a central air flow are provided. After mixing the slurry and air at the feeding end of the injector, they are fed into the tank via the nozzle. The air in the slurry entering the tank is divided into small pieces due to the decrease in pressure. Inside the tank, there is a vertical baffle opposite the nozzle. The slurry-air mixture that hits this baffle loses its injection effect. Thus, the mixture of air and slurry fed horizontally turns into a vertically ascending stream, during which the minerals connect to the bubbles and rise.

The tailing flows into the discharge line with the routing channels [17]. The model of the Davcra Cell is shown in Figure 12.

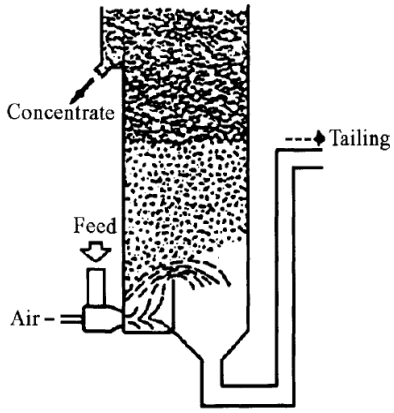


Figure 12. Davcra Flotation Cells [17]

3.2.2. Bahr Cells

Bahr cells consist of a conical lower part and a columnar tank in the upper part. A mixture of slurry and air is injected into the cell through a porous wall. As soon as the mixture enters the cell, it is included in the separation zone, and while the minerals which connected to bubbles rise, the tailing moves towards the exit with a spiral channel. Bahr cells, patented in Germany, have been used on coal and iron ores [18]. The sketch of the Bahr Cells is as in Figure 13.

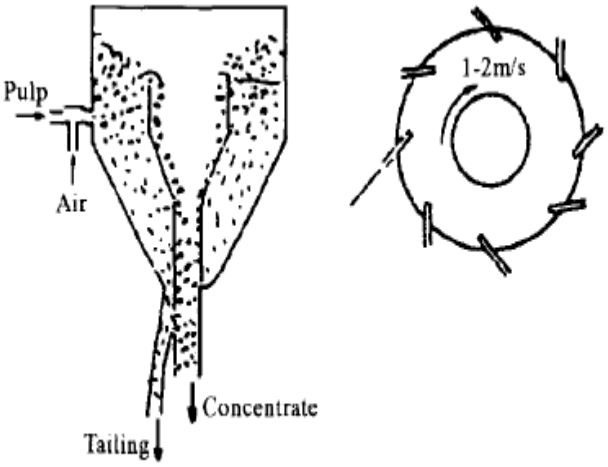


Figure 13. Bahr Cell [17]

3.2.3. Jameson Cells

Jameson cells feature a high-density column design that Professor Graeme Jameson has developed with Mount Isa Mines Limited. The slurry mixed with air through a downcomer is fed into the cell as a jet. Air bubbles inside the downcomer are already covered with minerals to be floated. After the slurry-air mixture inside is the cell, it is separated by rising in the disengagement zone. Tailings leaves the cell by flowing from the outlet. Like traditional columns, it provides 50% - 70% gain in a single stage, while 2 or 3 in series of cells are required. Thanks to its small and compact structure, it is easy to install and operate [18]. Figure 14 shows the Jameson Cell design.

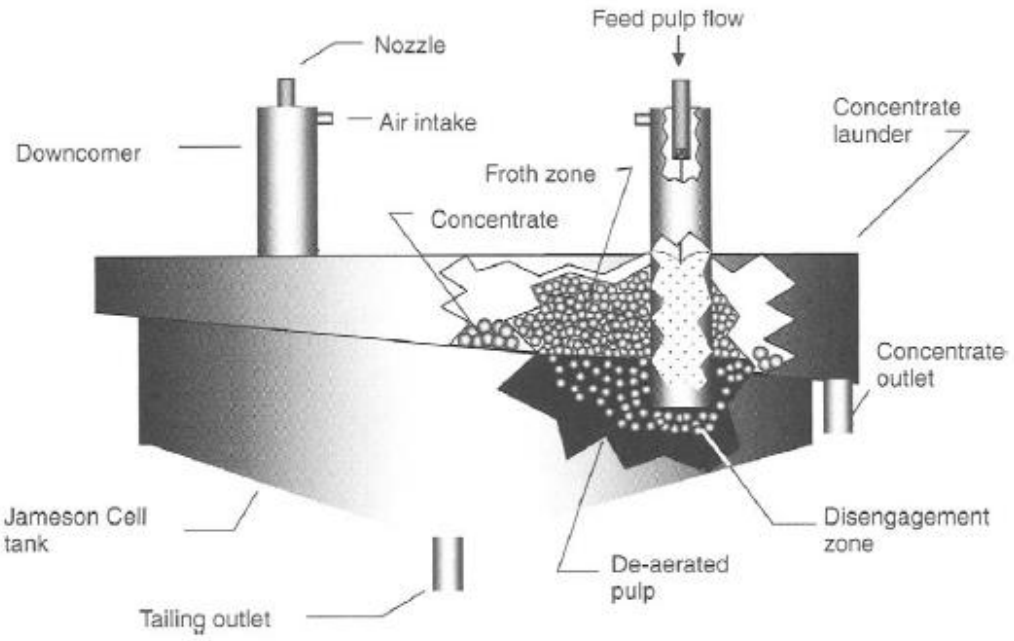


Figure 14. Jameson Cell [12]

3.2.4. Imhofloat Cells

In the IMHOFLOT V-Cell design, which began to be developed in the 1960s, slurry and air are mixed in a self-ventilation unit located above the cell. This mixture of slurry and air comes down a pipe into a diffuser box. The mixture is fed from this box to the separation tank by pipes with a nozzle system positioned upwards. Inside the separation tank, there is a loose clamping cone to adjust the mass pull. While the concentrate is taken to the this cone, the tailings leaves from the bottom of the tank [19] [20]. Imhofloat V-cell design can be seen in Figure 15.

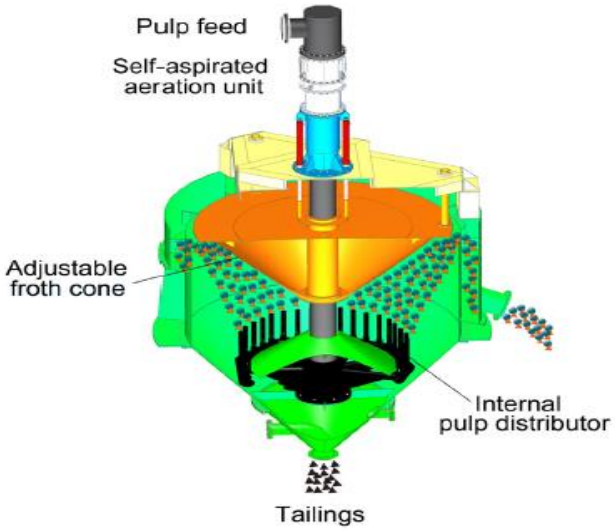


Figure 15. Imhofloat V-cell design [20]

The IMHOFLOT G-Cell design, which was introduced in 2001 and is an innovative technology, uses a self-ventilating system that is similar to the V-Cell design. The slurry coming out of the ventilation unit is fed into the distribution box tangentially. The cell has an inner launder system where the froth is collected [21]. Imhofloat V-cell design can be seen in Figure 16.

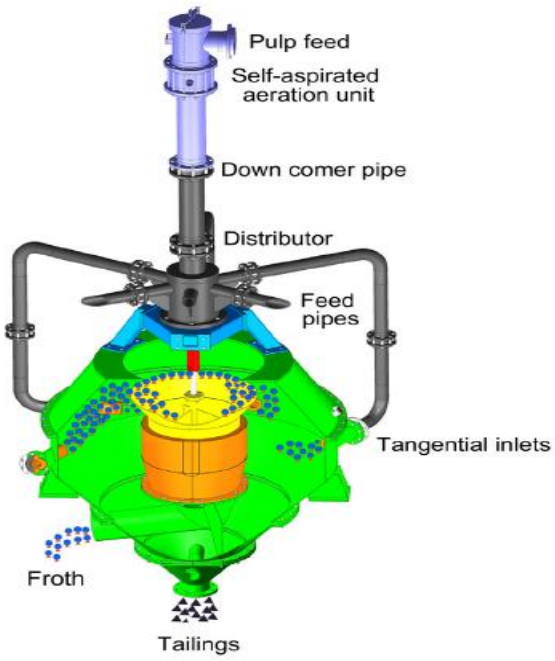


Figure 16. Imhofloat G-cell design [20]

4. COLUMN FLOTATION

The first flotation columns, in which washing water was not used and the bias effect was not fully understood, and this was not considered an important control parameter, started to be tested in the 1920s [22] [23]. However, modern flotation columns were patented by Boutin and Tremblay in 1960 in Canada. (Patent no: 680.576 and 694.547). The starting point of Pierre Boutin's thought is turbulence in the flotation cells, which is one of the biggest reasons why fine gangue particles come to concentrate. According to his opinion, if a high flotation cell without turbulence is fed from the upper side, gangue minerals escaping into the concentrate can be prevented. With the air to be given from the bottom of the column, the minerals and air bubbles will be met in opposite directions and while the gangue minerals will move to the bottom of the column due to their high precipitation rates, the concentrate will rise due to the bubbles [2]. In addition, the gangue minerals that are entrained into the concentrate with the use of washing water will turn back into a slurry and a high-grade concentrate can be obtained [24].

The advantages of flotation columns over conventional cells are:
Low investment and operating cost

- High froth zone height
- Providing ease of use because of the simple system
- Less footprint

High success especially in fine particles, because of the no turbulence [25] [26] [27].

Flotation columns have advantages as well as disadvantages. Coarse particles have a high settling rate and therefore their residence time in the cell is low. Fines, on the other hand, are affected by the flow lines around the bubbles and therefore less likely to collide with the bubbles [28]. In addition to these situations, some disadvantages of columns are as follows:

- Column height
- Maintenance of the bubble generating system
- Decreasing of the tailing solid percent
- Wash water cost

Not suitable for ore types that can oxidize quickly due to their long residence time [2] [25].
A typical flotation column design looks like Figure 17.

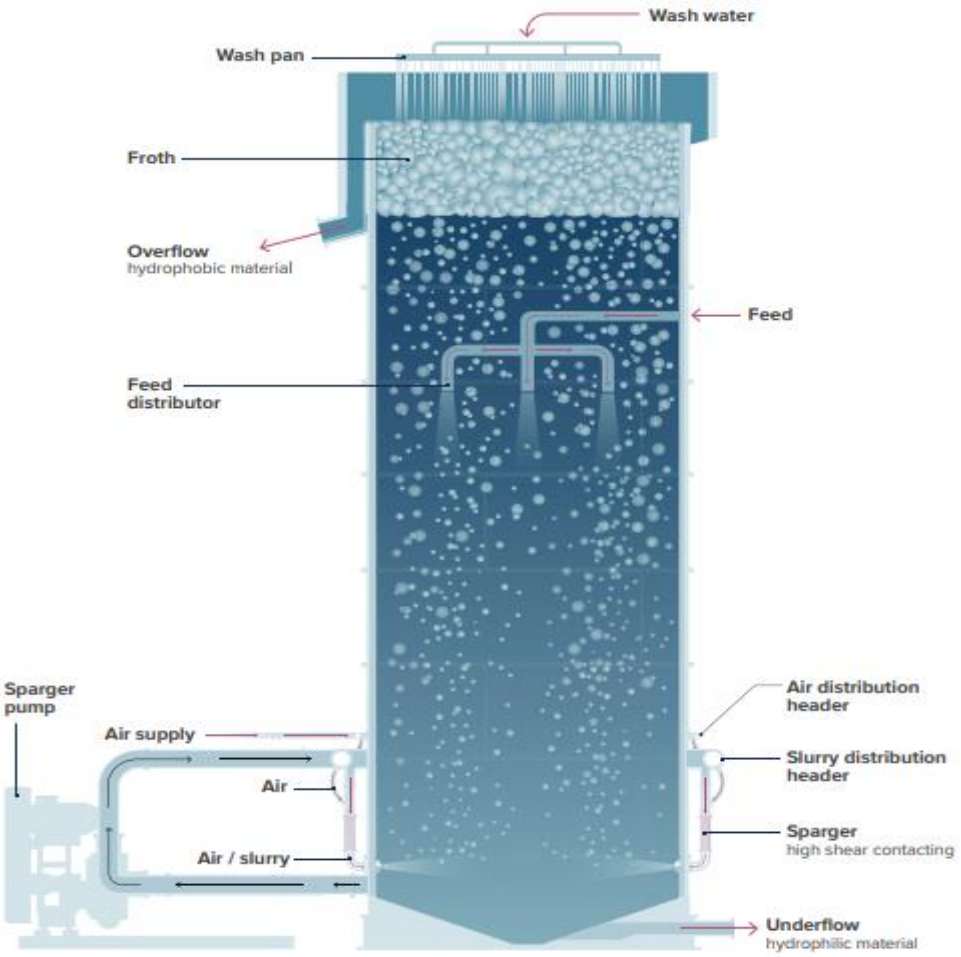


Figure 17. Flotation Column [29]

At its base, there are 2 zones in the flotation columns. One of them is the collection zone (recovery or pulp zone), which is below the feeding point and where the particles encounter the bubbles, and the other is the froth (washing or cleaning zone) starting from the pulp-froth interface to the top of the column and containing loaded bubbles [30]. The water between the bubbles moves towards the collecting zone and as a result, thinning occurs in the water layer surrounding the bubbles. As the interface between the liquid and air is approached, the bubbles combined their surface area decreases. If froths with reduced surface areas do not have sufficient stability to carry the particles, the particles separate from the froth and return to the collection zone Therefore, the recovery rate in the froth zone has a key place for column efficiency. Flotation column zones can be seen in Figure 18.

