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Effect of Desliming on Flotation Response of Kansanshi Mixed Copper Ore

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Abstract

The Kansanshi mixed copper sulfide-oxide ore contains significant proportions of fine material manifesting in a variety of processing challenges. This paper presents the work which was carried out to evaluate the efficacy of desliming in improving the flotation response of the ore. Two modes of desliming were investigated, namely; sieving and elutriation after which the deslimed material was subjected to Kansanshi standard laboratory flotation conditions. The minimum copper feed grade for the mixed copper ore was 0.5% Total Copper (TCu). The outcome of this work has shown that desliming improves the flotation response of the Kansanshi mixed copper ore. At a rougher copper concentrate grade of 8%, copper metal recoveries obtained with desliming were in excess of 70% compared to 58% achieved with baseline tests without desliming. It was further observed that desliming resulted in improved flotation rates for the sulfide minerals. For a flotation time of 3 minutes, recoveries of 69% and 74% were obtained with elutriation and sieving respectively compared to 58% recovery for baseline tests. From the same results it was also evident that, of the two modes of desliming investigated, sieving yielded better performance than elutriation.

Keywords

Kansanshi Sulfide Oxide Copper Ore, Flotation, Baseline, Desliming

1. Introduction

Flotation recovery of valuable minerals from ores is a function of particle size and it is therefore anticipated that variation in the granulometric characteristics of feed to a flotation process will affect flotation performance. Size of mineral particles plays a significant role in the sequence of events that leads to the flotation of a particle [1]. Further, recovery of metal bearing minerals and selectivity among the minerals is not only a function of mineral particle size but also particle size distribution in flotation feed material [2]. The fastest flotation rate and maximal metal recovery are associated with middle particle size region, while out of this region, the results are usually diminished. Also, an upper and lower particle size limit exists beyond which there is no flotation. For sulfide minerals of non-ferrous metals these limits were found to be different for each species [3].

Because of the extremely complicated physico-chemical-mechanical conditions existing in the flotation process, the problems associated with the presence of fine particles are most pronounced in flotation. Coarse and fine particles are often not recovered during the flotation processes or are poorly recovered. There is a general agreement that flotation decreases with a decrease in particle size in the fine particle range. However, it should be noted that in mineral processing, ores are ground to liberate minerals and not produce fine powder. As the ore is diminished from coarse to fine, minerals alter their status from being bonded by matrix to individual free particles and separated into concentrate and tailing based on their dissimilar physico-chemical properties. It is therefore, the degree of liberation and not the particle size distribution that is an important property of the product from the milling circuit. The efficiency of flotation process is much less for particles in the range of 0 to 10 microns. Authors [4] [5] observed a decrease in pyrite recovery and a lower flotation rate constant for fine particles. They suggested that the optimal size range for pyrite flotation in environmental desulphurization is between 35 microns (μm) and 120 μm. This is consistent with earlier observations on pyrite flotation reported by the authors [6] who defined the size range for maximum pyrite recovery to lie between 50 μm and 150 μm.

Increases in liberation are achieved by transferring material by means of size reduction from the coarse size ranges, where liberation is low, to finer size ranges where liberation is higher and mineral particles are most susceptible to flotation. The current flotation practice works best on average or medium size feed particles. The greater the amount of large or small particles, or of both large and small, the more difficult it is to achieve excellent flotation results [7]. In order to achieve acceptable feed particle size distributions, size reduction, in most milling operations, is carried out stage-wise to minimize both overgrinding and undergrinding. Narrow size distributions will usually result in placing more material in the intermediate fast-floating size classes, consequently increasing recovery of values. For the production of most of the valuable product, the grinding circuit objective for a simple single floatable mineral case is to produce as narrow a size distribution as possible in order to squeeze the maximum amount of the mineral value into the highest recovery size range. Communition techniques that result in closed sized distribution usually have a low proportion of ultra fine material in their communition products.

Fine particles in mineral processing are generated in at least two ways; accumulation in most of the ore bodies due to weathering and decomposition of certain rock components. These are referred to as primary slimes. Secondary slimes are subsequently generated at the mines due to the mechanization and automation in the mining processes. A further generation of fine particles is encountered during communition of an ore to its liberation size [8]. The accumulation of large volumes of slimes is attributed to increased tonnage of treating low grade ores even for mineralogically less complex ores. However, this is further aggravated by the occurrence of valuable minerals in a finely disseminated form which increases the need for fine grinding to attain acceptable degree of liberation of values and a need for techniques for beneficiation of fines.

As the high grades ores are exhaustible, the dependence on low grade ores to meet the growing global demand in the mineral industry is linked up with the more serious problem of the decrease in liberation and the direct consequence of this is more generation of fines [8]. As the quantity of the secondary slime is dependent on the liberation size and the natural breakage characteristics of ores and to some extent on the communition process used, attempts are possible to minimize the excess production of fines. Operators carefully avoid "overgrinding" and "sliming" of feed [9]. While nothing much can be done about the presence of primary slimes, proper control of communition circuits especially effective grinding circuit control may be effective in reducing the generation of secondary slimes.

1.1. Effects of Slimes on Flotation

The presence of slimes in flotation systems results in deleterious effects on recovery, selectivity and reagent consumption [10]. The problematic nature imposed by slimes on flotation is attributable to their 1) small mass, 2) high surface area and 3) high surface energy [8]. These are due to two characteristics which begin to dominate as the particle size is reduced namely that the specific surface becomes large and the mass of the particle becomes very small.

