EVALUATION AND OPTIMISATION OF THE GRINDING CIRCUIT AT KONKOLA CONCENTRATOR TO ENHANCE FLOTATION PERFORMANCE.

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ABSTRACT

The grinding plant at Konkola concentrator accepts 18.5 tonnes per hour of crushed ore to produce a grind of 84% passing 75µm. Although this grind and capacity were meeting the Konkola concentrator metallurgical requirements, the technical staff, however, sought for a more durable and lower down time equipment in the classification process of the grinding circuit in order to have an improved flotation performance and hence increase overall production.

Prior to this work, the technical staff of the concentrator carried out a test work in order to optimise the efficiency of grinding plant ball mills fitted with a 38.10 cm, 45.75 cm and 50.80 cm Multotec hydro cyclone. The results of the test work showed that the 38.10 cm and the 45.75 cm ***hydro cyclones were unsuitable*** *due to their low throughput and poor final product quality. The 50.80 cm Multotec hydro cyclone, on the other hand gave improved throughput and the required flotation product quality.*

Further investigations were done on the spigot, the throughput and the grind at which there was an enhanced flotation performance. The results revealed that the best flotation performance was at grinds above 83% passing 75µm. The spigot and the throughput were optimised at 100 mm and 20.59 tonnes per hour respectively.

Keywords: *Flotation, hydro cyclone, spigot, grinding*

INTRODUCTION.

The concentrator of Konkola Copper Mines Plc (KCM) in Chililabombwe has various metallurgical operating plants, which include washing plant, secondary and tertiary crushing and screening plants, grinding and flotation circuits.

Optimum mess of grind is very important to establish as the production of fine can be very expensive and not favourable for optimum flotation performance. (Warren, 1984) The design of the plant allows two different ores to be treated separately if they have different metallurgical characteristics. The concentrator receives ore from No. 1 shaft, No. 3 shaft and Nchanga mine and has a rated throughput capacity of 6500 tonnes per day.

Run-Of-Mine ore with about 3.5-4% TCu is concentrated to over 45% TCu according to requirements of the customers (i.e. the smelters).

The grinding plant has thirteen grinding units which consist of eleven small units each in closed circuit with a hydro cyclone while two bigger units are in closed circuit with two sets of a pump and two hydro cyclones. One set is always kept on standby.

For a long time, the plant has been using the 60.96 cm locally fabricated traditional hydro cyclones which were meeting the Konkola metallurgical requirements of 18.5 tph at 84 % passing 75µm product quality though they had a few setbacks.

The major setback of the traditional hydro cyclone (Wills, 1988) is the high internal wear rates, which consequently led to a high *down time*. The traditional hydro cyclone do not have a provision for warning when the rubber lining became worn out resulting to the wearing of the expensive metal housing.

In the search for better production equipment, Konkola concentrator Technical Staff first tested 38.10 cm Multotec hydro cyclones and later a 45.75 cm Multotec hydro cyclone which both gave not better results due to low throughput and poor product quality.

Subsequently, a 50.80 cm Multotec hydro cyclone obtained from South Africa, was tested in September 2001. This gave the required throughput and product. More orders were placed and the hydro cyclones were installed on all the thirteen units of the Konkola concentrator grinding plant.

Though the 50.80 cm Multotec hydro cyclone met the Konkola concentrator metallurgical requirements, there was need to be optimised on all the milling units of the entire grinding plant.
It was against this background that the Technical staff at Konkola concentrator formulated the research project, which was aimed at evaluating and optimising of the grind at the milling plant in order to enhance flotation performance.

TEST WORK.

In this test work the following were used; a laboratory scale 2.5 litre Denver flotation machine, eight 10-litre buckets and two 1-litre beakers, a series of screens with the following apertures: 106µm, 75µm, 53µm and 45µm and a laboratory balance. Reagents included Collector - Sodium Isopropyl Xanthate (SIPX) Sulphidiser - Sodium Hydrogen Sulphide (NaHS) and Frother – Flotanal 501 (F501).

EVALUATION OF THE GRIND.

Sample Collection and Preparation.

