Use of nitrogen in flotation for iron sulphide rejection

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I. ABSTRACT.

The rejection of iron sulphides (pyrite and pyrrhotite) using nitrogen was evaluated on three Canadian complex sulphide ores Laboratory scale work (at McGill and on-site) was conducted and, in addition, continuous minicell testwork was conducted on-site.

The promotion of pyrite flotation using nitrogen was confirmed. It was evaluated at Brunswick Mining ahead of zinc flotation (using the minicells), and at Kidd Creek Mines in a laboratory scale investigation of reverse flotation. This latter also showed enhanced chalcopyrite flotation with nitrogen.

The explanation of pyrite flotation using nitrogen was based on weakening of galvanic coupling with other sulphides.

Nitrogen proved more effective than air in depression of pyrrhotite in processing a pentlandite/chalcopyrite ore. Under some conditions, chalcopyrite was also depressed by nitrogen (apparently when the pulp potential was less than 0 V vs. S.H.E.).

The depressant action of nitrogen on pyrrhotite was interpreted as a reduction in the density of surface OH^- sites necessary for electrostatic interaction with xanthate ion in the Hodgson/Agar model of pyrrhotite floatability.

II. RÉSUMÉ

La dépression ou l'activation des sulfures de fer (pyrite et pyrrhotine) par utilisation de l'azote au conditionnement ou à la flottation a été évaluée avec trois minerais de sulfures complexes du Canada.

On a pu confirmer que l'azote peut promouvoir la flottation de la pyrite, par essais à Brunswick Mining en amont du circuit de flottation du zinc, avec les mini-cellules, et aux mines Kidd Creek, par flottation renversée dans une cellule de laboratoire. Dans ce dernier cas, nous avons pu constater que l'azote facilitait également la flottation de la chalcopyrite.

Nous expliquons l'effet de l'azote sur la flottation de la pyrite par une diminution des effets galvaniques entre la pyrite et les autres sulfures.

L'azote s'est montré plus efficace que l'air pour déprimer la pyrrhotine lors de la flottation des minerais de pentlandite et chalcopyrite de la région de Sudbury. Parfois, l'azote pouvait également déprimer la chalcopyrite (apparemment lorsque le potentiel de pulpe, tel que mesuré avec une électrode d'hydrogène standard, était inférieur à 0 V).

L'action déprimante de l'azote sur la pyrrhotine s'explique, nous

croyons, par la réduction de la densité des sites OH^- nécessaires a l'interaction électrostatique de la pyrrhotine avec l'ion xanthate, selon le modèle d'aptitude à la flottation de la pyrrhotine propose par Hodgson et Agar.

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1. INTRODUCTION.

Flotation is a physico-chemical process for separating finely divided particles of valuable minerals from waste minerals (gangue). It is based on the difference in the surface properties of the minerals present in the ore. Most species contained in the ore are hydrophilic but some, such as graphite, talc, sulphur and molybdenite are hydrophobic and others can be given a hydrophobic surface by reaction with suitable chemical compounds, including oxygen In a mixture of particles having different surface properties with respect to water the particles can be separated using the flotation process. The product containing the valuable minerals is the concentrate, the product containing the gangue is the tailings.

The application of electrochemical concepts in the flotation of sulphide minerals has helped to explain the formation of a range of surface species and demonstrates how the collector and oxygen interact at the mineral surface (1).

This work describes batch and continuous flotation tests using nitrogen as the flotation gas to modify the electrochemical reactions. The results are compared against air flotation.

Sections 2 and 3 present a review of the theoretical background and the use of gases other than air in flotation. Section 4 describes the operating practice at the three plants involved in the work.

The test procedures for the three ores investigated are given in Section 5.

As described in Section 6, continuous in-plant flotation tests at Brunswick Mining were conducted in order to test a technique already proven on bench scale to lower the pyrite content in the sphalerite concentrate generated from complex sulphide ores (2). The proposed process produces copper/lead concentrate in the conventional way, a pyrite concentrate using nitrogen as the flotation gas and a zinc concentrate from the pyrite tailings. Results indicate improved zinc rougher grade at comparable recoveries (2). Tests also show that low purity nitrogen (95%), sometimes available on-site at the smelter, can be used in the pyrite flotation step, making the process more attractive.

Laboratory testwork at Kidd Creek Mines demonstrated that nitrogen also enhanced the flotation of pyrite from zinc concentrate (zinc reverse flotation). Exploratory work at Kidd Creek demonstrated that using nitrogen in place of air on flotation circuit feed (primary grinding cyclone overflow) reduced the recovery of sphalerite to the concentrate without affecting the chalcopyrite recovery.

In the case of Falconbridge ore, nitrogen flotation proved superior to air flotation for pyrrhotite rejection. Under some conditions chalcopyrite was also depressed by nitrogen.

2. THEORETICAL BACKGROUND.

2.1. Flotation Mechanisms.

Three modes of flotation are generally recognized: natural flotation, collectorless (or self-induced) flotation and flotation induced by a collector. In the first case the mineral is naturally hydrophobic; in the second its surface is made hydrophobic by reaction with its environment.

The natural floatability of some sulphide minerals has always been questioned. However, flotation in the absence of collectors is observed in industry; for example, flotation of molybdenite. 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.

Collectorless flotation refers to the hydrophobicity acquired by electrochemical interaction with the environment. This can be induced through oxidation, for example, in the grinding circuit or by addition of chemicals such as sodium sulphide (3). Most sulphide minerals respond to collectorless flotation to varying degrees.

Up to now the common practice involves the use of collectors. The collector is the reagent which produces the hydrophobic film on the mineral particle.

Most collectors in current use are surface active compounds with a heteropolar structure, i.e. they contain a polar and a non-polar group. When attached to the mineral particle, such collector molecules are oriented so that the non-polar or hydrocarbon group extends outward. This results in the formation of a hydrophobic hydrocarbon-type film at the mineral surface.

2.2. Reactions Leading to the Formation of Sulphur.

Sulphide minerals apparently oxidize through a continuum of metal deficient sulphides of decreasing metal content up to elemental sulphur (4).

Many investigators have proved that most sulphide minerals, under certain conditions of pulp potential and pH, can be floated in the absence of collectors (3, 5, 6). They suggest that if not oxygen itself at least an oxidizing agent is required for collectorless flotation.

The collectorless flotation of chalcopyrite is probably the most widely studied. The key role of elemental sulphur in its collectorless flotation was verified by Gardner and Woods (7). Voltammetric studies indicated that floatability is associated with anodic oxidation of the surface of chalcopyrite to produce CuS, iron hydroxide and S in alkaline solutions.

$$CuFeS_2 + 2H_2O = 0.25Cu^{2+} + 0.75CuS + Fe(OH)_2 + 2H^+ + 1.25S^\circ + + 2.5e^-$$

Buckley, Hamilton and Woods (8) refuted these findings, arguing that voltammograms are not sufficiently discriminating to distinguish between the possible reactions and reiterating their confidence in the XPS spectra. They extended their studies to include galena, bornite, chalcopyrite and pyrrhotite and maintained that, for each mineral, oxidation involves the progressive removal of metal atoms leaving metal-deficient sulphides with sulphur lattices only marginally altered from the original structure.

Sulphides with stoichiometries differing markedly from those of established stable sulphide phases have been identified. The overall general reaction suggested is:

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

Galena, ground and classified under reducing conditions, is not floatable in the absence of a collector. Subsequent oxidation generates a hydrophobic reaction product (9), possibly via the following intermediate reaction:

$$PbS + xH_2O = Pb_{1-x}S + xPbO + 2xH^+ + 2xe^-$$

with further oxidation producing S and finally SO_4^{2-}.

Some workers suggest that the activation of sphalerite by copper results in the formation of a covellite (CuS) type surface (10) Further oxidation of the covellite leads to the formation of elemental sulphur:

$CuS + 2H_2O = S^\circ + Cu(OH)_2 + 2H^+ + 2e^-$

Xanthate could interact with $Cu(OH)_2$ to form a copper xanthate surface complex.

The oxidation of pyrrhotite is considered as a two stage process involving the sequential oxidation of Fe^{2+} and sulphur on the mineral surface (11). The following equations are proposed :

Partial oxidation of Fe^{2+} to form the hydroxide and of S (anionic) to the polysulphide:

$FeS + H_2O = [Fe(OH)_a](S) + (2-a)H^+ + (2-a)e^-$

Further oxidation of the polysulphide and complete hydroxylation of the $Fe(OH)_a$:

$[Fe(OH)_a](S) + (3-a)H_2O = Fe(OH)_3 + S^\circ + 3H^+ + 3e^-$

Sulphur will continue to oxidize to SO_4^{2-}.

Hamilton and Woods (12) found, through electrochemical studies, that pyrite oxidizes to both sulphur and sulphate. They found that the formation of sulphur is restricted to the order of a monolayer at values of pH over 9.

2.3. Electrochemical Reactions in Collector-Mineral Interaction.

Interaction of the collector species with the mineral surface is considered to take place via coupled charge transfer processes involving anodic oxidation of the collector to form a hydrophobic species and cathodic reduction of oxygen. The potential difference across the mineral-solution interface is the important factor determining the rate of such reactions and hence the flotation behavior exhibited by the mineral (13).

It is generally accepted that the interactions between the constituents of sulphide mineral flotation systems depend on the prior or simultaneous oxidation of the mineral. The potential difference at the mineral-solution interface is an important parameter because it determines the nature and the rate of reactions at the surface.

Salami and Nixon (14) suggested that chemical interactions on the surface would occur by an electrochemical mechanism Sulphides can act as a source or sink for electrons and consequently can support electron reactions at the surface. The concept of surface reactions proceeding by separate electrochemical steps is well established in many fields in which conducting materials are involved. Metal corrosion is a good example.

There are several known reactions. One example is oxidation of xanthate to dixanthogen,

$$4ROCS_2^- + O_2 + 4H^+ = 2(ROCS_2)_2 + 2H_2O$$

which occurs as the result of anodic oxidation

$$2ROCS_2^- = (ROCS_2)_2 + 2e^-$$

and the cathodic reduction of O_2

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

It is known that xanthates can react with sulphides by charge transfer reactions. For example, in the case of chalcopyrite, these reactions may lead to cuprous xanthate by the reaction:

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

or to cupric xanthate by the reaction:

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

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 (15).

Pyrite-xanthate interaction has been studied for many years. It is well known that xanthate is oxidized to dixanthogen in the presence of pyrite and oxygen. Fuerstenau and co-workers (16) proposed three possible electrochemical reactions for dixanthogen formation:

$$2X^- + (1/2)O_2 + H_2O = 2OH^- + X_2$$

$$2Fe^{3+} + 2X^- = 2Fe^{2+} + X_2$$

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

The first mechanism was ignored because oxidation of xanthate by dissolved oxygen has been shown to be negligible (17). The authors thus postulated that the oxidation of xanthate to dixanthogen was due to either Fe^{3+} or $Fe(OH)_3$.

According to Peters and Majima (18) pyrite is in a passive state after exposure to air, and the surface film formed may be adsorbed oxygen or an oxygen-sulphur compound. Assuming that adsorbed oxygen leads to passivity, Majima and Takeda (50) proposed the following mechanism for xanthate oxidation to dixanthogen:

$$(1/2)O_{2(ad)} + 2X^{-}_{(aq)} + H_2O = X_{2(ad)} + 2OH^{-}_{(aq)}$$

According to Majima and Takeda, electrochemical studies indicate that pyrite is acting as a catalyst in allowing the one redox system, $O_{2(ad)}$ and $OH^{-}_{(aq)}$, to equilibrate with another, i e. $X^{-}_{(aq)}$ and $X_{2(ad)}$.

In a recent study, Wang and Forssberg (19) show that dixanthogen is not the sole species responsible for the flotation behavior of pyrite. They confirmed the formation of Fe^{3+}-xanthate compounds on the pyrite surface by Fourier spectroscopic studies (FTIR). They also argue that above the Fe^{3+}-xanthate compound layer, dixanthogen and xanthate ions are co-adsorbed as multiple layers.

Hodgson and Agar (20) suggest that xanthate is adsorbed on the pyrrhotite surface forming an $Fe(OH)_2X$ product. They suggest that the adsorption of xanthate on the surface is through coulombic attraction. They proposed the following mechanism.

$FeS + H_2O = Fe(OH)[S]^+ + H^+ + 2e^-$ (e.g. oxidation)

$Fe(OH)[S]^+ + X^- = Fe(OH)[S]X$ (e.g. coulombic attraction)

Xanthate then oxidizes to dixanthogen through the reduction of oxygen at the pyrrhotite surface:

$2X^- = X_2 + 2e^-$

$(1/2)O_2 + H_2O + 2e^- = 2OH^-$

In contrast, unlike pyrrhotite, the formation of dixanthogen on pentlandite was considered to originate from chemisorbed xanthate on the surface, according to the equilibria (20):

$(Ni) + X^- = NiX + e^-$

$NiX + X^- = Ni(X)_2 = NiX_2 + e^-$

2.4. Importance of Pulp Potential.

The importance of electrochemical interaction in sulphide systems has led to the recognition of pulp potential as an important parameter in flotation.

The electrochemical potentials measured in the flotation pulp can be interpreted as indicators of the readiness of the sulphide minerals to float. Selective flotation is possible if the target mineral is made ready for flotation before the others.

The potential of a solid-solution interface is determined by the presence of oxidizing and reducing species in solution. In order to monitor redox properties, it is customary to measure the potential of an indicator electrode placed in the solution. A noble metal (Au or Pt) is generally used for this purpose because of their high resistance to corrosion.

Pulp potential control can be maintained by potentiostatic or chemical methods. With potentiostatic control the pulp potential can be readily manipulated to any desired value, whereas chemical control is less flexible. Chemical control, however, produces a more uniform electrochemical environment around the sulphide particles in the pulp, whereas potentiostatic control is very dependent on the efficiency of electrode-particle contact. Only chemical control is used in plant practice.

In presenting a detailed description of mineral-collector-oxygen interaction, it has often been suggested (1) that the mineral surface must first be oxidized to metal oxy-sulphur species which then exchange with collector ions in solution to form a metal collector compound.

2.5. Galvanic Interaction.

Sulphide minerals are semiconductors and they assume different potentials in an aqueous medium. If they make contact with each other or with other conducting surfaces an electrochemical cell will form. The mineral or conductor with the higher rest potential will act as a cathode while the anode will be the mineral or material with the lower rest potential. This particular type of electrochemical cell is called a galvanic cell and the electrochemical reaction occurring is called galvanic interaction.

Galvanic interactions occur between mineral and grinding medium like mild steel. An example is that of reaction between iron and sphalerite where electrons are drawn from the metal to the mineral and transferred to oxygen (Figure 3.1).

The presence of galvanic interactions in flotation may be beneficial or detrimental depending on the system. In Figure 3.1 the sphalerite tends to be depressed because of the galvanic couple. Iron ions being released form precipitates of iron hydroxide which coat the sphalerite (Learmont and Iwasaki argued this to explain depression of galena when contacted with mild steel(21)).

Among all sulphide minerals, pyrite has the highest rest potential. Electrochemical studies on pyrite (18) showed that the surface of pyrite, with a rest potential of +0.66 Volts, may be regarded as electrochemically passive. From this, when pyrite is in galvanic contact with other sulphide minerals, their oxidation will be enhanced. Such is the case of sphalerite when it is in contact with pyrite (Figure 3.2). The reaction that takes place on the surface of the sphalerite particles is the oxidation:

$$ZnS = Zn^{2+} + S^{\circ} + 2e^{-}$$

While on the more noble mineral, pyrite, oxygen is reduced by the reaction:

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

In this case, sphalerite may become hydrophobic even in the absence of collector because of the formation of elemental sulphur at the mineral surface.

Galvanic interactions in several sulphide systems have been studied in detail by Iwasaki and co-workers (21, 22).

Nakazawa and Iwasaki (23) showed that the flotation of pyrrhotite decreased with increased aeration and activity of oxygen. They suggested that the decrease in the recovery of pyrrhotite was attributable to the formation of a surface film such as iron hydroxide, on the pyrrhotite. However, the recovery of pyrrhotite increased when pyrite was present in the system. Since pyrite is more cathodic than pyrrhotite, it will act as a cathode and pyrrhotite as an anode upon contact. The galvanic interaction is dependent on the activity of oxygen in the slurry since oxygen is required as the electron accepting element.Galvanic contact accelerated the oxidation of pyrrhotite. Apparently, the precipitates of iron hydroxide were

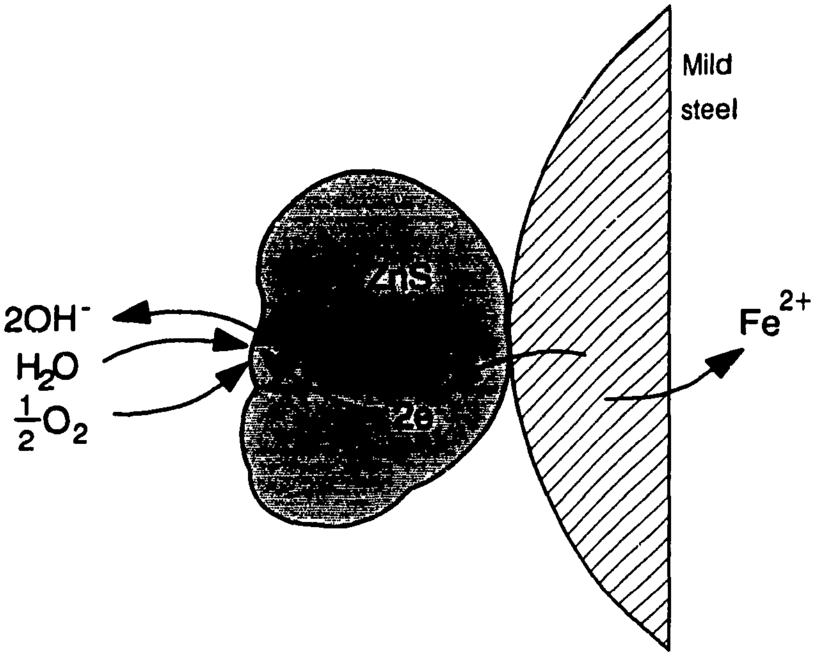


Fig 3.1 Galvanic interaction between sphalerite and grinding media.

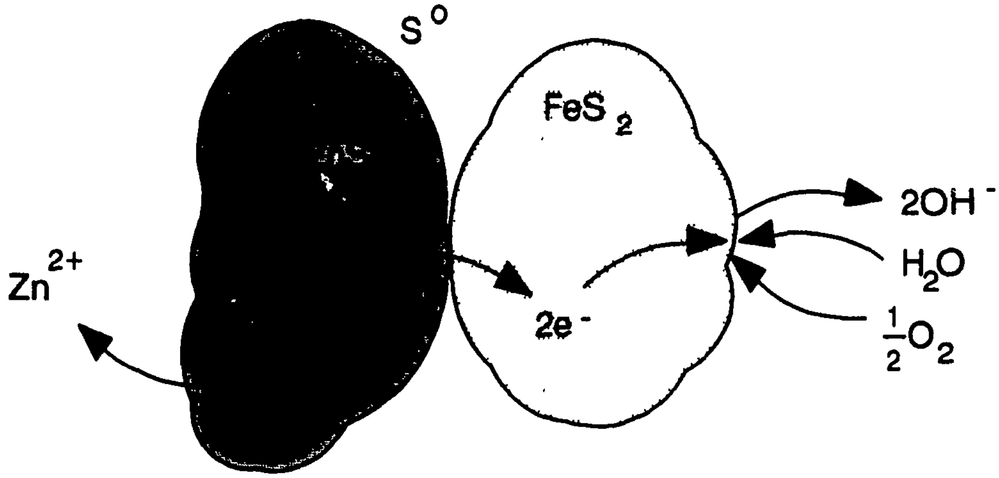


Fig 3.2 Galvanic interaction between sphalerite and pyrite particles.

formed by the reaction of iron ion released by pyrite itself. It has been demonstrated (24) that the type of grinding mill, grinding media, and chemical environment within the mill have a considerable effect on the flotation of sulphide minerals, in part because of the galvanic contacts made.

2.6. Factors Affecting Selectivity in Sulphide Flotation.

Rey and Formanek (25) investigated various factors affecting the selectivity of lead-zinc separation and reported that pure sphalerite ground in a porcelain mill, alone or in a mixture with silica, showed natural floatability. Grinding in an iron mill or in the presence of limestone decreased the floatability and addition of iron powder into the porcelain mill had the same effect as grinding in an iron mill. The suppression could be attributed to the strongly reducing environment in an iron ball mill, which is a result of the reactions:

$$Fe^{++} + 2e^{-} = Fe^{\circ} \qquad E^{\circ} = -0.440\ V$$

or

$$Fe(OH)_2 + 2e^{-} = Fe^{\circ} + 2OH^{-} \qquad E^{\circ} = -0.877\ V$$

This hypothesis is further supported by more recent work concerned with galena flotation. Batch tests (26) on an ore that contained lead, zinc and iron sulphides showed that, when the relatively low pulp potential that resulted from grinding was maintained in the cell by use of nitrogen as the flotation gas, galena flotation was suppressed and a mineralized froth with a high iron assay was obtained. Aeration of the pulp followed by flotation with air resulted in normal flotation of galena. In some tests in which air was the flotation gas but pre-aeration was not carried out, the content of froths collected during the first half minute of flotation was usually low in lead and high in iron, whereas subsequent froths assayed much higher in lead content. It is known that aeration rapidly raises the Eh of a reduced pulp (18) and it is inferred that the effects observed in the above batch tests were due to the Eh values established during the tests.

2.7. Mechanisms of Pyrite and Sphalerite Flotation.

Rao and Finch (27) found that when flotation is conducted in air from a galena-pyrite mixture galena is preferentially floated in the early stages and when flotation is conducted in nitrogen pyrite floats faster than galena. With a pyrite-sphalerite mixture, nitrogen enhanced pyrite flotation also. This preferential flotation of pyrite with nitrogen may be exploitable.

In plant practice galvanic interference appears to be beneficial since it favors pyrite depression by lowering the potential (and therefore the likelihood of dixanthogen formation) as well as promoting OH^- formation which competes with xanthate uptake.

Xanthate uptake by pyrite alone is low under conditions where the xanthate solution was prepared in nitrogenated water. Rao and Finch

(28) suggest that this may be related to the drop in the potential of pyrite due to the reduction of oxygen content (<0.8 ppm) upon nitrogenation. The results are different when the pyrite is mixed with xanthate in distilled water which has not been nitrogenated. In this case the potential of pyrite in contact with xanthate solution is not lowered. The introduction of nitrogen at the adsorption stage helps to weaken the galvanic interaction between pyrite and sphalerite. Since pyrite is nobler than sphalerite it functions as a cathode drawing electrons when the two minerals are in contact. The dissolved oxygen in water accepts these electrons to form hydroxide which competes with xanthate for adsorption sites. Upon nitrogenation the oxygen content is reduced, the galvanic interaction is interrupted and xanthate is able to adsorb more freely at pyrite.

3. USE OF GASES OTHER THAN AIR IN FLOTATION.

3.1. Introduction.

Use of gases other than air in froth flotation has been reported from time to time. For example, in the case of nonsulphides, Lin and Metzer (29) studied the effects of hydrocarbon gases. In the case of sulphides, nitrogen is now widely used in the separation of Mo/Cu, probably initiated by the work of Podobnik and Shirley (30) at Cuajone.

Two gases which are used as reagents in sulphide flotation are sulphur dioxide and oxygen.

Sulphur dioxide (SO_2) is employed as a sphalerite and pyrite depressant. One explanation for this depression is the formation of metal sulphites on the mineral surfaces, which inhibit collector adsorption (31). Other suggested mechanisms are xanthate decomposition due to a SO_2-promoted oxidation reaction, and retarded collector adsorption due to a shift of the redox potential to an unfavorable range (32).

The mineral-reagent oxidation reactions due to the presence of oxygen has been the subject of research and controversy for many years. The importance of oxygen in sulphide flotation using xanthate type collectors has been well recognized (33) and it has been found that its role varies from mineral to mineral.

3.2. The Use of Nitrogen.

Of all the gases, nitrogen would appear to hold the greatest practical promise because of its ready availability and chemical inertness. Burger (34) reports that a copper/molybdenum bulk concentrate is normally floated with the use of conventional collectors (xanthate and dithiophosphate). The natural hydrophobicity of molybdenite is taken as an advantage for its separation from chalcopyrite which is depressed by removing its collector coating with a strong reducing agent (e.g. sodium hydrosulphide or Nokes reagent).

Burger notes that sodium hydrosulphide, like any other strong reducing agent, is readily oxidized by dissolved oxygen during flotation and as a result up to 8 kg/tonne of sodium hydrosulphide is used which amounts to more than 300 times the 25 g/tonne that is needed according to stoichiometric considerations. The use of nitrogen in chalcopyrite/molybdenite separation, therefore, is largely to reduce the consumption of hydrosulphide. The same holds true in the case of Nokes reagent (a polysulphide compound) which is also used as a depressant for chalcopyrite.

Savings by using nitrogen are often considerable. Podobnik and Shirley (30) reported that the savings in reagent consumption were over 50%. Redfearn (35) reports that at Gibraltar Mines, the savings in sodium hydrosulphide consumption have been more than 75%. In The Soviet Union the use of nitrogen has been reported to reduce reagent costs by about 50% (36).

Metallurgically, nitrogen has two important properties: it lowers the dissolved oxygen in the pulp and, as a consequence, reduces the pulp potential of the system. Selective flotation of sulphides is affected by both of these changes.

The reduced concentration of oxygen in the pulp affects galvanic interactions between sulphide minerals (37, 38). As discussed in Section 2, pyrite is the most cathodic of the common sulphide minerals and draws electrons from less cathodic contacting minerals. The electron flow is maintained by the presence of oxygen which acts as an electron accepter to form hydroxyl ions. The hydroxyl ions at the pyrite surface make the mineral hydrophilic or, at least, less hydrophobic (1). In presence of nitrogen which lowers the activity of oxygen to <1 ppm this galvanic interaction is hindered by the elimination of the oxygen and the pyrite is rendered floatable.

The reduced pulp potential which accompanies the loss of oxygen may itself play a role in, for example, mineral-collector interaction. Generally, sulphide-xanthate interaction ceases below a certain potential value which depends on the mineral(s) system.

From a practical point of view nitrogen is becoming cheaper to produce and is often available on-site as a by-product from oxygen plants, with around 95% purity. It is chemically inert and poorly soluble in water. Recent developments in column flotation (including the Jameson cell) where usually less gas is required than in mechanical cells, and covered cells permitting gas recycling also give an incentive to investigation of gases other than air in the flotation process.

4. ORE SAMPLES AND PLANT PRACTICE.

4.1. Brunswick Mining and Smelting (BMS).

The ore is composed of approximately 80 percent sulphide minerals with the remaining 20 percent being mainly quartz, calcite and silicate minerals. Massive pyrite is the predominant mineral and is the main contaminant of the concentrates produced. The economic minerals are chalcopyrite, galena, sphalerite and silver minerals An approximate mineral composition of the Brunswick Mining and Smelting (BMS) ore is given in Table 4.1 (39).

TABLE 4.1. Mineral Composition of B.M.S. Mill Feed.

MINERAL	DISTRIBUTION, %
Sphalerite (ZnS)	15.0
Chalcopyrite ($CuFeS_2$)	1.0
Galena (PbS)	4.0
Pyrite (FeS_2)	60.0
Quartz (SiO_2)	12.0
Calcite ($CaCO_3$)	1.0
Pyrrhotite (Fe_7S_8)	1.0

Sphalerite from the Brunswick Mining ore contains approximately 6 percent Fe and 60 percent Zn as reported by Hendriks et.al. (39)

Some of the other minerals present in the ore body are arsenopyrite (FeAsS), marcasite (FeS_2), boulangerite ($5PbS \cdot 2Sb_2S_3$), stannite (Cu_2FeSnS_4), cassiterite (SnO_2), and pyrargyrite (Ag_3SbS_3). Silver is mainly present as tetrahedrite ($(Cu,Fe,Zn,Ag)_{12}Sb_4S_{13}$)

Mine production is entirely from underground. Currently the mill processes over 10,000 tonnes of run-of-mine ore per day (40).

Figure 4.1 presents the BMS general plant flowsheet. The crusher circuit generates a product of minus 1.2 cm. The grinding circuit consists of rod and ball mills and two stage cyclones size the mill products (41). The primary cyclones classify the rod mill discharge and are in closed circuit with the secondary ball mill. The secondary cyclones are in closed circuit with the tertiary ball mi

The Brunswick Mining ore is fine grained and the current practice is to carry out a rougher flotation at a fineness of about 65 percent passing 400 mesh (-37 µm). Cleaner circuit feeds need to be reground to about 95 percent under 37 µm in order to obtain satisfactory final concentrate grades.

Soda ash is added to maintain the pH at 8.8 in the grinding circuit and 150 g/tonne of sulphur dioxide are added to depress pyrite and sphalerite in the bulk flotation. The secondary cyclone overflow is

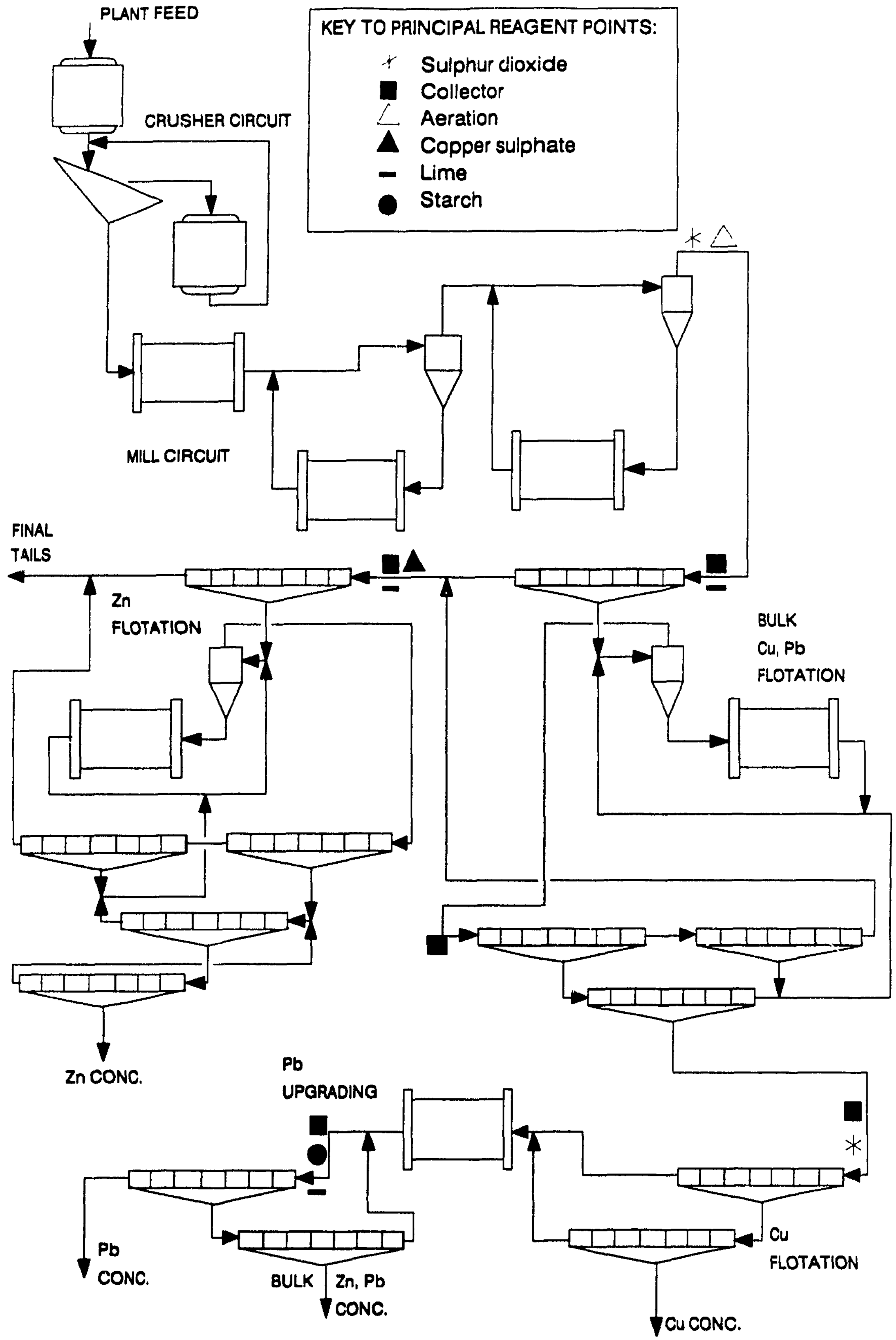


Figure 4.1 General plant flowsheet of Brunswick Mining and Smelting.

fed to an aeration tank with a retention time of 25 minutes to further depress pyrite flotation. Flotation is carried out by using 30 g/tonne of a mixture 80-20 of sodium isopropyl xanthate-sodium amyl xanthate and 20 g/tonne Cyanamid AEROFLOAT 241 (a dithiophosphate collector).

Bulk flotation produces a concentrate containing 18% combined copper and lead which is reground to ≅80% minus 400 mesh. Two stages of cleaning produce a concentrate containing 29% lead and 2.5% copper. The bulk cleaner tails together with the bulk scavenger tails form the zinc flotation feed.

The bulk cleaner concentrate is treated with activated carbon to remove the excess of collector and with starch which helps to depress the galena. Sulphur dioxide is then added to maintain the pH at 4.8 for copper flotation. 10-20 g/tonne of Cyanamid AERO 3894 is used for copper flotation which is followed by three stages of cleaning The final copper concentrate contains 22% copper and 6% lead.

Tails from copper flotation circuit are reground to ≅85% minus 400 mesh (-37 μm) before being fed to the zinc/lead separation circuit. The mill discharge is heated to 70°C to destroy the collector or starch remaining from the copper flotation circuit. The pulp is then cooled to 35°C.

Lime is added to raise the pH to 10.8 and 1000 g/tonne copper sulphate is added to activate the zinc minerals. A bulk rougher concentrate is then floated with xanthate and cleaned twice, producing a final bulk lead/zinc concentrate containing 34% zinc and 18% lead. The tailings from this circuit is the final lead concentrate assaying 35% lead and 5% zinc.

The zinc circuit is conditioned with 650 g/tonne of copper sulphate, 30 g/tonne of xanthate and 20 g/tonne of Cyanamid's AERO 3894. A zinc rougher concentrate assaying 34% zinc is obtained from 32 Outokumpu 8 m^3 cells. This concentrate is reground to 95% minus 400 mesh (-37 μm) and floated three times in a closed zinc cleaner circuit and a final zinc concentrate assaying 52% zinc is obtained.

4.2. The Kidd Creek Concentrator.

The Kidd Creek Concentrator is located 28 km east of Timmins, Ontario.

The concentrator until recently treated two types of ore: "A" type (a copper-zinc ore with low silver values) and "C" type (a copper-lead-zinc ore with high silver values) at a combined rate of over 12,000 tonnes per day. (Production from C-type has now ceased)

Flotation of the crushed and ground ore yields copper and zinc concentrates from the "A" type ore, and copper, lead and zinc concentrates from the "C" type ore. A general plant flowsheet is shown in Figure 4.2 for the "A" type ore and in Figure 4.3 for the "C" type ore.

The Kidd Creek ore body is separated into two distinct zones, the North Orebody and the South Orebody. The North Orebody includes copper ore, mainly chalcopyrite, and a massive sphalerite-pyrite ore containing native silver and galena. The South Orebody is characterized by a mixed ore containing massive sphalerite and chalcopyrite with lower silver, pyrite and galena concentrations.

Primary crushing at the minesite reduces the run-of-mine ore to minus 150 mm. The ore is transported to the fine crushing plant where two-stage crushing reduces the primary underground crushing product to minus 16 mm.

The grinding section comprises four parallel divisions. Three divisions ("A", "B" and "C") treat copper-zinc ore (known as "A" ore) while the fourth division ("D") treats either copper-zinc or copper-lead-zinc ore.

In the three grinding divisions treating copper-zinc ore, the ore is ground in an open-circuit rod mill. Rod mill discharge is combined with primary ball mill discharge to feed primary cyclones. Cyclone underflow is recirculated to the primary ball mill and cyclone overflow feeds the primary copper roughers. Primary rougher flotation tails are ground in the secondary ball mill before secondary copper and zinc flotation. Regrind mills treat copper and zinc flotation middlings.

The copper-lead-zinc ore (known as "C" ore) is ground in an open-circuit rod mill followed by primary and secondary ball milling operated in closed-circuit. A regrind mill treats zinc flotation middlings.

4.2.1. Copper-zinc ("A") Ore.

The primary grinding circuit product is sized at 50% minus 325 mesh (-44 µm). Conditioning with lime to pH 6.8, Cyanamid reagents 3418A and R208 as collectors, sodium cyanide (NaCN) and sulphur dioxide (SO_2) as pyrite and sphalerite depressants, and methyl isobutyl carbinol (MIBC) as frother prepare the mineral surface for flotation. Primary flotation is performed in three banks of five Wemco-Fagergren No. 66 machines each of 1.7 m^3 capacity.

Secondary rougher flotation feed, ground at 78% minus 325 mesh (-44 µm), is conditioned and aerated to produce a prefloat concentrate. Cyanamid's 3418A collector and MIBC are added to the secondary rougher feed prior to flotation.

Scavenger flotation follows secondary rougher flotation. Cyanamid's 3418A and R317 collectors are added.

Primary and secondary flotation concentrates are combined with the secondary cleaner tails to feed the first cleaning stage (Figure 4.2). The secondary rougher prefloat concentrate combines with the second cleaner concentrate for the third cleaning stage feed. First cleaner tails and scavenger concentrate are combined and sent to the regrind mill. After regrinding, this material combines with the secondary cyclone overflow as the prefloat conditioner feed. Third cleaner tails are recycled to feed the second cleaning stage.

