Information Circular 9420

Mechanical Excavation Systems

(In Three Parts)

2. Stope Leach and Stope Autoclave Mineral Recovery Systems

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PREFACE

The results of U.S. Bureau of Mines (USBM) research on mechanical excavation mining systems are presented in a three-part Information Circular (IC) series. Part 1 deals with narrow-vein and longwall mining using the USBM-developed radial-axial drill-split tool. Part 2 presents stope leaching and stope autoclave mineral recovery systems using the part-1-described drill-split narrow-vein and longwall mining methods. Part 3 is a review of potential environmental applications for the part-1-described drill-split narrow-vein and longwall mining methods, including the development of city-center "vertical landfills."

The USBM research presented in this series is conceptual in nature. The concepts were deemed technically feasible and costed. The costing of conceptual designs that use technologies that have no commercial track record is necessarily speculative. Wherever possible, cost estimating was guided by the "Bureau of Mines Cost Estimating System Handbook"* and USBM IC "Simplified Cost Models for Prefeasibility Mineral Examinations."** Where these cost sources were not applicable, the best available literature was examined and engineering judgments used. Costs are for comparative use only and in 1984 dollars. Conservative estimates of technology and technology combination productivities were sought for use in the cost-estimating exercises; however, only further development and demonstration of the proposed technologies can validate the cost estimates in the series. Although many components of the proposed mining systems are conceptual, the combination of all component costs that comprise a system should produce a reliable prefeasibility-type cost estimate. Prefeasibility cost estimates are best used in acceptance and rejection decisionmaking; for the mining systems proposed in the series, this level of decisionmaking makes single-component cost tracking inappropriate.

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*U.S. Bureau of Mines. Bureau of Mines Cost Estimating System Handbook (In Two Parts). 1. Surface and Underground Mining, IC 9142, 1987, 631 pp.; 2. Mineral Processing, IC 9143, 1987, 566 pp.

**Camm, T. W. Simplified Cost Models for Prefeasibility Mineral Evaluations. USBM IC 9298, 1991, 35 pp.

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

g/t	gram per metric ton
h	hour
ha	hectare
kPa	kilopascal
kW	kilowatt
L/min	liter per minute
L/s	liter per second
m	meter
m/yr	meter per year
m^2	square meter (area)
m^3	cubic meter (volume)
m^3/d	cubic meter per day
m^3/min	cubic meter per minute
Mm^3	million cubic meters
Mt	million metric tons
Pa	pascal
t	metric ton
t/d	metric ton per day
°C	degree Celsius
$/ha	dollar per hectare
$/t	dollar per metric ton

MECHANICAL EXCAVATION SYSTEMS

(In Three Parts)

2. Stope Leach and Stope Autoclave Mineral Recovery Systems

By John A. Lombardi[1] and Charles V. Jude[1]

ABSTRACT

In this U.S. Bureau of Mines report, in situ vat leach mineral recovery system concepts are described and cost compared with conventional mining and pad and autoclave leach processing systems. In the proposed systems, leachable ores are enclosed in place in reagent-impermeable in situ vat constructions. The ores are then rubbled and flood leached or flood oxidized and leached. The in situ vats are called stope leach or stope autoclave vessels. The report also proposes ore body-adjacent in situ vat constructions containing rechargeable processing stopes.

A series of ore body-specific, innovative, and conventional mineral recovery systems case study designs are described and prefeasibility level costed. Based on a comparison of these prefeasibility level cost estimates, the proposed systems are determined to be economically competitive with conventional mineral recovery systems.

The in situ vat leach mineral recovery systems concepts for leachable and refractory ores are developed around the drill-split narrow-vein and longwall mining methods described in part 1 of this Information Circular series.

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INTRODUCTION

The mission of the U.S. Bureau of Mines (USBM) is to help ensure the availability of minerals to the U.S. economy at acceptable social, environmental, and economic costs. Since many of the minerals crucial to the welfare of the Nation are domestically available only in low-grade, hard-rock resources, the USBM is engaged in the development of new hard-rock mining system technologies. Accordingly, the USBM initiated a broad review of the potential for hard-rock mechanical excavation mining. The specific goal of the research was to identify hard-rock "mining system" potentials unique to the mechanical excavation technique. An example of the kind of potential sought, taken from the soft-rock-coal industry, is the mine design revolution that followed the development of the longwall. In coal, safer, more productive, more economic, higher resource recovery mines resulted from this mechanical excavation innovation. This report identifies similar, revolutionary, mechanical excavator-based mine design concepts for the hard-rock mining of leachable and refractory ore. These hard-rock mining concepts are based on the potential of the USBM-developed radial-axial drill-split mechanical excavator in its narrow-vein and longwall mining modes, as proposed in part 1 of this Information Circular (IC) series.

The proposed mine designs are innovative and undemonstrated, and require proof of concept before technical and economic feasibilities are proven.

Technical descriptions and costs for the proposed drill-split mechanical excavation mineral recovery systems for leachable and refractory ores are presented in the body of the report. The proposed mineral recovery systems include (1) stope leaching, (2) stope autoclaving, and (3) underground leach plants. The systems assume that vertically extensive 0.6-m-wide sidewall channels can be constructed around ore zones using the drill-split narrow-vein miner. These channels, when backfilled with reagent-impermeable compounds, form solution-controlling in situ vat sidewalls. Reagent-impermeable vat bottoms and tops are assumed to be constructable by the drill-split-longwall mining and backfilling of ore zone undercuts and overcuts.