Samples were collected from the cyclone overflow by transversing the linear sampler across the cyclone overflow using the "all the stream part of the time sampling technique" (Chibwe, *et al*, 1999)

A quick wet screen analysis was done on each sample before any sample was accepted as representative. The quick wet screen analysis was done so as to facilitate the collection of a wide range of evenly distributed sample sizes to be taken for the evaluation of the grind. 1 litre of each collected sample was washed on a 75µm laboratory screen and the densities of the samples were taken before and after washing using the density scale. The percent passing 75 µm was calculated as follows (Wills, 1988):

$$\frac{\rho_b - \rho_a}{\rho_b - 1000} x100\% \qquad (1)$$

where:
ρ_b is the density of the pulp before washing (kg/m^3) and ρ_a is the density of the pulp after washing(kg/m^3)

Once a sample size was acceptable as representative, 20 liters of the cyclone overflow was collected at that throughput. The grinding mill throughput was varied each time a sample was collected. Seven samples were accepted. The accepted samples were homogenized and filtered. A small portion was taken from each sample, dried at 110 ^{0}C and a 100 g of the dried portion was then analyzed over the stack of the 106µm, 75µm, 53 µm and 45µm screens. The remaining dry portion was sent to the Analytical laboratories for the analysis of percent total copper and acid soluble copper.

The collected samples were stored in sealed plastics in order to avoid ageing and oxidation. Each plastic was opened a few minutes just before floating so that it's composition could be made homogenous by thorough mixing.

Experimental Procedure.

The flow sheet circuit shown in figure 1 was used on all samples. The sample was put into the 2.5-litre flotation cell. The cell was then filled up with water to the 2.5-litre mark. The reagents were prepared and added as follows:

Ten (10) grams of SIPX pellets were dissolved in 1-litre water to make a 1% solution of SIPX. 6 ml were used at the sulphide rougher stage and 1 ml at the oxide rougher stage. A 1% of NaHS solution was made by dissolving 10 grams of NaHS blocks in 1-liter water. 1 ml was used at the sulphide rougher stage and 20 ml at the oxide rougher stage. One (1) drop of the Frother, Flotanal 501, was found to weigh 0.06 grams. Seven (7) drops were used at the sulphide rougher stage and two (2) drops at the oxide rougher stage.

After three (3) minutes of conditioning time, the air was opened and four concentrate fractions were collected as shown in figure 1.

The fifth concentrate was collected after the addition of some more reagents and conditioning for three (3) minutes as shown in the Figure below. The collected concentrates were filtered, dried, weighed and taken to the analytical laboratories for the analysis of percent total copper and acid soluble copper.

