

THE UNIVERSITY OF ZAMBIA

SCHOOL OF MINES

DEPARTMENT OF METALLURGY AND MINERAL PROCESSING

MM 590-FINAL YEAR PROJECT REPORT

**THE EFFECT OF GRIND AND INCREASED RESIDENCE TIME ON
THE FLOATABILITY OF MUFULIRA COPPER ORE**

BY

SINKALA SUWILANJI

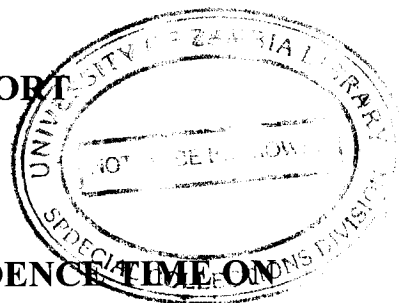
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**THE SUBMISSION OF THIS REPORT WAS THE PARTIAL
FULFILMENT TO OBTAIN THE BACHELOR'S DEGREE OF
MINERAL SCIENCE (B. Min.Sc.) OF THE UNIVERSITY OF
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BY

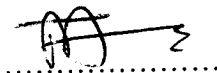
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DEDICATION

To my sweetest Mum and Dad for them being there for me, their patience and encouragement rendered through out my entire years of study.

My brothers, sisters and friends for giving me your support, care, inspiration and the confidence you thus gave me to go on.

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I would also like to express my appreciation to Mr. J.Manchisi, my project supervisor and lecturer, for sharing his time in giving me advice on the composition, and arrangement and making helpful and valuable comments of this project.

ABSTRACT

The purpose of this project was to evaluating the effect of grind time and increased residence time on the floatability of the Mufulira copper ore on a laboratory scale. The project was proposed mainly because of periodic fluctuations in the mineralogy of this complex copper sulphide ore. The aim was therefore to address this problem and generally improve the flotation performance of the Mufulira concentrator.

Systematic sampling was employed where copper ore samples were cut from six different selected conveyor belts each feeding the ball mills. Grinding time was done from 15 to 40 minutes at an interval of 5 minutes. Thereafter, flotation testworks were carried out to determine the effect of increased residence time on the flotation performance. However, 25 minutes grind time was the optimum mesh of grind with corresponding copper recovery of 94.8% and a grade of 24.9%. At 4 minutes rougher and 8 minutes scavenger was the optimum residence time with copper recovery and grade of 88.9% and 21.2% respectively. All other parameters were kept constant such as reagent dose rate at 30 gram per litre and pulp density except for time which was under evaluation. It is recommended that the optimum mesh of grind should be 25 minutes at flotation time of 4 minutes rougher and 8 minutes scavenger and testworks should be done at a decreased grind time and decreased residence time to determine the floatability of the Mufulira copper ore.

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ABBREVIATIONS

Cum.	Cumulative
TCu.	Total copper
AICu	Acid insoluble copper
ASCu	Acid soluble copper
Tails	Tailings
%wt	weight percent
Conc	concentrate
SIPX	Sodium isopropyl xanthate
PAX	Potassium Amyl xanthate
R	Rougher
S	Scavenger
MIN	Minutes

CHAPTER ONE

1.1 INTRODUCTION

For a long time, the study of size reduction was based on energy consumption during grinding. This is because of the fact that size reduction operation prior to flotation is accompanied by large cost due to the high power consumption.

Therefore, this report is based on laboratory grinding and flotation testwork carried at Mopani Copper Mine, in Mufulira.

1.2 BACKGROUND

Basically, Mufulira concentrator treats copper sulphide ore from the east section of the mine, at a rate approximately 250 000 to 300 000 tonnes of ore per month. The Mufulira ore consists of chalcopyrite, chalcocite and bornite as disseminated particle in fine grained, siliceous rock. Other minerals presenting small content in the ore are malachite, covellite, chrysocolla and some native copper.

The Mufulira concentrator produces high-grade copper concentrate which is subsequently sent to the Mufulira smelter. The operations of the concentrator are maintained and effectively controlled with the aim of achieving their target setting.

The project was formulated because of the fluctuations in the mineralogy of the complex copper sulphide ore over a short period of time. Therefore, the technical management at Mufulira concentrator formulated this study in order to evaluate the effect of grind time and the increased residence time on the floatability of the Mufulira copper ore in view to improving the flotation performance.

1.3 OBJECTIVE

The main objectives of this study were to

- determine the effect of grind time on the flotation performance, and subsequently
- determine the effect of increased residence time on the flotation performance.

1.4 PLANT DESCRIPTION

The Mufulira concentrator is sub-divided into the east crushing plant; the milling, flotation, filter plant as well as the tailing pump house and tailing dump.

1.4.1 Crushing Plant

The purpose of crushing is to reduce the run of mine ore to the size that can be milled cost effectively. Ore crushed by the primary crushers to less than 150mm underground is hoisted into the surface bin and a series of conveyors or networks facilitates the delivery of the ore from the shaft bins to standard bin. The smooth flow of the ore on the conveyors is influenced by the vibratory feeders under the bin. However, moisture content must be less than or equal to 2.00% H₂O for the ore hoisted in the surface bin. There are two types of crushers currently being used and these are the Hydrocone H6800 and the Standard Symons cone crusher. The ore is screened whereby the oversize is re-circulated in the closed circuit operation with the crusher. The screened undersize which is the final product of this operation is conveyed to the milling section in the fine storage bins.

1.4.2 Milling

The purpose of the milling is to reduce the crushing plant product to the size that can be floated cost effectively. The milling section receives crusher product at a minimum size of 70% passing 9.51mm. The product is fed into the ball mills which also operate in closed circuit with the hydrocyclone. The overflow from the cyclone goes for flotation

whereas the underflow is fed back to the ball mill for regrind. This is aimed at 55% passing 75microns with the pulp density in the range of 1240-1600 gram per litre (gpl).

1.4.3 Flotation

This section consists of a three stage process i.e. the rougher, cleaner and scavenger. It is in this section where reagents are utilized and these are sodium isopropyl xanthate (SIPX) and potassium amyl xanthate (PAX) for the rougher and scavenger respectively. Aero-froth 65 and beta-froth 65 are the frothers used. Lime is added as a modifier to raise the pH of the pulp in the range of 10.5-12.5.

THE CURRENT CONCENTRATOR CIRCUIT

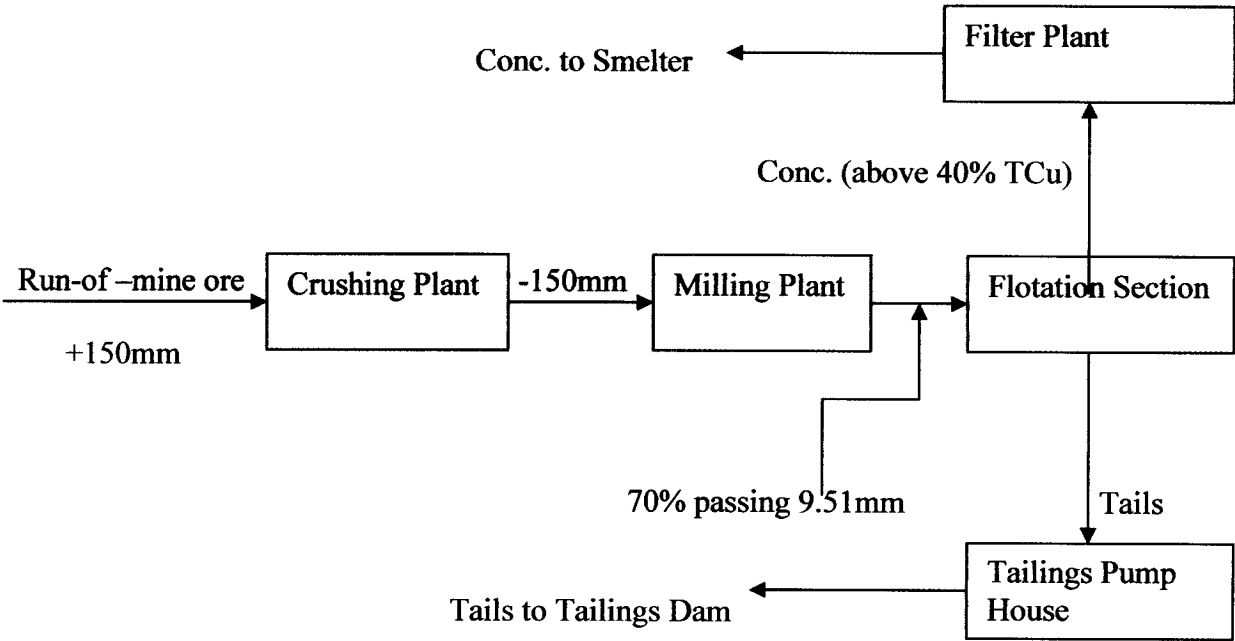


FIG.1. Material flow diagram for Mufulira Concentrator

CHAPTER TWO

2.0 Literature Review

2.1 Grinding

This is the last stage in the process of comminution where particles are reduced in size by a combination of impact and abrasion either wet or dry. It is performed in rotating cylindrical vessels mainly made of steel and are called tumbling vessels. These comprises of loose grinding media, which are free to move inside the mill, thus grinding the particle. The grinding media may be steel rods or balls hard rock or in some cases the ore it self (Wills, 1988).

Naturally, most minerals are finely disseminated and intimately associated with the gangue. Before separation can be under taken, the mineral must be initially “unlocked or liberated”. While, this is achieved by crushing and grinding, to be specific grinding prior to flotation size reduction is up to some extent. It has been observed that grinding should be done at the optimum mesh of grind which is the particle size at which there is maximum profitability (Deelder, 1996).

Grinding is the most energy intensive operation in mineral processing and accounts for about 50% of the concentrator’s energy consumption. Therefore over grinding and under grinding should be avoided to reduce difficulties faced in the flotation of slimes and coarse material respectively. Hence, there is a lower and upper size limit for the particle to be floated. Size reduction to a relatively coarse size does not liberate the mineral particle, flotation recoveries in this case will be poor as the particle size will not be small enough to be lifted by the air bubbles during flotation.

Grinding with a tumbling mill is influenced by a variety of factors which include speed of the mill load of grinding media and material, pulp level, size distribution of grinding media and solid-liquid ratio and circulating load.

i. Speed of mill.

The speed must not be too high because this brings about centrifuging of the charge and thus grinding does not take place. At the same time the speed must not be too low because this will promote the production of slimes. An action called cateracting action must result from the speed of the mill accompanying a free fall of grind media for favourable impact giving a high capacity and efficiency.

ii. Load of grind media and material.

Increasing these will require more power until the maximum value is achieved and the power requirements decreases with increasing load as the center of the load approaches the axis of rotation.

iii. Pulp level.

