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JERVOIS SULPHATES (N.T.) LTD - HEAP LEACHING INVESTIGATION

(Amdel Report No. 469)

by

D.F. Hall J.D. Hayton and B.H.J. Waters

The information contained in this report has been obtained by the Department of National Development, as part of the policy of the Commonwealth Government, to assist in the exploration and development of mineral resources. It may not be published in any form or used in a company prospectus without the permission in writing of the Director, Bureau of Mineral Resources, Geology and Geophysics.

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INTRODUCTION

Jervois Sulphates (N.T.) Ltd., was formed to produce copper sulphate for the domestic market by reaction of sulphuric acid with oxidised copper ore from the Jervois Range deposits followed by solar evaporation of the solution to produce copper sulphate crystals.

Initial attempts to bring the operation into commercial production using a ball mill grinding and counter current decantation system proved unsuccessful, and a change of management took place. After a review of the enterprise by the Bureau of Mineral Resources it was decided that the Australian Mineral Development Laboratories should investigate on alternative proposal based on heap leaching of the copper ore which the new management of the company believed could be used to bring about a commercially successful operation.

This report contains the results of these investigations and has been included in the Bureau's Record series so that it may enjoy the wider distribution offered by the "Open File" system. It has not been altered in any way.

2/1/11 February, 1966

BUREAU OF MINERAL RESOURCES

Amdel Report

No. 469

JERVOIS SULPHATES (NT) LTD
Heap Leaching Investigations

by

D. F. Hall, J. D. Hayton and B. H. J. Waters

Investigated by: Industrial Chemistry Section
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THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES
Adelaide South Australia

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SUMMARY

A request was received from the Bureau of Mineral Resources to assess the feasibility of a percolation leaching process for use by Jervois Sulphates (NT) Ltd. Mr J.D. Hayton discussed with officers of the Bureau of Mineral Resources, CSIRO and the Company, previous attempts to leach Jervois copper ore and later made a short visit to the Jervois mine.

An experimental programme was then drawn up and ten pilot-plant-scale leaching tests were carried out in the laboratories of Amdel. The results of these tests suggested that with careful control percolation leaching of Jervois ore could be carried out and an extraction efficiency of about 80% with an acid consumption of about 2.5 lb sulphuric acid per pound of copper could be expected.

Three larger scale leaching tests were then carried out at Jervois. Difficulty was experienced in compensating for the high rates of evaporation at Jervois, and the lack of adequate analytical facilities. However, the pilot scale results were generally confirmed. Using the split leaching technique which pilot-scale tests had indicated to be most appropriate for the existing conditions, extractions of 70-75% of the available copper with an acid consumption of about 2.5 lb sulphuric acid per pound of copper were obtained.

1. INTRODUCTION

The Bureau of Mineral Resources requested assistance from Amdel in assessing the efficiency of a heap leaching process aimed at producing copper sulphate pentahydrate ($CuSO_4.5H_2O$) at Jervois Sulphates (NT) Limited. Mr J.D. Hayton of Amdel discussed the problem with Mr G. Mead head of the Mining Engineering Branch of the BMR; Mr I. Sykes, a director of the Company and Messrs K. Blaskett and J. Woodcock of the CSIRO before visiting the Plant at Jervois.

A summary of the Company's operations prepared by Mr Mead indicated that the original intention was to leach finely ground ore and to separate the liquor in a counter current decantation system before evaporation. This process did not operate successfully, mainly because of failure of the CCD system. Various technical consultants reported that considerable expenditure would be involved in correcting the faults; therefore, because of the poor state of the Company's finances, some form of heap leaching was considered as an alternative. Some heap leaching had been done at Jervois since mid 1964, but the efficiency could not be assessed because no accurate measurement had been made.

Following these discussions a visit was made to the Jervois Treatment Plant by Mr Hayton, accompanied by Mr K.G. Johannsen of Alice Springs who is also a director of the Company. Further investigational work was recommended as a result of this visit.

A series of small scale leaching tests was carried out at Amdel and later a further series of larger scale leaching tests was carried out at Jervois.

The impressions gained during the visits and the results of the subsequent experimental leaching tests are discussed in this report.

2. MATERIALS EXAMINED

2.1 Laboratory Investigations

Three samples of Jervois ore were received, all from the Marshall Lode, namely:

Sample 1	947 lb of ore from cut 1
Sample 2	429 lb of ore from cut 2
Sample 3	424 lb of ore from cut 3

Two composite samples were prepared from the above samples:

Sample 4	consisted of 2 parts of Sample 1 and 1 part of Sample 2
Sample 5	consisted of equal quantities of Samples 1, 2 and 3 crushed to nominal $^3/_8$-inch mesh.