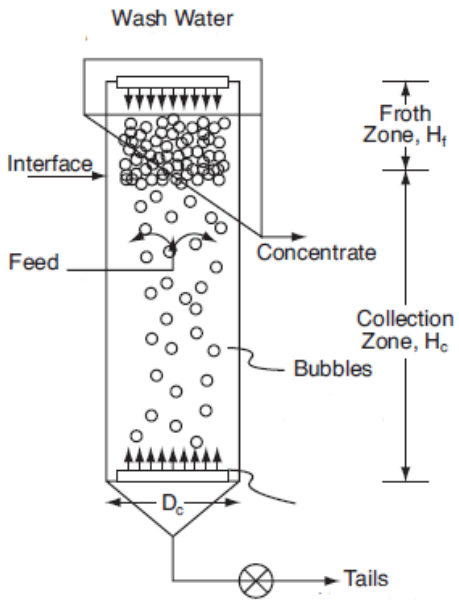


Figure 18. Flotation column zones [4]

4.1. Flotation Columns Air Generating Systems

Flotation columns do not contain parts such as rotor-stator that affect bubble production. Therefore, different systems are needed to produce bubbles. In the first designs of flotation columns, porous pieces such as perforated rubber or filter linen were added to the end of air pipes entering the column to produce bubbles in the column. However, the sparger systems, which were constantly dirty and clogged due to the particles, required constant maintenance and were costly. With the development of technology, patented designs such as SlamJet, SparJets, have been developed to produce bubbles [31]. Figure 19 shows the Slamjet design.

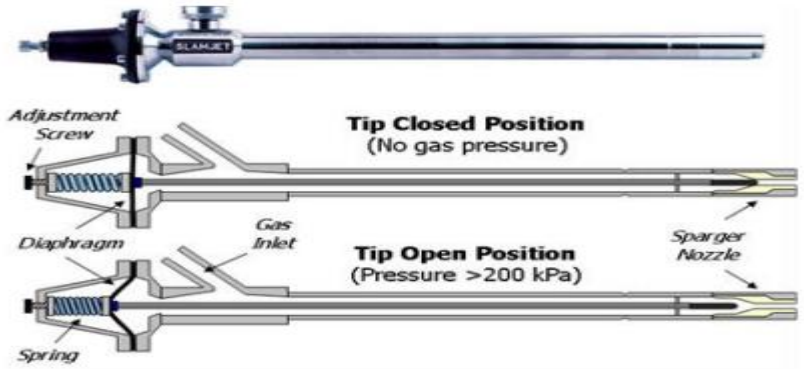


Figure 19. Eriez Slamjet Design [32]

In addition, another system used to produce air in flotation columns is cavitation tubes. As it is known, the formation of dynamic cavitation is the formation of bubbles in the liquid in low pressure regions that occur where the liquid is increased to high velocities. If the pressure of a moving liquid momentarily drops below the steam pressure, ultrafine bubbles are formed. These bubbles move towards areas of high pressure and create a vapor-like appearance. By using a similar system in flotation columns, bubbles are fed to the system. The air supplied with an external line to the slurry inserted into the cavitation tube with the help of a slurry pump creates an ultra-fine bubble, such as patented Microcell Spargers [33]. Figure 20 shows one of the samples CavTube design.

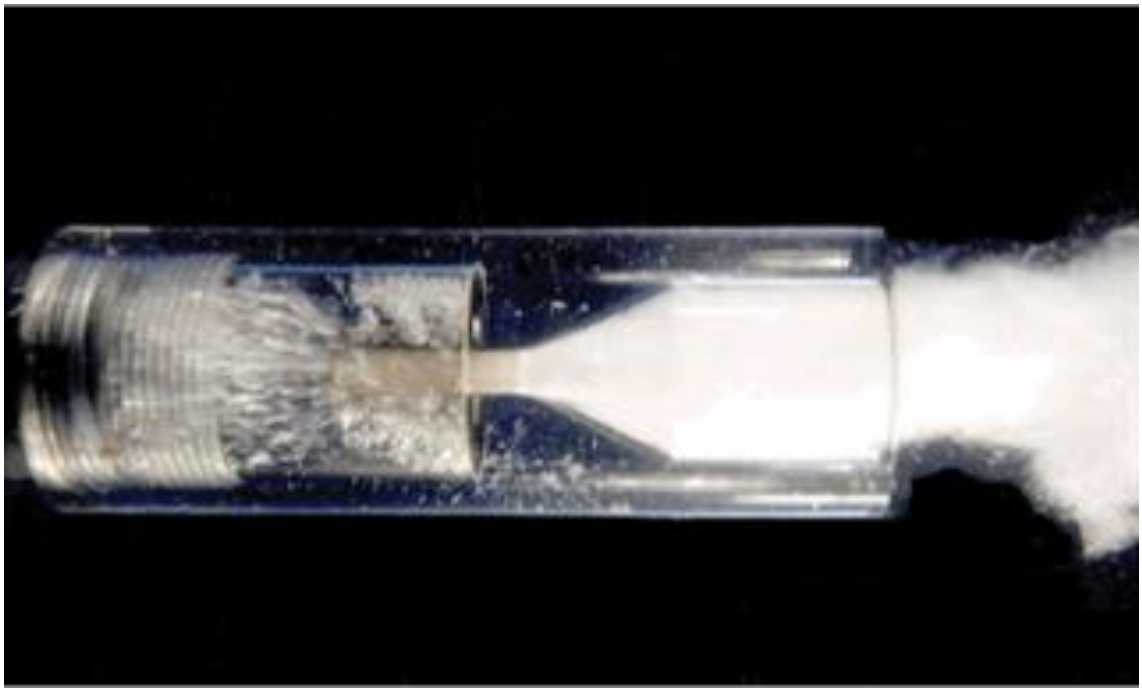


Figure 20. Eriez CavTube design [34]

4.2. Flotation Column Parameters

4.2.1. Carrying Capacity (C)

The carrying capacity, which is the upper limit of the particle collection process, is defined as the weight of the mineral floating in the cross-sectional area per unit of time. In other words, the payload refers to the amount of the maximum solid to be gained in a column [35]. The carrying capacity is obtained experimentally while the columns are operating, which changes the feed solid ratio until the maximum alignment is reached [31].

$C = 4\ Q_g\ D_p\ \rho\ \beta\ /\ D_f$

Where :

C	:	Carrying capacity
D_f	:	Buble diameter in the froth
β	:	Particle packing factor on the bubble surface
D_p	:	Diameter of particle in the froth
ρ	:	Specific gravity of the particle
Q_g	:	Flowrate of the gas

Figure 21 shows the carrying capacities of different mineral and coal flotation columns against the d80 and the specific gravity of the floating particles. Cmax is represented as the top limit (solid line) of all data points, which is similar to the values given by Espinosa-Gomez et al [36].

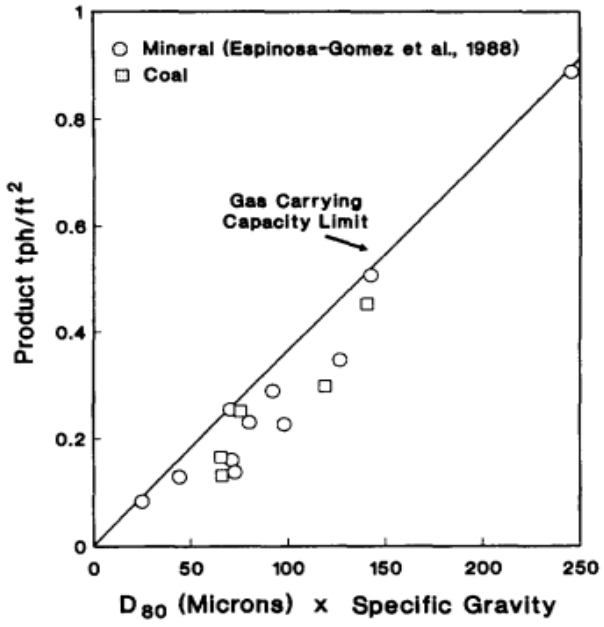


Figure 21. Carrying capacity and particle size-specific weight relation [36]

If the column works in speed-limited conditions, the efficiency of the column is significantly below the C_{max} value. In this case, the value of k is not at the optimum value. If this is detected, the bubble size, collector addition rate, or aeration rates should be examined.

According to Yoon, the most effective way to improve the k value is to change the bubble size [36]. The maximum carrying capacity according to the Sastri is calculated with the following formula [37].

$C_{max} = k \rho D_p$

Where :

k : Constant

D_p : Diameter of a particle in the froth

ρ : Specific gravity of the particle

Some of the carrying capacity experimental data is shown in Table 1.

Table 1. Carrying capacity experimental data [38]

Column diameter, m	Product size, d_{80} microns	Product density, ρ, g cm^{-3}	Carrying capacity, C_m g min^{-1} cm^{-2}	$C_m/(d_{80}\rho)$
0.025	35	2.6	5.13	0.056
0.051	6	4	1.4	0.058
	11	4	2.3	0.052
	14	5.8	4.1	0.050
	15	4.5	2.7	0.040
	16	4.2	4.2	0.067
	35	4	9.1	0.065
0.203	23	4.2	3.8	0.039
0.914	30	4.2	6.2	0.049
1.1	44	5.6	16.1	0.065

4.2.2. Gas Hold Up ($\mathcal{E}_g$)

Air hold, also known as air hold up, is defined as the volume occupied by air in flotation columns. While air hold up is between 5-30% in the collecting zone, it can go over 80% in the froth zone [39]. The gas hold up associated with bubble size is a function of the type of sponge, frother properties, solid ratio, airflow rate and slurry flow rate [40]. When there is no solid in the system, air hold up can be measured easily as the following equation according to Finch and Dobby, 1990.

$$\mathcal{E}_g = 1 - \Delta P / (\rho_{sl} \times g \times \Delta L)$$

Where :

ΔP : Difference of pressure between two points

ΔL : Vertical distance of two points

ρ sl : Density of the slurry

4.2.2.1. The JKMRC Gas Hold Up Probe

The JKMRC gas hold up probes are used widely to measure gas hold up, which was developed by JKMRC (Julius Kruttschnitt Mineral Research Center) within the scope of the AMIRA P9 project. This probe is a vertical cylinder placed under the froth-slurry interface and has either pneumatic or mechanical valves. Early versions had a valve design using compressed air, while modern designs have rotary mechanical or spring-loaded valves. The probe is lowered into the aerated slurry and the slurry is allowed to pass through it for at least 10 seconds. At the end of the period, the valves are closed and the Vs value, which is the volume of the slurry sample in the probe, is compared with the probe volume, Vp, between the two valves, and the value is calculated from the equation below. Although it is simple and fast, data on the whole cell cannot be obtained in this method. Therefore, at least 3 samples should be taken from different parts of the cell [41].

$$\varepsilon_g = \frac{V_p - V_s}{V_p}$$

Where:

V_s : Slurry volume

V_p : Probe Volume

The schematic of the probe is as follows, Figure 22.