This will require high power if the pulp level is high. The lower pulp level entails greater freedom of movement of grinding media hence more effective grinding. Therefore, the ball mills should be filled slightly more than half full with the materials and balls.

iv. Solid-liquid ratio (Pulp dilution)

Too dilute a pulp increases metal to metal contact, giving increased steel consumption and reduced efficiency. Therefore, ball mills should operate between 65 – 80% solids by weight depending on the ore. The viscosity of the pulp increases with the fineness of the particle, fine grind circuit may need lower pulp densities.

v. Circulating load.

In order to achieve grinding to required size in a single step might result in wasteful over grinding. To avoid over grinding and giving an effective grinding, the residence time for material to be in the mill must be short.

Generally, increased circulating load results due to the average size of entering feed becoming lower, interstitial loading is improved with closed adjustment of solid-liquid ratio.

In practice, ores are ground to an optimum mesh of grind determined by laboratory and pilot scale test work to produce an economic “degree of liberation” which is the percentage of the mineral occurring as the particle in the ore in relation to the total content. This can be high if there are weak boundaries between mineral and gangue particles. The optimum mesh of grind is the particle size at which the most economic recovery can be obtained. This depends not only on the grindability which is the ease with which the ore can be comminuted but also the ease with which the comminuted ore can be floated (Wills, 1988).

2.2 SIEVE ANALYSIS

Size control is important in order to check the quality of grinding, the extent to which values are liberated from the gangue at various particle sizes and to aid specific examination of ore constituents. Size analysis of the ground ore shows whether mineral particles are properly liberated and how grinding affect recovery of the mineral values from the ore.

Standard test sieves are used for sieve analysis. Laboratory sieves consists of woven wire, which results in square aperture with small tolerance (Kelly, 1982). There are different types of sieve series. These series differ mainly in their relationship between aperture and wire size. They include US sieve series, international test sieve series and British standard series. Sieve analysis is accomplished by passing a known weight of sample material through the stack of test sieves and the entire stack shaken. The weight retained

on each sieve is then used to determine the percentage weight in each size fraction. Sieving can be carried out on dry or wet material.

Sieving when applied to irregularly shaped particle is complicated because “near mesh” particles may pass only when presented in favorable position. This in many cases causes the blinding and obstruction of the sieve aperture and thus reduces the effective area of the sieving medium. Blinding is most serious with sieves of very small aperture size (Wills, 1988). Therefore, the effectiveness of a sieving test depends on the amount of material put on the sieve and the type of movement imparted to the sieve. The charge material should not be too large, so that each particle is given a chance to meet an aperture in the most favourable position, for sieving in a reasonable time. On the other hand, the charge must contain enough particles to be representative of the bulk sample (Wills, 1988).

2.3 FROTH FLOTATION

Froth flotation is a means of treating a pulp of finely ground ore so that valuable or desired mineral is obtained in a concentrate which will be amenable to further process. The process involves imparting a water repellent (hydrophobic) character to the desired mineral with the aid of chemicals that are called collectors or promoters under favourable conditions, these chemically coated particles becomes attached to the air bubbles rising through a pulp and will thus float to the surface.

This is the cheapest and most extensively used process for the separation of chemically similar mineral and to concentrate ore for economical smelting.

Floatable minerals can be classified into non polar and polar type. The difference between the two types is based on the surface bonding. The surfaces of the non polar mineral have relatively weak molecular bonds, difficult to hydrate, and in consequence such mineral are hydrophobic. The ores containing these minerals for beneficiation

usually require the addition of non specific collector to their pulp to aid the natural hydrophobicity of the floatable fraction.

Polar minerals have a strong covalent or ionic surface bonding and exhibit high free energies at their surface. These species are hydrophilic (water seeking) and need surface modifiers to render them amenable to flotation.

In froth flotation, the mineral is transferred to the froth or float fraction, leaving the gangue in the pulp. This is referred to as direct flotation which is the most widely practiced flotation process. The other types of flotation are bulk, reverse and differential flotation.

Froth flotation, therefore, consist of producing condition in which air bubbles form, or are introduced to the slurry of ground ore in water and the desired minerals are made to attach themselves to the bubbles. Air bubbles only continue to support the mineral if they form a stable froth, otherwise, they will burst and drop the mineral. These conditions can be achieved by using numerous chemical reagents.

2.3.1 Flotation reagents in general

For flotation to be effective, reagents must be added to the pulp. In general, the reagents are interfacial surface tension modifiers and/or flocculants (Crozier, 1992). Usually, these reagents are classified under five heading which are:

- Collector
- Frother
- Activator
- Modifier

i) Collector

Collectors are organic compounds used in flotation, which render selected mineral water repellent by absorption of molecules or ions on to the mineral surface, reducing the

stability of the hydrated layer, separating the mineral surface from the air bubble to the level that attachment of the particle to the bubble can be made on contact.

Collectors may be non-ionizing or ionizing compound. Ionizing collectors are mostly widely used in sulphide flotation. They are usually sodium and potassium salts of certain acids. They dissociate into ions in water and have chemical structures which are heteropolar: i.e. the molecules contain a non polar hydrocarbon group and a polar group.

The non polar parts of the molecules are oriented towards the water phase and the polar orient itself towards the mineral surface. The non polar hydrocarbon radical has pronounced water repellent properties and hence they render the mineral surface hydrophobic by keeping water away from the mineral surface. Non ionizing collectors which are non polar compounds are practically insoluble in water. They render the mineral water repellent by its surface with a thin film.

Collectors are used in small amounts, substantially those necessary to form a monomolecular layer on particle surface. An increase in concentration apart from the cost tends to float other minerals, thus, reducing selectivity. An excessive concentration of the collector can have an adverse effect on the recovery of the valuable mineral, due to the development of multi-layer on the particle, reducing the proportion of hydrocarbon radicals (non polar group) oriented into the bulk solution. The hydrophobicity of the particles is thus reduced, and hence their floatability (Wills, 1988).

ii) Frother

These are surface active reagents that aid the formation and stabilization of air induced flotation froths. They concentrate at the air-water interface, helping to keep the air bubbles dispersed and preventing their coalescence.

The other functions of frothers are to:

- reduce the surface tension of the air-liquid interface in order that a stable bubble is produced in the system.
- influence the kinetics of bubble-particle adhesion.

- thin the liquid layer by interacting with the collector.
- stabilizes the bubble-particle aggregates.(Kelly,1982)



iii) Activator

Activators are reagents that alter the chemical nature of the mineral surfaces so that they become hydrophobic due to the action of the collector. They are generally soluble salts, which ionizes in solution, the ion then reaching the mineral surface. A good example, if the mineral to be floated is an oxide in nature, an activator to form a pseudo-sulphide layer around the mineral can activate its surface. This will then render the mineral amenable to the action of a collector.

iv) Modifier

These are reagents used in flotation to modify the action of the collector, either by intensifying or reducing its water-repellent effect on the mineral surface. Thus, they make the collector action more selective towards certain minerals. These are classed as the regulating agents which also include the activators, depressants and pH modifiers i.e. lime. The same modifiers can perform as a depressant or activator or both depending on the condition under which it is used.

2.3.2 EFFECT OF GRINDING ON FLOTATION

Many factors influence the flotation process. According to Matis and Zouboulis, (19995) among the principal factors are particle size, pulp aeration, bubble mineralization, agitation intensity, residence time of bubble in the pulp, pulp density. Others include the specific gravity of the particle, presence of slimes, hydration of the surface, and adsorbed layer of flotation reagents (Matis and Zouboulis, 1995)

Grinding has the obligation exercise close control on product size. Undergrinding of the ore will of course result in the product that is too coarse with the degree of liberation too low for economic separation while over-grinding needlessly reduces the particle size of the substantially liberated major constituents usually the gangue and may reduce the particle size of the minor constituent (usually the mineral) below the size required for most efficient separation.

2.3.3 Flotation of large particles

In froth flotation, the particle size upper limit is 200-300 micron, whereas the lower limit is 5-10 micron with the desired mineral exposed at part of the particle's surface. Therefore, if the particle exceeds the upper limit in a given instance, they can not be floated under the prevailing condition. The upper size limit defines the size at which the air bubbles can no longer physically lift the particles to the surface. The probability of the particle being detached from the air bubble is high due to the heaviness of the particle as a result the larger particle are retained at the bottom of the flotation cell and this will lead to the loss of the valuable mineral in the tailing fraction. However, in order to recover the valuable mineral that reports to the tailing, a further regrind or liberation will be required and this entails an extra cost because grinding alone is the highest cost operation (Wills, 1988).

On the other hand there are essential conditions necessary for the flotation of large particles. This includes the following:

- Particle surfaces must be rendered as water repellent as possible by the use of increased amount of collector.
- Increased pulp aeration must be employed in order to produce favourable condition for “group” flotation of large particles by several bubbles.

2.3.4 Flotation of extreme fine particles

According to Glembskii (1972), the extremely fine slimes particles which are present in the pulp have profound effect on flotation.

The following phenomena are apparent when slimes are present in the pulp:

- It becomes significant with the increasing demand for minerals and the continuously declining grade of available ore.
- Low particle momentum
- Contamination of the froth product by small gangue fraction due to physical entrainment.
- Low probability of particle collision (P_c)

$$P_f = P_c \cdot P_a \cdot P_d$$

Where P_f = flotation probability

P_c = probability of particle-bubble collision

P_a = probability of particle-bubble attachment

P_d = probability of particle-bubble detachment

- There is undesirable coating of air-bubble surface by slime particle. This will prevent normal sized flotation particles from adhering to these coated air bubble thus hindering flotation of valuable.
- There is also an undesirable coating of a valuable particle with ultra fine gangue slime. This will render the valuable particle surface hydrophilic, thus suppressing flotation because slimes are generally hydrated in many ways.

2.3.5 Air

Air used in flotation is either induced by the action of the impeller of the flotation machine or super charged i.e. it is introduced through the internal blower. The role of air in flotation is to provide the necessary bubble, which in turn generate the froth which will act as separating medium to segregate and remove the valuable mineral from the gangue. (Arbiter, 1985)

CHAPTER THREE

3.0 EXPERIMENTAL PROCEDURE

The testwork was done on laboratory batch grinding and flotation basis for the Mufulira copper ore.

3.1 Apparatus

- One meter long sample cutter
- Buckets
- Scoops
- Electric balance
- Nest of sieves and sieve shaker machine
- Laboratory ball mill. Mill steel balls of 0.1153kg on average as grinding media and volume of 7.5litre.
- Laboratory Denver flotation machine (1500 RPM)
- Flotation cell 5000 cm³ capacity
- Drying oven
- Measuring cylinder (1000ml), stop watch, wash bottle, syringes, funnel, flotation collection plates, paddle, sample envelop, brush and washing detergent and plastic bags.