Copper flotation tails are the feed for the zinc flotation circuit. A conditioning stage for the zinc rougher feed is performed by adding Cyanamid's R317 as collector, Dowfroth 250 as frother, $CuSO_4$ as sphalerite activator and lime to adjust pH to 10.0. Rougher concentration is followed by scavenging which is split into two scavenger concentrates. Rougher and primary scavenger concentrates are conditioned with lime before three stages of cleaning at pH 10.5 to 11.0. Secondary scavenger concentrate and all cleaner tails are combined and fed to the regrind mill. $CuSO_4$, Dowfroth 250 and Cyanamid's R317 are added to the cyclone overflow, which returns to the rougher flotation bank.

4.2.2. Copper-lead-zinc ("C") Ore.

A grinding product of 73% minus 325 mesh (-44 μm) is fed to the flotation circuit ("D" division) as shown in Figure 4.3.

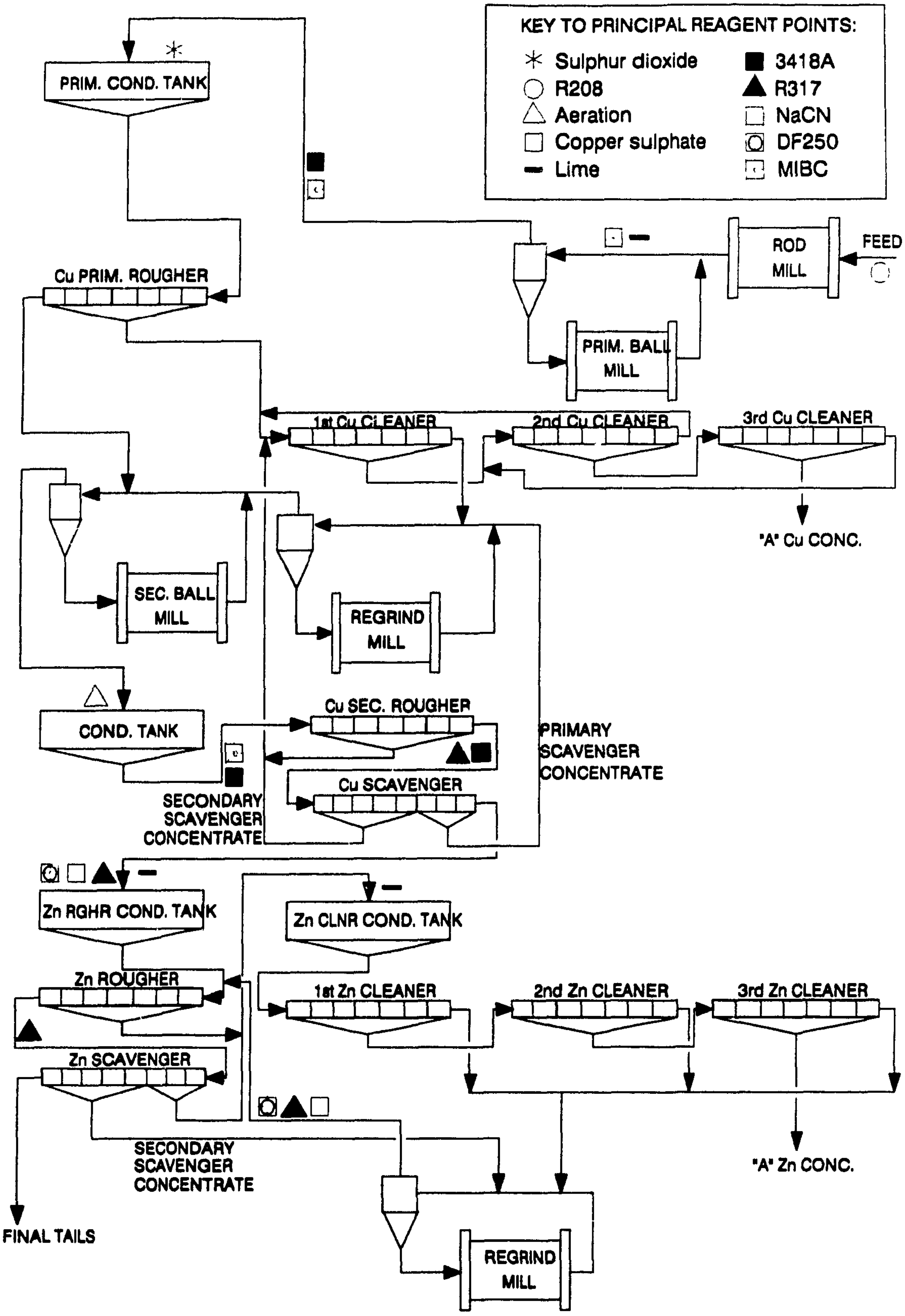


Figure 4.2 General plant flowsheet of Kidd Creek "A" ore (A, B & C divisions).

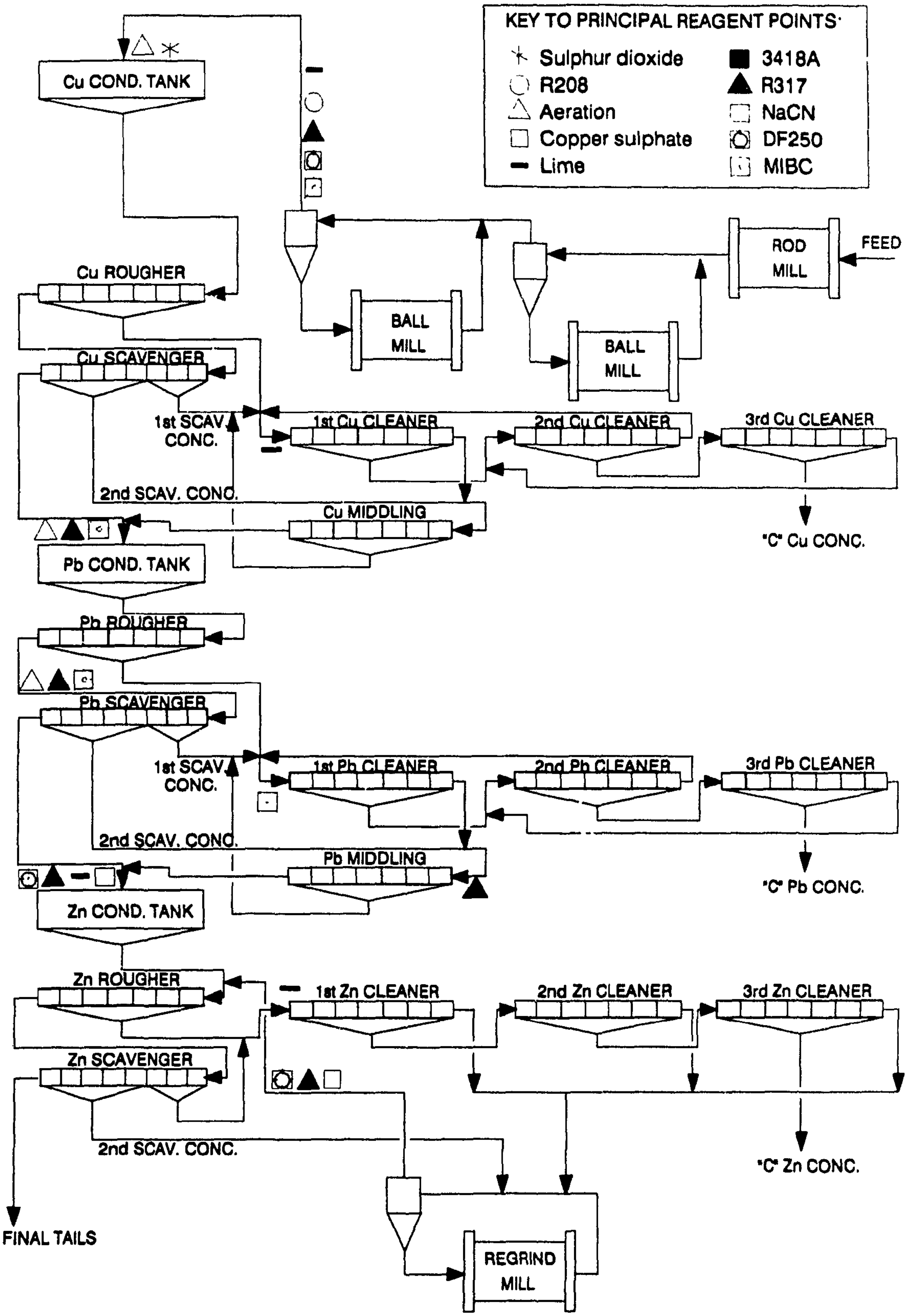


Figure 4.3 General plant flowsheet of Kidd Creek "C" ore (D division).

Copper flotation is carried out at pH of 8.5. Conditioning before flotation is done by aeration with the addition of SO_2, R317 and/or R208, MIBC or Dowfroth 250 and lime. The rougher and first scavenger concentrate are fed to a three-stage cleaning circuit after the addition of Dowfroth 250. The first cleaner tails and the second scavenger concentrate are sent to the copper middlings bank. The middlings concentrate is then combined with the copper cleaner feed. The middling tails, together with the scavenger tails are fed to the lead circuit conditioner. Tails from second and third cleaning stages are recycled to the head of the previous cleaning stage.

Lead flotation feed is conditioned at pH 8.2 adding R317, R208 and MIBC. The cell configuration is identical to that of the copper circuit.

The zinc flotation circuit of the "C" type ore is similar to that of the "A" type ore except that the pH is adjusted to 10 in the rougher stage and to 11.5 in cleaning to help in the depression of pyrite.

4.3. Falconbridge Limited (Strathcona mill).

Feed to the Strathcona mill comes from five mines located in the Sudbury area. The ore contains between 1 and 2 % Ni and between 0.8 and 1.1% Cu. The principal constituents of the ore processed at the Strathcona mill are given in Table 4.2.

TABLE 4.2. Principal Constituents of Strathcona Mill Ore.

Pentlandite	$(Fe, Ni)_9S_8$
Chalcopyrite	$FeCuS_2$
Nickelliferrous pyrrhotite: monoclinic hexagonal	Fe_7S_8
Pyrite	FeS_2
Feldspar	Aluminosilicates mainly of Ca, Na and K.
Quartz	SO_2
Mica	Hydrous aluminum silicate minerals with basal cleavage
Talc	$Mg_3Si_4O_{10}(OH)_2$
Chlorite	Silicate minerals with a cleavage parallel to (001)

Minerals of cobalt, selenium, silver, platinum, palladium, rhodium, iridium, and ruthenium are also found as secondary minerals.

The Strathcona mill flowsheet is shown in Figure 4.4. It consists of three separate grinding circuits. Circuits "A" and "B" consist of rod and ball milling in series, feeding two flotation circuits ("A" and "B"). Circuit "C" consists of ball milling in closed circuit which is used for a special high copper content ore.

The ore for circuits "A" and "B" is ground to 55-60% minus 200 mesh (-74 μm) to achieve liberation. Primary roughers generate a bulk

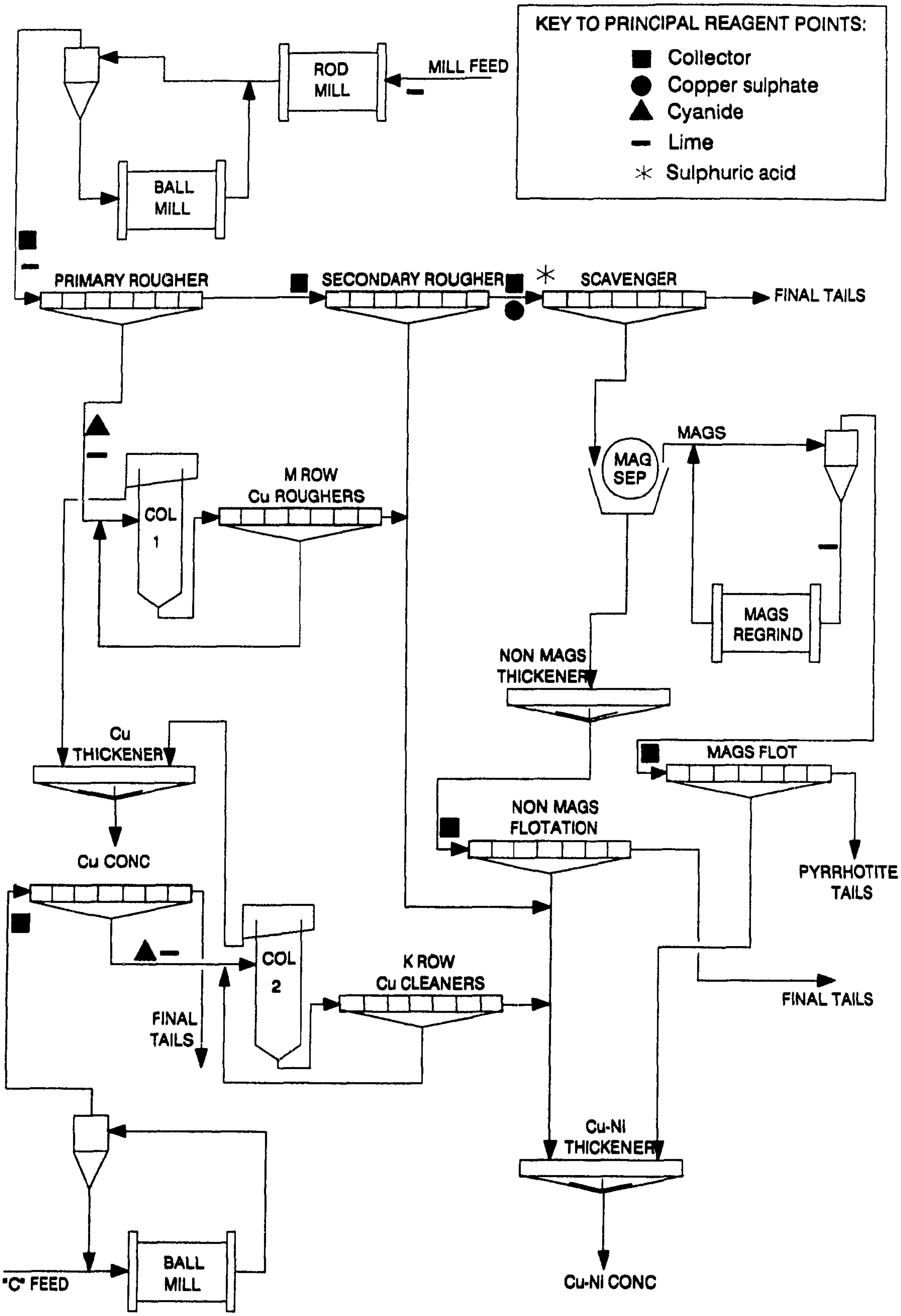


Figure 4.4 General plant flowsheet of Falconbridge Ltd. (Strathcona mill).

copper-nickel concentrate using 14 g/tonne of sodium isobutyl xanthate, 12 g/tonne of Dowfroth 250 and adjusting the pH to 9 2 with lime. Primary rougher concentrate from circuits "A" and "B" is fed to the flotation column 1 and rougher concentrate from "C" circuit is fed to the flotation column 2. Each of the columns are operated in closed circuit with a bank of flotation cells. The pH of the pulp fed to columns is adjusted at 12.3 using lime and cyanide is used to depress nickel and iron sulphides. Concentrates from columns 1 and 2 form the final copper concentrate.

The primary rougher tails are floated for nickel in the secondary rougher circuit at a pH of 9.2 adjusted with lime. This concentrate, which should contain small amounts of copper and pyrrhotite, is combined with tails from rows M and K (copper roughers and copper cleaners respectively) and shipped to the Falconbridge smelter after thickening.

The secondary rougher tails are scavenged for nickelliferrous pyrrhotite at a pH of 8.2 adjusted with H_2SO_4. Pyrrhotite is activated with $CuSO_4$ and floated using 50 g/tonne of sodium isobutyl xanthate and 12 g/tonne of Dowfroth 250. The scavenger concentrate is fed to two magnetic (0.1 Tesla) drum concentrators.

The magnetic product is reground to 80% minus 325 mesh (-44 μm) and then floated in the magnetics ("mags") circuit. The concentrate is combined with the copper-nickel concentrate and the tails discarded.

The non-magnetic fraction ("non-mags") is floated, after being thickened to about 28% solids, to recover nickelliferrous pyrrhotite at a pH of 8.2 and using sodium isobutyl xanthate as collector; the tails are discarded.

5. EXPERIMENTAL PROCEDURE.

Flotation testwork comparing nitrogen versus air as the flotation gas was carried out at the three plants, Brunswick Mining, Kidd Creek Mines, and Strathcona Mill (Falconbridge Limited).

5.1. Brunswick Mining Testwork.

Testwork on Brunswick Mining ore was done on bench scale at McGill University and in continuous minicells at the plant.

For bench scale flotation, the ore received from the plant was dried at 90°C to avoid oxidation of the ore and crushed using a Peacock Brothers cone-crusher with a synchronous motor of 5.6 kW. The crusher product was set to 100% minus 6 mesh (-3175 μm) by using a screen and returning the oversize to the crusher.

The crusher product was split into 1 kg samples using a rotary splitter model M12 from Desert Laboratories, Tucson, Arizona.

The ore was ground in a Research Hardware rod mill (Type L-1) at 65% solids. The mill was stainless steel and was 230 mm long by 178 mm diameter; the grinding media charge consisted of 15, 18mm by 225 mm rods, 2, 23 mm by 225 mm rods and 1, 30 mm by 225 mm rods, all stainless steel.

Grinding time was set to 30 minutes which produces a discharge sized at 65% minus 400 mesh (-37 μm) which is comparable to plant practice.

In the case of laboratory flotation tests performed at McGill University a 2.5 liters Leeds cell was used (Figure 5.1). The cell body in which the flotation takes place is mounted on top of the base unit. Gas control equipment is mounted on a post attached to the back on the base unit. The impeller speed is controllable.

The standard flotation procedure for the Brunswick Mining ore at McGill University is summarized in Table 5.1.

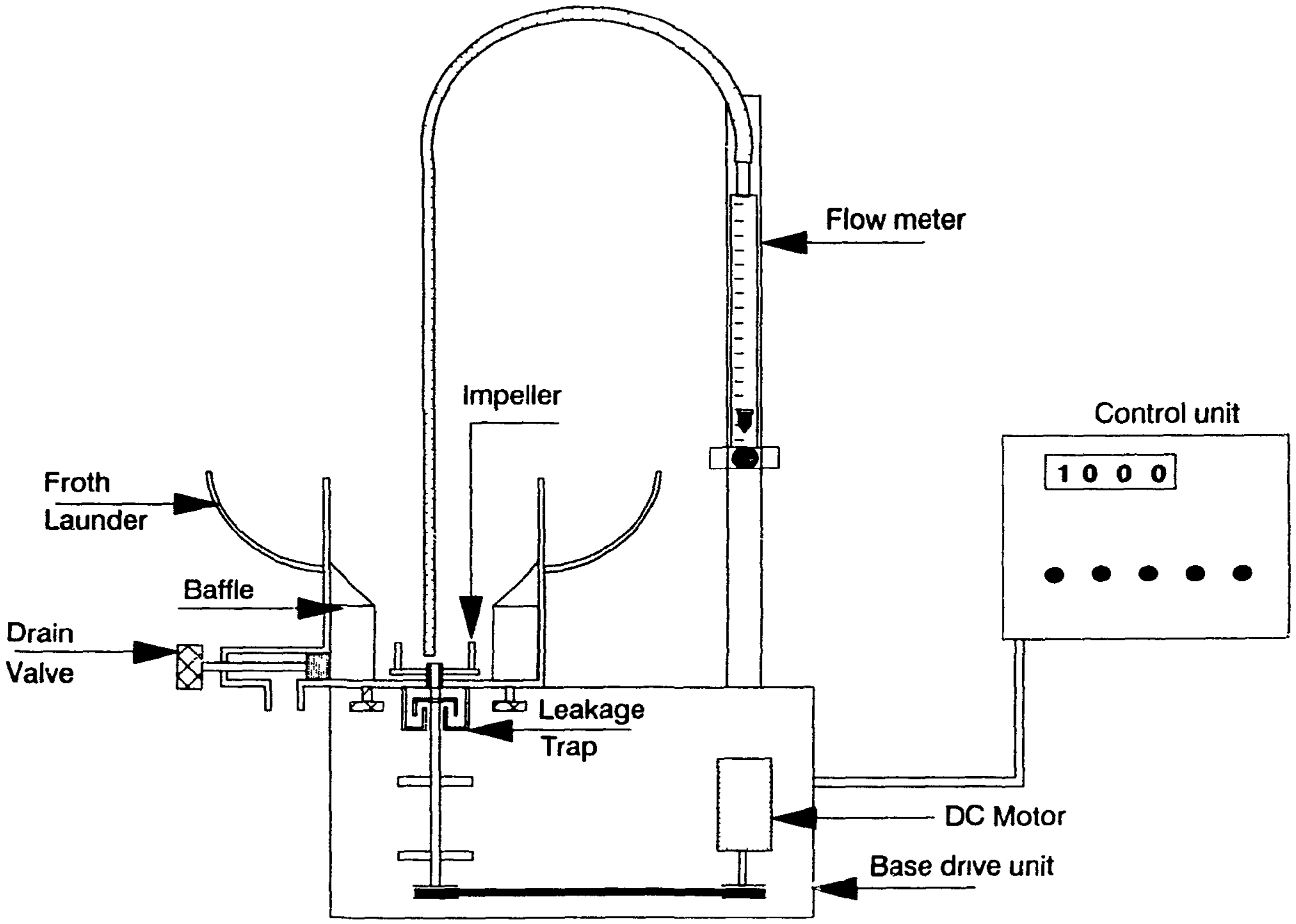


Fig 5.1 Leeds laboratory cell at McGill University.

TABLE 5.1. Test Procedure for B.M.S. Ore at McGill.

GRINDING:	
Mill	Rod
Media	Stainless Steel
Depresant (g/t)	$ZnSO_4$ (1200)
pH (modifier)	8.0 (lime)
Product size, % -37 µm	70
FLOTATION CELL:	
Type	Leeds
Capacity, l	2.5
PRE-AERATION:	
Time, min.	10
Air flowrate, l/min.	3
Impeller speed, rpm	1000
pH (modifier)	8.5 (lime)
Cu/Pb CONDITIONING:	
Time, min.	5
pH (modifier)	8.5 (lime)
Collector (g/t)	Sodium Isopropyl Xanthate (80)
Frother (g/t)	Methyl Isobuthyl Carbinol (10)
Flotation gas (l/min.)	Air (5)
Fe CONDITIONING:	
Time, min.	5
Conditioning gas (l/min.)	N_2 (3)
pH (modifier)	8.5 (lime)
Collector (g/t)	Sodium Isopropyl Xanthate (20)
Frother (g/t)	Methyl Isobuthyl Carbinol (10)
Flotation gas (l/min.)	N_2 (5)
Zn CONDITIONING 1:	
Time, min.	10
Activator (g/t)	$CuSO_4$ (1200)
pH (modifier)	10 (lime)
Zn CONDITIONING 2:	
Time, min.	2
Collector (g/t)	Sodium Isopropyl Xanthate (80)
Frother (g/t)	Methyl Isobuthyl Carbinol (10)
Flotation gas (l/min.)	Air (5)

For continuous on-site flotation tests, a sample was continuously pumped from the last cell of the copper-lead flotation bank. Flotation tests were carried out continuously in Noranda designed minicells.

The minicells are 5 liters capacity and have automatic level adjustment. Flowrate was controlled by calibrated plugs in the head tank with a diaphragm pump between the cells. The addition of reagents was done by a Cole-Parmer peristaltic pump model 7553-50 and three low flowrate pumps model C531-25T (George O. Hill Supply Ltd.)

Pulp potential (Ep) was measured using a Cole-Parmer gold electrode (model N-05994-25) and the oxygen content was obtained using an Orion oxygen electrode model 97-08-99. The pH was measured with a standard pH meter electrode (Hanna Instruments model HI-1210) All electrodes were connected to voltmeter/pH meter (Hanna Instruments model HI8424).

Figure 5.2 presents the details of a minicell and Figure 5.3 shows the flowsheet tested. The scheme consists of nitrogen flotation of pyrite ahead of sphalerite flotation. Tables 5.2 and 5.3 show the conditions followed in the different tests.

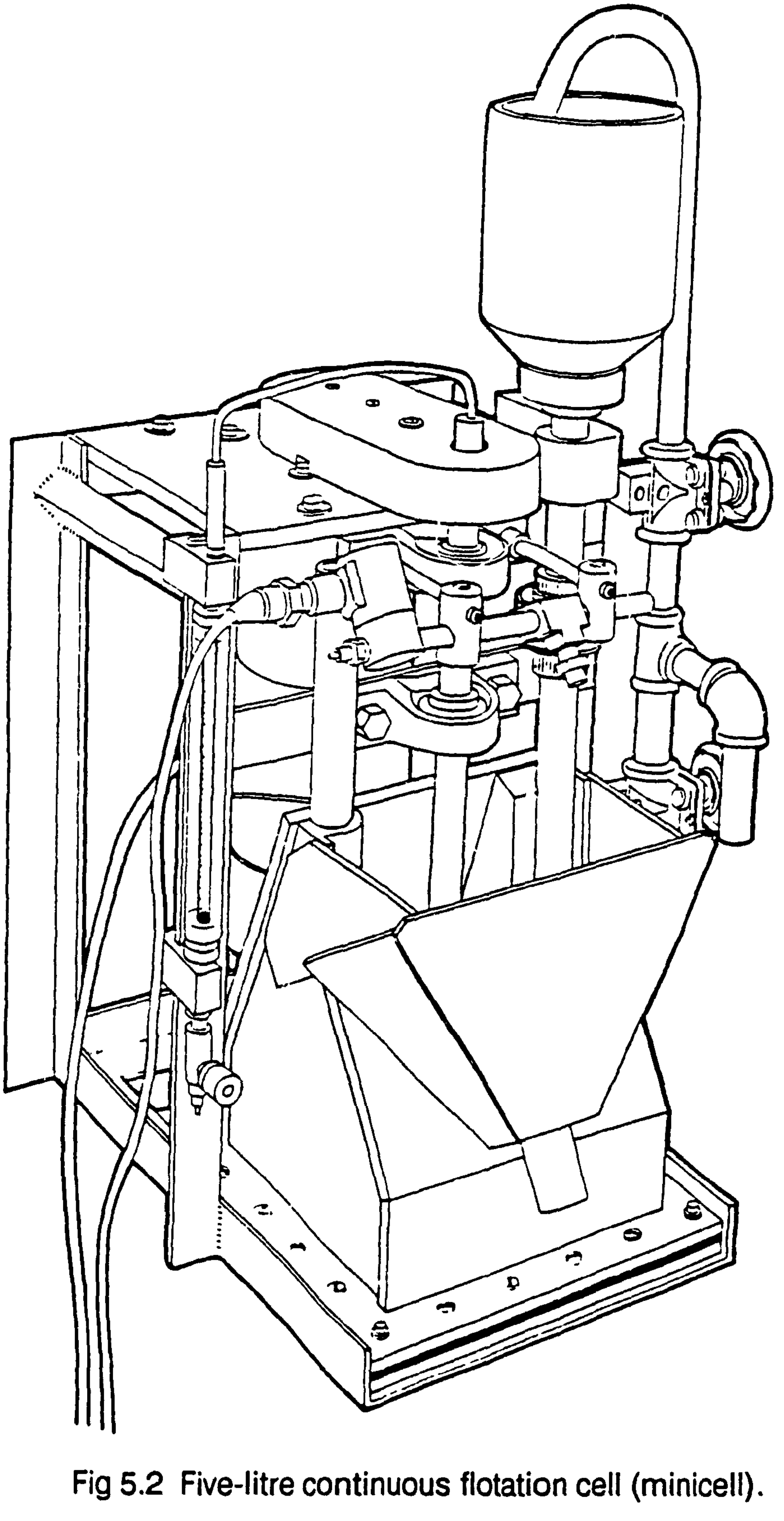


Fig 5.2 Five-litre continuous flotation cell (minicell).

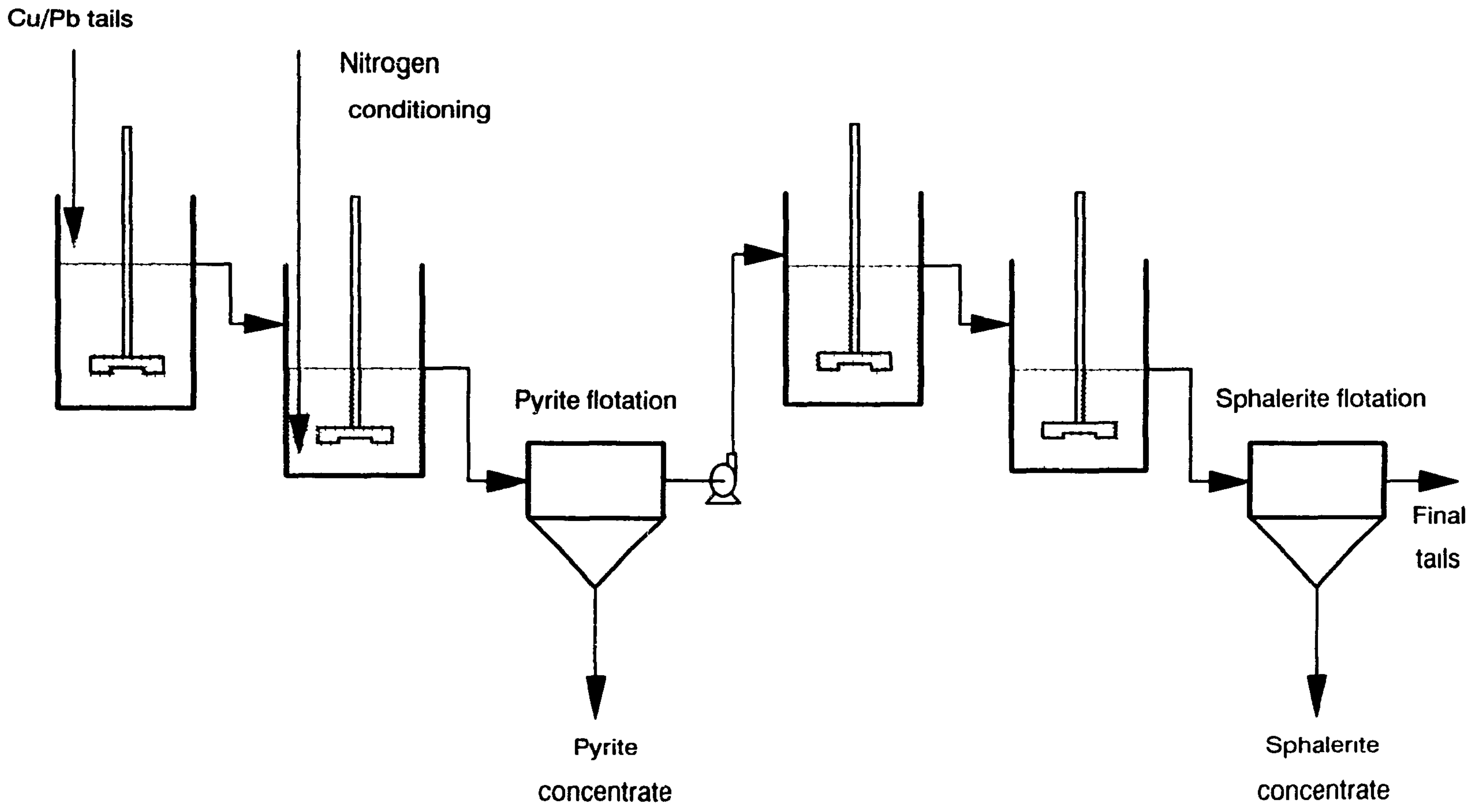


Fig 5.3 Circuit tested using mini-cells at Brunswick Mining.

TABLE 5.2. Pyrite Flotation Conditions.

Test #	Pulp Flowrate l/min.	Nominal Retention Time	Collector (g/t)
1	0.8	≃ 6 min.	No addition.
2	1.7	≃ 3 min.	No addition.
3	3.3	≃ 1.5 min.	No addition.
4	0.8	≃ 6 min.	80% X343 & 20% X350 (20)

TABLE 5.3. Sphalerite Flotation Conditions.

Option	Pulp Flowrate l/min.	$CuSO_4$ addition g/t	CaO addition
1	0.8	1100	No addition.
2	0.8	No addition.	to pH 10.5
3	0.8	1100	to pH 10.5

5.2. Kidd Creek Testwork.

Previous work at Kidd Creek had shown some promise for selective flotation of pyrite using nitrogen (2,42). Also it was found that even using low grade nitrogen (95% purity) it is still possible to obtain good selectivity of pyrite (43).

Based on the above findings, a series of tests were conducted to explore the application of nitrogen in a zinc-rich stream as well as in a primary grinding cyclone overflow at the Kidd Creek Mines

Concentrator.

Laboratory flotation testwork was performed on-site to evaluate zinc reverse flotation and chalcopyrite flotation.

Work on zinc reverse flotation was done at Kidd Creek Mines using samples from the final zinc concentrate stream. Flotation tests were also performed on samples taken from primary grinding cyclone overflow. Once again nitrogen was compared versus air.

Monitoring of pH, pulp potential and oxygen content variables was done by using the same equipment described in Chapter 5, Section 5 1.

The procedure followed for flotation of the cyclone overflow samples is summarized in Table 5.4.

Table 5.5 presents the procedure followed at Kidd Creek Mines for the zinc reverse flotation tests. Samples were taken in pairs to analyze the effect of the variables investigated.

TABLE 5.4. Test Procedure Followed at Kidd Creek Mines for Samples from Cyclone Overflow.

CONCEPT	AIR FLOTATION	N_2 FLOTATION
FLOTATION CELL:		
Tipe	Denver	Denver
Capacity, l	4.0	4.0
PRE-NITROGENATION:		
Time, min.	-----	5
N_2 flowrate, l/min.	-----	3
Impeller speed, rpm	1500	1500
PULP CONDITIONING:		
Time, min.	2	2
Collector (g/t)	R-3480A (100)	R-3480A (100)
Frother (g/t)	MIBC (20)	MIBC (20)

TABLE 5.5. Test Procedure Followed at Kidd Creek Mines for Zinc Reverse Flotation Experiments.

TEST	SAMPLE	CONDITIONS
1	Final Zn Conc. (circuit A)	Heat to 80°C, then SO_2 addition to pH = 4.5 and air flotation.
2	"	Same as 1 but using N_2.
3	"	Heat to 80°C, SO_2 addition and flotation without conditioning using N_2.
4	"	Same as 3 but using 20 min. of conditioning after SO_2 addition.
5	"	Same as 4 but using air.
6	"	Same as 5 but heating to 65°C.

5.3. Falconbridge (Strathcona mill) Testwork.

In the case of Strathcona mill ore, bench scale tests were performed on-site and at McGill University. Microflotation tests using 1 gram samples were also conducted on pyrrhotite isolated from a pyrrhotite-rich stream.

The tests completed on-site were done using a four litre Denver laboratory flotation cell and the conditions presented in Table 5.6 were followed using nitrogen as the flotation gas in one case and air in the other.

TABLE 5.6. Test Procedure Followed at Strathcona Mill.

	AIR FLOTATION	N_2 FLOTATION
FLOTATION CELL:		
Type	Denver	Denver
Capacity, l	4.0	4.0
PRE-NITROGENATION:		
Time, min.	-----	5
N_2 flowrate, l/min.	-----	3
Impeller speed, rpm	1500	1500
PULP CONDITIONING:		
Time, min.	2	2
Collector (g/t)	Xanthate (100)	Xanthate (100)
Frother (g/t)	MIBC (10)	MIBC (10)

The test procedure for bench scale tests at McGill University is described in Table 5.7. In this case a copper activated scavenger flotation step was carried out after the rougher flotation.

TABLE 5.7. Test Procedure for Bench Scale at McGill University.

	AIR FLOTATION	N_2 FLOTATION
FLOTATION CELL:		
Type	Leeds	Leeds
Capacity, l	2.5	2.5
PRE-NITROGENATION:		
Time, min.	-----	5
N_2 flowrate, l/min.	-----	3
Impeller speed, rpm	1000	1000
PULP CONDITIONING:		
Time, min.	2	2
ROUGHER FLOTATION:		
Time, min.	4	4
Collector (g/t)	Xanthate (60)	Xanthate (60)
Frother (g/t)	MIBC (10)	MIBC (10)
SCAVENGER FLOTATION:		
Time, min.	10	10
$CuSO_4$ (g/t)	30	30
Collector (g/t)	Xanthate (40)	Xanthate (40)
Frother (g/t)	MIBC (10)	MIBC (10)

Test procedure to evaluate if diethyl amine acts as a pyrrhotite depressant is summarized in Table 5.8. (This was tested to compare with nitrogen as a depressant: such amines are being considered as depressants for iron sulphides (44)).

TABLE 5.8. Test Procedure Followed at McGill for Pyrrhotite Depression Tests Using Diethyl Amine.

	Without amine pH		50 g/t amine pH	
	6.0	9.2	6.0	9.2
FLOTATION CELL:				
Type	Leeds	Leeds	Leeds	Leeds
Capacity, l	2.5	2.5	2.5	2.5
AMINE ADDITION:				
Addition rate, g/t	-----	-----	50	50
Conditioning time, min.	-----	-----	2	2
PULP CONDITIONING:				
Time, min.	2		2	
Collector (g/t)	Xanthate (100)		Xanthate (100)	
Frother (g/t)	MIBC (10)		MIBC (10)	

Concentrates were collected as a function of time for up to 13 min. The oxidation-reduction potential of the pulp was measured using a gold electrode while the oxygen content was obtained using an Orion oxygen electrode. The pH was measured using a standard pH meter electrode. In all cases, the electrodes were connected to volt/pH meters (Hanna Instruments model HI8424).

A Hallimond tube was used for microflotation tests on pyrrhotite obtained from the non-magnetic rougher flotation tails from Strathcona mill. The sample was prepared by washing it three times with deoxygenated hot water at pH ≈2 then with acetone (to remove collector coating) followed by drying at room temperature. Tests with and without collector (amyl xanthate) were run at pH 6.0, 9.0, and 10.5 using air as the flotation gas. Concentrate was collected for a total of 3 minutes in all cases.

6. RESULTS.

6.1. Brunswick Mining and Smelting.

These tests were done to follow up the work of Martin (40) on nitrogen flotation of pyrite ahead of sphalerite flotation, in particular by conducting continuous in-plant trials.

