THE NORANDA CONTRACT REPORTS ON THE PRE-FEASIBILITY STUDY OF IN-PLACE BACTERIAL LEACHING: A SUMMATION

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MINERALS RESEARCH PROGRAM
MINERAL SCIENCES LABORATORIES

CANMET REPORT 85-5E

SEPTEMBER 1985

Available in Canada through

Authorized Bookstore Agents
and other bookstores

or by mail from

Canadian Government Publishing Centre
Supply and Services Canada
Ottawa, Canada K1A 0S9

Catalogue No. M38-13/85-5E — Canada: $4.00
ISBN 0-660-12124-7 — Other Countries: $4.80

Price subject to change without notice

Disponible en français

FOREWORD

This report is a summary of the prefeasibility study carried out by Noranda Research Centre under contract to the Department of Energy, Mines and Resources. The original four-volume report was authored by A. Ismay, L. Rosata, and D. McKinnon of the Noranda Research Centre. Readers are cautioned that the costs and design of a bacterial leaching process are site-specific. The data and costs presented herein are specific for Noranda's Geco Mine and should not be transposed in totality to other mining properties. This study is presented as a guideline to the various factors and parameters that must be considered in assessing the potential of bacterial leaching for a particular property.

AVANT-PROPOS

Le présent rapport constitue un sommaire de l'étude préliminaire de faisabilité effectuée au Centre de recherche de Noranda dans le cadre d'un contrat avec le ministère de l'Énergie, des Mines et des Ressources. Le rapport initial en quatre volumes a été rédigé par A. Ismay, L. Rosata et D. McKinnon du Centre de recherche de Noranda. Les lecteurs doivent noter que les coûts et la conception du procédé de lixiviation bactérienne sont determinés par le site. Les données et les coûts cités dans le rapport s'appliquent à la Geco Mine de Noranda et ne devraient pas être appliqués globalement à d'autres propriétés minières. La présentation de cette étude vise à fournir des lignes directrices pour aider à déterminer les facteurs et les paramètres qui doivent être considérés lors de l'évaluation des possibilités de lixiviation bactérienne dans une propriété particulière.

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LIST OF ABBREVIATIONS

cm	—	centimetre
EW	—	electrowinning
FC	—	fixed cost
g/L	—	gram per litre
g/t	—	gram per ton
hp	—	horsepower
IE	—	installed equipment
IPBL	—	in-place bacterial leaching
kg	—	kilogram
kJ	—	kilojoule
lb	—	pound
L/m^2/h	—	litre per square metre per hour
M	—	million
mm	—	millimetre
m^3	—	cubic metre
m^3/h	—	cubic metre per hour
mt	—	metric ton
m/d	—	metre per day
NPV	—	net present value
PLS	—	pregnant leach solution
psi	—	pounds per square inch
RLCS	—	rubber-lined carbon steel
ROI	—	return on investment
RQD	—	rock quality designation
SX	—	solvent extraction
t	—	ton
t/d	—	ton per day
t/m^3	—	tonnes per cubic metre
y	—	year

INTRODUCTION

In 1982, the Canada Centre for Mineral and Energy Technology (CANMET) launched a program with the objective "*to develop biohydrometallurgical processes to extract and recover residual metal values from sulphide ores, specifically those portions of ore deposits that are usually left underground*".

One part of this program had a requirement for a "pre-feasibility analysis of bacterially assisted underground leaching of a Canadian copper sulphide deposit". The results of this pre-feasibility study would provide guidelines for the practicability of proceeding with the development of a biohydrometallurgical process to extract and recover copper from a Canadian sulphide deposit by bacterially assisted, underground in-place leaching. Accordingly, le Centre de Recherche Noranda was asked to submit a formal proposal to Supply and Services, Canada, re Request for Proposal 07SQ.23440-4-9071.

The pre-feasibility study was divided into four phases:

1. mine site selection;
2. pre-feasibility study of underground operations;
3. pre-feasibility study of metallurgical circuit;
4. overall assessment and sensitivity analysis.

Phase 1 — Mine Site Selection

In this phase seven Noranda mines were evaluated to establish the best one to use for this study.

Phase 2 — Pre-Feasibility of Underground Operations

This phase involved the design of the mining developments required before leaching. This design included the access to prepare the stopes, rubblization of ore in the stopes by drilling and blasting, and the excavations needed to collect the leach solution. A unit cost of $14.74 t of leachable ore was estimated for the mining activity. According to Noranda, this value is applicable to any production rate. It was based upon the assumption that development ore could be milled on-site, and that the revenues thus generated could be used as credit to the mining costs.

Phase 3 — Pre-Feasibility Study of Metallurgical Circuit

The leaching circuit and the recovery plant were designed and costed in Phase 2. On the basis of lower capital costs and greater flexibility for adapting the recovery plant to in-place leaching, particularly in the opening and closing years of the project, it was decided to use cementation as the recovery process instead of solvent extraction-electrowinning. The design capacity of the base case was fixed at 1000 t Cu/year, from pregnant leach solution pumped to surface with 1 g/L Cu.

Phase 4 — Overall Assessment and Sensitivity Analysis

In this phase, the costs developed for the mining development and the metallurgical circuits were used to establish the cost flows of the project for variations in:

— price of copper

— operating costs

— leaching rate and overall recovery

— grade of ore.

From the *Milestone Reports*, written by Noranda upon completion of each phase, pertinent data were abstracted and condensed into this report for the convenience of the reader. Should further details be desired, the reports may be obtained from:

Micromedia Limited
144 Front Street West
Toronto, Ontario
M5J 2L7

PHASE 1 — MINE SITE SELECTION

The Geco Mine in Manitouwadge, Ontario, was selected through a ranking process of criteria listed in Table 1, as the best of seven Noranda mines for the purposes of this study. The maximum weight assigned to each factor indicates its relative potential for positive or negative influence on the success of in-place bacterial leaching (IPBL). Thus, a potential ore zone (separated from current mining activities), with a steeply dipping ore body, a low precious metals content (separated from current mining activities) a more acid mine water, and a relatively high ambient rock temperature was assigned a high-ranking weight.

These weights reflect Noranda's priorities. Another company conducting the same exercise might use the same criteria, but might weigh each entirely differently.

The evaluation process is, necessarily, qualitative at this stage. A quantitative set of criteria will be possible only after completion of the pre-feasibility study.

DESCRIPTIONS OF MINE SITE SELECTION CRITERIA

General Description of Mine Site

The mine's location, surrounding topography, and climatology are considered to have, in most cases, little effect on its potential as a test site. However, when a mine is located in mountainous terrain and above the water table, the potential for contamination of local rivers and lakes becomes an important consideration.

Climatology may have an influence where test zones are below an open pit and, therefore, affected by air temperature, or where there is a correlation between the volumes of precipitation and pumped mine water. These situations all lead to a negative rating of the mine. The choice between a -1 rating and a -2 rating (for topography, for example) depended on the relative importance of topography at the particular mine site in question.

Table 1 — Mine site selection criteria

	Maximum weight
General description of mine site	
— Location	0
— Topography	2
— Climatology	1
Mine configuration (general)	
— Orebody dimensions and dip	3
— Mining method(s) used	0
— Tonnage handled	0
— Surface facilities and layout	2
Geology	
— Mineralogy, chemical analysis	3
— Structure	4
Mine water	
— Volume handled	2
— pH	4
Ventilation	
— Volume available	1
Potential zones for bacterially assisted leaching	
— Tonnage	2
— Grade of copper	3
— Grade of zinc	2
— Grades of precious metals	3
— Ground temperature	1
Site specific factors	
— For example: attitude of mine management, potential for damage to the environment, future prospects for mine using conventional techniques, access to potential test zones.	1 to 4 for each factor

Mine Configuration (General)

The mining method, or methods, in use at the mine and the tonnage handled were reported for information purposes, but did not influence the evaluation. A maximum weight of 3 was assigned to the orebody dimensions and dip.

A steeply dipping or vertical orebody merited a positive rating, as it would be more amenable to leaching than an orebody of shallow dip. A +1 rating for surface facilities and layout was awarded for a mine where there was ample space for a pilot plant (200 m × 200 m) near the shaft. A +2 rating was awarded when equipment for leaching (pregnant solution handling/treatment) existed on site, (e.g., vat leaching and cementation plant at Mines Gaspé).

Geology

The mineralogy and chemical composition of the material to be leached was assigned a maximum weight of 3. The presence of acid-consuming gangue in ore or waste zones warranted a negative rating. The less complex the mineralogy of a potential test zone, the more positive the rating for this factor.

Geological structure of the test zone and of the mine in general was weighted very heavily (maximum weight = 4) because the ability to contain and control a leaching solution is dependent on the number, spacing, and nature of fractures, joints, and faults in the rock mass. The more fractured the rock surrounding a potential test zone, the more negative the rating.

If structural mapping had been done, data for 'fractures per metre' might be available. The Rock Quality Designation (RQD), might also be used. The RQD is defined as *the sum length of all pieces of core greater than, or equal to, twice the core diameter divided by the total length of core recovered from a diamond drill hole.* RQD is usually expressed as a percentage.

Where no structural information was available, a rating had to be based on visual observation and an assessment of the mine's gangue.

Mine Water

The volume of mine water handled was an important consideration only if there was a correlation between daily volume pumped and precipitation. Such a situation might exist where the potential test zone is close to the surface and the rock mass is of moderate to high permeability. In such a case (e.g., Mines Gaspé), a negative rating was assigned to reflect the potential for migration of the leaching solution into the local groundwater system.

Mine water pH was regarded as a very important factor (maximum weight = 4). A positive rating was assigned where the pH was acidic, because this would indicate that the gangue is not acid-consuming.

Ventilation

The volume of ventilation available to the potential test zone (or zones) was recorded for information. At this stage, it was not possible to determine accurately the amount required for leaching reactions although it was expected that no additional ventilation over normal operation would be needed. Therefore, this factor was not heavily weighted.

Potential Zones for Bacterial Leaching

To ensure reasonable production rates of copper from low-grade zones, it is necessary to have large amounts of leaching material. Therefore, a positive rating was assigned where large tonnages were available.

The grade factor (for copper, zinc, and precious metals) were interrelated, because Canadian sulphide deposits are generally polymetallic in nature. The higher the copper grade, the more positive the rating. The precious metals grade factor was assigned a maximum weighting of 3 (the same as that of copper) because leaching does not allow recovery of gold or silver. Therefore, the higher the grade of these metals, the more negative the rating. Zinc is leached, but previous internal studies have shown that its recovery from dilute solution is not economic. For practical purposes, higher zinc grades were considered detrimental and resulted in a negative rating.

The ground temperature was not considered to have a significant effect on leaching potential because the solution could be externally heated if required. In addition, some recently reported tests have shown that bacterially assisted leaching might be effected at lower temperatures. However, positive ratings were assigned in the cases of Geco and Goldstream, which have high temperatures relative to the other mines visited.

Site-Specific Factors

Any given mine might have had one or more factors that substantially affected or even outweighed the combined influence of all the other factors. The maximum weight of a given site-specific factor depended on its relative importance at that particular mine.

Table 2 contains a summary of the evaluations conducted for each mine. Table 3 summarizes the main advantages and disadvantages of each site.

All of the mines considered shared the advantage of ready access to potential test zones by existing development, and the disadvantage of low ambient rock temperatures. The Geco mine had the greatest potential for in situ tests, primarily because of relatively high ambient rock temperatures, a steeply dipping orebody, and a potential test zone well away from current mining areas.

Two alternative sites, the Mattabi mine and the Heath Steele mine (in order of preference), were identified as backups to be reconsidered if Geco proved unsuitable.

Table 2 — Summary of mine site evaluations

	Mines Gaspé	Health Steele	Remnor	Goldstream	Geco	Mattabi	Pamour Porcupine (Copper Mine)
General description of mine site							
— Topography	−2	+1	+1	−2	+1	+1	+1
— Climatology	−1	0	0	0	0	0	0
Mine configuration (general)							
— Orebody dimensions and dip	−3	+2	+1	−2	+3	+2	+2
— Surface facilities and layout	+2	+1	+1	+1	+1	+1	+1
Geology							
— Mineralogy, chemical analysis	+3	−1	+1	−1	+3	+2	−2
— Structure	−3	−1	0	0	+2	+2	−3
Mine water							
— Volume handled	−2	+1	+1	0	−1	0	0
— pH	−4	+4	+4	−4	+4	+4	+4
Ventilation							
— Volume available	+1	+1	+1	+1	+1	+1	+1
Potential zones for leaching							
— Tonnage	+2	+1	+2	+2	+2	−1	+1
— Grade of copper	+3	+2	+1	+3	+1	+1	+1
— Grade of zinc	0	−1	0	−2	+1	−1	0
— Grade of precious metals	0	−1	−2	0	+1	−1	−1
— Ground temperature	0	−1	0	+1	+1	0	0
Sub-Total	−4	+8	+10	−3	+20	+11	+5
Site specific factors (See text for explanation)	0	0	−4	−4	0	0	0
Total score	−4	+8	+6	−7	+20	+11	+5

Site-specific factors affected the overall ratings for *two* mines. The *Remnor Project* was a gold mine with copper as an important secondary mineral. Potential test zones for copper sulphide leaching contained gold that was included in Remnor's reserves. The zones were also small and widely scattered within the mine, which would cause complicated pumping layouts and extensive development. These detrimental factors combined for a −4 rating.

The *Goldstream* deposit had high grades of copper and zinc relative to potential test zones in the other six mines. Goldstream had reasonable prospects for being reopened using conventional methods under improved market conditions. A rating of −4 was assigned to reflect this factor.

CONCLUSION

At Geco Mines, the regional geological structure consists of a broad easterly plunging synform. The orebody follows a dragfold in the southern limb of this synform. It is one vertical, lenticular, continuous zone of mineralization that is interrupted by several diabase dykes and is offset by the Fox Creek Fault.

The average horizontal length on any level is about 730 m, and the average width is about 20 m. The bottom of the orebody plunges to the east at an average of 35° as it follows the S-shaped dragfold that exists west of the orebody on each level.

The orebody is made up of a core of massive sulphides consisting of pyrite, pyrrhotite, sphalerite, chalcopyrite, galena, and minor amounts of gold. Appreciable silver is present, associated with the chalcopyrite and galena. The remainder of the massive core is made up of wall rock inclusions. Some 36 M tonnes of ore have been mined since production started in 1957.

The zone selected for this study is an area between the 2450- and 3250-ft levels which is a relatively large, low-grade zone of copper sulphide in a serecite schist with little or no precious metal content (see Fig. 1). The zone contains $\sim 2.5 \times 10^6$ t of ore grading 0.5% Cu, is competent, and is of low permeability.