Ore body-specific case studies using the proposed systems were developed and costed. Costs are also developed for the conventional mining and milling of the case study ore bodies. Costs and the timing of expenditures for the proposed and conventional systems were compared to determine the economic feasibilities for the proposed systems. Detailed stope leach-autoclave mineral recovery system and conventional mineral recovery system cost comparators are shown in the tables in the appendixes.

LEACH MINERAL RECOVERY OVERVIEW

Most metals can be dissolved by one or more aqueous chemicals. Once in solution, these metals can be selectively precipitated to effect a recovery of the metal in a highly concentrated form. Metal recoveries from ores using dissolution-precipitation processes are called "leaching" operations. The easiest metals to recover by this method are directly leachable with single-pass reagent contacting; difficult ores require alternate, and/or elevated temperature-and-pressure chemical treatment prior to leach reagent use and metals dissolution. The treatment required by these latter "refractory" ores is accomplished in pressure-retentive reaction vessels called autoclaves.

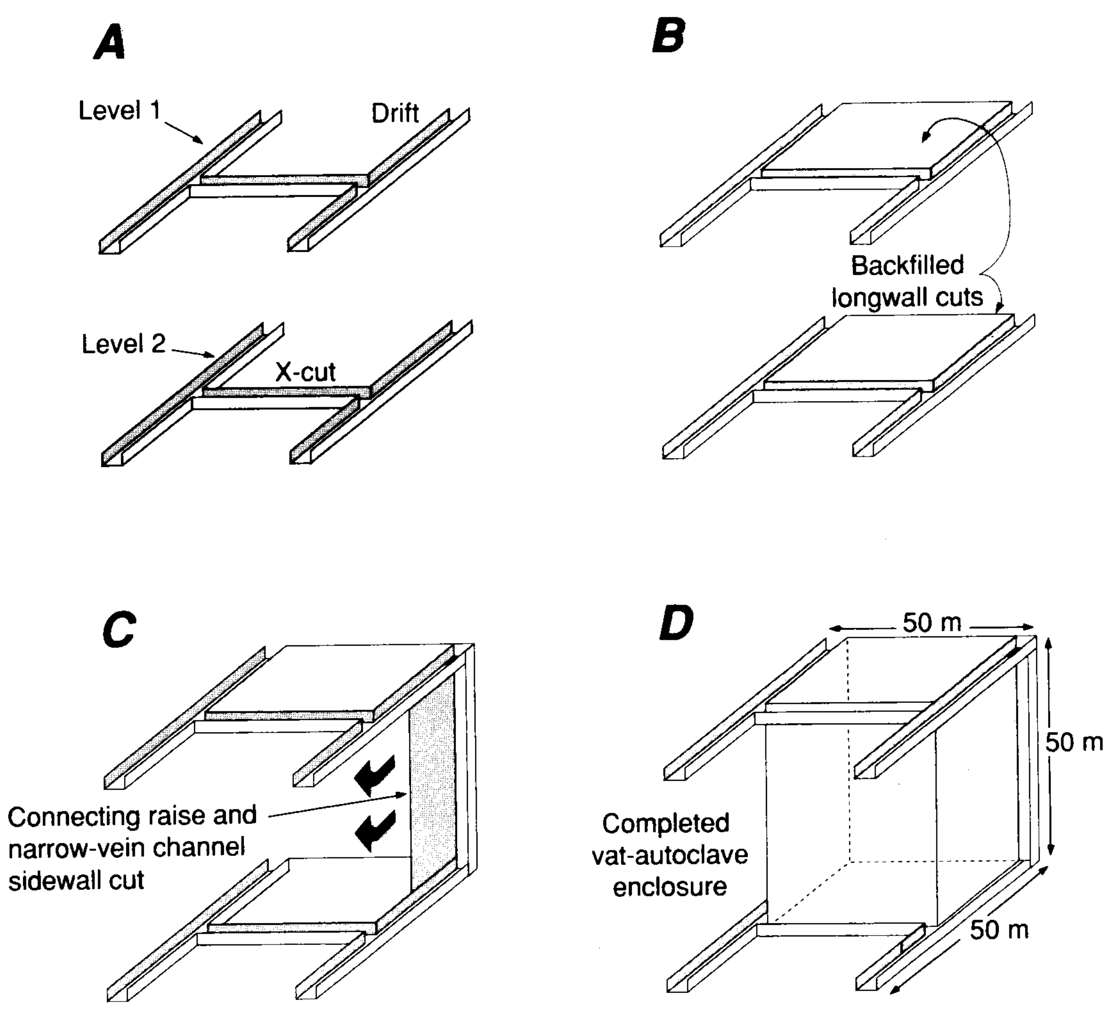
The rate of dissolution of metals from ores is, in part, determined by ease of leach chemical-metals contacting, i.e., a long soak time is required for solution penetration of lump ore—the larger the lump, the longer the time. Ore crushing to facilitate chemical penetration and effect a rapid metals dissolution is the common industrial practice. The solution control and crushing infrastructure required for successful leach processing is usually permanently sited away from, but adjacent to, the active mining area. Run-of-mine ores are hauled and hoisted to these ore body-adjacent processing facilities. The chemicals used for the dissolution of metals are universally toxic, and solution controls are environmentally mandated.