Sizing, moisture and copper content of the samples are given in Table 1. Both total copper and copper soluble in 5% sulphuric acid were determined, the latter being referred to in the text as "extractable" copper. Leaching efficiencies were calculated with respect to the extractable copper content of each sample of ore.

2.2 Field Investigations

Samples used in this work were taken from the working face of the Marshall Lode. Sample 6 was crushed to a nominal size of $^3/_8$ inch and the fines were removed by passing over a screen (approximately 8 mesh).

Samples 7 and 8 were crushed to a nominal size of $^3/_8$ inch but without removal of fines. Sizing moisture and copper content of the samples are given in Table 2.

3. ANCILLARY MATERIALS

Commercial grade concentrated sulphuric acid was used for all leaching tests. Reservoir water was used for laboratory work and Jervois bore water for leaching tests carried out in the field. The bore water had a pH of 8.4 and a residue on evaporation of 1430 ppm of which 295 ppm was chloride and 793 ppm bicarbonate.

4. EQUIPMENT

The laboratory leaching tests were carried out in a $9^1/_2$-inch diameter rubber lined vertical tube using a 3 ft deep bed of ore, as shown in Figure 1.

The first field test was carried out in a 10- by $7^1/_2$-foot vessel constructed from wooden planks inside a ground level thickener and with a bed depth of 2 ft 6 inches. Because loading and unloading of this vessel in the thickener proved very time consuming, a 10-ft-square plastic-lined wooden tank was assembled on a concrete base on a level area of ground and used for the remaining leaching tests. The depth of bed in this vessel was 3 ft and the general arrangement is shown in Figure 2.

5. VISITS TO TREATMENT PLANT

5.1 Ore Supplies

The Bell-bird, Green Parrot, Marshall and Reward lodes were inspected. At present ore is being mined only from the Marshall lode and there is a small stockpile at grass. The Reward lode is the nearest to the Treatment Plant apart from the Marshall lode and is said to contain ore of similar grade and composition to that being used at present. Sufficient ore is thought to be available from the Marshall and Reward lodes to enable production of from 1500 to 2000 tons of $CuSO_4.5H_2O$ per year to be maintained for at least 3 years.

The Green Parrot lode is not being considered at present, and ore from the Bell-bird will not be used unless insufficient ore is available elsewhere, because this ore contains a higher proportion of acid soluble impurities than that from the Marshall and Reward lodes.

5.2 Mining

Ore is broken from the open cut by drilling, blasting and hand breaking. The working face of the Marshall lode No. 1 lens is approximately 15 ft square. Broken ore is loaded by a front-end loader into trucks for transportation to the crushing Plant.

5.3 Crushing

The ore crushing and storage facilities are in good condition and appear to be adequate.

5.4 Leaching

At present the ore is crushed to a size of approximately 1 inch and leached by percolation in an area just south of the crushing equipment. The heap for leaching is approximately 40 ft long 12 ft wide and 3 ft deep and contains approximately 100 tons of ore. The walls on either side are left to act as retaining buttresses. Sulphuric acid of 10 to 12% concentration(a) is distributed over the top of the heap from a rigid polythene grid. Liquor percolating through the ore and a 6-inch layer of sand on which it rests, drains at the bottom from an old PVC surface covering. The area slopes slightly to the east, allowing liquor to be collected in a plastic lined sump.

Leach liquor is pumped for a distance of approximately half a mile to the crystallizing area on the east side of a small outcrop.

5.5 Evaporating and Crystallizing

The plastic reinforced fibre glass evaporating areas are not at present usable as evaporating ponds, having been damaged by winds while not securely fastened to the ground. There are two small concrete ponds adjacent to the product washing plant which can hold the sulphate liquors. Most of the crystallization occurs in one of these ponds. Liquor is evaporated until yellow crystals begin to appear on the surface of the solid blue crystals. At this stage the remaining liquor is run off to the second pond where some attempt is made to recover the remaining copper by scrap iron cementation. The copper sulphate in cakes about 2 inches thick is dug out manually and transported to the washing plant.

5.6 Washing and Drying

The copper sulphate cake is fed into a pulverizing drum. From the drum it is pumped as a slurry containing the copper sulphate broken up into small crystals to a screw conveyor which moves the crystals counter-current to the washing water. At the top of the conveyor the crystals are fed to a continuous centrifuge where most of the water is spun out. The crystals then move on belts through two horizontal troughs where they are further dried, and if necessary exposed to hot air in a small furnace. The near dry crystals are finally discharged through a chute to a bagging shed and collected in plastic lined jute bags. Each bag stands on a scale while being filled to ensure that the correct weight of crystals is collected. Some of the saturated

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(a) All acid strengths reported are on a weight/volume basis.

copper sulphate washings are returned to the crystal pulverizing drum while the remainder is fed into the crystallizing pond.