Figures 6.1 and 6.2 show the influence of nitrogen purity on pyrite flotation in the laboratory. The highest iron (and weight) recoveries were obtained with the use of pure nitrogen. It is also seen that using a mixture 95%-5% nitrogen-oxygen the recoveries are reduced but by a margin probably within experimental error On the other hand, the use of a mixture 90%-10% nitrogen-oxygen generated recoveries which were much lower. In this particular case, for example, the iron recovery was around 43% less than when using 100% nitrogen.

These results reconfirm those of Martin, and also reveal that the purity of nitrogen is important, mixtures of 95% nitrogen or greater are required for good results in the pyrite flotation step.

The results of the minicell, continuous tests are summarized in Tables 6.1 and 6.2.

During the minicell pilot tests, some flotation of pyrite without further collector addition was initially observed and pursued in

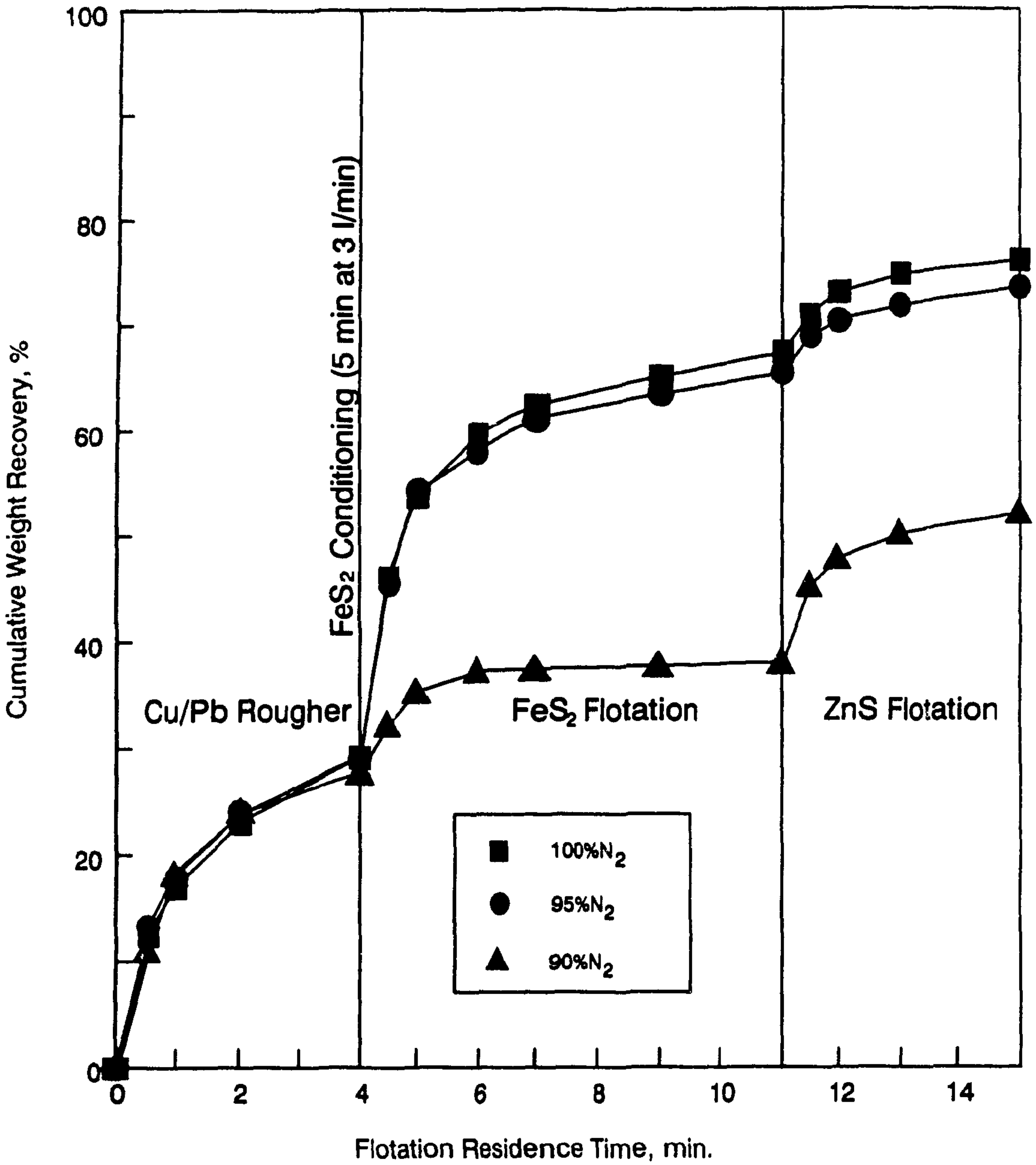


Fig 6.1 Laboratory flotation test on B.M.S. ore. Influence of nitrogen composition.

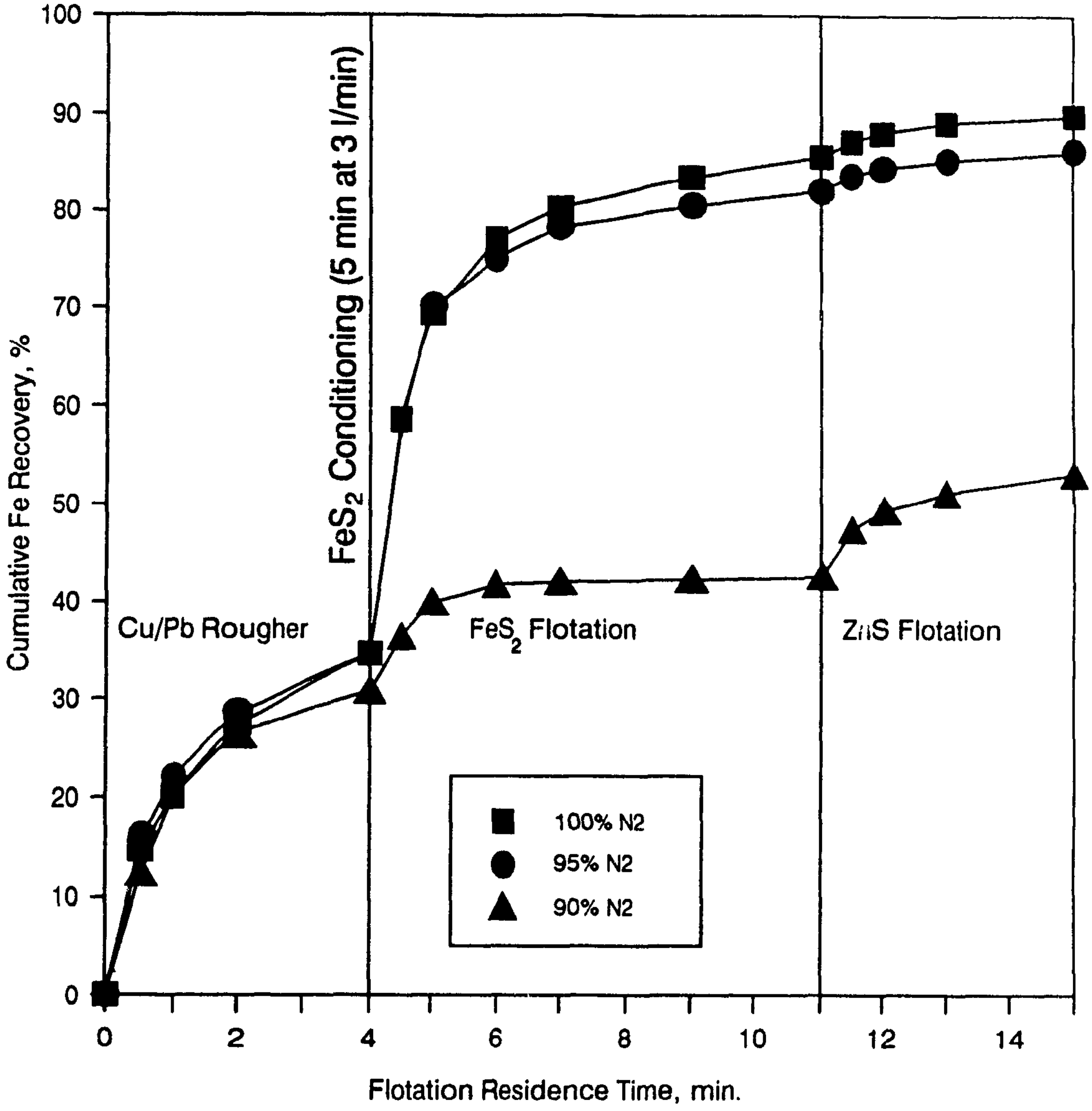


Fig 6.2 Laboratory flotation test on B.M.S. ore. Influence of nitrogen composition.

tests 1, 2, and 3 (Table 6.1). However, the results showed only around 3% Fe recovery.

TABLE 6.1. Pilot Flotation Tests of Pyrite Using Minicells.

Test #	%Fe	%Zn	%Fe Rec.	%Zn Rec.
1	33.0	9.2	3.0	2.3
2	32.7	12.1	2.8	2.8
3	32.5	12.1	3.3	3.7
4	38.6	6.0	37.3	16.0

By using a small amount of collector as indicated in Test #4(Table 5.2), significant selective pyrite flotation was observed. Figure 6.3 presents the flowsheet of the test run which includes sphalerite flotation. These results are comparable to those previously obtained in the laboratory tests (Figure 6.4).

As indicated in Table 5.3, three options were tested in the sphalerite flotation step: results are shown in Table 6.2.

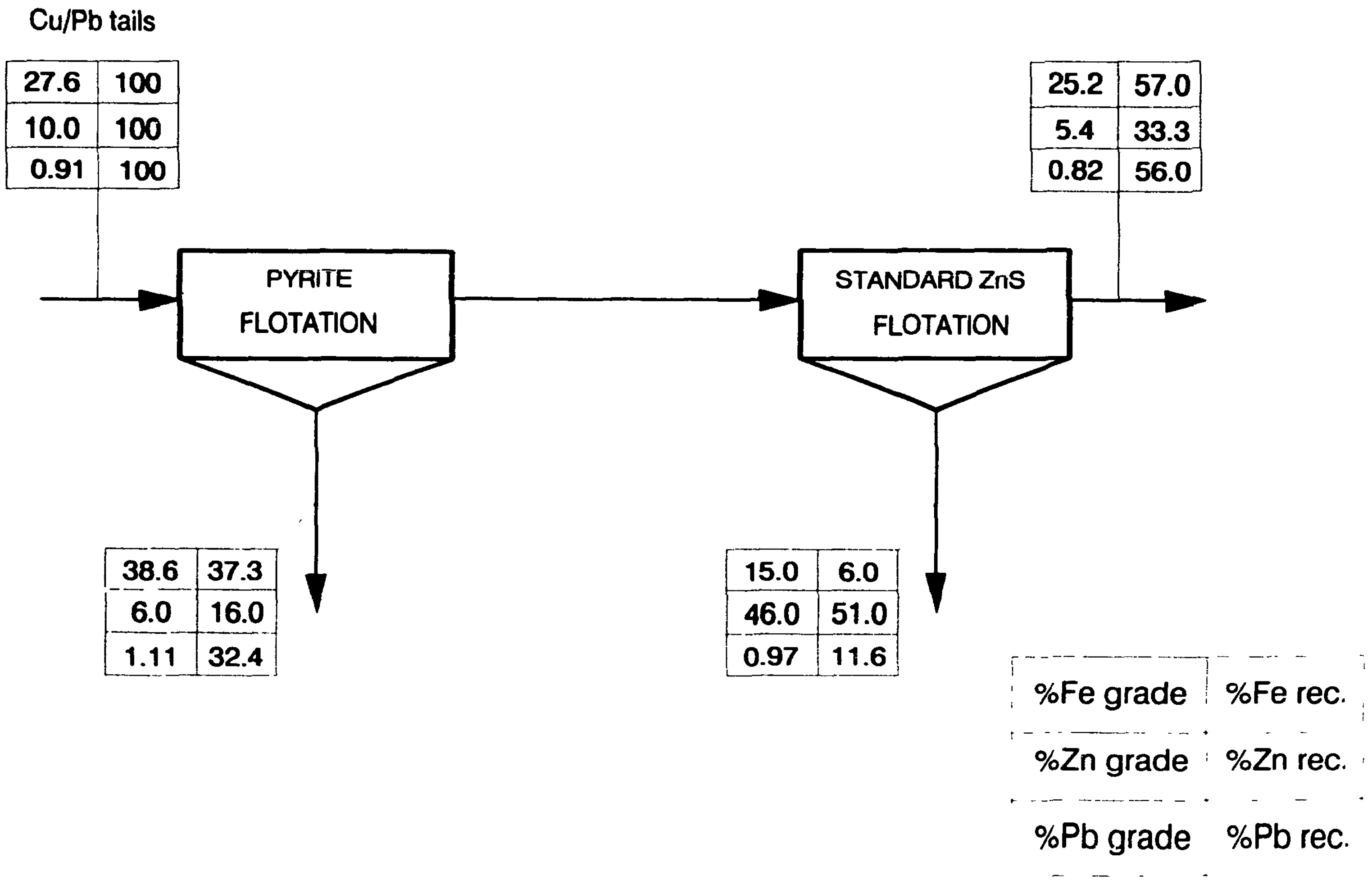


Fig 6.3 Nitrogen flotation of pyrite followed by standard sphalerite flotation (option 3, Table 6.2).

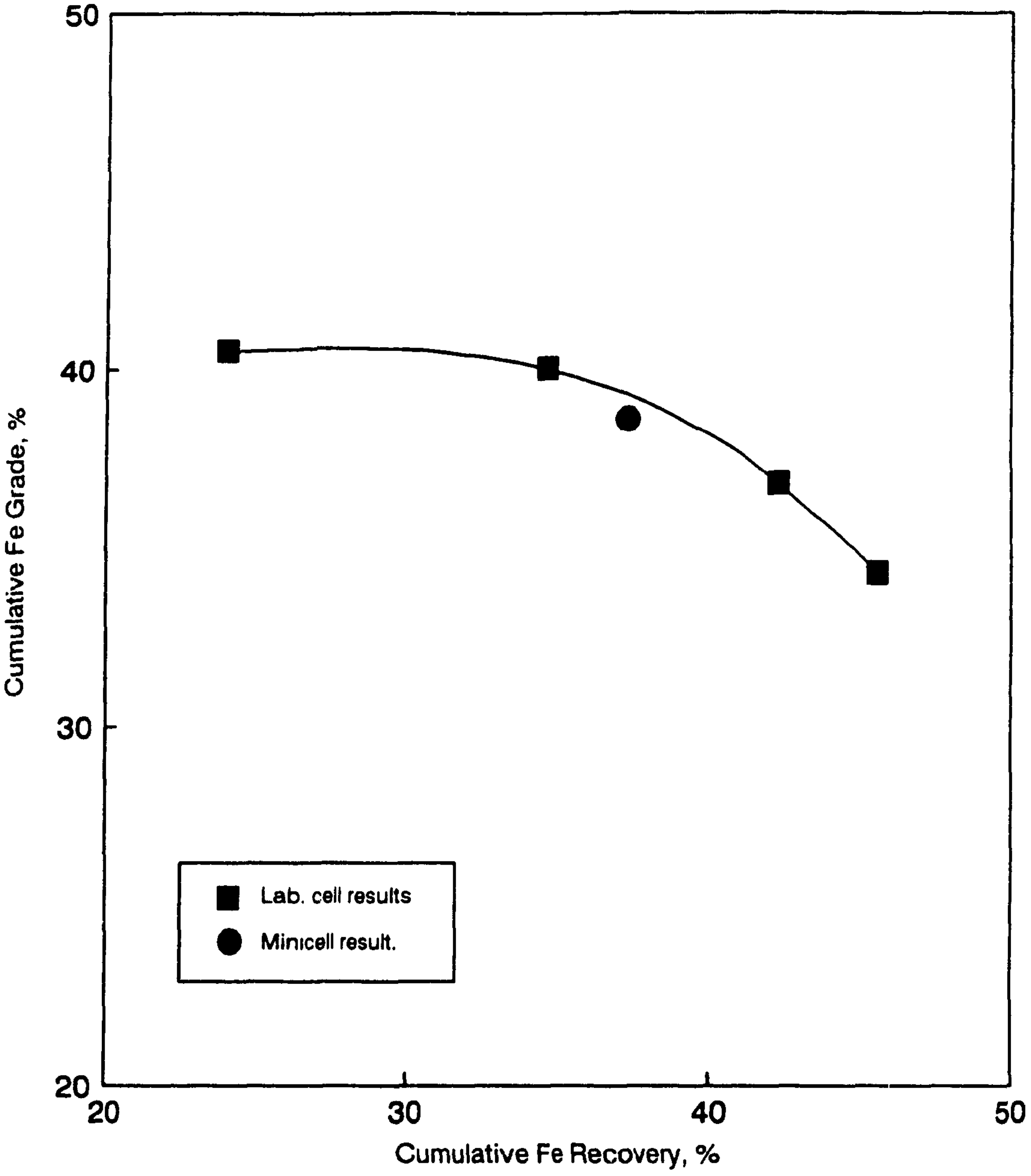


Fig 6.4 Results using laboratory (batch) and minicell (continuous) cells.

TABLE 6.2. Sphalerite Flotation Results

	Option 1	Option 2	Option 3
%Zn grade	41.0	40.5	46.0
%Zn rec.	69.0	44.0	51 0
%Fe grade	18.0	16.5	15.0
%Fe rec.	11.0	6.5	6 0

Option 1 suggests that pH adjustment to 10.5 might not be necessary for a successful sphalerite flotation. Option 2 generated a concentrate with a comparable zinc grade to Option 1 but a lower recovery. The observation that copper activation in alkaline medium is not always essential for sphalerite flotation was noted before in laboratory work (45); those results are confirmed here. Similar to the observation in the laboratory, flotation rate is lower without copper activation. The standard procedure for sphalerite flotation (i.e. Option 3) gave the highest zinc grades with recoveries intermediate to the other two options.

Electrochemical variables such as pulp potential, pH and oxygen content for tests shown in Table 5.2 are presented in Table 6.3.

TABLE 6.3. Pulp Conditions During Pyrite Flotation.

Test #	pH	Oxygen, ppm	Pulp Potential (Au vs. Ag/AgCl)
1	8.7 to 9	0.2 to 0.5	-40 to -35
2	8.7 to 9	0.2 to 0.5	-30 to -25
3	8.7 to 9	0.2 to 0.5	-10 to 0
4	8.7 to 9	0.2 to 0.5	-52 to -45

The pulp conditions were quite stable and the only parameter that changed with the pulp flowrate was the pulp potential going from more negative (reducing conditions) at the lowest flowrate to close to 0 mV at the highest flowrate The difference in pulp potential between tests #1 and #4 is possibly due to the use of collector in test #4 which creates more reducing conditions

6.2. The Kidd Creek Concentrator.

6.2.1. Zinc Reverse Flotation Testwork.

A sphalerite-rich stream at the Kidd Creek Mines Concentrator was studied in order to explore any possible benefit of using nitrogen (in place of air) in reverse flotation.

The separation in a zinc reverse flotation process is achieved by heating and conditioning with SO_2. These conditions cause depression of sphalerite and flotation of pyrite but at a relatively high cost.

The principal purpose of zinc reverse flotation is to separate pyrite from a sphalerite-rich stream. The pyrite reports to the float product while the sphalerite reports to the non-float stream as final concentrate.

It is known that the principal attraction of the zinc reverse flotation process is that the ratio of the pyrite rate constant to the sphalerite rate constant is much higher than the equivalent inverse ratio for conventional sphalerite cleaning (46)

The variables studied included temperature, SO_2 addition and conditioning time. Samples were taken in pairs to analyze the effect of the variables investigated. Table 5.5 shows the variables studied in the zinc reverse flotation testwork.

Figures 6.5 and 6.6 show that larger amounts of pyrite (and chalcopyrite) are recovered in the float product per unit of yield using nitrogen. This leads to higher grade of sphalerite in the non-float product (Figures 6.7 and 6.8).

Effect of conditioning time on the flotation of pyrite and chalcopyrite using nitrogen is shown in Figures 6.9 and 6 10 Conditioning time is beneficial to recover larger amounts of pyrite and chalcopyrite. This leads to improved zinc grade-recovery in the non-float product (Figures 6.11 and 6.12) which clearly has an

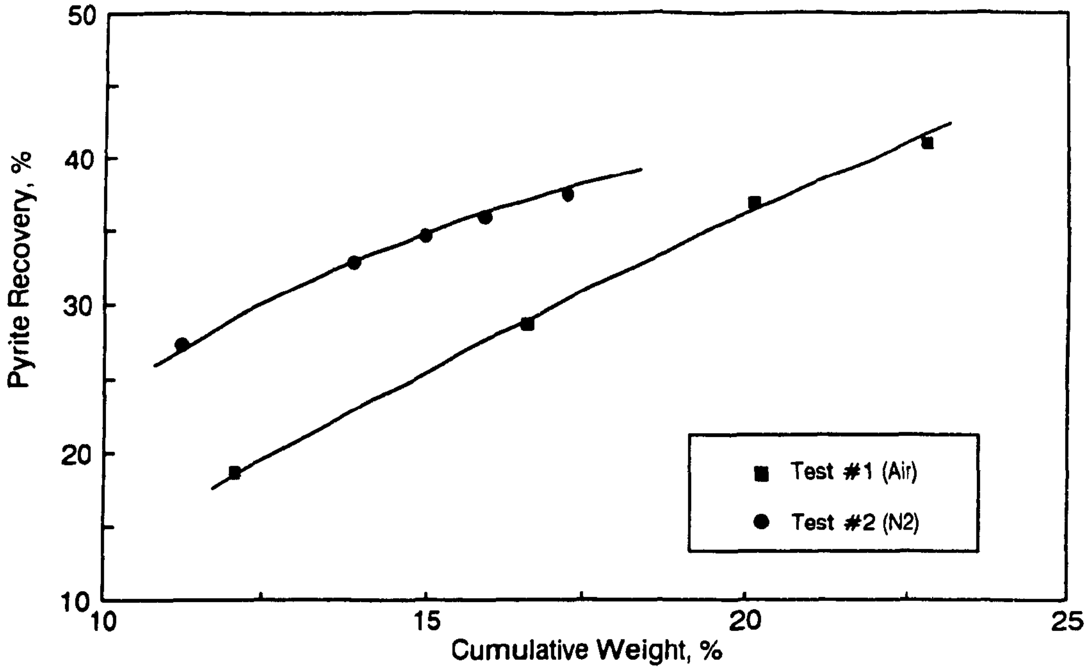


Fig 6.5 Recovery-Yield curves for pyrite in the float product (zinc reverse flotation tests). Effect of flotation gas.

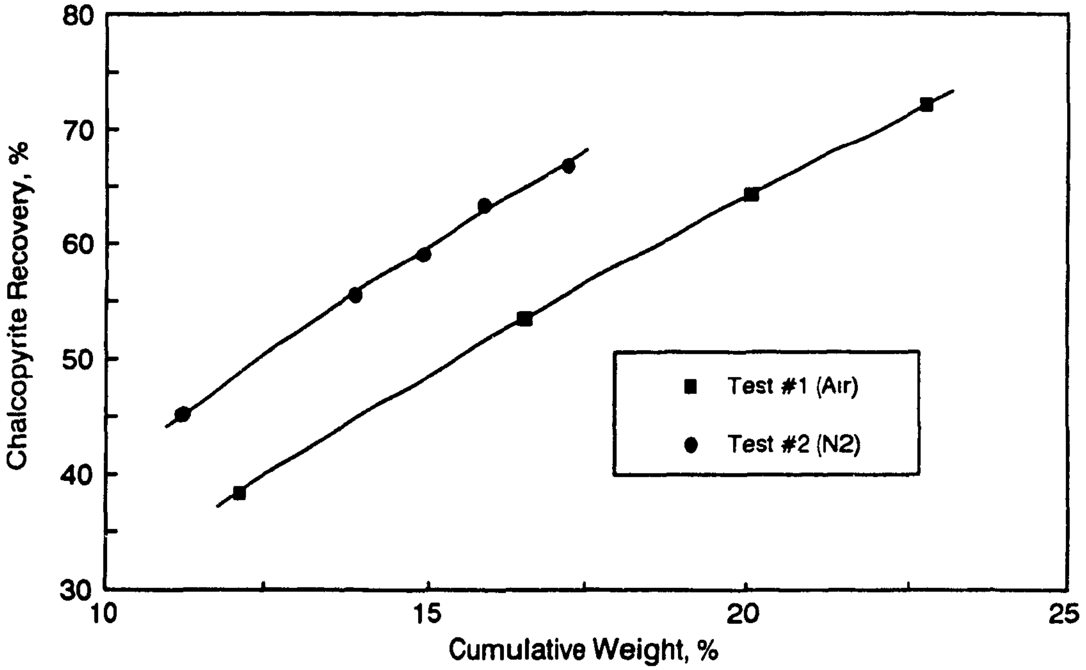


Fig 6.6 Recovery-Yield curves for chalcopyrite in the float product (Zn reverse flotation tests). Effect of flotation gas.

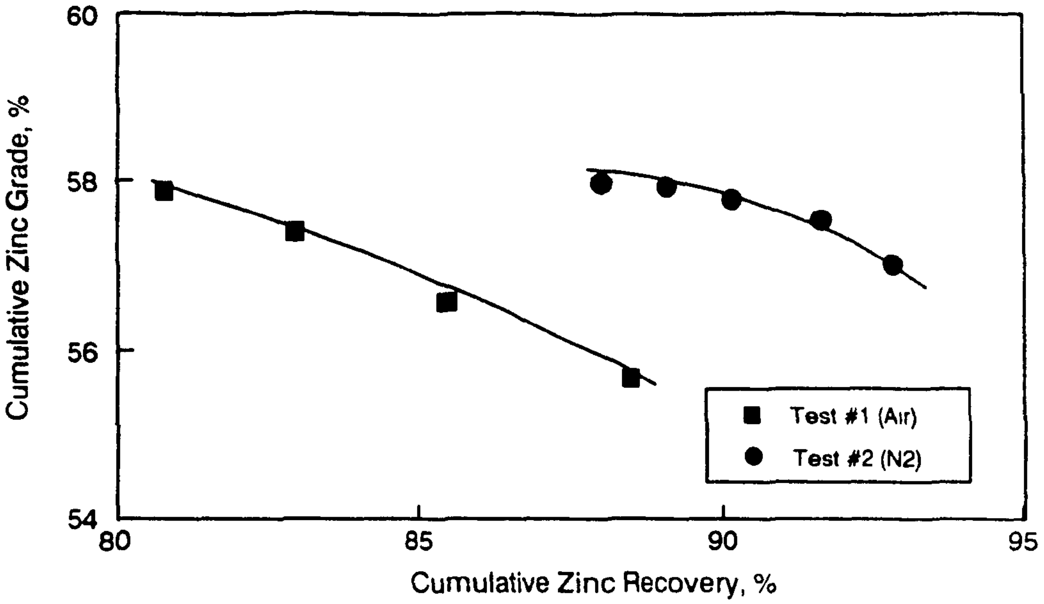


Fig 6.7 Grade-Recovery curves for reverse zinc flotation
(T to 80 'C, then SO_2addition).

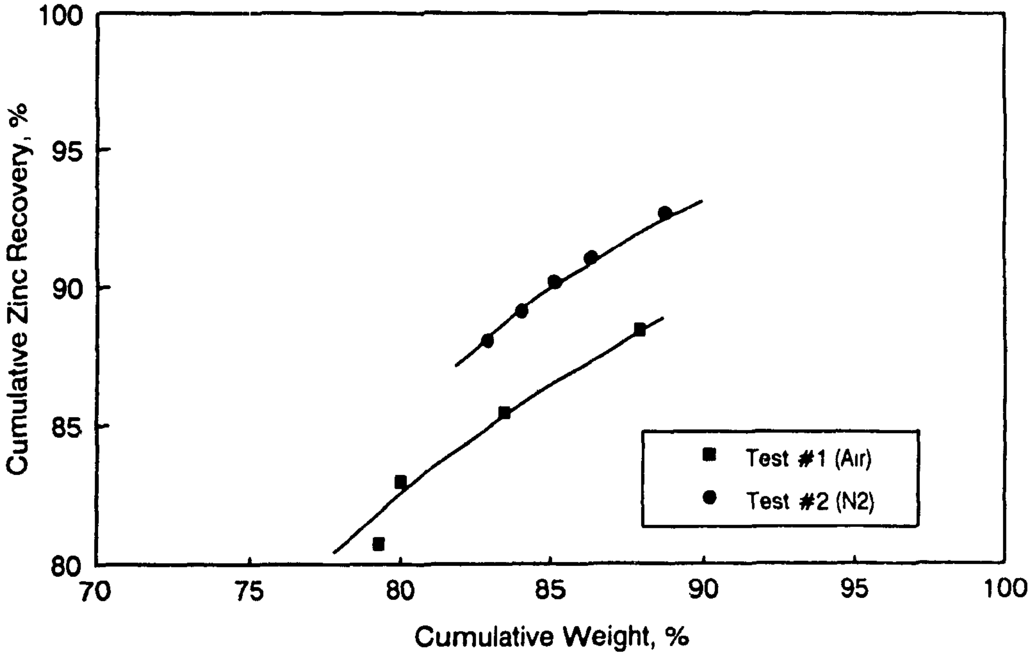


Fig 6.8 Recovery-Yield curves for reverse zinc flotation
(T to 80 'C, then SO_2addition).

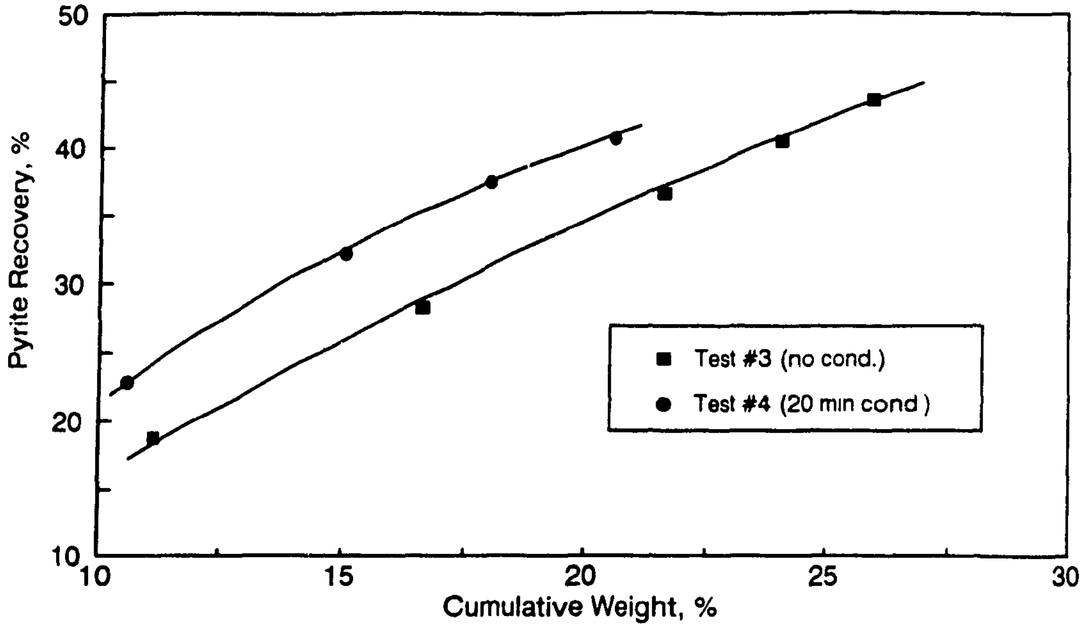


Fig 6.9 Recovery-Yield curves for pyrite in the float product (zinc reverse flotation tests). Effect of conditioning time.

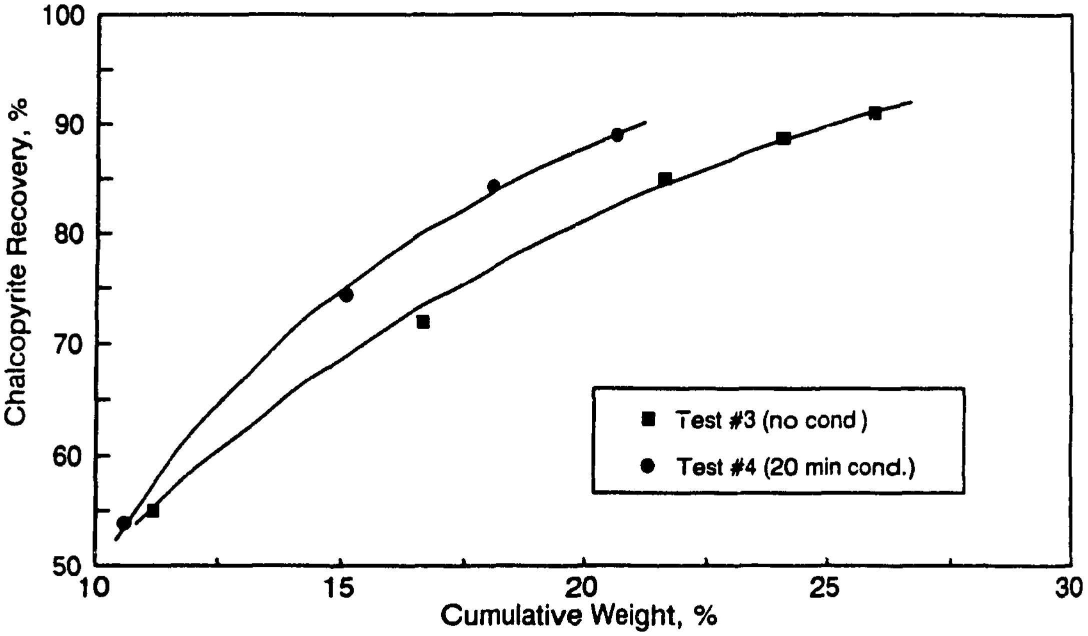


Fig 6.10 Recovery-Yield curves for chalcopyrite in the float product (zinc reverse flotation tests). Effect of conditioning time.

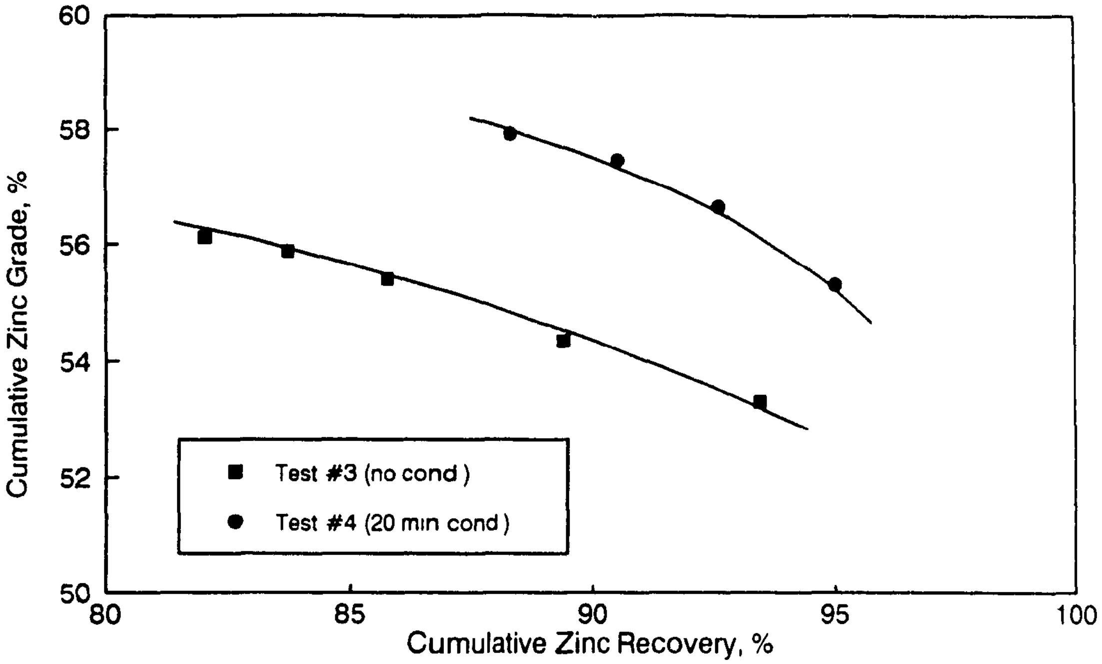


Fig 6.11 Grade-Recovery curves for zinc reverse flotation (T to 80 'C, SO_2 addition and nitrogen flotation).

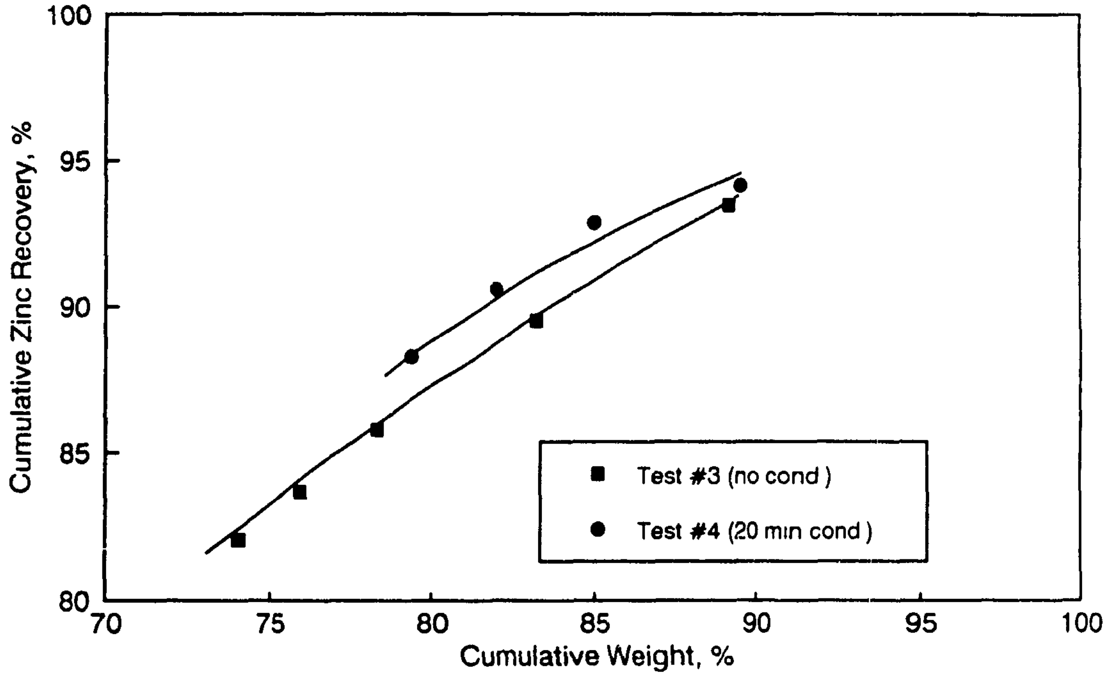


Fig 6.12 Recovery-Yield curves for reverse zinc flotation (T to 80 'C, SO_2 addition and nitrogen flotation).

economic impact.