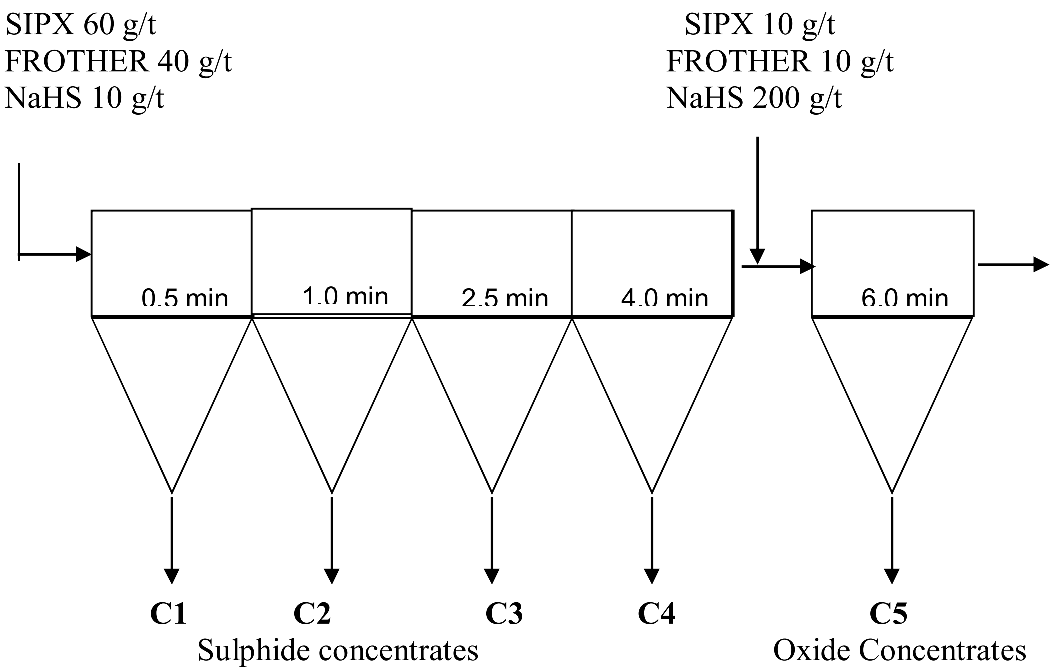

Figure 1: Konkola Concentrator Standard laboratory Flotation flow sheet.

OPTIMIZATION OF THE SPIGOT

Sample Collection and Preparation

Samples were collected from the cyclone feed, overflow and underflow and the ball mill discharge. The samples were collected on an hourly basis for an entire shift. The densities of the samples from all the four streams were measured using laboratory pulp density scale. The collected samples were homogenized and then filtered through the laboratory compressed air filter.

The filtered samples were dried at 110 ^{0}C in the laboratory oven, after which they were homogenized. A 100 g portion was first washed over the 45µm sieve in order to avoid blinding. The filtrate was dried again and then screened over the 106µm, 75µm, 53µm and 45µm stack of screens. Each of the fractions was weighed and the percentage passing each of the screen sizes were calculated.

Four spigot sizes of 80 mm, 90 mm, 100 mm and 110 mm were tested.

OPTIMIZATION OF THE THROUGHPUT

Sample Collection and Preparations

The mill was allowed to stabilize for an hour and thirty minutes at one throughput each time before any sample was collected.

The throughput was determined by taking a belt-cut of 1 meter and weighing it on the industrial scale.

The same procedure performed in the optimisation of spigot was also followed here: Samples were collected from the cyclone feed, overflow and underflow and the mill discharge at three different throughputs. The densities of the samples from all the four streams were measured using the laboratory pulp density scale. The collected samples were homogenized and filtered through the laboratory compressed air filter.

The filtered samples were dried at 110 ^{o}C in the laboratory oven, after which they were homogenized. A 100 g portion was first washed over the 45µm sieve in order to avoid blinding. The filtrate was dried again and then screened over the 106µm, 75µm, 53µm and 45µm stack of screens. Each of the fractions was weighed and the percentage passing each of the screen sizes was calculated.

The test was done on the following throughputs: 19.20 tph, 20.11 tph and 21.07 tph.

RESULTS AND DISCUSSIONS

Evaluation of the Grind

The assays of the two sets of seven differently sized samples analysed showed that the total copper contained in the samples was between 2.96 % and 3.52 % respectively

The flotation performance of copper recoveries for the seven samples is shown in Figure 2.

It showed that the recoveries stabilized between 91.5% and 92% at grinds that were above 83% passing 75µm. Though a test was not done on grinds above 90% passing 75µm, the trend of the results showed that the recoveries would still be within the same range even at finer grinds. Moreover finer grinds than 90 % would result in very low throughput. Extreme fines are however not desired because they tend to consume a lot of reagents and usually result in low recoveries (Warren, 1984).