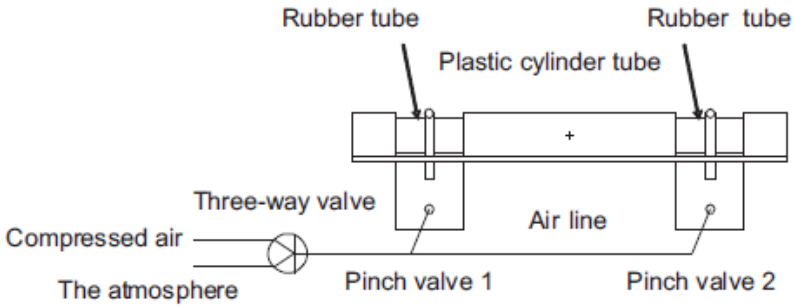


Figure 22. JKMRC ε_g probe schematic [42]

4.2.2.2. The McGill Gas Hold Up Probe

The McGill gas hold up the probe, which is a different system used for measurement, measures by using the conductivity difference between gas and slurry. As can be seen in the schematic in the figure, it consists of two tubes, and the open tube measures the conductivity of the aerated slurry while the siphon cell measures the conductivity of the airless slurry. The results found are evaluated using the following equation [43].

$$\varepsilon_g = \frac{1 - \frac{k_o}{k_s}}{1 + 0.5\frac{k_o}{k_s}}$$

Where:

k_o : Slurry conductivity with bubbles

k_s : Slurry conductivity without bubbles

The schematic of the probe is Figure 23.

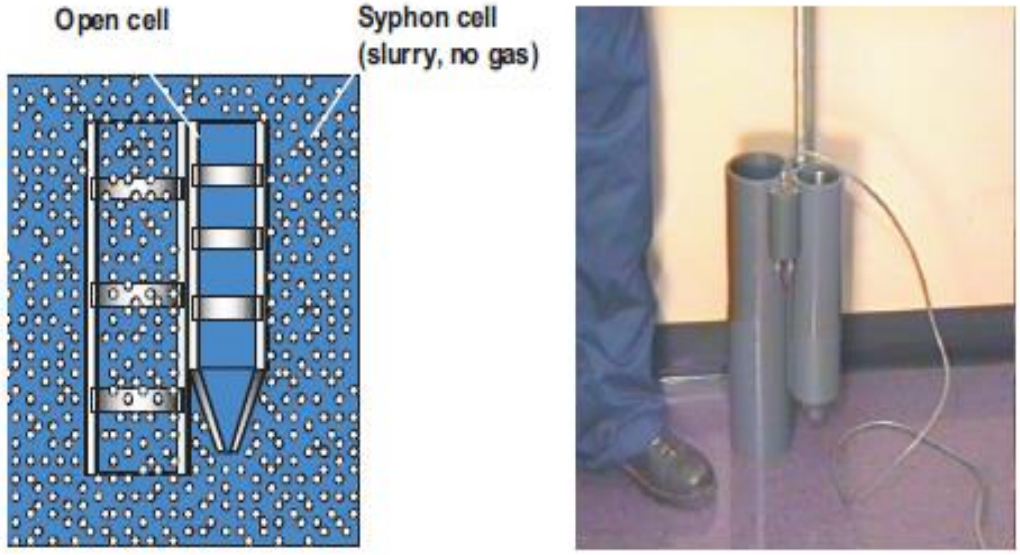


Figure 23. McGill Ɛg probe schematic [44]

Radioactive systems [45], acoustic emissions [46], ultrasonic techniques [47] [48], and pressure transducers are also different techniques used for air measurement [42].

According to Power, Franzidis and Manlapig, the gas hold value varies between 6% and 25% in both mechanical and column cells. However, if the air flowrate is insufficient, this value may drop below 2%. On the other hand, if the slurry viscosity is high, this value may exceed 50% [49]. Value ranges can be seen in Figure 24 below.

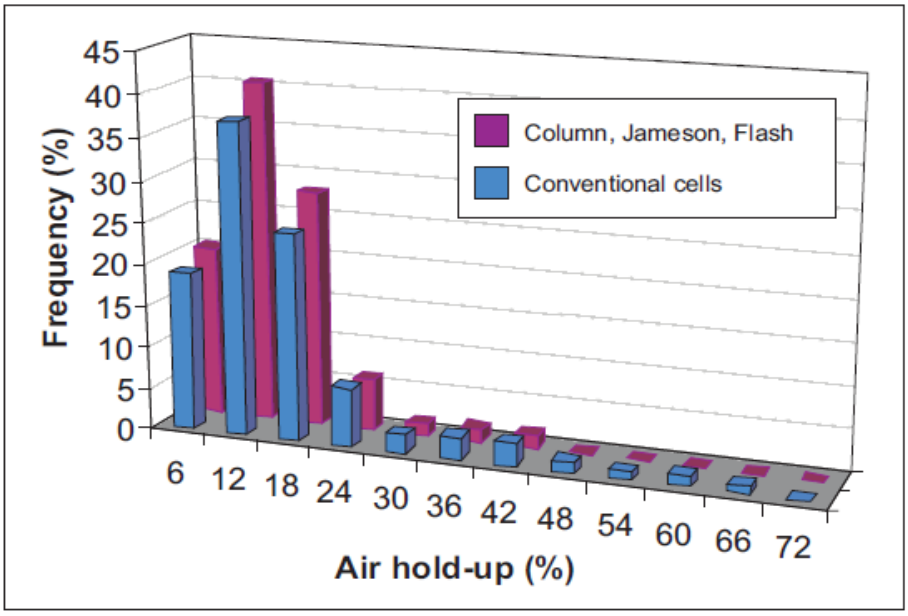


Figure 24. Air hold up a range of different flotation cells [42]

4.2.3. Bubble Surface Area Flux (S_b)

Bubble surface area flux, S_b, which includes the combination of bubble size and surface gas velocity parameters, is a measure of the bubble surface area rising per unit time in a unit cross section in the column [42]. Bubble surface area flux is an important parameter for generating flux models and characterizing kinetics. Bubble surface area flux is more difficult to calculate, although it is a more fundamental parameter compared to air hold up by Jameson et all. Bubble surface area flux is calculated by the following equation [50].

$$Sb = 6 \times J_g / d_b$$

Where :

S_b : Bubble surface area flux

J_g : Superficial gas velocity

d_b : Bubble diameter

Typically, the range of the bubble surface area flux is between 30 s^{-1} – 70 s^{-1} and as shown in Figure 25.

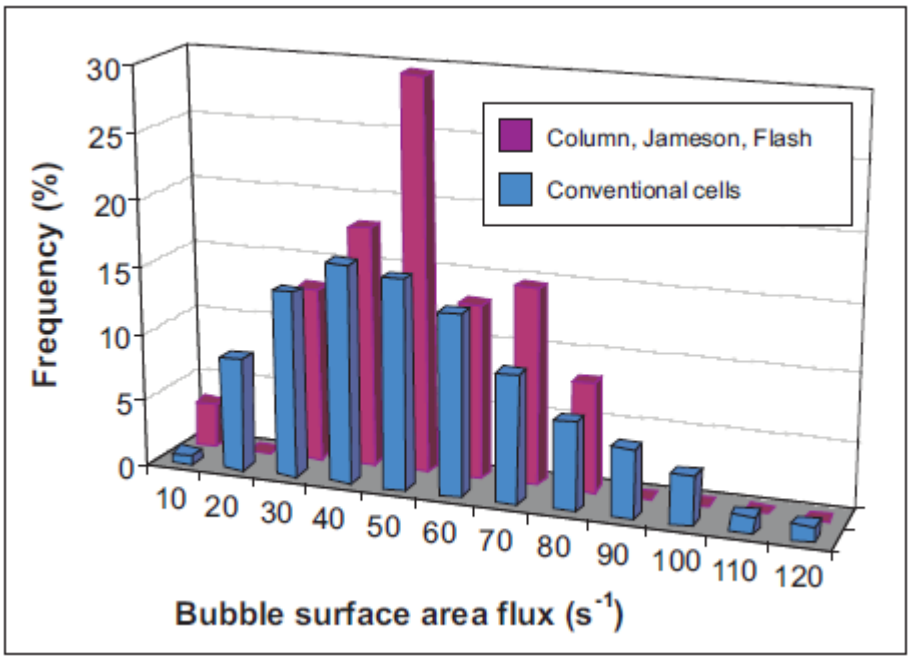


Figure 25. Bubble surfaces are flux values range of different flotation cells [42]

To examine the relationship between air hold up and bubble surface area flux, a series of tests were conducted at Bowater's Gatineau Mill in Quebec, Canada. Columns with dimensions of 10 x 470 cm as laboratory scale and 50 x 150 cm as pilot scale were used in the tests. The test procedure was to remove the ink from the paper. Different systems such as porous steel, cloth filter, and spray sparger were used in the columns, and experiments were carried out on gas rate, retention time, and froth depth. The gas rate was precisely measured and controlled with a flow scale. The air hold up was measured by both conductivity (Cortes-Lopez) and pressure (Hardie and Leichtle) methods, and bubble size was calculated by drift flux analysis (Banisi and Finch, 1994). The results were taken from 128 and 3 different sources in total and were obtained with a slope difference of 3% and a confidence interval of 95 %. The graph prepared according to the test results is as follows *[51]*. Figure 26 shows the gas hold up and bubble surface flux comparison in the de-inking process.

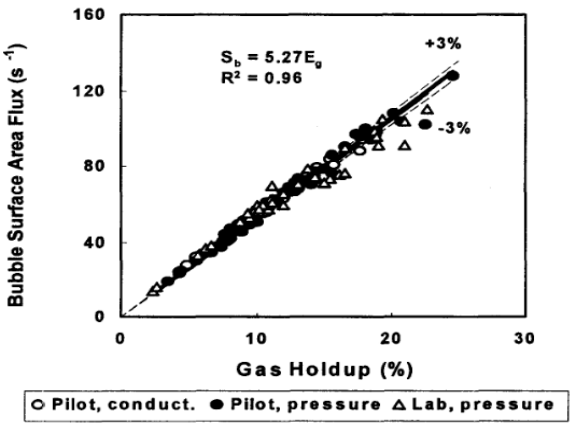


Figure 26. Gas hold up and bubble surface are flux comparisons in the de-inking process [51]

4.2.4. Superficial Gas Velocity (J_g)

Superficial gas velocity is a measure of a cell's aeration capability and is specifically linked to flotation kinetics [52]. A high superficial gas velocity causes excess gangue mineral in the froth by entrainment and decreases the stability of the froth. It is measured in cross-sectional areas of different heights in the cell to show how homogeneously the air is distributed inside the cell [53]. Superficial gas velocity is calculated by 2 methods. The first method, is calculated with the following formula by using the air flow rate value fed into the cell. This method may produce erroneous results if the flow meter calibrations are not done correctly.

$J_g = Q / A$

Where :

J_g : Superficial gas velocity
Q : Volumetric air flow rate
A : Flotation cell cross-sectional area

The second method used in superficial gas velocity calculations is The JKMRC mechanical Jg probe, which was developed by JKMRC and McGill Online J_g probe which are part of the AMIRA P9 project.

JKMRC Jg probe is a persprex tube consisting of a valve providing water inlet and air discharge at one end and a pneumatic valve at the other end. JKMRC Jg probe is a persprex tube consisting of a valve providing water inlet and air discharge at one end and a pneumatic valve at the other end. The probe, which is lowered under the froth-slurry interface, is filled with water. The air release valve is in the closed position and the pinch valve at the lower end is opened and the probe is filled with the gas coming from the cell. Gas fills the probe with the gas flowing into the cell at an equal rate, and the elapsed time is recorded until the water level reaches a predetermined distance. Necessary adjustments are made according to the depth in which the probe is immersed and the probe length [49]. The schematic and real image of the probe is as follows, Figure 27.

