ISSN No: -2456-2165 https://doi.org/10.38124/ijisrt/25oct823

Flotation Tests of Poor Copper and Cobalt Tailings from Dike 3 of Kipushi

Mungwa Kalundu Gaylord[1,2,4*]; Ipang Ruwej Rebecca[5]; Ndala Mbavu Bavon[1,2,4]; Zeka Mujinga[1,3,4,5]

[1]University of Lubumbashi, Polytechnic Faculty, Department of Industrial Chemistry, Lubumbashi, Democratic Republic of Congo
[2]Kolwezi Higher Institute of Applied Technology, Department of Applied Chemistry and Metallurgy, Kolwezi, Democratic Republic of Congo
[3]Gécamines Metallurgical Research Department, Lubumbashi, Democratic Republic of Congo
[4]Zebra Research and Expertise Center (CreZ), Kolwezi, Democratic Republic of Cong
[5]University of Kolwezi, Polytechnic Faculty, Department of Industrial Chemistry, Kolwezi, Democratic Republic of Congo

Corresponding Author: Mungwa Kalundu Gaylord*

Publication Date: 2025/11/06

Abstract: Current technological developments are pushing companies to recover waste and recycle refuse. With this in mind, the treatment of waste from dam 3 represents a significant opportunity to recover the metals contained in this waste and thus maximise the use of these resources.This recovery is achieved by increasing the content of recoverable metals in the tailings, based on a characterisation of our sample, which leads us to choose a concentration technique that is slightly more suitable for this type of tailings.

Having chosen froth flotation as the technique, the first part of the experiment focused on characterising the sample through mineralogical analysis, chemical analysis and granulochemical analysis. The results showed that total copper is at 0.94% and oxide at 0.85%. The main mineral is malachite, with traces of chrysocolla. The gangue consists of quartz, iron oxides, pyrite and dolomite; it is therefore dolomitic in terms of the Cutotal/CaOsoluble ratio.The reject is crushed to a D70 of 75 microns in 20 minutes.

The second part focused on laboratory flotation tests, with reagent doses such as NaHS and KAX maintained at 3000 g/t and 300 g/t for a recovery yield of 44.8% with a concentrate of 1.11%.We realised that the sulphide portion could be minimised to perform mixed flotation, so we moved on to oxide flotation by sulphurisation, where we varied the doses of all our reagents and obtained the following best doses.

- **NaHS and KAX: 500g/t and 50g/t, with an estimated content of 1.12%;**
- **mixture: 200g/t, with a rough grade of 1.27%;**
- **silicate: 100g/t, with a rough grade of 1.17%.**

These results simply show that the initial goal could not be achieved, but with this we can try to feed the leaching reject.

Keywords: *Tailings Valorization, Oxide Flotation, Sulfidization, Malachite, Chrysocolla, Mine Waste Treatment, Copper Recovery, Flotation Reagents, Nahs, KAX, Leaching, Dam 3 Tailings, Ore Beneficiation, Dolomite, Dolomitic Gangue*.

How to Cite: Mungwa Kalundu Gaylord; Ipang Ruwej Rebecca; Ndala Mbavu Bavon; Zeka Mujinga (2025). Flotation Tests of Poor Copper and Cobalt Tailings from Dike 3 of Kipushi. *International Journal of Innovative Scienceand Research Technology*,10(10), 2597-2603. https://doi.org/10.38124/ijisrt/25oct823

I. INTRODUCTION

In the mining industry, mineral concentration is one of the most important steps in the process of extracting metals such as copper and cobalt.

Current technological developments are pushing companies to recover waste and recycle refuse; it is with this in mind that the treatment of waste from dyke 3 represents a significant opportunity to recover the metals contained in this waste and thus maximise the use of these resources.

This recovery is achieved by increasing the content of recoverable metals in the waste, starting with a characterisation of our sample, which leads us to choose a concentration technique that is slightly more suited to this type of waste. We have therefore opted for froth flotation simply because the nature of our gangue, the initial content of our recoverable metals and the size of our particles do not allow us to move directly to another concentration technique such as stirred tank or heap leaching.

Flotation is a separation process in which particles of different minerals constituting the ore are separated after selectively rendering the minerals to be floated hydrophobic. The hydrophobic particles attach themselves to rising air bubbles and are thus carried upwards and collected in a froth on the surface of the pulp, while the hydrophilic particles remain in the pulp and constitute the tailings (CORNEILLE, 1973).