Firstly, small mass leads to; 1) low particle momentum, 2) particle entrainment in concentrates (e.g. froth), 3) low probability of collision with a bubble and 4) difficulty in overcoming the energy barrier between particle and particle, and particle and bubble barrier. Because of the small mass and momentum of fine particles, they may be carried into the froth after getting either entrained in the liquid or mechanically entrapped with particles being floated. Some authors [9] have shown that fine particles suspended in the water are transferred to the froth phase in the wake of ascending air bubble. When such particles are of gangue minerals, the deportment of gangue phases to the concentrate results in concentrate dilution.

Secondly, high Surface area leads to; 1) a high dissolution rate in water, 2) adsorption of large quantities of chemicals, 3) rigidity of froth, 4) high pulp viscosity and 5) undesirable coating of the valuable particles by ultrafine gangue particles. The large specific surface area of fine particles increases the adsorption capacity for reagents when considered on a mass basis. Thus, a significant proportion of the reagent may not be available for the flotation of larger particles with a resultant decrease in recovery of values [11]. The undesirable coating of the valuable particles by ultrafine gangue particles can significantly change the flotation behavior of the valuable mineral particle. Intuitively, the slimes coated on value mineral surface form a hydrophilic "armor" preventing the value mineral from direct contact with collectors and/or air bubbles, lowering flotation recovery [12]. Author [13] showed that slime coatings of alumina on galena reduced the latter's floatability. Apart from these particle-particle interaction effects, slimes are in themselves notoriously difficult to separate by flotation, a consequence of their very fine size. Authors [14] have shown that fine particles (−10 μm) float at a slower rate than intermediate (−50 + 10 μm) particles at the same surface coverage of the hydrophobic collecting species. They suggested that this was because the kinetic process of thin film rupture and drainage, occurring during bubble-particle attachment was longer for fine particles. The investigations by authors [14] demonstrated that the critical surface coverage required for flotation of fine particles, is more than twice that required to induce flotation of the intermediate size particles. If the slimes were of a valuable mineral, it would open up for a possibility of separate treatment of fines [10].

Thirdly, high surface energy per unit area due to imperfect crystallization, increases cracks, dislocations, edges and other high energy sites can lead to the following difficulties 1) nonspecific adsorption of reagents, 2) increased hydration, 3) rapid surface reaction and 4) increased solubility [8]. Of particular importance is the effect of slimes on sulfide flotation, predominantly the surface oxidation of newly ground particles, which occurs during mining, ore handling, crushing and storage. Some fine particles are inevitably produced prior to grinding and, since substantial surface oxidation is anticipated, selectivity during flotation is expected to be affected. Authors [15] have shown that surface oxidation adversely affects the selectivity of chalcopyrite/galena separations due to the combined effect of accidental heavy metal ion activation and hydrophilic surface coatings generated from metal ion hydrolysis products. The presence of oxidation of the slimes produced prior to grinding, should affect the flotation selectivity of the particles produced during grinding [10].

1.2. Treatment Options of Slimes in Flotation

Treatment options of ore materials with significant proportion of slimes to improve flotation recovery of values can be classified into two categories;

1) Processes which are based on more favorable change in the energetic of bubble-particle contact.

2) Processes which are based primarily on increasing the probability of collision between air bubbles and mineral particles.

The first category consists of methods where flotation is depended on collision and adhesion. For fine particles this is achieved when fine bubbles are generated. This accounts for successes for fine particle treatment with such devices as flotation columns and dissolved air flotation cells.

The second category mainly consists of those methods which tend to increase particle size. The probability of collision between air bubbles and mineral particles increases when latter is presented as agglomerates (floc flotation) or is attached to larger (hydrophobic) particles which act as carrier particles (carrier flotation). If flotation of fines is poor because the particle size is too small, then "aggregation" of the fine particles into larger particles, in one way or the other, would be the solution. To increase the probability of collision between air bubbles and mineral particles, pre-segregation of fines into bigger particles has great potential in the beneficiation of fines.

The aggregation of fine particles into bigger particles can proceed either by flocculation or coagulation. Coagulation occurs when repulsive force between the particles is reduced either by charge neutralization or reduction of the electrical double layer [16]. Flocculation on the other hand involves the use of polymers. A polymer is a long chain adsorbate with several active sites on it and induces aggregation by attaching itself to two or more particles consequently providing bridging between particles.

In summary, treatment options for slimes are achieved by either aggregation of fine particles or fine bubble generation techniques.

1.3. Methods of Aggregation of Fine Particles

There at least two methods commonly used for aggregation of fines into bigger particles in mineral beneficiation. These are selective polymer flocculation and hydrophobic aggregation.

Selective Polymer Flocculation

A polymer induces aggregation by attaching itself to two or more particles consequently providing bridging between particles. It is, in particular, a preferential adsorption of a polymer either onto the valuable or inert/impure mineral [17]. Hence the success of the selective flocculation process depends on the identification of proper polymer flocculants that are selective with respect to one of the components to be separated. Selective adsorption of polymer molecules can be achieved by adjusting the chemical composition of the suspending medium and thereby the surface potential of the mineral or by introducing active functional groups into the polymer, which will form complexes with the surface species [17]. Selective flocculation can also be achieved if there are differences in the rate of polymer adsorption on various components.

Separation of the flocculated particles from the others can then be achieved by using conventional techniques such as flotation, elutriation or sedimentation with care to produce minimum re-dispersion of flocs during such processes. In the case of flotation, it is also necessary to take into account the interactions of flocculants with flotation agents [17]. However, a major problem with selective polymer flocculation is that most of the currently available long chain polymers are bulk flocculants and lack the desired specificity. For most ores, selective flocculation is not easily achieved even under conditions when excellent selectivity is expected [17].