Table 3 — Mine sites considered for in-place bacterial leaching tests

Mine	Advantages	Disadvantages
Recommended site		
Geco (Total score = +20)	— Steeply dipping orebody. — Relatively high ambient rock temperature (13°). — Potential material for leaching (1 000 000 t at <1% Cu) is isolated from current mining areas, and is considered *waste*.	— Tonnage of potential material for leaching is limited for a long term operation.
Alternative sites		
1) Mattabi (Total score = +11)	— Reasonable grades (1-1.5% Cu). — Mine water is acidic (pH between 3.2 and 3.4).	— Limited amount of leachable material (375 000 t in the Upper Mine). — Mainly a zinc mine, and the zinc and silver values would be lost. — Blasting in high Cu areas could affect current mining areas.
2) Heath Steele (Total score = +8)	— Accessible zone beneath open pit, could be drilled from surface. — Reasonable grades (1.46% Cu). — Mine water is naturally acidic (pH ≤ 4).	— Polymetallic massive sulphide ore, silver values would be lost. — Low ambient rock temperatures, (5-8°C). — Blasting of crown pillar would affect the mine ventilation system and create problems with spring runoff entering the mine.
Sites rejected (order not significant)		
Mines Gaspé (Total score = −4)	— Existing vat leaching equipment could possibly be used.	— All potential material for leaching is in pillars the removal of which could endanger mine stability. Rock mass is quite permeable. Mine water is basic (pH>7).
Remnor (Total score = +6)	— Mine water is acidic (pH = 3)	— Small pockets of ore for leaching. — Low grades (about 0.5% Cu). — Loss of gold values (>4.5 g/t Au).
Goldstream (Total score = −7)	— Reasonably high ambient ground temperature (10-15°C) relative to other mines.	— Copper grades high enough (about 3.5%) for possible re-opening of the mine in the foreseeable future. — Fairly flat orebody (dip = 33°). — Mine water is basic (pH = 7.9). — Carbonate in the ore may be disadvantageous for leaching.
Copper Mine (Pamour) (Total score = +5)	— Existing access to a number of small potential test areas (e.g., stopes containing broken ore).	— Low copper grades (0.47% Cu) and loss of gold values (1.37 g/t Au). Ground is fractured and heavily foliated in potential leaching zone, and leach solution may migrate. Presence of carbonate (acid-consuming) is a possible disadvantage.

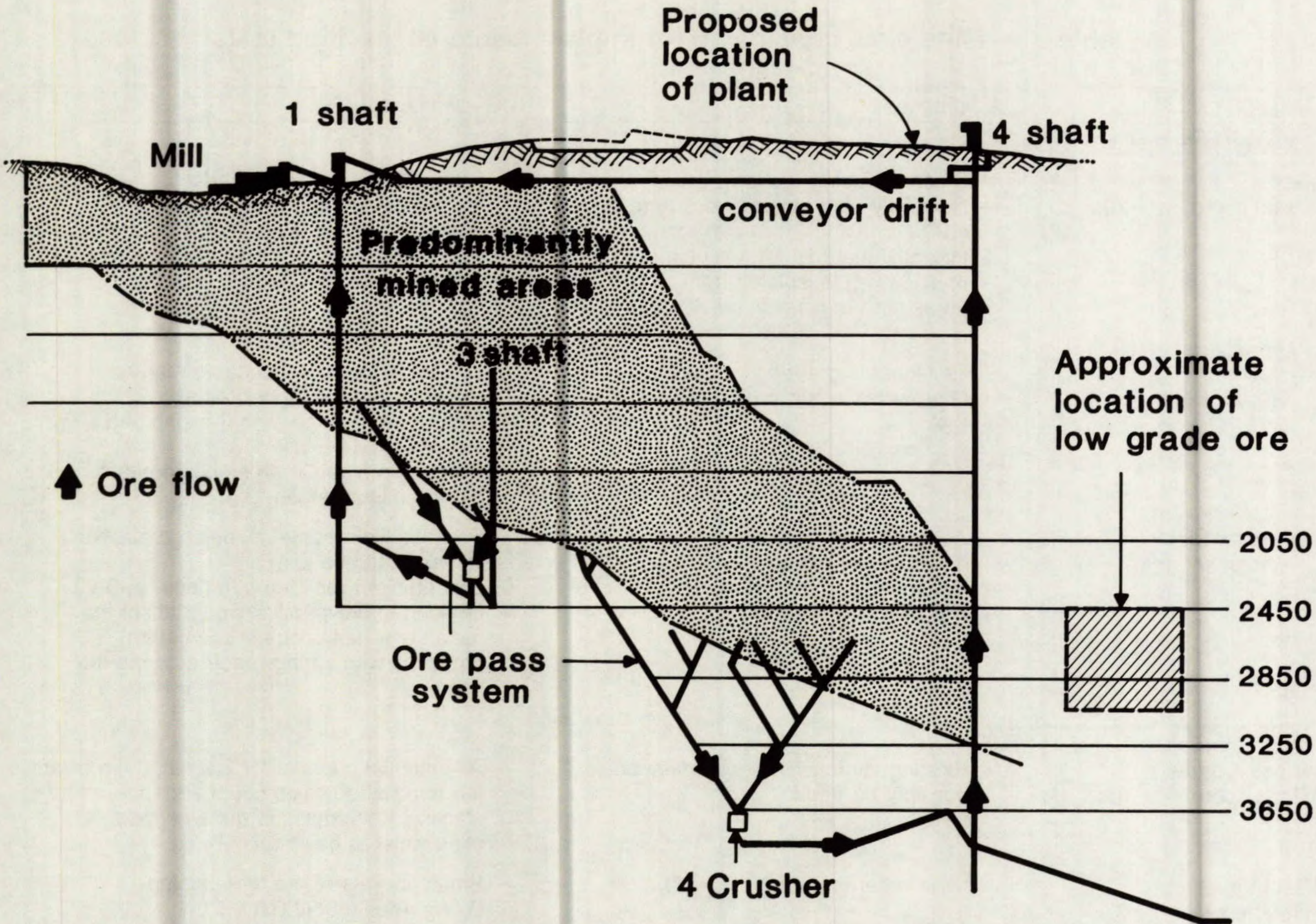

Fig. 1 — Schematic view of Geco mining activities and proposed site zone selected for IPBL

PHASE 2 — PRE-FEASIBILITY OF UNDERGROUND OPERATIONS

The purpose of Phase 2 of the investigation was to estimate the costs to prepare stopes for leaching in the Geco Mine (selected by Noranda in Phase 1 as being the most suitable candidate). These estimates would then be used for the Sensitivity Analysis in Phase 4 of this project.

MINING METHODS AND THEIR APPLICABILITY TO IN-PLACE LEACHING

The information in this section was abstracted *ad verbatim* from the paper "Engineering Pre-Feasibility for In-Place Bacterial Leaching of Copper" by A. Ismay, L. Rosato, and D. McKinnon, Noranda Research Centre, 240 Hymus Boulevard, Pointe Claire, Quebec, H9R 1G5. This paper was presented at the 6th International Symposium on Biohydrometallurgy in Vancouver, British Columbia, August 21 to 24, 1985.

An overview of the different methods used in underground mining is presented in Table 4, including the circumstances in which they were used and the relative costs. It is difficult to present specific information for each of the mining methods, because in most situations the basic concepts have to be modified and adapted to suit a particular orebody and its characteristics. For example, level intervals, powder factors, stone dimensions, drilling and blasting patterns, and haulage methods are site specific and depend on the mine management's expertise and on the orebody itself.

Initial development is needed in all of them in order to:

- access the ore zones;
- create draw points for removing the ore (except room and pillar);
- allow for the swelling of the material during blasting (~25% of the volume to be blasted);
- drill and blast.

Because in-place leaching requires that the leach solution percolate through large volumes of highly fragmented ore, cut and fill, square set, and room and pillar techniques cannot be used. These techniques are based on the immediate removal of relatively small quantities of ore. At no time in the mining sequence does enough ore exist in a stope to justify the installation of the IPBL infrastructure. Shrinkage mining — in which 30 to 40% of the stope's current volume must be removed continuously to allow for further drilling and blasting — is not compatible with long leaching periods.

Caving methods are amenable to IPBL leaching. Also, holes can be drilled from surface or from an underground level to the top of a caved area to percolate leachant through the broken ore. However, because the use of block caving requires that the ore and host rock be relatively incompetent and fractured, and the use of sublevel caving requires that host rock be incompetent, a major drawback for using IPBL in these situations is the potential loss of leachant.

Furthermore, in block caving, there exists little or no control over the size distribution of the fragmented ore, and low metal recoveries can result. The maintenance of wellholes can also prove to be a problem as surrounding rock begins to cave. Other drawbacks result from the fact that up to 40-50% of the in-place ore must be removed before caving begins, and caving is a continuous, rather than a batch, operation. Therefore, IPBL can be used only after a deposit has been completely caved.

Sublevel stoping, though usually more costly than caving techniques, is also amenable to IPBL and allows better control of the leaching parameters. Fragmentation can be done in a more controlled manner, and stope height can be designed to permit downward percolation of the leachant and air flow through the stope. Another advantage is that sublevel stoping is used in competent ground, and solution losses can then be avoided.

Hydrofracing is a relatively new technology being studied for the enhancement of the ore permeability for in-place leaching but, although it would certainly be less expensive than any of the mining methods mentioned previously, it was not considered in this study because of the low original permeability (10 – 9 to 10 – 8 cm/s) (7) of the highly consolidated ore at Geco and also because, in massive disseminated sulphides, only a very small amount of metal mineralization would be expected on the surfaces of existing fractures that ultimately define where the ore is to break.

Hydrofracing is more amenable to leaching copper oxide ores or other secondary mineralization where the mineral exists on fractured surfaces.

DESIGN OF MINING DEVELOPMENT FOR IN-PLACE LEACHING

Sublevel stope mining has accounted for 95% of the ore extracted at Geco. Primary stopes are 21-m wide, 91-m high transverse slices at 37, and 46-m intervals along the strike. These have all been mined and filled with waste rock.

Sublevel stoping was also selected for preparing the ore for IPBL on the basis that:

— The equipment and expertise is available at the mine.
— It is the most applicable technique in the area selected for study.
— It has moderate costs.
— It provides good control over the size distribution of blasted ore.
— It results in well-prepared overcuts for the leachant sprinkler system.

To arrive at the cost of preparing stopes for IPBL at Geco in the serecite schist zone, three representative stopes (Fig. 2) were designed between levels 2650 and 2850.

The design included:

— access for the main levels to the stoping area;
— overcuts for drilling and blasting, and, ultimately, the installation of the sprinkler system;
— undercuts for removal of the ore that would have to be mined conventionally to open a slot to allow for swell upon blasting, and to handle solution flow. (This necessitated the installation of four bulkheads and one sump.)
— raises for access to the mid-sections of the stopes for sublevel development;
— sublevel development for drilling and blasting;
— drilling and blasting patterns, and a suitable powder factor for proper fragmentation.

Tonnages of ore blasted and removed in each of the operations needed in the development phase were calculated, and cost estimates were made using data from Geco. For the purpose of this study, it was assumed that existing mine equipment could be used for commissioning ten new stopes per year in addition to the current mining activity, and that no capital was required for the mine shaft and mill.

To fracture ore in the stopes to an average size of −15 cm, it was estimated that the 3.4- to 3.7-m drilling pattern at present used in the mine should be reduced to 1.5 m × 3 m. Thus, the powder factor used for blasting was 0.7 kg/t ore.

In the early stages of the project, it was observed that of the total known tonnage (2.5 M t at 0.5% Cu), 25% would remain as pillars and roofs to maintain mine stability, and 25% would be removed to allow for swelling during blasting. With these values and the assumed 55% copper recovery over ten years (see Leaching Reactions), this project could only produce 390 t copper per year, which was considered uneconomic for an investment in a new recovery plant.

Consequently, it was decided to set the design capacity arbitrarily at 1000 t/y copper, and the ore zone reserves were therefore increased to 3.8 M t at 0.9% Cu. All other data of Geco (mineralogy, rock temperature, ventilation, mine geology) have been respected. When fully developed, the orebody would contain 60 equally sized stopes measuring 31 m long, 9 m wide, and 61 m deep, containing 35 000 t each of leachable ore.

CONCLUSIONS

The pertinent conclusions of the report were:

— Mining costs incurred to prepare stopes for bacterially assisted leaching are $14.74 t.
— After stope preparation, an average of about 31 500 t stope at about 0.8% Cu remains for leaching.
— Total revenues generated by processing development ore are almost $750 000 assuming a base case copper price for $2.20/kg.
— The three stopes will be ready for leaching after 16 months.
— Permeability testing showed that an insignificant loss of leachate can be expected.

Further details can be obtained by consulting Noranda's "Milestone 2" Report (Micromedia #MON 86-01474/4 Fiche) and *Proceedings of the International Symposium on Biohydrometallurgy* (in press).

Table 4 — Overview of underground mining methods and requirements for their application

Type	Method	Required ore and waste features	Relative costs	Comments
CAVING	Block caving	— Inherent structural weakness and low compressive, tensile, and shear strength of both ore and waste. — Massive, steeply dipping ore zone. — Well-defined contacts, uniform grade	Low	— Undercut is drilled and blasted and material begins to cave into void. This continues as material is removed from the stope. — 100% recovery. — Waste rock above and beside; also caves into the stope.
CAVING	Sublevel caving	— Moderate to high strength ore contained in low strength waste. — 60° to 90° dip. — Uniform grade and good contacts.	Low to moderate	— Sublevels have to be excavated every 9 to 12 m for drilling and blasting of ore. — As more blasted ore is removed and caved, waste rock caves.
NATURALLY SUPPORTED STOPES	Room and pillar	— Relatively flat orebody (up to a dip of 30°). — Moderate strength. — Relatively uniform thickness and grade	Moderate	— Pillars are left at regular or irregular intervals to support the surrounding rock. — Pillars result in a recovery of only 35 to 90%. — Selective (follows ore contacts). — Requires minimum development.
NATURALLY SUPPORTED STOPES	Sublevel stoping	— Vertical or steeply dipping orebody. — Competent ore and waste rock. — Regular contacts.	Moderate	— Pillars are left between stopes. — Requires extensive development in the form of sublevels every 30 m or so for drilling and blasting. — Large stopes are sequentially blasted once a slot representing 25% of the stope's volume has been removed. — Low production costs and the fact that development takes place in ore offset the high development costs incurred. — Sometimes fill is used.
NATURALLY SUPPORTED STOPES	Vertical crater retreat	— Similar to sublevel stoping.	Low to moderate	— Eliminates need of sublevel development using large diameter (165 mm) drill rigs. — Uses spherical blasting charges.
ARTIFICIALLY SUPPORTED STOPES	Shrinkage	— Steeply dipping orebody with regular contacts. — Ore cannot be susceptible to oxidation. — Relatively competent ore and waste rock.	Moderate	— Stope is first undercut and then drilled and blasted vertically upward. — Blasted ore is used as a working platform, therefore, only a limited amount of ore can be removed. — Selective and often used to mine veins. — Filled stopes. — Limited productivity.
ARTIFICIALLY SUPPORTED STOPES	Cut and fill	— Steeply dipping, relatively firm ore	Moderate to high	— Ore is not tied up in stope like in shrinkage because the stope is filled as it is mine and the fill is used as a floor. — Must maintain an ore pass in the fill. — Selective. — Bottom of stope must be undercut like shrinkage.
ARTIFICIALLY SUPPORTED STOPES	Square set	— Irregular, incompetent, high grade ores.	High	— Of limited importance. — Timber used as ore is mined. — Good recovery.

