Flotation Column Testwork at
Les Mines Camchib Inc.
Chibougamau, Québec

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ABSTRACT

A stage of reprocessing a reground flotation middling is currently being carried out at the Mines Camchib Inc. concentrator using three parallel banks of six 1.4 m^3 Fagergren cells. A research program was initiated by McGill University to determine the feasibility of replacing the Fagergren cells with a column flotation cell. Laboratory test work with a Leeds mechanical flotation cell showed that pre-aeration with sodium sulphite, a pyrite depressant, yielded the best selectivity. The actual comparison was performed with a 5.7 cm diameter column installed with a PI controller to maintain a stable interface. A poorer separation resulted from the use of the column than the plant cells. This poorer performance may be attributed to a significantly lower air addition rate than in the plant cells (approximately 8 times larger). This lower rate may be inadequate for pyrite depression.

A 3.8 cm diameter column was first tested in both batch and continuous operation. The interface level was controlled by allowing part of the column tails to overflow through a 3 cm pipe running up the column to a location slightly below the desired interface level. This method of control did not produce a stable level in continuous mode. As a result, a more conventional approach using a PI controller was employed. The interface level was controlled by adjusting the column tails flowrate such that a constant hydrostatic pressure in the column results.

Résumé

Une étape de retraitage d'un produit mixte rebroyé se fait au concentrateur des Mines Camchib Inc. dans trois bancs de six cellules mécaniques Fagergren de 1.4 m^3 chaque. Un programme de recherche avec des colonnes de flottation de petite taille a été conçu pour évaluer la faisabilité de remplacer ces cellules par une colonne. La séparation de la chalcopyrite de la pyrite était particulièrement difficile; des essais en laboratoire avec une cellule Leeds ont indiqué que la pré-aération avec déprimant, le sulfite de sodium, maximisait la sélectivité de la flottation.

Nous avons utilisé une première colonne de 3.8 cm de diamètre en discontinu d'abord, puis en continu. L'interface pulpe-écume était ajustée avec un tuyau de 3 cm de diamètre, par lequel les rejets s'écoulaient. Cette technique a permis d'obtenir une interface très stable en discontinu, mais très instable en opération continue. La comparaison de la performance des cellules mécaniques avec la flottation en colonne s'est finalement faite avec une colonne de 5.7 cm de diamètre, dont l'interface était ajustée par une boucle de régulation PI qui maintenait la pression hydrostatique de la colonne constante en modifiant le débit des rejets. Avec ou sans déprimant, la colonne a produit une séparation inférieure à celle des cellules. Nous avons expliqué ce résultat par le taux d'addition d'air spécifique de la colonne, de huit fois inférieur à celui des cellules, et insuffisant pour déprimer la pyrite et activer la chalcopyrite adéquatement.

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CHAPTER 1

INTRODUCTION

1.1 Location and Mineralogy

The copper-gold operations Les Mines Camchib Inc. are located near Chibougamau, Québec, Canada, approximately 700 km north-northwest of Montreal. Geological studies by Hawkins (1960) indicated that the sulfide minerals in the Chibougamau region are mainly pyrite, pyrrhotite, chalcopyrite, and traces of sphalerite and galena. The pyrite-pyrrhotite ratio varies from deposit to deposit. The host matrix contains chlorite, quartz, carbonates (calcite, siderite, ankerite) with minor occurrences of chloritoid, actinolite, and talc. Gold is the major valuable component.

At the time of the study, the Camchib mill was treating the Cedar Bay and Meston ores. The Cedar Bay ore assays 0.10-0.18 oz/st Au, and 0.5-1.0% Cu while the Meston ore assays 0.22-0.45 oz/st Au, and 0.3-0.5% Cu. Meston is the dominant source of ore for the Camchib mill. Gold in the Meston ore occurs in two generations. First generation gold, about 25% of total gold, mostly associated with silicates, contains little silver, and is very fine (1-10 µm). Due to its fineness, complete liberation is impractical; cyanidation must assist flotation and gravity recovery. Second generation gold, associated with sulfides and silver-bearing, is very coarse, easily liberated, and responds readily to gravity recovery (Laplante et al., 1989).

1.2 Description of the Camchib Mill Circuit

In the Camchib Mill, the circuit incorporates gravity concentration, flotation, and cyanidation, as shown in Figure 1.1. After three stages of crushing, the third in closed circuit with 1.3 cm screens, the ore is ground in a 3.4 m x 4.0 m rod mill. The rod mill discharge is pumped to two 76 cm cyclones in closed circuit with two 3.1 m x 3.7 m ball mills (Côté, 1978). The cyclone overflow is fed to the flotation circuit. Gold is recovered from the ball mill discharge using sluices, Knelson concentrators, and a shaking table. The tails of the sluices and the Knelson concentrators are returned to cyclones. Details of gravity recovery are given in Liu (1989). The flotation circuit produces a copper-gold concentrate, assaying 17-19% Cu with 82% Au and 95% Cu recovery, for shipment to the Noranda Minerals Horne smelter and a gold-bearing pyrite concentrate for cyanidation. The cyanidation circuit also treats the coarse tails from the flotation circuit, produced by two stages of cyclones. Total gold recovery is 88%, of which 35-40% is in the gravity concentrate, 35-40% in the flotation concentrate, and 10% in the Merrill-Crowe precipitate (Laplante et al., 1989).

The Camchib conventional flotation circuit is divided into 4 parts: rougher, scavenger, cleaner, and retreater. The cyclone overflow is fed to four Denver DR 300 rougher flotation cells, and then to two six-cell banks of Denver DR 30. The rougher tails is distributed to four six-cell banks of Fagergren 66 scavengers. The scavenger tails are upgraded in a two-stage cycloning circuit (previously used to produce backfill), and the coarse fraction cyanided. The rougher concentrate is upgraded through two stages of two six-cell banks of Denver DR 24 cleaners. The scavenger

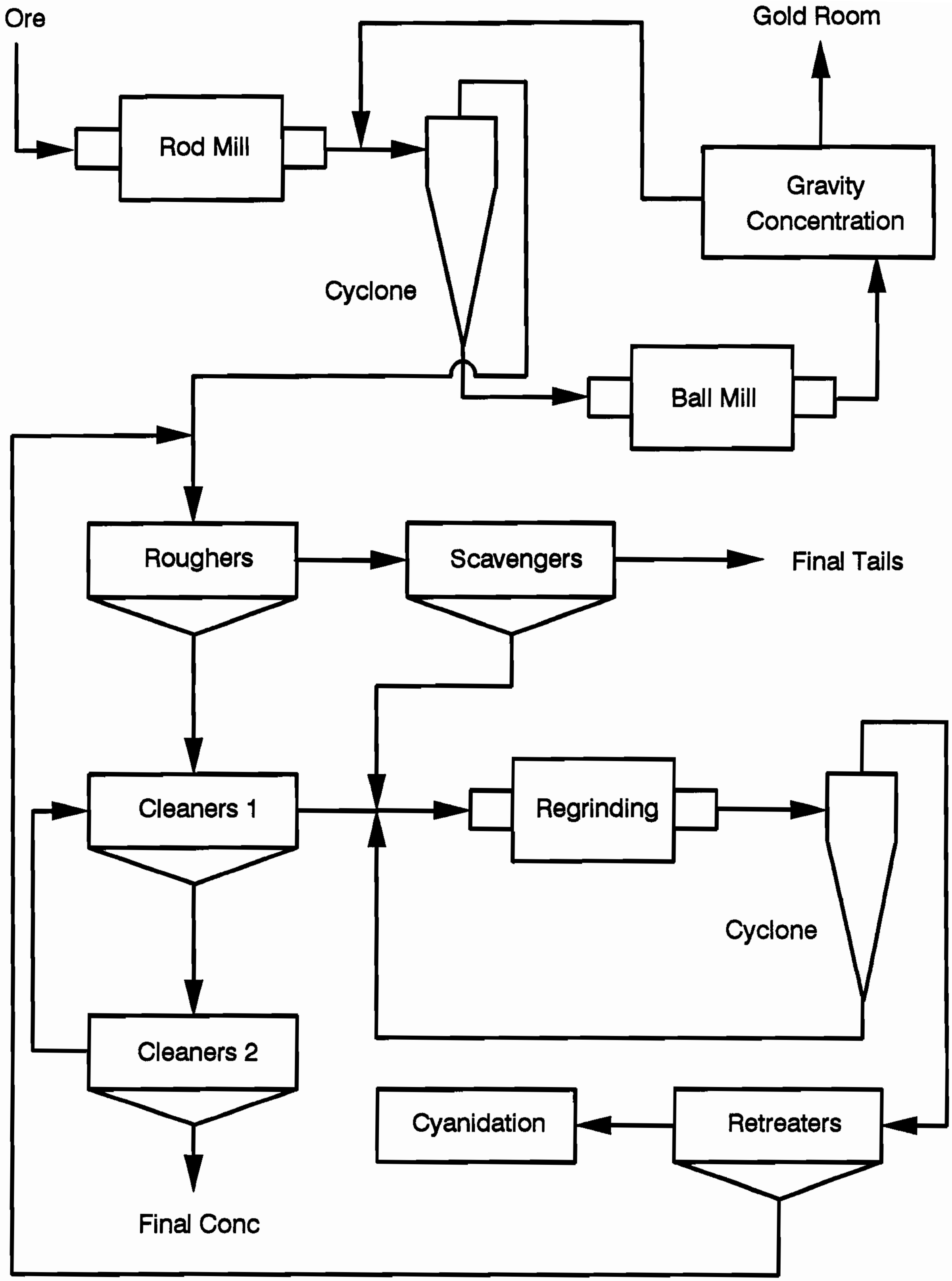

Figure 1.1 Camchib plant flowsheet

concentrate and the cleaner tails are cycloned in closed circuit with a 2.2 m x 2.2 m regrind mill to obtain better mineral liberation and create fresh surfaces needed for selective flotation; the cyclone overflow is then fed to three six-cell banks of Fagergren 66 retreaters for recleaning. Selective flotation in the retreater stage shows high copper and gold recoveries, in a low grade concentrate returned to the roughers. Gold-bearing iron sulfides in the tails are cyanided. At this point, the selectivity of chalcopyrite from gold-bearing pyrite in the retreater circuit should be as high as possible, because gold-bearing pyrite recovered in the retreater concentrate could be lost in scavenger final tails. It would then be too fine (because of the regrinding step) to be recovered in the coarse tails and would altogether escape cyanidation. Replacing the retreaters with a column is attractive if a higher concentrate grade can be achieved. First, less gold and pyrite would be returned to the head of the flotation circuit, resulting in lower gold losses. Second, these losses due to recycling could be altogether eliminated if retreater concentrate grade was high enough to be directed to the cleaner circuit. The flotation circuit would then be operated in such a way that any particle floated at least once would report either to the final concentrate or the retreater tails, to be cyanided. This can be achieved only if retreater flotation is selective enough to prevent the build-up of a pyrite circulating load in the retreater/cleaner loop, whilst keeping copper concentration in the retreater tails low enough for efficient cyanidation. Historical data at camchib suggest that the copper content of cyanidation feed below 0.08% is critical.

1.3 Objectives

Flotation columns have been claimed to give better separation than mechanical cells, particularly when operating on fine materials (Wills, 1988; Sivamohan, 1990; Soto, 1989). A significant development in recent years has been the increasing industrial use of flotation columns, primarily in Canada. Originally, most industrial flotation columns were used by large mining companies like INCO, Cominco, Noranda Metals, and Falconbridge. These companies have well-developed engineering groups capable of significant in-house developments, and in fact have contributed to the advancement of flotation column engineering (e.g. column flotation at INCO's matte separation plant, Feeley et al., 1987; Pb/Zn circuit at Cominco Sullivan Ltd., Egan et al., 1988 and Fairweather, 1991; Cu/Ni separation in cleaner stage at Falconbridge Ltd., Huls et al., 1989 and 1991). Small mining groups with less or no technical support have lagged behind. One objective of this project is to initiate a study with a smaller producer to demonstrate the applicability of column technology for a specific separation. The exercise would then be described in details to serve as a case study. The specific application is the replacement of the retreater cells with a column for increased selectivity. This is perceived as an ideal application of the column because of the fineness of the retreater feed (90% -53 µm). The technical objectives of this work are:

c. To determine how small a column can be and how simply the interface level of a column can be controlled whilst yielding reliable metallurgical results.

a. To study the metallurgical performance of flotation column and to compare with mechanical cells.

b. To optimize flotation selectivity by using pyrite depressants.

1.4 Organization of the Thesis

The thesis consists of 7 chapters. The theoretical background of column flotation fundamentals and sulfide flotation is briefly reviewed in chapter 2 and 3.

Chapter 4 describes preliminary testwork with a laboratory Leeds cell to justify the choice of iron sulfides depressant for good chalcopyrite selectivity from pyrite.

Chapter 5 describes the experimental set-up of a 3.8 cm diameter flotation column using a rubber pipe which runs up to a desired interface level, and presents testwork in batch and continuous operations. The addition of collectors is tested in batch operation. Continuous operation at the Camchib plant will reveal if stable operation can be achieved.

Chapter 6 describes the experimental set-up of a 5.7 cm diameter flotation column whose interface level is controlled with a pressure transducer to measure hydrostatic pressure above a given point and a PI controller whose 4-20 mA output drives a varistaltic pump. Column flotation is tested in batch operation with the addition of sodium sulfite. In continuous operation, the column runs parallel with the retreater mechanical circuit to compare metallurgical performance.

Most results are summarized and discussed in chapter 7. Conclusions and suggestions for future work are also presented.

CHAPTER 2

COLUMN FLOTATION FUNDAMENTALS

2.1 Introduction

The flotation column was invented in Canada in the early 1960s (Wills, 1988; Wheeler, 1983). The first commercial flotation column was installed in 1981 at Les Mines Gaspé (Québec, Canada) by Column Flotation Company of Canada. Three columns replaced 13 conventional cleaners in a molybdenum upgrading circuit, with superior results (Coffin and Miszczak, 1982; Moon and Sirois; 1983; Mathieu, 1972). In recent years the number of column flotation applications has increased substantially. For example, columns proved very effective on bulk Cu/Mo cleaning at Gibraltar Mines (Dobby et al., 1985), in the Pb circuit at the Polaris Mine (Kosick et al., 1988), for Pb/Zn cleaning at Mount Isa Mines (Espinosa-Gomez et al., 1989), and ultrafine coal at BHP-UTAH Coal Limited's Riverside Mine (Nicol et al., 1988). Pilot-plant test work has also shown superior performance of the column for oxide flotation, such as chromite (McKay, 1986).

The interest in column flotation is continuing to grow. Examples of improved metallurgy using column were described at the International Column Flotation Symposium in Phoenix, Arizona, in January 1988 (Ed. Sastry, 1988). A monograph detailing fundamentals of column flotation was recently published (Finch and Dobby, 1990). A forthcoming international conference on column flotation (Column '91) in Sudbury, Ontario, in June

1991, also testifies to the importance of columns.

It is probable that columns may find increasing use in the future for treating ores that need extensive fine grinding. The column is particularly attractive for applications involving multi-stage cleaning and can often upgrade in a single stage compared with several stages of mechanical cells. However, it is highly unlikely that full circuits will ever replaced with a single column stage.

2.2 Column Characteristics

A flotation column is shown schematically in Figure 2.1. The column differs dramatically from conventional mechanical flotation machines, both in design and operation. Three principal design features distinguish the column from a mechanical cell: first, the use of wash water; second, the absence of mechanical agitation; and third, the bubble generation system.

The column consists of two distinct zones, the collection zone (also known as the bubbling zone, pulp/slurry zone, or recovery zone) below the interface, where particles from the feed slurry are contacted countercurrent with a rising swarm of bubbles produced by a sparger near the bottom base. Floatable particles collide with and attach to the bubbles and are transported to the cleaning zone (also known as the froth zone and the secondary upgrading zone) above the interface. Non-floatable particles are removed from the bottom of the column as tailings. Wash water is added just above or below the lip of the column, thus providing a net downward liquid flowrate called a positive bias. The existence of a positive bias prevents gangue entrainment into the concentrate. Overall flotation column performance is influenced by interaction between the

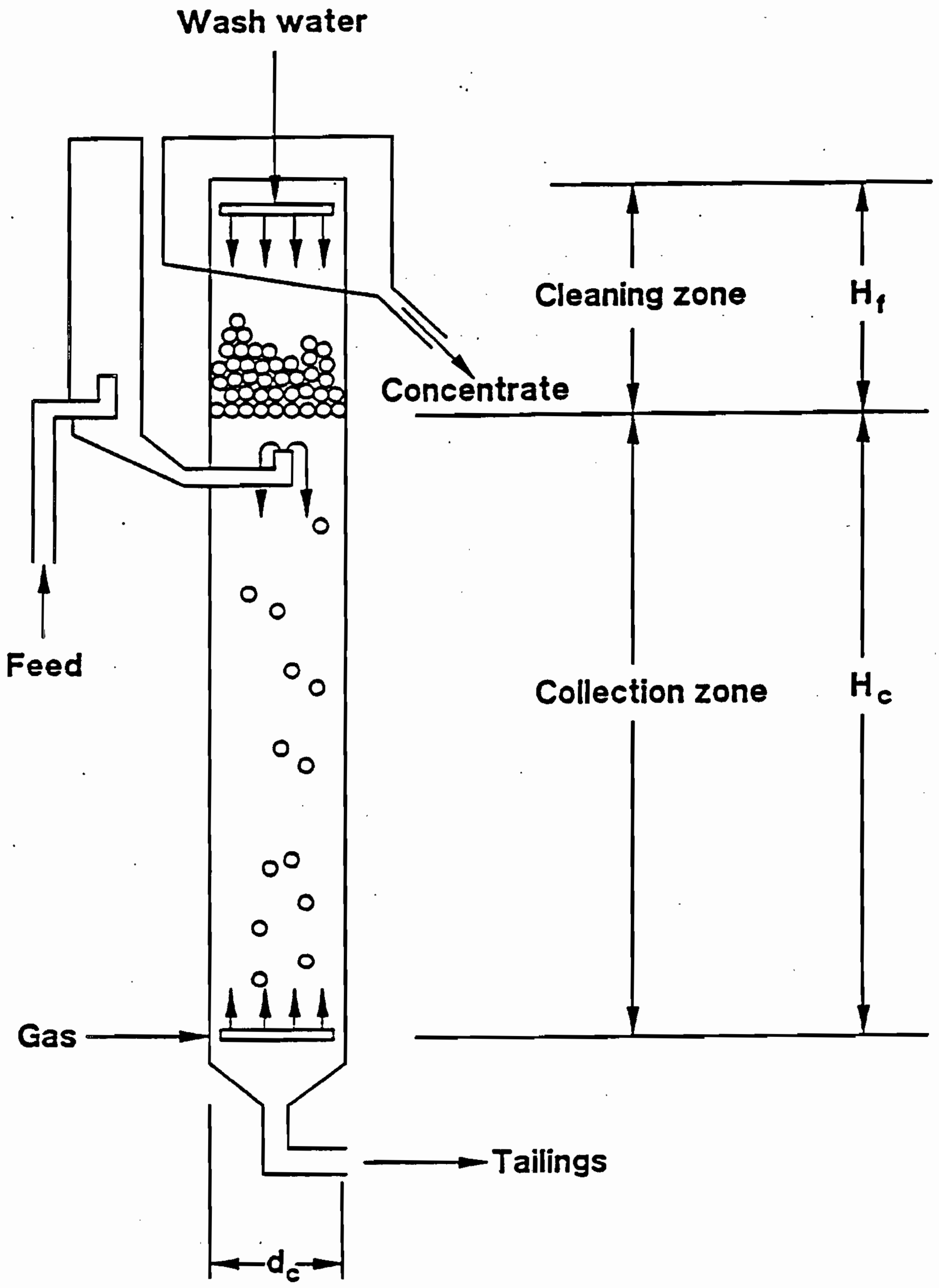

Figure 2.1 Schematic of flotation column.

collection and cleaning zones (Finch and Dobby, 1990).

2.3 Column Operating Variables

A large number of variables affect column performance. A description of these variables is given below.

2.3.1 Gas Holdup and Gas Rate

Gas is introduced into a flotation column as bubbles generated internally or externally. When gas is introduced into a column liquid or slurry is displaced. The volumetric fraction displaced is called the gas holdup, ε_g, which is used to characterize the hydrodynamics of the column. The complement $(1-\varepsilon_g)$ is the liquid holdup (if the solids holdup is negligible).

Superficial gas rate, J_g (cm/s), is commonly used to characterize operation since it can be compared for different column sizes. It is defined as the volumetric gas flowrate per unit column cross-section. Typical operating conditions are J_g = 1-3 cm/s (Yianatos, 1987).

Gas rate is one of the most important operating variables. It is usually found to have a pronounced effect on performance. The required gas flowrate must be determined in each case. The superficial gas rate should be kept as low as possible (normally below 1 cm/s) for better upgrading because of minimized entrainment and longer residence time of particles in the froth, promoting coalescence and better secondary upgrading (Yianatos et al., 1987).

Gas rate directly affects the flotation rate constant, k, and indirectly affects it by changing bubble size. This effect can be expressed from the dependence of k on J_g, bubble diameter, d_b, and bubble collection efficiency, E_k, as follow (Dobby et al., 1985):

$$k \propto \frac{J_g \; E_k}{d_b} \qquad (2.1)$$

Gas holdup varies directly with gas rate and inversely with bubble size. The maximum holdup is about 20-24% (Dobby et al., 1985). Above this, gas holdup becomes unstable and the flow rapidly changes from homogeneous bubbly flow to a churn-turbulent flow regime which is characterized by large bubbles rising rapidly, displacing water and small bubbles downward (Shah et al., 1982; Finch and Dobby, 1990).

The limiting gas rate can be determined by the deviation from a linear gas holdup versus gas rate relationship, called loss of bubbly flow. In addition, gas rate limitations can be also defined by the loss of interface or positive bias (Xu et al., 1989).

2.3.2 Wash Water

Wash water is essential for optimum column operation. It increases both froth stability and depth and displaces water from the recovery zone in the froth. The net effect is an elimination of the entrainment of fine gangue particles into the concentrate (Yianatos, 1987). Wash water is added into or above the froth, usually from an array of perforated pipes located just below the overflow lip to provide a downward flow of water which prevents hydraulic entrainment of non-floatable minerals.

An estimation of the wash water requirement, J_w (cm/s), is

$$J_w = \frac{J_g^* \; E_c}{1 - E_c} + J_b \tag{2.2}$$

where

$$E_c = \frac{J_c}{J_c + J_g^*} \tag{2.3}$$

and represents the fractional holdup of concentrate at the top of the column (Yianatos, 1987); J_g^* is the superficial gas rate at standard (atmospheric) conditions, i.e. at the column overflow, J_b the superficial bias rate and J_c the superficial concentrate rate. In plant operation typical E_c values are in the range 0.1-0.2. Figure 2.2 gives an estimate of wash water addition based on equations (2.2) and (2.3), with $J_b = 0.15$ cm/s, a typical value.

It has been experimentally observed that a strong increase in superficial wash water rate ($J_w > 0.5$ cm/s) can drastically change the near plug flow regime of the bubble bed to a more mixed behaviour including severe channelling and recirculation. The correct practice is to maintain sprinkling water for the whole range of water rates, typically 0.1-0.5 cm/s, rather than spraying or jetting (Dobby et al., 1985; Yianatos et al., 1986).

2.3.3 Bias Rate

Bias rate is defined as the net difference of water flow between the

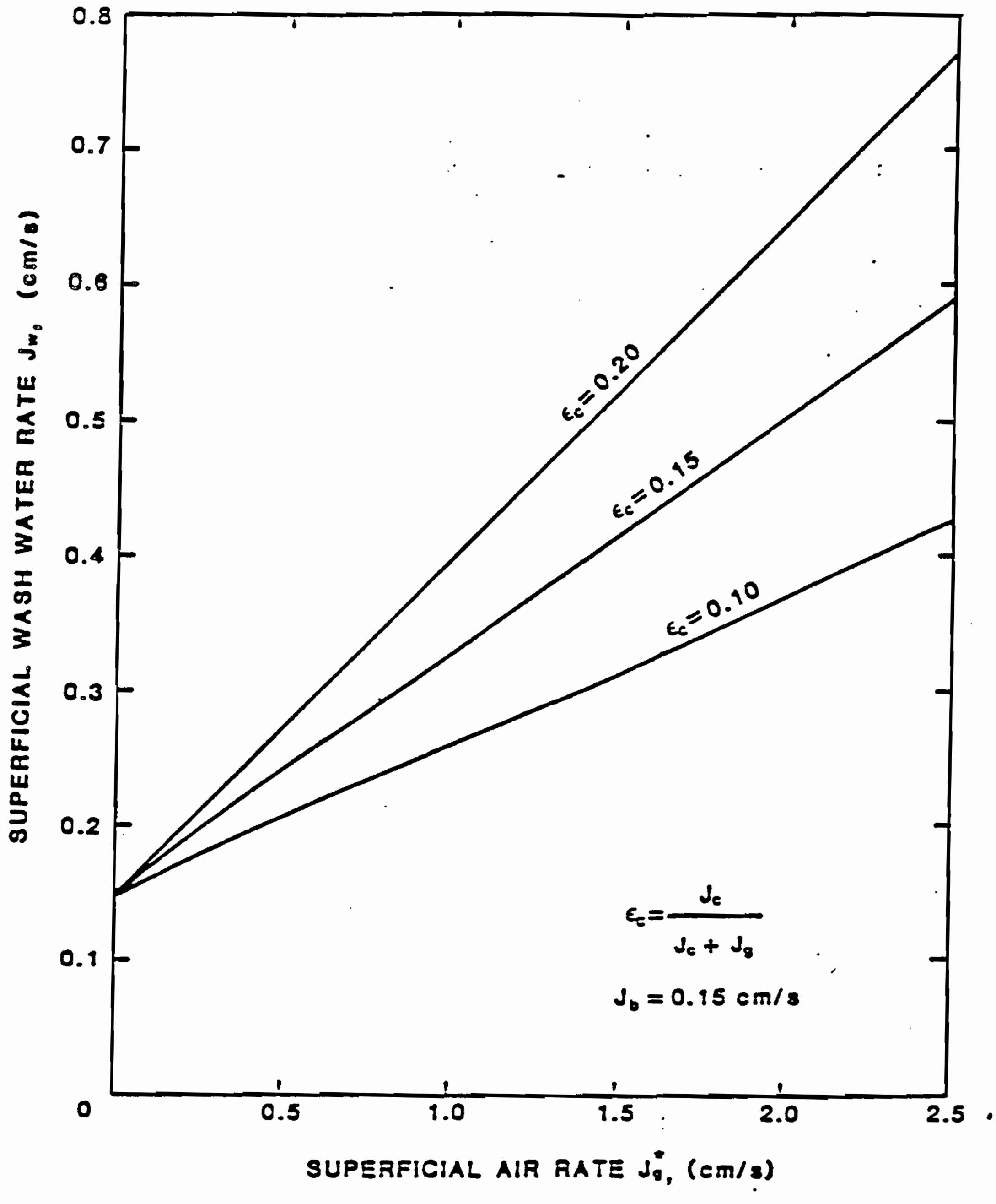

Figure 2.2 Wash water estimation (Yianatos, 1987).

tailings and feed. In superficial velocity (i.e. dividing by column cross-sectional area) this difference J_B is

$$J_B = J_T - J_F \tag{2.4}$$

Column flotation is normally performed with a positive bias, i.e. tailings flowrate is greater than feed flowrate. However, experimental work has shown that using a negative bias can be an attractive alternative to upgrade coarsely sized feeds (Soto, 1989).

Bias rate is one of the stabilizing control loops typically employed. Superficial bias rate J_B = 0.1-0.6 cm/s are used in plant practice (Yianatos, 1987). Nevertheless, excessive bias rates cause mixing in the upper froth region and loss of cleaning. Therefore, bias rate should be as low as possible to minimize water consumption and maximize collection zone retention time.

2.3.4 Collection Zone Height

Collection zone height has a significant impact on column performance. This zone must have sufficient length to provide adequate retention time for settling particles to attach to the rising bubbles (McKay et al., 1987). Illustrative experiments on column flotation of fluorite showed a decrease in recovery when particle retention time was not sufficient as the collection zone was shortened.

Yianatos et al. (1987) suggested that mineral recovery can be significantly improved by increasing the collection zone height to column diameter (H_c/d_c) ratio for a constant column volume and feed flowrate, while concentrate grade decreases only marginally. A common H_c/d_c value is about 10:1.

2.3.5 Froth Depth

Froth depths in plant operation are around 1.0-1.5 m. At laboratory scale data have been reported in the range 0-1.0 m (Dobby and Finch, 1986a; Yianatos, 1987). No general rule has emerged regarding froth depth. However, froth depth should be maintained while still conserving sufficient collection zone height to provide the proper particle retention time, i.e. to maintain mineral recovery (McKay et al., 1987).

Huls et al.(1989) examined the effect of froth depth upon metallurgical performance, and showed that the grade improved with a higher froth depth, but recovery dropped. Similar results are normally achieved with mechanical cells.

2.4 Collection Zone

2.4.1 The Particle Collection Process

A mineral particle in a flotation column is collected by a gas bubble through one of the two processes: (1) particle-bubble collision followed by attachment due to the hydrophobic nature of the mineral surface; or (2) entrainment of the particle within the boundary layer and the wake of the bubble (Finch and Dobby, 1990).

The collection process have been modelled in the column by considering a countercurrent of particle and bubble (Dobby and Finch, 1986b). Collection efficiency, E_K, is defined as the fraction of all particles swept out by the projected area of the bubble that collide with, attach to and remain attached to the bubble until reaching the cleaning zone. For a given system, collision efficiency, E_C, is the fraction of particles in the path of the rising bubble that collide with it and attachment efficiency, E_A, is the fraction of colliding particles that attach to it; therefore,

$$E_K = E_C \; E_A \tag{2.5}$$

The collection efficiency is a complex function of many parameters, the most obvious being particle size, bubble size, and particle hydrophobicity. Nevertheless, E_K is directly related to the collection rate constant, k, by

$$k = \frac{1.5 \; J_g \; E_K}{d_b} \tag{2.6}$$

where J_g is the superficial gas velocity, and d_b is the bubble diameter. The collection rate mechanism is first-order with respect to particle concentration. Figure 2.3 shows simulation results from Dobby and Finch (1986b) illustrating the effect of gas rate on collection efficiency, the rate constant, and bubble size. There exists a peak in collection rate constant for a range of gas rate.

Particle size plays an important role in collection efficiency. Figure 2.4 illustrates the relationship between collection efficiency and particle size for a given induction time. The first feature of note is

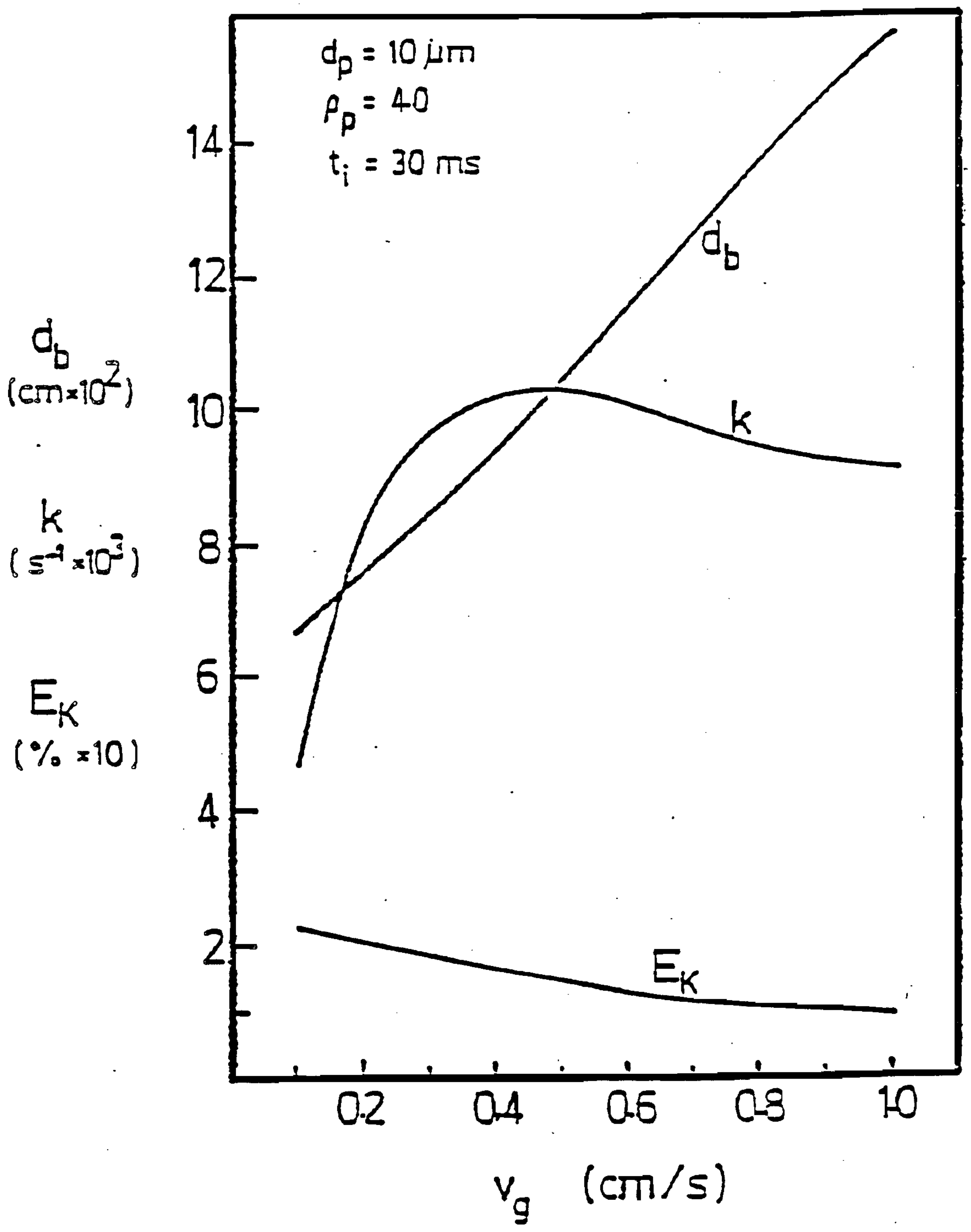

Figure 2.3 Simulation example of effect of gas rate on d_b, E_k, and K for opering system where $d_b = C \cdot V_g^n$, n = 0.53 d_p=10 µm, T_i=30 ms, V_l=0 (Dobby, 1984)

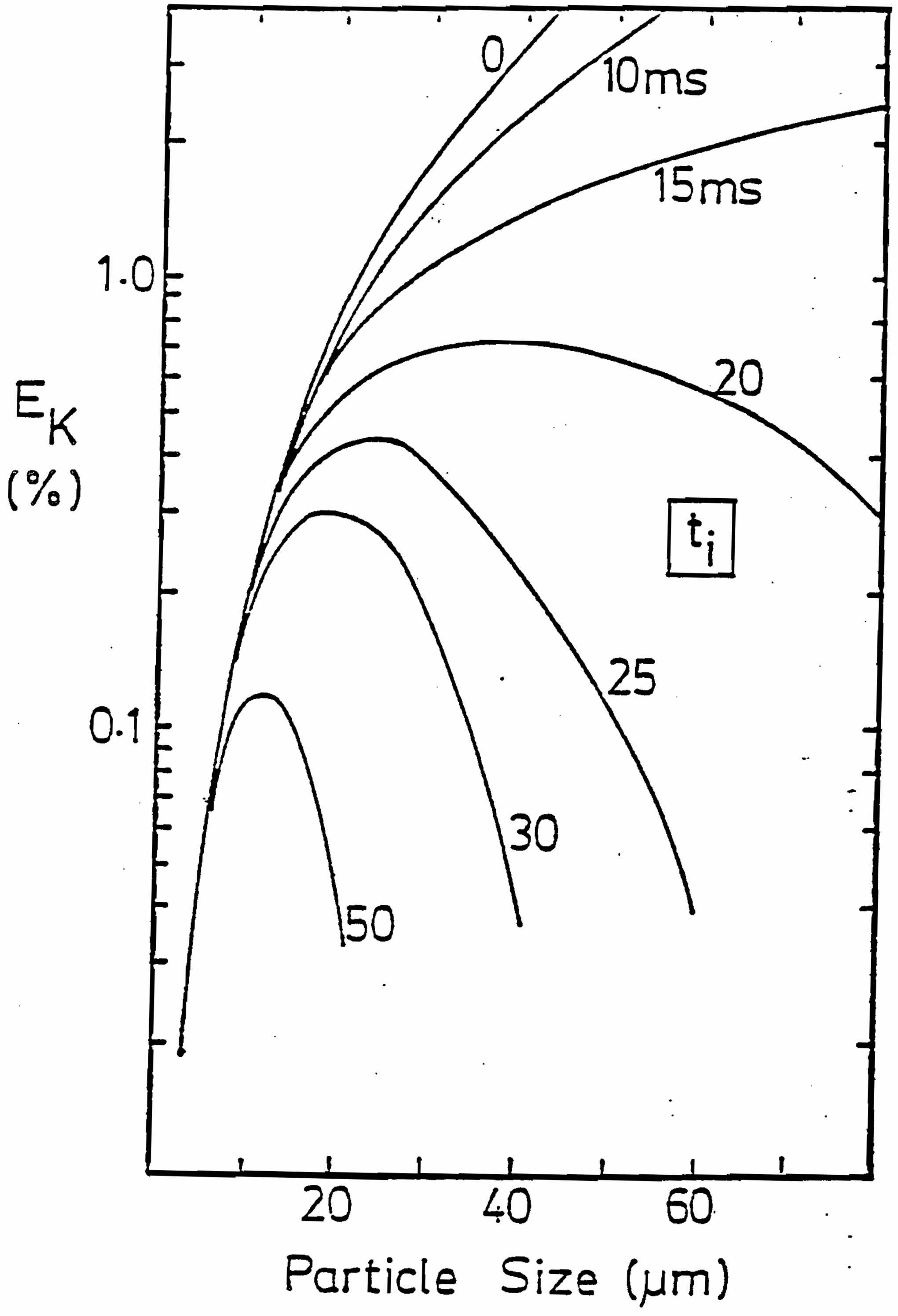

Figure 2.4 Collection efficiency vs. particle size and induction time for the flotation model d_b=0.1 cm, U_i=10cm/s, μ=0.01 poise particle s.g. = 4.0, gas holdup ϵ_g = 0 (Dobby, 1984)

that collection efficiency passes through a maximum. The maximum is explained by the opposing effect of particle size upon collision and attachment efficiency; as particle size increases, collision efficiency increases, but attachment efficiency decreases. The second feature is that the peak shifts to smaller particle size as induction time increases.