The attachment of the particles to the gas bubbles results in a lighter complex that rises and floats to the surface of the pulp. Thus, fundamentally, separation occurs due to density, but the main characteristic of this separation remains the adhesion of the particles to the gas bubbles.

Flotation therefore appears to be a physicochemical process because there is an interaction between the physical and chemical properties of the three phases present: gas, liquid and solid particles (KALENGA, 1992).

Foam flotation (the most commonly used method) is based on the formation of an aggregate between an air bubble and one or more solids rendered hydrophobic and aerophilic by the action of specific chemical reagents called collectors. The loaded bubbles are subjected to tear-off forces as they rise through the machine. If the bubble-particle adhesion force is less than the tear-off force, the particle separates from the bubble and falls back down. These particles can be picked up by other bubbles, separated again, and so on. Equally visibly, the grains undergoing these successive phenomena gradually lose their hydrophobicity, either through simple desorption of the collector reagent or through attrition. This loss of individual hydrophobicity ultimately results in a lower flotation rate, which eventually becomes zero.

The principle of flotation is based on the hydrophobic and hydrophilic properties of solid surfaces. These properties may be natural or stimulated by adding a suitable reagent to the water in which the solid particles are suspended. When air is introduced in the form of small bubbles into such a medium, selective transport of the hydrophobic particles occurs. Particles with hydrophobic surfaces attach themselves to air bubbles when they collide with them. This phenomenon is due to the high affinity of hydrophobic surfaces for air, which is non-polar in nature. The air bubbles carry these particles to the surface of the pulp, where they form a charged foam. However, particles with hydrophilic surfaces do not bind to the air bubbles and remain suspended in the pulp. The products of flotation are the concentrate and the tailings.

With regard to the tailings from dam 3, the literature states that the tailings from the Kipushi dam 3 are a mixture of the sulphides deposited by the former Kipushi concentrator and the oxides stored by the CMSK company. In 1995, the estimated tailings from dam 3 were 639,330 $m^{(3)}$ ·or approximately 1,534,390 dry tonnes. The results of the mineralogical analysis show that there is a mixture of sulphides and oxides, which means that we are dealing with a mixture.

This work will result in a feed for leaching, which has led us to study leaching techniques in order to determine the one that will be best suited to the characteristics of our feed.

Leaching consists of dissolving the desired metal(s) into solution in ionic form. The aim is to determine the optimal type of leachate in terms of reagent consumption and cost, minimal solubilisation of impurities and equipment maintenance.

The leaching reaction will therefore be carried out with the lowest possible reagent consumption and minimal contamination of the solution with impurities, which is why the solvent used must be selective (C.B GILL FAYETTE, 1980). The size and grade of the ore and the ease with which it dissolves in the solvent are the factors that determine the choice of leaching method to be adopted (FATHI HABASHI, 1970), while varying the particle size of the ore, the leaching time, the acid concentration and the reducing agent content.

II. MATERIALS AND METHODS

The tailings sample used in our study came from dam 3 in Kipushi; it is a mixed copper-cobalt tailings sample.

The sample was taken systematically so as to be as representative as possible. A quantity of 200 kg was taken and then stored at EMT in crates in a warehouse after mechanical preparation.

➢ *The Flotation Test Results Will be Assessed on the Basis of the Following Factors:*

- The copper content of the concentrate;
- Copper recovery yield.

ISSN No: -2456-2165 https://doi.org/10.38124/ijisrt/25oct823

- *The Flotation Tests Carried Out Were Aimed at the Following Objectives:*

- The influence of the dose of sodium sulphhydrate (NaHS), potassium amylxanthate (KAX), sodium silicate and the mixture on the discharge from dam3.

- *The Following Equipment was Used for the Laboratory Flotation Tests:*

- A laboratory ball mill;
- A METSO laboratory flotation machine;
- 2.5-litre capacity flotation cells;
- An oven for drying samples;
- A SARTORIUS electronic balance ;
- Panels to collect the foam laden with floating particles;
- A stopwatch;
- Graduated pipettes and syringes for measuring out reagents.