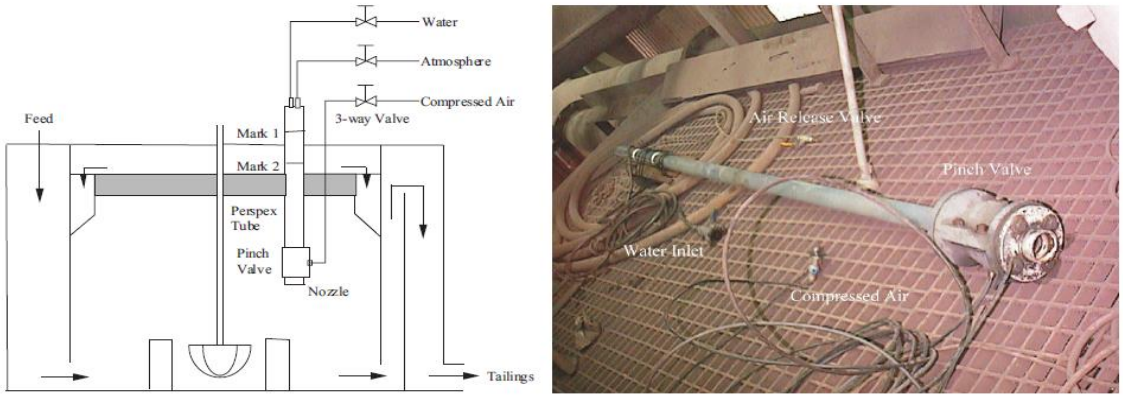


Figure 27. JKMRC J_g probe schematic and set up [44]

The McGill online J_g probe is a system consisting of two empty plastic tubes placed under the froth-slurry interface, similar to the JKMRC probe. The probe works according to the principle of pressure change, which is created by pushing down the gas coming from the cell and the slurry in the first probe. The second tube is used to control the slurry level and a time-pressure relationship is established through the first tube. The following equation is used for this system.

$$J_g = \frac{\left(\frac{1033(2P - P_O)}{\rho_b}\right)}{P_L} \frac{dP}{dt}$$

Where :

P_o : Atmospheric pressure

P_L : Pressure changing depending on the level at the bottom of the pipe

ρ_b : Aerated pulp density

The schematic view of the probe is as Figure 28.

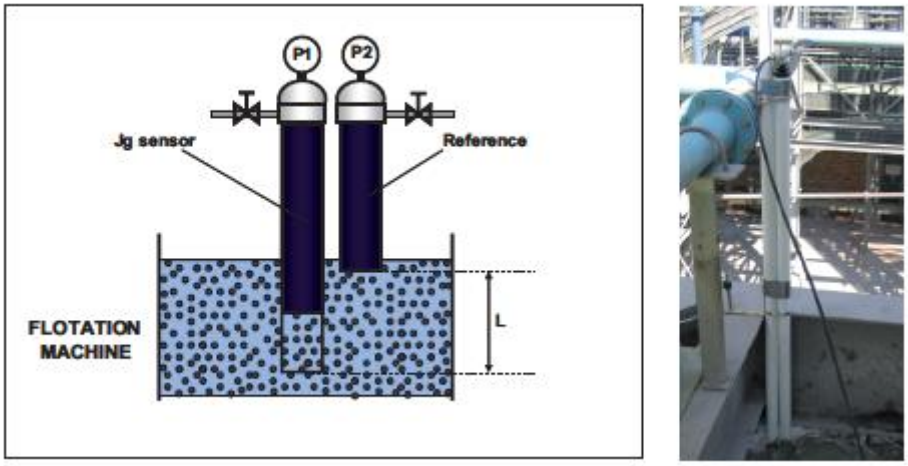


Figure 28. McGill Jg probe schematic and set up [44]

Depending on the plant's operational strategy, the surface gas velocity can vary greatly. In certain cases, values as low as 0.1 cm/s were recorded, whereas values as high as 3 cm/s were obtained for mechanical floatation cells. Generally, readings ranging from 1.0 cm/s to 2.0 cm/s are considered to be within an acceptable limit. When Jg is less than 1.0 cm/s, it is generally recommended to examine raising the air flow rate to optimize pulp phase recovery. Flooding can occur when Jg exceeds 3.0 cm/s, leading to pulp being recovered in the concentrate launder. This demonstrates a physical limit to raising air flow rates, while the influence on froth phase performance should also be examined. Jg data from over 1000 flotation cells are displayed as a guide [42]. An example of superficial gas velocity values ranges of different flotation cells is in Figure 29.

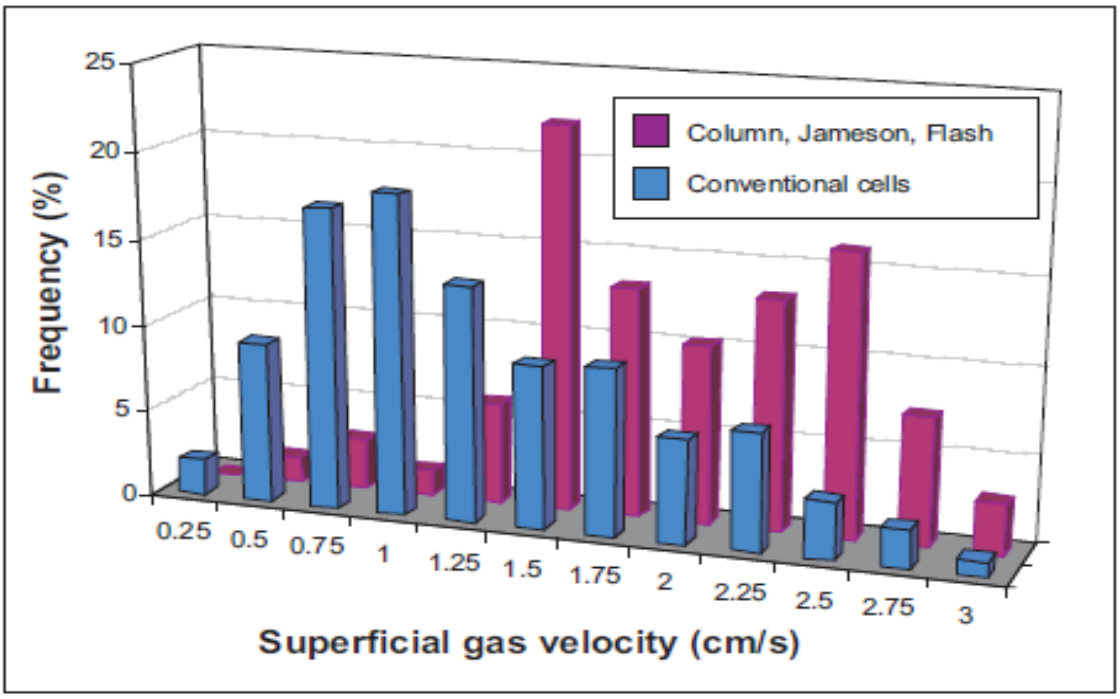


Figure 29. Superficial gas velocity values range different flotation cells [42]

4.2.5. Feed Rate

There are different results in the literature regarding the effect of feed rate. The general opinion is that with the decrease in the feeding rate, the residence time increases, and accordingly the efficiency increases [54] [55] [56]. On the other hand, according to Goodall and Conner, on the contrary, they think that the yield increases due to the increase in the feeding speed.

4.2.6. Wash Water (J_w)

The purpose of the washing water used as one of the biggest advantages of column flotations is to ensure that the gangue minerals that come to the concentrate with entrainment are washed from the froth and returned to the pulp. In general, the washing water given up to an optimum point increases the concentrate grade and causes these values to decrease after reaching the optimum values [57]. As is known, the purpose of wash water, which is directly related to froth stability in flotation columns, is to increase drainage and accordingly reduce entrainment. However, when high ratios of washing water are used, it leads to an inefficient cleaning with the increase of water short circuits to the concentrate and froth mixture, and

the metallurgical efficiency may be limited. The original wash water design was a piping system developed by the Canadian Flotation Column Corporation and located 10-20 cm below the lip level. The tubes were a system that fed water into the slurry in a double row, with holes of 2 to 3 mm in diameter, and at a right angle to the slurry. Although the advantage of this system was that it provided a downward and clear flow and thus the concentrate achieved a higher solids ratio, its disadvantage was that the holes in contact with the froth were clogged and the water regime deteriorated [58]. Another design is a system in which water is fed from a tank with holes of 3 mm in diameter, similar to sprinkling, and approximately 30 cm above the froth. Thus, the washing water can be observed clearly [59]. However, since weak froths can be affected by this water flow and quickly collapse, it may not be suitable for use in applications with an unstable froth structure.

4.2.7. Froth Height

The concentrate grade is related to froth height and it increases with the froth height. Gang minerals coming to the concentrate with entrainment are cleaned and returned to the slurry, depending on the decrease in water yield in the froth and the effect of washing water [60] [61].

4.2.8. Bias

Bias velocity is one of the most important characteristics of flotation columns and is defined as the net water flow downstream of the froth, or the net water flows difference between the equivalent tailing and feed streams [54]. If the amount of water entering the column is more than the water recovery of the concentrate, this situation is called positive bias, while in the opposite case, if the water moves towards the concentrate, it is called negative bias [58]. More precisely, if the amount of water in the tailings is more than the amount of water in the feed, positive bias occurs, while in the opposite case, negative bias occurs. In the case of positive bias, the reason that creates the difference is the wash water. While some of the washing water fed to the system from the froth area meets the positive bias, some of it is taken from the concentrate. Thus, gangue minerals coming to the concentrate with entrainment are prevented [62] [63] [64].

According to Trahar, one of the most important reasons for decreasing the concentration grade in mechanical flotation cells is entrainment [68]. The reason for this is that mechanical

flotation cells generally work with negative bias. On the other hand, flotation columns generally work with a positive bias, that is, the froth is washed, and entrainment is prevented with the excessive froth height [2] [62] [63] [65] [66].

As is known, flotation columns work with positive bias. However, according to Oteyaka and Soho, negative bias can also be used in some cases. In a laboratory scaled column flotation test of quartz and quartz-calcite mixture, they obtained high recoveries with particle sizes varying between 126-714 microns. In this system with a negative bias, where there is coarse particle size and no froth layer, remarkably high phosphate, potash, and quartz recovery were achieved due to the fast flotation kinetics. They experienced about the suitability of a column with a length of 1 meter or less in this system, where the minerals do not stay in this system for more than a few seconds. According to the experimental results, they suggest that it is an idea to be developed in flotations to be applied in flash, coarse, or scavenger circuits to be made in coarse particle size [67].

Using a flowmeter as a way of measuring bias is a difficult method as it assumes steady-state operation. In addition, using multiple measuring devices increases error propagation and causes high standard deviations [54]. For this reason, Uribe-Salas et al propose a method that uses steady-state conductivity values as a more practical method. In this method, conductivity values of feed, concentrate, tailings, and washing water are used and the formula is as follows [68].

$$J_b = J'_t\left(\frac{k'_f - k'_t}{k'_f - k_w}\right) - J'_c\left(\frac{k'_c - k_w}{k'_f - k_w}\right)$$

Where :

J_b : Bias

J_c : Superficial concentrate rate

J_t : Superficial tailings rate

k_f : Feed conductivity

k_c : Concentrate conductivity

k_t : Tailings conductivity

k_w : Wash water conductivity

5. MATERIALS AND CIRCUIT

Küre deposits are similar to massive sulfide deposits with ophiolite side rocks, which are defined as “Cyprus-type Massive Sulfide Deposits” due to their environment. There are massive ore bodies both inside these basalts and at their contact with black shales. Due to overturning, small, massive ore bodies located in Bakibaba and the section above 780 m of the Mağaradoruk deposit are observed on the black stuff. In the detailed mineralogical studies carried out in the Küre Aşıköy and Bakibaba deposits, these deposits mainly consist of pyrite, chalcopyrite, cobalt, and to lesser extent marcasite, sphalerite, covelline, neodigenite, malachite, azurite, fahlerz, little bravoite, lineite, limonite, hematite, and traces of chromite, rutile. anatase, chalcosine, cuprite, tenorite, magnetite, pyrrhotite, valleriite, bornite, galena, native copper, and native gold are observed. The main gangue minerals are quartz, siderite-ankerite, calcite, dolomite, and chlorite. The mineralogy of the Caverdoruk deposit is similar to the mineralogy of the Asikoy and Bakibaba deposits. However, unlike them, cobalt minerals, magnetite, hematite, native copper, and native gold are more abundant in the Mağaradoruk deposit. In the "Cyprus type" deposits, the main ore minerals are pyrite, chalcopyrite, sphalerite, pyrrhotite, and magnetite [69] [70].

The flotation circuit feed contains 2-2.5% copper, 0.32-0.40% cobalt and 25-35% total sulphur. Cyclone overflow P80 value is around 46 microns. In the plant, lime is used as pH regulator, DOW-250 as frother and Hostaflot-X231 and KAX as collector. Lime addition is made in the conditioner tanks before the rougher flotation and the pH value is adjusted to the value of 11. X-231 and Dow-250 additions are made in the next tanks. X-231 and KAX are added to BU 2 and BU 3 cells. Finally, Hostaflot X-231 is added to the conditioner tank before the first cleaning circuit and no other reagents are added. In the studies performed with column flotation, no chemical addition was made.