2.4.2 Mixing Characteristics and Recovery

The particle collection process in a column is considered to follow first-order kinetics. The recovery of a component from a first-order rate process depends on three variables: the rate constant, mean residence time, and the mixing conditions in the collection zone (Finch and Dobby, 1990).

One extreme of mixing is plug flow, where the residence time of all elements of the fluid (and all mineral particles) is the same. Plug flow in a column generates a concentration gradient of floatable mineral along the axis of the column. The other extreme is a perfectly mixed flow, where there is a distribution of retention time and the concentration is uniform throughout the system. For a first-order rate process with a rate constant k, exhibiting plug flow transport and having a retention time t, recovery R is given by

$$R = 1 - \exp(-kt) \tag{2.7}$$

and for a system exhibiting perfect mixing with a mean residence time τ

$$R = \frac{k\tau}{(1 + k\tau)} \tag{2.8}$$

The flow conditions in a laboratory flotation column approaches plug flow transport, while the liquid and solids in plant columns are transported under conditions between those of plug flow and perfectly mixed flow, but closer to the latter. Therefore, it is important to know the degree of mixing of flotation columns and to relate that mixing to recovery. The degree of mixing is quantified by the axial dispersion coefficient, E. The dimensionless vessel dispersion number, N_d, is given by

$$N_d = \frac{E}{u\ H_c} \tag{2.9}$$

where u is either the liquid interstitial velocity or the particle velocity, H_c is the collection zone of height. $N_d = 0$ corresponds to plug flow and $N_d = \infty$ corresponds to perfectly mixed flow.

The objective of measuring the mixing parameters is to quantify the mixing effect upon recovery. The relationship is given by

$$R = 1 - \frac{4\ a\ \exp\left(\frac{1}{2\ N_d}\right)}{(1 + a)^2 \exp\left(\frac{a}{2\ N_d}\right) - (1 - a)^2 \exp\left(\frac{-a}{2\ N_d}\right)} \tag{2.10}$$

where

$$a = (1 + 4 k \tau N_d)^{\frac{1}{2}} \qquad (2.11)$$

Dobby and Finch (1986b) examined the mixing characteristics in the collection zone of flotation columns by a uniaxial dispersion model and proposed that the dispersion coefficient of the solid particles, D_p, is the same as that of the liquid, D_l, and is linearly dependent upon column diameter, d_c, expressed as

$$E_1 = E_p = 0.063\, d_c \left(\frac{J_g}{1.6}\right)^{0.3} \qquad (2.12)$$

Mixing conditions were re-examined by Laplante et al. (1988). They suggested that $E_l \alpha d_c^{1.3}$ and that E_l decreases as slurry density increases.

2.5 Froth Zone

2.5.1 Froth Structure

The flow conditions in the froth zone are not well understood; they are dependent upon such factors as gas rate, bias rate, solid content, and the split between wash water reporting to concentrate and tailings.

Yianatos et al. (1986) proposed the froth structure shown in Figure 2.5 based on the experimental observation and measurements in the two dimensional column. Basically, the froth zone consists of three regions: (a) an expanded bubble bed (next to the interface), (b) a packed bubble bed above that, and (c) a conventional draining froth above the wash water

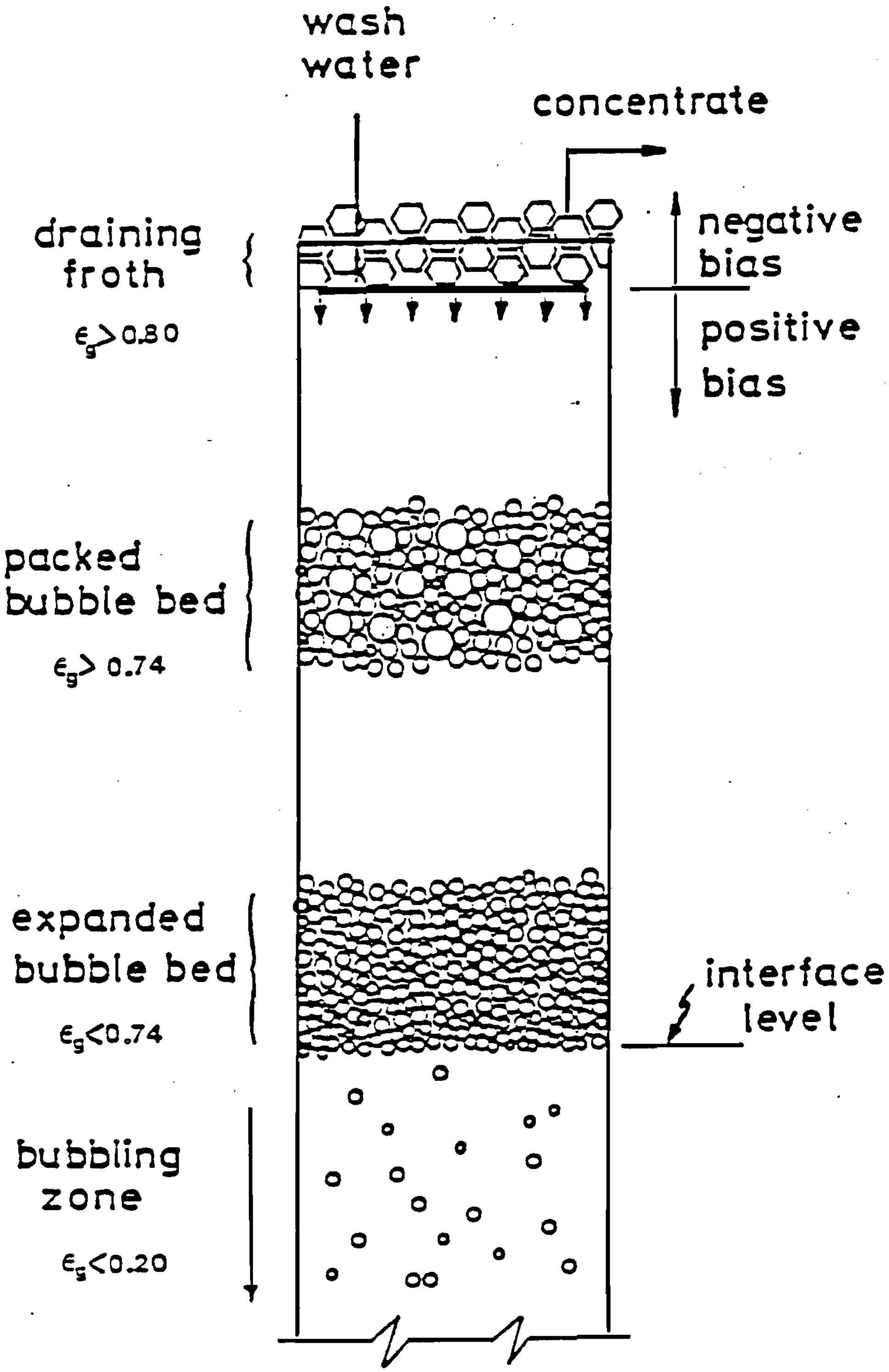

Figure 2.5 Froth structure (Yianatos et al, 1986).

inlet.

(a) The expanded bubble bed: bubbles travel upward from the collection zone and enter the expanded bubble bed after collision with the first layer of bubbles which define a very distinct interface. At this stage, the bubbles have a small homogeneous size and remain spherical. Bubble collisions against the interface generate shock pressure waves which promote additional collisions up through the expanded bubble bed region. This phenomenon seems to be the main cause of bubble coalescence in a region where the high fractional liquid content ($(1-\varepsilon_g)>0.26$) makes natural drainage unlikely as the primary cause of film thinning and rupture.

(b) The packed bubble bed: this region extends to the wash water inlet level. The fractional liquid content is lower than 0.26 and bubbles remain relatively spherical but with an increasing range in size. Most bubbles move upward close to plug flow against a well distributed flow of wash water. In this region the rate of coalescence is lower and appears to be mainly due to collisions caused by the faster rising larger bubbles.

(c) The conventional drainage froth: this region occurs immediately above the wash water inlet level and consists of a froth draining conventionally -- i.e. as the froth of mechanical cells. The main purpose of this region is to convert vertical motion into horizontal motion to recover the solids. Typical fractional liquid contents are lower than 0.2.

2.5.2 Froth Dropback

Dropback is the fraction (or percentage) of the solids entering the froth that are rejected back into the collection zone. Falutsu and Dobby (1989) investigated the froth dropback in a laboratory column by isolating the froth zone from the collection zone. Recovery in the froth zone was found to be more affected by gas rate. At low gas rate (<1.2 cm/s) the froth zone recovery becomes more sensitive to changes in froth depth or froth dosage.

Figure 2.6 shows the interaction between the cleaning and collection zones. Let R_c be the collection zone recovery and R_f the froth zone recovery (drop back then is $1-R_f$). Thus, the overall flotation column recovery R_{fc} is given by

$$R_{fc} = \frac{R_c R_f}{R_c R_f + (1-R_c)} \tag{2.13}$$

which assumes that the collection rate constant is the same in subsequent recollection events as in the original collection event (Finch and Dobby, 1990).

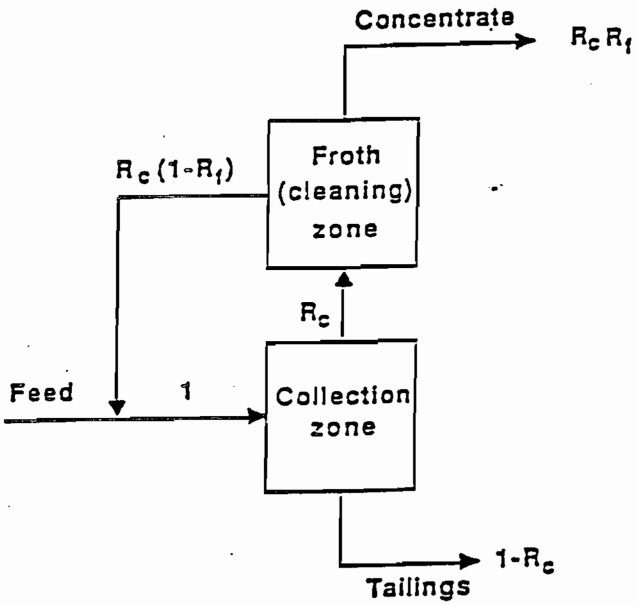

Figure 2.6 Interaction of the two zones.

CHAPTER 3

BRIEF SURVEY OF SULFIDE FLOTATION

3.1 Flotation Mechanisms

The property responsible for the attachment of a mineral particle to an air bubble is either a natural or an acquired characteristic of the mineral surface. Three modes of flotation are recognized:

(1) natural flotation: molybdenite is the most important mineral possessing this property naturally. The hydrophobic character of MoS_2 originates from the crystal structure of the mineral and its preferred cleavage plane. The weakest bond is the S-S bond and the crystal is broken between two sulphur layers upon impact in the size reduction step. The sulphur atoms, which are hydrophobic, are then exposed on the surface.

(2) collectorless flotation: the mineral surface is made hydrophobic by electrochemical interaction with the environment. This can be induced through oxidation-reduction in the flotation pulp by aeration, nitrogenation (Rao and Finch, 1987), and oxidizing or reducing agents, such as sodium sulphite, or sodium hypochlorite (Johnson et al., 1982; Heyes et al., 1987). Most sulfide minerals respond to collectorless flotation to varying degrees.

(3) flotation induced by a collector: for sulfides except molybdenite as well as non-sulfide minerals the surface condition required for flotation is obtained by specific reagents called collectors. Collectors are made up of a polar group, and one or more non-polar

hydrocarbon groups. When attached to the mineral particle, such collector molecules are oriented so that the non-polar group extends outward. This results in the formation of a hydrophobic hydrocarbon-type film at the mineral surface.

3.2 Electrochemistry of Flotation

The interaction of sulfide mineral with collectors was first explained by Salamy and Nixon (1953, as reported by Woods, 1976). Electrochemical mechanisms are based on charge transfer reactions between the mineral and collectors, or other chemical species in solution or at the surface of solids in the slurry. Such reactions are best known in sulfide mineral systems. They have been recognized in the following three situations:

(i) interaction of sulfide mineral with oxygen dissolved in water leading to the formation of elemental sulfur.

(ii) interaction of sulfide mineral with thiol type collectors, i.e. xanthate, leading to the formation of an adsorbed metal-xanthate complex or the formation and adsorption of dixanthogen.

(iii) interaction of the mineral with the griding medium (galvanic interaction) and also between two sulfide minerals which could chemically alter the mineral surface and affect its flotation properties.

1. Interaction of sulfide mineral with oxygen dissolved in water.

Most sulfide minerals interact with oxygen dissolved in water by charge transfer mechanism of the type:

$$MS + H_2O \Longrightarrow MOH + H^+ + S + e \qquad \text{(anodic component)}$$

$$H_2O + \tfrac{1}{2}O_2 + 2e \Longrightarrow 2OH^- \qquad \text{(cathodic component)}$$

$$2MS + H_2O + \tfrac{1}{2}O_2 \Longrightarrow 2MOH + 2S \qquad \text{(overall reaction)}$$

The oxidation reaction results in the formation of metal hydroxides and elemental sulfur on the mineral surface. Elemental sulfur renders the mineral floatable, i.e. it makes mineral surface hydrophobic.

Buckley et al. (1985) mentioned that oxidation involved the progressive removal of metal atoms leaving metal-deficient sulfides with sulfur lattices only marginally altered from the original structure. Sulfides with stoichiometries differing markedly from those of established stable sulfide phases have been identified. The overall general reaction suggested is

$$MS + xH_2O == M_{1-x}S + xMO + 2xH^+ + 2xe^-$$

2. Interaction of sulfide minerals with thiol type collector.

Chemisorption of thiol collectors at sulfide mineral surface has been recognized in the formation of hydrophobic film involving an oxidation step and reduction step -- i.e. charge transfer reactions which are facilitated on an electronic conductor acting as a source or a sink for electrons. Sulfide minerals are electronic conductors and fulfil this role (Woods, 1976).

There are two mains charge transfer reactions which are relevant to the flotation of sulfides.

(1) Adsorption of xanthate at sulfide mineral surfaces.

(2) Oxidation of xanthate to dixanthogen.

A reaction such as the oxidation of a xanthate to dixanthogen is

$$2ROCS_2^- + \frac{1}{2}O_2 + 2H^+ \quad ===> \quad (ROCS_2)_2 + H_2O$$

which occurs as the result of anodic reduction

$$2ROCS_2^- \quad ===> \quad (ROCS_2)_2 + 2e^-$$

and the cathodic reduction of O_2

$$\frac{1}{2}O_2 + 2H^+ + 2e^- \quad ===> \quad H_2O$$

In this way the sulfide surface can catalyze the formation of dixanthogen.

3. Galvanic Interaction

When a mineral is ground with iron balls there is interaction between their surfaces (Rao et al., 1976). This has been identified to be a charge transfer reaction. Such electrochemical reactions are known as galvanic interactions. An example is that of the reaction between mild steel and sphalerite where electrons are drawn from the metal to the mineral and transferred to oxygen by the reaction:

$$\frac{1}{2}O_2 + H_2O + 2e^- \quad ===> \quad 2OH^-$$

In this case, iron ions being released form precipitates of iron hydroxide which coat the sphalerite and hinder its flotation.

Kocabag and Smith (1985) demonstrated that the type of grinding mill, grinding media, and chemical environment within the mill have a considerable effect on the flotation of sulfide minerals due to the galvanic interactions.

Galvanic interactions also occur when two sulfide minerals are present together in an ore. The mineral which is more cathodic draws electrons from the less cathodic one. This alters the surface of the minerals and affects their floatabilities.

Among all sulfide minerals, pyrite is the most cathodic. When pyrite is in galvanic contact with other sulfide minerals, their oxidation will be enhanced, such as in the case of pyrrhotite - pyrite: pyrite will act as a cathode and pyrrhotite as an anode upon contact. Nakazawa and Iwasaki (1985) showed that the galvanic contact improved the floatability of pyrrhotite, but adversely affected that of pyrite. In this case, pyrrhotite may become hydrophobic due to the formation of elemental sulfur at the mineral surface.

3.3 Flotation of Chalcopyrite

Chalcopyrite is readily floated with sulphydryl collectors (Cooper et al., 1985; Glembotskii, 1963). Fuerstenau (1982) reported that complete flotation of chalcopyrite with ethyl xanthate is possible in the pH range of 3 to 12.

When xanthate is added as collector, chalcopyrite can react with xanthate by charge transfer reactions. These reactions may lead to cuprous xanthate by the reaction:

$$Cu(OH)_2 + X^- + e^- \implies CuX + 2OH^-$$

or to cupric xanthate by the reaction:

$$Cu(OH)_2 + 2X^- \implies CuX_2 + 2OH^-$$

where at longer times cupric xanthate is unstable and decomposes to cuprous xanthate and dixanthogen. There is a good correlation between flotation and the adsorption reactions and between the decrease in flotation and the reduction of the adsorbed xanthate product (Richardson et al., 1984).

In an electrochemical investigation, Allison et al. (1972) observed dixanthogen, a product of xanthate oxidation, on the surface of chalcopyrite.

It can be assumed that both electrochemical oxidation of xanthate to dixanthogen as well as chemisorption of xanthate on chalcopyrite are responsible for flotation.

Plaksin (1959) found that the floatabilities of sulfide minerals depended on the concentration of oxygen in solution. The recovery of chalcopyrite increased with increasing oxygen content in the water. Heyes and Trahar (1977) demonstrated that the natural floatability of chalcopyrite depended on the gas used for flotation and the presence of reducing or oxidizing agents in the pulp. Chalcopyrite was found to float naturally in the more oxidizing environments. Gardner and Woods (1979) confirmed that chalcopyrite became naturally floatable at oxidizing potentials. Presumably, superficial iron forms ferrous hydroxide and hydrophobic sulfur is formed at the mineral surface. Spira and Rosenblum (1978) mentioned that pre-flotation aeration of sulfide pulps assisted in the depression of pyrite and pyrrhotite and the activation of chalcopyrite. Therefore, the presence of oxygen in the pulp is essential for chalcopyrite flotation.

3.4 Flotation of Pyrite and Pyrrhotite

Pyrite is readily floated and, if not heavily oxidized, can be completely floated using frother only (Glembotskii, 1963).

The mechanisms by which pyrite is floated with sulphydryl collectors are well understood. The species responsible for pyrite flotation in the presence of short-chained xanthate is only dixanthogen (Fuerstenau, 1982;

Cooper et al., 1985; Ball and Rickard, 1976). The presence of dixanthogen on the pyrite surface was confirmed by infrared spectroscopy. Fuerstenau et al. (1968) proposed three possible electrochemical reactions for dixanthogen formation:

$$2X^- + \tfrac{1}{2}O_2 + H_2O \Longrightarrow 2OH^- + X_2$$

$$2Fe^{3+} + 2X^- \Longrightarrow 2Fe^{2+} + X_2$$

$$2Fe(OH)_3 + 6H^+ + 2X^- \Longrightarrow 2Fe^{2+} + 6H_2O + X_2$$

where X^- represents xanthate ion and X_2 represents dixanthogen. The first mechanism was ignored because oxidation of xanthate by dissolved oxygen has been shown to be negligible (Finkelstein, 1967). Ball and Rickard (1976) thus postulated that the oxidation of xanthate to dixanthogen was due to either Fe^{3+} or $Fe(OH)_3$.

It has been found that pyrite shows poor floatability in very acidic solutions, as xanthate becomes unstable and decomposes rapidly. Above pH 11 dixanthogen is not stable with respect to the xanthate ion. Thus, pyrite can be depressed due to the lack of dixanthogen (Ball and Rickard, 1976).

Another collector has been shown to function similarly in the pyrite system, and this is dithiophosphate. Dithiophosphate is more difficult to oxidize to dithiophosphatogen than xanthate is to dixanthogen (Woods, 1976, as reported by Cooper et al., 1985). That is,

$$X_{2(l)} + 2e^- \Longrightarrow 2X^- \qquad E^\circ = -0.06 \text{ volt}$$

$$(DTP)_{2(l)} + 2e^- \Longrightarrow 2DTP^- \qquad E^\circ = 0.25 \text{ volt}$$

where X^- and DTP^- represent xanthate and dithiophosphate, respectively.

Pyrrhotite has much in common with pyrite; however, the flotation properties of pyrrhotite are different (Glembotskii, 1963). Sutherland and Wark (1955) reported that by using potassium ethyl xanthate, the critical pH values - below which any given mineral will float and above which it will not - are 6 for pyrrhotite and 10.5 for pyrite. Therefore, the flotation of pyrrhotite with ethyl xanthate is more readily prevented with lime than that of pyrite.

Pyrrhotite can exist in its normal state only up to a certain maximum oxidation point; as soon as this point is reached, the crystalline state of pyrrhotite alters swiftly and suddenly, due to the formation of the twin sulfur anion. Pyrrhotite is then transformed into pyrite and subsequently oxidizes as pyrite.

In copper-pyrrhotite flotation, the copper minerals are not floated until pyrrhotite, which adsorbs large amounts of oxygen from the pulp, is oxidized completely. Thus, pulp aeration before flotation may have a considerable effect, depressing pyrite and producing favourable conditions for copper flotation (Glembotskii, 1963).

Nakazawa and Iwasaki (1985) showed that the flotation of pyrrhotite decreased with increased aeration and activity of oxygen. They reported that the decrease in pyrrhotite recovery was ascribable to the formation of a surface film, such as iron hydroxide, on the pyrrhotite.

CHAPTER 4

PRELIMINARY TESTWORK WITH A LEEDS CELL

4.1 Introduction

The Camchib ore has chalcopyrite associated to some degree with gold, pyrrhotite, and pyrite in a gangue that contains quartz and various silicates. Flotation separation of chalcopyrite and gold from the iron sulfides and silicate gangue is the objective of this work. Preliminary flotation was performed in a laboratory Leeds Cell, with samples of the retreater feed (RF1) averaging 0.3 % Cu and 0.3 oz/st Au, to test the effect of aeration rate and iron sulfides (mostly pyrite) depressants on selectivity.

4.2 Depression of Pyrite

Taggart (1945) mentioned that cyanide was a powerful depressant for pyrite. A detailed study was undertaken by Elgillani and Fuerstenau (1968) to establish which species in the pyrite-cyanide-xanthate system is responsible for depression and then to determine the conditions under which this species will adsorb. They found that ferrocyanide and cyanide were effective depressants for pyrite. When pyrite is added to a cyanide solution, the species $Fe(CN)_6^{4-}$ can form, and the distribution of uncomplexed ferrous ion and ferrocyanide ion can be calculated as a function of pH by means of the equation:

$$Fe^{2+} + 6HCN \Longrightarrow Fe(CN)_6^{4-} + 6H^+$$

This calculation showed that above pH 6.4, almost all of the iron exists as the ferrocyanide complex, $Fe(CN)_6^{4-}$, which is the species responsible for depression.

The formation of ferric ferrocyanide on the surface of pyrite is proposed to occur by the following electrochemical reaction (Elgillani and Fuerstenau, 1968; Leja, 1982):

$$7Fe^{2+} + 18HCN \Longrightarrow Fe_4[Fe(CN)_6]_3 + 18H^+ + 4e$$

This reaction represents the overall reaction which must occur in several steps. The mineral must produce some ferrous ion in solution which reacts with CN^- to form $Fe(CN)_6^{4-}$. Most pyrites yield sufficient iron in solution especially after oxidation. Cyanide becomes a much more effective depressant when pyrite has been oxidized, which supports this model of cyanide depression.

It was known that lime and cyanide when used jointly are effective depressants for iron sulfides. Recently, Agar (1989) investigated the optimum addition of lime and cyanide to maximize the separation of chalcopyrite from pyrrhotite.

Sulfur dioxide, SO_2, is also used as a sphalerite and pyrite depressant. The depression mechanism was explained by the formation of metal sulfites on the mineral surface, which inhibited collector adsorption (Bayzabi, 1979). Misra et al. (1985) suggested that the mechanisms were xanthate decomposition due to a SO_2-promoted oxidation reaction, and the retardation of collector adsorption due to a shift of the redox potential to an unfavourable range.

Sulfite is as well used as a pyrite depression. Since dixanthogen is the species responsible for pyrite flotation, reagents which are more reducing than the dixanthogen-xanthate couple should function as depressants for pyrite. The standard cell potential of the dixanthogen-xanthate half cell is -0,06 volt (IUPAC sign convention):

$$X_2 + 2e \implies 2X^- \qquad E^\circ = -0{,}06 \text{ volt}$$

while the standard cell potential of the sulfate-sulfite couple in basic medium is - 0,93 volt (IUPAC):

$$SO_4^{2-} + H_2O + 2e \implies SO_3^{2-} + 2OH^- \qquad E^\circ = -0{,}93 \text{ volt}$$

Thus, sulfite should effectively function as a depressant (Fuerstenau, 1982).

Recently, metabisulphite (MBS) was used to depress sphalerite, pyrite, and pyrrhotite. With the use of MBS and a dithiophosphate collector at the pulp temperature of 40°c to 45°c, galena was recovered more selectively from sphalerite and pyrite (Rohner et al., 1989).

In addition, hydroxyl, sulfide, hyposulfite, and hydrosulfite are used to depress pyrite while floating lead and copper minerals (American Cyanamid Company, 1976).

Besides the addition of reagents for pyrite depression, the pre-flotation aeration of sulfide pulp helps in the activation of chalcopyrite and the depression of pyrite (as mentioned in chapter 3).

4.3 Experimental Procedure

There is a number of options to depress iron sulfides as mentioned above. Generally, cyanide is the preferred depressant for iron sulfides; however, it is not used at Camchib because gold dissolves in cyanide solution. Potassium ferrocyanide, another effective pyrite depressant, is already a stable complex and less likely to dissolve gold. Its use was tested and flotation solutions were assayed for dissolved gold. The following iron sulfides depressants were tested:

(1). Potassium ferrocyanide ($K_4Fe(CN)_6$).
(2). Sodium sulfite (Na_2SO_3).
(3). Pre-flotation aeration.
(4). Sulfur dioxide (SO_2).

A sample of the retreater feed from the Camchib flotation circuit were extracted, filtered, and sent to McGill University. The sample was assayed for copper, gold, and size distribution. The size analysis was obtained by wet screening at 53 µm (270 mesh), and 37 µm (400 mesh) followed by microscreening at 25 µm, 15 µm, and 10 µm, respectively.

For bench scale flotation, 1 kg samples of the retreater feed were ground with lime and depressant in a laboratory rod mill at 65% solids. The stainless steel mill, 24 cm in length and 18 cm in diameter, was charged with fifteen 18 mm by 225 mm, two 23 mm by 225 mm, and one 30 mm by 225 mm stainless steel rods (using stainless steel rod mill rather than mild steel reduces the effect of galvanic interactions as mentioned in chapter 3). Grinding time was set for 20 minutes to create fresh surfaces needed for pyrite depression. A 2.5 litre Leeds Cell was used for the laboratory flotation tests (Figure 4.1). The pulp was conditioned. Air flow rate and the impeller speed were controlled.

The flotation procedure is summarized in Table 4.1.

Table 4.1 The flotation procedure used for pyrite depression

Grinding:	
Mill	Rod
Media	Stainless Steel
% Solids	65%
Depressant	$K_4Fe(CN)_6$, or Na_2SO_3, or or SO_2
pH (modifier)	lime
Flotation Cell:	
Type	Leeds
Capacity, litre	2.5
Impeller speed, rpm	1000
Conditioning:	
Time, min	5
pH	11.2 (lime)
Collector, g/t	Aerofloat 208*, 100
Frother, g/t	Dowfroth 250C**, 10
Flotation Gas, L/min	Air, 5

* Aerofloat 208 is the mixture of sodium diethyl and sodium di-secondary butyl dithiophosphate
** Dowfroth 250C is polypropylene glycol methyl ether

With pre-flotation aeration, the pulp was pre-aerated with 3 L/min of air about 30 minutes, until the oxygen content in the pulp had reached 7.0 ppm, at a pH of 11.2 regurated with lime. After pre-aeration, collector was added, followed by 5 minutes of conditioning and addition of frother. The oxygen content was measured using an Orion oxygen electrode model 97-08-99. The pH was measured by a standard pH meter electrode

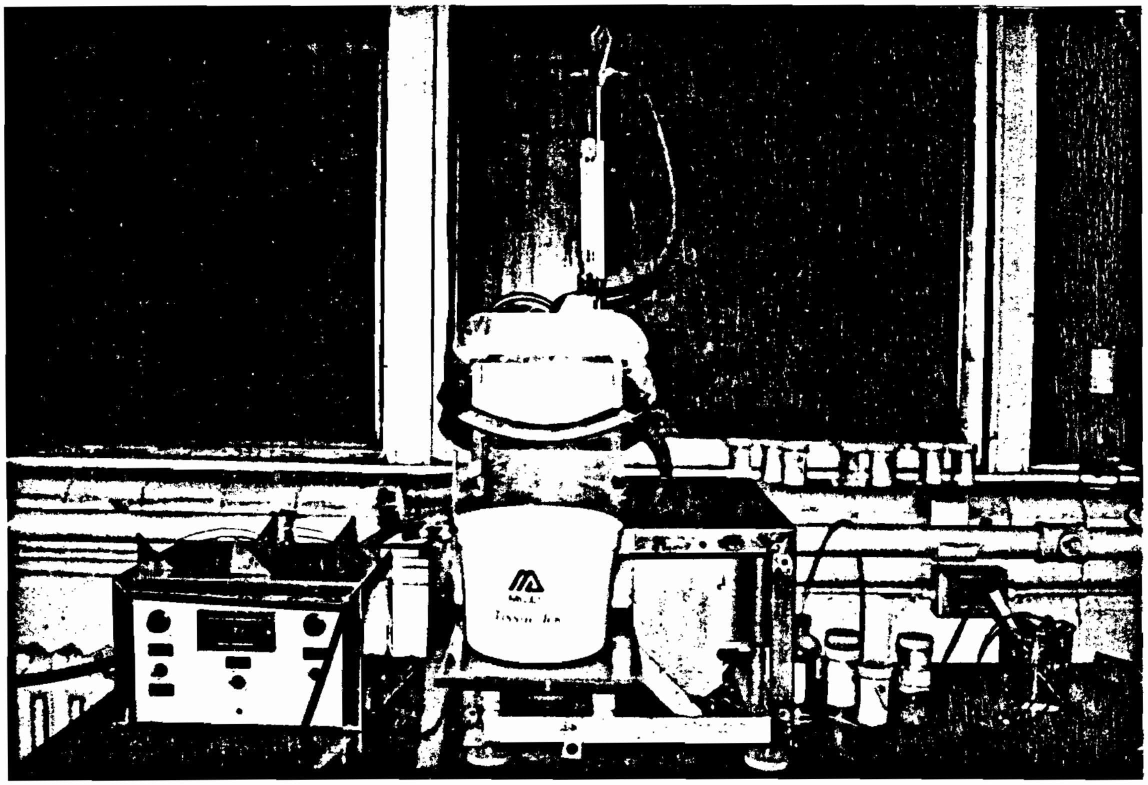

Figure 4.1 A 2.5 litre laboratory Leeds cell

(Hanna Instruments model HI-1210). Both electrodes were connected to a Hanna Instruments model HI-8424 voltmeter/pH meter.