- *The Following Reagents were Used in Our Tests:*

- *Potassium Amylxanthate (KAX):*

Main collector prepared at 1.2%, i.e. 1.2 g of KAX in 100 ml of water;

- *Sodium Silicate (Na_2SiO_3):*

Depressant and dispersant, brought to the laboratory in the form of a 50% solution and diluted to 30% by adding water;

- *Sodium Sulphhydrate (NaHS):*

Sulphurising agent, prepared as a 12% solution, i.e. 12 g of NaHS in 100 ml of water;

- *Mixture 90/10:*

Secondary collector or auxiliary collector, prepared in the form of a 1.2% by weight solution. It consists of a mixture of 90% diesel fuel and 10% Rinkalore 10, to which a few drops of sodium carbonate are added to stabilise the emulsion;

- *Senfroth (G41):*

A foaming agent used in its pure form, i.e. as sold commercially.

- *Procedure*

This section includes the procedures for implementing the adopted methodology. During the flotation tests, the following operating procedure was followed:

- Weigh 1200 g of ore previously crushed to -10 mesh (1.7 mm);
- Place this mass of ore in the ball mill;
- Add 1200 mLq of water to obtain a pulp with a dilution ratio of 1:1;
- Grind the resulting pulp for time x;
- Collect the discharge from the grinding in a 2.5 litre capacity flotation cell;
- Add the sodium silicate, two and a half minutes later, the mixture of diesel oil and Rinkalore four minutes later, add the sodium sulphhydrate and collector in fractions; at the fifth minute, add a few drops of foaming agent (G41), then open the air intake valve in the pulp;
- Float the ore in five fractions for oxides and seven fractions for mixed ores, following the diagram.

III. RESULTS AND DISCUSSION

The aim is to increase the copper and cobalt content in our tailings in order to improve their leaching, so here the aim is to present the results of the various flotation tests carried out on our sample. These results will be presented in tables and illustrated by figures, then interpreted based on the evaluation criteria, i.e. the copper and cobalt content in the concentrates, as well as the recovery yields of these metals.

- *Chemical Analysis of the Sample*

The chemical analyses, carried out at EMT by atomic absorption spectrometry, gravimetric methods and ICP (Inductively Coupled Plasma), gave the results shown in Table 1.

Table 1 Results of the Chemical Analysis of the Sample

Elements	**Content (%)**	**Elements**	**Content(%)**
Cu tot.	0,94	Al_2O_3	-
Cu ox.	0,85	SiO_2	41,64
Cu mal.	-	Fe tot.	1,58
Co tot.	0,39	P_2O_5	-
Co ox.	0,28	Au	0,06 ppm
S	0,46	Cd	4,00 ppm
CaO tot.	1,49	Pb	20 ppm
CaO sol.	0,38		
MgO tot.	8,33		
MgO sol.	2,62		
Mn	0,043		
Zn	0,04		

ISSN No: -2456-2165 https://doi.org/10.38124/ijisrt/25oct823

An examination of these tables shows that this sample is therefore a dolomitic oxide with a total Cu/Ca Sol ratio of 2.41.

➢ *Mineralogical Analysis of the Sample*

Observation under a binocular optical microscope of the sample consisting of grains of – 10 mesh (1.7 mm) revealed the presence of the following minerals:

- Malachite ($Cu_2CO_3(OH)_2$) is the main copper mineral, with traces of chrysocolla (CuSiO3.nH2O).
- Chalcopyrite (CuFeS2), bornite (Cu5FeS4), chalcocite (Cu_2S) as trace sulphide minerals ;
- The gangue consists of quartz (SiO_2), iron oxides (Fe_2O_3), pyrite (FeS_2) and dolomite ($MgCa_2$).

➢ *Preflotation Tests*

A series of four tests was carried out using different doses of sulphhydrate (NaHS) (500, 1000, 2000 and 3000 g/t). The doses of sodium silicate were 200 g/t and SIBX 100 g/t. The KAX collector was taken in the same 1/10 ratio of NaHS.

