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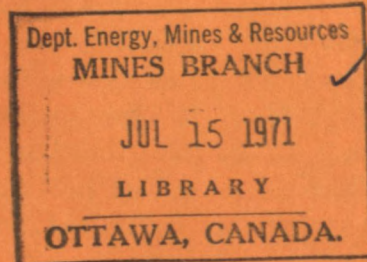
DEPARTMENT OF
ENERGY, MINES AND RESOURCES
MINES BRANCH
OTTAWA

ACID PRESSURE LEACHING A PENTLANDITE-CHALCOPYRITE-PYRRHOTITE CONCENTRATE

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EXTRACTION METALLURGY DIVISION

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ACID PRESSURE LEACHING A PENTLANDITE-CHALCOPYRITE-PYRRHOTITE CONCENTRATE

by

J. A. Vezina*

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SUMMARY

Test work was carried out to determine the feasibility of treating a pentlandite-chalcopyrite-pyrrhotite concentrate by pressure oxidation for the extraction of nickel, copper, and cobalt. Conditions for a practical method of extraction were determined which resulted in >98 per cent extractions of the nickel and cobalt and >90 per cent of the copper as sulphated in aqueous pulp of pH about 1.4. These conditions were that the concentrate was ground to 100 per cent minus 325 mesh and pulped with water or acidified water to contain 30 per cent solids; the pulp was then subjected to a 400-psig pressure with oxygen at 230°F for 14 hours.

An aqueous pulp having these characteristics and containing a substantial quantity of ferric iron could be treated directly by a solvent-in-pulp technique for the recovery of the copper. Subsequently, the copper-free pulp could be treated for the removal of the iron by increasing the pH to about 2 with calcium carbonate. After removing the precipitated solids by filtration, the resulting solution could then be processed for the recovery of the cobalt and nickel by using a liquid-liquid extraction method.

The cost of the reagents to dissolve >98 per cent of the nickel and cobalt and >90 per cent of the copper originally contained in the concentrate was estimated at about $10.00 per ton of concentrate.

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Direction des mines

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LIXIVIATION ACIDE SOUS PRESSION D'UN CONCENTRÉ DE

PENTLANDITE-CHALCOPYRITE-PYRRHOTINE

par

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RÉSUMÉ

L'auteur a effectué des essais en vue de déterminer la
praticabilité du traitement par oxydation sous pression d'un
concentré de pentlandite-chalcopyrite-pyrrhotine, aux fins
d'extraction du nickel, du cuivre et du cobalt. Il a déterminé
dans quelles conditions une méthode pratique d'extraction peut
permettre d'extraire plus de 98 pour cent du nickel et due cobalt
et plus de 90 pour cent du cuivre sous la forme de sulfates
constituant une pulpe aqueuse dont le pH est de 1.4 environ. Ces
conditions sont les suivantes: le concentré doit être moulu
uniformément de façon à traverser le tamis de 325 mailles et mis
en suspension dans de l'eau ou dans de l'eau acidifiée pour former
une pulpe contenant 30 pour cent de solides. Cette pulpe est
alors mise pendant 14 heures en présence d'oxygène à 230°F sous
une pression manométrique de 400 livres par pouce carré.

Une pulpe aqueuse possédant ces caractéristiques et
contenant une quantité importante de fer ferrique peut être
traitée directement au moyen d'une technique du type solvant-sur-
pulpe afin d'en récupérer le cuivre. La pulpe sans cuivre peut
ensuite être traitée pour en éliminer le fer, en faisant monter
le pH jusqu'à 2 environ à l'aide de carbonate de calcium. Aprés
avoir éliminé par filtration les solides précipités, on peut alors
traiter la solution en employant une méthode d'extraction du type
liquide-liquide afin de récupérer le cobalt et le nickel.

L'auteur évalue à environ $10.00 par tonne de concentré
le coût des réactifs nécessaires pour dissoudre plus de 98 pour
cent du nickel et du cobalt et plus de 90 pour cent du cuivre
contenus, à l'origine, dans ce concentré.

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INTRODUCTION

It has been shown that it is possible to decompose pyrrhotite, pentlandite and chalcopyrite in dilute sulphuric acid by pressure oxidation to produce metal sulphates in solution, hydrated ferric oxides, and elemental sulphur [1,2]. The respective decomposition reactions may be expressed by the following idealized equations :

$$FeS + \tfrac{1}{2}O_2 + H_2SO_4 \longrightarrow FeSO_4 + S^o + H_2O \tag{1}$$

$$(FeNi)_9S_8 + 4\tfrac{1}{2}O_2 + 9H_2SO_4 \longrightarrow 4\tfrac{1}{2}NiSO_4 + 4\tfrac{1}{2}FeSO_4 + 8S^o + 9H_2O \tag{2}$$

$$CuFeS_2 + O_2 + 2H_2SO_4 \longrightarrow CuSO_4 + FeSO_4 + 2S^o + 2H_2O \tag{3}$$
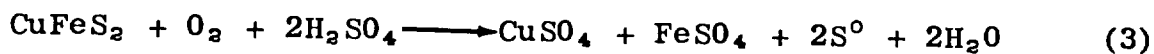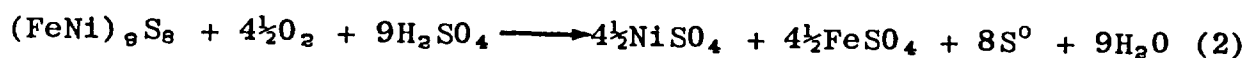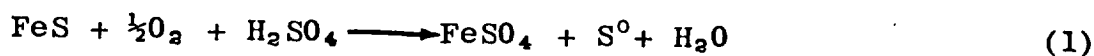
Part of the sulphuric acid required for these reactions is generated by the oxidation of ferrous sulphate and the hydrolysis of ferric sulphate, as follows:

$$2FeSO_4 + \tfrac{1}{2}O_2 + H_2SO_4 \longrightarrow Fe_2(SO_4)_3 + H_2O \tag{4}$$

$$Fe_2(SO_4)_3 + 6H_2O \longrightarrow 2Fe(OH)_3 + 3H_2SO_4 \tag{5}$$
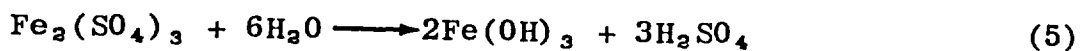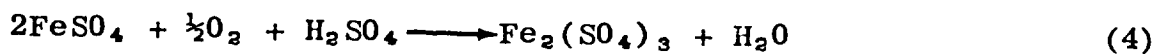
Equations 1, 4 and 5 show that the decomposition of pyrrhotite in dilute sulphuric acid by pressure oxidation is self-sufficient in sulphuric acid, but Equations 2, 3, 4 and 5 show that the decomposition of pentlandite and chalcopyrite is not. The decomposition of pentlandite and chalcopyrite by pressure oxidation requires that sulphuric acid must be added to the leach pulps for the reactions to take place. An important consideration of these equations is that sulphur dioxide, a well-known polluting

gas, is not a product of the reactions.