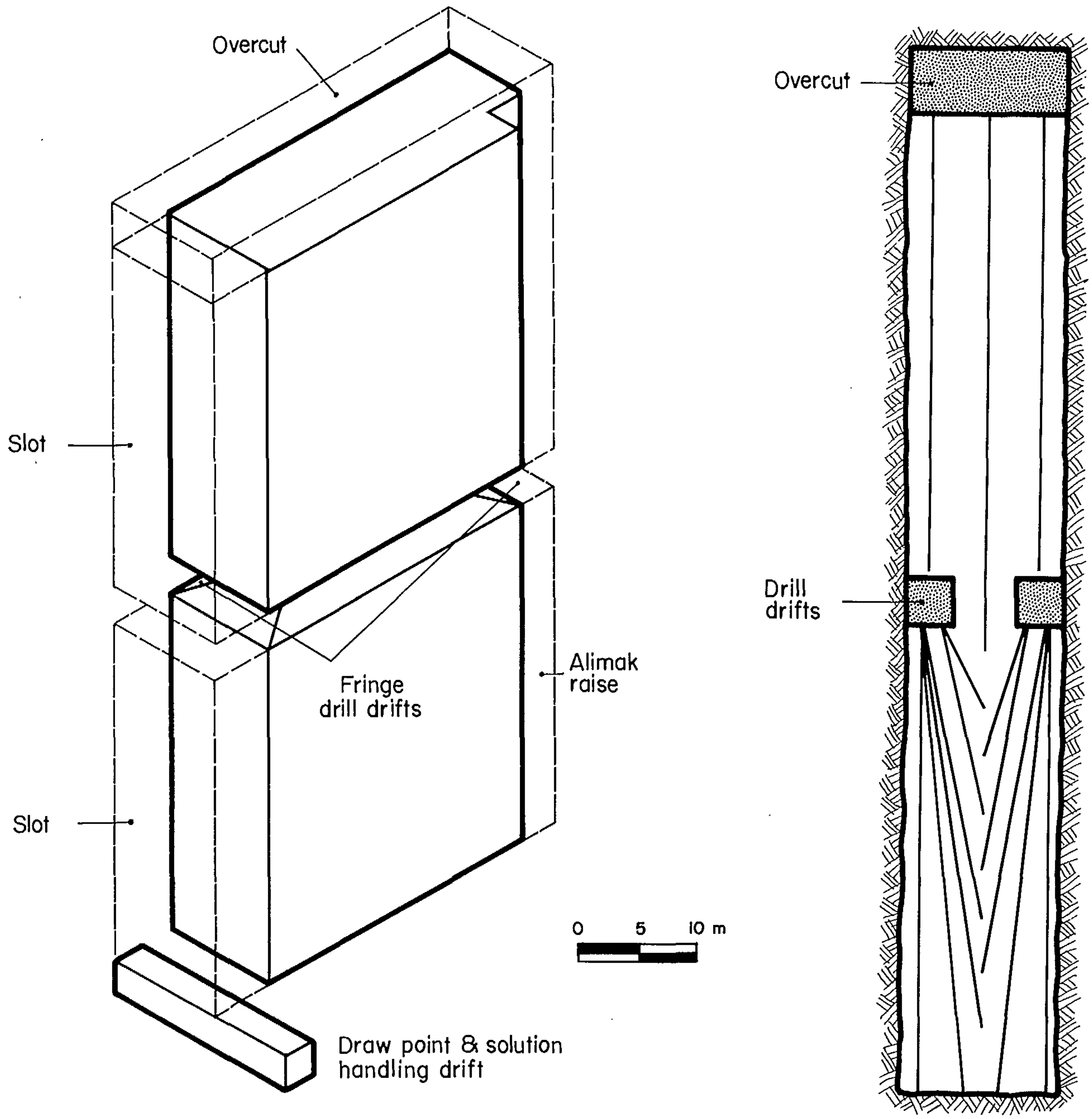

Fig. 2 — Isometric view of stope preparation and schematic cross-section of a drilled stope

PHASE 3 — PRE-FEASIBILITY OF METALLURGICAL CIRCUIT

INTRODUCTION

This phase presents the design and cost estimates of the metallurgical circuit required for producing 1000 t/y of copper by in-place bacterial leaching of a low-grade (0.9% Cu) sulphide zone, 1000 m below surface, and using some of the services of an operating mine.

With a recovery rate of 55% over ten years, this project would produce only 390 t copper per year. This value was considered uneconomic for an investment in a new recovery plant and, to complete the feasibility study, it was decided to assume that the orebody contained sufficient copper to produce 1000 t/y. All other data of Geco (mineralogy, rock temperature, ventilation and mine geometry) have been respected.

Flowsheets have been designed for two recovery systems: cementation and solvent extraction-electrowinning (SX-EW).

Material and heat balances were estimated, based on the mineralogy of Geco ore and the characteristics of the recovery processes. There were no experimental data on this project, and no metallurgical tests done with this ore. The design criteria for the pre-feasibility study were decided on the basis of published information pertaining to:

- bacterial and work done in situ by others at the laboratory and/or pilot plant scale;
- operating experience in related systems.

GENERAL DESCRIPTION OF IN-PLACE BACTERIAL LEACHING

Ore Rubblization

The orebody was divided into sections measuring ~40 m long, 15 m wide, and 70 m deep. A stope of rubblized material was prepared within each of these, which measured 31 m long, 9 m wide, and 61 m deep. Basically, the task consisted in opening accesses from the existing shaft to the top and bottom levels of the future stopes, removing slots of ore from the lateral sides of the stope so that the remaining rock could swell when blasted, and drilling and blasting the ore to rubblize it to an average maximum size of 15 cm.

Leaching

Leach solution, containing acidified iron sulphate and bacteria, (barren from the plant) was brought from the surface down to the level at the top of the stope and transferred through pipes to sprinklers that are layed down on top of the rubblized ore in the stopes. The solution was sprayed on and was allowed to percolate through the rubblized ore, leaching the copper from it, and to exit at the bottom of the stope where it collected in sumps to be transferred to surface, or recirculated to other stopes.

Recovery

Pregnant leach solution was pumped to surface and was processed in the recovery plant to obtain a saleable copper product. Barren solution was adjusted to the desired acidity and iron content, and was sent underground for leaching.

LIMITATIONS OF REAL CASE STUDY

Because about 25% of the deposit, of 2.5×10^6 t at a 0.5% Cu grade, was removed by conventional mining to allow for swelling during blasting and 25% remained in pillars and roofs between stopes to maintain mining stability, the total contained copper available for leaching was only 7031 t.

Copper Extraction

No testwork has been carried out with Geco for this study, nor is there any commercial application of in-place bacterial leaching. A maximum copper extraction of 55% over a ten-year period was therefore assumed, based upon published laboratory and pilot plant studies for chalcopyrite ores.

ESTABLISHMENT OF BASE CASE CONDITIONS FOR FEASIBILITY STUDY

Production Tonnage

To complete the pre-feasibility study, it was decided to assume a base case with a production of 1000 t of copper per year. It is also assumed that the ore reserves required for this tonnage are available in the same orebody at Geco by increasing the reserves to 3.8×10^6 t and improving the grade to 0.9% Cu. It was, therefore, possible to establish mine development costs based on actual data and to use the mineralogy of this orebody in studying the chemistry and heat balances of the leaching process.

The production tonnage selected was a compromise. It was still small for SX-EW plants but only three times greater than estimated tonnages of low-grade ore at Geco.

Pregnant Leach Solution

Based on available information from bacterial leaching studies and dump leach operations, the copper concentration in the pregnant leach solution pumped to surface was fixed at 1 g/L.

Recovery Plant Alternatives

Both cementation and solvent extraction-electrowinning processes were considered because an annual production of 1000 t is smaller than commercial SX-EW plants. The recovery plant interacted with the leaching process by generating or consuming acid, and by producing iron so that each alternative was studied in conjunction with the underground operation and solution treatment/residue impoundment alternative.

Process Flowsheet

Figure 3 presents the generalized flowsheet considered for this study for the in-place bacterial leaching of low-grade chalcopyrite ore followed by copper recovery from the leach liquor by either cementation of solvent extraction-electrowinning.

LEACHING PROCESS DESIGN

Geco Ore Composition

The mineralogical and chemical composition of the rock was available. It was a zone of disseminated sulphides consisting of chalcopyrite, pyrrhotite, and pyrite in a quartz host rock.

For the purpose of this study, it was assumed that all stopes have the same chemical and mineralogical composition.

Leaching Reactions

The principal reactions considered to occur during the acid-bacterial leaching of low-grade rock containing separate inclusions of chalcopyrite, pyrrhotite, and pyrite are given in Table 5, together with the estimated heats of reaction.*

These reactions were used for calculating the acid and iron balances, for estimating the oxygen requirements (to establish if air injection was required), and for predicting the solution temperature within the stopes.

Rates of Extraction

There were no metallurgical data available for this ore, and therefore all extraction rate values were assumed, based upon a review of published information on bacterial leaching in dumps and experimental laboratory and large-column tests.

*For this study, it has been assumed that all sulphide minerals that react are oxidized completely to ferric sulphate, copper sulphate, and sulphuric acid.

For the base case study, it was assumed that chalcopyrite leaches at an average rate of 0.015% per day to give a maximum recovery of 55% copper over a ten-year period. Pyrrhotite leaches much more readily than chalcopyrite, and it was assumed that 100% dissolution would occur over ten years. Pyrite oxidation was assumed to be the same as chalcopyrite at 55% recovery over ten years.

The annual copper and iron extraction were estimated on a per-stope basis from typical leaching-rate profiles. These are shown in Table 6.

Stope Development Schedule

It was assumed that:

- total production of copper would be 10 000 t;
- target annual production would be 1000 t per year;
- each stope would be leached for ten years.

At a grade of 0.9% Cu and 55% copper recovery, the minimum ore required was 3.8×10^6 t. The total number of stopes required was estimated to be 57. For ease of calculation, however, it was assumed that 60 stopes would be developed.

Several schedules were evaluated for the development of the 60 stopes. To achieve the target annual copper production of 1000 t in the shortest time and not to interfere with the hoisting capacity of Geco's No. 4 shaft, it was decided to adopt the schedule in Table 7. With this alternative, the project would produce 1000 t/y copper in the fourth year of operation and would require that about 215 t/day of development ore be removed, which represents approximately 10% of the hoisting capacity at Geco. Because there are normal variations of 20% in this parameter, the selected schedule is compatible with present mine activities.

Rate of Leach Solution Application

An application rate of 10.2 L/m²/h was adopted for this study. This value is typical of sprinkler systems for dump leaching.

The low application rate was adopted to obtain the highest copper concentrations in the pregnant leach solution (PLS), thus compensating for the slow rates of reaction, and to maintain as much void volume as possible so that a proper aeration would exist.

Lower application rates also result in lower pumping costs. Depending on the PLS concentration, the solution is either transferred to other stopes or pumped to surface for copper recovery.

Table 5 — Heats of reactions considered for bacterial leaching of sulphide minerals

Reaction			Heat of reaction kJ
	bact.		
$FeS_2 + 3.5\ O_2 + H_2O$	$\rightarrow FeSO_4 + H_2SO_4$	(1)	−1 443.19
$FeS_2 + Fe_2(SO_4)_3$	$\rightarrow 3FeSO_4 + 2S°$	(2)	+8.039
$FeS + 2\ O_2$	$\rightarrow FeSO_4$	(3)	−903.93
$4FeSO_4 + O_2 + 2H_2O$	$\rightarrow 2Fe_2(SO_4)_3 + 2H_2O$	(4)	−413.66
$CuFeS_2 + 2Fe_2(SO_4)_3$	$\rightarrow CuSO_4 + 5FeSO_4 + 2S°$	(5)	+5.82
$2CuFeS_2 + 8.5O_2 + H_2SO_4$	$\rightarrow 2CuSO_4 + Fe_2(SO_4)_3 + H_2O$	(6)	−3 511.05
$2\ S° + 3\ O_2 + 2H_2O$	$\rightarrow 2\ H_2SO_4$	(7)	−1 244.32
$3\ Fe_2(SO_4)_3 + 10\ H_2O$	$\rightarrow 2\ Fe_3(SO_4)_2(OH)_5 + 5H_2SO_4$	(8)	+333.69
$CaCO_3 + H_2SO_4 + H_2O$	$\rightarrow CaSO_4.2H_2O + CO_2$	(9)	−14.40

Table 6 — Estimated rates of copper and iron dissolution per stope*

Year	Chalcopyrite copper		Chalcopyrite iron**		Pyrrhotite iron***		Pyrite iron****		Total iron	
	Annual mt	Cumulative mt	Annual mt	Cumulative mt	Annual mt	Cumulative mt	Annual mt	Cumulative mt	Annual mt	Cumulative mt
1	31.8	31.8	27.9	27.9	116.9	116.9	24.6	24.6	169.4	169.4
2	28.7	60.5	25.2	53.1	105.2	222.1	22.1	46.7	152.5	321.9
3	25.5	86.0	22.4	75.5	93.5	315.6	19.6	66.3	135.5	457.4
4	22.3	108.3	19.6	95.1	81.8	397.4	17.2	83.5	118.6	576.0
5	19.1	127.4	16.8	111.9	70.1	467.5	14.7	98.2	101.6	677.6
6	16.0	143.4	14.1	126.0	58.4	525.9	12.3	110.5	84.8	762.4
7	12.7	156.1	11.2	137.2	46.7	572.6	9.8	120.3	67.7	830.1
8	9.5	165.6	8.4	145.6	35.1	607.7	7.4	127.7	50.9	881.0
9	6.4	172.0	5.6	151.2	23.4	631.1	4.9	132.6	33.9	914.3
10	3.1	175.0	2.7	153.9	11.7	642.8	2.4	135.0	16.8	931.1

*35 356 mt rock per stope at 0.9% Cu, 3.3% Fe
**Total dissolution of chalcopyrite : 55%
***Total dissolution of pyrrhotite : 100%
****Total dissolution of pyrite : 55%