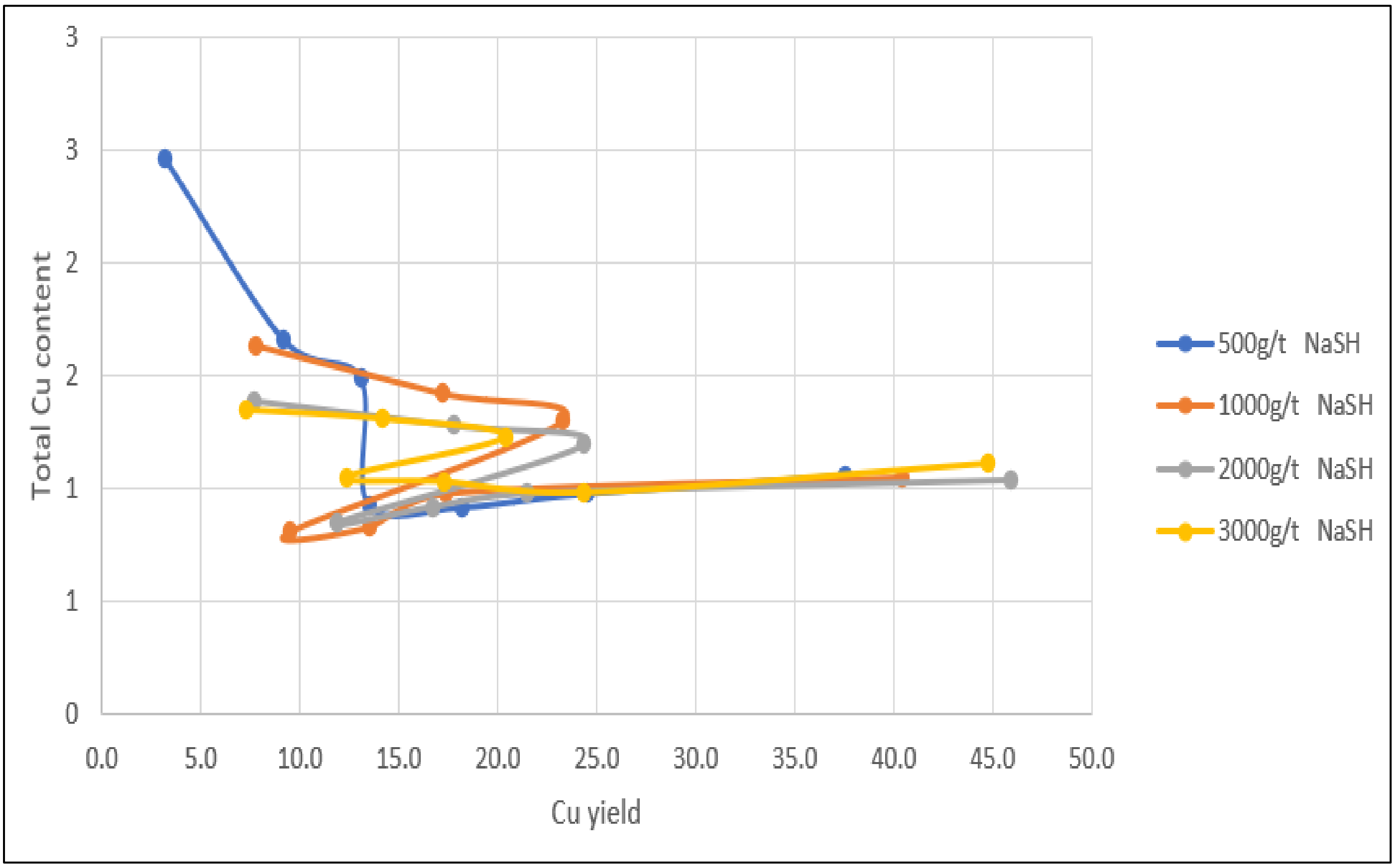

Fig 1 Copper Flotation at Different Doses of NaHS and KAX

The results obtained here merely guided us by showing that the reject was heavily oxidized.

➢ *Tests Without Preflotation*

Here we studied the influence of all our reagents while varying the doses.

Table 2 Variation in Reagent Doses.

DOSE	SILICATE	MIXTURE	NaHS	KAX
1	100	100	500	50
2	200	200	1000	100
3	400	300	2000	200
4	500	400	3000	300

A series of 12 tests was carried out here to determine the best doses to maintain.

➢ *Influence of silicate*

After analysing the curves, the best dose to maintain is as follows:

- Silicate 100g/t, with a recovery yield of 37% and a rough concentrate of 1.17%.