On the basis of Equations 1 to 5 above, it appeared feasible that Canadian nickel sulphide concentrates, which usually contain chalcopyrite, pyrrhotite and cobalt, could be decomposed to yield nickel, copper, and cobalt in the leach solution and elemental sulphur, hydrous ferric orxies, and the gangue materials in the residue.

The iron can be removed from the leach solution by precipitation with calcium carbonate [3,4], while the nickel, copper and cobalt can be separated from the leach solution by liquid-liquid extraction [5,6] and then can be reduced to metals by electrolysis or by hydrogen reduction methods. The sulphur can be separated from the leach residue by flotation [1,2], for subsequent storage and handling. The hydrous ferric oxides in the final residue, which also contain the gangue, can be de-hydrated to oxides by heating and then be disposed of as backfill, etc., thus causing no pollution problem.

These considerations led to the investigation described in this report, the object of which was to determine the feasibility of a practical process for treating chalcopyrite-bearing Canadian nickel sulphide concentrates by pressure oxidation. The cost of the reagents and the heat required for the process were also evaluated.

PROCEDURE

The test work was carried out on chalcopyrite-bearing nickel sulphide concentrates provided by Consolidated Canadian Faraday Ltd., Werner Lake, Ontario. The chemical analyses carried out at the Mines Branch are reported in Table 1.

TABLE 1

Chemical Analyses of Consolidated Canadian Faraday Ltd. Nickel Sulphide Concentrates

(In per cent)

Mines Branch Reference Numbers:	8/63-1	11/68-2	11/68-4
Nickel	8.87	10.8	10.1
Copper	3.94	5.64	4.29
Cobalt	0.34	0.34	0.30
Iron	30.3	27.9	27.8
Sulphur	23.8	28.0	24.3

A mineralogical analysis of concentrate 8/63-1 (Table 1) showed that the only nickel-bearing mineral present was pentlandite. This mineral was also the most abundant metallic mineral in the concentrate. Chalcopyrite, pyrrhotite and pyrite were the other major sulphide minerals present. No cobalt mineral was identified; the cobalt probably occurred in solid solution in pentlandite and/or pyrrhotite.

The tests were carried out in a 3-litre stainless steel autoclave. The autoclave was heated externally with an open-flame propane gas burner. A stainless-steel coil, inside the autoclave, was used for circulating cold water through the charge for cooling. The agitation was provided by a three-blade agitator which was rotated at a constant 950 rpm by an electric motor. The temperature of the charge was controlled at 230°F $\pm$ 0.5°F to stay below the melting temperature of sulphur and reduce the chance of making sulphate instead of elemental sulphur. The oxygen was supplied by compressed-gas cylinders. A stream of gases from the autoclave was cooled to room temperature to condense the water vapours and then analysed with a Beckman on-stream oxygen analyser. The cooled stream contained about 100 per cent oxygen in every test, indicating that the autoclave atmosphere during leaching contained only oxygen and water vapours.

The leached charges were filtered on a Büchner funnel and the residues were repulped with water, filtered again, and dried. The leach products were analysed by standard analytical laboratory methods.

To precipitate the iron from the leach solution, the solutions were heated to 176°F and calcium carbonate was added to raise the pH in the range of 1.8 to 2.1. The pulps were then agitated vigorously for 2 hours and filtered. The residues were washed with cold water that had been acidified to pH 3 with sulphuric acid and then were dried overnight in an oven at 100°C.

RESULTS

The effect of the oxygen pressure on the decomposition of the sulphide minerals was investigated, using pressures ranging from 20 to 400 psig oxygen, this latter being the maximum permissible design pressure of the autoclave used in this work. The particle size of the concentrate was 80-85 per cent minus 200 mesh, the leaching temperature was 230°F, the pulp density was 15 per cent solids, the retention time was 2 hours, and the impeller speed was 950 rpm. The results of these tests are shown in Figure 1.

With these conditions, the nickel, cobalt, and copper extractions all increased to maximum values of 97, 93, and 34 per cent, respectively, when the oxygen pressure was raised from 20 to 400 psig. The chalcopyrite did not respond as readily to the increased oxygen pressure as did the other metal sulphides, because only 34 per cent of the copper was extracted when the oxygen pressure was 400 psig.

A series of tests was then done to investigate the effect, on the metal extractions, of increasing the pulp density from 15 to 30 and 60 per cent solids. The oxygen pressure was 110 psig, and, as before, the temperature was maintained at 110°C and the concentrate was as received. The results are shown in Figure 2. Increasing the pulp density decreased the rate of decomposition of all of the sulphide minerals present. However, because of the much smaller autoclave volume needed for a given tonnage of

Figure 1. Pressure leaching Consolidated Canadian Faraday
Limited nickel sulphide concentrate. Effect of
varying oxygen pressure.

Conditions: Particle size, 80-85% minus 200 mesh; retention
time, 2 hours; temperature, 230°F; pulp density,
15% solids; agitation, 950 rpm.

Figure 2. Pressure leaching Consolidated Canadian Faraday
Limited nickel sulphide concentrate. Effect of
varying pulp density.

Conditions: Particle size, 80-85% minus 200 mesh; oxygen
pressure, 110 psig; retention time, 2 hours;
temperature, 230°F; agitation, 950 rpm.

dry ore at 30 per cent solids than for ore at 15 per cent solids and the small increase in the chalcopyrite decomposition realized by using a pulp containing 15 per cent solids instead of 30 per cent, it was decided to do all subsequent tests with pulps containing 30 per cent solids.

The effect on metal extraction of increasing the retention time in leaching is shown in Figure 3. The results show that, at an oxygen pressure of about 200 psig, increasing the retention time from 2 to 8 hours increased the nickel and cobalt extractions by about 5 per cent to over 95 per cent. At the same time, the copper extraction increased from 32 per cent to 60 per cent. Further work showed that increasing the oxygen pressure to 400 psig and extending the retention time to 14 hours increased the copper extraction to only 75 per cent.

