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TREATMENT OF OXIDE COPPER ORES

by

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† Computer sheets bound separately.

SUMMARY

Background

At discussions between Messrs R. Adam and R. Love, S.A. Department of Mines, and Messrs P. Dixon, J. Gooden, L. Goldney, J. Lackey and G.B. Warburton at Amdel, Thebarton on 6 October 1971, a programme to investigate the treatment of South Australian oxide copper ores was developed. Three projects were defined:

1. Collation of leaching data available in Amdel records on various copper minerals, with supplementary tests where required.
2. Electrowinning of copper.
3. Equipment for small-scale production operations.

The general scheme of treatment methods available for copper ores shown in Fig.1 was tabled, and in this project it was proposed to consider the three stages which are important for small-scale producers treating oxide ores, viz. leaching, solids-liquid separation and cementation. Other methods of copper recovery via solvent extraction and electrowinning would be beyond the reach of smaller producers who generally lack the required technical background and have not the larger financial resources required for the more sophisticated operations. Refining of the cement copper would be the function of a central refinery. (Note: LPF is considered in a separate Amdel Report to be published later.)

This Report gives the results of Projects 1 and 3 above, Project 2 having been separately reported (Amdel Report No.857).

Objectives

The objective of the first part of this project was to record the extensive unpublished experience accumulated at Amdel over the years in the leaching of a range of mineralogical types of copper ores.

The objective of the second part of the project was to outline treatment processes for South Australian oxidised copper ores which have sufficiently low capital and operating costs, and which can be operated with limited technical expertise, in order to be applicable by small-scale operations in South Australia.

During the course of the project, work was directed by the Department of Mines towards testing a range of specific ore samples which were supplied.

Summary of Work Done

The following work has been carried out:

(1) A review and summary of all data previously accumulated by Amdel in relation to the leaching of oxidised copper ores was prepared.

(2) An investigation of the agitation and percolation leaching of a series of South Australian oxidised copper ores with sulphuric acid in laboratory-scale tests has been carried out. Suitable leaching conditions for most of the various ore types tested were determined.

(3) A pilot-scale investigation of the agitation (mechanical and air) and percolation leaching of two selected South Australian ores was carried out. Sufficient data were obtained in this work and the work covered by (2) above to permit economic comparison of the various treatment methods. Solids-liquid separation of the leached pulp was investigated using pulp from the pilot-plant tests.

(4) A survey of the literature relating to the cementation of copper from leach solutions with iron was prepared.

(5) Capital and operating costs for the various processing alternatives were prepared. Comparison of the costs of each process were made and the sensitivity of these costs to changes in the most important variables examined.

Conclusions

(1) Both percolation and agitation leaching with sulphuric acid were successful for the treatment of the oxidised copper ores tested on a laboratory scale.

(2) Pilot-scale percolation leaching of both Mt Gunson and Paratoo ores achieved good leaching rates with low acid consumptions. Air Pachuca leaching on a pilot-scale was found to be unsatisfactory. Pilot-scale mechanical agitation leaching on ores ground to 80 to 90% minus 177 micrometres (85 mesh BSS) was satisfactory for both Mt Gunson and Paratoo ores.

(3) Settling tests on Mt Gunson and Paratoo ores showed rather poor settling characteristics; in particular, the low solids density of the settled slurry.

(4) Due to their simplicity and low installed cost, gravity flow launders appear to be the most attractive cementation equipment for small-scale copper producers, from examination of Amdel Report No 810.

(5) A general cost study on the leaching of a hypothetical ore showed that percolation leaching would be marginally more attractive than agitation leaching because of the lower capital requirements. Both methods gave about the same returns on investment. For a plant of 10 000 tonnes ore/annum operating over a period of 3 years, ore of at least 4% Cu grade would be required at a copper price of $1000/tonne. Similarly, for 20 000 tonnes/annum over 3 years, ore of at least 3% grade would be required. The minimum economic plant size is approximately halved if the price of copper rises by 20%. The effect of increasing plant life to 4 years has about half this effect.

Recommendations

While general treatment recommendations for South Australian oxide copper ores have been developed, it is still necessary, of course, to examine each orebody with regard to the following aspects:

(1) Ore tonnage.

(2) Average ore grade.

(3) Ease of mining the orebody.

(4) Ore crushing and grinding characteristics.

(5) Ore leaching characteristics for agitation and percolation operations.

(6) Leach residue washing, including settling and filtration characteristics of leached ground ore.

(7) Availability and on-site cost of chemicals (e.g., sulphuric acid and scrap iron).

(8) Availability of services (e.g., power and water).

(9) Availability of established infrastructure (housing and amenities).

1. INTRODUCTION

At discussions between Messrs R. Adam and R. Love, Department of Mines and Messrs P. Dixon, J. Gooden, L. Goldney, J. Lackey and B. Warburton of Amdel, a programme to investigate the treatment of South Australian oxide copper ores was developed. The routes available were set out as shown in Fig.1 and three projects were defined:

1. Collation of leaching data available in Amdel records on various copper minerals with supplementary tests where required.
2. Electrowinning of copper.
3. Equipment for small-scale production operations.

The general scheme of treatment methods available for copper ores shown in Fig.1 was tabled, and in this last project it was proposed to consider the three stages which are important for small-scale producers treating oxide ores, viz. leaching, solids-liquid separation and cementation. Other methods of copper recovery via solvent extraction and electrowinning would be beyond the reach of smaller producers who generally lack the required technical background and have not the larger financial recources required for the more sophisticated operations. Refining of the cement copper would be the function of a central refinery. During the course of the project, work was directed by the Department of Mines towards testing a range of specific ore samples which were supplied.

Parts A and B respectively of this Report give the results of Projects 1 and 3 above. Project 2 was reported separately (Amdel Report No.857 — 'Electrowinning of Copper', by G.B. Warburton, May 1972). (Note: LPF is considered in a separate Amdel Report to be published later.).

PART A: REVIEW OF LEACHING BEHAVIOUR OF OXIDE COPPER MINERALS

1. INTRODUCTION

As part of Amdel's work on treatment of oxidised copper ores for the South Australian Department of Mines it was considered desirable to collate the results of leaching tests carried out by Amdel on oxidised copper ores from a variety of sources.

Some data on the leaching of pure minerals have been obtained from the literature. This information is valuable but it must be borne in mind when dealing with an ore, that:

(a) the copper is usually present in two or more minerals,

(b) the copper minerals are perhaps only 1 or 2% of the ore and the reactivity of the gangue towards the leaching agent is important, and

(c) the particle size of the copper minerals and their location in the gangue will decide how much grinding is necessary to expose the minerals to the lixiviant. Finer grinding usually increases the ease of leaching of the copper but can also increase the reactivity of gangue minerals, resulting in both increased acid consumption and the presence of other elements in the leach liquor.

2. OXIDE COPPER MINERALS

The following are the oxidised copper minerals which most commonly occur and which were present in the ores described later:

Cuprite	Cu_2O		
Tenorite	CuO		
Malachite	$Cu_2(OH)_2$	or	$Cu(OH)_2.CuCO_3$
Azurite	$Cu_3(OH)_2(CO_3)_2$	or	$Cu(OH)_2.2CuCO_3$
Atacamite	$Cu_2(OH)_3Cl$	or	$3Cu(OH)_2.CuCl_2$
Chrysocolla	$CuSiO_3.2H_2O$		

van Arsdale (1953) has summarised some of the results obtained by Sullivan (1933) on acid leaching of copper minerals.

He states that all of these minerals (except cuprite) dissolve readily in dilute sulphuric acid according to the following equations:

Tenorite	$CuO + H_2SO_4 \longrightarrow Cu^{2+} + SO_4^{2-} + H_2O$
Malachite	$Cu(OH)_2.CuCO_3 + 2H_2SO_4 \longrightarrow 2Cu^{2+} + 2SO_4^{2-} + CO_2 + 3H_2O$
Azurite	$Cu(OH)_2.2CuCO_3 + 3H_2SO_4 \longrightarrow 3Cu^{2+} + 3SO_4^{2-} + 2CO_2 + 4H_2O$
Atacamite	$3Cu(OH)_2.CuCl_2 + 3H_2SO_4 \longrightarrow 4Cu^{2+} + 3SO_4^{2-} + 2Cl^- + 6H_2O$
Chrysocolla	$CuSiO_3.2H_2O + H_2SO_4 \longrightarrow Cu^{2+} + SO_4^{2-} + SiO_2 + 3H_2O$

Cuprite (cuprous oxide) is the only oxidised mineral which contains copper as copper (I). In the absence of oxidising agents it dissolves only partially, as expressed in the disproportionation reaction:

$$Cu_2O + H_2SO_4 \longrightarrow Cu^{2+} + SO_4^{2-} + Cu + H_2O$$

In presence of a suitable oxidising agent (e.g., air or iron (III) salts) all the copper may be dissolved.

$$Cu + 2H^+ + \tfrac{1}{2}O_2 \longrightarrow Cu^{2+} + H_2O, \quad \text{or}$$

$$Cu + 2Fe^{3+} \longrightarrow Cu^{2+} + 2Fe^{2+}$$

The results are summarised in Table 1.

3. LEACHING BEHAVIOUR OF PARTICULAR ORES

A number of copper ores are considered in turn. Their mineralogies are summarised and results of leaching tests are given.

In all cases *Acid Leaching refers to Sulphuric Acid* and for Cu^{2+} the minimum H_2SO_4 consumption is 1.54 kg/kg Cu. Acid consumption may be expressed in two forms:

(i) kg/tonne, which is kg acid per tonne of ore, and

(ii) kg/kg Cu, which is kg acid per kg copper extracted.

All leaching tests were conducted at ambient temperatures.

3.1 Ore I from the Roebourne-Port Hedland Area of Western Australia

The ore was generally fine-grained quartz. Opaque mineral occured in some cases as large irregular patches of disseminated grains, and sometimes filling the cracks and fissures of the quartz.

Opaque mineral was mainly cuprite with occasional relics of chalcocite extensively corroded and rimmed with cuprite. Some malachite was present as small patches and fissure fillings and some chrysocolla was also present with the malachite. Goethite was present in small amounts.

Copper minerals therefore were principally cuprite, with some malachite and a lesser quantity of chrysocolla.

A composite sample containing 1.6% Cu was used for both agitation and percolation leaching tests. The results are given in Table 2.

Because of the inability to achieve more than about 80% extraction in any test the residues were examined for copper minerals. The only copper mineral found was *Turquoise* $Cu(Al, Fe)_6(PO_4)_4(OH)_8.4H_2O$ and this may have accounted for most of the unleached copper.

3.2 Ore II from the Vicinity of Gladstone, Queensland

Copper was present in this ore largely as *Tenorite* with some *Malachite*. Gangue minerals were described as follows:

Predominantly silicates and goethite.

Abundant: Chlorite
Common: Clay minerals, sericite, epidote, goethite
Minor: Amphibole, quartz, feldspar, haematite
Trace: Carbonate (other than malachite)

The tenorite was mostly associated with or finely interground with goethite. It also occurred in numerous small veinlets, where it was free from iron oxides, along fractures through silicate rock fragments. Malachite was mostly very fine-grained, and occurred as small grains, patches and veinlets.

The results of acid leaching are given in Table 3.

3.3 Ore III from Burra, South Australia

The ore sample examined here was described as a composite of limonitic silts and quartzose breccias. Main minerals were quartz, alkali feldspar, chlorite and sericitic matrix.

Azurite and *Malachite* were the only identifiable copper minerals. They occurred principally as late infillings in cracks and in recrystallised quartz lenses.

The following analyses were reported:

Copper	1.58%
Iron	2.04%
Sulphide	0.11%
Sulphate	0.01%
Carbonate	0.34%

An extensive series of acid leaching tests was made.

Table 4 shows the effects of variable acid additions.

Table 5 shows the effect of particle size on the rate of leaching.

Ammonia leaching on this ore gave the following results:

Lixiviant	0.85M ammonia and 0.8M ammonium chloride		
Particle size	-300 µm (52#)	-2.05 mm (8#)	-2.05 mm (8#)
Time (h)	1	4	24
Extraction (%)	34	30	34

Ammonia leaching results were not good; although, from results obtained with similar ores, these extractions may have been increased to perhaps 50 to 70% by using 2 to 3M ammonium carbonate instead of ammonium chloride.

Acid leaching gave high recoveries at reasonable acid consumption: 90% extraction was possible using 100 lb acid per ton ore; i.e., 3.0 to 3.5 kg/kg Cu extracted. The relative contents of Cu, CO_2 and S in the head sample (i.e., low CO_2 relative to Cu and negligible S) suggested that at least 30 to 40% of the copper could not be present as carbonate (azurite or malachite) and was probably present as chrysocolla.

3.4 Ore IV, also from Burra, South Australia

Another sample of ore, also from the Burra area, was examined in leaching tests.

The mineralogy was similar to Ore III although there was considerably more carbonate gangue. This resulted in acid consumption being prohibitively high; consequently ammonia leaching was examined more closely. Results are shown in Table 6.

With ammonia leaching an overall extraction efficiency for the whole of this ore of 60 to 65% was found. Despite considerable variation in the head grade, the residues contained ~0.6% Cu.

Mineralogical examination of the residues indicated mainly limonitic stained quartz, clear quartz, minor green flaky micaceous silicates and limonitic grains dark brown/red or yellowish in colour.

Electron-probe microanalyses of one large silicate grain indicated 13% Cu. Several limonitic grains showed fairly uniform copper to 5.5%, and several composite grains of clay, silica and limonite contained up to 2% Cu.

It was concluded that about 50% of the residual unleached copper occured in limonite, possibly in atomic form, and the remaining 50% occurred as silicate — probably chrysocolla. The chrysocolla grains may have been partially leached.

3.5 Ore V from Mount Gunson, South Australia

Detailed mineralogy was not recorded for this ore, it being described merely as siliceous. A closer examination was made of the copper-containing minerals, which were estimated to be present in the following proportions:

Chrysocolla	73%
Atacamite	17%
Malachite	8%
Opaques	2%

Acid leaching data are given in Table 7.

These results show very high extraction of copper with low acid consumption.

3.6 Ore VI from Ukaparinga, South Australia

This ore was described as an intensely weathered and sheared paragneiss. Examination was difficult but a visual estimation from a thin section gave the following composition:

Quartz		35%
Clay minerals, muscovite	each	20%
Microcline, tourmaline	each	10%
Magnetite, zircon, apatite	Accessory	
Malachite, iron oxides, chlorite	Secondary	

Two samples were used for leaching tests, one from the surface and one from underground. The surface ore was more weathered and contained a larger proportion of chloritic material. The mined ore showed a predominance of quartz and feldspar and a lack of micaceous material.

Malachite was the only copper mineral identified. Acid leaching results from both ore samples are given in Table 8.

A later mineralogical examination showed that a considerable amount of copper was present in biotite, either in solid solution or as disseminated submicron inclusions of a discrete copper mineral. Probably there was more copper in this 'inaccessible' form in the surface ore than the underground ore.

It was subsequently learned that leaching with hot dilute acid was quite effective for this ore.

3.7 Ore VII from the Upper Flinders Ranges, South Australia

Detailed mineralogy of this ore was not available. Copper minerals present were *Azurite, Malachite* and *Chrysocolla*. Three samples, nominally minus 6.35 mm (-¼ in.), were subjected to agitation acid leaching, each at three different pH values. Results are given in Table 9.

Sample C was screened at 850 micrometres (18 mesh). The two fractions were analysed and leached separately as shown in Table 10.

The distribution of carbonate in size fractions corresponded with the copper distribution, suggesting that no carbonate minerals other than copper carbonates were present. This was supported by the relatively low acid consumptions. The actual ratios of carbonate to copper are less than the the theoretical for the accepted formulae of malachite and azurite.

Ammonia leaching of Samples A, B and C minus 6.35 mm (-¼ in.) was done with 2M ammonia/1M ammonium carbonate.

The following result was obtained:

	Sample		
	A	B	C
Time (h)	7.5	10	8
Extraction (%)	73	68	38

The poor extraction from Sample C with ammonia leaching indicated that much of the copper mineral present was silicate or was associated with gangue silicates. The better extractions from A and B supported the premise that they contained more carbonate copper and less silicate copper than did C.

3.8 Ore VIII

This deposit is located in the mid-north of South Australia. Seven pieces of drill-core examined had copper contents ranging from 0.67% Cu to 1.45% Cu.

The ore is composed of quartz, mica, chlorite and potash feldspar, as a yellow, fine-grained clayey matrix ($\geqslant$ 80% of the ore). Electron microprobe analysis showed that the copper was associated with iron, magnesium, aluminium and silicon, and it was tentatively concluded that the copper is substituted in the lattice of the mineral chlorite.

Leaching tests using sulphuric acid showed that the copper is relatively difficult to extract from this ore. After 7 h at pH 1 and ambient temperature, less than 10% of the copper had been leached, and acid consumption was high.

By leaching at elevated temperature the rate of copper extraction was greatly improved. At 80°C and pH 1, 95% extraction of copper was achieved in 7 h, but acid consumption was excessive (about 22 kg H_2SO_4 per kg Cu extracted).

Acid consumption was reduced to 5 kg H_2SO_4 per kg Cu extracted by raising the pH to 2. However, 11 h at 80°C was necessary to attain 80% extraction of copper.

PART B: SMALL-SCALE LEACHING OF SOUTH AUSTRALIAN OXIDE COPPER ORES

1. MATERIAL EXAMINED

Table 11 gives details of samples from the S.A. Department of Mines used in the laboratory-scale agitation and percolation leaching tests. Appendix E gives a mineralogical study of these samples.

For pilot-plant leaching tests two ore samples were used:

1 tonne of Paratoo ore,

1 tonne of Mt Gunson ore.

These samples were provided by the S.A. Department of Mines. The two large samples were not characterised prior to their use.

2. LABORATORY-SCALE LEACHING TESTS

Laboratory-scale agitation and percolation leaching tests with sulphuric acid were carried out on samples of ore of varying mineral types, selected by the S.A. Department of Mines as representative of South Australian oxide copper deposits.

2.1 Agitation Leaching: 1st Series

2.1.1 Sample Preparation

For agitation leaching, the samples were crushed and ground in stages to 100% minus 1.67 mm (10 mesh BSS) producing a minimum of fines. After each pass through the crushing or grinding circuit, plus 1.67 mm (10 mesh BSS) material was screened and recycled to the crushing circuit until all the material was ground to minus 1.67 mm (10 mesh BSS).

2.1.2 Procedure

A 500-g sample of ore, ground to minus 1.67 mm (10 mesh BSS), with an equal weight of water, was leached with sulphuric acid in a 1.5-litre beaker. The slurry was agitated with a stainless-steel impeller which was rotated fast enough to keep all the solids suspended. The pH of the solution was controlled either at 1 or 2 by adding concentrated sulphuric acid from a burette. Residue samples for controlled pH 1 leaches were taken at ½, 1, 2, 4 and 8 hours after commencement of leaching, and 2, 4 and 8 hours for controlled pH 2 leaches. Most of the leaches were stopped after 8 hours.

Samples A53 and A54, both from Dome Rock Mine, contained significant sulphide minerals (chalcocite), so oxidising agents (manganese dioxide and ferric sulphate) were added and pH was controlled at 1.

2.1.3 Results and Discussion

Table 12 reports the results of the leaching tests.

Leaching efficiencies were found to be satisfactory for all except the Dome Rock ore samples (A53, A54) and the Elsie Adair sample (A57) at pH 1. These contained chalcocite, which is much slower in leaching under the combined effects of an oxidising agent and acid. Sample A54 was controlled at pH 1 for the first hour and this leached rather rapidly. It was then controlled at pH 2 for the rest of the leach, but very little additional extraction of copper was achieved. This suggests that chalcocite needs to be leached with acid at least as strong as pH 1. Extraction efficiencies at pH 2 were much lower than at pH 1 for Samples A51, A52, A55 and A56.

For Samples A50, A51, A52, A55 and A56, acid consumption at pH 1 is not excessive, so that it is more satisfactory to leach at the higher acid concentration rather than at pH 2. This is supported by calculation of the 'net value' of the copper leached, i.e., value of copper (taken at refined copper price) less the cost of the acid consumed.

In Table 12 some of the samples indicate that copper content of the residues increases slightly with time — for example, in the pH 1 leach of Sample A51, the copper content of the residue increased from 0.24% to 0.29% between 2 hours and 4 hours leaching; in the pH 1 leach of Sample A53 (no oxidising agent), the copper content of the residue increased from 2.50% to 2.85% between 4 hours and 8 hours.

This can be explained by non-representative samples being withdrawn from the slurry during leaching. It is very difficult when only 10 to 20 g of solids are removed from 500 g of solids to get a completely representative sample. It is possible to overcome this problem by doing a series of leaches which are completed after, say ½ hour, 1 hour, 2 hours, 4 hours and 8 hours, and samples are taken from the final residues. However, this method is much more time-consuming and is unwarranted for first tests. In the consecutively-sampled tests done, the final assay, which is based on the total residue remaining, should be used for overall recovery calculations.

2.2 Agitation Leaching: 2nd Series

2.2.1 Introduction

In the agitation leaching tests described in Section 2.1 of this Report, standard grind sizes of 100% minus 1.67 mm (10 mesh BSS), and ambient temperature were used.

While these conditions were satisfactory for some of the leaches, it was considered that there might be some economic merit in operating at higher temperatures and finer grinds. Sample A56 was chosen as it leached more slowly than most of the others and had a reasonably high copper value in the residue (see Table 12).

Tests were carried out at two different grind sizes (100% minus 1.67 mm, or 10 mesh BSS, and 100% minus 300 micrometres, or 52 mesh BSS) and two different temperatures (20° C and 50° C).

2.2.2 Procedure

Four 500-g samples of A56 ore, two ground to minus 1.67 mm (10 mesh BSS) and two to minus 300 micrometres (52 mesh BSS), with equal weights of water, were leached with sulphuric acid in 1.5-litre beakers. The slurries were agitated with stainless-steel impellers which rotated fast enough to keep all the solids suspended. Two of the leaches were operated at room temperature (approximately 20° C) and two at 50° C (in a hot-water bath). The pH of the solution was controlled at 1.0 by adding concentrated sulphuric acid from a burette. Residue samples were taken at ½ hour, 1 hour, 2 hours and 4 hours after the commencement of the leach.

2.2.3 Results and Discussion

Table 13 and Fig.2 present the results of the leaches. Finer grinding of this ore from minus 1.67 mm (10 mesh BSS) to minus 300 micrometres (52 mesh BSS) increases the leaching rate and reduces the final residue value by the greater liberation or exposure of the copper minerals. Similarly, increasing the temperature also increases the leaching rate, but the effect is less pronounced at the finer grind size.

These tests show that, for this ore, a grind finer than minus 1.67 mm (10 mesh BSS) results in higher copper extractions and decreased leaching times. In using a grind finer than minus 1.67 mm (10 mesh BSS) it is probably not worth increasing the temperature. Although grind size with this material is important, other ores treated (Section 2.1) leached satisfactorily at the coarser grind.

Acid consumptions were very low, with only a maximum of 25% excess acid being consumed with the finer grind at the higher temperature. There is little acid-consuming gangue in Sample A56.

Some of the sample assays indicate that the copper content of the residues increases slightly with time. This is probably due to the difficulty in withdrawing representative samples from the slurry during leaching. This is more fully discussed elsewhere (Section 2.1).

2.3 Agitation Leaching: 3rd Series

2.3.1 Introduction

Batch leaching by small operators who have the minimum of equipment, manpower and expertise is best carried out as simply as possible. Leaching without pH control was therefore included in the present study. Tests were carried out on the one-tonne samples of two ores — Paratoo and Mt Gunson, three leaching tests being carried out at ambient temperature on each:

(i) Controlled pH 1.0.

(ii) Total acid: one addition at commencement of leach; acid requirement same as (i).

(iii) Repeat (ii) with 50% additional acid.

2.3.2 Procedure

(i) The same procedure as in Section 2.2.2 was used with the solids being ground to 100% minus 1.67 mm (10 mesh BSS) and leached at ambient temperature.

(ii) The acid consumption was calculated from (i) and this quantity of acid was added to the ore slurry in one addition at the start of the leach.

(iii) The same procedure was used as (ii) except 50% additional acid was added at the start of the leach.

2.3.3 Results and Discussion

Table 14 and Figs 3 to 6 present the results. These show that both ores leached more quickly when all the acid was added at the start of the leach. Approximately the same residue values were obtained for the three leaches of each ore, indicating that it is unnecessary to add any additional acid above that used in (i) and (ii).

The single-acid-addition method for batch leaching is acceptable for both these ores and offers a much simpler operation commercially. The acid-consuming gangue is attacked in both ores at low free acidities as well as at high free acidities, with no significant increase in attack at higher acidities. This would not be true of all ores and each ore would have to be tested individually.

Based on the above results, the single acid addition method was used for leaching Paratoo and Mt Gunson ores on the pilot-plant scale.

2.4 Percolation Leaching

2.4.1 Sample Preparation

For percolation leaching, samples were crushed to 100% minus 6.35 mm (¼ in.) in a jaw crusher. All minus 300 micrometres (52 mesh BSS) material was removed from the sample and only the minus 6.35 mm (¼ in.), plus 300 micrometres (52 mesh BSS) material used in percolation-leaching tests. (To minimise production of minus 300 micrometres (52 mesh BSS) material in the crushing cycle, the minus 6.35 mm (¼ in.) fraction was screened out and the plus 6.35 mm (¼ in.) fraction recycled to the jaw crusher until all the sample was minus 6.35 mm (¼ in.).) Table 15 shows the proportion of plus 300 micrometres (52 mesh BSS) material to minus 300 micrometres (52 mesh BSS) material.

2.4.2 Procedure

150 g of minus 6.35 mm (¼ in.) plus 300 micrometres (52 mesh BSS) ore was placed in a glass column with 250 ml of water. Sulphuric acid was added to the liquor, controlling the pH at 1. The liquor was circulated by an air lift so that it flowed downwards through the packed bed of solids. Only A50, A51, A52, A53, A55 and A56 were percolation-leached over a period of about 48 hours.

2.4.3 Results and Discussion

Table 16 details the results. In no case did the solids break down during leaching to cause a blockage of liquor flow, and all showed acceptable extraction rates except for A53 which contains copper sulphides (chalcocite). (No oxidising agent was used.)

Acid consumption was not recorded as it was considered that this would be similar to the agitation-leaching tests.

It appears, from the results, that percolation leaching with sulphuric acid was successful for the oxide copper ores on a laboratory scale. While agitation leaching can be scaled-up to full-size equipment with reasonable confidence of obtaining similar results to the laboratory-scale leaching tests, for percolation leaching the wall effects and bed weight can produce different results in the larger equipment due to particle breakdown, bed blockage or short-circuiting of the recycle liquor.

Hence larger-scale percolation leaching tests need to be carried out before the success of this method can be fully established.

3. PILOT-SCALE LEACHING TESTS

3.0 Equipment

Crushing:	Denver Crushing rolls	
Grinding:	0.3-m (1-ft) dia. × 0.6 m (2 ft) rod mill.	
Leaching:	(1)	360-l (80-gal.) capacity rubber-lined Pachuca, dia. 0.6 m (2 ft); operating height 1.5 m (5 ft).
	(2)	360-l (80-gal.) capacity rubber-lined vessel with stainless-steel agitator, dia. 0.71 m (2.3 ft). for mechanical agitation.
	(3)	0.3-m (1-ft) dia. × 1.8 m (6 ft) high perspex column for percolation leaching.

3.1 Percolation Leaching

3.1.1 Introduction

As described in Section 2.4.3 of this Report, a series of laboratory-scale percolation-leaching tests showed that satisfactory leaching efficiencies could be obtained by this means of leaching a number of oxide copper ores with sulphuric acid. The tests indicated that, on laboratory scale, there was no significant breakdown of solids during leaching which would obstruct liquor flow.

There were doubts as to whether the ores tested on laboratory scale would percolate satisfactorily on a pilot-plant scale, so two ores (one comparatively hard, Mt Gunson, and one comparatively soft, Paratoo) were chosen for larger-scale percolation tests. The objectives were to check the minimum cut-off size required for percolation of these two ores of different character, and to verify the laboratory-scale results for circulation and leaching rates.

3.1.2 Procedure

A 150 to 200 kg sample of each ore was crushed to 100% minus 6.35 mm (¼ in.) in a jaw crusher. Paratoo samples were screened at 350 micrometres (44 mesh BSS), 252 micrometres (60 mesh BSS) and 177 micrometres (85 mesh BSS), with the undersize being discarded. The Mt Gunson sample was screened at 177 micrometres (85 mesh BSS) discarding the undersize.

The oversize material was placed into a 0.30 m (12 in.) diameter × 1.83 m (6 ft) high perspex column. The solids were flooded with water and the liquor circulated by an air lift. Concentrated sulphuric acid was added to control the pH at about 1. The leach was run continuously for one week. Liquor samples were taken, and flow-rates measured, daily.

3.1.3 Results and Discussion

Tables 17, 18 and 19 and Figs 7 and 8 show the results of percolation leaching. All leached satisfactorily (Table 18 shows final extractions and Fig.8 shows the variation of extraction with time). Table 17 indicates the proportion of the ore discarded in the undersize, showing that a significant amount of the Paratoo ore is in this undersize, but that at the lower split of 177 micrometres (85 mesh BSS) the discarded fraction was reduced significantly (26.0% at minus 350 micrometres (44 mesh BSS), 19.5% at minus 252 micrometres (60 mesh BSS) and 15.4% at minus 177 micrometres (85 mesh BSS)). Since leaching was satisfactory in all these tests it may be feasible to split at even finer screen sizes and still get satisfactory liquor circulation and leaching rates. This would reduce the quantity of the discarded fraction even further.

The liquor circulation flow-rates, which give no overall pattern, are shown in Fig.7. Paratoo L1 (split at 350 micrometres (44 mesh BSS)) contained coarser material than Paratoo L2 (split at 252 micrometres (60 mesh BSS)) yet the flow-rate was lower. It was considered that the solids were not properly distributed in the column and that there was a partial concentration of fines in the upper region of the column. This caused a lowering in the liquor circulation rate. The extraction rates of copper with time are given in Fig.8, and again no overall pattern is evident from the three Paratoo leaches. This indicates that the extraction is not a function of liquor flow-rate, within the flow-rate ranges obtained in this study, and considerably lower liquor circulation rates would be acceptable.

Acid consumptions (Table 19) in the percolation leaching were considerably lower than for agitation leaching.

The liquor circulation rates in all leaches were satisfactory, so it is not possible to predict a limiting minimum particle size for percolation leaching. However, both ores percolated satisfactorily at a size of plus 177 micrometres (85 mesh BSS). This gives a small increase in the percentage of the total ore available for percolation compared with the quantity with a size cut-off of plus 300 micrometres (52 mesh BSS) as used in previous tests.

Operations in the Western United States where vat leaching is used either separate the slimes (Inspiration Consolidated Copper Co. where slimes of 85% minus 76 micrometres (200 mesh BSS) material are removed by rake classifiers and cyclones) or agglomerate the fines (Anaconda Co. at Weed Heights — see Appendix D).

The acceptable particle size range for percolation leaching is dependent on ore characteristics, and if leaching does not cause excessive particle breakdown it is probably only necessary to remove the slimes (minus 76 micrometres (200 mesh BSS)). For satisfactory percolation leaching on different ores it would be necessary to test each ore for breakdown during leaching.

As the percolation leaching proved so satisfactory with good leaching efficiency and low acid consumption for Paratoo and Mt Gunson ores, it is recommended that further work be considered. The biggest drawback with this process is the ore undersize wastage. Percolation techniques which will enable more of the fine ore fraction to be used need investigation — for example, liquor upflow through the bed of solids or pulsed liquor flow.

3.2 Pachuca and Mechanical Agitation Leaching

3.2.1 Introduction

A comparison of leaching by mechanical agitation and air-agitated Pachuca was made. It was intended to compare the power requirements for each type of equipment against the leaching efficiency.

Section 2.3 indicated that efficient leaching of copper could be achieved without fine-grinding of the ore. Since this would cut down the equipment required by the small producer, leaches on crushed but unground ore were carried out.

3.2.2 Procedure

Leaching was carried out on 6 samples (3 each of Mt Gunson and Paratoo). Four of the leaches (L1 to L4) were on material that was crushed in a rolls crusher to 100% minus 1.40 mm (12 mesh BSS) and the other two (L5 and L6) on material that was crushed to 100% minus 1.40 mm (12 mesh BSS) and then ground in a rod mill to nominally 50% minus 76 micrometres (200 mesh BSS). Sizings were taken on these samples. Each leach on Paratoo ores was duplicated using Mt Gunson ore.

Batch Mechanical Agitation Leach L1 and L3. A sample of 200 kg (450 lb) of ore was added to water to give a 50% solids slurry (w/w) in a 360-litre (80 gallon) mixing vessel which was mechanically agitated. A single addition of acid was added at the leach commencement. The quantity of concentrated sulphuric acid required was determined from work in Section 2.3 on laboratory-scale leaching. The stirrer speed was controlled at a speed at which adequate mixing was observed. A turbulent motion on the surface of the slurry was used as a guide to adequate mixing.

Samples of the slurry were taken at regular intervals (2, 4 and 8 hours) for settling tests — see Section 3.3. Residue samples were also taken at ½, 1, 2, 4 and 8 hours. At the conclusion of the leach, power requirements were measured over a wide range of stirring speeds by a spring balance connected to the vessel which was mounted on a turntable. Measurements of pH were also made throughout the run.

Batch Pachuca Leaches L2 and L4. A sample of 200 kg (450 lb) of ore was loaded into a 360-litre (80-gallon) capacity Pachuca containing an equal quantity of water to give a solids content of the slurry of 50% w/w. The same method of acid addition as used for leaches L1 and L3 was used — i.e., all acid added at the start of the leach. Circulation of the slurry in the Pachuca was maintained by an airlift in a draught tube. The air flow-rate was maintained so that a satisfactory level of slurry circulation was achieved.

Samples of the slurry were taken at 2, 4 and 8 hours for settling tests — see Section 3.3. Residue samples were taken and pH measurements made at ½, 1, 2, 4 and 8 hours.

Batch Mechanical Agitation Leaches L5 and L6. Except for acid addition, the same conditions of leaching were used for these leaches as for L1 and L3.

The same amount of acid was added at the beginning of the leach as in L1 and L3 but, with the ore ground finer, more acid was consumed by gangue material. Additional acid was added during the leach to control the acidity at pH 1.

3.2.3 Results and Discussion

Tables 20 to 23 show the sizing analyses of ores used in the leaches L1 to L6. Mt Gunson ore needed much more grinding than Paratoo ore to achieve the required 50% minus 76 micrometres (200 mesh BSS); since in the crushed ore only 15% of the former ore was in this size range whereas about 35% of the latter was in this range.

Table 25 and Figs 10 and 11 show the leaching characteristics of the two ores under the different conditions, and Table 26 shows the pH of the leaching Test L1 to L4 as a function of time. Very little difference in leaching efficiency was obtained in all the tests in which the variables were either mechanical or air agitation and coarse or finely ground ore. This highlights the fact that, in this size range, the rate of leaching of copper from these ores is rapid and is not the limiting factor in choosing a particle size range of ore for leaching or the method of agitation.

This satisfactory leaching of fairly coarse material confirmed the indications obtained at laboratory scale (Section 2.3, where the same size range as is given in Tables 20 and 22 was used).

Table 24 and Fig.9 show the power requirements for mechanical agitation. While the power-requirement curves for L1 and L6 leaches of Paratoo ore do not show any great variation, a much lower stirrer speed gave adequate mixing with the ground sample. A similar result was found for the Mt Gunson ore where it was possible to reduce the power requirement by a factor of three with the ground ore. The absolute values of kilowatt/kilolitre, although slightly high due to the wall effects in the small 360-l (80-gal.) test tank, are a good indication of the comparative power requirement.

A commonly-used particle sizing requirement in agitated mixing vessels and Pachucas is 100% minus 1.67 or 1.40 millimetres (10 or 12 mesh BSS) and 50% minus 76 micrometres (200 mesh BSS). The reason for having this size range is usually because of leaching characteristics. However, it was found with the unground samples (Tables 20 and 22), which contained considerably less minus 76 micrometres (200 mesh BSS) material, that handling problems occurred due to difficulty in keeping the ore suspended. The handling difficulties were not because of the low amount of minus 76 micrometres (200 mesh BSS) material but because of the greater-than-tolerable amount of material in the range of 1.40 mm to 177 micrometres (12 to 85 mesh BSS). It is believed that a sizing criterion of 50% minus 76 micrometres (200 mesh BSS) puts the emphasis on the wrong part of the sizing analysis. A minimum of 80 to 90% minus 177 micrometres (85 mesh BSS) and 100% minus 1.67 or 1.40 millimetres (10 or 12 mesh BSS) would be more applicable for solids suspension in agitated systems. It should be emphasised that, where grinding is used, and given 50% minus 76 micrometres (200 mesh BSS) as a requirement, the requirement of 80 to 90% minus 177 micrometres (85 mesh BSS) should also be met. However, there are cases where this might not be so; for example, sedimentary type ores where the gangue is composed of several distinct mineral phases differing in physical properties.

Physical handling problems were experienced on the unground ore leaches L1 to L4 (L1 and L3 — mechanical agitation; L2 and L4 — air agitation).

In Paratoo ore, there tends to be a variation in copper content with particle size, the larger particles containing a higher percentage of copper. In the mechanically agitated leaches, the size of solids in samples taken from the upper portion of the vessel was finer than the average particle size, as is indicated by the head sample (2.10% Cu). This indicates that the larger particles were circulating around the bottom of the vessel and not uniformly throughout the vessel.

Pachuca leach L2 operated quite differently. Here, difficulty was experienced with physical handling,and generally slurry circulation did not function satisfactorily because of the large amount of solids that settled to the bottom. Blocking of the bottom of the draught tube by solids occurred and the air-flow stopped. An additional airhose had to be inserted into the cone region to stir the settled solids and allow circulation to recommence.

The slurry recirculated was not typical of the vessel's contents as it contained a much higher than average solids to liquor ratio, and the solids were much coarser than average. The circulation pattern was characterised by the larger particles at the bottom of the cone being forced up the draught tube and discharged at the top. These particles then settled to the bottom very quickly where they were recirculated.

Finer particles would be circulated too, but at a much slower rate. The air-rates required to circulate this coarse slurry were far in excess of realistic commercial operating conditions. This parallels the greater power requirements for mechanical agitation on the unground ore. Samples were taken of the circulating stream which contained a higher proportion of coarse ore. The leaching rate of the coarser fraction for both L2 and L4 did not appear to be much less than the finer particles when comparing L1 to L2, and L3 to L4 in Figs 10 and 11.

Because Mt Gunson ore was even coarser than Paratoo ore in the unground leaches, even more trouble was experienced with this ore. In leach L3, the impeller did not have sufficient power to lift the settled material off the bottom of the tank and this material was obviously far too coarse for satisfactory operations. A power failure would cause the solids to settle and an air or water lance would have to be used before mixing could be restarted in the vessel.

Pachuca leach L4 required more air to drive the slurry up the draught tube than L2 and was generally more temperamental. Air stoppages twice caused the solids to settle and it took a considerable amount of time to get the slurry recirculating. Remarks made about the operations of L2 are equally applicable to L4. Leaches L5 and L6 operated without any problems and these represent conditions which would be needed for commercial operations.

Pachuca leaching does not appear feasible on a batch scale with unground ore. Finely ground ore would be required if Pachucas are considered, but it is recommended that mechanical agitation be used in preference to air agitation.

3.3 Settling Tests

3.3.1 Procedure and Results

Samples were taken from each leaching test after 2, 4 and 8 hours, except for the Mt Gunson ore Pachuca leaching test and the 8-h sample on the ground Paratoo ore mechanical agitation leaching tests. Normally, 4 samples were taken at each time in 250-ml beakers, the volume of the samples being of 150 to 200 ml.

Each sample was washed into a weighed 500-ml measuring cylinder which was then filled to approximately 500 ml with tap water. After being shaken to thoroughly mix the contents, each cylinder was slowly inverted six times over a period of one minute. During this period the desired dose of flocculant was added, usually in three additions. The cylinder was then set down, the stopper removed, and the level of the interface between clear liquid and pulp noted at various time-intervals. No flocculant was added to one cylinder in each batch, which acted as a reference standard. The remaining cylinders in each batch were treated with different proportions of flocculant in order to obtain an indication of the optimum quantity of flocculant required. In all tests, Superfloc 16, which is recognised as effective in acid pulps, was the flocculant used. No other types were tested, although some may be more effective under some conditions.

The interface level after settling for one hour was noted, together with the total clear liquor volume. The pulp was then filtered, dried and weighed. The pulp density before and after settling, the filter cake moisture content, and the flocculant addition rates were calculated, the results being shown in Tables 27 to 40. Settling curves for each test are shown in Figs 12 to 25. A summary of the results from the settling tests is given in Table 41.

3.3.2 Discussion

Examination of the pulp densities achieved after settling for one hour shows that, with the exception of the Paratoo Leach No.1, increased time of agitation gave a lower settled pulp density. In the case of the Paratoo Leach No.1 there was no significant change in pulp density with leaching time. For both ore types, the more-highly-ground sample, agitated mechanically, gave better settling results than the lesser-ground sample. Settling tests on the two samples from the Pachuca were not comparable with those from mechanical agitation, as they contained far more coarse material than the samples from the mechanically-agitated leaches. The difference in material size in the samples taken from the top of the suspensions is no doubt due to the mechanical agitator's inability thoroughly to disperse the coarse particles throughout the pulp.

The percentage of dissolved copper which can be recovered from the leached pulp is a function of the dilution of the pulp and the density of the settled solids after flocculation. Curves have been drawn (Fig.26) showing the recovery at various typical dilutions and pulp densities.

The recovery from, say, the Paratoo Leach No.1, which was diluted to 15% solids w/w and settled to 35% solids w/w (Table 28), would therefore be 67%. If the settled pulp were rediluted, re-flocculated and resettled, a further amount of dissolved copper would be recovered. Curves showing the total recovery after initial dilution, settling and decantation of the clear liquor, followed by re-dilution, settling and decantation are given in Fig.27. In the example given above, recovery would be increased to 89%.

However, it should be noted that this wash considerably dilutes the copper concentration, unless the wash liquor is kept separate from the initial dilution liquor. If this is done, the wash liquor may then be used as the dilution liquor for the next batch of leached ore. This would have the effect of raising the copper concentration in the following wash liquor, and the total recovery would be slightly better.

3.4 Discussion and Conclusions

Percolation leaching of both Mt Gunson and Paratoo ores achieved good leaching rates with low acid consumptions. More work is recommended to find ways of using all the ore rather than just the coarser material. This could include a more fundamental approach in determining some design criteria for percolation leaching — for example, finding the lower limit of liquor circulation which produces satisfactory leaching of the ore.

Air Pachuca leaching on the batch scale was found to be unsatisfactory due to the difficulty of keeping solids circulated. Mechanical agitation on ores ground to 80 to 90% minus 177 micrometres (85 mesh BSS) was satisfactory for both Mt Gunson and Paratoo ores. However, on both these ores it appears from the pilot-scale tests that percolation would be preferred.

Settling data are reported for Mt Gunson and Paratoo ores. These show rather poor settling characteristics — in particular, the low solids density of the settled slurry. Larger flocculant dosages, or alternatively different types of flocculant, could improve the settling characteristics.

4. SURVEY OF CEMENTATION EQUIPMENT

The most common method of precipitating copper from acid leach solutions has been the gravity flow launder charged with scrap iron as the precipitant. As a general figure, this type of plant requires 159 metres (500 ft) of launder, 1.27 metres (4 ft) wide × 1.27 metres (4 ft) deep, to process 3630 litres (800 gal) per minute, of copper-bearing solution. A launder this size can effectively recover over 90% of the copper in solution.

Three chemical reactions, each of which consumes iron, have long been recognised as of importance in copper cementation. These reactions may be shown by the following equations:

$$CuSO_4 + Fe \rightarrow Cu + FeSO_4 \quad .. \quad .. \quad .. \quad (1)$$

$$Fe_2(SO_4)_3 + Fe \rightarrow 3FeSO_4 \quad .. \quad .. \quad .. \quad (2)$$

$$H_2SO_4 + Fe \rightarrow FeSO_4 + H_2 \quad .. \quad .. \quad .. \quad (3)$$

The capital and operating costs of launders suitable for the small-scale operator have been set out in Amdel Report No.810 'Economics of Hydrometallurgical Recovery of Copper' by J.M. Clayton and R.E. Wilmshurst. The relevant section on cementation is reproduced in Appendix A to this Report.

More sophisticated types of cementation equipment — namely cones, drum precipitators and activated launders — are described in Appendix B and Appendix C to Amdel Report No.793 'Evaluation of Kapunda Ore, Part II', by L.H. Goldney. The appendixes are reproduced as Appendix B and C to this Report.

Pages 32 to 44 of I.C. 8341, U.S. Bureau of Mines 'Copper Leaching Practices in the Western United States' present a good coverage of cementation. This extract is reproduced as Appendix D to this Report.

5. ECONOMICS OF OXIDE COPPER ORE TREATMENT

5.1 Introduction

Laboratory- and pilot-scale leaching tests for the extraction of copper from oxide copper ores with sulphuric acid by mechanical agitation and by percolation have shown that both methods are applicable for most ores tested in the course of this project. An economic comparison of the methods for commercial-scale operations by a small producer is made in this Section. This examination is a general comparison, as no specific ore is considered and some assumptions on ore grade, location, crushing, grinding, leaching and cementation characteristics are made.

The object is to consider the simplest types of equipment and cheapest possible way in which to operate a plant on a small scale.

5.2 Assumptions

As this is a general comparison with no specific ore deposit considered, a number of assumptions must be made for costing purposes:

(1) The operation is near a small town where houses can be rented or some suitable accommodation arrangements can be made.

(2) The township is on the ETSA grid and ETSA will extend power lines to the operation at no cost to the consumer. This electricity can be purchased at 2¢/kWh.

(3) There is a water supply nearby and water can be purchased at 11¢/kilolitre (50¢/1000 gal).

(4) No capital is necessary for office, showers, change-room, toilet, lunchroom, etc.

(5) No expense is involved in building a tailings dam.

(6) Ore grade is 2%.

(7) Ore is mined by open-cut techniques and there is a minimum of overburden.

(8) Ore is mined and treated at the rate of 10 000 tonnes/annum.

(9) Labour requirements are:

Mining and Carting	1 man
Crushing	½ man
Leaching	1½ men
Cementation	1 man
	4 men

(10) Capital is to be depreciated over 3 years because of the uncertainty of the ore deposit.

(11) Operating day is 8 hours for 6-day week (270 day/year) single shift, producing 40 t/d of treated ore.

(12) Copper is produced as cement copper (70% Cu) which is worth 97¢/kg contained copper at the treatment-plant site; this assumes the Australian market price for copper to be $1.20/kg.
The payment for 70% cement at Port Kembla after deductions and tariffs is $1.005/kg of contained copper, and freight on cement copper is $24 per tonne (3.5¢/kg of contained Cu).

(13) Agitation leaching gives an 80% overall efficiency.

(14) Percolation leaching (operating on crushed ore minus 6.35 mm (¼ in.) plus 300 micrometres (50 mesh BSS)) uses 80% of ore which is 80% efficient; i.e., the overall efficiency is 64%.

(15) Wages for operators are $5000 per year for six-day week.

(16) Scrap-iron consumption is 2.5 kg Fe/kg Cu and costs 4.4¢/kg (2¢/lb).

(17) Acid consumption for both agitation leaching and percolation leaching is 200% theoretical, and acid consts $40/tonne H_2SO_4.

5.3 Flow Sheet Description

5.3.1 Agitation Leaching

The flow sheet used for the mechanical agitation leaching circuit is shown in Fig.28. Ore is fed through two stages of crushing and one stage of grinding to produce a feed material to the leaching circuit of 100% minus 1.67 mm and 30 to 50% minus 76 micrometres.

Ore is fed into a batch leaching tank which is sized to carry out two batch leaches per operating day. A leaching time of two hours is envisaged as being sufficient for most copper ores provided adequate acid is added (e.g., controlled pH at 1 or sufficient acid added at the start of the leach to give a final pH around 1). The tank is rubber-lined and has a mechanically-driven stainless-steel turbine agitator.

The slurry from the leaching tank is pumped into a batch washing and settling tank. This has associated with it a series of wash liquor storage tanks so that the solids are washed countercurrently. There are two batch washing-settling circuits — one for each of the batch leaching operations carried out per day. The washing-settling tanks are mild-steel rubber-lined vessels with stainless-steel turbine agitators.

A settling time of one hour is considered sufficient to leave a layer of settled solids of approximately 35% by weight. The clear liquor is pumped back to the appropriate liquor storage tank.

This circuit is operated as follows:

(a) Leach slurry is pumped to washing-settling tank.

(b) Liquor is pumped from the first wash storage tank to the washing-settling tank to reduce solids to about 20% by weight; this slurry is stirred to consistency and allowed to settle for 1 hour.