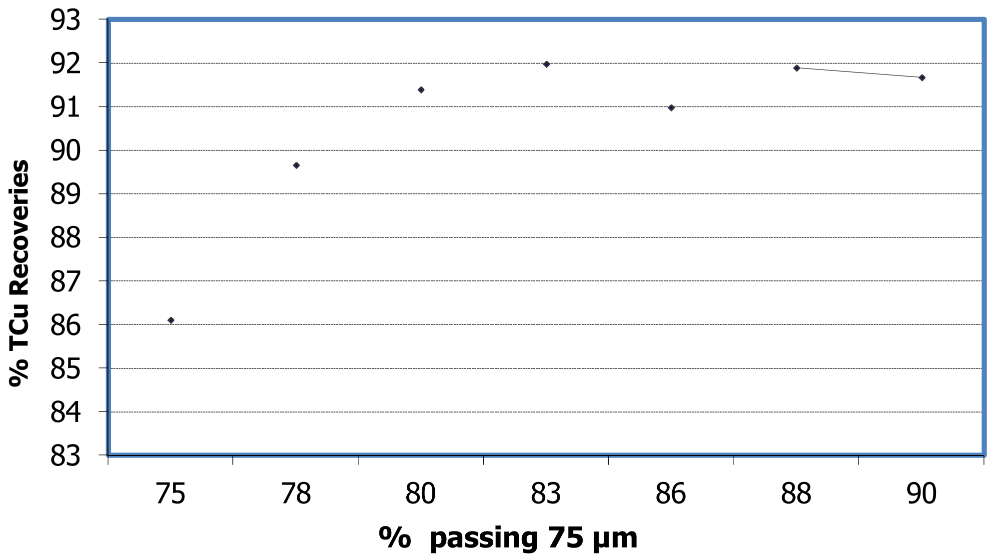

Figure 2: Effect of Grind on Float Performance

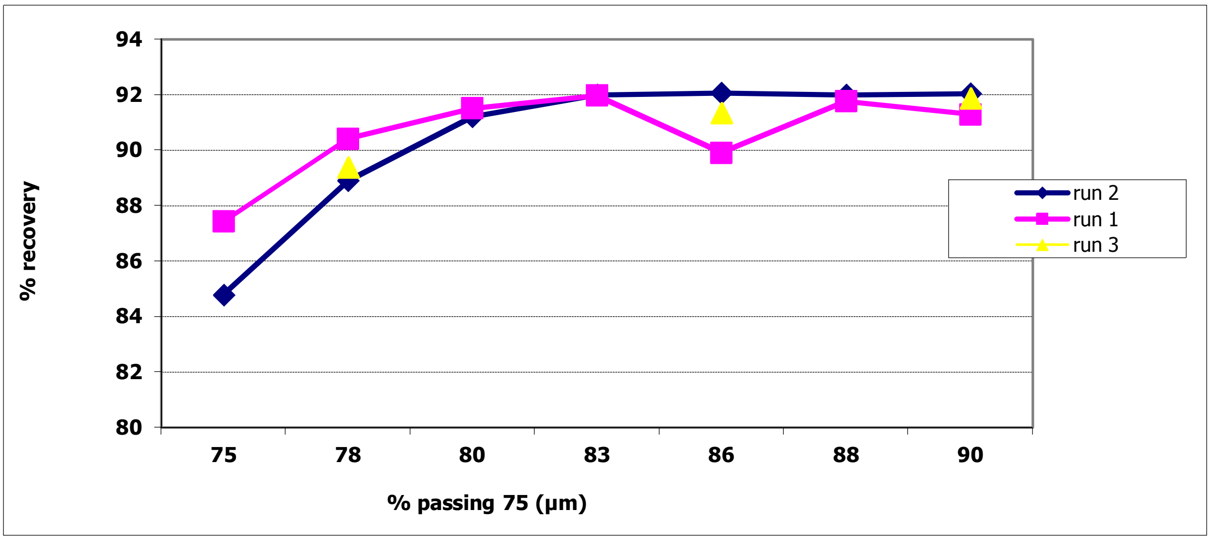

Figure 3: Cumulative Recovery copper against mesh of Grind

The recovery at 86% passing 75µm for the second run caused a depression in the graph at that grind. This prompted the running of a third test on three grinds. The triplicate test was done on 90%, 86% and 78% passing 75µm. The results of the three different runs were shown in figure 3.

The three tests done on the various percentages passing 75µm showed no significant differences. This meant that there was no incentive in grinding above 86% passing 75µ m

Optimization of the Spigot

The samples were analysed from all the four streams, cyclone feed, underflow and overflow and mill discharge in order to determine the circulating load. The cyclone feed sample was, however, not a true representation of the actual cyclone feed because it was cut from the mill discharge dilution sump box, which is before the pump. There was no provision for the sample to be cut at any point after the pump and before the hydro cyclone inlet pipe. The cyclone-feed variable was therefore not used in the calculations as it could have given wrong results.