Since high oxygen pressures and long retention times did not result in satisfactory copper extraction, the effect on copper extraction of fine grinding was studied. The concentrate, which was 80-85 per cent minus 200 mesh as received, was ground to 100 per cent minus 325 mesh in a ball mill and leached at an oxygen pressure of 400 psig for 14 hours at 230°F. Under these conditions, 90 per cent of the copper and more than 95 per cent of the nickel and cobalt were extracted.

In all the tests described to this point, no acid was added to the leach although the reacted pulps were all acidic, being about pH 1.4 for the tests of Figure 3. Therefore,
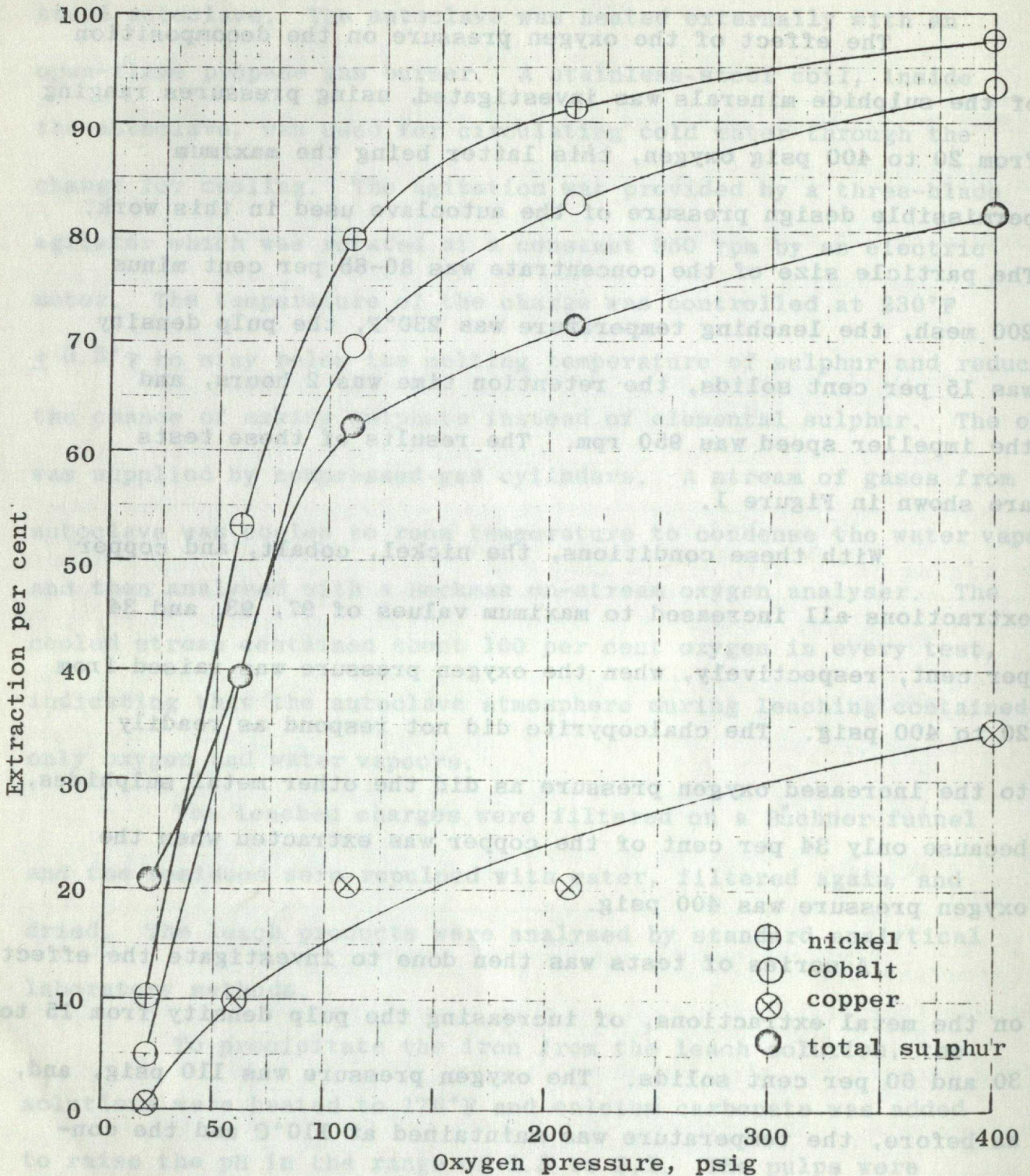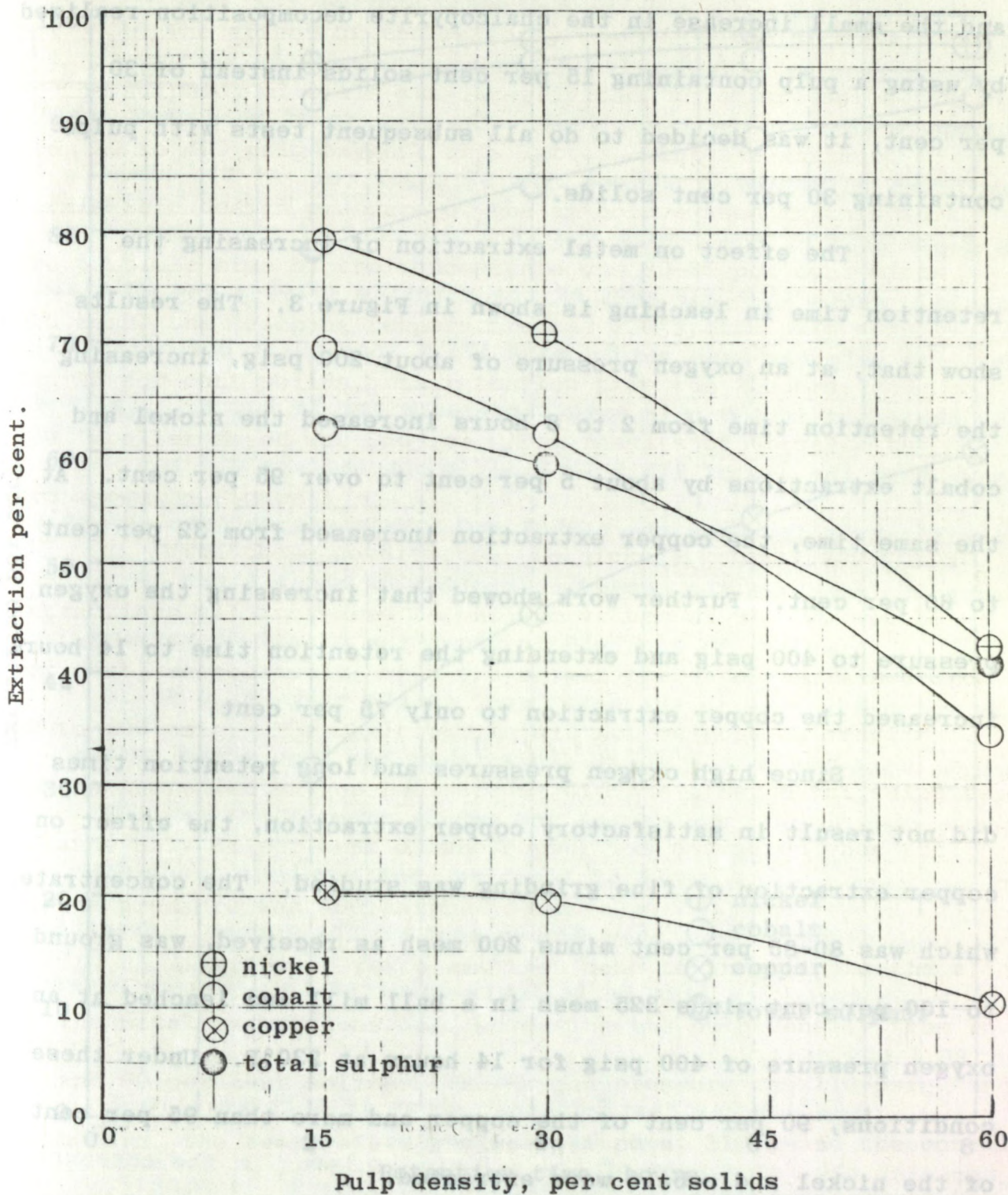
Figure 3. Pressure leaching Consolidated Canadian Faraday
Limited nickel sulphide concentrate. Effect of
varying retention time.