This section of the plant is in good order, is working efficiently and should be able to treat a considerable output of copper sulphate crystals if sufficient liquor becomes available. However, further evaporation and crystallization areas will be required since the original plastic reinforced fibre glass area is useless for this purpose.

6. EXPERIMENTAL PROCEDURE AND RESULTS

6.1 Laboratory Investigations – Leaching

Each leaching test was carried out on a 3 ft deep bed of ore contained in a vertical cylindrical vessel $9\frac{1}{2}$-inches in diameter (see Fig. 1). Approximately 150 lb of ore (ore Sample 4) having a volume of $1\frac{1}{2}$ cubic feet, was used in each of the first seven tests. The ore sizing for the first five tests was nominally 1 inch; for Tests 6 and 7 the ore sizing was a nominal 100% minus $\frac{1}{2}$ inch and 100% minus $\frac{1}{4}$ inch respectively.

The ore was underlain by a layer of sand and gravel to permit drainage. All leaching was conducted by percolation of acid downward from the top. In the first four tests the acid was added manually in small portions. No acid was added overnight. For the later tests a more uniform addition rate was maintained by using a pump to deliver the acid to the top of the column.

The conditions under which the tests were carried out were:

Test 1. A total of 11.5 litres (0.27 bed volume) of 1% sulphuric acid (pH 1.7) was passed through the bed. The effluent was reacidified and recycled through the column. The total leaching time for the two passes of liquor was 19 hours.

Test 2. This test was conducted in a similar manner to Test 1 but 3% sulphuric acid was used and the liquor was recycled four times. Before the liquor was recycled it was reacidified to maintain the acid concentration at approximately 3%. The total leaching time for the five passes of liquor was 28 hours.

Test 3. The sulphuric acid concentration was initially 5%. After the liquor had been recycled twice, the copper content had risen to a high level, so fresh acid was used. After recycling twice this liquor was also replaced by a third batch of fresh acid, which was in turn recycled a further three times. Following each acid pass (except 9 and 11) acid was added to maintain the concentration at 5%. The total leaching time for the eleven passes was 53 hours.

Test 4. The bed of ore from Test 1 was used since the quantity of copper extracted in that test was very small. The procedure adopted was similar to that described for Test 1, but the acid concentration was maintained at 7%. Seven cycles of liquor were passed through the column, fresh liquor being used at the first, fourth and seventh cycles. Acid was added to all recycle liquors excepting on the fifth cycle.

Test 5. The bed previously leached of a small part of its copper in Test 2 was used for this test. A total of 30 gallons of acid, maintained at a concentration of 5% by the addition of concentrated acid when necessary, was recycled for a total period of 257 hours at a flow rate of approximately 85 ml per minute.

Test 6. A portion of ore Sample 4, crushed to a nominal size of 100% minus 1/2 inch, was leached by the single passage of 5% sulphuric acid at the rate of 75 ml per minute. The total amount of acid passed through the bed was calculated to be sufficient to dissolve all the soluble copper in the ore.

Test 7. This test was similar to Test 6 except that the ore was crushed to a nominal size of 100% minus 1/4 inch. A flow rate of 85 ml per minute was maintained.

The liquors were analysed for copper by a method(a) which gives copper plus ferric iron. Ferric iron determinations on 5% acid leach liquors showed that approximately 5% of the indicated copper was present as iron. The analyses for all 5%, and the 7% liquor, have been adjusted on this basis.

The results of leaching Tests 1 to 7 are given in Table 3.

The initial extraction rate in any particular test appeared to be approximately constant, hence the rates shown in Table 3 should be a satisfactory basis for comparison. The overall rates shown for Tests 5 and 7 are probably low because the rates did actually become slower when most of the copper had been extracted.

Test 8. A portion of ore Sample 5 having a dry weight of 143.5 lb and containing 8.75 lb of extractable copper was leached with 5% sulphuric acid. The acid requirement was assumed to be 25% in excess of the theoretical acid required to leach the available copper, i.e. a total of approximately 250 lb per ton of ore. One third of the total acid was made up as a 5% aqueous solution and passed through the ore. The effluent was reacidified with 80% of a third of the total acid to restore the concentration to approximately 5%. This liquor was recycled through the ore, reacidified and again recycled. The effluent then had the remaining acid added and was again passed through the bed. The results are given in Table 4.

(a) A 5-ml sample was diluted with 15 ml distilled water and ammonium hydroxide was added until a blue precipitate formed. The precipitate was dissolved by adding 5 ml of 10% acetic acid solution. Approximately 1 g of potassium iodide was added and the liberated iodine titrated with 0.1N sodium thiosulphate solution. Starch solution was added near the end of the titration to obtain a satisfactory end point.