After flotation, the concentrates and tails were dried, weighed, and assayed for Au, Cu, and Fe.

When testing ferrocyanide, a single concentrate was extracted, after 10 minutes of flotation. This procedure was later modified; concentrate was recovered incrementally to generate grade-recovery curves.

4.4 Results and Discussion

The retreater feed assays 0.30 oz/st Au and 0.28% Cu. The size distribution of the retreater feed as sampled from the plant is shown in Table 4.2.

Table 4.2 Size distribution of the Camchib retreater feed (as received)

Size (µm)	Mass (%)	Cum. Mass (%)
53	9.21	9.21
37	5.77	14.98
25	17.10	32.08
15	14.77	46.85
10	10.79	57.64
-10	42.36	100.00

However, the ore was reground at McGill University to create fresh surfaces for effective pyrite depression, to 97% minus 37 µm (400 mesh).

4.4.1 Potassium ferrocyanide as a depressant

Using potassium ferrocyanide ($K_4Fe(CN)_6$) as a depressant yielded results summarized in Table 4.3. The optimum addition (yielding the highest Cu grade, 1.47% Cu) is at 50 g/t. The recovery ratio of Chalcopyrite:Pyrite is about 5 and that of Chalcopyrite:non-sulfide gangue (NSG) about 6. At 100 g/t $K_4Fe(CN)_6$, pyrite recovery equals NSG recovery, which suggests a near complete depression and recovery by incomplete liberation or hydraulic entrainment; however, this was achieved at the expense of copper recovery, which dropped to 49% from 70%.

Table 4.3 Effect of potassium ferrocyanide on grade and recovery

$K_4Fe(CN)_6$ (g/t)	Stream	Mass (%)	Grade Au (oz/st)	Cu (%)	* Fe (%)	Recovery (%) Au	Chal	Py	NSG
0	Conc	15.60	1.40	1.15	20.59	72	66	18	14
	Tails	84.40	0.10	0.11	17.50	28	34	82	86
	Feed	100.00	0.30	0.27	18.17	100	100	100	100
30	Conc	13.52	1.41	1.30	17.41	63	70	14	13
	Tails	86.48	0.13	0.09	16.77	37	30	86	87
	Feed	100.00	0.30	0.25	16.86	100	100	100	100
50	Conc	11.77	1.48	1.47	19.29	54	66	13	10
	Tails	88.23	0.17	0.10	17.16	46	34	87	90
	Feed	100.00	0.32	0.26	17.41	100	100	100	100
100	Conc	12.11	0.98	0.99	16.58	44	49	11	12
	Tails	87.89	0.17	0.14	18.58	56	51	89	88
	Feed	100.00	0.27	0.24	18.34	100	100	100	100

* Fe in chalcopyrite already deducted

The effect of $K_4Fe(CN)_6$ concentration on recovery and grade is shown in Figure 4.2 and Figure 4.3, respectively. Increasing $K_4Fe(CN)_6$ concentration lowers the recovery of gold, copper, and iron sulfides. Gold and copper grade increases with increasing $K_4Fe(CN)_6$ concentration

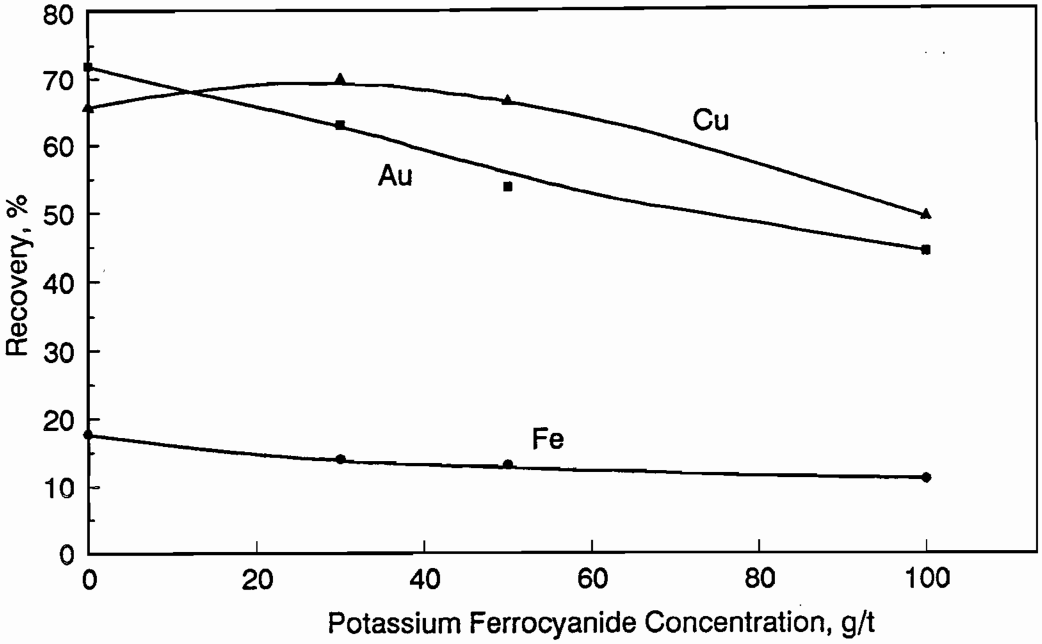

Figure 4.2 Effect of potassium ferrocyanide concentration on recovery

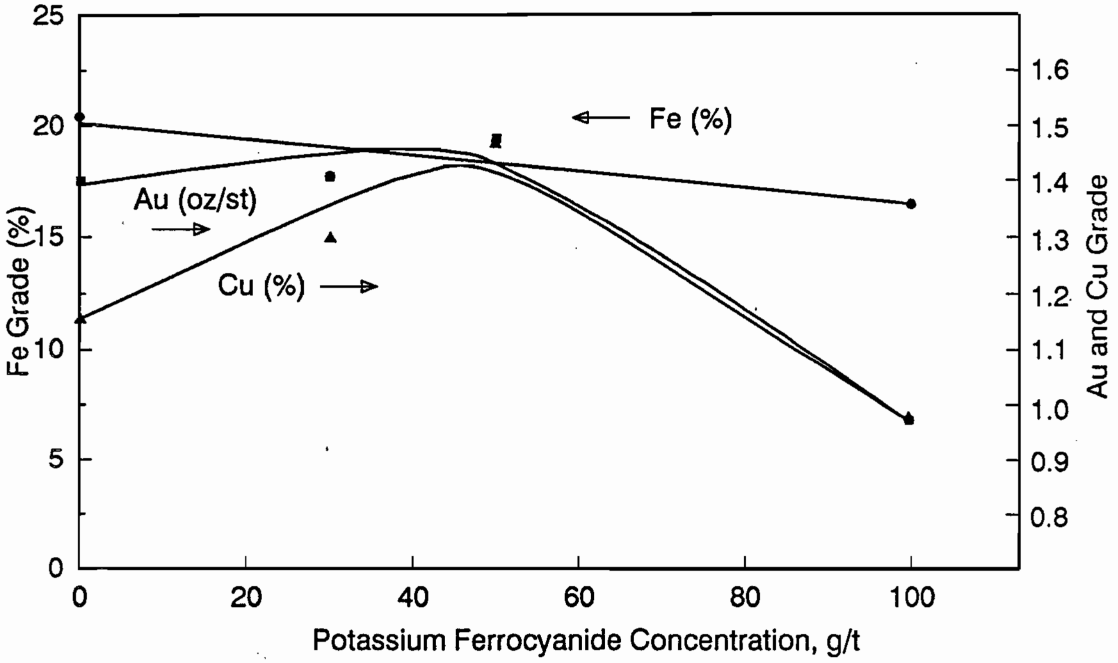

Figure 4.3 Effect of potassium ferrocyanide concentration on grade

from 0-50 g/t, but decreases at 100 g/t. Iron grade decreases with increasing depressant concentration. Presumably, at higher concentration of $K_4Fe(CN)_6$, gold, chalcopyrite, and iron sulfides are depressed.

Because gold dissolved in the flotation circuit at Camchib is not recovered, it is necessary to determine whether gold is dissolved by $K_4Fe(CN)_6$. Test procedure and results are presented in Appendix A. Dissolution tests showed that between 0.03 to 0.07 g/t Au (0.001-0.002 oz/st) was dissolved at 50 or 100 ppm $K_4Fe(CN)_6$; this could lead to significant gold losses if ferrocyanide was recycled in the flotation circuit solutions. The use of $K_4Fe(CN)_6$ is therefore unadvisable at Camchib. Further testing with $K_4Fe(CN)_6$ was discontinued.

4.4.2 Sodium sulfite as a depressant

The grade-recovery curves obtained at different sodium sulfite (Na_2SO_3) concentrations are shown in Figure 4.4, 4.5, and 4.6 for gold, chalcopyrite, and pyrite, respectively. Detailed results are shown in Appendix B. Gold and chalcopyrite grades increase while pyrite grade decreases with higher Na_2SO_3 concentration; Na_2SO_3 depresses pyrite. At 30 and 50 g/t Na_2SO_3, cumulative grade-recovery curves for gold and chalcopyrite are unusual, as both concentrate grade and recovery increase with time at the beginning of flotation (the first incremental concentrate should be the richest grade for the typical grade-recovery curve). These would be explained by the presence of "fast floating" gangue minerals (sulfides and non-sulfides) which bring the grade of the early concentrates down. The species are termed fast floating only inasmuch as they are recovered early in the flotation tests, before most of the

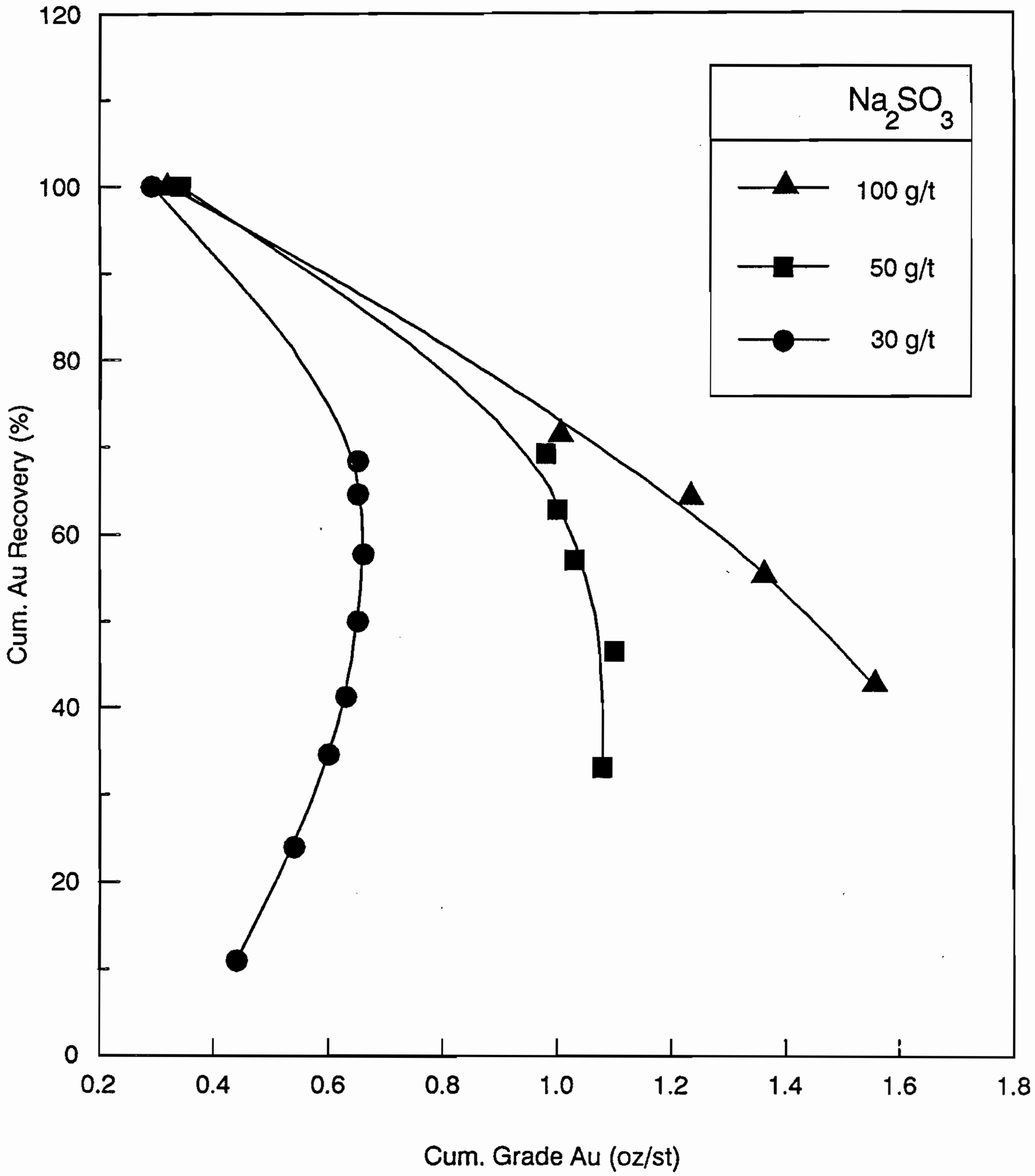

Figure 4.4 Gold Grade-Recovery curves using sodium sulfite as a depressant.

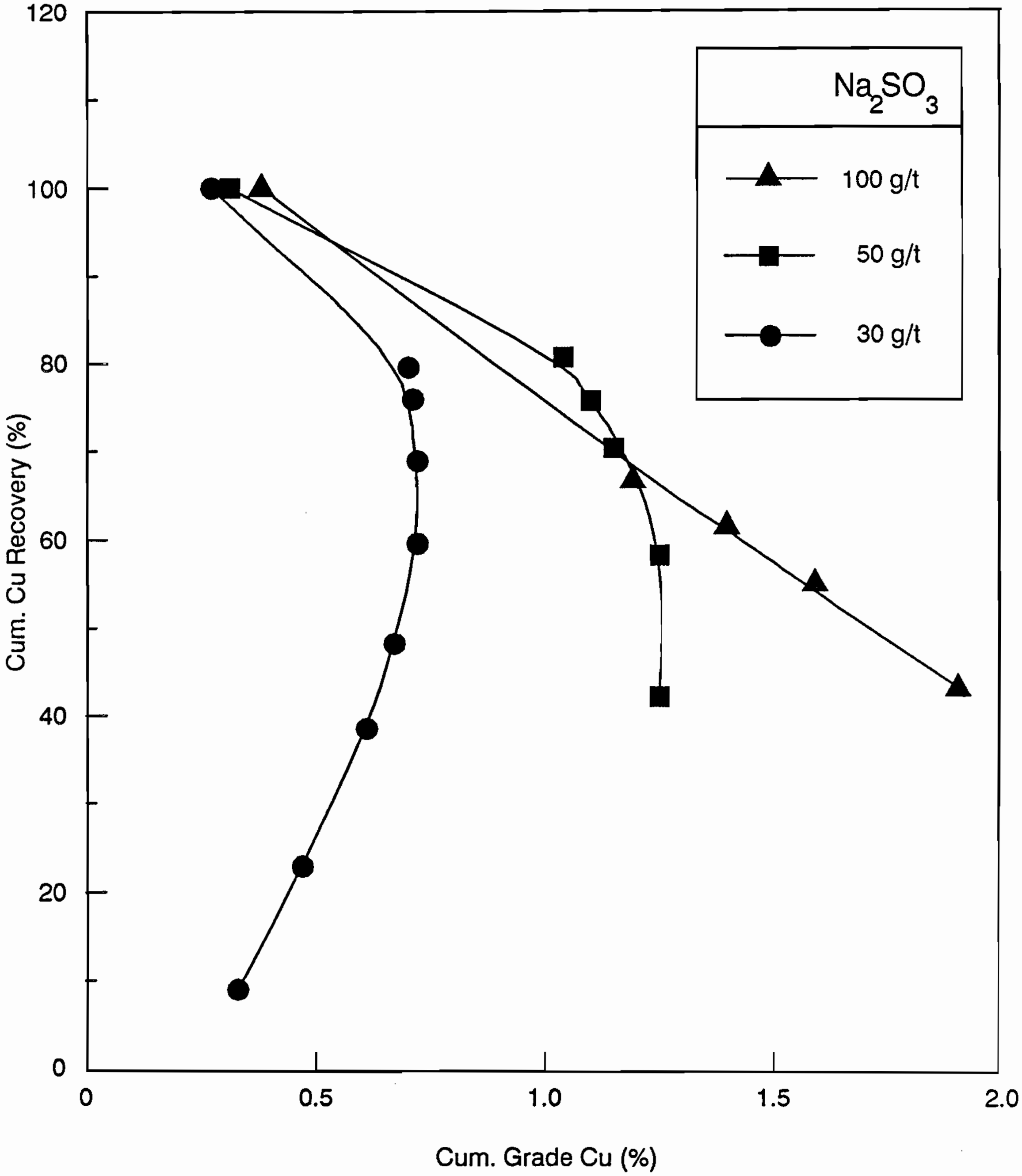

Figure 4.5 Chalcopyrite Grade-Recovery curves using sodium sulfite as a depressant.

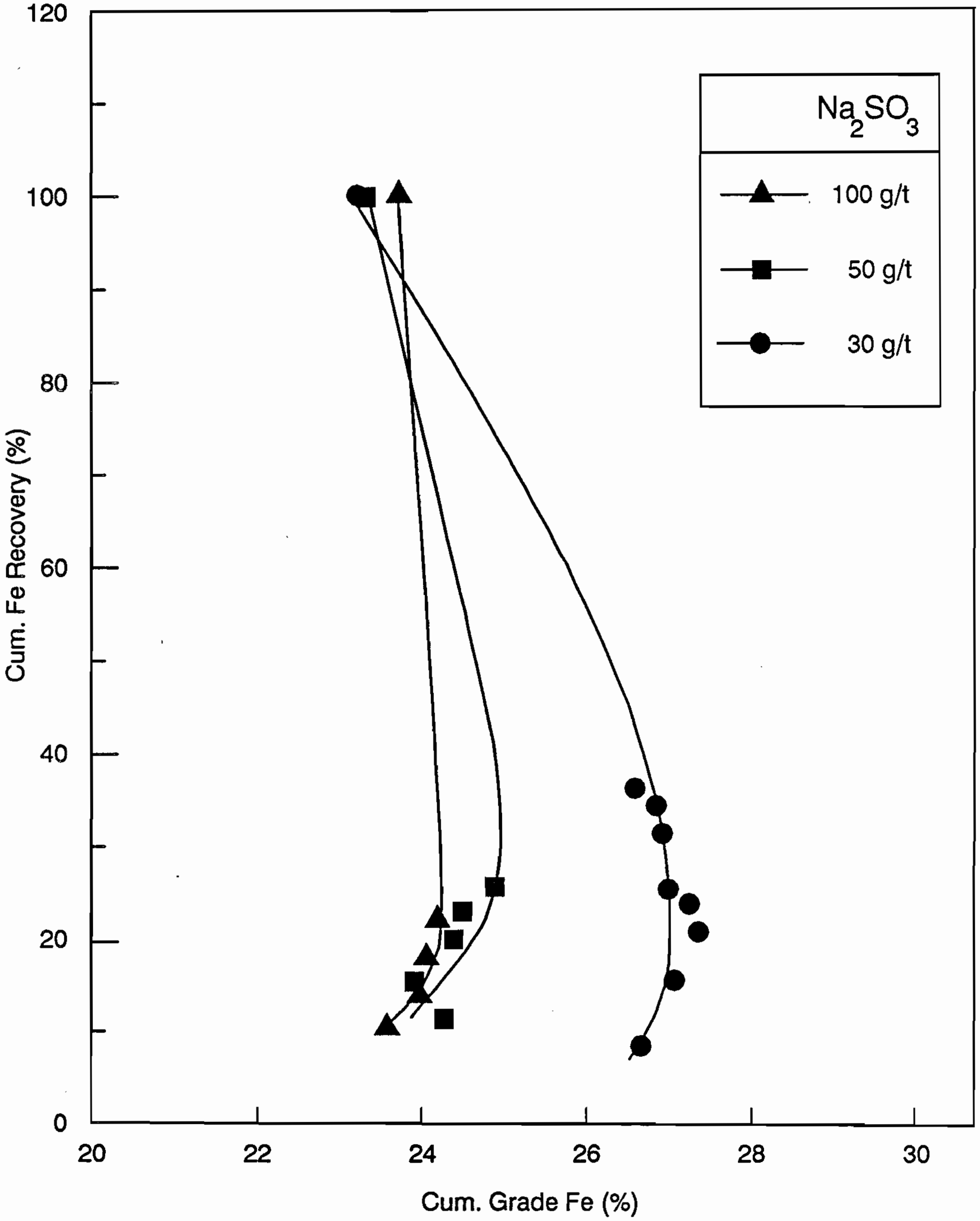

Figure 4.6 Fe Grade-Recovery curves using sodium sulfite as a depressant.

chalcopyrite and gold is. This would happen if the chemical conditions in the pulp, such as oxygen content, are not appropriate for chemical adsorption of the collectors on the valuable minerals at the start of flotation.

The grade-recovery curves of iron show very little change in grade; assays between 24 and 27% Fe correspond to pyrite contents of 52 to 58%, which implies that the concentrate contains a significant fraction of NSG.

4.4.3 Pre-flotation aeration

The effect of pre-aeration with and without Na_2SO_3 as a pyrite depressant on chalcopyrite and gold flotation is presented in Figure 4.7. Detailed results are in Appendix C. Pre-aeration with Na_2SO_3 gives much higher grades and slightly higher recoveries of gold and copper than pre-aeration or Na_2SO_3 alone. The effect of pre-aeration and Na_2SO_3 addition is clearly synergetic. As mentioned in chapter 3, the concentration of oxygen in the pulp plays an important role in chalcopyrite flotation. Chalcopyrite flotation is enhanced in oxidizing environments while pyrite is depressed. This can be explained by galvanic interactions, as reviewed in section 3.2.

The important effect of aeration can be substantiated by the low concentration of oxygen in the slurry just after grinding, 0.17 ppm. If no pre-aeration is used, selectivity actually increases after air is turned on for flotation. Oxygen content increases significantly with time as shown in Figure 4.8. This increase explains the unusual shapes of the grade-recovery curves of Au and Cu (i.e. Figure 4.4, 4.5, and 4.7). It is likely that pre-aeration not only improves the depression of pyrite, but

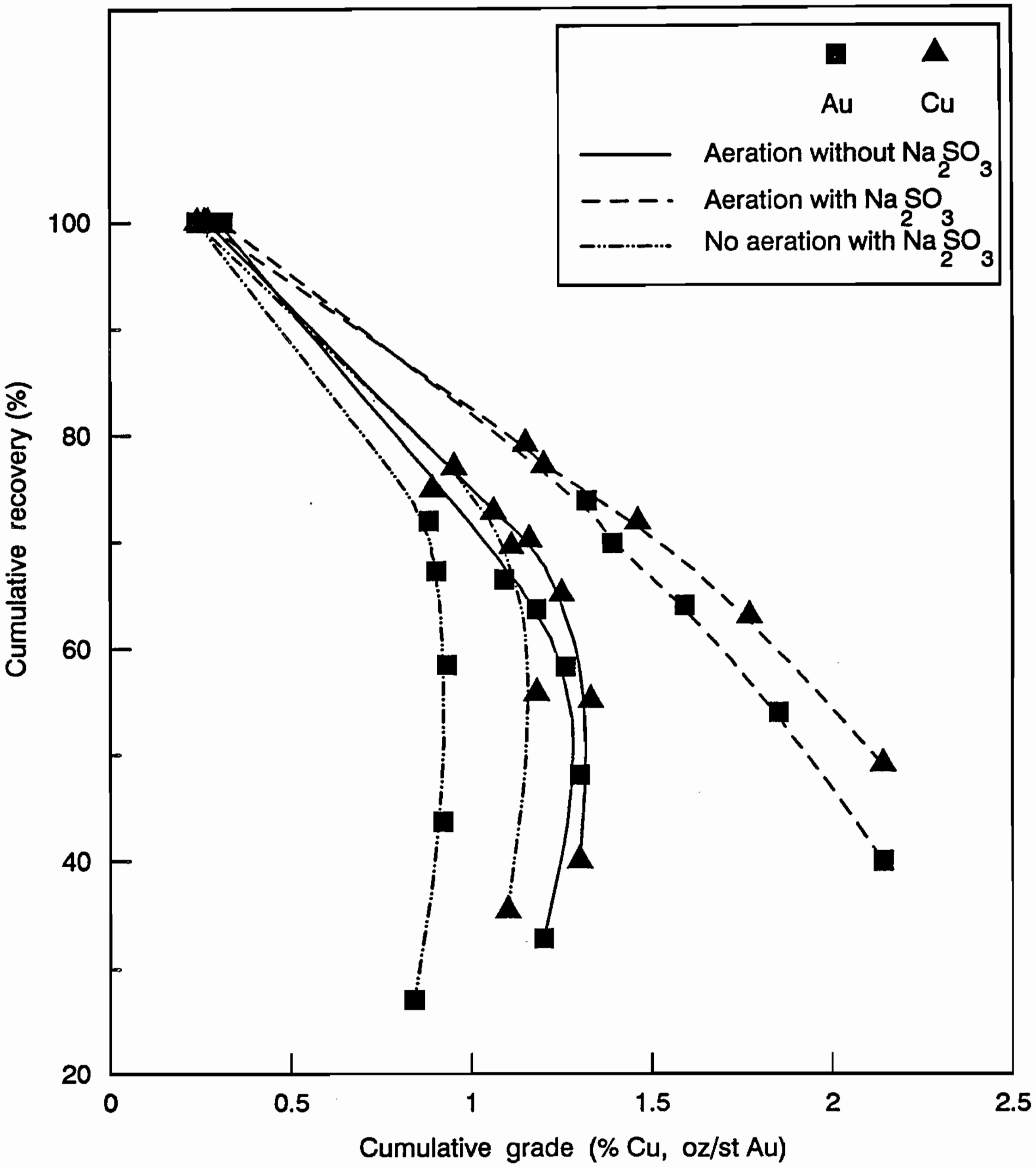

Figure 4.7 Effect of pre-aeration flotation with and without sodium sulfite (50 g/t).

also the activation of chalcopyrite, as collector adsorption is then more selective.

4.4.4 Sulfur dioxide as a depressant

The effect of sulfur dioxide (SO_2) as a pyrite depressant is presented in Figure 4.9. Detailed results are in Appendix D (Table D1 and D2 with and without aeration, respectively). This figure also presents the curves of pre-aeration without adding depressant (results from Table C1). Pre-aeration without depressant results in the highest Au and Cu grades and recoveries. When adding SO_2, Cu grade decreases, especially when pre-aeration is omitted. This again confirms that pre-aeration promotes chalcopyrite/pyrite selectivity. However, gold grade with pre-aeration is lower than without pre-aeration; this is partly due to the lower gold content in the sample (0.24 oz/st vs 0.32 oz/st).

In Figure 4.7, pre-aeration flotation without depressant results in lower grade and recovery than those of pre-aeration with Na_2SO_3 while in Figure 4.9, the former presents higher grade and recovery than those of pre-aeration with SO_2. It can be concluded that SO_2 in this case failed to enhance chalcopyrite/pyrite selectivity, possibly because dosage was too high (3 g of saturated SO_2 per kg ore). An excessive dosage would depress chalcopyrite as well as pyrite, thus failing to achieve selectivity. SO_2, because of safety and health considerations, would not be used at Les Mines Camchib Inc., unless absolutely essential to achieve selectivity. This is not substantiated by the present results.

4.5 Summary

Potassium ferrocyanide ($K_4Fe(CN)_6$) is an attractive iron sulfide depressant, but it dissolves gold, and its use is therefore unadvisable at Camchib.

Sodium sulfite (Na_2SO_3) can be used as a pyrite depressant; it is much more effective when combined with pre-aeration. In fact, pre-aeration to an oxygen content of 7.0 ppm is in itself more effective than Na_2SO_3 without pre-aeration.

Sulfur dioxide (SO_2) as a pyrite depressant was also tested, but proved less effective than either pre-aeration or Na_2SO_3. It is possible that sulfur dioxide dosage (3 g SO_2/kg ore) needs adjusting. The amount of SO_2 should be controlled due to its strong effect on gold and chalcopyrite flotation. This may prove difficult in plant.

The testwork of this section has identified Na_2SO_3 as a promising depressant, whose use will be tested in columns. Furthermore, the role of oxygen to foster pyrite depression has been confirmed. Differences in oxidation levels in column vs. mechanical cells could therefore yield more or less chalcopyrite/pyrite selectivity. This is particularly important when a product is fed to flotation after regrinding without an oxidizing condition step; such is the case at Camchib.

CHAPTER 5

FLOTATION TESTWORK WITH A 3.8 CM DIAMETER COLUMN

5.1 Introduction

In this chapter, a simple approach to column flotation is formulated and tested. The typical pilot-plant flotation column is a 5 cm diameter unit operated continuously (at steady-state) with some form of automatic level control (Amelunxen et al., 1988; Stephanopoulos, 1984; Considine, 1985; Hunter, 1987). Three possible simplifications are the use of a smaller diameter column, batch tests, and simpler interface control.

For smaller column, a 3.8 cm diameter unit will be tested. This is to determine how small of the column can be operated and yield reliable results.

For batch tests, the froth phase is first stabilized, the mineral sample is then fed, and all of the products are recovered.

In column operation, froth depth (interface level) has to be maintained because it affects metallurgical performance. In this work, the interface level will be controlled by adjusting the height of a large diameter tails pipe which runs up from the column tails section. Because of the special design of the column, its operational stability will be monitored closely.

5.2 Column Description and Operation

The flotation column is made of Plexiglas, 3.8 cm in diameter, 325 cm in height for batch operation at McGill University or 510 cm in height for continuous operation at the plant. The column consists of three sections. The top section, 185 cm high, allows for free overflow of the concentrate into a launder for removal of material. Wash water is introduced 5 cm below the column lip through a perforated copper tube, and feed 88 cm below, by using Tygon 9.5 mm (3/8") Cole-Parmer Masterflex tubing. The middle section, 140 cm high, is connected to the tails or bottom section, 35 cm in height, 20 cm in diameter. The sparger is located at the top of the tails section. In continuous test, another section, 185 cm high, is connected between the top and the middle sections of the batch test; the feed point is 150 cm below the lip. The interface level is controlled by adjusting the height of a 3 cm (1¼") rubber pipe which runs from the tails section up the column to achieve the required froth depth. The fine tails overflows through this rubber pipe. The coarse tails is accumulated in the tails section and removed either continuously using a varistaltic pump (continuous operation) or at the end of each test (batch operation). The experimental set-up is shown in Figure 5.1.

Feed and wash water are added to the column by varistaltic pumps which give good control of the flow rate. Wash water is pumped to a calibrated rotameter (Cole-Parmer model N044-40G). Air is admitted through the sparger, and set to the pressure of 135 kPa (20 psi) at the desired flow rate, which is manually controlled with a calibrated rotameter (Cole-Palmer model N102-05G). Wash water and gas flowmeter calibration curves are shown in Appendix E.

5.3 Batch Operation

The objectives of batch operation were to monitor the column operational stability with the special design of the interface level control and simultaneously, to study flotation response when changing reagent addition, especially with respect to selectivity of chalcopyrite and pyrite.

The scavenger concentrate (sample SC1) and the cleaner tails (samples CT1 and CT2) from the Camchib flotation circuit were extracted separately, filtered, and sent to McGill University. The samples were screened for size distribution and assayed for gold, copper, and iron.

Since the samples extracted from mill stream were partially oxidized and stripped of their collector-bearing solution, the addition of both collectors and Na_2SO_3 as a pyrite depressant was first be tested. Important considerations are (a) the ability to reject pyrite and NSG, (b) Cu recovery, and (c) the different behaviour of the three feed samples (CT1, CT2, and SC1).

5.3.1 Experimental Procedure

The column was first filled with water already mixed with the desired concentration of Dowfroth 250C and lime. Wash water was added to the column. The froth depth was set at 20 cm below the column lip. Before feeding material to the column, froth depth was set around 17 cm below the lip to compensate for the increase of slurry density in the collection zone, which caused higher froth depth and pulled the interface level down to 20 cm.

For column feed preparation, 500 grams of cleaner tails or scavenger concentrate was ground in a laboratory stainless steel rod mill at 65% solids for 30 minutes to achieve 90% -37 µm (400 mesh). The ground product was conditioned with lime as pH regulator, and/or Na_2SO_3 as a pyrite depressant for 5 minutes. Next, collector was added and the slurry was conditioned for 5 minutes. Dowfroth 250C (Polypropylene glycol methyl ether) was then added as frother. Finally, the slurry was fed to the column.

To recover all of the froth, the rubber tail pipe was lifted to move the interface up to the froth discharge level, about 2 minutes after all of the material had been fed. During column operation, all of the concentrate and the fine tails (from the discharge pipe) were collected. At the end of the test, the balance of the material still in the column was then recovered as coarse tails when the column was flushed. All products were dried, weighed, and assayed for Au, Cu, and Fe.

5.3.2 Experimental Design

The experiment was divided into three series. Each series used different feed material; the elemental analysis of the three samples are shown in Table 5.1. The second cleaner tails is richest in gold, copper, and iron sulfides, whilst the scavenger concentrate is poorest. The size distribution of the cleaner tails is finer than the scavenger concentrate, as shown in Table 5.2.

Table 5.1 Analysis of cleaner tails and scavenger concentrate samples

Test series	Au (oz/st)	Cu (%)	Fe (%)
1st: CT1 (Cleaner tls)	0.18	0.42	19.0
2nd: CT2 (Cleaner tls)	0.23	0.50	29.0
3rd: SC1 (Scav. Conc)	0.10	0.25	13.0

Table 5.2 Size distribution of cleaner tails and scavenger concentrate (cleaner tails average of CT1 and CT2)

	Cleaner tails		Scavenger concentrate	
Size (µm)	Mass (%)	Cum. Mass (%)	Mass (%)	Cum. Mass (%)
150	6.76	6.76	5.55	5.55
100	6.07	12.83	4.39	9.94
75	10.29	23.12	11.29	21.23
53	6.39	29.51	17.61	38.84
37	9.60	39.11	11.60	50.44
-37	60.89	100.00	49.56	100.00

<u>**First Test Series:**</u> The objective was to study the effect of Aero 208 dosage on grade and recovery. For tests 1-5, a first cleaner tails sample was used as feed. Superficial wash water, feed, and gas flow rate were maintained at 0.43, 0.35, and 0.88 cm/s, respectively. Frother concentration and pH were kept constant at 10 ppm and 11.0, respectively. For test 1, no collector was added in the feed tank but between 10 and 40 g/t of Aero 208 was added for tests 2-5.