Conditions: Particle size, 80-85% minus 200 mesh; oxygen
pressure, 210 psig; temperature, 230°F; pulp
density, 30 per cent solids; agitation, 950 rpm.

a series of tests was done to determine whether the rate of decomposition of chalcopyrite could be increased by adding sulphuric acid to the leach feed. The results of this work, which are given in Figure 4, showed that 99.2 per cent of the nickel, 98.2 per cent of the cobalt and 95.1 per cent of the copper were leached, while 97.7 per cent of the sulphide sulphur was oxidized when the pulp was made acidic with 175.2 lb H_2SO_4/t concentrate and subjected to 400 psig oxygen pressure for 14 hours. Figure 4 shows that the use of 175.2 lb H_2SO_4/t concentrate did not significantly increase the nickel and cobalt extractions and only served to increase the copper extraction by about 5 per cent.

Finally, in Figure 5 is shown the effect of varying the retention time in the autoclave, in the range of 4 to 14 hours, when 175.2 lb H_2SO_4/t concentrate had been added to the leach feed pulps. The nickel and cobalt extractions were over 95 per cent after 4 hours of leaching, but a 9-hour leach was required to leach more than 90 per cent of the copper. Extending the leach retention time to 14 hours resulted in a copper extraction of 95.1 per cent.

The sulphur "extractions" shown in Figures 1 to 5 represent the percentage of the sulphide sulphur in the concentrate that was not determined in the leach solution and residue as elemental and sulphate sulphur.

Figure 4. Pressure leaching Consolidated Canadian Faraday
Limited nickel sulphide concentrate. Effect of
varying sulphuric acid additions.