3.2 Reagents

Xanthate collectors were used, especially for easy to treat ore where selectivity is not an issue. These were in powder and pellet form, and are readily soluble in water and were made up to 6ml at 1% strength. The following are the reagents which were used:

- Sodium Isopropyl Xanthate (SIPX) the most widely used in the flotation of sulphide minerals of copper.
- Potassium Amyl Xanthate (PAX) the most powerful and least selective xanthate. Often used as a scavenger collector following a more selective rougher collector.

- Aero-frother 65 is a polyglycol that exhibits strength and longevity in flotation circuits.

3.3 Sample preparation

The technique employed during sampling was called a systematic sampling. Copper ore samples were cut from six different selected conveyor belts each feeding the ball mills. A one meter sample cutter was struck along the stopped belt and all the ore confined within the one meter was collected and put into the buckets. Samples were taken to the Analytical Laboratory Service for crushing to 100% passing 2mm and after which the product was stored in covered buckets to prevent contamination, oxidation and surface ageing of the minerals. Homogenization of the fresh ore was done manually using a scoop upon pouring the ore on the preparation bench.

3.4 Grind and Sieving

Wet grinding was carried out in the laboratory ball mill. The charge to the mill included of 2000g of ore, 1500ml of water and mill balls of different sizes. The pulp density of 57.1% solid was maintained or kept constant for every test. Grinding time evaluation was done at the following times; 15, 20, 25, 30, 35 and 40 minutes for every 2000g respectively. After grinding, the ball mill discharge was wet screened passing through 45 microns which provided a removal of slimes and loss of dust. The oversize (what was retained on the +45 micron) was then dried, weighed and screened on the nest of sieve (250, 180, 150, 125, 106, 90, 75, 63, 45 microns) were arranged in the decreasing order from the top to the bottom on the sieve shaker machine. The undersize (-45 microns) was however weighed and then discarded.

From the sieve analysis, the cumulative weight percent passing was computed. All the tests work were done in duplicates.

3.5 Flotation Testwork

The Denver laboratory flotation machine was used for each test. Ground ore sample from the ball mill was immediately placed in a 5000ml flotation cell and more water was added to make up to the mark with the pulp density of 33.3% solids by weight. This was followed by vigorous agitation to keep all solid in suspension, at an impeller speed of 1500 rpm.

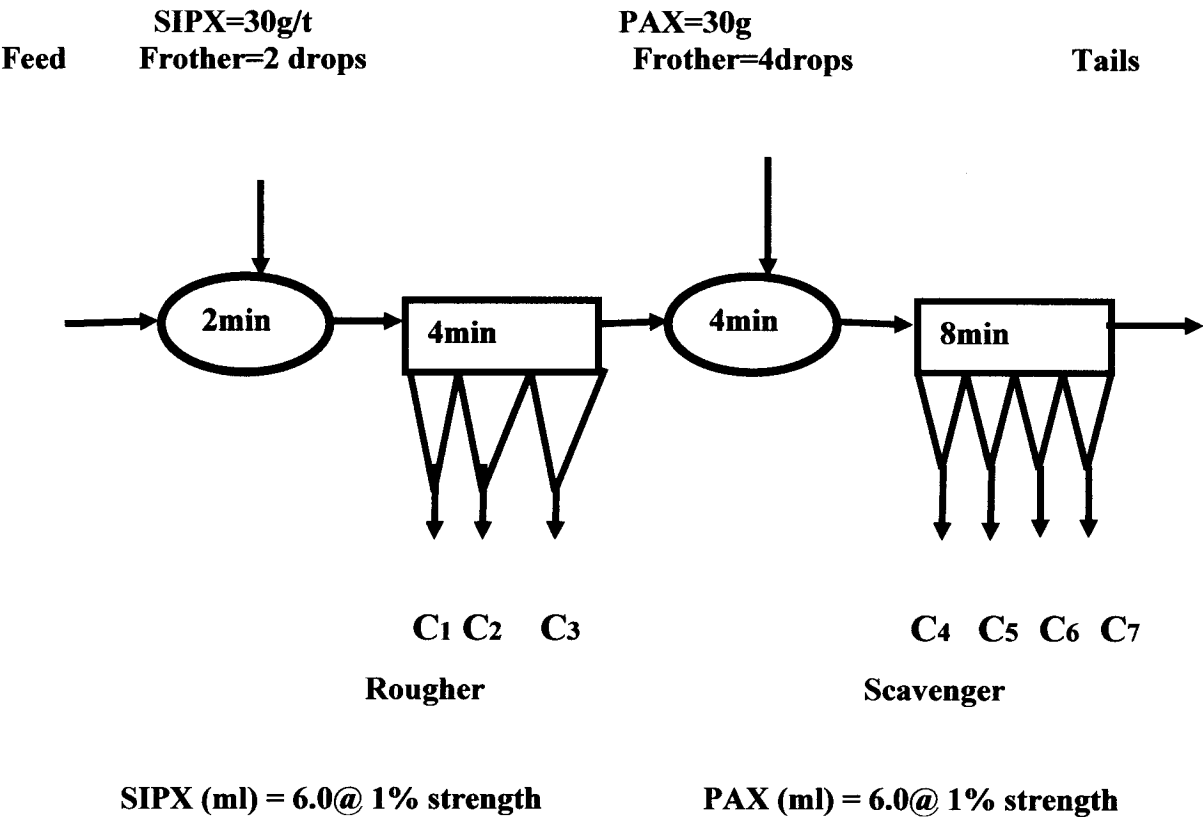


Figure 3.1: The Mufulira standard differential laboratory flowsheet for copper ore.

To simulate plant practice a differential mode for the Mufulira ore was used. This is where sulphides are recovered in the rougher and scavenger stages under the addition of different chemical reagents.

Sulphide Roughing

Initially, the collector was prepared and the required amount was accurately measured and added to the slurry. The collector sodium isopropyl xanthate (SIPX) was added then conditioned for the desired period (standard is 2min) and with only a minute remaining of the collector conditioning time, the desired amount of sulphides frother (aero-frother65 with only 2drops) was added.

After the conditioning time, air was introduced into the cell by opening the valve on the stand pipe.

However, using plastic paddles, the froth being generated into the cell was pulled by running the paddles across the pulp surface (the froth falls into the plates) for the desired period of 10 seconds interval. Water was added occasionally to the cell to maintain the pulp level; hence, froth collection period was for 4, 5, 6 and 7 minutes for every 2000g ground ore respectively.

Air addition to the cell was removed by closing the air valve on the stand pipe. Samples collected in the plates as the sulphide concentrate using appropriate prepared labels were identified and taken for drying in oven.

Sulphide scavenging

The procedure employed in roughing stage was the same as in scavenging stage but only differed in terms of chemical reagents addition and the conditioning time. Potassium Amyl Xanthate (PAX) and aero-froth 65 were added and 4min of conditioning time. The sulphide scavenger concentrate was collected for 8, 9, 10 and 11 minutes.

Samples generated from the laboratory were analyzed for total copper (TCu) and acid soluble (ASCu), which are analytical estimates for the total copper content in the ore. The

copper sulphide minerals are estimated as acid insoluble which is the difference between the TCu and ASCu estimate.

CHAPTER FOUR

4.0 RESULTS AND DISCUSSION

This chapter presents a discussion of results and the main findings that were obtained in this study. The main parameters that were considered are the effect of grind time and increased residence time on flotation performance of Mufulira copper sulphide ore.

4.1 SEIVE ANALYSIS

From the graph below, an estimated optimum grinding time was approximately 17 minutes which gave 55% solids passing 75 μm . It also shows that, any increase in grind time results in more finely particles being produced i.e. more particles passes 75 μm , therefore size distribution is important prior to flotation.

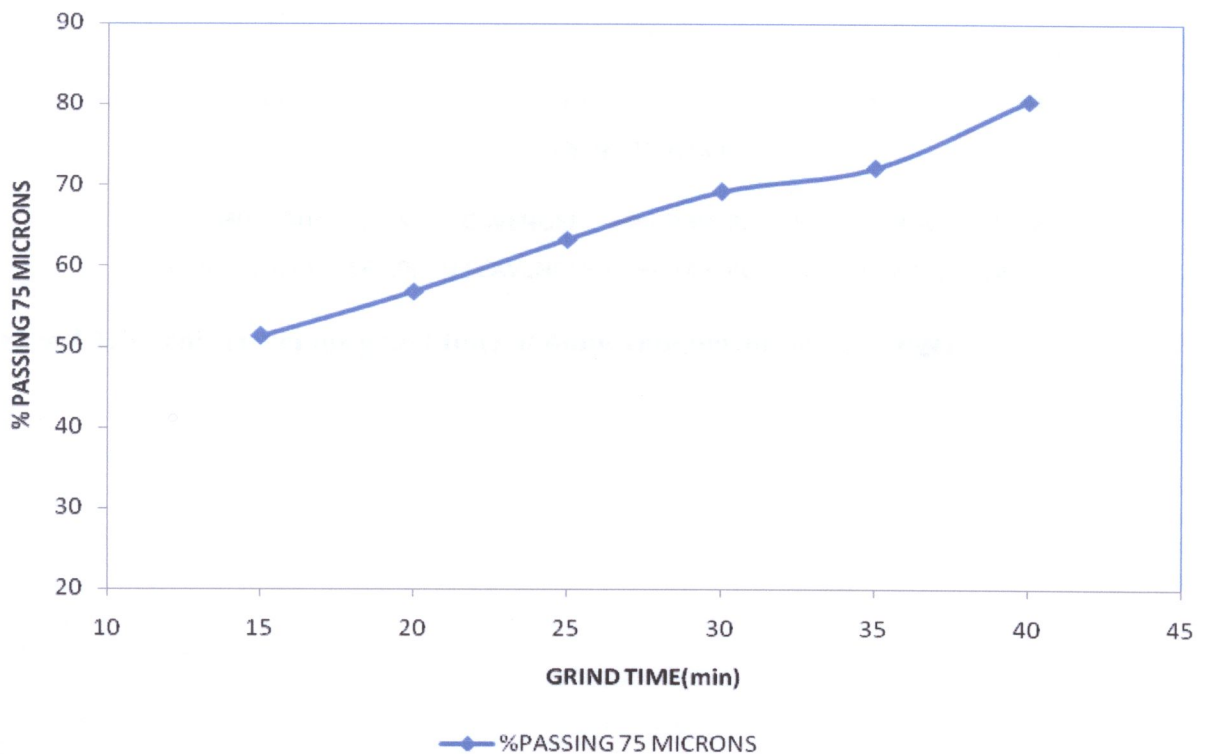


Figure 4.1: Percentage of solids passing 75 μm plotted against grind time

4.2 THE EFFECT OF GRIND TIME ON FLOTATION PERFORMANCE

Figures 4.1-4.3 below show, the effect of grind time on copper recovery and grade at constant residence time of 4 minutes rougher and 8 minutes scavenger.