Second Test Series: The objective was to study the effect of collectors combination on grade and recovery. Superficial wash water, feed, and gas flow rate were set at the same values as the first test series. Frother concentration was constant at 10 ppm for tests 6-9 but 15 ppm for test 10. pH was maintained at 11.0. For tests 6 and 7, Aero 3501 was added at 10 g/t and Aero 208 was increased from 10 g/t (test6) to 15 g/t (test7). 20 g/t of Aero 208 was added with 10 g/t of SIPX in test 8, and with 10 g/t of Aero 355 in tests 9 and 10.

Third Test Series: The objectives were to study the behaviour of the scavenger concentrate on column flotation, and the effect of Na_2SO_3 on pyrite depression. Superficial wash water, gas flow rate, and feed rate were maintained at 0.43, 1.11, and 0.25 cm/s, respectively. The increased gas rate and lower feed rate (from the first two test series) were aimed at increasing copper recovery. Frother concentration and pH were kept constant at 15 ppm and 11.0, respectively. A combination of Aero 208 and Aero 355 with the ratio of 2:1 was used. Na_2SO_3 was added between 0 and 10 g/t.

5.3.3 Results and Discussion

The height of discharge rubber pipe was fixed throughout the operation. Upon feeding material, the interface level was pulled down by the increasing of pulp density in the collection zone, and froth thickness increased from 17 cm at the beginning of each test to 20 cm at the end. The interface level was well behaved throughout each test. As only 500 g of solids was fed for each test, the impact on slurry and froth density was small.

Table 5.3 presents the effect of Aero 208 dosage on grade and recovery. Cu and Au recovery in the concentrate is too low (with a maximum of only 50%), and Cu and Au grade in the tails remains high compared to the feed. The ratio of chalcopyrite:pyrite (Chal:Py) recovery in the concentrate is about 3:1 while the ratio of chalcopyrite:NSG is about 11:1.

Table 5.3 First test series: the effect of Aero 208 concentration on grade and recovery

Test	Aero 208 (g/t)	Stream	Mass (%)	Grade		*	Recovery			
				Au (o/st)	Cu (%)	Fe (%)	Au (%)	Chal (%)	Py (%)	NSG (%)
1	0	conc	5	0.70	2.54	32.65	21	29	10	2
		c.tls	86	0.14	0.33	17.20	71	63	83	88
		f.tls	9	0.14	0.36	14.04	8	8	7	10
		feed	100	0.17	0.45	17.70	100	100	100	100
2	10	conc	10	0.58	1.92	31.06	33	48	17	5
		c.tls	60	0.14	0.23	17.59	47	34	59	62
		f.tls	30	0.12	0.25	14.64	20	18	24	33
		feed	100	0.18	0.41	18.07	100	100	100	100
3	20	conc	9	0.68	2.25	30.07	36	46	14	4
		c.tls	68	0.12	0.25	18.38	49	40	67	70
		f.tls	23	0.11	0.26	15.24	15	14	19	26
		feed	100	0.17	0.43	18.69	100	100	100	100
4	30	conc	9	0.70	2.69	30.60	37	50	16	3
		c.tls	67	0.11	0.27	18.16	42	38	65	68
		f.tls	24	0.15	0.26	15.64	21	12	19	29
		feed	100	0.17	0.41	18.23	100	100	100	100
5	40	conc	9	0.78	2.27	30.76	41	49	15	4
		c.tls	73	0.11	0.23	18.20	47	40	70	76
		f.tls	18	0.11	0.26	15.64	12	11	15	20
		feed	100	0.17	0.42	18.88	100	100	100	100

Aero 208 - Sodium diethyl and sodium di-secondary butyl dithiophosphate
Dowfroth 250C - Polypropylene glycol methyl ether
Experimental conditions
Dowfroth 250C 10 ppm, pH 11.0
Superficial wash water rate 0.43 cm/s, J_w
Superficial gas rate 0.88 cm/s, J_g
Superficial feed rate 0.35 cm/s, J_f
Froth depth 20 cm
c.tls : coarse tails
f.tls : fine tails
* Fe in chalcopyrite already deducted

The effect of Aero 208 concentration on grade and recovery is shown in Figure 5.2 and 5.3, respectively. Figure 5.2 shows that Au and Cu

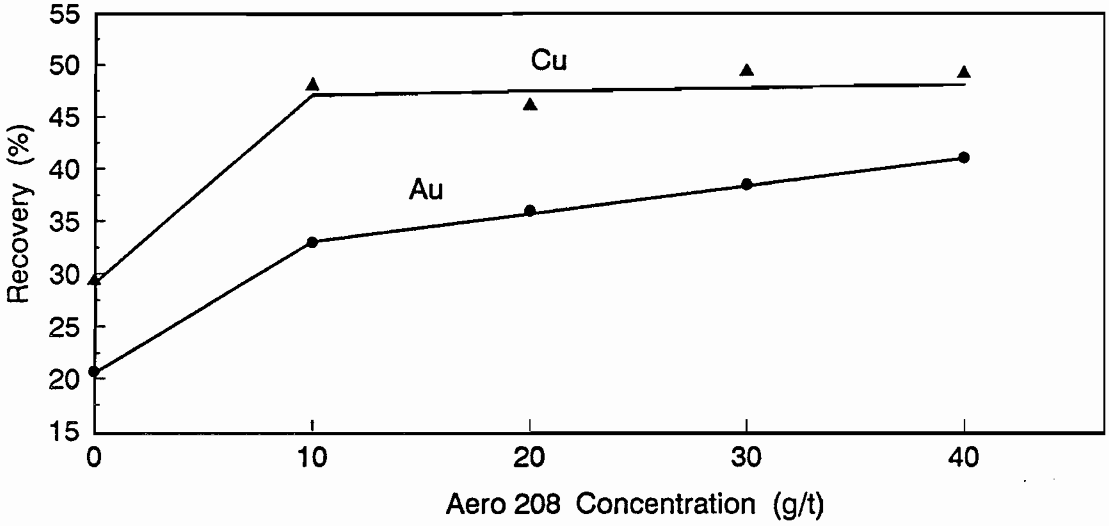

Figure 5.2 Effect of collector concentration on flotation recovery.

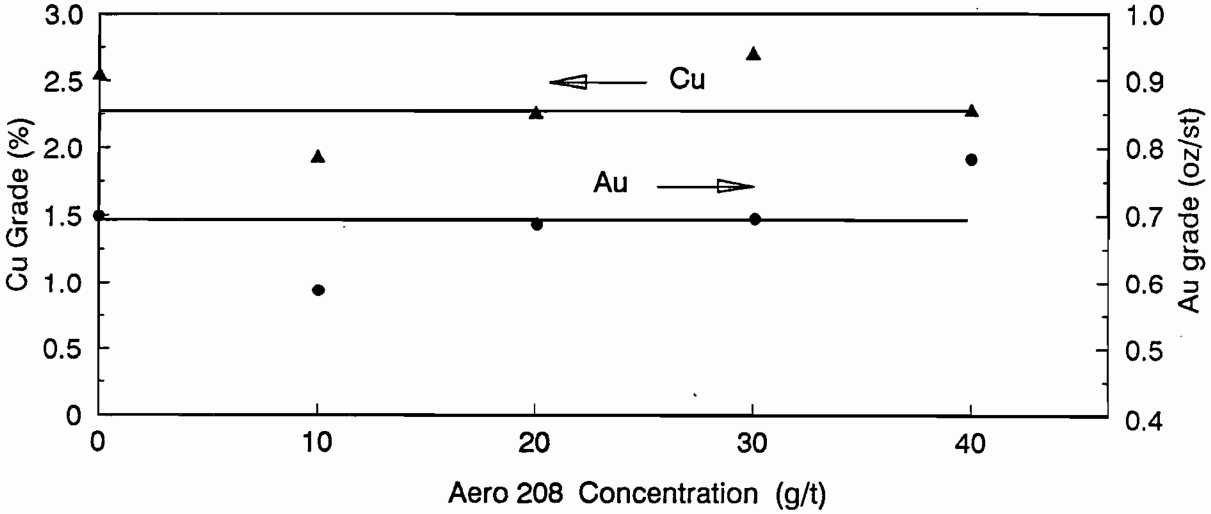

Figure 5.3 Effect of collector concentration on concentrate grade.

recovery increases significantly with increasing collector concentration from 0 to 10 g/t, but only slightly for higher dosages. Figure 5.3 shows that Cu concentrate grade varies from 2.0 to 2.7% and Au grade from 0.6 to 0.8 oz/st with no detectable trend.

Table 5.4 presents the effect of collector combination on grade and recovery. Au and Cu recoveries are still below 50%. The concentrate grades are in the range of 3-4 %Cu and 1.0-1.2 oz/st Au. The second test series shows a better chalcopyrite and pyrite selectivity than for the first. The ratio of Chal:Py recovery in concentrate is between 6:1 and 9:1 (3:1 in the first test series). However, the ratio of Chal:NSG recovery in concentrate is about 7:1 (11:1 in the first test series). This could be because the feed is different (more sulfides in the second test series than in the first).

The collector combination gave better Cu and Au grades than using Aero 208 alone. To compare better the effectiveness of using only Aero 208 and collector combinations, a plot of the collector dosage versus gold and copper enrichment ratio is constructed, as shown in Figure 5.4 (the enrichment ratio is the ratio of the grade of the concentrate to the grade of the feed, and is related to the efficiency of the process). Tests 9 and 10 present the highest enrichment ratio of Au and Cu with a combination of Aero 208 and Aero 355, which are both used in the Camchib flotation circuit. Frother concentration does not affect grade or recovery (10 ppm in test 9 while 15 ppm in test 10).

Figure 5.4 also shows that increasing collector concentration up to 30 g/t actually enhances selectivity, for at least two samples of the cleaner tails. This information, however, will be difficult to apply to plant practice, as the cleaner tails and scavenger concentrate are reground in the collector rich solution, and no collector is actually added prior to the retreater stage.

Table 5.4 Second test series: the effect of collector combination on grade and recovery

Test	Aero:Other 208 Col (g/t)	Stream	Mass (%)		Grade	*		Recovery		
				Au (o/st)	Cu (%)	Fe (%)	Au (%)	Chal (%)	Py (%)	NSG (%)
6	10 : 10	conc	8	0.90	2.95	26.32	35	49	8	8
	3501	c.tls	80	0.16	0.28	28.97	57	44	81	78
		f.tls	12	0.14	0.30	25.72	8	7	11	14
		feed	100	0.22	0.51	28.36	100	100	100	100
7	15 : 10	conc	7	1.02	3.49	26.15	30	46	6	6
	3501	c.tls	79	0.18	0.30	29.26	59	45	81	77
		f.tls	14	0.17	0.33	25.59	11	9	13	17
		feed	100	0.23	0.52	28.52	100	100	100	100
8	20 : 10	conc	6	0.96	3.80	26.28	26	44	5	6
	SIPX	c.tls	79	0.18	0.30	30.37	62	47	82	76
		f.tls	15	0.18	0.32	26.10	12	9	13	18
		feed	100	0.22	0.51	29.49	100	100	100	100
9	20 : 10	conc	6	1.02	3.90	25.25	29	47	5	7
	355	c.tls	82	0.17	0.29	30.38	62	45	84	79
		f.tls	12	0.17	0.33	26.40	9	8	11	14
		feed	100	0.22	0.52	29.58	100	100	100	100
10	20 : 10	conc	6	1.20	3.60	26.06	30	46	6	6
**	355	c.tls	79	0.18	0.28	30.28	59	45	81	77
		f.tls	14	0.18	0.32	26.71	11	9	13	17
		feed	100	0.24	0.50	29.51	100	100	100	100

Aero 208 - Sodium diethyl and sodium di-secondary butyl dithiophosphate
Aero 3501 - Sodium diisoamyl dithiophosphate
SIPX - Sodium isopropyl xanthate
Aero 355 - Sodium amyl xanthate

Experimental conditions
Dowfroth 250C 10 ppm, pH 11.0
Superficial wash water rate 0.43 cm/s, J_w
Superficial gas rate 0.88 cm/s, J_g
Superficial feed rate 0.35 cm/s, J_f
Froth depth 20 cm

* : Fe in chalcopyrite already deducted
** : Dowfroth 250C 15 ppm, pH 10.0

The metallurgical results of the third test series using the scavenger concentrate as a feed are shown in Table 5.5. Gold and copper recoveries are higher than for the first two series. Cu and Au concentrate grades do not change much. The ratio of Chal:Py recovery in concentrate is between 3:1 and 4:1, as for first test series. The third test series yields higher enrichment ratios, about 8-10 for Au and Cu, and low NSG recoveries in the 2-4% range as same as the first test series. The ratio of Chal:NSG recovery is very high, about 32:1. Scavengers are operated at Camchib with very thin froths and unsteady level control, resulting in high water recovery and hence entrainment. The hydrophilic gangue thus recovered is easily rejected in the column.

Table 5.5 Third test series: the effect of Na_2SO_3 on iron sulfides depression

Test	208:355 (g/t)	SDSF (g/t)	Stream	Mass (%)	Grade		*	Recovery			
					Au (o/st)	Cu (%)	Fe (%)	Au (%)	Chal (%)	Py (%)	NSG (%)
11	20: 10	0	conc	6	0.82	2.58	33.72	47	64	15	2
			c.tls	64	0.06	0.09	11.95	38	25	60	66
			f.tls	30	0.05	0.11	10.31	15	14	24	32
			feed	100	0.10	0.24	12.71	100	100	100	100
12	20: 10	5	conc	8	0.78	2.30	35.69	59	65	22	2
			c.tls	74	0.05	0.10	11.64	31	27	66	78
			f.tls	18	0.06	0.13	9.39	10	8	13	20
			feed	100	0.11	0.28	13.16	100	100	100	100
13	20: 10	10	conc	7	0.80	2.38	32.89	65	63	19	2
			c.tls	61	0.04	0.10	12.04	23	22	56	64
			f.tls	32	0.04	0.14	10.19	12	15	25	34
			feed	100	0.09	0.29	13.02	100	100	100	100

208 - Sodium diethyl and sodium di-secondary butyl dithiophosphate
355 - Sodium amyl xanthate
SDSF - Sodium sulfite

Experimental conditions
Dowfroth 250C 15 ppm, pH 11.0
Superficial wash water rate 0.43 cm/s, J_w
Superficial gas rate 1.11 cm/s, J_g
Superficial feed rate 0.25 cm/s, J_f
Froth depth 20 cm

* : Fe in chalcopyrite already deducted

The effect of Na_2SO_3 concentration on Au and Cu recovery and grade is shown in Figure 5.5 and 5.6, respectively. Cu recovery is constant but Au recovery increases with increasing Na_2SO_3 concentration, at constant gold and copper grade in the concentrate. However, this is largely due to minor changes in the tails gold assay, which are within the accuracy of fire assay at these low grades (0.04-0.06 oz/st). Na_2SO_3 seems to have no effect on pyrite depression; it is probably due to its low addition rate, only 5-10 g/t. Further testwork will use rates of 50 to 100 g/t as suggested by the Leeds cell results.

5.4 Continuous Operation

The objectives are to study the column performance by increasing the collection zone height and to monitor its operational stability and metallurgical performance with a continuous of freshly ground material.

5.4.1 Experimental Procedure

The continuous operation on flotation column was done at the Camchib mill. The column was filled with water of pH 11.0 using lime and 10 ppm of Dowfroth 250C. Launder water was used to break the froth through a flexible tubing and to provide an adequate flow rate of concentrate. The coarse tails were extracted continuously using a varistaltic pump, and the interface level was maintained at 110 cm below the column lip.

The column could be accessed first at the ground level for manually controlling the flow rate of wash water and gas using calibrated rotameters (see Appendix E). Air was admitted through the sparger, and

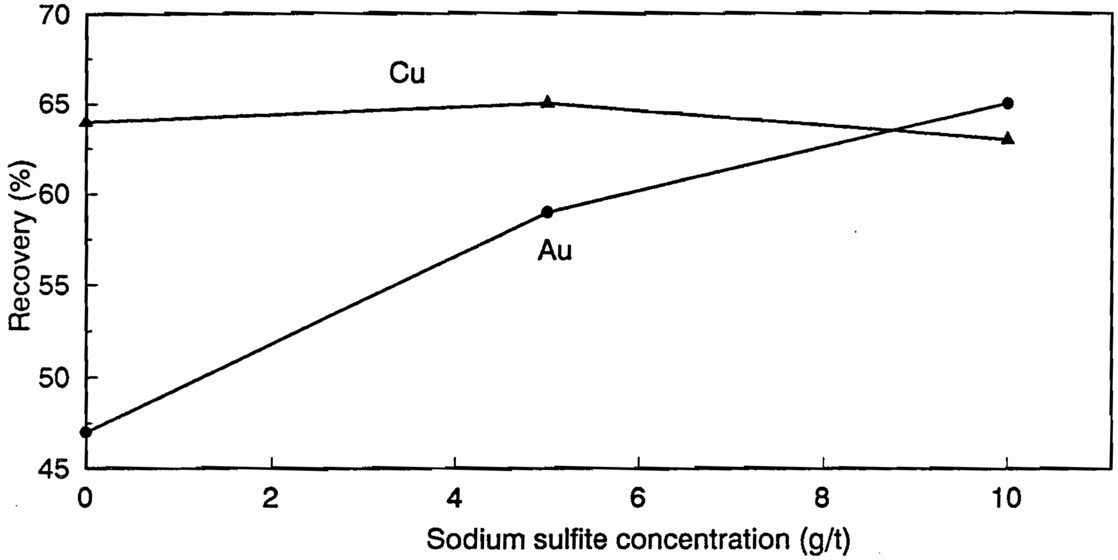

Figure 5.5 Effect of sodium sulfite concentration on Au and Cu recovery.

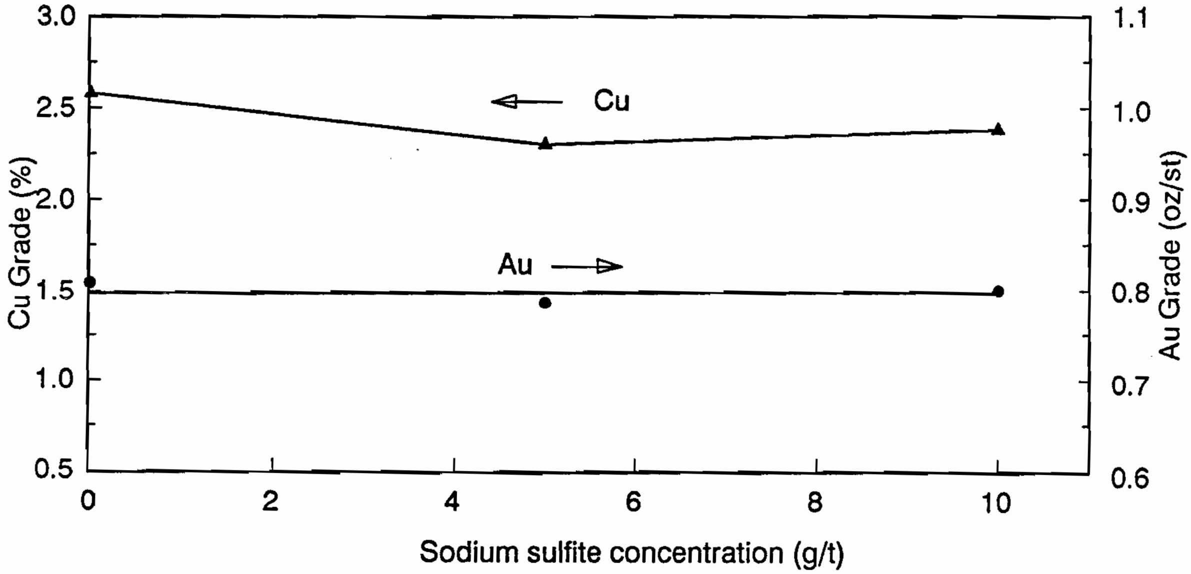

Figure 5.6 Effect of sodium sulfite concentration on Au and Cu grade.

set to the pressure of 135 kPa (20 psi). Superficial wash water and gas rate were maintained at 0.5 cm/s and 1.3 cm/s, respectively. During the test, gas flow rate was continuously monitored and adjusted, as it slightly fluctuated. The coarse tails was also collected on the ground level. The coarse tails flow rate was adjusted around 180 ml/min (0.26 cm/s). The second level was accessed for feeding the material, and collecting the concentrate and fine tails. The superficial feed rate was maintained at 0.35 cm/s. The interface level was maintained by lifting or lowering the rubber pipe along the column. Water was fed to the concentrate launder on the top level with the flow rate of 480 ml/min. The experimental set-up at the plant can be seen in Figures 5.7 and 5.8.

For column feed preparation, the retreater feed stream of Camchib flotation circuit was sampled, filtered, and assayed for Au and Cu. The filtered retreater feed material was slurried at 10% for RUN No.1 and 5% solids for RUN No.2. Then, pulp pH was adjusted to 11.0 with lime, and conditioned with 100 g/t of Aero 208 for 10 minutes. Next, Dowfroth 250C was added as frother. The pulp was then fed to the column.

As soon as the material was fed, the concentrate, coarse and fine tails were collected simultaneously, with a 10 minute frequency. The first sample was collected for 10 minutes, the second for 9 minutes, and the others for 2 minutes. The collected samples were immediately processed in the Camchib laboratory to determine percent solids and dried for chemical analysis.

5.4.2 Results and Discussion

The retreater feed assayed 0.24 oz/st Au and 0.21 %Cu. The results

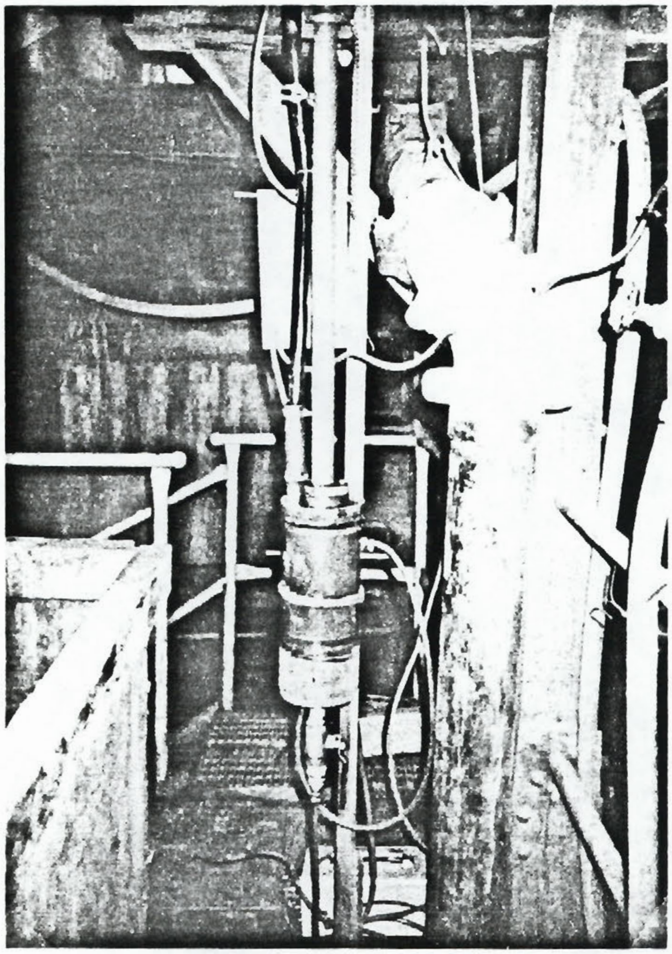

Figure 5.7 Experimental set-up of the flotation column at the Camchib plant showing the coarse tails section at the ground floor and the rubber pipe which runs up to the desired interface level.

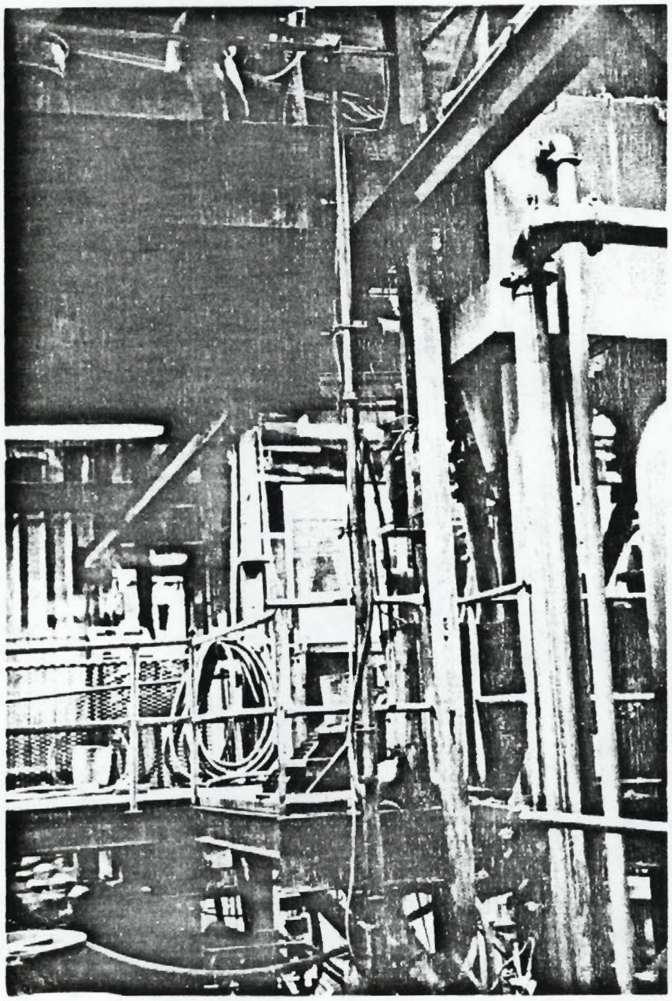

Figure 5.8 Experimental set-up of the flotation column at the Camchib plant showing the second and the top access levels to the column.

of the continuous operation at 10% and 5% solids feed are shown in Tables 5.6 and 5.7, respectively.

Table 5.6 The metallurgical results of column continuous operation at 10% solids feed (RUN No.1)

Time	Slurry (g)	Solid (g)	%Solids	Flow rate (ml/min)	Grade		Recovery	
					Au (oz/st)	Cu (%)	Au (%)	Cu (%)
	*			**CONCENTRATE**				
0-10	1800	69	3.81	195	0.86	1.23	63	82
10-19	1886	103	5.46	201	0.88	1.11	75	85
19-21	379	19	5.01	183	0.98	1.10	70	81
29-31	447	26	5.81	214	0.82	1.06	57	77
39-41	482	30	6.20	230	0.84	1.05	65	78
				COARSE TAILS				
0-10	1874	179	9.55	175	0.19	0.10	36	18
10-19	1763	245	13.90	177	0.12	0.08	25	15
19-21	387	54	13.95	175	0.13	0.08	26	17
29-31	407	79	19.41	177	0.16	0.08	33	17
39-41	399	79	19.80	173	0.13	0.09	27	18
				FINE TAILS				
0-10	1867	0.75	0.04	187	0.32	0.17	0.26	0.12
10-19	1835	0.58	0.03	204			0.15	0.07
19-21	420	3.17	0.76	209			4.00	2.10
29-31	318	11.75	3.70	155			9.82	5.45
39-41	300	10.69	3.56	146			8.83	4.52

* laundry water already deducted from slurry concentrate

From Table 5.6, it can be seen that the flowrate of coarse tails is relatively constant while those of concentrate and fine tails fluctuate. Percent solids in the concentrate does not change much but in the coarse and fine tails starts increasing after 30 minutes. The recovery and grade of Cu and Au in the concentrate are shown in Figure 5.9 and 5.10 for a period of 50 minutes.

In Figure 5.9, the recovery of Cu is high for the first 20 minutes and decreases slightly later on while Au recovery oscillates without any

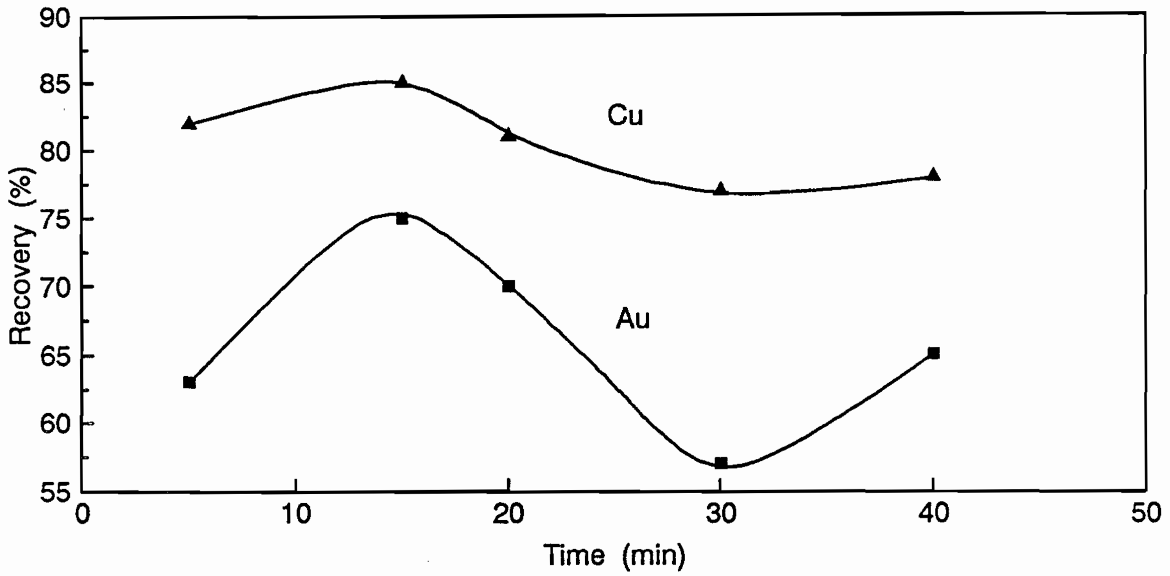

Figure 5.9 Recovery of Cu and Au in Concentrate for RUN 1.

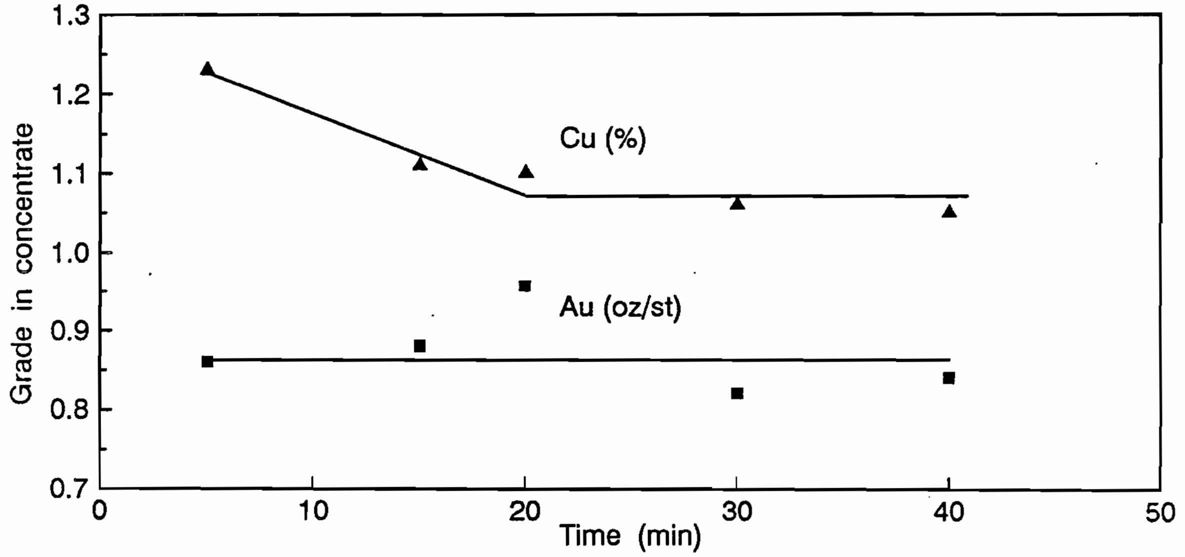

Figure 5.10 Grade of Cu and Au in Concentrate for RUN 1.

detectable trend. The grade of Cu decreases at first and then stabilizes after 20 minutes while the Au grade does not change much (see Figure 5.10).

RUN No.2, at5% solids, yielded unstable operations, as shown in Table 5.7.

Table 5.7 The metallurgical results of column continuous operation at 5% solids feed (RUN No.2)

Time	Slurry (g)	Solid (g)	%Solids	Flow rate (ml/min)	**Grade**		**Recovery**	
					Au (oz/st)	Cu (%)	Au (%)	Cu (%)
	*		**CONCENTRATE**					
0-10	1371	103	7.51	130	0.81	1.34	81	97
10-19	1286	97	7.54	135	0.75	1.14	86	95
19-21	281	20	7.12	133	0.78	0.92	86	93
29-31	269	21	7.81	127	0.78	0.60	60	77
39-41	210	17	8.10	99	0.76	1.00	46	71
49-51	243	20	8.23	114	1.14	0.50	64	69
			COARSE	**TAILS**				
0-10	1157	69	5.96	111	0.29	0.06	19	3
10-19	1039	104	10.01	108	0.11	0.05	14	5
19-21	242	23	9.50	113	0.11	0.06	14	7
29-31	650	64	9.85	303	0.12	0.05	28	20
39-41	**pump**	**leaks**						
49-51	561	69	12.30	257	0.12	0.05	23	25
			FINE	**TAILS**				
0-10	1571	0.23	0.01	157			0.11	0.01
10-19	1275	0.71	0.06	142			0.40	0.05
19-21	**							
29-31	377	7.01	1.86	186	0.48	0.09	12.22	3.78
39-41	**							
49-51	341	9.70	2.85	167			12.93	5.93

* laundry water already deducted from slurry concentrate

** No fine tails

The flow rate of concentrate, coarse and fine tails fluctuated considerably. The interface level varied significantly, and the fine tail pipe had to be adjusted to stabilize it. By doing this, at time 20 and 40 minutes in RUN No. 2, there was no fine tails overflow and the interface level was lost. It is found that the interface level control was very sensitive to the level of the tail pipe, as even a small adjustment resulted in a significant shift of the interface level in the column. For this reason, the interface level control by fine tails overflow is not recommended in continuous operation. However, the column worked well on batch operation. It is believed that in continuous operations, fluctuations in slurry and froth density make level control extremely difficult. In short batch tests, relatively little material (500 g) is fed slowly, and does not affect slurry or froth density significantly.

5.5 Summary

Using a rubber pipe for tails overflow up to the desired interface level worked well for batch operation because of the short operation time, small amount of feed, and lower column height. However, with this operational design, the column cannot achieve stable interface and yields unsteady results for continuous operation, since the interface level is very sensitive to adjustments of the rubber pipe level or changes in slurry density.