(c) The clear liquor is decanted to the pregnant liquor tank.

(d) Liquor is pumped from the 2nd wash storage tank and (b) is repeated.

(e) This clear liquor is decanted to the 1st wash tank.

(f) Water from the 3rd wash storage tank is pumped and (b) is repeated.

(g) The clear liquor is decanted to the 2nd wash storage tank.

(h) The solids are pumped to the tailings dam.

The pregnant liquor is pumped continuously from its storage tank to the cementation launder. Here scrap iron is added to the launder to precipitate the copper. A liquor residence time of 1½ hours (see Appendix A) is used, with the scrap being added batchwise. Barren liquor is then pumped to a disposal area.

5.3.2 Percolation Leaching

The flow sheet for the percolation or vat leaching circuit is shown in Fig.29. Ore from mining is fed through two stages of crushing which produce 100% minus 6.35 mm (¼ in.) material. The crushed ore is wet-separated by DSM screening at 300 micrometres (52 mesh BSS). The DSM undersize is discarded while the oversize fraction is fed to the leaching vats.

Leaching of the minus 6.35 mm (¼ in.) plus 300 micrometres (52 mesh BSS) ore is carried out in vats over 6 days. Each vat holds one day's ore production and eight vats are needed — six for leaching and one each for washing and loading. Liquor flows through the vats in a countercurrent fashion; i.e., water contacts ore that has been leached for six days and the liquor cycles through with acid additions to the ore which has been leached for the least amount of time. The overflow from this vat is pumped to a pregnant-liquor holding tank before cementation.

Pregnant liquor from the holding tank is pumped continuously into a cementation launder. Here scrap iron is added to the launder to precipitate copper. A liquor residence time of 1½ hours is used, with the scrap being added batchwise. Barren liquor is then pumped to a disposal area.

5.4 Economic Analysis

5.4.1 Cost Analysis

Appendix F gives the plant sizing and capital cost of equipment for the two methods, agitation leaching and percolation leaching. The equipment cost was $93 500 for agitation leaching and $56 000 for percolation leaching. Buchanan and Sinclair (1966) give various methods of determining the installed cost from the equipment cost; for a plant with only the minimum of complexity and no big back-up facilities, 250% in addition to the equipment cost is a good estimate of installed cost. This gives an installed cost of $327 000 for agitation leaching and $196 000 for percolation leaching.

Operating costs are given in Table 42. On the assumptions made, the price of copper will have to rise by 20% before percolation leaching breaks even and 32% before agitation leaching breaks even. Looking at the operating costs, capital depreciation is the largest item. For both plants, crushing equipment represents a major item — 36% for agitation leaching and 52% for percolation leaching. Operations are only planned for a 3-year period, so a plant like this would probably be considered feasible only if second-hand crushing equipment could be purchased. Similarly, the mining would be carried out with second-hand equipment; e.g., 10-tonne truck and front-end loader. This could reduce the mining costs below $2/tonne.

Other capital items for either plant are not likely to be available second-hand. Equipment costs for percolation leaching could not be reduced by much more than approximately $16 000 (purchasing second-hand crushing equipment for $10 000). This would give an equipment cost of $40 000 and an installed cost of $160 000 (4 times equipment cost). If mining can be carried out for $1/tonne with second-hand equipment ($10 000 per year total), this and the savings on capital depreciation with second-hand crushing equipment could reduce the overall operating costs to $126 000 for percolation leaching (small loss of $2000).

By rearranging the operations of the agitation leaching and working the washing-settling cycle on a two-shift basis (i.e., mining, carting, crushing, leaching and washing on day-shift and washing and cementation on afternoon-shift), one of the washing-settling tanks would not be needed, and neither would one liquor transfer pump and half the clarified liquor tanks (savings of \$22 200). However, one additional operator would be required. It is possible to save a total of approximately \$43 500 if second-hand crushing equipment were also purchased and operations were worked on a two-shift basis. This would give a capital cost of \$50 000 (installed cost \$200 000). If mining could be carried out for \$1/tonne with second-hand equipment (\$10 000 per year total), the overall operating costs could be reduced to \$152 000 (small profit of \$2000). Operating the agitation leach circuit on two shifts represents a more economic proposition than on a single shift, so all reference to agitation leaching will be with this condition.

5.4.2 Sensitivity Analysis

The costing analysis shown in Table 42 indicates that a plant of 10 000 tonnes/annum of 2% copper ore operating over three years is not economic, whether new or second-hand equipment is used, at the present selling price of copper, which is approximately \$1.20/kg Cu.

A sensitivity analysis was made of the important variables, ore grade (Table 43), selling price of copper (Table 44), plant throughput (Tables 45 and 46), and plant or orebody life (Table 47). This was to establish minimum economic plant sizes on the type of operations described in Section 4 and make a comparison between percolation and agitation leaching.

Over a fairly wide range of conditions, the economics of both operations (i.e., percolation and agitation leaching) appeared to give about the same returns. This would favour the percolation leaching, as it requires less capital — which is very important to the small producer who invariably finds raising of capital a difficult proposition.

On the basis of the assumptions made using second-hand crushing equipment, the following was found on the minimum economic size at present copper prices:

(1) 4% copper ore ~ 10 000 t/annum over three years (Table 43)

(2) 3% copper ore ~ 20 000 t/annum over three years (Table 45)

(3) 2% copper ore ~ 40 to 80 000 t/annum over three years (Table 46).

If the price of copper were to rise by 20%:

(1) 3% copper ore ~ 10 000 t/annum over three years (Table 44)

(2) 2% copper ore ~ 20 000 t/annum over three years.

The economics are significantly altered if it is possible to increase the operating life of the plant to longer than three years. The return is almost acceptable if 3% copper ore is mined at 10 000 t/annum over 4 years (Table 47: about 37 to 40% profit/investment for both agitation and percolation leaching with second-hand crushing equipment).

5.5 Discussion and Conclusions

Conventional costing techniques have been used in the calculation of the minimum economic sizes of plants for processing of copper ores with some second-hand equipment. Percolation leaching was found to be marginally more attractive than agitation leaching because of the lower capital requirements. Both gave about the same returns on investment. For a plant of 10 000 t/annum operating over 3 years, ore of at least 4% grade is required. Similarly, for 20 000 t/annum over 3 years, ore of at least 3% grade is required. The minimum economic plant size approximately halves if the price of copper rises by 20%. The effect of increasing plant life to 4 years has about half of this effect.

However, a project of this nature would have a much better financial future if unconventional construction techniques could be applied. If the plant operators were able to construct more of the plant and not accept wages in that time for a share of the operation's profits later, then the investment would be much less, since installation by conventional techniques amounts to about 50 to 60% of capital requirements. It is very difficult to estimate economic plant sizes under these circumstances, as plant operators might do considerable damage to the equipment in constructing it. The final capital requirements would be a function of the skill and ingenuity of these operators to put second-hand bits-and-pieces and new equipment together into a working plant.

6. CONCLUSIONS

(1) Both percolation and agitation leaching with sulphuric acid were successful for the treatment of the oxidised copper ores tested on a laboratory scale.

(2) Pilot-scale percolation leaching of both Mt Gunson and Paratoo ores achieved good leaching rates with low acid consumptions. Air Pachuca leaching on a pilot-scale was found to be unsatisfactory because of the difficulty in keeping the solids circulating. Pilot-scale mechanical agitation leaching on ores ground to 80 to 90% minus 177 micrometres (85 mesh BSS) was satisfactory for both Mt Gunson and Paratoo ores.

(3) Settling tests on Mt Gunson and Paratoo ores showed rather poor settling characteristics, in particular the low solids density of the settled slurry. No explanation could be found for these results.

(4) From examination of Amdel Report No.810, gravity flow launders appear to be the most attractive cementation equipment for small-scale copper producers.

(5) A general cost study on the leaching of a hypothetical ore showed that percolation leaching in vats would be marginally more attractive than agitation leaching because of the lower capital requirements. Both methods gave about the same returns on investment. For a plant of 10 000 tonnes ore/annum operating over a period of 3 years, ore of at least 4% Cu grade would be required. Similarly, for 20 000 tonnes/year over 3 years, ore of at least 3% grade would be required. The minimum economic plant size is approximately halved if the price of copper rises by 20%. The effect of increasing plant life to 4 years has about half this effect.

7. RECOMMENDATIONS

While general treatment recommendations for South Australian oxide copper ores have been developed it is still necessary, of course, to examine each orebody with regard to the following aspects:

(1) Ore tonnage.

(2) Average ore grade.

(3) Ease of mining the orebody.

(4) Ore crushing and grinding characteristics.

(5) Ore leaching characteristics for agitation and percolation operations.

(6) Leach residue washing, including settling and filtration characteristics of leached ground ore.

(7) Availability and on-site cost of chemicals (e.g., sulphuric acid and scrap iron).

(8) Availability of services (e.g., power and water).

(9) Availability of established infrastructure (housing and amenities).

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9. ACKNOWLEDGEMENT

L.A. Pilgrim carried out a considerable amount of the experimental leaching work.

APPENDIX A

EXTRACT FROM AMDEL REPORT NO.810 'ECONOMICS OF HYDROMETALLURGICAL RECOVERY OF COPPER' by J.M. Clayton and R.E. Wilmshurst (Nov.1971)

6.2 Cementation

Cementation is the most common method of recovering copper from copper sulphate solutions in small plants. The primary reaction is between the copper sulphate in solution and metallic iron, and may be expressed as follows:

$$CuSO_4 + Fe \rightarrow Cu + FeSO_4$$

In an ideal situation this reaction goes almost to completion, and the iron consumption is less than 1 lb for each pound of copper recovered. In practice, side reactions occur which increase the amount of iron required:

$$Fe_2(SO_4)_3 + Fe \rightarrow 3FeSO_4$$

$$H_2SO_4 + Fe \rightarrow FeSO_4 + H_2$$

The presence of ferric sulphate or too much free sulphuric acid in the liquor is therefore undesirable. Fig. A-1, a plot of free acid concentration against copper concentration, indicates the probable practical limits of these two variables between which cementation would be suitable.

Leach liquors containing copper sulphate solution are passed through long vats containing metallic iron. The reaction rate is a function of the surface area of the iron, temperature and acidity; the most commonly used source of iron for cementation is detinned and shredded tin cans and baled car bodies may be used. As the iron goes into solution, copper is precipitated out. When a sufficient quantity of copper has collected in the bottom of the vat it is removed from service, the liquor is run out and the excess iron removed. The precipitated copper may then be shovelled or hosed onto a drying apron and the vat refilled. Recoveries of up to 98% of the dissolved copper can be achieved, with product grades ranging from 70 to 80% copper. Impurities in the product include any impurities on the iron and small pieces of iron which detach from the main mass. The cost estimates which follow are based on liquors containing 0.4 to 2.0 g of copper per litre of solution.

6.2.1 Capital Costs

It is assumed that the cementation plant draws clear liquor from a storage tank which forms part of a leaching plant to be considered separately. The liquor is pumped through one or more launders 4 feet deep by 3 feet wide, operating at an effective cross-sectional area of 10 quare feet. A 'standard' launder is taken to be 60 feet in length, constructed of 2-inch timber with suitable steel supports. Sufficient launder capacity is provided to allow a 2-hours' residence of liquor in the absence of steel scrap — actual residence time would be in the vicinity of 1½ hours. The cost of a single launder, with pumps and other associated equipment, is estimated to be $1200, and to be constant regardless of the number of units in a multiple installation.

The cost represents only the purchased cost of the materials, and it is assumed that small operators would construct the launders themselves and installation charges would not apply.

For the larger plants, however, construction would probably be by paid labour, and installation charges would apply.

The capital costs given in Table A-1, for plants of various capacities with pregnant liquor of various grades, assume the use of paid labour for installation of five or more launders.

Companies with better access to capital may prefer to spend more on more-durable launders in preference to paying high depreciation rates.

6.2.2 Operating Costs

In calculating operating costs, the following assumptions have been made:

Scrap Iron	Consumed at the rate of 2 lb per pound of copper produced, and charged at 2¢ per pound of scrap.
Operating Labour	Operators are charged at $2.50 per hour, or $20 per 8-h day. An integral number of operators is assumed in every case; i.e., operators have no other duties outside the plant.

The labour required is taken to be:

Up to	2	launders	1 man	
for	3-5	launders	2 men	
for	6-15	launders	2 men,	with one fork-lift truck not capitalised but charged at $10 per day.
for	17-55	launders	3 men,	with one large fork-lift truck not capitalised but charged at $15 per day.

Power	This is required only for pumping, and is small. Estimated cost is 0.1¢ per pound of copper.
Depreciation	The life of the launders is taken as 4 years, and the capital is written off at the rate of 25% annually.
Maintenance	It is expected that substantial maintenance of the launders will be required, and 10% of the installed cost is allowed annually for this purpose.

The estimated operating costs are shown in Table A-2 and in Fig.A-2. Because of the assumptions of integral numbers of standard launders and of operators, the figures in Table A-2 show some discontinuities when plotted; Fig.A-2 therefore shows a range of operating costs. Plants treating the lower grade (0.4 g/litre) liquors will be near the top edge of this range, and those treating more concentrated liquors (2.0 g/litres) will be near the bottom.

TABLE A-1: CAPITAL COSTS — CEMENTATION

Copper Concentration in Pregnant Liquor, g/litre	Installed Costs, $			
	300[*]	1000[*]	3000[*]	10 000[*]
0.4	2400[Φ]	18 000[†]	57 600[†]	198 000[†]
0.7	1200[Φ]	3 600[Φ]	36 000[†]	115 200[†]
1.0	1200[Φ]	2 400[Φ]	25 200[†]	79 200[†]
2.0	1200[Φ]	2 400[Φ]	3 600[Φ]	39 600[†]

[*] Production rate, lb Cu/day.

[Φ] Costs are for launders only.

[†] Costs are for launders + installation.

TABLE A-2: OPERATING COSTS — CEMENTATION

Copper Concentration in Pregnant Liquor, g/litre	Operating Costs, ¢/lb Cu			
	300[*]	1000[*]	3000[*]	10 000[*]
0.4	11.6[Φ]	10.4[†]	8.3[†]	7.4[†]
0.7	11.2[Φ]	8.5[Φ]	7.3[†]	6.3[†]
1.0	11.2[Φ]	6.4[Φ]	6.9[†]	5.9[†]
2.0	11.2[Φ]	6.4[Φ]	5.5[Φ]	5.1[†]

[*] Production rate, lb Cu/day.

[Φ] Costs include depreciation charges for launders installed by owner at no cost.

[†] Costs include depreciation charges for launders installed by paid labour.

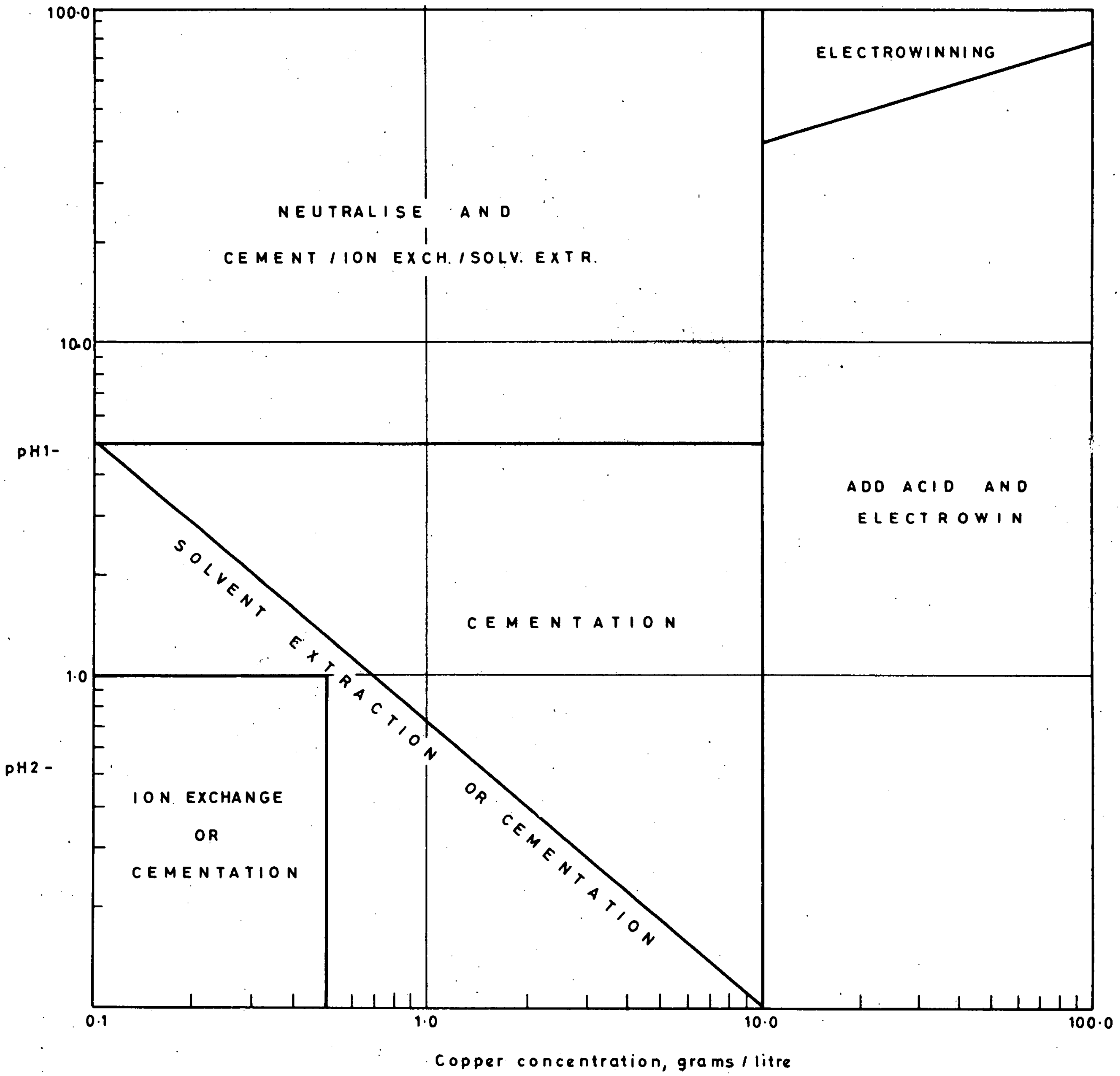

FIG. A1 : METHODS FOR COPPER RECOVERY FROM SOLUTION

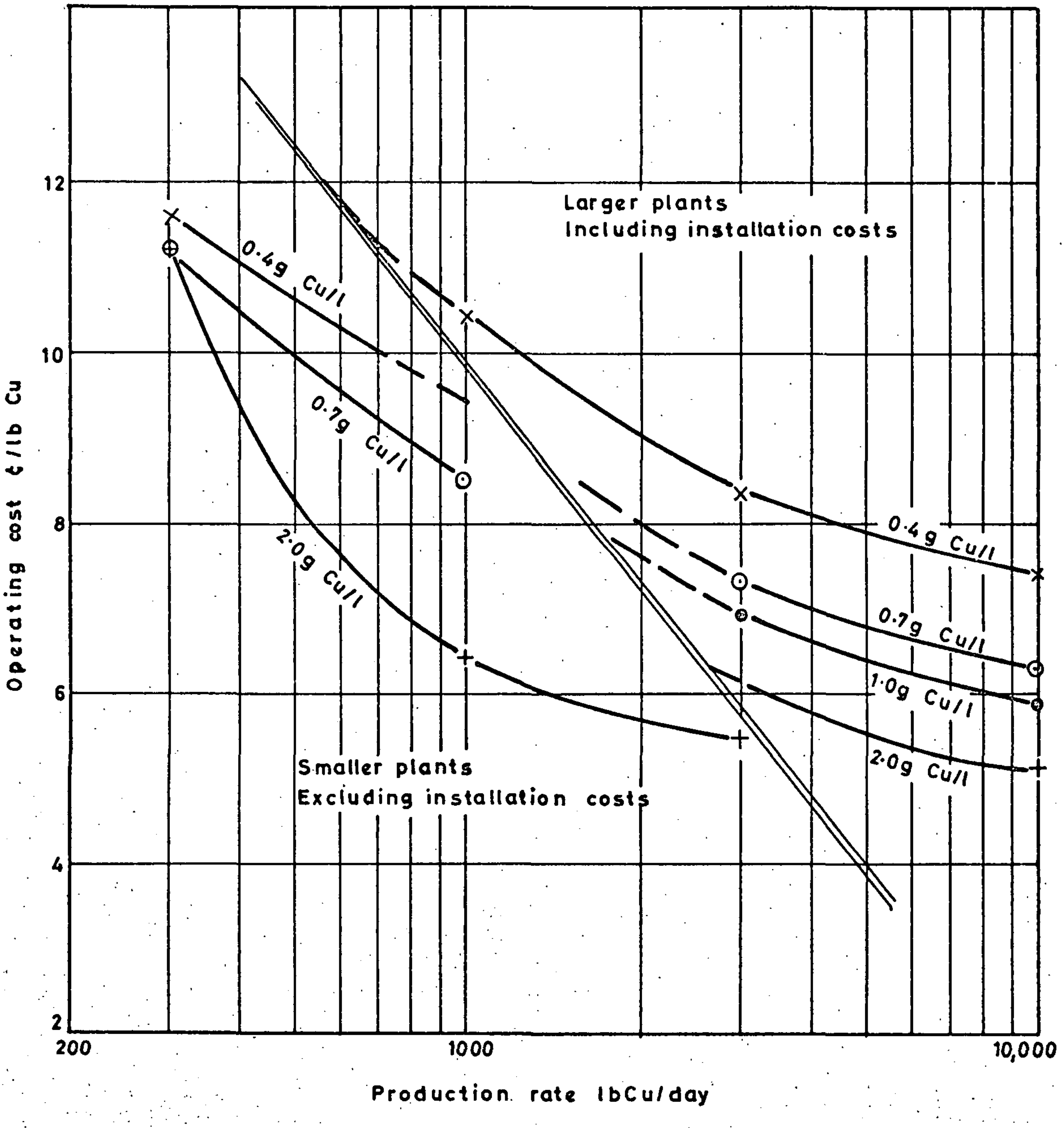

FIG. A2: OPERATING COSTS-CEMENTATION

USE OF CONE-TYPE COPPER PRECIPITATORS TO RECOVER COPPER FROM COPPER-BEARING SOLUTION

by H. R. Spedden, E. E. Malouf, and J. D. Prater

With the expansion of copper leaching of the various mine wastes at the Kennecott properties, additional copper recovery facilities have been required. A research and development program has been pursued to determine a suitable economic approach to the processing of the additional leach solutions contemplated. Investigations on laboratory and pilot plant scales covered electrowinning, solvent extraction, launder and vat type cells, activated launders, precipitation drums, and precipitation cones. This paper contains information on the development and use of a new precipitation-cone-type copper recovery plant, which provides many advantages over older systems.

The application of research findings to the old art of leaching copper from copper-bearing mine waste has resulted in a significant contribution of copper to over-all copper production. For example, at the completion of the current expansion program at the Kennecott properties in the United States, copper produced from waste dump leaching will amount to about 25% of the total production. Formerly only 2% was derived from this source. This planned expansion of copper leaching focused attention on the problem of developing more efficient methods for the recovery of copper from greatly increased volumes of copper-bearing solutions.[1]

The detailed investigations which have been made in laboratory pilot plant and plant tests for the recovery of copper from copper-bearing solutions have included electrowinning, solvent extraction, ion exchange and cementation with iron in launders, precipitation drums, activated launders, and precipitation cones. The results of these investigations have led to the development of a new precipitation cone-type copper recovery system providing many advantages over older methods.

ELECTROWINNING OF COPPER FROM COPPER-BEARING SOLUTIONS

The direct electrowinning of copper from relatively dilute solutions of copper-bearing mine water, namely solutions containing 10 to 20 lbs of copper per 1000 gal, has always been an attractive possibility.[2,3] By this method, high-purity copper powder can be recovered with a power consumption of 3.5 kwh per lb of copper. However, once the solution strength de-

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H. R. SPEDDEN is Research Director, E. E. MALOUF is Project Development Engineer, and J. D. PRATER is Chief, Hydrometallurgical Section, Western Mining Div. Research Center, Kennecott Copper Corp., Salt Lake City, Utah. TP 66B86. Manuscript, February 1966. New York Meeting, February 1966. Discussion of this paper, submitted in duplicate prior to Mar. 5, 1967, will appear in SME Transactions, June 1967, and AIME Transactions, 1967, vol. 238.

creases to approximately 5 lbs of copper per 1000 gal, the current efficiency drops rapidly, requiring the use of other methods to recover the remainder of the copper economically. These results combined with the high capital cost make this approach economically unsound under most conditions.

SOLVENT EXTRACTION OF COPPER FROM COPPER-BEARING SOLUTIONS

Another technically feasible process is using various organic reagents for the solvent extraction and concentration of copper from dilute acidic solutions.[4] The majority of solutions obtained from leaching rather heterogeneous mine waste dumps, however, contain substantial quantities of ions other than copper as well as suspended insoluble materials. These various ions and suspended gangue can result in a combination that may cause emulsification and costly loss of the solvent. Although substantially improved liquid ion-exchange or solvent extraction reagents which resist emulsification to extremely low levels are now available, enough experience with these new reagents has not yet been obtained to safely justify a major installation. Presently available reagents have a rather low loading factor and thus would require a large capital expenditure for plants of the size now being constructed. A small plant and, in particular, one which does not have a readily available, low-cost source of scrap iron would appear to be the logical next step in this development. The application of the process to copper metallurgy on a significant scale is, nevertheless, a most stimulating goal.

ION-EXCHANGE RECOVERY OF COPPER FROM COPPER-BEARING SOLUTIONS

The use of ion-exchange resins for the concentration of copper from copper-bearing mine solutions has not proven feasible. The non-selectivity of the resins and the fouling of the resins with iron and aluminum ions has precluded the use of this approach. Even the carboxylic-type resins, which are quite specific for copper, display an unsatisfactorily low loading capacity when used for extraction from acidic solutions; thus projected capital costs appear to be unduly high.[5]

CHEMICAL PRECIPITATION OF COPPER FROM COPPER-BEARING SOLUTIONS

Precipitation of copper from dilute copper-bearing solutions using various chemical precipitants has been a source of continuing investigations by several research groups.[6,7] Consideration has been given to precipitating the copper as a sulfide, as a cyanide, as a thiocyanate, and even as a hydroxide using lime. The recovery of an extremely fine chemical precipitate, with the inherent difficulties of settling and filtering, is a common problem in essentially all processes employing chemical precipitants. The products of chemical precipitation also usually require additional processing steps to obtain the copper in a form readily obtained by cementation on iron.

PRECIPITATION OF COPPER BY IRON

Gravity Launders: The most common method of precipitating copper from leach solutions, and the oldest method, has been the gravity-flow launder charged with scrap iron as the precipitant. As a general figure, this type of plant requires 500 ft of launder, 4 ft wide by 4 ft deep, to process 1000 gpm of copper-bearing solution. A launder of this size can effectively recover over 90% of the copper in solutions. However, iron consumption will generally vary between two to four times that amount theoretically required to precipitate the contained copper, depending on the ferric iron and free sulfuric acid contents of the solution. Fig. 1 is a photograph of the gravity launder-type copper precipitation plant at Kennecott's Utah Copper Div.

44 ~~37~~ minutes
for 3' deep
solution
NB. U.S. gals.

Launder plants, although simple to construct and operate, require much hand labor and produce an impure cement copper which is usually blended with concentrates as a feed to a smelter. Efforts to improve the method have led to numerous modifications, mostly involving variations in the mechanical handling of the scrap iron or the precipitates.

Drum Precipitators: Rotating drum precipitators have been used industrially in place of launders. The major problem encountered in this system is that of maintaining a large mechanical device in which the total mass of scrap iron is tumbled constantly. Our own test work has shown that the tumbling action breaks the copper precipitates into fine particles, much of it even colloidal in size, thus presenting a further operational problem. Labor requirements for charging scrap iron and for the periodic removal of unconsumed trash likewise make these units generally unsatisfactory.

Activated Launders: Gravity launders have been modified by laying one or more nozzle manifolds along the bottom of the launder to inject the copper-bearing solutions into the mass of iron.[8,9] Studies involving the precipitation of copper in this type of launder have indicated definite improvements over the gravity-type plant, both as to iron factor and volumes of solution treated. However, here again, cleaning of this type of unit requires removing the accumulation of copper precipitates, cans, and trash with consequently high hand labor requirements.

Development of a New Precipitation System: Although most of the recovery methods thus investigated are practical, each one seemed to be lacking in at least one of several desirable features for relatively large scale production. As with most chemical processing methods, an efficient copper recovery system should

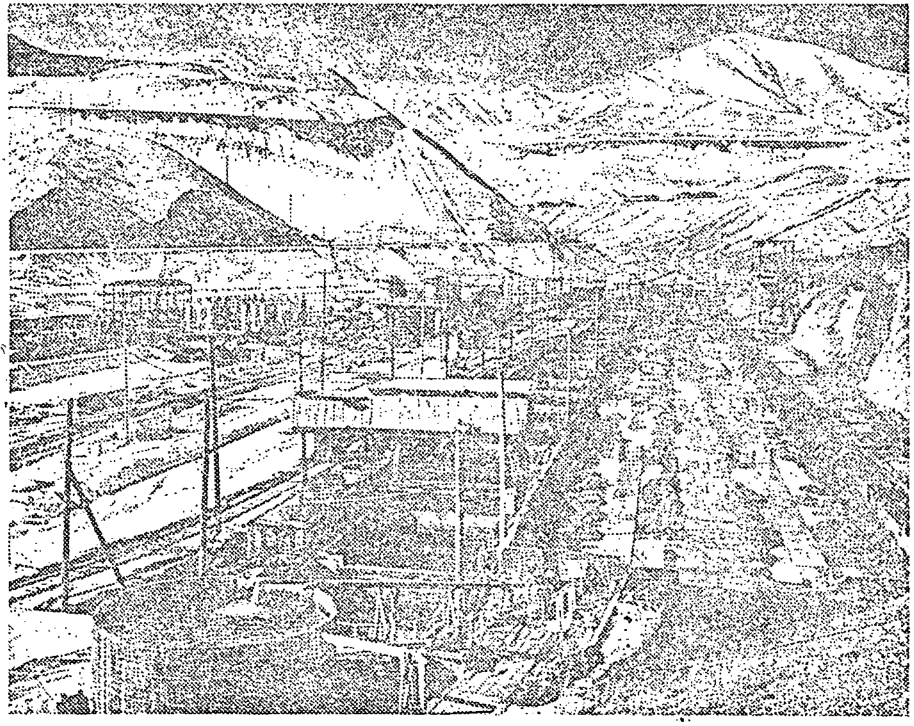

Fig. 1 – Photograph of gravity launder-type copper precipitation plant at Kennecott's Utah Copper Div., Bingham Canyon Mine.

provide for high volumetric capacity, be able to treat solutions of variable concentrations with high recovery, and be amenable to a substantial degree of mechanical handling and automatic control.

Three chemical reactions, each of which consumes iron, have long been recognized as of importance in copper cementation. These reactions may be shown by the following equations

$$CuSO_4 + Fe \longrightarrow Cu + FeSO_4 \qquad [1]$$

$$Fe_2(SO_4)_3 + Fe \longrightarrow 3\ FeSO_4 \qquad [2]$$

$$H_2SO_4 + Fe \longrightarrow FeSO_4 + H_2 \qquad [3]$$

Under quiescent conditions, as represented in a launder plant, these reactions will reach equilibrium. If, however, powdered iron of high surface area is used, the copper precipitation reaction is found to be predominant and may be essentially completed before excessive amounts of iron have been consumed by the other two reactions.

Since powdered iron is thus an effective precipitant and may be produced at reasonable cost as a by-product of a base-metal mining and smelting complex, a research program was initiated to develop an efficient vessel in which to utilize such a precipitant. The successful accomplishment of this objective has been reported separately by A. E. Back.[10]

A pressing need for more copper production, combined with the necessity for a prolonged delay to bring a source of powdered iron into production, dictated that a further effort be made to improve precipitation methods employing available scrap iron. Using the basic cone configuration, a new vessel was designed for a scrap iron feed; this vessel has now been found to have advantages not available in other precipitating processes. Its operation, design, and copper product differ from that of the powdered iron cone. It is a compact unit lending itself to automatic control, low iron consumption, and self-cleaning of copper precipitates. Furthermore, it results in the production of a higher purity cement copper than that resulting from a launder system. The operating characteristics of the cone precipitator are based on a high-velocity, rapid through-put of copper-bearing solutions and an intimate contact of the solution with clean active iron for precipitation of copper.

Our experience comparing the relative effectiveness of powdered iron and scrap iron as precipitants clearly indicated that the three basic reactions proceed at different rates. Furthermore, these reactions are temperature dependent, another characteristic of rate reactions. Fig. 2 shows the effect of temperature on increasing the copper precipitation rate. Thus, a rapid contact of solution with iron surfaces promotes reaction 1, the copper precipitation reaction, by removing the diffusion layer. The resulting suppression of the acid on iron reaction (Eq. 3) brings about a very real and profitable reduction in iron consumption.

Wadsworth and co-workers have defined the mechanisms involved in copper precipitation on iron[11] as:

1) diffusion of reactants to the surface,
2) adsorption of reactants on the surface,
3) chemical reaction at the surface,
4) desorption of products from the surface, and
5) diffusion of products away from the surface.

They have quantified item 1) for laboratory conditions in the terms of stirring speed of a mechanical agitator. We have likewise determined an apparent optimum range of conditions in the terms of rate of

APPENDIX B

USE OF CONE-TYPE COPPER PRECIPITATORS TO RECOVER COPPER FROM COPPER-BEARING SOLUTION

by

H.R. Spedden, E.E. Malouf, and J.D. Prater

Note: *The gallons referred to in this Appendix are US gallons:*
1 US gal = 0.83268 Imp. gal = 3.7854 litres

(Transactions of Society of Mining Engineers, December 1966)

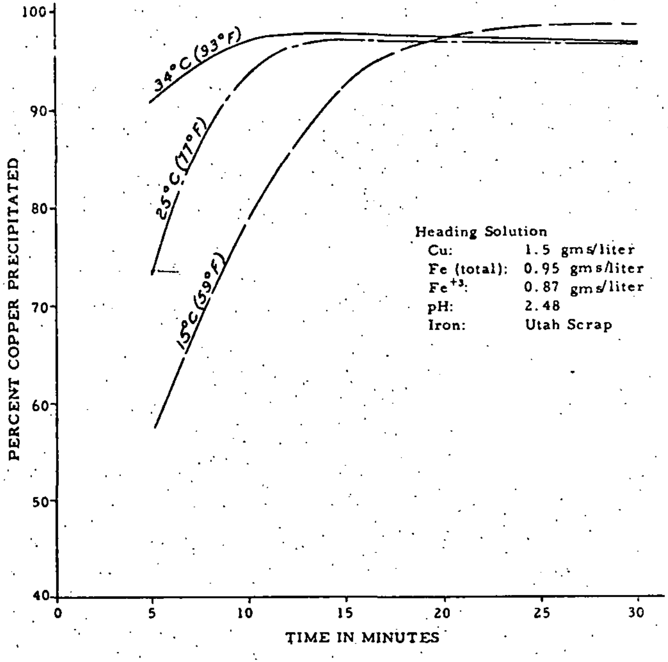

Fig. 2 – Copper cementation as a function of temperature.
**Conversion factor 1 g = 8.345 lbs per 1000 gal of solution.*

solution flow through full production size vessels of varying sizes. They further have found that the copper precipitation reaction is a first order rate reaction and that even under some conditions the reaction seems to follow zero-order kinetics. Thus, with adequate agitation, rapid precipitation will occur and acid consumption by iron will be minimized.

The nucleation mechanism has also been studied by Wadsworth[11] with the following conclusion: "The precipitated copper adheres to the iron as a spongy layer at low speeds, peels off in the form of bright strips at medium speeds, and as a fine powder at high speeds." The low speed agitation results defined by Wadsworth correspond to the action in a gravity launder. Medium speed results are of the same magnitude as those we have found optimum for promoting the precipitation reaction, and thus a dual advantage is gained by dynamic contact of solution with iron.

The theory of diffusional control is further enhanced by experiments employing iron turnings as a precipitant. Clean turnings, extensively laced with submicron cracks, present a larger surface area to the small hydrogen ion than to the much larger copper ion. Thus, acid attack by pore diffusion is possible in the cracks. Turnings in this form yielded copper precipitation at a consumption of over 3 lbs iron per lb copper, as compared with 1.5 lbs per lb of copper when using shredded iron in a precipitation cone.

Another portion of these turnings was crushed in a hammer mill to powder size. In a similar copper-bearing solution, the iron consumption was only 1.2 lbs iron per lb copper. The cracks had thus been opened, exposing all surfaces to the faster copper precipitation reaction.

The granular and dense type of copper produced under dynamic precipitation conditions results in a product that can be filtered readily to a low moisture content. This is in contrast to the thixotropic-type of precipitates produced in the launder-type plant which, after filtration, may contain 35% to 40% moisture.

The experimental precipitation cone as developed to employ these principles at the Utah Copper Div. of the Kennecott Copper Corp. is capable of processing high volumes of copper-bearing solutions. Fig. 3 is a photograph of a cross-sectional area of the precipitation cone. The vessel consists of a 14-ft diam tank, 24 ft tall, into which is mounted an inverted cone 10 ft in diam by 10 ft high. The outer 14 ft diam tank contains a 45-degree sloped false-bottom floor from one side of the tank to a bottom side discharge at the opposite side. The annular space between the inner cone and the tank is covered by a heavy gauge stainless steel screen. The screen is mounted as a continuation of the cone and is anchored to the cone and tank. The cone supports a pressure manifold that consists of six vertical legs with each leg containing a series of nozzles directed inward from the tangent to the cone and upward from the angle of the legs of the manifold. The nozzles are arranged in such a manner as to create a vortex when the copper-bearing solutions are pumped through the manifold into the cone. The inner cone and the area of the tank above

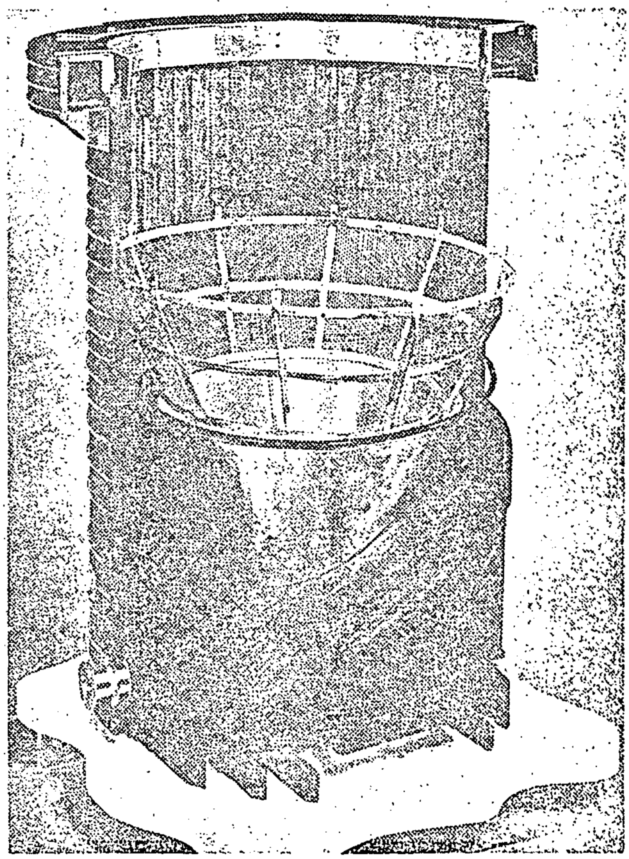

Fig. 3 – Photograph of a cross-sectional area of Kennecott's cone precipitator (patent pending). (Photograph courtesy Bechtel Corp.)

the stainless steel screens are filled with shredded detinned iron scrap, such as is commonly used in the precipitation of copper. The shredded iron is 'coned' to the top of the tank. This large mass of iron in a confined vessel has proved to be an effective heat retaining medium, thus enhancing the reaction kinetics.

Copper-bearing solutions are pumped through the manifold with the nozzles injecting the copper-bearing solutions into the mass of iron. The injection of the solutions has the effect of not only rapidly precipitating copper, but also removing the metallic copper from the iron surface, thereby exposing clean, fresh iron. Fig. 4 is a diagrammatic sketch of the cone precipitator showing solution inflow and copper precipitate discharge.

The precipitation cone is a continuously operated unit that is self-cleaning as to copper precipitates and eliminates the need for the conventional approach of labor with fire hoses to wash the copper precipitates from the precipitator, as is the practice in many launder-type plants. The pressure and velocity of the solutions in the lower conical section tend to move the copper precipitates in the same manner as an elutriation column, upward and out of the cone into the reduced velocity zone created by the larger diam of the holding tank. The copper precipitates settle down through the stainless steel screen and accumulate on the sloped false-bottom of the tank. The copper can then be discharged intermittently with the use of a pneumatically operated valve on a time cycle or bled continuously through a small diameter pipe into a thickener or holding basin. Fig. 5 is a photograph of the experimental cone-type copper precipitator at the Kennecott Copper Corp., Utah Copper Div. The copper precipitates produced in this manner are of substantially higher grade than the conventional cement copper produced in a launder-type plant. They typically will analyze 90% to 95% copper, 0.1% to 0.2% iron, 0.1% to 0.2% silica, and 0.1% to 0.2% alumina with the balance of the impurity being primarily oxygen.

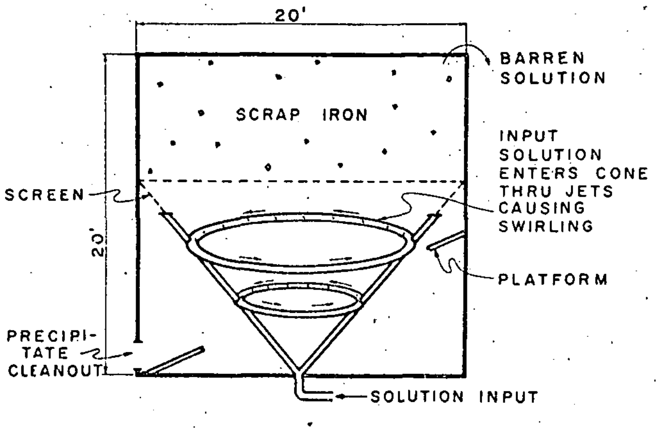

Fig. 4 – Sketch of cone precipitator showing solution inflow and copper precipitate discharge.

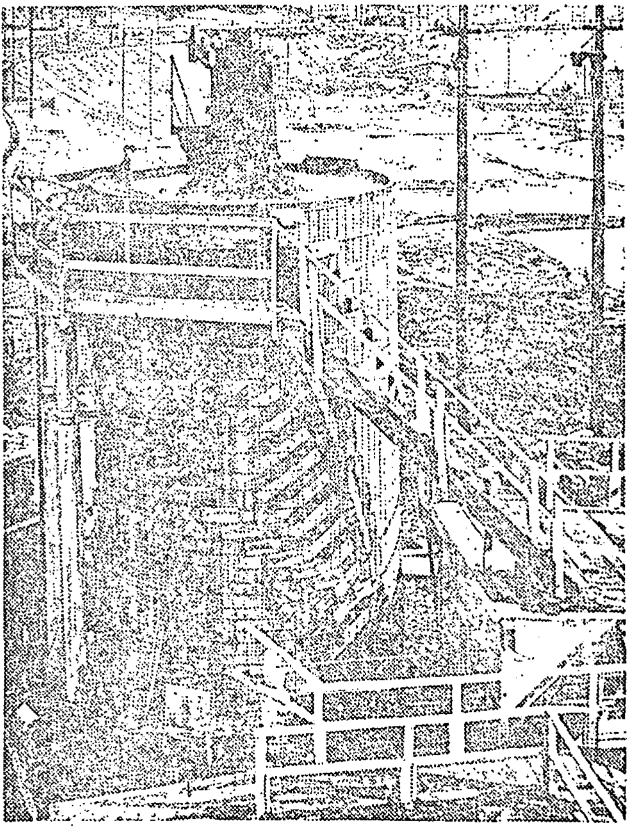

Fig. 5 – Photograph of experimental precipitator at Kennecott Copper Corp., Utah Copper Div.

The prototype precipitation cone was operated continuously for seven weeks in one test. After this test run, using commercial shredded iron identical to the material used in the launder plant, the unit was shut down and examined. A total of 18 in. of trash remained in the bottom of the inner cone. This residue consisted of pieces of concrete, rocks, granular copper, and some chunks of copper-plated steel shafting. This small amount of residue was striking evidence that most of the trash inherent in the scrap iron had been masticated by the dynamic action in the cone and discharged in the tailing solutions. In comparison, a launder plant usually requires daily washing with high pressure hoses.

During seven weeks of continuous operation of the experimental cone, copper recovery averaged 89.7% with inclusion of data for periods in which wide fluctuations in the addition of shredded iron was experienced. Sustained periods during optimum operating conditions resulted in copper recoveries exceeding 95% in the single cone.

Results comparing the performance of an experimental precipitation cone operating at approximately double the flowrate of the standard-sized gravity launder with which it is compared are presented in Table I. These results represent daily averages for the same operating periods.

As a result of the successful operation of the experimental cone at the Utah Copper Div., a cone-type precipitation plant is now being constructed which will contain 26 cone precipitator units. The plant will be of modular construction arranged in a manner to permit the solution flow to pass through two cones in series. This will provide an operational safety factor for optimum stripping. Fig. 6 is a photograph of a model of the new copper precipitation plant under construction at Kennecott's Utah Copper Div. The structure rising above the cone tanks is a movable scrap iron feeder fed by a conveyor belt from the scrap storage yard. Shredded scrap iron is added to each cone intermittently as required.

Table I. Typical Data (Daily Average) Comparing The Operating Results of an Experimental Precipitation Cone with a Gravity Launder

Heading	Cone Precipitator		Heading	Launder Precipitator	
Lbs Cu/ 1000 Gal	Cu Recovery, %	Soluble Fe Factor	Lbs Cu/ 1000 Gal	Cu Recovery, %	Soluble Fe Factor
17.6	92.4	1.13	15.1	98.3	2.85
14.8	93.2	1.45	15.4	89.4	2.15
14.8	88.2	1.83	16.1	85.1	2.04
13.3	94.3	1.60	15.7	87.5	2.26
12.9	91.0	1.49	14.4	85.8	2.30
13.3	92.4	1.77	14.8	90.7	2.17
15.8	93.6	1.86	14.6	93.7	2.16
16.2	93.3	1.74	14.7	91.3	2.22
15.9	95.3	1.54	15.1	91.5	2.25
16.4	96.4	1.74	14.6	84.6	2.40
16.2	95.4	1.10	13.7	84.4	2.47
13.9	94.0	1.72	14.0	95.8	2.70
			Average		
15.1	93.3	1.58	14.9	89.9	2.33

Two cones have been operated in series to determine the effect of the second cone on total iron consumption when only partial stripping has been accomplished in the first cone. This two stage treatment can provide a better over-all control without increasing iron consumption beyond that required by a single stage treatment yielding the same recovery. Variations in the copper content of the influent solutions to the second cone have ranged, for test purposes, from 1.5 lbs of copper per 1000 gal to as high as 15 lbs per 1000 gal, yielding the same low content of copper in the tailing from this second cone. The copper-bearing solutions are chemically conditioned in passing through the first cone, thus resulting in rapid and effective stripping of the remainder of the copper in the second unit.

A further variation of the two-stage system is effectively employed in the recently expanded precipitation plant at the Kennecott Chino Mines Div. Larger cone units, each 20 ft in diam and 24 ft high, and thus capable of processing substantially greater volumes of solution, are used to recover over 80% of the copper in the first stage. The conditioned cone tailing is then passed through the pre-existing launder plant for final stripping. A single launder cell, which previously had a capacity of 300 gpm, can now strip copper from 1000 gpm of the conditioned solution at a relatively low iron consumption. Thus, the combination of cones and launders has provided the technical advantages of the cone precipitator in a plant of greatly expanded capacity at minimum cost.