Effect of temperature on zinc reverse flotation is shown in Figures 6.13 and 6.14. These figures indicate that change in temperature over the range 65-80 °C does not seem to have a significant effect on the zinc reverse flotation operation.

Results of these preliminary investigations indicate that, compared to air, flotation using nitrogen leads to greater removal of pyrite and chalcopyrite from the sphalerite concentrate. The increased removal of pyrite and chalcopyrite lowers the iron content from 10-11 to 6-7 percent iron.

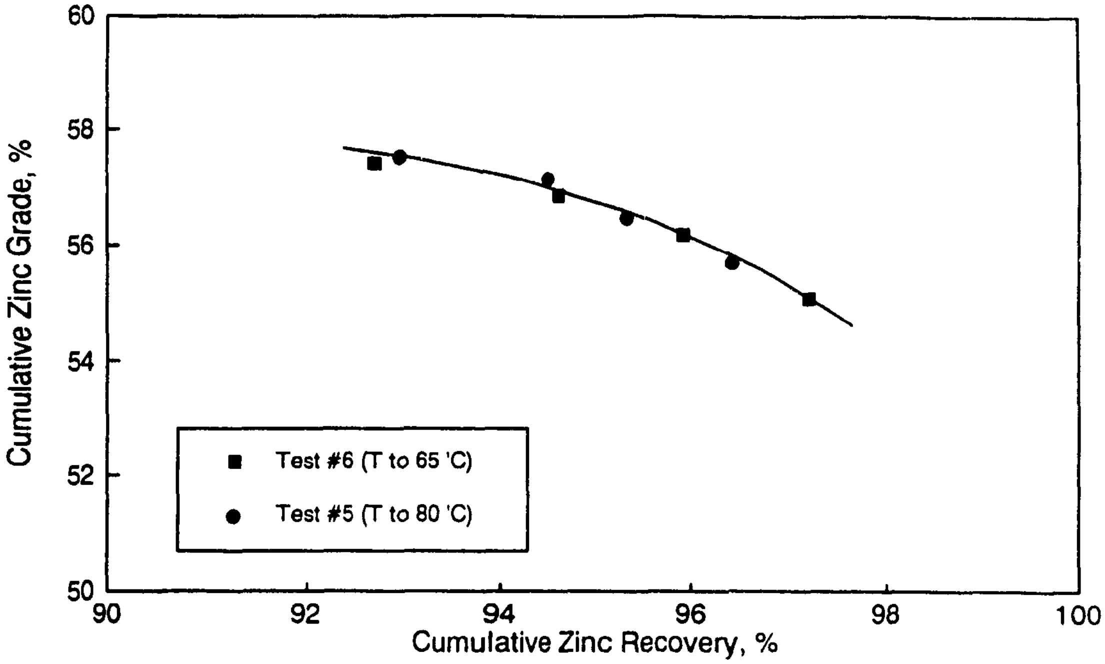


Fig 6.13 Grade-Recovery curves for reverse zinc flotation (20 min conditioning and flotation with air).

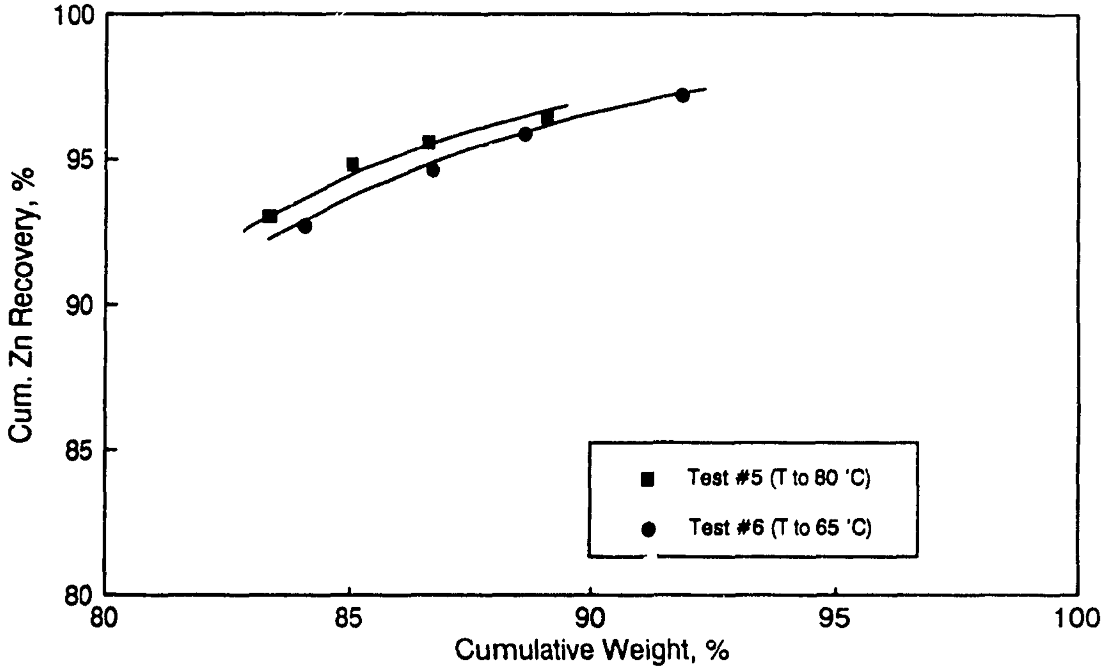


Fig 6.14 Recovery-Yield curves for reverse zinc flotation (20 min conditioning and flotation with air).

6.2.2. Chalcopyrite-Sphalerite Separation.

Samples from primary grinding cyclone overflow were tested in order to compare the results obtained using nitrogen vs. air as the flotation gas.

The oxidation of complex sulphide minerals leads to the formation of soluble salts. Among the metallic ions that originate from these soluble salts, copper ions in particular can lead to have an unfavorable influence on accidental activation of sphalerite. To solve this problem it is necessary either to eliminate copper ions from the flotation pulp or to deactivate the copper-activated film on the sphalerite surface.

Since mild steel is used as grinding media in many cases, the oxygen contained in the pulp is consumed by the media during grinding which retards the oxidation of the sulphide minerals. Subsequently oxygen is introduced into the pulp, however, either deliberately through an aeration stage (not used at Kidd Creek) or via pumping or by introducing fresh water to the pulp. Some degree of sulphide mineral oxidation must then be expected.

Since oxygen is consumed during the grinding operation, if the sample for flotation is taken directly from the cyclone overflow it will be in its least oxidized state. In particular, the quantity of dissolved copper ions from chalcopyrite oxidation will be minimum and accidental sphalerite activation less likely (47) If nitrogen gas is used, oxidation is further hindered, and sphalerite flotation should be kept low.

Figure 6.15 confirms that sphalerite is less floatable when nitrogen is used as the flotation gas. It is also observed, in these particular experiments, that chalcopyrite recovery is not affected when nitrogen is used instead of air. This may suggest a promising method of improved selectivity in the flotation of chalcopyrite Pulp potential of the sample was found to be ≅ -10 mV referred to Ag/AgCl (212.5 mV vs. S.H.E.).

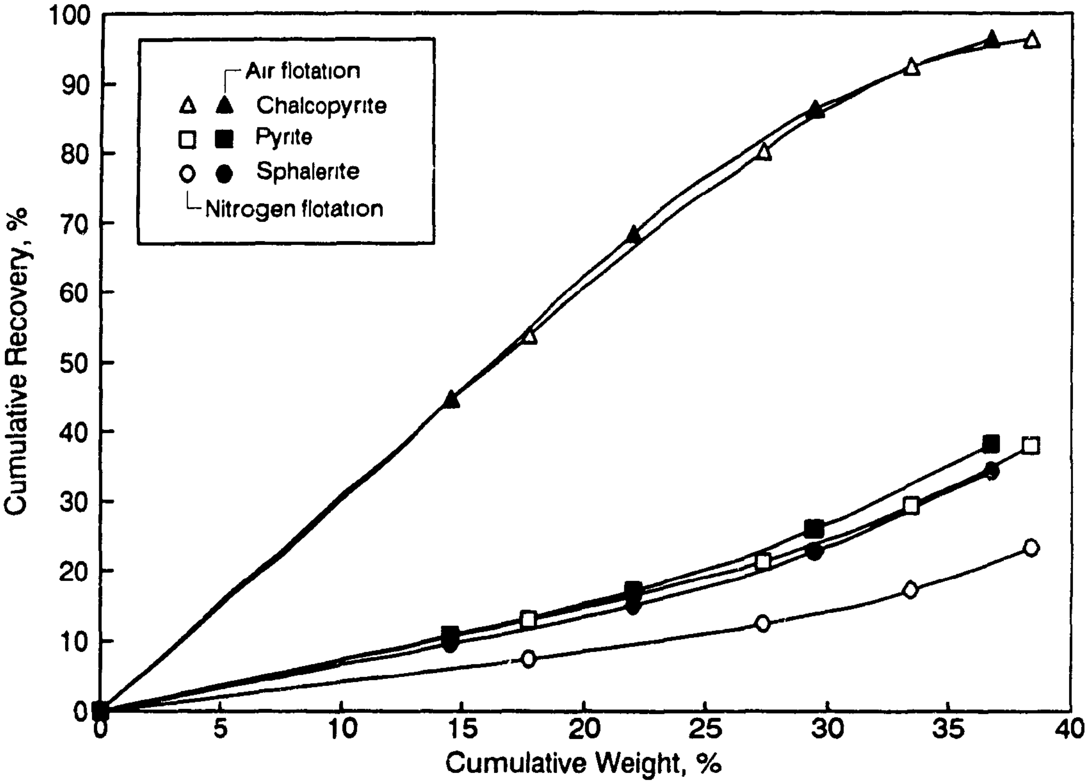


Fig 6.15 Cu/Zn separation at Kidd Creek Mines.

Samples from cyclone overflow primary grinding.

6.3. Falconbridge Limited (Strathcona mill).

6.3.1. On-site Bench Scale Flotation Tests.

Four streams were tested using nitrogen and air at Strathcona Mill Primary Rougher Feed (PRF), Magnetic Flotation Feed (MFF), Magnetic Flotation Concentrate (MFC), and Non-magnetic Flotation Concentrate (NMFC).

Only in the case of the PRF were large differences between flotation using nitrogen and air indicated. As is shown in Figure 6.16, the pentlandite grade at 70% pentlandite recovery was up to three times higher with nitrogen than with air. However, the recovery of chalcopyrite (another economic mineral) in flotation with nitrogen was below 10%.

The recovery-yield curve presented in Figure 6.17 shows excellent separation of pentlandite from chalcopyrite and pyrrhotite when nitrogen is used.

A program of additional tests on samples of the PRF stream was designed to repeat these tests and, as additional objectives, to test collectorless flotation and to monitor the electrochemical variables pH, pulp potential and oxygen content. Table 6.5 shows the characteristics of the tests completed in this additional program.

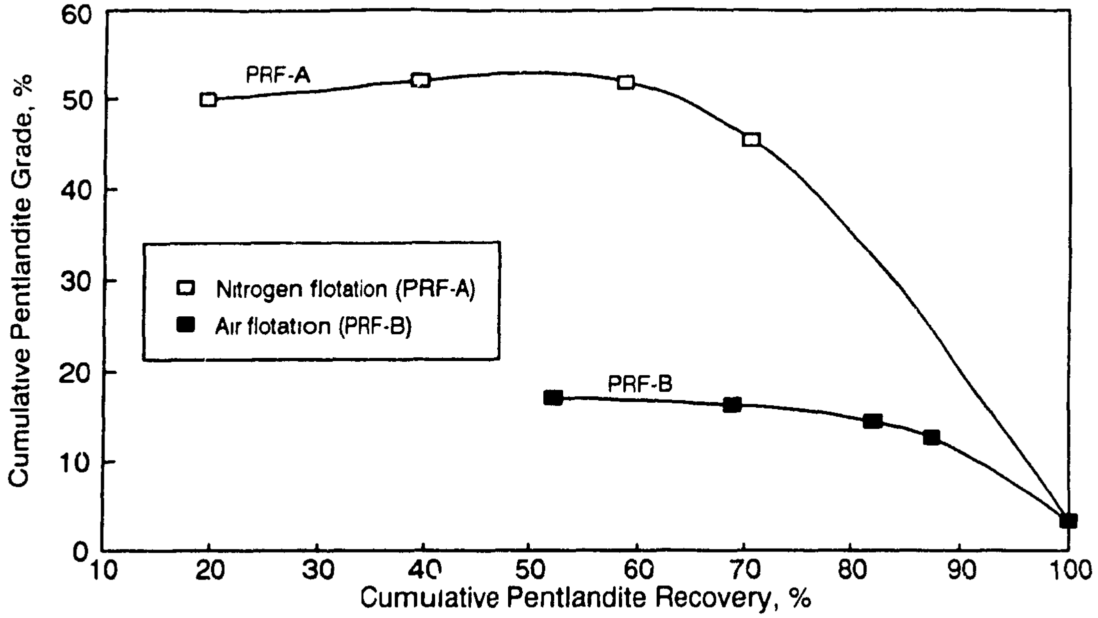


Fig 6.16 Cumulative Grade-Recovery curves for Primary Rougher Feed (PRF) at Strathcona mill.

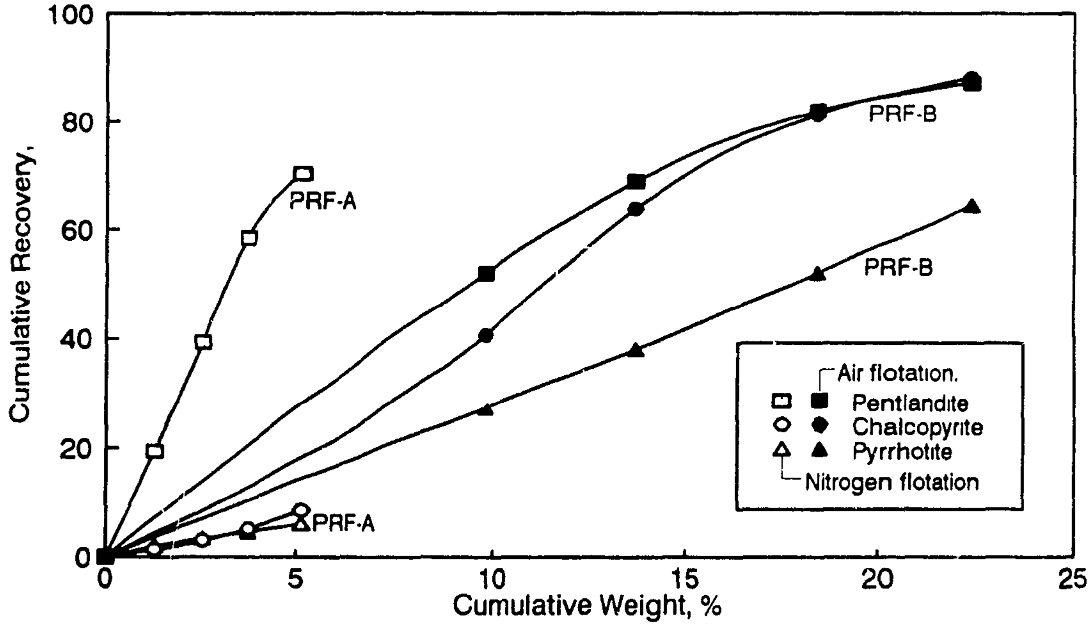


Fig 6.17 Recovery-Yield curves for Primary Rougher Feed (PRF) at Strathcona mill.

Preliminary work was also initiated on design of a flotation column for gas recycle, which would greatly influence the economics of using nitrogen.

TABLE 6.4. Characteristics of Laboratory Flotation Tests on PRF

TEST	CONDITIONS
PRF-01	Air flotation with collector
PRF-02	Nitrogen flotation with collector
PRF-03	Air flotation with collector.
PRF-04	Nitrogen flotation with collector
PRF-05	Air flotation with collector
PRF-06	Nitrogen flotation with collector
PRF-07	Collectorless nitrogen flotation followed by nitrogen flotation with collector.
PRF-08	Air flotation with collector
PRF-09	Nitrogen flotation followed by air flotation (both with collector).
PRF-10	Collectorless air flotation followed by air flotation with collector.

Tests PRF-1 and PRF-2 showed higher grades with nitrogen than with air (Figure 6.18). This confirms the results shown in Figure 6.16 and 6.17. However, grades were much lower than before. Figure 6.19 shows that pentlandite is separated from pyrrhotite but, contrary to the previous tests, chalcopyrite is also separated from pyrrhotite and floated at the same rate as pentlandite. Absolute recoveries were lower possibly due to a lower speed used in this test (1200 rpm instead of the normal 1500 rpm used in all others). As observed in Figure 6.20, the variation of pulp potential during the flotation indicated no effect of nitrogen conditioning on the ORP while the addition of collector decreased it. In nitrogen flotation, the pulp potential remained constant and, in air flotation, the pulp potential increased significantly with time.

Similar results were obtained in tests PRF-03 and PRF-04. Figure 6.21 shows again higher grades obtained with nitrogen than with air. The results of Figure 6.22 show that the rejection of pyrrhotite and the recovery of pentlandite were more effective with nitrogen than with air. No appreciable differences between nitrogen and air were observed in the flotation of chalcopyrite. Pulp potential variations during these tests presented the same trends described for runs PRF-01 and PRF-02 (Figure 6.23).

Tests PRF 05 and PRF-06 were completed using air and nitrogen respectively and tests PRF-07 was performed to investigate the applicability of collectorless nitrogen flotation. Tailings from run PRF-07 were conditioned with collector and subsequently floated using nitrogen. Grade-recovery curves (Figure 6.24) show that, with nitrogen, flotation concentrate grades are much higher than with air. Collectorless nitrogen flotation concentrate grades were also higher than the air flotation concentrate grades but were not as high as those with nitrogen flotation plus collector. Figure 6.25 shows that pyrrhotite was rejected more effectively when the flotation was done

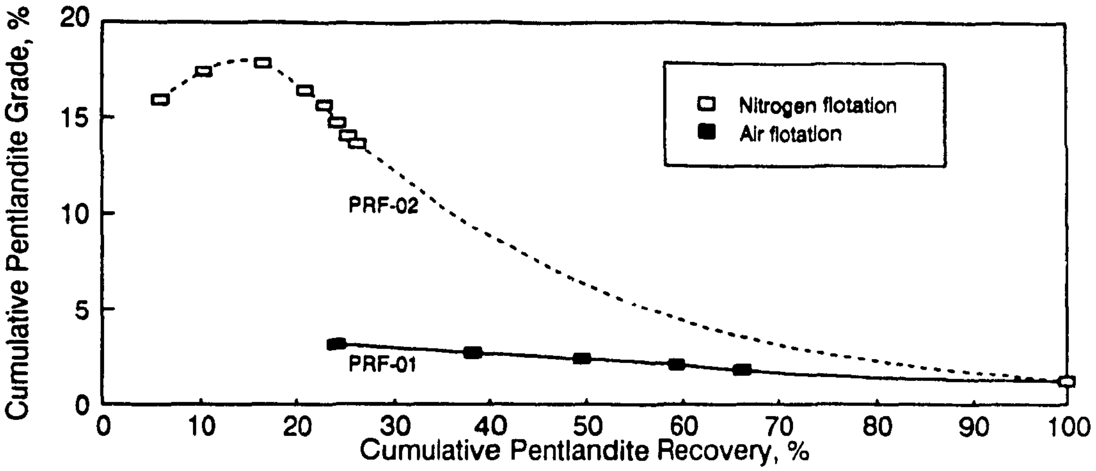


Fig 6.18 Cumulative Grade-Recovery curves for tests PRF-01 and PRF-02.

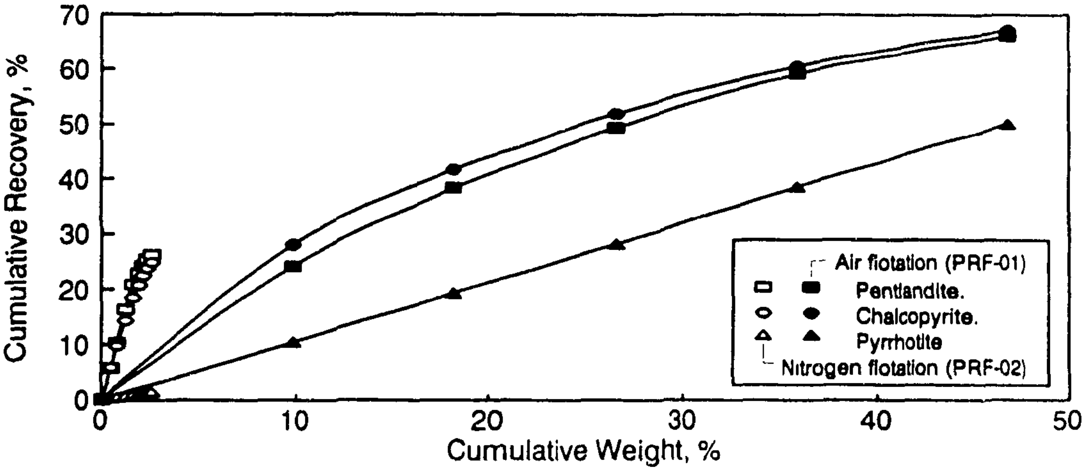


Fig 6.19 Cumulative Recovery-Yield curves for tests PRF-01 and PRF-02.

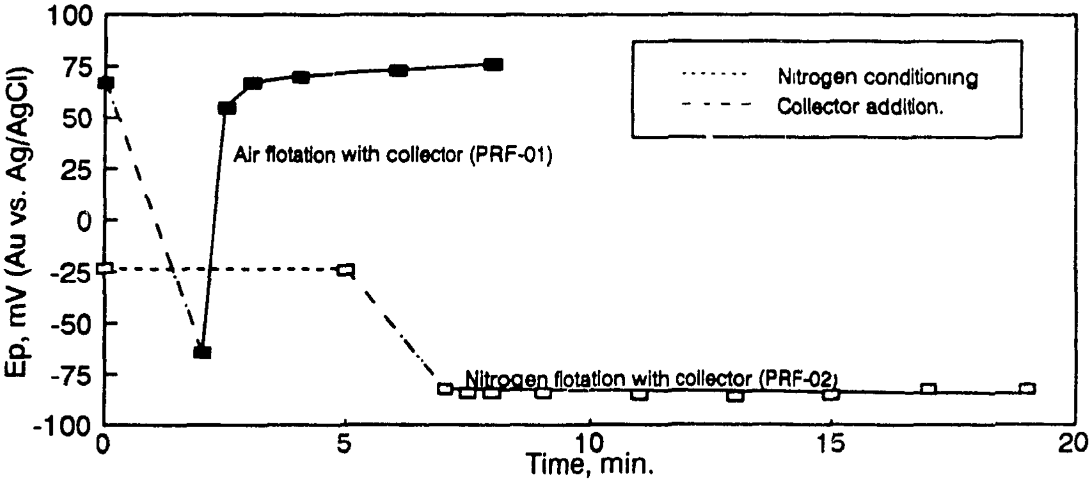


Fig 6.20 Pulp potential variations during tests PRF-01 and PRF-02.

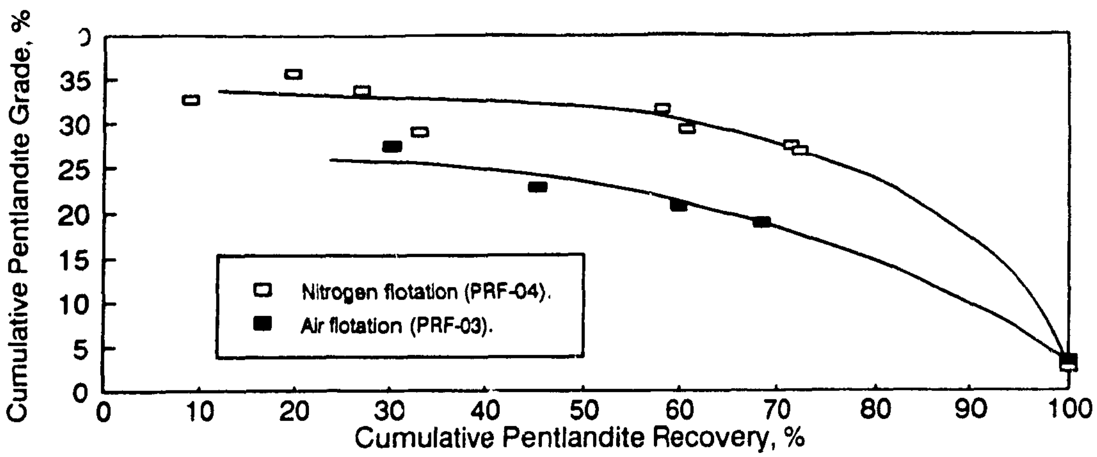


Fig 6.21 Cumulative Grade-Recovery curves for tests PRF-03 and PRF-04.

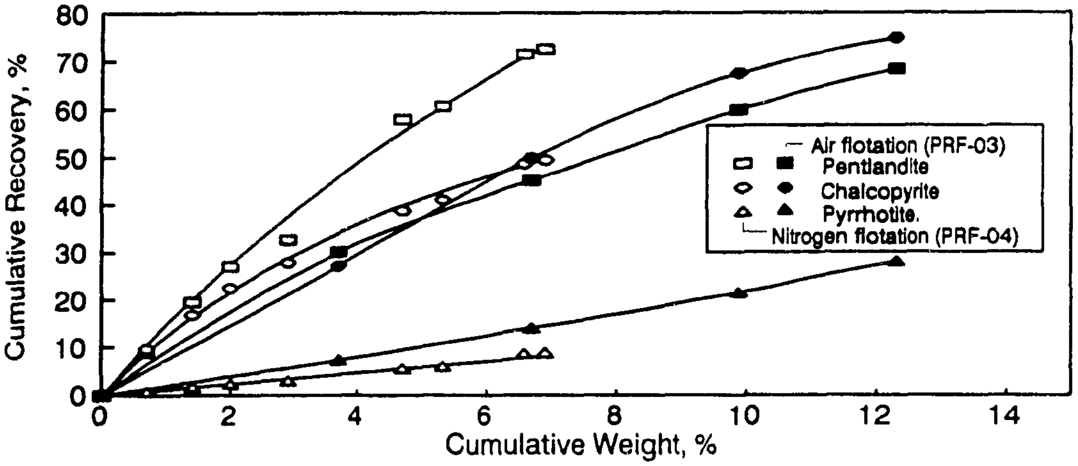


Fig 6.22 Cumulative Recovery-Yield curves for tests PRF-03 and PRF-04.

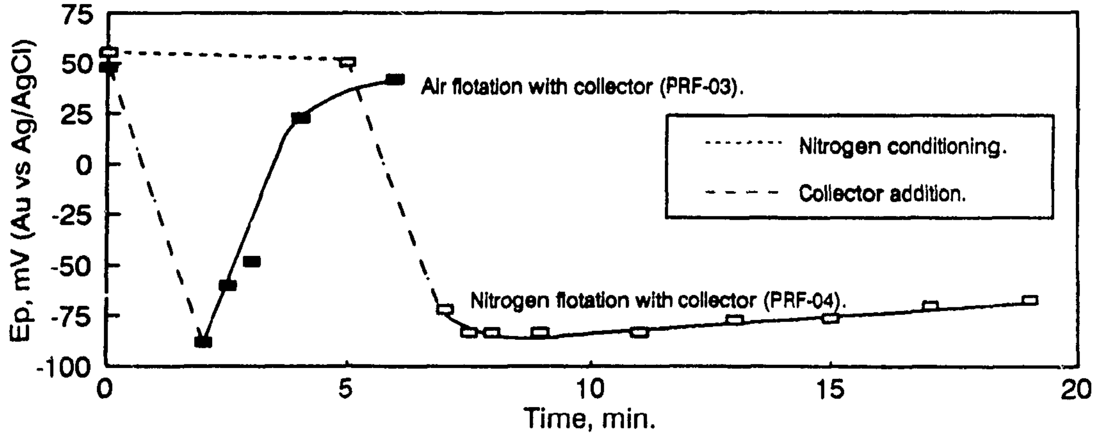


Fig 6.23 Pulp potential variations during tests PRF-03 and PRF-04.

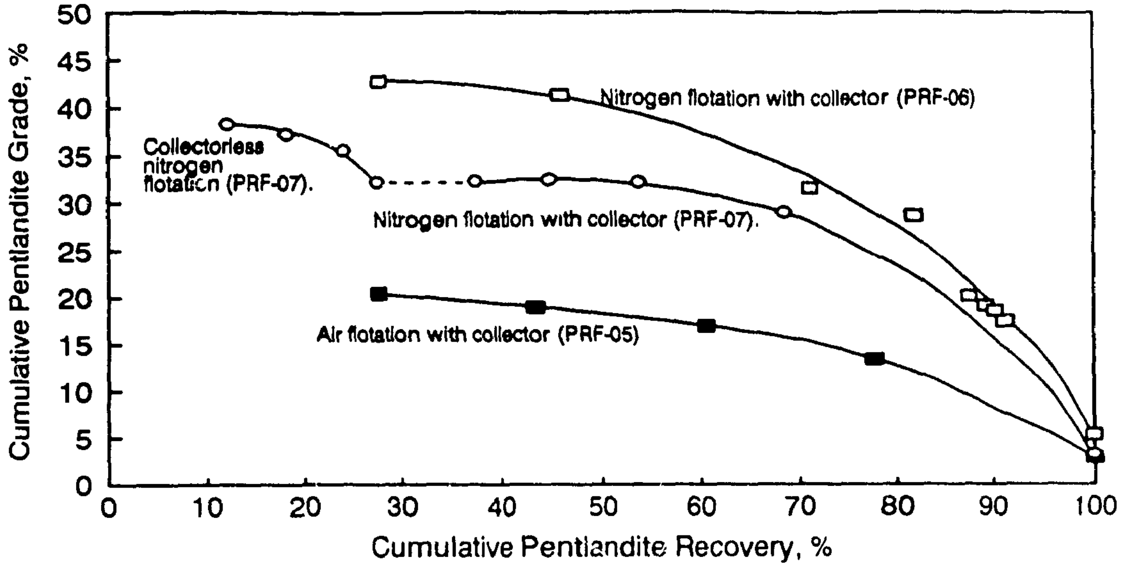


Fig 6.24 Cumulative Grade-Recovery curves for tests PRF-05, PRF-06, and PRF-07.

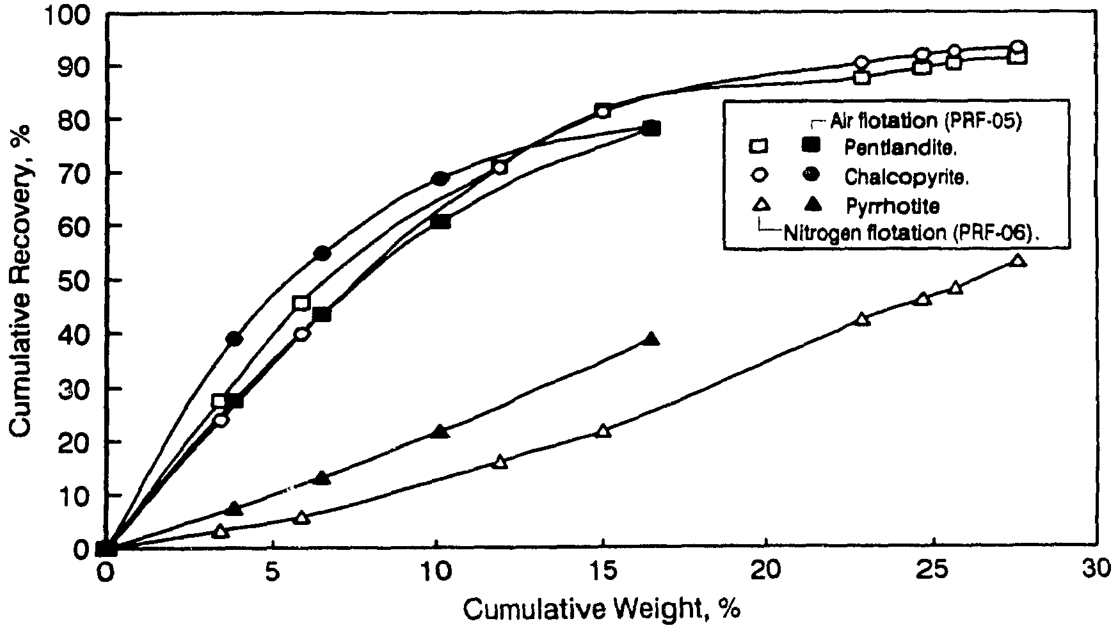


Fig 6.25 Cumulative Recovery-Yield curves for tests PRF-05 and PRF-06.

using nitrogen and pentlandite and chalcopyrite were floated at about the same rate in both air and nitrogen flotation. Pyrrhotite rejection, as well as pentlandite and chalcopyrite recovery, were also more effective in collectorless nitrogen flotation than in air flotation (Figure 6.26). Pulp potential variations were consistent with the variations observed in previous tests as shown in Figure 6.27.

Grade-recovery curves for tests PRF-08, PRF-09 and PRF-10 show once again that nitrogen flotation resulted in concentrates with higher nickel content than air flotation or collectorless air flotation (Figure 6.28). The grade-recovery curves of subsequent air flotation of the tailings from nitrogen flotation and from collectorless air flotation, approximated that of the air flotation. Figure 6.29 shows that nitrogen flotation produced concentrates with higher pentlandite and lower chalcopyrite and pyrrhotite contents than air flotation. As noted in Figure 6.28, after the nitrogen was switched to air, the curves from Figure 6.29 approached those obtained in the air flotation. Pyrrhotite flotation was slightly higher when collectorless air flotation was used as compared to air flotation (Figure 6.30). Chalcopyrite was removed at a higher rate with collectorless air flotation, but pentlandite presented the opposite trend. Variations in pulp potential were consistent again with previous tests except that ORP during nitrogen conditioning was much lower than before as is shown in Figure 6.31.

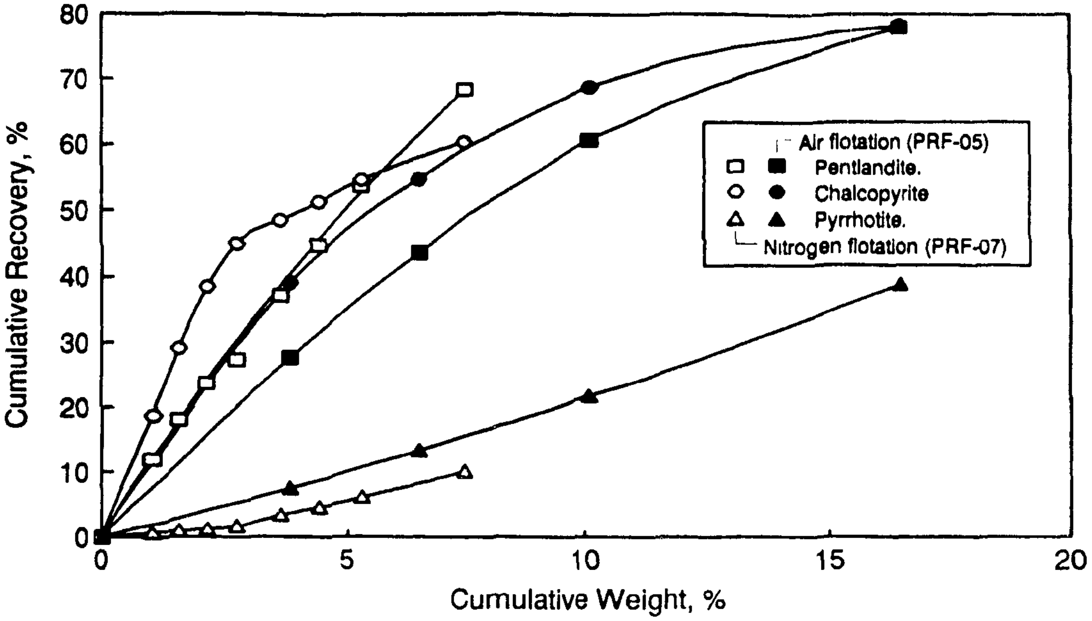


Fig 6.26 Cumulative Recovery-Yield curves for tests PRF-05 and PRF-07.

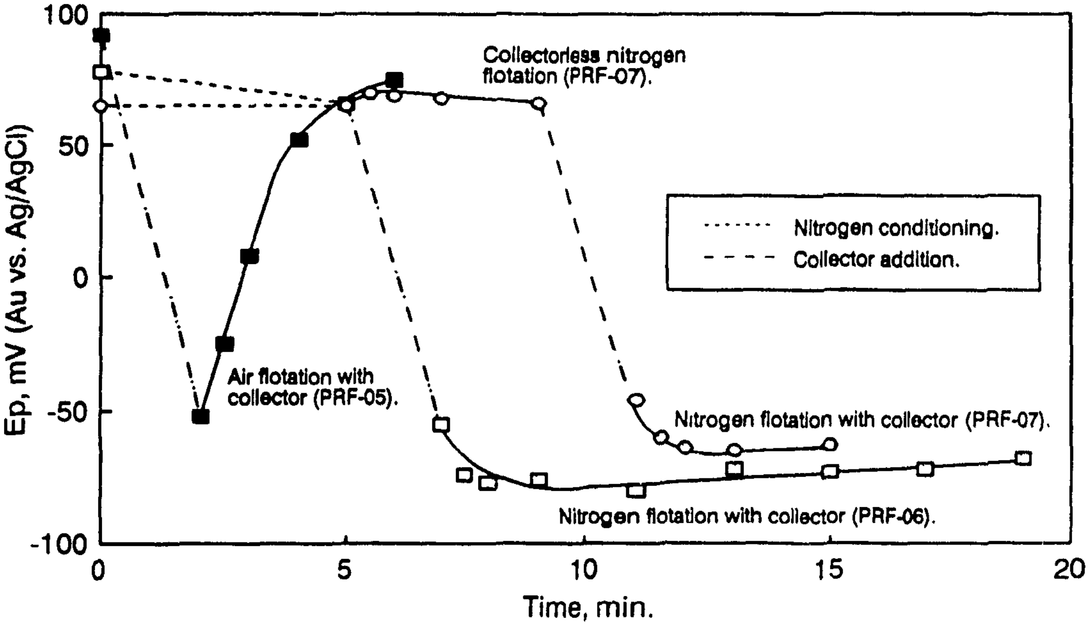


Fig 6.27 Pulp potential variations during tests PRF-05, PRF-06, and PRF-07.