Batch operation showed that additional collector was necessary for increasing recovery in the concentrate since the Camchib ore which was sent to McGill University was partially oxidized, and stripped of its collector-bearing solution. Combining Aero 208 with a second collector gave better concentrate grades than Aero 208 alone, at constant copper recovery. The optimum addition rate was 30 g/t total collector. The collector combination of Aero 208 and Aero 355 gave the best grade among the collector combinations. This combination is used at the Camchib plant.

However, the improvement of flotation selectivity (chalcopyrite vs pyrite) and NSG is function not only of the choice of collectors but also the feed material. The three samples tested, first cleaner tails (CT1), second cleaner tails (CT2), and scavenger concentrate (SC1), have a slightly different behaviour, with the sample having the highest NSG content yielding the lowest NSG recovery (SC1), and the one with the highest pyrite content (CT2) yielding the lowest pyrite recovery. The ability to reject pyrite is evidenced by the high ratio of chalcopyrite recovery to pyrite recovery, which is at 3:1 for CT1, between 6:1 and 9:1 for CT2, and between 3:1 and 4:1 for SC1. The Leeds cell had generated ratios of 3:1 with a more finely ground retreater feed sample. NSG rejection is excellent in the column with all three samples, with gangue recoveries in the 2-4% range with the CT1 and SC1 samples, and 6-8% with the CT2 sample. The Leeds had yielded recoveries in the 10-14% range. The column, with its reduced height and retention time, yielded lower copper recoveries, 45-50% for the cleaner tails and 60-65% for the scavenger concentrate, vs. 65-70% for the retreater feed in the Leeds.

The effect of Na_2SO_3 from 0 to 10 g/t, was limited to an increase in gold recovery. Further testwork over a wider addition range will be necessary to confirm results.

CHAPTER 6

FLOTATION TESTWORK WITH A 5.7 CM DIAMETER COLUMN USING AUTOMATIC LEVEL CONTROL

6.1 Introduction

In chapter 5, a 3.8 cm diameter column with tails overflow interface control was tested. It gave unstable continuous operation and a steady interface level was not achieved. For future work, a 5.7 cm diameter column will be tested, with the conventional and more accurate method of interface control by hydrostatic pressure measurement with tail flowrate adjustments.

In this chapter, testwork will also consist of batch and continuous operation. The operational objective will be to assess the stability of interface control. In batch-scale tests, at McGill, Na_2SO_3 addition for pyrite depression will be tested while in continuous operation, the laboratory column will be run side by side with the retreater mechanical circuit. The metallurgical results of column and retreater cells will then be compared.

6.2 Column Description and Operation

The column used for batch operation at McGill University is made of plexiglas, 5.7 cm in diameter, 370 cm in height. The column consists of two sections. The upper section is fitted with a launder for concentrate removal. The lower section has a conical bottom for tailings removal and a port for a compressed air tube to feed a sparger. In addition, the lower section has another port for a pressure transducer. Feed is introduced 120 cm below the column lip through Tygon 9.5 mm (3/8") Cole-Parmer Masterflex tubing, and wash water 5 cm below the lip through a perforated copper tube. Wash water flow rate is measured with a calibrated rotameter (Cole-Parmer model FM-082-03). Air is admitted through the sparger and set to the pressure of 135 kPa (20 psi) at the desired flow rate which is manually controlled with a calibrated rotameter (Cole-Parmer model N102-05G). Wash water and air flowmeter calibration curves are shown in Appendix E. The experimental set-up is shown in Figure 6.1.

The interface level is controlled with a pressure transmitter (Foxboro model 843DP-H2I1NS-AM); it uses the principle of the resonant wire, whose two ends are connected to the low and high pressure diaphragms of the sensor. Its tension affects the resonance frequency of the wire, which is accurately measured with an oscillator. In this case, the low pressure end is connected to the atmosphere or vented, which makes it a suppressed-zero range application. The high pressure end is connected to the column through a 9.5 mm (3/8") vertical aluminum pipe, 20 cm high, filled with tetrabromoethane (specific gravity 2.9) in order to increase the hydrostatic pressure since the transmitter minimum pressure range is 190 cm, or 75 inches of water; the tetrabromoethane can also limit the migration of solids to the transmitter. The output signal (4-20 mA dc) of the transmitter is proportional to the hydrostatic pressure and directed to a PI controller (Foxboro single station micro controller model 761CNA-

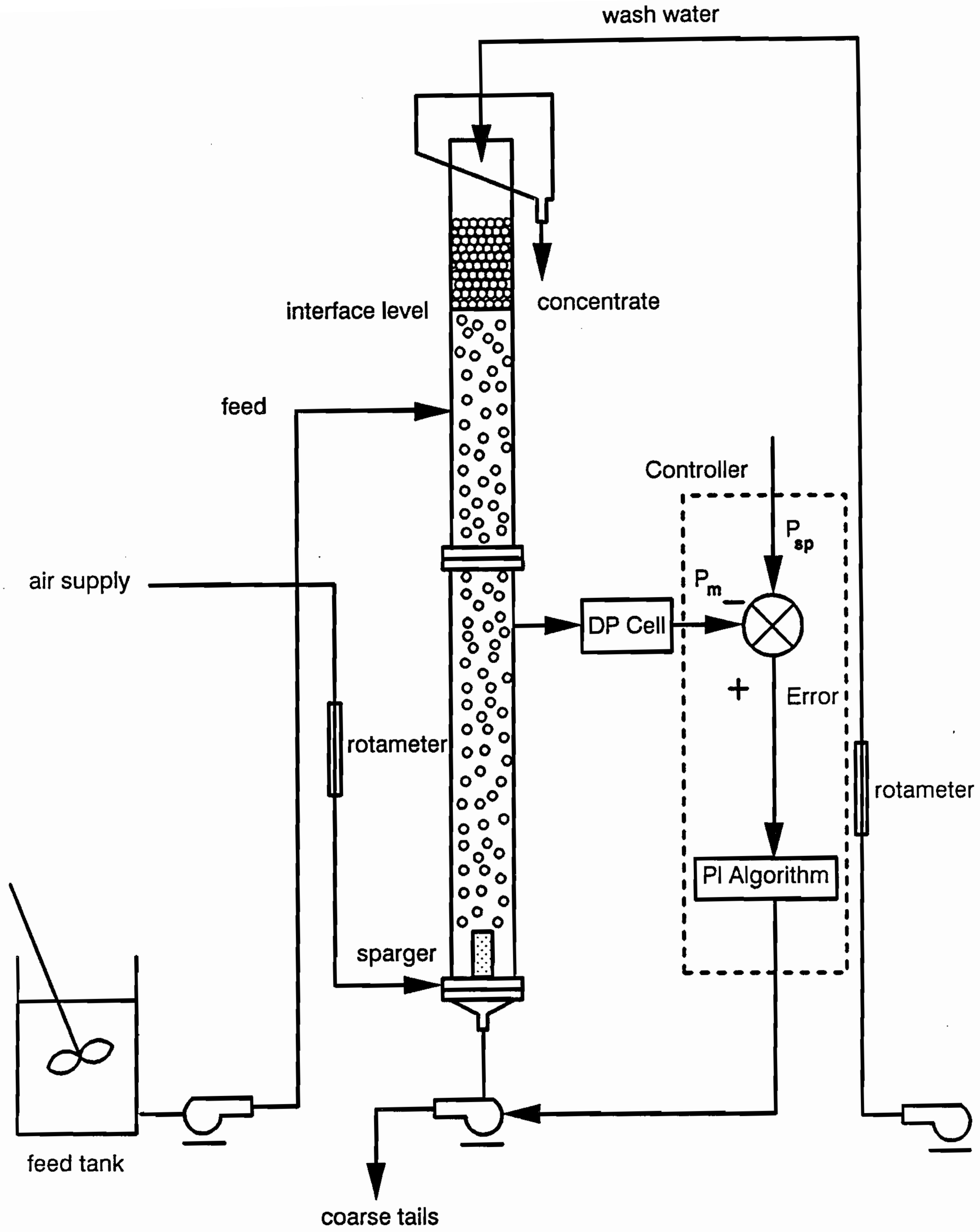

Figure 6.1 Experimental Set-up of a 5.7 cm diameter column with interface level controlled by a DP cell and a PI controller.

AT-1). The controller compares the input signal with the set-point and strives to eliminate the error which is the difference between pressure measurement and the set-point. Then, the controller sends the output signal (4-20 mA) which sets the speed of a varistaltic pump. Figure 6.2 shows the column experimental set-up with automatic control using a pressure transmitter and a PI controller.

In continuous operation at the Camchib plant, the column was 555 cm high (another section of 185 cm high was connected between the upper and lower sections of the batch operation).

6.3 Batch Operation

The cleaner tails and the scavenger concentrate streams from the Camchib flotation circuit were sampled, filtered at Camchib and sent to McGill University. The cleaner tails assayed 1.65 %Cu and 1.20 oz/st Au while the scavenger concentrate assayed 0.20 %Cu and 0.15 oz/st Au.

The objective of batch-scale testwork is to find adequate PI gains and assess interface stability. Two additional samples of cleaner tails (CT3) and scavenger concentrate (SC2) were tested to estimate Na_2SO_3 dosage.

6.3.1 Experimental Procedure

The column was first filled with water of 20 ppm of Dowfroth 250C and pH adjusted to 11.0 with lime. Wash water rate, feed rate, and air rate were kept constant at 0.53, 0.45, and 1.4 cm/s, respectively. The interface level was maintained at 70 cm below the column lip by setting

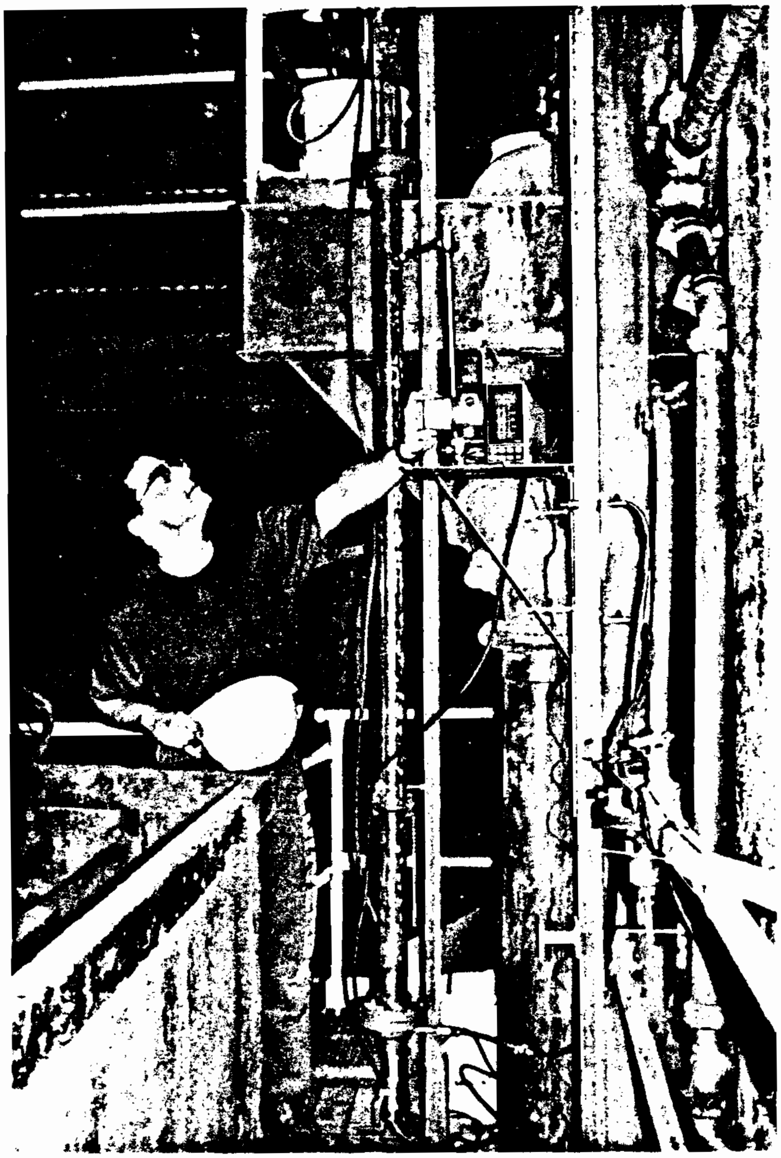

Figure 6.2 Column experimental set-up with automatic level control using a pressure transmitter and a PI controller.

the set-point in the controller set mode. The controller uses the concept of control band width, which is the inverse of the proportional gain. The band width and integral gains were set at 5% and 3 minutes, respectively.

For column feed preparation, 500 grams of the cleaner tails or the scavenger concentrate was ground with sodium sulfite in a laboratory stainless steel rod mill at 65% solids for 30 minutes to create fresh surface and obtain better mineral liberation. The slurry density was adjusted to 10% solids and a pH of 11.0 with lime, then conditioned for 5 minutes with 60 g/t of Aero 208 and 30 g/t of Aero 355 and for an additional 1 minute after Dowfroth 250C addition completed feed preparation (the high collector addition rate, above the optimum determined in Chapter 5, was aimed at achieving higher copper recoveries). The slurry was then fed to the column. Two minutes after feeding has been completed, the interface was raised to recover the froth content as concentrate, and the column was then flushed to recover the remaining tails. The concentrate and tails were filtered, dried, and assayed for Au and Cu.

6.3.2 Results and Discussion

The 5.7 cm diameter column with PI interface control system proved very reliable; the interface could be controlled within ±10 cm with no supervision and within ±5 cm with just occasional visual monitoring of the interface and set-point adjustment.

Results of the batch flotation using sodium sulfite are presented in Table 6.1. The effect of Na_2SO_3 on recovery and grade of the cleaner tails (CT3) and the scavenger concentrate (SC2) is shown in Figure 6.3 and 6.4, respectively.

Table 6.1 The effect of sodium sulfite on grade and recovery

Na_2SO_3 (g/t)	Stream	Mass (%)	Grade Au (oz/st)	Grade Cu (%)	Recovery Au (%)	Recovery Cu (%)
			Cleaner Tails (CT3)			
15	**Conc**	26	2.02	3.18	44	49
	Tails	74	0.91	1.17	56	51
	Feed	100	1.20	1.69	100	100
30	**Conc**	27	2.07	3.42	48	56
	Tails	73	0.84	1.01	52	44
	Feed	100	1.17	1.66	100	100
50	**Conc**	29	2.32	3.61	53	61
	Tails	71	0.82	0.92	47	39
	Feed	100	1.25	1.69	100	100
75	**Conc**	33	2.06	3.12	58	63
	Tails	67	0.74	0.90	42	37
	Feed	100	1.18	1.64	100	100
			Scavenger Concentrate (SC2)			
15	**Conc**	8	1.25	0.79	52	48
	Tails	92	0.10	0.07	48	52
	Feed	100	0.18	0.13	100	100
30	**Conc**	7	1.55	0.99	57	49
	Tails	93	0.09	0.08	43	51
	Feed	100	0.20	0.15	100	100
50	**Conc**	7	1.56	1.00	57	50
	Tails	93	0.09	0.08	43	50
	Feed	100	0.20	0.15	100	100

* Calculated feed grade

Na_2SO_3 addition increases copper and gold recovery, up to 30 g/t for the scavenger concentrate and at least 70 g/t for the cleaner tails. Concentrate grade is also improved, up to 30 g/t for the scavenger concentrate and 50 g/t for the cleaner tails. The higher requirement of Na_2SO_3 for the cleaner tails is likely attributable to its higher sulfide content.

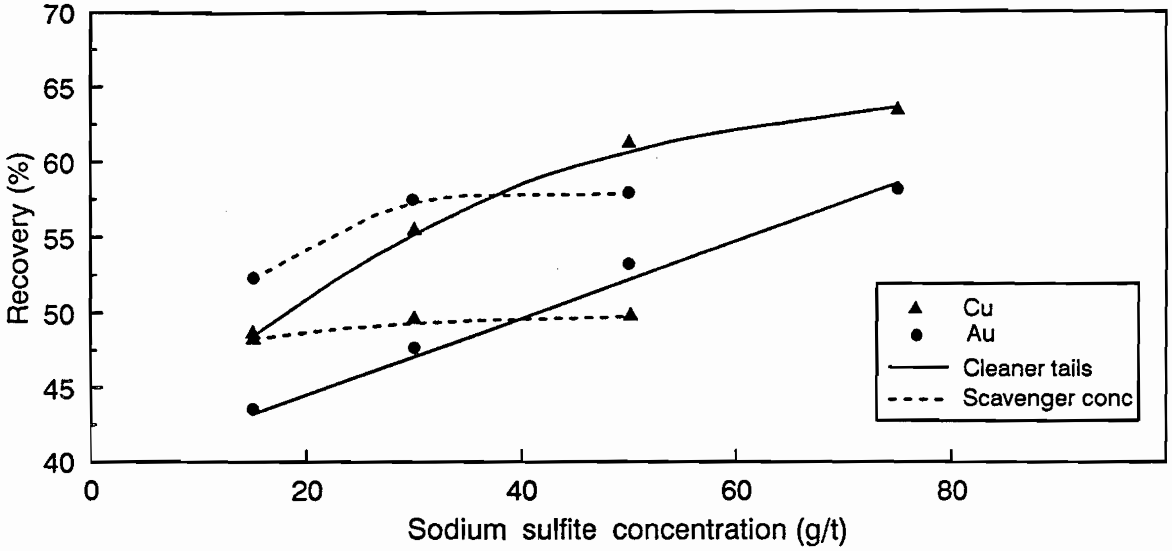

Figure 6.3 Effect of sodium sulfite addition rate on recovery.

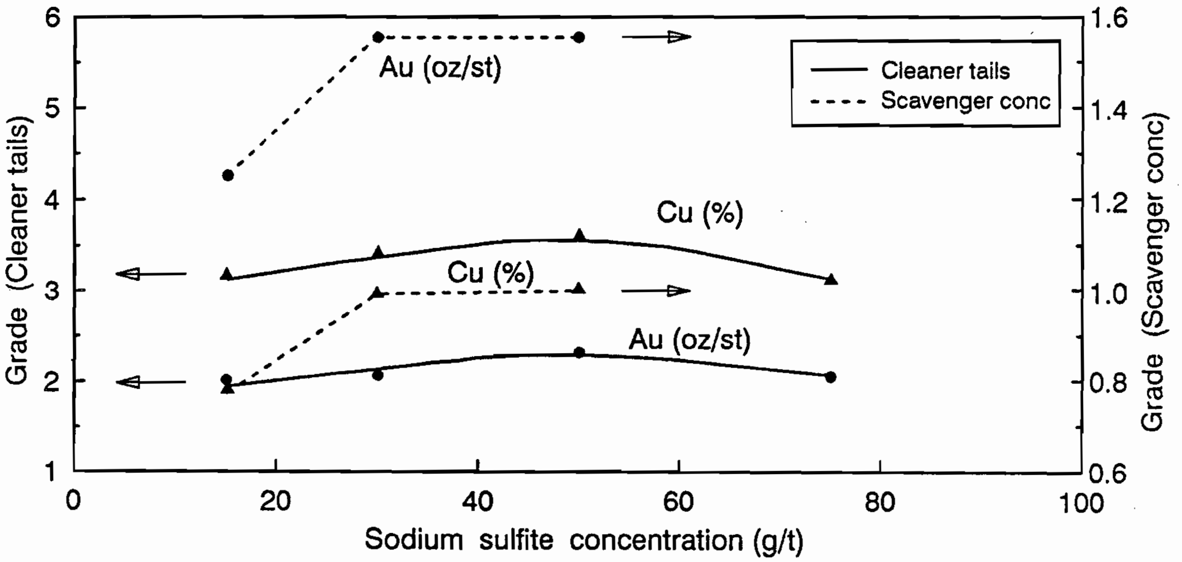

Figure 6.4 Effect of sodium sulfite addition rate on grade.

The enrichment ratio is much higher for the scavenger concentrate than the cleaner tails (e.g. 7:1 compared to less than 2:1 for Au). Significant amounts of non-sulfide gangue are entrained into the scavenger concentrate. This gangue would be readily rejected in the column, to yield the observed high enrichment ratio.

6.4 Continuous Operation

6.4.1 Experimental Procedure

Continuous operation was tested at Camchib, by tapping the retreater feed box to feed the column without any collector addition since the collector combination of Aero 208 and Aero 355 is added at the roughers and the scavengers at Camchib. However, 20 ppm of Dowfroth 250C was added to achieve a stable froth phase (MIBC - methyl isobuthyl carbinol - is used at Camchib); slurry pH was already adjusted at 11.0, for plant operation. Air was fed to the sparger at 135 kPa (20 psi). Wash water rate was kept constant at 0.35 cm/s. For each test, timed samples of the column concentrate and tails and untimed samples of the retreater feed, concentrate, and tails (all three banks) were extracted over a period of 30 minutes. Samples extracted were weighed, dried, weighed again, and analyzed for Cu and Au.

Column gas and feed rate and froth depth were systematically varied for eleven tests, with levels shown in Table 6.2. The objective was to run a 2^3 factorial design. Two additional tests at sodium sulfite addition rates of 50 and 100 g/t of retreater feed were completed (Tests 12 and 13).

Table 6.2 Operating variables for column experimental design

Variable → Level ↓	Froth depth (cm)	Feed rate (cm/s)	Gas rate (cm/s)
High	100	0.6	1.3
Medium	60	0.5	1.0
Low	25	0.3	0.8

Wash water rate is constant at 0.35 cm/s

6.4.2 Results and Discussion

The comparative metallurgical results of flotation column and mechanical cells are summarized in Table 6.3 for Cu and Table 6.4 for Au, respectively. Grades and flow rates were adjusted using the Noranda **NORBAL2** software (Spring, 1986) to satisfy known material balance constraints (Details are in Appendix F).

Table 6.3 Comparative metallurgical results of Cu with a 5.7 cm column and plant mechanical cells

Test	Grade (Cu, %)						Recovery		Flot.col yield (%)
	Mechanical cell			Flotation column *			Mech. cell (%)	Flot. column (%)	
	Feed	Conc	Tails	Feed	Conc	Tails			
1	0.29	0.97	0.05	0.26	1.81	0.10	86	68	9.14
2	0.31	1.57	0.03	0.33	1.15	0.11	91	73	20.74
3	0.32	1.30	0.06	0.29	1.22	0.11	85	72	16.30
4	0.30	1.76	0.04	0.28	1.74	0.13	90	63	10.15
5	0.62	3.57	0.05	0.50	1.56	0.14	93	85	26.69
6	0.39	2.30	0.05	0.40	1.46	0.13	89	74	19.92
7	0.38	2.09	0.05	0.35	1.58	0.11	90	76	15.52
8	0.33	2.40	0.05	0.33	1.29	0.13	87	67	17.32
9	0.45	3.53	0.06	0.45	1.67	0.15	88	73	19.59
10	0.38	3.28	0.06	0.27	1.07	0.13	87	75	16.28
11	0.57	4.43	0.07	0.55	2.02	0.19	89	73	19.64
12	0.43	2.99	0.04	0.36	3.10	0.15	91	69	7.48
13	0.26	2.01	0.03	0.31	2.48	0.10	90	64	8.23

* Calculated feed

Table 6.4 Comparative metallurgical results of Au with a 5.7 cm column and plant mechanical cells

Test	Grade	(Au, %)					Recovery		Mech. yield (%)
	Mechanical cell			Flotation column *			Mech. cell (%)	Flot. column (%)	
	Feed	conc	Tails	Feed	Conc	Tails			
1	0.43	1.42	0.09	0.38	2.54	0.18	84	63	25.41
2	0.44	1.88	0.12	0.43	1.52	0.17	77	68	18.01
3	0.46	1.70	0.13	0.41	1.65	0.18	78	68	20.85
4	0.41	2.04	0.11	0.41	2.33	0.18	77	62	15.40
5	0.57	2.56	0.19	0.50	1.40	0.15	72	82	16.04
6	0.49	2.40	0.15	0.48	1.76	0.18	74	71	15.04
7	0.47	2.13	0.14	0.41	1.85	0.16	74	72	16.40
8	0.39	2.34	0.12	0.40	1.40	0.18	72	61	11.86
9	0.49	3.20	0.15	0.48	1.71	0.20	72	68	11.20
10	0.46	3.06	0.17	0.35	1.33	0.15	66	77	10.00
11	0.58	3.67	0.19	0.57	2.05	0.21	72	72	11.43
12	0.46	2.71	0.12	0.40	3.23	0.16	78	68	13.23
13	0.33	2.06	0.10	0.38	2.97	0.14	73	60	11.74

* Calculated feed

The effect of froth depth, feed rate, and gas rate on recovery, yield, and enrichment ratio is shown in Figure 6.5 for Cu and Figure 6.6 for Au (Test 12 and 13 are not included). Both figures present the same trends. All points lie along an operating band which predicts that Cu and Au recoveries increase and enrichment ratios decrease with increasing yield. At high froth depth, high feed rate, and low gas rate (Point A), Cu and Au recoveries and yields are lowest while the enrichment ratio is highest but at high values of those parameters (Point D) yields and recoveries are highest while enrichment ratio is lowest. The variable which affects yield and recovery most significantly is gas rate. A high gas rate pushes more material into the froth zone; therefore, yield and recovery increase. Obviously, gangue minerals are also pushed to the froth zone; hence, the enrichment ratio drops. At medium froth depth, feed rate, and gas rate, recovery, yield, and enrichment ratio are

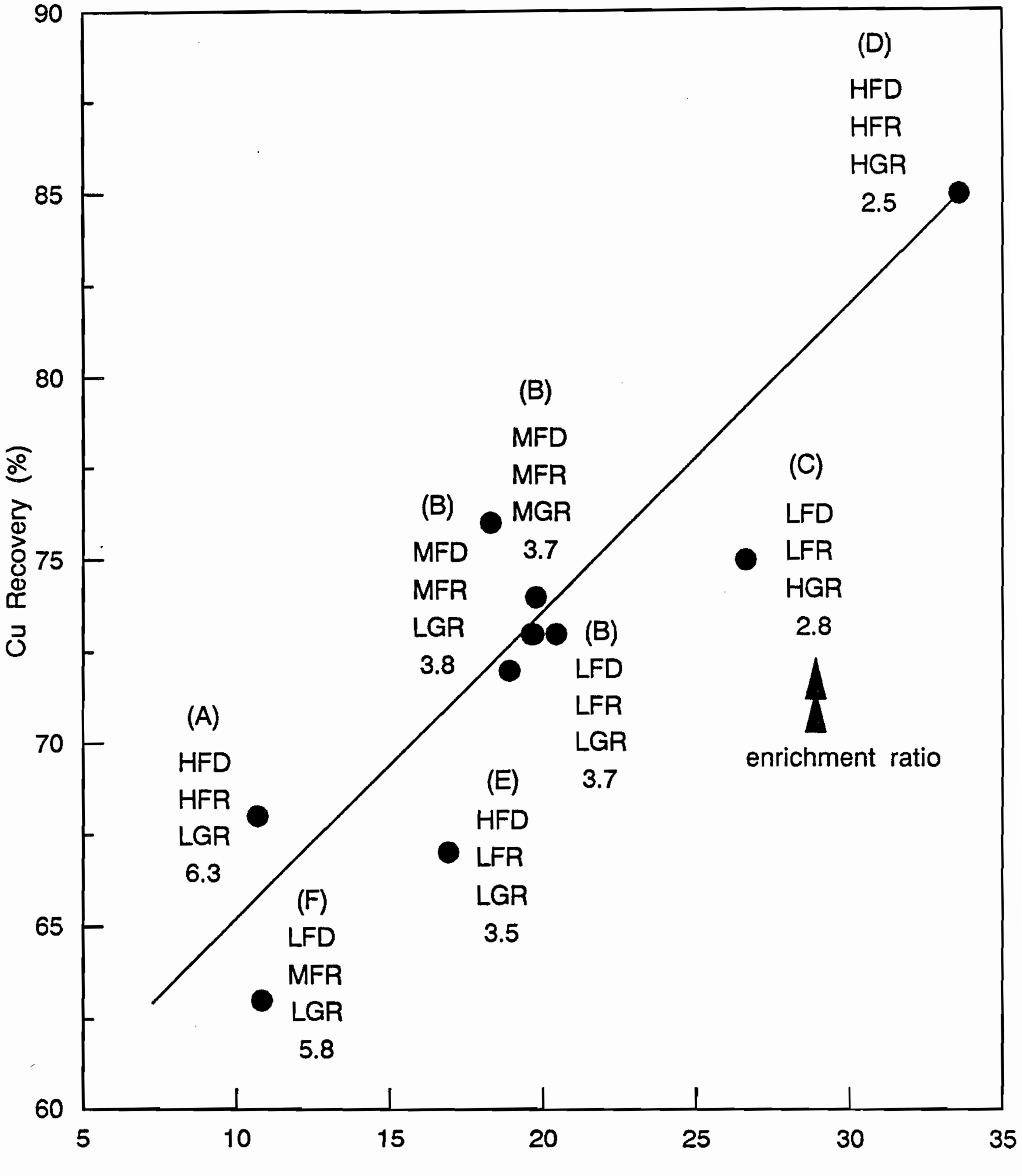

Figure 6.5 Effect of froth depth (FD), feed rate (FR), and gas rate (GR) on Cu recovery, yield, and enrichment ratio.

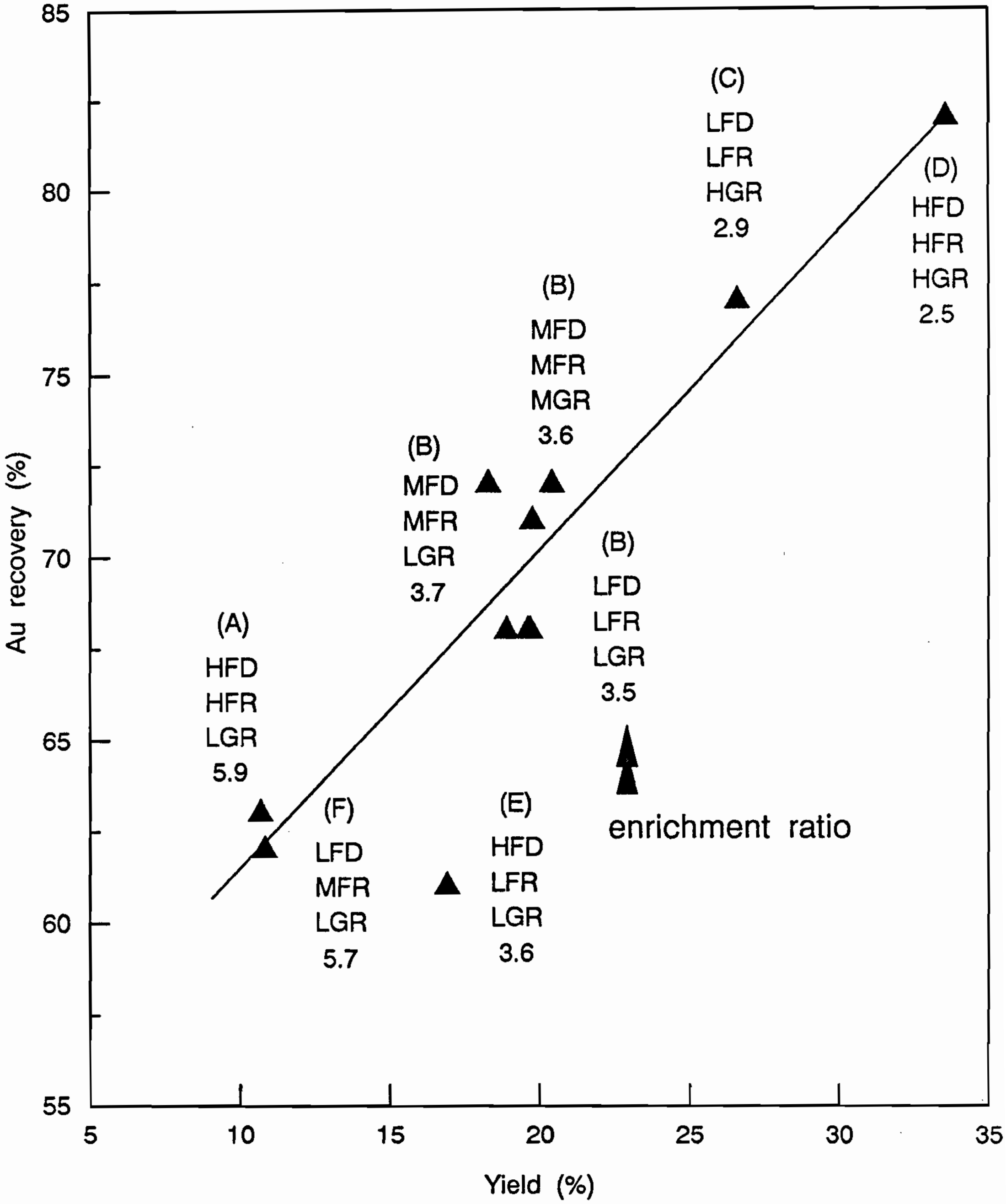

Figure 6.6 Effect of froth depth (FD), feed rate (FR), and gas rate (GR) on Au recovery, yield, and enrichment ratio.

intermediate. Metallurgical results are intermediate at the low values of those parameters as well (Group B). At low froth depth, low feed rate, and high gas rate (Point C), recoveries and yields are lower while the enrichment ratio is higher than at Point D. All points with the low feed rate are below the operating lines (except Point C in Figure 6.6), which suggests that high feed rates are beneficial to selectivity. An increased froth depth decreases recovery, but increases grade. Decreasing feed rate increases retention time in the collection zone; therefore, recovery increases.

Figures 6.7 and 6.8 compare Cu and Au recovery for the column and mechanical cells of the plant, respectively. The mechanical cells clearly outperform the column, achieving higher copper recoveries and similar gold recoveries, but at lower yield. The difference of metallurgical performance is high for Cu and lower for gold, as additional pyrite flotation in the column contributes to gold recovery. Adding Na_2SO_3 does yield more selective flotation, but at the expense of Cu recovery, more so at 100 g/t than 50. Mechanical cells also yield more consistent metallurgy, but much of the variability of the column data stems from the systematic variation of feed and gas rate and froth depth. Column recovery is significantly lower than that of mechanical cells at the same yield, especially for Cu, and to a lesser extent for Au.