Table 7 — Copper production schedule (Development of 10 stopes/year) tonnes/year

Stopes Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
10	318	287	255	223	191	160	127	95	64	32								
20		318	287	255	223	191	160	127	95	64	32							
30			318	287	255	223	191	160	127	95	64	32						
40				318	287	255	223	191	160	127	95	64	32					
45					—	159	144	128	111	96	80	64	48	32	16			
50							159	144	128	111	96	80	64	48	32	16		
55								159	144	128	111	96	80	64	48	32	16	
60									159	144	128	111	96	80	64	48	32	16
Total Cu Prod mt/y	318	605	860	1083	956	988	1004	1004	988	796	605	446	319	224	160	96	48	16
Stopes in operation	10	20	30	40	40	45	50	55	60	60	50	40	30	20	20	15	10	5

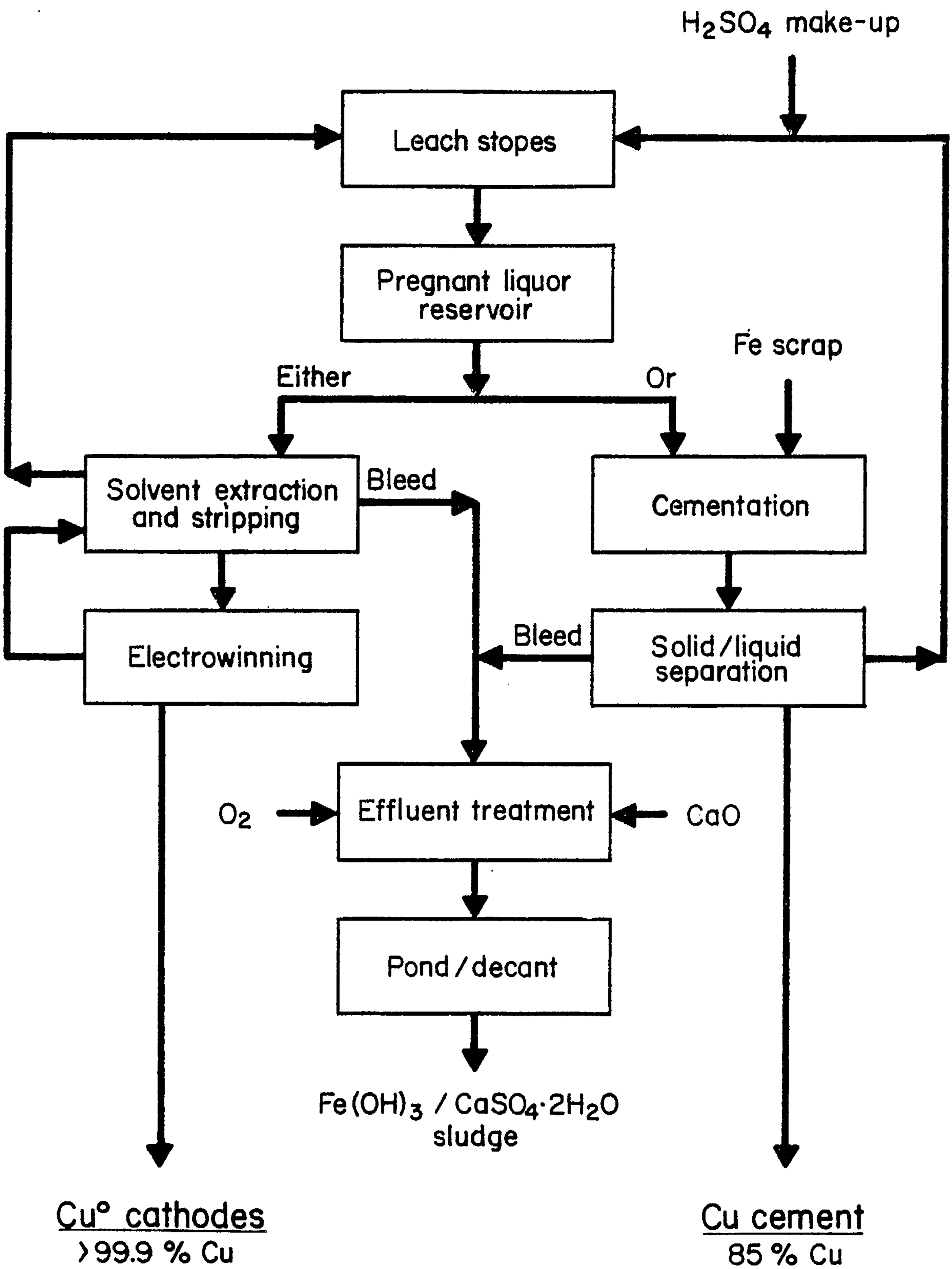

Fig. 3 — Generalized flowsheet for in-place leaching and copper recovery

Pregnant Leach Solution

It was assumed that the average PLS composition would be 1 g/L Cu so that the design volume of solution feed to the recovery plant would be 114 m^3/h, based on an annual copper production of 1000 t.

The PLS volume fixes the design capacity of the copper-recovery plant.

Solution Flow Management

Because new stopes were to be commissioned for leaching in each of the first few years of the project (10 stopes/year in the first 4 years) and were to be taken off leaching at the end of the project, and also because it was assumed that the solution application rate would remain constant during the lifetime of the project, there would be variable volumes of PLS to pump to surface, and the PLS from certain stopes would have to be recirculated through others during the middle years of the project.

The first factor, the variable volumes of PLS, affects the selection of the recovery plant. In this aspect, it was felt that the cementation plant offered the best solution because it could be designed for the maximum flow. Smaller volumes would only result in unnecessary longer residence times, whereas, with solvent extraction, PLS in the first and last years of operation would have to be mixed with barren solution to maintain the aqueous flow close to the designed volume, which would cause a certain dilution equivalent to what would occur if larger application rates were used to leaching.

The second factor was controlled by assuming that a group of ten stopes leaches at the same rate (as a block) and by building PLS sumps at a collection point from the ten stopes, so that, if the PLS copper concentration were low, it could be pumped from this sump to another group of ten stopes. Thus, each ten-stope group was provided with one sump and two pumps (one stand-by).

Permeability and Irrigation Rate

The void space or pore volume within each stope was assumed to be equivalent to the rock swell during blasting, which is 25%. A fully saturated stope would therefore hold 4120 m^3 of lixiviant.

At an application rate of 10.2 $L/m^2/h$, the vertical flow velocity will be 0.25 m/d and under good ventilation conditions, the pore volume should contain sufficient oxygen to maintain the reactions assumed in Table 5.

In many dump leaching operations, the solution flow is interrupted for certain periods to allow the ore to "rest". During this time air is restored to the void space and higher extraction rates are observed when solution is re-applied.

Because, in this case, the bacteria population was an important factor to maintain proper leaching rates, it was felt that rest periods may be counter-active and therefore, continuous solution application was adopted. As a result, the irrigation rate was the same value as the application rate.

It was also assumed that, because the leach solution flow was maintained constant, there would not be any decrease in stope permeability during the leaching period of the stope.

Acid Balance

The acid balances for both the leach SX-EW and cementation alternatives were calculated based upon the reactions and upon the assumption that all dissolved iron was oxidized to the ferric state, and that sulphur from the reactions was oxidized to sulphuric acid.

It was also assumed that the overall leaching system would be self-buffering between pH 2-2.8 and, thus, would create a stable environment for bacterial activity as well as for chalcopyrite and ferrous iron oxidation. To create such conditions, it was necessary to assume that the ore contained acid-consuming gangue.

Based upon the chemistry of this system, sulphuric acid would be consumed by:

- dissolution of chalcopyrite;
- oxidation of ferrous iron to ferric iron;
- dissolution of the acid-consuming gangue:

and would be generated by:

- oxidation of sulphur to sulphate;
- precipitation of ferric iron as basic iron sulphate.

For the SX-EW alternative, acid was also generated by the extraction of copper to the organic phase.

Iron Balance

Within the pH limits 2-2.8, created by the self-buffering leaching system, the total maximum ferric iron concentration was ~3-4 g/L. It was assumed that all the iron was oxidized to the ferric state, so that the total iron concentration in the pregnant liquor was also at ~3-4 g/L.

It was also assumed that, under steady-state conditions, the net iron precipitation would be equivalent to the iron released by sulphide dissolution plus iron introduced from cementation if iron precipitation of copper were used.

Oxygen Requirements

The total oxygen required to oxidize 55% of the contained chalcopyrite, 100% of the contained pyrrhotite, and 55% of the contained pyrite within the 60 stopes to copper sulphate, sulphuric acid, and ferric sulphate was estimated to be 89 551 mt. An additional 3544 mt O_2 are required to oxidize the iron introduced for the cementation alternative for copper recovery.

It was assumed that air circulation within the stopes, which contain 25% voids, is free-flowing and that no air injection system is required in addition to the normal mine ventilation.

Heat Balance

The average net amount of heat generation, as calculated from the standard heats of reaction (see Table 5) and assuming that all of the iron was oxidized to the ferric state, was 11.7 kJ. The most significant source of heat generation was pyrrhotite oxidation which contributed more than 58% of the total heat, whereas the main heat losses were:

- rubblized ore in the stope;
- pregnant leach solution that was pumped to surface;
- air circulating through voids in the rubblized ore which then entered the main ventilation system of the mine.

For both recovery alternatives, it was estimated that the heat losses in the transport of barren solution to the sprinklers and pregnant solution from the underground sumps to the recovery plant (most of which was done through 2000-m pipes in the No. 4 shaft), resulted in a temperature drop of 5-7°C. The initial rock temperature at Geco was 15°C.

The heat balances for both recovery alternatives indicated that there would be temperature variations within each stope of between 15 and 45°C, and that the average temperature of the leach solution leaving the leached stopes could be between 27 and 29°C. The temperature of the PLS entering the recovery plant would average 22°C.

These calculations indicated that the underground bacterial leaching system and the recovery plant could operate without external solution heating.

Water Balance

It was assumed that there would be no solution losses within the underground leaching system, and that the only bleed would be through the cement copper (15-20% moisture) in which case an equivalent amount of water would be added during the acidification stage.

Commencing in year 12 onwards, the volume of barren solution recycled to leaching would be decreased to maintain solution application rates at 10.2 L/m^2/h and would be directed towards lime neutralization for iron and gypsum precipitation.

Description of Leaching System

Barren solution was transferred through a 15-cm diameter, rubber-lined, carbon steel (RLCS) pipe from the recovery plant, down the No. 4 shaft to the 2650 level where it entered a 10-cm diameter pipe layed in the access drift to the orebody development. Solution was distributed through header pipes to each stope development access and was transferred to 25-mm diameter pipes layed in the centre of each rubblized stope. Five Senninger No. 6 wobblers were joined to the distribution line at 6-m intervals. These sprinklers were impulse-type, and discharged solution continuously over a 10-m diameter circle.

Solution emerging from the draw points under each stope were collected in drainage ditches that conveyed it to a collection sump. Because it was estimated in the stope development schedule that the best commissioning option was ten stopes per year, the solution emerging from each of these should have had the same concentration of copper. Collection samples from each group of ten or five stopes were connected to pump sumps. Each of these was provided with two 15-hp pumps to either transfer the PLS to the sprinkler system on another group of ten stopes, or to a common sump from which solution was pumped to the surface.

The pumping of 114 m^3/h of PLS from the 2850 level to the plant at the surface was done through 15-cm diameter, RLCS tubing in three stages of pumping, each of which was provided with two 150-hp pumps (one standby).

A schematic drawing of the underground piping system is presented in Figure 4.

Pressure relief valves were installed every 70 m between the surface and the distribution lines in the stopes to reduce pressure from ~800 psi down to the 50-100 psi required in the sprinkler headers.

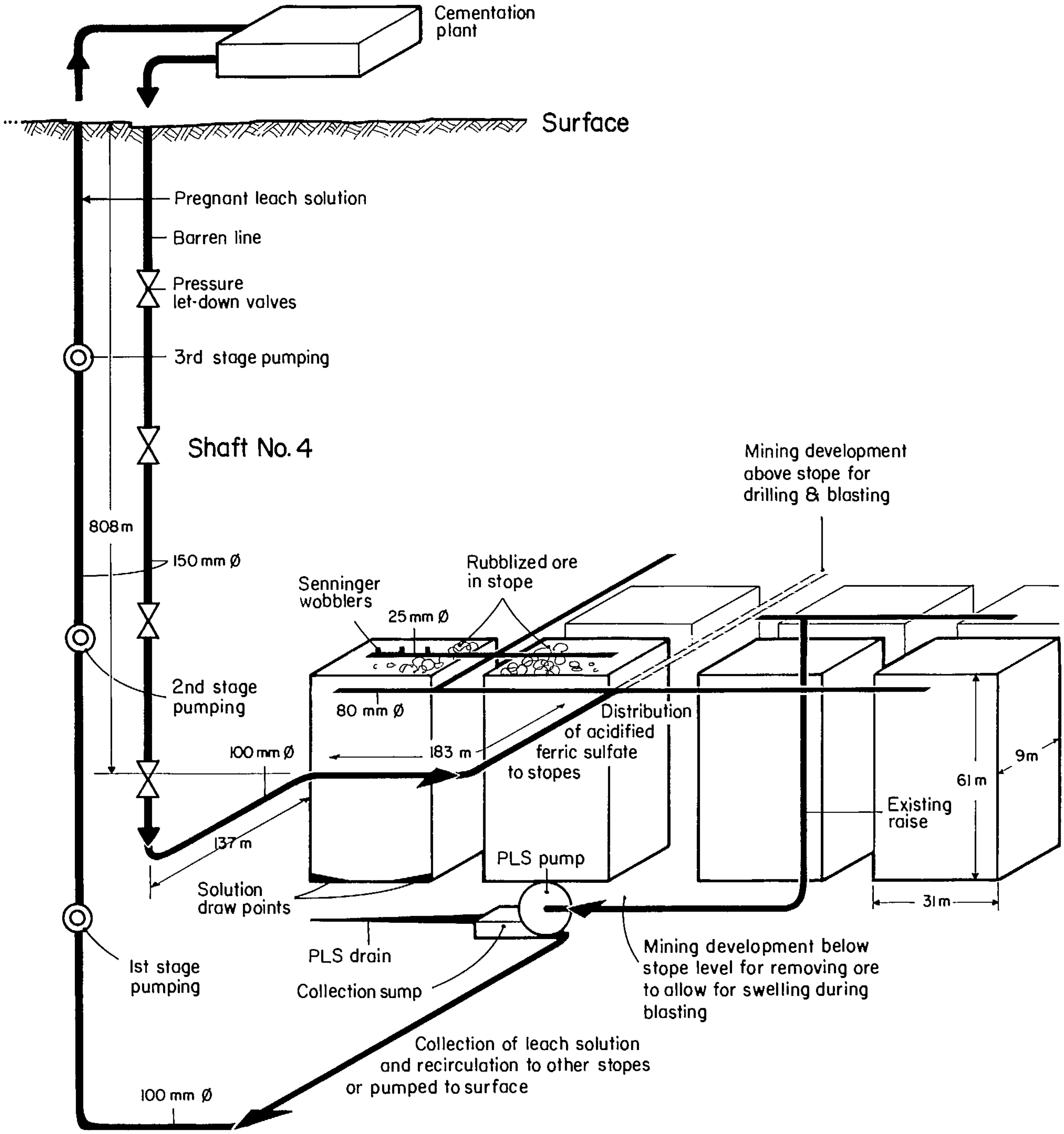

Fig. 4 — Schematic piping network for in-place bacterial leaching

Labour Requirements

Because of the small scale of the recovery plant and the low labour requirements in the leaching system, it was necessary that the operators perform multiple assignments. The main areas, in addition to the recovery plant, were:

- leach field operations
- laboratory
- control room
- maintenance.