Hydrophobic Aggregation

Hydrophobic aggregation proceeds in at least two consecutive stages namely; selective hydrophobization and hydrophobic bonding. Selective hydrophobization is achieved by adding special collectors to selectively render particular mineral particles hydrophobic upon their adsorption. Particle hydrophobicity can be enhanced by addition of non-polar oil due to its spreading on hydrophobic surfaces. Meanwhile, hydrophobic bonding is achieved by addition of energy to the pulp mixture to overcome the potential energy barrier between the particles resulting into hydrophobic bonding [4] [18] [19]. The energy addition is achieved by some form of high intensity agitation. Hydrophobic bonding can be achieved by shear flocculation, carrier flotation or oil agglomeration to mention but a few of such methods. The size and density of hydrophobic flocs depend on particle hydrophobicity and non-polar oil addition. Hydrophobic bonding methods include the following:

Shear Flocculation

Shear Flocculation is usually performed in a mechanical mixing tank, through which kinetic energy is provided mechanically to hydrophobic particles to collide with each other and surmount energy barrier between them due to electric double layer repulsion and water films. In this stage, hydrophobic flocs form as a result of hydrophobic interaction between particles and kinetic energy input. Hydrophobic bonding takes place between two hydrophobic minerals. Substantial activation energy is therefore required to bring the hydrophobic mineral particles sufficiently close together for it to be invoked. This activation energy is supplied by shear forces in a fluid medium at a convenient stirring regime. Therefore, for particles to aggregate when they are dispersed, the corresponding potential energy barrier must be overcome by applying extra energy to the particles, usually by intense stirring [4] [18] [20].

Carrier Flotation of Fine Particles

Carrier flotation can be achieved via the enhanced aggregation between fine (carried) and coarse (carrier) particles under intense agitation and followed by froth flotation. Fine particles to be floated from slime coatings on the auxiliary or carrier material and the coated particles are then floated together. The general conditions necessary for successful removal of these colloidal particles include coagulation of the particles onto the surface of the large size particles (carrier mineral), finding a suitable collector, frother, and floating the agglomerates in a flotation cell [11].

Oil Agglomeration

Oil agglomeration, also called spherical agglomeration, is an extension of shear flocculation. It already has practical use in fine coal recovery but recent laboratory work was performed on other minerals [21] [22] [23]. The process consists of the formation of agglomerate by mixing hydrophobic solids with immiscible oil [21] [22] [23]. The requirements are hydrophobic particles, immiscible bridging oil and high intensity agitation. Oil consumption is high historically, in the range of 10% by weight of solids, recent work is in the 0.5% to 2% range [22]. Due to its natural hydrophobicity, coal is the choice material for oil agglomeration, but other minerals can become suitable candidates provided they are rendered hydrophobic by surfactant addition [22].

1.4. Fine Bubble Generation

It is generally accepted that the main reason for the low flotation response of fine particles is the decrease in the probability of collision between particles and bubbles as the particle size is decreased [24] [25] [26]. To improve the flotation recovery of fine particles, the use of small bubbles is warranted since it will increase the probability of collision between the particles and the bubbles [24] [25]. Several methods exist to generate bubbles in a flotation machine. Some of these methods that can produce small bubbles are dispersed air flotation, dissolved air and induced air flotation, hydrodynamic cavitation and electro-flotation.

The poor recovery in fine particle sizes with conventional mechanical flotation cells is attributed to the turbulent conditions inside the cell [24]. Collision between bubbles and particles are relatively random [27]. Flotation columns are a type of machine that evolved from these limitations. A conventional column cell has a deep froth zone and wash water sprays that allow achieving a high concentrate grade by reducing entrainment [28] [29]. The feed is added in the top third of the column and the particles settle among rising bubbles in a quiescent state, providing good conditions for increased fine particle—bubble collision probability [27] [28] [29]. Fine particles benefit specifically from this countercurrent interaction since their settling rate is lower and their residence time in the column is higher, which enhances the probability of particle—bubble collision [27] [28]. Many plants replaced mechanical cells by columns in the cleaning stage and obtained higher recovery of fine particles [30]. The bubble sizes are usually smaller in flotation columns and it is a function of the gas rate [28] [31] and the type of sparger used [25].

1.5. Desliming

The adverse effect of slimes on flotation of values has already been alluded to. The options for treatment of fines discussed in the preceding sections are partly based on the assumption that for a given particle size distribution both the fine and coarse particles contain values from which metal values need to be recovered. However, if the slime material is essentially gangue material, elimination by desliming would be the most plausible option. The slime coatings on coarse particles which interfere with bubble-mineral contact and collector adsorption on the valuable coarse particle are eliminated resulting in improved flotation response of the valuable coarse particles. Since the slimes exhibit great detrimental effects on flotation efficiency, it is desirable to remove the slimes before flotation [32]. Removing fine particles from coal using a hydrocyclone significantly improved the coal flotation performance [33] [34]. Desliming using a hydrocyclone also gave improved flotation performance of pentlandite [35].

However, if the slime fraction does contain high concentration of value minerals, separate flotation circuits for slimes and coarse fractions should be considered. The authors [36] treated fluorite ores by separate coarse and fine flotation at a cut size of 15 μm in laboratory tests, and the results showed that the overall recovery of fluorite increased by 18 percentage points. These separate flotation circuits have been used in Mt Keith nickel plant in Western Australia [37].

The work described in this report focuses on investigating the effect of desliming on the flotation response of Kansanshi Ore.