The better metallurgical performance of the mechanical cells is partly due to a higher retention time, about 40 minutes (Wibisono, 1985), compared to 8 to 25 minutes in the column. However, the cells are more selective, as equivalent Cu recoveries can be achieved at much lower yields. This is further confirmed when Na_2SO_3 is used in the column to increase selectivity. Yield can then be dropped below that of the cells,

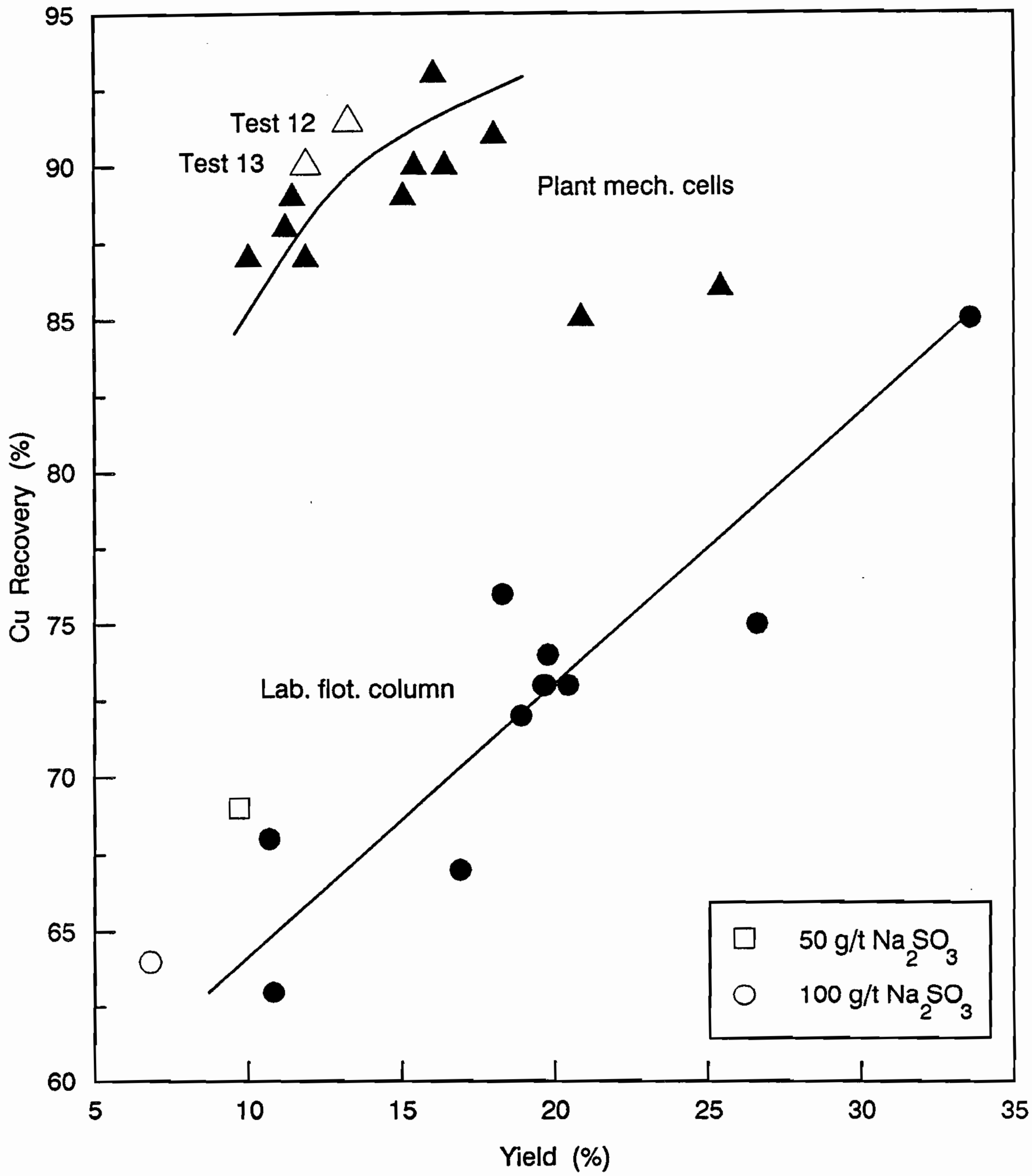

Figure 6.7 Comparative metallurgy of the flotation column and mechanical cells for copper.

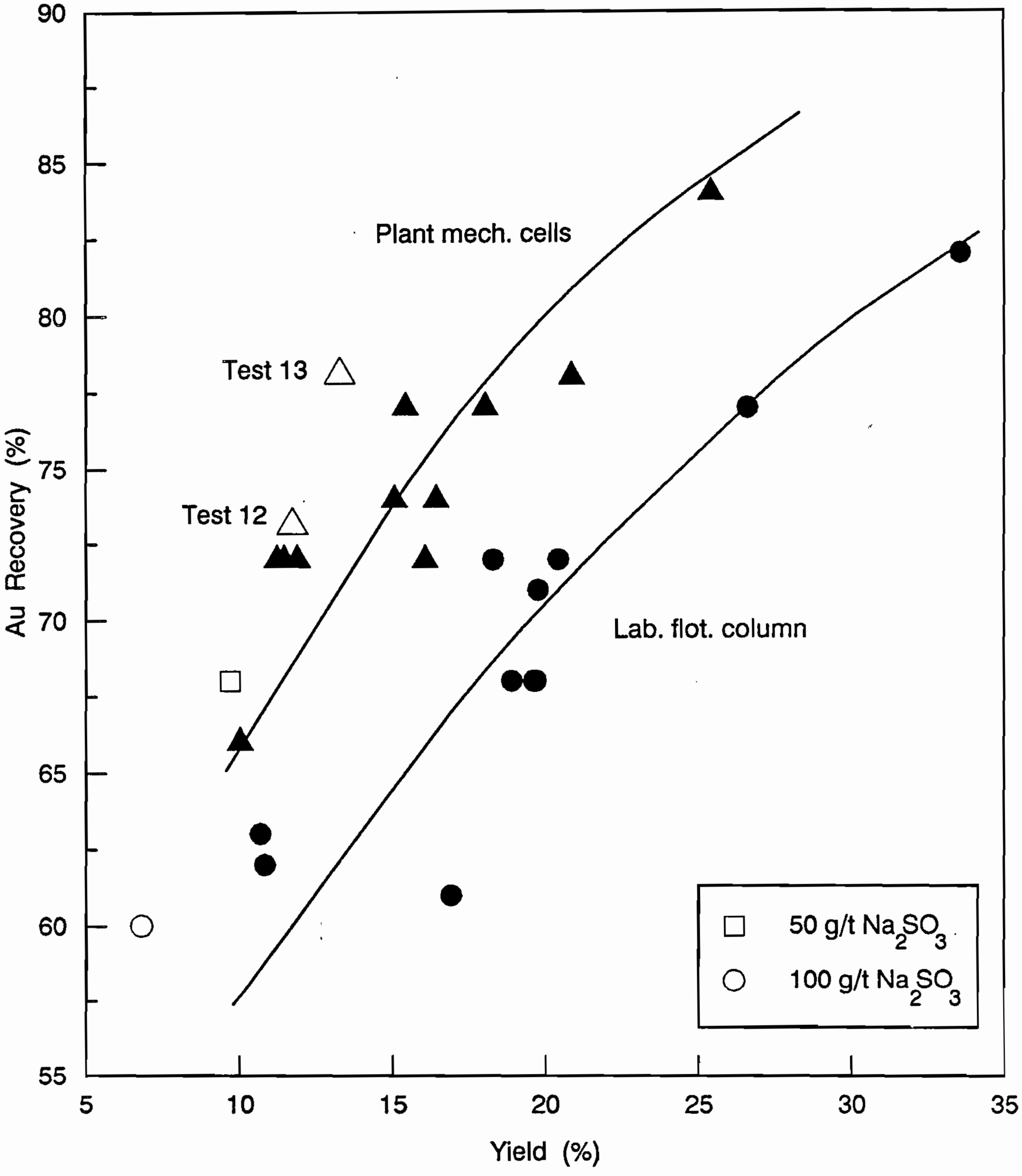

Figure 6.8 Comparative metallurgy of the flotation column and mechanical cells for gold.

but at the expense of Cu and Au recovery.

The different level of aeration in both systems most likely plays a more important role than retention time. Testwork with the Leeds cell has already demonstrated the effectiveness of pre-aeration to enhance chalcopyrite/pyrite selectivity. Table 6.5 shows that aeration, expressed in air flowrate per effective volume of slurry, is on average eight times lower in the column than in the mechanical cells; 1.18 min^{-1} in Fagergren 66 (Harris, 1976), compared to only 0.14 min^{-1} in the 5.7 cm column.

Table 6.5 Comparative aeration of the plant cells and the 5.7 cm diameter column

Equipment	Aeration	Effective volume	Aeration per effective vol. (min^{-1})
Fagergren 66 mech. cell	60 ft^3/min	51 ft^3	1.18
5.7 cm Column	1515 cm^3/min (J_g = 1 cm/s)	11 L	0.14

Aeration has a pronounced effect on chalcopyrite flotation, particularly in an oxidizing environment, and if selectivity from pyrite is sought. The adsorption of collector on chalcopyrite surface is probably inhibited in a reducing environment or at low oxygen concentration. The results in chapter 4 show that metallurgical performance is improved with pre-flotation aeration; presumably the retreater cells, with a higher aeration rate, achieve better chalcopyrite flotation with increased selectivity from pyrite.

6.5 Mineralogy

The liberation of pyrite and chalcopyrite in the column and cells concentrate was studied with the optical microscope. The 53-75 µm (200-270 mesh) size fraction is shown in Figure 6.9 while the 37-53 µm (270-400 mesh) fraction is in Figure 6.10. Chalcopyrite is softer and yellower than pyrite; the colour and the surface damage on chalcopyrite distinguishes it readily from pyrite. For both sizes, most of pyrite and chalcopyrite is liberated. This confirms that poor pyrite/chalcopyrite separation is a selectivity rather than a liberation problem.

Gold-bearing pyrite in the column concentrate is shown in Figure 6.11 (back-scattered image with the scanning electron microscope). In the Camchib flotation circuit, the retreater concentrate is returned to the roughers. Then, gold-bearing pyrite particles such as the one shown could be lost in scavenger tailings instead of being recovered by cyanidation.

Selected samples of the column concentrate and the retreater concentrate (tests 5 to 12) were assayed for sulfur. Iron and NSG contents were calculated. Data are presented in Table 6.6.

Table 6.6 Analysis of the column and the retreater concentrate

Test	Column concentrate				Retreater concentrate			
	S (%)	Cu (%)	Fe (%)	NSG (%)	S (%)	Cu (%)	Fe (%)	NSG (%)
5	37.86	1.56	31.66	27.55	35.14	3.57	27.52	30.63
6	34.25	1.46	28.60	34.41	31.24	2.30	25.23	39.21
7	32.95	1.58	27.36	36.72	30.91	2.09	25.13	40.03
8	35.71	1.29	30.03	31.83	34.00	2.40	27.55	33.94
9	34.08	1.67	28.27	34.51	32.26	3.53	25.04	36.07
10	33.67	1.07	28.44	35.88	32.78	3.28	25.71	35.35
11	32.82	2.02	26.86	36.52	31.48	4.43	23.56	36.64
12	21.54	3.10	16.06	56.58	20.26	2.99	15.04	59.08

Figure 6.9 Liberated pyrite and chalcopyrite in the 53-37 micron (200-270 mesh).

Figure 6.10 Liberated pyrite and chalcopyrite in the 37-53 micron (270-400 mesh).

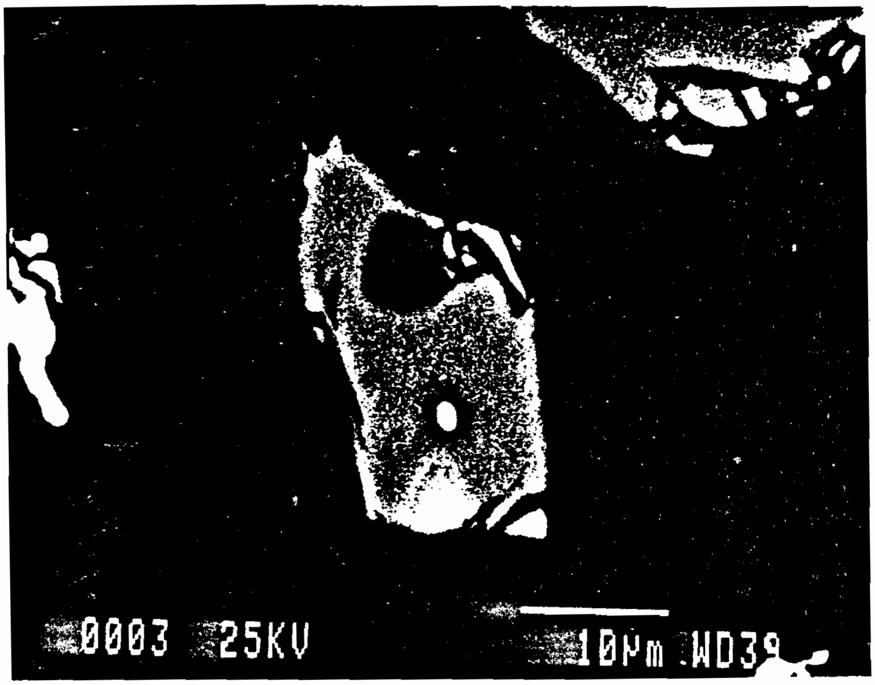

Figure 6.11 Gold-bearing pyrite in the column concentrate using backscattered electron imaging.

Significantly, the sulfur content of the column concentrates was systematically higher than that of the cells, by about 2%, despite the higher column yields; this confirms that column can reject hydrophilic NSG more effectively than the cells, but fails to be as effective with pyrite. The column and the retreater concentrates of test 5 to test 11 assayed 31 to 38% sulfur while in test 12 sulfur content was low, about 21%. More NSG was recovered in test 12. The grade of the retreater feed varied considerably. X-ray diffraction analysis was used to determine the mineral content of the column concentrate from tests 5 and 12; spectra are shown in Appendix G. The concentrates consisted of pyrite, chalcopyrite, pyrrhotite (minor amounts), and NSG which were mostly quartz and lesser amounts of chlorite. The concentrate of test 12, which had a much lower sulfur content, had lower pyrite peaks than test 5, but higher peaks corresponding to quartz and chlorite. Quartz does not have good natural floatability, even in the presence of thio collectors; it is therefore likely that most quartz recovered is unliberated. Chlorite, on the other hand, is a phyllosilicate with a platy or flaky habit, and a dominant cleavage plane, along the sheet structure. Cations are bonded to the sheet structure (Klein and Hurlbut, 1985); when breakage takes place along the cleavage plane, chlorite can behave as a weakly hydrophobic mineral, which could explain its recovery. NSG content in the cell and column concentrate is strongly correlated (coefficient of correlation: 0.96), which would suggest that it is more function of solution chemistry (identical in both case) or mineralogy, rather than flotation physics.

6.6 Cyanidation of Tails

Cyanidation tests were performed on all column and mechanical cells tails, at a pH of 11.0, a cyanidation of 5 kg/t, and a cyanidation time of 12 hours. Cyanidation data are summarized in Table 6.7.

Cyanide consumption is higher for the column tails, 2.1 kg/t vs 1.6 kg/t for the retreater tails, but not as high as the increase in Cu content would suggest, 0.13% vs 0.05% Cu. Interestingly, gold recovery is also slightly higher for the column tails, 73% vs 71% for the retreater cells tails, which implies that if the additional gold rejected to the column tails is associated with Cu, it has no deleterious effect on gold recovery. Because gold content in the column tails is higher, more gold is dissolved from the column tails 5.8 vs 4.4 g/t (0.17 vs 0.13 oz/st).

Table 6.7 Summary of cyanidation testwork

	Column Tails				Retreater Tails			
Test	%Cu	NaCN Cons. (kg/t)	Au Diss. (oz/st)	Au Rec. (%)	%Cu	NaCN Cons. (kg/t)	Au Diss. (oz/st)	Au Rec. (%)
1	0.110	1.608	0.159	70	0.051	1.043	0.109	72
2	0.110	1.643	0.142	75	0.033	1.470	0.114	73
3	0.110	1.678	0.180	77	0.060	1.776	0.099	70
4	0.120	1.816	0.173	73	0.035	1.856	0.086	67
5	0.150	2.681	0.198	74	0.055	1.631	0.171	72
6	0.130	2.537	0.171	72	0.049	1.458	0.141	71
7	0.120	1.493	0.152	72	0.046	1.343	0.091	64
8	0.130	1.943	0.170	78	0.048	1.216	0.120	71
9	0.160	2.797	0.203	73	0.062	1.764	0.171	76
10	0.100	2.208	0.158	77	0.057	1.989	0.135	71
11	0.190	2.318	0.196	66	0.068	1.706	0.267	74
12	0.130	1.891	0.164	73	0.043	2.220	0.098	68
13	0.110	2.145	0.155	74	0.031	1.885	0.074	70
Average	0.128	2.058	0.171	73.4	0.049	1.643	0.129	70.7

6.7 Summary

The failure of the overflow level control system led to the use of a more conventional approach: monitoring and controlling hydrostatic pressure in the column with tails flow rate. Although this method is not entirely satisfactory for plant operation, as shifts in slurry or froth density can cause the interface to drift, it is perfectly acceptable for lab or pilot operation since the interface can be monitored visually and corrections to the pressure set-point can offset observed drift in interface height.

The metallurgical performance of the flotation column was inferior to that of the cells. This was attributed to a selectivity problem (chalcopyrite from pyrite) rather than poor liberation, as confirmed by observation under the microscope and X-ray diffraction that the pyrite reporting to the flotation concentrate of the column and cells was largely liberated, hence hydrophobic. Silicates reporting to the concentrate were mostly quartz, probably unliberated, and chlorite, which may have a low floatability. The column rejected hydrophilic NSG more effectively than the cells, despite its failure to be as effective with pyrite. It can therefore be concluded that the column yielded more selective physical flotation, but a less selective chemical environment than the mechanical cells.

Aeration rate is an important parameter to consider since it increases oxygen concentration in the pulp which causes an oxidizing environment and activates chalcopyrite flotation, whilst depressing pyrite. Na_2SO_3 showed good potential to depress iron sulfides. Its use should be ideally combined with pulp pre-aeration before feeding to the

Operating variables should be controlled, since they affect metallurgical results. As expected, increasing gas rate increases recovery, but pushes more gangue into the concentrate. Decreasing feed rate gives the same results, because of the higher retention time in the collection zone; weakly hydrophobic particles have more chance to attach to bubbles and are carried into the froth zone. Froth depth as well should be controlled to have sufficient collection zone height; however, the effect of froth depth in the column was small, which suggests that accurate control is unnecessary.

CHAPTER 7

SUMMARY, DISCUSSION, AND CONCLUSIONS

7.1 Summary

7.1.1 Testwork with a Leeds cell

Salient results of this work are presented in Table 7.1. The sample used was the retreater feed assaying 0.30 oz/st Au and 0.28 %Cu. The objective was to maximize the selectivity of chalcopyrite from pyrite. To justify the choice of pyrite depressant for good selectivity, preliminary testwork was done with a Leeds cell. It was found that potassium ferrocyanide ($K_4Fe(CN)_6$) is effective pyrite depressant, with a ratio of chalcopyrite:pyrite recovery of about 5. The chalcopyrite:NSG recovery is about 5.5. At 30-50 g/t $K_4Fe(CN)_6$ showed the best selectivity with an enrichment ratio of 5.5 for Cu and 4.6 for Au. However, it also dissolved gold (about 0.03 to 0.07 g/t); this precludes its use at Camchib since gold could be lost in the flotation circuit if ferrocyanide-bearing solution was recycled.

Sodium sulfite 's potential for pyrite depression was almost as good as $K_4Fe(CN)_6$ when used with pre-aeration. More importantly, higher copper recoveries were achieved, 77-79%, compared with 70% with $K_4Fe(CN)_6$. The selectivity of chalcopyrite from pyrite or NSG was improved with pre-aeration, which promotes chalcopyrite flotation.

Sulfur dioxide (SO_2) showed some capacity for pyrite depression.

When SO_2 was used, pre-aeration did not show improved chalcopyrite/pyrite or chalcopyrite/NSG selectivity (section 4.4.4). The performance of SO_2 was not as good as either $K_4Fe(CN)_6$ or Na_2SO_3.

Consequently, Na_2SO_3 was selected as a pyrite depressant for further testwork.

Table 7.1 Comparative results of using pyrite depressants with a Leeds cell on enrichment ratio and recovery

Pyrite Depressant	Enrichment ratio		Recovery (%)				Recovery ratio	
	Au	Cu	Au	Chal	Fe	NSG	Chal:Fe	Chal:NSG
$K_4Fe(CN)_6$								
0 g/t	4.7	4.3	72	66	18	14	3.7	4.7
30 g/t	4.7	5.2	63	70	14	13	5.0	5.4
50 g/t	4.6	5.7	54	66	13	10	5.1	6.6
100 g/t	3.6	4.1	44	49	11	12	4.5	4.1
Na_2SO_3								
30 g/t	2.2	2.6	68	79	36	24	2.2	3.3
50 g/t	2.7	3.4	69	80	26	22	3.1	3.6
100 g/t	3.4	3.2	68	65	21	19	3.1	3.4
Na_2SO_3								
0 g/t, aeration	3.6	4.0	66	73	23	15	3.2	4.9
50 g/t, aeration	4.3	4.4	74	79	22	14	3.6	5.6
50 g/t, no aeration	3.7	4.0	72	77	24	16	3.2	4.8
Sulfur dioxide (20 ml SO_2/kg ore)								
pre-aeration	2.5	2.8	73	82	34	26	2.4	3.2
no aeration	2.9	2.8	70	68	27	23	2.5	3.0

7.1.2 Testwork with column flotation

A number of variables in column flotation operation have to be controlled. Gas rate has considerable affect on metallurgical results. With high gas rates, the entrainment of gangue mineral increases while with low gas rate, recovery is low. Hence, superficial gas rate should be controlled. Another variable is interface level or froth depth. In this thesis, the interface level was first controlled by using a large diameter (3 cm) rubber pipe of tails overflow running up to the desired froth depth. Column operation achieved stable interface for batch-scale tests, but not for continuous tests since the interface level was very sensitive to the adjustment of the large rubber pipe level. A second approach was to control the interface level of a 5.7 cm diameter column by varying tails flowrate with a PI controller, with a hydrostatic measurement. This seems to be the simpler column set-up for metallurgical testing that can yield reliable results. Although continuous operation showed that the interface drifted slightly, periodic adjustment of the set-point achieved very tight interface control (within 5 cm).

The samples tested at McGill University had been stripped of their collector-bearing solution and partially oxidized. The samples were reground and collector addition was tested. The collector combination of Aero 208 and Aero 355 gave the best grade among the other collector combinations.

With a 3.8 cm diameter column flotation, Na_2SO_3 was tested with scavenger concentrate sample (SC1) assaying 0.10 oz/st Au and 0.28 %Cu. The results showed high selectivity of chalcopyrite with respect to NSG. Enrichment ratios were very high, about 8 (see Table 7.2). This is probably due to mode of operation of the scavenger cells at Camchib which recover significant free gangue by hydraulic entrainment. Low additions of Na_2SO_3, 5 to 10 g/t, did not alter column performance significantly.

Results from column flotation with Na_2SO_3 addition are summarized in Table 7.2.

Table 7.2 Results from column flotation with Na_2SO_3 addition on enrichment ratio and recovery

Feed material	Na_2SO_3 (g/t)	Enrichment ratio		Recovery (%)	
		Au	Cu	Au	Cu
With 3.8 cm column (batch test)					
scavenger conc.(SC1) 0.1 o/t Au, 0.28 %Cu	0	8.2	11.0	47	64
	5	7.1	8.2	59	65
	10	8.9	8.2	65	63
With 5.7 cm column (batch test)					
cleaner tails (CT3) 1.2 o/t Au, 1.65 %Cu	15	1.7	1.9	44	49
	30	1.8	2.1	48	56
	50	1.9	2.1	53	61
	75	1.7	1.9	58	63
scavenger conc.(SC2) 0.15 o/t Au, 0.2 %Cu	15	7.0	6.1	52	48
	30	7.8	6.6	57	49
	50	7.8	6.7	57	50
With 5.7 cm column (continuous test)					
retreater feed 0.45 o/t Au, 0.4 %Cu (test 12)	50	7.0	7.2	68	69
0.3 o/t Au, 0.27 %Cu (test 13)	100	8.9	9.4	60	64

Na_2SO_3 at higher concentrations, up to 100 g/t, was also added for both batch and continuous operation of a 5.7 cm diameter column with automatic level control. Results confirmed that the scavenger concentrate (SC2) would be upgraded more easily than the cleaner tails (CT3, batch operation). In continuous operation with the retreater feed, enrichment ratios were high and recoveries of Au and Cu were higher than for batch operation.

Comparative metallurgical results were generated by running the flotation column parallel with the retreater mechanical circuit. The column was outperformed by the cells both in grade and recovery (see Tables 6.3 and 6.4).

Microscopic examination of the flotation products confirmed the pyrite reporting to the concentrate of the cells and column was mostly liberated. This indicated that lower pyrite/chalcopyrite selectivity in the flotation column was responsible for the poorer performance. Aeration rate per effective volume, 0.14 min^{-1} in the column while 1.18 min^{-1} in the cells, is the probable cause. The adsorption of collector on chalcopyrite is possibly inhibited at low oxygen concentration. X-ray diffraction showed that silicates (largely quartz, probably unliberated, and chlorite, which may be weakly hydrophobic) also report to the flotation concentrate.

By changing operating variables, such as froth depth, feed rate, and gas rate, as a 2^3 factorial design, it was found that gas rate was the most significant variable in this system, affecting yield and the recovery of copper and gold. Increasing gas rate pushed more particles (including gangue particles) into the froth zone; hence, yield and recovery increased but grade dropped. Changes in froth depth did not affect metallurgical performance significantly. Decreasing feed rate yielded higher retention times in the collection zone; therefore, the recovery increased.

7.2 Discussion

7.2.1 Interface Stability

Failing to achieve a stable interface with the continuous overflow discharge column makes this option unattractive. The increased solid loading upon continuous feeding is partly responsible for this failure; another major cause is the increased height of the column. The plug flow transport in the recovery zone compounds the problem, as the froth phase first, and then the recovery zone become filled with solids before the tails line does. This causes a significant shift in the interface, which worsens with increasing column height. Using a pressure transducer located just below the interface minimizes the impact of recovery zone density, and yields adequate interface stability.

7.2.2 Effect of Aeration

The better metallurgical performance of the mechanical cells is partly due to a higher retention time, about 40 minutes (Wibisono, 1985), compared to 8 to 25 minutes in the column. However, the level of aeration also plays a significant role. Testwork with the Leeds cell has already demonstrated the effectiveness of pre-aeration to enhance chalcopyrite/pyrite selectivity. Plaksin (1959) reported that the floatability of chalcopyrite increased with increasing oxygen. Similar findings were reported by Heyes and Trahar (1977) and Gardner and Woods (1979). Pre-aeration can also assist in depressing pyrite (Spira and Rosenblum, 1978).

The level of aeration, expressed in air flowrate per volume of slurry (min^{-1}), is on average eight times lower in the column than in the mechanical cells. To that lower specific aeration rate must be added the sitting time in the feed tank of the flotation column, which was about 2 minutes at the highest feed rate, but 5 to 10 minutes at the lowest feed rate. If both effects are significant, it follows that high feed rates and gas rates should yield better separations. To test this hypothesis, a multiregression was performed in the column results to characterize copper recovery (R_{Cu}) as a cause is the increased height of the column. The plug flow transport in the recovery zone compounds the problem, as the froth phase first, and then the recovery zone become filled with solids before the tails line does. This causes a significant shift in the interface, which worsen with increasing column height. Using a pressure transducer located just below the interface minimizes the impact of recovery zone density, and yields adequate interface stability.

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$$R_{Cu} = 46.5 + (0.566 \pm 0.176) * Y + (10.9 \pm 5.3) * J_g + (14.2 \pm 8.1) * J_f$$

Although this regression has a low correlation coefficient, 0.81, it fits copper recovery with a mean residual of 2.9%. More importantly, it agrees with assumed effect of gas and feed rates; the two regression coefficients for these two variables are significant, at 95% for gas rate and 90% for feed rate.

7.2.3 Improved Plant Operation

The column's inability to reject pyrite more effectively than the retreater cells makes the column option unattractive. The column can reject NSG more effectively, but the improvement is marginal, and has little economic impact, as the retreater concentrate is further cleaned thrice prior to reporting in the final concentrate (roughers, cleaners, and recleaners). However, results opened some avenues worth exploring. First, the production of a higher copper content tails does not appear to increase cyanide consumption unduly, 0.5 kg/t of retreater tails. Gold recovery is actually higher for the higher copper content column tails, which implies that gold recovery is not affected either. This is in line with the findings of Stamboliadis et al. (1976), who concluded that as long as free cyanide levels were appropriate, gold cyanidation was not affected by the presence of dissolved copper. There is little doubt that the increase in pyrite rejection warrants the increased cyanide consumption, if we compare the net smelter value of the material cyanided, typically 5 g/t Au and 0.10% Cu. For example, with a transport and smelter charge of $200/t and a gold payment of 96.5% at $C400/oz., the NSV is actually negative, as the $C62 payment for gold cannot offset the $C200 treatment charge. Cyaniding this material with a gold recovery of 70% would yield gold values of $C45, which would more than offset the cyanidation cost (evaluated at $C10/t at Camchib).

Greater pyrite rejection to the cyanidation plant can be achieved in two ways. First, the use of a depressant in the retreater stage should be investigated. Pre-aeration may be the cheapest depressant, as a conditioning tank is already available on site, in the vicinity of the regrinding circuit and retreater cells. Pre-aeration could also strip the pyrite of adsorbed collectors (dixanthogen and dithiophosphate) to make it more responsive to cyanidation (Mrkusic et al., 1968).

7.3 Conclusions

The present case study has shown that the choice of column flotation to replace mechanical cells can at times be determined by factors other than the ability of column to reject hydrophilic fines. Whereas selectivity (i.e. iron sulfide rejection) has been reported to improve in some columns, the opposite can actually happen, when aeration requirements are high. It is interesting to note that the first successful commercial application of columns, at Mines Gaspé, was in a $MoS_2/CuFeS_2$ separation circuit for which oxygen plays a deleterious role, as it consumes the $CuFeS_2$ depressant, Na_2S. It has been suggested by some that this separation could be achieved with nitrogen flotation, to minimize Na_2S consumption.

The use of the overflow technique for interface control should be restricted to batch tests with limited feed mass, as the technique proves unstable when enough feed is introduced to change slurry density in the recovery and froth zones. Stability becomes particularly difficult to achieve when the froth and recovery zone height is extended. Severe problems were experienced when the column height was extended to 5 m in continuous operation.

7.4 Suggestions for Future Work

The column with tails overflow interface control proved easy to use for small batch tests; this use should be further investigated.

Testwork with pre-flotation aeration indicates that oxygen concentration in the pulp has a pronounced effect on chalcopyrite recovery. In future testwork, the slurry should be pre-aerated in a conditioning tank before feeding to the column. In addition, oxygen concentration and pulp potential should be closely monitored.

To obtain higher selectivity of chalcopyrite versus pyrite, the use of a depressant in the retreater circuit at Les Mines Camcib Inc. should be investigated. Na_2SO_3 is an attractive pyrite depressant. Its use should be examined with pre-aeration flotation to regain additional selectivity. Moreover, pre-aeration could strip the pyrite of adsorbed collector to render it more responsive to cyanidation.

To achieve better selectivity, a higher copper content in the tails must be accepted; however, this does not seem to increase cyanide consumption unduly. Interestingly, gold recovery is higher for the higher copper content in the tails (section 6.6). The scale of both the cyanide consumption and gold recovery increases should be determined at full plant scale.

Metabisulphite, hydroxyl, and hyposulfite are other pyrite depressants, and could also be tested to improve the selectivity of chalcopyrite versus pyrite.

The retreater concentrate averages a copper grade of 2 to 3%, which is much higher than the rougher feed, typically 0.3% Cu. Further, its yield is quite low, in the 10 to 15% range, which is much less than the present feedrate to the cleaners. This suggests that even without increasing retreater selectivity, its concentrate can be directed to the first cleaners, rather than roughers. This would eliminate altogether the possibility of retreater concentrate escaping cyanidation. This change should be implemented in the plant and carefully monitored.

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APPENDIX A

Investigation on gold dissolution by potassium ferrocyanide

Investigation on gold dissolution by potassium ferrocyanide.

Test procedure

1. Put 100 gram ore and 100 ml water into volumetric flask.
2. Adjust the pulp pH to 11.
3. Add 0.05 g or 0.1 g potassium ferrocyanide $K_4Fe(CN)_6$ into the pulp to give a strength of 0.05% and 0.1% respectively.
4. Start stirring the flasks.
5. After 30 minutes, remove the flask from the stirrer, filter the pulp.
6. Repeat procedure 5 for times of 1, 2, and 4 hours.
7. Analyze the solution for gold.

Results

Table A1 Effect of potassium ferrocyanide on gold dissolution.

$K_4Fe(CN)_6$ concentration (%)	Retention Time (hr)	Dissolved gold (oz/st)
0.05	0.5	0.0023
	1.0	0.0013
	2.0	0.0010
	4.0	0.0013
0.10	0.5	0.0017
	1.0	0.0013
	2.0	0.0019
	4.0	0.0013

APPENDIX B

The effect of sodium sulfite as a depressant on the Camchib ore using a Leeds Cell

Table B1 Metallurgical results of flotation test at 30 g/t Na_2SO_3 on the Camchib retreater feed

Conditioning:	Time	5 minutes
	pH	11.2 with lime
	Collector	Aerofloat 208, 100 g/t
	Frother	Dowfroth 250C, 10 g/t
	Gas used	Air, 5 L/min

Product	Time (min)	Mass (%)	Grade Au (oz/st)	Cu (%)	(**) Fe (%)
Conc 1	0.5	7.34	0.44	0.33	26.71
Conc 2	1.0	5.67	0.67	0.66	27.77
Conc 3	1.5	3.89	0.80	1.08	28.25
Conc 4	2.0	2.44	0.80	1.08	26.45
Conc 5	3.0	3.07	0.84	1.00	26.12
Conc 6	5.0	3.34	0.68	0.75	26.24
Conc 7	7.0	3.25	0.62	0.58	26.09
Conc 8	10.0	1.68	0.66	0.58	25.39
Tails		69.32	0.13	0.08	20.78
Feed (*)		100.00	0.29	0.27	22.64

Metallurgical results

Product	Cumulative grade Au (oz/st)	Cu (%)	Fe (%)	Cumulative recovery Au (%)	Cu (%)	Fe (%)
Conc 1	0.44	0.33	26.71	11.00	8.97	8.66
Conc 2	0.54	0.47	27.17	23.94	22.84	15.62
Conc 3	0.60	0.61	27.42	34.54	38.41	20.47
Conc 4	0.63	0.67	27.30	41.19	48.18	23.32
Conc 5	0.65	0.72	27.14	49.98	59.56	26.86
Conc 6	0.66	0.72	27.02	57.71	68.84	30.74
Conc 7	0.65	0.71	26.92	64.58	75.83	34.48
Conc 8	0.65	0.70	26.83	68.36	79.45	36.37
Tails	0.29	0.27	26.64	100.00	100.00	100.00
Feed (*)	---	---	---	----	----	----

(*) Calculated.

(**) Fe in chalcopyrite already deducted.