The first copper cleaning circuit of the plant contains 7% to 11% copper, while it also contains cobalt between 0.33% and 0.68%. While the second copper cleaning circuit contains 11-16% copper, it contains 0.27-0.5% cobalt.

The copper cleaning circuit of the plant is shown in Figure 30.

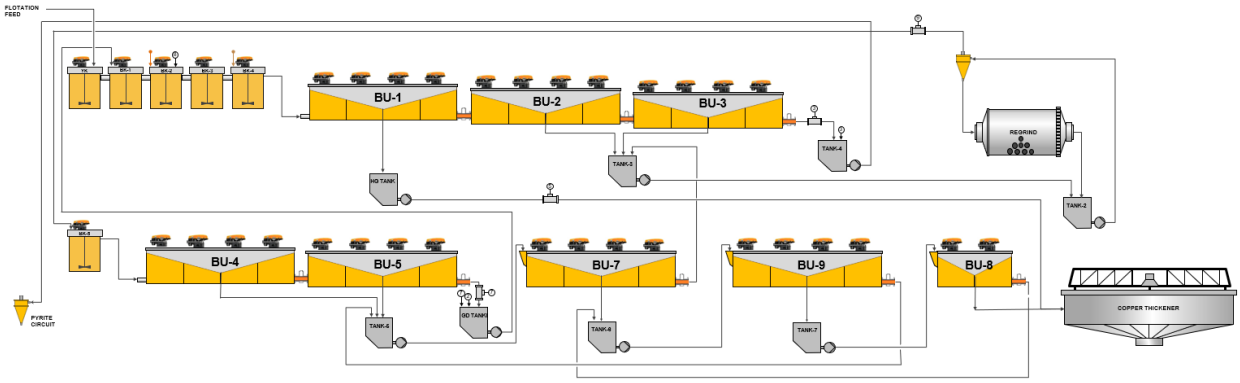


Figure 30. Copper cleaning circuit flow diagram

Kure Plant copper flotation circuit cells had been designed as 14.6 m^3 capacity and is 4 series cells. Only the BU 8 cell, from which the plant's final concentrate is taken, has 2 series and 7.3 m3. In the plant, the BU 1 cell is a coarse flotation circuit and is a system that works like flush flotation. The concentrate of BU cells is taken directly as the final concentrate and sent to the copper thickener. BU 2 and BU 3 cells work like scavengers and the concentrate is fed to BU 4, which is the primary cleaning circuit. BU 2 and then BU 3 waste are fed to the regrind mill. The regrind unit has a conventional closed-circuit design. The overflow of the cyclone is fed into the cleaning circuit, while the underflow of the cyclone returns to the mill. Currently, the BU 5 cell is not working and only BU 4 cells constitute the first cleaning cycle. While the tailings of the first cleaning circuit return to the rougher circuit conditioners, its concentrate combines with the tailings of the third cleaning circuit to form the feed of the second cleaning circuit. While the tailings of the second cleaning cycle return to the regrind mill, its concentrate combines with the tailing of the fourth cleaning circuit to form the feed of the third cleaning circuit. The concentrate of the third cleaning circuit directly constitutes the feed of the fourth cleaning circuit without combining with any lines. The concentrate of the fourth cleaning cycle is taken as the final concentrate. Flotation is done with conventional mechanical cells in the plant and FLoatForce designed rotor-stator systems are used on the circuit.

6. METHOD AND EXPERIMENTAL STUDIES

6.1. Pilot Scale Column Specifications

The column used in the experiments is a pilot scale column of Eriez company with a volume of 0.84 m3. The column height is 4.1 meters, and its diameter is 0.51 meters. There are 2 pressure sensors on the column, one at the bottom of the slurry feed line and one slightly above the tailing line. The froth height is calculated by reading the pressure difference between these two points with an empirical formula.

Two different systems produce bubbles in the system. The first of these is the Sparger, which is located at the bottom of the column and directly fed with system air, and the other is the cavitation tube located at the connection point of the column on the discharge line of the circulation pump. While the Sparger generally produces larger bubbles, the bubbles created by the cavitation tube are smaller. The air entering the column is divided into 2 lines. While the first line goes directly to the tailing valve, the second line goes to the flowmeter, which will measure the cavitation and sparger air amounts. The tailing valve works with a solenoid valve, according to the information from the sensor on the column, open command comes to the solenoid valve and the normally closed tailing valve opens. There are no separate flowmeters for Sparger and cavitation. Instead, a single flowmeter is located on it, which measures the air flow from the system. However, the air pipelines to the Sparger and the cavitation tube have separate manual valves. To adjust the air going to these two lines, first of all, a valve is closed or brought to a certain value, then the air flowrate to be supplied to the other line is adjusted.

The circulation pump is a centrifugal pump with 5.5 kW power. This is the only part that moves in the column system and needs energy. The pump has a VFD system, and the motor speed can be changed by changing the operating frequency with the help of a switch on the column panel. The circulation pump used in the column has 2 purposes. Firstly, this pump prevents slurry from settling into the column by making suction from the bottom part of the column. Secondly, the output line of the pump is slightly higher than the suction line, there is also a cavitation tube on it which is one of the bubbles generating systems.

Washing water is supplied to the system as sprinkling with a pipe system on the column. There is a flowmeter to measure the flow amount with a manual valve. In the system where

process water is used as washing water, launder water is also added to blast the froth when necessary and its values are recorded. The pilot scale flotation column can be seen in Figure 31.

Figure 31. Pilot scale flotation column

Pilot scale column flotation experiments were carried out in the first and second circuits of the copper circuit of the plant. Although the second cleaning circuit is mainly aimed, a series of tests were carried out in the first stage to understand the column dynamics and parameters.

For the first cleaning circuit studies, it is aimed to provide a regular and homogeneous slurry flow by placing a sample box on the feed line of the BU 4 cleaning circuit of the plant. The

slurry was taken from the sample box to an intermediate tank and fed from this tank to the column feeding conditioner tank with the help of a pump. In order to keep the slurry level in the column feeding conditioner tank constant, an overflow line was added on the conditioner, and it was aimed to keep the pressure depending on the height constant. In the first cleaning circuit studies, only cavitation air tests were carried out in the column and no tests were made with the sparger. In the first cleaning circuit studies, BU 4 first cleaning circuit) was sampled once a day and the results were compared with the column tests performed on the same day.

For the second cleaning circuit studies, the column feed slurry was taken to the intermediate tank through a sample box added to the feed line of the BU 7 cell. Similar to the first cleaning circuit, the slurry is fed from the intermediate tank into the column feeding conditioner tank by a pump. In the second cleaning circuit, in addition to cavitation and air supply, sparger studies were also carried out, and the results were examined by feeding at different rates.

The tests were carried out by varying the froth height, the amount of washing water, the air flowrate, and the speed of the circulation pump. One parameter was changed in each test series. The optimum values found were kept constant in the next test series and tests were carried out for the second parameter. In the second cleaning circuit studies, BU 7 (second cleaning circuit) and BU 8 (third cleaning circuit) were sampled once a day and the results were compared with the column tests performed on the same day.

6.2. First Cleaner Circuit Studies

Eti Bakir A.S Kure Corporation flotation plant copper cleaning circuit and the schema of the column study in the first cleaning circuit is as Figure 32.

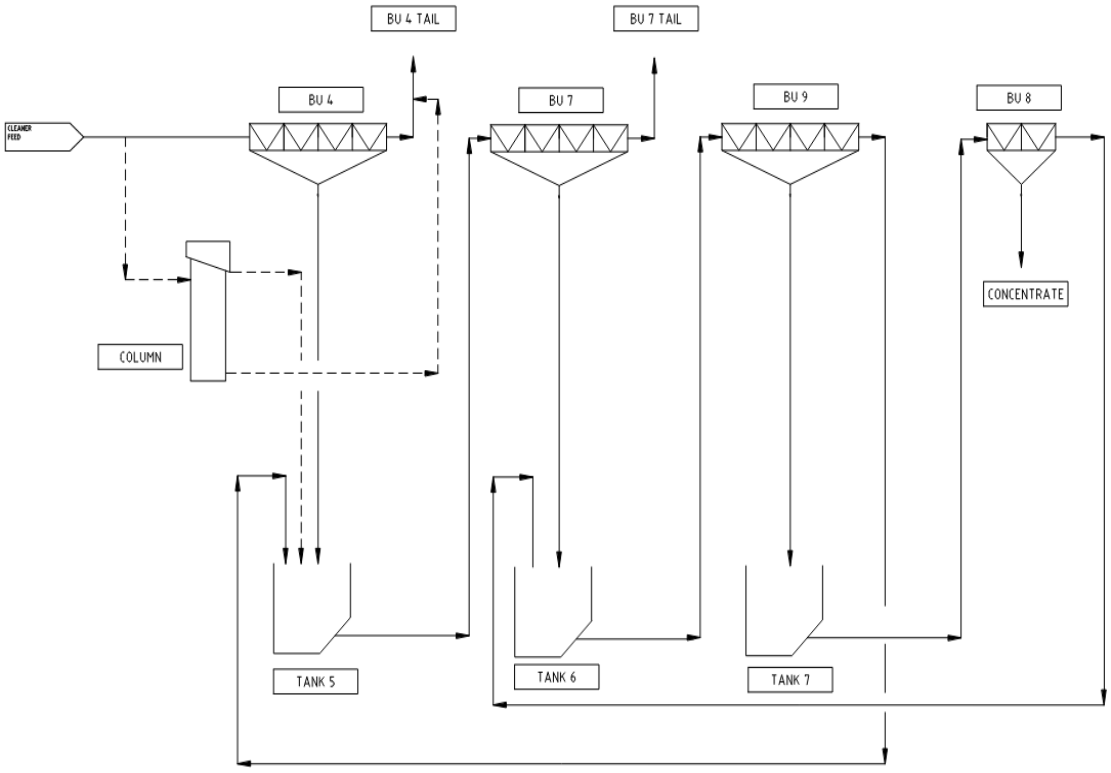


Figure 32. Copper flotation cleaning circuit

The tests in the same table in both circuits were completed on the same day and in a single series. For example, test 5, test 6 and test 7 were completed in the same day, test 8, test 9 and test 10 were completed in a different series the next day.

6.2.1. Froth Height

The first parameter to study for column flotation is froth height. The pressure sensor located on the upper part of the column is connected to the column panel and the tailings valve. To adjust the froth height, a set value is entered on the panel and if the process value of the pressure sensor is higher than the set value, the command is sent to the tailings valve to open. In the opposite case, the valve is closed, and the froth height is reduced.

The froth height is calculated by using an empirical formula of the values of the 2 pressure sensors located on the column. However, since the sensors on the system are constantly clogged and the stability of the column deteriorates during cleaning, the measurements were made with the help of a manual floating tool.

While the froth height tests were carried out, the height differences were kept high, as can be seen from the test conditions. The reason for this is that the valve works as on-off, not positional. As a result, froth overflow and tailings discharge are not provided continuously. It works as a batch system. While this difference in froth height in the open-close duration may vary between 2 and 3 cm, it has also been observed that it varies between 7 and 9 cm. For this reason, the distances between the froth height tests were kept high and the errors that could be caused by small height differences were tried to be minimized. The reason why the tests were not carried out above 45 cm is that the froth stability could not be ensured due to the closeness of the slurry height to the column feeding valve. In the first tests, the feed rate was 0.5 m3/h, the density was 1250 gr/lt, the air flowrate was 9 Nm3/h, the washing water was 10 lt/min and the motor speed was 80%. The first test results of the froth height studies can be found in Table 2.

Table 2. Froth height tests results - 1

Test No	Froth Height (cm)	Flows	Cu Grade (%)	Co Grade (%)
5	20	**Concentrate**	21.7	0.31
		Tail	7.89	0.53
		Feed	11.24	0.49
6	35	**Concentrate**	21.74	0.32
		Tail	7.48	0.54
		Feed	11.06	0.49
7	45	**Concentrate**	23.11	0.28
		Tail	7.7	0.52
		Feed	11.11	0.48

In the second and third tests, tests were carried out by increasing the feed rate to 1 m3/h and other parameters were kept at density 1250 gr/lt, air flowrate 9 Nm3/h, washing water 10 lt/min and engine speed at 80% and the results can be seen in Tables 3 and 4.