Two, single-stage cone precipitators of the same basic design as the Utah cones, are also producing copper at Kennecott's Nevada Mines Div. In addition,

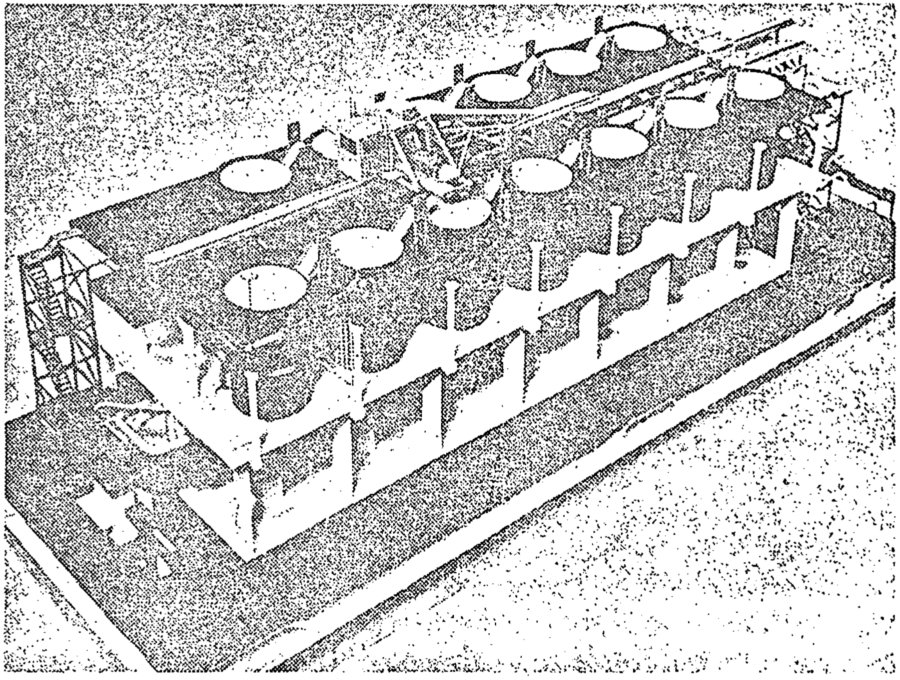

Fig. 6 – Photograph of a model of the cone precipitation plant now being constructed. Each module contains 13 cone precipitators. (Photograph courtesy Bechtel Corp.)

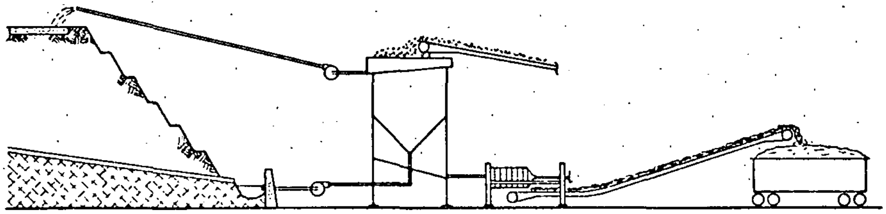

Fig. 7 – Diagrammatic sketch of the copper leaching and precipitation system as used at Kennecott Copper Corp.

a modified cone, using gravity flow from a hillside head tank at a relatively low injection pressure is under development at the Kennecott Ray Mines Div. A further variation in this particular design provides for cyclical operation with periodic dumping of the copper through the bottom clean-out valve.

In summary, operating experience with precipitation cones has demonstrated that the application of kinetic principles results in the production of precipitate copper of a more granular, higher purity form and at a lower iron consumption than is possible with the older launder methods. High capacity, versatile, precipitating vessels are now available with features permitting automatic control and mechanized materials handling. An old art has been modernized. Fig. 7 is a diagrammatic sketch of this new copper leaching and precipitation system.

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APPENDIX C

USE OF PARTICULATE IRON IN THE PRECIPITATION OF COPPER FROM DILUTE SOLUTIONS

by

A.E. Back

Note: *The gallons referred to in this Appendix are US gallons:*
1 US gal = 0.83268 Imp. gal = 3.7854 litres

(Transactions of SME, March 1967, Vol.238)

USE OF PARTICULATE IRON IN THE PRECIPITATION OF COPPER FROM DILUTE SOLUTIONS

by A. E. Back

A method is described in which particulate iron, as distinguished from high purity iron powders used in powder metallurgy, is a precipitant for copper contained in dilute solutions. A new precipitation apparatus utilizes particulate iron precipitants to recover copper more efficiently than is possible in a conventional launder precipitation plant.

In the search for lower cost methods of recovering copper from mine water, the use of sponge iron or particulate iron (as distinguished from iron powder used in powder metallurgy) as precipitants in place of tin cans, detinned scrap iron, or scrap iron is an intriguing possibility. The relatively faster copper precipitation rate obtained with particulate iron as compared to scrap iron promises economic and processing advantages when, and if, particulate iron becomes competitive cost-wise with available scrap iron. Kennecott has developed a precipitation cone,[1] see Fig. 1, utilizing various particulate iron precipitants and has demonstrated the process successfully in a prototype at a flow rate of approximately 1000 gal of solution per min. Essentially complete precipitation of copper is obtained, the cone overflow solution is clear and contains no particulate copper, and the iron factor is more favorable than in a conventional launder plant.

Initially, sponge iron produced at the Ray Mines Div. Kennecott Copper Corp. for use in a Leach-Precipitation-Flotation process was used in exploratory tests to develop a suitable apparatus to take advantage of the rapid precipitation rate and efficient iron utilization for the recovery of copper from dilute solutions. Here it was demonstrated that, when sponge iron was added to a launder, the particles of iron collected in the bottom of the launder and tended to cement together. This resulted in incomplete precipitation of the copper and inefficient utilization of the iron. Next, sponge iron was suspended in glass columns to study the dynamics of the precipitation reaction. Again it was observed that the particles tended to cement together at low solution flow rates, while at higher flow rates large amounts of the precipitant overflowed the column, resulting in loss of precipitant and an inefficient process. Next, the precipitant was suspended in an inverted cone in a rising column of solution. Dynamic suspension of the solids was maintained at relatively rapid solution flow rates with instantaneous and complete precipitation of copper and with improved iron utilization. The first tests were conducted on a batch basis and it was determined that a dynamically suspended bed of precipitant was necessary for satisfactory operation. More than 99% of the metallic iron could be converted to metallic copper by careful control of the residence time in the cone and copper precipitation was essentially complete, even when the available iron precipitant was almost depleted.

Several cones were built, ranging in capacity from 1 to 200 gal of feed solution per min, and finally a prototype cone was constructed which was 20-ft high

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A. E. BACK is Assistant to the Research Director, Western Mining Div., Kennecott Copper Corp., Research Center, Salt Lake City, Utah. TP 66B87. Manuscript, February 1966. New York Meeting, February 1966. Discussion of this paper, submitted in duplicate prior to June 15, 1967, will appear in SME Transactions, September 1967, and AIME Transactions, 1967, vol. 238.

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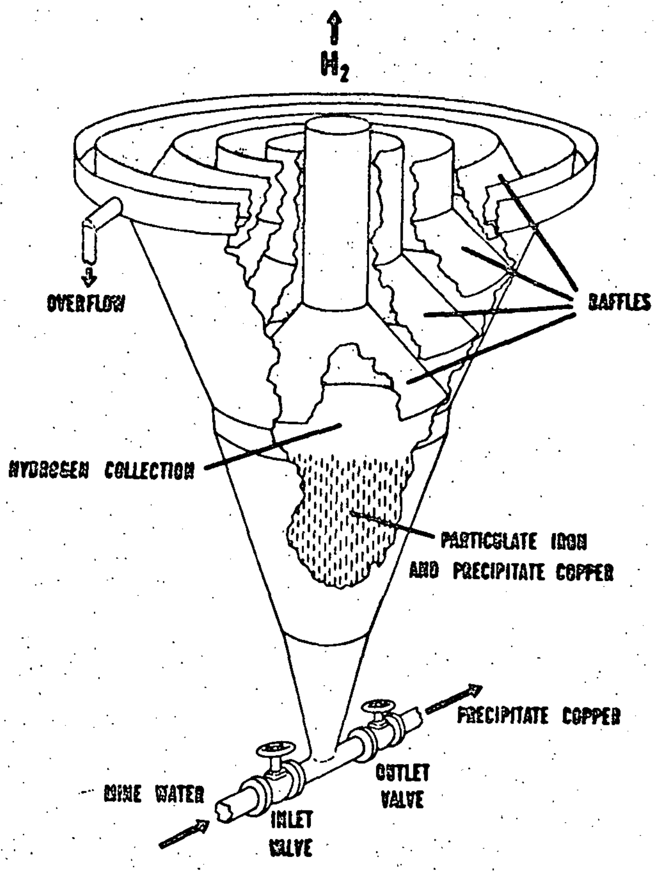

Fig. 1 – Precipitation cone, developed by Kennecott.

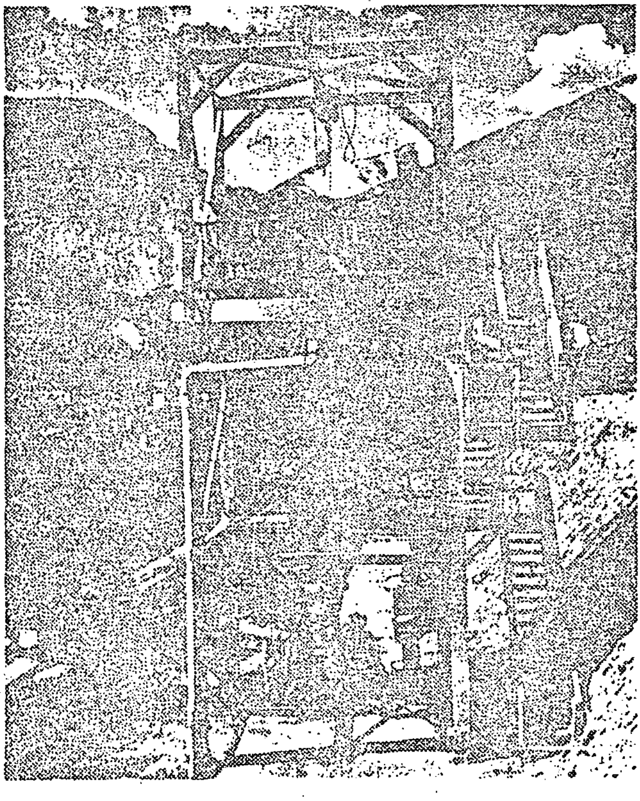

Fig. 2 – Prototype cone; approximate capacity, 1000 gal of solution per min.

and 20 ft in diam, with an approximate capacity of 1000 gal of solution per min. This cone is illustrated in Fig. 2. Tonnage samples of particulate iron have been evaluated in the prototype cone to confirm the results of small scale cone tests. The effectiveness of a precipitant using 10-lb samples in a small cone precipitator can be determined with the same confidence as testing tonnage lots in the prototype precipitator.

DESCRIPTION OF COPPER CEMENTATION PROCESS

The following competitive reactions occur in the precipitation of copper from dilute acid solutions using metallic iron as the precipitant

$$\text{Fe (metal)} + CuSO_4 \rightarrow \text{Cu (metal)} + FeSO_4 \qquad [1]$$

$$\text{Fe (metal)} + Fe_2(SO_4)_3 \rightarrow 3\ FeSO_4 \qquad [2]$$

$$\text{Fe (metal)} + H_2SO_4 \rightarrow H_2 + FeSO_4 \qquad [3]$$

Reactions [1] and [2] proceed almost simultaneously in the cone precipitator because the metallic copper produced in [1] catalyzes reaction [2]. Reaction [3] proceeds more slowly than [1] and [2]. In the solutions studied, the bulk of the iron is used to precipitate copper because of the relatively small concentration of ferric iron present. Because of the intimate contact in the cone between the metallic iron and the solution, copper precipitation is complete with less neutralization of the acid [3] by the iron than in a conventional launder plant. Thus, the cementation reaction is more efficient with respect to iron consumption and an iron factor (weight ratio of iron consumed to copper precipitated) approaching more closely the theoretical value of 0.9 is obtained. In addition, advantage is taken of reaction [3] to monitor the course of the precipitation reaction [1] by measuring the change in the rate of hydrogen evolution. Hydrogen evolution decreases as the amount of available metallic iron is diminished, and provides a sensitive measure for process control without chemical analysis. Addition of a measured batch of precipitant to the cone, with continuous discharge of barren solution at the overflow and intermittent discharge of the precipitates from the apex of the cone, was found to be most desirable.

During the period of this investigation, some 14 different samples of particulate iron from various companies and individuals have been evaluated as copper precipitants. Various samples of directly reduced iron, iron powder, iron turnings and chips, and granulated iron have been submitted. Raw material source, reductant used, and typical analyses are given in Table I.

The samples contained variable amounts of copper, sulfur, carbon, and silicon which are not listed. The grade of precipitate theoretically recoverable is

Table I. Typical Analysis of Particulate Iron Samples Tested and Grade of Copper Precipitate

Raw Material	Reductant	Analysis, %		Reduction, %	Calculated Grade of Copper Precipitates with 0.9 Iron Factor, %
		Fe (Metal)	Fe (Total)		
Commercial iron powder	–	97.6	99.2	98.4	97.8
Commercial iron powder	–	96.0	97.1	98.9	96.4
Magnetite concentrates	Reformed gas	93.8	94.4	99.4	94.4
Magnetite concentrates	Solid carbonaceous	93.8	96.0	97.7	94.4
Iron turnings	–	93.0	98.0	94.9	93.7
Pelletized magnetite concentrates	Solid carbonaceous	92.0	94.4	97.5	92.8
Pelletized magnetite concentrates	Solid carbonaceous	86.8	90.7	95.7	88.0
Pelletized magnetite concentrates	Solid carbonaceous	86.6	93.0	93.1	87.8
Pelletized magnetite concentrates	Solid carbonaceous	83.6	97.4	85.8	86.6
Magnetite concentrates	Solid carbonaceous	79.4	87.5	90.6	81.0
Reverberatory slag	Carbon	65.4	83.4	78.4	67.7
Pyrite cinders	Solid carbonaceous	55.2	71.6	77.1	57.8
Refinery residues	Reformed gas	48.8	64.2	76.0	51.4

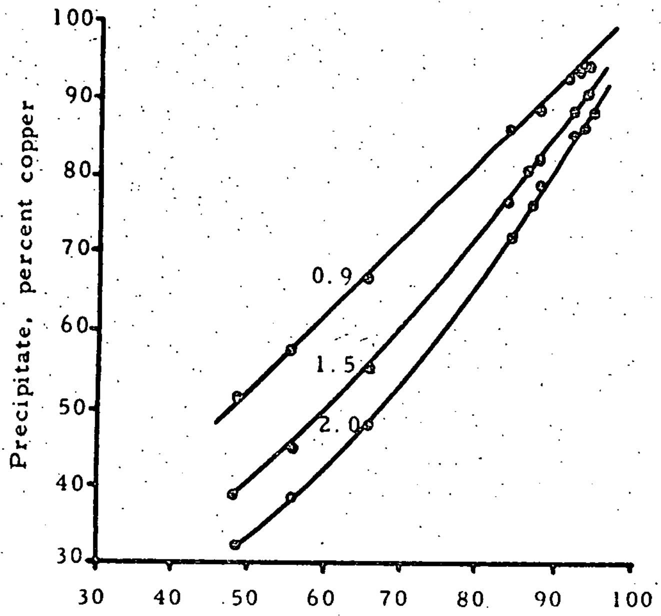

Fig. 3 – Relationship of grades of precipitant and precipitate at different iron factors.

shown graphically in Fig. 3 with theoretical 0.9, 1.5, and 2.0 iron factors. Any copper present in the precipitant is recovered in the resulting precipitates at a grade higher than that indicated in Table I. This means that the copper specification required for iron blast furnace raw materials is not critical, which could result in a lower cost precipitant, and a high grade precipitate would be smelted, which could decrease the amount of inert material required during smelting.

In general, each of the particulate iron precipitants is satisfactory for the recovery of copper from dilute solutions in a precipitation cone. The dense iron samples require finer grinding, i.e. to –35 mesh, than do the spongy iron samples, which are suitable if ground to –10 mesh. An excessive amount of –200 mesh material is undesirable because this favors reaction [3] slightly over reaction [1]. However, as long as the bed of precipitant is maintained in dynamic suspension by adjusting the rate of solution flow into the cone, variations in particle size can be accommodated and reaction [1] is favored. A high degree of reduction of the iron precipitant is desirable because of the resultant high grade precipitate. However, the cost of obtaining such a precipitant often prices it out of the market and a compromise between reduction costs for the precipitant and smelting costs for the precipitate must be resolved.

The prototype cone that is now in operation at the Utah Div. precipitation plant, Fig. 2, is equipped with a variable speed feeder so that the precipitant feed rate can be varied to meet the requirements of the process. The facility is automated so that the progress of the consumption of the available iron can be followed by the decrease in the evolution of hydrogen from the reaction,

$$H_2SO_4 + Fe\ (metal) \longrightarrow FeSO_4 + H_2. \qquad [3]$$

As the hydrogen evolution decreases and comes to a predetermined low value, a system of valves becomes operative thus shutting off the incoming solution and opening a discharge valve for approximately 30 sec to remove the precipitate from the lower portion of the cone. Then, the valve closes and fresh iron is fed into the cone simultaneously with the influx of fresh copper-bearing solution. The precipitates are filtered and then transferred to a conveyor for subsequent treatment. Channeling the hydrogen or reaction gases toward the center of the cone into a central exhaust system is essential because particles of copper adhere to the hydrogen bubbles and, if they are near the overflow of the cone, the particulate copper is transported into the tailings. Thus, it is necessary to install a battery of baffles in the upper part of the cone to divert the gas towards the center and allow the bubbles to burst and release the particulate copper to settle back into the precipitation zone. As this copper settles back into the apex of the cone, it is subjected to attrition and peening so that it is necessary to discharge the bed of the cone periodically to prevent attrition of the particulate copper to form a colloid which would not settle and would report as a loss in the tailings overflow.

It has been found that to obtain the best operation of the precipitation cone on a batch-continuous basis approximately 50% of the iron requirement should be fed into the cone during a 5 min time interval. The remainder of the iron is fed during the next fifteen minute period at a gradually decreasing rate. In this manner, a bed is built rapidly within the cone so that effective fluidization of the particulate iron and efficient utilization of the metallic iron as a copper precipitant are obtained. Under these conditions, the competing reactions between copper, acid, and metallic iron are regulated, the precipitation reaction being favored over the iron dissolution-hydrogen evolution reaction. In this manner, the pH range of the effluent solution in a precipitation cone is from 2.5 to 3.0 pH while in a conventional launder plant it might range from 3.0 to 3.5 pH.

ADVANTAGES OF CONE PRECIPITATOR USING PARTICULATE IRON

The advantages of a cone precipitation plant over a launder precipitation plant are many. For example, a cone precipitation plant capable of treating 10 million gal of solution per day would require an area of 10,000 sq ft whereas a launder plant similar to the Utah precipitation plant at Bingham Canyon would require an area approximately 10 times as great. Handling of the precipitant is time consuming in a launder plant because of the large bulk per unit weight of iron. The bulk density of the particulate iron samples tested varies from approximately 75 to 200 lbs per cu ft, and the material is free flowing so that it can be handled with conventional materials handling equipment. In a cone precipitation plant, the copper precipitates are discharged automatically into a filtration system while in the launder plant the precipitates are washed manually from the launders. It has been found generally that the iron efficiency in the cone plant is superior to the launder plant; in fact, in treating the same solution through the launder plant and through the prototype cone, it was found that savings of 0.5 to 1.0 lb of iron per lb of copper could be realized. In addition to this saving in iron, essentially all of the soluble copper is precipitated from the solutions so that a circulating load of dilute copper solution to the leaching system is eliminated.

In the operation of the prototype precipitation cone, it has been demonstrated conclusively that the decrease in the rate of hydrogen evolution is a sensitive indirect measure of the completeness of the precipitation reaction which can be used for process control. The only chemical analyses required are for accountability because of the completeness of the precipitation reaction within the cone. Another advantage in collecting the hydrogen in a central exhaust system is to operate under a slight pressure of atomic hydrogen, which seems to accelerate the copper precipitation rate and to prevent reoxidation of ✓ the ferrous iron.

It is interesting to note that when particulate iron is used as the precipitant, copper recoveries from solution are in the range of 95% before a bed of dynamically suspended precipitant is established and, after the bed has been established, recoveries of more than 99% are common regardless of the type of precipitant used and the amount of metallic iron available for precipitation. Thus, it is possible to precipitate 99% of the copper even when less than 1% metallic iron is available in the partially spent precipitant, which insures excellent iron efficiency. After a batch of precipitant has been added and is essentially utilized, a system of valves is actuated in which the solution inflow is shut off and the precipitate that collects in the lower portion of the cone flows out of the valve into a receiving tank from which it flows by gravity into a filter. The precipitate slurry is dewatered in a filter from approximately 15% to 50 to 60% solids and the filter cake is then discharged as a moist cake onto a conveyor for transport to the next processing step.

SOURCES OF PARTICULATE IRON

Sponge iron produced by the direct reduction of magnetite concentrates, iron ore, or pyrite cinders may be considered as sources of high grade reactive precipitants. Some ten different sources of this directly reduced iron have been made available for testing at the Research Center and at the Utah Copper Div. in the prototype cone. All of these precipitants were found to be suitable for the process. Factors of cost and tonnage availability remain to be resolved before any of these are used as the source precipitant. Commercial, high purity iron powder is also satisfactory; however, the cost of such a precipitant is not competitive with other indicated sources. Detinned scrap or burned tin cans are suitable precipitants for launder plants or for the cone precipitator described by Spedden, Malouf, and Prater.[2] Because of the necessity of keeping the entire bed of precipitant in dynamic suspension and maintaining a clear overflow, it is not possible to suspend large pieces of scrap iron satisfactorily. Cast iron or steel chips have been considered a potentially inexpensive source of precipitant; however, as produced, they contain grease and are relatively coarse. Degreasing of the chips and grinding them to approximately 10 mesh is required to prepare them for use in the precipitation cone, thus adding to the cost of iron. This type of precipitant has also been tested in the pilot plant.

At the present time, Kennecott is investigating the recovery of a suitable precipitant from reverberatory slag. Selective reduction of the slag with coke and pyrite yields a metallic product that is relatively brittle. Products of varying composition have been tested in the prototype cone, and again it was determined that this type of iron is a suitable precipitant with the same favorable characteristics of copper precipitation and iron utilization as the other iron samples that have been evaluated. Several potential producers of directly reduced iron have graciously supplied samples, some of them being tonnage samples for evaluation in the prototype cone. Other smaller samples have been evaluated in a cone precipitator with flow rates of about one liter per min. In these small scale tests, fluidization characteristics of the particulate iron samples are determined and the particle size of the precipitant is evaluated. Generally it is necessary to have a particulate iron of -35 mesh size. However, when sponge iron is used and a spongy product with a relatively high specific surface area is used, a 10 mesh product has been found satisfactory.

SUMMARY

At the present time, a prototype precipitation cone is being operated to establish operating parameters and to develop engineering data. A cone precipitation plant utilizing particulate iron as the precipitant to recover some 400,000 lbs of copper per day at the Utah Copper Div. of Kennecott Copper Corp. will be installed as soon as an economic source of particulate iron precipitant can be developed. Several potential suppliers have submitted samples which were found to be satisfactory. However, no firm contract has been made because of the difficulty of producing a particulate iron precipitant competitive with current scrap iron prices. Kennecott is now evaluating the previously mentioned process for the recovery of iron precipitant from copper reverberatory slag which, as a captive source, would make it unnecessary to rely upon outside suppliers.

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APPENDIX D

I.C. 8341 U.S. BUREAU OF MINES

COPPER LEACHING PRACTICES IN THE WESTERN UNITED STATES

COPPER RECOVERY

Copper is presently recovered from leach liquors almost entirely by means of precipitation by metallic iron. Copper in the vat leach liquors at Inspiration Consolidated Copper Co is recovered by electrolytic deposition. Other methods of precipitation have been suggested by Baarson and Ray (6) and Croasdale (20). Numerous investigators, including Forward (31), Peters and Hahn (59), and Schaufelberger (65), have shown that at elevated temperatures and pressures, copper can be precipitated from solution by hydrogen reduction.

Bagdad Copper Corp. is constructing a plant to produce copper powders by application of hydrogen reduction principles. Other methods of recovering copper from solution, suggested by Agers and coworkers (1-2), Fletcher and Flett (29), Quarm (62), and Swanson and Agers (80), are by solvent extraction and ion exchange. At present, however, copper recovery is accomplished in practice only by the methods of precipitation by metallic iron and electrolytic deposition, and the following descriptions focus attention on these methods.

Precipitation by Iron

Metallic iron precipitates copper from solution according to the well-known reaction:

$$CuSO_4 + Fe = Cu + FeSO_4$$

In practice copper is recovered from copper-bearing pregnant liquors almost entirely by application of the principle of this equation.

Sources of Iron

Shredded scrap cans (Fig. 23) are the chief source of iron used as a precipitant in the recovery of copper from copper-bearing solutions. They are

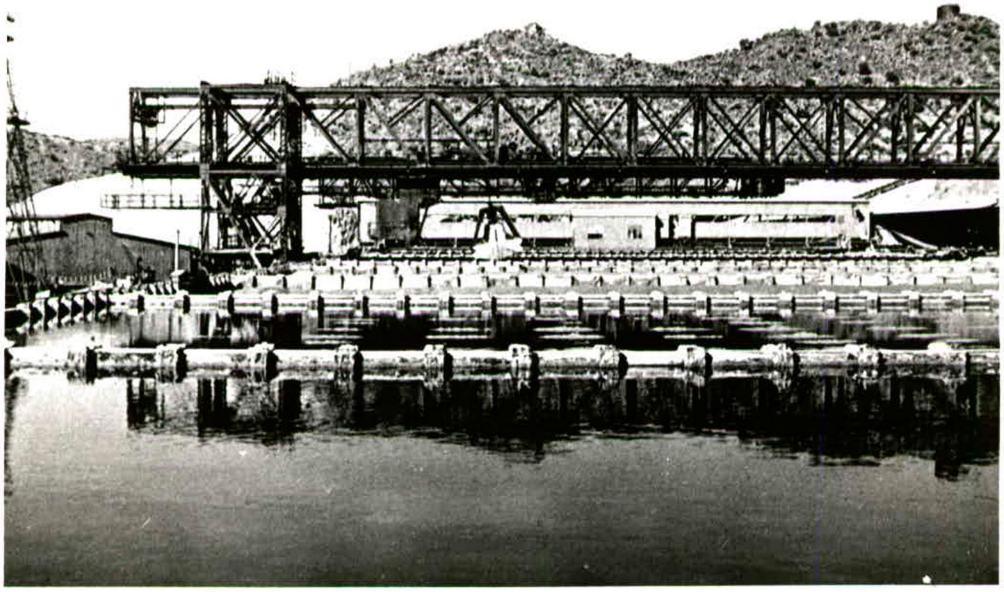

FIGURE 22. - Vat Leaching Operation, Inspiration Consolidated Copper Co. Intense blue solution contains as much as 30 grams per liter of copper.

FIGURE 23. - Stockpile of Partially Oxidized, Burned, and Shredded Cans at Mineral Park Precipitation Plant, Duval Corp. Brown color is typical of the oxidized detinned and shredded cans.

first burned to remove paper labels and organic material. The tin content of scrap cans is often removed by a caustic leach process. The principal Western suppliers of shredded, detinned cans are Proler Steel Corp., with plants at Houston and El Paso, Tex., and San Francisco, Calif., and the Los Angeles By-Products Co., with plants at Bakersfield and Los Angeles, Calif.

Scrap punchings and clippings produced during the manufacture of cans has attained significant importance during recent years as a source of iron for copper precipitation. These materials are good precipitators for copper because of their relative uniformity in size and absence of deleterious impurities, such as organic materials and other solids. Proler Steel Corp. has recently erected a shredding plant near Copperton, Utah, to supply the Bingham Canyon precipitation operation of Kennecott Copper Corp. The product will be used in the newly installed cone precipitators.

Another source of iron that has been used as a precipitant is turnings from lathes and other cutting machines. This type of precipitant has proved unsatisfactory, however, because of inclusions of oil and other lubricants. An additional deterrent to the use of turnings is the slow precipitation of copper attained because of the low surface-to-weight ratio of the iron.

OK in agitators.

Heavy scrap iron has been used as a precipitant for copper in several operations. Low cost of the scrap iron obtained from the mine and milling operations has made this type of precipitant attractive. Deterrents to using heavy scrap as a precipitant, however, are the difficulty in handling the scrap during loading and cleaning of the precipitation launders and the small surface area per unit of weight available for reaction. A few operators have used heavy scrap in the head section of the launders to reduce the ferric iron content of the pregnant solution and thereby reduce the shredded can consumption in the remaining cells, since ferric sulfate and sulfuric acid are responsible for the iron consumed in the launders according to the reactions:

$$Fe_2(SO_4)_3 + Fe \rightleftharpoons 3FeSO_4;$$

$$H_2SO_4 + Fe \rightleftharpoons H_2 + FeSO_4.$$

Finely divided sponge iron is an effective precipitant of copper, although at present it is not competitive in cost with available shredded scrap cans. The principal advantage in using sponge iron seems to be the relatively faster copper precipitation rate obtained with particulate iron as compared to that obtained with more massive scrap iron products. According to Back (7), sponge iron is most effective as a precipitant for copper when used in conjunction with cone-type precipitators. High-grade magnetite, iron ore, and pyrite cinders are the chief sources for producing sponge iron suitable as a reactive precipitant for copper. Kennecott Copper Corp. is investigating the feasibility of producing sponge iron for copper precipitation by the reduction of the iron in copper reverberatory slag from the firm's smelter.

Types of Precipitation Plants

Copper is precipitated from solutions in a series of gravity launders or in cone precipitators. Descriptions and examples of each type currently employed are given in addition to illustrations of certain aspects of operating practices.

Precipitation Launders

Current precipitation practice is typified by the use of gravity launders. Typical top, end, and side views of a series of gravity launders are shown in figure 24. Launder dimensions at numerous operations are given in table 1. where is it?

Ballard (9), Jacky (44), and Power (60) have described the precipitation launders of The Anaconda Company and the American Smelting and Refining Co. Many launders are of concrete construction, with wooden planks to protect the top of the launder walls. Plywood and/or stainless steel gratings are used to support the iron precipitant within the launders.

The launders of the precipitation plants are charged with cans by means of different mechanical devices. The launders of the Copper Cities precipitation plant of the Miami Copper Co. are loaded with cans that are discharged from a belt conveyor by a moving tripper. A crane-mounted magnet is used to charge and clean launders at the Silver Bell precipitation plant of the American Smelting and Refining Co. A gantry-mounted clamshell bucket is used to load cans into the launders at the Butte precipitation plant of The Anaconda Company. Front-end loaders are used at the Morenci precipitation plant of Phelps Dodge Corp., and at the Mineral Park plant of the Duval Corp. Part of the cells at the latter plant are loaded by a belt conveyor (fig. 25). Kennecott Copper Corp. at its Bingham Canyon original launder precipitation plant uses a forklift loader to load the launders with cans.

Copper-bearing liquors are introduced into the upper end of gravity launder plants through a solution feed launder and allowed to trickle downward through the shredded cans. Launder arrangement is such that solution flow is in series. Baffles and pumps are utilized in order that any launder may be bypassed and solutions returned to any particular launder. The copper, ferric iron, and free sulfuric acid content of the pregnant liquors and the surface area of iron available for reaction are the more important factors in determining the rate and degree of recovery of copper in each particular launder. Copper usually is washed three times a week from the first few launders, which in most plants precipitate more than 60 percent of the total copper. The copper precipitated in the remaining launders is removed usually from once a week to once a month. Tailing (barren) solutions discharge by gravity from the lower ends of the plants.

The "Yerington-type" plant differs from those described in that the pregnant solution flow is upward through the iron precipitant. The pregnant liquor is introduced into the launders under pressure through three parallel, 4-inch plastic tubes which lie in gutters in the floors of the launders. The tubes have been perforated with 5/8-inch holes spaced from 1 to 3 feet apart

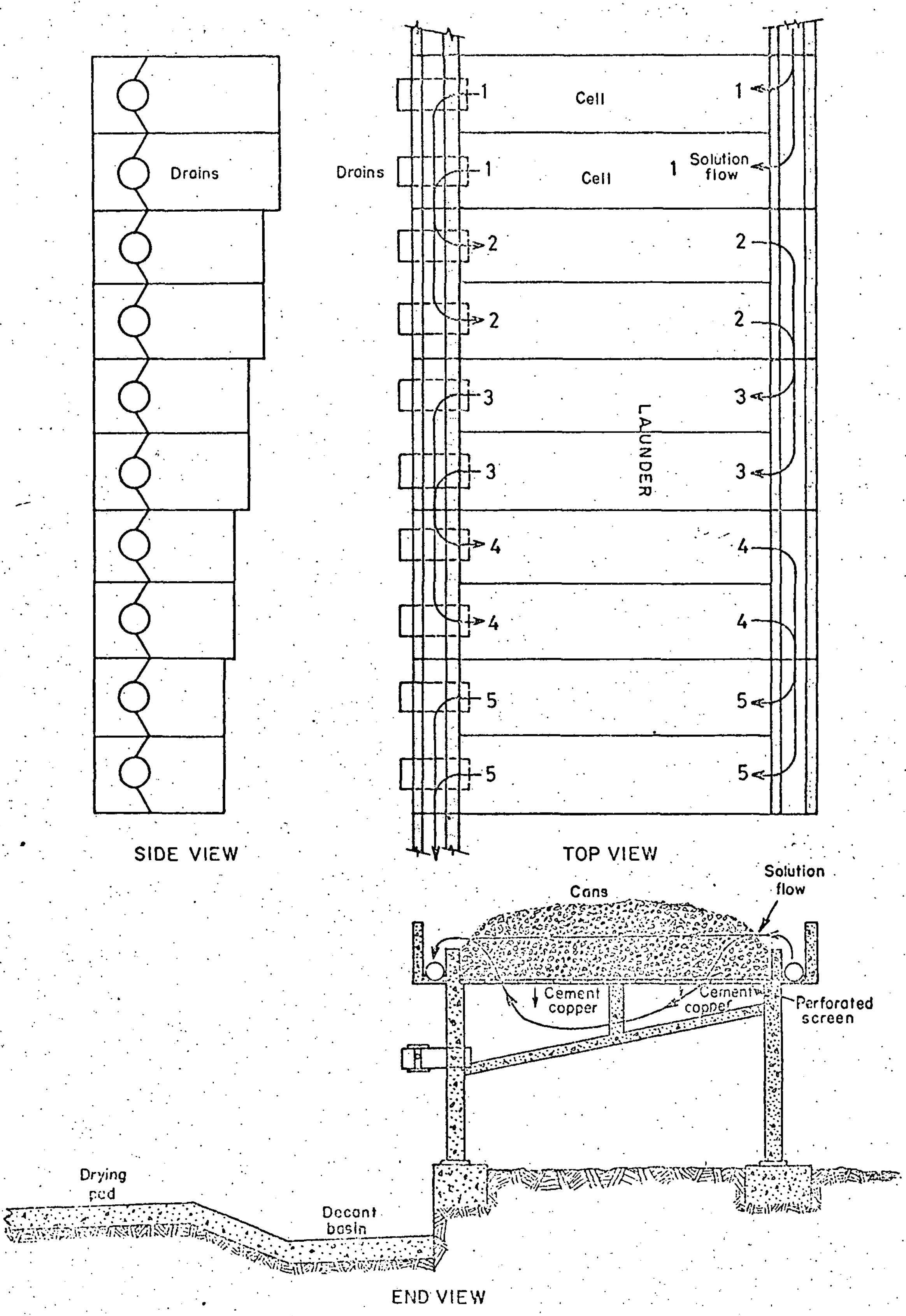

FIGURE 24. - Typical Design of Gravity Launder-Precipitation Plant.

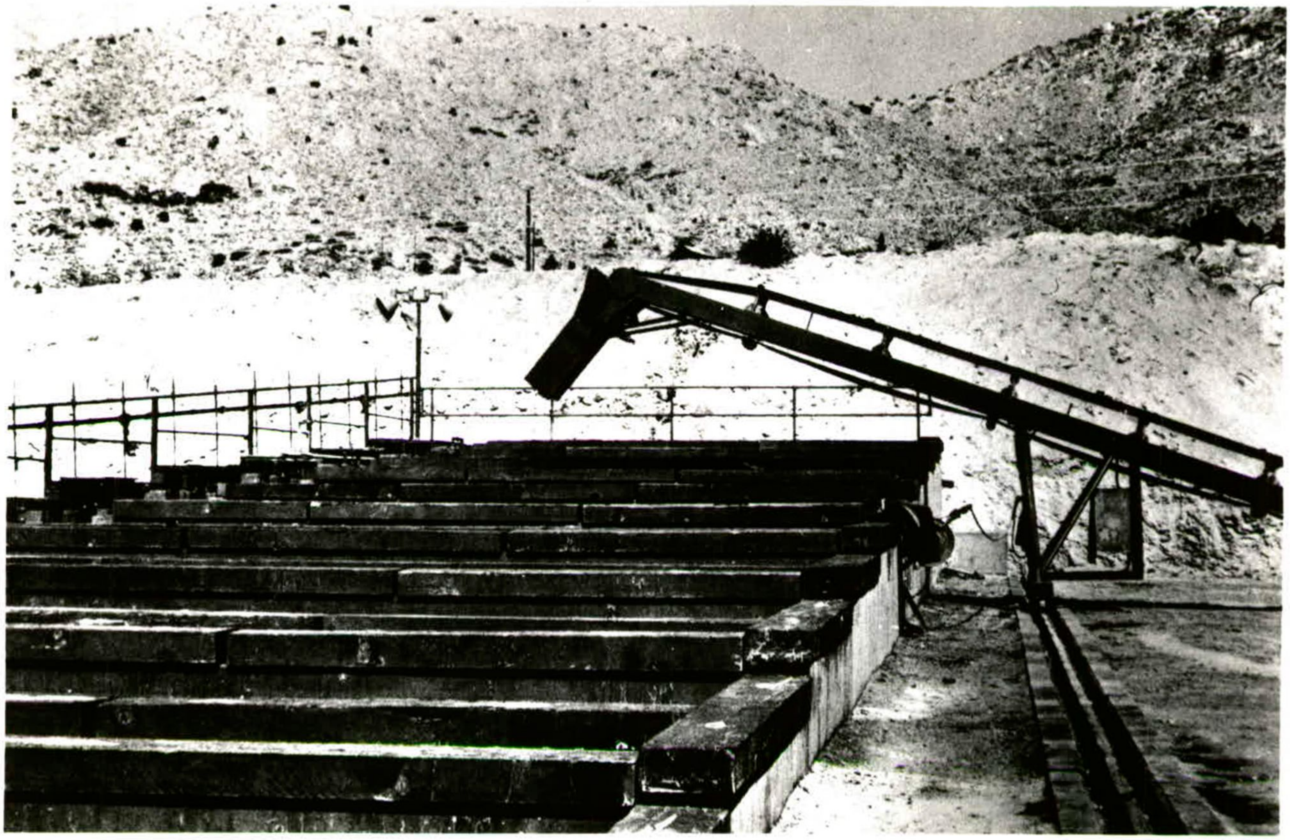

FIGURE 25. - Sliding Conveyor Used to Load Cans Into Cells of Precipitation Plant at Mineral Park Property, Duval Corp.

(fig. 26). High-solution velocities are maintained in the launders and a coarser cement copper product is formed. Higher purity and lower moisture content are synonymous with this coarser product.

Pregnant liquors introduced into, and tailing (barren) solutions discharged from, precipitation plants are sampled continuously. The solutions are analyzed for pH and copper, iron, and acid content by wet chemical methods, although other methods are being adopted such as X-ray, electrolytic, and atomic absorption. Typical analyses of such solutions from numerous precipitation plants in the Western States are given in table 1. The pH of the pregnant liquors varies from 1.4 to 3.5, copper content from 0.75 to 7.0 grams per liter, ferrous iron content from 0.00 to 3.60 grams per liter, ferric iron content from 0.05 to 3.00 grams per liter, and free acid content from 0.04 to 7.50 grams per liter. The recovery of copper is high in all operations, averaging well above 90 percent. The copper content of the tailing solutions therefore is low and ranges from less than 0.01 to 0.36 gram per liter. The pH of the tailing solutions is higher, in the range of 2.4 to 4.4, as a result of the decrease in acid content. The decrease in acid and an increase in total iron content (almost entirely ferrous iron) is the result of the several iron-consuming reactions that take place during precipitation.

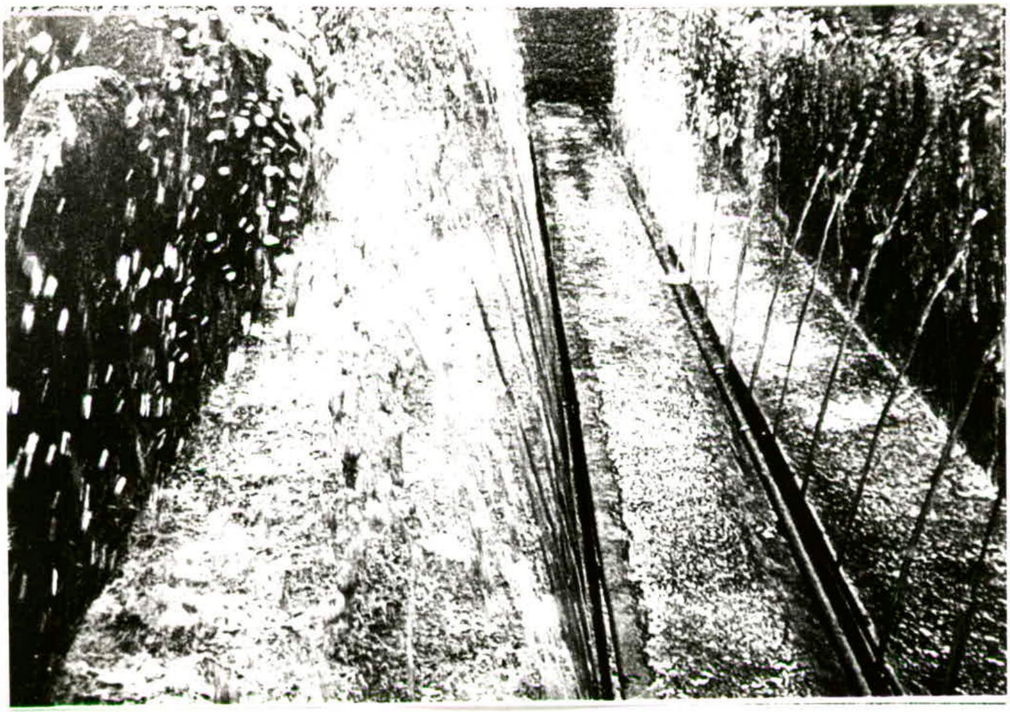

FIGURE 26. - View of Empty Launder at Weed Heights Precipitation Plant of The Anaconda Company Showing Pregnant Solution Introduction Under Pressure Through Perforated Plastic Pipes. Blue-green color of solution is indicative of copper content.

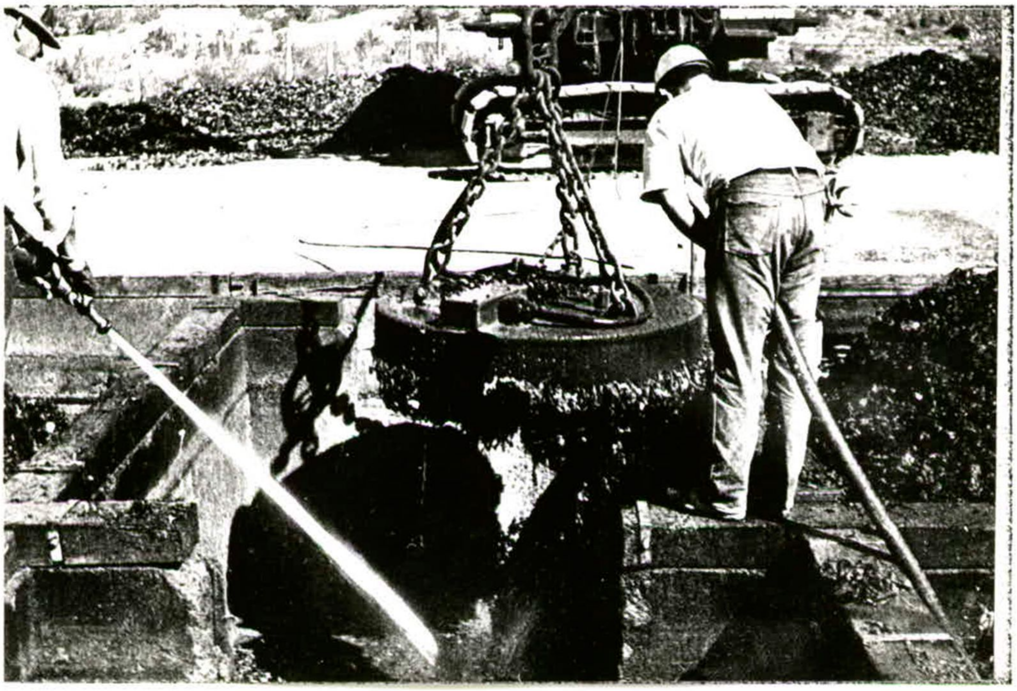

FIGURE 27. - Removing Red-Brown Cement Copper From Cells by High-Pressure Hoses at the Silver Bell Precipitation Plant, American Smelting and Refining Co.

The amount of iron consumed per pound of copper precipitated (commonly known as the "can factor") is an important concern in all plants. Iron consumption varies from operation to operation, but is always higher than the theoretical 0.88 pound of iron per pound of copper required for the iron replacement by copper reaction. The can factors range from a low of 1.2 to a high of 2.5 at the various operations. A list of can factors is given in table 1.

Cone Precipitators

A recent innovation in the recovery of copper from copper-bearing solutions is the cone-type precipitator developed by the Kennecott Copper Corp. (8), and described by Spedden and coworkers (68). A cutaway diagram of the precipitation cone is shown in figure 28. Copper-bearing solution is pumped through a manifold system in the bottom of the cone and is injected through nozzles into shredded, detinned iron scrap, which is semicontinuously loaded into the inner cone above a heavy-gage, stainless steel screen. The injection of the solution under pressure into the iron material has a dual effect. Copper is rapidly precipitated and quickly removed from the iron surfaces by the turbulent action, creating fresh, clean surfaces of iron for continued precipitation of the copper.

Methods of Handling Cement Copper

Various methods are used to handle the cement copper following its precipitation in the launders or in the cones. The purpose in each instance is the same, to obtain as pure a product as possible with minimum moisture content.

Removal From Precipitation Unit

The most frequently used method for removing cement copper from precipitation launders is with high-pressure streams of water. The water is pumped at about 100 pounds of pressure to wash hoses equipped with quick shutoff nozzles (fig. 27). The cement copper is washed from the unreacted iron, and the resultant slurry is emptied into decant basins through drain valves in the bottoms of the cells.

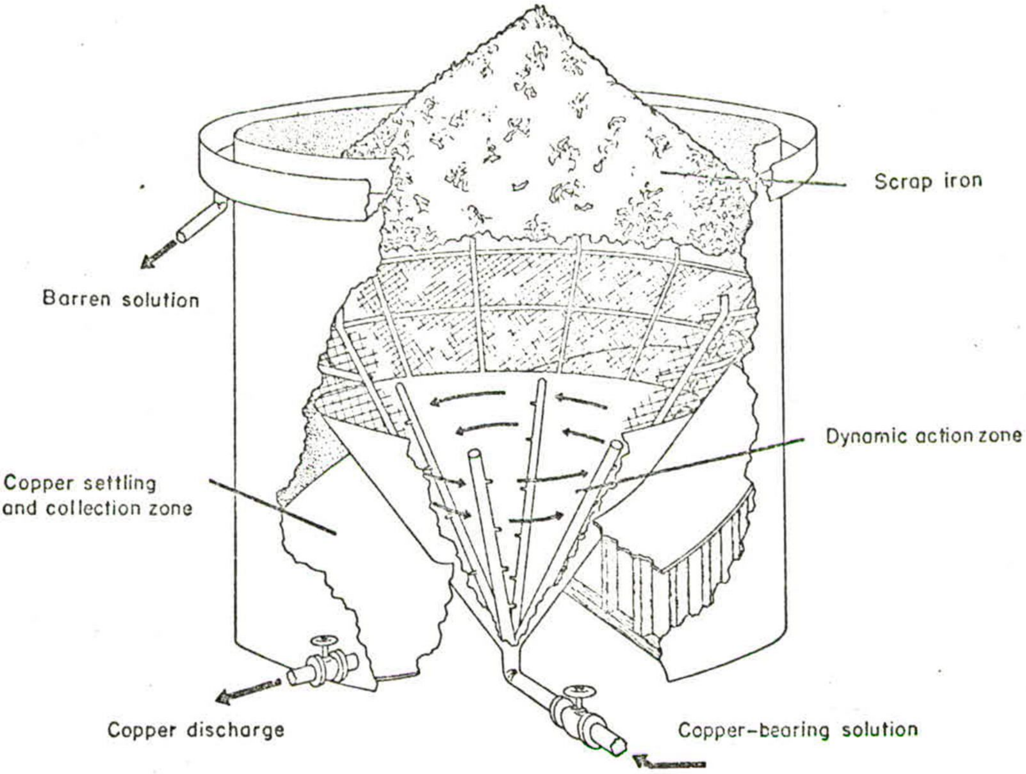

FIGURE 28. - Cutaway Diagram of Cone Precipitator Designed by Kennecott Copper Corp. (Neal Bishop, Kennecott Research Center.)