Table B2 Metallurgical results of flotation test at 50 g/t Na_2SO_3 on the Camchib retreater feed

Conditioning:	Time	5 minutes
	pH	11.2 with lime
	Collector	Aerofloat 208, 100 g/t
	Frother	Dowfroth 250C, 10 g/t
	Gas used	Air, 5 L/min

Product	Time (min)	Mass (%)	Grade Au (oz/st)	Grade Cu (%)	(**) Fe (%)
Conc 1	1.0	10.52	1.08	1.25	24.25
Conc 2	2.0	4.04	1.14	1.25	23.00
Conc 3	4.0	4.49	0.81	0.84	25.86
Conc 4	6.0	2.54	0.79	0.66	25.37
Conc 5	9.0	2.66	0.82	0.58	26.64
Tails		75.75	0.14	0.08	22.83
Feed (*)		100.00	0.34	0.31	23.29

Metallurgical results

Product	Cumulative grade Au (oz/st)	Cumulative grade Cu (%)	Cumulative grade Fe (%)	Cumulative recovery Au (%)	Cumulative recovery Cu (%)	Cumulative recovery Fe (%)
Conc 1	1.08	1.25	24.25	33.03	42.07	10.96
Conc 2	1.10	1.25	23.90	46.42	58.23	14.95
Conc 3	1.03	1.15	24.37	56.99	70.30	19.94
Conc 4	1.00	1.10	24.48	62.83	75.66	22.70
Conc 5	0.98	1.04	24.72	69.17	80.61	25.75
Tails	0.34	0.31	23.29	100.00	100.00	100.00
Feed (*)	---	---	---	----	----	----

(*) Calculated.

(**) Fe in chalcopyrite already deducted.

Table B3 Metallurgical results of flotation test at 100 g/t Na_2SO_3 on the Camchib retreater feed

Conditioning:	Time	5 minutes
	pH	11.2 with lime
	Collector	Aerofloat 208, 100 g/t
	Frother	Dowfroth 250C, 10 g/t
	Gas used	Air, 5 L/min

Product	Time (min)	Mass (%)	Grade Au (oz/st)	Grade Cu (%)	Grade (**) Fe (%)
Conc 1	2.0	8.72	1.56	1.90	23.93
Conc 2	4.0	4.57	0.99	1.00	23.02
Conc 3	6.0	3.17	0.65	0.66	25.17
Conc 4	8.0	3.79	0.45	0.42	25.83
Tails		79.75	0.13	0.17	23.55
Feed (*)		100.00	0.32	0.38	23.70

Metallurgical results

Product	Cumulative grade Au (oz/st)	Cumulative grade Cu (%)	Cumulative grade Fe (%)	Cumulative recovery Au (%)	Cumulative recovery Cu (%)	Cumulative recovery Fe (%)
Conc 1	1.56	1.90	23.93	42.16	43.17	8.81
Conc 2	1.36	1.59	23.62	56.18	55.08	13.25
Conc 3	1.23	1.41	23.92	62.57	60.53	16.62
Conc 4	1.08	1.23	24.27	67.87	64.68	20.75
Tails	0.32	0.38	23.70	100.00	100.00	100.00
Feed (*)	---	---	---	----	----	----

(*) Calculated.

(**) Fe in chalcopyrite already deducted.

APPENDIX C

The effect of Pre-aeration flotation with and without sodium sulfite as a depressant on the Camchib ore using a Leeds Cell

Table C1 Metallurgical results of pre-aeration flotation test without Na_2SO_3 on the Camchib retreater feed

Pre-aeration 3 litres/min air about 30 minutes until > 7.0 ppm O_2 in pulp

Conditioning:	Time	5 minutes
	pH	11.2 with lime
	Collector	Aerofloat 208, 100 g/t
	Frother	Dowfroth 250C, 10 g/t
	Gas used	Air, 5 L/min

Product	Time (min)	Mass (%)	Grade Au (oz/st)	Cu (%)	(**) Fe (%)
Conc 1	1.0	8.29	1.20	1.30	20.21
Conc 2	2.0	2.89	1.59	1.40	21.42
Conc 3	4.0	2.83	1.10	0.95	27.82
Conc 4	7.0	2.28	0.71	0.60	27.92
Conc 5	10.0	2.15	0.40	0.33	27.62
Tails		81.55	0.13	0.09	17.42
Feed (*)		100.00	0.30	0.27	18.52

Metallurgical results

Product	Cumulative grade Au (oz/st)	Cu (%)	Fe (%)	Cumulative recovery Au (%)	Cu (%)	Fe (%)
Conc 1	1.20	1.30	20.21	32.80	40.01	9.05
Conc 2	1.30	1.33	20.52	47.97	55.05	12.40
Conc 3	1.26	1.25	21.99	58.23	65.03	16.65
Conc 4	1.18	1.16	22.83	63.58	70.12	20.09
Conc 5	1.09	1.06	23.38	66.39	72.76	23.30
Tails	0.30	0.27	18.52	100.00	100.00	100.00
Feed (*)	---	---	---	----	----	----

(*) Calculated.

(**) Fe in chalcopyrite already deducted.

Table C2 Metallurgical results of pre-aeration flotation test with 50 g/t Na_2SO_3 on the Camchib retreater feed

Pre-aeration 3 litres/min air about 30 minutes until > 7.0 ppm O_2 in pulp

Conditioning:	Time	5 minutes
	pH	11.2 with lime
	Collector	Aerofloat 208, 100 g/t
	Frother	Dowfroth 250C, 10 g/t
	Gas used	Air, 5 L/min

Product	Time (min)	Mass (%)	Grade Au (oz/st)	Cu (%)	(**) Fe (%)
Conc 1	1.0	5.87	2.14	2.14	18.32
Conc 2	2.0	3.27	1.34	1.10	27.18
Conc 3	4.0	3.51	0.90	0.65	27.73
Conc 4	7.0	3.10	0.60	0.44	28.01
Conc 5	10.0	1.88	0.67	0.28	28.21
Tails		82.38	0.10	0.07	18.54
Feed (*)		100.00	0.31	0.26	19.61

Metallurgical results

Product	Cumulative grade Au (oz/st)	Cu (%)	Fe (%)	Cumulative recovery Au (%)	Cu (%)	Fe (%)
Conc 1	2.14	2.14	18.32	39.94	48.93	5.48
Conc 2	1.85	1.77	21.49	53.85	62.92	10.01
Conc 3	1.59	1.46	23.22	63.90	71.81	14.97
Conc 4	1.39	1.20	24.16	69.81	77.13	19.40
Conc 5	1.32	1.15	24.60	73.81	79.14	22.10
Tails	0.31	0.26	19.61	100.00	100.00	100.00
Feed (*)	---	---	---	----	----	----

(*) Calculated.

(**) Fe in chalcopyrite already deducted.

Table C3 Metallurgical results of no aeration flotation test with 50 g/t Na_2SO_3 on the Camchib retreater feed

Conditioning:		
	Time	5 minutes
	pH	11.2 with lime
	Collector	Aerofloat 208, 100 g/t
	Frother	Dowfroth 250C, 10 g/t
	Gas used	Air, 5 L/min

Product	Time (min)	Mass (%)	Grade Au (oz/st)	Grade Cu (%)	Grade (**) Fe (%)
Conc 1	1.0	7.85	0.84	1.10	26.93
Conc 2	2.0	3.67	1.10	1.35	18.91
Conc 3	4.0	3.74	0.96	0.90	19.21
Conc 4	7.0	2.96	0.72	0.44	18.66
Conc 5	10.0	1.51	0.76	0.34	18.76
Tails		80.27	0.09	0.07	17.33
Feed (*)		100.00	0.24	0.24	18.28

Metallurgical results

Product	Cumulative grade Au (oz/st)	Cumulative grade Cu (%)	Cumulative grade Fe (%)	Cumulative recovery Au (%)	Cumulative recovery Cu (%)	Cumulative recovery Fe (%)
Conc 1	0.84	1.10	26.93	27.00	35.42	11.57
Conc 2	0.92	1.18	24.38	43.63	55.74	15.37
Conc 3	0.93	1.11	23.11	58.40	69.53	19.29
Conc 4	0.90	0.89	22.39	67.19	74.88	22.32
Conc 5	0.88	0.95	22.11	71.90	76.95	23.87
Tails	0.24	0.24	18.28	100.00	100.00	100.00
Feed (*)	---	---	---	----	----	----

(*) Calculated.

(**) Fe in chalcopyrite already deducted.

APPENDIX D

The effect of sulfur dioxide (SO_2) as a pyrite depressant with and without aeration on the Camchib ore using a Leeds cell

Table D1 Metallurgical results of flotation test with 3 g/kg of saturated SO_2 and aeration (3 ml/min) on the Camchib retreater feed

Conditioning:		
	Time	5 minutes
	pH	11.0 with lime
	Collector	Aerofloat 208, 100 g/t
	Frother	Dowfroth 250C, 10 g/t
	Gas used	Air, 5 L/min

Product	Time (min)	Mass (%)	Grade Au (oz/st)	Cu (%)	(**) Fe (%)
Conc 1	2.0	13.63	0.75	1.02	18.02
Conc 2	4.0	8.32	0.51	0.79	18.13
Conc 3	6.0	4.05	0.40	0.40	19.02
Conc 4	8.0	3.40	0.33	0.26	18.87
Tails		70.60	0.09	0.07	14.89
Feed (*)		100.00	0.24	0.28	15.90

Metallurgical results

Product	Cumulative grade			Cumulative recovery		
	Au (oz/st)	Cu (%)	Fe (%)	Au (%)	Cu (%)	Fe (%)
Conc 1	0.75	1.02	18.02	43.39	49.79	15.45
Conc 2	0.66	0.93	18.06	61.40	73.33	24.94
Conc 3	0.62	0.85	18.24	68.27	79.13	29.83
Conc 4	0.59	0.78	18.31	73.04	82.30	33.87
Tails	0.24	0.28	15.90	100.00	100.00	100.00
Feed (*)	---	---	---	----	----	----

(*) Calculated.

(**) Fe in chalcopyrite already deducted.

Table D2 Metallurgical results of flotation test with 3 g/kg of saturated SO_2 and no aeration on the Camchib retreater feed

Conditioning:	Time	5 minutes
	pH	11.0 with lime
	Collector	Aerofloat 208, 100 g/t
	Frother	Dowfroth 250C, 10 g/t
	Gas used	Air, 5 L/min

Product	Time (min)	Mass (%)	Grade Au (oz/st)	Grade Cu (%)	(**) Fe (%)
Conc 1	0.5	3.54	0.67	0.37	15.83
Conc 2	1.5	7.23	1.08	0.55	17.42
Conc 3	4.5	6.19	0.92	0.95	17.62
Conc 4	9.5	7.53	0.88	0.95	19.22
Tails		75.51	0.13	0.12	15.80
Feed (*)		100.00	0.32	0.27	16.29

Metallurgical results

Product	Cumulative grade Au (oz/st)	Cumulative grade Cu (%)	Cumulative grade Fe (%)	Cumulative recovery Au (%)	Cumulative recovery Cu (%)	Cumulative recovery Fe (%)
Conc 1	0.67	0.37	15.83	7.34	4.78	3.44
Conc 2	0.95	0.49	16.90	31.53	19.52	11.17
Conc 3	0.94	0.66	17.16	49.17	41.30	17.86
Conc 4	0.92	0.75	17.79	69.59	67.82	26.75
Tails	0.32	0.27	16.29	100.00	100.00	100.00
Feed (*)	---	---	---	----	----	----

(*) Calculated.

(**) Fe in chalcopyrite already deducted.

APPENDIX E

Gas and Wash Water Flowmeters Calibration

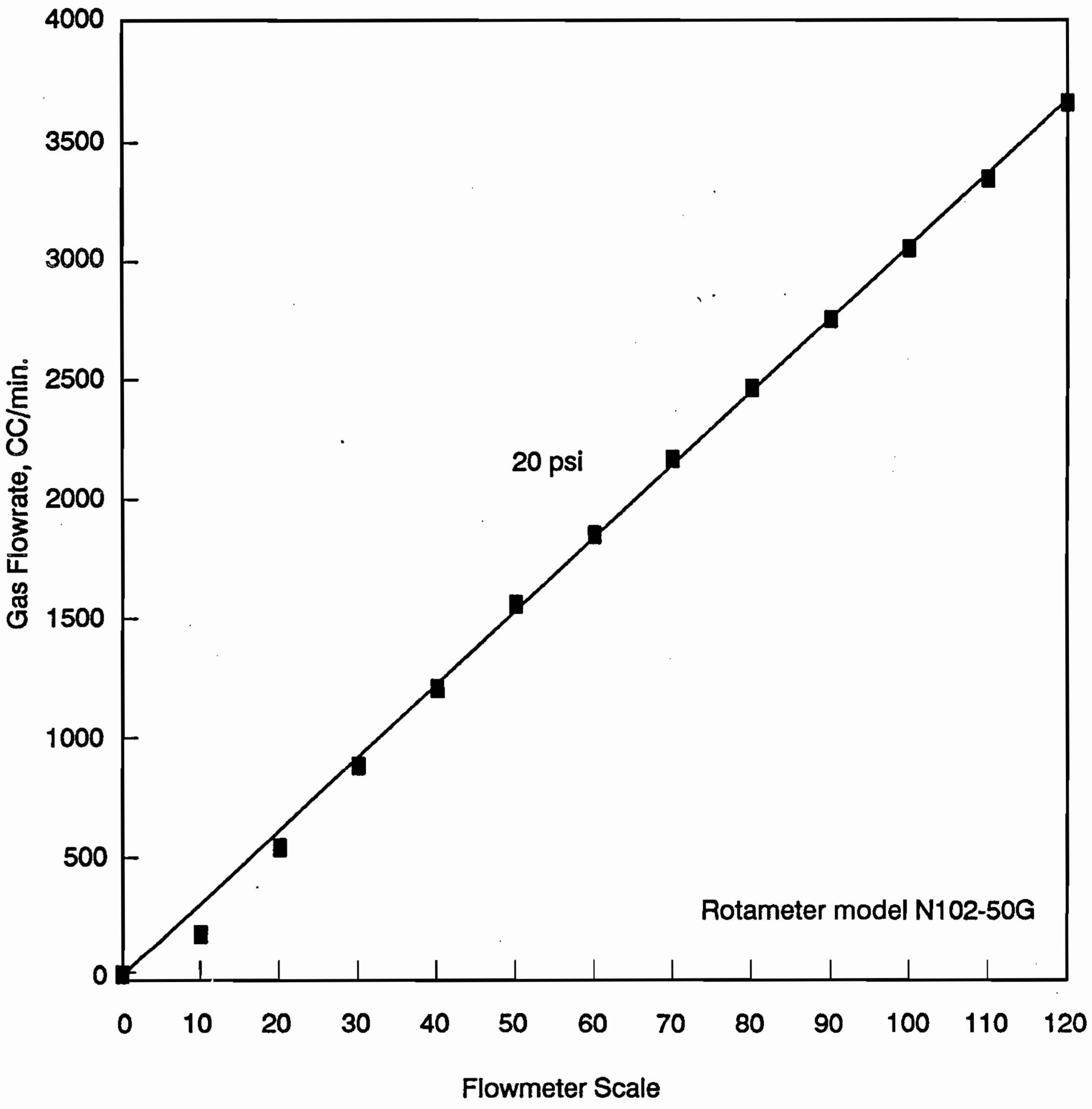


Figure E1. Gas Flowmeter Calibration.
Calibrated with Wet Test Meter

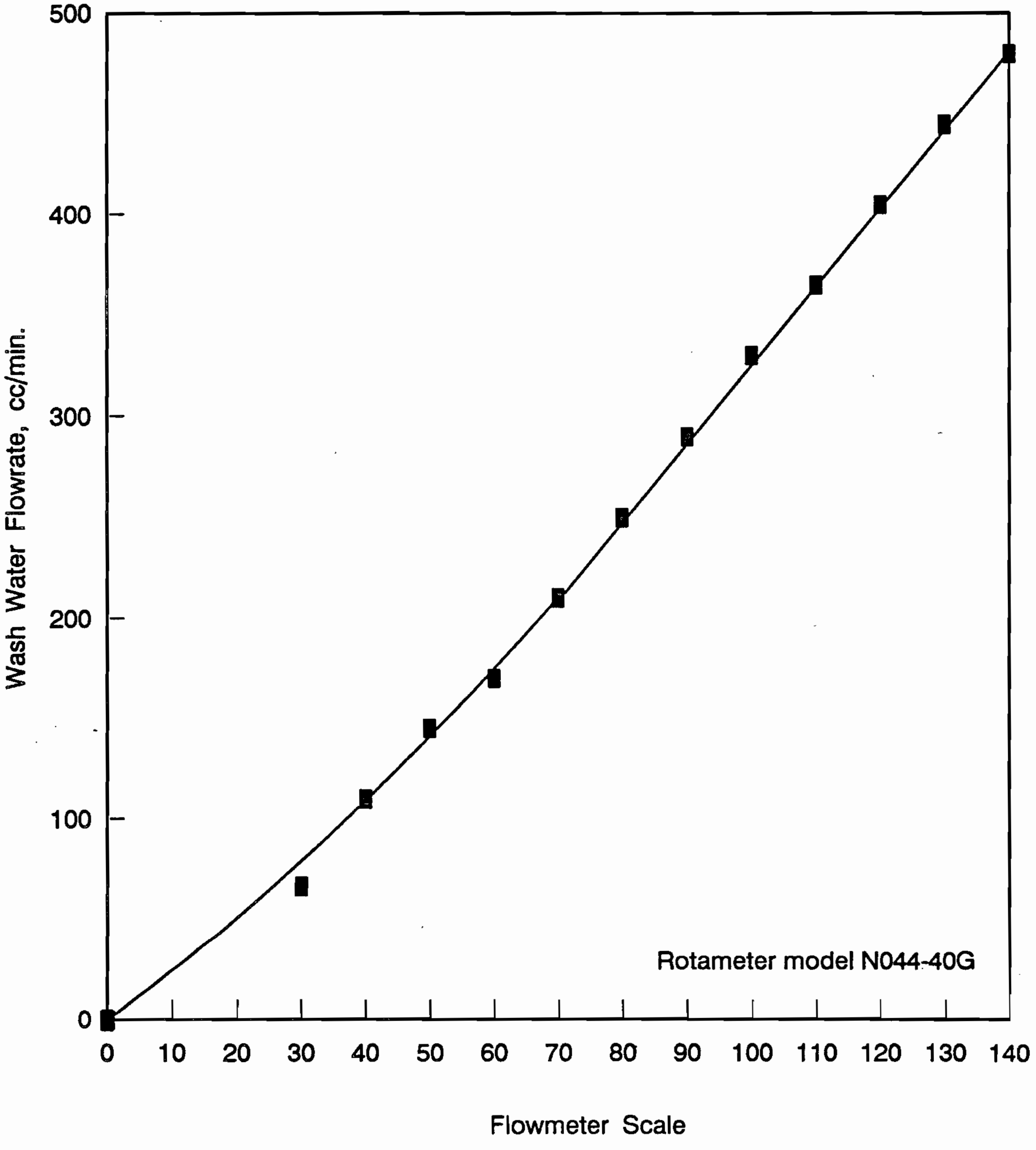


Figure E2. Wash Water Flowmeter Calibration.

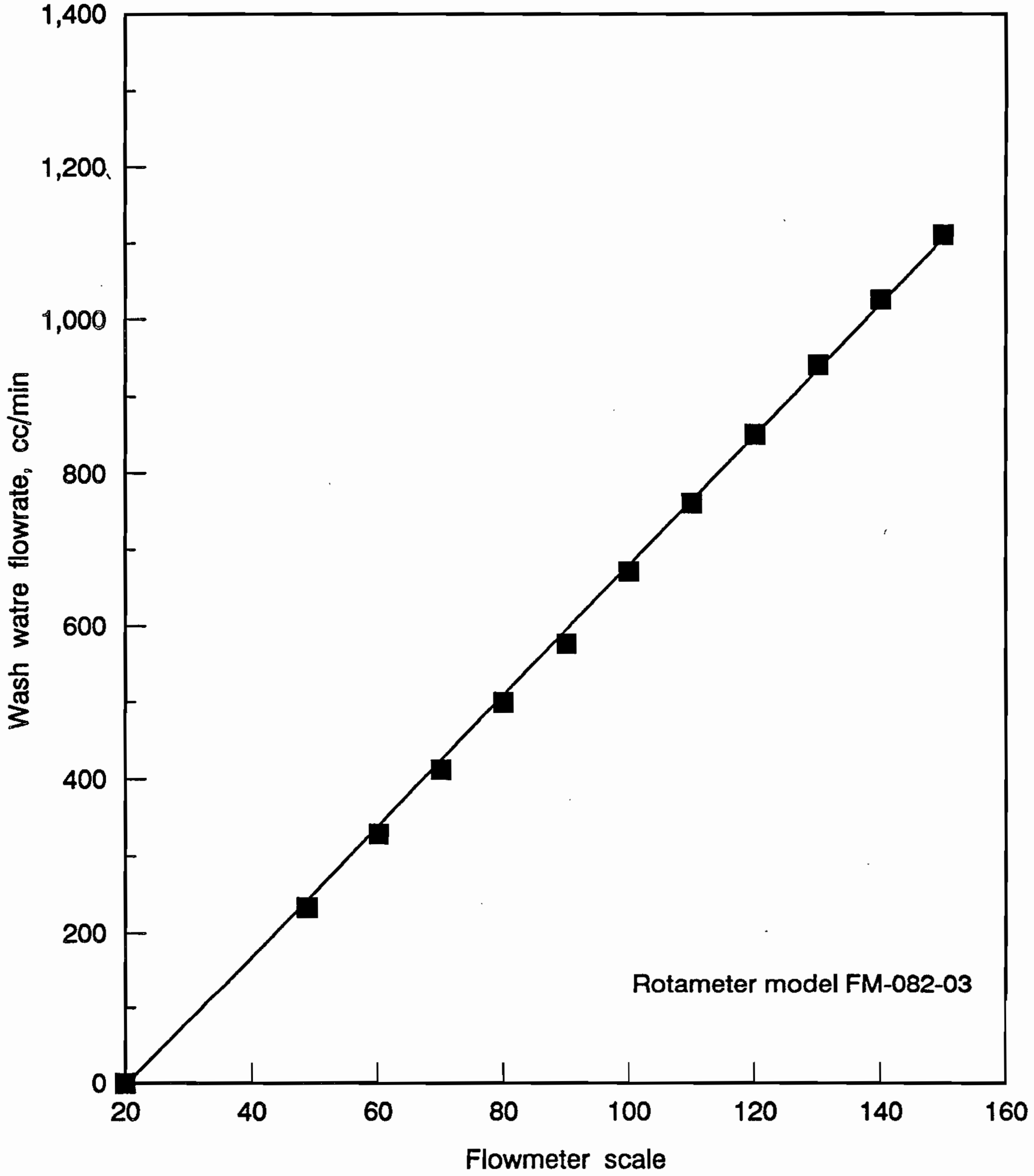


Figure E3. Wash Water Flowmeter Calibration.

APPENDIX F

Adjustment of Assays Using the Noranda NORBAL2 Software

Adjustment of Assays using the Noranda NORBAL2 Software

NORBAL2 is a least squares regression program for mineral processing circuit data. When there are sufficient data to overdefine the mass balance in a circuit, NORBAL2 uses material balance constraints to detect and rectify erroneous data. The differences between the adjusted values and the raw data are such that the weighted sum of squared differences is minimal (Details are in Spring, 1986). The basis of adjustment is the law of mass conservation.

Constraint: Mass in = Mass out
for copper and gold.

In this campaign: 5 streams were collected with 2 nodes, where nodes are defined as junctions between two or more streams. The retreater feed was arbitrarily assigned a flow rate value of 1. Thus in the adjustment program, the other streams were allowed to vary to minimized the sum of the square. The diagram is shown in Figure F1.

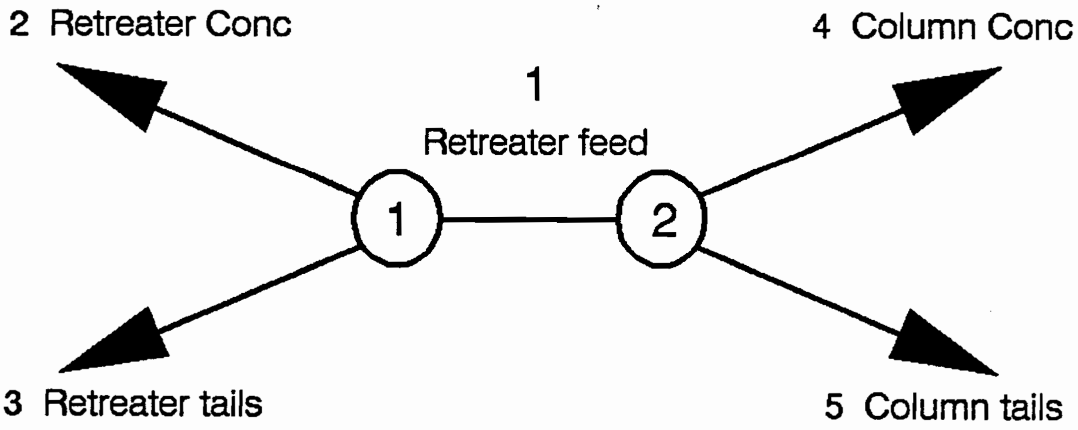


Figure F1 The mass conservation of retreater feed.

```
                    Test 1, Column vs. Retreater     (June 11, 1990)
                    ----------------------------

Residual sum of squares: 2.395431                    Final Results

                  | Absolute Solids |        Pulp Mass Flowrate         |
   Stream         |    Flowrate     |  Meas  |  Calc  |  S.D.  | Adjust |
==========================================================================
 1  Retr. feed    |     101.75      |        | 101.7  |        |        |
 2  Retr. conc    |      25.85      |  25.0  |  25.9  |   7.0  |   0.9  |
 3  Retr. tail    |      75.90      |        |  75.9  |        |        |
 4  Col. conc     |      10.85      |   9.0  |  10.9  |   5.0  |   1.9  |
 5  Col. tail     |      90.90      |  91.0  |  90.9  |  10.0  |  -0.1  |

                  | Relative Solids |
   Stream         |    Flowrate     |
=====================================
 1  Retr. feed    |     100.00      |
 2  Retr. conc    |      25.41      |
 3  Retr. tail    |      74.59      |
 4  Col. conc     |      10.66      |
 5  Col. tail     |      89.34      |


                          Assay Data
                          ----------

Copper (%) |   Meas.   |   Calc.   | Std. Dev. |  Adjust.  | % Rec |
======================================================================
Retr. feed |   0.280   |   0.285   |   0.030   |   0.005   |  100  |
Retr. conc |   0.950   |   0.967   |   0.030   |   0.017   |   86  |
Retr. tail |   0.051   |   0.052   |   0.005   |   0.001   |   14  |
Col. conc  |   1.840   |   1.809   |   0.060   |  -0.031   |   68  |
Col. tail  |   0.110   |   0.103   |   0.010   |  -0.007   |   32  |

Gold (o/t) |   Meas.   |   Calc.   | Std. Dev. |  Adjust.  | % Rec |
======================================================================
Retr. feed |   0.420   |   0.429   |   0.030   |   0.009   |  100  |
Retr. conc |   1.440   |   1.420   |   0.040   |  -0.020   |   84  |
Retr. tail |   0.095   |   0.091   |   0.010   |  -0.004   |   16  |
Col. conc  |   2.517   |   2.543   |   0.080   |   0.026   |   63  |
Col. tail  |   0.163   |   0.177   |   0.020   |   0.014   |   37  |
```

<u>Conditions:</u>

gas rate 0.8 cm/s
wash water rate 0.35 cm/s
feed rate 0.5 cm/s
pH 11.0
Froth depth 100 cm (set point: 47.5)

Test 2, Column vs. Retreater (June 11, 1990)

Residual sum of squares: 2.147463

Final Results

Stream	Absolute Solids Flowrate	Pulp Mass Flowrate Meas	Calc	S.D.	Adjust
1 Retr. feed	99.87		99.9		
2 Retr. conc	17.99	16.8	18.0	7.0	1.2
3 Retr. tail	81.88		81.9		
4 Col. conc	19.65	20.7	19.6	5.0	-1.1
5 Col. tail	80.22	79.3	80.2	10.0	0.9

Stream	Relative Solids Flowrate
1 Retr. feed	100.00
2 Retr. conc	18.01
3 Retr. tail	81.99
4 Col. conc	19.67
5 Col. tail	80.33

Assay Data

Copper (%)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.290	0.310	0.030	0.020	100
Retr. conc	1.550	1.570	0.050	0.020	91
Retr. tail	0.033	0.033	0.003	0.000	9
Col. conc	1.160	1.148	0.030	-0.012	73
Col. tail	0.110	0.105	0.010	-0.005	27

Gold (o/t)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.420	0.436	0.040	0.016	100
Retr. conc	1.905	1.876	0.060	-0.029	77
Retr. tail	0.125	0.120	0.012	-0.005	23
Col. conc	1.500	1.517	0.050	0.017	68
Col. tail	0.160	0.171	0.020	0.011	32

Conditions:

gas rate 0.8 cm/s
wash water rate 0.35 cm/s
feed rate 0.3 cm/s
pH 11.0
Froth depth 25 cm (set point: 43.5)

Test 3, Column vs. Retreater (June 11, 1990)

Residual sum of squares: 1.218584 Final Results

	Stream	Absolute Solids Flowrate	Pulp Mass Flowrate			
			Meas	Calc	S.D.	Adjust
1	Retr. feed	101.78		101.8		
2	Retr. conc	21.22	20.8	21.2	7.0	0.4
3	Retr. tail	80.56		80.6		
4	Col. conc	19.22	16.3	19.2	5.0	2.9
5	Col. tail	82.56	83.7	82.6	10.0	-1.1

	Stream	Relative Solids Flowrate
1	Retr. feed	100.00
2	Retr. conc	20.85
3	Retr. tail	79.15
4	Col. conc	18.88
5	Col. tail	81.12

Assay Data

Copper (%)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.300	0.318	0.030	0.018	100
Retr. conc	1.300	1.298	0.040	-0.002	85
Retr. tail	0.060	0.060	0.006	-0.000	15
Col. conc	1.220	1.216	0.040	-0.004	72
Col. tail	0.110	0.109	0.010	-0.001	28

Gold (o/t)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.480	0.456	0.048	-0.024	100
Retr. conc	1.705	1.698	0.050	-0.007	78
Retr. tail	0.130	0.128	0.013	-0.002	22
Col. conc	1.640	1.651	0.050	0.011	68
Col. tail	0.170	0.178	0.020	0.008	32

Conditions:
gas rate 0.8 cm/s
wash water rate 0.35 cm/s
feed rate 0.5 cm/s
pH 11.0
Froth depth 60 cm (set point: 46.0)

Test 4, Column vs. Retreater (June 12, 1990)

Residual sum of squares: 6.817735 Final Results

Stream	Absolute Solids Flowrate	Pulp Mass Flowrate			
		Meas	Calc	S.D.	Adjust
1 Retr. feed	91.67		91.7		
2 Retr. conc	14.12	14.5	14.1	7.0	-0.4
3 Retr. tail	77.55		77.6		
4 Col. conc	9.90	22.0	9.9	5.0	-12.1*
5 Col. tail	81.77	78.0	81.8	10.0	3.8

Stream	Relative Solids Flowrate
1 Retr. feed	100.00
2 Retr. conc	15.40
3 Retr. tail	84.60
4 Col. conc	10.80
5 Col. tail	89.20

Assay Data

Copper (%)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.280	0.300	0.030	0.020	100
Retr. conc	1.750	1.757	0.050	0.007	90
Retr. tail	0.035	0.035	0.004	0.000	10
Col. conc	1.750	1.739	0.050	-0.011	63
Col. tail	0.130	0.126	0.010	-0.004	37

Gold (o/t)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.400	0.409	0.040	0.009	100
Retr. conc	2.045	2.036	0.060	-0.009	77
Retr. tail	0.115	0.113	0.012	-0.002	23
Col. conc	2.325	2.330	0.070	0.005	62
Col. tail	0.173	0.177	0.020	0.004	38

Conditions:
gas rate 0.8 cm/s
wash water rate 0.35 cm/s
feed rate 0.5 cm/s
pH 11.0
Froth depth 25 cm (set point: 43.7)

Test 5, Column vs. Retreater (June 18, 1990)

Residual sum of squares: 6.526973

Final Results

	Stream	Absolute Solids Flowrate	Pulp Mass Flowrate			
			Meas	Calc	S.D.	Adjust
1	Retr. feed	94.73		94.7		
2	Retr. conc	15.20	18.3	15.2	7.0	-3.1
3	Retr. tail	79.53		79.5		
4	Col. conc	31.79	26.7	31.8	5.0	5.1*
5	Col. tail	62.94	73.3	62.9	10.0	-10.4*

	Stream	Relative Solids Flowrate
1	Retr. feed	100.00
2	Retr. conc	16.04
3	Retr. tail	83.96
4	Col. conc	33.56
5	Col. tail	66.44

Assay Data

Copper (%)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.680	0.618	0.070	-0.062	100
Retr. conc	3.650	3.568	0.110	-0.082	93
Retr. tail	0.055	0.054	0.006	-0.001	7
Col. conc	1.510	1.556	0.050	0.046	85
Col. tail	0.140	0.144	0.010	0.004	15

Gold (o/t)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.635	0.571	0.064	-0.064*	100
Retr. conc	2.510	2.561	0.080	0.051	72
Retr. tail	0.190	0.190	0.002	0.000	28
Col. conc	1.420	1.402	0.040	-0.018	82
Col. tail	0.160	0.151	0.020	-0.009	18

<u>Conditions:</u>

gas rate 1.3 cm/s
wash water rate 0.35 cm/s
feed rate 0.6 cm/s
pH 11.0
Froth depth 100 cm (set point: 51.0)

Test 6, Column vs. Retreater (June 18, 1990)

Residual sum of squares: 2.407615 Final Results

	Stream	Absolute Solids Flowrate	Pulp Mass Flowrate Meas	Calc	S.D.	Adjust
1	Retr. feed	100.79		100.8		
2	Retr. conc	15.16	14.8	15.2	7.0	0.4
3	Retr. tail	85.63		85.6		
4	Col. conc	19.90	19.9	19.9	5.0	0.0
5	Col. tail	80.89	80.1	80.9	10.0	0.8

	Stream	Relative Solids Flowrate
1	Retr. feed	100.00
2	Retr. conc	15.04
3	Retr. tail	84.96
4	Col. conc	19.75
5	Col. tail	80.25

Assay Data

Copper (%)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.380	0.388	0.040	0.008	100
Retr. conc	2.250	2.296	0.070	0.046	89
Retr. tail	0.049	0.050	0.005	0.001	11
Col. conc	1.480	1.459	0.040	-0.021	74
Col. tail	0.130	0.125	0.010	-0.005	26