Conditions: Particle size, 100% minus 325 mesh; oxygen
pressure, 400 psig; retention time, 14 hours;
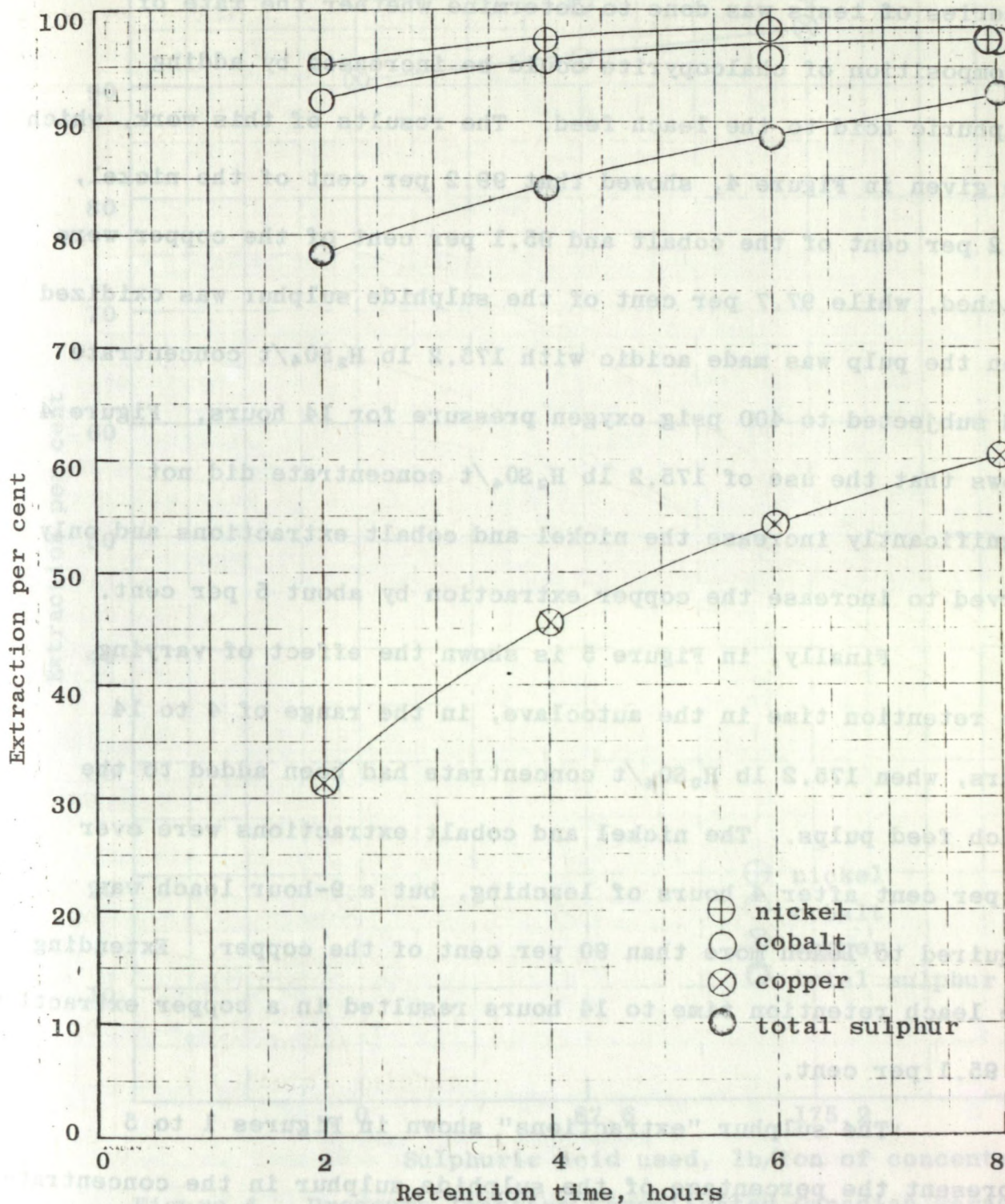pulp density, 30% solids; agitation, 950 rpm;
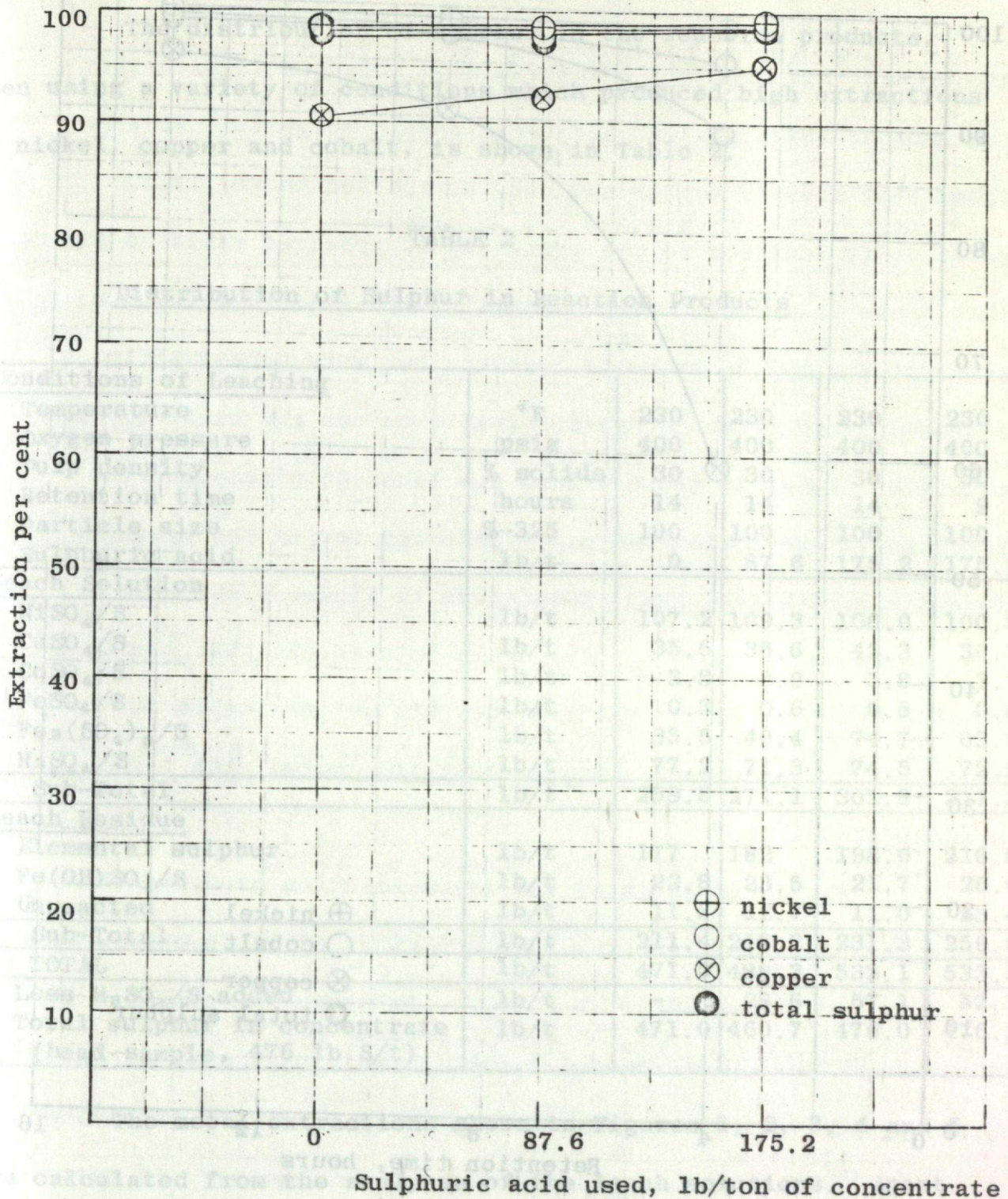temperature, 230°F.
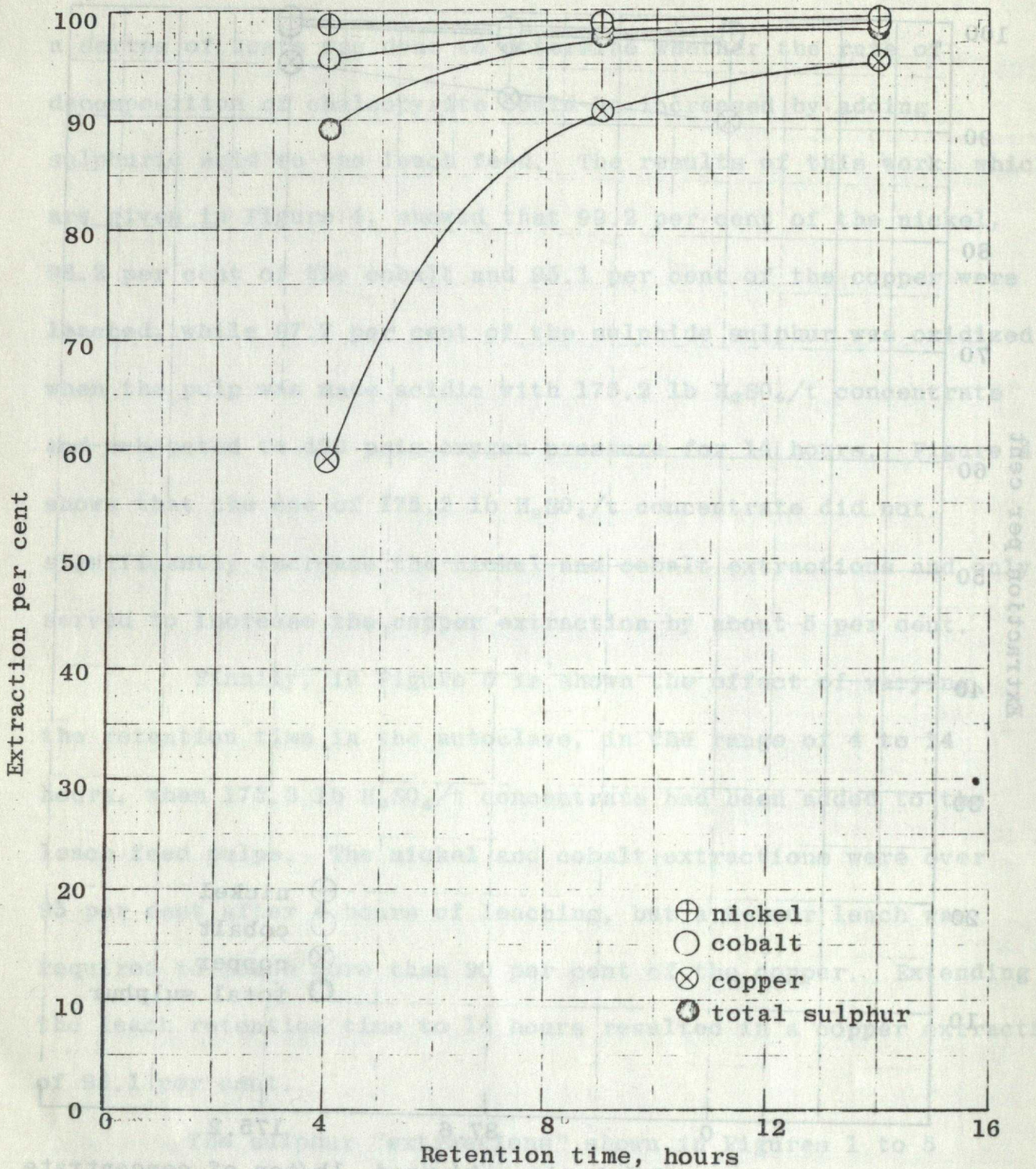
Figure 5. Pressure leaching Consolidated Canadian Faraday
Limited nickel sulphide concentrate. Effect of
varying retention time with sulphuric acid addition
of 175.2 lb/ton.