STOPE LEACH MINERAL RECOVERY METHOD

The drill-split narrow-vein and longwall mining methods presented in part 1 of this IC series are proposed to construct in situ leaching vats and autoclaves.

Drill-split narrow-vein methods previously described are proposed to cut 0.6-m-wide vertical channels around ore body zones. Backfilling these cuts with a reagent-impermeable compound forms an ore body-enclosing sidewall. Drill-split longwall methods previously described are proposed to excavate top and bottom cuts to link the sidewalls. Backfilling these cuts with a reagent-impermeable compound creates an ore body-encompassing zone of hydrologic isolation. This rock mass enclosure process is shown graphically in figure 1. Ore body zone encapsulation is the physical infrastructure required for the control of toxic, aqueous leach reagents.

Figure 1

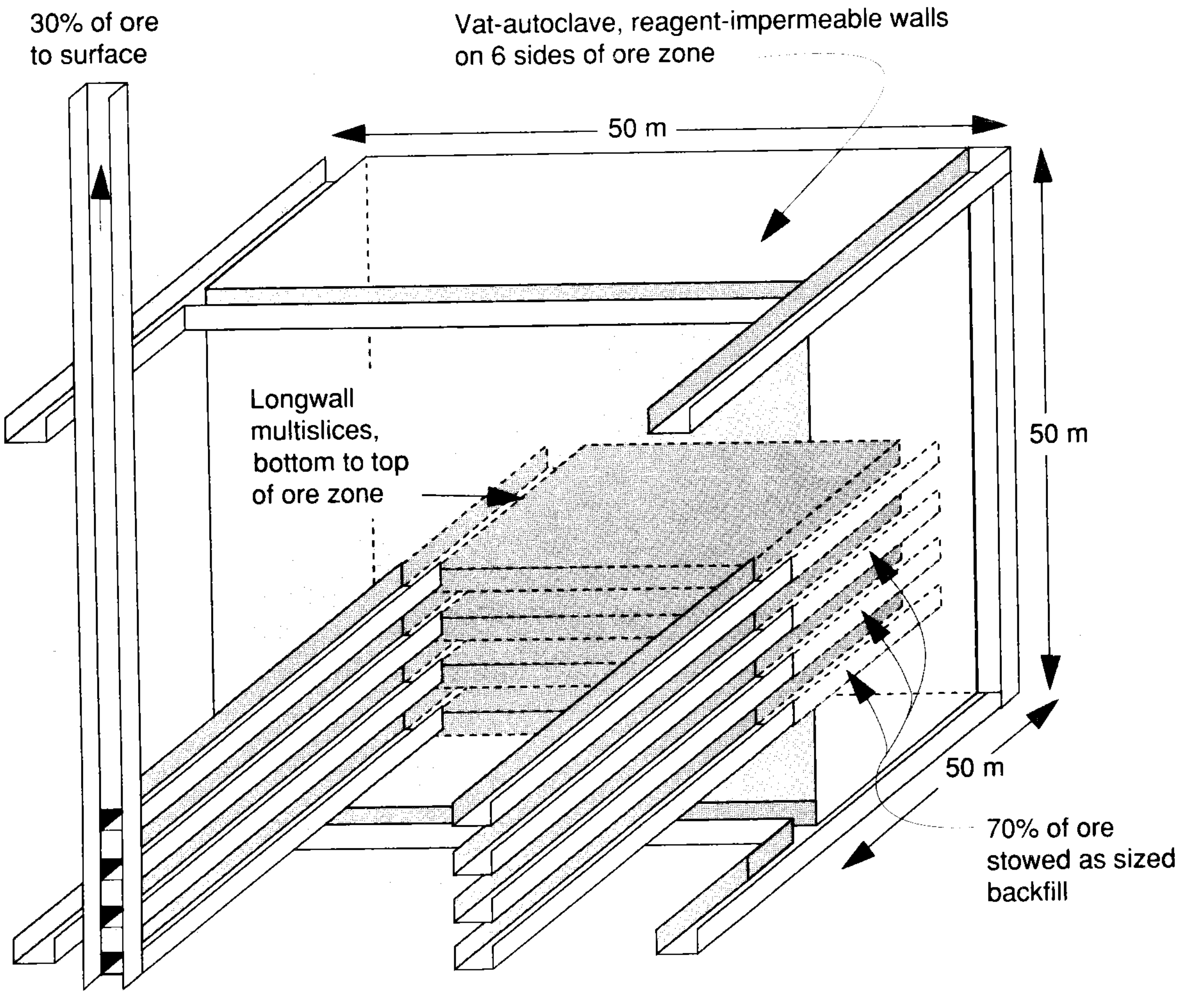


In situ vat-autoclave construction steps. A, Parallel drifts on two levels with connecting, longwall access crosscuts; B, longwall mine bottom cut (vat bottom) and top cut (autoclave lid) and backfill cuts with impermeable compound; C, drive raise to connect levels and channel-cut perimeter sidewalls (using narrow-vein jumbo); D, backfill sidewall cuts with impermeable compound complete six-sided vat-autoclave enclosure. (Dimensions shown are typical.)