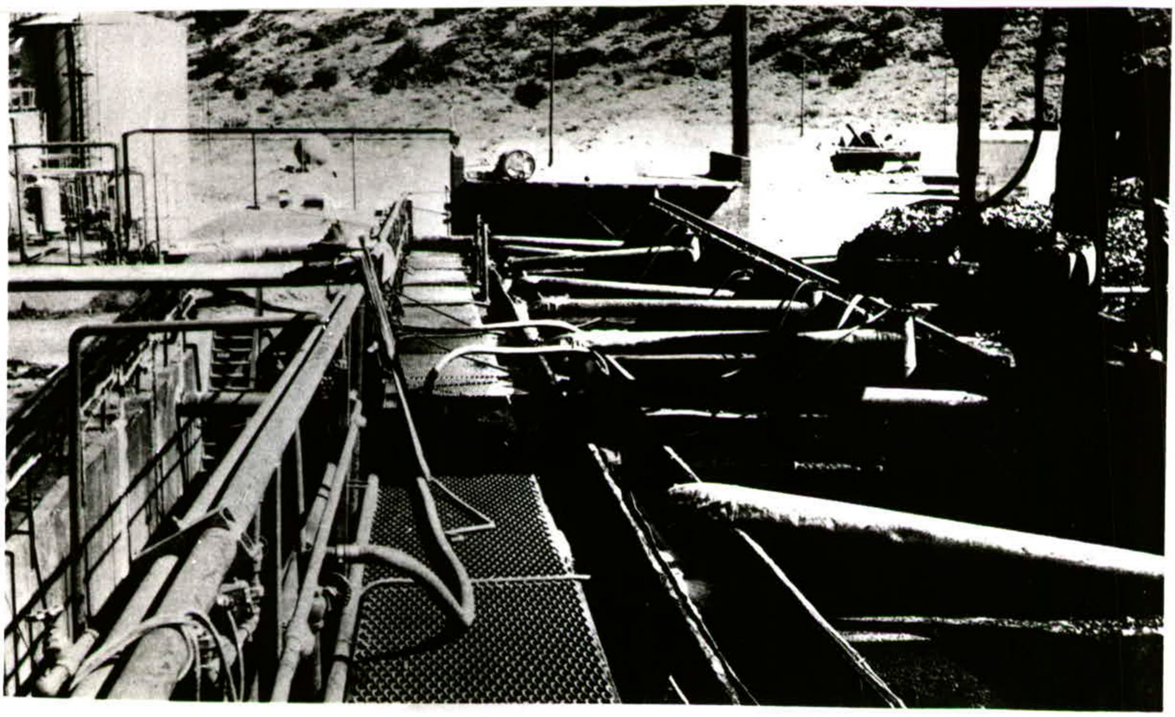

FIGURE 29. - Airlift Pumps Used for Removing Cement Copper From Precipitation Cells, Bagdad Copper Corp.

Airlift pumps (fig. 29) are used by Bagdad Copper Corp. to remove the cement copper from the bottom of the launders after the cement copper has been washed from the unreacted cans. The slurry of cement copper and water, after being discharged from the airlift pumps, flows by gravity into settling basins (fig. 30).

Hydraulic slushers (fig. 31), consisting of a trussed bridge which spans all six of the parallel launders at The Anaconda Company's Butte precipitation plant, are used to agitate the cans in the launders, remove the cement copper from the cans, and slush (sweep) the cement copper along the launders to drop tanks. After the drop tanks are nearly full, the water is decanted by pumping and the cement copper is loaded into railroad cars by an overhead clamshell bucket.

A few operations use a gantry-operated clamshell bucket to unload the cement copper and unreacted cans from the precipitation launders. The combined material then is charged into a trommel (fig. 32) fitted with stainless steel screens having small-diameter openings to separate the cement copper and unreacted cans from one another. The unreacted cans are returned to the precipitation launders and the cement copper is discharged into settling basins.

The cement copper produced in Kennecott's cone precipitators settles through the stainless steel screen, drops to the bottom of the tank, and is removed intermittently through a pneumatically operated valve. Higher grade products, analyzing 90 to 95 percent copper, are obtained by this method of precipitation. Other benefits from this method are a substantially lower consumption of iron and the elimination of the need for the usual labor involved in a launder-type plant.

Decanting

The cement copper is allowed to settle in basins. The clear water is decanted and returned by pumping to the precipitation launders to prevent the loss of any suspended fine particles of copper.

Drying

The cement copper is removed from the settling basins by various mechanical devices, such as front-end loaders and crane-mounted clamshell buckets. The cement copper removed from the settling basins is dried before shipment to smelters or other markets. Concrete drying pads are generally used, and in most instances are constructed adjacent to the settling basins. The cement copper at the Weed Heights, Nev., operation of The Anaconda Company is dried on large gas-fired hotplates ~~(fig. 34)~~. not presented here .

Atmospheric drying of the cement copper reduces the moisture content from about 50 percent to between 25 and 30 percent. The cement copper dried by hotplates at Weed Heights contains 15 percent moisture.

Plate-and-frame filter presses are used to reduce the moisture content of the cement copper produced in the cone precipitators at the Bingham Canyon

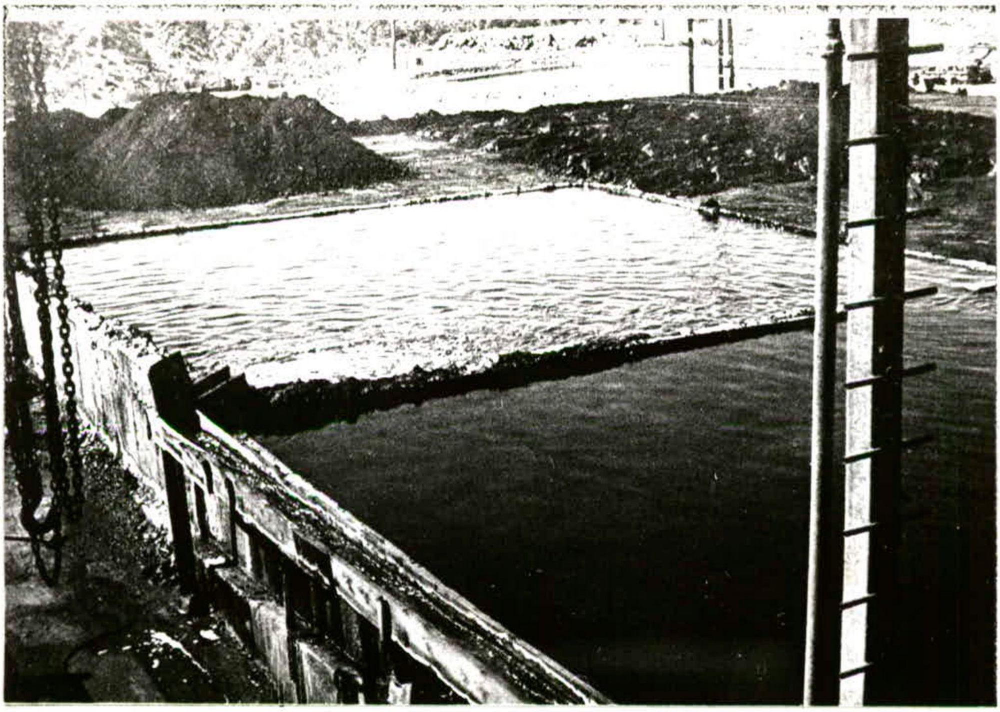

FIGURE 30. - Decant Basin Adjacent to Precipitation Plant, Bagdad Copper Corp. Blue is copper-bearing water in decant basin; wooden trough in left foreground is delivering red-brown slurry of cement copper and water from the plant's precipitation cells to decant basin.

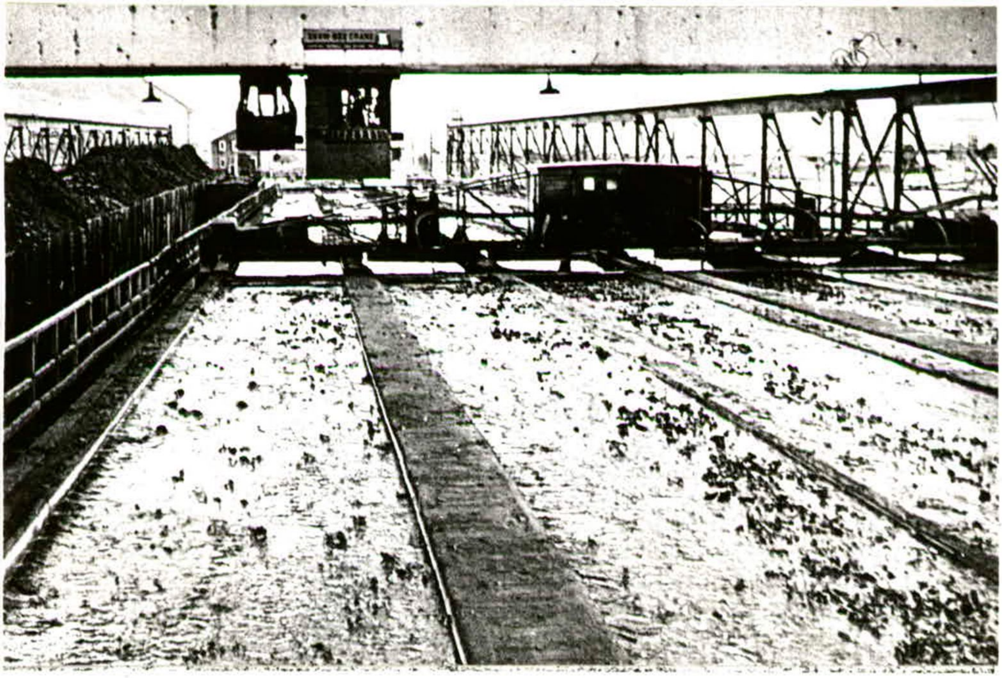

FIGURE 31. - Hydraulic Slushers Used To Remove Cement Copper From the Cells of the Butte Precipitation Plant, The Anaconda Company. The copper-red color of the cement copper in the cells can be seen as copper replaces the iron in the cans.

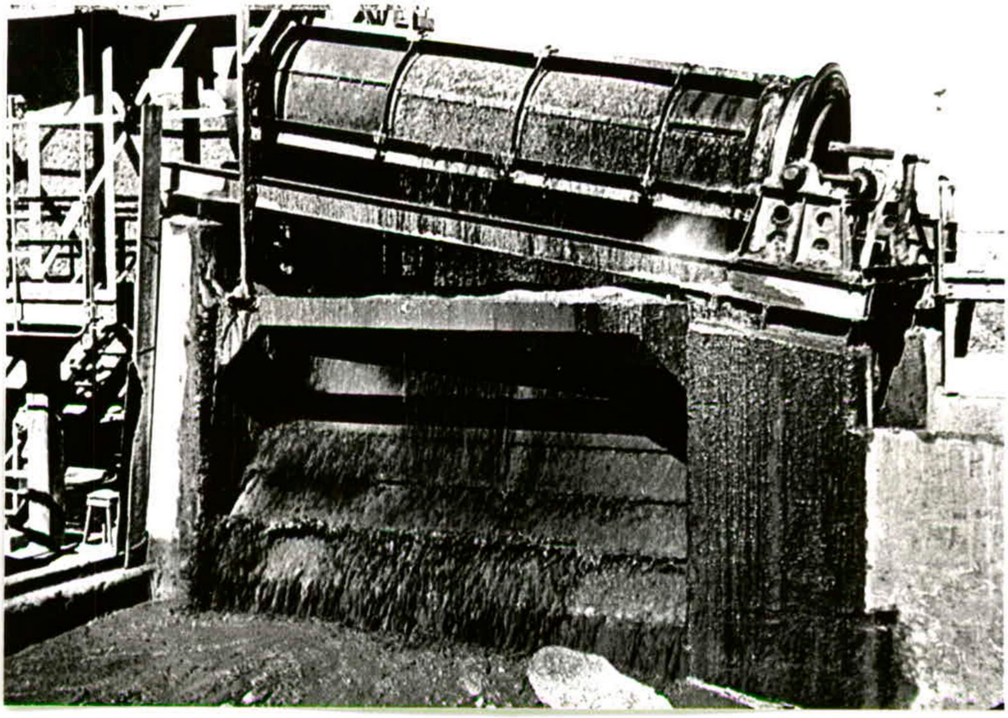

FIGURE 32. - Trommel Used for Screening Cement Copper at Weed Heights Precipitation Plant, The Anaconda Company. Red-brown cement copper is passing through the screen sections on the trommel.

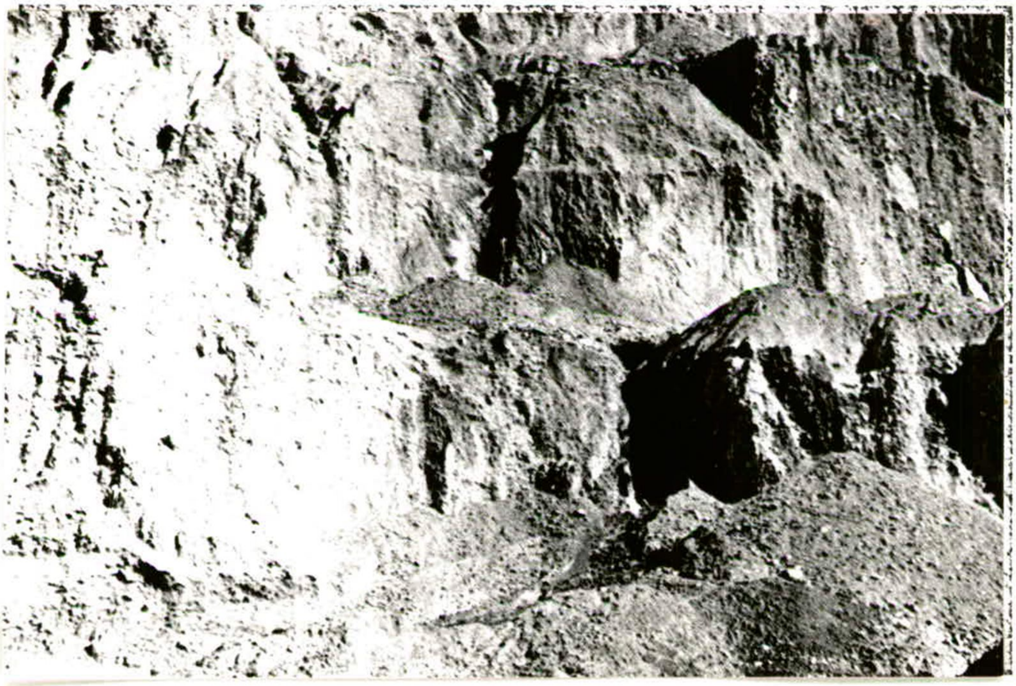

FIGURE 33. - Impervious Layers of Brown Iron Salts Above Blue-Green Copper Minerals in Leached, Block-Caved Area of Underground Mine, Inspiration Consolidated Copper Co.

operation of Kennecott Copper Corp. The final product contains from 8 to 10 percent moisture.

Additional Treatment

The cement copper produced in a few plants is processed further to increase the purity of the final product. The cement copper is screened through small-mesh vibrating screens by the Inspiration Consolidated Copper Co. Flotation techniques are used at the Bingham Canyon operation of Kennecott Copper Corp. to improve the quality of part of the cement copper from the launder plant for direct sale to consumers.

Electrolytic Deposition

The Inspiration Consolidated Copper Co. recovers copper from vat leach liquors by electrolytic deposition using insoluble anodes. The process is termed electrowinning. Liquors containing from 25 to 30 grams per liter of copper serve as the electrolyte for the electrowinning process. A description of the electrowinning process and operating data for the Inspiration plant have been given by McMahon (56).

The principal advantages of this process are that electrolytically refined copper does not require additional treatment--that is, the usual smelting operation required for copper production is eliminated--and sulfuric acid and ferric sulfate needed as solvents during the leaching operation are regenerated.

RESEARCH

Most companies currently engaged in copper leaching are researching or are planning to research the following: (1) The solvent extraction of copper from leach liquors; (2) the use of other precipitants such as sponge iron and aluminum; (3) bacterial leaching; (4) the nature of dumps (by drilling and analysis of drill samples); (5) chemical relationships of dump materials and solutions; (6) effectiveness of acid and other solvent additions to the leaching solution; and (7) dump emplacement techniques.

SUMMARY

A more rapid and more complete recovery of copper from leach dumps is the ultimate concern of all companies engaged in copper leaching. Close attention must be given to numerous problems inherent in the leach cycle to achieve a satisfactory leach rate and recovery of the copper.

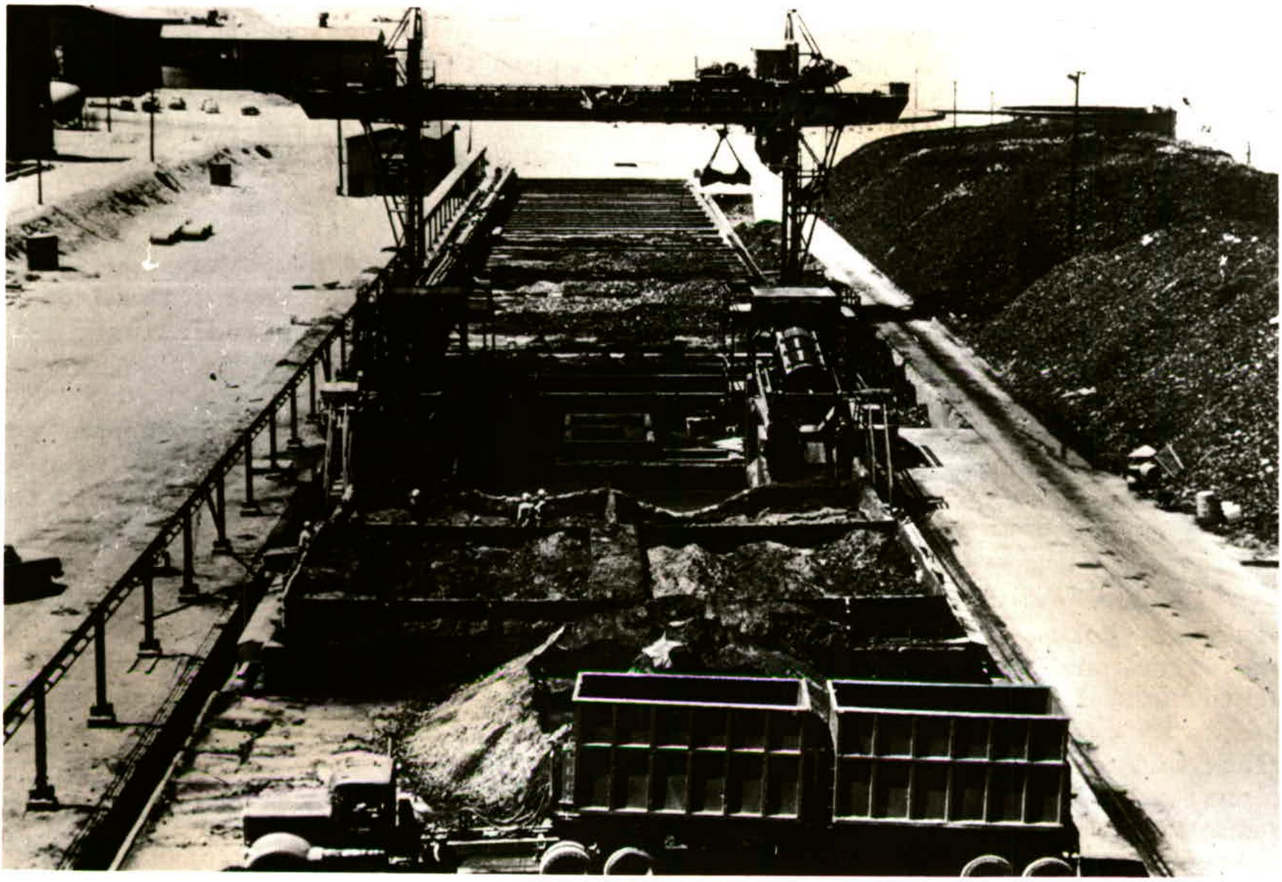

FIGURE 34. - Hotplates (in Foreground) Used for Drying Cement Copper at Weed Heights Precipitation Plant, The Anaconda Company.

Iron salt precipitation on the surface or within leach dumps is one of the principal problems encountered in leaching. Impervious layers formed within leach dumps act like blankets through which leach solutions cannot pass. Large volumes of copper-bearing material are not contacted by the leach solutions. Examples of this phenomenon can be seen in the Ohio Copper Co. mine presently exposed in the open-pit mining operation of Kennecott Copper Corp. at Bingham, Utah, and at the Inspiration Consolidated Copper Corp. at Inspiration, Ariz. (fig. 33). A partial solution to this problem seems to be to minimize the iron salt precipitation within the dumps by maintaining a low solution pH by acid additions to the leach solutions.

Channeling is created in many leach dumps as a result of the methods employed to build the dumps. Solution flow often is horizontal following alternating layers of coarse and fine material; hence, large volumes of mineralized material may not be contacted by the leach solutions. Close attention to the method of constructing the leach dump is needed to minimize size segregation.

TABLE 1. - Data from copper leaching and precipitation

Company	Source materials			Leaching					
					Leach material				
	Host rock	Principal copper minerals	Minor copper minerals	Type of leaching	Quantity 1,000 tons (est.)	Geometry: Area, 1,000 sq ft	Geometry: Maximum height, ft	Ground preparation	Method of emplacement
American Smelting and Refining Co., Silver Bell unit, Silver Bell, Ariz.	Alaskite, dacite porphyry, monzonite.	Chalcocite and chrysocolla.	Chalcopyrite, azurite, malachite, cuprite.	Dump......	[1]30,000	5,650	200	Leach material is deposited on the existing topography.	Leach material is hauled and dumped by trucks. Edges and surfaces of dumps are leveled by bulldozer.
The Anaconda Company:									
Butte, Mont..............	Quartz monzonite......	Chalcocite..........	Chalcopyrite, bornite, azurite, malachite.	....do....	10,000 [5]23,000	390 860 400	195 175 50	The main dump is underlain by an impervious pad. See complete description in body of report under ground preparation for dump emplacement.	..................do..................
Yerington mine, Weed Heights, Nev.	..........do..........	Chrysocolla.........	Tenorite, malachite, cuprite, azurite.	Dump and vat.[10]	30,000	20	165	The leach dumps have been deposited on a dry lake bed which was leveled by bulldozer and compacted by sheepsfoot rollers.	..................do..................
Bagdad Copper Corp., Bagdad, Ariz.	Monzonite porphyry....	Chrysocolla, malachite, azurite.	Tenorite and cuprite....	Dump......	[1]40,000	2,390	240	Leach material is deposited on the existing topography.	..................do..................
Duval Corp.:									
Esperanza mine, Sahuarita, Ariz.	Quartz monzonite, rhyolite flows, quartz diorite.	Chalcocite, some chalcopyrite.	Cuprite, malachite, azurite, tenorite.	....do....	19,000	830	220	................do................	..................do..................
Mineral Park, Ariz........	Quartz porphyry and quartz monzonite.	Chalcocite..........	Chalcopyrite, covellite, turquoise.	....do....	[1]5,500	340	250	................do................	..................do..................
Inspiration Consolidated Copper Co., Inspiration, Ariz.	Schist and granite porphyry.	Chrysocolla, azurite, malachite.	Chalcocite and chalcopyrite.	Dump, inplace, vat.[10]	30,000	750	200	................do................	..................do..................
Kennecott Copper Corp.:									
Utah Copper Division, Bingham Canyon, Utah.	Quartz monzonite......	Chalcopyrite........	Chalcocite, covellite, bornite; oxide copper minerals 0.05-0.07 pct.	Dump......	[16]4,000,000	31,000	1,200	................do................	Leach material is hauled and dumped by trucks and train. Edges and surfaces of dumps are leveled by bulldozer.
Chino Mines Division, Santa Rita, N. Mex.	Granodiorite porphyry.	Chalcocite..........	Chalcopyrite and nonsulfide copper minerals.	....do....	425,000	28,000	300	................do................	..................do..................
Ray Mines Division, Ray, Ariz.	Schist and diabase....	.........do.........	Cuprite, native copper, chalcopyrite, chrysocolla, azurite, malachite.	Dump and inplace.	94,000 31,000 44,000 [2,4]17,500	12,000 2,000 6,000 NA	125 80 85 NA	................do................	Leach material is hauled and dumped by trucks. Edges and surfaces of dumps are leveled by bulldozer.
Miami Copper Co., Miami, Ariz.:									
Castle Dome unit..........	Quartz monzonite and granite porphyry.	Chalcopyrite and chalcocite.	Covellite, cuprite, azurite, malachite, chalcanthite, turquoise.	Dump......	48,000	NA	NA	................do................	..................do..................
Copper Cities unit........	Quartz monzonite......	.........do.........	Covellite, turquoise, malachite, azurite.	....do....	NA	NA	150	................do................	..................do..................
Miami unit................	Schist and granite porphyry.	Chalcocite..........	Chalcopyrite, bornite, covellite, malachite, azurite, chrysocolla, cuprite, native copper.	Inplace...	NA	([28])	NA	Surface topography on which solutions are introduced is a depression resulting from block-caving operations beneath.	Copper minerals remain in block-caved stopes, pillars, and capping.
Phelps Dodge Corp.:									
Bisbee, Ariz..............	Quartz monzonite and conglomerate.	Chalcocite, some azurite and malachite.	Some bornite and turquoise.	Dump......	47,000	3,850	170	Leach material is deposited on the existing topography.	Leach material is hauled and dumped by trucks. Edges and surfaces of dumps are leveled by bulldozer.
Morenci, Ariz.............	Quartz monzonite porphyry.	Chalcocite..........	Chalcopyrite, covellite, oxide minerals.	....do....	NA	NA	NA	Strip material is sometimes deposited on preexisting dumps that were deposited on the existing topography.	Leach material is hauled and dumped by side-dump railroad cars.
Ranchers Exploration and Development Corp., Bluebird mine, Miami, Ariz.	Granite porphyry......	Malachite and azurite	None..................	Heap......	500	100	80	Ground is dressed, soil is cemented and covered with diluted tar for curing and sealing purposes.	Leach material is hauled by bottom-dump scrapers to the heaps. Motor grader levels the heap area.
J. H. Trigg Co., Tyrone, N. Mex.:									
Property No. 1............	Quartz monzonite......	Chrysocolla and azurite.	Malachite and tenorite.	....do....	150	20	75	Leach material is deposited on the existing topography.	Blasted material is loaded by scoopers into dumpsters. The dumpsters unload the material on a pile. Bulldozer levels the dump material.
Property No. 2............	..........do..........	Malachite...........	Chrysocolla and azurite.	....do....	100	30	40	................do................	..................do..................
Zontelli Western Mining Co., Page, Ariz.	Sandstone.............	Malachite, azurite, chrysocolla.	Some chalcocite.........	....do....	500	300	40	A special pad is prepared. See complete description in body of report under ground preparation for dump emplacement.	Leach material is hauled and dumped by trucks. The surfaces of the heaps are leveled by bulldozer.

NA--Not available.
[1] In 2 dumps.
[2] Sloping.
[3] Launder tailing solution.
[4] Barren solution sent to dumps.
[5] Main dump is divided into high and low dump sections.
[6] July and August.
[7] January and February.
[8] March.
[9] 6 parallel rows of launders, 5 cells per row.
[10] Data presented in table do not apply to vat leaching. See descriptions of vat leaching in text.
[11] Double cells.
[12] Approximately 2.5 pounds of acid per pound of copper precipitated.
[13] Maximum.
[14] December 1963.
[15] 4-compartment cells; cell dimensions are for each compartment.
[16] Chiefly in East Dump.
[17] Present.
[18] Future.

operations in the Western United States

operation											Precipitation operation															
Method of introduction of leach solutions	Leach solutions								Area seasonal temperature, ° F		Type of precipitant used	Can factor, lb Fe/ lb Cu	Precipitation units				Analyses of solutions, gpl									
	H_2SO_4 addition, gpl	Flow rate to dumps, gpm	Influent			Effluent							Number of cells	Cell dimensions, ft			Influent					Effluent				
			pH	Copper content, gpl	Temperature, ° F	pH	Copper content, gpl	Temperature, ° F	Summer	Winter				Length	Width	Depth	Cu	Fe^{++}	Fe^{+++}	Free acid	pH	Cu	Fe^{++}	Fe^{+++}	Free acid	pH
Ponding and trenching	None	1,000	3.4	<0.01	NA	2.4-2.8	1.09	NA	85-95	50-60	Burned, shredded tin cans.	1.6-2.0	10	12	8	[2]6-1/2-11-1/2	1.09	0.01	0.57	0.60	2.3	[3]0.01 [4].01	2.08 1.67	Trace 0.04	0.08 .06	3.6 3.3
Solutions introduced through perforated plastic pipes spaced 100 ft apart on grid.	0.1	5,000	1.9	.11	85	2.2-2.3	.80-1.00	67	[6]80-90	[7]20 [8]20-45	Burned, compacted tin cans.	1.5	[9]30	100	8-1/2	2-1/2	.87	.09	.41	NA	2.3	.02	1.50	.05	NA	2.8
Spraying............	7.0-8.0	4,000	1.2-1.3	.08	60	1.9	1.20	55	80	53	Burned, shredded tin cans and discarded clippings from can manufacture.	1.3	[11]20	60	20	4	1.20-1.30	.00	1.00	3.90	1.9	.08	1.30-1.40	.00-.20	1.50	3.0
.........do.........	(12)	3,300	1.4-1.6	.02	70	2.1-2.4	1.18	70	95	35	Mine scrap and burned, shredded tin cans.	1.8	22	10	10	6	1.00-1.25	.09	2.00	1.50-2.50	<2.0	.02	3.80-3.90	.10-.20	NA	2.4
.........do.........	.4	[13]1,600	2.5	<.01	[14]46	2.3-3.3	1.32-1.56	[14]63	70-85	45-55	Burned, shredded tin cans.	1.4	[15]12	12	4	[2]5-10	1.60	<.10	.20	NA	2.6	<.01	1.90	Trace	NA	3.9
.........do.........	None	750	3.5	<.01	NA	2.4	1.24	NA	75	40	........do.........	1.4	[15]12	12	5	10	1.24	.01	.47	NA	2.3	<.01	2.45	.01	NA	3.7
Spraying, ponding, and trenching.	2.0	1,700	2.6-3.0	.10	NA	>2.0	1.10	NA	100-102	50-60	........do.........	1.2-1.3	[11]4	60	20	5	1.50	.56	.46	1.00	NA	.01	3.30	Trace	Trace	3.5
Solutions introduced into strips or channels.	.1	[17]8,000-10,000 [18]35,000	2.8-3.0	.12-.18	92-94	2.5	1.80	110-125	75-85	20-25	Burned, shredded tin cans. Shredded clippings	2.5 1.5-1.8	[19]48 [20]26	80 (20)	4 (20)	4 (20)	1.80 1.80	3.36 3.36	1.44 1.44	NA NA	2.5 2.5	.12-.18 .02-.05	8.39 7.00	.00 .00	NA NA	3.5-3.8 2.9-3.0
Ponding and trenching	Varies	13,000-15,000	3.5	.08-.36	70-78	2.5	1.40	90-95	[21]66	[22]43	Detinned scrap (tin cans, light trimmings and punchings, some shreadded auto bodies).	1.0-1.8	30 [24]4	40 (23)	5 (23)	4 (23)	1.32-2.16	NA	.05-1.08	.04-.84	2.5	.08-.36	NA	.06-.24	.08-.22	NA
Ponding.............	Small amounts.	7,000	2.5-3.8	<.06	73-85	2.1-3.0	.90	73-100	85-95	50-60	Clippings discarded from can production. Some burned, shredded tin cans.	1.7-1.8	[25]48-72	40	5	5	.90	<.05	.36-.60	NA	1.5-3.0	.06	2.16	.00	NA	3.0-3.8
Spraying and ponding.	None	NA	2.7	.04-.32	NA	2.3	.85	NA	85-95	50-60	Burned, shredded tin cans.	1.8	5	52	10	4-1/4	.85	.02	1.06	.15	2.3	[3].02 [4].07	2.20 1.26	.06 .30	Trace .05	3.0 2.7
Spraying............	.4	1,800	2.6	.01	73-74	2.5	.75-2.25	73-80	85-95	50-60	........do.........	1.5	24	27	4-1/2	3	.75-2.25	.02	.32	.25	2.5	<.01	1.02	.03	Trace	3.6
Spraying of surface above block-caved areas	4.0	2,000	1.4	.02	NA	2.4	1.75-2.00	NA	85-95	50-60	........do.........	1.4	24	27	4-1/2	3	2.14	1.15	1.05	.25	2.4	.02	4.60	.10	Trace	4.2
Ponding.............	None	2,300	2.8-3.1	.03-.07	[27]90-95	2.0-2.2	.96-1.80	[27]100	85-95	50-60	Burned, shredded cans. Sponge iron from Douglas smelter. Heavy scrap.	2.5	24	25	8	[2]3-10	1.32	3.60	3.00	NA	2.0	.03	8.77	.06	NA	3.1
.........do.........	None	[17]300 [18]5,000	3.8-4.2	.01-.02	NA	2.8-3.2	1.00-3.60	NA	75-85	40-50	Burned, shredded tin cans, heavy scrap, and turnings from shop.	1.3	4	20	20	3	NA	.01	.27	NA	3.0	NA	2.01	<.01	NA	4.4
Percolated through plastic pipes spaced 8 ft apart on surface of the heaps.	[28]50.0 [29]20.0	(30)	2.5	[31].00 [32]3.00	50	1.4-2.8	2.00-8.00	70-85	85-95	50-60	Burned, shredded tin cans, baling wire, shredded car bodies being evaluated.	1.5-1.7	Several	30	3-1/2	8	3.00-7.00	NA	NA	NA	1.4-2.8	NA	NA	NA	NA	NA
Ponding.............	1.7-1.8	1,000	2.2	.18	40-60	3.0-3.5	1.80	40-60	64-95	23-54	Burned, shredded tin cans.	1.5-2.0	4	35	5	4-1/2	1.80	.60-1.80	.24-.72	.30	3.0-3.5	.12	2.40	.06	.06	NA
NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	........do.........	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
Ponding.............	(33)	NA	NA	.10	[7]38	NA	1.00	[7]45	84-88	33-37	Scrap iron and scrap tin cans.	1.3	13	28	4	8	1.00	NA	NA	7.50	NA	.10	NA	NA	6.80	NA

[19] 4 parallel rows of launders, 12 cells per row.
[20] Cone precipitators. Dimensions, 14 ft diam by 24 ft deep.
[21] April through September.
[22] October through March.
[23] Cone precipitators. Dimensions, 18 ft diam by 34 ft deep.
[24] Total for other 3 dumps.
[25] 6 units of 4 to 6 pairs of cells.
[26] Block-caved stopes, pillars, and capping.
[27] Summer.
[28] Initial.
[29] Minimum.
[30] Each needle valve, 2 ft apart along the pipes, feeds at rate of 180 cu cm/min.
[31] Fresh water.
[32] Recirculated solutions.
[33] From 6.5 to 7.0 pounds of acid per pound of copper precipitated.

APPENDIX E

MINERALOGICAL EXAMINATIONS

This Appendix comprises the results of three mineralogical examinations and is accordingly presented in three parts.

Part 1	deals with five of the samples of secondary ores detailed in Table 11 of this Report.
Part 2	deals with the remaining three samples detailed in Table 11.
Part 3	deals with further work on one of the samples included in Part 2.

PART 1

MINERALOGICAL EXAMINATION OF FIVE SECONDARY COPPER ORES

1. INTRODUCTION

Five of the samples of secondary copper ores detailed in Table 11 of this Report, labelled A50 to A54 inclusive, were initially submitted for mineralogical examination.

Work required was the following:

(1) Identification of the copper mineral.

(2) Semi-quantitative analysis of the proportions of minerals.

(3) Comments on the size range of the minerals.

2. PROCEDURE

One quarter of each sample as received was riffled out. A polished thin section was prepared from this material, the remainder being retained for reference. The remaining three-quarters was crushed to wholly pass 0.42 mm (36 mesh). From this crushed material, one quarter was riffled out and ground in a Sieb mill, in order to provide a sample for chemical analysis and for X-ray diffractometry.

The remaining -0.42 mm (-36 mesh) material was wet sieved at 0.045 mm (350 mesh) and the -0.42+0.045 mm and -0.045 mm fractions were obtained for each sample.

The -0.42+0.045 mm fractions were separated in diluted tetrabromoethane (sp.gr. 2.5) and full-strength tetrabromoethane (sp.gr. 2.96) in order to separate chrysocolla (sp.gr. ~ 2.2) from the other copper minerals thought to be present [malachite (sp.gr. 4.0) and chalcocite (sp.gr. 5.8)] — and to give information of copper liberation (essentially liberated copper minerals other than chrysocolla report into the >2.96 sp.gr. product). The separates so obtained were riffled in half in order to provide portions for mineralogical examination and chemical analysis.

The following products from each sample were analysed for 'oxide' and acid-soluble copper.

(a) a head sample

(b) the -0.44 + 0.045 mm material with sp.gr. < 2.5

(c) the -0.42 + 0.045 mm material with sp.gr. > 2.5 and <2.96

(d) the -0.42 + 0.045 mm material with sp.gr. > 2.96

(e) the -0.45 mm material.

The weight and assay data so obtained were computer processed in order to determine the distribution of the 'oxide' and total or acid-soluble copper. The analysed head sample provided a check on the calculated head assay obtained by combining the weight/assay/distribution data of the fractions.

Polished thin sections of the coarser fragments (P.T.S. 27877 and 27881, A50 and A54) were prepared and examined in transmitted and reflected light; the heavy liquid separation products were mounted in oil and examined in transmitted light. From the microscopic examination, and using information obtained from the X-ray diffractometry examination, the minerals present were identified and their approximate proportions and size ranges determined.

3. ASSAY DATA DISCUSSION

In the following discussion of analyses, three types of analysis were obtained:

(a) 'Oxide' copper. This was determined by a 5% sulphuric acid digestion in the presence of ascorbic acid.

(b) Total copper. This was carried out on head samples only and an HF digestion was used.

(c) Acid-soluble copper. This was carried out by Amdel Scheme F1 and included a hot perchloric acid digestion.

The acid-soluble copper values are always rather lower than the total copper. For Samples A50, A51 and A52 it is also below the values for oxide copper. It is clear that, in these samples, oxide copper and acid-soluble copper should be effectively the same. The discrepancies in values are due mainly to inaccuracies in the determination of acid-soluble copper.

4. DESCRIPTION OF SAMPLES

Sample A50: P.T.S. 27877 (Paratoo)

An estimate of the constituents gave the following:

	Vol. %
Malachite	8-10
Dolomite	3- 6
Quartz	30-40
Sericite/muscovite	10-30
Hematite/goethite	3- 6
Kaolin	20-30
Feldspar	5-10

The rock is fine-grained, sedimentary rock, composed of quartz, feldspar, hematite and muscovite in a fine, largely irresolvable, micaceous, and clayey matrix. The grains of quartz, feldspar and hematite are up to 0.05 mm across.

The copper mineral in this rock is malachite. This was determined from examination of the polished thin section and from an examination of the heavy liquid separates in temporary oil mounts.

The heavy liquid separation/assay data, shown in the tabulations for sample A50 reveal that all the copper in the rock is 'oxide' and not 'sulphide' copper, consistent with the evidence that the copper mineral is malachite.

Furthermore, it can be seen that the malachite is relatively coarse and easily liberated, 50% of the copper in the -0.42+0.045 mm fractions reporting in the >2.96 sp.gr. product. This is consistent with what can be seen in the polished thin section, for the malachite occurs in veins and patches with dolomite, and these veins and patches are up to several millimetres across. The malachite also occurs as finer material replacing the matrix of the rock.

Comparison with other samples submitted suggests that the malachite may be associated with small amounts of chrysocolla and paratacamite (see Sample A52: Kanyaka), but neither of these minerals could be positively identified in this Sample due to fineness of the material and the closely intergrown nature of the minerals. However, the low proportion of the copper reporting in the <2.5 sp.gr. product suggests that chrysocolla is a minor component of the sample.

Sample A51: P.T.S. 27878 (Yappala)

An estimate of the constituents gave the following:

	Vol. %
Malachite	1- 3
Quartz	30-40
Feldspar	10-20
Chlorite	15-25
Muscovite/sericite	10-20
Opaques (goethite/hematite)	3- 6

This is a sedimentary rock similar to Sample A50 (Paratoo) just described, but finer-grained, visible grains of quartz being less than 0.02 mm across and most of the rock, which is semi-opaque matrix, being irresolvable when examined with the transmitted-light microscope.

The proportions of the minerals were determined largely by X-ray diffraction analysis.

From examination of the polished thin section and the heavy liquid separates in temporary oil mounts, it appears that malachite is the major copper mineral present. As in the previous sample, it occurs in veins, but much of it is in the matrix of the rock and is very fine-grained. Such fine-grained material is difficult to identify positively and some chrysocolla and paratacamite may also be present [cf. Samples A52 (Kanyaka) and A55 (Mt Gunson)]. However, the proportion of the 'oxide' copper reporting into the <2.5 sp.gr. products indicates that chrysocolla is a minor component only.

The heavy liquid separation data shown in the attached tables†, are consistent with the mineralogical data in that they show all the copper is in 'oxide' (non-sulphide) form. Furthermore, the proportion of the copper reporting in the >2.96 sp.gr. product of the -0.42+0.045 mm fractions is only 19%, 78% reporting in the 2.5 to 2.96 sp.gr. product. These features are consistent with the fine-grained nature of the rock and the presence of the copper minerals dispersed through the matrix. These results contrast sharply with those obtained on Sample A50 where 60% of the copper in the -0.42+0.045 mm fraction is in the >2.96 sp.gr. product and about 38% is in the 2.5 to 2.96 sp.gr. product.

In both samples, the low proportion of copper in the <2.5 sp.gr. products suggests that chrysocolla is virtually absent.

Sample A52: P.T.S. 27879 (Kanyaka)

An estimate of the constituents gave the following:

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† Computer sheets bound separately.

	Vol. %
?Conichalcite	Trace
Azurite	Trace
Malachite	1- 3
Quartz	35-45
Gypsum	10-30
Kaolin	35-45
Muscovite/sericite	Trace-2
Paratacamite	1- 3

This sample differs from Sample A50 (Paratoo) and A51 (Yappala) in that it is coarser, of sandstone grade.

The grains of quartz, which are well-rounded to subangular in shape and between 0.1 and 0.5 mm in diameter, make up 35 to 45% of the rock.

The remainder of the rock consists of matrix (muscovite/sericite), copper minerals and coarse particles of gypsum.

The major copper minerals present are malachite and paratacamite, which occur in approximately equal amounts. The formula for paratacamite is $Cu_2(OH)_3Cl$, and its identity was checked with an X-ray diffraction photograph. There are also trace amounts of azurite and possibly minor amounts of conichalcite [$CaCuAsO_4(OH)$] a mineral positively identified in Sample A53 (Dome Rock).

The size/assay/distribution data in the Sample A52 tabulation† are consistent with the mineralogy in that the copper is present in the 'oxide' form and not in sulphides. Most of the copper is present in the +0.045 mm material but is equally divided between the >2.96 sp.gr. product and the 2.50 to 2.96 sp.gr. product. This distribution is consistent with the mineralogy; paratacamite, in particular, occurs as a matrix surrounding quartz grains and much of the crushed material consists of composite grains of copper minerals and quartz. These composites have a lower density than the liberated copper minerals and hence report in the 2.5 to 2.96 sp.gr. product of the -0.42+0.045 mm material.

Gypsum makes up 10 to 30% of the sample and occurs as particles of quite coarsely crystalline material up to 5 mm across. Together with the presence of paratacamite, the abundance of gypsum suggests that this copper deposit is close to a salt lake or playa.

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† Computer sheets.

Sample A53: P.T.S.27880 (Dome Rock)

An estimate of the constituents gave the following:

	Vol. %
Quartz	35-45
Feldspar (?albite)	2- 4
Kaolin Montmorillonite }	20-30
Dolomite	10-20
?Calcite	3- 6
Muscovite/sericite	Trace-2
Chalcocite	1- 2
Malachite	2- 4
Conichalcite	Trace
Covellite	Trace
Goethite/hematite	2- 4
Pyrite	Trace

The chips of rock representative of this sample are quite variable but it would appear that the host rock or dominant rock-type of this copper deposit is a fine-grained sedimentary rock, with a grain size of about 0.05 mm. However, there are other chips rich in carbonate which are coarser grained, with crystals up to 0.3 mm and chips rich in clay or (rarely) opaques.

Copper materials recognised include chalcocite, covellite, malachite and conichalcite, with the chalcocite and malachite predominating.

The chalcocite has an earthy, porous habit and has partly replaced pyrite. Patches of chalcocite up to 2 mm across were noted and remnants of pyrite within these patches are up to 0.8 mm across. Covellite is intergrown with the chalcocite in minor or trace amounts. Locally there are minor amounts of another opaque copper phase present (?tenorite), but this has not been positively identified.

The assay data indicate that copper in non-sulphide minerals makes up only 40% of the total copper content, the remaining 60% being in sulphide minerals. As already indicated, chalcocite is the dominant copper sulphide mineral present.

Of the oxide copper minerals, malachite predominates and conichalcite [$CaCuAsO_4(OH)$] a green-coloured secondary copper mineral, identified with an X-ray diffraction photograph, occurs in minor amounts. Some chrysocolla may be present but it has not been positively identified.

The distribution data indicates that the chalcocite is quite well liberated, 91.6% of the sulphide (total oxide) copper occurring in the >2.96 sp.gr. product of the -0.04+0.045 mm fraction and this being about 55% of the total copper content of the sample. The high content of oxide copper in the <2.5 sp.gr. product suggests that chrysocolla is present, but examination of this fraction in a temporary mount revealed no chrysocolla-like grains. On the other hand, the oxide copper distribution data can be interpreted as showing that the oxide copper minerals are poorly liberated and this is partly true, for many chips rich in clay and/or quartz and secondary copper minerals are very fine-grained.

Sample A54: P.T.S. 27881 (Dome Rock)

An estimate of the constituents gave the following:

	Vol. %
Chalcocite	25-35
Pyrite	25-35
Quartz (variety chalcedony)	25-35
Unidentified secondary copper mineral	2- 5

This is a sample rich in chalcocite. This mineral has an earthy habit and is intergrown with, or has replaced, pyrite in patches up to several millimetres across.

Another major mineral present is chalcedony, which occurs in patches and veins, some of which are several millimetres across.

The bulk of the remaining material is pyrite.

The distribution data show that the chalcocite is well liberated, or at least that most grains of the crushed material are chalcocite-rich and report in the >2.96 sp.gr. product; this is confirmed by examination of the >2.96 sp.gr. product.

According to the assay data, about 10% of the copper is in the form of secondary non-sulphide minerals, the remaining 90% being sulphide copper (sulphide copper = acid-soluble — oxide copper), but some, at least, of the non-sulphide copper will have been derived from slight leaching of the chalcocite. From examination of the polished thin section and the heavy liquid separates in temporary oil mounts, it is not clear what this secondary mineral is. It is possibly chrysocolla which looks similar to chalcedony in certain instances and may occur as a thin coating about patches of chalcocite.

Investigation and Report by: R. Cooper

X-ray Diffraction Photograph Interpretation by: R. Cooper & Dr R.N. Brown

Chemical Analyses by: H. Fishman, A. Francis & J. Powell

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PART 2

MINERALOGICAL EXAMINATION OF THREE SECONDARY COPPER ORES

1. INTRODUCTION

Three samples of secondary copper ore, labelled A55 to A57 inclusive, were submitted for mineralogical examination.

The work required was identical to that for the five samples (A50-A54) discussed in Part 1.

2. PROCEDURE

The procedure adopted was similar to that taken with the previous five samples.

3. DESCRIPTION OF SAMPLES

Sample A55: P.T.S. 28009(Mt Gunson)

An estimate of the constituents gave the following:

	Vol. %
Quartz	85-95
Muscovite/sericite	2- 4
Gypsum	Trace
Paratacamite	1- 2
Malachite	1- 2
Brochantite	Trace
Chalcocite	Trace-0.5
Chrysocolla	2- 4

The rock is a sandstone similar to Sample A52 (Kanyaka). The grains of quartz are well-rounded to subangular, sometimes slightly elongate as a result of compaction and ?pressure solution of grain boundaries, and they range in size from 0.1 mm to 1.5 mm with most being about 0.5 mm in diameter. Quartz grains constitute 60 to 80% of each chip in the polished section.

The other minerals occur in the interstices between the quartz grains or, less commonly, in veins and patches which are rarely as much as 0.5 mm across.