Conditions: Particle size, 100% minus 325 mesh; pulp density,
30% solids; temperature, 230°F; oxygen pressure,
400 psig; agitation, 950 rpm; sulphuric acid addition
175.2 lb/ton.

The distribution of sulphur in the reaction products, when using a variety of conditions which produced high extractions of nickel, copper and cobalt, is shown in Table 2.

TABLE 2

Distribution of Sulphur in Reaction Products

Conditions of Leaching					
Temperature	°F	230	230	230	230
Oxygen pressure	psig	400	400	400	400
Pulp density	% solids	30	30	30	30
Retention time	hours	14	14	14	9
Particle size	%-325	100	100	100	100
Sulphuric acid	lb/t	0	87.6	175.2	175.2
Leach Solution					
$NiSO_4/S$	lb/t	107.2	109.3	106.0	100.5
$CuSO_4/S$	lb/t	35.6	36.6	42.3	36.3
$CoSO_4/S$	lb/t	3.8	3.9	3.8	3.7
$FeSO_4/S$	lb/t	0.3	0.6	0.5	0.8
$Fe_2(SO_4)_3/S$	lb/t	35.5	49.4	74.7	63.8
H_2SO_4/S	lb/t	77.2	71.3	74.5	72.2
Sub-Total	lb/t	259.6	271.1	301.8	277.3
Leach Residue					
Elemental sulphur	lb/t	177	182	198.6	216.0
$Fe(OH)SO_4/S$	lb/t	22.8	23.5	21.7	26.6
Unreacted	lb/t	11.6	12.7	11.0	13.4
Sub-Total	lb/t	211.4	218.2	231.3	256.0
TOTAL	lb/t	471.0	489.3	533.1	533.3
Less H_2SO_4/S added	lb/t	-	28.6	57.1	57.1
Total sulphur in concentrate (head sample, 476 lb S/t)	lb/t	471.0	460.7	476.0	476.2

The metal extractions shown in Figures 1, 2, 3, 4 and 5 were calculated from the analyses of the leach solutions. Apart from the respective pressure leaching tests, the leach residues from the three tests reported in Figure 4 were further treated, first, with carbon disulphide to dissolve the elemental sulphur,

and second, with 300 g/l sulphuric acid solution to dissolve the hydrous ferric oxides and basic ferric sulphate. It was found that nickel, copper and cobalt reported in the leach solution in addition to the iron, and it appeared valid to assume that these values had been originally dissolved from the concentrate during the pressure leaching but were subsequently coprecipitated with the hydrous ferric oxides and basic ferric sulphate. Thus, if these values are taken into account, the total metal extractions actually effected during the pressure leaching were higher than those shown in Figure 4 and have been compiled in Table 3. It may be pointed out that the acid-soluble metal values in the residues are not available for recovery in the process, because they are locked with the precipitated iron.

The composition of the leach solution produced when leaching the concentrate for 14 hours at an oxygen pressure of 400 psig and adding 175.2 lb H_2So_4/t, is shown in Table 4. A solution with a pH of about 1.4, and containing a substantial quantity of ferric iron, may be treated by a solvent extraction technique[5] for the recovery of the copper; however, it must be free of iron for subsequent treatment by solvent extraction for the recovery of the cobalt and nickel[6].

TABLE 3

Effect of Treating the Leach Residue of Figure 4
with Carbon Disulphide and Sulphuric Acid

(As percentage of total in concentrate)

	NICKEL			COPPER			COBALT		
Sulphuric acid added, lb/t	0	87.6	175.2	0	87.6	175.2	0	87.6	175.2
Recovered in leach solution	98.6	98.1	98.0	90.5	92.2	95.4	97.7	97.8	98.2
Recovered from leach residue*	>1.3	>1.7	>1.9	8.1	6.9	3.0	>2.2	>2.1	>1.7
Total extracted from concentrate	>99	>99	>99	98.6	99.1	98.4	>99	>99	>99

*The leach residue was treated with carbon disulphide and
 sulphuric acid. The values recovered were those found in the
 sulphuric acid leach solution.