2. Materials and Methods

2.1. Sample Collection and Preparation

The Kansanshi mixed copper ore feed material was sampled by cutting from the cyclone overflow of the grinding circuit at 30 minutes interval for a duration of 3 hours for each day for 3 days. The dried composite sample of 60 kg was homogenized using a cube mixer and split into 1kg lots for test works.

2.2. Feed Characterization

Particle size characterization of the feed sample was carried out by sieving to fractionate the sample using a nest of sieves. Chemical analysis for total copper (TCu) and Acid Soluble Copper (ASCu) was carried out on each size fraction generated. Mineralogical data was obtained to compliment metal assay results.

2.3. Standard Flotation Test (Baseline Tests)

The standard flotation test procedure for Kansanshi mixed copper ore is shown in **Figure 1**. A labtech Essa flotation machine with 2.5 liters cell was used for 1kg sample feed at a pulp density of 32% solids. The first stage involved the flotation of sulfide minerals (analyzed as Acid Insoluble Copper-AICu), 50 g/t of Sodium Isobutyl Xanthate (SIBX) Collector and 20 g/t Aero froth 68 frother were added and conditioned for 2 minutes. Froth collection began immediately after 3 minutes conditioning time elapsed and was carried out for 3 minutes. The rate of scrapping concentrate from the cell was done every 15 seconds and collected on trays.

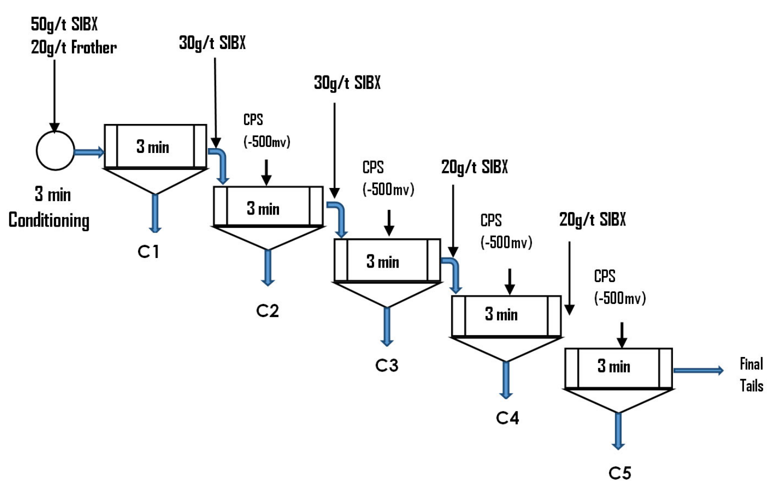

Figure 1. Standard laboratory flowsheet for Kansanshi ore.

The second stage involved flotation of oxide minerals (analyzed as Acid Soluble Copper, ASCu). Activation of oxide minerals prior to flotation with xanthate collector was achieved by sulfidization of the oxide minerals using Sodium Hydrogen Sulphide (NaHS) as a sulfidizing agent. The sulfidizing agent was administered using Controlled Potential Sulfidization (CPS) technique. The process uses the sulfide ion specific electrode to control the addition rate of the sulfidizing reagent and maintain a constant pulp potential (Es value) during condition. In this test work, the probe meter was immersed in the pulp and the addition of the sulfurdizing agent, Sodium Hydrogen Sulfide (NaHS), was done for a duration of 2 minutes while maintaining −500 mV (Es value) on the probe meter. Collector and frother were added for each stage and conditioned for another 3 minutes, scavenger concentrates were collected for 3 minutes each. The concentrates were filtered using a vacuum filter. The samples were analyzed for total copper (%TCu), acid insoluble copper (%AICu), acid soluble copper (%ASCu) and Iron (%Fe). The flotation tails were also filtered and dried in a moderate oven and their weights noted. The concentrates and flotation tails were also analyzed for %TCu, %AICu, %ASCu and %Fe.

2.4. Desliming Tests

Desliming prior to flotation tests was carried out on separate tests by screening using a sieve and by elutriation. The deslimed fraction from each test was subjected to the standard flotation test.

Screening Tests

2 kg of the sample was screened on 38 micron (μm) sieve to generate 1 kg of dry +38 μm sample for flotation. Both the +38 μm and −38 μm samples from the screening test were analyzed for %TCu, %AICu, %ASCu and %Fe.

Elutriation Tests

A 2 kg test sample was placed in a bucket containing an opening at the bottom through which water was injected at a predetermined rate. An agitator running at 450 rpm was used to keep the solids in suspension.

The upward flow of water which was allowed to run for an hour, transported a portion of the solids into the overflow (overflow fraction). The flow of water was shut off and the remaining material (underflow fraction) settled to the bottom of the bucket. Elutriation tests were carried out at different levels of water flowrate, namely 12 ml/s, 16.45 ml/s, 24.27 ml/s, and 45.45 ml/s on each portion of 2 kg test sample. For each water flowrate, the two fractions, overflow and underflow, were dried and separated into size fraction by screening on a sieve shaker.

Flotation tests were thereafter carried out on deslimed material generated from the screening and elutriation tests.

3. Results and Discussion

3.1. Feed Characterization

The particle size distribution of the float to feed material, effect of varying water flowrate on elutriation and variation of metal content with particle size are presented in Figures 2-4 respectively.

Particle Size Distribution

The 80% passing size (P80) for Kansanshi Ore is 150 µm. The test sample was slightly coarser at 74% passing 150 µm and about 40% of the material in the sample was less than 38 µm.