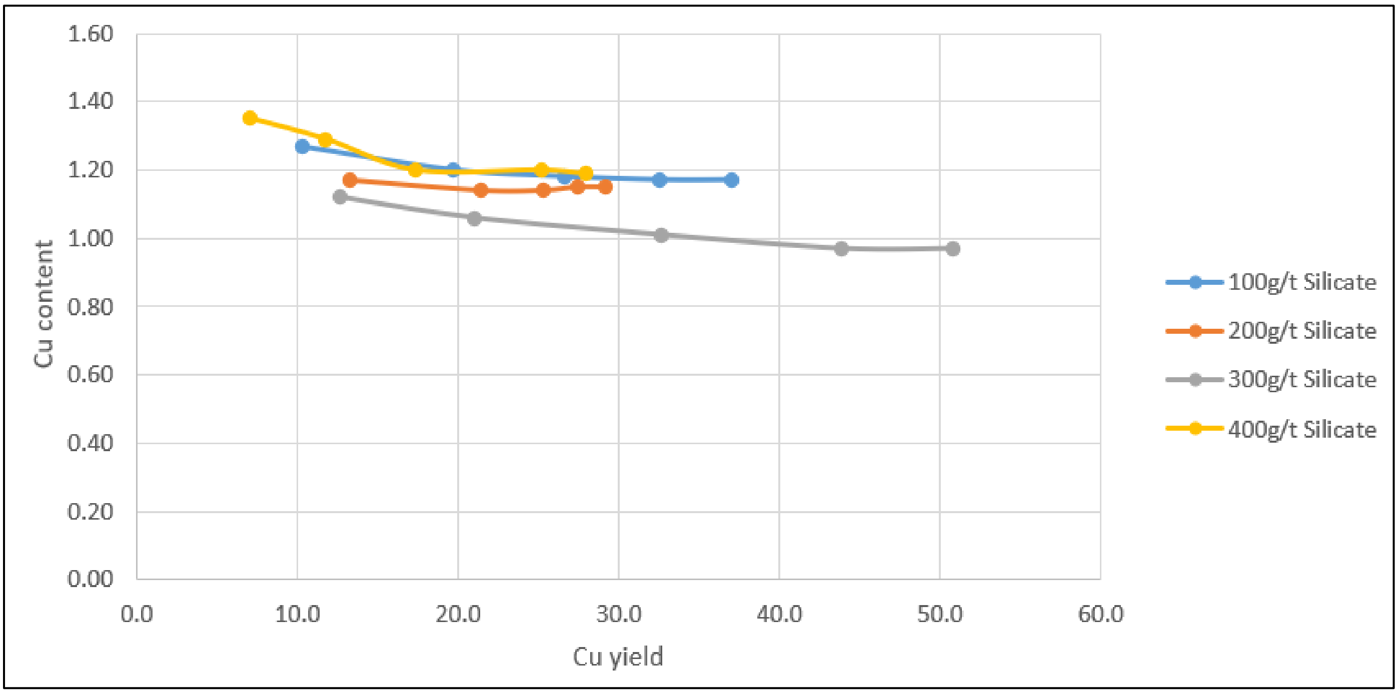

Fig 2 Copper Flotation at Different Silicate Doses

➢ *Influence of the Mixture (90/10)*
After analysing the curves, the best dose to maintain is as follows:

- Mixture 200g/t, with a recovery yield of 37% and a rough concentrate of 1.27%

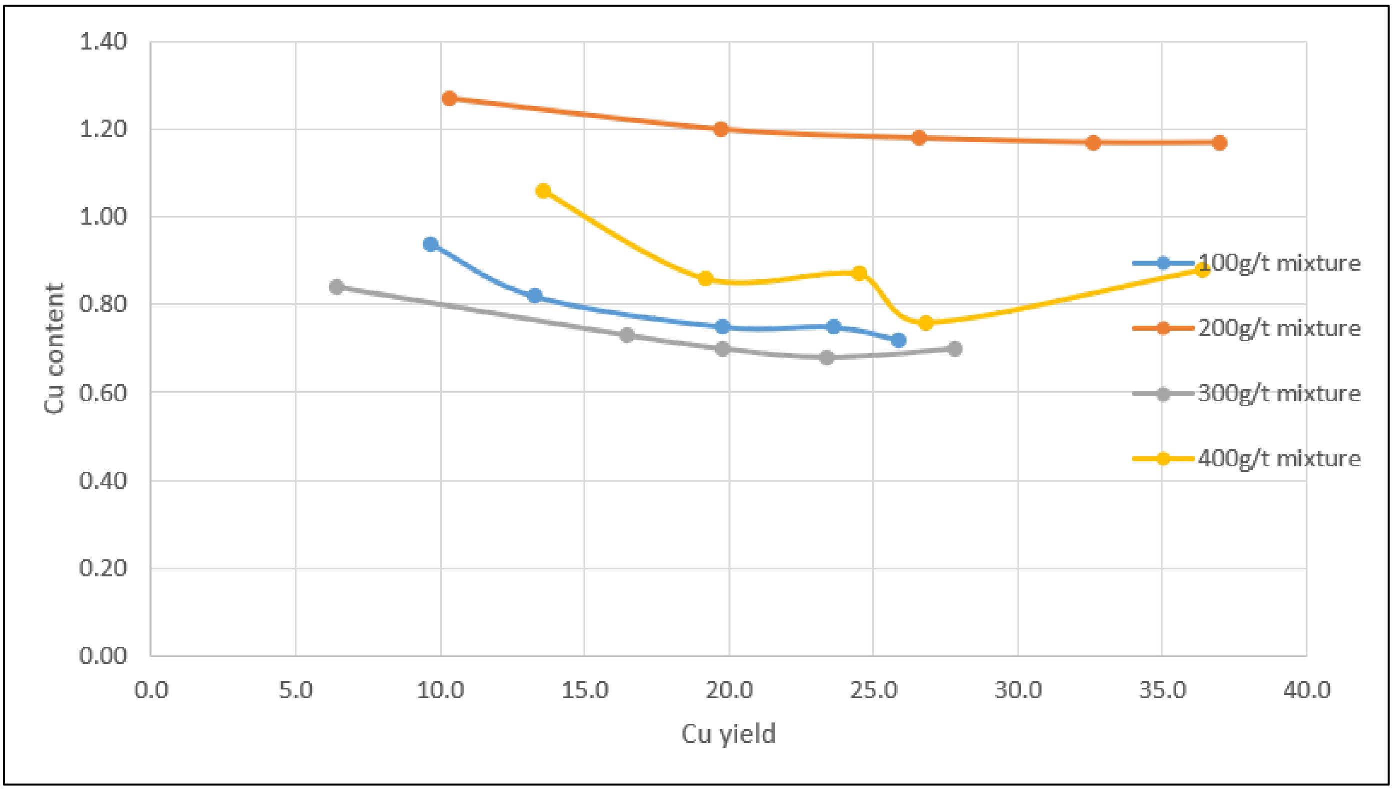

Fig 3 Copper Flotation at Different Mixture Doses

➢ *Influence of NaSH and KAX*
After analysing the curves, the best doses to maintain are as follows:

- NaHS 500g/t and KAX 50g/t, with a recovery yield of 50.9 and concentrate of 1.12%.