TABLE 4

Analysis of Leach Solution (Wash Included)

(Grams per litre)

pH (leach solution only)	1.1
Nickel-iron ratio	2.6
Nickel g/l	29.2
Copper "	10.9
Cobalt "	1.03
Ferric iron "	11.24
Ferrous iron "	0.16
Sulphate sulphur "	43.3

To investigate the removal of iron from a typical leach solution by neutralization, four tests were carried out with the leach solution of Table 4, whereby the solution was treated with calcium carbonate at various pH values, using a procedure that was found successful previously[3,4] to remove the bulk of the iron from nickel and copper sulphate solutions. The results of these tests are tabulated in Table 5, and show that neutralizing the leach solution to pH 2-2.1 required about 180 lb $CaCO_3$/t of concentrate and precipitated about 90 per cent of the iron; the amounts of nickel and copper entrained in the precipitates were reasonably small.

TABLE 5

Effect of Treating Leach Solutions of Table 4
with Calcium Carbonate

(Test conditions: temperature, 176°F; retention time, 2 hours.)

pH	Treated Solutions	Fe Precipitated	Per Cent Losses		$CaCO_3$ Used
	Ni/Fe	Per Cent	Ni	Cu	lb/t concentrate
1.8	2.5	0	-	-	53
1.9	2.3	0	-	-	59
2.0	19.1	88	0.05	0.2	177
2.1	23.9	90	0.1	0.4	189

With an oxygen pressure of 400 psig, the temperature

of the charge had to be raised by external heating to about

180°F for the oxidation reactions to start. After this temperature

was reached, the heat generated by the reactions was intense,

and the charge had to be cooled for 15 minutes with cold water

to regulate the temperature to the required 230°F. From then

on, the temperature of the charge was maintained constant by

using both external heating and water cooling as required. This

arrangement was necessary because the autoclave used was not

insulated. It is likely that with a properly insulated autoclave

the only heat required for the process would be that needed to

raise the temperature of the leach feed pulp to 180°F. Approximate

calculations of this heat requirement have been made (see Appendix A)

and it is shown that this would amount to 522,524 BTU/t concentrate.

DISCUSSION

The results of the work described in this report have

provided a set of leaching conditions which resulted in over

90 per cent extraction of the nickel, copper and cobalt present

in a chalcopyrite-pentlandite-pyrrhotite concentrate. When the

concentrate was ground to 100 per cent minus 325 mesh and leached

for 14 hours at a temperature of 230°F with a solid/liquid ratio

equivalent to 30 per cent solids by weight in an autoclave under an

oxygen pressure of 400 psig, 90 per cent of the copper and >95 per

cent of nickel and cobalt were extracted into the leach solution.

When about 175 lb H_2SO_4/t concentrate was added to the leach, the copper extraction was increased to about 95 per cent but the nickel and cobalt extractions were unaffected.

The leach solutions produced by the above leaching conditions would be amenable to successive solvent extraction operations to separate and recover the copper, nickel and cobalt. Since the copper can be extracted in the presence of iron, the leach pulp could be treated directly to recover the copper from the nickel, cobalt, and iron [5]. The copper-depleted solution could then be treated to remove all the iron, after which it could be treated by solvent extraction to separate and recover the cobalt and the nickel. The metal values in the hydrous ferric oxide precipitate produced in the neutralization step would be negligible.

The cost of the reagents required, using four sets of leach conditions that produced high yields of nickel, copper and cobalt, have been compiled in Table 6. On the basis of reagent unit costs used, Table 6 shows that there is a slight economic advantage in not adding sulphuric acid to the leach feed pulp, although the copper recovery is lower when acid is not added. This is because the value of the additional copper recovery obtained when acid is added is less than the combined cost of the oxygen and the sulphuric acid which must be added to the leach feed pulp to obtain the higher recovery. However, this economic advantage is marginal, because it is based on unit costs of sulphuric acid and oxygen which vary widely according to local conditions.

TABLE 6

. Cost of Reagents

Conditions of Leaching			1	2	3	4
Temperature		°F	230	230	230	230
Oxygen pressure		psig	400	400	400	400
Pulp density		% solids	30	30	30	30
Retention time		hours	14	14	14	9
Particle size		%-325 mesh	100	100	100	100
Sulphuric acid		lb/t	0	87.6	175.2	175.2
Metal Extractions						
Nickel		%	98.6	98.6	99.2	98.4
Copper		%	90.5	92.2	95.1	90.8
Cobalt		%	97.7	97.8	98.2	97.5
Cost of Reagents	Unit Cost					
Sulphuric acid**	$34/t [14]	$/t	–	1.49	2.98	2.98
Oxygen (see Appendix B)	$15/t [15]	$/t	5.62	5.48	5.35	5.11
Heat (see Appendix A)		$/t	0.60	0.60	0.60	0.60
Copper losses*	$0.60/lb [16]	$/t	2.17	1.37	–	2.03
TOTAL	.	$/t	8.39	8.94	8.93	10.72

* Basis: maximum copper extraction, 95.1 per cent.
** The basic unit cost of sulphuric acid used is probably much
higher than can be purchased locally; e.g. 'Sudbury, Ontario,
$10.00/t approximately[17].

It is significant that the leached solution produced

using the set of conditions in column 1, Table 2, contained less

iron than the solutions produced with the other three sets of

conditions which involved sulphuric acid additions. This resulted

in a lower calcium carbonate consumption for neutralizing the solution

to remove the bulk of the iron. In all four tests shown in Table 6,

the sulphur and iron extractions from the concentrate were not

significantly different and were greater than 98 per cent. The

differences in the oxygen consumption between the four tests are in

the quantities of elemental sulphur produced (see Table 2). For
reasons that we do not know, it appears that increasing the leach
retention time from 9 to 14 hours increased the oxidation of
elemental sulphur, and that decreasing the amount of sulphuric
acid added to the feed increased the oxidation of the elemental
sulphur (see Table 2).

CONCLUSIONS

The technical feasibility of treating a pentlandite-
chalcopyrite-pyrrhotite concentrate by a hydrometallurgical method
for the recovery of nickel, copper and cobalt has been established.
High metal extractions from a concentrate similar to that used in
this work can be achieved by leaching the concentrate, ground to 100
per cent minus 325 mesh, at 30 per cent solids, with or without
sulphuric acid added, at an oxygen pressure of 400 psig for
14 hours at a temperature of 230°F. The results of this work
present a variety of possibilities for the treatment of chalcopyrite-
bearing concentrates by hydrometallurgical processes. No estimate
of total operating costs and capital costs of the indicated process
were made, but reagent and heat costs would not be excessive--being
in the order of $0.03 to $0.04 per pound of metal (exclusive of
iron) recovered.