The assay data indicate that about 10% of the copper is in the form of sulphides. Examination of the polished thin section revealed two small patches of chalcocite, the largest 0.5 mm across. From the distribution data for sulphide (total-oxide) copper it can be seen that 39% of the sulphide is in the >2.96 sp.gr. product of the -0.42+0.045 mm material and 49% in the 2.5 to 2.96 sp.gr. product. These data are consistent with the chalcocite being finely divided and not well liberated.

The oxide copper distribution data on the -0.42+0.045 mm heavy liquid products indicate either that the copper minerals are very poorly liberated (very fine and dispersed) or that there is a copper mineral with very low specific gravity such as chrysocolla present. The latter interpretation appears correct, for in the <2.5 sp.gr. product there are numerous coarse grains of a pale-blue-coloured copper mineral.

The other major secondary copper minerals are paratacamite and malachite which are concentrated in the 2.50 to 2.96 and >2.96 sp.gr. products. There are also minor amounts of brochantite present.

To summarise, the copper minerals in this sample in order of importance are: chrysocolla, paratacamite, malachite, chalcocite and brochantite.

Sample A56: P.T.S. 28010 (Paratoo)

An estimate of the constituents gave the following:

		Vol. %
Quartz		20-30
Feldspar (potash)	ratio uncertain	5-10
Kaolin		30-40
Mica		2- 4
Feldspar (albite)		5-10
Malachite		10-15
Chrysocolla		2- 4

The host rock of this copper sample is a fine-grained siltstone containing grains of quartz up to 0.05 mm in a largely irresolvable matrix of clay (Kaolin) and feldspar.

The copper minerals occur in a network of veins and patches running through the host rock and also have replaced the matrix along a few bands or laminae.

The assay and distribution data indicate that this sample contains only 'oxide' copper minerals.

The most important of these is malachite, but there is also certainly some chrysocolla, for the <2.5 sp.gr. product of the −0.42+0.045 mm material assays 2.70% oxide copper. Examination of this product in transmitted light on a temporary mount reveals clay-rich grains and grains which appear to be a mixture of clay and a green secondary copper mineral. There are also composites of ?clay and malachite.

Examination of the distribution data shows that about 25% (overall) of the total (≃ acid soluble copper) content is in either the 2.50 to 2.96 sp.gr. product or the <2.50 product of the −0.42+0.045 mm material. This is either locked malachite or chrysocolla. However, the malachite in the >2.96 sp.gr. product is mostly coarse and liberated and should be amenable to leaching.

Sample A57: P.T.S. 28011 (Elsie Adair)

An estimate of the constituents gave the following:

		Vol. %
Quartz		30-40
Feldspar (potash and plagioclase) Kaolin Mica	irresolvable matrix	50-60
Olivenite		1- 2
Chrysocolla		Trace-2
Malachite		Trace

This sample is very similar to Sample A56 (Paratoo) described previously, being a fine-grained siltstone weakly mineralised with copper.

The copper mineral(s) occur in veins and in the matrix of the rock. In the latter situation they are very fine and well dispersed.

The assay and distribution data indicate that the copper minerals are not well liberated. Considering the −0.42+0.045 mm material, in terms of acid-soluble copper, approximately 60% of the copper in this material is in the <2.96 sp.gr. products. This distribution is to be expected considering the overall fine grain size of the material.

The 'oxide' copper assays are equal to the acid-soluble copper assays, indicating that no 'sulphide' copper minerals are present.

From an examination of the polished thin section and the heavy liquid products in temporary oil mounts, the 'oxide' copper minerals are known to rank in importance as follows:

(a) Chrysocolla
(b) Malachite
(c) Olivenite $(CuAsO_4)OH$

The latter mineral was identified by X-ray diffraction techniques.

4. DISCUSSION

With reference to the tables[†] of 'oxide' copper distribution for Samples A55 to A57, it should be noted that, except for Sample A55 (Mt Gunson), the distribution is the same as for 'acid-soluble' copper distribution. Some difficulty was initially encountered in assaying for 'oxide' copper, but repeat analyses for 'oxide' copper on the >2.96 sp.gr. products confirmed their virtual identity with the 'acid-soluble' copper values and this has been extrapolated to the other specific gravity products. This procedure is considered to be valid, as any sulphide copper present in the samples would be expected to concentrate in the >2.96 sp.gr. product and its absence in this product indicates its absence in the other products.

Investigation and Report:	R. Cooper
X-ray Diffraction Photograph Interpretation by:	R. Cooper & Dr R.N. Brown
Chemical Analyses by:	H. Fishman, A. Francis & J.Powell
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[†] Computer sheets bound separately.

PART 3

MINERALOGICAL EXAMINATION OF A SAMPLE OF OXIDE COPPER ORE FROM MT GUNSON WITH SPECIAL ATTENTION TO THE NATURE OF THE CHRYSOCOLLA

1. INTRODUCTION

Earlier mineralogical work in Part 2 of this Appendix on a set of oxide copper samples had shown that one, Sample A55 (Mt Gunson), contained approximately 40% of its copper in the form of chrysocolla.

Subsequent leaching tests on this sample, however, produced apparently anomalous results, the ore leaching far better than might be anticipated from the quantity of chrysocolla it contained.

It was requested that the copper mineralogy of the Mt Gunson oxide copper ore be confirmed and, if possible, an explanation be given as to how the chrysocolla in this sample differed from 'normal' chrysocolla.

The sample submitted for this work was from the same Mt Gunson ore batch as A55 but is not representative of the whole of the batch.

2. SUMMARY

Comparison of the approximate mineralogy of this sample of Mt Gunson oxide copper ore, as determined by optical and X-ray diffraction techniques, with that obtained for the earlier sample, labelled A55, reveals the following:

	This Sample Wt %	A55 Wt %
Chrysocolla	6- 8	2
Atacamite/paratacamite	1- 2	1- 2
Malachite	4- 6	1- 2
Brochantite	nd*	Trace
Chalcocite	Trace	Trace-0.5
Quartz	88-90	85-90
Muscovite/sericite	Trace	2- 4
Gypsum	-	Trace
Kaolin	Trace	-

* nd = Not detected.

The samples are mineralogically very similar, the major copper minerals being chrysocolla, malachite and atacamite/paratacamite.

Comparison of the size-assay analysis data obtained from this sample with similar data obtained on Sample A55 reveals the following:

Size Fraction (mm) Heavy Liquid Product	Weight %	Copper Assay %	Copper Distribution %
This Sample			
+0.045 <2.5 sp.gr.	4.4	15.4	9.3
+0.045 >2.5 sp.gr.	78.4	5.8	62.4
-0.045	17.2	12.0	28.3
	100.0	(7.3)*	100.0
Sample A55			
+0.045 <2.5 sp.gr.	16.4	3.1	26.7
+0.045 >2.5 sp.gr.	72.1	1.35	51.5
-0.045	11.8	3.6	21.8
	100.0	(1.9)*	100.0

* Calculated head assays.

1. The copper content of this sample (7.3% Cu), is considerably higher than that of the earlier sample (1.9% Cu).
2. Overall the +0.045 mm <2.5 sp.gr. product of this sample contains a smaller proportion of both the weight and copper than the equivalent product of Sample A55.
3. In this sample a maximum of 25 to 35% of the copper is present as chrysocolla, whereas in Sample A55 at least 40% of the copper was present as chrysocolla.

X-ray diffraction analysis of the chrysocolla in this sample has shown that it is not crystalline.

Electron-probe microanalysis of the chrysocolla in this sample has shown that it is relatively pure, most grains containing 30% or more copper. Principal impurities are kaolin and quartz.

3. COMMENTS ON CHRYSOCOLLA

Chrysocolla can be regarded as a gel of copper oxide, silica and water which approximates in composition to the formula $CuSiO_3.2H_2O$. This formula implies that pure chrysocolla has a copper content of ∿36% and a silicon content of ∿16%. However, because chrysocolla commonly contains impurities such as silica and alumina (kaolin), the copper content is usually lower.

In this sample of Mt Gunson oxide copper ore the chrysocolla had the following features which may explain the unusually good leaching response:

1. The normal weak X-ray diffraction pattern could not be obtained, suggesting that the chrysocolla is not crystalline. Lack of crystallinity is usually associated with increased chemical reactivity and may account for the unexpectedly good leaching response.

2. The copper contents of chrysocolla grains in this sample, as determined by electron-probe microanalysis, ranged from 23.5% to 42.7%. These values are relatively high and close to the theoretical value for $CuSiO_3.2H_2O$ (36% Cu). The purity of the chrysocolla in this sample of Mt Gunson oxide copper ore may have been beneficial to the leaching process.

4. PROCEDURES

The sample as received was crushed to 0.42 mm (36 mesh BSS) then sized at 0.045 mm (350 mesh BSS). The -0.42+0.045 mm fraction was separated in diluted tetrabromoethane (sp.gr. 2.491), and portions of the 'sinks' and 'floats' products obtained, as well as a portion of the -0.045 mm material, were assayed for copper. Chrysocolla, (sp.gr. $\sim$2.2) present in liberated grains will report in <2.49 sp.gr. product.

X-ray diffraction analysis was carried out on several chips of the uncrushed sample and on a portion of the -0.42+0.045 mm <2.491 sp.gr. product.

Portions of the size/heavy liquid products were examined optically and the heavy liquid products of the -0.42+0.045 mm fraction were briquetted (PS21743). This briquette was examined with the electron-probe microanalyser and individual chrysocolla grains analysed for copper, silicon and aluminium.

5. RESULTS

5.1 Size-assay Analysis

The following size-assay analysis was obtained.

TABLE E-1: DISTRIBUTION OF COPPER IN THIS SAMPLE OF MT GUNSON OXIDE COPPER ORE

Size Fraction (mm)/ Heavy Liquid Product	Weight %	Copper Assay %	Copper Distribution %
-0.42+0.045 <2.49 sp.gr.	4.4	15.4	9.3
-0.42+0.045 >2.49 sp.gr.	78.4	5.8	62.4
-0.045	17.2	12.0	28.3
	100.0	(7.3)*	100.0

* Calculated head assay.

The following conclusions can be derived from Table E-1:

1. The calculated head assay of 7.3% copper is considerably higher than that of the previous sample, 1.9% copper (see comparative data, Section 2).

2. The -0.42+0.045 mm <2.49 sp.gr. product, which contains essentially liberated chrysocolla (sp.gr. 2.2 to 2.3) and other <2.49 sp.gr. minerals, contains 4.4% of the sample weight, assays 15.4% copper and contains 9.3% of the copper in the sample.

3. The -0.42+0.045 mm >2.49 sp.gr. product, which contains liberated and locked copper minerals with specific gravities >2.49 and small amounts of chrysocolla in composites with gangue minerals of specific gravity >2.49 (e.g., quartz sp.gr. 2.65), contains 78.4% of the sample weight, assays 5.8% copper, and contains 62.4% of the copper in the sample.

4. The -0.045 mm material constitutes 17.2% of the sample by weight, assays 12.0% copper, and contains 28.3% of the copper in the sample.

5. The maximum amount of copper that could be present as chrysocolla in this sample is calculated to be about 25 to 35%. This contrasts with Sample A55 where about 40% of the copper was present as chrysocolla.

The calculation is made by:

(a) Considering all the copper in the -0.42+0.045 mm <2.49 sp.gr. product to be in the form of chrysocolla.

(b) Making allowance for chrysocolla in the -0.045 mm slimes on the basis that it is present in similar proportions to the chrysocolla in the +0.045 mm material.

(c) Making allowance for chrysocolla being present in small amounts in the -0.42+0.045 mm >2.49 sp.gr. product locked with >2.49 sp.gr. minerals, especially quartz (sp.gr. 2.65).

5.2 Results of Optical Examination and X-ray Diffraction Analysis

Mineralogically this sample of Mt Gunson oxide copper has the following approximate composition:

	Wt %
Chrysocolla	6- 8
Atacamite/paratacamite	1- 2
Malachite	4- 6
Quartz	85–90
Muscovite/sericite	Trace
Kaolin	Trace

In the initial Sample A55 of Mt Gunson oxide copper ore traces of gypsum, chalcocite, and brochantite were also detected.

Two interesting mineralogical facts emerged as a result of the X-ray diffraction work:

1. No X-ray diffraction patterns for chrysocolla could be obtained from either the bulk sample or from individual grains in the -0.45+0.045 mm product. This suggests that the chrysocolla in this sample is not crystalline.

2. Data in the ASTM index for atacamite and paratacamite are inadequate to distinguish the two minerals. The phase(s) is so finely crystalline in this sample that it is not possible to make a distinction optically. (Atacamite and paratacamite are polymorphs of basic copper chloride $Cu_2(OH)_3Cl$.)

5.3 Electron-probe Microanalyser Results

Electron-probe microanalysis of chrysocolla grains in the -0.42+0.045 mm fraction for copper, silicon and aluminium produced the following results:

	% Cu	% Si	% Al
1	32.9	16.4	-
2	36.8	20.6	-
3	26.0	21.9	-
4	42.7	17.1	(3.6) average value
5	23.5	20.4	-
6	38.3	16.4	-
Range	23.5-42.7	16.4-21.9	-

It was discovered that the higher the copper content of the chrysocolla grains, the softer they were, and the more affected they were by the beam of the electron-probe microanalyser. This feature was used to establish the range of copper values occurring in chrysocolla grains, and the grains chosen for analysis were not a random or representative selection but were selected to establish the range.

Aluminium was erratically distributed through the chrysocolla grains and the figure of 3.6% is an average from a traverse across several grains.

6. CONCLUSIONS

1. The major copper minerals in the Mt Gunson oxide copper ore appear to be chrysocolla, malachite and atacamite/paratacamite.
2. The chrysocolla is not crystalline and, presumably, is particularly reactive chemically.
3. The chrysocolla grains in the -0.42+0.045 mm <2.49 sp.gr. product had relatively high copper contents, as determined by electron-probe microanalysis, suggesting that much of the chrysocolla in the sample is relatively pure.

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Electron-probe microanalysis by:	P.K. Schultz
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                 SAMPLE A50 WITH SLIMES


    SUMMARY OF INPUT DATA ....


    SEPARATION TYPE  1        NO. OF SIZE FRACTIONS  2        NO. OF PRODUCTS  3

    SIZE FRACTIONS ..
           .045    .000

    SIZE ANALYSIS ..
          34.59    7.03

    SPLITTING VALUES ..
           2.50    2.96

    WEIGHTS IN PRODUCTS ..
           2.05   29.10    5.49
```

```
SAMPLE  A 50  ACID  SOLUBLE  COPPER  DISTRIBUTION

  SIZE FRACTION    SPEC GRAV   ---  IN SIZE FRACTIONS ---       IN HEAD SAMPLE                CUMULATIVE DATA
  (MILLIMETRES)    FRACTION    WT  %   ---  TOTAL CU  --        WT  %   TOTAL CU          WEIGHT    ASSAY     DISTN
                                       ASSAY %    DISTN %               DISTN %              %         %          %


         +  .045   < 2.50        5.6     2.90        2.3         4.65     1.92            4.65      2.90       1.92
                   < 2.96       79.4     3.40       39.1        66.01    32.02           66.01      3.40      32.02
                   > 2.96       15.0    27.00       58.6        12.45    47.98           12.45     27.00      47.98
                   TOTAL       100.0 (   6.91)     100.0        83.11    81.92           83.11      6.91      81.92


  TOTALS FOR ABOVE < 2.50        5.6   ( 2.90)       2.3
  SIZE FRACTIONS   < 2.96       79.4   ( 3.40)      39.1
                   > 2.96       15.0   (27.00)      58.6

  OVERALL TOTALS..   SANDS      83.1   ( 6.91)      81.9
                     SLIMES     16.9     7.50       18.1
                     TOTAL     100.0   ( 7.01)     100.0                        CALCULATED HEAD ASSAY=  7.01%
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```
SAMPLE  A 50  OXIDE  COPPER  DISTRIBUTION

  SIZE FRACTION    SPEC GRAV  --- IN SIZE FRACTIONS ---        IN HEAD SAMPLE               CUMULATIVE DATA
  (MILLIMETRES)    FRACTION   WT  %   ---  OXIDE CU  --        WT  %   OXIDE CU         WEIGHT     ASSAY     DISTN
                                      ASSAY %    DISTN %                DISTN %            %          %          %

        +  .045    < 2.50       5.6     3.05        2.3        4.65      1.91            4.65       3.05       1.91
                   < 2.96      79.4     3.45       37.0       66.01     30.68           66.01       3.45      30.68
                   > 2.96      15.0    30.00       60.7       12.45     50.34           12.45      30.00      50.34
                   TOTAL      100.0 (   7.41)     100.0       83.11     82.93           83.11       7.41      82.93

  TOTALS FOR ABOVE < 2.50       5.6   ( 3.05)       2.3
  SIZE FRACTIONS   < 2.96      79.4   ( 3.45)      37.0
                   > 2.96      15.0   (30.00)      60.7

  OVERALL TOTALS..   SANDS     83.1   ( 7.41)      82.9
                     SLIMES    16.9     7.50       17.1
                     TOTAL    100.0   ( 7.42)     100.0                   CALCULATED HEAD ASSAY=  7.42%
```

```
                    SAMPLE A51 WITH SLIMES


     SUMMARY OF INPUT DATA ....


     SEPARATION TYPE  1          NO. OF SIZE FRACTIONS  2          NO. OF PRODUCTS  3

     SIZE FRACTIONS ..
              .045     .000

     SIZE ANALYSIS ..
             19.98     9.21

     SPLITTING VALUES ..
              2.50     2.96

     WEIGHTS IN PRODUCTS ..
               .85    18.76      .37
```

```
SAMPLE  A 51  ACID  SOLUBLE  COPPER  DISTRIBUTION

   SIZE FRACTION    SPEC GRAV   ---  IN SIZE FRACTIONS ---       IN HEAD SAMPLE                 CUMULATIVE DATA
   (MILLIMETRES)    FRACTION    WT  %   ---  TOTAL CU  --        WT  %   TOTAL CU            WEIGHT     ASSAY      DISTN
                                        ASSAY %   DISTN %                 DISTN %               %          %          %


          + .045    < 2.50        4.3     1.30       2.8          2.91     1.93              2.91      1.30       1.93
                    < 2.96       93.9     1.60      77.2         64.27    52.37             64.27      1.60      52.37
                    > 2.96        1.9    21.00      20.0          1.27    13.56              1.27     21.00      13.56
                    TOTAL       100.0 (   1.95)    100.0         68.45    67.86             68.45      1.95      67.86

   TOTALS FOR ABOVE < 2.50        4.3   ( 1.30)      2.8
   SIZE FRACTIONS   < 2.96       93.9   ( 1.60)     77.2
                    > 2.96        1.9   (21.00)     20.0

   OVERALL TOTALS..    SANDS     68.4   ( 1.95)     67.9
                       SLIMES    31.6     2.00      32.1
                       TOTAL    100.0   ( 1.96)    100.0                     CALCULATED HEAD ASSAY=  1.96%
```

```
SAMPLE  A 51  OXIDE  COPPER  DISTRIBUTION

  SIZE FRACTION    SPEC GRAV   ---  IN SIZE FRACTIONS ---     IN HEAD SAMPLE               CUMULATIVE DATA
  (MILLIMETRES)    FRACTION    WT  %   ---  OXIDE CU  --      WT  %   OXIDE CU           WEIGHT    ASSAY     DISTN
                                     ASSAY %    DISTN %                DISTN %              %         %         %


           + .045  < 2.50       4.3     1.30       2.7        2.91     1.84              2.91      1.30      1.84
                   < 2.96      93.9     1.73      78.0       64.27    54.05             64.27      1.73     54.05
                   > 2.96       1.9    21.80      19.4        1.27    13.43              1.27     21.80     13.43
                   TOTAL      100.0 (   2.08)    100.0       68.45    69.32             68.45      2.08     69.32


  TOTALS FOR ABOVE < 2.50       4.3   ( 1.30)      2.7
  SIZE FRACTIONS   < 2.96      93.9   ( 1.73)     78.0
                   > 2.96       1.9   (21.80)     19.4

  OVERALL TOTALS..   SANDS     68.4   ( 2.08)     69.3
                     SLIMES    31.6     2.00      30.7
                     TOTAL    100.0   ( 2.06)    100.0                        CALCULATED HEAD ASSAY=  2.06%
```

```
                SAMPLE A52 WITH SLIMES

SUMMARY OF INPUT DATA ....

SEPARATION TYPE  1        NO. OF SIZE FRACTIONS  2        NO. OF PRODUCTS  3

SIZE FRACTIONS ..
         .045    .000

SIZE ANALYSIS ..
        50.40    9.51

SPLITTING VALUES ..
         2.50    2.96

WEIGHTS IN PRODUCTS ..
         3.20   44.80    2.40
```

```
SAMPLE  A 52  ACID  SOLUBLE  COPPER  DISTRIBUTION

  SIZE FRACTION    SPEC GRAV    ---  IN SIZE FRACTIONS ---         IN HEAD SAMPLE               CUMULATIVE DATA
  (MILLIMETRES)    FRACTION     WT  %   ---  TOTAL CU  --          WT  %   TOTAL CU         WEIGHT    ASSAY      DISTN
                                         ASSAY %    DISTN %                DISTN %             %         %          %

          + .045   < 2.50        6.3     4.10        9.6            5.34     8.21            5.34      4.10       8.21
                   < 2.96       88.9     1.35       44.5           74.78    37.85           74.78      1.35      37.85
                   > 2.96        4.8    26.00       45.9            4.01    39.05            4.01     26.00      39.05
                   TOTAL       100.0 (   2.70)     100.0           84.13    85.12           84.13      2.70      85.12


  TOTALS FOR ABOVE < 2.50        6.3   ( 4.10)       9.6
  SIZE FRACTIONS   < 2.96       88.9   ( 1.35)      44.5
                   > 2.96        4.8   (26.00)      45.9

  OVERALL TOTALS..   SANDS      84.1   ( 2.70)      85.1
                     SLIMES     15.9     2.50       14.9
                     TOTAL     100.0   ( 2.67)     100.0                       CALCULATED HEAD ASSAY=  2.67%
```

```
    SAMPLE  A 52  OXIDE  COPPER  DISTRIBUTION

      SIZE FRACTION    SPEC GRAV    ---  IN SIZE FRACTIONS ---     IN HEAD SAMPLE                CUMULATIVE DATA
      (MILLIMETRES)    FRACTION     WT  %    ---  OXIDE CU --      WT  %    OXIDE CU          WEIGHT    ASSAY      DISTN
                                             ASSAY %   DISTN %             \DISTN %             %         %          %

             + .045    < 2.50         6.3     4.55     10.0        5.34     8.66              5.34      4.55       8.66
                       < 2.96        88.9     1.44     44.4       74.78    38.37             74.78      1.44      38.37
                       > 2.96         4.8    27.60     45.6        4.01    39.40              4.01     27.60      39.40
                       TOTAL        100.0 (   2.88)   100.0       84.13    86.43             84.13      2.88      86.43

      TOTALS FOR ABOVE < 2.50         6.3   ( 4.55)    10.0
      SIZE FRACTIONS   < 2.96        88.9   ( 1.44)    44.4
                       > 2.96         4.8   (27.60)    45.6

      OVERALL TOTALS..   SANDS       84.1   ( 2.88)    86.4
                         SLIMES      15.9     2.40     13.6
                         TOTAL      100.0   ( 2.81)   100.0                     CALCULATED HEAD ASSAY=  2.81%
```

```
                SAMPLE AS3 WITH SLIMES


SUMMARY OF INPUT DATA ....


SEPARATION TYPE  1        NO. OF SIZE FRACTIONS  2        NO. OF PRODUCTS  3

SIZE FRACTIONS ..
         .045    .000

SIZE ANALYSIS ..
        35.89    7.61

SPLITTING VALUES ..
         2.50    2.96

WEIGHTS IN PRODUCTS ..
         8.45   20.68    6.76
```

```
SAMPLE  A 53  ACID  SOLUBLE  COPPER  DISTRIBUTION

  SIZE FRACTION     SPEC GRAV    ---  IN SIZE FRACTIONS ---         IN HEAD SAMPLE                  CUMULATIVE DATA
  (MILLIMETRES)     FRACTION     WT  %   ---  TOTAL CU  --          WT  %   TOTAL CU             WEIGHT    ASSAY      DISTN
                                         ASSAY %    DISTN %                 DISTN %                %         %          %

          +  .045   < 2.50       23.5      2.70       13.5          19.43    12.21               19.43      2.70      12.21
                    < 2.96       57.6      1.20       14.7          47.54    13.28               47.54      1.20      13.28
                    > 2.96       18.8     18.00       71.9          15.54    65.13               15.54     18.00      65.13
                    TOTAL       100.0 (    4.72)     100.0          82.51    90.63               82.51      4.72      90.63


  TOTALS FOR ABOVE  < 2.50       23.5    ( 2.70)      13.5
  SIZE FRACTIONS    < 2.96       57.6    ( 1.20)      14.7
                    > 2.96       18.8    (18.00)      71.9

  OVERALL TOTALS..    SANDS      82.5    ( 4.72)      90.6
                      SLIMES     17.5      2.30        9.4
                      TOTAL     100.0    ( 4.29)     100.0                             CALCULATED HEAD ASSAY=  4.29%
```

```
SAMPLE  A 53  OXIDE  COPPER  DISTRIBUTION

 SIZE FRACTION    SPEC GRAV   --- IN SIZE FRACTIONS ---        IN HEAD SAMPLE               CUMULATIVE DATA
 (MILLIMETRES)    FRACTION    WT  %   ---  OXIDE CU --        WT  %    OXIDE CU          WEIGHT    ASSAY     DISTN
                                      ASSAY %    DISTN %                DISTN %            %         %         %

         + .045   < 2.50      23.5     2.55       31.6         19.43    27.33            19.43     2.55     27.33
                  < 2.96      57.6      .85       25.8         47.54    22.29            47.54      .85     22.29
                  > 2.96      18.8     4.30       42.6         15.54    36.87            15.54     4.30     36.87
                  TOTAL      100.0 (   1.90)     100.0         82.51    86.49            82.51     1.90     86.49

 TOTALS FOR ABOVE < 2.50      23.5   ( 2.55)      31.6
 SIZE FRACTIONS   < 2.96      57.6   (  .85)      25.8
                  > 2.96      18.8   ( 4.30)      42.6

 OVERALL TOTALS..   SANDS     82.5   ( 1.90)      86.5
                    SLIMES    17.5     1.40       13.5
                    TOTAL    100.0   ( 1.81)     100.0                          CALCULATED HEAD ASSAY=  1.81%
```

```
SAMPLE  A53  SULPHIDE  (TOTAL-OXIDE)  COPPER  DISTRIBUTION

 SIZE FRACTION    SPEC GRAV    ---  IN SIZE FRACTIONS ---       IN HEAD SAMPLE                  CUMULATIVE DATA
 (MILLIMETRES)    FRACTION     WT  %   ---  SULPH. CU --        WT  %   SULPH. CU           WEIGHT     ASSAY      DISTN
                                       ASSAY %    DISTN %               DISTN %                %          %          %

          + .045  < 2.50       23.5      .15        1.3         19.43     1.17              19.43       .15       1.17
                  < 2.96       57.6      .35        7.2         47.54     6.70              47.54       .35       6.70
                  > 2.96       18.8    13.70       91.6         15.54    85.78              15.54     13.70      85.78
                  TOTAL       100.0 (   2.82)     100.0         82.51    93.66              82.51      2.82      93.66


 TOTALS FOR ABOVE < 2.50       23.5   (  .15)       1.3
 SIZE FRACTIONS   < 2.96       57.6   (  .35)       7.2
                  > 2.96       18.8   (13.70)      91.6

 OVERALL TOTALS..   SANDS      82.5   ( 2.82)      93.7
                    SLIMES     17.5      .90        6.3
                    TOTAL     100.0   ( 2.48)     100.0                          CALCULATED HEAD ASSAY=  2.48%
```

```
                SAMPLE A54 WITH SLIMES


SUMMARY OF INPUT DATA ....


SEPARATION TYPE  1        NO. OF SIZE FRACTIONS  2        NO. OF PRODUCTS  3

SIZE FRACTIONS ..
         .045    .000

SIZE ANALYSIS ..
        33.83    3.88

SPLITTING VALUES ..
         2.50    2.96

WEIGHTS IN PRODUCTS ..
         2.24    3.30   28.29
```

```
SAMPLE  A 54  ACID  SOLUBLE  COPPER  DISTRIBUTION

  SIZE FRACTION    SPEC GRAV   ---  IN SIZE FRACTIONS ---       IN HEAD SAMPLE              CUMULATIVE DATA
  (MILLIMETRES)    FRACTION    WT  %   ---  TOTAL CU  --        WT  %   TOTAL CU        WEIGHT    ASSAY     DISTN
                                       ASSAY %    DISTN %               DISTN %            %        %         %

          + .045   < 2.50       6.6     5.50        1.3         5.94     1.19            5.94      5.50      1.19
                   < 2.96       9.8     8.50        3.0         8.75     2.72            8.75      8.50      2.72
                   > 2.96      83.6    32.00       95.7        75.02    87.64           75.02     32.00     87.64
                   TOTAL      100.0 (  27.95)     100.0        89.71    91.55           89.71     27.95     91.55


  TOTALS FOR ABOVE < 2.50       6.6   ( 5.50)       1.3
  SIZE FRACTIONS   < 2.96       9.8   ( 8.50)       3.0
                   > 2.96      83.6   (32.00)      95.7

  OVERALL TOTALS..   SANDS     89.7   (27.95)      91.5
                     SLIMES    10.3    22.50        8.5
                     TOTAL    100.0   (27.39)     100.0                CALCULATED HEAD ASSAY= 27.39%
```

```
SAMPLE  A 54  OXIDE  COPPER  DISTRIBUTION

 SIZE FRACTION    SPEC GRAV    ---  IN SIZE FRACTIONS ---        IN HEAD SAMPLE                    CUMULATIVE DATA
 (MILLIMETRES)    FRACTION     WT  %   ---   OXIDE CU --         WT  %    OXIDE CU              WEIGHT    ASSAY      DISTN
                                       ASSAY %    DISTN %                 DISTN %                  %         %          %

          + .045  < 2.50        6.6     1.99        4.9           5.94     4.29                 5.94     1.99       4.29
                  < 2.96        9.8     1.78        6.5           8.75     5.65                 8.75     1.78       5.65
                  > 2.96       83.6     2.85       88.6          75.02    77.56                75.02     2.85      77.56
                  TOTAL       100.0 (   2.69)     100.0          89.71    87.50                89.71     2.69      87.50

 TOTALS FOR ABOVE < 2.50        6.6   ( 1.99)       4.9
 SIZE FRACTIONS   < 2.96        9.8   ( 1.78)       6.5
                  > 2.96       83.6   ( 2.85)      88.6

 OVERALL TOTALS..   SANDS      89.7   ( 2.69)      87.5
                    SLIMES     10.3     3.35       12.5
                    TOTAL     100.0   ( 2.76)     100.0                       CALCULATED HEAD ASSAY=  2.76%
```

```
SAMPLE  A54  SULPHIDE  (TOTAL-OXIDE)  COPPER  DISTRIBUTION

  SIZE FRACTION    SPEC GRAV    ---  IN SIZE FRACTIONS ---        IN HEAD SAMPLE                CUMULATIVE DATA
  (MILLIMETRES)    FRACTION     WT  %    ---  SULPH. CU --        WT  %   SULPH. CU          WEIGHT    ASSAY     DISTN
                                          ASSAY %   DISTN %                DISTN %             %         %         %

            + .045  < 2.50         6.6      3.51         .9        5.94        .85              5.94      3.51       .85
                    < 2.96         9.8      6.72        2.6        8.75       2.38              8.75      6.72      2.38
                    > 2.96        83.6     29.15       96.5       75.02      88.64             75.02     29.15     88.64
                    TOTAL        100.0 (  25.26)      100.0       89.71      91.87             89.71     25.26     91.87


  TOTALS FOR ABOVE  < 2.50         6.6   ( 3.51)         .9
  SIZE FRACTIONS    < 2.96         9.8   ( 6.72)        2.6
                    > 2.96        83.6   (29.15)       96.5

  OVERALL TOTALS..    SANDS       89.7   (25.26)       91.9
                      SLIMES      10.3    19.50         8.1
                      TOTAL      100.0   (24.67)      100.0                       CALCULATED HEAD ASSAY= 24.67%
```

```
                    SAMPLE  A55  WITH  SLIMES

      SUMMARY OF INPUT DATA ....

      SEPARATION TYPE  1          NO. OF SIZE FRACTIONS  2          NO. OF PRODUCTS  3

      SIZE FRACTIONS ..
               .045     .000

      SIZE ANALYSIS ..
              31.86     4.14

      SPLITTING VALUES ..
               2.50     2.96

      WEIGHTS IN PRODUCTS ..
               5.11    21.96       .55
```

```
SAMPLE- A 55  ACID  SOLUBLE  COPPER  DISTRIBUTION

  SIZE FRACTION     SPEC GRAV    ---  IN SIZE FRACTIONS ---         IN HEAD SAMPLE                CUMULATIVE DATA
  (MILLIMETRES)     FRACTION     WT   %  ---  TOTAL  CU --          WT  %  TOTAL  CU          WEIGHT    ASSAY      DISTN
                                         ASSAY %   DISTN %                DISTN %                %         %          %

          +  .045   < 2.50        18.5    3.10      34.2         16.37    26.73             16.37      3.10     26.73
                    < 2.96        79.5     .60      28.4         70.36    22.23             70.36       .60     22.23
                    > 2.96         2.0   31.50      37.4          1.76    29.23              1.76     31.50     29.23
                    TOTAL        100.0 (  1.68)    100.0         88.50    78.20             88.50      1.68     78.20

  TOTALS FOR ABOVE  < 2.50        18.5  ( 3.10)     34.2
  SIZE FRACTIONS    < 2.96        79.5  (  .60)     28.4
                    > 2.96         2.0  (31.50)     37.4

  OVERALL TOTALS..    SANDS       88.5  ( 1.68)     78.2
                      SLIMES      11.5    3.60      21.8
                      TOTAL      100.0  ( 1.90)    100.0                  CALCULATED HEAD ASSAY=  1.90%
```

```
SAMPLE  A 55  OXIDE  COPPER  DISTRIBUTION

 SIZE FRACTION    SPEC GRAV   ---  IN SIZE FRACTIONS ---        IN HEAD SAMPLE              CUMULATIVE DATA
 (MILLIMETRES)    FRACTION    WT  %   ---  OXIDE CU  --         WT  %   OXIDE CU          WEIGHT    ASSAY     DISTN
                                      ASSAY %    DISTN %                 DISTN %             %         %         %

       +  .045    < 2.50       18.5     3.00      36.6          16.37    28.16            16.37      3.00     28.16
                  < 2.96       79.5      .50      26.2          70.36    20.17            70.36       .50     20.17
                  > 2.96        2.0    28.30      37.2           1.76    28.59             1.76     28.30     28.59
                  TOTAL       100.0 (   1.52)    100.0          88.50    76.92            88.50      1.52     76.92

 TOTALS FOR ABOVE < 2.50       18.5   ( 3.00)     36.6
 SIZE FRACTIONS   < 2.96       79.5   (  .50)     26.2
                  > 2.96        2.0   (28.30)     37.2

 OVERALL TOTALS..  SANDS       88.5   ( 1.52)     76.9
                   SLIMES      11.5     3.50      23.1
                   TOTAL      100.0   ( 1.74)    100.0                      CALCULATED HEAD ASSAY=  1.74%
```

```
SAMPLE  A 55  SULPHIDE  (TOTAL-OXIDE)  COPPER  DISTRIBUTION

  SIZE FRACTION    SPEC GRAV   ---  IN SIZE FRACTIONS ---       IN HEAD SAMPLE                 CUMULATIVE DATA
  (MILLIMETRES)    FRACTION    WT  %   ---  SULPH. CU --        WT  %   SULPH. CU           WEIGHT    ASSAY     DISTN
                                       ASSAY %    DISTN %              DISTN %                 %         %         %

          +  .045  < 2.50       18.5       .10      11.4         16.37     9.86             16.37       .10      9.86
                   < 2.96       79.5       .10      49.2         70.36    42.35             70.36       .10     42.35
                   > 2.96        2.0      3.20      39.4          1.76    33.95              1.76      3.20     33.95
                   TOTAL       100.0  (    .16)    100.0         88.50    86.16             88.50       .16     86.16

  TOTALS FOR ABOVE < 2.50       18.5   (  .10)      11.4
  SIZE FRACTIONS   < 2.96       79.5   (  .10)      49.2
                   > 2.96        2.0   ( 3.20)      39.4

  OVERALL TOTALS..   SANDS      88.5   (  .16)      86.2
                     SLIMES     11.5      .20       13.8
                     TOTAL     100.0   (  .17)     100.0                      CALCULATED HEAD ASSAY=   .17%
```

```
                    SAMPLE  A56  WITH  SLIMES


SUMMARY OF INPUT DATA ....


SEPARATION TYPE  1          NO. OF SIZE FRACTIONS  2          NO. OF PRODUCTS  3

SIZE FRACTIONS ..
        .045     .000

SIZE ANALYSIS ..
       10.13     6.11

SPLITTING VALUES ..
        2.50     2.96

WEIGHTS IN PRODUCTS ..
        2.60     3.67     1.38
```

```
SAMPLE  A 56  OXIDE  COPPER  DISTRIBUTION

  SIZE FRACTION    SPEC GRAV  ---  IN SIZE FRACTIONS ---       IN HEAD SAMPLE                    CUMULATIVE DATA
  (MILLIMETRES)    FRACTION    WT  %   ---  OXIDE  CU --       WT  %   OXIDE  CU             WEIGHT    ASSAY     DISTN
                                       ASSAY %    DISTN %              DISTN %                  %         %         %

          + .045   < 2.50      34.0      2.60        9.7       21.20      6.49               21.20      2.60      6.49
                   < 2.96      48.0      5.40       28.5       29.92     19.01               29.92      5.40     19.01
                   > 2.96      18.0     31.20       61.8       11.25     41.30               11.25     31.20     41.30
                   TOTAL      100.0 (   9.10)      100.0       62.38     66.80               62.38      9.10     66.80

  TOTALS FOR ABOVE < 2.50      34.0    ( 2.60)       9.7
  SIZE FRACTIONS   < 2.96      48.0    ( 5.40)      28.5
                   > 2.96      18.0    (31.20)      61.8

  OVERALL TOTALS..   SANDS     62.4    ( 9.10)      66.8
                     SLIMES    37.6      7.50       33.2
                     TOTAL    100.0    ( 8.50)     100.0                      CALCULATED HEAD ASSAY=  8.50%
```

```
SAMPLE  A 56  ACID  SOLUBLE  COPPER  DISTRIBUTION

  SIZE FRACTION    SPEC GRAV    --- IN SIZE FRACTIONS ---       IN HEAD SAMPLE                  CUMULATIVE DATA
  (MILLIMETRES)    FRACTION    WT  %   ---  TOTAL  CU --        WT  %   TOTAL  CU          WEIGHT     ASSAY      DISTN
                                        ASSAY %    DISTN %                DISTN %               %          %          %

             + .045 < 2.50      34.0     2.60        9.7        21.20    6.49               21.20      2.60       6.49
                    < 2.96      48.0     5.40       28.5        29.92   19.01               29.92      5.40      19.01
                    > 2.96      18.0    31.20       61.8        11.25   41.30               11.25     31.20      41.30
                    TOTAL      100.0 (   9.10)     100.0        62.38   66.80               62.38      9.10      66.80

  TOTALS FOR ABOVE  < 2.50      34.0   ( 2.60)       9.7
  SIZE FRACTIONS    < 2.96      48.0   ( 5.40)      28.5
                    > 2.96      18.0   (31.20)      61.8

  OVERALL TOTALS..    SANDS     62.4   ( 9.10)      66.8
                      SLIMES    37.6     7.50       33.2
                      TOTAL    100.0   ( 8.50)     100.0                        CALCULATED HEAD ASSAY=  8.50%
```

```
                SAMPLE  A57  WITH  SLIMES


SUMMARY OF INPUT DATA ....

SEPARATION TYPE  1          NO. OF SIZE FRACTIONS  2         NO. OF PRODUCTS  3

SIZE FRACTIONS ..
         .045    .000

SIZE ANALYSIS ..
         9.79   16.77

SPLITTING VALUES ..
         2.50    2.96

WEIGHTS IN PRODUCTS ..
         1.89    4.93       .60
```

SAMPLE A 57 OXIDE COPPER DISTRIBUTION

SIZE FRACTION (MILLIMETRES)	SPEC GRAV FRACTION	--- IN SIZE FRACTIONS --- WT %	--- OXIDE CU -- ASSAY %	DISTN %	IN HEAD SAMPLE WT %	OXIDE CU DISTN %	CUMULATIVE DATA WEIGHT %	ASSAY %	DISTN %
+ .045	< 2.50	25.5	.30	11.9	9.39	5.76	9.39	.30	5.76
	< 2.96	66.4	.45	46.6	24.49	22.52	24.49	.45	22.52
	> 2.96	8.1	3.30	41.5	2.98	20.10	2.98	3.30	20.10
	TOTAL	100.0	(.64)	100.0	36.86	48.38	36.86	.64	48.38
TOTALS FOR ABOVE SIZE FRACTIONS	< 2.50	25.5	(.30)	11.9					
	< 2.96	66.4	(.45)	46.6					
	> 2.96	8.1	(3.30)	41.5					
OVERALL TOTALS..	SANDS	36.9	(.64)	48.4					
	SLIMES	63.1	.40	51.6					
	TOTAL	100.0	(.49)	100.0					

CALCULATED HEAD ASSAY= .49%

```
SAMPLE  A 57  ACID  SOLUBLE  COPPER  DISTRIBUTION

 SIZE FRACTION    SPEC GRAV   --- IN SIZE FRACTIONS ---      IN HEAD SAMPLE              CUMULATIVE DATA
 (MILLIMETRES)    FRACTION   WT  %   ---  TOTAL  CU --       WT  %  TOTAL  CU         WEIGHT    ASSAY     DISTN
                                     ASSAY %    DISTN %              DISTN %             %          %         %

         + .045   < 2.50     25.5       .30       11.9        9.39     5.76             9.39      .30      5.76
                  < 2.96     66.4       .45       46.6       24.49    22.52            24.49      .45     22.52
                  > 2.96      8.1      3.30       41.5        2.98    20.10             2.98     3.30     20.10
                  TOTAL     100.0 (     .64)     100.0       36.86    48.38            36.86      .64     48.38

 TOTALS FOR ABOVE < 2.50     25.5   (   .30)      11.9
 SIZE FRACTIONS   < 2.96     66.4   (   .45)      46.6
                  > 2.96      8.1   ( 3.30)       41.5