Test 9. A sample of dry ore (Sample 5) weighing 141.3 lb and containing 8.62 lb of extractable copper was leached with 5% acid. The total amount of acid was approximately 15% in excess of the theoretical amount required to leach the available copper, i.e., a total of approximately 200 lb per ton of ore. One half of the total acid was made up as a 5% aqueous solution and passed through the ore. The effluent was reacidified with the remaining acid and recycled. The effluent was recycled a second time without further acid addition. The results are given in Table 5.

Test 10. A sample of dry ore (Sample 5) weighing 143.3 lb and containing 8.74 lb of extractable copper was used as the "trailing" bed in a split leaching test with 5% acid. The "leading" bed contained the residue from Test 8 which had 32.4% of the original extractable copper remaining. The total amount of acid was approximately 25% in excess of the theoretical requirement and was passed through the leading and trailing beds in series. The effluent from the leading bed was sampled frequently. The results are given in Table 6.
The leading bed which would become the trailing bed of the next cycle had a somewhat higher residual copper content than that used during this test, therefore the overall extraction in practice might be slightly more favourable than indicated by the results in Table 6.

Probably 60% of the extractable copper would be dissolved by passing the total amount of acid through the ore in a single leaching stage. This is consistent with the yield of earlier tests on ½-inch and ¼-inch ore. A 6 bed-volume water displacement wash would recover a further 5% of the available copper.

Although only one extended displacement wash series was carried out, the data are considered to be applicable to all leaching tests providing the copper content of the first bed volume of the displacement wash is known. Six bed volumes of wash are considered necessary where the final effluent liquor concentration is about 40 g per litre. In this context a "bed-volume" is defined as the quantity of liquor held up in the ore after draining. For lower liquor concentrations fewer bed volumes displacement would be required to give effective copper recovery.

6.2 Field Investigations

Test 11. A sample of dry ore (ore Sample 6) weighing 16,500 lb and containing 653 lb copper was leached with 5% acid. The acid requirement was assumed to be 15% in excess of the theoretical acid required to leach the available copper, i.e. a total of approximately 140 lb per ton of ore.
One half of the total acid (1.050 gal) was made up as a 5% aqueous solution and passed through the bed at a flow rate of 4 gallons per ft^2 per hour. The effluent was reacidified with the remaining acid and recycled. This liquor was recycled through the bed once more without further acid addition.
The results are given in Table 7.

Test 12. Ore Sample 7 of dry weight 30,500 lb and containing 1308 lb of extractable copper was leached with 4150 gallons of 4.22% acid. This corresponds to 120 lb of acid per ton of ore or a total quantity of acid 13% less than the theoretical requirement.(a)

The acid distribution was considered less satisfactory than desirable, and to remedy this problem it was decided to use a slightly smaller quantity of ore for Test 13. Since the recycle effluent from Test 12 was to be used for Test 13, the volume of fresh acid used for the second half of the leaching process in Test 12 was 3850 gallons. The first 200 gallons out was added to the first effluent which was mixed, sampled, and pumped to the crystallizers. A sample of this liquor was evaporated to dryness and analysed spectrographically. The results are given in Table 8.

A shortage in volume of approximately 400 gallons was noted of which not more than 100 gallons could be accounted for by leakage or hold up in the ore.

The remainder of the second effluent was collected separately. Two hundred gallons of wash water was passed into the bed and the effluent added to the second effluent collected. A shortage of 850 gallons was observed of which losses from leakage and liquor remaining in the bed would not exceed 100 gallons. Approximately 150 gallons were lost from 3000 gallons of liquor overnight (18 hours) and the losses are thus attributed to evaporation. The ambient shade temperature was approximately 100°F and the tanks were exposed to strong hot winds.

The results of Test 12 are given in Table 9.

Test 13. Ore Sample 8 weighing 29,000 lb and containing 1186 lb of extractable copper was leached with 2850 gallons of recycle effluent from Test 12, containing 7.4 g Cu per litre. The bed was then leached by passing through it 3850 gallons of 4.17% acid which contained 90% of the theoretical acid requirement for a fresh bed of ore. The first 200 gallons of effluent from the drained bed was added to the product from Test 12, mixed, sampled and sent for recovery of copper sulphate. Following the passage of the fresh acid 500 gallons of wash water was used and allowed to mix with the leach liquor. This quantity of wash water was chosen since previous washing had not appeared to be adequate, and also to compensate to some extent for the water lost by evaporation. The results of Test 13 are given in Table 10.

To attain an equilibrium condition in a series of split leaching tests it would be desirable to make up sufficient volume of fresh acid to contain the full theoretical requirement of acid, and to add water to the recycle effluent to compensate for evaporation. This could most profitably be carried out by adding wash water from the bed following passage of the fresh acid. The reduction in volume during the passage of the recycle liquor through the bed should prove advantageous since it provides a preliminary concentration of the liquor before delivery to the crystallizers.