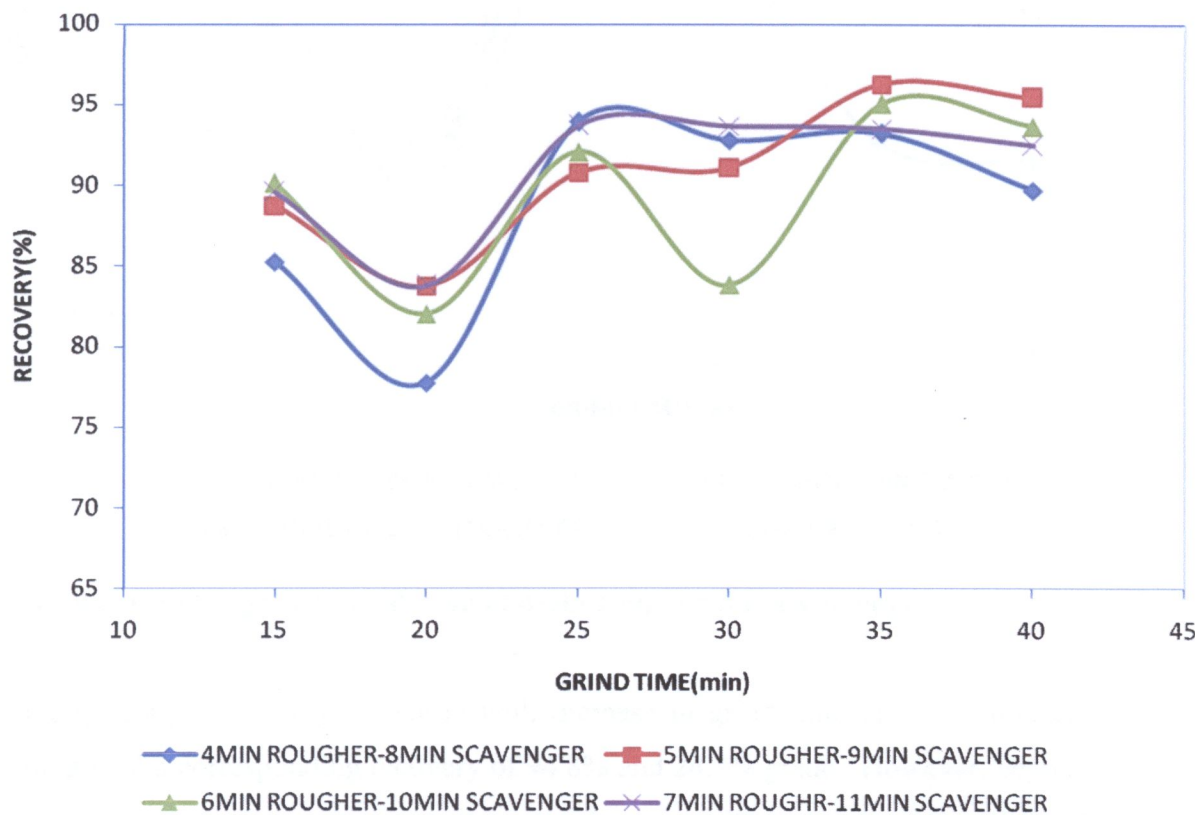


Figure 4.1: Recovery versus grind time at 4min rougher-8min scavenger

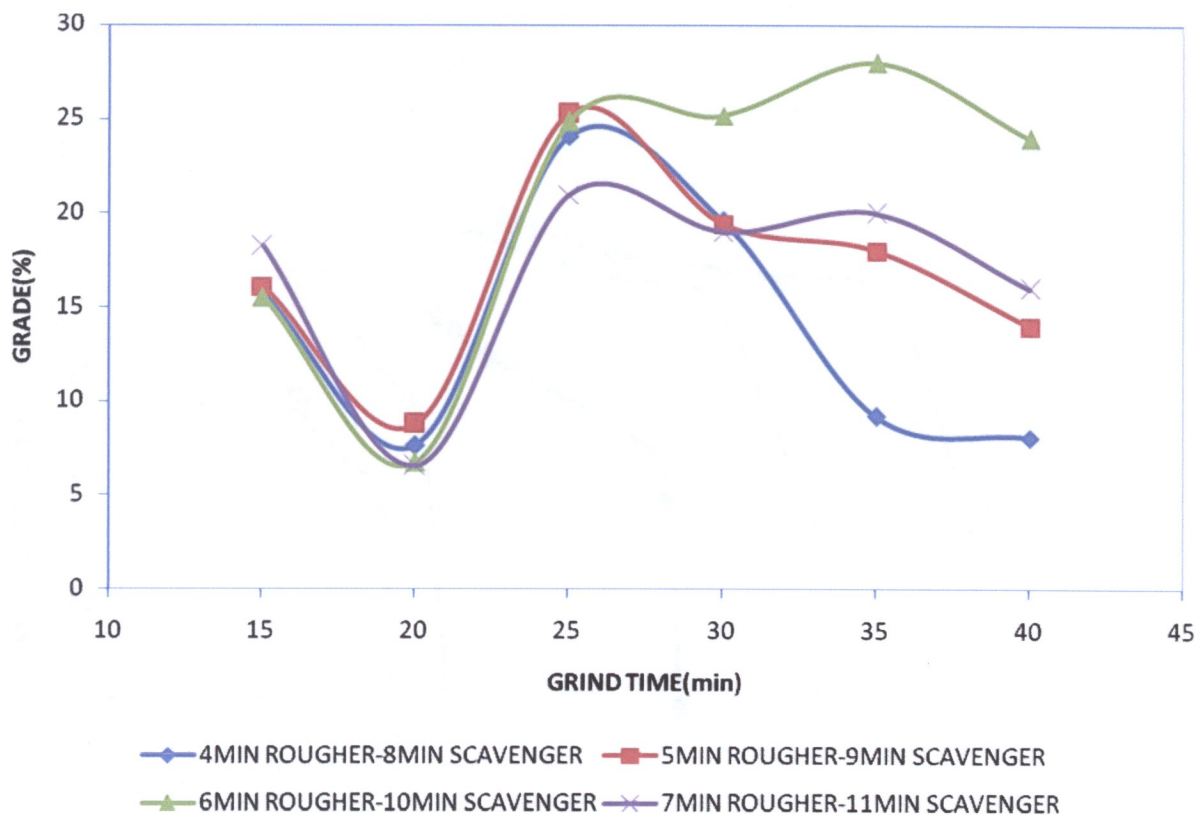


Figure 4.2: Grade against grind time at 4min rougher-8min scavenger

Generally, copper recovery increased with increase in grind time up to 25 minutes of grinding with a corresponding recovery of 94.8% and 24.9% grade. However, beyond 25 minutes grind time, copper recovery began to decrease with increasing grind time and it also occurred at 35 minutes grind time, which gave a recovery of 94.6% but a low grade of 9.3%. This result in high cost of energy consumption hence slimes production which has a deleterious effect on flotation performance. A drop at 20 minutes grind time with copper recovery of 76% could be as a result of the hardness of the ore and also the discrepancies incurred during the test. Figure 4.3, the recovery and grade graph below shows that, at six different grind times from 15 – 40 minutes at an interval of 5 minutes, copper recovery decreases with an increase in grade. The 25 minutes grind time gave the highest recovery indicating that the degree of liberation is highest at this time having a copper recovery of 88.9% and a grade of 21.2% at a fixed residence time of 4 minutes rougher and 8 minutes scavenger.

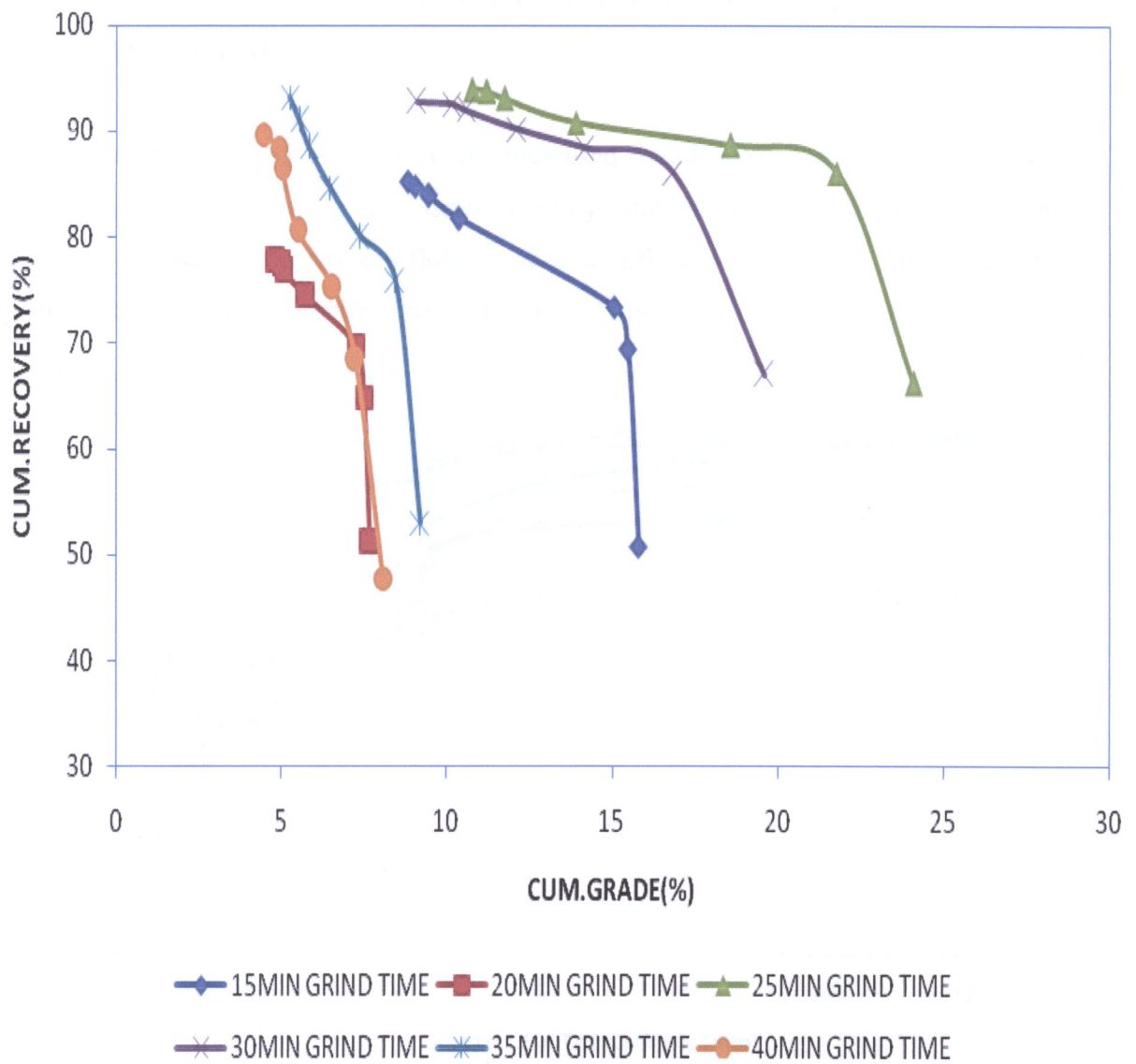


Figure 4.3: Copper recovery plotted against grade at 4 minutes rougher - 8 minutes scavenger

4.3 THE EFFECT OF INCREASED RESIDENCE TIME ON FLOTATION PERFORMANCE

In the determination of the effect of increased residence time on the flotation performance, flotation kinetics were determined by establishing the relationship between cumulative recovery of copper and flotation time at different grind times. The results for copper recovery and grade are shown below in Figures 4.4 and 4.5.