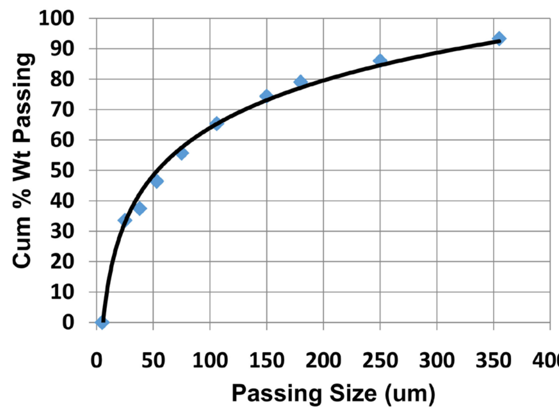

Figure 2. Particle size distribution of float to feed material.

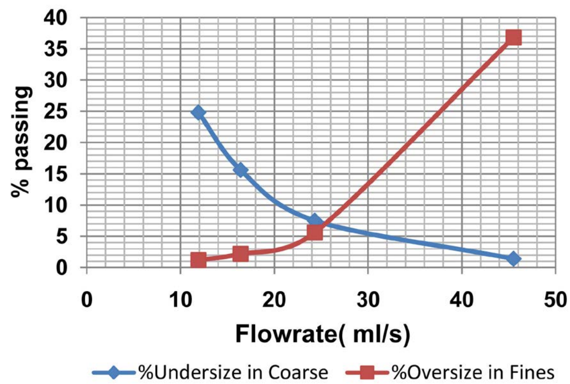

Figure 3. Effect of varying water flow rate on elutriation.

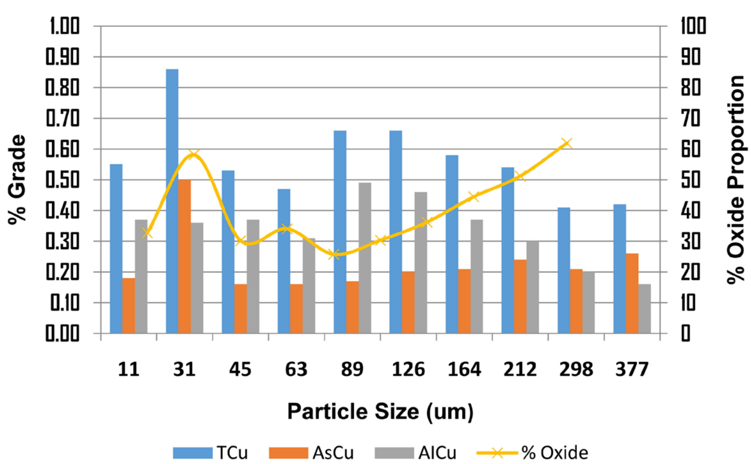

Figure 4. Variation of metal content with particle size in float to feed material.

Effect of Water Flow Rate on Elutriation

Figure 3 shows the optimum flow rate giving the best separation between the oversize and undersize to be 25 ml/s. This was equivalent to filling up the 500 ml beaker in 20 seconds in order to get a flow rate of 25 ml/s.

Variation of Metal Content with Particle Size

The relationship between metal grade and particle size is shown in Figure 4. The overall grade of the feed is about 0.5% TCu. The grades for the size fractions vary from 0.40% to 0.86% in TCu, with the highest grade obtaining in the finer size fractions (minus 38 μm). The proportion of oxide values in each size fraction is relatively higher in the fine size and the coarser size ranges.

The increased proportion of oxide values in the coarser size ranges is indicative of either significant locking of the oxide mineral phases to gangue or that the oxide mineral phases are coarse grained.

The metal distribution per size fraction is shown in Figure 5. Close to 35% of the values are in the minus 11 μm size range, whilst the rest of the copper values are fairly distributed between 31 μm and 380 μm and account for 65% of the metal distribution. It is therefore envisaged that a desliming process with a cut point at approximately 40 μm particle size, will deport about 30% of the values to the slimes fraction with an average grade of 0.60% TCu. This would entail a separate treatment of the slimes fraction to recover the values. For high throughput plants this would be an important treatment option.

3.2. Flotation Test Results

The grade-recovery and flotation kinetics profiles are presented in the Figures 6-11 for standard flotation test and desliming by sieving and elutriation.

3.2.1. Standard Flotation Test (Baseline Tests)

The results of standard flotation tests without desliming (baseline tests) are graphically depicted in Figure 6 (grade-recovery profiles) and Figure 7 (flotation kinetics). The acid soluble copper (ASCu) and acid insoluble copper (AICu) assays represent analytical estimates for copper oxide and sulfide minerals respectively.

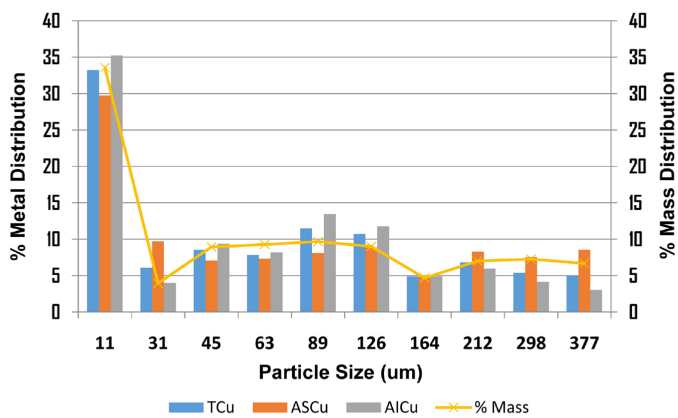

Figure 5. Variation of metal distribution with particle size in float to feed material.