 OVERALL TOTALS..   SANDS    36.9   (   .64)      48.4
                    SLIMES   63.1       .40       51.6
                    TOTAL   100.0   (   .49)     100.0                 CALCULATED HEAD ASSAY=   .49%
```

APPENDIX F

PLANT SIZING AND COSTING OF EQUIPMENT

1. AGITATION LEACHING

1.1 Crushing and Grinding

For 180 t/d plant (1 shift operation) to 100% minus 1.67 mm (10 mesh BSS), 50% minus 76 micrometres (200 mesh BSS):

$92 000 (Amdel internal records).

Equivalent for 40 t/d = $92\ 000 \times \left(\frac{40}{180}\right)^{0.67}$ $34 000

1.2 Batch Leaching Vessel

This tank holds ½ day's production of 20 t solids and 20 t liquor. Assuming sp.gr. solids at 2.7, volume of slurry in tank is 25 kl (5500 gal.). Use 3.4 m (11 ft) diameter, 3.7 m (12 ft) high rubber-lined tank with stainless-steel agitator including drive and motor (capacity 34 kl: 7150 gal).

Cost (Amdel internal records): $ 9 000

1.3 Batch Settling Tanks

Number required is two. When fully loaded, tanks will hold 20 t solids and 80 t liquor (20 weight % solids). Volume of this slurry is 80 kl (17 500 gal). Using 90 kl (20 000 gal) rubber-lined tank with stainless-steel agitator, drive and motor and extrapolating from 1.2.

Cost of 2 settling tanks = $2 \times 9000 \times \left(\frac{90}{34}\right)^{0.67}$ $35 000

1.4 Clarified Liquor Tanks

Eight tanks of 72 kl (16 000 gal) capacity are required. Swimming pools are cheaper storage for light duty. In 25 kl (6000 gal) capacity at approximately $150 each, 24 are required.

Cost = 24 × 150 $ 4 000

1.5 Cementation

Hold up for cementation launders is about 1½ hours (Appendix A). Maximum size of launders is 17 kl (600 ft^3) costing approximately \$1250. Liquor production rate is 145 kl/day (32 000 gal/day) so operating time per day for a single launder

$$= \frac{145 \times 1.5}{17} \approx 13 \text{ hours.}$$

2 launders (operating 6½ h each per day), cost \$ 2 500

1.6 Pumps

Cost

(a)	Slurry pump of 8.5 kl/min (300 gal/min) to pump slurry from leach vessel to settler and underflow from settler to tailings dam:	\$2700	
(b)	Liquor transfer between wash tanks and decanted wash tanks 2 × 14 kl/min (500 gal/min) \$2700 ea.	\$5400	
(c)	Cementation pump from pregnant liquor storage to launders 1.4 kl/min (50 gal/min)	\$ 600	
		\$8700	\$ 9 000

Total of Equipment Costs \$93 500

2. PERCOLATION LEACHING

2.1 Crushing

For 180 t/d plant (1 shift operation) crushing to minus 6.35 mm (¼ in.): \$77 000 (Amdel internal records).

Equivalent for 40 t/d $= 77\ 000 \times \left(\frac{40}{180}\right)^{0\cdot 67}$ \$29 000

2.2 Screening

DSM Wet Screen and Stand \$ 1 000

2.3 Leaching Vats

Cost for 700-tonne capacity vat = \$15 000 (Amdel internal records), so equivalent for 40 t/d (with 8 vats 9 m (30 ft) × 3 m (10 ft) × 1.5 m (5 ft) high).

8 vats cost is $8 \times \left(\frac{40}{700}\right)^{0\cdot 67} \times 15\ 000$ \$18 000

2.4 Cementation

For a retention time of 1½ hours (Appendix A), size of cementation launder - 11 kl (400 ft^3).

Cost	$ 1 000

2.5 Liquor Storage

Pregnant liquor storage capacity of 150 kl (36 000 gal). Using 6 × 25 kl (6000 gal) swimming pools at $150

Cost 6 × 150	$ 1 000

2.6 Pumps

(a)	Slurry pump for screening operation, 28 kl/min (100 gal/min)	$1500	
(b)	8 leach vat pumps and 1 cementation of 6 kl/min (20 gal/min) capacity, 9 × $500 each	$4500	$ 6 000
	Total of Equipment Costs		$56 000

TABLES 1 to 47

FIGS 1 to 29

TABLE 1: DISSOLUTION RATES OF COPPER MINERALS AT ROOM TEMPERATURE (after Sullivan)

Mineral	Particle Size	Solvent % H_2SO_4	Extraction %	Time h
Tenorite	−147 +74 μm	1	98	1
Malachite	−147 +74 μm	1–5	100	1
Azurite	−147 +74 μm	1–5	100	1
Chrysocolla	−13.3 +0.68 mm	5	100	30
	− 1.65+0.59 mm	5	97	6

TABLE 2: LEACHING OF ROEBOURNE-PORT HEDLAND AREA ORE (1.6% Cu)

		Agitation Leaching				
Particle size,	micrometres	−152	−152	−600	−600	−600
Constant pH		1.0	2.0	1.0	2.0	2.0
Time,	h	2	2	2	2	5
Extraction	%	80	80	75	54	73
Acid consumption,	kg/kg Cu	6.5	3.3	4.2	2.1	2.3
		Percolation Leaching				
Particle size,	mm		12.7		6.35	
Stage 1: Constant	pH		1.0		1.0	
Time,	h		50		50	
Extraction,	%		57		58	
Stage 2:* Constant	pH		0.5		1.0	
Time,	h		166		160	
Extraction,	%		17		21	
Stage 3:* Constant	pH		–		1.0	
Time,	h		–		215	
Extraction,	%		–		3.5	
Total extraction			74		82.5	
Acid consumption,	kg/kg Cu		6.2		3.2	

* Stages 2 and 3 used fresh liquor for further leaching of the ore from Stage 1.

TABLE 3: ACID LEACHING OF ORE FROM THE VICINITY OF GLADSTONE, QUEENSLAND (0.26% Cu)

Agitation Leaching

Particle size,	micrometres	-420	-420
Constant pH		1.7	1.0
Time,	h	6	6
Extraction,	%	30	54
Acid consumption:	kg/tonne	16	33
	kg/kg Cu	21	24

Ammonia Leaching

The same ore was leached with an ammoniacal liquor containing 2M ammonia and 2M ammonium carbonate. After agitation for 6 hours, fresh lixiviant was used for another 6 hours.

Total Extraction: 12%.

The high acid consumption indicated that chlorite was probably being attacked. The low extractions might have been improved by finer grinding but are uneconomic due to the high acid consumption.

TABLE 4: ACID LEACHING OF A BURRA ORE
Variable acid addition
Agitation leaching on 200 g ore
Particle size: -76 micrometres
Pulp density: 40% solids

		Test Number						
		1	2	3	4	5	6	7
Acid addition,	g	20	10	10	3.4	4.0	6.8	9.0
Final pH*		0.72	1.37	1.41	3.5	3.5	2.5	2.0
Time,	h	4	2	4	4	6	4	4
Extraction,	%	91	83	94	23	31	56	68
Acid consumption,	kg/tonne	101	48.5	50.5	17	20.5	34.5	45
	kg/kg Cu	7.0	3.8	3.4	4.7	4.2	4.0	4.2

* Tests 1 to 3 all acid was added at commencement of leaching.
In Tests 4 to 7 acid was added sufficient to maintain the indicated pH.

TABLE 5: ACID LEACHING OF A BURRA ORE
Variable particle size
Agitation leaching on 200 g ore
Pulp density: 40% solids
Acid added: 10 g

		Particle Size		
		-76 micrometres	-300 micrometres	-6.35 mm
Time,	h	1	4	6
Final pH		1.7	1.5	1.3
Extraction,	%	89	91	92
Fe extraction,	%	12	1.5	1.3
Acid consumption,	kg/tonne	50	50	50
	kg/kg Cu	3.4	3.1	3.2

TABLE 6: LEACHING OF A SECOND BURRA ORE

Acid Agitation Leaching
Head grade: 1.20% Cu
Particle size: -152 micrometres
Constant pH: 2.0

Time,	h	0.5	1	2
Extraction,	%	31	42	44
Acid consumption,	kg/tonne	95	110	122.5
	kg/kg Cu	26	22	24

Ammonia Agitation Leaching
Head grade: 1.90% Cu
Particle size: -152 micrometres
Pulp density: 33% solids

NH_3 Molarity	1	1.5	1.5	2	3
$(NH_4)_2CO_3$ Molarity	0.5	0.5	0.75	1	1.5
Total equivalent NH_3	2	2.5	3	4	6
Extraction %,					
Time, h: 0.5	20	28	37	48	-
1.0	24	34	45	52	-
2.5	29	41	51	55	-
6	-	-	-	56	59
12*	-	-	-	64	67

Ammonia Percolation Leaching
Head grade: 1.67%

Particle size,	-6.35 mm +105 micrometres
Lixiviant	0.5M NH_3 and 2.5M $(NH_4)_2CO_3$
Time,	2 days, then leached with fresh liquor for a further 1 day
Total extraction, %	73

*Two 6-hour leaches, using fresh liquor for the second.

TABLE 7: LEACHING OF A MOUNT GUNSON ORE
Head grade: 1.2%
Particle size: −500 micrometres
Acid (30 kg/tonne) was added over 0.5 h and samples were taken at intervals subsequently

		Time, h			
		0.5	1	2	3
Extraction,	%	96	97	98	98
Acid consumption,	kg/tonne	30	−	−	−
	kg/kg Cu	2.6	−	−	−

TABLE 8: ACID LEACHING OF A UKAPARINGA ORE
Agitation leach
Pulp density: 60% solids

		micrometres	micrometres		mm
Underground Ore — Head grade: 1.27% Cu					
Particle size,		−152	−152		−6.35
pH		1.5	2.0		1.3
Time,	h	1	1		1
Extraction,		77	73		65
Acid consumption,	kg/tonne	65	60		25
	kg/kg Cu	6.6	6.5		3.0
Surface Ore — Head grade: 0.96% Cu					
Particle size,		−300	−300	−1.67	−1.67
pH		1.3	1.5	1.0	1.5
Time,	after steady pH	2	2	2	2
Extraction,	%	38	33	35	25
Acid consumption,	kg/tonne	50	15	120	35
	kg/kg Cu	14	5	36	15

TABLE 9: ACID LEACHING OF A FLINDERS RANGES ORE
Agitation leaching
Particle size: -6.35 mm

		Constant pH		
		1.0	1.5	2.0
Sample A — Head grade: 2.5% Cu, 3.1% CO_2				
Time,	h	8	8	8
Time to consume 90% of acid,	h	4	5	6
Extraction,	%	89	84	68
Acid consumption:	kg/tonne	85	70	67.5
	kg/kg Cu	3.4	3.0	4.0
Sample B — Head grade: 3.1% Cu, 0.93% CO_2				
Time,	h	10	10	10
Time to consume 90% of acid,	h	5.5	6	7
Extraction,	%	87	79	51
Acid consumption:	kg/tonne	32.5	31.5	23.5
	kg/kg Cu	2.7	2.6	2.6
Sample C — Head grade: 3.1% Cu, 0.53% CO_2				
Time,	h	5	5	5
Time to consume 90% of acid,	h	nd*	1	1.5
Extraction,	%	96	82	75
Acid consumption,	kg/tonne	65	52.5	47.5
	kg/kg Cu	2.0	1.6	1.6

* nd = Not determined.

TABLE 10: ACID LEACHING OF A FLINDERS RANGE ORE
Sample C, screened

		Particle Size, micrometres	
		+850	-850
Assay,	Cu %	3.8	2.1
	CO_2 %	0.65	0.35
Constant pH		1.5	1.5
Time,	h	5	5
Time to consume 90% of acid,	h	1	1
Extraction,	%	97	94
Acid consumption,	kg/tonne	55	50
	kg/kg Cu	1.5	2.3

TABLE 11: SAMPLE DETAILS: LABORATORY-SCALE TESTS

Sample	Location	Sample Size kg	Comments
A50 A412/71	Paratoo Mine	1.4	Hand-picked
A51 A413/71	Yappala Mine, Bandioota	8.7	-
A52 A414/71	Kanyaka Mine	6.5	-
A53 A415/71	Dome Rock Mine	18.0	-
A54 A416/71	Dome Rock Mine	1.3	Hand-picked chalcocite
A55 A417/71	Mt Gunson	5.0	-
A56 A420/71	Paratoo Mine	7.7	-
A57	Elsie Adair Mine, Copley	1497	-

TABLE 12: AGITATION LEACHING TESTS; LABORATORY TESTS AT 50% SOLIDS

Sample	% Cu	Acid Consumption		pH	Residue % Cu and (% Extraction)					Final Extraction %	Main Cu Minerals	Comments	Ore Value as Extracted Cu Less Cost of Acid $/tonne†
		kg H_2SO_4/t	/Theoretical* %		½ h	1 h	2 h	4 h	8 h				
A50 Paratoo	7.10	143.1	135.8	1	3.65 (48.4)	2.45 (65.5)	2.20 (69.0)	1.28 (82.0)	0.39 (95.0)	95	Malachite	-	75.3
A51 Bandioota Yappala	1.81	47.5	177.1	1	0.51 (71.8)	0.24 (86.7)	0.24 (86.7)	0.29 (84)	0.10 (94)	94	Malachite	-	18.5
A52 Kanyaka	3.25	56.2	112.0	1	0.97 (69.7)	0.53 (83.4)	0.47 (85.5)	0.11 (96.6)	0.05 (98.4)	98.4	Malachite Paratacamite	-	36.1
A53 Dome Rock Ore	4.70	164.2	575.0	1	3.25 (30.8)	3.05 (35.2)	3.25 (30.8)	2.50 (46.8)	2.85 (39.4)	39.4	Malachite Chalcocite	No oxidising agent for the chalcocite	15.6
A54 Dome Rock Ore	25.0	245.5	196.4	1	17.6 (29.6)	16.8 (32.8)	16.9 (32.4)	18.9 (24.4)	16.9 (32.4)	32.4	Chalcocite Malachite	Started with pH 1 but after 1 h was controlled at pH 2	87.2
A55 Mt Gunson	2.14	48.6	161.5	1	0.67 (68.7)	0.42 (80.4)	0.27 (87.4)	0.18 (91.6)	0.22 (89.7)	89.7	Chrysocolla Malachite Atacamite	-	21.5
A56 Paratoo, more representative	7.5	94.9	129.3	1	3.2 (57.4)	3.0 (60.0(	2.45 (67.3)	1.40 (81.3)	1.73 (76.9)	76.9	Malachite Chrysocolla	-	65.2
A57 Elsie Adair	0.51	25.6	856.0	1	0.44 (14.7)	0.33 (35.3)	0.29 (43.2)	0.29 (43.2)	0.32 (37.3)	37.3	-	-	1.3
A51 Yappala	1.81	30.8	138	2	- -	- -	0.80 (55.6)	0.66 (63.3)	0.38 (78.9)	78.9	-	-	16.0
A52 Kanyaka	3.25	27.4	~100	2	-	-	1.80 (44.6)	1.50 (53.8)	1.35 (58.5)	58.5	-	-	21.7
A53 Dome Rock	4.70	269.3	491.7	1	- -	- -	2.6 (44.7)	2.1 (55.3)	1.9 (59.6)	1.15 (48 h) 75.5	-	Oxidising agent; redox electrode maintained at 400 mV	31.8
A55 Mt Gunson	2.14	21.0	135	2	- -	- -	1.60 (25.3)	1.40 (34.6)	1.15 (46.3)	46.3	-	-	11.1
A56 Paratoo	7.5	60.7	106	2	- -	- -	5.10 (32)	4.95 (34)	3.85 (48.6)	- 48.6	-	-	41.3

† Copper at $1200 per tonne. Acid at $40 per tonne.