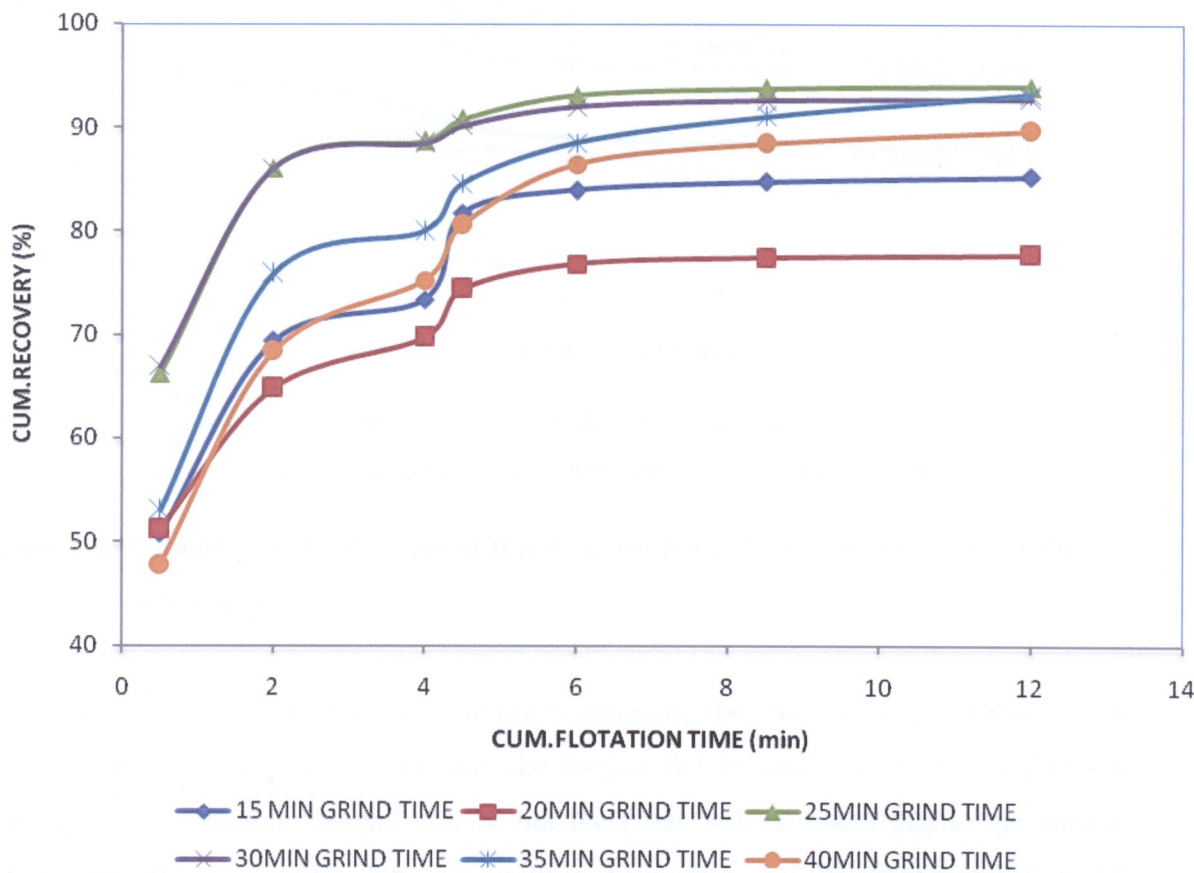


Figure 4.4: Cumulative recovery vs. flotation at a fixed residence time of 4 minutes
Rougher-8 minutes scavenger

The uppermost curve which is the 25 minutes grind time represents the best flotation conditions with corresponding copper recovery of approximately 94.5% while the 20 minutes is the bottom most with approximately 79% copper recover. After 25 minutes of

grind time recovery significantly starts to decrease because of slime formation and this have a huge effect on the recovery even as residence time is increased.

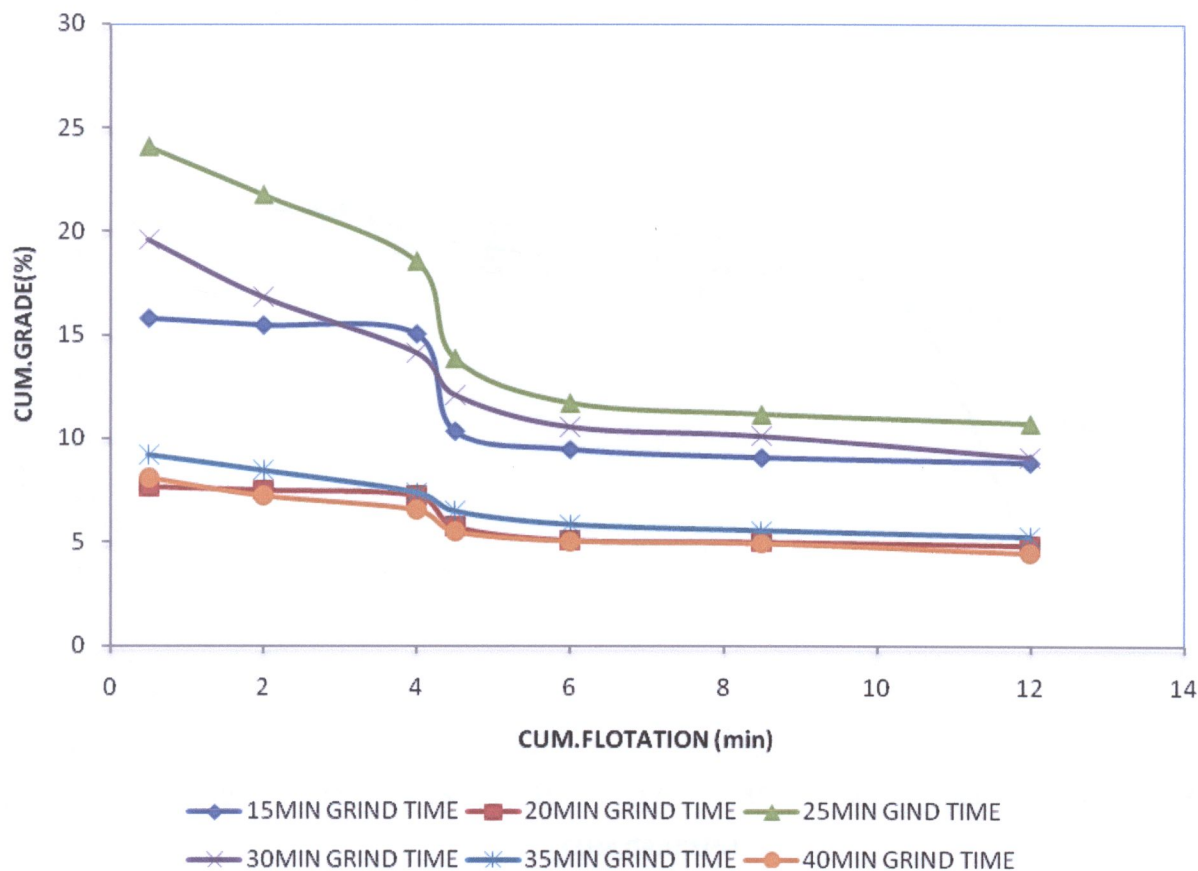


Figure 4.6: Cumulative grade against flotation time for TCU at 4min rougher-8min scavenger.

The figure above shows that, after 4 minute of roughing the recovery of copper increased with the decrease in grade. However, the longer the residence time the higher the recovery of the valuable mineral will be but the grade will be lower due to the gangue minerals that will be floated along side with it. This can be clearly shown in Figure 4.7 where 4 minutes rougher and 8 minutes scavenger has a copper recovery and grade of 88.9% and 21.2%TCu, respectively. While for the residence times of 5 minutes rougher – 9 minutes scavenger and 6 minutes rougher – 10 minutes scavenger the corresponding recoveries are 85.5% and 85.3 respectively. 7 minutes rougher – 11 minutes scavenger significantly decreased in both recovery and grade because the mineralized sulphide froth

which was carefully skimmed off became partially barren meaning that what remained in the flotation cell were literally tailings.

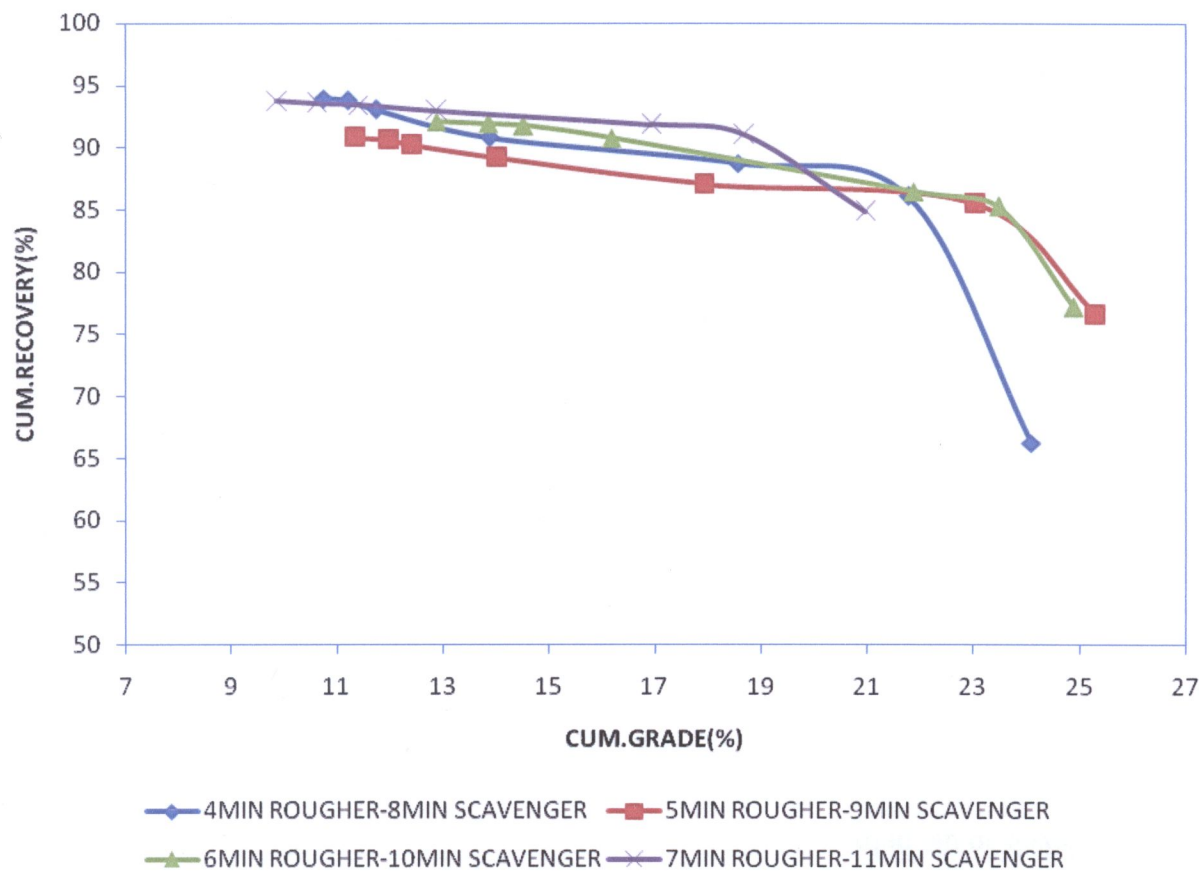


Figure 4.7: Relationship between cumulative copper recovery and grade at four different residence times.