It was assumed that for the base case, at 1000 t/y copper, the administration and engineering activities required for this project would be handled by the mine-mill personnel. In addition to the above activities, the project had one single supervisor in charge of all project activities.

Total labour requirements for the leaching section in the base case study were estimated as one person, including operation and maintenance.

The main activities in the leach system were:

- pipe layout for sprinklers
- pump maintenance
- sampling of stope effluents.

Some of these activities would require a two-person team, but staffing then would require personnel from other areas. It was assumed that no security personnel would be required for this size of plant.

RECOVERY PLANT DESIGN

Plant Size

Both recovery alternatives were designed to treat 114 m^3/h of PLS solution at a 1 g/L copper concentration.

Cementation

The pregnant leach solution was fed to a series of launders containing detinned cans, and copper was cemented out from the solution according to the following reaction:

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

at an iron consumption of 2.5 kg per kg copper precipitated.

The cement passed through the false bottom of the launders to a settling tank. The barren solution overflow, containing 0.03 g/L Cu, 5-6 g/L Fe at pH 3 was acidified with sulphuric acid to 0.3 g/L H_2SO_4 and recycled to the stopes for further leaching. The cement copper was reslurried with water, and was filtered on a plate and frame filter press, and the cement cake, which contained 15-20% moisture, was transported to a smelter. A cement copper grade of 85% was assumed.

The cementation plant flowsheet and equipment diagram is presented in Figure 5.

Solvent Extraction-Electrowinning

Copper was extracted in two stages from the pregnant leach solution with 5% LIX 64N in kerosene and was stripped from the organic phase in one stage with spent electrolyte containing 170-180 g/L H_2SO_4 and 30 g/L Cu. The raffinate contained <0.01 g/L Cu, 4 g/L Fe, and ~1.5 g/L H_2SO_4 and was recycled to the stopes directly without pH adjustment.

The pregnant electrolyte containing 45 g/L Cu and 150-160 g/L H_2SO_4 was fed to 14 commercial electrowinning cells that use lead/calcium/tin anodes, and copper was electrowon at a current density of 215 A/m^2 and at a current efficiency of 85%. It was assumed that the quality of the copper cathodes is >99.9% and that they can be directly sold on the LME.

The SX-EW flowsheet and equipment diagram is presented in Figure 6, and a plant layout for both solvent extraction and electrowinning is shown in Figure 7.

Advantages of Cementation Over SX-EW

As mentioned before, the recovery plant would either have to treat smaller PLS volumes than the designed capacity, during the first four years and the last two to six years of operation, or mix barren solution with PLS to pass a volume of aqueous solution to solvent extraction that was within a working range from the design value.

On this matter, a cementation plant offered an advantage over SX-EW plants not only because the designed residence time could easily be achieved by by-passing a certain amount of launders, but also because the major reagent consumption (iron) was a function of the copper tonnage recovered and was less dependent on the volume of solution treated. On the other hand, in solvent extraction, the aqueous flow would need to be similar to the designed volume (maximum) and the major consumption (organic losses) was a function of the volume of solution treated.

At relatively comparable costs, the cementation plant should be selected on the basis that it adapts better to an in-place leaching operation.

Also, as discussed later on when dealing with the final years of the project, operations in the recovery plant were progressively reduced (the same operations in shorter time periods to recover decreasing copper tonnages) to maintain the operating costs closer to the period of full production. Since cementation

— has less unit operations than SX-EW;

— requires less supervision;

— has a higher proportion of batch vs continuous activities;

it was easier to transform to a single-shift operation and it adapted better to cost reductions.

Labour Requirements

Cementation

Most of the activities of a cementation plant of this capacity could be handled by a single operator in one shift. The batch filtration operation could be carried out in two shifts (one operator in each) at the size and with the equipment costed in the capital cost estimate. However, idle times will exist, and both the operators in the launders and the filter-presses could be used in other activities.

The small scale of the plant would demand that operators perform different activities. In the case of the cementation alternative, to have one person in the recovery plant at all times, the operators could also perform duties in the laboratory and watches in the control room. The filter presses would be manned by two operators, with some functions in the maintenance department to be divided with the operator of the leaching section. One supervisor was included in the estimate of the recovery plant, but would also be responsible for the overall project.

Labour requirements for the cementation plant were as follows:

	No. operators/shift	Number shifts	Days/week	Total number operators
Cementation (also laboratory, control room and leaching)	1	1	7	4
Filtration (maintenance)	1	2	5	2
Supervision (security, planning and leaching section)	1	1	5	1
		Total Recovery Plant		7
Leaching (and maintenance)	1	1	5	1
		Total in project		8

Solvent extraction-electrowinning

As in the case of cementation, operators were required to perform several activities in different unit operations. The SX circuit requires a closer supervision than the cementation launders, but still operators, would have some idle time. It was proposed that they would also perform routine analysis in the laboratory and watches in the control room, but unlike cementation, they would not be available for work in the leaching section.

In electrowinning, it was necessary to have one operator per shift in the cellhouse, and part of his activities could be interchanged with personnel from the SX plant, to free an operator for additional work in the leaching section.

Cathode stripping was handled by two operators during one shift. One supervisor was included in the estimate of the recovery plant, but was responsible for the whole project.

Labour requirements for the SX-EW plant were as follows:

	No. operators/shift	Shifts/day	Days/week	Total number operators
Solvent extraction (laboratory, control room)	1	3	7	4
Cellhouse (and SX)	1	3	7	4
Cathode stripping	2	1	7	3
Supervision	1	1	5	1
			Total in SX-EW plant	12
Leaching	1	1	5	1
			Total operators in project	13