Gold (o/t)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.490	0.489	0.049	-0.001	100
Retr. conc	2.435	2.402	0.070	-0.033	74
Retr. tail	0.160	0.150	0.016	-0.010	26
Col. conc	1.738	1.760	0.050	0.022	71
Col. tail	0.162	0.176	0.020	0.014	29

<u>Conditions:</u>

gas rate 1.0 cm/s
wash water rate 0.35 cm/s
feed rate 0.5 cm/s
pH 11.0
Froth depth 60 cm (set point: 48.0)

Test 7, Column vs. Retreater (June 18, 1990)

Residual sum of squares: 3.175698 Final Results

Stream	Absolute Solids Flowrate	Pulp Mass Flowrate			
		Meas	Calc	S.D.	Adjust
1 Retr. feed	98.68		98.7		
2 Retr. conc	16.18	16.6	16.2	7.0	-0.4
3 Retr. tail	82.50		82.5		
4 Col. conc	18.02	15.5	18.0	5.0	2.5
5 Col. tail	80.66	84.5	80.7	10.0	-3.8

Stream	Relative Solids Flowrate
1 Retr. feed	100.00
2 Retr. conc	16.40
3 Retr. tail	83.60
4 Col. conc	18.27
5 Col. tail	81.73

Assay Data

Copper (%)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.360	0.382	0.040	0.022	100
Retr. conc	2.050	2.089	0.060	0.039	90
Retr. tail	0.046	0.047	0.005	0.001	10
Col. conc	1.620	1.584	0.050	-0.036	76
Col. tail	0.120	0.114	0.010	-0.006	24

Gold (o/t)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.500	0.470	0.050	-0.030	100
Retr. conc	2.145	2.130	0.060	-0.015	74
Retr. tail	0.150	0.144	0.016	-0.006	26
Col. conc	1.830	1.847	0.050	0.017	72
Col. tail	0.150	0.162	0.020	0.012	28

Conditions:

gas rate 0.8 cm/s
wash water rate 0.35 cm/s
feed rate 0.5 cm/s
pH 11.0
Froth depth 100 cm (set point: 50.5)

Test 8, Column vs. Retreater (June 19, 1990)

Residual sum of squares: .6505708

Final Results

Stream	Absolute Solids Flowrate	Pulp Mass Flowrate Meas	Calc	S.D.	Adjust
1 Retr. feed	102.31		102.3		
2 Retr. conc	12.13	11.9	12.1	7.0	0.2
3 Retr. tail	90.17		90.2		
4 Col. conc	17.28	17.3	17.3	5.0	-0.0
5 Col. tail	85.03	82.7	85.0	10.0	2.3

Stream	Relative Solids Flowrate
1 Retr. feed	100.00
2 Retr. conc	11.86
3 Retr. tail	88.14
4 Col. conc	16.89
5 Col. tail	83.11

Assay Data

Copper (%)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.320	0.326	0.030	0.006	100
Retr. conc	2.400	2.395	0.070	-0.005	87
Retr. tail	0.048	0.048	0.005	-0.000	13
Col. conc	1.290	1.291	0.040	0.001	67
Col. tail	0.130	0.130	0.010	0.000	33

Gold (o/t)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.390	0.387	0.039	-0.003	100
Retr. conc	2.320	2.340	0.070	0.020	72
Retr. tail	0.120	0.124	0.012	0.004	28
Col. conc	1.408	1.399	0.040	-0.009	61
Col. tail	0.192	0.181	0.020	-0.011	39

Conditions:

gas rate 0.8 cm/s
wash water rate 0.35 cm/s
feed rate 0.3 cm/s
pH 11.0
Froth depth 100 cm (set point: 50.4)

Test 9, Column vs. Retreater (June 19, 1990)

Residual sum of squares: .911097 Final Results

Stream	Absolute Solids Flowrate	Pulp Mass Flowrate			
		Meas	Calc	S.D.	Adjust
1 Retr. feed	100.00		100.0		
2 Retr. conc	11.20	11.2	11.2	7.0	0.0
3 Retr. tail	88.80		88.8		
4 Col. conc	19.60	19.6	19.6	5.0	0.0
5 Col. tail	80.40	80.4	80.4	10.0	0.0

Stream	Relative Solids Flowrate
1 Retr. feed	100.00
2 Retr. conc	11.20
3 Retr. tail	88.80
4 Col. conc	19.60
5 Col. tail	80.40

Assay Data

Copper (%)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.450	0.451	0.050	0.001	100
Retr. conc	3.500	3.527	0.110	0.027	88
Retr. tail	0.062	0.063	0.006	0.001	12
Col. conc	1.680	1.670	0.050	-0.010	73
Col. tail	0.160	0.153	0.020	-0.007	27

Gold (o/t)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.500	0.494	0.050	-0.006	100
Retr. conc	3.230	3.196	0.100	-0.034	72
Retr. tail	0.160	0.153	0.016	-0.007	28
Col. conc	1.690	1.706	0.050	0.016	68
Col. tail	0.188	0.198	0.020	0.010	32

Conditions:
gas rate 1.0 cm/s
wash water rate 0.35 cm/s
feed rate 0.5 cm/s
pH 11.0
Froth depth 60 cm (set point: 49.0)

Test 10, Column vs. Retreater (June 19, 1990)

Residual sum of squares: 13.9027

Final Results

	Stream	Absolute Solids Flowrate	Pulp Mass Flowrate			
			Meas	Calc	S.D.	Adjust
1	Retr. feed	106.32		106.3		
2	Retr. conc	10.62	12.3	10.6	7.0	-1.7
3	Retr. tail	95.69		95.7		
4	Col. conc	28.26	16.3	28.3	5.0	12.0*
5	Col. tail	78.05	83.7	78.1	10.0	-5.6

	Stream	Relative Solids Flowrate
1	Retr. feed	100.00
2	Retr. conc	9.99
3	Retr. tail	90.01
4	Col. conc	26.59
5	Col. tail	73.41

Assay Data

Copper (%)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.470	0.377	0.050	-0.093*	100
Retr. conc	3.350	3.281	0.100	-0.069	87
Retr. tail	0.057	0.055	0.006	-0.002	13
Col. conc	1.040	1.065	0.030	0.025	75
Col. tail	0.120	0.128	0.010	0.008	25

Gold (o/t)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.505	0.462	0.050	-0.043	100
Retr. conc	3.010	3.059	0.090	0.049	66
Retr. tail	0.160	0.174	0.016	0.014	34
Col. conc	1.350	1.332	0.040	-0.018	77
Col. tail	0.160	0.147	0.020	-0.013	23

Conditions:

gas rate 1.3 cm/s
wash water rate 0.35 cm/s
feed rate 0.3 cm/s
pH 11.0
Froth depth 25 cm (set point: 47.0)

Test 11, Column vs. Retreater (June 19, 1990)

Residual sum of squares: .2793634

Final Results

Stream	Absolute Solids Flowrate	Pulp Mass Flowrate			
		Meas	Calc	S.D.	Adjust
1 Retr. feed	101.27		101.3		
2 Retr. conc	11.58	11.6	11.6	7.0	0.0
3 Retr. tail	89.69		89.7		
4 Col. conc	20.67	19.6	20.7	5.0	1.0
5 Col. tail	80.60	80.4	80.6	10.0	0.2

Stream	Relative Solids Flowrate
1 Retr. feed	100.00
2 Retr. conc	11.43
3 Retr. tail	88.57
4 Col. conc	20.41
5 Col. tail	79.59

Assay Data

Copper (%)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.580	0.566	0.060	-0.014	100
Retr. conc	4.450	4.433	0.130	-0.017	89
Retr. tail	0.068	0.067	0.010	-0.001	11
Col. conc	2.010	2.019	0.060	0.009	73
Col. tail	0.190	0.194	0.020	0.004	27

Gold (o/t)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.590	0.584	0.060	-0.006	100
Retr. conc	3.680	3.667	0.110	-0.013	72
Retr. tail	0.190	0.187	0.020	-0.003	28
Col. conc	2.042	2.051	0.062	0.009	72
Col. tail	0.205	0.209	0.020	0.004	28

Conditions:

gas rate 0.8 cm/s
wash water rate 0.35 cm/s
feed rate 0.5 cm/s
pH 11.0
Froth depth 60 cm (set point: 49.3)

Test 12, Column vs. Retreater (June 19, 1990)

Residual sum of squares: .7972715

Final Results

Stream	Absolute Solids Flowrate	Pulp Mass Flowrate			
		Meas	Calc	S.D.	Adjust
1 Retr. feed	102.69		102.7		
2 Retr. conc	13.59	13.1	13.6	7.0	0.5
3 Retr. tail	89.11		89.1		
4 Col. conc	9.93	7.5	9.9	5.0	2.4
5 Col. tail	92.76	92.2	92.8	10.0	0.6

Stream	Relative Solids Flowrate
1 Retr. feed	100.00
2 Retr. conc	13.23
3 Retr. tail	86.77
4 Col. conc	9.67
5 Col. tail	90.33

Assay Data

Copper (%)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.440	0.432	0.040	-0.008	100
Retr. conc	3.000	2.985	0.090	-0.015	91
Retr. tail	0.043	0.043	0.004	-0.000	9
Col. conc	3.080	3.095	0.090	0.015	69
Col. tail	0.140	0.147	0.020	0.007	31

Gold (o/t)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.445	0.459	0.040	0.014	100
Retr. conc	2.700	2.710	0.080	0.010	78
Retr. tail	0.115	0.116	0.010	0.001	22
Col. conc	3.250	3.230	0.100	-0.020	68
Col. tail	0.170	0.163	0.020	-0.007	32

Conditions:

gas rate 0.8 cm/s
wash water rate 0.35 cm/s
feed rate 0.5 cm/s
pH 11.0
Froth depth 60 cm (set point: 48.0)
* sodium sulfite 50 g/t

Test 13, Column vs. Retreater (June 19, 1990)

Residual sum of squares: 1.788938 Final Results

	Stream	Absolute Solids Flowrate	Pulp Mass Flowrate			Adjust
			Meas	Calc	S.D.	
1	Retr. feed	107.88		107.9		
2	Retr. conc	12.66	11.3	12.7	7.0	1.4
3	Retr. tail	95.21		95.2		
4	Col. conc	7.33	8.2	7.3	5.0	-0.9
5	Col. tail	100.55	91.8	100.5	10.0	8.8

	Stream	Relative Solids Flowrate
1	Retr. feed	100.00
2	Retr. conc	11.74
3	Retr. tail	88.26
4	Col. conc	6.79
5	Col. tail	93.21

Assay Data

Copper (%)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.270	0.264	0.030	-0.006	100
Retr. conc	2.000	2.012	0.060	0.012	90
Retr. tail	0.031	0.031	0.003	0.000	10
Col. conc	2.490	2.483	0.070	-0.007	64
Col. tail	0.110	0.102	0.020	-0.008	36

Gold (o/t)	Meas.	Calc.	Std. Dev.	Adjust.	% Rec
Retr. feed	0.310	0.334	0.030	0.024	100
Retr. conc	2.070	2.063	0.060	-0.007	73
Retr. tail	0.105	0.104	0.010	-0.001	27
Col. conc	2.975	2.970	0.089	-0.005	60
Col. tail	0.145	0.141	0.020	-0.004	40

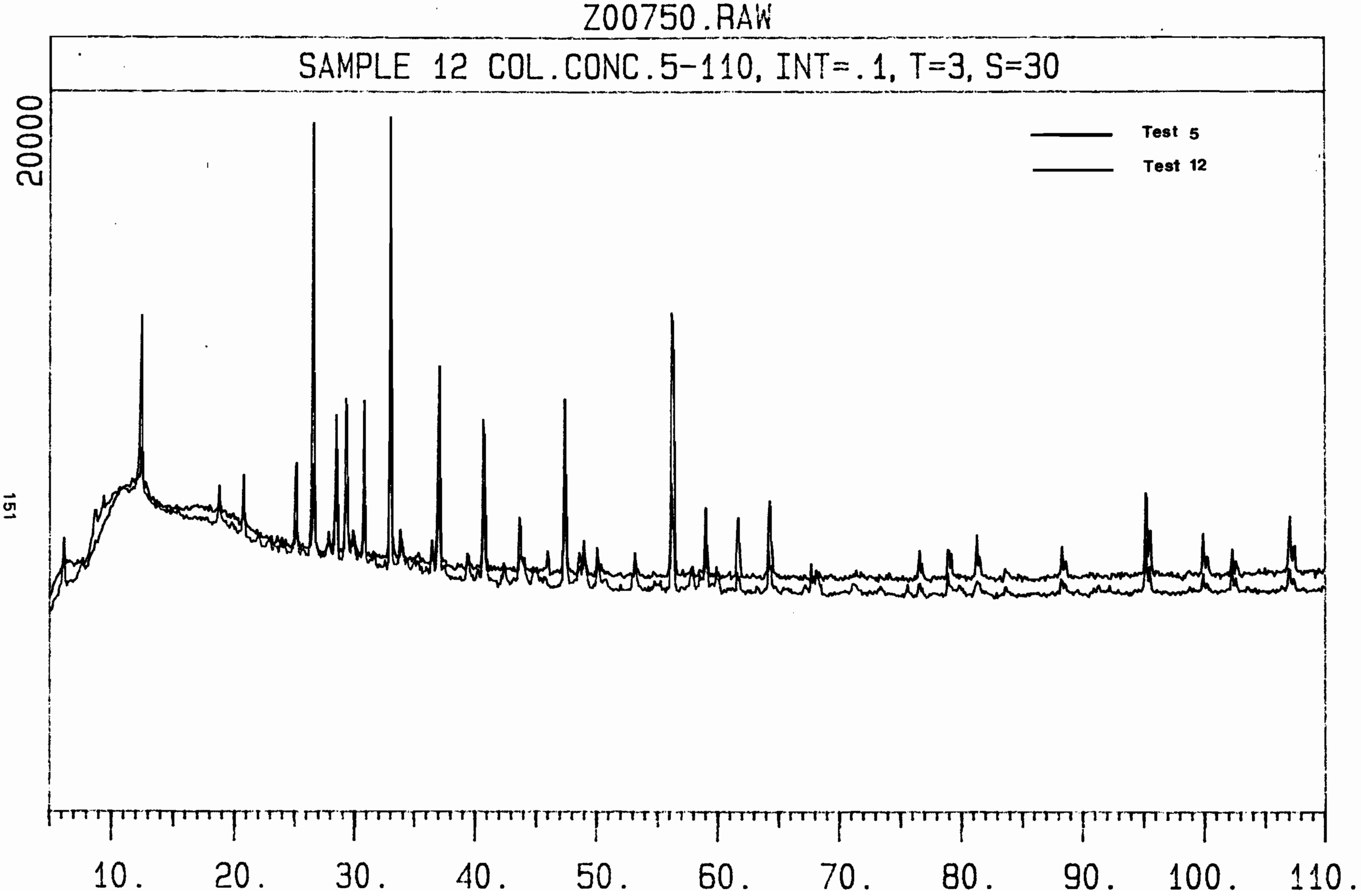
Conditions:

gas rate 0.8 cm/s
wash water rate 0.35 cm/s
feed rate 0.3 cm/s
pH 11.0
Froth depth 60 cm (set point: 48.0)
* sodium sulfite 100 g/t

APPENDIX G

Comparing the X-Ray Diffraction Spectra of Column Concentrate (Tests 5 and 12) to those of Specific Minerals

- Chalcopyrite
- Pyrite
- Pyrrhotite
- Quartz
- Chlorite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Test 5
Test 12
10.
20.
30.
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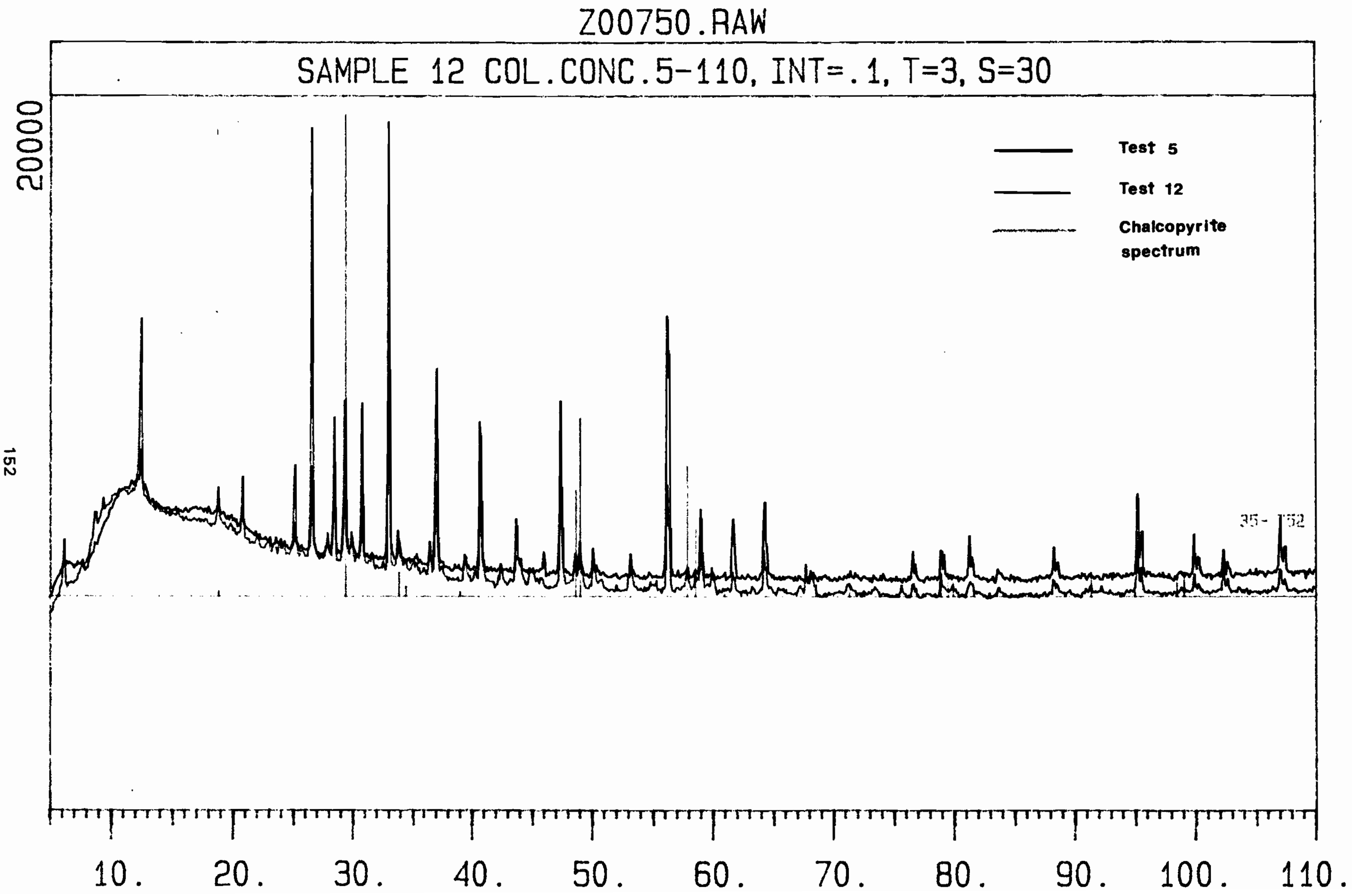
70.
80.
90.
100.
110.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Test 5
Test 12
Chalcopyrite spectrum
35-752
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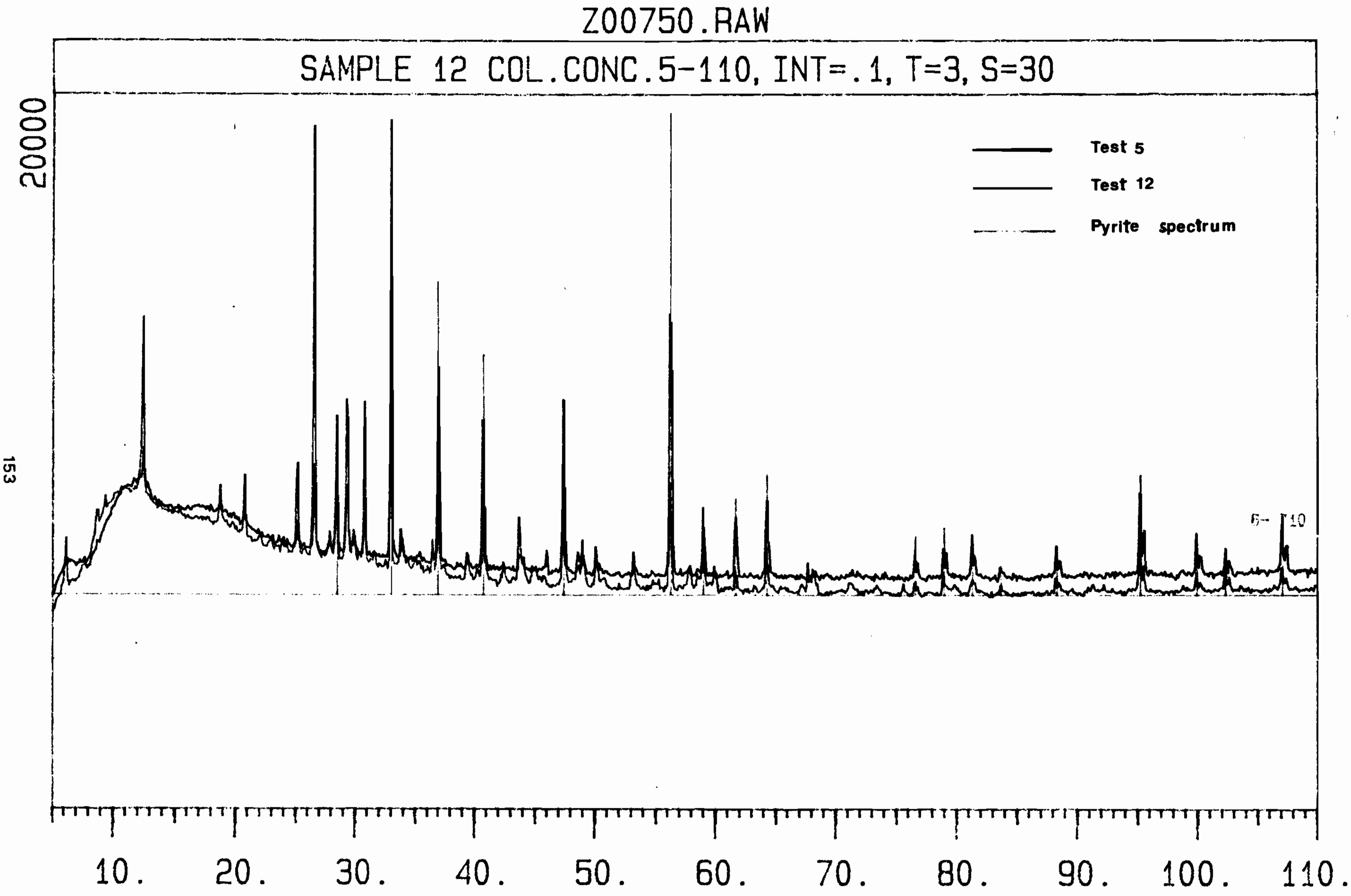
20.
30.
40.
50.
60.
70.
80.
90.
100.
110.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Test 5
Test 12
Pyrite spectrum
10.
20.
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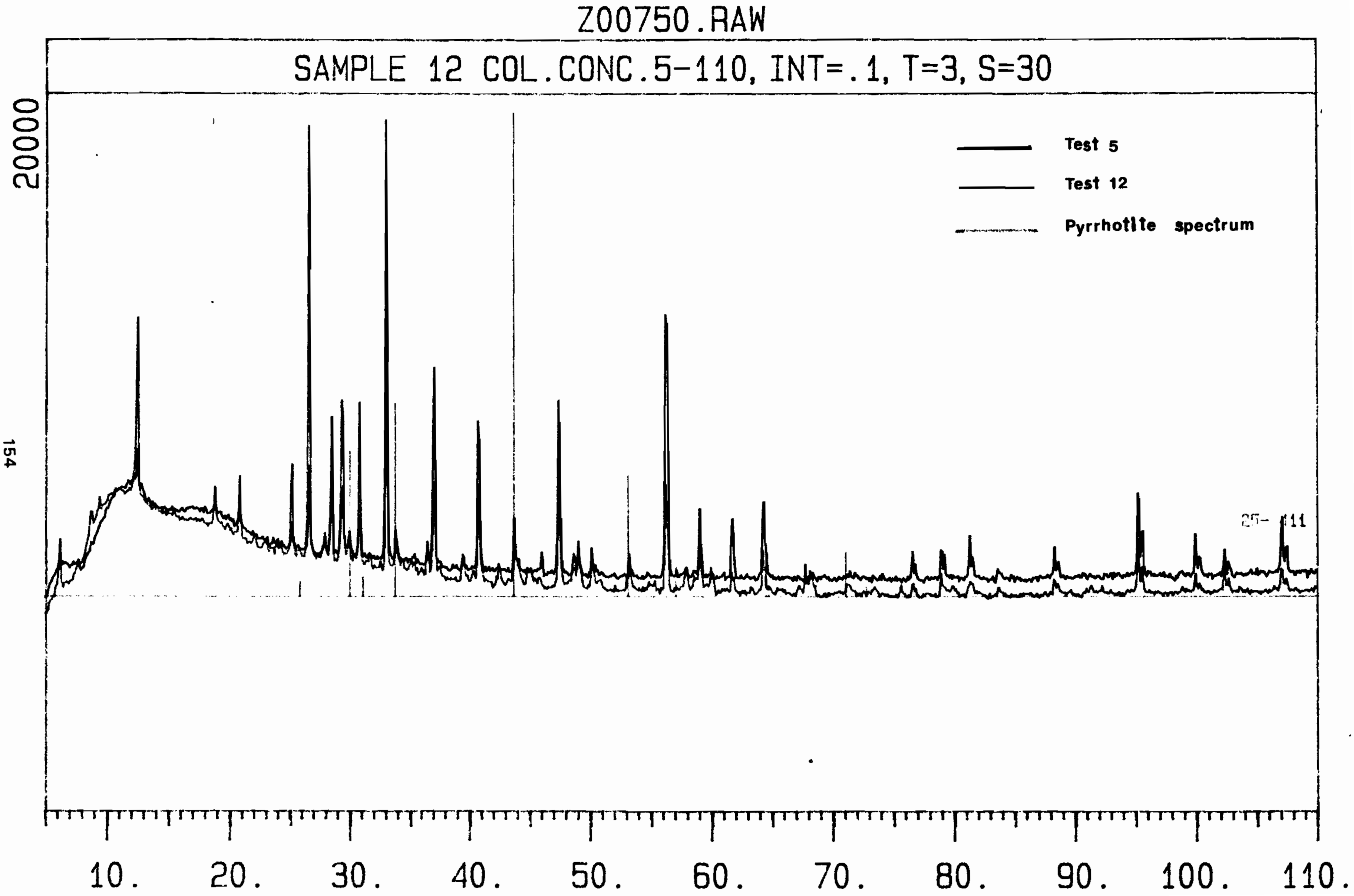
70.
80.
90.
100.
110.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20000
Test 5
Test 12
Pyrrhotite spectrum
25- 411
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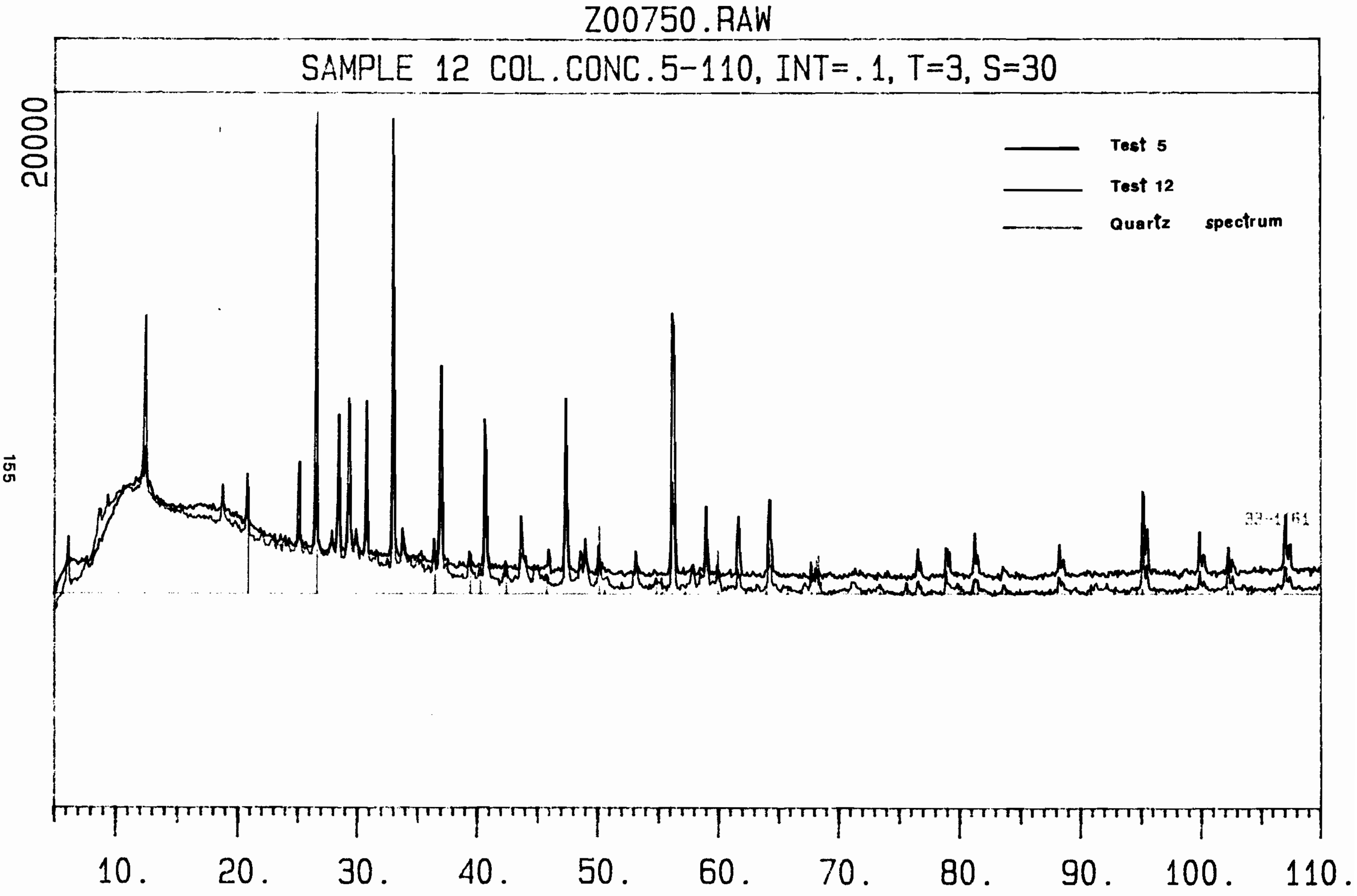
70.
80.
90.
100.
110.



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20000
Test 5
Test 12
Quartz spectrum
33-1161
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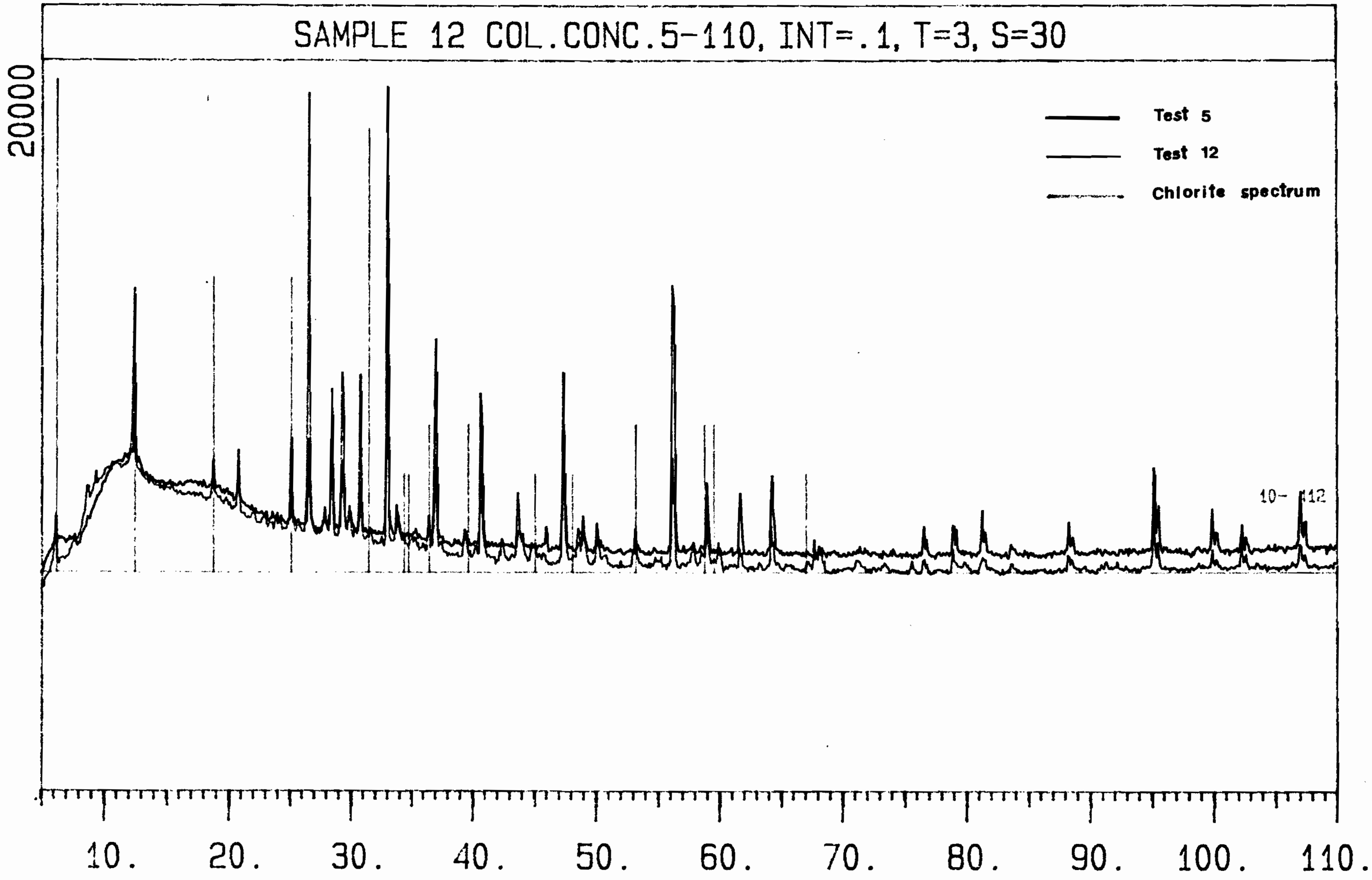
90.
100.
110.

Z00750.RAW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Test 5
Test 12
Chlorite spectrum
10- 412
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20.
30.
40.
50.
60.
70.
80.
90.
100.
110.