However, the 4 minutes roughing and 8 minutes scavenging with recovery and grade of 88.9% and 21.2% respectively is relatively better than the other graphs. The other graphs show good recoveries which could be nearly the targets of the plant, but these cannot be the optimum residence time because of the lower grades obtained. The low grades could have been due to entrainment of gangue particles as sufficiently hydrophobic surface conditions could not be developed on the particle surfaces to overcome the hydrophilic tendency, thus lowering the floatability. These could also have been due to:

,

(a) When batch laboratory flotation tests are carried out, a number of operations are undertaken and consequently experimental errors are inevitable. Thus a number of duplicate flotation tests always produce results which show some significant variance, as was the case in this test work. Some of the operations, which could have led to the discrepancies, are:

(i) Variation in trying to skim or scoop the froth volume at a constant rate after every ten minutes. Hence, different concentration weights and grades of the repeat tests were obtained.

(ii) Continuous variations in the pulp density from the beginning to the end of flotation as solids were continuously being removed with the froth and water added to maintain the cell pulp level. This resulted in a change in concentration of all the reagents and pH levels in the pulp.

CHAPTER FIVE

5.1 CONCLUSION

It can be concluded that;

- 1) The mesh of grind was determined to be 25 minutes grind time. This gave copper recovery of 94.8 % with a grade of 24.4% at a fixed residence time of 4 minutes rougher and 8 minutes scavenger.
- 2) The optimum residence time was determined to be 4 minutes rougher and 8 minutes scavenger, with copper recovery of 88.9% with a grade 21.3% at constant grind time of 25 minutes.

5.2 RECOMMENDATIONS

- It is recommended that for an increased grind time, 25 minutes should be the optimum while for increased residence time, 4 minutes rougher and 8 minutes scavenger flotation time should be maintained.
- Mineralogical examinations should be carried out on the Mufulira copper ore together with its associated concentrates and final tailing. This will aid not only in positive identification of minerals but also the behavior of the same constituent minerals or how they will respond to ore treatment.
- Since the optimum grind time is dependent on the speed of mill, amount and size distribution of the grinding media, they must be a consistence in replacing the grinding media in order to improve the efficiency of grinding and also on the reduction of power consumption.
- Testworks should be done at a decreased grind time and decreased residence time to determine the floatability of the Mufulira copper ore.

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APPENDIX 1

REAGENT ADDITION CALCULATIONS

ADDITION OF COLLECTOR (SIPX)

Given,

1% solution strength

Dose rate of 30g/ton

Basing on 2000g of dry solids in the cell

$$\text{Volume of reagents} = \frac{[\text{Doserate} \times \text{Mass of solids}]}{[\%strength \times 10^6]}$$

$$= \frac{[30 \times 2000]}{[0.01 \times 10^6]}$$

$$= 6\text{ml}$$

Note; this calculation is employed also for the addition of PAX

APPENDIX 2

TABLES FOR SEIVE ANALYSIS

SEIVE ANALYSIS

15 MINUTES GRIND TIME

Sieve size (microns)	weight (g)	weight (%)	cum. weight %	% passing
250	20.00	3.86	3.86	100.00
180	57.00	11.00	14.86	89.00
150	37.00	7.14	22.01	81.85
125	47.00	9.07	31.08	72.78
106	40.00	7.72	38.80	65.06
90	52.00	10.04	48.84	55.02
75	19.00	3.67	52.51	51.35
63	33.00	6.37	58.88	44.98
45	13.00	2.51	61.39	42.47
-45	200.00	38.61	100.00	3.86
Total	518.00	100.00		

20 MINUTES GRIND TIME

Sieve size (microns)	weight (g)	weight (%)	cum. weight %	% passing
250	3.00	0.60	0.60	100.00
180	21.00	4.17	4.77	95.83
150	27.00	5.37	10.14	90.46
125	43.00	8.55	18.69	81.91
106	43.00	8.55	27.24	73.36
90	61.00	12.13	39.36	61.23
75	22.00	4.37	43.74	56.86
63	44.00	8.75	52.49	48.11
45	17.00	3.38	55.86	44.73
-45	222.00	44.14	100.00	0.60
Total	503.00	100.00		

25 MINUTES GRIND TIME

Sieve size (microns)	weight (g)	weight (%)	cum. weight %	% passing
250	1.00	0.19	0.19	100.00
180	7.00	1.34	1.53	98.66
150	13.00	2.49	4.02	96.18
125	32.00	6.12	10.13	90.06
106	34.00	6.50	16.63	83.56
90	81.00	15.49	32.12	68.07
75	25.00	4.78	36.90	63.29
63	42.00	8.03	44.93	55.26
45	17.00	3.25	48.18	52.01
-45	271.00	51.82	100.00	0.19
Total	523.00	100.00		

30 MINUTES GRIND TIME

Sieve size (microns)	weight (g)	weight (%)	cum. weight %	% passing
250	0.00	0.00	0.00	100.00
180	2.00	0.39	0.39	99.61
150	5.00	0.98	1.37	98.63
125	14.00	2.75	4.12	95.88
106	26.00	5.10	9.22	90.78
90	80.00	15.69	24.90	75.10
75	30.00	5.88	30.78	69.22
63	50.00	9.80	40.59	59.41
45	21.00	4.12	44.71	55.29
-45	282.00	55.29	100.00	0.00
Total	510.00	100.00		

35 MINUTES GRIND TIME

Sieve size (microns)	weight (g)	weight (%)	cum. weight %	% passing
250	0.00	0.00	0.00	100.00
180	1.00	0.19	0.19	99.81
150	2.00	0.39	0.58	99.42
125	9.00	1.75	2.33	97.67
106	20.00	3.89	6.23	93.77
90	73.00	14.20	20.43	79.57
75	38.00	7.39	27.82	72.18
63	49.00	9.53	37.35	62.65
45	22.00	4.28	41.63	58.37
-45	300.00	58.37	100.00	0.00
Total	514.00	100.00		

40 MINUTES GRIND TIME

Sieve size (microns)	weight (g)	weight (%)	cum. weight %	% passing
250	0	0.00	0.00	100.00
180	1.00	0.20	0.20	99.80
150	1.00	0.20	0.40	99.60
125	5.00	0.99	1.38	98.62
106	15.00	2.96	4.35	95.65
90	52.00	10.28	14.62	85.38
75	25.00	4.94	19.57	80.43
63	61.00	12.06	31.62	68.38
45	27.00	5.34	36.96	63.04
-45	319.00	63.04	100.00	0.00
Total	506.00	100.00		

APPENDIX 3

TABLES AND GRAPHS ON FLOTATION PERFORMANCE

30MINUTES GRIND TIME 4MINUTES ROUGHER-8MINUTES SCAVENGER

[illegible]

35MINUTES GRIND TIME 4MINUTES ROUGHER-8MINUTES SCAVENGER

[illegible]

40MINUTES GRIND TIME 4MINUTES ROUGHER-8MINUTES SCAVENGER

[illegible]

[illegible][illegible]

15MINUTES GRIND TIME 6MINUTES ROUGHER-10MINUTES SCAVENGER

[illegible]

20MINUTES GRIND TIME 6MINUTES ROUGHER-10MINUTES SCAVENGER

[illegible]

25MINUTES GRIND TIME 6MINUTES ROUGHER-10MINUTES SCAVENGER

[illegible]

20MINUTES GRIND TIME 7MINUTES ROUGHER-11MINUTES SCAVENGER

25MINUTES GRIND TIME 7MINUTES ROUGHER-11MINUTES SCAVENGER

[illegible]

CUM.GRADE-RECOVERY vs FLOTATION TIME FOR TCu(5R-9S)