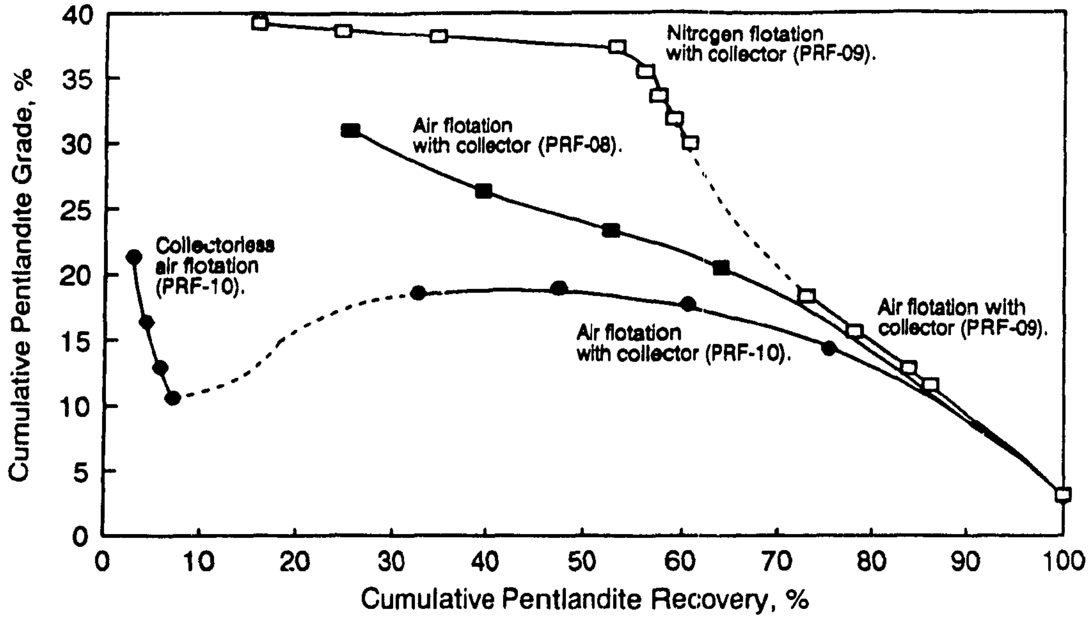


Fig 6.28 Cumulative Grade-Recovery curves for tests PRF-08, PRF-09, and PRF-10.

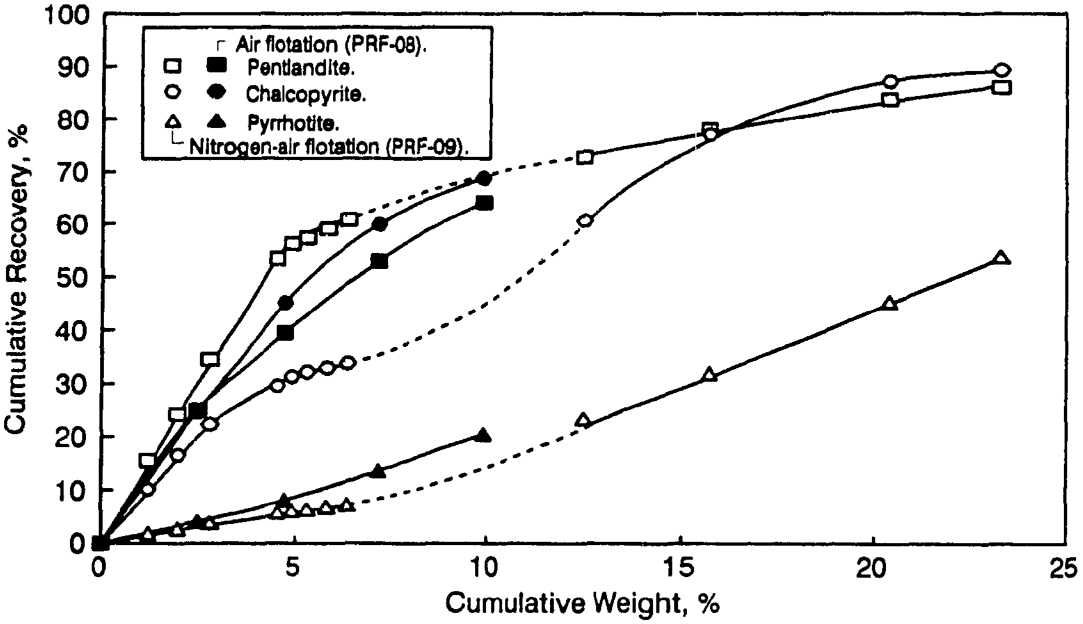


Fig 6.29 Cumulative Recovery-Yield curves for tests PRF-08 and PRF-09.

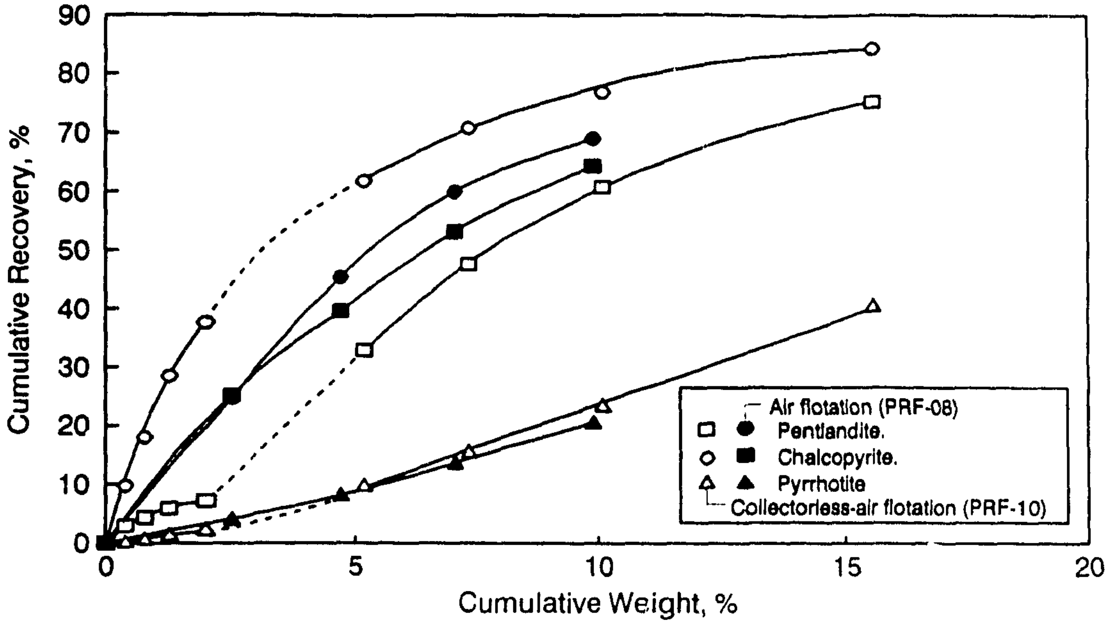


Fig 6.30 Cumulative Recovery-Yield curves for tests PRF-08 and PRF-10.

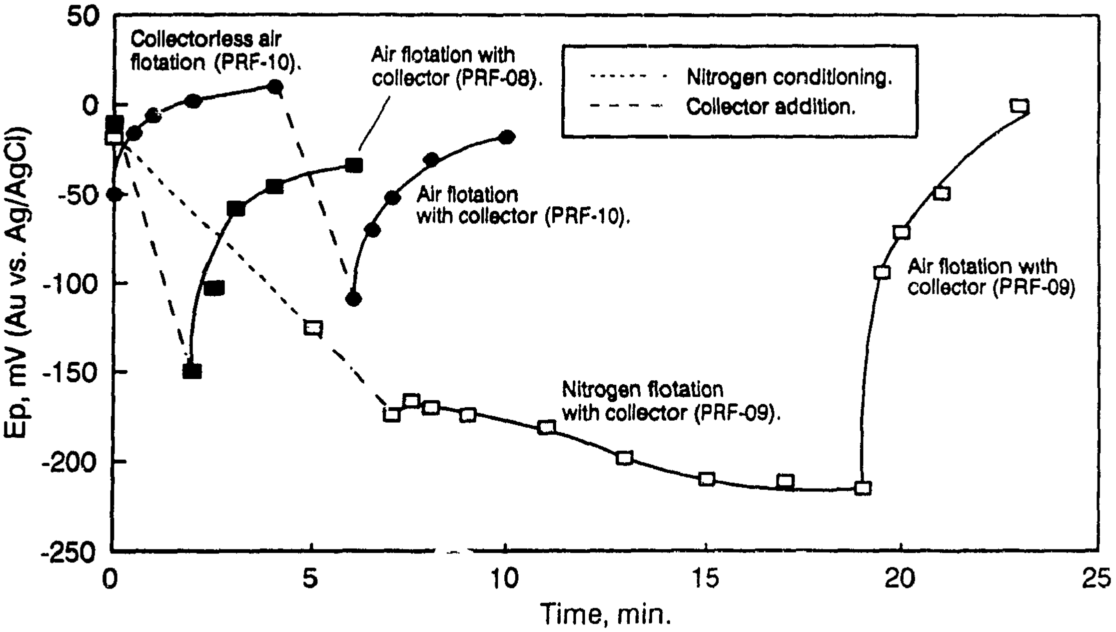


Fig 6.31 Pulp potential variations during tests PRF-08, PRF-09, and PRF-10.

6.3.2. Survey at Strathcona Mill.

The previous section (Section 6.3.1) showed that differences in pulp potential values correlated with the flotation response. These results are also reported elsewhere (48, 49). A general observation is that pentlandite/pyrrhotite selectivity was better at lower pulp potential values (the pyrrhotite being depressed).

In order to determine the range in ORP values in the plant, a survey was performed at the mill during the period December 3rd to 10th, 1989. Data on pH and oxygen content taken manually were also included.

This survey presented a picture of the changes in ORP as well as in dissolved oxygen and pH at different points in the mill. This knowledge will be of use in future on-site N_2 work designed to determine if, by having a reducing pulp, pyrrhotite is selectively depressed as was demonstrated in the laboratory flotation tests.

Points included in the survey were the following:

1 Rod mill discharge circuit "A".

2 Ball mill discharge circuit "A".

3 Pump sump of rod & ball mill discharge circuit "A".

4 Pump sump of rod & ball mill discharge circuit "B".

5 Cyclone overflow of primary grinding circuits "A", "B" & "C".

6 Primary rougher flotation circuits "A" & "B".

7 Primary rougher feed at the chemical analysis unit (PCXQ).

8 Process and recycled water.
9 Pump sump of regrinding mill discharge circuits "A" & "B"
10 Cyclone overflow of regrinding circuits "A" & "B".
11 Magnetic flotation.

Oxidation-reduction potential values of most of the points monitored are in the range -75 mV to +50 mV (Au vs. Ag/AgCl).

Grinding mill discharges showed the most reducing values, the process water stream was the most oxidizing.

6.3.2.1. Methodology.

A computerized system developed at McGill University to register ORP continuously was installed near the selected sampling point; in some cases, the data was continuously collected for 15 hours or more.

Oxidation-reduction potential (ORP) and oxygen content were obtained using the electrodes described in Section 5.1. In this particular case pH monitoring was done by using a Cole Parmer pH electrode model N-0993-99.

Points such as cyclone overflow, the flotation cells and the recycled water tank were easy to sample and the values collected were consistent. On the other hand, mill discharge presented practical problems for electrode immersion due to the high percent solids and high flowrate of the pulp.

Oxidation-reduction potential (ORP) values are referred to the silver/silver chloride reference electrode while the dissolved oxygen is expressed as percent of saturation.

6.3.2.2. Results.

Table 6.5 shows the different streams included in the survey. Tables 6.6 and 6.7 present the streams that were monitored and the specific characteristics followed in each case. Table 6.6 includes the streams that were monitored for a period of time while Table 6.7 refers to single values manually taken in some locations around the mill.

Figure 6.32 presents a general picture of changes found during the survey in ORP, dissolved oxygen (DO), and pH around circuit "A". The numbers shown in this diagram are the average of the values taken over a period of time.

Figure 6.33 shows the approximate variations found during the survey for those sampling points monitored over a period of time. As is observed in the figure, most of the sampling points are in the range -75 to +50 mV referred to silver/silver chloride as the reference electrode.

TABLE 6.5 Sampling points in the survey.

IDENTIFICATION	MEANING
PWT	Process water
RWT	Recycled water
RMD"A"	Rod mill discharge circuit "A"
BMD"A"	Ball mill discharge circuit "A"
R&BMD"A"	Pump sump of the rod & ball mills circuit "A"
R&BMD"B"	Pump sump of the rod & ball mills circuit "B"
COF"A"	Cyclone overflow primary grinding circuit "A"
COF"B"	Cyclone overflow primary grinding circuit "B"
COF"C"	Cyclone overflow primary grinding circuit "C"
PRF at PCXQ	Primary rougher feed stream at PCXQ
PRFLO"A"	Primary rougher flotation circuit "A"
PRFLO"B"	Primary rougher flotation circuit "B"
RDIS"A"	Pump sump of the regrinding mill circuit "A"
RDIS"B"	Pump sump of the regrinding mill circuit "B"
RCOF"A"	Cyclone overflow regrinding mill circuit "A"
RCOF"B"	Cyclone overflow regrinding mill circuit "B"
MFLO	Magnetic flotation

TABLE 6.6 Time spent in monitoring the different points during the survey.

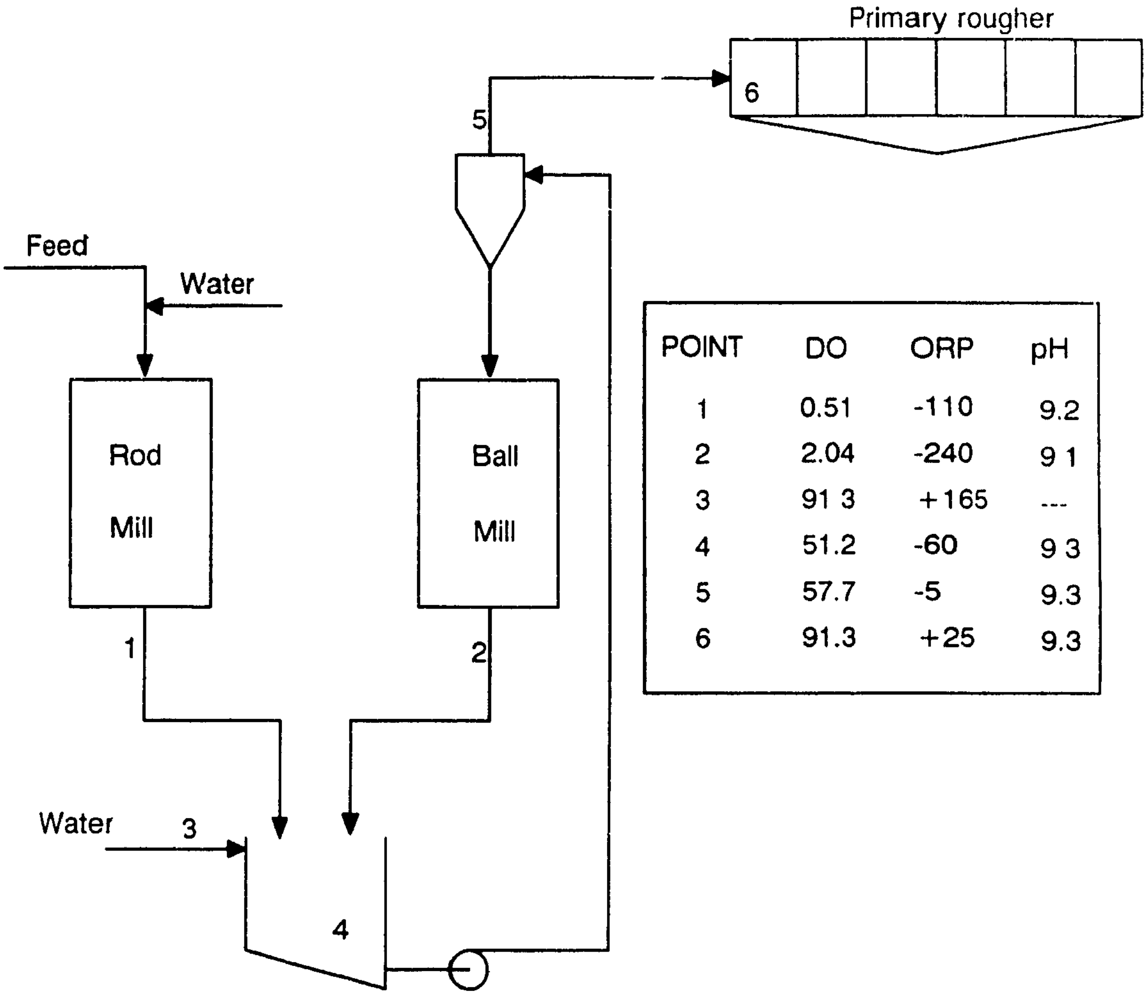
Stream	Date	Time(*) (hr)	ORP, mV (Ag/AgCl)	DO (ppm)	pH	Temp. ('C)
PWT	7/12/89	1	By computer	Manually	Manually	6
RWT	9/12/89	2	By computer	Manually	Manually	9.5
RMD"A"	7/12/89	1	By computer	-----	-----	--
BMD"A"	7/12/89	1	By computer	-----	-----	--
R&BMD"A"	9/12/89	0.5	Manually	-----	-----	--
R&BMD"B"	9/12/89	0.25	Manually	-----	-----	--
	9/12/89	2	By computer	Manually	Manually	12
COF"A"	5/12/89	-------	Manually	Manually	Manually	14
	6/12/89	17	By computer	Manually	Manually	14
COF"B"	5/12/89	-------	Manually	Manually	Manually	14
	5/12/89	13	By computer	-----	-----	--
	6/12/89	2	By computer	Manually	Manually	13.5
COF"C"	5/12/89	-------	Manually	Manually	Manually	14
PRF at PCXQ	9/12/89	0.5	Manually	Manually	-----	15
PRFLO"A"	7/12/89	15	By computer	-----	-----	14
	8/12/89	0.5	Manually	Manually	Manually	13.5
	9/12/89	-------	Manually	Manually	Manually	14
PRFLO"B"	6/12/89	-------	Manually	Manually	Manually	14
RDIS"A"	8/12/89	0.25	Manually	-----	-----	--
RDIS"B"	8/12/89	0.25	Manually	-----	-----	--
RCOF"A"	8/12/89	2	By computer	Manually	Manually	16
RCOF"B"	8/12/89	2	By computer	Manually	Manually	15
MFLO	8/12/89	17	By computer	Manually	Manually	16

(*) The time showed is referred to the one spent in ORP monitoring either manually or by computer.

TABLE 6.7 Single values manually taken in some locations of the Mill.

STREAM	DATE	TIME	ORP (*) (Ag/AgCl)	DO (**)	pH	TEMP 'C	POINT OF MEASUREMENT
PWT	7/12/89	---	+375	---	---	8.5	Launder water of magnetic flotation bank.
RWT	7/12/89	---	-45	10.1	---	14	Reclaim water tank
50%RWT-50%PWT	7/12/89	---	-25	10.2	---	14	A mixture of equal parts of the above locations.
RMD"A"	7/12/89	---	---	0.05	---	15	Discharge of grinding mill.
BMD"A"	5/12/89	---	-350 to -300	0.20	9.06	15	Discharge of grinding mill.
PRF at PCXQ	7/12/89	11:30	+40	---	8.90	15	PRF sampler at PCXQ.

(*) Measured with a combined electrode of Au (plate) and Ag/AgCl as the reference electrode.



POINT	DO	ORP	pH
1	0.51	-110	9.2
2	2.04	-240	9 1
3	91 3	+165	---
4	51.2	-60	9 3
5	57.7	-5	9.3
6	91.3	+25	9.3

Fig 6.32 Sampling points selected at the Strathcona circuit "A" to monitor ORP, DO, and pH changes.

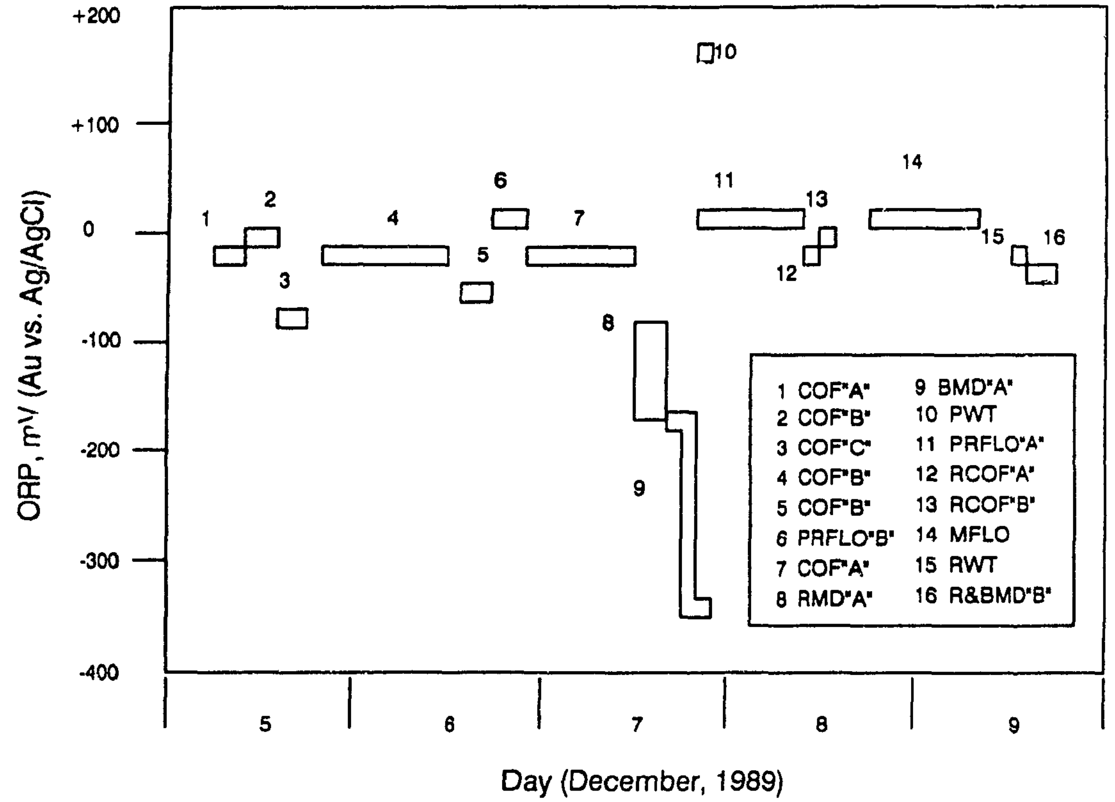


Fig 6.33 Approximate variations in some of the streams monitored during the survey.

Figures 6.34 to 6.37 present the ORP variations of the streams that were monitored for more than twelve hours. Figure 6.34 corresponds to primary rougher flotation circuit "A" (PRFLO"A") and Figure 6 35 to the magnetic flotation bank (MFLO). In both cases the values were taken from the first cell of the bank. Figures 6.36 and 6.37 correspond to primary grinding cyclone overflow circuits "A" and "B" (COF"A" and COF"B") respectively. All of them present very consistent values having a variation of ±10 to ±15 mV except for the last hour in the stream COF"B" which presented higher variations as observed in Figure 6.37.

Figure 6.38 shows that large changes in ORP occurred in the mill discharges. It is worth reminding that practical problems were encountered in monitoring these streams due to the high percent solids and high flowrate at these sampling points. It will be necessary to construct a device to protect the electrode and to ensure better reproducibility. The ORP variations of process water (PWT), recycled water (RWT), COF"B", and cyclone overflow circuits "A" and "B" of the regrinding step (RDIS"A" and RDIS"B") are also shown in the same figure. All except PWT which is more oxidizing showed values between 0 to -50 mV with a variation of about ±10 mV.

Figure 6.39 presents data from some sampling points which were collected manually; Figure 6.40 includes the pulp potential changes across the whole of flotation circuit "A". As observed in Figure 6.40, the variations found from bank to bank were from around 0 mV to

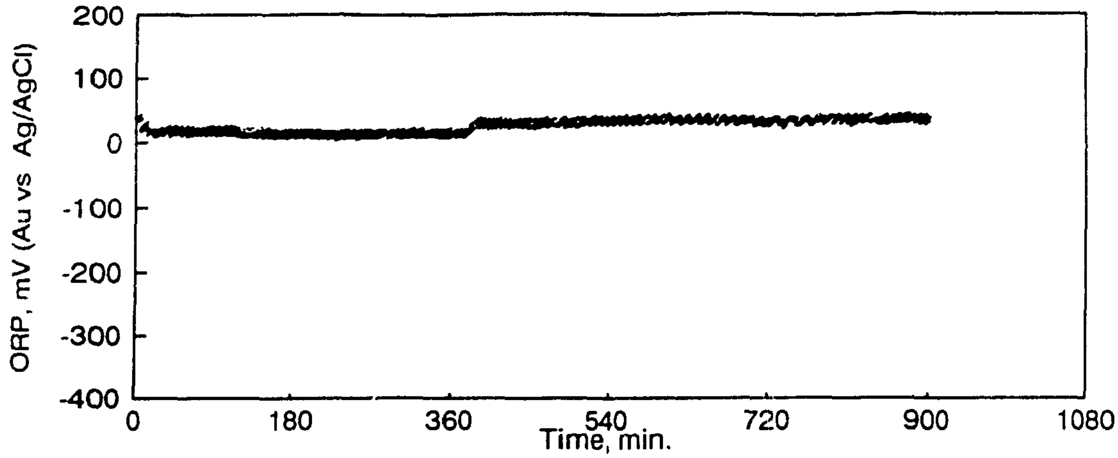


Fig 6.34 Pulp potential variations in the first cell of rougher flotation circuit "A" for a period of 15 hours.

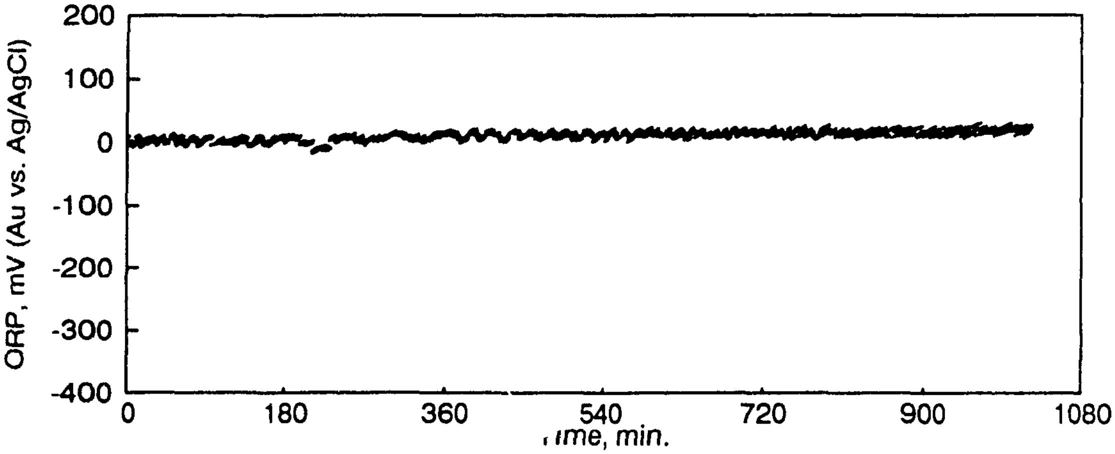


Fig 6.35 Pulp potential variations in the first cell of magnetic flotation for a period of 17 hours.

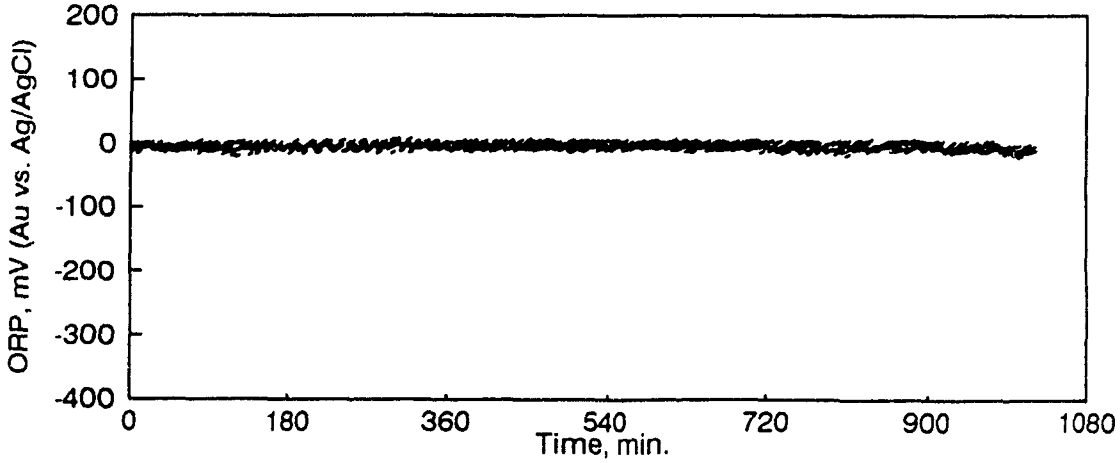


Fig 6.36 Pulp potential variations in grinding mill cyclone overflow circuit "A" for a period of 17 hours.

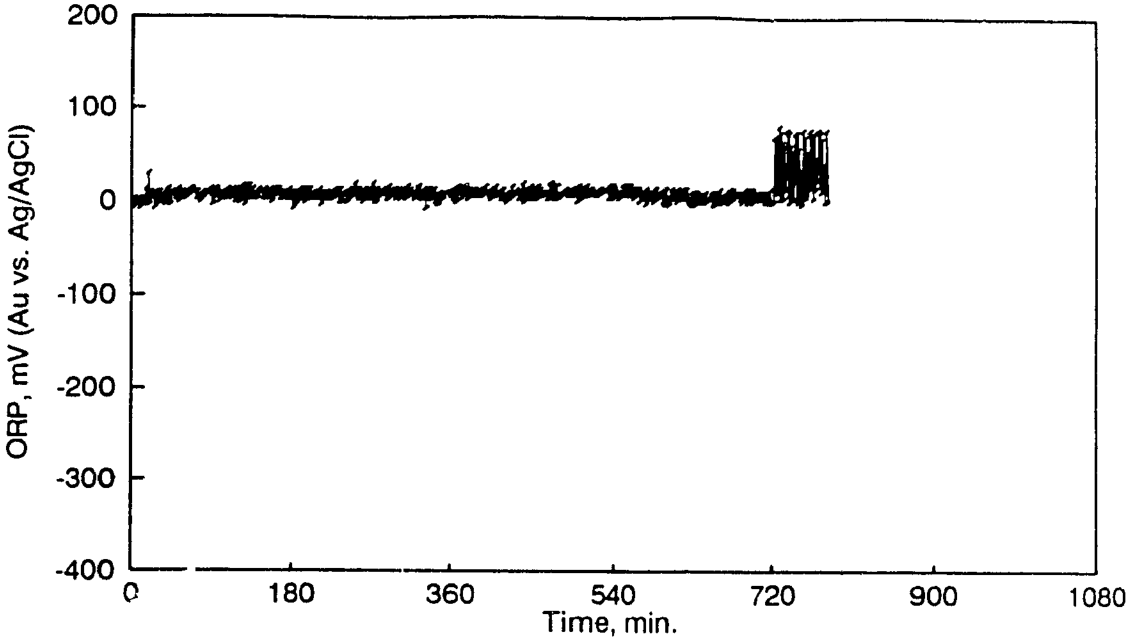


Fig 6.37 Pulp potential variations in primary grinding cyclone overflow circuit "B" for a period of 13 hours.

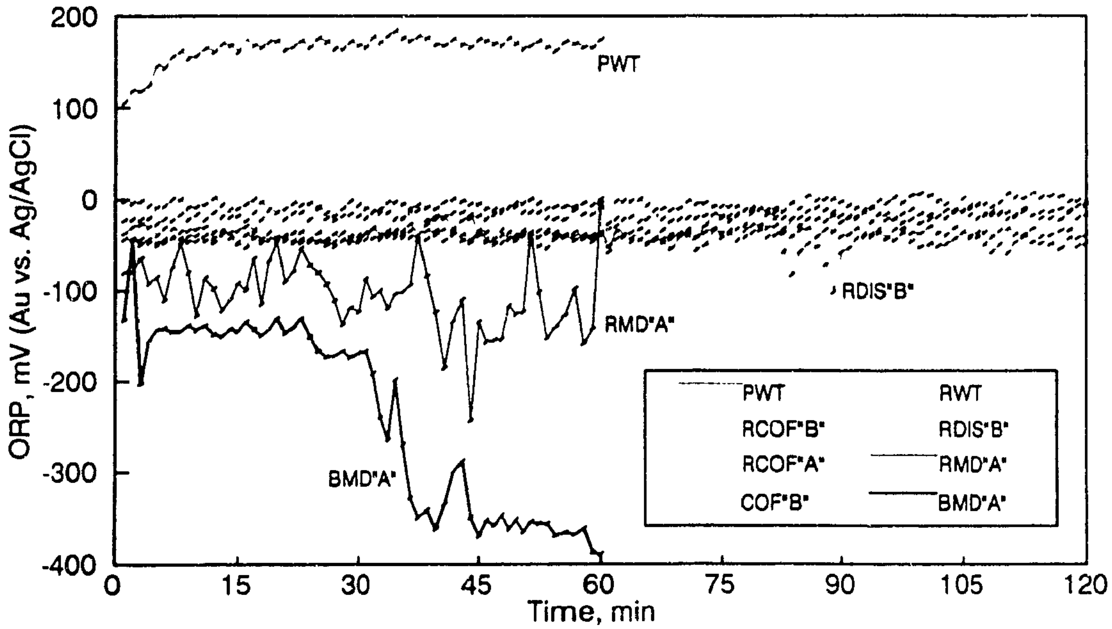


Fig 6.38 Variations in ORP detected during two hours or less in the locations indicated.

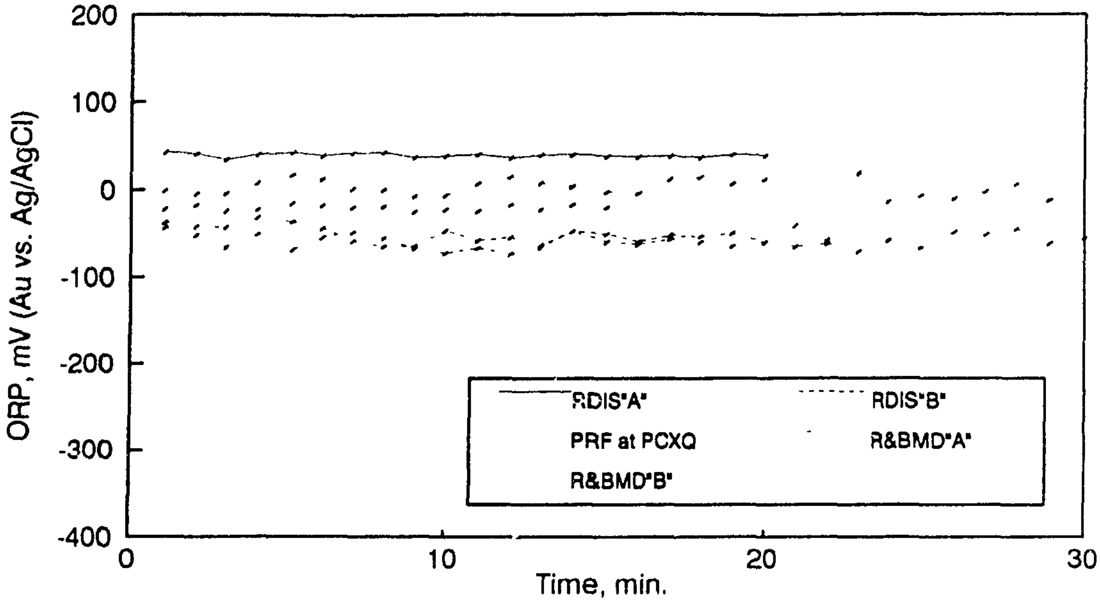


Fig 6.39 Data collected manually in the points indicated.