(a) Owing to lack of suitable facilities in the field difficulty was experienced in obtaining accurate analytical results. Later analysis carried out by the Analytical Section of Amdel showed that the copper concentration in the ore was higher, and the acid concentration was lower than assumed. It had been intended to use slightly more than the theoretical acid requirement in both Tests 12 and 13.

7. DISCUSSION

Tests 1 and 2 showed that although copper could be extracted with little more than the theoretical requirement of acid, the rates of extraction using either 1% or 3% sulphuric acid were too low to be of any practical value.

Tests 3 and 5 in which the copper ore was leached by 5% acid for 53 and 257 hours respectively showed that most of the copper could be extracted if prolonged leaching were feasible. In an attempt to reduce the leaching time further tests were carried out using more concentrated acid, and ore crushed to a finer size.

Test 4, with 7% acid failed to show any marked advantage over Test 3 while the acid consumption per pound of copper was slightly higher.

The rate of extraction was increased in Test 6, in which the single passage of the theoretical amount of acid extracted half the available copper. Since the leaching efficiency had been improved by reducing the ore size from 1 inch to $\frac{1}{2}$ inch, Test 7 was done on ore crushed to minus $\frac{1}{4}$ inch. The extraction rate was higher than for any previous test and 82% of the extractable copper was leached.

Information was received from Jervois that the practical limit of crushing was approximately $\frac{3}{8}$ inch, so to obtain a more representative sample equal portions of Samples 1, 2, and 3 were mixed.

Test 8 was conducted on the new ore sample, using a leaching technique which was expected to give a good copper recovery and a high liquor concentration. This expectation was confirmed, but prolonged washing of the residue was required to displace all the liquor. To overcome this difficulty a split leach was planned in the expectation that contacting the depleted ore with fresh acid would result in more efficient extraction and reduce the quantity of water required to wash the ore. The presence of large quantities of low grade liquors seemed undesirable in view of the difficulty of recovering the copper values.

Test 10 which employed the split leaching technique showed the most favourable copper extraction of any test except Test 5 in which the leaching time was impracticably long.

During the course of the leaching tests various flow rates were used. Variation in flow rate within the range 0.6 to 1.3 gallon per ft^2 per hour had little effect on the rate of extraction.

The first leach at Jervois made use of the recycle technique, but the results were unfavourable. This is attributed to the practice at Jervois of rejecting the fines from the crushed ore (some 20% of the total) on the assumption that percolation would otherwise be hindered. Since the ore rejected contains the most readily extractable copper, it is evident that this procedure will markedly reduce the copper recovery.

The second field test made use of ore from which the fines had not been removed. The rate at which acid would percolate through the ore was found to be adequate even though flow rates twice those used in laboratory testing were used, to permit good distribution of acid over the surface of the ore. The mechanical arrangement of the leaching vessel was considerably better than in the first leaching test, although some slight losses due to leakage occurred in the early stages of the test.

The extraction efficiency (based on head and residue assay) was 74.8%. Taking into account the losses by leakage, this is considered to be in good agreement with the recovery calculated from liquor analyses of 70.4%.

Spectrographic analysis of the residue obtained by evaporation of the leach liquor to dryness indicated that the only significant impurity was 2% calcium, equivalent to approximately 8% gypsum. Since the solubility of gypsum is 0.3 lb per 100 lb of liquor, compared with 24.3 lb of copper sulphate per 100 lb of liquor, most of the gypsum would be precipitated before crystals of copper sulphate were formed. The remaining metallic impurities in the crystalline product would not exceed 1.5%.

In the third test, the effluent from the previous test was recycled in an endeavour to simulate the equilibrium condition of a series of split leaching operations. The ideal condition was not attained because of high evaporation losses, but an average leaching efficiency of 69.4% was achieved (residue, 66.3%; liquor, 72.5% efficiency). The acid consumption was equivalent to 2.48 lb acid per pound copper (0.62 lb per lb $CuSO_4.5H_2O$) compared with the theoretical 1.54 lb acid per pound copper.

The effluent contained approximately 1% free acid at this stage and was therefore sent to the evaporation section since the acid concentration was too low to be of further value in leaching.

8. RECOMMENDATIONS

From the results of experimental work in the laboratory and in the field, recovery of 70-75% of the extractable copper in the ore could be expected by application of a simple 2-stage ("split") leaching procedure. The acid consumption would be in the order 2.5 lb H_2SO_4 per lb Cu. The main conditions to be adhered to in application of the leaching procedure are:

1. Ore crushed to a nominal size of minus $^3/_8$ inch and the fines retained.
2. A "split leaching" technique to be applied in 2-stages as described in Section 6.2 (Test 13), using 4.5% w/v sulphuric acid. The amount of acid required is determined by "extractable" copper in the ore, i.e. sufficient acid is used to convert all the extractable Cu to sulphate.
3. Acid flow rate through a 3 ft deep bed: 2 gallons per square foot per hour.
4. Size of ore bed recommended: 10 ft x 10 ft x 3 ft. (see Fig. 2). A bed having these dimensions contains approximately 30,000 lb of ore and is easily managed. The arrangement is simple and beds of this size could be easily duplicated to allow any desired output of copper sulphate to be achieved. The bed dimensions are not mandatory and could be varied to suit operational requirements.