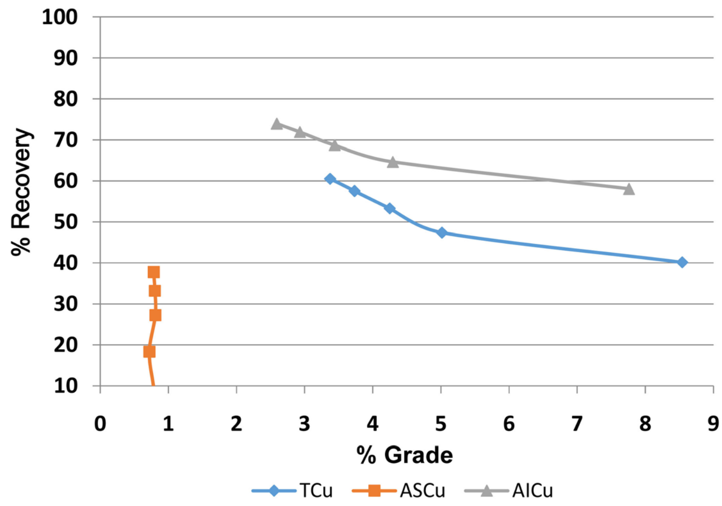

Figure 6. Grade-recovery profile for baseline flotation tests (without desliming).

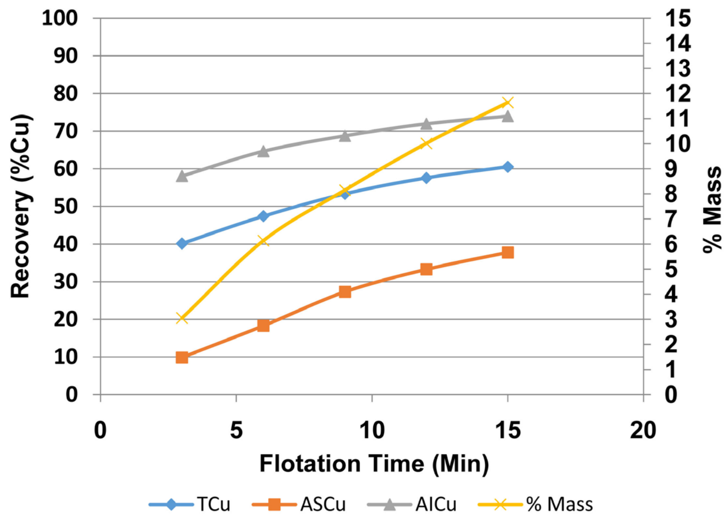

Figure 7. Flotation kinetics for baseline flotation tests (without desliming).

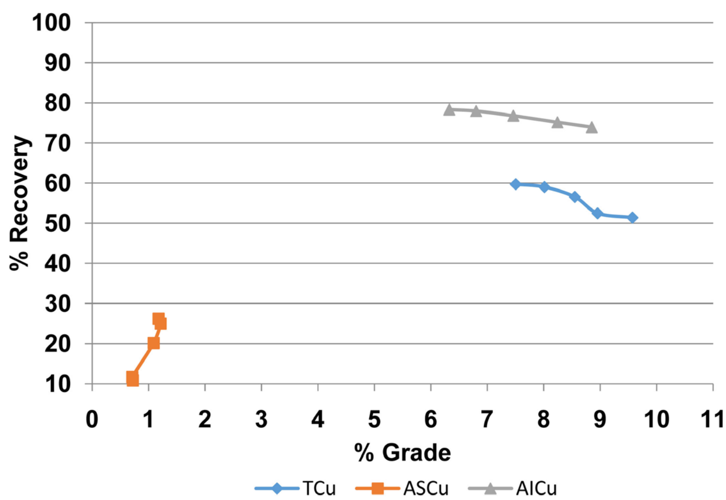

Figure 8. Grade-recovery profile for flotation with desliming by sieving.

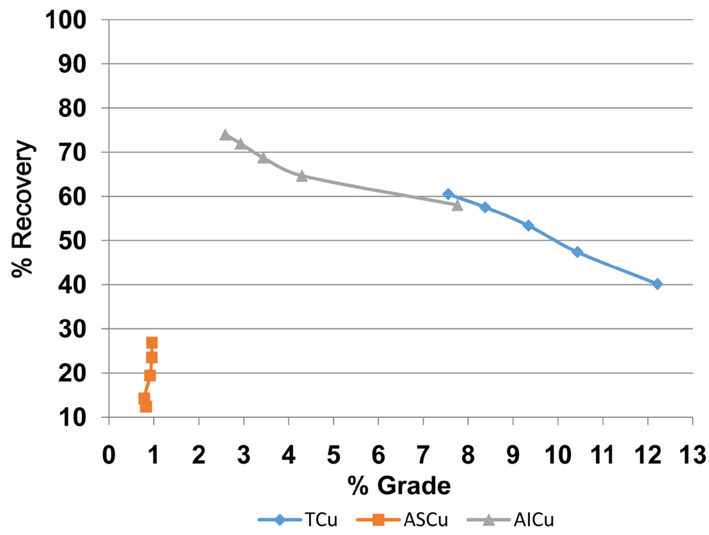

Figure 9. Grade-recovery profile for flotation with desliming by elutriation.