Consequent to the formation of these zones, the ore within the zone is drill-split spiral or multislice longwall excavated and face crushed, and 70% to 80% of the ore stream is stowed behind the advancing shield line. (Twenty to thirty percent of the ore must be sent to the surface to accommodate swell.) In this way, the encapsulated zone is entirely filled with coarsely crushed ore. The longwall multislice mining of the rock mass within a vat or autoclave-enclosed zone is shown in figure 2.

Upon the completion of these two basic leach mineral recovery criteria, i.e., that of solution control and sizing ore, the ore can be flooded for long or short periods of time to effect leaching with either ambient or elevated temperature aqueous reagents. The ores can also be gas sparged to oxygenate the reactions and/or create enclosed stope pressurization. Zone pressurization can also be used to repeatedly force flush reagent or wash fluids through the chamber. One scheme for vat or autoclave lixiviant

Figure 2



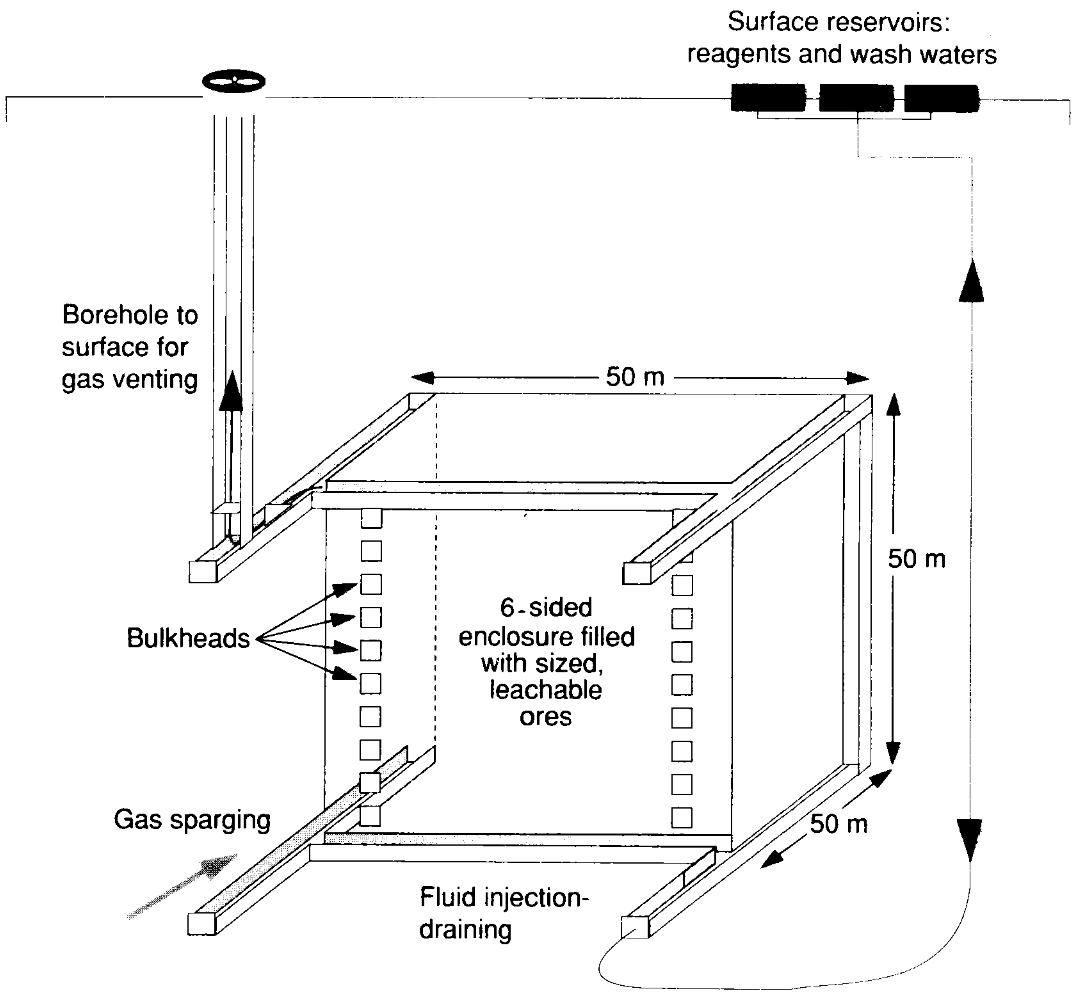
Multislice longwall rubbelization of vat-autoclave contained ores. (Dimensions shown are typical.)

and wash water plumbing and/or gas sparge and product gas ventilation is shown in figure 3.

Ore body zone encapsulations for leaching and elevated temperature-and-pressure autoclave oxidation-leaching are called "stope leach" and "stope autoclave" mining systems, respectively.