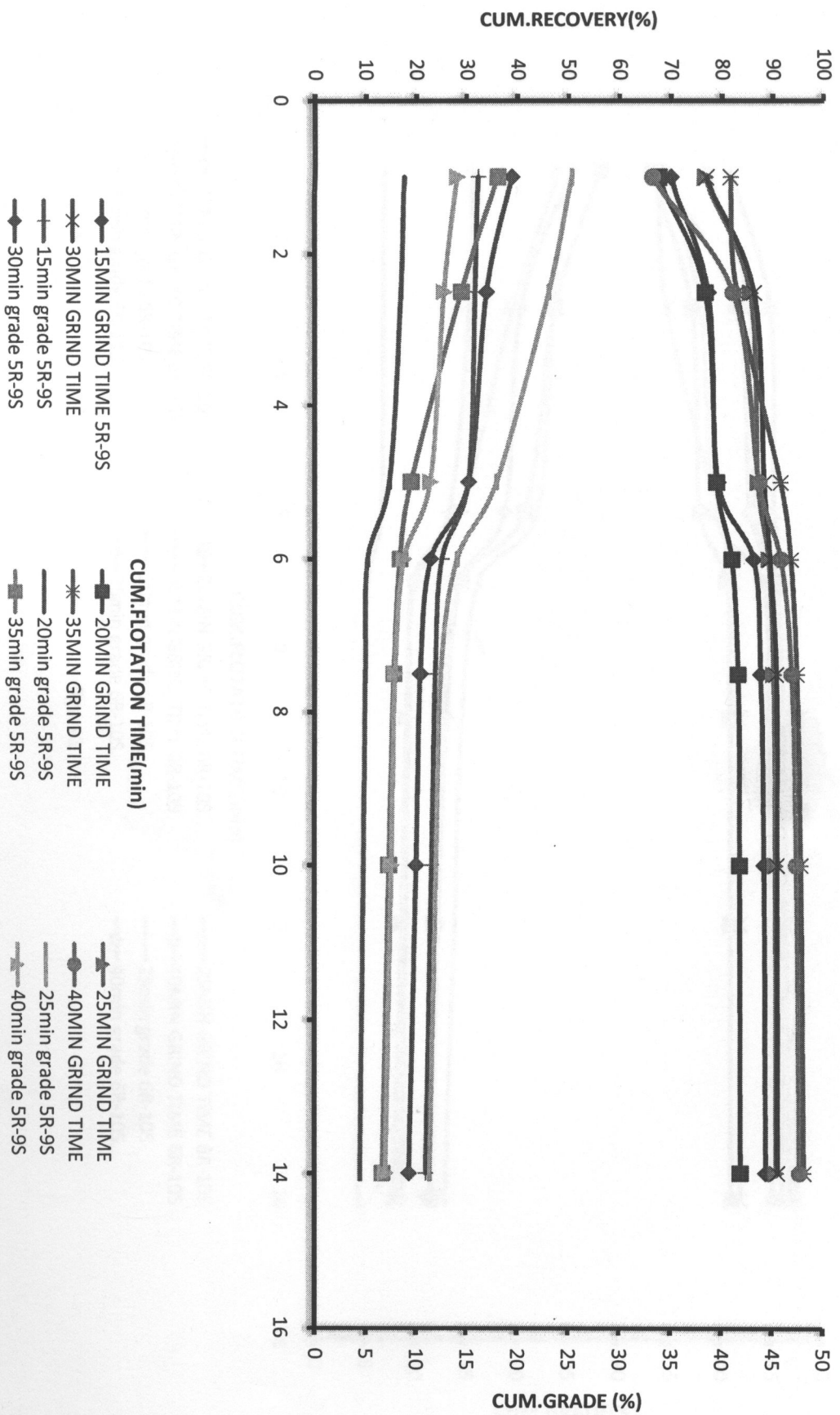


figure 1...

CUM.GRADE-RECOVERY VS FLOTATION TIME FOR TCu(6R-10S)

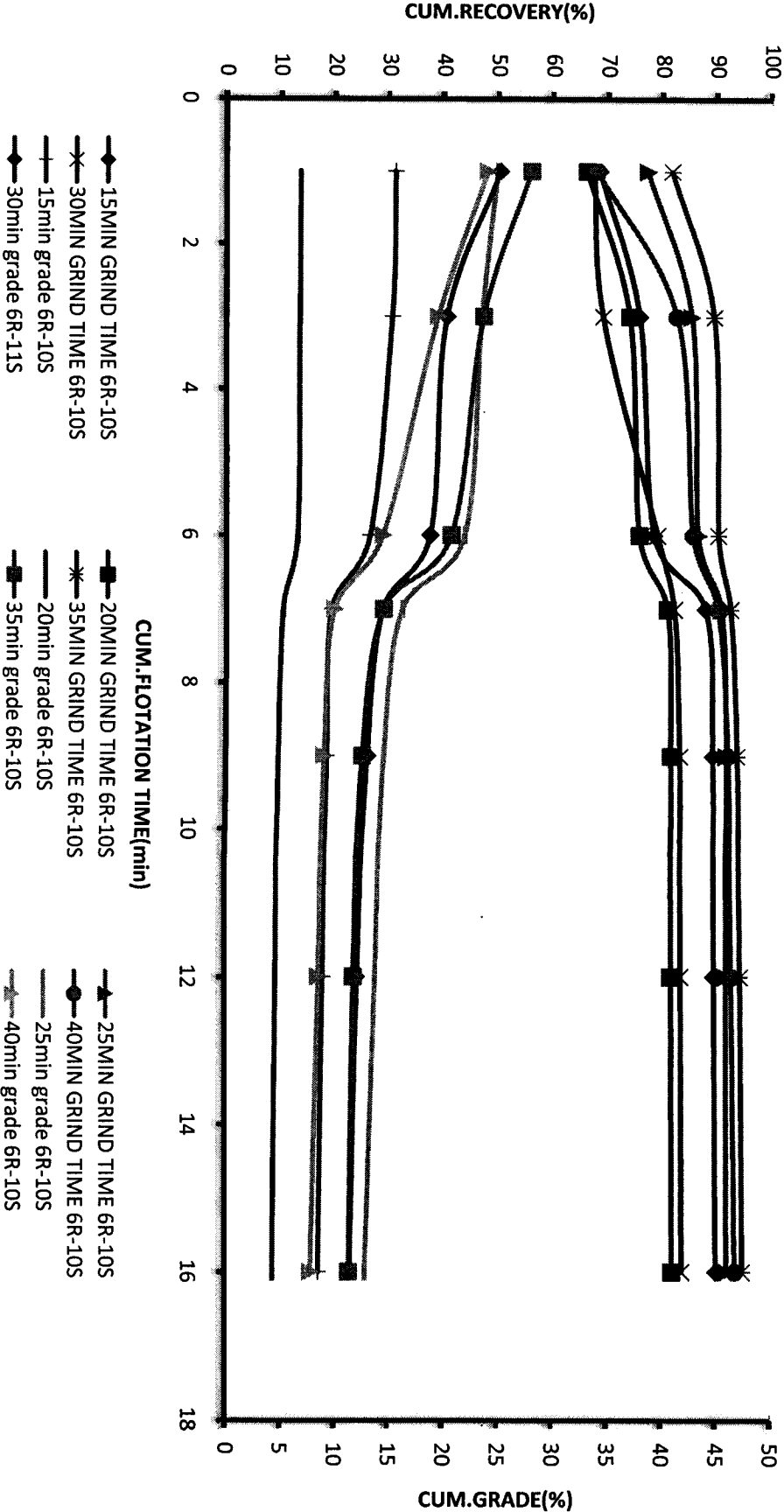


figure2

CUM.GRADE-RECOVERY vs FLOTATION TIME FOR TCu(7R-11S)

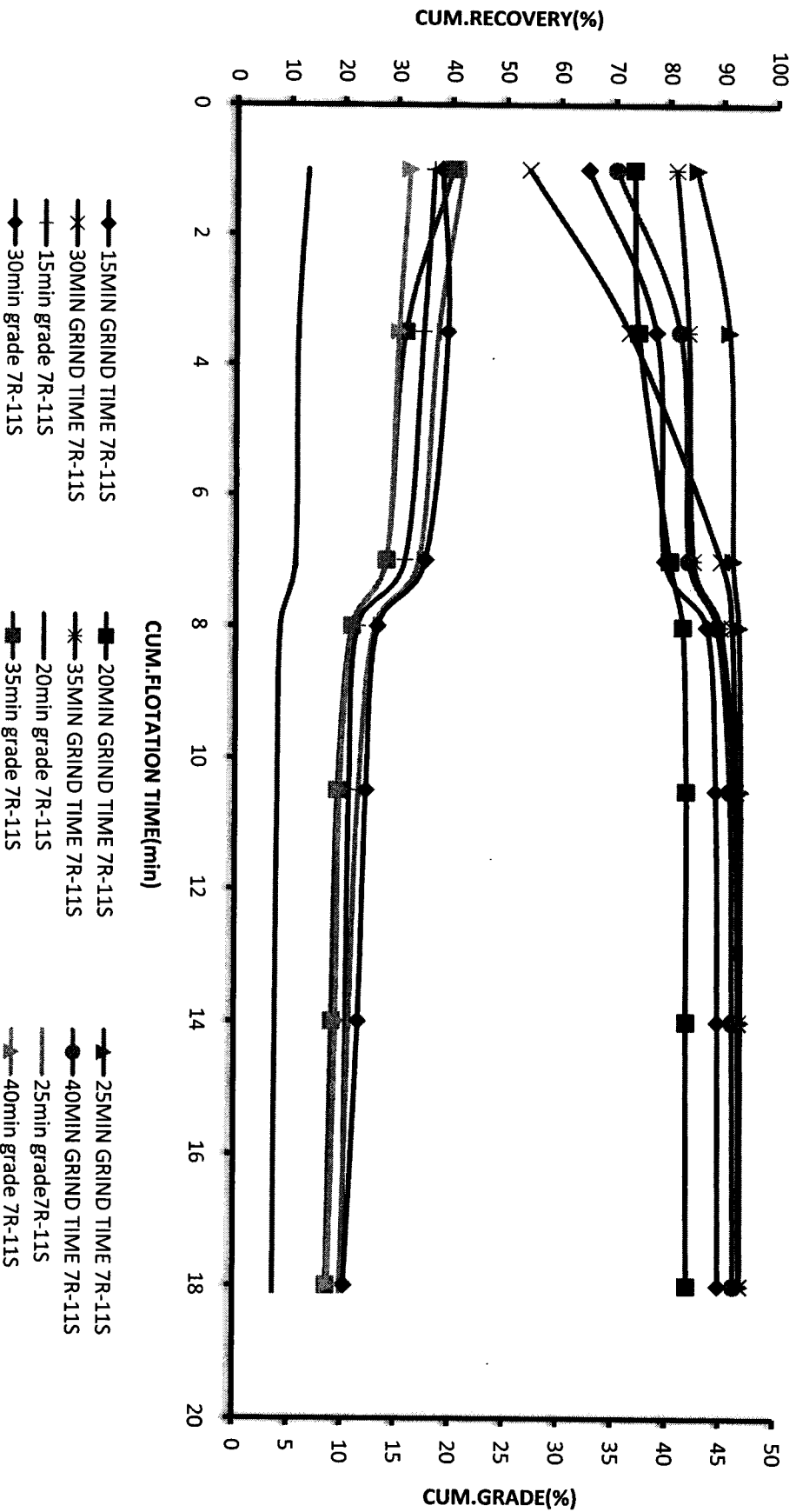


figure3

CUM.RECOVERY-GRADE FOR TCu(5R-9S)