Table 3. Froth height tests results - 2

Test No	Froth Height (cm)	Flows	Cu Grade (%)	Co Grade (%)
8	15	**Concentrate**	12.47	0.47
		Tail	3.22	0.56
		Feed	8.03	0.51
9	30	**Concentrate**	16.89	0.42
		Tail	4.22	0.59
		Feed	7.73	0.54
10	45	**Concentrate**	17.89	0.39
		Tail	4.62	0.59
		Feed	7.92	0.53

Table 4. Froth height tests results - 3

Test No	Froth Height (cm)	Flows	Cu Grade (%)	Co Grade (%)
11	35	**Concentrate**	18.19	0.34
		Tail	5.94	0.57
		Feed	9.01	0.52
12	45	**Concentrate**	19.56	0.36
		Tail	4.7	0.6
		Feed	7.48	0.54

6.2.2. Air

Air is supplied to the column in 2 ways. The first is the cavitation tube which is connected to the discharge line of the circulation pump, and the second is the sparger, which is connected to the column from the bottom. Only cavitation tube studies were performed in the first cleaning circuit. Air is provided from the air pipeline connected to the plant. As mentioned before, the column main air pipeline continues by dividing into two. The first line is directly connected to the tailings valve by a solenoid valve. The second line is divided into two as cavitation and sparger air lines. There is only one flow meter for both air pipelines

and the flow rate is adjusted via the flow meter with a manual valve. The results of the air tests conducted is shown in tables 5.

Table 5. Air rate test results

Test No	Air Rate (Nm3/h)	Flows	Cu Grade (%)	Co Grade (%)
13	3	**Concentrate**	23.77	0.31
		Tail	8.25	0.47
		Feed	9.36	0.47
14	12	**Concentrate**	19.51	0.32
		Tail	5.69	0.53
		Feed	9.43	0.45

6.2.3. Wash Water

Since the minimum value of the flowmeter is 8 lt/min, 0 – 8 lt/min tests could not be performed. For this reason, the tests were completed at 8 lt/min and above in order to stay in safe conditions. Wash water test results are shown in Table 6.

Table 6. Wash water tests results

Test No	Wash Water (lt/min)	Flows	Cu Grade (%)	Co Grade (%)
22	0	**Concentrate**	19.59	0.33
		Tail	5.01	0.53
		Feed	9.2	0.49
23	8	**Concentrate**	19.75	0.29
		Tail	6.02	0.53
		Feed	9.69	0.42
24	10	**Concentrate**	17.56	0.3
		Tail	5.45	0.51
		Feed	11.00	0.45
25	12	**Concentrate**	17.15	0.31
		Tail	5.63	0.51
		Feed	9.87	0.44

6.2.4. Circulating Pump Speed

The slurry circulation pump used produces fine sized air bubbles through the cavitation tube in the discharge line. Moreover, the pump also prevents the precipitation of slurry on the bottom of the column. The speed of this pump is adjusted by changing the frequency with a switch on the panel. As the pump speed increases, the amount of slurry passing through the cavitation also increases. Accordingly, different types of bubbles occur at different speeds. An air called cavitation air is supplied to the discharge line of this pump and there is a cavitation tube at the point where this line connects to the column. The results of the pump speed tests performed in two series can be examined in table 7 and table 8.

Table 7. Pump speed test results - 1

Test No	Pump Speed (%)	Flows	Cu Grade (%)	Co Grade (%)
26	90	**Concentrate**	19.19	0.30
		Tail	4.24	0.48
		Feed	7.64	0.43
27	80	**Concentrate**	16.01	0.32
		Tail	3.5	0.46
		Feed	7.49	0.41
28	70	**Concentrate**	14.23	0.29
		Tail	3.22	0.41
		Feed	7.16	0.33

Table 8. Pump speed test results - 2

Test No	Pump Speed (%)	Flows	Cu Grade (%)	Co Grade (%)
29	60	**Concentrate**	16.34	0.42
		Tail	3.78	0.60
		Feed	6.39	0.56
30	70	**Concentrate**	14.82	0.43
		Tail	3.28	0.58
		Feed	6.26	0.54
31	80	**Concentrate**	18.32	0.37
		Tail	4.06	0.58
		Feed	7.1	0.53
32	90	**Concentrate**	20.93	0.30
		Tail	5.62	0.57
		Feed	8.06	0.53

6.3. Second Cleaner Circuit

The second cleaning circuit in the plant is the BU – 7 cells and its supply is combined with the concentrate of the BU 4 cells (first cleaning circuit) and the tailings from the BU 9 (third cleaning circuit) cells. The column feed is separated by a sample box placed on the BU 7 feed line. In the tests made in this circuit, the main problem is density. For this reason, the density was reduced by adding water to the column conditioner feeding tank. In case the slurry density exceeds 1300 gr/lt, the amount of concentrate floated through the column was too high and the froth overflow problem occurred in the launder. Although the water line was drawn on the launder and the froth were tried to be extinguished, the froth could not be extinguished, and the densities were reduced below 1300 gr/lt and the tests were completed.

While BU 4 sampling was done once in the same series in the first cleaning circuit, BU 7, and BU 8 (final concentrate circuit) cells were sampled in the same way in the second cleaning circuit. Sparger tests were added to the air tests while sampling the second cleaning circuit. Separate studies of cavitation and sparger air sources were examined and then their effects on the system in case of joint operation were tested. The scheme of the column, which was studied in the second copper cleaner circuit, is shown in figure 33.

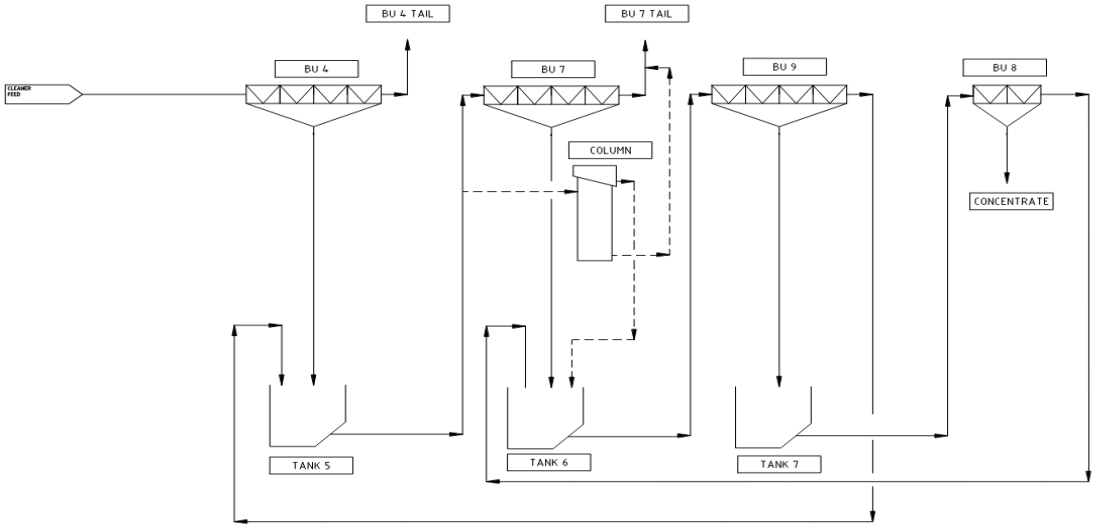


Figure 33. Second cleaning circuit column flotation studies

6.3.1. Froth Height

The test procedures were completed similarly to the first cleaning stage, and the first parameter tested was the froth height. Froth height test results are shown in Table 9.

Table 9. Froth height test results

Test No	Froth Height (cm)	Air Rate (Nm3/h)		Flows	Cu Grade (%)	Co Grade (%)
		Sparger	Cavitation			
33	15	0	3	**Concentrate**	12.29	0.36
				Tail	4.08	0.47
				Feed	10.94	0.38
34	30	0	3	**Concentrate**	15.27	0.33
				Tail	6.52	0.44
				Feed	10.91	0.38
35	45	0	3	**Concentrate**	17.55	0.29
				Tail	6.89	0.44
				Feed	11.41	0.39

6.3.2. Wash Water

Since the feed of the second cleaning circuit is cleaned in one step, it has been found that it is suitable for experiments with low air volume and high froth height. For this reason, a

series of wash water tests were carried out at low air volume. Froth height test results are shown in Table 10.

Table 10. Wash water test results

Test No	Air Rate (Nm3/h)		Wash Water (lt/min)	Flows	Cu Grade (%)	Co Grade (%)
	Sparger	Cavitation				
39	0	2	8	Concentrate	22.84	0.25
				Tail	8.08	0.41
				Feed	11.21	0.38
40	0	2	11	Concentrate	21.81	0.26
				Tail	7.70	0.42
				Feed	10.99	0.39
41	0	2	14	Concentrate	21.62	0.23
				Tail	6.18	0.43
				Feed	11.76	0.36

6.3.3. Cavitation Air

Cavitation and sparger air were tested separately in the first stage, and the tests to be given into the column in the form of a mixture were carried out in the next tests.

In the tests performed, a consistent decrease in the concentrate grade was observed while the feed grade fluctuated. Concentrate grade is higher in low cavitation air. Even if tests at 2 Nm3/h are carried out, the stability of the froth can easily deteriorate. For this reason, the optimum amount of cavitation air was determined as 3 Nm3/h. The test results in which only cavitation air is given can be found in Table 11.

Table 11. Cavitation air test result

Test No	Air Rate (Nm3/h)		Flows	Cu Grade (%)	Co Grade (%)
	Sparger	Cavitation			
42	0	3	Concentrate	24.61	0.21
			Tail	12.79	0.36
			Feed	16.06	0.33
43	0	6	Concentrate	22.90	0.22
			Tail	11.40	0.39
			Feed	15.08	0.33
44	0	9	Concentrate	21.64	0.26
			Tail	11.85	0.41
			Feed	15.63	0.35
45	0	12	Concentrate	18.75	0.28
			Tail	10.79	0.37
			Feed	14.27	0.36

6.3.4. Pump Speed Tests

Pump speed tests, which are effective on cavitation air, were carried out in 2 series. While the test results can be accessed from Tables 12 and 13.

Table 12. Pump speed tests results - 1

Test No	Pump Speed (%)	Flows	Cu Grade (%)	Co Grade (%)
46	80	Concentrate	21.11	0.21
		Tail	9.35	0.36
		Feed	12.39	0,30
47	90	Concentrate	22.71	0.20
		Tail	9.21	0.36
		Feed	14.39	0.30

Table 13. Pump speed tests results - 2

Test No	Air Rate (Nm3/h)		Pump Speed (%)	Flows	Cu Grade (%)	Co Grade (%)
	Sparger	Cavitation				
48	0	3	70	**Concentrate**	19.59	0.24
				Tail	10.2	0.43
				Feed	16.58	0.36
49	0	3	90	**Concentrate**	20.45	0.29
				Tail	8.86	0.45
				Feed	16.10	0.37

6.3.5. Feed Flowrate Tests

The effect of the feed flowrate made by a manual valve over the column conditioner on the system was examined with 2 data sets. Since the pump speed did not have a large effect, the first tests were performed at 70% pump speed, while the second tests were conducted at 80% pump speed. While the test results can be accessed from Tables 14 and 15.

Table 14. Feed flowrate tests results - 1

Test No	Flowrate (m3/h)	Air Rate (Nm3/h)		Flows	Cu Grade (%)	Co Grade (%)
		Sparger	Cavitation			
50	1	0	3	**Concentrate**	21.16	0,20
				Tail	11.8	0.31
				Feed	15.78	0.27
51	1.5	0	3	**Concentrate**	22.21	0.17
				Tail	13.36	0.30
				Feed	15.47	0.27

Table 15. Feed flowrate tests results - 2

Test No	Flowrate (m3/h)	Air Rate (Nm3/h)		Flows	Cu Grade (%)	Co Grade (%)
		Sparger	Cavitation			
52	0.5	0	3	Concentrate	26.30	0.16
				Tail	15.85	0.35
				Feed	16.39	0.33
53	1	0	3	Concentrate	22.66	0.23
				Tail	13.26	0.41
				Feed	16.05	0.34
54	1.5	0	3	Concentrate	20.97	0.27
				Tail	11.66	0.43
				Feed	15.05	0.36

It was observed that the concentrate grade was higher at a low feed rate. However, a few negative situations were encountered. It is difficult to adjust the flow rate in tests performed initially at a low feed flow rate (0.5 m3/h). It causes blockages on the feeding line and the condition deteriorates during the test. For this reason, it has been tried to keep it constant by measuring the flow rate at regular intervals, but it creates a problem in long-term test studies. Another negative situation is that the incoming ore floats directly when the grade is high, resulting in an irregular tailings flow.