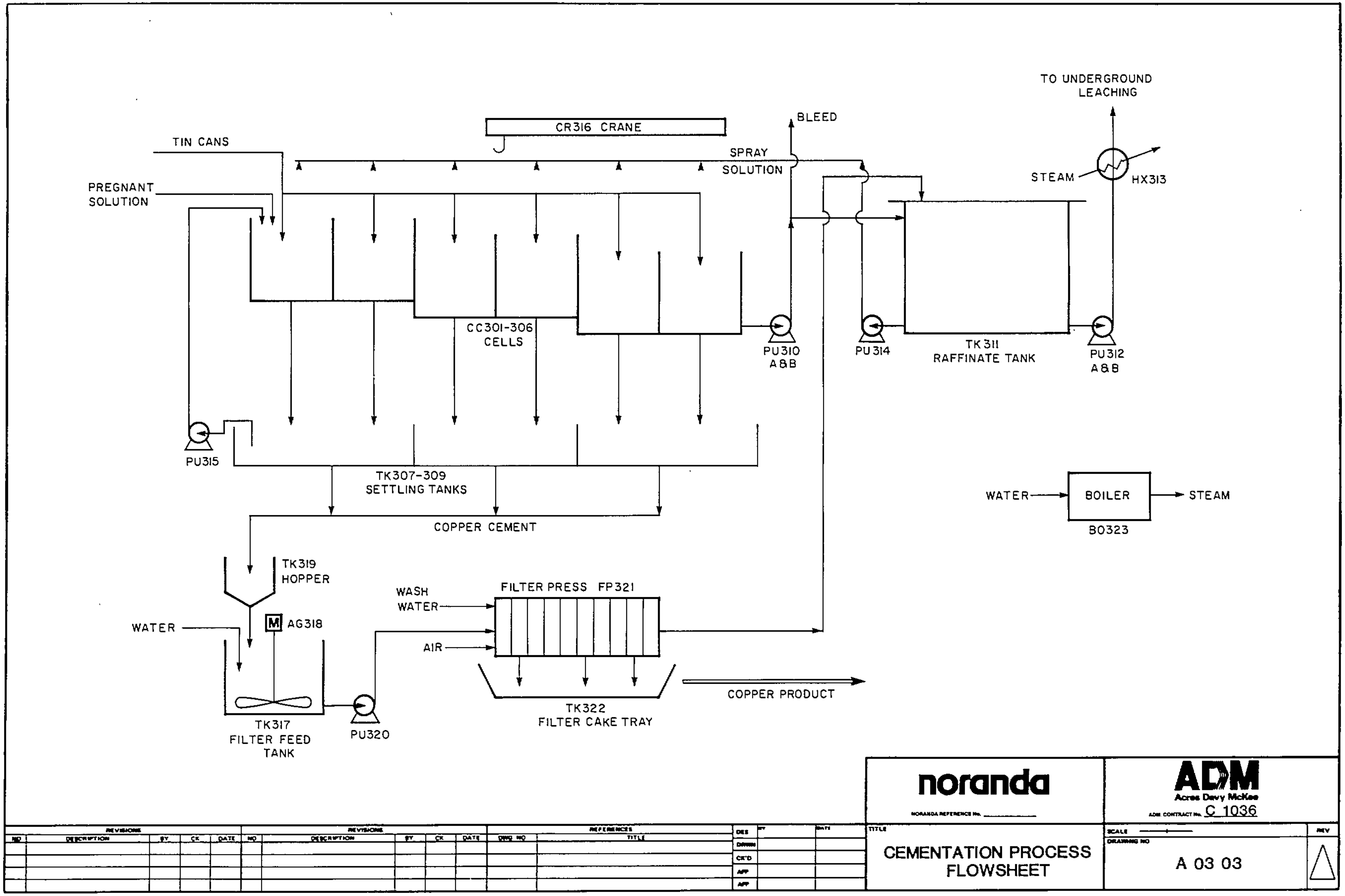

Fig. 5 — Cementation plant flowsheet and equipment diagram

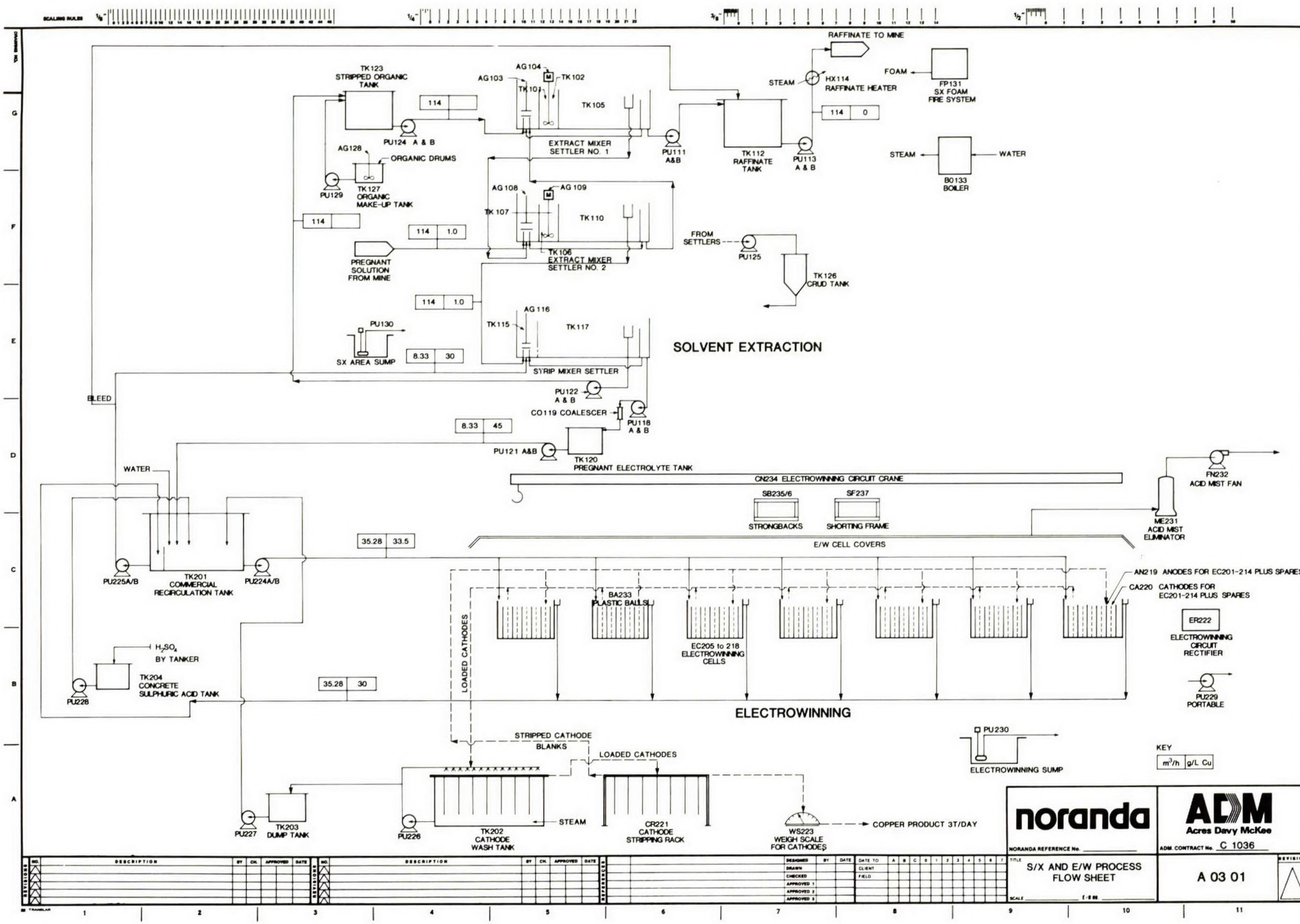

Fig. 6 — SX-EW flowsheet and equipment diagram

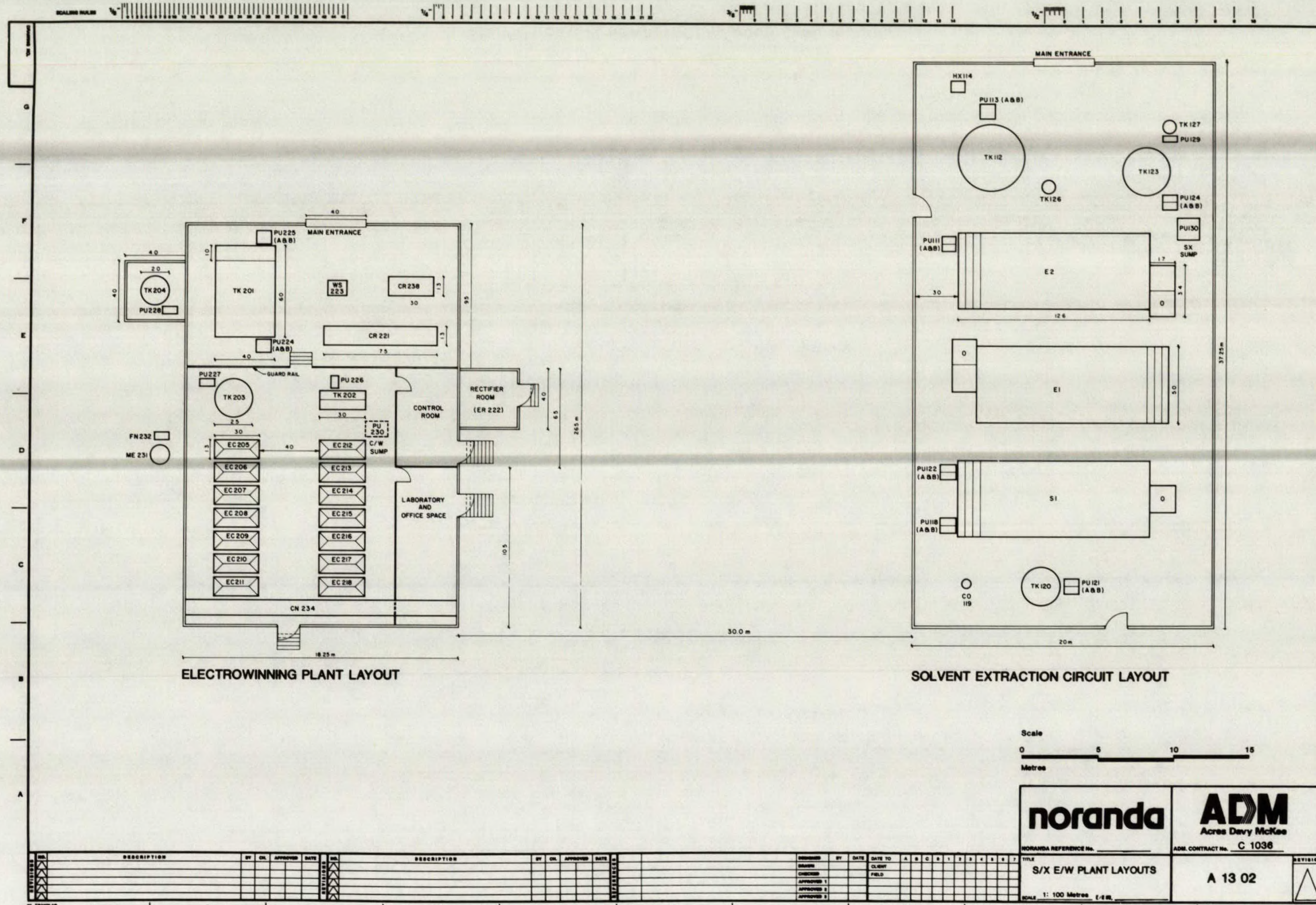

Fig. 7 — SX-EW plant layout

ECONOMICS

Noranda sub-contracted Acres Davy McKee (ADM) to cost the underground pumping and piping system and the cementation and SX-EW recovery plants. All costs were based upon 1985 Canadian dollars. Operating cost estimates for the leach plant and effluent treatment plant were not included in ADM's estimates and were carried out by Noranda Research Centre.

The recovery of copper by solvent extraction-crystallization was also costed by the ADM. This alternative was not considered in this evaluation because it does not appear to offer major advantages in recovery costs, and does not offset the high mining development costs.

Capital Cost Estimate

Leaching section

The solution distribution system drawing of the Geco orebody and equipment requirements described in the previous section, were used to estimate the capital cost of the leaching section. The cost breakdown for the initial capital investment is given in Table 8. Additional annual capital costs for commissioning new stopes (up until year 10 of the project) are shown in Table 9. Other plant components, costs (such as the electrical installation), instrumentation, laboratory, and control room were estimated using factors on the major equipment investment. A start-up cost equivalent to 10% of the fixed capital investment was incorporated.

Table 8 — Capital costs of leaching section

				Cost $
Direct costs*				
1.	Piping, includes	a.	Recovery Plant to Stopes	
		b.	Pregnant Solution to Surface	
		c.	Solution Distribution	
		d.	Sprinklers	475,000
2.	Pumps, includes	a.	Solution Transfer Pumps to Surface 3 stages, 2-150 hp pumps each, one stand-by	
		b.	Solution Recirculation	233,000
3.	Bleed & neutralization circuit, includes	a.	Neutralization Tank	
		b.	Solution Transfer Pump	
		c.	Agitator, piping	40,000
4.	Valves			15,000
			Total installed equipment	763,000
5.	Pond for neutralized iron residue ($5/t residue)			150,000
6.	Electrical (6% of initial cost)			46,000
7.	Instrumentation (4% of initial capital)			30,000
8.	Control room (5% of initial capital)			38,000
9.	Laboratory (8% of initial capital)			61,000
				1,088,000
10.	Start-up costs (10% of fixed capital investment)			108,000
			Total direct costs	1,196,000
Indirect costs				
	Field indirect costs (3% of direct costs)			36,000
	Engineering and construction management (15% direct costs)			179,000
	Construction insurance 1 year			14,000
	Contingency 15%			211,000
			Total indirect costs	440,000
			Total capital cost leaching section	$1,636,000

*Costs of sumps and drain ditches have been estimated in the Mine Development Section.

Table 9 — Additional capital costs per year of stope commission

Pipes and couplings	$ 4,000
Pumps (2)	9,600
Sprinklers and attachments	400
Valves	500
	$14,500
Contingency (15% of direct costs)	2,175
	$17,000

Estimate

The total capital cost for the leaching section was estimated to be $1.64 M. This value contains a contingency allowance of 15%.

List of exclusions

The following items were not included in capital cost estimate:

— power lines or water supply to site; (These are understood to be available from the mill and mine operations.)

— all sumps and drains; (These have been considered as part of the mining development cost.)

— taxes or duties;

— financing during construction;

— working capital, spare parts or tools.

Recovery plant

The plant drawings for cementation and solvent extraction shown in Figures 5 and 6, and equipment accounts described in the previous sections were used to estimate the capital cost of the recovery plant.

Capital cost estimates exclude solution heating equipment, because it was estimated that the heat of reactions in leaching and the heat losses in the pumping of the PLS to surface would result in an average temperature of the pregnant solution of 22°C. Although the temperature was lower than desired for the operation of the SX circuit, it was found acceptable and resulted in significant cost reductions, particularly in the operating costs.

Estimate

The total expected capital cost of a 1000 t/y copper recovery plant was calculated to be $1.73 M and $5.09 M for the cementation and SX-EW alternatives, respectively. The cost breakdown for SX-EW was $2.82 M for the solvent extraction section and $2.27 M for the electrowinning section.

List of exclusions

The following items were not included in capital cost estimate:

— extensive site preparation;

— power lines and water supply to site;

— storage of cement copper or cathodes;

— taxes or duties;

— working capital or tools;

— start-up costs;

— financing during construction.

Operating Costs

Leaching section

Operating costs of the leaching section, including bleed neutralization and residue disposal, have been estimated at 12.03¢/lb of produced copper in the case of a SX-EW recovery plant, and 12.67¢/lb copper for the alternative with a cementation plant. A summary of the operating costs for both alternatives is given in Table 10.

Major operating costs in the leaching section were the pumping of the pregnant leach solution to surface and reagent consumption. A labour component equivalent to one operator has been included in this estimate, although this operator would not be required full-time in this section.

The reagents costs varied according to which type of recovery plant was adopted. In cementation, a small amount of acid was required to adjust the acidity of barren solution from approximately 0 to 0.3 g/L before it was transferred underground. In the SX-EW option, the acidity of the raffinate was sufficient for direct injection into the leaching section. A breakdown of the operating cost components is shown for both alternatives in Table 11.

Recovery Plant

Cementation

The annual operating costs for the cementation plant were estimated to be $0.86 M or 38.8¢/lb Cu (Table 12).

These costs reflect an increase of scrap-iron costs delivered to Geco from $90 t to $180 t, and exclude the energy requirements for solution heating.

Table 10 — Operating costs — leaching section*

	SX-EW		Cementation	
	$/year	¢/lb	$/year	¢/lb
Labour	45,000	2.05	45,000	2.05
Reagents	34,000	1.55	48,250	2.19
Supplies	5,500	0.25	5,500	0.25
Power	167,000	7.59	167,000	7.59
	251,000	11.44	266,000	12.08
Residue disposal	13,000	0.59	13,000	0.59
Total operating cost	264,000	12.03	279,000	12.67

*1985 Cdn $

Table 11 — Breakdown of operating costs — leaching section*

Cost item			
SX-EW		**Cementation**	
Reagents	$/year		$/year
H_2SO_4		215 t/y × $50/t	10,750
Lime for bleed neutralization		398 t/y × $\frac{1}{0.85}$ × $80/t	37,500
362 t/y × $\frac{1}{0.85}$ × $80/t =	34,000		48,250
Labour			
No. of operators/year = 1	45,000		45,000
Power			
Consumption			
455 hp × 0.745 × $\frac{1}{0.65}$ × 8000 h/y			
= 4.17 × 10^6 kWh/y × 4¢/kWh	167,000		167,000
Supplies	5,500		5,500
Residue disposal			
~1500 t/y (solids) × 2.86 (t/m^3) × $3/t	13,000		13,000
Total operating cost $/year	269,500	$/year	279,000
¢/lb Cu	12.02	¢/lb Cu	12.67

*1985 Cdn $

Table 12 — Recovery plant operating costs — cementation*

	Cementation	
	$/year	¢/lb
Labour	270,000	12.25
Reagents	450,000	20.45
Supplies	62,000	2.81
Power	15,000	0.68
	797,000	36.19
Supervision and planning	58,000	2.64
Total operating costs	856,000	38.83

*1985 Cdn $

Solvent extraction-electrowinning

The annual operating costs for the solvent extraction-electrowinning plant were estimated to be $0.86 M or 38.80¢/lb Cu. Details are presented in Table 13.

Cementation vs solvent extraction-electrowinning

The total capital cost of this project (leaching and recovery) has been estimated to be $3.4 M if copper were recovered by cementation, or $6.7 M if solvent extraction-electrowinning were selected. Capital costs breakdown for the two options is shown in Table 14.

Table 13 — Recovery plant operating costs — SX-EW*

	SX-EW	
	¢/lb	$/year
Solvent extraction		
Labour	8.16	180,000
Reagents	5.22	114,960
Supplies	1.00	22,050
Power	0.74	16,390
	15.12	334,400
Electrowinning		
Labour	14.29	315,000
Reagents	0.77	17,000
Supplies	1.81	40,000
Power	4.17	92,000
	21.04	464,000
Supervision and planning	2.64	58,000
Total operating costs	38.80	856,000

*1985 Cdn $

Total operating costs have been estimated at 50.83¢/lb and 51.52¢/lb for cementation and SX-EW, respectively. A breakdown of operating costs is given in Table 15. In the case of producing copper cement, it was necessary to add costs of transportation of the cement to the smelter at $38.50 t and smelting and refining charges of 13.18¢/lb (which includes transportation). In the SX-EW case, because high-purity cathode is produced in remote locations such as Geco, it was necessary to add cathode transportation costs, estimated at 4.09¢/lb. Thus, the total cost of producing copper by cementation is 66.70¢/lb, and by SX-EW is 54.92¢/lb.

Because the additional $3.4 M capital cost required for the SX-EW option could not be offset by 11.81¢/lb lower production costs, and also because the cementation plant would offer significant technical advantages over SX-EW in adapting to variations in pregnant solution volume and lower copper production, the cementation option was selected for the overall project evaluation in Phase 4.

FACTORS AFFECTING BACTERIAL LEACHING TECHNOLOGY

Because no experimental data exist for this deposit, many important design parameter values were assumed based upon our criteria and available literature. Even after the optimistic assumptions that:

- a 55% copper recovery could be obtained within ten years from a 6-in. diameter chalcopyrite ore;
- sufficient permeability existed within the blasted stopes to allow the 10 $L/m^2/h$ solution application rate;

the following additional parameters were considered essential to the success of the project.

Table 14 — Capital cost comparison of cementation and SX-EW

Cementation		SX-EW	
Leaching	$1,636,000	Leaching	$1,636,000
Cementation	1,731,000	SX	2,819,000
		Electrowinning	2,271,000
Capital cost of project	$3,367,000		$6,726,000

Table 15 — Operating cost comparison of cementation and SX-EW (¢/lb Cu)

Cost item	Cementation		SX-EW
Leaching	12.10		11.44
Supervision and planning	2.64		2.64
Recovery plant — cementation	36.19	SX	15.12
		electrowinning	21.04
Residue disposal	0.59		0.59
Transportation of cement copper	2.06	Transportation of cathodes	4.09
Smelting and refining charges	13.18		
Total production cost	66.8		54.9

Oxygen Availability

It was assumed that there was enough oxygen in the stopes, between oxygen-carrying leach solution and diffusiveness of air into the void spaces of the broken ore in the stopes, to allow proper oxidation of the sulphides in the presence of bacteria. This assumption was made because all mined-out areas of the deposit are connected into the mine ventilation system. If this were not the case, air injection into the lower part of the stope would be required at an additional cost.

Mineralogy

The presence of higher sulphide contents could lead to unacceptably high temperatures (>45°C) within the stopes being leached at the rates of solution application considered for this study, which would result in a decrease of bacterial activity and an interruption of the leaching process until the process re-equilibrated.

It was also assumed that the ore contained a sufficient quantity of acid-consuming material to neutralize the acid produced by oxidation of sulphides. In addition to unacceptably high temperatures, therefore, the presence of higher sulphide content, or lower acid-consuming gangue, or both, would also result in low pH values which would cease upon decreased bacterial activity.

Iron Precipitation

It was assumed that all of the iron introduced into solution, both by the dissolution of sulphide minerals and cementation of copper, would precipitate as basic iron sulphate within the stopes to maintain an equilibrium iron concentration of approximately 4 g/L. It was assumed that the iron precipitate would not coat the ore, which would interfere with leaching rate or create channeling. If this assumption were not the case, a volume of solution equivalent to the iron introduced would require to be bled and treated in an effluent treatment pond at an additional cost of 16¢/lb copper.