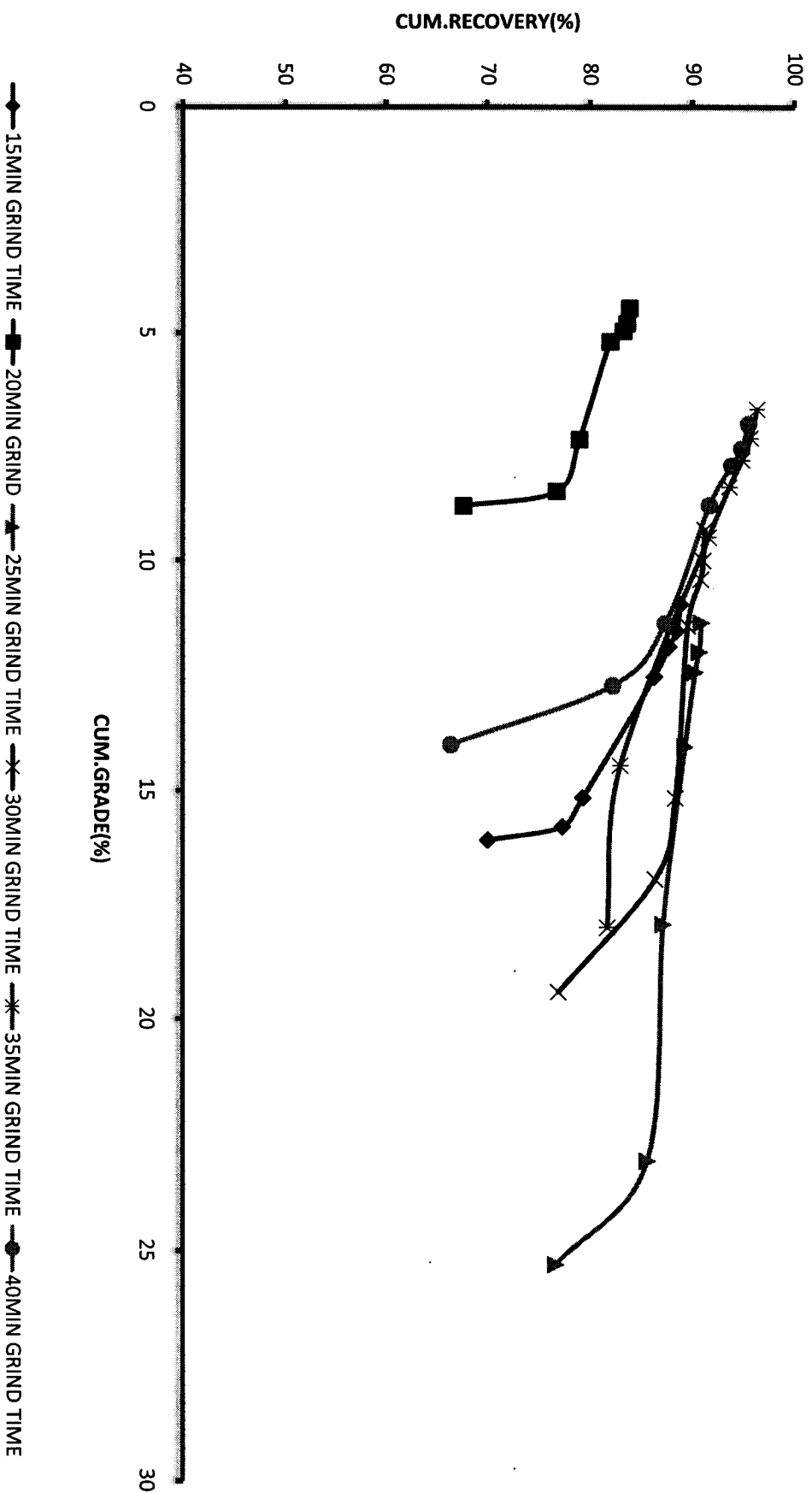
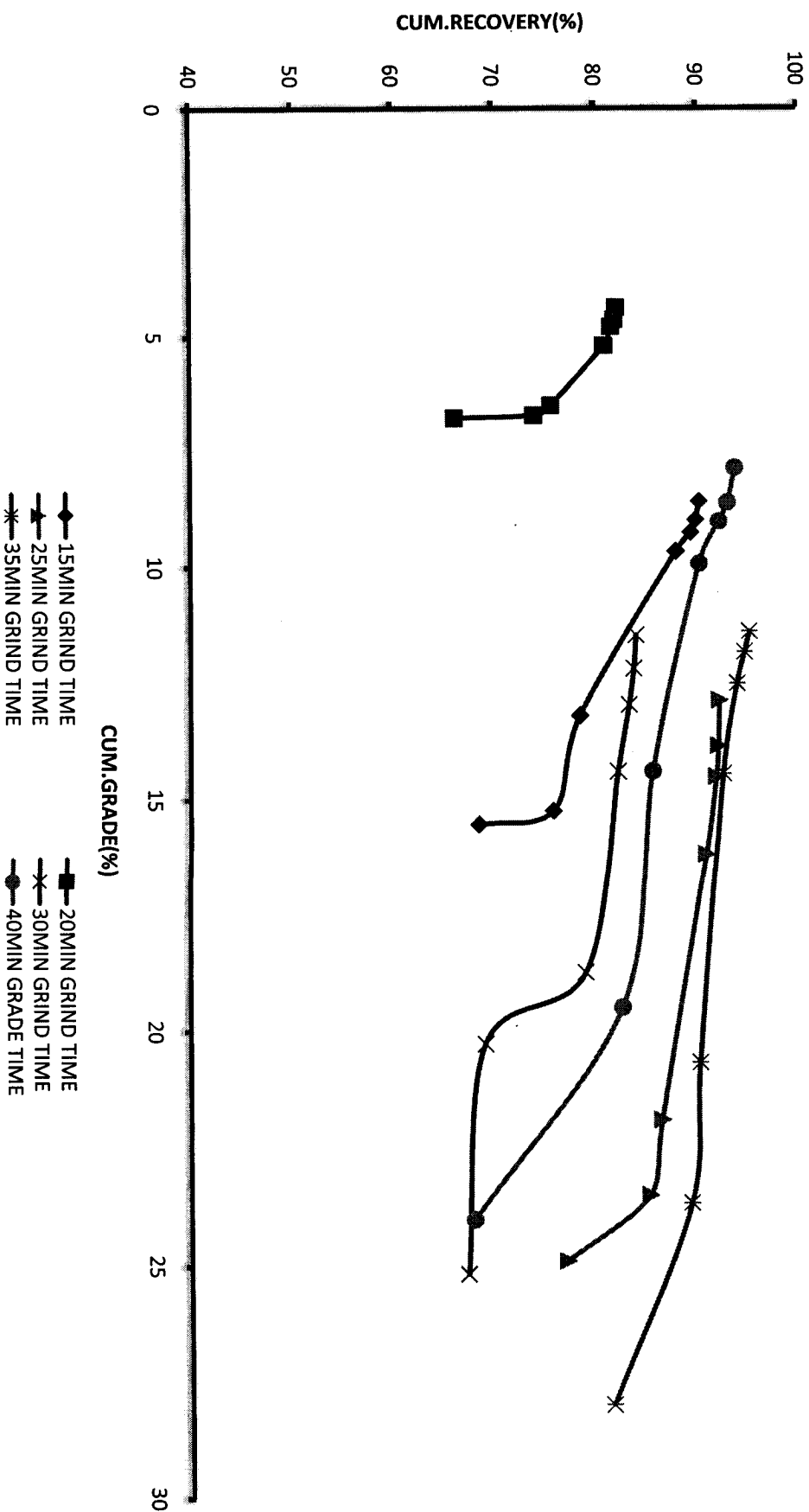


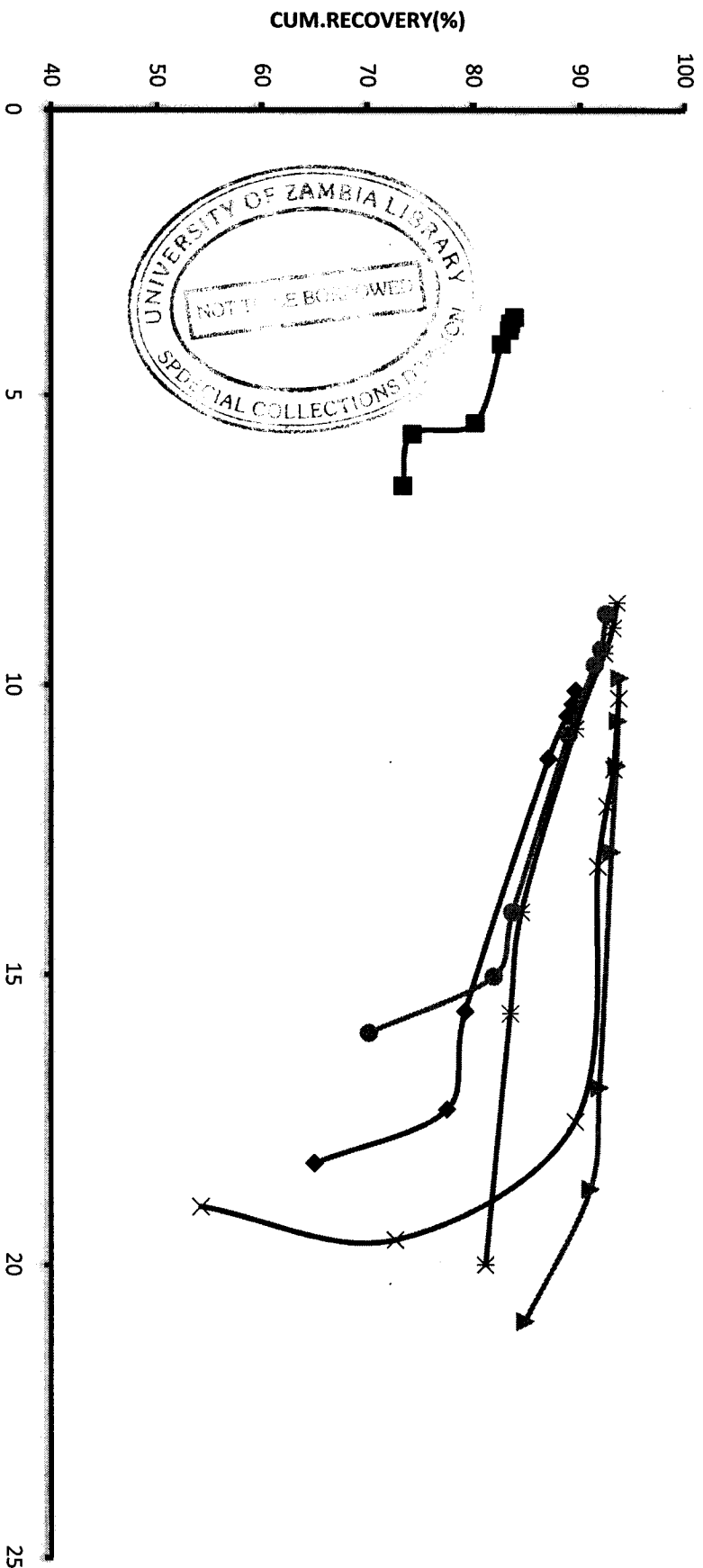
figure 4

CUM.RECOVERY-GRADE FOR TCu(6R-10S)



figures5

CUM.RECOVERY-GRADE FOR TCu(7R-11S)



◆ 15MIN GRIND TIME
 ▲ 25MIN GRIND TIME
 * 35MIN GRIND TIME

■ 20MIN GRIND TIME
 × 30MIN GRIND TIME
 ● 40MIN GRIND TIME

CUM.GRADE(%)

CUM.RECOVERY(%)

figure 6