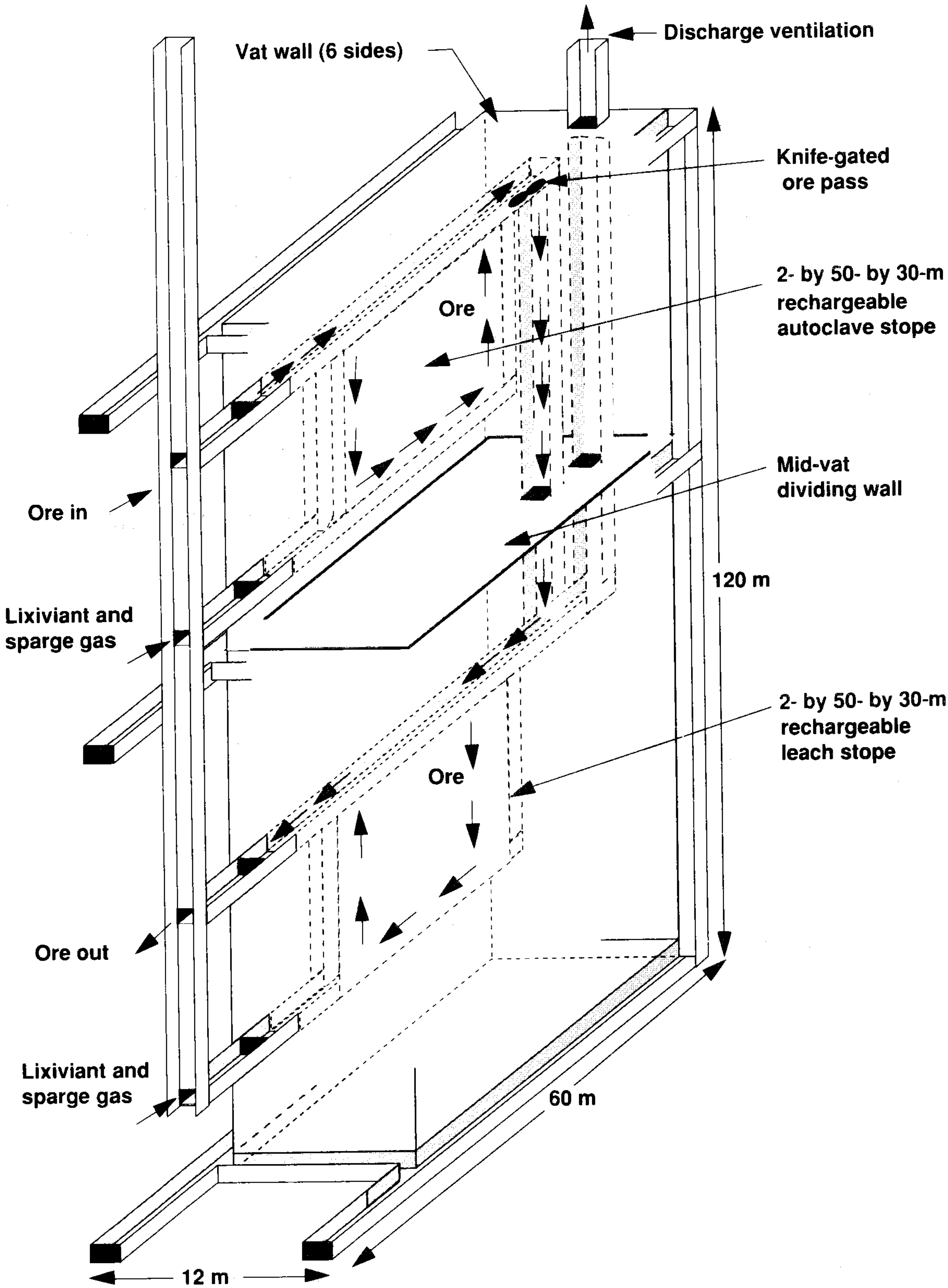
A derivation of the stope leach-stope autoclave method is the ore body-adjacent, underground leach plant method. In this system, reagent-impermeable walls are constructed around open stopes that are equipped for ore and reagent chargings and dischargings. The stopes are, in effect, rechargeable vat leaching vessels. Ores are brought to the encapsulated processing vats, treated, then returned to the mining areas for use as backfill. A two-stope refractory ore treatment plant module is shown graphically in figure 4.

Figure 3



In situ vat-autoclave plumbing and ventilation. (Dimensions shown are typical.)

Figure 4



Underground leach plant (two-stage, refractory ore treatment). Top chamber can be pressurized independently of lower chamber. All ore removed from stopes by airlifts. (Dimensions shown are typical.)

STOPE LEACH MINERAL RECOVERY SYSTEMS

Mineral recovery systems presented in this report are defined as including all aspects of ore body access, ore extraction and treatment, and tails disposal. Mining systems are evaluated on the basis of safety, environmental impact, and economy.

SAFETY

The standards and principles of safe activities in regards to the mining aspects of the proposed leach mining systems, e.g., the number, size, and construction sequencing of headings, etc., are well developed. There is, however, no industrial use history available to guide the design and operation of in situ processing vats.

The principle design concept chosen to effect work force protection, relative to the operation of the proposed in situ chemical reaction containment vessels, is isolation. In the isolation principle, the ore body is divided into discrete, modular, pillar and bulkhead separated ore blocks. The sequenced exploitation of liquid and gas isolated ore blocks to keep chemically active processing stopes physically distant from the personnel performing other mine duties, is presumed to further safeguard the work force.