TABLES 1 TO 10

FIGURES 1 AND 2

TABLE 1: ANALYSIS OF JERVOIS COPPER ORE
Laboratory Test Samples

	Ore Sample Number				
	1	2	3	4	5
Moisture, %	1.5	1.4	nd(a)	1.5	1.5
Total copper, %	6.95	5.15	7.90	6.35	6.67
Copper soluble in 5% sulphuric acid, (b)	6.25	4.75	7.30	5.75	6.10
Sizing weight %: + 1 inch	2.2	4.3	nd	2.9	45.2
- 1 + ¾ "	12.2	14.2	nd	12.9	
- ¾ + ½ "	32.0	33.8	nd	32.6	
- ½ + ¼ "	27.0	25.7	nd	26.6	
- ¼ +100 mesh	24.0	22.0	nd	25.0	54.8
-100	2.6				
Bulk density, lb/cu ft	nd	nd	nd	98.6	nd

(a) nd - not determined.

(b) Determined by agitating a small sample of finely ground ore with a large excess of 5% w/v sulphuric acid, and analysis of the residue.

TABLE 2: ANALYSIS OF JERVOIS COPPER ORE
Field Test Samples

			Ore Sample Number	
		6	7	8
Moisture,	%	0.7	nd(b)	nd
Total copper,	%	4.42	4.78	4.45
Copper soluble in 5% sulphuric acid,	%	3.96	4.26	4.09
Sizing weight %: + 1/2 inch		(a)	3.8	7.8
-1/2 + 5/16 "		-	18.8	24.4
-5/16 + 1/4 "		-	9.9	10.0
-1/4 + 8 mesh		-	25.0	25.0
- 8 + 14 "		-	10.1	8.9
-14 + 25 "			7.9	5.7
-25 "		-	24.5	18.2

(a) Approximate sizing:

+ 5/16	inch	40%
-5/16 +20	mesh	55%
-20	"	5%

TABLE 3: SUMMARY OF LABORATORY LEACHING TEST RESULTS
Tests 1 to 7

Test No.	Extraction %	Extraction Rate %/hr(a)	Acid Consumption lb acid/lb copper
1	2	0.1	1.7
2	15	0.5	1.7
3	49	1.0	2.0
4	41	1.6	2.2
5	90	0.3	2.5
6	49	1.5	-
7	82	3.1	-

(a) Average over whole test.

TABLE 4: LABORATORY LEACHING TEST RESULTS
Test 8

Cycle	Liquor Concentration Cu g/l	Extraction %
1st effluent	25.4	28.0
2nd effluent	36.4	41.4
3rd effluent	44.0	52.0
4th effluent	50.2	61.9
Displacement wash, 1.4 bed volume	-	67.6

TABLE 5: LABORATORY LEACHING TEST RESULTS
Test 9

Cycle	Liquor Concentration Cu g/l	Extraction %
1st effluent	23.8	40.0
2nd effluent	35.1	58.3
3rd effluent	41.2	65.5
Displacement wash, 5 bed volume	-	73.1

TABLE 6: LABORATORY LEACHING TEST RESULTS
Test 10

Cycle	Range of Liquor Concentrations Cu g/l	Extraction %
Effluent from leading column	10- 4	21.1
Effluent from trailing column	33-16	60.8
1 bed volume displacement wash column 1	-	1.0
Overall	-	82.9
6 bed volume displacement wash (calc)	-	2.0

TABLE 7: FIELD LEACHING TEST RESULTS
Test 11

Cycle	Liquor Concentration		Extraction	Cum Extraction Available Cu
	Cu g/l	free acid g/l	lb Cu	%
1st effluent	18.3	79(a)	159	24.2
2nd effluent	28.9	57	231	35.4
3rd effluent	34.4	48	253	38.8
Wash water: 1	24.7	9.4	45	7.2
2	6.1	nd(b)	6	0.9
Total Cu extracted	-	-	-	46.9

(a) Includes second acid addition prior to recycle.
(b) nd - Not determined.