Leaching Temperature

Rock temperature at Geco, as in most Canadian mines, is 15°C. Solutions to and from the recovery plant were to be transferred through the mine shaft. The heat balance calculated shows that no external solution heating was required in the form of a submerged combustor, thereby decreasing the operating costs by ~20¢/lb copper.

Production Slowdown

Production slowdown was one of the most overlooked areas of chemical mining. Once the last stopes were commissioned for leaching, the project would enter a period of reduced recoveries. To continue copper production at average costs, it was necessary to accommodate all the unit operations to the progressively smaller tonnages, which meant reducing labour, re-scheduling shifts, and having personnel work at several different tasks. The decision on the type of recovery process, as discussed previously, had also to be made considering the last years of operation.

It was very likely that, based upon economic considerations, the project would be shut down before the end of the projected schedule, and as a result there would be less than the 55% copper extraction from the last groups of stopes to be commissioned.

RECOMMENDATIONS FOR COST REDUCTION

The optimization of the leaching and recovery process was outside the scope of this project. The cementation process was selected on the basis of economical and practical grounds. One change that could significantly reduce costs would be to install the cementation launders underground and to haul the cement copper to surface. It was worth noting that 87% of the major equipment costs and 66% of the operating costs were dedicated to pumping the 1 g/L copper solution to surface.

The cementation plant for 1000 t/y copper production was estimated to be 23 m wide × 30 m long × 6 m high and could be installed within a worked-out area of two stopes.

For further details, consult Noranda's "Milestone 3" Report (Micromedia #MON 86-01474/4 Fiche).

PHASE 4 — OVERALL ASSESSMENT AND SENSITIVITY ANALYSIS

In this phase, the costs calculated for the mining development and the metallurgical circuits were used to establish the cash flows of the project for variations in:

- price of copper
- operating costs
- leaching rate and overall recovery
- grade of ore.

In the base case study, cumulative net present values (NPVs) were estimated using a 15% discount rate, and a comparison between these values was then used in the sensitivity analysis. Cash flow schedules spanned a 19-year period (total time required to leach the ore) in constant 1985 Canadian dollars. Taxation was estimated at 40% of positive cash flows.

Two other case studies were presented to help clarify some of the more striking conclusions made in the study of bacterially assisted underground leaching.

In Case 2, it was assumed that the ore grade was 1.8% Cu, double the value used in the base case, which resulted in a 50% reduction in mining development costs.

In Case 3, in addition to improving the ore to 1.8% Cu, it was assumed that; (i) the project could obtain 75% overall copper recovery; (ii) pregnant leach solutions could be concentrated to 2 g/L Cu (instead of 1 g/L used in the base case), so that the recovery plant could double its capacity for a slight increase in the capital cost; and (iii) unit mining costs could be reduced by ~30% using larger drilling equipment.

These two cases (particularly the latter) should be regarded as optimistic, but they highlight the cost of this technology even under the best circumstances.

Conventional mining costs and cash-flow schedules were also calculated so that a comparison between the two technologies could be made.

Both leaching and conventional mining calculations were based on the fact that the zone to be mined is part of an operating mine and that, consequently, milling facilities exist. As a result, the mining development costs in the leaching case were reduced and the need for any capital expenditures in the conventional mining case were eliminated.

ECONOMICS OF CASE STUDIES

Base Case Study

The main variables assumed for the base case study were:

—	Leaching ore	2.1 million tonnes
—	Ore grade	0.9% Cu
—	Total copper recovery	55% (10 356 mt)
—	Rate of copper extraction	10% first year 1% less each subsequent year
—	Total leaching time	18 years
—	Pregnant leach liquor concentration	1 g/L Cu
—	Designed annual copper production	1000 mt
—	Mine development cost	\$14.74/t
—	Concentrate transportation	\$38.58/t
—	Smelter deductions	11 kg*/t of concentrate
—	Smelting and refining charges for cement copper	\$0.29/kg
—	Leaching and recovery plant capital (cementation route)	\$3.37 M
—	Leaching and recovery plant operating cost (cementation route)	\$1.13/kg Cu (51.5¢/lb Cu)
—	Price of copper	\$2.20/kg Cu (\$1/lb Cu)

Under these base case conditions, the NPV was estimated to be −\$22.1 million. A zero NPV (which implies a 15% ROI) is only obtained by:

- reducing the overall operating costs by 87% to 19¢/kg Cu (8.6¢/lb Cu) (see Fig. 1);
- increasing the selling price of copper to \$9.22/kg Cu (\$4.18/lb Cu) (see Fig. 2).

These figures indicate clearly that bacterially assisted in-place leaching of chalcopyrite is not economically viable under the base case conditions.

The effect of the mining costs on the NPV was also examined. The analysis shows that even if the mining costs were reduced to zero, the NPV of the base case would still be negative (−\$0.5 million), indicating that at current copper prices, not even the leaching and recovery circuits could be operated profitably for an in-place bacterial leaching project.

*Cost is dependent on smelter tolling charges.

Contrary to what would be expected, the effects of ore grade (0.9 to 1.8% Cu) and overall copper recoveries (55-100%) at constant leaching rates on the NPV show that neither of these variables significantly affect the economics of in-place bacterial leaching.

Based upon these analyses, it was concluded that the most significant variable affecting the economic viability of this project is the slow (and incomplete) copper extraction that results in slow realization of revenues. In effect, if the cash flow were discounted as in this study, the 55% copper recovery would be equivalent to a present recovery of only 28.9%.

The negligible effect of higher ore grade and overall recoveries on the NPV was further exemplified by the other two studies.

CASE 2 Study

The main variables differing from the base case study (Section 2) were:

—	ore grade	1.8% Cu; and
—	leachable ore	1.1 million tonnes

All other variables were kept constant.

The main advantage of this over the base case is that the number of stopes required for leaching are reduced by 50% thus reducing mining development costs by 50%.

The NPV for this case was estimated to be −\$11.5 million. A zero NPV is obtained by increasing the selling price of copper to \$6/kg (\$2.72/lb).

CASE 3 Study

The main variables differing from the base case study were:

—	ore grade	1.8% Cu
—	leachable ore	1.1 million tonnes
—	total copper recovery	75%
—	pregnant leach liquor concentration	2 g/L Cu
—	mine development cost	\$10.58/t
—	leach/recovery operating cost	\$1.20/kg Cu (54.4¢/lb Cu)

The reduction in mine development costs from \$14.74 to \$10.57/t was based upon the use of 152-mm drill holes instead of 114-mm drill holes for the base case.

The main advantages for this case over the base case study were:

— The unitary mining development costs are reduced by 27%.

— The overall mining development costs are reduced by 62%.

— The leaching circuit operating costs are reduced by pumping a more concentrated solution to surface.

— The annual rate of copper recovery and overall extraction are increased over the ten-year period.

The NPV for this case was estimated to be −\$8.1 million. For this project to be viable, even under these highly (or overly) optimistic conditions, the price of copper would have to be \$3.63/kg (\$1.65/lb).

CONVENTIONAL MINING

The economics of producing copper by conventional mining and milling were also examined. For this study, it was decided to use the same ore grade and production tonnage as for the base case. As is the case at Geco, it was assumed that the project would not require any capital and that the existing mill could process the ore hauled to surface.

Unitary mining development costs of \$17.76/t were used for conventional mining instead of \$14.74/t used for the in-place leaching project. This difference was largely a result of the credit from the copper recovered in the mill from development ore for the in-place leaching project (removed to allow for swelling). Costs for milling, smelting, refining, and transportation were the same as those used in Phase 2 and in the base case of this report.

It was estimated that the 2.1 million tonnes of ore could be mined conventionally at Geco within six years so that the cash flow schedule for this case only spans six years.

It was shown that this case generated an NPV of −\$25.3 million; only −\$3.2 million less than the base case for bacterial leaching. A zero NPV was obtained, however, by:

— reducing the overall operating costs by 45% (see Fig. 6); or by

— increasing the selling price of copper to \$3.94/kg Cu (\$1.8/lb Cu) (see Fig. 7).

The effect of ore grade on conventional mining NPV's was also examined and is shown in Figure 4. It can be seen that, unlike bacterial in-place leaching, ore grade has a significant effect on the economics of the project, reaching an NPV of zero at an ore grade of only 1.9% Cu.

COMPARISON OF THE BASE CASE AND CONVENTIONAL MINING

Figure 6 shows that conventional mining is much more sensitive than in-place bacterial leaching to variations in

Table 16 — Conditions required for conventional mining and in-place bacterial leaching to generate a zero NPV*

Conventional mining	In-place bacterial leaching
— At an ore grade of 0.9% Cu, a selling price of $3.94/kg Cu is required.	— At an ore grade of 0.9% Cu, a selling price of $9.22/kg Cu is required.
— At a Cu selling price of $2.20/kg minimum ore grade required is 1.9% Cu.	— At a Cu selling price of $2.20/kg Cu, ore grade does not have a significant effect on NPV's within the limits investigated.
— At a copper ore grade of 0.9% and selling price of $2.20/kg Cu, total operating costs should be reduced by 45%.	— At a copper ore grade of 0.9% Cu and selling price of $2.20/kg Cu, the total operating costs (mining, leaching and recovery) should be reduced by 87%.

*A zero NPV requires 15% ROI (see text).

copper price, operating costs, and grade. Table 16 summarizes the conditions required for both techniques to generate a zero NPV and thus a 15% ROI.

Although neither conventional mining nor in-place bacterial leaching of low-grade sulphide ores are economically attractive under present economic conditions, the former requires lower copper prices and ore grades to generate a zero NPV. It should be stressed again that this comparison is only valid when a mine and mill exist and have been amortized.

It is difficult to estimate what potential exists for reducing the operating costs of either conventional mining or in-place bacterial leaching to a point that could make each alternative economically viable. It could be argued that it is more likely that greater cost reduction will be obtained in an emerging technology, such as in-place bacterial leaching, than in one that has existed for years. However, because up to 66% of the operating costs for in-place bacterial leaching are due to the mining development needed for adequate rubblization of ore, it should be expected that major improvements in in-place bacterial leaching, resulting from more economic rubblization methods, may also reduce the costs of conventional mining.

Only if the leaching aspects are improved significantly, particularly the rate of copper extraction, could in-place bacterial leaching technology become more attractive than conventional mining technology.

The overall assessment and sensitivity analysis was summarized in an excellent manner by Ismay et al. in their paper, and are reproduced here *ad verbatim*.

COST ESTIMATES

The scope of the contract was to examine IPBL for a deposit within the domain of an existing mine. This scenario had the advantage that the existing mine shaft and drilling equipment could be used in the development of stopes for leaching, and capital investment would only be required for the leach and recovery plant. In the case used for this study at Geco, there was an operating mill, and it was assumed that the development ore could be concentrated on site and sold as copper concentrate.

As a result of these specific factors, all mining development costs were considered as an operating cost, and the revenues obtained in the mill by producing copper concentrate from ore removed to allow for swelling during the rubblization of the stopes for leaching, were considered as credits to the mining development costs. In consequence, the average development cost of $16.09/t (Table 17) of ore to be leached, which was estimated from the design of the three stopes described above, was reduced by $1.35/t. This credit was calculated on the basis that it costs $0.5, $1.1, and $5.5 to muck, hoist, and mill each tonne of ore removed in the mining development phase, respectively, and that to recover the copper, it is necessary to cover concentrate transportation costs of $38.50/t and smelting and refining charges of $0.85/kg copper in concentrate. A copper price of $2.20/kg was used to calculate the credit from development ore.

On the basis of the stope commissioning plan development for this project, (see Table 7), total mining costs will be $5.2 M/y in the first four years and $2.6 M/y from years 6 to 9 of the project.

Order of magnitude capital cost estimates (±25%) were developed for the leaching section and the two recovery plant alternatives. Major equipment was sized and costed by subcontract to Acres Davy McKee, and other services were factored on the installed equipment costs. The capital cost breakdown for each section is shown in Table 18.

Indirect costs include engineering and construction management, construction insurance, and a contingency factor. The capital cost required in the first year of the project is estimated to be $3.4 M for the cementation plant option and $6.7 M for the SX-EW plant option. These costs do not include solution heating equipment

Table 17 — Mining development costs

Development item	Access	Overcuts	Undercuts	Raise	Sublevel development	Slot-drilling blasting	Stope drilling	Stope blasting
$/tonne of leachable ore	3.45	2.34	0.20	2.20	1.84	1.74	1.71	2.61

Total mining and development cost	$16.09/t
Credit from development ore	$ 1.35/t
Net cost of mining	$14.74/t

Table 18 — Capital costs for in-place bacterial leaching ($ × 1000)

Leaching section Direct costs	
1. Piping	490
2. Pumps	233
3. Bleed circuit	40
Installed equipment	763
4. Iron residue pond	150
5. Electrical (6% of IE)	46
6. Instrumentation (4% of IE)	30
7. Process control (5% of IE)	38
8. Laboratory (8% of IE)	61
Fixed capital	1,088
9. Start-up costs (10% FC)	108
Total direct costs	1,196
Indirect costs	440
Total	1,636

Recovery plant section Direct costs	Cementation	Solvent extraction-electrowinning SX	EW	Total
1. Sitework, structure, foundations and building	670	805	663	1468
2. Equipment	370	777	500	1277
3. Electrical	100	128	215	343
4. Instrumentation	25	50	100	150
5. Piping	75	250	110	360
6. Miscellaneous	25	50	50	100
Total direct costs	1,265	2,060	1,638	3,698
Indirect costs	467	759	633	1,392
Total	1,732	2,819	2,271	5,090

or ponds for continuous disposal of iron or neutralization residues for the cementation and SX-EW options, respectively, which would increase substantially both the capital and operating costs.

The operating costs for the leaching section and the two recovery plant alternatives are shown in Table 19. Because personnel in this project must perform more than one duty, maintenance labour was included in the labour costs for leaching. The three major costs were:

— labour (principally in the SX-EW option);

— reagent (particularly scrap iron in the cementation option);

— power, most of which is used for pumping the pregnant solution to surface.

It can be seen that the cost of producing cement copper (85% Cu) is about the same as for the production of

Table 19 — Operating costs for in-place bacterial leaching (¢/kg copper)

Leaching section: Cost component	Cementation	SX-EW
Labour	4.51	4.51
Reagents	4.82	3.41
Supplies	0.55	0.55
Power	16.70	16.70
Residue disposal	1.30	1.30
	27.9	26.8

Recovery plant: Cost component	Cementation	SX	EW	Total SX-EW
Labour	26.95	17.95	31.44	49.39
Reagents	44.99	11.48	1.69	13.17
Supplies	6.18	2.20	3.98	6.18
Power	1.50	1.63	9.17	10.80
	79.62	33.26	46.28	79.54
Supervision	5.84			5.84
	85.4			85.4

Unit costs of reagents on-site		Other unit costs	
Sulphur acid	$50/tonne	Electricity	4¢/kWh
Lime	$80/tonne	Residue disposal	$3/tonne
Scrap iron	$180/tonne	Labour	$45,000/year

cathode copper (>99.9% Cu). However, costs of transporting cement copper to a smelter at \$38.50/t and smelting and refining charges of 29¢/kg for copper must be added to the cementation option, raising its total production cost to \$1.47/kg. A freight charge of 9¢/kg to transport cathodes to market must be added to the SX--EW option, raising the cost to \$1.21/kg. Both costs are exclusive of mining costs.