ENVIRONMENTAL IMPACT

In each of the proposed stope leach mineral recovery systems, 70% to 80% of the ore is left underground as washed backfill and enclosed on five sides by water-impermeable barriers (in situ vat remnants). In the stope autoclave mineral recovery system, the washed backfill is totally encapsulated by impermeable barriers (in situ autoclave remnants). Water-impermeable wall constructions surrounding a washed backfill redundantly protect the site ground water from reagent contamination.

When stope leach mineral recovery is chosen over surface mining, the volume of waste rock and tails discarded on the surface is reduced by an order of magnitude. For example, in a 1.7:1 stripping ratio open-pit mine, 1 m^3 of waste rock and tailing is discarded on the surface for every metric ton of ore processed; only 0.10 m^3 of tailing per metric ton of ore processed is disposed of on the surface by the stope leach technique. For a 4:1 stripping ratio ore body, the open-pit technique discards 20 times more material on the surface than the stope leach approach, and so on. The open-pit mining method also leaves a glory hole, whereas the stope leach technique does not.

The reduced surface disposal characteristic of the stope leach systems reduces the potential for waste pile acid formation and drainage.

The reduced surface disruption characteristic of the stope leach systems greatly reduces the visual pollution of the operation. In some locales, reduced visual impacts may be the key to permit attainment.

COSTS DISCUSSION

The standard industrial measure of project attractiveness is the value resulting from a time-value-of-money weighted discounted-cash-flow-rate-of-return (DCFROR) analysis; the higher the value, the more economically attractive the project. In a DCFROR calculation, the highest DCFROR values are obtained when early years expenditures are minimized and revenues are maximized. From a systems perspective, the proposed stope leach mineral recovery operations potentially maximize early years revenues by (1) being flexible in "targeting" the highest grade zones of the ore body for early years extraction, and (2) being flexible by sort segregating, on a daily basis, the highest grade 20% to 30% of the ore for delivery to high-recovery surface processing. These high grading flexibilities advantage the stope leach techniques over (1) open-pit mines, which can access high grades only on a systematic, ore body top-to-bottom planing fashion; and (2) blasting operations of all types, which mix high-grade and low-grade ores in the blast process and lose the potential to selectively mine and mill ores by grade and/or type for return-on-investment optimized milling.

The proposed stope leach mineral recovery systems minimize environmental permit and bonding costs because they discard significantly less waste on the surface. Similarly, the reduced size surface mill (only 20% to 30% of the full mine production is surface treated) is proportionately less environmentally disruptive and lower in cost to permit and bond.

In addition to the early years revenue maximization and permit cost minimization characteristics, the economics of the proposed mineral recovery systems will be largely determined by (1) the costs of mechanical excavation mining and face crushing (see part 1 of this IC series), (2) the costs savings associated with the avoided haul-hoist costs for the 70% to 80% of the ore retained underground, (3) the costs of in situ stope vat, autoclave, or processing zone constructions, and (4) the costs associated with the leach and oxidize-leach chemistries suited to stope processing. Of these, the costs of containment construction and the costs associated with whole ore treatment chemistries are noted as crucial to the ore body-specific applicability of the proposed stope leach mineral recovery systems.

In the proposed in situ vat construction technique, a 0.6-m-wide drill-split narrow-vein channel is cut around an ore body zone, 2-m-thick drill-split longwall top and bottom cuts are made, and all cuts are backfilled with reagent-impermeable compound. The ideal backfill material would be a chemically inert substance that is easily placed and autogenously healing (i.e., cracks rebond) or cellular (i.e., crushes but does not break). There are a number of substances in the marketplace that ostensibly meet these criteria. Sand-sodium silicate mixtures, sand-polyurethane mixtures, and hot mix asphalt can all theoretically be used to construct reagent-impermeable in situ walls. The costs associated with placement and use of these materials underground are not well developed, but are conservatively estimated at $50/t in place underground.

In many proposed applications, the cost competitiveness of stope leach mineral recovery systems will be determined by the identification of whole ore, vat leach chemistries. Ambient temperature-and-pressure leaching of gold ores in ore body zone stope vats or processing zone rechargeable vats has a low level of whole ore treatment risk because the pertinent leach chemistries have been well demonstrated in the heap leach gold industry. Copper oxide leaching with sulfuric acid represents another low-risk whole ore chemistry, again by virtue of widespread industrial demonstration.

Significant uncertainty exists in the whole ore, stope autoclave treatment chemistries for refractory ores. Conventional autoclaves operate at high temperatures and pressures, using caustic and corrosive chemicals on finely ground materials. This "extreme conditions" autoclave chemistry, although common, does not translate well to the coarse ore autoclave treatment methods described by either stope autoclaving (up to 750,000-t batches) or rechargeable stope autoclaving (typically 5,000 t per batch with 12- to 36-h treatment times). The specific difficulties encountered in treating stope leach mineral recovery ores with extreme conditions chemistries include:

- Temperature: Stope vat temperatures approaching 100 °C appear achievable through heated reagent circulation systems. Temperatures greater than 100 °C represent a significant British thermal unit-delivery problem and may only be achievable by the triggering of ore-based exothermic reactions within the stope autoclaves. Relative to the extreme conditions autoclave chemistries used by industry, 100 °C is a low temperature, and reaction rates and percentages of reaction completions will be comparatively low.
- Pressure: Gas sparging can be used to deliver oxygen, chlorine, etc., and chemical reactions within the vessels can themselves be gas evolving. In either case, the gas buildup within the stope can be contained and pressure allowed to rise. The upper limit on the pressurization of stopes is unknown but may be a function of the gas permeability of the in situ vessel wall materials.
- Material sizing: Fine-grind rod and ball mills are complex, circulating load systems that are not practical additions to a mobile face-crushing circuit. It is most likely that a "coarse sand" size, the result of a final stage of rolls crushing at the face, will be the minimum stope autoclave feed size.
- Caustic-corrosive: Wall rock reactions and/or in situ vat wall corrosion susceptibility will determine the kinds of chemicals available to the stope autoclave chemistries designer. Ore body-specific and wall-materials-specific data are necessary to authoritatively address these issues.

In combination, these factors indicate the need for aqueous oxidation-leaching chemistries that exhibit high recovery at low temperature from coarse materials (implying a long retention time). Long retention time, low-temperature autoclave chemistries are largely undeveloped, but there are precedents. As shown in the case studies in this report, these precedents show whole ore stope autoclave systems economics to be attractive when the chemistries are available.

The field of bioleaching in ambient temperature-and-pressure environments is too new to apprise for whole ore treatment chemistry potential, but the early indications are that a wide variety of metals may eventually be obtainable in this manner.

One of the more exciting, presently identified biotreatment opportunities is the two-step, ambient pressure and temperature, bio-oxidation-cyanide leaching chemistry proposed for the double refractory (carbonaceous ore-sulfide encapsulated values) deep Carlin Belt ores (*1*).[2] Ores of this type would appear to be an ideal match with the proposed in situ vat, stope leach mineral recovery technology.

CASE STUDIES

Four case studies were developed for different types of ore bodies using the proposed stope leach mineral recovery systems. The designs were costed and compared with cost projections for the conventional drill-blast mining and surface processing of the specified case study ore bodies. Specific mine designs and cost estimate details are contained in the appendixes.

[2]Italic numbers in parentheses refer to items in the list of references preceding the appendixes at the end of this report.

The following four case study mining system designs were developed to test the economic viabilities of the proposed stope leach mining methods:

1. Case study 1: The stope vat treatment of cyanide (CN) leachable ores, tested on a surface-exposed ore body (typical oxidized Carlin Belt deposit).
2. Case study 2: The stope vat bio-oxidation-CN-leaching of a refractory ore, tested on a buried, but near-surface ore body that would normally be open-pit mined (typical double refractory deep Carlin Belt deposit).
3. Case study 3: The stope vat hypochlorite oxidation-CN-leaching of a refractory ore, tested on a deeply buried medium-width vein (typical deep sulfide or telluride vein, parallel vein, or vein swarm zone).
4. Case study 4: A processing zone is created adjacent to the ore body; lime-oxygen, low pressure-low temperature-long-residence-time (12 h), autoclave oxidize-CN leach; tested on a deeply buried medium-width vein ore (typical deep sulfide or telluride vein, parallel vein, or vein swarm zone).

In the case studies of the proposed methods, some significant cost-saving design potentials were not used in order to create a generic, baseline definition of stope vat, stope autoclave, and encapsulated leach plant costs and economic viabilities, including the following: (1) The ore was disseminated, i.e., not distributed bimodally; hence, high-grade ore blocks were not targetable for early years exploitation, and high-grade ores were not targetable daily at the mining face for high-recovery processing; (2) the 0.6-m-wide asphalt stope vat walls with asphalt at $50/t in place (1984 dollars) is not the lowest cost impermeable wall construction material inferable from the literature; and (3) the $10/t for hypochlorite and lime-oxygen ore oxidation is a high estimated process chemical charges. The proposed stope leach mineral recovery methods are new and untested, and, based on the further development and demonstrations of the techniques, systems optimizations in the above cited areas will significantly affect utilization of the proposed systems for specific ore deposits.

CASE STUDY 1: STOPE CYANIDE LEACH

In case study 1, the hypothesized ore body is 120 by 360 by 120 m high and surface exposed. The ore is assumed to carry 2.6 g/t Au and 75% recoverable by CN leaching.

The conventional mining system approach to the ore body was assumed to be open-pit mining with ore crushing, agglomeration, and pad leaching. With 45° pit walls, the ore body had a 1.7:1 life-of-mine stripping ratio.

The stope leach mineral recovery system chosen for the ore body removed 20% of the ore to the surface for pad leaching and left 80% of the ore underground in open-topped vat, flood leach vessels. The ore within the in situ vats was spiral-longwall mined. In spiral longwalling, an asphalt isolated ore block is raise bored in the center, and the drill-split longwall is spiraled up around the raise. Ores are dropped down the raise, mucked and crushed, then returned behind the shields (80%) for eventual stope leaching, or sent to the surface (20%) for pad leaching.

As shown in table 1, the open-pit approach is marginally more economic than the underground stope leach system approach for the case study ore body. Table 1 cost estimate details are contained in appendix A.

A true assignment of environmental permitting and bonding costs for the open-pit technique would adjust the results more in favor of the less surface disruptive stope leach mineral recovery system.