* Actual acid consumption/theoretical acid consumption for copper dissolved.

TABLE 13: EFFECTS OF TEMPERATURE AND GRIND SIZE ON AGITATION LEACHING AT 50% SOLIDS
Sample A56 (Paratoo)
Head grade: 7.95% Cu

Sample	Grind	Temperature	Acid Consumption		Actual Addition /Theoretical for Cu Dissolved	Residue % Cu and (% Extraction)			
		°C	kg/tonne	lb/ton		½ h	1 h	2 h	4 h
1	-1.65 mm (-10 mesh)	20	109.3	244.8	1.05	4.90 (38.4)	4.50 (43.4)	2.10 (74.6)	1.36 (82.9)
2	-300 µm (-52 mesh)	20	144.0	322.5	1.20	1.70 (78.6)	0.78 (90.2)	0.38 (95.2)	0.30 (96.2)
3	-1.65 mm (-10 mesh)	50	135.2	302.8	1.21	2.30 (71.1)	1.55 (80.5)	0.64 (91.9)	0.84 (89.4)
4	-300 µm (-52 mesh)	50	150.0	336.1	1.25	0.44 (94.5)	0.30 (96.2)	0.25 (96.9)	0.32 (96.0)

TABLE 14: EFFECTS ON LEACHING OF DIFFERENT METHODS OF ACID ADDITION

Sample (1-tonne)	Acid Add^{n}.	Acid Consumption		Actual Addition /Theoretical for Cu Dissolved	Head	Residue % Cu and (% Extraction)				
		kg/tonne	lb/ton		Cu %	½ h	1 h	2 h	4 h	6 h
Paratoo	(1)	68.1	152.6	1.94	2.40	0.46 (80.8)	0.30 (87.5)	0.20 (91.7)	0.13 (94.6)	0.12 (95.0)
	(2)	68.1	152.6	1.92	2.40	0.28 (87.3)	0.17 (92.9)	0.11 (95.4)	0.12 (95.0)	0.10 (95.8)
	(3)	102.1	228.9	2.93	2.40	0.20 (91.6)	0.14 (94.2)	0.14 (94.2)	0.09 (96.2)	0.14 (94.2)
Mt Gunson	(1)	48.8	109.2	1.63	2.08	0.37 (82.2)	0.23 (88.9)	0.18 (91.3)	0.18 (91.3)	0.13 (93.7)
	(2)	48.8	109.2	1.63	2.08	0.21 (89.9)	0.19 (90.9)	0.16 (92.3)	0.16 (92.3)	0.13 (93.7)
	(3)	73.2	163.8	2.38	2.08	0.21 (89.9)	0.14 (93.3)	0.13 (93.7)	0.15 (92.8)	0.11 (94.7)

(1) Controlled pH 1.

(2) Same total acid addition as (1), all added at start of leach.

(3) 50% additional acid to (2), all added at start of leach.

TABLE 15: SAMPLE FINES (% FINES REMOVED PRIOR TO PERCOLATION LEACHING)
Splitting of sample crushed to -6.35 mm (¼ in.) into -6.35 mm (¼ in.) to +300 μm (52 mesh BSS) and -300 μm (52 mesh BSS)

Sample		% -300 μm (52 mesh BSS) by Weight
A50	Paratoo	5.8
A51	Yappala	6.5
A52	Kanyaka	22.5
A53	Dome rock	7.6
A54	Dome rock (not leached)	6.0
A55	Mt Gunson	18.3
A56	Paratoo	9.0
A57	Elsie Adair (not leached)	13.3

TABLE 16: PERCOLATION LEACHING TESTS
Percolation leach: 48 h at pH 1
on -6.35 mm (¼ in.) +300 μm (52 mesh BSS)

Sample	Head % Cu	Residue % Cu	Extraction %	Value of Cu Extracted/t of Ore Leached, $*	Fraction <300 μm (52 mesh BSS) %	Comments
A50	7.10	1.58	78	66	5.8	-
A51	1.81	0.27	85	18	6.5	-
A52	3.25	0.22	93	36	22.5	-
A53	4.7	4.3	8.5	5	7.6	(No oxidant)
A55	2.14	0.17	92	24	18.3	-
A56	7.5	1.65	78	70	9.0	-

Note: All percolated well - none showed any signs of breaking up and blocking liquid circulation.

* Copper at $1200 per tonne.

TABLE 17: UNDERSIZE DISCARDED BEFORE PERCOLATION LEACHING

Ore		Split		Undersize Wt, %
L1	Paratoo	355 μm	(44 mesh BSS)	26.0
L2	Paratoo	250 μm	(60 mesh BSS)	19.5
L3	Paratoo	180 μm	(85 mesh BSS)	15.4
L4	Mt Gunson	180 μm	(85 mesh BSS)	9.0

TABLE 18: FINAL EXTRACTIONS FROM PERCOLATION LEACH

Ore		Head Grade % Cu	Residue %	Total Extraction % from Fraction Leached
L1	Paratoo	2.4	0.22	90.7
L2	Paratoo	2.4	0.21	91.3
L3	Paratoo	2.4	0.24	90.0
L4	Mt Gunson	2.08	0.10	95.2

TABLE 19: SULPHURIC ACID CONSUMPTION

Ore		Acid Consumption		Consumption:Theoretical for Cu Dissolved	Comparative Acid Consumption by Mechanical Agitation kg/tonne
		kg/tonne	lb/ton		
L1	Paratoo	46.2	103.5	1.38	68.1
L2	Paratoo	51.3	115.0	1.52	68.1
L3	Paratoo	48.1	107.8	1.45	68.1
L4	Mt Gunson	34.2	76.6	1.11	48.8

TABLE 20: SIZING OF PARATOO ORE BEFORE GRINDING
(Material used for L1 and L2)

Screen Sizing		Weight Retained			
Nominal Size	Equivalent Mesh BSS	g	%	Cumulative %	
				Passing	Retained
1.40 mm	12	0.32	0.15	99.85	0.15
1.00 mm	16	8.41	4.00	95.85	4.15
710 micrometres	22	30.70	14.62	81.23	18.77
500 micrometres	30	26.77	12.75	68.48	31.52
355 micrometres	44	21.72	10.34	58.14	41.86
250 micrometres	60	14.30	6.81	51.33	48.67
180 micrometres	85	10.46	4.98	46.35	53.65
125 micrometres	120	11.16	5.32	41.03	58.97
90 micrometres	170	8.70	4.14	36.89	63.11
63 micrometres	240	9.53	4.54	32.35	67.65
45 micrometres	350	9.32	4.44	27.91	72.09
	-350	58.61	27.91	-	100.00

TABLE 21: SIZING OF PARATOO ORE AFTER GRINDING
(Material used for L6)

Screen Sizing		Weight Retained		
Nominal Size, micrometres	Equivalent Mesh BSS	g	%	Cumulative % Retained
710	22	9.55	5.36	5.36
500	30	8.40	4.72	10.08
355	44	7.32	4.11	14.19
250	60	4.80	2.70	16.89
180	85	4.08	2.29	19.18
125	120	6.14	3.45	22.63
90	170	7.84	4.40	27.03
75	200	5.68	3.19	30.22
63	240	7.50	4.21	34.43
45	350	15.20	8.53	42.96
-45	-350	101.59	57.04	100.00

TABLE 22: SIZING OF MT GUNSON ORE BEFORE GRINDING
(Material used for L3 and L4)

Screen Sizing		Weight Retained		
Nominal Size	Equivalent Mesh BSS	g	%	Cumulative % Retained
1.40 mm	12	0.20	0.10	0.10
1.00 mm	16	5.85	2.93	3.03
710 micrometres	22	24.37	12.20	15.23
500 micrometres	30	28.42	14.23	29.46
355 micrometres	44	35.05	17.55	47.01
250 micrometres	60	29.32	14.68	61.69
180 micrometres	85	21.61	10.82	72.51
125 micrometres	120	15.58	7.80	80.31
90 micrometres	170	7.85	3.93	84.24
63 micrometres	240	5.98	3.00	87.24
45 micrometres	350	4.15	2.08	89.32
	-350	21.32	10.68	100.00

TABLE 23: SIZING OF MT GUNSON ORE AFTER GRINDING (Material used for L5)

Screen Sizing		Weight Retained			
Nominal Size, micrometres	Equivalent Mesh BSS	g	%	Cumulative %	
				Retained	Passing
355	44	1.00	0.55	0.55	99.45
250	60	5.32	2.89	3.44	96.56
180	85	11.98	6.51	9.95	90.05
125	120	23.94	13.01	22.96	77.04
90	170	23.90	12.99	35.95	64.05
75	200	12.50	6.79	42.74	57.26
63	240	10.66	5.79	48.53	51.47
45	350	20.44	11.11	59.64	40.36
	-350	74.26	40.36	100.00	

TABLE 24: POWER REQUIREMENTS FOR MECHANICAL AGITATION

Agitator Speed rev/min	L1 Paratoo hp/1000 gal	L3 Mt Gunson hp/1000 gal	L5 Mt Gunson hp/1000 gal	L6 Paratoo hp/1000 gal
27.5	0.214	-	-	-
55.0	0.55	-	-	-
82.5	1.07	-	0.73	-
110	2.52	3.18	1.71	1.83
137.5	4.22	4.43	3.46	4.33
165	7.21	6.29	6.47	6.53
192.5	11.26	9.90	10.26	11.34
220	16.53	14.66	-	17.34
247.5	23.08	17.95	-	24.73
275	30.53	-	-	-
Operating hp/1000 gal	11.3	16.3	5.5	4.3
Stirrer speed rev/min*	192.5	233.8	159.5	137.5

* Operating speed required for good mixing during leaching tests.

TABLE 25: LEACHING EXTRACTION EFFICIENCY AS A FUNCTION OF TIME

Sample		Acid Consumption			Head	Residue % Cu and (% Extraction)					Method of
		kg/tonne	lb/ton	Actual/Theoretical for Cu Dissolved	% Cu	½ h	1 h	2 h	4 h	8 h	Agitation
Paratoo (coarse)	L1	68.1	152.6	2.22	2.10	0.19 (90.9)	0.15 (92.9)	0.15 (92.9)	0.11 (94.8)	0.11 (94.8)	Mechanical
Paratoo (coarse)	L2	68.1	152.6	1.87	2.60	0.32 (87.7)	0.18 (93.1)	0.18 (93.1)	0.23 (91.5)	0.24 (90.8)	Pachucha
Mt Gunson (coarse)	L3	48.8	109.2	1.59	2.08	0.20 (90.4)	0.16 (92.3)	0.11 (94.7)	0.10 (95.2)	0.08 (96.2)	Mechanical
Mt Gunson (coarse)	L4	48.8	109.2	1.63	2.08	– –	0.15 (92.8)	0.12 (94.2)	0.13 (93.8)	0.14* (93.3)	Pachucha
Mt Gunson (fine)	L5	61.7	138.1	2.07	2.08	0.32 (84.6)	0.12 (94.2)	0.10 (95.2)	0.10 (95.2)	0.08** (96.2)	Mechanical
Paratoo (fine)	L6	74.3	166.3	2.11	2.40	0.13 (94.6)	0.12 (95.0)	0.10 (95.8)	0.12 (95.0)	0.11* (95.4)	Mechanical

* 5½ h. ** 7 h.

TABLE 26: pH OF SINGLE ACID ADDITION LEACHES (L1-L4) AS A FUNCTION OF TIME

	pH at Time				
	½ h	1 h	2 h	4 h	8 h
L1 Paratoo	0.6	0.68	0.78	0.85	0.90
L2 Paratoo	0.55	0.60	0.67	0.85	0.95
L3 Mt Gunson	0.8	0.95	1.1	1.2	1.3
L4 Mt Gunson	-	0.95	1.05	1.15	1.20*

* 5½ h.

TABLE 27: RESULTS OF SETTLING TESTS - PARATOO LEACH NO.1 MECHANICAL AGITATION FOR 2 HOURS (FIG.12)

Test No.		1	2	3
Flocculant		Nil	Superfloc 16	
Dose rate	g/t	-	29	61
Pulp weight	g	569	570	569
Pulp volume	ml	504	504	507
Feed density	wt % solids	13	15	14
Settled vol. after 1 h	ml	246	225	202
Solids in settled pulp	%	28	29	33
Dry solids wt	g	84	82	86
H_2O in filter cake	%	-	-	25
Liquor clarity			Slightly Cloudy	

TABLE 28: RESULTS OF SETTLING TESTS - PARATOO LEACH NO.1 MECHANICAL AGITATION FOR 4 HOURS (FIG.13)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	28	56	83
Pulp weight	g	567	575	583	588
Pulp volume	ml	501	507	513	517
Feed density	wt % solids	15	15	15	15
Settled vol. after 1 h	ml	252	238	212	190
Solids in settled pulp	%	28	30	32	35
Dry solids weight	g	87	89	89	91
H_2O in filter cake	%	26	25	26	28
Liquor clarity			Slightly Cloudy		

TABLE 29: RESULTS OF SETTLING TESTS - PARATOO LEACH NO.1 MECHANICAL AGITATION FOR 8 HOURS (FIG.14)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	27	58	84
Pulp weight	g	572	572	571	581
Pulp volume	ml	507	504	507	513
Feed density	wt % solids	15	16	15	15
Settled vol. after 1 h	ml	253	255	214	185
Solids in settled pulp	%	28	29	32	36
Dry solids weight	g	87	92	86	90
H_2O in filter cake	%	24	25	26	25
Liquor clarity		Slightly Cloudy			

TABLE 30: RESULTS OF SETTLING TESTS - PARATOO LEACH NO.2 AIR AGITATION FOR 2 HOURS (FIG.15)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	13	28	41
Pulp weight	g	628	643	639	640
Pulp volume	ml	503	514	514	514
Feed density	wt % solids	28	29	29	28
Settled vol. after 1 h	ml	275	273	240	232
Solids in settled pulp	%	45	47	51	51
Dry solids weight	g	178	187	182	181
H_2O in filter cake	%	18	20	18	21
Liquor clarity		Cloudy	Sl. Cloudy	Clear	Clear

TABLE 31: RESULTS OF SETTLING TESTS - PARATOO LEACH NO.2
AIR AGITATION FOR 4 HOURS (FIG.16)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	18	33	53
Pulp weight	g	609	609	614	626
Pulp volume	ml	501	510	506	520
Feed density	wt % solids	25	23	25	23
Settled vol. after 1 h	ml	263	228	205	195
Solids in settled pulp	%	42	44	50	48
Dry solids weight	g	154	141	152	143
H_2O in filter cake	%	22	21	18	20
Liquor clarity		Slightly Cloudy		Clear	

TABLE 32: RESULTS OF SETTLING TESTS - PARATOO LEACH NO.2
AIR AGITATION FOR 8 HOURS (FIG.17)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	19	32	53
Pulp weight	g	598	597	625	617
Pulp volume	ml	512	504	512	516
Feed density	wt % solids	21	23	25	23
Settled vol. after 1 h	ml	242	225	257	207
Solids in settled pulp	%	38	43	44	47
Dry solids weight	g	123	135	159	143
H_2O in filter cake	%	19	14	25	18
Liquor clarity		Slightly Cloudy		Clear	

TABLE 33: RESULTS OF SETTLING TESTS - PARATOO LEACH NO.3
MECHANICAL AGITATION FOR 2 HOURS (FIG.18)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	30	60	92
Pulp weight	g	629	630	631	616
Pulp volume	ml	514	516	516	500
Feed density	wt % solids	26	26	26	26
Settled vol. after 1 h	ml	310	277	260	254
Solids in settled pulp	%	39	43	44	45
Dry solids weight	g	166	165	163	163
H_2O in filter cake	%	24	27	27	32
Liquor clarity		Slightly Cloudy		Clear	

TABLE 34: RESULTS OF SETTLING TESTS - PARATOO LEACH NO.3
MECHANICAL AGITATION FOR 4 HOURS (FIG.19)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	29	58	95
Pulp weight	g	612	629	626	637
Pulp volume	ml	500	502	500	524
Feed density	wt % solids	26	27	28	25
Settled vol. after 1 h	ml	310	298	280	257
Solids in settled pulp	%	38	41	43	43
Dry solids weight	g	158	172	174	158
H_2O in filter cake	%	31	26	28	27
Liquor clarity		Slightly Cloudy		Clear	

TABLE 35: RESULTS OF SETTLING TESTS - MT GUNSON LEACH NO.1
MECHANICAL AGITATION FOR 2 HOURS (FIG.20)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	15	51	63