ECONOMIC ANALYSIS

The cash flow for production of ~10 000 t/y of copper from 3.8 M t of ore grading 0.9% Cu, was calculated for the production schedule shown in Table 3, using the cementation alternative for the recovery plant. To simplify calculations, it was assumed that the unit leaching and recovery costs were constant throughout the life of the project. The cumulative net present value (NPV) was then determined for a 15% discount rate, in constant 1985 Canadian dollars. A 40% tax rate was applied to all positive, year-end cash flows instead of a more complicated formula incorporating capital cost allowance and depreciation.

The NPV for the IPBL project, under the base case conditions (55% copper recovery in ten years, 0.9% Cu grade, 21 M t of leachable ore) and assuming a copper price of \$2.20/kg, is −\$22 M. A reduction of 87% of all operating costs or an increase in the price of copper to \$9.22/kg would be required to achieve a zero NPV.

For comparison, the NPV for the conventional mining of this low-grade ore under the conditions existing at Geco (also assuming a copper price of \$2.20/kg) was estimated to be −\$25 M. However, in this case, a zero NPV is obtained for a 45% reduction of all operating costs or an increase in the price of copper to \$3.94/kg.

These figures show that neither IPBL nor conventional mining are economically feasible, but that the latter is more sensitive to parameters that affect copper production. It should be stressed that, in the case of conventional mining, it was assumed that no capital was required for the mine shaft and mill, and that ten stopes could be mined each year. Costs and charges used to estimate the cash flow for this case are \$17.76/t ore for mining, \$5.51/t ore for milling, \$38.50/t concentrate as freight, and a smelter deduction of 11 kg Cu/t concentrate.

The sensitivity of in-place bacterial leaching and conventional mining was examined for variations in ore grade, operating costs, leaching rate, and the price of copper. Figure 8 shows that for very low grades, the NPV of IPBL is less negative than conventional mining, but that the latter is much more sensitive to ore grade. Although this last fact has been claimed as an advantage for recovering copper from low-grade deposits by in-place leaching, it can only have a positive effect if the project is economically feasible.

Other observations in Figure 8 are:

— At \$2.20/kg Cu, conventional mining is economically feasible only for ore grades higher than 2% Cu.

— Faster rates of leaching improve the NPV of IPBL, but the incremental difference is much more significant when reducing the leaching period from ten to five years than for subsequent reduction.

— IPBL is not economically feasible even when no capital costs are incurred for the leaching section and the recovery plant.

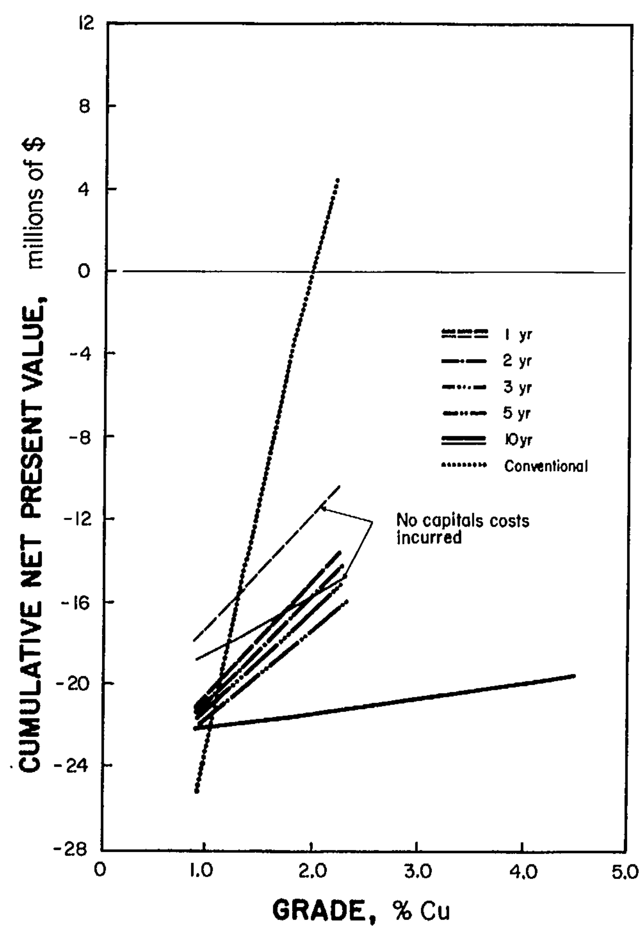

Fig. 8 — Comparative sensitivity analysis: copper grade

These comparisons indicate that, although the rate of copper recovery in IPBL is slow, which results in a slow realization of revenues, the major cause that hinders its economics are the high mining development costs.

An estimate of all the operating costs over the life of the project shows that 66% correspond to the mining development phase, whereas only 5% are attributed to leaching, and 29% to the recovery section.

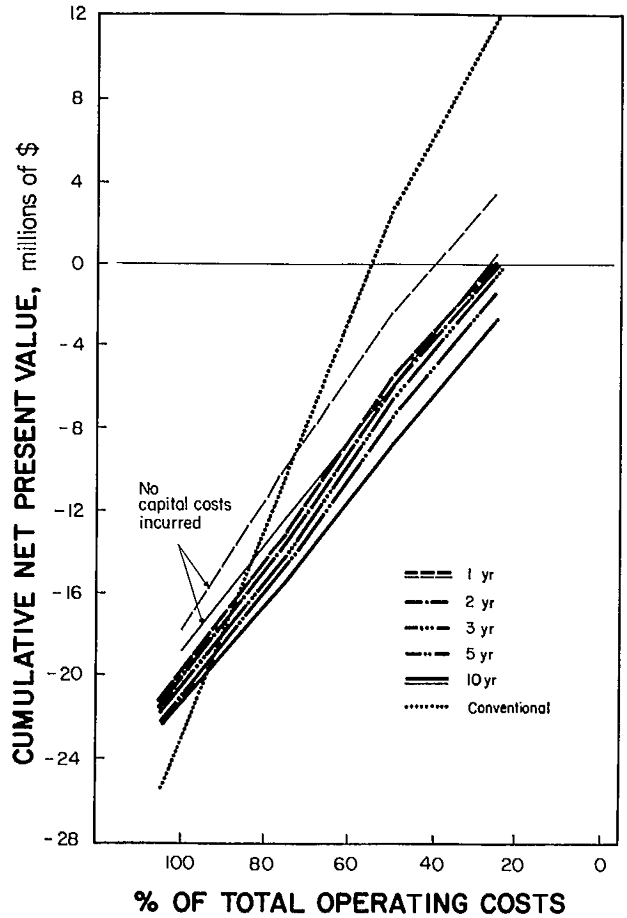

Fig. 9 — Comparative sensitivity analysis: total operating costs

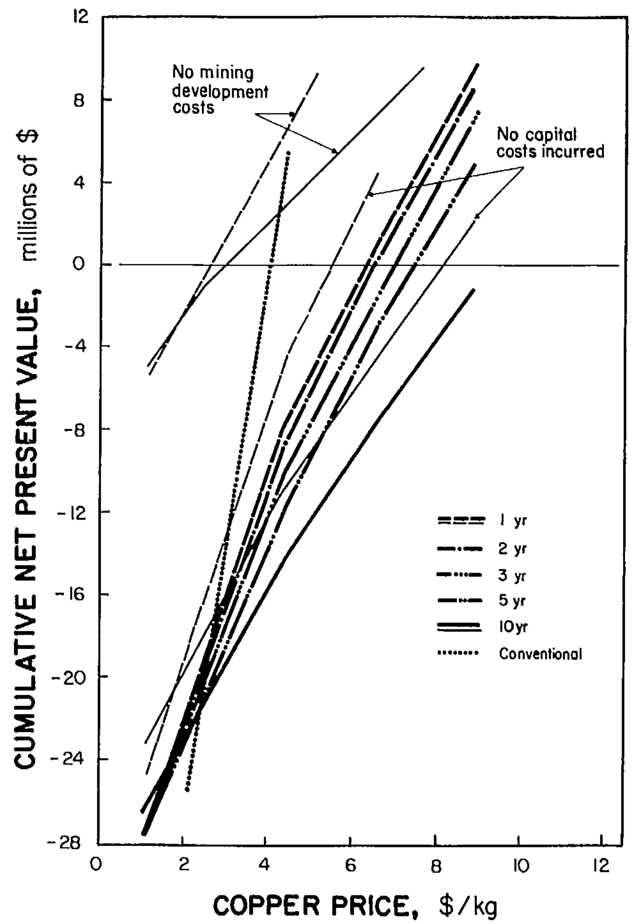

Fig. 10 — Comparative sensitivity analysis: copper price

The relative effect between the reduction of operating costs and the rate of leaching on the NPV (Fig. 9) shows that improvements in this parameter will not render the project attractive as long as there are high costs associated with mining development. The influence of the high costs in the preparation phase is more evident in Figure 10, which shows that a project that did not require mining development costs would be economically feasible at a copper price of $2.80/kg even when capital expenditure is needed for the leaching section and the recovery plant, and the rate of extraction, as in the base case, is 55% over ten years.

RECOVERABLE COPPER RESERVES

The amount of copper to be recovered from the 3.8 M t orebody is expected to be 16 000 t in the IPBL project and 21 000 t if mined conventionally (assuming 85% recovery of mined material and 50% overall recovery in leaching). Furthermore, of the total tonnage produced in IPBL, 45% will be recovered from the development ore in only six years.

CONCLUSIONS

This study has shown that IPBL is not economically feasible for recovering copper from a 3.8 M t ore zone with a 0.9% Cu grade, 1000 m below surface, within the boundaries of an existing mine with an operating mine shaft and a mill. The capital cost required to recover 1000 t/y Cu (assuming 55% recovery over ten years), is estimated to be $3.4 M and the operating costs/kg copper are $2.98 for mining development, $0.28 for leaching, and $1.19 for the recovery plant.

The sensitivity analysis has shown that faster rates of leaching, increased recovery, and higher ore grade have little effect on the economics of IPBL. The principal reason is the high,, mining development costs that are only marginally lower ($14.74/t) than conventional mining ($17.76/t) for which revenues for 85% of the contained copper are realized within three months, whereas IPBL entails a delay of several years to achieve only a 55% copper recovery.

Although these costs were established for a specific case at Geco, they are applicable to most Canadian mines. The fracturing of a highly consolidated orebody in preparation for in-place leaching requires that part of the ore be removed (to allow for swelling during blasting) and milled or leached on-site (depending on whether or not a mill exists).

At present, there are no known mining methods that could reduce the development cost of deep, disseminated, sulphide deposits to a point of making IPBL economically feasible. Furthermore, it must be kept in mind that less-expensive methods of fracturing rock in place could also benefit conventional mining.

The application of IPBL to mined-out stopes (roofs and walls) for copper recovery is also not feasible because, in most cases, these stopes contain limited copper tonnages that would not justify a major investment in a new recovery plant.

The technical problems associated with IPBL were:

— There is no proven inexpensive reagent that dissolves copper and precious metals simultaneously.

— The concentration of copper in the pregnant solution pumped to surface (1-4 g/L Cu) is lower than that in the ore hoisted to surface.

— The existence of large amounts of pyrite and/or pyrrhotite can lead to excessively high temperatures that are detrimental to the microorganisms and that generate large amounts of iron (which have to be precipitated) and acid (which requires neutralization).

The conclusion of this study is that in the short term, IPBL should only be considered for deposits that:

— require no conventional mining during the development phase;

— have a hydrometallurgical recovery plant on-site;

— have mixed oxide-sulphides;

— have a relatively small amount of iron.

In the long term, research in bacterial leaching should be coupled with the development of less-expensive ore fragmentation methods and underground recovery systems.

Further details on this phase may be obtained from Noranda's "Milestone 4" Report (Micromedia #MON 86-01474/4 Fiche).

GLOSSARY OF TERMS

Alimak Raise: (See Raise) A raise driven using a retractable platform that runs on rails attached to the host rock.

Back: The roof or overhead surface of an underground excavation.

Benching: A sequential method of blasting.

Bulkhead: A barrier constructed in an underground opening to restrict access or the flow of material.

Burden: The distance between consecutive rows of drill holes.

Crosscut: A nearly horizontal or horizontal underground drift to intersect an orebody, normally used for access and, subsequently, for drilling and blasting.

Charge: The density (w/w) of explosive used to fragment rock.

Diabase: A hypabyssal rock of the composition of gabbro, but having an ophitic texture and consisting of labradite laths in a matrix of augite, with magnetite a common accessory.

Dragfold: Used to define adjacent anticlinal (concave) and synclinal (convex) folds on the limb of a larger fold.

Drawpoints: A place where ore can be loaded and, subsequently, removed from the bottom of a stope.

Drift: Horizontal or nearly horizontal underground opening typically used as a roadway.

Dyke: A discordant tabular body of igneous rock that cuts across the structure of the adjacent country rocks, and which usually has a high angle of dip.

Incompetent: Soft or fragmented rocks in which an opening, such as a borehole or an underground working place, cannot be maintained unless artificially supported by casing, cementing, or timbering.

Lenticular: Shaped approximately like a double convex lens.

Muck: Broken ore or waste.

Mucking: The removal of muck from a stope or development drift.

Overcut: An opening made on the top of a stope for drill set-up.

Raise: Underground opening that is driven upwards from one level to another by one of a variety of methods.

Rockbolts: Long, self-anchoring steel bolts that are inserted into pre-drilled holes in rock and secured for the purpose of ground control.

Rock Quality Design: The sum length of all pieces of core greater than, or equal to, twice the core diameter divided by the total length of core recovered from a diamond drill hole. RQD is usually expressed as a percentage.

Schist: A rock that occurs in thin layers.

Sericite: A white potash mica very similar to, if not identical with, muscovite mica.

Slash: Increase the dimensions of an underground opening with further drilling and blasting.

Slot: Vertical or inclined ore section that is excavated in a stope to allow for further blasting and the resulting ore swell.

Slot Raise: A raise driven where the slot is to be opened up to allow for blasting of the slot.

Spacing: The distance between the toes of two drill holes.

Stope: An underground excavation that is made by removing ore from the surrounding rock.

Sub-level: A system of secondary horizontal openings situated between main levels, developed for drilling and blasting of stopes.

Syncline: A fold in rocks in which the strata dip inward from both sides toward the axis.

Synform: A term used to describe a syncline when the stratigraphical succession is unknown.