The opposite situation is encountered in high feed flow tests. Due to the high incoming flow, the pressure on the sensor causes the set value entered to adjust for the froth height to rise rapidly and causes the froth height to go out of the reference range. In addition, an increase was observed in the flowrate of tailings. For these reasons, the feed rate was not changed, and the optimum value was determined as 1 m3/h.

6.3.6. Sparger Air Tests

The sparger system, which was not tested in the first cleaning circuit and produces larger air bubbles than the cavitation air, was tested in the second cleaning circuit. In the tests carried out with Sparger, the froth stability is quite unstable and even the washing water opening in drops is sufficient for the froth to disappear completely. For these reasons, no washing water was used in the tests performed with the sparger. Table 16 shows the sparger air test results.

Table 16. Sparger air tests results

Test No	Air Rate (Nm3/h)		Flows	Cu Grade (%)	Co Grade (%)
	Sparger	Cavitation			
60	3	0	Concentrate	20.87	-
			Tail	10.56	-
			Feed	10.84	-
61	6	0	Concentrate	19.63	0.26
			Tail	8.54	0.45
			Feed	10.47	0.44
62	9	0	Concentrate	17.84	0.29
			Tail	8.12	0.46
			Feed	8.78	0.44
63	6	0	Concentrate	19.01	-
			Tail	8.09	-
			Feed	8.58	-
64	9	0	Concentrate	18.69	0.25
			Tail	7.90	0.46
			Feed	8.49	0.42

6.3.7. Cavitation - Sparger Air Tests

The working conditions of cavitation and sparger air are investigated. In the test performed with 2 Nm^3\h sparger air and 1 Nm^3\h cavitation, the washing water was closed because it completely extinguished the froth. While the test results can be accessed from Tables 17 and 18.

Table 17. Cavitation - sparger air tests results - 1

Test No	Air Rate (Nm3/h)		Wash Water (lt/min)	Flows	Cu Grade (%)	Co Grade (%)
	Sparger	Cavitation				
56	0	3	8	Concentrate	20.96	0.22
				Tail	9.93	0.39
				Feed	14.74	0.31
57	2	1	0	Concentrate	22.38	0.17
				Tail	12.40	0.35
				Feed	12.77	0.33

Table 18. Cavitation - sparger air tests results - 2

Test No	Air Rate (Nm3/h)		Flows	Cu Grade (%)	Co Grade (%)
	Sparger	Cavitation			
58	1	2	**Concentrate**	22.61	0.24
			Tail	11.38	0.50
			Feed	13.68	0.46
59	2	2	**Concentrate**	22.65	0.28
			Tail	10.60	0.55
			Feed	12.38	0.50

6.4. Final Tests

As a result of the tests of the parameters and observations, the parameters promising satisfactory results were determined and the final tests were carried out on these parameters in the final test series. In this context, tests performed at low air volumes have positive results on the concentrate grade of the cleaning circuit. Although sparger air at low froth height gives positive results, only cavitation or sparger-cavitation mixtures at high froth height have been the preferred parameters since they provide more stability and stable froth. The final test parameters and results can be found in Table 19 and Table 20, respectively.

Table 19. Final tests parameters

Test No	Flowrate (m3/h)	Density (gr/l)	Froth Height (cm)	Air Rate (Nm3/h)		Wash Water (lt/min)	Pump Speed (%)
				Sparger	Cavitation		
65	1	1260	20	3	0	0	80
66	1	1258	38	0	3	<8	80
67	1	1245	38	2	2	<8	80
68	1	1255	38	2	3	<8	80

Table 20. Final parameters tests results

Test No	Flows	Cu Grade (%)	Co Grade (%)
65	Concentrate	24.05	0.21
	Tail	13.24	0.41
	Feed	14.19	0.40
66	Concentrate	21.64	0.26
	Tail	10.32	0.47
	Feed	14.82	0.39
67	Concentrate	23.5	0.16
	Tail	13.17	0.42
	Feed	13.84	0.40
68	Concentrate	23.13	0.21
	Tail	12.00	0.46
	Feed	14.13	0.40

7. DISCUSSIONS

The test results set by changing the froth height were compatible with the literature [71]. When tests 33, 34 and 35 were evaluated, it was observed that the copper concentrate grade increased with the increase in froth height and a decrease in the cobalt grade entrainment into the concentrate. Comparative results for samples taken simultaneously from the second cleaning circuit which is BU 7 cells and the column are shown in Table 21.

Table 21. Froth height comparative test results

Test No	Flows	Cu Grade (%)	Co Grade (%)
33	**Concentrate**	12.29	0.36
	Tail	4.08	0.47
	Feed	10.94	0.38
34	**Concentrate**	15.27	0.33
	Tail	6.52	0.44
	Feed	10.91	0.38
35	**Concentrate**	17.55	0.29
	Tail	6.89	0.44
	Feed	11.41	0.39
BU 7	**Concentrate**	15.01	0.34
	Tail	4.14	0.46
	Feed	12.33	0.39

According to the column flotation results, column flotation concentrate grade result is higher compared to the existing mechanical cells of the plant. Figure 34 shows the graph of the results.

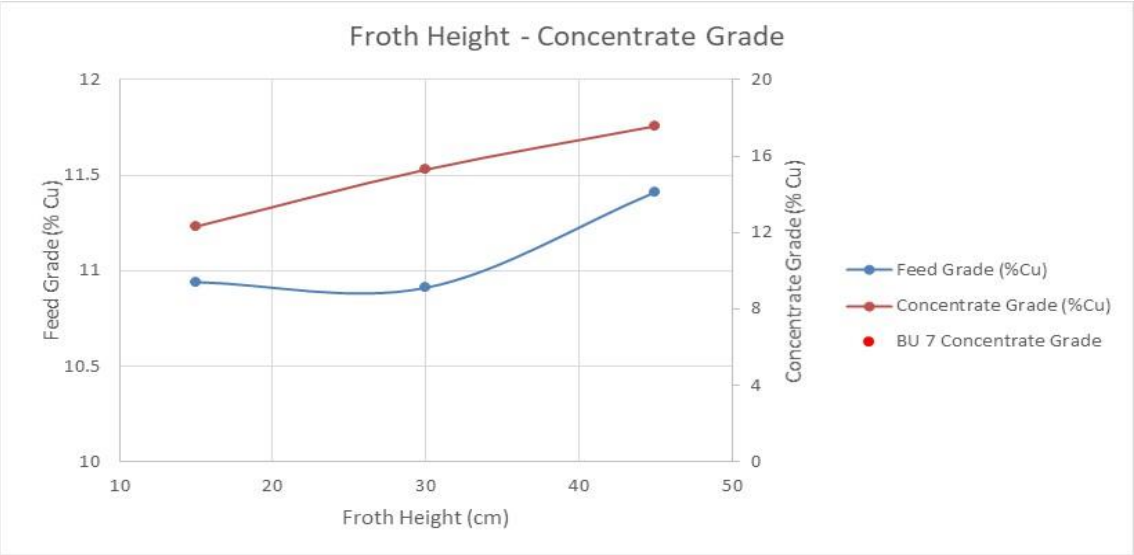


Figure 34. Froth height tests grade relations

When the literature is reviewed, according to Jera and Bhondayi, increasing the amount of washing water up to an optimum point cleans the froth and increases the concentrate grade [72]. It cleans the froth and ensures that the gangue minerals carried to the froth with entrainment return back into the pulp. During the column flotation tests, the maximum flow was determined as 8 lt/min. Although it was desired to conduct experiments with less amounts of washing water, unfortunately it was not possible because of the flowmeter. However, as can be seen from Table 22, with the increase of washing water after the optimum point, a decrease in copper grade is observed. Also, Table 22 shows the results of the second cleaning circuit (BU 7) and the final fourth cleaning (BU 8) circuits taken from the plant simultaneously.

Table 22. Wash water comparative test results

Test No	Flows	Cu Grade (%)	Co Grade (%)
39	**Concentrate**	22.84	0.25
	Tail	8.08	0.41
	Feed	11.21	0.38
40	**Concentrate**	21.81	0.26
	Tail	7.70	0.42
	Feed	10.99	0.39
41	**Concentrate**	21.62	0.23
	Tail	6.18	0.43
	Feed	11.76	0.36
BU 7	**Concentrate**	15.97	0.31
	Tail	7.69	0.42
	Feed	10.71	0.37
BU 8	**Concentrate**	20.16	0.25
	Tail	14.5	0.34
	Feed	-	-

As can be seen from Figure 35, the results obtained with column flotation are higher than the results obtained from the plant mechanical cells.

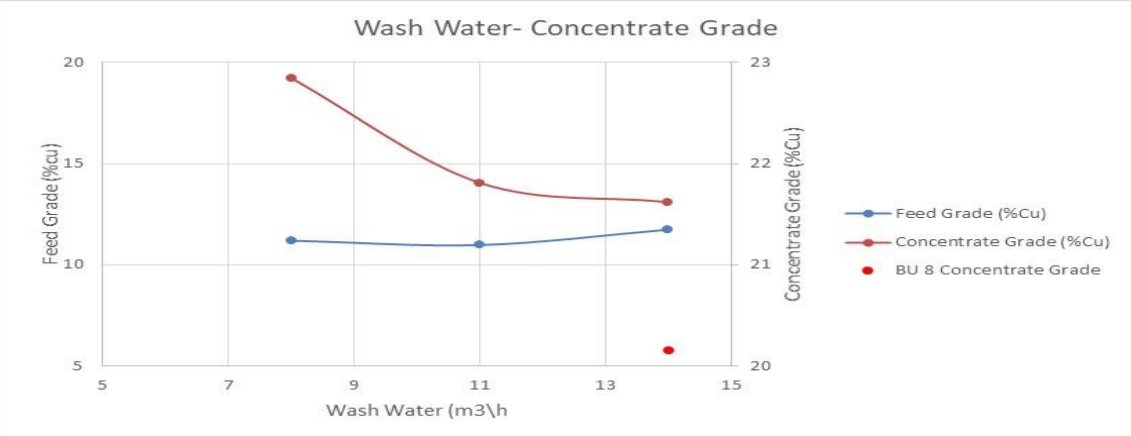


Figure 35. Wash water tests grade relations

While there is consensus that bubble size reductions will improve recovery due to increased surface area and therefore higher probability of contact between bubbles and particles, the effect has not been measured over a wide range of operating conditions. Data on the effect of this variable on selectivity are sparse, mainly due to difficulties in determining bubble size [73]. In the studies, when the sparger and cavitation air suppliers are used separately, decreases in grade were observed with the increase in the air flowrate. However, in case both air types are fed into the system together, higher of the sparger air increases the grade, while higher cavitation air increases the recovery. This provides great convenience in order to adjust the concentrate quality as desired. The results of the samples taken simultaneously from the column and the plant can be found in Tables 23 and 24, and the graphic drawings in Figures 36 and 37.