Pulp weight	g	625	621	620	629
Pulp volume	ml	506	503	513	520
Feed density	wt % solids	27	27	24	25
Settled vol. after 1 h	ml	373	367	348	310
Solids in settled pulp	%	35	35	33	38
Dry solids weight	g	169	170	148	159
H_2O in filter cake	%	23	28	22	26
Liquor clarity		Slightly Cloudy		Clear	

TABLE 36: RESULTS OF SETTLING TESTS - MT GUNSON LEACH NO.1
MECHANICAL AGITATION FOR 4 HOURS (FIG.21)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	30	60	96
Pulp weight	g	600	624	638	643
Pulp volume	ml	500	512	524	536
Feed density	wt % solids	24	27	26	24
Settled vol. after 1 h	ml	383	392	388	365
Solids in settled pulp	%	30	33	34	34
Dry solids weight	g	143	166	168	157
H_2O in filter cake	%	25	21	25	18
Liquor clarity		Slightly Cloudy		Clear	

TABLE 37: RESULTS OF SETTLING TESTS - MT GUNSON LEACH NO.1
MECHANICAL AGITATION FOR 8 HOURS (FIG.22)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	71	100	124
Pulp weight	g	614	619	635	656
Pulp volume	ml	505	524	536	550
Feed density	wt % solids	26	23	24	25
Settled vol. after 1 h	ml	435	393	376	377
Solids in settled pulp	%	29	29	32	34
Dry solids weight	g	158	140	150	162
H_2O in filter cake	%	24	25	25	25
Liquor clarity		Slightly Cloudy		Clear	

TABLE 38: RESULTS OF SETTLING TESTS - MT GUNSON LEACH NO.3
MECHANICAL AGITATION FOR 2 HOURS (FIG.23)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	33	68	92
Pulp weight	g	601	618	627	647
Pulp volume	ml	500	514	528	536
Feed density	wt % solids	24	24	23	25
Settled vol. after 1 h	ml	322	318	270	263
Solids in settled pulp	%	34	36	41	44
Dry solids weight	g	142	151	147	162
H_2O in filter cake	%	25	23	27	25
Liquor clarity		Slightly Cloudy		Clear	

TABLE 39: RESULTS OF SETTLING TESTS - MT GUNSON LEACH NO.3
MECHANICAL AGITATION FOR 4 HOURS (FIG.24)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	30	63	101
Pulp weight	g	573	584	608	628
Pulp volume	ml	457	470	500	528
Feed density	wt % solids	28	28	26	24
Settled vol. after 1 h	ml	338	328	307	269
Solids in settled pulp	%	35	37	39	41
Dry solids weight	g	159	164	160	148
H_2O in filter cake	%	20	24	24	30
Liquor clarity		Slightly Cloudy		Clear	

TABLE 40: RESULTS OF SETTLING TESTS - MT GUNSON LEACH NO.3
MECHANICAL AGITATION FOR 8 HOURS (FIG.25)

Test No.		1	2	3	4
Flocculant		Nil	Superfloc 16		
Dose rate	g/t	-	33	64	116
Pulp weight	g	611	627	640	657
Pulp volume	ml	510	522	534	540
Feed density	wt % solids	24	24	24	20
Settled vol. after 1 h	ml	352	350	336	246
Solids in settled pulp	%	32	33	35	36
Dry solids weight	g	144	150	156	129
H_2O in filter cake	%	20	22	22	29
Liquor clarity		Slightly Cloudy		Clear	

TABLE 41: SUMMARY OF SETTLING TESTS

Leach Sample		% Solids W/W		Superfloc 16 Dose g/t
		Feed	Settled Pulp	
Paratoo No. 1,	2 h	14	33	60
	4 h	15	35	90
	8 h	15	36	85
Paratoo No. 2,	2 h	28	51	40
	4 h	23	48	50
	8 h	23	47	50
Paratoo No. 3,	2 h	26	45	90
	4 h	25	43	95
Mt Gunson No. 1,	2 h	25	38	60
	4 h	24	34	100
	8 h	25	34	125
Mt Gunson No. 3,	2 h	25	44	90
	4 h	24	41	100
	8 h	24	36	65

TABLE 42: OPERATING COSTS 10 000 t/a 2% Cu Ore

		Percolation $/a	Agitation $/a
1.	Capital Depreciation (over 3 years)	65 000	109 000
2.	Acid, at $40/ton H_2SO_4 assuming consumption 200% theoretical	16 000	20 000
3.	Labour, 4 men × $5000/a	20 000	20 000
4.	Scrap Fe 5¢/lb Cu, approximately 2.5 lb Fe/lb Cu	14 000	18 000
5.	Mining, $2/ton	20 000	20 000
6.	Water, (50¢/1000 gal)	1 000	4 000
7.	Power, (2¢/kWh)	2 000	4 000
8	Minor Maintenance and Overheads	10 000	10 000
	Total Operating Costs	148 000	204 000
	Product Value at 97¢/kg Cu	124 000	154 000
	Loss on Operation	24 000	50 000

TABLE 43: OPERATING COSTS FOR 3% AND 4% COPPER ORE — 10 000 t/a

	Percolation		Agitation	
	$*	$**	$*	$**
Capital Cost of Items	56 000	40 000	93 500	50 000
Installed Capital Cost	196 000	160 000	327 000	200 000
Operating Cost for 2% Ore	148 000†	126 000	204 000†	152 000
3% Ore				
Additional acid	8 000	8 000	10 000	10 000
Additional scrap	7 000	7 000	9 000	9 000
Total Operating Costs	163 000	141 000	223 000	171 000
Product Value @ 97¢/kg Cu	188 000	188 000	235 000	235 000
Surplus	25 000	47 000	12 000	64 000
Surplus/Investment	12.7%	29%	3.7%	32%
4% Ore				
Additional acid	16 000	16 000	20 000	20 000
Additional scrap	14 000	14 000	18 000	18 000
Total Operating Costs	178 000	156 000	242 000	190 000
Product Value @ 97¢/kg Cu	248 000	248 000	308 000	308 000
Surplus	70 000	92 000	66 000	118 000
Surplus/Investment	36%	58%	22%	59%

* New crushing equipment.

** Second-hand crushing equipment.

† From Table 42.

TABLE 44: SELLING PRICE OF COPPER GOES UP BY 20% ($1.20 to $1.44/kg Cu) — 10 000 t/a Ore

	Percolation		Agitation	
	$*	$**	$*	$**
Operating Cost for *2% ore*	148 000	126 000	204 000	152 000
Product Value	149 000	149 000	185 000	185 000
Surplus	1 000	23 000	−19 000	33 000
Surplus/Investment	0.5%	14%	-	16.5%
Operating Cost for *3% ore*	163 000	141 000	223 000	171 000
Product Value	226 000	226 000	282 000	282 000
Surplus	63 000	85 000	59 000	111 000
Surplus/Investment	32%	53%	18%	55%

* New crushing equipment.
** Second-hand crushing equipment.

TABLE 45: DOUBLING THROUGHPUT TO 20 000 t/a FOR 2% AND 3% ORE

	Percolation		Agitation	
	\$*	\$**	\$*	\$**
2% Ore				
Installed Capital Cost Cost $\propto$ (size)$^{0.67}$	311 000	254 000	520 000	318 000
Depreciation over 3 years	104 000	85 000	173 000	106 000
Acid	32 000	32 000	40 000	40 000
Labour: 6 men percolation 7 men agitation	30 000	30 000	35 000	35 000
Scrap Fe	28 000	28 000	36 000	36 000
Mining, 80¢/tonne	16 000	16 000	16 000	16 000
Water	2 000	2 000	8 000	8 000
Power	4 000	4 000	8 000	8 000
Maintenance	16 000	16 000	16 000	16 000
Total Operating Costs	232 000	213 000	332 000	265 000
Product Value	248 000	248 000	308 000	308 000
Surplus	16 000	35 000	-24 000	43 000
Surplus/Investment	5%	14%	-	14%
3% Ore				
Total Operating Costs	262 000	243 000	370 000	303 000
Product Value	376 000	376 000	470 000	470 000
Surplus	114 000	133 000	100 000	167 000
Surplus/Investment	37%	52%	19%	53%

* New crushing equipment.

** Second-hand crushing equipment.

TABLE 46: 40 000 t/a PLANT THROUGHPUT — 2% ORE

	Percolation		Agitation	
	\$*	\$**	\$*	\$**
Installed Capital Cost Cost ∝ (size)$^{0 \cdot 67}$	495 000	404 000	827 000	506 000
Depreciation over 3 years	165 000	135 000	276 000	169 000
Acid	64 000	64 000	80 000	80 000
Labour: 6 men percolation 7 men agitation	30 000	30 000	35 000	35 000
Scrap Fe	56 000	56 000	72 000	72 000
Mining, 75¢/ton	30 000	30 000	30 000	30 000
Water	4 000	4 000	16 000	16 000
Power	8 000	8 000	16 000	16 000
Maintenance	30 000	30 000	30 000	30 000
Total Operating Costs	383 000	353 000	547 000	440 000
Product Value	496 000	496 000	616 000	616 000
Surplus	113 000	143 000	69 000	176 000
Surplus/Investment	23%	35%	8%	35%

* New crushing equipment.

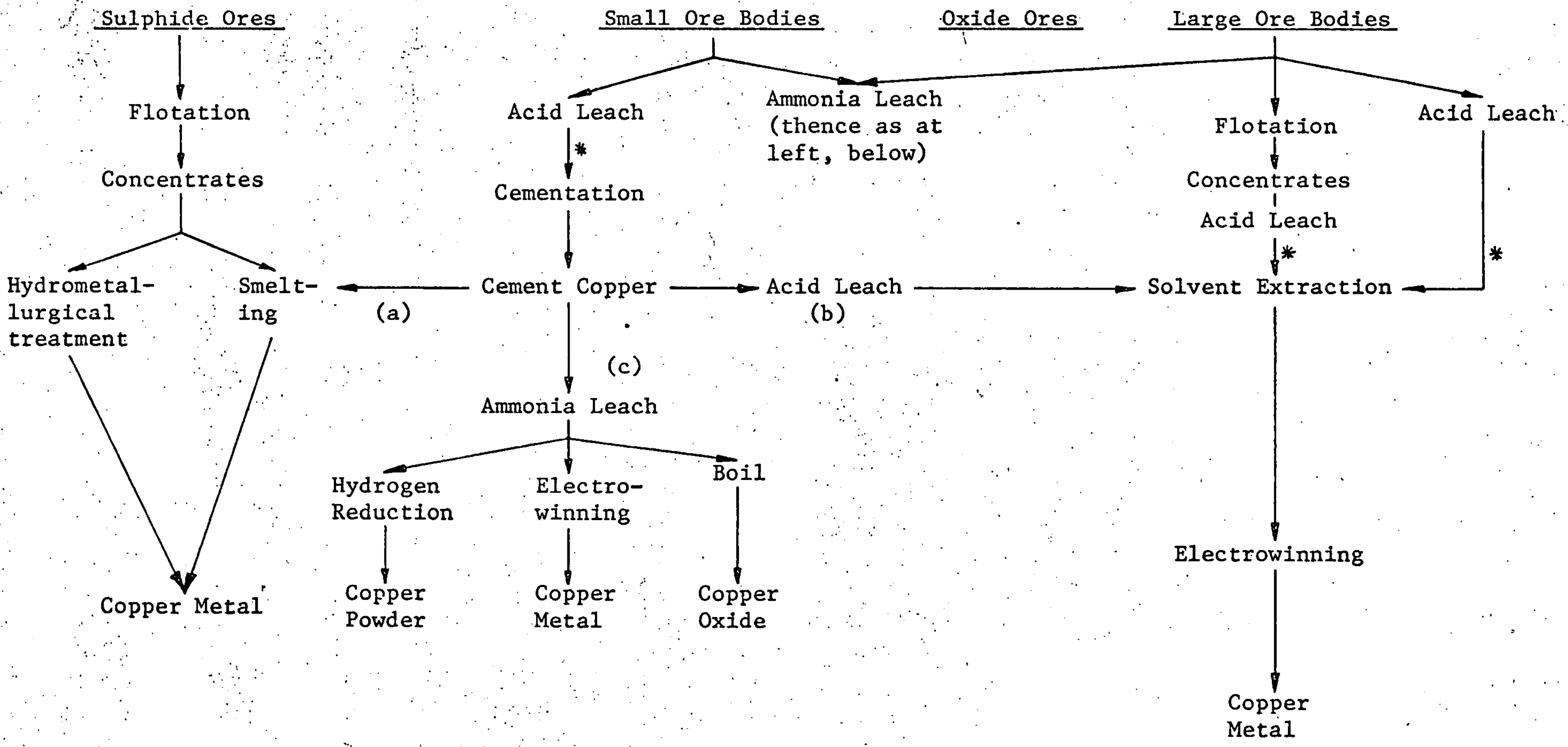
** Second-hand crushing equipment.

TABLE 47: PLANT OPERATING OVER 4 YEARS FOR 2% AND 3% ORE

	Percolation		Agitation	
	$*	$**	$*	$**
	2% Ore			
Operating Costs over 3 years	148 000	126 000	204 000	152 000
Less Lower Depreciation Rate	16 000	13 000	27 000	17 000
Operating Cost over 4 years	132 000	113 000	177 000	135 000
Value Product	124 000	124 000	154 000	154 000
Surplus	- 8 000	11 000	-23 000	19 000
Surplus/Investment	-	7%	-	9.5%
	3% Ore			
Operating Cost over 3 years	163 000	141 000	223 000	171 000
Less Lower Depreciation Rate	16 000	13 000	27 000	170 000
Operating Cost over 4 years	147 000	128 000	196 000	154 000
Value Product	188 000	188 000	235 000	235 000
Surplus	41 000	60 000	39 000	81 000
Surplus/Investment	21%	37.5%	12%	40.5%

* New crushing equipment.

** Second-hand crushing equipment.



* At these points cementation in pulp may be considered with subsequent copper recovery by flotation i.e. L.P.F.

FIG. 1: TREATMENT SCHEME FOR SOUTH AUSTRALIAN COPPER ORES

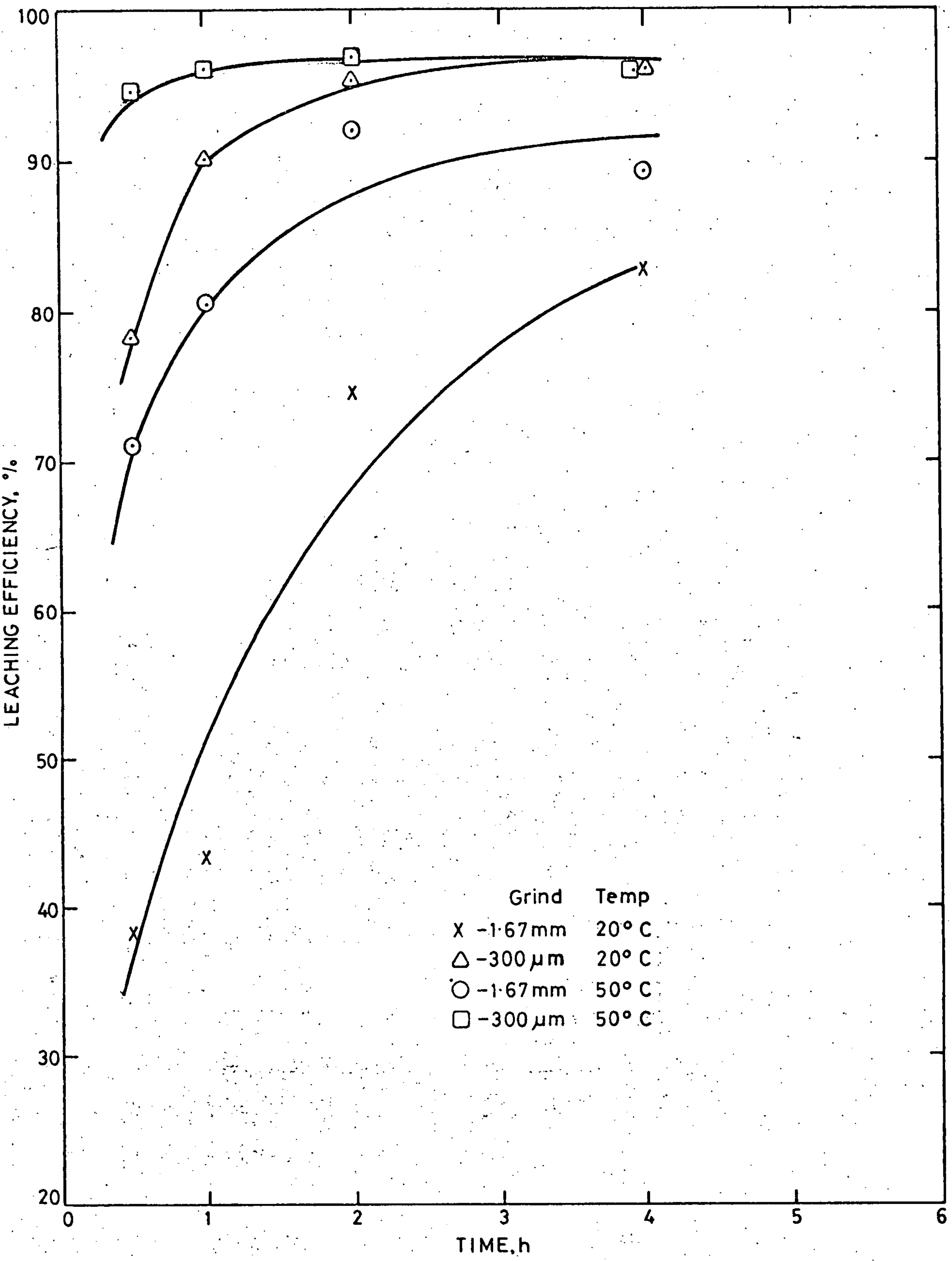


FIG. 2 : LEACHING EFFICIENCY v TIME FOR SAMPLE A56 PARATOO ORE

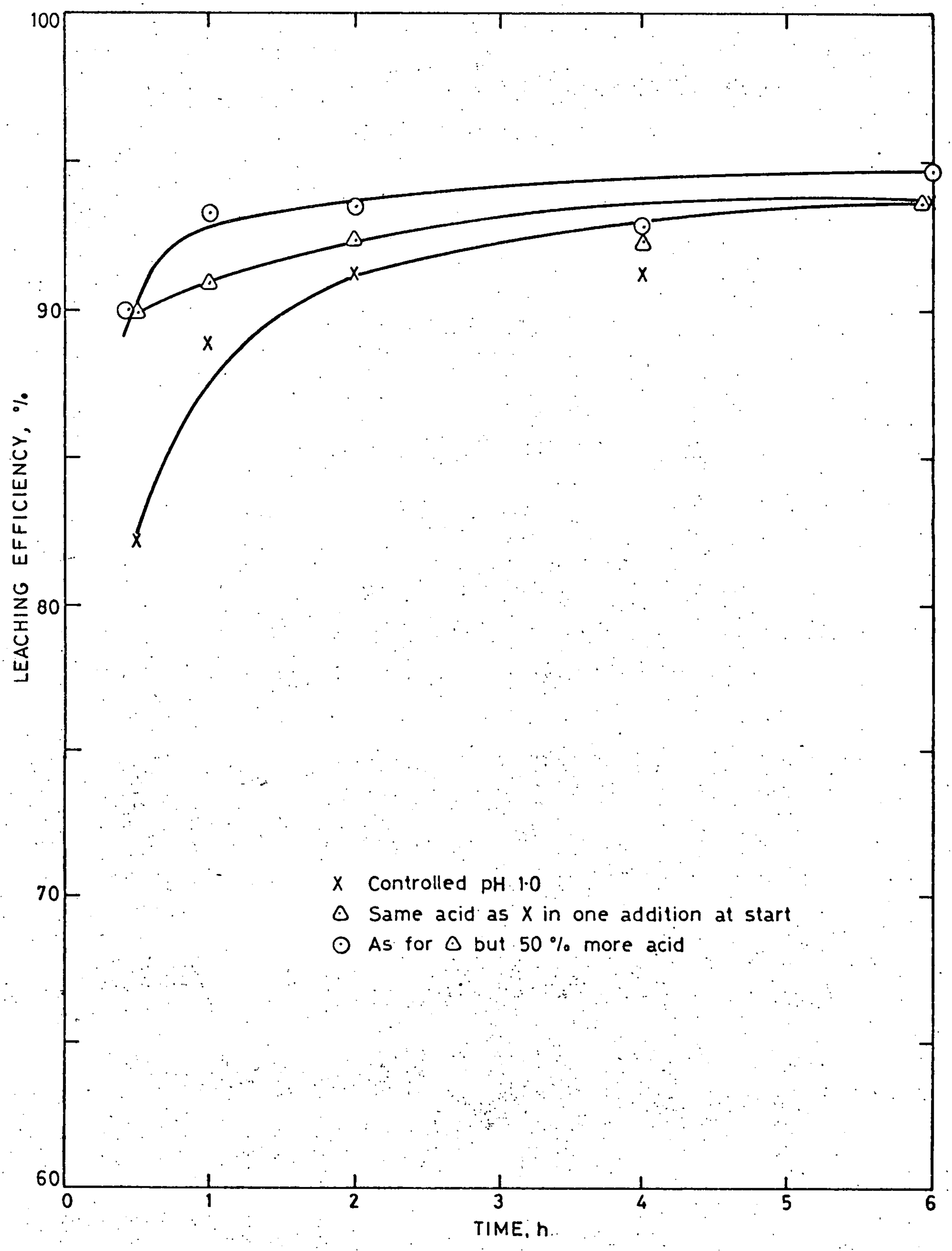


FIG.3: LEACHING EFFICIENCY v TIME FOR MT GUNSON ORE (1-tonne sample)

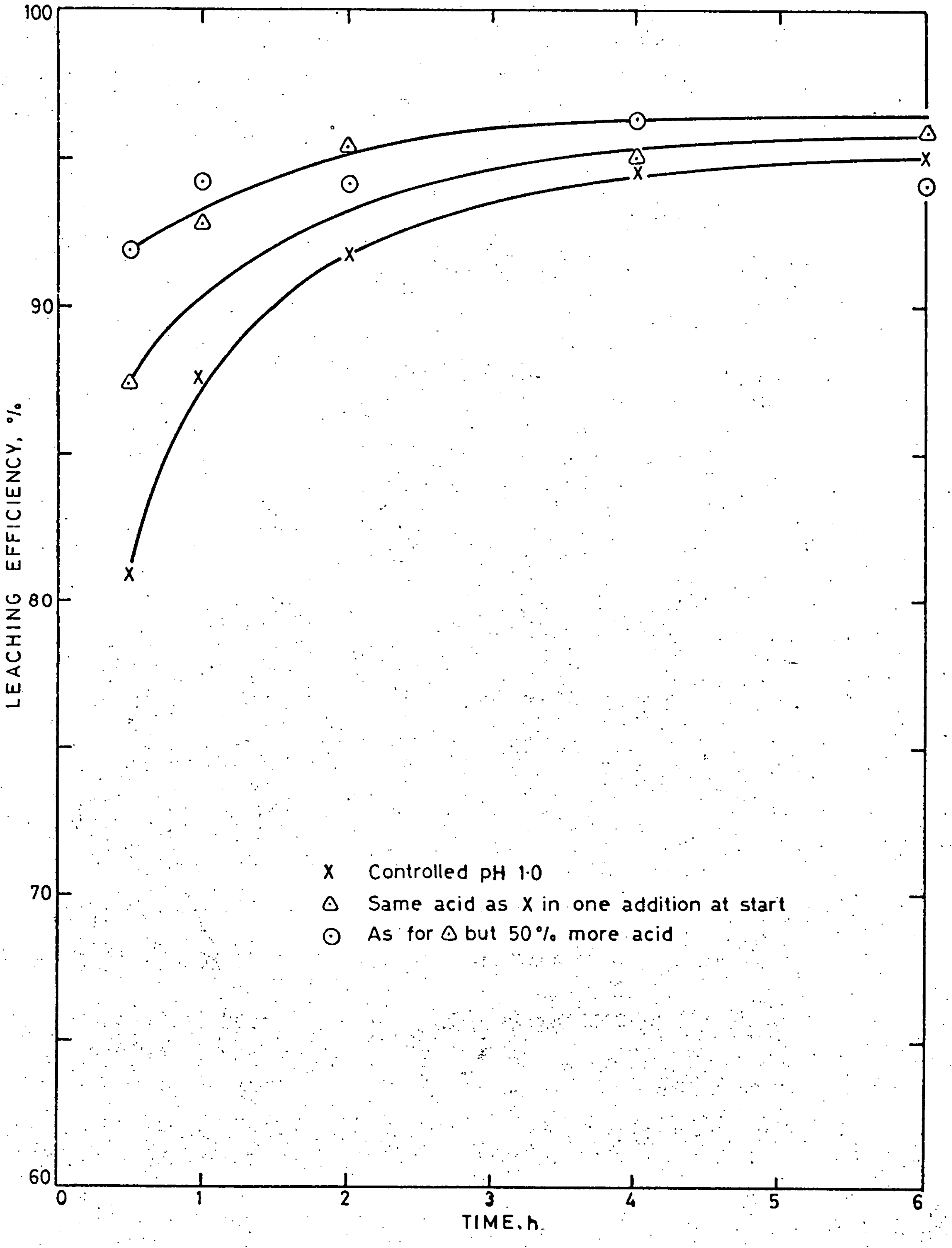


FIG. 4: LEACHING EFFICIENCY v TIME FOR PARATOO ORE (1-tonne sample)

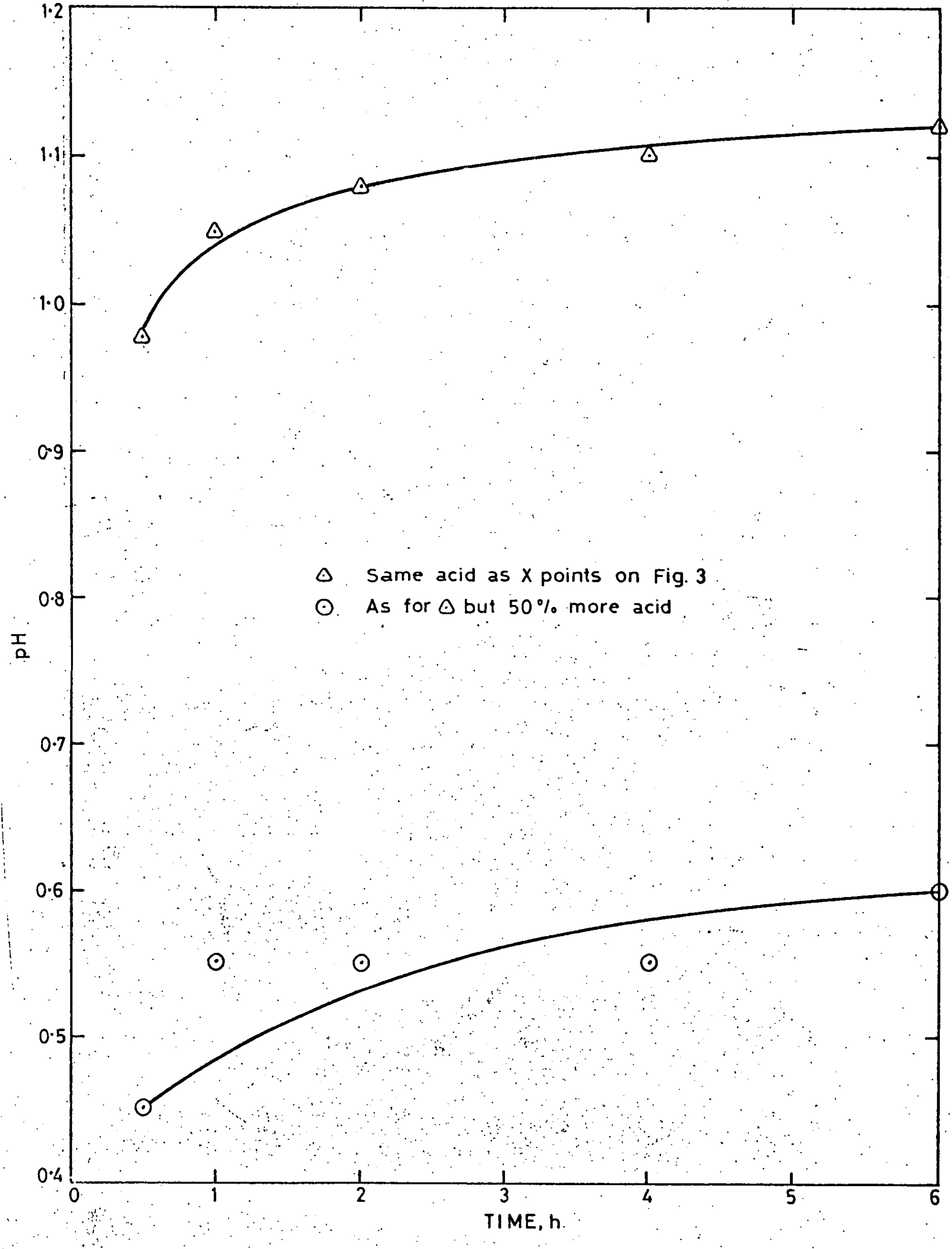


FIG. 5: CHANGE OF pH WITH TIME FOR LEACHING OF MT GUNSON ORE (1-tonne sample)

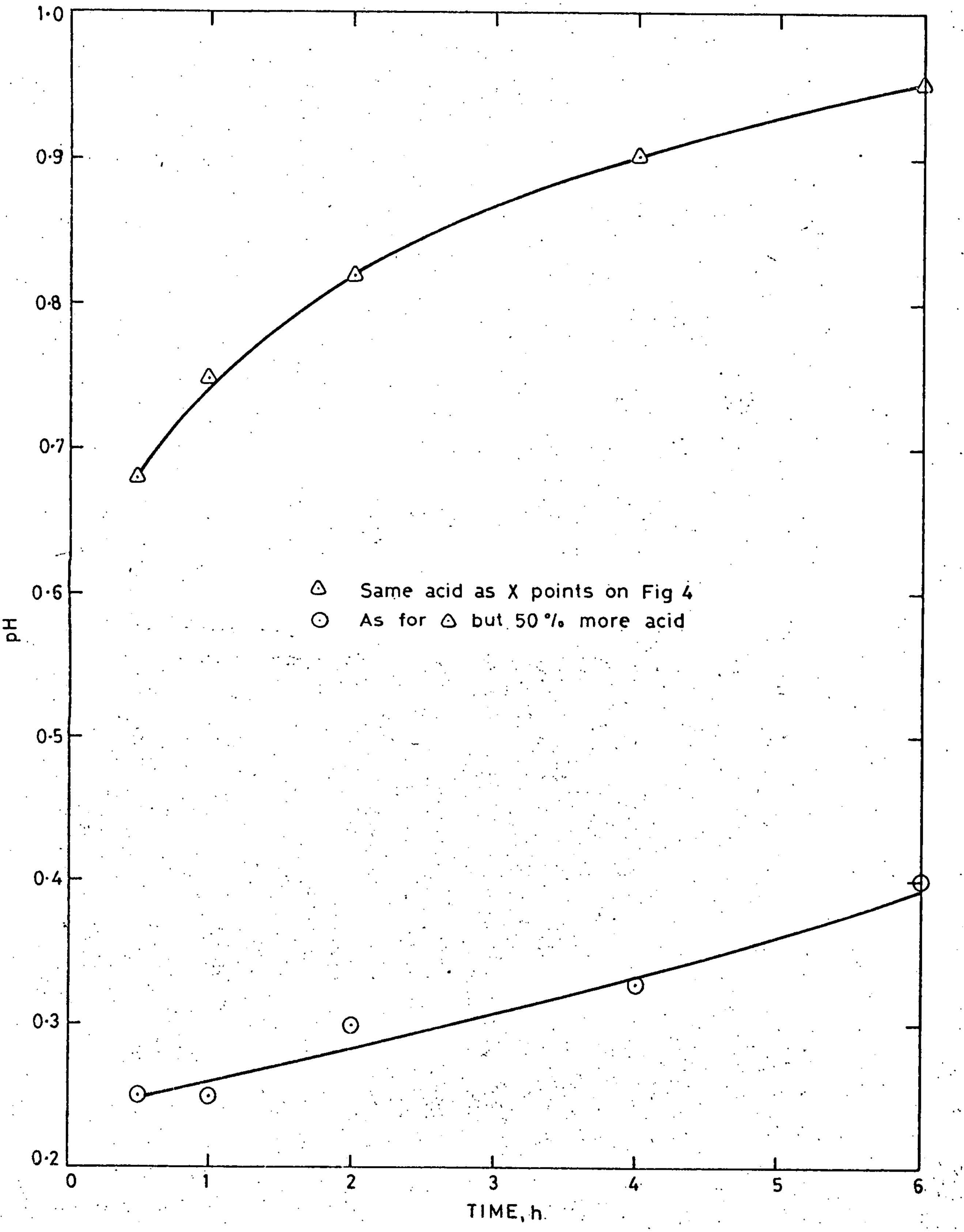


FIG. 6: CHANGE OF pH WITH TIME FOR PARATOO ORE (1-tonne sample)

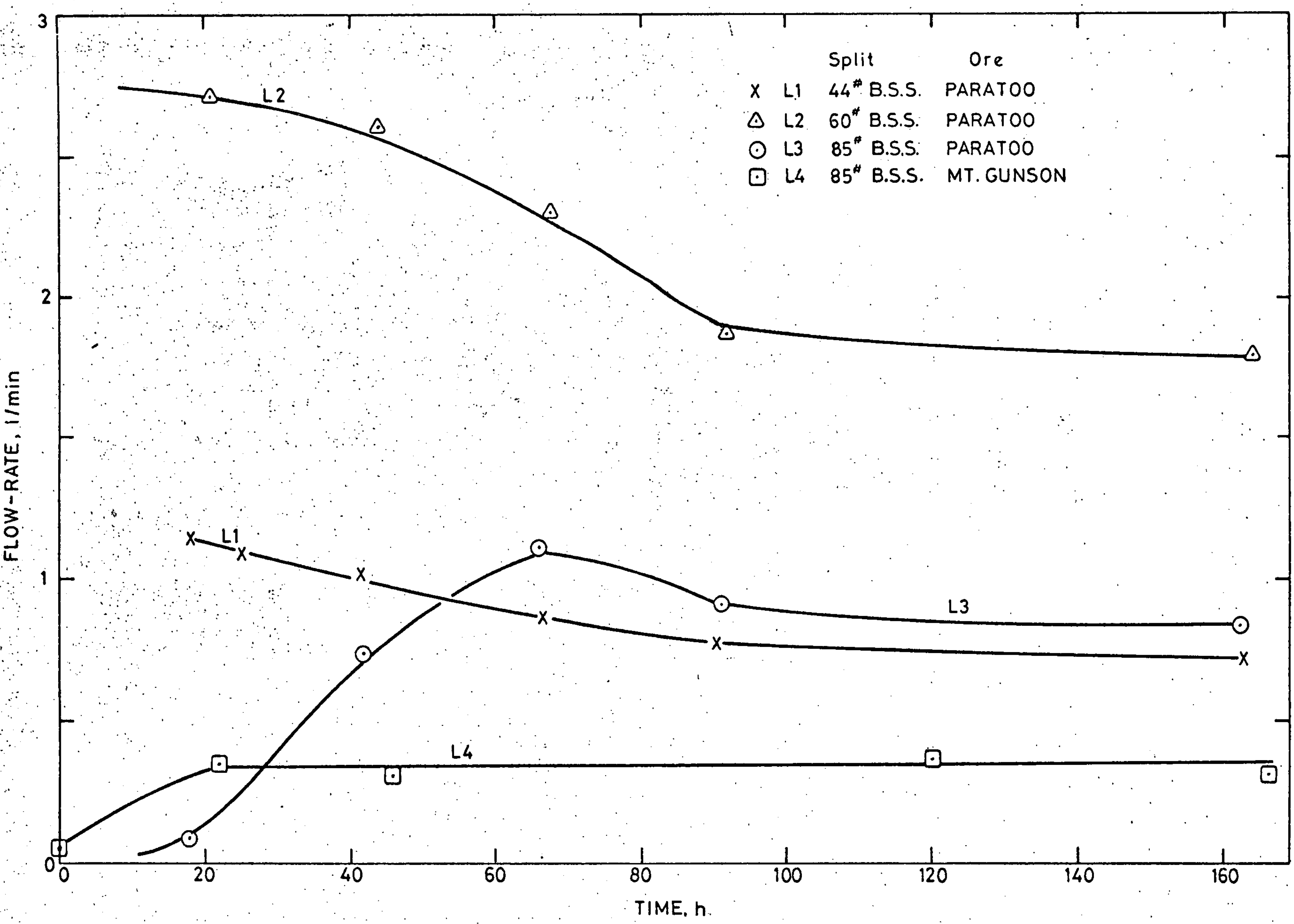


FIG.7: PERCOLATION LEACH: LIQUOR CIRCULATION RATE WITH TIME

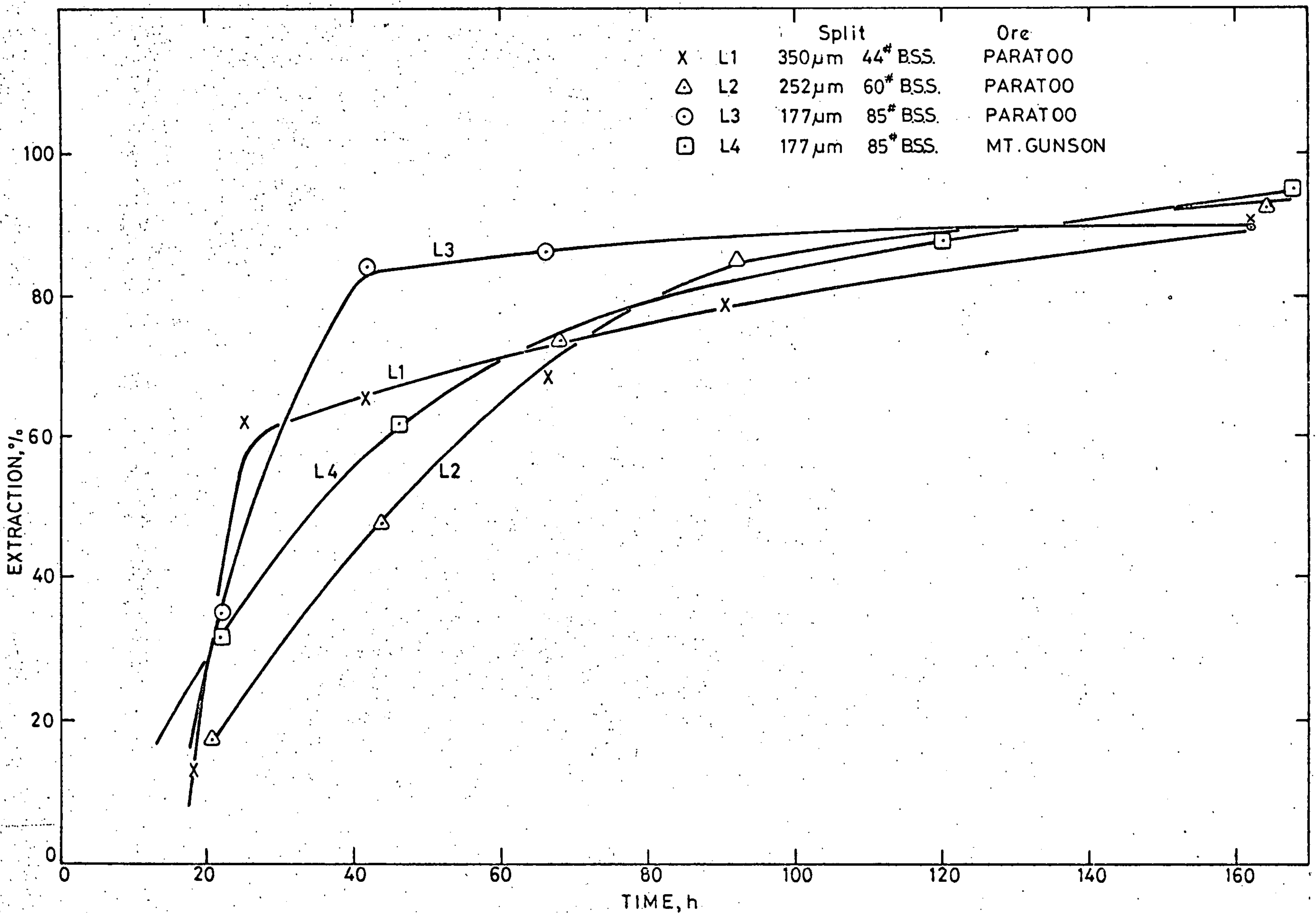


FIG. 8: PERCOLATION LEACH: COPPER EXTRACTION WITH TIME

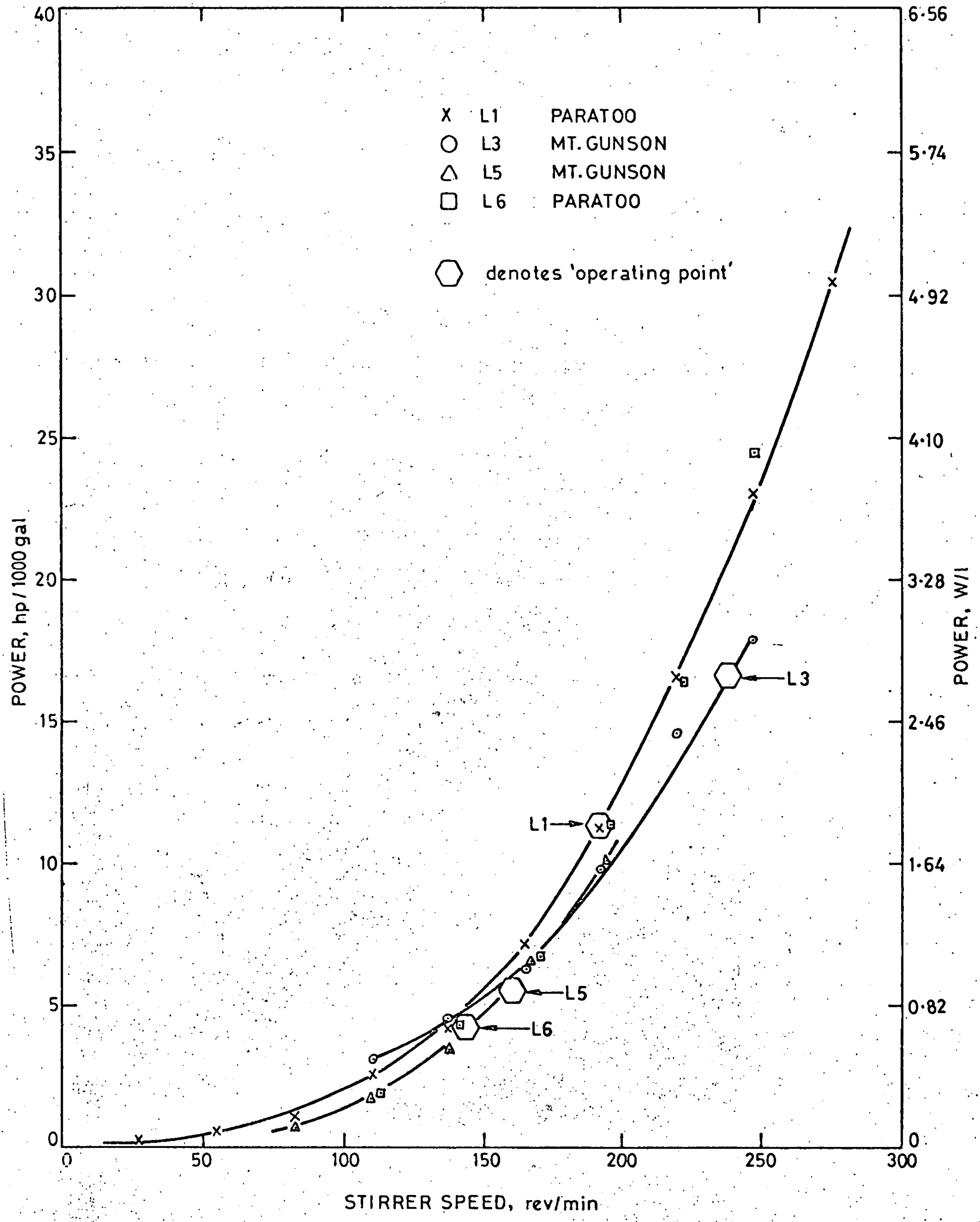


FIG. 9: POWER v STIRRER SPEED

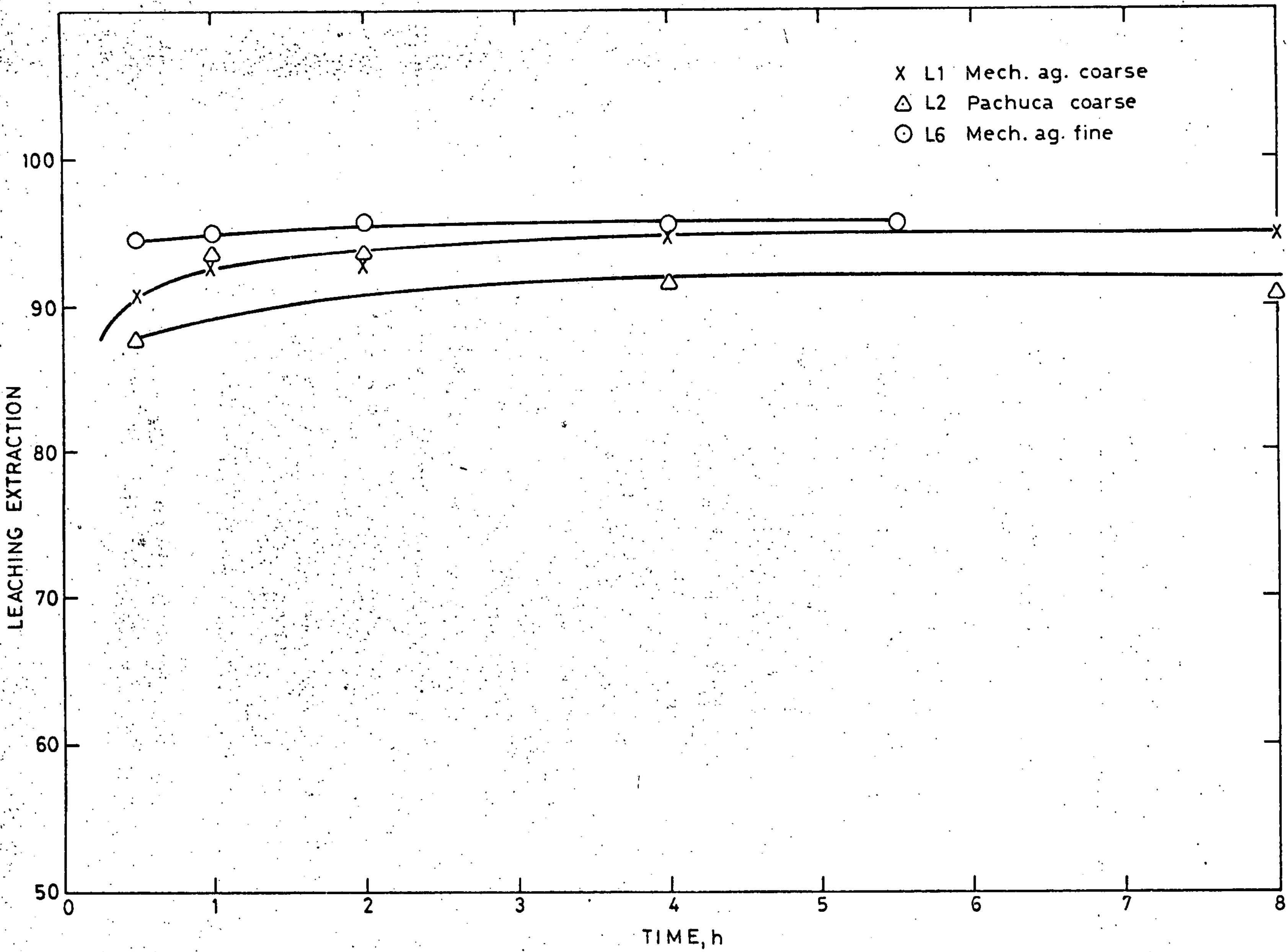


FIG.10: LEACHING OF PARATOO ORE BY MECHANICAL AGITATION AND PACHUCA

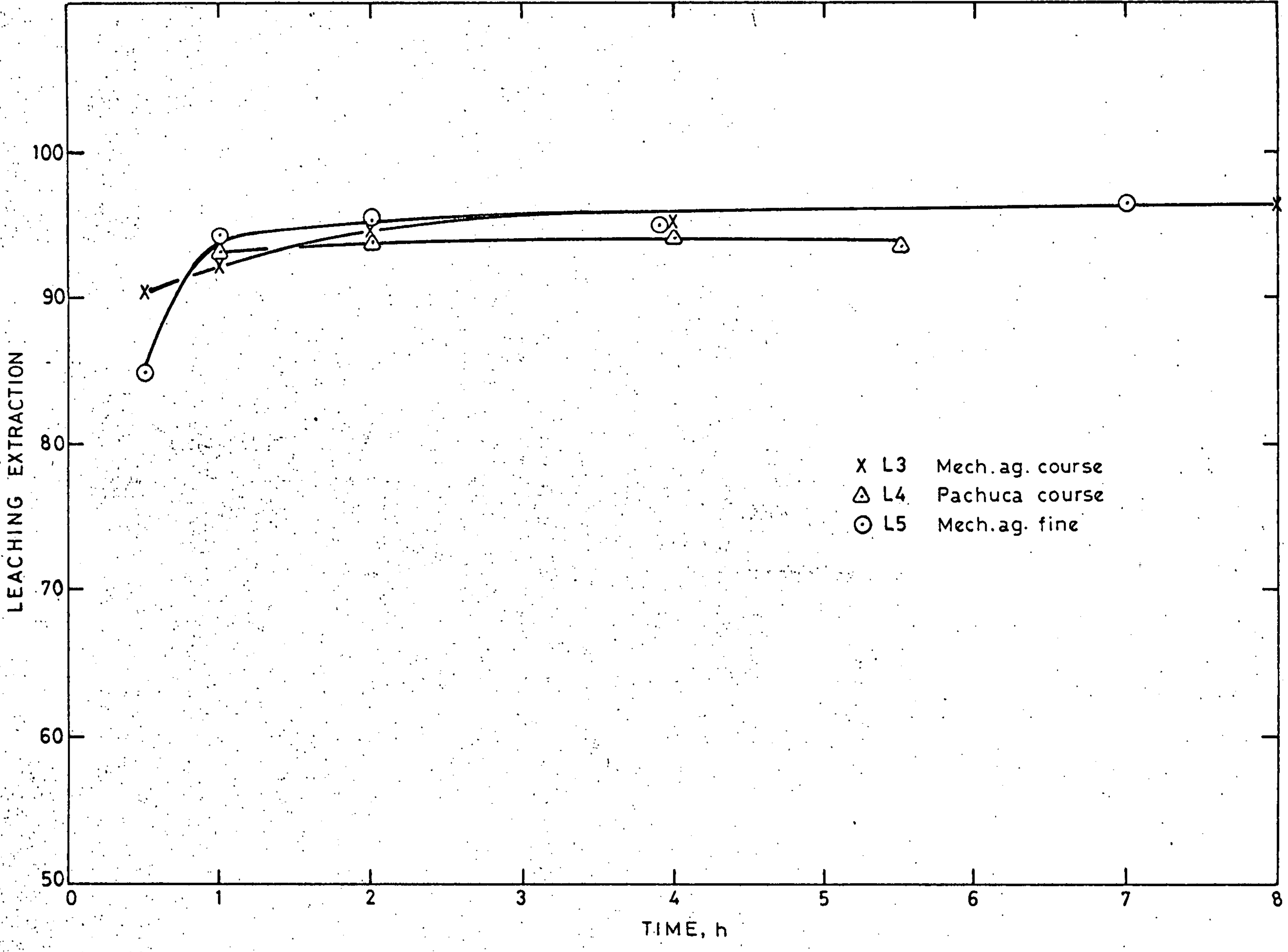


FIG. 11: LEACHING OF MT. GUNSON ORE BY MECHANICAL AGITATION AND PACHUCA

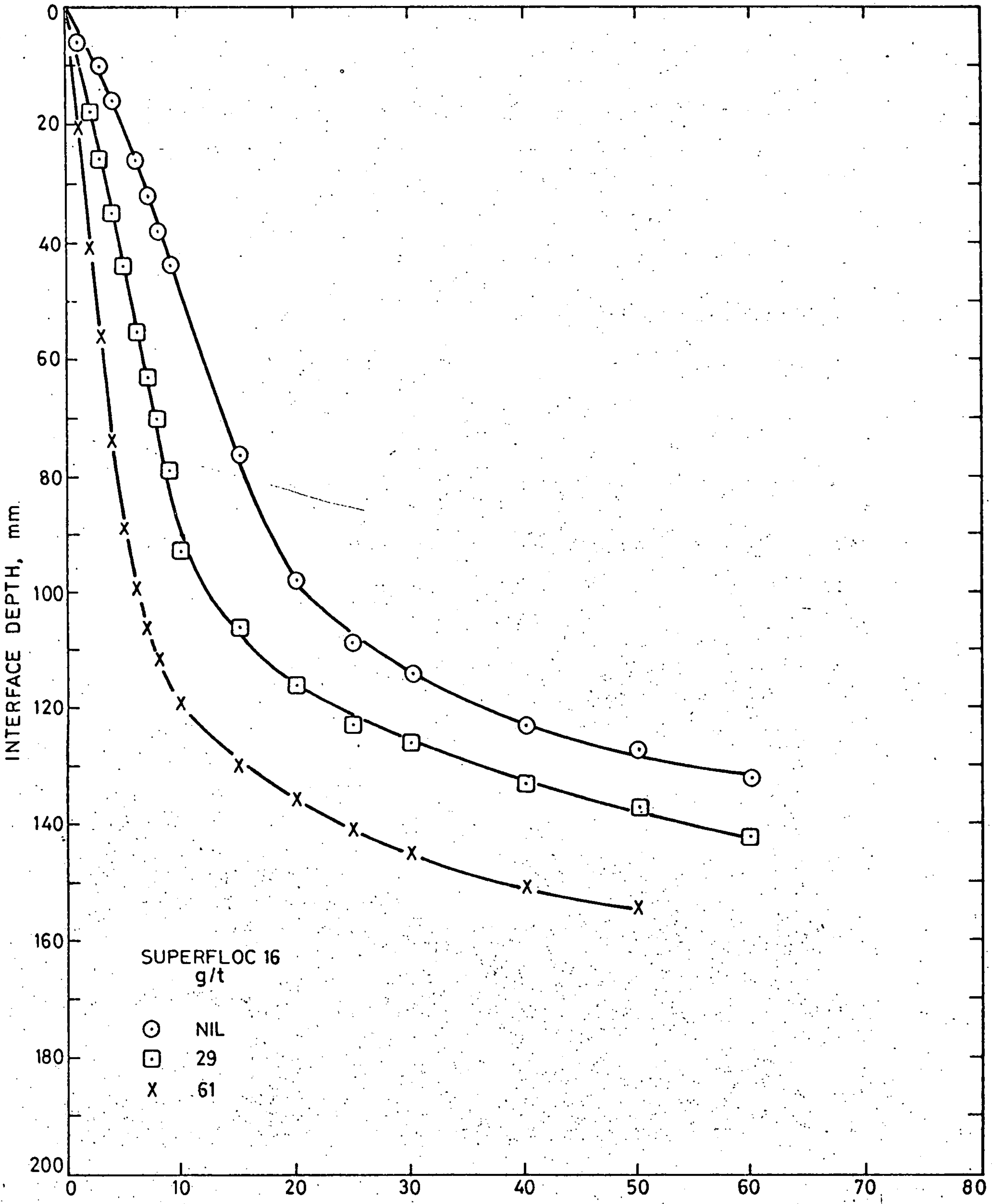


FIG. 12: SETTLING CURVES. L1: PARATOO LEACH No.1
2 HRS. MECHANICAL AGITATION

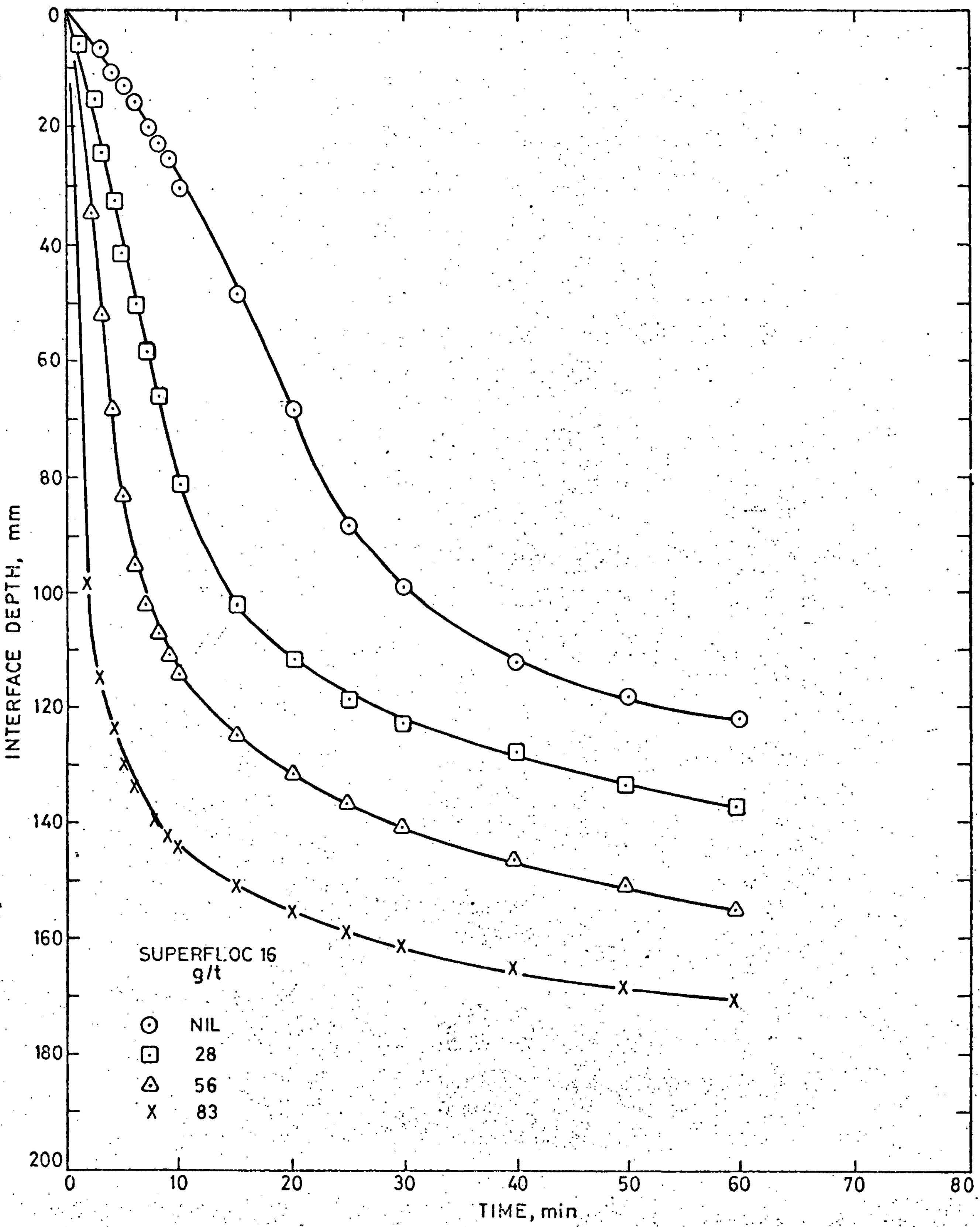


FIG. 13: SETTLING CURVES. L1 : PARATOO LEACH No.1
4 HRS MECHANICAL AGITATION

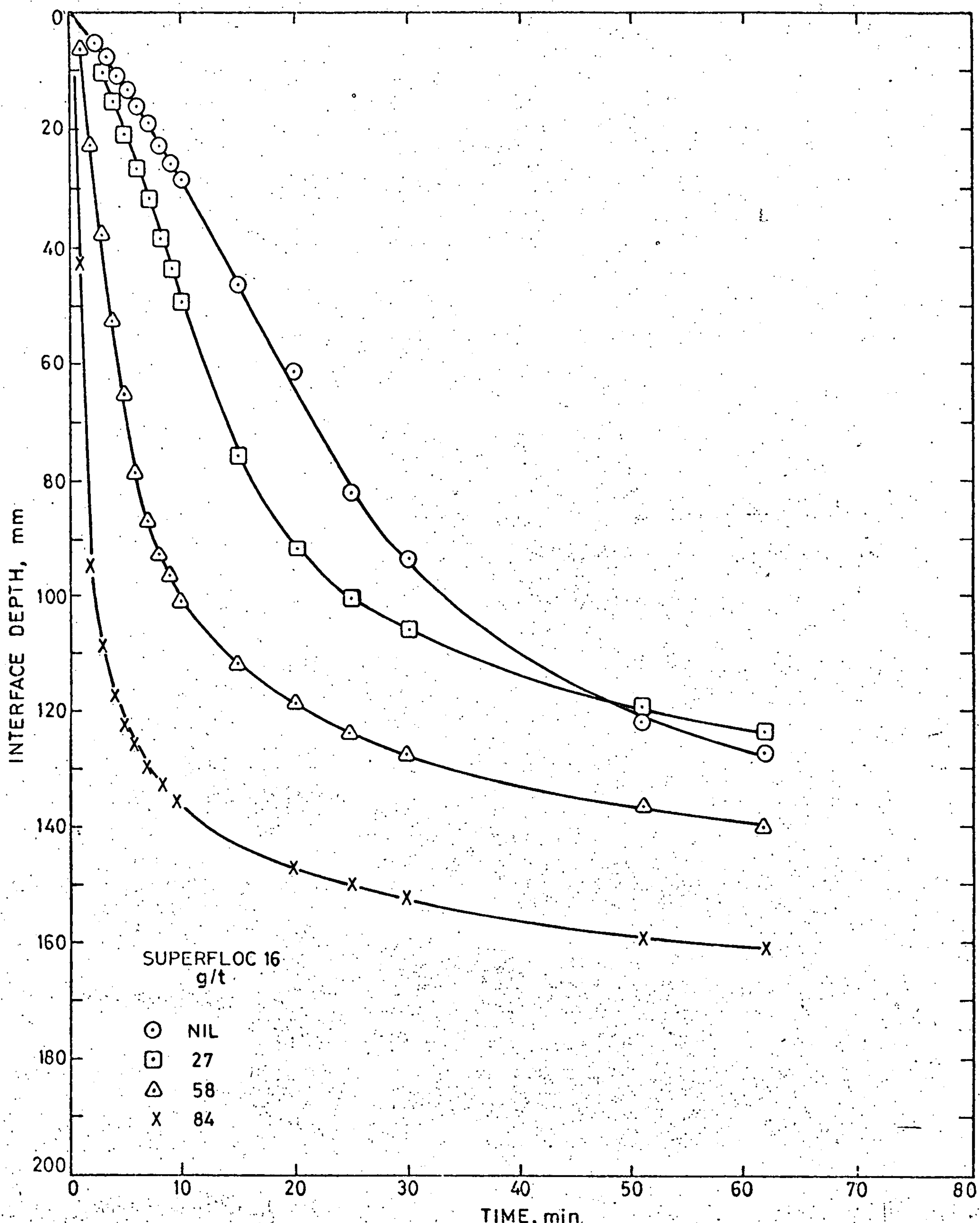


FIG.14: SETTLING CURVES. L1: PARATOO LEACH No.1
8 HRS MECHANICAL AGITATION

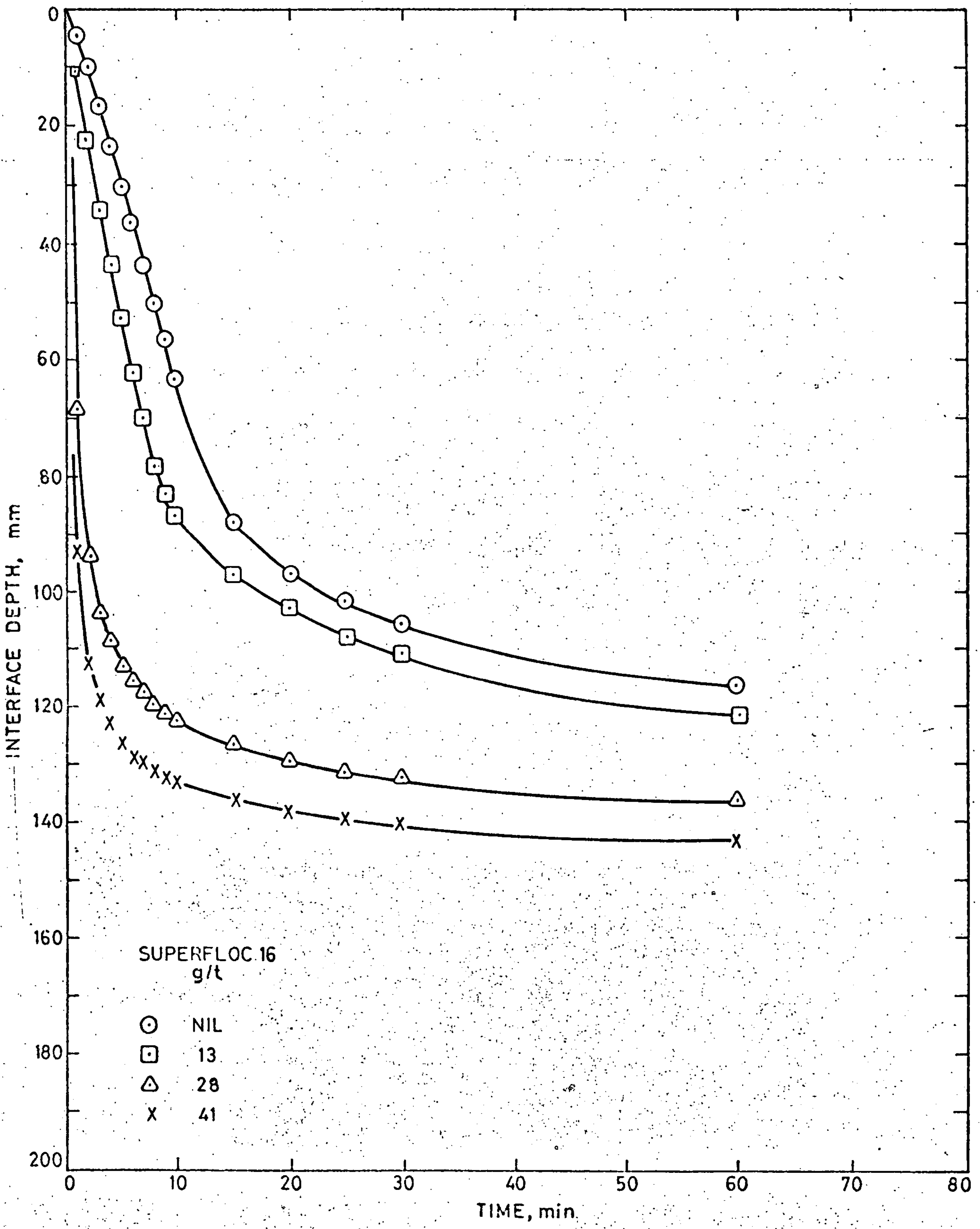


FIG. 15: SETTLING CURVES. L2: PARATOO LEACH No. 2
2 HRS AIR AGITATION

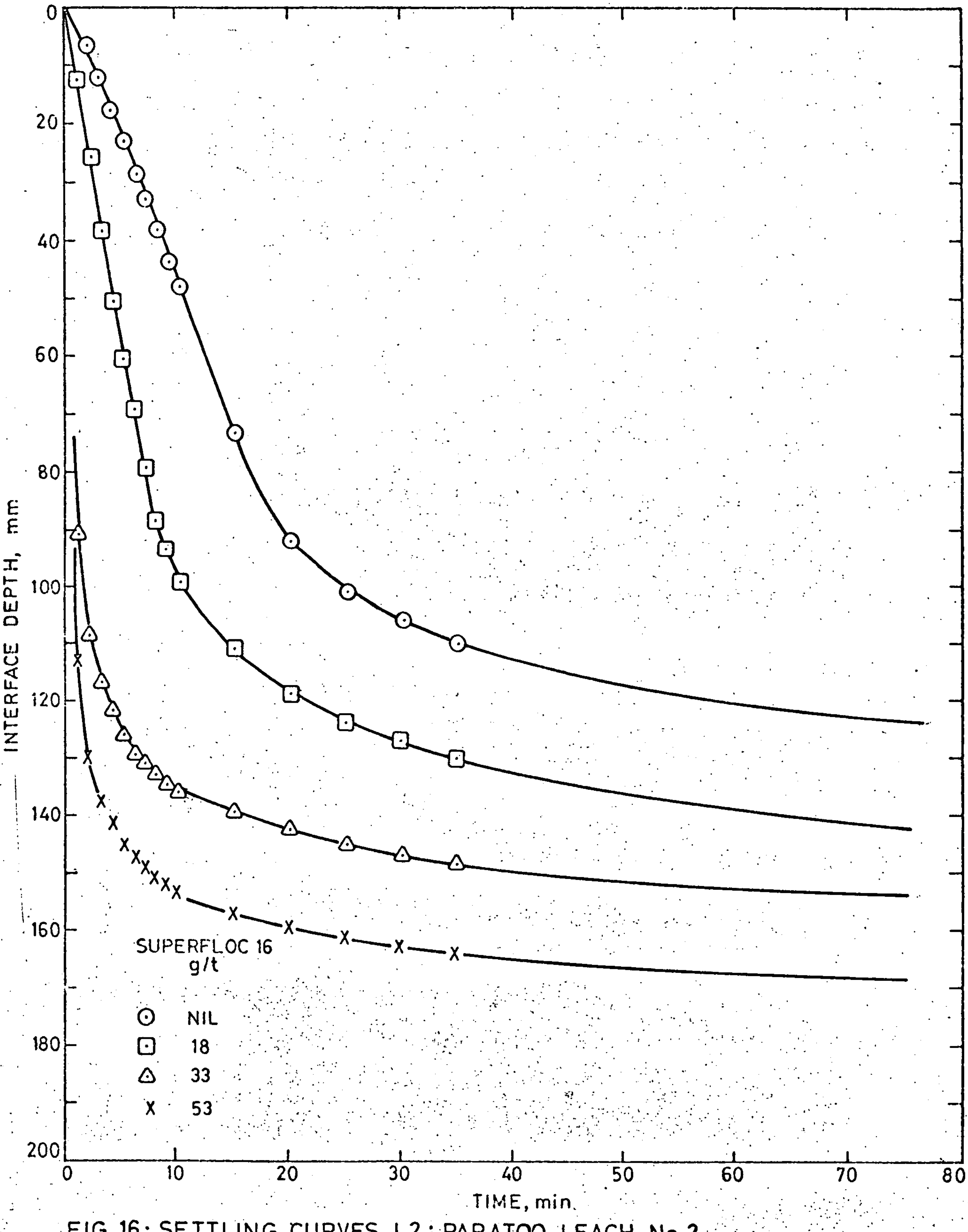


FIG. 16: SETTLING CURVES. L2: PARATOO LEACH No.2. 4 HRS AIR AGITATION

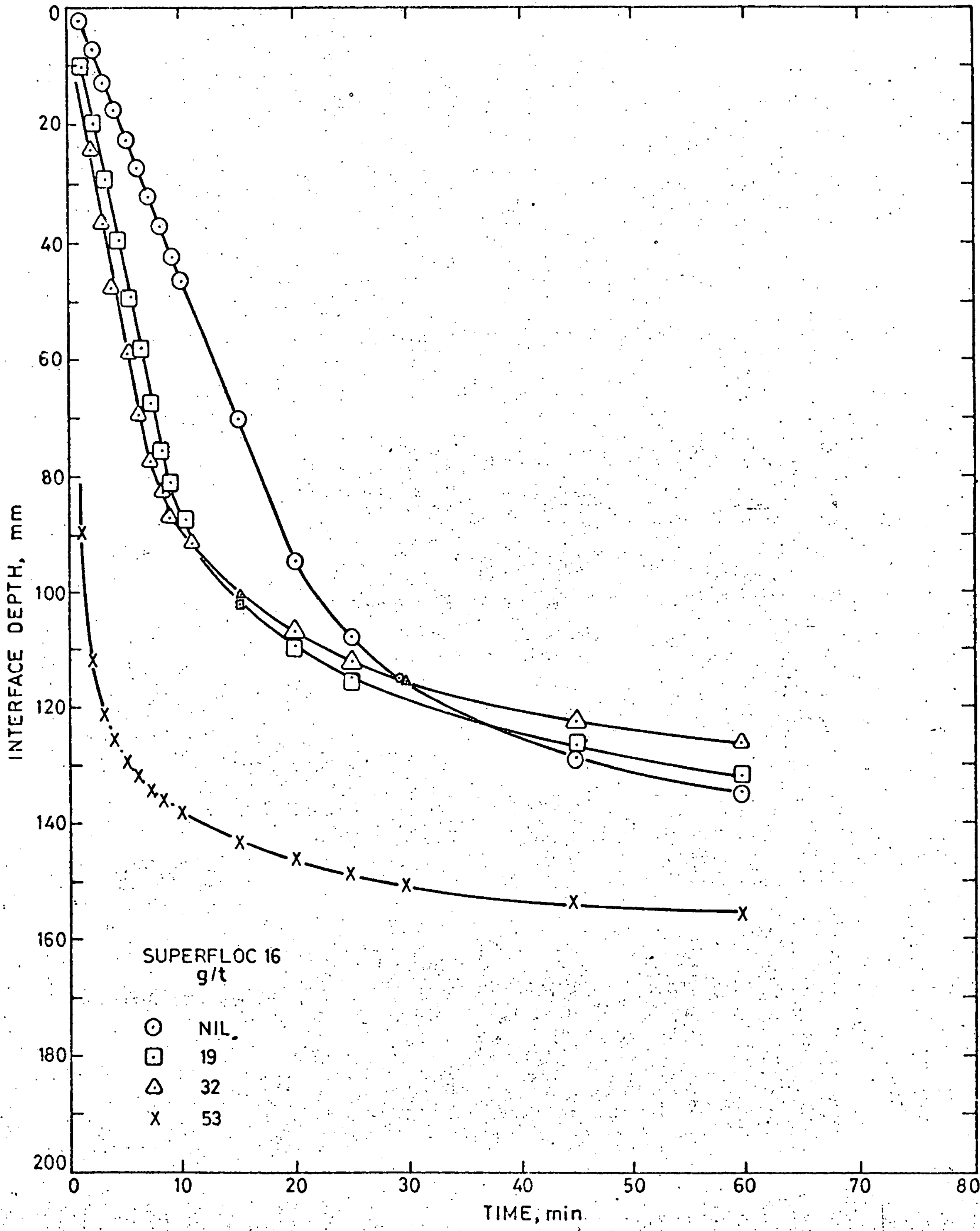


FIG.17: SETTLING CURVES. L2: PARATOO LEACH No.2
8 HRS AIR AGITATION

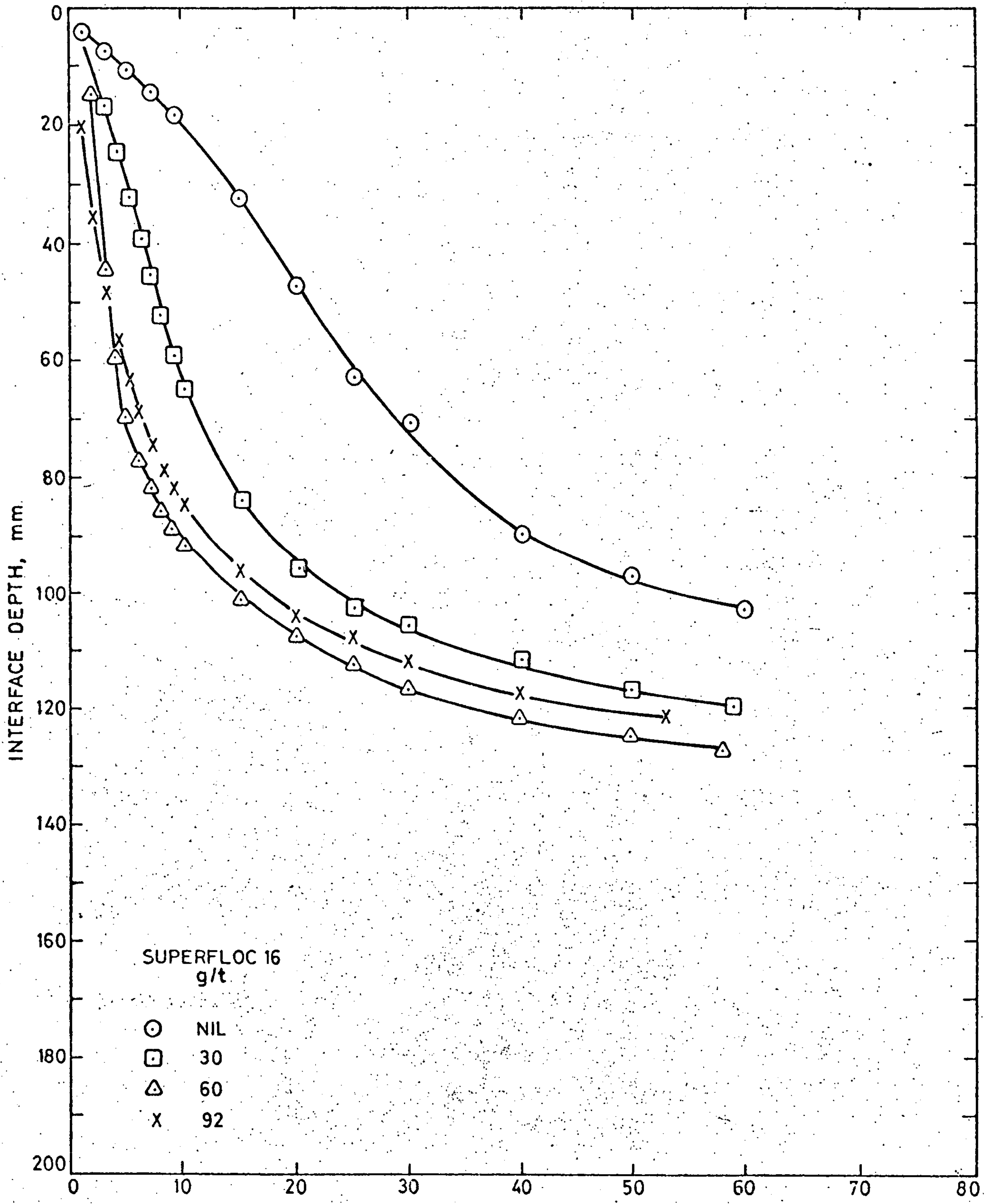


FIG.18: SETTLING CURVES. L6: PARATOO LEACH No.3
2 HRS. MECHANICAL AGITATION

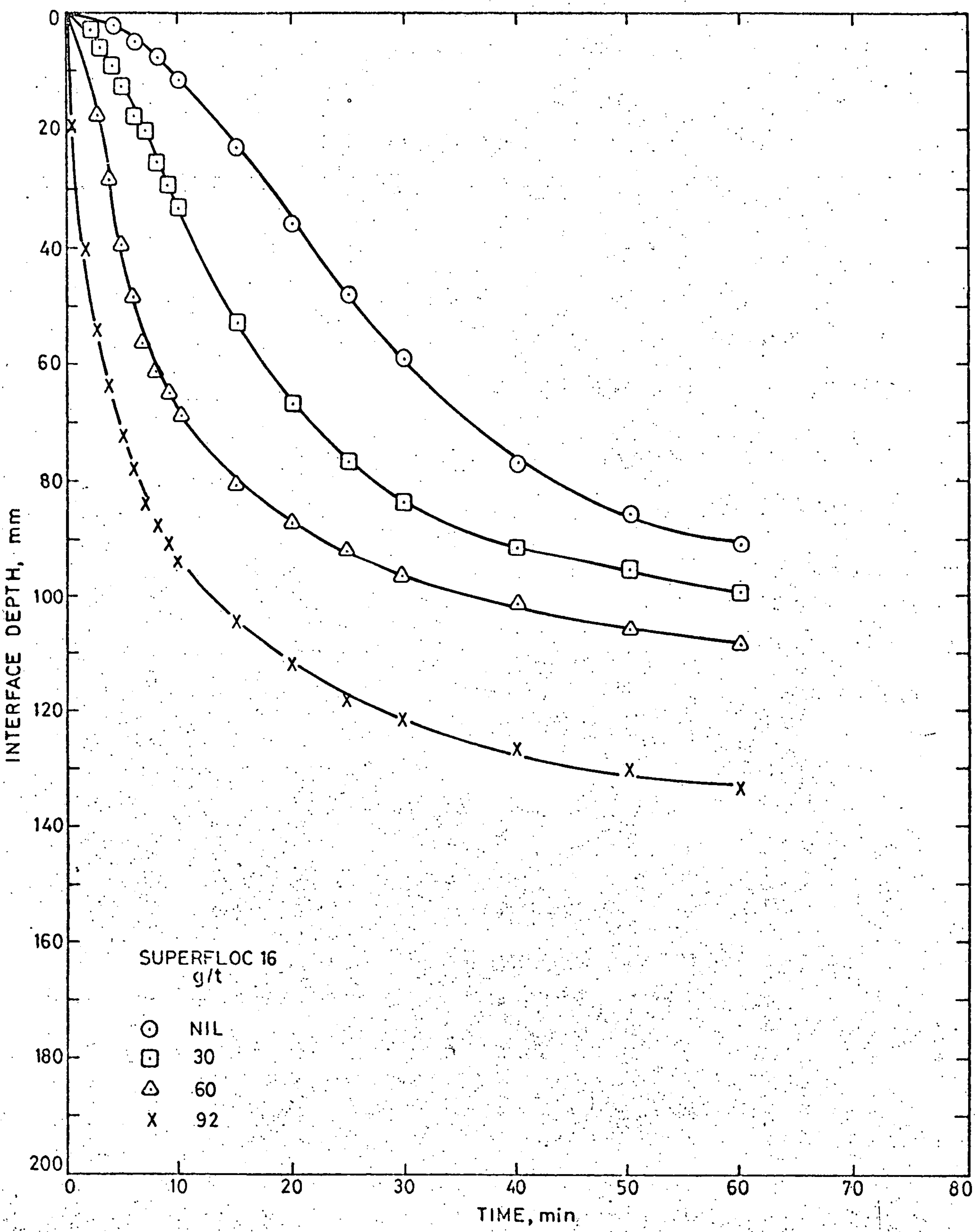


FIG.19: SETTLING CURVES. L6: PARATOO LEACH No.3
4 HRS MECHANICAL AGITATION

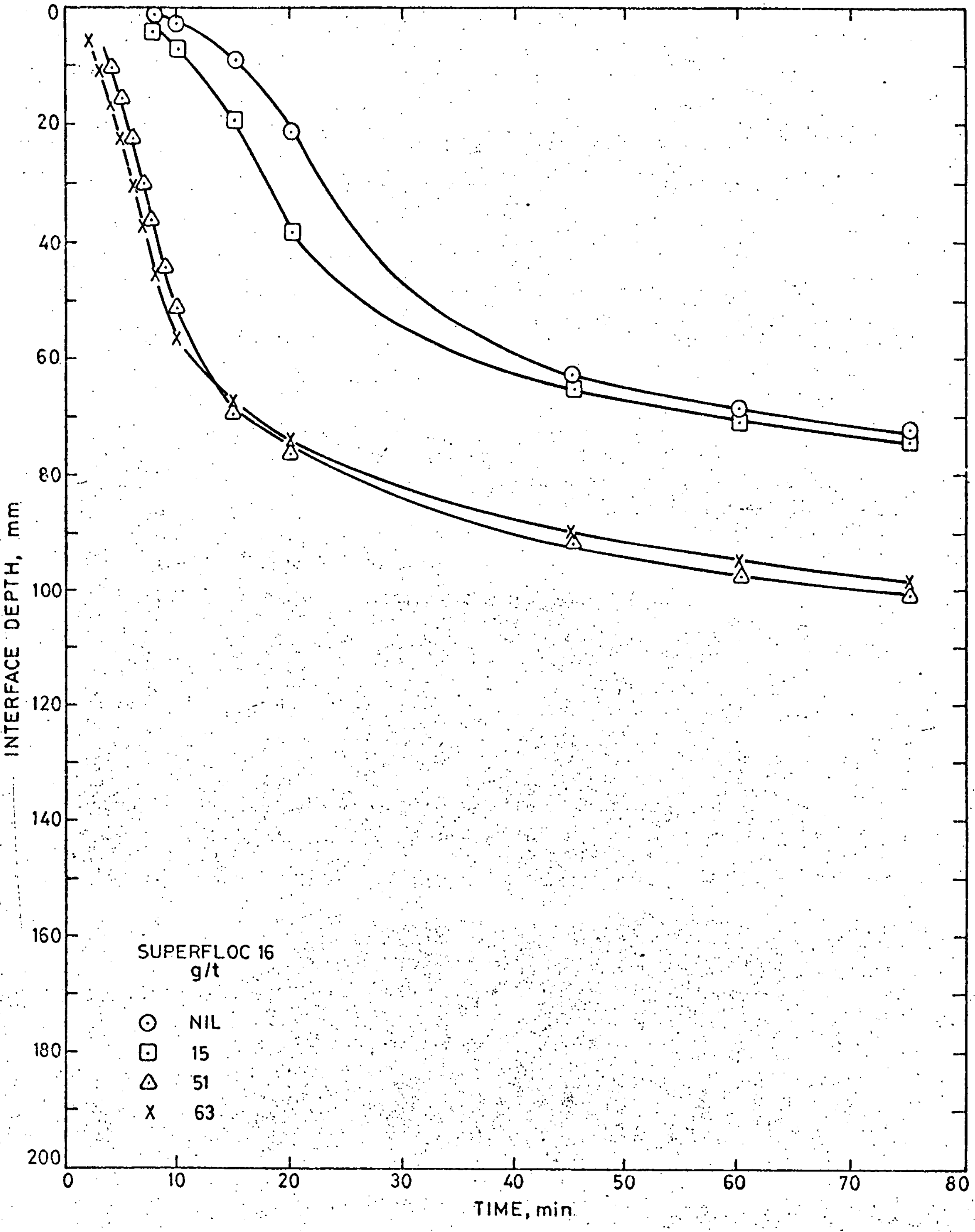


FIG. 20: SETTLING CURVES. L3: MT. GUNSON LEACH No.1
2HRS MECHANICAL AGITATION

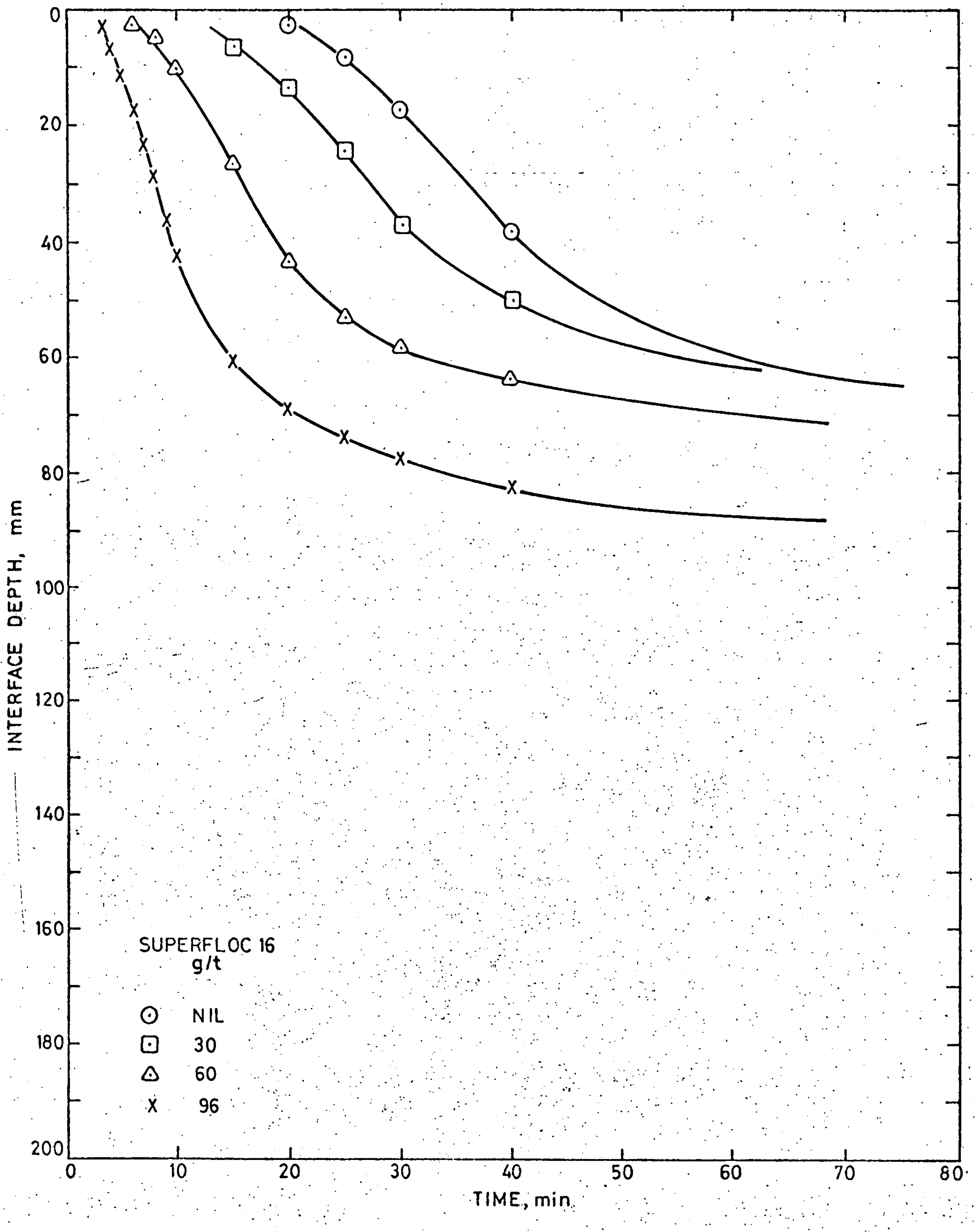


FIG. 21: SETTLING CURVES. L3: MT GUNSON LEACH No.1 4 HRS MECHANICAL AGITATION

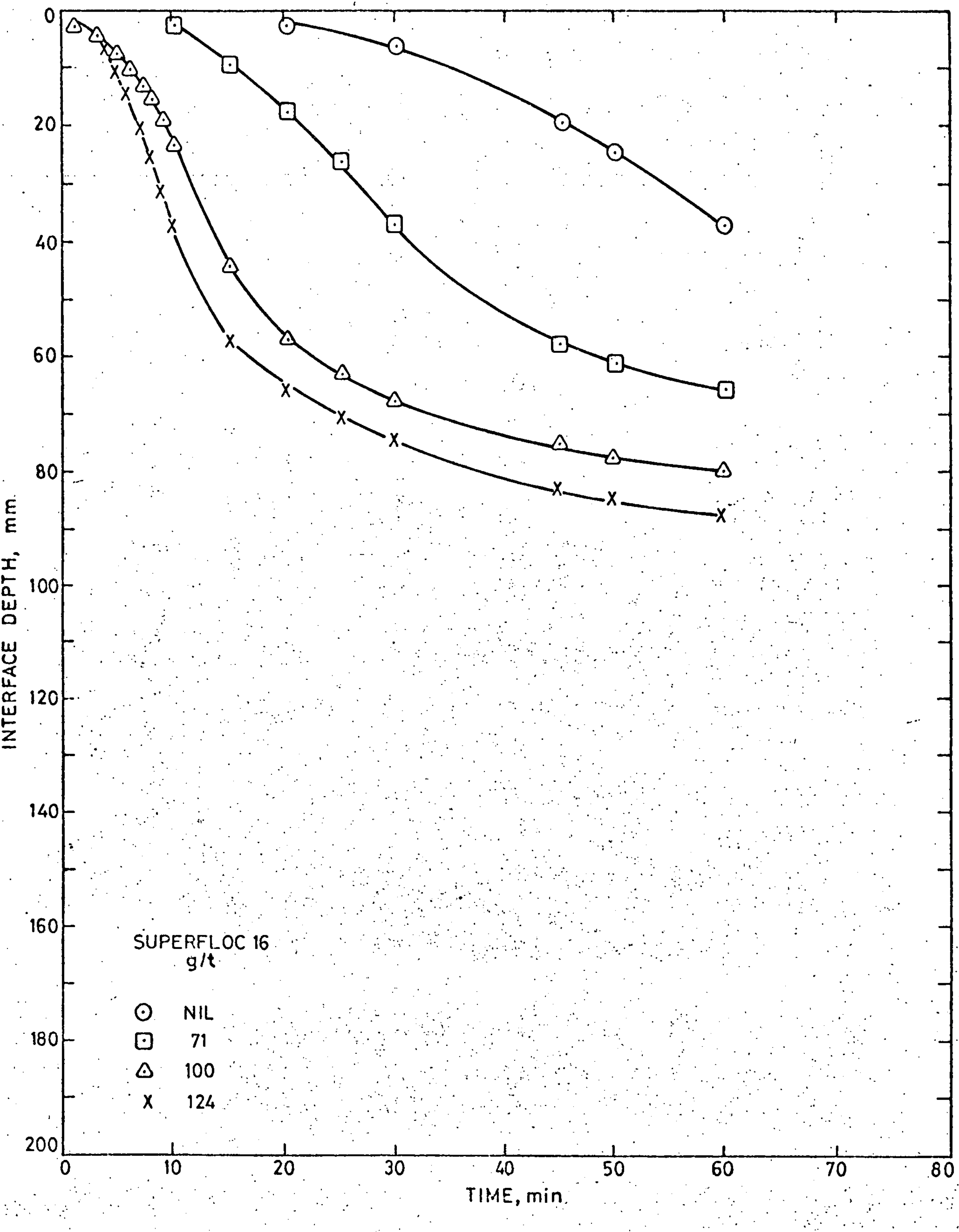


FIG. 22: SETTLING CURVES. L3: MT GUNSON LEACH No.1
8 HRS MECHANICAL AGITATION

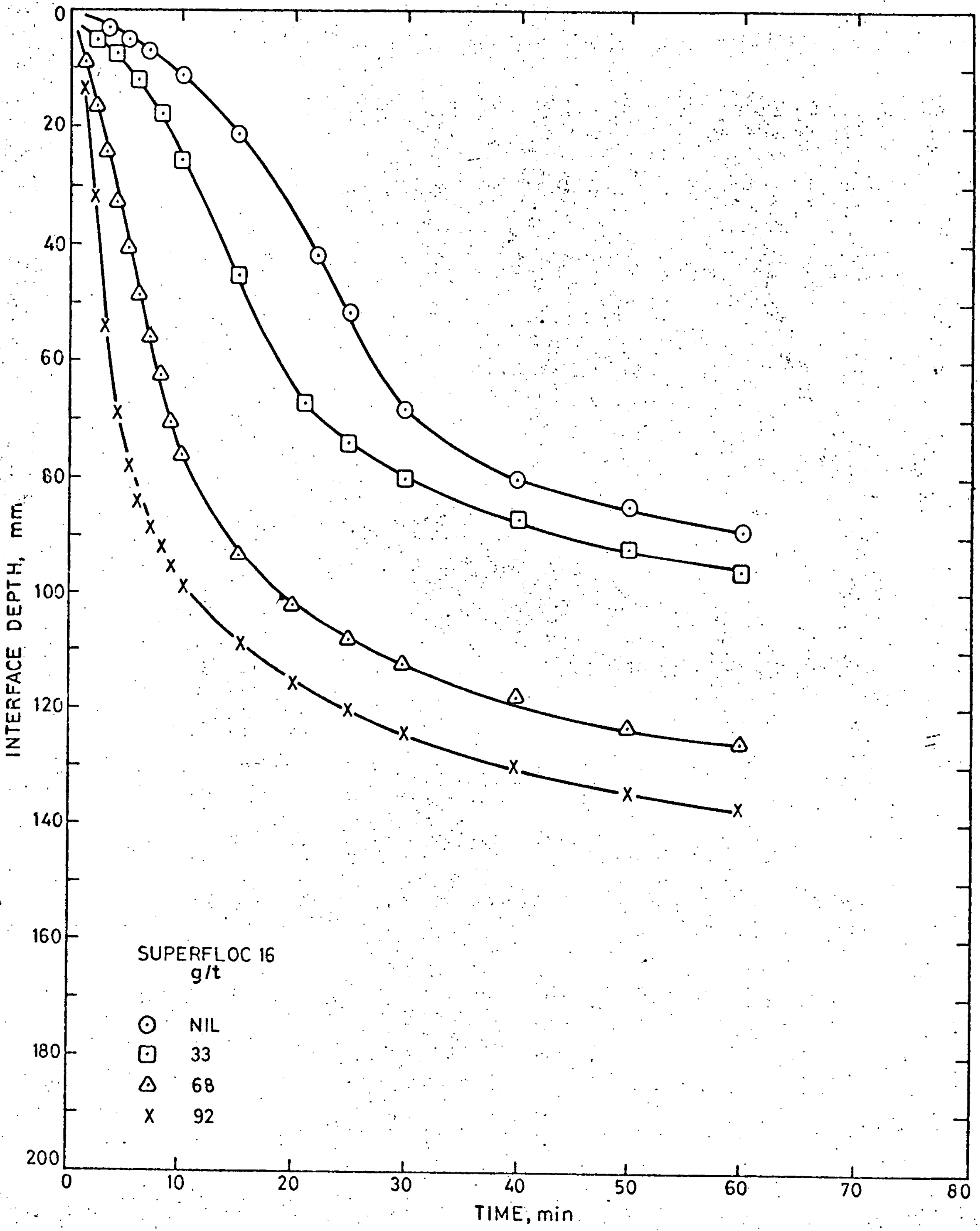


FIG. 23: SETTLING CURVIS. L5: MT. GUNSON LEACH No.3
2 HRS. MECHANICAL AGITATION

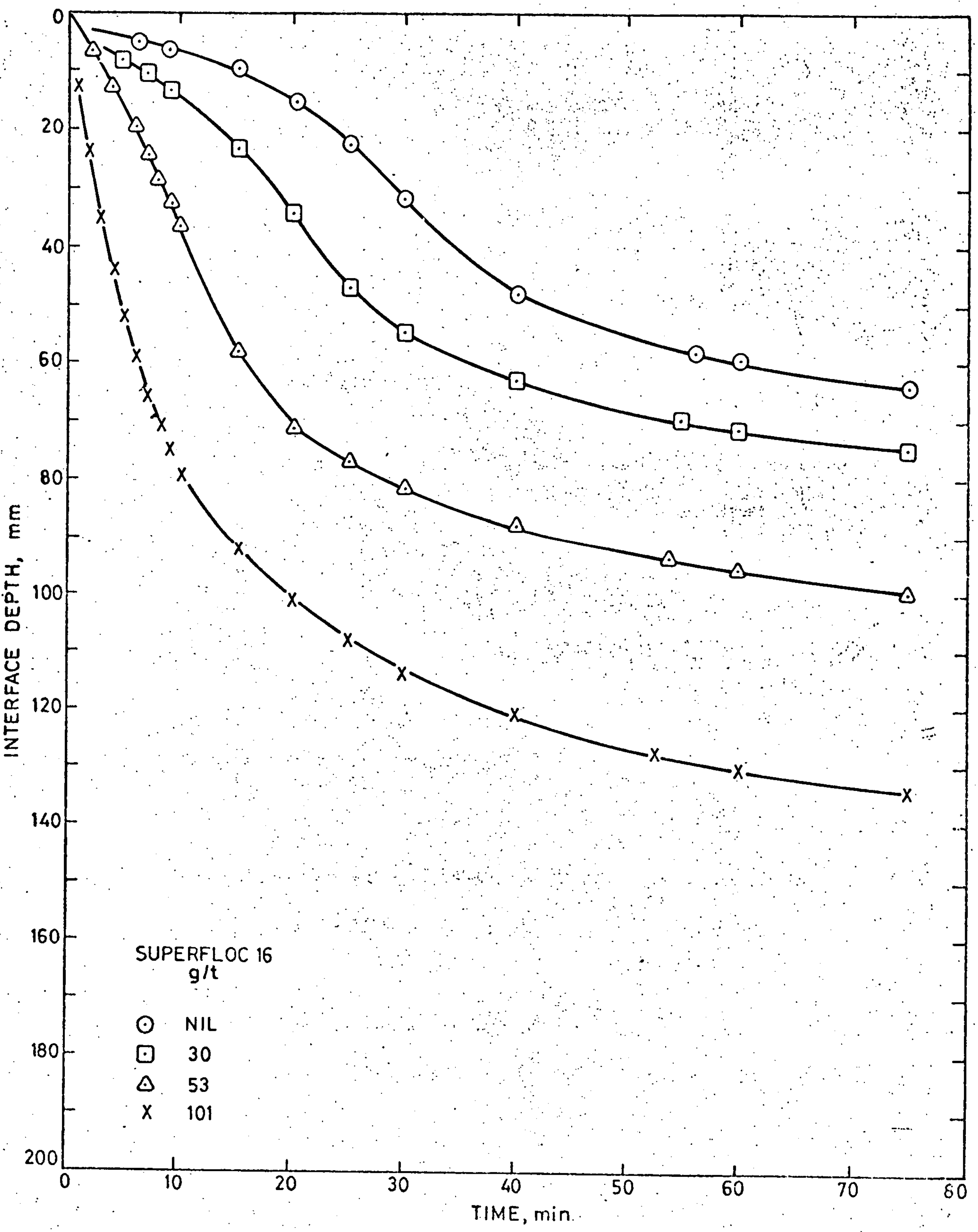


FIG.24: SETTLING CURVES. L5: MT. GUNSON LEACH No.3
4 HRS. MECHANICAL AGITATION

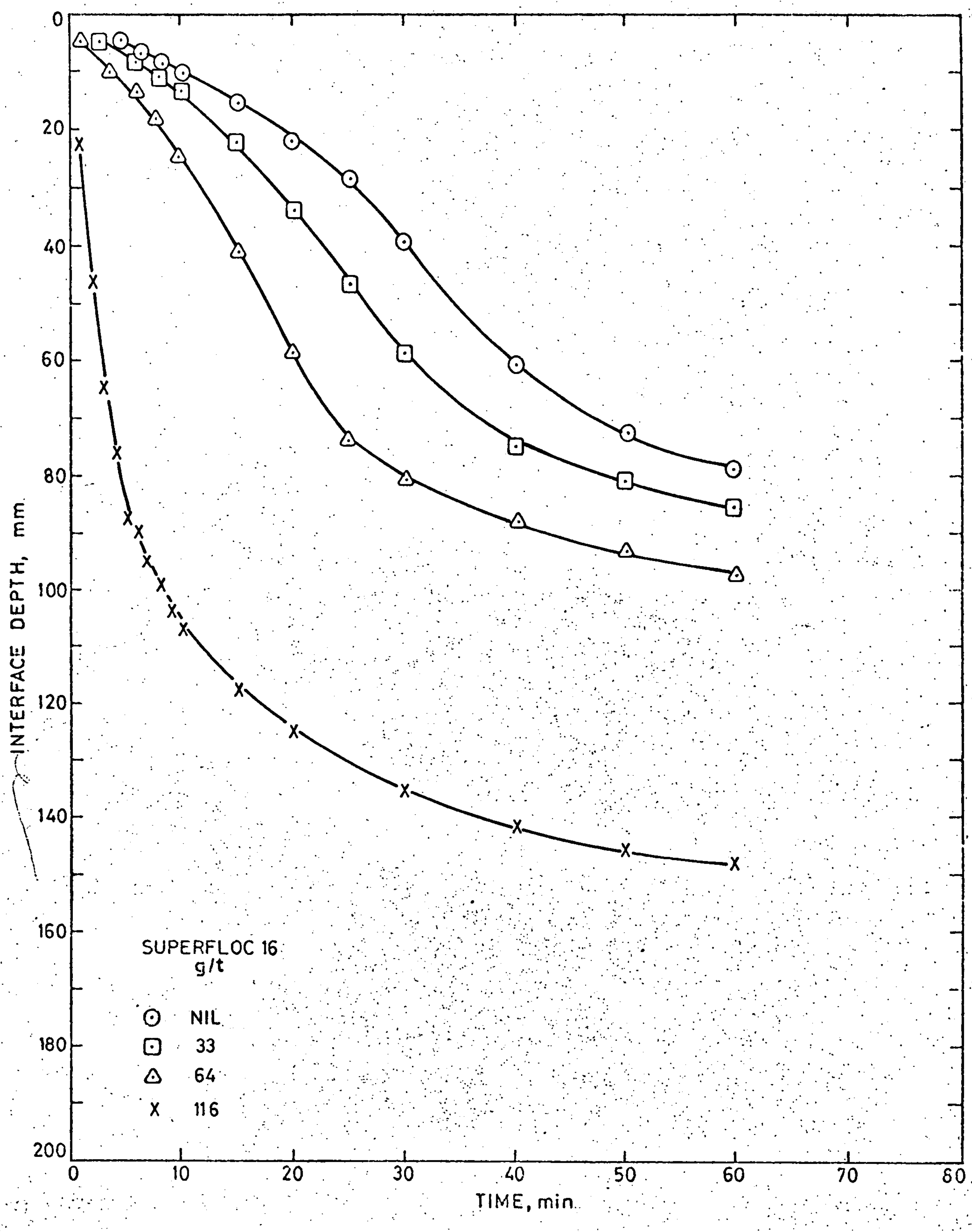


FIG. 25: SETTLING CURVES. L5: MT GUNSON LEACH No.3
8 HRS. MECHANICAL AGITATION

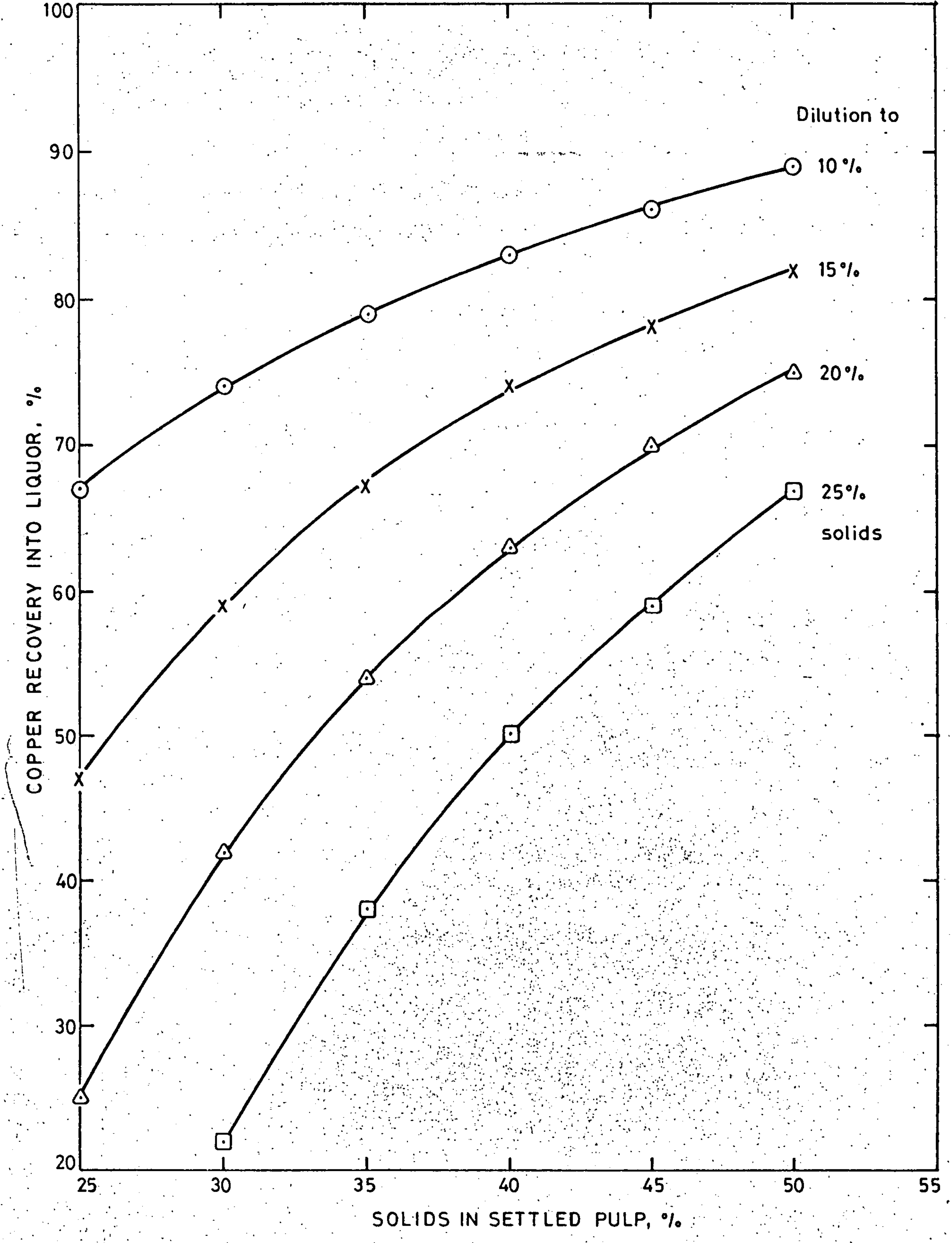


FIG. 26 : RECOVERY OF DISSOLVED COPPER FROM SOLIDS AFTER DILUTION AND DECANTATION FROM SETTLED SOLIDS

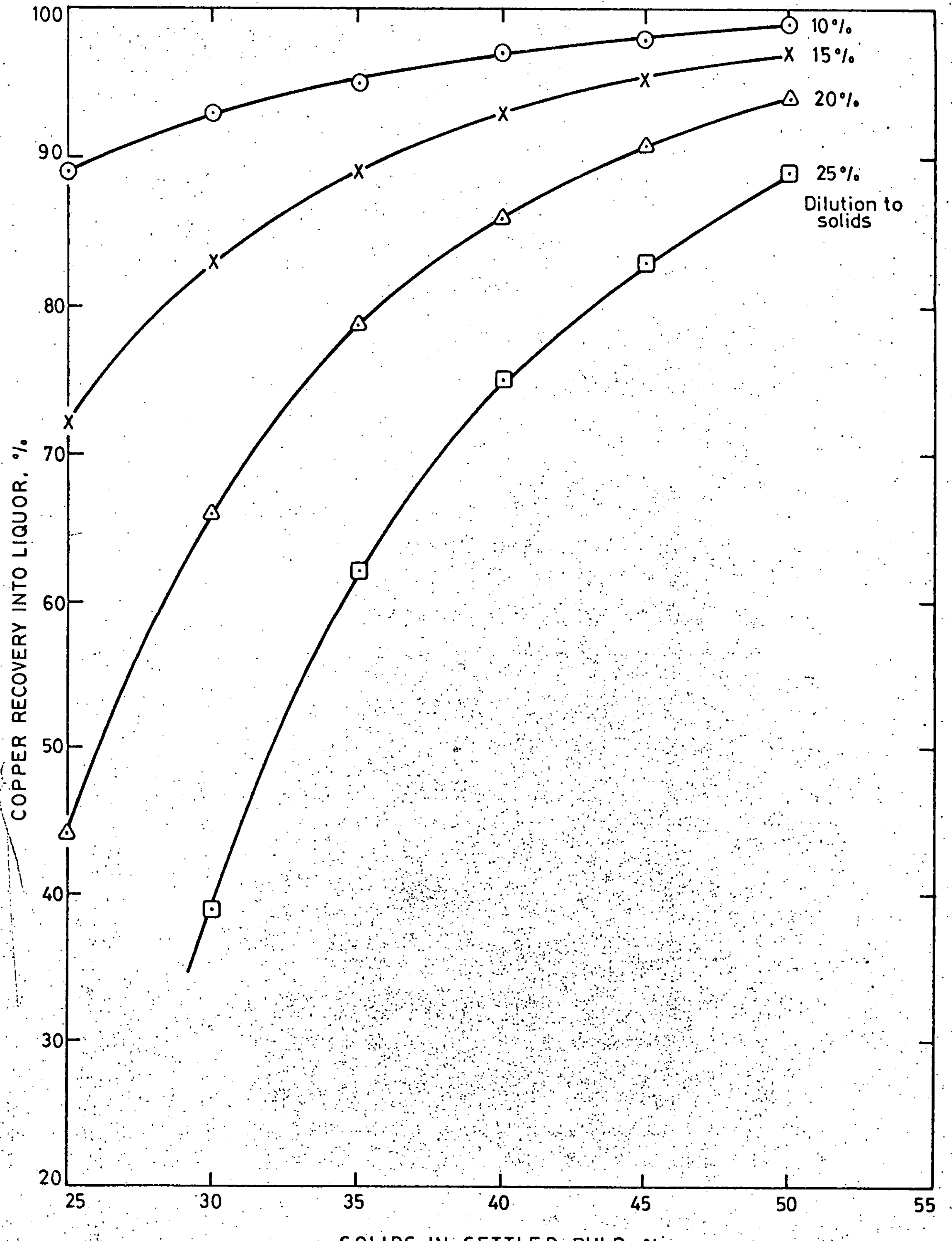


FIG. 27: TOTAL RECOVERY OF DISOLVED COPPER FROM SOLIDS AFTER INITIAL DILUTION, DECANTATION AND ONE WASH

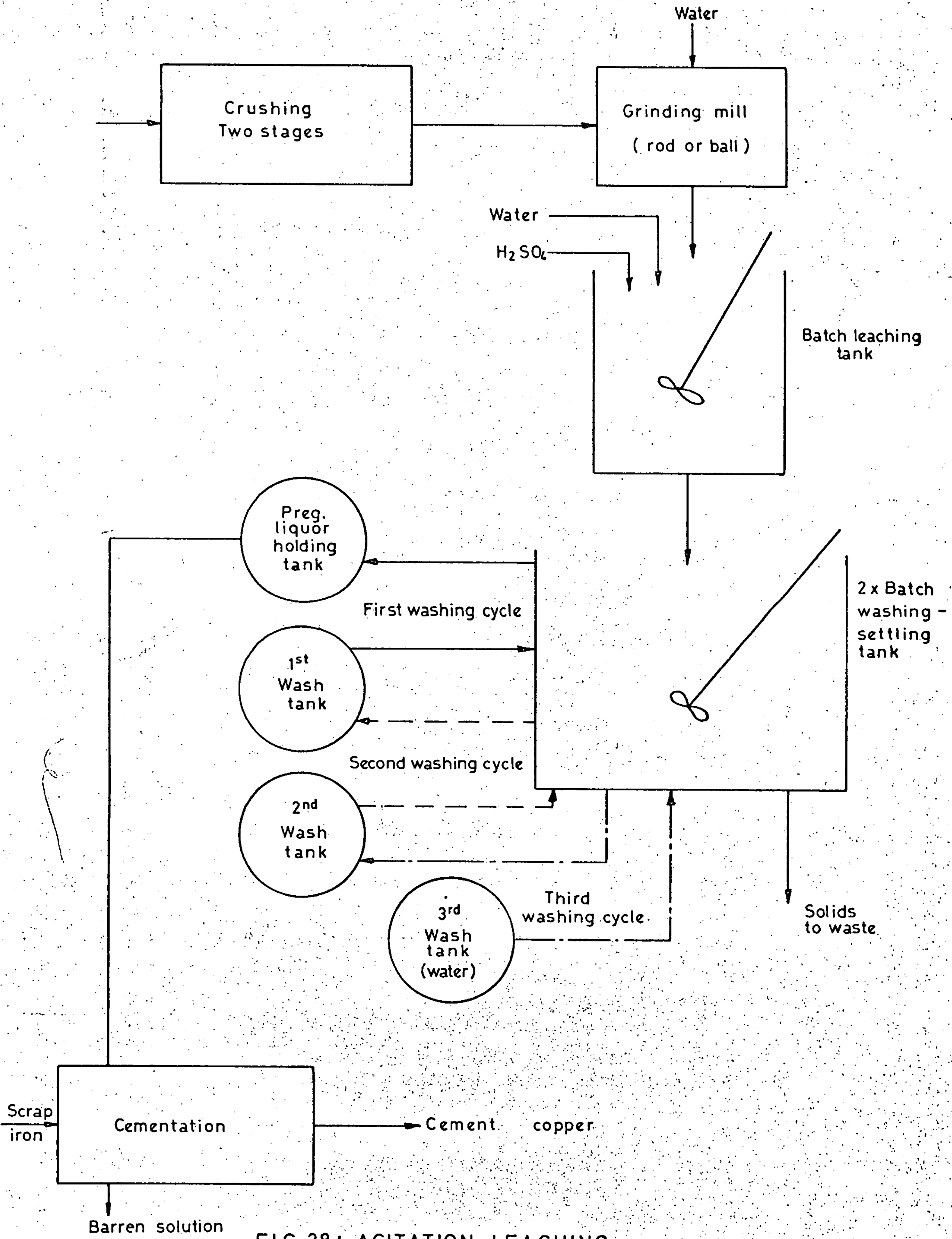


FIG.28: AGITATION LEACHING

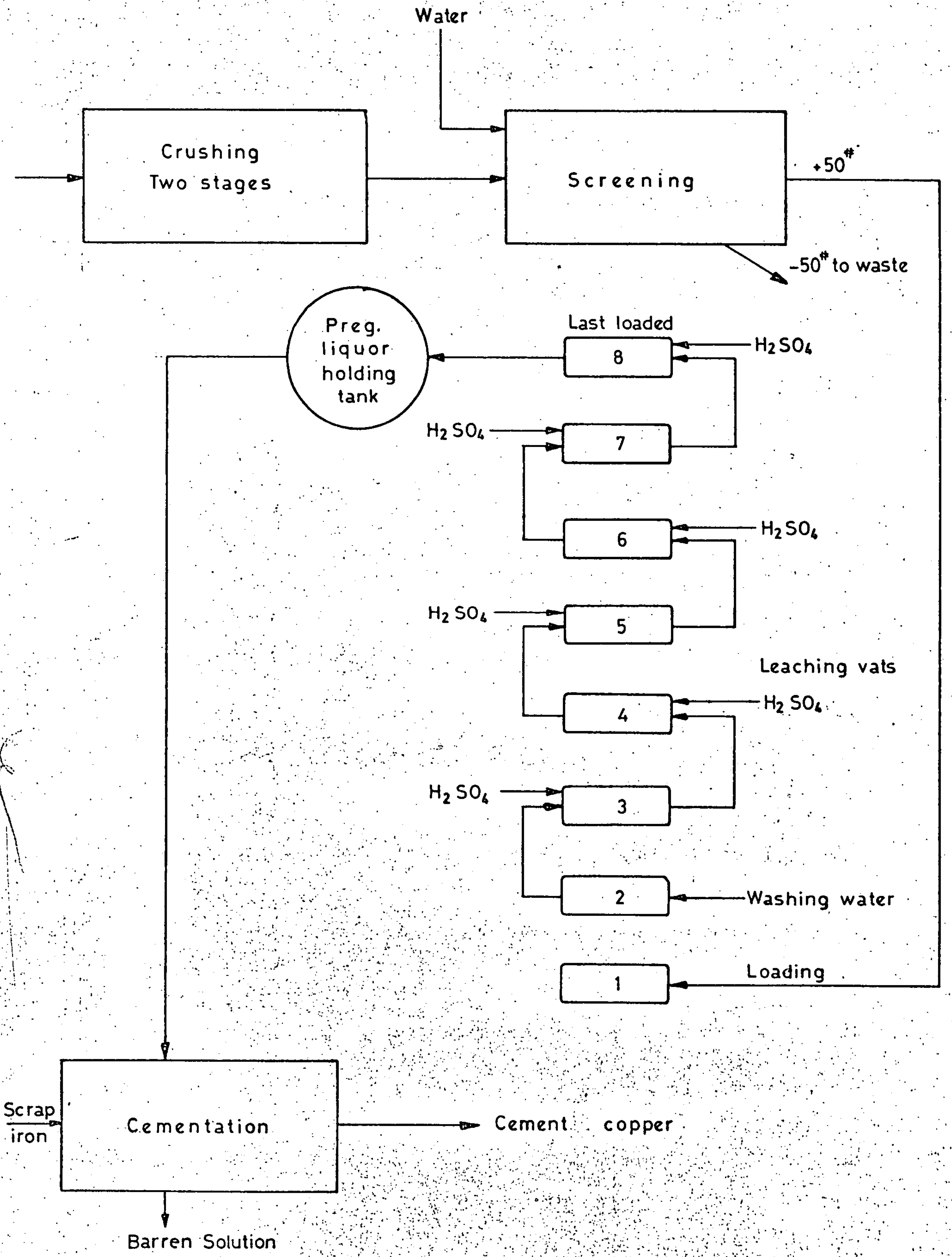


FIG. 29: VAT LEACHING