ACKNOWLEDGEMENTS

Metallurgy Division: Mr. W. A. Gow, as head of the Hydrometallurgy Section, made periodic technical reviews of the project and assisted in the preparation of this report, and Messrs. J. J. Laliberté and A. Poulin carried out the test work. The analytical work was carried out by the Chemical Analysis Section personnel under the direction of Mr. J. C. Ingles, Section Head. The mineralogical analyses were performed by personnel of the Mineralogy Section under the supervision of Mr. S. Kaiman, Section Head. The advice of Mr. L. Shaheen, Mill Foreman, was often sought during the test work. Finally, to Mr. H. W. Smith, Division Planning Officer, further thanks are due for sharing his knowledge and files on base metals hydrometallurgy.

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APPENDIX A - Consideration of Heat Requirements

Basic Thermochemical Data:			
			Ref.
FeS	Cp (77-200°F)	0.17 BTU/lb/°F	(8)
FeS_2	Cp (400-298.15°K)	0.14 cal/gram/°C	(13)
$(FeNi)_9S_8$ *	Cp (77-200°F)	0.16 BTU/lb/°F	(11)
$CuFeS_2$	Cp (0-48°C)	0.129 cal/gram/°C	(10)
$CoFeS_2$ *	Cp (77-200°F)	0.16 BTU/lb/°F	(11)
Gangue **	Cp (77-200°F)	0.20 BTU/lb/°F	(11)
Water	Cp (77-150°F)	1 BTU/lb/°F	(11)
H_2SO_4 (100%)	Cp (35°C)	1.414 joule/gram	(12)

*The heat capacities for pentlandite and cobalt sulphide were
not found in literature. Half the sum of the heat capacities
for NiS and FeS, and CoS and FeS, respectively, was used to
approximate the heat capacities for pentlandite and cobalt
sulphide.

**The heat capacity of the gangue was not known; the heat capacity
of quartz was used instead.

Heat to Pyrrhotite

Pyrrhotite in concentrate: 588.7 lb/t
Heat capacity   : 0.17 BTU/lb/°F
Heat required   : 588.7 x 0.17 x (180-77)=
                       10,290 BTU/t

Heat to Pyrite

Pyrite in concentrate   : 7.6 lb/t
Heat capacity   : 0.14 BTU/lb/°F
Heat required   . : 7.6 x 0.14 x (180-77)=
                       110 BTU/t

Heat to Pentlandite

Pentlandite in concentrate: 518.5 lb/t
Heat capacity   : 0.16 BTU/lb/°F
Heat required   . : 518.5 x 0.16 x (180-77)=
                       8,550 BTU/t

Heat to Chalcopyrite

Chalcopyrite in concentrate: 227.4 lb/t
Heat capacity   : 0.129 CTU/lb/°F
Heat required   (: 227.4 x 0.129 x (180-77)=
                       3,020 BTU/t

Heat to Cobalt Sulphide

Cobalt sulphide in concentrate: 20.6 lb/t
Heat capacity : 0.16 BTU/lb/°F
Heat required : 20.6 x 0.16 x (180-77)=
340 BTU/t

Heat to Gangue

Gangue in concentrate : 637.2 lb/t
Heat capacity : 0.20 BTU/lb/°F
Heat required : 637.2 x 0.20 x (180-77)=
13,120 BTU/t

Heat to Sulphuric Acid

Sulphuric acid used : 175.2 lb/t
Heat capacity : 0.3377 BTU/lb/°F
Heat required : 175.2 x 0.3377 x (180-77)=
6,094 BTU/t

Heat to Water

Water required : 4,670 lb/t
Heat capacity : 1 BTU/lb/°F
Heat required : 4,670 x 1 x (180-77)=
481,000 BTU/t

Summary of Heat Requirements:

Pyrrhotite	10,290 BTU/t
Pyrite	110 BTU/t
Pentlandite	8,550 BTU/t
Chalcopyrite	3,020 BTU/t
Cobalt sulphide	340 BTU/t
Gangue	13,120 BTU/t
Sulphuric acid	6,094 BTU/t
Water	481,000 BTU/t
Total	522,524 BTU/t

Heat Cost:

On the basis that 1 imperial gallon of fuel oil costs
$0.16 and has a heat equivalent of 185,000 BTU, and that steam
generation has an efficiency of 75 per cent, the cost of the
heat is as follows:

$$\frac{522,524 \times 0.16 \times 100}{185,000 \quad x \quad 75} = \$0.60/t \text{ concentrate.}$$

For the calculation of heat requirement, only that
necessary to raise the temperature of the leach feed pulp from
77°F to 180°F has been considered. It is realized that some of
the heat from the exothermic chemical reactions might be utilized.
However, the heat cost is less than 6% of the total cost of
reagents therefore any estimate of heat credit would have only
a negligible effect on the statistical significance of the estimate
of the total cost of reagents. Any heat credit was therefore
omitted.

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APPENDIX B — Calculation of the Cost of the Oxygen Required in Pressure Leaching Consolidated Canadian Faraday Ltd. Nickel Sulphide Concentrate with Oxygen

Sets of conditions (see Table 2)		1	2	3	4
Total sulphur in leach products	lb/t	471	461	476	476
adjusted to head assay	476 lb/t	476	476	476	476
Total sulphur reacted	lb/t	465	463	465	463
Elemental sulphur produced	lb/t	178.8	187.8	198.6	216

All reacted sulphur is first oxidized to S^o according to the following equation:

$$M''S + \tfrac{1}{2}O_2 + H_2SO_4 \longrightarrow M''SO_4 + S^o + H_2O \qquad (1)$$

Oxygen required for reaction (1)	lb/t	232	231	232	231

Part of the elemental sulphur is oxidized further to sulphate as follows:

$$S^o + 3/2\ O_2 \longrightarrow SO_3 \qquad (2)$$

Oxygen required for reaction (2)	lb/t	428	412	399	368

The dissolved iron is oxidized to ferric iron as follows: (Fe^{++} in leach solution negligible)

$$2FeSO_4 + \tfrac{1}{2}O_2 + H_2SO_4 \longrightarrow Fe_2(SO_4)_3 + H_2O \qquad (3)$$

Oxygen required for reaction (3)	lb/t	89	88	82	82
Total oxygen required	lb/t	749	731	713.5	681
Cost of oxygen ($15.00/t) [15]	$/t	5.62	5.48	5.35	5.11

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JAV: (PES) dmp