ISSN No: -2456-2165 https://doi.org/10.38124/ijisrt/25oct823

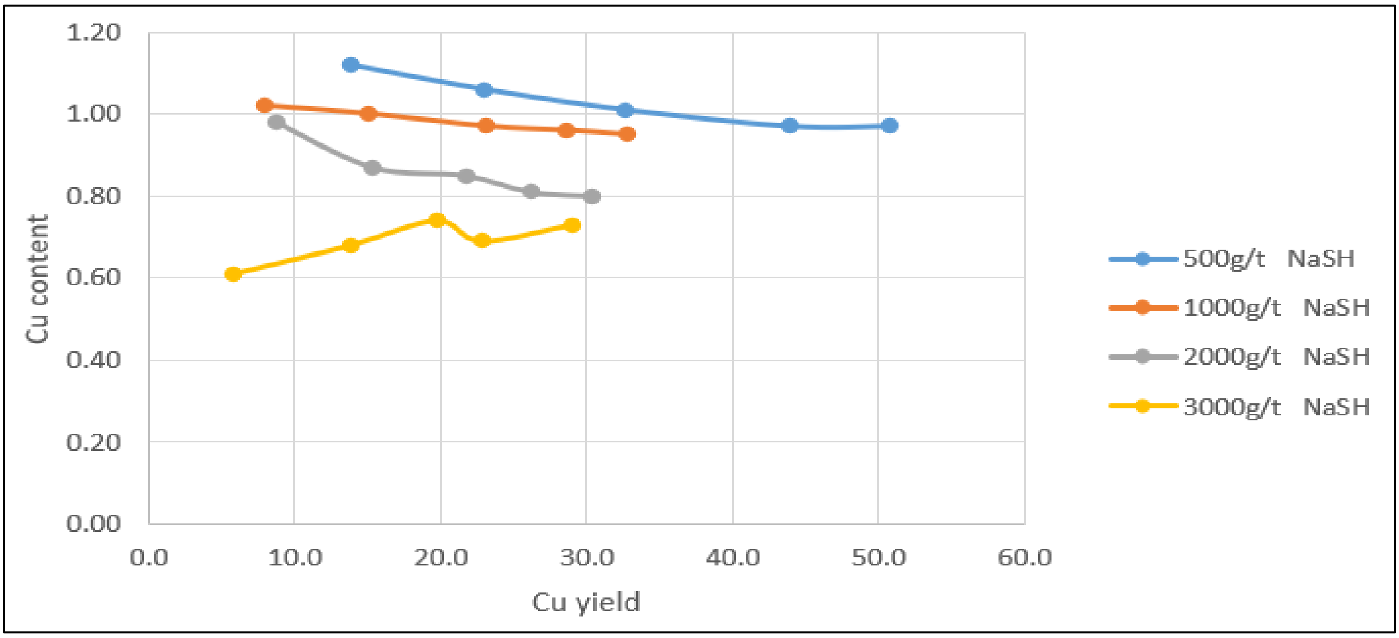

Fig 4 Copper Flotation at Different Doses of NaHS and KAX

- *Comparative Analysis of Tests Carried Out with and Without Pre-Flotation*

A comparative analysis of the series of tests carried out without pre-flotation and those carried out with pre-flotation reveals that the pre-flotation operation has a negative influence on the flotation kinetics of these tailings.

We therefore suggest flotation without pre-flotation for the concentration of this type of tailings, considering doses of 200 g/t of mixture, 100 g/t of silicate, sodium sulphhydrate and potassium amylxanthate 500 g/t and 50 g/t.

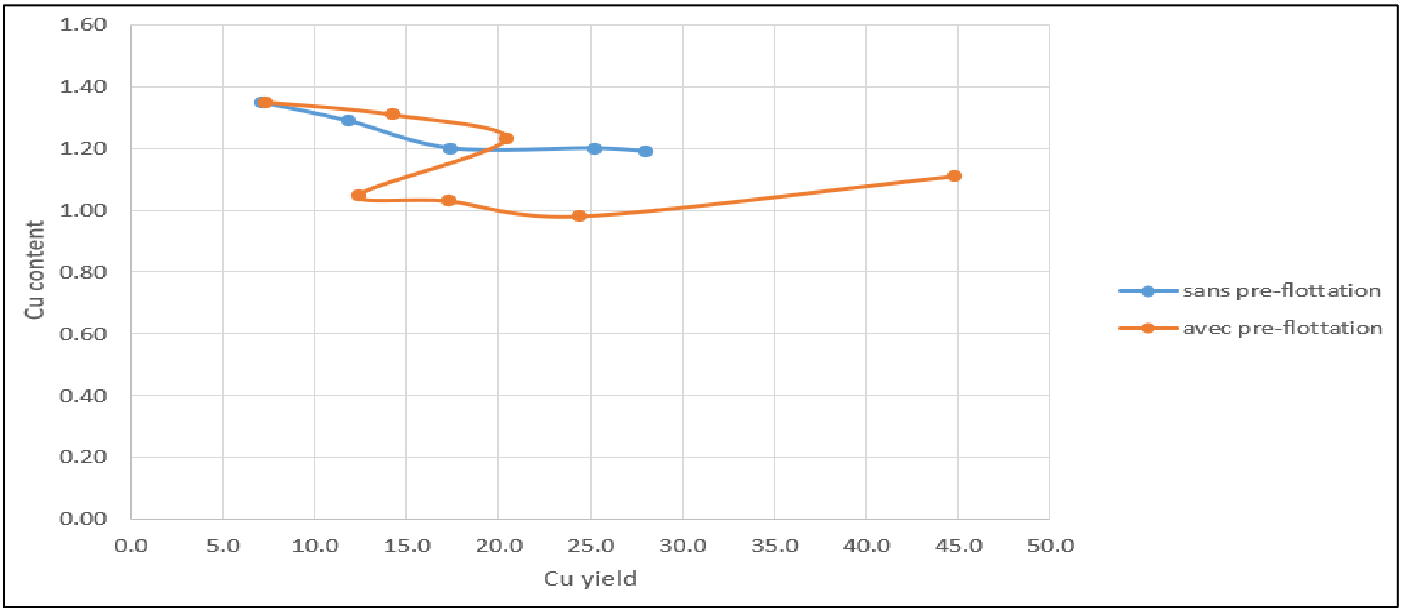

Fig 5 Synthèse De Meilleurs Résultats Des Deux Flottations

IV. CONCLUSION

Our goal was to recover the metals contained in the tailings from the Kipushi 3 dam by increasing their copper and cobalt content to better feed them into the leaching process.