TABLE 8: SPECTROGRAPHIC ANALYSIS OF PRODUCT LIQUOR

Element	Concentration %
Cu	Major component
Ca	2
Fe	0.3
Na	0.3
Mg	0.2
Zn	0.2
Al	0.1
Mn	0.06
Si	0.05
Pb	0.02
Co	0.02
Bi	0.02
Ni	<0.01
Sn	<0.01
Cd	<0.01
Au	<0.01
Ag	<0.01
Cr	<0.01
V	<0.01
W	<0.01
Be	<0.01
Mo	<0.01
As	<0.01
Sb	<0.01

< Less than.

TABLE 9: RESULTS OF FIELD LEACHING
Test 12

Cycle	Volume	Liquor Concentration		Extraction	
	gal	Cu g/l	free acid g/l	lb Cu	% of avail.Cu
Fresh acid	4150	-	42.2	-	-
Product effluent	3750	18.6	18.5	698	53.4
Fresh acid	3850	-	41.7	-	-
Recycle effluent	3000	7.4	30.2	222	17.0
Total Cu Extracted		-	-	-	70.4

TABLE 10: RESULTS OF FIELD LEACHING
Test 13

Cycle	Volume	Liquor Concentration		Extraction	
	gal	g/l Cu	g/l free acid	lb Cu	% of avail.Cu
Recycle acid	2850	7.4	30.4	211	-
Product effluent	2700	24.0	10.5	648	36.9(a)
Fresh acid	3850	-	41.7	-	-
Recycle effluent	3300	12.8	22.1	422	35.6
Total Cu Extracted		-	-	-	72.5

(a) Based on total Cu in product effluent minus that originally present in recycle acid.