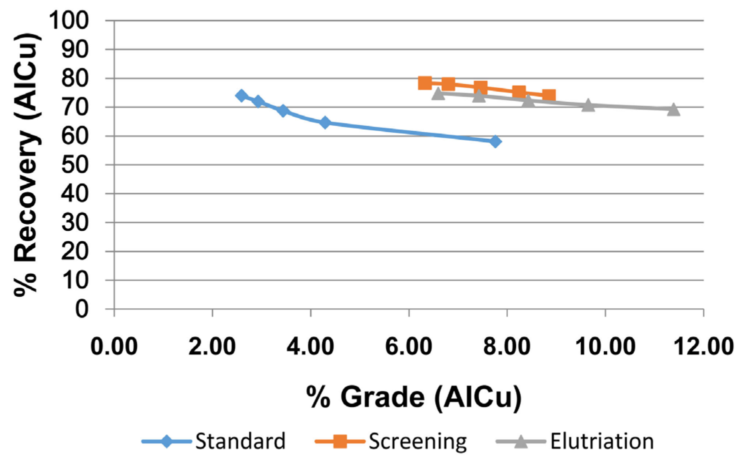

Figure 10. Grade-recovery profiles comparing the standard with desliming by screening and elutriation.

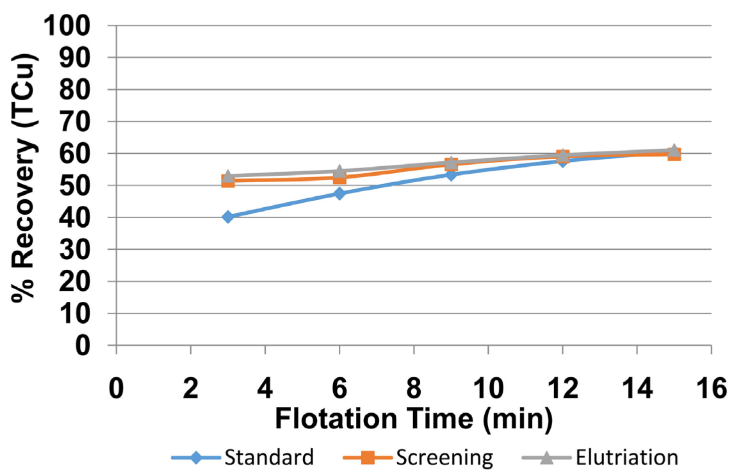

Figure 11. Flotation kinetics comparing the standard with desliming by sieving (screening) and elutriation.

The flotation response of sulfide minerals in the Kansanshi mixed oxide-sulfide copper ore is superior compared to oxide minerals, for the given flotation conditions. The grade-recovery profiles and the flotation kinetics show improved results for copper sulfide minerals than oxide minerals. Figure 6 shows that oxide values were not enriched during flotation despite the use Controlled Potential Sulfidization (CPS) and exhibited poor flotation kinetics (Figure 7). At 10 minutes of flotation time, oxide recoveries were approximately 30% compared to 70% for sulfide minerals. Comparing recovery profile for oxide values (ASCu) with mass recoveries to the concentrates (Figure 7), particle entrainment should be responsible for deportment of oxide values to the concentrate and not by true flotation.

3.2.2. Flotation Tests on Deslimed Material

The flotation test results on the effect of desliming using sieving and elutriation is depicted in Figure 8 and Figure 9 respectively. Observations on the flotation behavior of oxide values compared to the sulfide values are similar to that obtained under standard flotation tests. The grade-recovery profiles and flotation kinetics are then compared with the standard flotation tests as shown in Figure 10 and Figure 11 respectively. The indication is that the effect of desliming either by sieving or elutriation has a significant benefit on the flotation response of sulfide minerals than oxide minerals. The poor flotation behavior of oxide minerals in the present investigation cannot be attributed to or without the presence of fines but rather the prevailing chemical environment in the pulp and specifically the efficacy of sulfidization on oxide minerals.

The results also indicate that desliming by sieving yields better results than elutriation. At a rougher concentrate grade of 8%, the corresponding recovery values yielded 58% for the baseline test and 73% and 77% for elutriation and screening tests respectively (Figure 10). Desliming also improved the rate of flotation. For a flotation time of 3 minutes, recoveries of 69% and 74% were obtained with elutriation and screening respectively compared to 58% recovery for baseline tests. It is therefore evident that desliming improves the flotation response of the Kansanshi mixed copper ore. The difference in results between elutriation and screening can be attributed to the way each desliming method fractionates a given particulate material. Sieving separates particles strictly according to particle size. A single test sieve separates a particulate material into two fractions of which one is retained by the sieving medium and the other passes through its apertures. The particle size for such material can be expressed in terms of the characteristic dimension of the material that can best represent the state of the subdivision or its constituting particles which in this case is the sieve diameter.

In the process of elutriation, particles falling in a rising fluid can be classified into size fractions. When the fluid in a sorting column is rising with a certain velocity, the particles having terminal velocities higher than this velocity, settle at the bottom of the sorting column. Meanwhile, the particles with lower terminal velocities are lifted to the top of the sorting column and carried away to the overflow constituting the undersize fraction while the settled particles make up the oversize fraction. Terminal velocities of the particles falling in a fluid under laminar conditions can be expressed by Stokes' law. The settling behavior of a particle is thus a function of three physical characteristics namely, particle size, particle density and particle shape. Under such conditions, it is possible that a small particle with a higher specific gravity may end up in the same fraction with a bigger but light particle which is not possible with sieving. Consequently, the oversize product of sieving will be more of a close size particle size distribution compared to one generated by elutriation. When such materials are subjected to flotation, they will have a relatively higher adverse effect on flotation compared to the oversize product from sieving.

The results presented in this study show that generally, desliming improved the flotation response of Kansanshi copper ore and in particular the flotation results on sieved material has shown better performance than results obtained with elutriated material.