The study was carried out on a sample from Kipushi Dam 3 with an initial grade of 0.94% copper and 0.39% cobalt.

- *Therefore, with the Aim of Recovering at Least 3–4% Copper and 1% Cobalt Using Standard Laboratory Reagents, Namely:*

- Sodium silicate ;
- A mixture of Rinkalore 10 and diesel fuel;
- Potassium amyl xanthate ;
- Sodium sulphhydrate.

ISSN No: -2456-2165 https://doi.org/10.38124/ijisrt/25oct823

After characterising the sample and knowing that we were dealing with mixed minerals, a series of four tests was carried out with pre-flotation in order to better guide us.

This guidance showed us that the reject was oxidised, with a grade of 0.94%, which led us to move directly to flotation itself.

➢ *We Began Our Evaluation by Studying the Influence of the Doses of Our Various Reagents, Varying Them as Follows:*

- NaHS from 500g/t to 3000g/t;
- KAX from 50 g/t to 300 g/t;
- Rinkalore 10 mixture and diesel oil from 100g/t to 400g/t;
- sodium silicate from 100g/t to 500g/t.

➢ *The Best Doses Obtained are as Follows:*

- NaHS and KAX: 500g/t and 50g/t, with a rough content of 1.12%;
- mixture: 200g/t, with a rough grade of 1.27%;
- silicate: 100g/t, with a rough content of 1.17%.

Therefore, analysing the results obtained, we can see that we were unable to achieve the target set at departure, but we have achieved an acceptable level in the leaching standard in order to be supplied.

➢ *The Initial Goal of Our Work was not Achieved Due to Several Constraints, Including:*

- The nature of the gangue;
- The reagents used;
- The working conditions. In addition, we suggest that they:
- Testing the use of new reagents, such as foaming agents;
- Testing production in a closed circuit in order to obtain a minimum concentrate of ±3%.

ACKNOWLEDGEMENTS

We would like to thank the Gécamines Metallurgical Studies Laboratory (EMT) for their support in carrying out this work.

We would also like to thank engineer Bhawa Kushi Christian for his collaboration in the development of this work.

REFERENCES

[1]. Blazy, P., & El-Aid, J. (2000a). Flottation : Aspects pratiques. Techniques d'ingénieur, Procédés Chimie Agro Bio, Procédés unitaires, j3360.
[2]. Blazy, P., & El-Aid, J. (2000b). Flottation : Mécanismes et réactifs. Techniques d'ingénieur, Procédés Chimie Agro Bio, Procédés unitaires, j3350.
[3]. Kitobo Samsoni, W. (2009). *Dépollution et valorisation des rejets miniers sulfurés du Katanga : Cas des tailings de l'Ancien Concentrateur de Kipushi* (Thèse de doctorat, Université de Liège, Belgique).
[4]. Kongolo, M. (1991). *Interaction de l'amaylxanthate de potassium avec la galène et la pyrite finement broyées : Conséquences sur la flottation* (Thèse de doctorat, Institut National Polytechnique de Lorraine, Vandœuvre-lès-Nancy, France).
[5]. Wills, B. A., & Napier-Munn, T. (2006). *Mineral processing technology: An introduction to the practical aspects of ore treatment and mineral recovery* (7e éd.). Elsevier Science & Technology Books.
[6]. Abette, & Khitine. (1970). *Manuel de minéralurgie descriptive*. Éditions Mir, Moscou.
[7]. Corneille, E. K., & Masson, A. (1973). *Cours de minéralurgie et préparation des minerais*. Éditions Derroux, Liège.
[8]. Blazy, P. (1970). *La valorisation des minerais*. Presse Universitaire de France, Paris.
[9]. Rickard, T. A. (2001). *Concentration by flotation*. New York.
[10]. Nouioua, A. (2018). *Lixiviation et extraction du fer à partir de la roche ferrique de la mine de l'Ouenza*.
[11]. Mbweb, P. (2017). *Étude de la lixiviation en tas du minerai cupro-cobaltifère dans les mines de la province du Katanga en RDC.*