The results of the spigot optimisation are as shown in figure 4.

The 80 mm spigot gave a very thick underflow and required constant operator attention with a hose to assist the flow of the underflow back into the mill. This was an indication that the spigot size was too small. As a result, coarse particles or oversize materials were caused to report to the overflow. The oversize was very thin in texture and contained a lot of coarse particles.

The 90 mm spigot was not causing as much “roping” as the 80 mm spigot, the overflow, however, had a lot of evenly distributed coarse particles. Though the underflow was not as thick as with the 80 mm spigot, the 90 mm spigot also required occasional operator’s assistance to allow the underflow to flow back into the mill.

The 110 mm spigot gave 90 % of the cyclone overflow to pass through 75µm. The apex was however spraying and the circulating load was too high. The 110 mm spigot was also causing the pump sump to sand up. Though increasing the spigot size would increase the grind, the separation would not be effective because even the already fine material would unnecessarily be recycled into the mill thus increasing the power consumption per unit of material being treated and it may also lead to over grinding of some particles.

The 100 mm spigot gave a grind of 87% passing 75µm. The apex discharge had an acceptable conical shape and the cyclone underflow was freely gravitating back into the mill without the need of constant operator’s assistance, as was the case with the 80 mm and 90 mm spigots. The level of the dilution water in the mill discharge sump had to be maintained at three quarters full. After the dilution water level was increased, the apex began to spray.

Throughput Optimisation

Any slight change in the throughput was affecting the grind very significantly. The results of the throughput showed a relatively course grind of 79% passing 75µm at 21.07 tph. There was 87% and 86% passing 75µm at 19.20 tph and 20.11 tph respectively as shown in the figure 5.

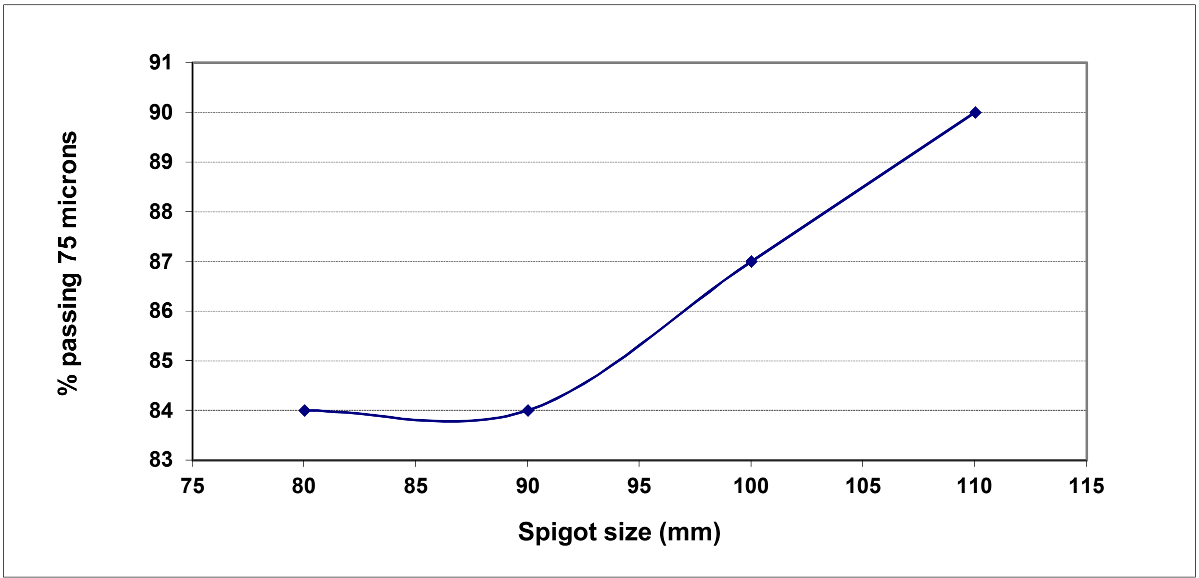

Figure 4: % Passing 75 µm against Spigot Size.