In terms of industrial application, the implementation of fine screening as part of the grinding circuit or any other section in a concentrator is widely practiced. However, devices that are more suitable than screens for a full scale high throughput operating plant like Kansanshi Mining, are hydrodynamic based devices such as classifiers and in particular the application of hydrocyclones in milling circuits. The desliming technique described in this work is generally referred to as conventional desliming and the focus is on mitigating the two-fold detrimental effect of slimes on flotation, namely; 1) high reagent consumption because of specific surface of particles and 2) slime coatings on granular particles which interfere with bubble-mineral contact. In conventional desliming there is no consideration for the distribution of metal values in the various particle size ranges and in cases where the deslimed undersize fraction contains significant proportions of metal values is discarded, it would result in loss of metal values. In this work, close to 35% of the values are in the minus 11 micron (μm) size range, while the rest of the copper values are fairly distributed between 31 μm and 380 μm and account for 65% of the metal distribution. This entails that conventional desliming with a cut point at approximately 40 μm particle size will deport about 30% of the values to the slimes fraction with an average grade of 0.06% TCu. For high throughput plants like Kansanshi Mine treating an average 5000 tonnes per hour of ore, discard of the slimes fraction of this nature would translate into significant loss of values.

In view of the above, the efficacy of the desliming process will depend on the dominant effect of slimes on flotation, whether this be high reagent consumption by slime material in which case a significant proportion of the reagent may not be available for the flotation of mineral particles with a resultant decrease in recovery or slimes coating value mineral surface preventing the value mineral from direct contact with collectors and/or air bubbles, thus lowering flotation recovery.

In the work carried out by authors [38] to improve Iron ore beneficiation using flotation demonstrated that in cationic reverse flotation of siliceous gangue at near natural pH, clay slimes are the reagent consuming species, but if enough amine collector is added so that the solution concentration is maintained, the clay slimes do not interfere with flotation. Conversely, goethite slimes do not consume the amine collector, but they seriously interfere with the flotation of quartz particles because goethite slimes coat the particles and prevent attachment of air bubbles to quartz surfaces.

It follows therefore that where the detrimental effect of slimes on flotation is predominantly reagent consumption, desliming may result in unnecessary loss of values to the slimes fraction. Replenishing collector concentration by increasing collector dosage would suffice. To prevent loss of values to the slimes fraction where the predominant role of slimes is interference with the flotation of values, the flotation stage may be preceded by selective desliming or split conditioning.

Selective desliming consist of a selective flocculation stage where preferential adsorption of the flocculant either onto the valuable or inert/impure mineral takes place. The resulting flocculated particles are separated from other phases using flotation. As already pointed out, the success of the selective flocculation depends on the availability of flocculants that are selective with respect to one of the components to be separated.

Split conditioning technique requires that subsequent to a desliming stage, the resulting oversize (coarse) and undersize (fine) fractions of the flotation feed be conditioned separately and then combined prior to flotation. Authors [39] applied split conditioning to the flotation of coarse pyrite particles from a cleaner tailings product using potassium nonyl xanthate and showed that pyrite recovery did not only improve but the grade of concentrate obtained was far superior to that obtained with tests without split conditioning. Further, work by authors [40] [41] indicated that split conditioning can improve the flotation efficiency of slime (−37 μm) and coarse particles (−105 + 44 μm) due to hydrophobic aggregation between coarse particles and slimes from the Maton rock phosphate plant, India.

Notwithstanding the potential benefits of these techniques, the application on industrial scale are yet to be fully realized. Selective desliming process has been in commercial operation on Michigan hematitic ore [42]. Although split conditioning has shown promise in the laboratory, there is marked lack of published information on its practical implementation. Possibly this is because separation by cyclones, which would be required in practice, is not effective, resulting in the over-conditioning of the fines in the cyclone underflow and poor grades of concentrate. In addition, the difficulties in the operation of a split-conditioning circuit may not warrant the marginal improvement found in practice. The use of this technique could be justified only if the overall recoveries were high −90 per cent and more, [39].

Conventional desliming still remains the most widely practiced desliming technique to resolve the detrimental effect of slimes on flotation. The benefits of desliming described in this report are supported by similar works done elsewhere. For instance, in the work conducted by authors [43], desliming was performed with the tailings from the first stage of the multi-stage flotation system using a 44 μm screen to remove the −44 μm slime fraction. The flotation was continued from second to fourth stage using the +44 μm fraction. The results of the flotation test showed that after the desliming process, Copper (Cu) and Molybdenum (Mo) recoveries significantly increased to 1.63% Cu and 0.24% Mo. Authors [44] [45] demonstrated similar observations.

4. Conclusions

The outcome of this work has shown that desliming improves the flotation response of the Kansanshi mixed copper ore. At a rougher copper concentrate grade of 8%, metal recoveries obtained with desliming were in excess of 70% compared to 58% achieved with baseline tests without desliming.

It was further observed that desliming resulted in improved flotation rates for the sulfide minerals. For a flotation time of 3 minutes, recoveries of 69% and 74% were obtained with elutriation and sieving respectively compared to 58% recovery for baseline tests. From the same results it was also evident that, of the two modes of desliming investigated, sieving yielded better performance than elutriation.

Although flotation on sieved material has shown a better performance than the corresponding material from elutriation, desliming devices that are more suitable than screens for a full scale plant operation such as classifiers (e.g. hydrocyclones) are recommended. It should also be noted that the metal distribution in the feed material indicating that close to 35% of the values are in the minus 11 μm size range, a desliming process with a cut point at approximately 40 μm particle size targeted in the present work, will deport about 30% of the values to the slimes fraction with an average copper grade of 0.60% TCu. This would entail a separate treatment of the slimes fraction such as fine particle processing or leaching to recover the values, like Kansanshi Mine. For high throughput plants this would be an important treatment option.

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Conflicts of Interest

The authors declare no conflicts of interest regarding the publication of this paper.

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