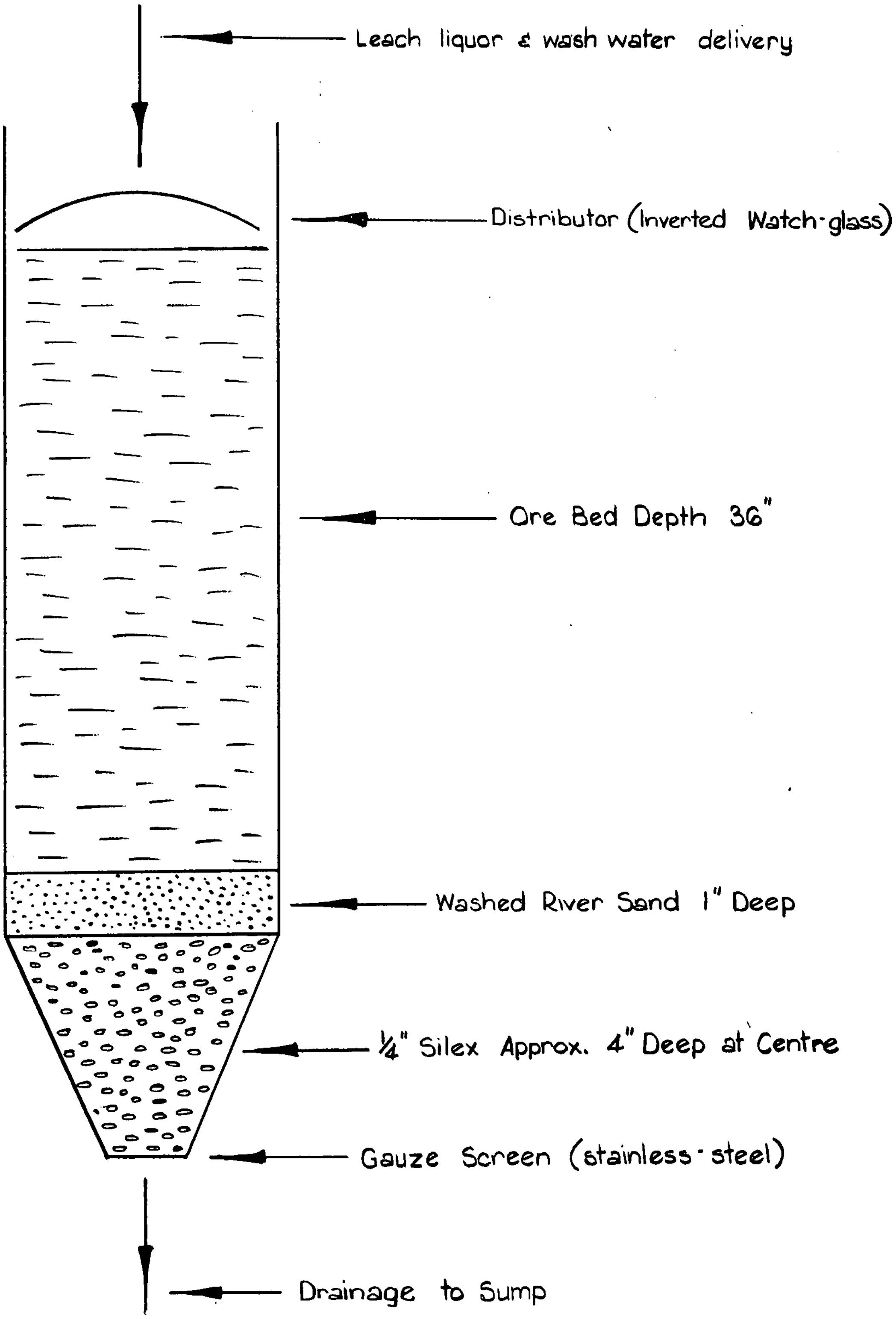
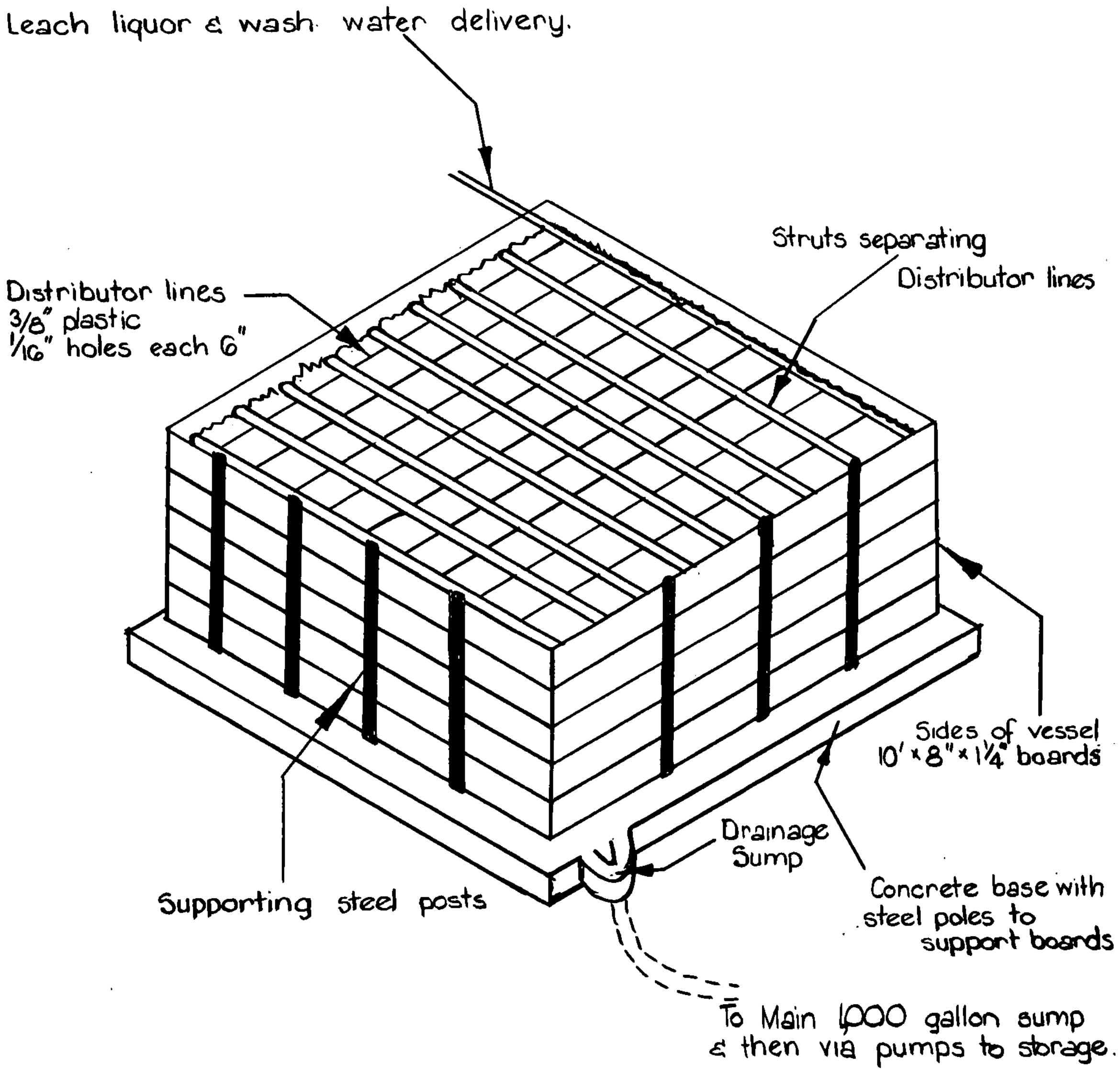


FIG. 1: LABORATORY LEACHING TEST VESSEL
Not to scale



Floor of Leaching Tank covered with strips of conveyor belt rubber overlying polythene sheet. Walls of tank covered by polythene sheeting with additional doubled sheeting in corners.

FIG. 2: FIELD TEST LEACHING VESSEL