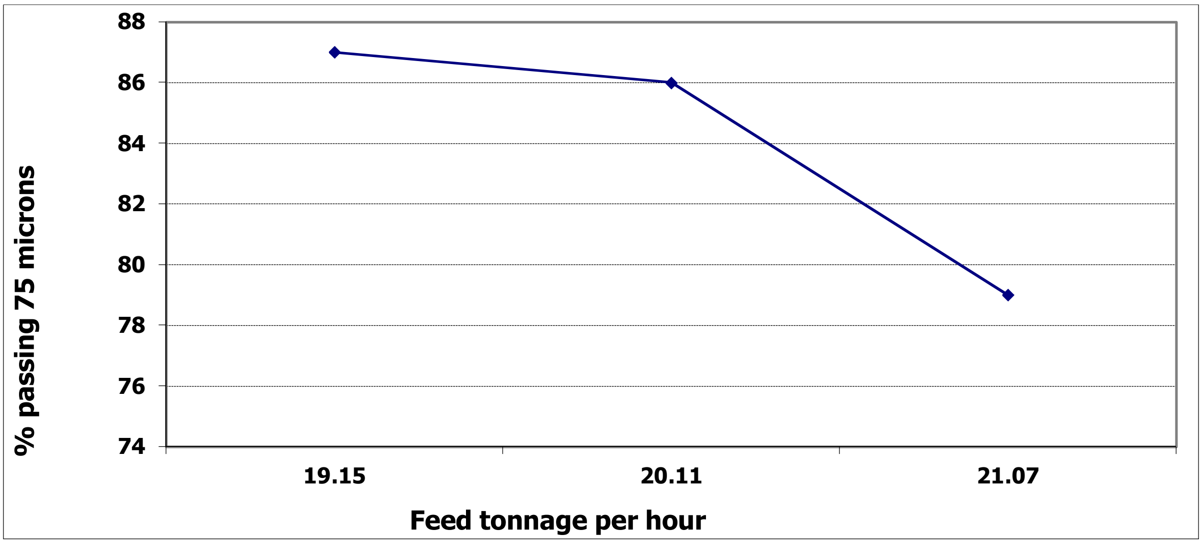

Figure 5: % Passing 75µm against Feed (Tonnes/hour).

The grind was becoming coarser as the feed tonnage was increasing. This was because the residence time in the mill was reduced as the feed was increased while all the other parameters were held constant. Figure 5 shows that the optimum throughput would be between 20.11 tph and 21.07 tph. But since the evaluation of the grind revealed that optimum performance would be above 83% percent passing 75µm, extrapolating a line on the vertical axis at 83% passing 75µm gave a limit of the throughput to be at 20.59 tph as shown in figure 6.