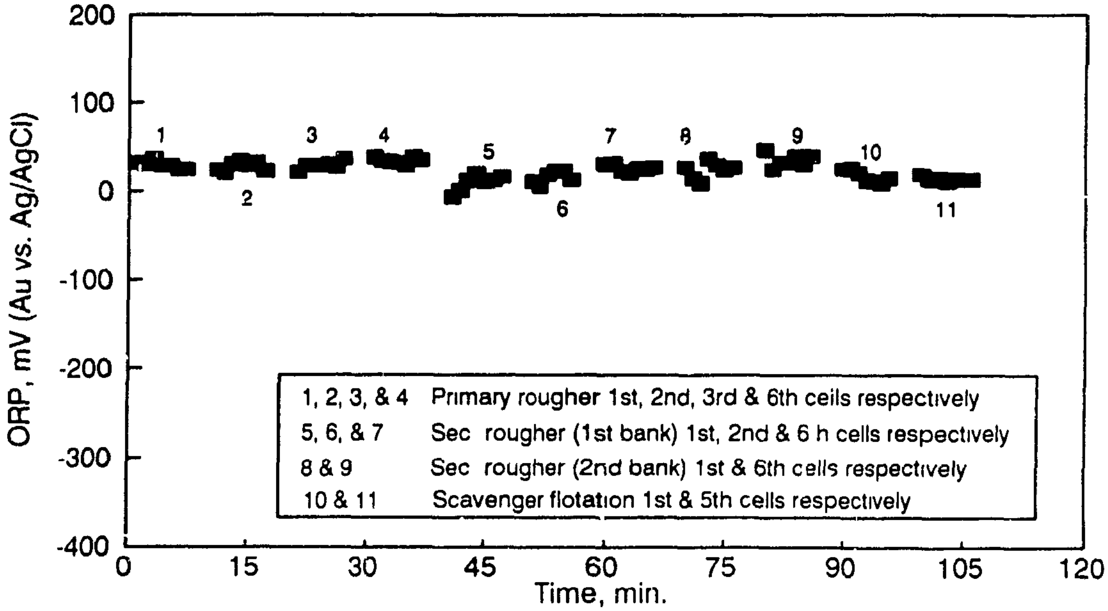


Fig 6.40 Pulp potential variations in the whole circuit "A".

+50 mV.

Figures 6.41 and 6.42 show the dissolved oxygen (DO) data collected manually for a period of time at some points around circuits "A" and "B".

Figures 6.43 and 6.44 show the changes in dissolved oxygen and pH respectively for circuit "A". Figure 6.43 shows that the dissolved oxygen progressively increases from the roughers to the scavengers Figure 6.44 shows that the pH progressively drops from 9 4 in the first cell of primary rougher flotation to 8 9 in secondary rougher and scavenger flotation. Also it is observed that the first cell in any of the banks has the highest pH value while the last cell has the lowest one.

Figure 6.45 contains the pH data manually collected at some points around the mill. From all these sampling points only R&BMD"B" showed relatively large variations.

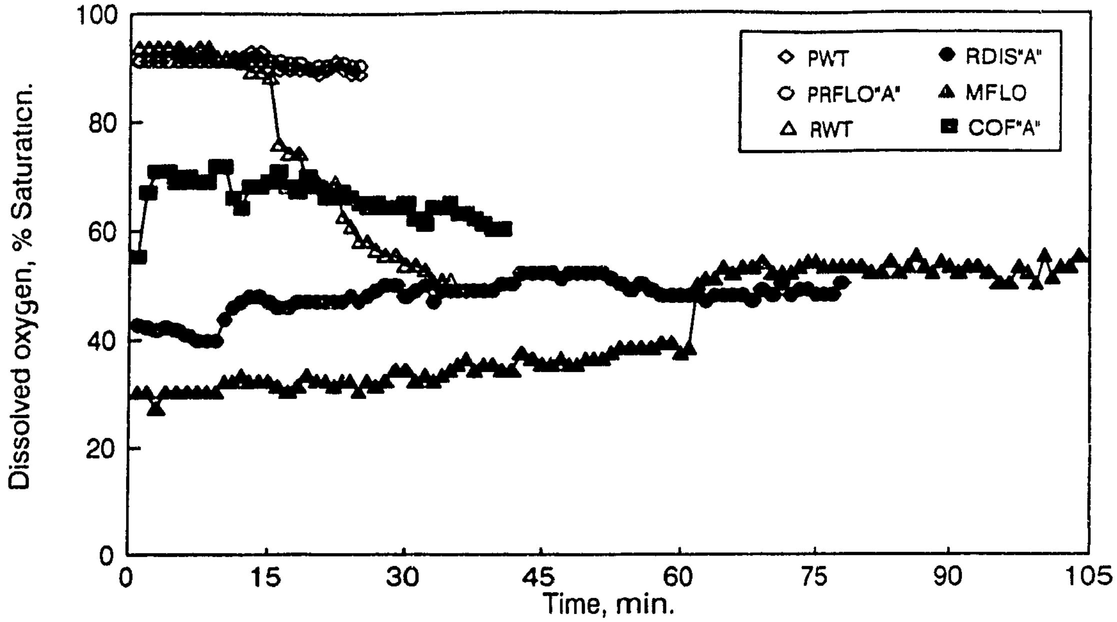


Fig 6.41 Oxygen content detected in some points around circuit "A".

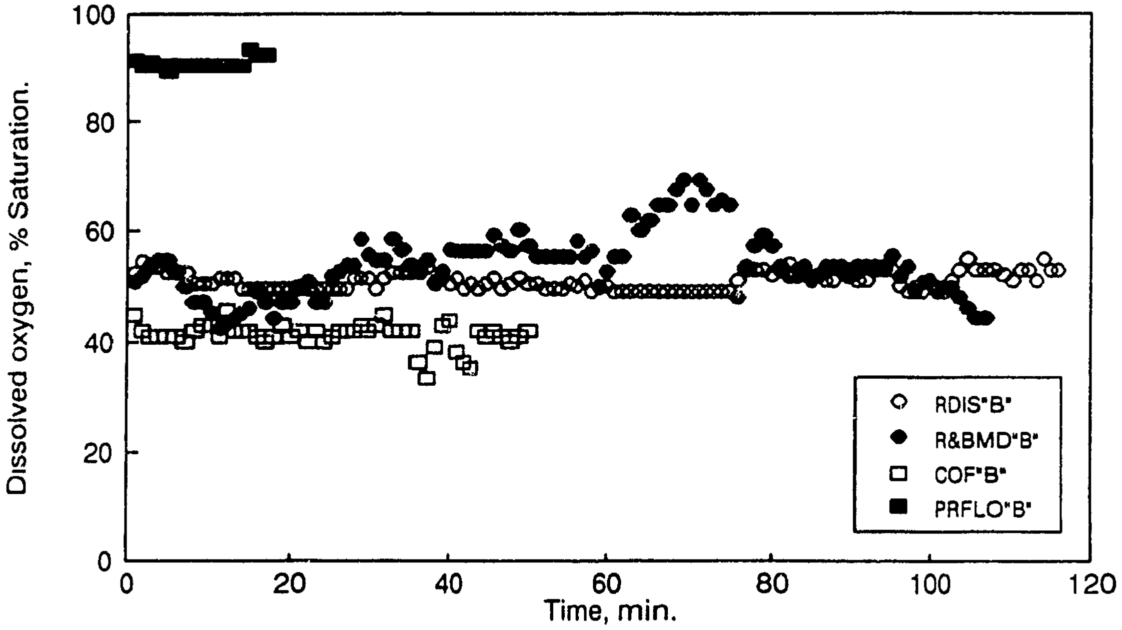


Fig 6.42 Dissolved oxygen variations in some points around circuit "B".

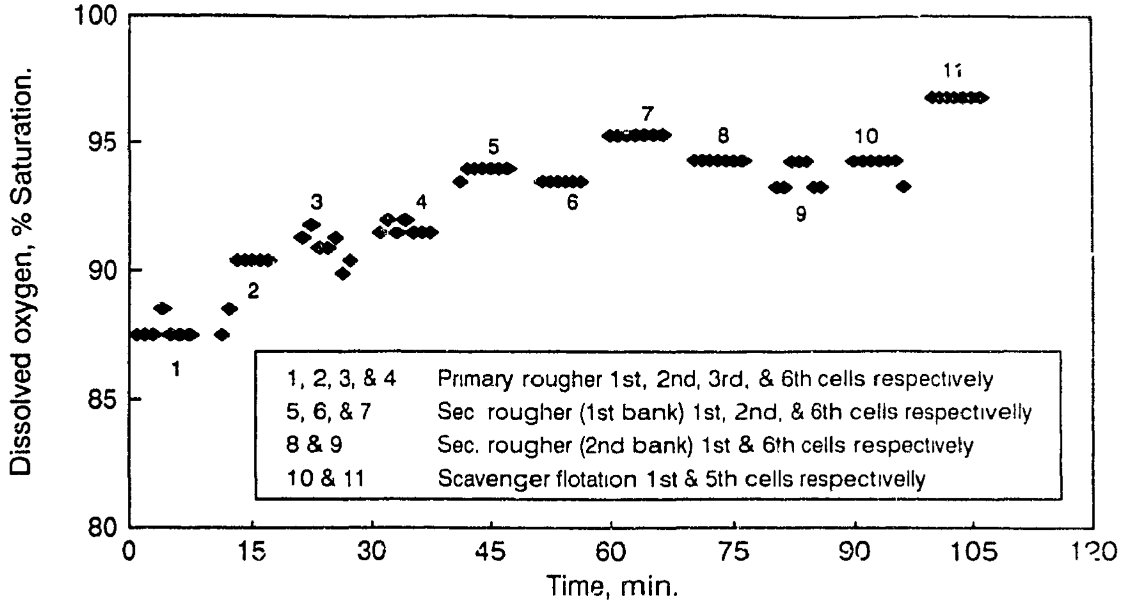


Fig 6.43 Dissolved oxygen variations in the whole circuit "A"

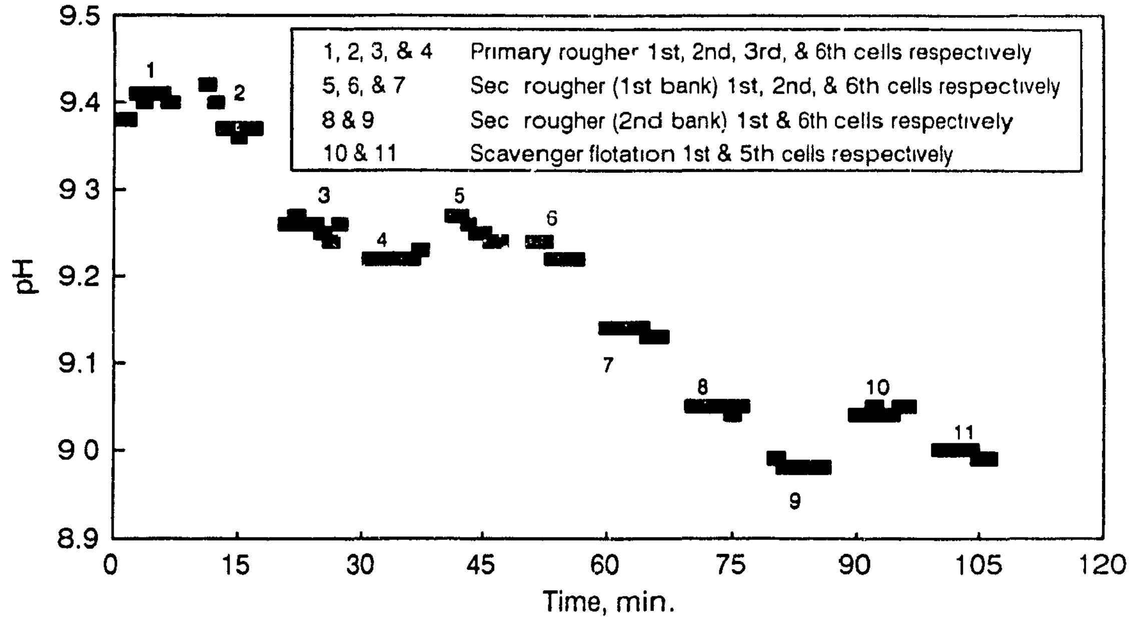


Fig 6.44 pH variations in the whole circuit "A".

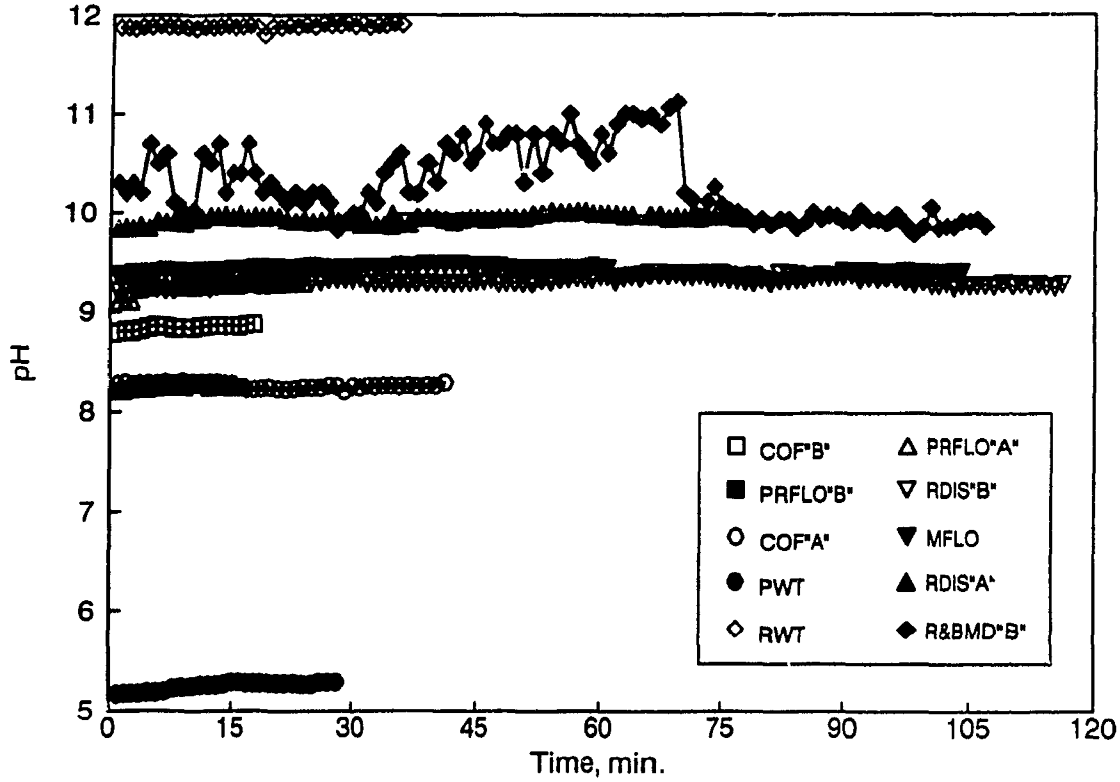


Fig 6.45 pH variations detected in some points around the mill.

6.3.3. Flotation Tests at McGill University.

6.3.3.1. Tests Using Falconbridge Procedure.

In order to check reproducibility of results at McGill University with those previously obtained on-site, flotation tests using air and nitrogen were carried out on a sample taken from the rod mill feed at Strathcona mill. Cumulative grade-recovery curves for chalcopyrite and pentlandite are shown in Figure 6.46. This figure confirms the results obtained on-site; that is, nitrogen flotation is superior to air flotation in the rougher stage (the first four points of the curves) presenting better pentlandite and chalcopyrite grades at comparable recoveries. At the beginning of the rougher stage, nitrogen generates a pentlandite concentrate grade significantly greater than that using air. As recovery increases, it seems that the grade tends to reach equivalent values using both air and nitrogen. Also it is shown that in the scavenger stage (the last five points) there is no difference between using air and nitrogen. Figure 6.47 shows the recovery-yield curves for pentlandite, chalcopyrite, and pyrrhotite.

Figure 6.48 shows that nitrogen retards the pyrrhotite flotation compared with air in the rougher stage. Scavenger flotation presents no difference in using air or nitrogen as the flotation gas possibly due to the copper sulphate added as activator at the scavenger stage.

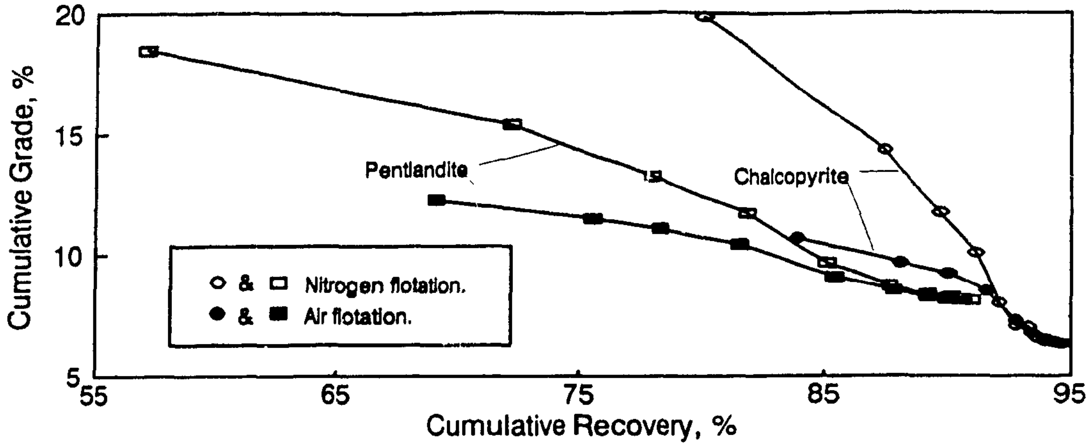


Fig 6.46 Grade-Recovery curves for Strathcona mill ore at McGill. (First four points represent roughing, last five scavenging).

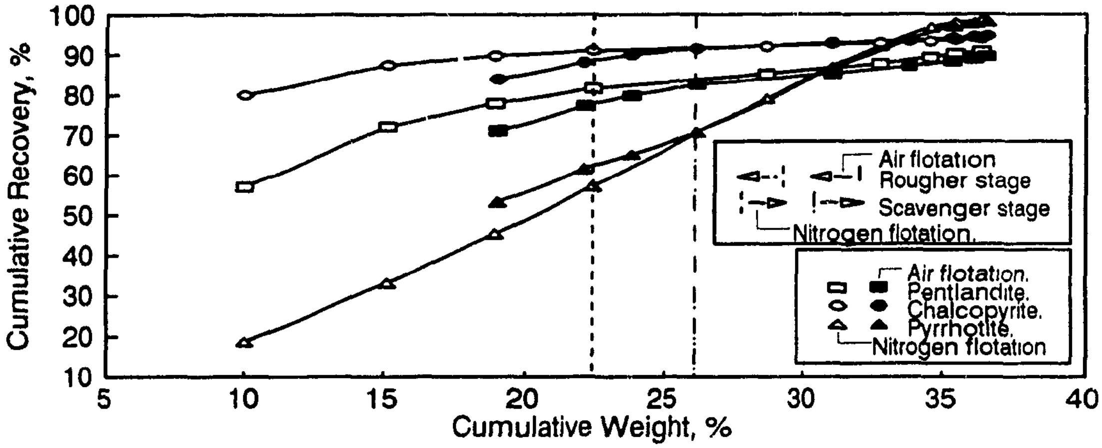


Fig 6.47 Cumulative Recovery-Yield curves for Strathcona mill ore at McGill.

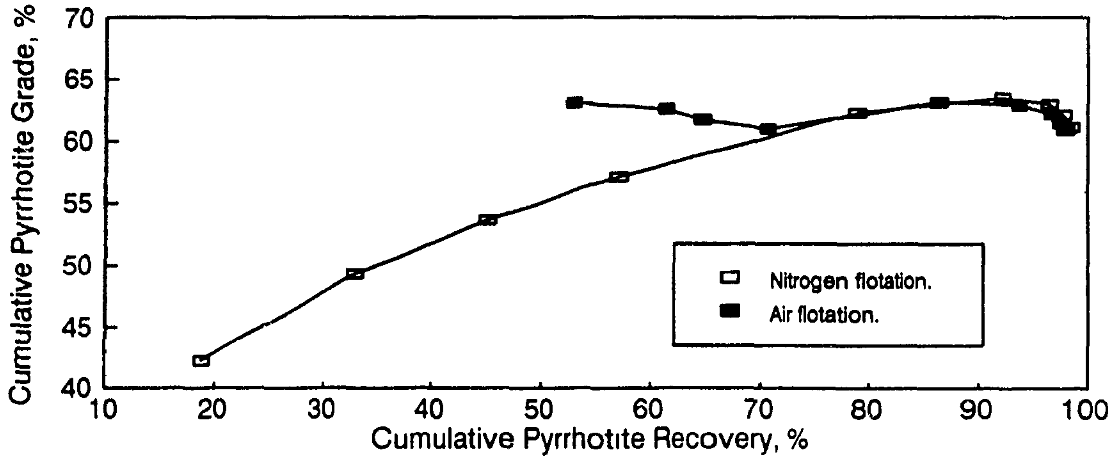


Fig 6.48 Pyrrhotite Grade-Recovery curves for Strathcona mill ore at McGill University.

6.3.3.2. Testing Diethyl Amine as Pyrrhotite Depressant.

As part of the research at McGill trying to eliminate iron sulphides, some bench scale flotation tests were performed using diethyl amine. This reagent has shown some promise as a pyrrhotite depressant in microflotation tests (44) and it is of interest to compare its effectiveness with nitrogen.

Grade-recovery curves for pyrrhotite at different conditions are shown in Figure 6.49. The figure shows that diethyl amine seems to have no effect on pyrrhotite depression for this particular ore at the conditions studied. On the other hand pH seems to have certain effect on pyrrhotite depression getting lower grades at pH 9.2. Figure 6.50 presents the recovery-yield curves. Comparable results were obtained with and without the use of diethyl amine; pH 9.2 seems to have slightly better pyrrhotite depressant action over pH 6.0.

The oxidizing conditions are known to be important in the use of these amines for pyrrhotite depression. This variable was not examined in these preliminary trials.

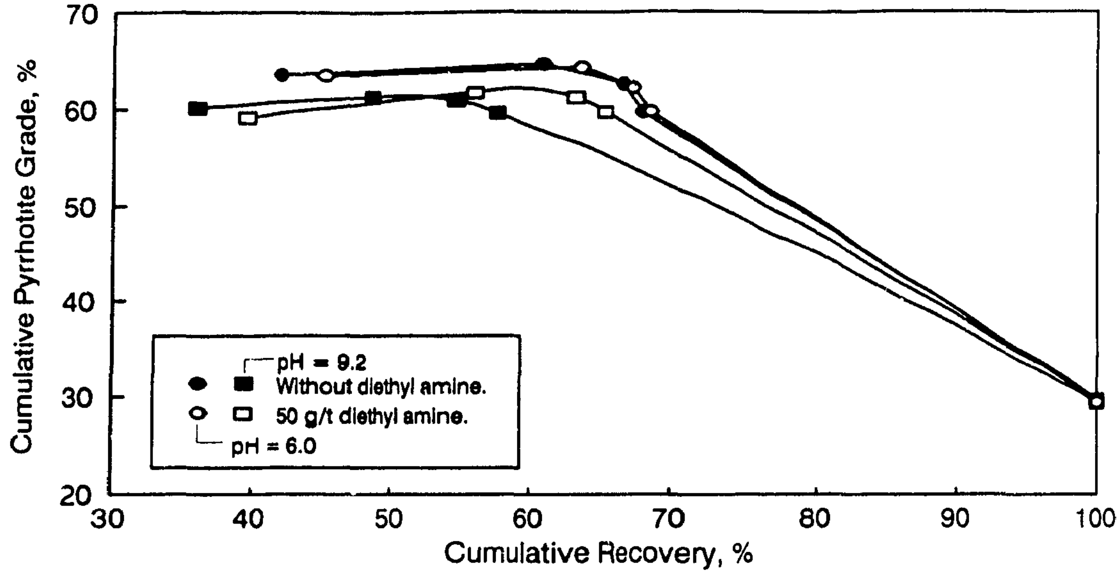


Fig 6.49 Pyrrhotite Grade-Recovery curves using diethyl amine as depressant.

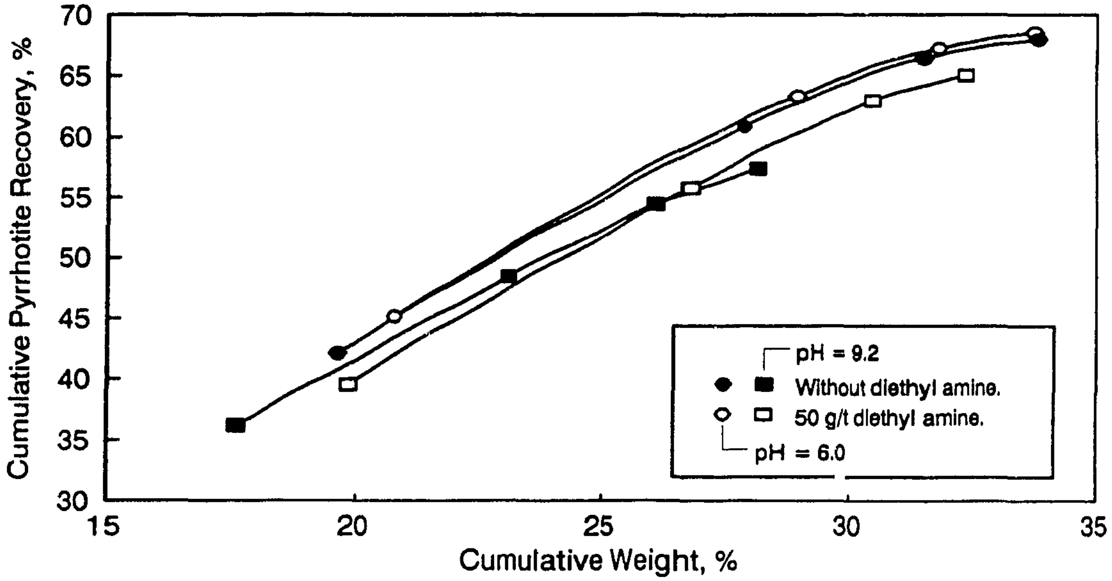


Fig 6.50 Recovery-Yield curves using diethyl amine as depressant for pyrrhotite.

6.3.3.3. Microflotation Tests on Pyrrhotite from Falconbridge.

Microflotation experiments were performed on pyrrhotite mineral with a particle size of 100% -200 +400 mesh (-74 +37 μm). The results of these measurements are given in Figure 6.51. From these mono-mineral experiments it can be observed that higher recoveries are obtained at pH 9.0 for both collectorless and with collector flotation.

6.3.4. Flotation Column with Gas Recycle.

Martin (40) found that 0.04 tonnes of fresh nitrogen gas is needed per tonne of ore treated. For economic reasons, some form of recirculation system is necessary. Preliminary work on the design of a flotation column for gas recycle was initiated. Figure 6.52 shows the design. It includes a container for gas/concentrate separation and a compressor to recirculate the gas to the column. Wash water addition could be another source of oxygen introduction to the system thus it is necessary to design a device to deoxygenate this water stream before it enters the flotation column. Figures 6.53 and 6.54 show the details of both devices. These devices are intended for use in a laboratory column of 6.4 cm (2.5") in diameter.

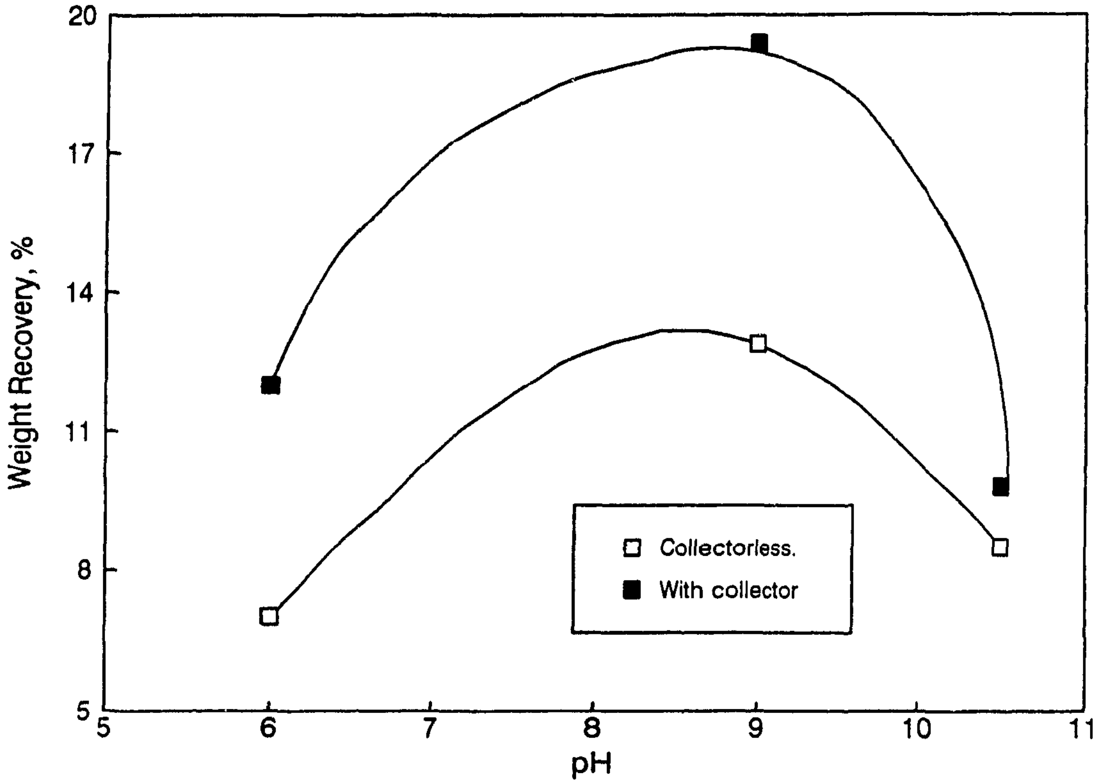


Fig 6.51 Microflotation tests of pyrrhotite as a single mineral.

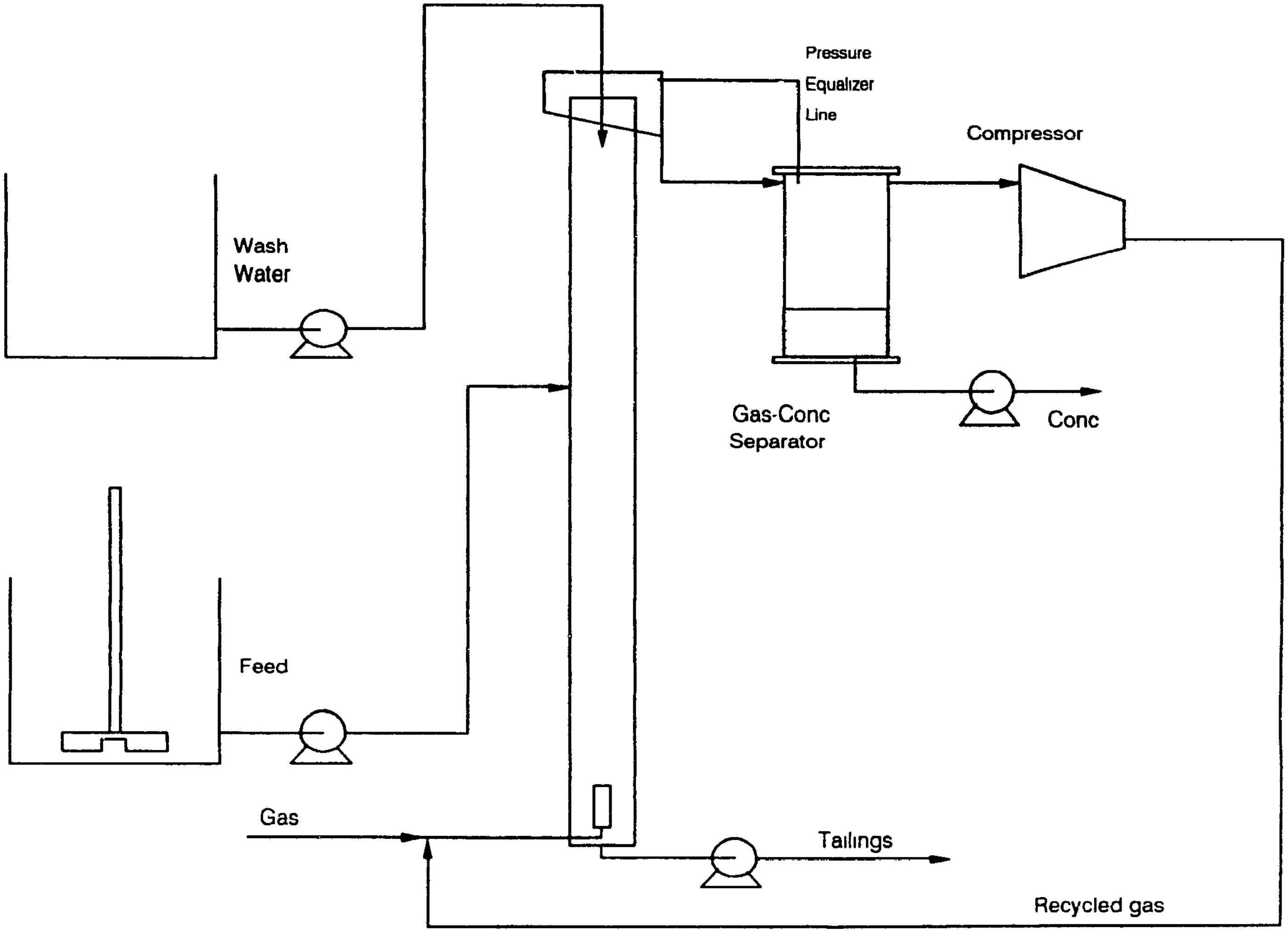
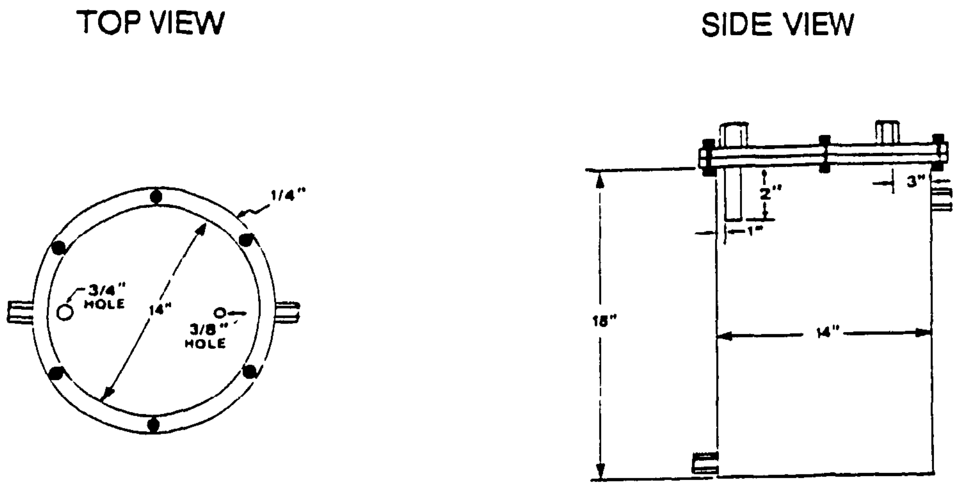


Fig 6.52 Flotation column with gas recycle.



TOP VIEW

SIDE VIEW

Fig 6.53 Detail of gas/concentrate separator.

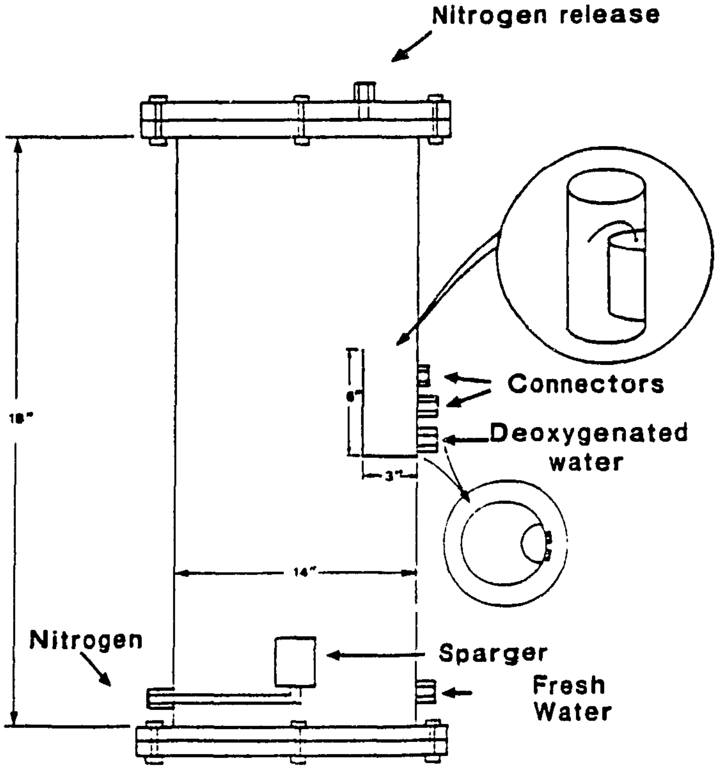


Fig 6.54 Detail of washwater deoxygenation tank.

6.4. Discussion.

6.4.1. Pyrite Flotation.

Majima (50) found that among all the sulphide minerals, pyrite has the highest rest potential. This suggests that when pyrite is in galvanic contact with other sulphide minerals, oxidation of the other sulphides will be enhanced. He found that the oxidation of galena, sphalerite and chalcopyrite was accelerated in the presence of pyrite. The presence of pyrite also contributes to the oxidation of xanthate collectors to dixanthogen which may then adsorb non-selectively and reduce flotation selectivity (51). These observations may explain some of the difficulties in selective flotation of pyritic ores.

Section 3.4 described the mechanisms governing galvanic interaction. There it was also discussed how the introduction of nitrogen hinders this galvanic coupling and promotes pyrite flotation.

This thesis proves that using nitrogen -down to 95% purity at least- promotes pyrite flotation both before and after zinc flotation (Figures 6.2, 6.6, and 6.10). In tests performed on-site at Brunswick Mining it was demonstrated that the technique developed in the laboratory for pyrite flotation also worked in continuous minicell pilot tests. This work has been published (52).

In the case of zinc reverse flotation (performed at Kidd Creek) not only was pyrite promoted by nitrogen, so also was chalcopyrite Chalcopyrite has the next highest rest potential (50) thus it too may be responding to the same breaking of galvanic contact in the presence of nitrogen as argued for pyrite.

In at least one operation, zinc reverse flotation was used to yield a copper concentrate (53); nitrogen may be more effective.

An apparently contradictory observation at Kidd Creek in the Cu/Zn separation testwork on cyclone overflow, was the fact that the behaviour of pyrite was the same in nitrogen as in air flotation.

In this case even air flotation gave high pyrite recovery. Perhaps the oxygen content is low enough in Kidd Creek cyclone overflow (i.e. immediately after grinding) that pyrite flotation is already favoured, and that further exclusion of oxygen (by nitrogen) is not necessary. Martin (40) found that air flotation of pyrite after nitrogen conditioning still gave high pyrite recovery suggesting that any de-oxygenating system (such as grinding mills) will activate pyrite.