Table 23. Cavitation-sparger air first tests comparative results

Test No	**Flows**	**Cu Grade (%)**	**Co Grade (%)**
56	**Concentrate**	20.96	0.22
	Tail	9.93	0.39
	Feed	14.74	0.31
57	**Concentrate**	22.38	0.17
	Tail	12.40	0.35
	Feed	12.77	0.33
BU 7	**Concentrate**	15.88	0.27
	Tail	6.75	0.42
	Feed	12.61	0.33
BU 8	**Concentrate**	19.33	-
	Tail	14.31	-
	Feed	18.92	-

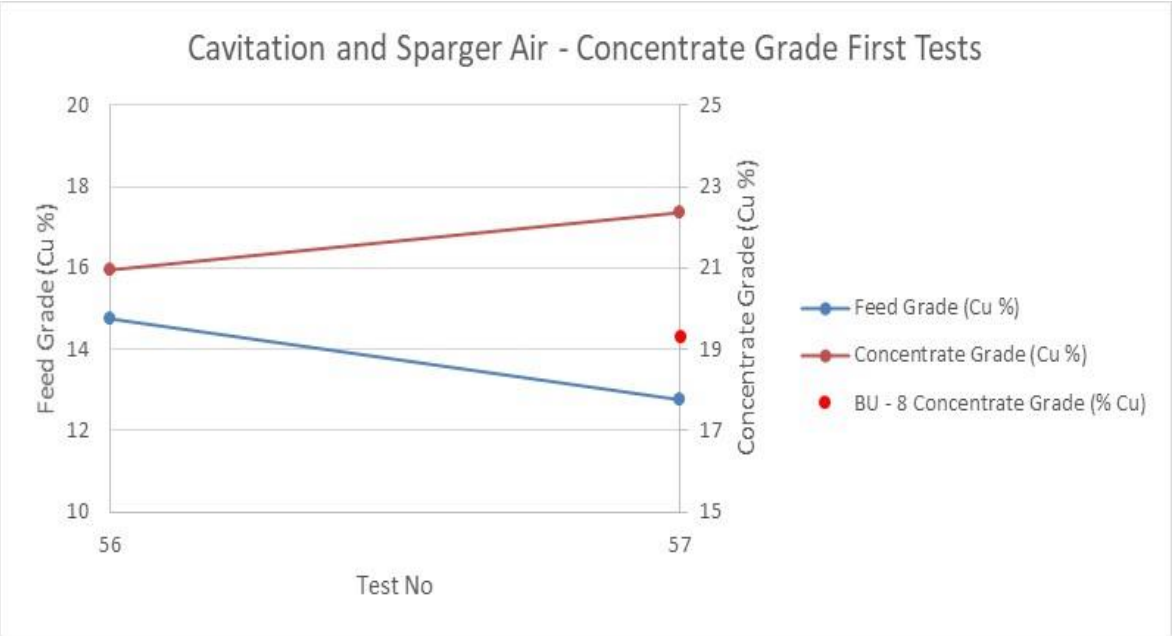


Figure 36. Cavitation - sparger air first tests grade relations

Table 24. Cavitation-sparger air second tests comparative results

Test No	Flows	Cu Grade (%)	Co Grade (%)
58	**Concentrate**	22.61	0.24
	Tail	11.38	0.50
	Feed	13.68	0.46
59	**Concentrate**	22.65	0.28
	Tail	10.60	0.55
	Feed	12.38	0.50
BU 7	**Concentrate**	17.35	0.39
	Tail	7.40	0.56
	Feed	13.23	0.47
BU 8	**Concentrate**	19.02	-
	Tail	12.87	-
	Feed	18.49	-

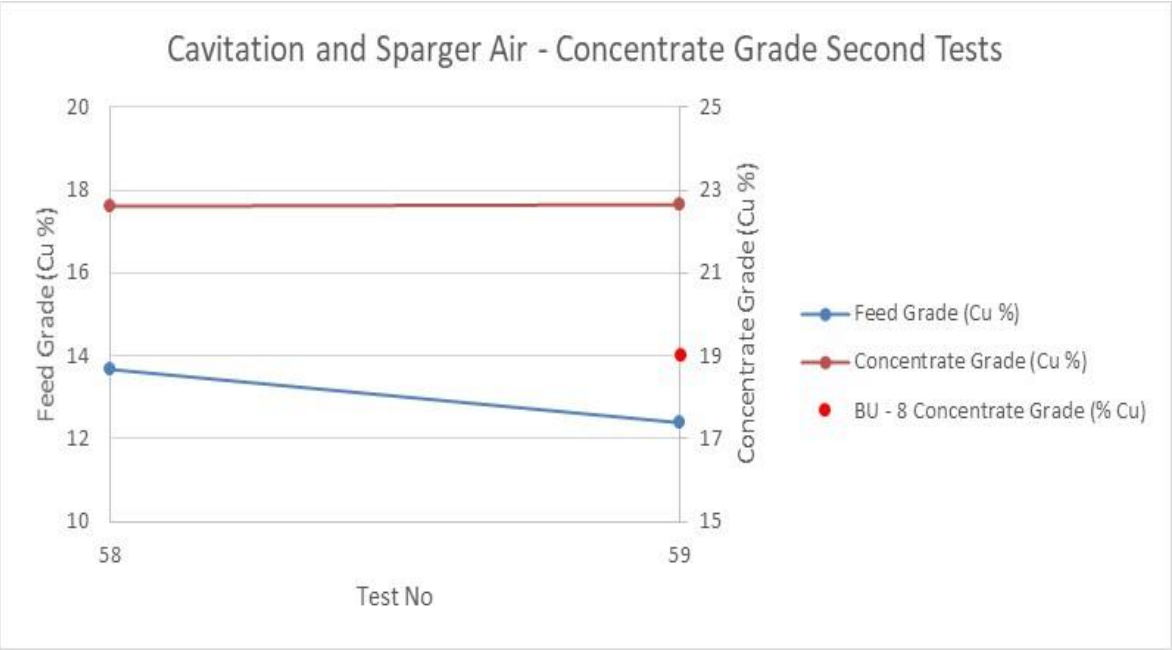


Figure 37. Cavitation - sparger air second tests grade relations

After the parameters were determined, final tests were carried out with parameters that gave good and promising results. The purpose of these tests is to see if a higher grade can be obtained by comparing them with samples taken simultaneously from the processing plant mechanical flotation cells. Table 25 shows the results of the final tests and the results for the second and fourth cleaning circuits from the plant. Graphic of the data can be found in Figure 38.

Table 25. Final tests comparative results

Test No	Flows	Cu Grade (%)	Co Grade (%)
65	Concentrate	24.05	0.21
	Tail	13.24	0.41
	Feed	14.19	0.40
66	Concentrate	21.64	0.26
	Tail	10.32	0.47
	Feed	14.82	0.39
67	Concentrate	23.5	0.16
	Tail	13.17	0.42
	Feed	13.84	0.40
68	Concentrate	23.13	0.21
	Tail	12.00	0.46
	Feed	14.13	0.40
BU 7	Concentrate	17.38	0.35
	Tail	7.45	0.51
	Feed	13.88	0.39
BU 8	Concentrate	18.27	-
	Tail	12.44	-
	Feed	17.91	-

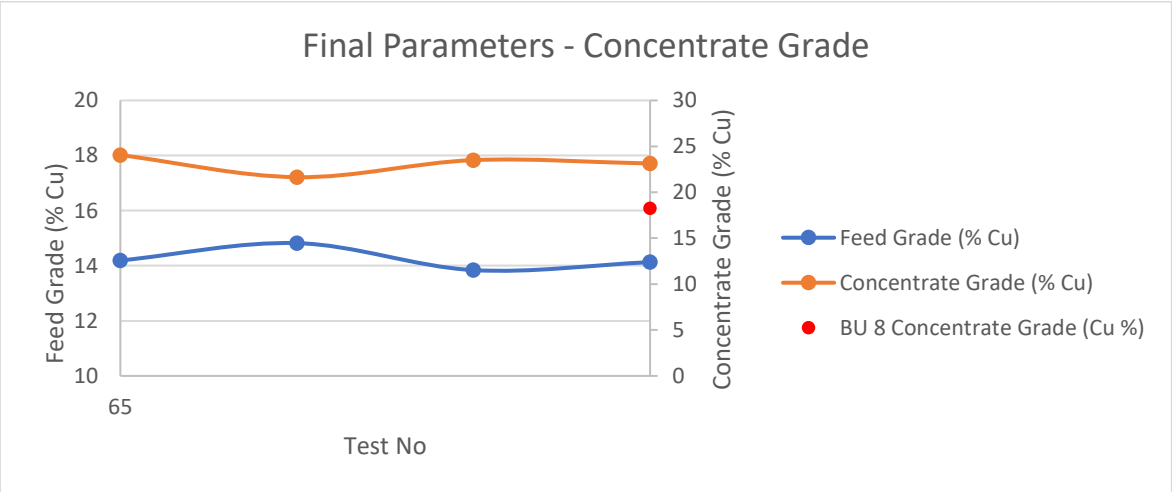


Figure 38. Final tests grade relations

Although tests have been carried out on pump speed with different values and sets, no significant effects of pump speed on the second cleaning circuit were observed.

Polished sections were prepared from the samples taken from the column concentrate. Although there was no quantitative analysis, a decrease was observed in the free pyrite particles in the column concentrate, despite the plant's final concentrate.

As a result of the column flotation tests applied in the processing plant, higher grade concentrate can be obtained despite the decrease in recovery. The suggested flow diagram is shown in Figure 39.

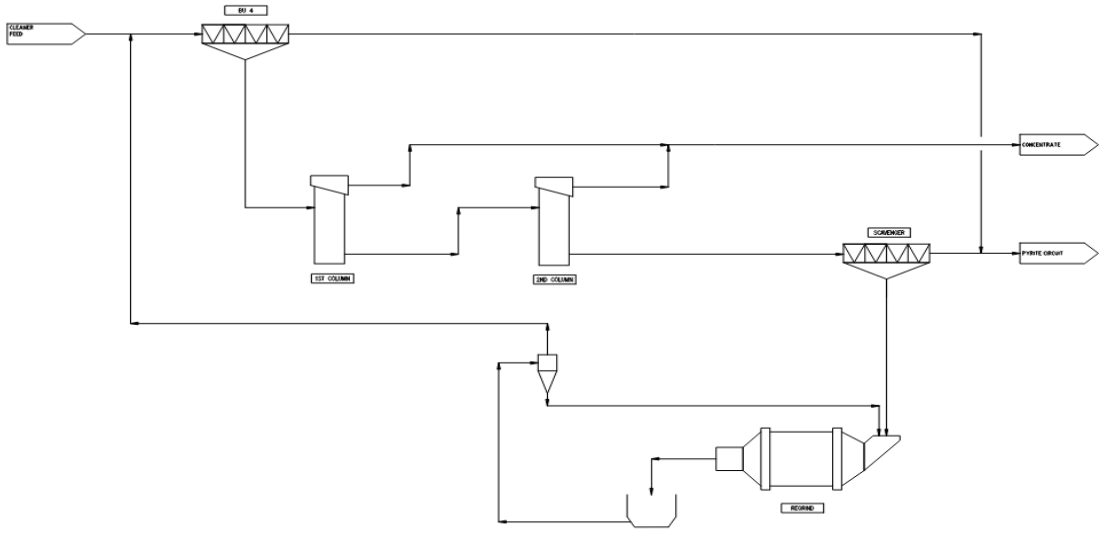


Figure 39. Suggested flowsheet of the copper cleaning circuit

8. RESULTS AND RECOMMENDATIONS

In the studies carried out with the pilot scale column flotation cell, the copper grade concentrated in the 3-stage cleaning circuit in the Eti Bakir Corporation Kure Plant can be gained in a single circuit. According to the results;

- Tests with low air volumes give high results on the final concentrate grade of the cleaning circuit.
- An increase in the concentrate grade was observed until the optimum value. After the amount of washing water exceeded an optimum value, it showed a negative effect.
- The copper grade in the concentrate increased with increasing the froth height. Moreover, a decrease was observed in the cobalt grade which gained to the concentrate with entrainment.

In tests with the final parameters, an average of 26.33% higher concentrate was obtained than the concentrate grade of BU 8 final concentrate cells on the same day. As it is common in the industry, a second cleaning column and a scavenger circuit are recommended.

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APPENDICES

Test No	Cu (%)			Co (%)			Total S (%)		
	Feed	Concentrate	Tail	Feed	Concentrate	Tail	Feed	Concentrate	Tail
Test 65	14.27	24.27	13.32	0.4015	0.1969	0.4032	45.98	40.91	46.45
Test 66	14.76	21.84	10.06	0.3957	0.2701	0.4767	46.1	42.16	49.48
Test 67	14.41	27.04	12.64	0.4028	0.1565	0.3985	46.41	39.55	44.74
Test 68	13.95	24.25	11.47	0.4046	0.2064	0.4408	46.07	41.37	46.67
BU 7	14.63	17.94	7.24	0.4055	0.3415	0.5329	46.95	45.15	50.98
BU 8	18.97	19.18	12.58	0.3225	0.3230	0.4334	44.99	44.42	47.36

Appendices 1.Final tests detailed analysis results

Test 41 Analyses Results					
Analyses	**Feed**	**Column Concentrate**	**Column Tail**	**BU 7 Concentrate**	**BU 8 Final Concentrate**
Cu (%)	11,91	21,75	7,45	15,25	20,45
Co (%)	0,36	0,23	0,42	0,32	0,25
Total S (%)	46.83	43.54	49.42	45.64	42.54

Appendices 2. Test 41 detailed analysis results