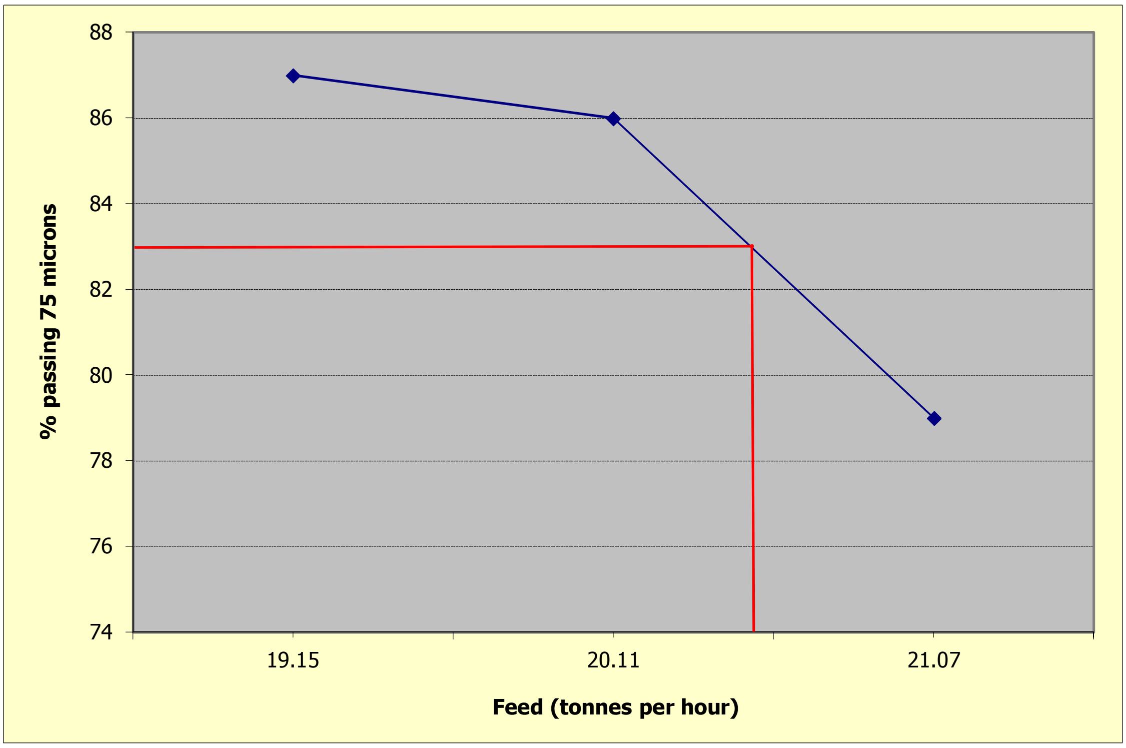

Figure 6: Limit of the Feed (Tonnes/hour)

The optimised spigot size and throughput was tested on unit No. 5 but it did not give as good a product as on unit No. 7. Upon investigation it was discovered that the thirteen units were not operating under the same conditions. The pump speeds, mill ball levels and mill revolutions were all different. A further investigation was then done on all the thirteen units and the findings will be reported elsewhere.

Further investigation revealed that each mill unit had a different feed-belt length and speed. Moreover feed-belt settings on Units No. 3 and No. 5 were changed but the technical staff did not enter the new variables into the throughput calculation table used by the operators to determine the throughput. This called for a thorough investigation because mill unit No. 3 was suspiciously giving a very fine grind of 90% passing 75 microns at 21 tonnes per hour. The investigation revealed that mill unit No. 3 was over-rated while mill unit No. 5 was under-rated. Some differences were also observed between the updated throughput calculation table and that which was being used in the plant before the findings of this investigation.

CONCLUSIONS

The flotation is undoubtedly the most widely used technique to upgrade lean ores (Fuerstenau, 1962, Pyror, 1965). Flotation performance in both the sulphide and oxide rougher stages is enhanced by an increase in the grind. Grinds that are higher than 83% passing 75μm are required for optimum flotation performance. The feed rate to the mill, to the hydro cyclone and the pulp density must be kept as close as possible to the optimum values. Any fluctuation in these parameters results in a very significant change on the grind.

There is an interaction between the cyclone feed pressure, mill ball level, throughput, pulp density, spigot size and the grind at constant mill speed, vortex finder and cyclone type. With all the parameters shown below kept constant, the 100 mm spigot gives an acceptable grind of 86 % passing 75μm. A slight change in the throughput affects the grind greatly. An optimized throughput of 20.59 tph gave an acceptable grind.

In order to have optimum performance in the flotation plant, the grinding plant should produce a grind that is above 83 %. The grind was optimised at 86% passing 75μm with a 100 mm spigot and at a throughput of 20.59 tph.

The test work permitted the following recommendations to be made:

In order to have optimum results from all he thirteen units in the grinding plant, all the units should either be set to the same technical and operating parameters (*mill speed, pump speed, mill ball level and pulp density*) or the optimisation test should be done on each single unit.

A vortex finder optimisation test should be done once the different sizes are made available (*only the 189 mm vortex finder was available at the time of the test work*).

The grinding plant operators should always maintain the mill running conditions to the optimised parameters.

A float valve should be installed to the water supply in the pump sump. This will help to keep the water level in the sump at optimised levels and avoid empting the sump, causing a momentary air lock and sudden fluctuations in the pumped volume.

What is also very important is to understand the surface chemistry of the minerals in the ore. (Leja, 1982).

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