A potentially important finding in the reverse flotation was that good results could be achieved by heating the pulp at only 65°C if it is preceded by 20 minutes of aeration. Further work is required to confirm these results.

6.4.2. Pyrrhotite Rejection.

Using the Hodgson/Agar model (described in Section 2.3), the use of nitrogen instead of air could hinder the formation of OH ions on the pyrrhotite surface which are necessary for xanthate uptake The use of nitrogen then would promote pyrrhotite depression which corresponds to the present observation which showed, in all cases, that nitrogen was superior to air in pyrrhotite depression. Microflotation tests on single mineral pyrrhotite showed that higher recoveries are obtained at pH close to 9 Lower pyrrhotite recoveries at lower pH may be due to the lack of OH ions, at higher pH excess OH^- may exist on the surface thus tipping the balance in favor of hydrophilicity.

Pulp potential values registered on cyclone overflow and on rougher flotation during the survey were in the range -10 to +30 mV vs. Ag/AgCl (+210 to +250 mV vs. S.H.E.). The ORP values registered in the laboratory were in the range +40 to +60 mV vs. Ag/AgCl (+260 to +280 vs. S.H.E.) in the case of air flotation and -50 to -30 mV vs. Ag/AgCl (+170 to +190 mV vs. S.H.E.) in the case of nitrogen flotation. The plant, therefore, tends to operate at potential values which favours pyrrhotite flotation, not depression.

In most cases pentlandite/chalcopyrite was promoted using nitrogen except for tests PRF-A and PRF-B where chalcopyrite was also depressed. This chalcopyrite depression may be explained by the much lower initial pulp potentials for tests PRF-A and PRF-B (≅ -25 mV vs. S.H E) compared with those for tests PRF-01 to PRF-10 (+75 - +175 mV vs. S.H.E.)

The influence of pulp potential in chalcopyrite flotation was investigated by Trahar (54) and he found that flotation is retarded below 0 mV (vs. S.H.E.) but becomes pronounced above 100 mV (vs. S.H.E.). The plant survey showed a pulp potential range of +210 to +250 mV vs. S.H.E. which clearly favours chalcopyrite flotation.

6.4.3. Advantages of Pyrite and Pyrrhotite Rejection.

Rejection of iron sulphides, primarily pyrite and pyrrhotite, in processing sulphide ores remains a major challenge. Their selective rejection by the use of nitrogen has several advantages:

- The production of higher grade base metal concentrates compared with the conventional operation.

- The production of iron sulphide concentrates which may have certain economic value, or could be separately impounded for improved control of acid drainage.

- Possible savings in reagents consumption such as lime and copper sulphate. Lime is typically used to depress pyrite in, for example, zinc flotation. If pyrite is removed (using nitrogen) there would be less pyrite to depress. If less lime is used probably less $CuSO_4$ would be needed.

* A positive environmental impact since less sulphur would be sent to the smelter, cutting down SO_2 emissions.

7. CONCLUSIONS AND SUGGESTIONS FOR FUTURE WORK.

7.1. Brunswick Mining and Smelting.

* Laboratory flotation tests show that mixtures of 95% nitrogen or greater are required for pyrite flotation.

* Continuous on-site tests at Brunswick Mining confirmed the technical feasibility of using nitrogen gas for pyrite flotation ahead of sphalerite flotation.

* Results from the continuous tests suggested that pH adjustment might not be necessary for a successful sphalerite flotation. It was also confirmed that copper activation in alkaline medium is not essential for sphalerite flotation.

* Up to now there is no doubt that pyrite is promoted by nitrogen flotation even at 95% purity and plants with a smelter on-site can perform industrial scale tests to check whether pyrite flotation is economically feasible this way.

* Enclosed cell (e.g. flotation column) flotation tests using nitrogen should be performed to check if pyrite can still be floated this way which would give another competitive route for the separation of complex sulphides.

7.2. Kidd Creek Mines.

* Nitrogen was found to be more efficient than air in the zinc reverse flotation stage. These investigations show that, compared to air, flotation with nitrogen leads to greater removal of pyrite and chalcopyrite from the sphalerite concentrate. Further work should be conducted on-site to verify the action of nitrogen gas on the depression of zinc during chalcopyrite flotation.

* Flotation tests on primary grinding cyclone overflow indicate that sphalerite is less floatable when nitrogen is used as the flotation gas and chalcopyrite recovery is not affected. This may suggest a promising method of improved selectivity in the flotation of chalcopyrite.

* Results using air as the flotation gas show that heating to 65 °C (instead of the 80 °C commonly practiced) might be sufficient to have good zinc recoveries in the zinc reverse flotation.

* Continuous in plant flotation tests using minicells should be performed on zinc reverse flotation to evaluate the results obtained on bench scale.

7.3. Falconbridge (Strathcona mill).

* Pentlandite recovery was consistently better with nitrogen than with air flotation.

* The improvement in grade-recovery curves was a consequence of more effective pyrrhotite rejection using nitrogen.

* Nitrogen flotation proceeded at an almost constant pulp potential while it increased exponentially with time during air flotation.

* Even though collectorless flotation with nitrogen was more effective than air flotation with collector, it was less effective than nitrogen flotation with collector.

* The use of continuous minicells for in plant testing is recommended. Two minicells would be desirable for parallel testing of nitrogen and air as the flotation gases.

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9. APPENDIX 1 BRUNSWICK MINING ORE TESTWORK

LABORATORY FLOTATION

TABLE A.1 Flotation test at McGill University to evaluate nitrogen purity over flotation performance. Test run with pure nitrogen.

TABLE A.2 Flotation test at McGill University to evaluate nitrogen purity over flotation performance. Test run with 95% nitrogen.

TABLE A.3 Flotation test at McGill University to evaluate nitrogen purity over flotation performance. Test run with 90% nitrogen.

CONTINUOUS FLOTATION

TABLE B.1 Pyrite flotation using minicells at Brunswick Mining.

TABLE A.1 Brunswick Mining ore flotation tests at McGill University.
Testing nitrogen purity over flotation performance.

RUN IDENTIFICATION
****

Run number 1
Nitrogen purity 100 %

Sample	Time (min)	Solids (%)	Gas used	Ph	Grade (%)			
					Fe	Cu	Pb	Zn
Feed (*)	----	100.00	----	----	28.52	0.32	3.01	7.31
Bulk Con1	0.50	12.45	Air	8.50	33.80	0.99	9.99	6.50
Bulk Con2	1.00	4.60	Air	8.50	33.60	0.98	9.00	7.44
Bulk Con3	2.00	5.96	Air	8.50	34.30	0.67	5.08	8.08
Bulk Con4	4.00	6.21	Air	8.50	33.60	0.40	3.63	10.60
Fe Conc 1	4.50	16.87	N2	8.50	40.50	0.12	1.27	3.51
Fe Conc 2	5.00	7.59	N2	8.50	40.00	0.11	1.27	4.24
Fe Conc 3	6.00	5.98	N2	8.50	36.80	0.11	1.36	6.93
Fe Conc 4	7.00	2.74	N2	8.50	34.30	0.11	1.56	8.18
Fe Conc 5	9.00	2.66	N2	8.50	32.50	0.11	1.62	8.25
Fe Conc 6	11.00	2.26	N2	8.50	28.50	0.11	1.75	11.00
Zn Conc 1	11.50	3.58	Air	10.00	12.00	0.17	1.09	41.00
Zn Conc 2	12.00	1.95	Air	10.00	13.60	0.22	1.40	30.30
Zn Conc 3	13.00	1.83	Air	10.00	15.50	0.28	1.85	27.80
Zn Conc 4	15.00	1.33	Air	10.00	18.10	0.31	1.98	14.70
Tails	----	23.99	----	----	12.10	0.09	0.75	0.99

(*) Calculated

METALLURGICAL RESULTS

Sample	Cumul. Solids (%)	Cumulative grade				Cumulative recovery			
		Fe (%)	Cu (%)	Pb (%)	Zn (%)	Fe (%)	Cu (%)	Pb (%)	Zn (%)
Bulk Con1	12.45	----	0.99	9.99	----	----	38.77	41.32	----
Bulk Con2	17.05	----	0.99	9.72	----	----	52.95	55.08	----
Bulk Con3	23.01	----	0.91	8.52	----	----	65.51	65.14	----
Bulk Con4	29.22	----	0.80	7.48	----	----	73.32	72.63	----
Fe Conc 1	46.09	12.10	----	----	----	23.95	----	----	----
Fe Conc 2	53.68	40.34	----	----	----	34.60	----	----	----
Fe Conc 3	59.66	39.65	----	----	----	42.31	----	----	----
Fe Conc 4	62.40	39.21	----	----	----	45.61	----	----	----
Fe Conc 5	65.06	38.71	----	----	----	48.64	----	----	----
Fe Conc 6	67.32	38.10	----	----	----	50.90	----	----	----
Zn Conc 1	70.90	----	----	----	41.00	----	----	----	20.07
Zn Conc 2	72.85	----	----	----	37.23	----	----	----	28.15
Zn Conc 3	74.68	----	----	----	34.88	----	----	----	35.11
Zn Conc 4	76.01	----	----	----	31.79	----	----	----	37.78
Feed	100.00	28.52	0.32	3.01	7.31	100.00	100.00	100.00	100.00

TABLE A.2 Brunswick Mining ore flotation tests at McGill university.
Testing nitrogen purity over flotation performance.

RUN IDENTIFICATION
****

Run number 2
Nitrogen purity 95 %

Sample	Time (min)	Solids (%)	Gas used	Ph	Grade (%)			
					Fe	Cu	Pb	Zn
Feed (*)	----	100.00	----	----	28.08	0.28	2.67	7.67
Bulk Con1	0.50	13.08	Air	8.50	33.60	0.80	8.10	6.80
Bulk Con2	1.00	5.34	Air	8.50	33.50	0.77	7.24	7.67
Bulk Con3	2.00	5.36	Air	8.50	33.90	0.56	4.30	9.53
Bulk Con4	4.00	5.42	Air	8.50	31.90	0.37	3.34	10.30
Fe Conc 1	4.50	16.19	N2	8.50	40.10	0.11	1.16	4.11
Fe Conc 2	5.00	8.80	N2	8.50	38.70	0.11	1.29	4.96
Fe Conc 3	6.00	3.89	N2	8.50	36.60	0.13	1.62	6.55
Fe Conc 4	7.00	3.04	N2	8.50	30.60	0.14	1.90	9.12
Fe Conc 5	9.00	2.32	N2	8.50	26.50	0.15	2.21	11.20
Fe Conc 6	11.00	1.89	N2	8.50	23.40	0.14	2.06	13.00
Zn Conc 1	11.50	3.49	Air	10.00	12.00	0.18	1.05	44.80
Zn Conc 2	12.00	1.52	Air	10.00	13.50	0.24	1.43	36.00
Zn Conc 3	13.00	1.35	Air	10.00	15.30	0.29	1.78	31.20
Zn Conc 4	15.00	1.77	Air	10.00	16.20	0.28	1.83	16.80
Tails	----	26.54	----	----	14.80	0.09	0.71	1.26

(*) Calculated.

METALLURGICAL RESULTS

Sample	Cumul. Solids (%)	Cumulative grade				Cumulative recovery			
		Fe (%)	Cu (%)	Pb (%)	Zn (%)	Fe (%)	Cu (%)	Pb (%)	Zn (%)
Bulk Con1	13.08	----	0.80	8.10	----	----	37.18	39.63	----
Bulk Con2	18.42	----	0.79	7.85	----	----	51.79	54.10	----
Bulk Con3	23.78	----	0.74	7.05	----	----	62.45	62.72	----
Bulk Con4	29.20	----	0.67	6.36	----	----	69.58	69.49	----
Fe Conc 1	45.39	14.80	----	----	----	23.12	----	----	----
Fe Conc 2	54.19	39.61	----	----	----	35.24	----	----	----
Fe Conc 3	58.08	39.20	----	----	----	40.31	----	----	----
Fe Conc 4	61.12	38.38	----	----	----	43.63	----	----	----
Fe Conc 5	63.44	37.58	----	----	----	45.81	----	----	----
Fe Conc 6	65.33	36.84	----	----	----	47.39	----	----	----
Zn Conc 1	68.82	----	----	----	44.80	----	----	----	20.38
Zn Conc 2	70.34	----	----	----	42.13	----	----	----	27.51
Zn Conc 3	71.69	----	----	----	39.81	----	----	----	33.01
Zn Conc 4	73.46	----	----	----	34.80	----	----	----	36.88
Feed	100.00	28.08	0.28	2.67	7.67	100.00	100.00	100.00	100.00

TABLE A.3 Brunswick Mining ore flotation tests at McGill University.
Testing nitrogen purity over flotation performance.

RUN IDENTIFICATION
****
Run number 3
Nitrogen purity 90 %

Sample	Time (min)	Solids (%)	Gas used	Ph	Grade (%) Fe	Cu	Pb	Zn
Feed (*)	----	100.00	----	----	28.53	0.35	3.64	8.08
Bulk Con1	0.50	11.43	Air	8.50	30.80	1.17	11.90	8.20
Bulk Con2	1.00	6.96	Air	8.50	31.70	1.17	11.20	8.66
Bulk Con3	2.00	5.81	Air	8.50	31.90	0.72	7.62	11.20
Bulk Con4	4.00	3.85	Air	8.50	31.20	0.47	4.88	12.40
Fe Conc 1	4.50	4.35	N2	8.50	36.10	0.20	2.90	7.70
Fe Conc 2	5.00	3.18	N2	8.50	32.00	0.19	2.79	9.89
Fe Conc 3	6.00	1.87	N2	8.50	28.80	0.17	2.40	10.50
Fe Conc 4	7.00	0.24	N2	8.50	28.30	0.19	2.59	11.80
Fe Conc 5	9.00	0.28	N2	8.50	27.40	0.20	2.79	11.70
Fe Conc 6	11.00	0.28	N2	8.50	26.40	0.17	2.68	12.60
Zn Conc 1	11.50	7.23	Air	10.00	19.50	0.14	1.22	33.40
Zn Conc 2	12.00	2.63	Air	10.00	19.60	0.20	1.53	32.00
Zn Conc 3	13.00	2.28	Air	10.00	23.80	0.21	1.85	21.40
Zn Conc 4	15.00	2.05	Air	10.00	28.00	0.21	1.95	11.50
Tails	----	47.56	----	----	28.10	0.07	0.79	1.02

(*) Calculated.

METALLURGICAL RESULTS

Sample	Cumul. Solids (%)	Cumulative grade Fe (%)	Cu (%)	Pb (%)	Zn (%)	Cumulative recovery Fe (%)	Cu (%)	Pb (%)	Zn (%)
Bulk Con1	11.43	----	1.17	11.90	----	----	37.96	37.39	----
Bulk Con2	18.39	----	1.17	11.64	----	----	61.08	58.82	----
Bulk Con3	24.20	----	1.06	10.67	----	----	72.95	70.99	----
Bulk Con4	28.05	----	0.98	9.88	----	----	78.09	76.15	----
Fe Conc 1	32.40	36.10	----	----	----	5.50	----	----	----
Fe Conc 2	35.58	34.37	----	----	----	9.07	----	----	----
Fe Conc 3	37.45	33.26	----	----	----	10.96	----	----	----
Fe Conc 4	37.69	33.14	----	----	----	11.20	----	----	----
Fe Conc 5	37.97	32.98	----	----	----	11.46	----	----	----
Fe Conc 6	38.25	32.79	----	----	----	11.72	----	----	----
Zn Conc 1	45.48	----	----	----	33.40	----	----	----	29.90
Zn Conc 2	48.11	----	----	----	33.03	----	----	----	40.32
Zn Conc 3	50.39	----	----	----	30.84	----	----	----	46.37
Zn Conc 4	52.44	----	----	----	28.05	----	----	----	49.29
Feed	100.00	28.53	0.35	3.64	8.08	100.00	100.00	100.00	100.00

TABLE B.1 Pyrite flotation using minicells at Brunswick Mining.

PYRITE FLOTATION:
Pulp flowrate = 0.8 l/min.
N2 addition to cond. tank = 6 l/min.
Collector in cell #1 (g/t) = 80-20 X343 and X350 (20)
Flotation gas in cell #1 (l/min) = N2 (5)

SPHALERITE FLOTATION:
Collector in cell #2 (g/t) = 80-20 X343 and X350 (80)
Activator in cell #2 (g/t) = CuSO4 (1100)
CaO addition to cell #2 = To pH of 10.5
Flotation gas in cell #2 (l/min) = Air (6)

LABORATORY DATA

Sample	Solids (g)	Solids (%)	Gas used	pH	ORP, mV (Ag/AgCl)	Oxygen (ppm)	Grade (%) Zn	Grade (%) Fe
Feed(*)	512.15	100.00	-	-	-	-	10.04	27.57
FeS2 Conc	136.60	26.67	N2	8.80	-50	0.11	6.02	38.57
ZnS Conc.	56.15	10.96	Air	10.50	-	-	46.40	14.32
Tails	319.40	62.36	-	-	-	-	5.36	25.19

METALLURGICAL RESULTS

Sample	Cumul. solids (%)	Mineral grade Sphal. (%)	Mineral grade Pyrite (%)	Recovery Sphal. (%)	Recovery Pyrite (%)
FeS2 Conc.	26.67	8.97	82.86	16.00	37.32
ZnS Conc.	10.96	69.16	30.76	50.69	5.70
Tails	62.36	7.99	54.11	33.31	56.99
Feed(*)	100.00	14.96	59.22	100.00	100.00

(*) Calculated

10. APPENDIX 2 KIDD CREEK MINES

ZINC REVERSE FLOTATION

TABLE C.1 Standard zinc reverse flotation on final zinc concentrate using air.

TABLE C.2 Standard zinc reverse flotation on final zinc concentrate using nitrogen.

TABLE C.3 Zinc reverse flotation on final zinc concentrate without conditioning and using air.

TABLE C.4 Zinc reverse flotation on final zinc concentrate with 20 min conditioning and using air.

TABLE C.5 Zinc reverse flotation on final zinc concentrate at 80 °C and using air.

TABLE C.6 Zinc reverse flotation on final zinc concentrate at 65 °C and using air.

TESTS ON CYCLONE OVERFLOW

TABLE D.1 Flotation test on cyclone overflow primary grinding using air.

TABLE D.2 Flotation test on cyclone overflow primary grinding using nitrogen.

TABLE C.1 Zinc reverse flotation tests at Kidd Creek.
Samples from final zinc concentrate stream (circuit "A").

RUN IDENTIFICATION
****

Run number	1
Gas used	Air

Sample	Time	Solids	Grade (%)				
	(min)	(%)	Fe	Zn	Cu	FeS2(*)	CuFeS2(*)
Conc 1	0.50	12.10	15.30	53.10	2.28	28.56	6.58
Conc 2	1.00	4.41	21.80	37.50	2.47	42.17	7.13
Conc 3	1.50	3.55	21.90	38.20	2.20	42.89	6.35
Conc 4	2.50	2.75	16.30	45.10	2.07	31.11	5.98
Tails	----	77.19	6.86	57.90	0.26	14.74	0.75
Feed(*)	----	100.00	9.33	55.37	0.72	19.07	2.08

METALLURGICAL RESULTS

Sample	Cumulative grade					Sample Tails (*)	
	Fe (%)	Zn (%)	Cu (%)	FeS2(*) (%)	CuFeS2(*) (%)		Zn grade (%)
Conc 1	15.30	53.10	2.28	28.56	6.58	Feed	-----
Conc 2	17.04	48.93	2.33	32.20	6.73	Tail 1	55.68
Conc 3	17.90	47.03	2.31	34.09	6.66	Tail 2	56.64
Conc 4	17.71	46.80	2.28	33.73	6.58	Tail 3	57.46
Feed(*)	9.33	55.37	0.72	19.07	2.08	Tail 4	57.90

Sample	Cumulative recovery					Sample Tails (*)	
	Fe (%)	Zn (%)	Cu (%)	FeS2(*) (%)	CuFeS2(*) (%)		Zn rec. (%)
Conc 1	19.83	11.60	38.29	18.12	38.29	Feed	-----
Conc 2	30.14	14.59	53.42	27.89	53.42	Tail 1	88.40
Conc 3	38.47	17.04	64.26	35.87	64.26	Tail 2	85.41
Conc 4	43.27	19.28	72.15	40.35	72.15	Tail 3	82.96
Tails	100.00	100.00	100.00	100.00	100.00	Tail 4	80.72
Feed(*)	-----	-----	-----	-----	-----	-----	-----

(*) Calculated.

TABLE C.2 Zinc reverse flotation tests at Kidd Creek.
Samples from final zinc concentrate stream (circuit "A").

RUN IDENTIFICATION
****

Run number	2
Gas used	Nitrogen

Sample	Time (min)	Solids (%)	Grade (%)				
			Fe	Zn	Cu	FeS2(*)	CuFeS2(*)
Conc 1	0.50	11.19	23.20	35.60	3.42	43.38	9.88
Conc 2	1.00	2.71	20.40	37.90	3.23	37.73	9.33
Conc 3	1.50	0.99	16.60	42.70	3.02	29.96	8.72
Conc 4	2.50	1.04	14.90	49.40	3.45	25.49	9.96
Conc 5	4.50	1.27	11.10	50.00	2.29	19.52	6.61
Tails	----	82.80	6.59	58.00	0.34	13.51	0.98
Feed(*)	----	100.00	9.07	54.61	0.85	17.88	2.44

METALLURGICAL RESULTS

Sample	Cumulative grade					Sample Tails (*)	
	Fe (%)	Zn (%)	Cu (%)	FeS2(*) (%)	CuFeS2(*) (%)		Zn grade (%)
Conc 1	23.20	35.60	3.42	43.38	9.88	Feed	-----
Conc 2	22.65	36.05	3.38	42.28	9.77	Tail 1	57.00
Conc 3	22.25	36.49	3.36	41.46	9.70	Tail 2	57.60
Conc 4	21.77	37.33	3.36	40.42	9.72	Tail 3	57.77
Conc 5	20.98	38.27	3.29	38.87	9.49	Tail 4	57.88
Feed(*)	9.07	54.61	0.85	17.88	2.44	Tail 5	58.00

Sample	Cumulative recovery					Sample Tails (*)	
	Fe (%)	Zn (%)	Cu (%)	FeS2(*) (%)	CuFeS2(*) (%)		Zn rec. (%)
Conc 1	28.63	7.29	45.20	27.15	45.20	Feed	-----
Conc 2	34.73	9.17	55.53	32.87	55.53	Tail 1	92.71
Conc 3	36.53	9.95	59.05	34.52	59.05	Tail 2	90.83
Conc 4	38.25	10.89	63.30	36.01	63.30	Tail 3	90.05
Conc 5	39.81	12.06	66.75	37.40	66.75	Tail 4	89.11
Tails	100.00	100.00	100.00	100.00	100.00	Tail 5	87.94
Feed(*)	-----	-----	-----	-----	-----	-----	-----

(*) Calculated.

TABLE C.3 Zinc reverse flotation tests at Kidd Creek.
Samples from final zinc concentrate stream (circuit "A").

RUN IDENTIFICATION
****

Run number	3
Gas used	Air
Conditioning	-----

Sample	Time (min)	Solids (%)	Grade (%) Fe	Zn	Cu	FeS2(*)	CuFeS2(*)
Conc 1	0.50	11.14	21.30	29.50	8.18	30.31	23.62
Conc 2	1.00	5.58	18.80	36.10	5.06	30.83	14.61
Conc 3	1.50	4.96	17.90	38.10	4.36	30.22	12.59
Conc 4	2.50	2.41	15.60	42.50	2.56	28.68	7.39
Conc 5	4.50	1.89	15.80	44.30	2.04	30.09	5.89
Tails	----	74.02	6.59	56.10	0.20	13.78	0.58
Feed(*)	----	100.00	9.86	50.58	1.66	18.06	4.79

METALLURGICAL RESULTS

Sample	Cumulative grade Fe (%)	Zn (%)	Cu (%)	FeS2(*) (%)	CuFeS2(*) (%)	Sample	Tails (*) Zn grade (%)
Conc 1	21.30	29.50	8.18	30.31	23.62	Feed	-----
Conc 2	20.47	31.70	7.14	30.49	20.62	Tail 1	53.22
Conc 3	19.88	33.16	6.50	30.43	18.78	Tail 2	54.37
Conc 4	19.45	34.10	6.11	30.25	17.64	Tail 3	55.40
Conc 5	19.19	34.84	5.81	30.24	16.79	Tail 4	55.81
Feed(*)	9.86	50.58	1.66	18.06	4.79	Tail 5	56.10

Sample	Cumulative recovery Fe (%)	Zn (%)	Cu (%)	FeS2(*) (%)	CuFeS2(*) (%)	Sample	Tails (*) Zn rec. (%)
Conc 1	24.07	6.50	54.97	18.71	54.97	Feed	-----
Conc 2	34.71	10.49	72.00	28.24	72.00	Tail 1	93.50
Conc 3	43.70	14.22	85.03	36.54	85.03	Tail 2	89.51
Conc 4	47.52	16.25	88.75	40.37	88.75	Tail 3	85.78
Conc 5	50.55	17.90	91.07	43.52	91.07	Tail 4	83.75
Tails	100.00	100.00	100.00	100.00	100.00	Tail 5	82.10
Feed(*)	-----	-----	-----	-----	-----	-----	-----

(*) Calculated.

TABLE C.4 Zinc reverse flotation tests at Kidd Creek.
Samples from final zinc concentrate stream (circuit "A").

RUN IDENTIFICATION
****

Run number 4
Gas used Air
Conditioning 20 min.

Sample	Time	Solids			Grade (%)		
	(min)	(%)	Fe	Zn	Cu	FeS2(*)	CuFeS2(*)
Conc 1	0.50	10.55	25.80	25.10	2.28	51.12	6.58
Conc 2	1.00	4.51	24.20	26.80	2.47	47.32	7.13
Conc 3	1.50	3.00	19.50	36.20	2.20	37.74	6.35
Conc 4	2.50	2.61	13.20	44.90	2.07	24.45	5.98
Tails	----	79.33	6.66	57.90	0.26	14.31	0.75
Feed(*)	----	100.00	10.03	52.05	0.68	20.65	1.96

METALLURGICAL RESULTS

Sample		Cumulative grade				Sample	Tails (*)
	Fe (%)	Zn (%)	Cu (%)	FeS2(*) (%)	CuFeS2(*) (%)		Zn grade (%)
Conc 1	25.80	25.10	2.28	51.12	6.58	Feed	-----
Conc 2	25.32	25.61	2.34	49.98	6.75	Tail 1	55.23
Conc 3	24.35	27.37	2.31	47.95	6.68	Tail 2	56.73
Conc 4	22.94	29.58	2.28	44.98	6.59	Tail 3	57.49
Feed(*)	10.03	52.05	0.68	20.65	1.96	Tail 4	57.90

Sample		Cumulative recovery				Sample	Tails (*)
	Fe (%)	Zn (%)	Cu (%)	FeS2(*) (%)	CuFeS2(*) (%)		Zn rec. (%)
Conc 1	27.15	5.09	35.47	26.12	35.47	Feed	-----
Conc 2	38.03	7.41	51.89	36.46	51.89	Tail 1	94.91
Conc 3	43.86	9.50	61.61	41.94	61.61	Tail 2	92.59
Conc 4	47.31	11.75	69.59	45.03	69.59	Tail 3	90.50
Tails	100.00	100.00	100.00	100.00	100.00	Tail 4	88.25
Feed(*)	-----	-----	-----	-----	-----	-----	-----

(*) Calculated.

TABLE C.5 Zinc reverse flotation tests at Kidd Creek.
Samples from final zinc concentrate stream (circuit "A").

RUN IDENTIFICATION
****

Run number	5
Gas used	Air
Temperature	80 C.

Sample	Time (min)	Solids (%)	Grade (%)				
			Fe	Zn	Cu	FeS2(*)	CuFeS2(*)
Conc 1	0.50	10.86	36.00	17.30	4.62	68.61	13.34
Conc 2	1.00	2.37	31.80	21.50	3.94	60.87	11.38
Conc 3	1.50	1.64	27.90	27.50	3.37	53.57	9.73
Conc 4	2.50	1.95	17.40	39.90	2.31	33.02	6.67
Tails	----	83.19	6.75	57.50	0.07	14.50	0.20
Feed(*)	----	100.00	11.07	51.45	0.75	22.48	2.18

Sample	Cumulative grade					Sample Tails (*)	
	Fe (%)	Zn (%)	Cu (%)	FeS2(*) (%)	CuFeS2(*) (%)		Zn grade (%)
Conc 1	36.00	17.30	4.62	68.61	13.34	Feed	-----
Conc 2	35.25	18.05	4.50	67.23	12.99	Tail 1	55.61
Conc 3	34.44	19.10	4.37	65.72	12.63	Tail 2	56.54
Conc 4	32.46	21.50	4.13	61.93	11.94	Tail 3	57.10
Feed(*)	11.07	51.45	0.75	22.48	2.18	Tail 4	57.50

Sample	Cumulative recovery					Sample Tails (*)	
	Fe (%)	Zn (%)	Cu (%)	FeS2(*) (%)	CuFeS2(*) (%)		Zn rec. (%)
Conc 1	35.29	3.65	66.57	33.14	66.57	Feed	-----
Conc 2	42.10	4.64	78.96	39.56	78.96	Tail 1	96.35
Conc 3	46.23	5.52	86.30	43.47	86.30	Tail 2	95.36
Conc 4	49.29	7.03	92.27	46.33	92.27	Tail 3	94.48
Tails	100.00	100.00	100.00	100.00	100.00	Tail 4	92.97
Feed(*)	-----	-----	-----	-----	-----	-----	-----

(*) Calculated.

TABLE C.6 Zinc reverse flotation tests at Kidd Creek.
Samples from final zinc concentrate stream (circuit "A").

RUN IDENTIFICATION
****

Run number	6
Gas used	Air
Temperature	65 C.

Sample	Time (min)	Solids (%)	Grade (%) Fe	Zn	Cu	FeS2(*)	CuFeS2(*)
Conc 1	0.50	8.27	37.80	17.90	4.39	72.91	12.68
Conc 2	1.00	3.04	34.30	21.60	4.30	65.57	12.42
Conc 3	1.50	2.17	25.30	30.50	3.52	47.70	10.17
Conc 4	2.50	2.55	18.80	39.00	2.66	35.36	7.68
Tails	----	83.97	6.80	57.40	0.31	14.61	0.90
Feed(*)	----	100.00	10.91	51.99	0.90	22.23	2.59

Sample	Cumulative grade Fe (%)	Zn (%)	Cu (%)	FeS2(*) (%)	CuFeS2(*) (%)	Sample Tails (*)	Zn grade (%)
Conc 1	37.80	17.90	4.39	72.91	12.68	Feed	-----
Conc 2	36.86	18.90	4.37	70.94	12.61	Tail 1	55.06
Conc 3	35.00	20.76	4.23	67.20	12.21	Tail 2	56.21
Conc 4	32.42	23.67	3.98	62.12	11.49	Tail 3	56.86
Feed(*)	10.91	51.99	0.90	22.23	2.59	Tail 4	57.40

Sample	Cumulative recovery Fe (%)	Zn (%)	Cu (%)	FeS2(*) (%)	CuFeS2(*) (%)	Sample Tails (*)	Zn rec. (%)
Conc 1	28.65	2.85	40.39	27.12	40.39	Feed	-----
Conc 2	38.21	4.11	54.95	36.09	54.95	Tail 1	97.15
Conc 3	43.25	5.38	63.46	40.75	63.46	Tail 2	95.89
Conc 4	47.65	7.30	71.03	44.81	71.03	Tail 3	94.62
Tails	100.00	100.00	100.00	100.00	100.00	Tail 4	92.70
Feed(*)	-----	-----	-----	-----	-----	-----	-----

(*) Calculated.

TABLE D.1 Results of laboratory flotation test on cyclone overflow primary grinding at Kidd Creek Mines.

LABORATORY DATA

Sample	Time (min)	Solids (g)	Solids (%)	Gas used	pH	ORP, mV (Ag/AgCl)	Oxygen (ppm)	Grade (%) Zn	Grade (%) Cu	Grade (%) Fe
Feed	-	-	-	-	7.90	85	6.10	-	-	-
Head(*)	2.0	938.40	100.00	-	7.80	80	6.10	7.45	7.36	19.30
Conc-1	2.5	136.40	14.54	Air	7.90	25	9.40	4.91	23.50	30.20
Conc-2	3.0	70.30	7.49	Air	8.00	28	8.20	5.37	21.60	29.70
Conc-3	4.0	69.80	7.44	Air	7.90	40	8.30	7.79	17.80	31.00
Conc-4	6.0	68.10	7.26	Air	8.10	65	8.90	11.90	10.20	30.60
Tails	-	593.80	63.28	-	-	-	-	7.73	0.42	12.90

(*) calculated, conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

Sample	Cumul. solids (%)	Cumulative grade Zn (%)	Cumulative grade Cu (%)	Cumulative grade Fe (%)	Cumulative mineral grade Sphal (%)	Cumulative mineral grade Chalco (%)	Cumulative mineral grade Pyrite (%)	Cumulative mineral grade Gangue (%)	Cumulative recovery Sphal (%)	Cumulative recovery Chalco (%)	Cumulative recovery Pyrite (%)	Cumulative recovery Gangue (%)
Conc-1	14.5	4.91	23.50	30.20	7.32	67.87	20.51	4.31	9.6	46.4	10.8	1.6
Conc-2	22.0	5.07	22.85	30.03	7.55	66.00	21.36	5.08	15.0	68.4	17.1	2.8
Conc-3	29.5	5.75	21.58	30.27	8.58	62.32	24.30	4.81	22.8	86.3	26.0	3.5
Conc-4	36.7	6.97	19.33	30.34	10.39	55.82	28.68	5.11	34.3	96.4	38.2	4.7
Feed	100.0	7.45	7.36	19.30	11.11	21.27	27.57	40.06	100.0	100.0	100.0	100.0

TABLE D.2 Results of laboratory flotation test on cyclone overflow primary grinding at Kidd Creek Mines.

LABORATORY DATA

Sample	Time (min)	Solids (g)	Solids (%)	Gas used	pH	ORP, mV (Ag/AgCl)	Oxygen (ppm)	Grade (%) Zn	Grade (%) Cu	Grade (%) Fe
Feed	-	-	-	-	8.00	-18	6.10	-	-	-
Head(*)	2.0	971.50	100.00	-	8.10	-20	5.70	12.31	7.41	19.56
Conc-1	2.5	170.50	17.55	N2	8.10	-10	3.40	5.20	22 70	29.60
Conc-2	3.0	94.30	9.71	N2	8.20	-8	3.30	6.33	20.20	29.10
Conc-3	4.0	60.90	6.27	N2	8.20	-12	3.30	9.50	14.50	29.20
Conc-4	6.0	47.20	4.86	N2	8.10	-11	3.10	15.60	5.97	28.60
Tails	-	598.60	61.62	-	-	-	-	15.30	0.43	13.50

(*) calculated, conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

Sample	Cumul. solids (%)	Cumulative grade Zn (%)	Cumulative grade Cu (%)	Cumulative grade Fe (%)	Cumulative mineral grade Sphal (%)	Cumulative mineral grade Chalco (%)	Cumulative mineral grade Pyrite (%)	Cumulative mineral grade Gangue (%)	Cumulative recovery Sphal (%)	Cumulative recovery Chalco (%)	Cumulative recovery Pyrite (%)	Cumulative recovery Gangue (%)
Conc-1	17.6	5.20	22.70	29.60	7.75	65.56	20.73	5.96	7.4	53.8	13.0	3.2
Conc-2	27.3	5.60	21.81	29.42	8.35	62.99	22.03	6.64	12.4	80.2	21.4	5.6
Conc-3	33.5	6.33	20.44	29.38	9.44	59.04	24.52	7.01	17.2	92.5	29.3	7.3
Conc-4	38.4	7.50	18.61	29.28	11.19	53.75	27.76	7.30	23.4	96.4	38.0	8.7
Feed	100.0	12.31	7.41	19.56	18.35	21.40	28.03	32.23	100.0	100.0	100.0	100.0

11. APPENDIX 3 FALCONBRIDGE LIMITED (STRATHCONA MILL)

ON-SITE FLOTATION TESTS

TABLE E.1	Laboratory flotation test on Primary Rougher Feed using air (Test PRF-B).
TABLE E.2	Laboratory flotation test on Primary Rougher Feed using nitrogen (Test PRF-A).
TABLE F.1	Laboratory flotation test on Primary Rougher Feed using air (Test PRF-01).
TABLE F.2	Laboratory flotation test on Primary Rougher Feed using nitrogen (Test PRF-02).
TABLE F.3	Laboratory flotation test on Primary Rougher Feed using air (Test PRF-03).
TABLE F.4	Laboratory flotation test on Primary Rougher Feed using nitrogen (Test PRF-04).
TABLE F.5	Laboratory flotation test on Primary Rougher Feed using air (Test PRF-05).
TABLE F.6	Laboratory flotation test on Primary Rougher Feed using nitrogen (Test PRF-06).
TABLE F.7	Laboratory flotation test on Primary Rougher Feed using nitrogen (Test PRF-07).
TABLE F.8	Laboratory flotation test on Primary Rougher Feed using air (Test PRF-08).
TABLE F.9	Laboratory flotation test on Primary Rougher Feed using air and nitrogen (Test PRF-09).

TABLE F.10 Laboratory flotation test on Primary Rougher Feed using air (Test PRF-10).

FLOTATION TESTS AT McGILL

TABLE G.1 Laboratory flotation test according to Falconbridge procedure using nitrogen.

TABLE G.2 Laboratory flotation test according to Falconbridge procedure using air.

TABLE E.1 Results of laboratory flotation test on Primary Rougher Feed (Test PRF-B) at Strathcona mill.

LABORATORY DATA

Sample	Time	Solids		Gas	pH	ORP, mV	Oxygen	Grade (%)		
	(min)	(g)	(%)	used		(Ag/AgCl)	(ppm)	Ni	Cu	S
Feed	-	-	-	-	9.56	-18	0.80	-	-	-
Head(*)	2.0	2152.28	100.00	-	7.91	-260	1.40	1.32	0.72	10.95
Conc-1	2.5	211.95	9.85	Air	8.31	-10	1.60	6.60	2.96	34.10
Conc-2	3.0	83.47	3.88	Air	8.21	15	3.00	5.50	4.30	34.40
Conc-3	4.0	100.58	4.67	Air	8.13	38	4.70	3.70	2.67	33.30
Conc-4	6.0	85.28	3.96	Air	8.07	52	6.10	2.00	1.21	31.30
Tails	-	1671.00	77.64	-	-	-	-	0.26	0.11	4.46

(*) calculated, conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

	Cumul.	Cumulative grade			Cumulative mineral grade				Cumulative recovery			
Sample	solids (%)	Ni (%)	Cu (%)	S (%)	Pentla (%)	Chalco (%)	Pyrrho (%)	Gangue (%)	Pentla (%)	Chalco (%)	Pyrrho (%)	Gangue (%)
Conc-1	9.8	6.60	2.96	34.10	17.08	8.58	63.94	10.40	52.1	40.7	27.4	1.4
Conc-2	13.7	6.29	3.34	34.18	16.22	9.68	63.90	10.19	69.0	64.0	38.1	2.0
Conc-3	18.4	5.63	3.17	33.96	14.38	9.19	65.30	11.13	82.0	81.4	52.2	2.9
Conc-4	22.4	4.99	2.82	33.49	12.59	8.18	66.49	12.74	87.2	88.1	64.6	4.0
Feed	100.0	1.32	0.72	10.95	3.23	2.08	23.01	71.69	100.0	100.0	100.0	100.0

TABLE E.2 Results of laboratory flotation test on Primary Rougher Feed (Test PRF-A) at Strathcona mill.

LABORATORY DATA

Sample	Time (min)	Solids (g)	Solids (%)	Gas used	pH	ORP, mV (Ag/AgCl)	Oxygen (ppm)	Grade (%) Ni	Grade (%) Cu	Grade (%) S
Feed	-	-	-	-	7.88	-208	1.30	-	-	-
Feed(*)	5.0	-	-	-	7.88	-208	1.30	-	-	-
Head(**)	7.0	1896.55	100.00	-	7.91	-236	1.30	1.33	0.72	10.68
Conc-1	7.5	24.30	1.28	N2	7.99	-225	1.20	18.30	0.79	31.10
Conc-2	8.0	22.82	1.20	N2	7.98	-240	1.20	19.80	0.94	29.30
Conc-3	9.0	23.31	1.23	N2	7.99	-246	1.20	18.80	1.29	27.30
Conc-4	11.0	26.27	1.39	N2	7.99	-245	1.30	10.30	1.71	20.10
Tails	-	1799.85	94.90	-	-	-	-	0.51	0.69	9.82

(*) after 5 min nitrogen conditioning; (**) calculated, conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

Sample	Cumul. solids (%)	Cumulative grade Ni (%)	Cumulative grade Cu (%)	Cumulative grade S (%)	Cumulative mineral grade Pentla (%)	Cumulative mineral grade Chalco (%)	Cumulative mineral grade Pyrrho (%)	Cumulative mineral grade Gangue (%)	Cumulative recovery Pentla (%)	Cumulative recovery Chalco (%)	Cumulative recovery Pyrrho (%)	Cumulative recovery Gangue (%)
Conc-1	1.3	18.30	0.79	31.10	49.88	2.29	34.63	13.21	19.5	1.4	2.0	0.2
Conc-2	2.5	19.03	0.86	30.23	51.95	2.50	30.52	15.02	39.4	3.0	3.4	0.5
Conc-3	3.7	18.95	1.00	29.26	51.79	2.91	27.86	17.43	58.7	5.2	4.6	0.9
Conc-4	5.1	16.60	1.20	26.77	45.33	3.47	26.50	24.70	70.5	8.5	6.1	1.7
Feed	100.0	1.33	0.72	10.68	3.28	2.08	22.30	72.35	100.0	100.0	100.0	100.0

TABLE F.1 Results of laboratory flotation test on Primary Rougher Feed (Test PRF-01) at Strathcona mill.

LABORATORY DATA

Sample	Time (min)	Solids (g)	Solids (%)	Gas used	pH	ORP, mV (Ag/AgCl)	Oxygen (ppm)	Grade (%) Ni	Grade (%) Cu	Grade (%) S
Feed	-	-	-	-	9.20	67	0.70	-	-	-
Head(*)	2.0	2198.41	100.00	-	9.20	-64	0.70	0.98	0.10	32.74
Conc-1	2.5	215.87	9.82	Air	9.30	55	7.60	1.70	0.28	35.70
Conc-2	3.0	184.00	8.37	Air	9.10	67	8.20	1.34	0.16	35.40
Conc-3	4.0	181.82	8.27	Air	8.90	70	8.70	1.18	0.12	35.30
Conc-4	6.0	209.64	9.54	Air	8.70	73	9.20	1.04	0.09	35.50
Conc-5	8.0	236.96	10.78	Air	8.60	76	9.40	0.85	0.06	34.60
Tails	-	1170.12	53.23	-	-	-	-	0.78	0.06	30.50

(*) calculated, conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

Sample	Cumul. solids (%)	Cumulative grade Ni (%)	Cumulative grade Cu (%)	Cumulative grade S (%)	Cumulative mineral grade Pentla (%)	Cumulative mineral grade Chalco (%)	Cumulative mineral grade Pyrrho (%)	Cumulative mineral grade Gangue (%)	Cumulative recovery Pentla (%)	Cumulative recovery Chalco (%)	Cumulative recovery Pyrrho (%)	Cumulative recovery Gangue (%)
Conc-1	9.8	1.70	0.28	35.70	3.17	0.81	86.35	9.67	24.2	28.1	10.5	5.4
Conc-2	18.2	1.53	0.22	35.56	2.71	0.65	86.53	10.11	38.4	41.8	19.4	10.5
Conc-3	26.5	1.42	0.19	35.48	2.40	0.56	86.66	10.38	49.5	52.0	28.3	15.7
Conc-4	36.0	1.32	0.17	35.49	2.11	0.48	86.98	10.43	59.3	60	38.7	21.5
Conc-5	46.8	1.21	0.14	35.28	1.82	0.41	86.78	11.00	66.2	67.5	50.2	29.4
Feed	100.0	0.98	0.10	32.74	1.28	0.28	80.94	17.50	100.0	100.0	100.0	100.0

TABLE F.2 Results of laboratory flotation test on Primary Rougher Feed (Test PRF-02) at Strathcona mill.

LABORATORY DATA

Sample	Time (min)	Solids (g)	Solids (%)	Gas used	pH	ORP, mV (Ag/AgCl)	Oxygen (ppm)	Grade (%) Ni	Grade (%) Cu	Grade (%) S
Feed	-	-	-	-	9.28	-23	0.80	-	-	-
Feed(*)	5.0	-	-	-	9.26	-24	0.80	-	-	
Head(**)	7.0	2300.94	100.00	-	9.29	-82	0.70	0.96	0.10	31.42
Conc-1	7.5	10.63	0.46	N2	9.28	-84	0.30	6.05	1.32	23.70
Conc-2	8.0	6.85	0.30	N2	9.27	-84	0.20	7.43	1.35	24.30
Conc-3	9.0	9.35	0.41	N2	9.25	-84	0.20	7.07	1.17	24.00
Conc-4	11.0	10.40	0.45	N2	9.23	-85	0.20	4.94	0.95	23.60
Conc-5	13.0	5.56	0.24	N2	9.22	-86	0.20	4.20	1.00	25.50
Conc-6	15.0	5.39	0.23	N2	9.20	85	0.20	3.15	0.77	25.00
Conc-7	17.0	4.85	0.21	N2	9.16	-82	0.20	2.85	0.64	23.70
Conc-8	19.0	3.46	0.15	N2	9.16	-82	0.20	3.00	0.72	26.60
Tails	-	2244.45	97.54	-	-	-	-	0.85	0.08	31.60

(*) after 5 min nitrogen conditioning; (**) calculated, conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

Sample	Cumul. solids (%)	Cumulative grade Ni (%)	Cumulative grade Cu (%)	Cumulative grade S (%)	Cumulative mineral grade Pentla (%)	Cumulative mineral grade Chalco (%)	Cumulative mineral grade Pyrrho (%)	Cumulative mineral grade Gangue (%)	Cumulative recovery Pentla (%)	Cumulative recovery Chalco (%)	Cumulative recovery Pyrrho (%)	Cumulative recovery Gangue (%)
Conc-1	0.5	6.05	1.32	23.70	15.93	3.83	42.93	37.31	5.8	5.9	0.3	0.8
Conc-2	0.8	6.59	1.33	23.94	17.44	3.86	42.24	36.46	10.4	9.7	0.4	1.3
Conc-3	1.2	6.76	1.28	23.96	17.90	3.70	42.06	36.34	16.4	14.3	0.6	2.0
Conc-4	1.6	6.25	1.18	23.86	16.48	3.44	43.22	36.87	20.9	18.5	0.9	2.9
Conc-5	1.9	5.98	1.16	24.07	15.72	3.37	44.45	36.47	23.0	20.8	1.1	3.3
Conc-6	2.1	5.67	1.12	24.18	14.83	3.24	45.56	36.37	24.4	22.5	1.2	3.7
Conc-7	2.3	5.41	1.07	24.13	14.11	3.11	46.16	36.62	25.5	23.8	1.4	4.1
Conc-8	2.5	5.26	1.05	24.28	13.69	3.05	46.95	36.31	26.4	24.9	1.5	4.3
Feed	100.0	0.96	0.10	31.42	1.27	0.30	77.62	20.80	100.0	100.0	100.0	100.0

TABLE F.3 Results of laboratory flotation test on Primary Rougher Feed (Test PRF-03) at Strathcona mill.

LABORATORY DATA

Sample	Time	Solids		Gas	pH	ORP, mV	Oxygen	Grade (%)		
	(min)	(g)	(%)	used		(Ag/AgCl)	(ppm)	Ni	Cu	S
Feed	-	-	-	-	9.60	48	0.70	-	-	-
Head(*)	2.0	2751.03	100.00	-	9.70	-88	0.70	1.39	0.63	12.02
Conc-1	2.5	102.04	3.71	Air	9.90	-60	4.80	10.30	4.61	34.00
Conc-2	3.0	83.39	3.03	Air	9.95	-48	5.90	6.50	4.72	33.00
Conc-3	4.0	85.82	3.12	Air	9.90	23	7.30	6.20	3.52	34.00
Conc-4	6.0	67.05	2.44	Air	9.86	42	8.20	4.63	1.93	33.00
Tails	-	2412.73	87.70	-	-	-	-	0.58	0.18	9.00

(*) calculated, conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

	Cumul.	Cumulative grade			Cumulative mineral grade				Cumulative recovery			
Sample	solids	Ni	Cu	S	Pentla	Chalco	Pyrrho	Gangue	Pentla	Chalco	Pyrrho	Gangue
	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
Conc-1	3.7	10.30	4.61	34.00	27.52	13.37	50.81	8.31	30.1	27.2	7.3	0.4
Conc-2	6.7	8.59	4.66	33.55	22.75	13.51	53.51	10.22	45.2	49.9	14.0	1.0
Conc-3	9.9	7.83	4.30	33.69	20.61	12.47	56.57	10.35	59.9	67.4	21.6	1.5
Conc-4	12.3	7.20	3.83	33.56	18.82	11.11	58.91	11.16	68.3	74.9	28.1	2.0
Feed	100.0	1.39	0.63	12.02	3.39	1.82	25.78	69.01	100.0	100.0	100.0	100.0

TABLE F.4 Results of laboratory flotation test on Primary Rougher Feed (Test PRF-04) at Strathcona mill.

LABORATORY DATA

Sample	Time (min)	Solids (g)	Solids (%)	Gas used	pH	ORP, mV (Ag/AgCl)	Oxygen (ppm)	Grade (%) Ni	Grade (%) Cu	Grade (%) S
Feed	-	-	-	-	9.70	56	0.80	-	-	-
Feed(*)	5.0	-	-	-	9.74	51	0.70	-	-	-
Head(**)	7.0	2197.24	100.00	-	9.80	-72	0.70	1.05	0.52	9.50
Conc-1	7.5	15.20	0.69	N2	9.78	-83	0.70	12.00	7.18	26.30
Conc-2	8.0	15.55	0.71	N2	9.71	-83	0.70	14.20	5.18	28.70
Conc-3	9.0	14.02	0.64	N2	9.71	-83	0.70	10.90	4.67	23.20
Conc-4	11.0	18.30	0.83	N2	9.72	-83	0.70	6.43	3.37	15.70
Conc-5	13.0	39.96	1.82	N2	9.68	-77	0.70	13.00	3.09	26.40
Conc-6	15.0	13.03	0.59	N2	9.71	-76	0.70	4.33	1.93	11.00
Conc-7	17.0	29.81	1.36	N2	9.68	-70	0.70	7.68	2.91	26.00
Conc-8	19.0	5.47	0.25	N2	9.68	-67	0.70	3.68	1.81	11.40
Tails	-	2045.90	93.11	-	-	-	-	0.40	0.28	8.50

(*) after 5 min nitrogen conditioning; (**) calculated, conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

Sample	Cumul. solids (%)	Cumulative grade Ni (%)	Cumulative grade Cu (%)	Cumulative grade S (%)	Cumulative mineral grade Pentla (%)	Cumulative mineral grade Chalco (%)	Cumulative mineral grade Pyrrho (%)	Cumulative mineral grade Gangue (%)	Cumulative recovery Pentla (%)	Cumulative recovery Chalco (%)	Cumulative recovery Pyrrho (%)	Cumulative recovery Gangue (%)
Conc-1	0.7	12.00	7.18	26.30	32.74	20.82	20.58	25.86	8.9	9.6	0.7	0.2
Conc-2	1.4	13.11	6.17	27.51	35.75	17.89	23.69	22.66	19.6	16.7	1.6	0.4
Conc-3	2.0	12.42	5.70	26.16	33.85	16.53	23.07	26.54	27.1	22.5	2.3	0.7
Conc-4	2.9	10.68	5.02	23.13	29.09	14.57	21.12	35.21	32.8	27.9	3.0	1.3
Conc-5	4.7	11.58	4.27	24.40	31.52	12.39	24.20	31.88	58.0	38.8	5.5	2.0
Conc-6	5.3	10.77	4.01	22.89	29.30	11.63	22.94	36.13	60.7	41.0	5.9	2.5
Conc-7	6.6	10.14	3.79	23.53	27.49	10.98	26.61	34.92	71.6	48.7	8.6	3.1
Conc-8	6.9	9.90	3.71	23.09	26.86	10.77	26.22	36.15	72.6	49.5	8.8	3.3
Feed	100.0	1.05	0.52	9.50	2.55	1.50	20.45	75.51	100.0	100.0	100.0	100.0

TABLE F.5 Results of laboratory flotation test on Primary Rougher Feed (Test PRF-05) at Strathcona mill.

LABORATORY DATA

Sample	Time	Solids		Gas	pH	ORP, mV	Oxygen	Grade (%)		
	(min)	(g)	(%)	used		(Ag/AgCl)	(ppm)	Ni	Cu	S
Feed	-	-	-	-	9.80	92	1.00	-	-	-
Head(*)	2.0	2324.02	100.00	-	9.96	-52	0.90	1.20	0.54	12.61
Conc-1	2.5	88.79	3.82	Air	10.07	-25	9.10	7.73	5.50	34.10
Conc-2	3.0	61.52	2.65	Air	10.10	8	9.70	6.52	3.21	33.30
Conc-3	4.0	84.06	3.62	Air	10.05	52	10.10	5.23	2.08	32.80
Conc-4	6.0	149.57	6.44	Air	9.98	75	10.30	3.24	0.81	32.70
Tails	-	1940.08	83.48	-	-	-	-	0.40	0.14	8.55

(*) calculated, conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

	Cumul.	Cumulative grade			Cumulative mineral grade				Cumulative recovery			
Sample	solids	Ni	Cu	S	Pentla	Chalco	Pyrrho	Gangue	Pentla	Chalco	Pyrrho	Gangue
	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
Conc-1	3.8	7.73	5.50	34.10	20.36	15.95	54.75	8.94	27.6	39.0	7.5	0.5
Conc-2	6.5	7.23	4.56	33.77	18.94	13.23	57.49	10.33	43.5	54.7	13.3	1.0
Conc-3	10.1	6.52	3.67	33.42	16.91	10.65	60.57	11.87	60.6	68.7	21.8	1.8
Conc-4	16.5	5.24	2.56	33.14	13.29	7.42	65.71	13.58	78.0	78.3	38.8	3.3
Feed	100.0	1.20	0.54	12.61	2.82	1.56	27.98	67.64	100.0	100.0	100.0	100.0

TABLE F.6 Results of laboratory flotation test on Primary Rougher Feed (Test PRF-06) at Strathcona mill.

LABORATORY DATA

Sample	Time (min)	Solids (g)	Solids (%)	Gas used	pH	ORP, mV (Ag/AgCl)	Oxygen (ppm)	Grade (%) Ni	Grade (%) Cu	Grade (%) S
Feed	-	-	-	-	9.84	78	1.20	-	-	-
Feed(*)	5.0	-	-	-	9.84	66	0.70	-	-	-
Head(**)	7.0	1734.27	100.00	-	9.90	055	0.40	2.10	1.05	14.35
Conc-1	7.5	59.28	3.42	N2	9.88	-74	0.30	15.70	7.35	32.70
Conc-2	8.0	42.32	2.44	N2	9.85	-77	0.20	14.40	6.84	32.20
Conc-3	9.0	105.26	6.07	N2	9.83	-76	0.20	8.30	5.33	32.30
Conc-4	11.0	54.13	3.12	N2	9.82	-80	0.20	6.76	3.48	30.20
Conc-5	13.0	137.61	7.93	N2	9.80	-72	0.20	1.91	1.20	32.30
Conc-6	15.0	29.45	1.70	N2	9.81	-73	0.20	2.38	0.92	28.40
Conc-7	17.0	17.35	1.00	N2	9.81	-72	0.20	2.28	0.72	26.00
Conc-8	19.0	34.87	2.01	N2	9.80	-68	0.20	1.47	0.38	29.60
Tails	-	1254.00	72.31	-	-	-	-	0.35	0.10	7.80

(*) after 5 min nitrogen conditioning; (**) calculated, conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

Sample	Cumul. solids (%)	Cumulative grade Ni (%)	Cu (%)	S (%)	Cumulative mineral grade Pentla (%)	Chalco (%)	Pyrrho (%)	Gangue (%)	Cumulative recovery Pentla (%)	Chalco (%)	Pyrrho (%)	Gangue (%)
Conc-1	3.4	15.70	7.35	32.70	42.82	21.32	27.84	8.02	27.7	24.0	3.3	0.4
Conc-2	5.9	15.16	7.14	32.49	41.30	20.70	29.12	8.88	45.8	39.9	5.9	0.8
Conc-3	11.9	11.67	6.22	32.39	31.49	18.03	39.38	11.10	71.1	70.8	16.2	2.1
Conc-4	15.0	10.65	5.65	31.94	28.64	16.38	42.05	12.92	81.6	81.1	21.8	3.1
Conc-5	23.0	7.63	4.11	32.06	20.11	11.93	53.36	14.59	87.5	90.2	42.3	5.3
Conc-6	24.7	7.27	3.89	31.81	19.10	11.29	54.13	15.47	89.3	91.7	46.1	6.1
Conc-7	25.7	7.08	3.77	31.59	18.56	10.93	54.33	16.18	90.3	92.4	48.1	6.6
Conc-8	27.7	6.67	3.52	31.44	17.42	10.22	55.54	16.82	91.3	93.1	53.1	7.4
Feed	100.0	2.10	1.05	14.35	5.28	3.04	28.99	62.69	100.0	100.0	100.0	100.0

TABLE F.7 Results of laboratory flotation test on Primary Rougher Feed (Test PRF-07) at Strathcona mill.

LABORATORY DATA

Sample	Time (min)	Solids (g)	Solids (%)	Gas used	pH	ORP, mV (Ag/AgCl)	Oxygen (ppm)	Grade (%) Ni	Grade (%) Cu	Grade (%) S
Feed	-	-	-	-	9.77	65	0.70	-	-	-
Head(*)	5.0	1829.68	100.00	-	9.77	65	0.70	1.34	0.56	13.77
Conc-1	5.5	18.12	0.99	N2	9.79	70	0.27	14.00	10.60	31.50
Conc-2	6.0	10.09	0.55	N2	9.79	69	0.24	12.90	10.50	29.40
Conc-3	7.0	10.49	0.57	N2	9.77	68	0.22	11.40	9.12	26.70
Conc-4	9.0	10.56	0.58	N2	9.79	66	0.21	7.37	6.36	20.30
Head(**)	11.0	-	-	-	9.84	-46	0.32	-	-	-
Conc-5	11.5	17.14	0.94	N2	9.82	-60	0.24	12.10	2.08	34.90
Conc-6	12.0	13.34	0.73	N2	9.81	-64	0.22	12.30	2.18	34.50
Conc-7	13.0	16.88	0.92	N2	9.80	-65	0.21	11.50	2.00	31.70
Conc-8	15.0	39.91	2.18	N2	9.77	-63	0.24	8.13	1.50	31.00
Tails	-	1693.15	92.54	-	-	-	-	0.58	0.24	12.40

(*) calculated conditions after 5 min nitrogen conditioning

(**) conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

Sample	Cumul. solids (%)	Cumulative grade Ni (%)	Cumulative grade Cu (%)	Cumulative grade S (%)	Cumulative mineral grade Pentla (%)	Cumulative mineral grade Chalco (%)	Cumulative mineral grade Pyrrho (%)	Cumulative mineral grade Gangue (%)	Cumulative recovery Pentla (%)	Cumulative recovery Chalco (%)	Cumulative recovery Pyrrho (%)	Cumulative recovery Gangue (%)
Conc-1	1.0	14.00	10.60	31.50	38.26	30.74	20.36	10.64	12.0	18.7	0.7	0.2
Conc-2	1.5	13.61	10.56	30.75	37.19	30.64	19.45	12.72	18.1	29.1	1.0	0.3
Conc-3	2.1	13.01	10.17	29.65	35.55	29.50	19.06	15.90	23.8	38.4	1.3	0.5
Conc-4	2.7	11.80	9.36	27.65	32.22	27.13	18.87	21.78	27.4	44.9	1.7	0.9
Conc-5	3.6	11.88	7.48	29.52	32.26	21.68	28.31	17.74	37.1	48.4	3.4	1.0
Conc-6	4.4	11.95	6.59	30.35	32.39	19.11	32.55	15.95	44.7	51.2	4.6	1.1
Conc-7	5.3	11.87	5.79	30.59	32.12	16.79	35.41	15.69	53.7	54.5	6.1	1.3
Conc-8	7.5	10.78	4.54	30.71	29.00	13.15	41.50	16.36	68.5	60.4	10.1	1.9
Feed	100.0	1.34	0.56	13.77	3.16	1.63	30.53	64.68	100.0	100.0	100.0	100.0

TABLE F.8 Results of laboratory flotation test on Primary Rougher Feed (Test PRF-08) at Strathcona mill.

LABORATORY DATA

Sample	Time (min)	Solids (g)	Solids (%)	Gas used	pH	ORP, mV (Ag/AgCl)	Oxygen (ppm)	Grade (%) Ni	Grade (%) Cu	Grade (%) S
Feed	-	-	-	-	9.57	-10	0.60	-	-	-
Head(*)	2.0	3434.18	100.00	-	9.69	-150	0.40	1.30	0 78	12.32
Conc-1	2.5	87.20	2.54	Air	9.77	-103	5.90	11.50	7.57	35.00
Conc-2	3.0	74.46	2.17	Air	9.77	-58	9.50	7.84	7.39	33 70
Conc-3	4.0	83.68	2.44	Air	9.78	-46	10.20	6.68	4 74	34.00
Conc-4	6.0	93.34	2.72	Air	9.74	-34	10.30	5.10	2.55	33.40
Tails	-	3095.50	90.14	-	-	-	-	0.60	0.27	9 94

(*) calculated, conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

Sample	Cumul. solids (%)	Cumulative grade Ni (%)	Cumulative grade Cu (%)	Cumulative grade S (%)	Cumulative mineral grade Pentla (%)	Cumulative mineral grade Chalco (%)	Cumulative mineral grade Pyrrho (%)	Cumulative mineral grade Gangue (%)	Cumulative recovery Pentla (%)	Cumulative recovery Chalco (%)	Cumulative recovery Pyrrho (%)	Cumulative recovery Gangue (%)
Conc-1	2.5	11.50	7.57	35.00	30.97	21.95	42.92	4.16	25.1	24.6	4.1	0.2
Conc-2	4.7	9.81	7.49	34.40	26.27	21.71	45.54	6.48	39.5	45.1	8.1	0.4
Conc-3	7.1	8.75	6.55	34.26	23.24	19.00	50.10	7.67	53.0	59.9	13.6	0.8
Conc-4	9.9	7.74	5.45	34.03	20.39	15.80	54.68	9.14	64.2	68.8	20.5	1.3
Feed	100.0	1.30	0.78	12.32	3.13	2.26	26.35	68.25	100.0	100.0	100.0	100 0

TABLE F.9 Results of laboratory flotation test on Primary Rougher Feed (Test PRF-09) at Strathcona mill.

LABORATORY DATA

Sample	Time	Solids		Gas	pH	ORP, mV	Oxygen	Grade (%)		
	(min)	(g)	(%)	used		(Ag/AgCl)	(ppm)	Ni	Cu	S
Feed	-	-	-	-	9.56	-18	0.80	-	-	-
Feed(*)	5.0	-	-	-	9.56	-125	0.70	-	-	-
Head(**)	7.0	2545.82	100.00	-	9.60	-174	0.30	1.31	0.72	12.81
Conc-1	7.5	31.47	1.24	N2	9.56	-166	0.20	14.50	5.95	34.80
Conc-2	8.0	18.34	0.72	N2	9.55	-170	0.20	13.90	6.31	33.00
Conc-3	9.0	21.83	0.86	N2	9.54	-174	0.20	13.70	4.81	31.20
Conc-4	11.0	41.74	1.64	N2	9.52	-181	0.20	13.20	3.20	28.50
Conc-5	13.0	12.41	0.49	N2	9.53	-198	0.20	6.61	2.52	17.70
Conc-6	15.0	9.61	0.38	N2	9.52	-210	0.20	3.74	1.67	12.10
Conc-7	17.0	11.50	0.45	N2	9.51	-211	0.20	4.19	1.35	13.50
Conc-8	19.0	13.91	0.55	N2	9.50	-215	0.20	3.70	1.00	13.00
Conc-9	19.5	156.53	6.15	Air	9.65	-94	9.28	2.69	3.13	34.30
Conc-10	20.0	82.70	3.25	Air	9.68	-71	9.92	2.31	3.66	34.40
Conc-11	21.0	120.39	4.73	Air	9.63	-49	10.16	1.85	1.53	34.10
Conc-12	23.0	73.39	2.88	Air	9.61	-34	10.30	1.43	0.53	33.90
Tails	-	1952.00	76.67	-	-	-	-	0.31	0.10	6.95

(*) after 5 min nitrogen conditioning; (**) calculated, conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

Sample	Cumul. solids	Cumulative grade			Cumulative mineral grade				Cumulative recovery			
		Ni	Cu	S	Pentla	Chalco	Pyrrho	Gangue	Pentla	Chalco	Pyrrho	Gangue
	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
Conc-1	1.2	14.50	5.95	34.80	39.30	17.26	39.60	3.84	15.6	10.2	1.8	0.1
Conc-2	2.0	14.28	6.08	34.14	38.72	17.64	38.09	5.55	24.3	16.6	2.7	0.2
Conc-3	2.8	14.10	5.69	33.24	38.25	16.51	37.22	8.02	34.6	22.3	3.8	0.3
Conc-4	4.5	13.77	4.78	31.50	37.35	13.85	35.91	12.89	53.5	29.6	5.8	0.9
Conc-5	4.9	13.06	4.55	30.14	35.43	13.21	34.66	16.71	56.3	31.3	6.2	1.2
Conc-6	5.3	12.40	4.35	28.86	33.62	12.61	33.46	20.30	57.5	32.2	6.4	1.6
Conc-7	5.8	11.76	4.11	27.65	31.87	11.93	32.50	23.70	59.1	33.0	6.8	2.0
Conc-8	6.3	11.06	3.84	26.39	29.96	11.15	31.59	27.30	60.8	33.8	7.2	2.6
Conc-9	12.5	6.93	3.49	30.29	18.21	10.13	52.08	19.59	72.9	60.6	23.4	3.6
Conc-10	15.7	5.98	3.53	31.14	15.49	10.23	56.38	17.90	78.2	77.1	31.9	4.2
Conc-11	20.4	5.02	3.06	31.82	12.77	8.89	61.54	16.80	83.9	87.2	45.3	5.1
Conc-12	23.3	4.58	2.75	32.08	11.50	7.98	64.04	16.48	86.2	89.3	53.8	5.7
Feed	100.0	1.31	0.72	12.81	3.11	2.08	27.77	67.03	100.0	100.0	100.0	100.0

TABLE F.10 Results of laboratory flotation test on Primary Rougher Feed (Test PRF-10) at Strathcona mill.

LABORATORY DATA

Sample	Time (min)	Solids (g)	Solids (%)	Gas used	pH	ORP, mV (Ag/AgCl)	Oxygen (ppm)	Grade (%) Ni	Grade (%) Cu	Grade (%) S
Feed	-	-	-	-	9.58	-50	0.60	-	-	-
Head(*)	-	2613.87	100.00	-	9.58	-50	0.60	1.22	0.64	11.26
Conc-1	0.5	10.50	0.40	Air	9.58	-16	8.79	7.87	15.50	30.50
Conc-2	1.0	9.70	0.37	Air	9.46	-6	9.12	4.14	14.20	28.50
Conc-3	2.0	14.91	0.57	Air	9.40	2	9.63	3.19	11.90	26.70
Conc-4	4.0	16.50	0.63	Air	9.36	10	9.76	2.31	9.34	26.10
Head(**)	6.0	-	-	-	9.57	-109	1.06	-	-	-
Conc-5	6.5	84.03	3.21	Air	9.66	-70	9.45	8.82	4.83	34.80
Conc-6	7.0	56.72	2.17	Air	9.68	-52	10.00	7.54	2.66	34.40
Conc-7	8.0	70.92	2.71	Air	9.65	-31	10.11	5.63	1.46	34.40
Conc-8	10.0	143.29	5.48	Air	9.58	-18	10.24	3.33	0.87	33.50
Tails	-	2207.30	84.45	-	-	-	-	0.42	0.12	7.20

(*) calculated; (**) conditions after additions of xanthate and frother (2 min conditioning)

METALLURGICAL RESULTS

Sample	Cumul. solids (%)	Cumulative grade Ni (%)	Cumulative grade Cu (%)	Cumulative grade S (%)	Cumulative mineral grade Pentla (%)	Cumulative mineral grade Chalco (%)	Cumulative mineral grade Pyrrho (%)	Cumulative mineral grade Gangue (%)	Cumulative recovery Pentla (%)	Cumulative recovery Chalco (%)	Cumulative recovery Pyrrho (%)	Cumulative recovery Gangue (%)
Conc-1	0.4	7.87	15.50	30.50	21.36	44.95	19.45	14.24	2.9	9.7	0.3	0.1
Conc-2	0.8	6.08	14.88	29.54	16.36	43.14	22.78	17.72	4.3	17.9	0.7	0.2
Conc-3	1.3	4.85	13.61	28.33	12.92	39.47	25.83	21.78	5.9	28.4	1.4	0.4
Conc-4	2.0	4.04	12.25	27.62	10.62	35.51	29.42	24.44	7.2	37.6	2.4	0.7
Conc-5	5.2	7.00	7.65	32.07	18.50	22.19	45.72	13.59	32.8	61.7	9.8	1.0
Conc-6	7.4	7.16	6.18	32.76	18.85	17.92	50.90	12.33	47.4	70.7	15.5	1.3
Conc-7	10.1	6.75	4.91	33.20	17.62	14.23	56.27	11.87	60.6	76.8	23.4	1.7
Conc-8	15.6	5.54	3.49	33.30	14.18	10.11	63.02	12.69	75.3	84.3	40.5	2.8
Feed	100.0	1.22	0.64	11.26	2.93	1.87	24.22	70.99	100.0	100.0	100.0	100.0

TABLE G.1 Strathcona ore flotation tests at McGill University

LABORATORY DATA

Sample	Time	Solids		Gas	pH	ORP, mV	Oxygen		Grade (%)	
	(min)	(g)	(%)	used		(Ag/AgCl)	(ppm)	Cu	Ni	S
Feed(*)	-	999.90	100.00	-	9.00	52	0.36	0.86	1.32	10.89
Conc.1	0.50	99.80	9.98	N2	9.20	47	0.21	6.87	6.97	29.90
Conc.2	1.00	51.30	5.13	N2	9.00	47	0.23	1.24	3.82	29.40
Conc.3	2.00	38.80	3.88	N2	9.15	49	0.21	0.52	2.21	30.30
Conc.4	4.00	35.40	3.54	N2	9.20	51	0.19	0.32	1.74	31.60
Conc.5	5.00	60.20	6.02	N2	9.20	50	0.21	0.14	1.19	33.20
Conc.6	7.00	41.30	4.13	N2	9.00	48	0.25	0.16	1.18	29.40
Conc.7	9.00	19.00	1.90	N2	9.20	48	0.18	0.25	1.34	23.00
Conc.8	11.00	8.90	0.89	N2	9.10	51	0.19	0.35	1.38	13.00
Conc.9	13.00	8.20	0.82	N2	9.05	47	0.18	0.32	1.19	8.63
Tailings	-	637.00	63.71	-	-	-	-	0.08	0.17	0.42

METALLURGICAL RESULTS

Sample	Cumul.	Cumulative grade			Cumulative mineral grade			Cumulative recovery		
	solids	Cu	Ni	S	Chalco	Pyrrho	Pentla	Chalco	Pyrrho	Pentla
	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
Conc.1	9.98	6.87	6.97	29.90	19.86	42.28	18.48	79.98	18.76	57.09
Conc.2	15.11	4.96	5.90	29.73	14.33	49.27	15.41	87.40	33.10	72.06
Conc.3	18.99	4.05	5.15	29.85	11.71	53.66	13.25	89.76	45.30	77.88
Conc.4	22.53	3.47	4.61	30.12	10.02	57.13	11.71	91.08	57.22	81.68
Conc.5	28.55	2.76	3.89	30.77	7.99	62.27	9.63	92.06	79.03	85.11
Conc.6	32.68	2.44	3.55	30.60	7.04	63.48	8.67	92.83	92.22	87.65
Conc.7	34.58	2.32	3.43	30.18	6.69	63.00	8.34	93.39	96.85	89.26
Conc.8	35.47	2.27	3.37	29.75	6.55	62.15	8.21	93.75	98.00	90.17
Conc.9	36.29	2.22	3.33	29.27	6.42	61.16	8.09	94.06	98.67	90.92
Feed(*)	100.00	0.86	1.32	10.89	2.48	22.50	3.23	100.00	100.00	100.00

(*) Calculated

TABLE G.2 Strathcona ore flotation tests at McGill University

LABORATORY DATA

Sample	Time (min)	Solids (g)	Solids (%)	Gas used	pH	ORP, mV (Ag/AgCl)	Oxygen (ppm)	Grade (%) Cu	Grade (%) Ni	Grade (%) S
Feed(*)	-	1034.00	100.00	-	9.20	102	0.35	0.84	1.34	10.94
Conc.1	0.50	197.40	19.09	Air	9.27	112	8.40	3.69	4.85	32.90
Conc.2	1.00	32.40	3.13	Air	9.00	106	8.50	1.12	2.77	27.00
Conc.3	2.00	16.40	1.59	Air	9.30	105	8.45	1.03	2.27	22.80
Conc.4	4.00	24.80	2.40	Air	9.00	101	8.49	0.51	1.76	23.00
Conc.5	5.00	50.30	4.86	Air	9.15	103	8.42	0.23	1.11	30.60
Conc.6	7.00	28.00	2.71	Air	9.30	110	8.41	0.22	1.19	24.70
Conc.7	9.00	16.30	1.58	Air	9.16	110	8.34	0.29	1.25	19.70
Conc.8	11.00	7.10	0.69	Air	9.18	108	8.33	0.42	1.48	11.70
Conc.9	13.00	4.30	0.42	Air	9.10	106	8.28	0.55	1.55	8.92
Tailings	-	657.00	63.54	-	-	-	-	0.07	0.20	0.50

METALLURGICAL RESULTS

Sample	Cumul. solids (%)	Cumulative grade Cu (%)	Cumulative grade Ni (%)	Cumulative grade S (%)	Cumulative mineral grade Chalco (%)	Cumulative mineral grade Pyrrho (%)	Cumulative mineral grade Pentla (%)	Cumulative recovery Chalco (%)	Cumulative recovery Pyrrho (%)	Cumulative recovery Pentla (%)
Conc.1	19.09	3.69	4.85	32.90	10.66	63.07	12.27	83.92	53.24	70.98
Conc.2	22.22	3.33	4.56	32.07	9.62	62.57	11.47	88.10	61.49	77.24
Conc.3	23.81	3.17	4.40	31.45	9.18	61.75	11.06	90.04	65.01	79.83
Conc.4	26.21	2.93	4.16	30.68	8.47	60.97	10.41	91.50	70.65	82.68
Conc.5	31.07	2.51	3.68	30.67	7.25	63.14	9.05	92.83	86.75	85.24
Conc.6	33.78	2.32	3.48	30.19	6.72	62.86	8.50	93.54	93.89	87.07
Conc.7	35.36	2.23	3.38	29.72	6.46	62.13	8.24	94.09	97.14	88.33
Conc.8	36.04	2.20	3.35	29.38	6.36	61.43	8.15	94.43	97.91	89.09
Conc.9	36.46	2.18	3.33	29.14	6.30	60.93	8.11	94.70	98.23	89.59
Feed(*)	100.00	0.84	1.34	10.94	2.43	22.62	3.30	100.00	100.00	100.00

(*) Calculated