Table 1.—Case study 1 cost comparison: stope leaching and open-pit-pad leaching

	Stope	Open-pit-pad
Mining rate ... t/d ..	5,000	13,500
Mill equipment capital cost, 10^6 $:		
Surface	2.975	18.700
Underground	3.225	NAp
Milling recovery ... % ..	75	75
Mill operating cost, $/t:		
Surface	6.10	4.43
Underground	4.60	NAp
Mine equipment capital cost ... 10^6 $ ..	11.725	7.100
Mine operating cost ... $/t ..	7.50	[1]4.60
Mine preproduction cost ... 10^6 $ ..	5.650	1.500
Mining recovery ... % ..	100	100

NAp Not applicable.
[1]Ore.

CASE STUDY 2: STOPE BIO-OXIDIZE AND CYANIDE LEACH

In case study 2, the hypothesized ore body is 360 m wide, 750 m long, 30 m high, and under 120 m of cover. The ore grade is identified to be 3.9 g/t Au and 80% recoverable with sequential bio-oxidation and CN leach treatments.

The conventional mining system approach to this ore body is open-pit mining with bio-oxidation and CN leaching done on the surface on separate pads. With 45° pit walls, the life-of-mine stripping ratio was determined to be 4:1.

The stope leach mineral recovery system chosen was stope bio-oxidation and CN leaching. Eighty percent of the ore was treated in stope vats; 20% of the ore was treated on surface pads. In situ asphalt vats were created to segment the ore body underground. The ores within the in situ vats were then drill-split longwall multislice mined. Ores in the underground vats were flood bio-oxidized, washed, CN treated, and washed again.

No high-grade vats were identified to take economic advantage of ore body grade zonation. The open-pit technique spoiled 20 times more rock on the surface and left a large glory hole, but no significant differential environmental permit and bonding costs were assigned to these disruptions.

As shown in table 2, the stope leach mineral recovery system approach is decidedly more economic than the open-pit-pad treatment approach. Table 2 cost estimate details are contained in appendix B.

Table 2.—Case study 2 cost comparison: stope oxidation-leaching and open-pit-pad oxidation-leaching

	Stope	Open-pit-pad
Mining rate . . . . t/d . .	8,000	35,500
Mill equipment capital cost, 10^6 $:		
Surface.	4.950	23.650
Underground	3.525	NAp
Milling recovery . . . . %. .	80	80
Mill operating cost, $/t:		
Surface.	9.60	8.15
Underground	6.30	NAp
Mine equipment capital cost . . . 10^6 $ . .	17.350	12.400
Mine operating cost . . . . $/t . .	5.40	[1]5.10
Mine preproduction cost . . . . . . 10^6 $ . .	7.400	4.900
Mining recovery . . . . % . .	100	100

NAp Not applicable.
[1]Ore.

CASE STUDY 3: STOPE HYPOCHLORITE OXIDIZE AND CYANIDE LEACH

In case study 3, the hypothesized ore body is a 13.3-m-wide, steeply dipping vein, 500 m deep, 400 m long, and under 100 m of cover. The ore grade is assumed to be 7.8 g/t Au and 90% recoverable in a fine grind, autoclave-carbon-in-leach (CIL) circuit, and 70% recoverable in a hypochlorite oxidation-CN leaching circuit with $10/t hypochlorite consumption (*2*).

The conventional mining system chosen for the ore body is longhole blast stoping. The stope leach mineral recovery system chosen was a multislice "shortwall" mining of the ore body, bottom-to-top, and the division of the ore body into 16 separate stope vats. Narrow channel excavations were made on the hanging wall and footwall and filled with asphalt to form the sidewalls for the in situ vats. Vat bottoms and tops were constructed by the asphalt backfilling of shortwall cuts. The vats were flooded with hypochlorite for 6 weeks to effect ore oxidation; the hypochlorite was then drained and the ore washed. Cyanide was then used to effect leaching and the ore was washed again. Seventy percent of the ore body was stowed as pea-gravel-sized backfill in the in situ vats; 30% of the ore was treated on the surface in a fine-grind, high-recovery, autoclave-CIL circuit.

As shown in table 3, the stope leach mineral recovery technique is more costly than the drill-blast mining and autoclave-CIL processing technique. Table 3 cost estimate details are contained in appendix C.

Table 3.—Case study 3 cost comparison: stope oxidation-leaching and longhole stoping-autoclave-CIL circuit

	Stope oxidation-leaching	Longhole stoping-autoclave-CIL
Mining rate . . . . t/d . .	2,300	2,500
Mill equipment capital cost, 10^6 $:		
Surface.	13.225	37.250
Underground	5.575	NAp
Milling recovery, %:		
Surface.	[1]90	90
Underground	[2]75	NAp
Mill operating cost, $/t:		
Surface.	19.75	15.50
Underground	[3]25.40	NAp
Mine equipment capital cost . . . . 10^6 $ . .	5.225	6.575
Mine operating cost . . . . $/t . .	7.63	6.70
Mine preproduction cost . . . . 10^6 $ . .	[4]16.550	5.750
Mining recovery . . . . % . .	90	[5]92

NAp Not applicable.
[1]Autoclave-CIL.
[2]Stope oxidation leaching.
[3]Hypochlorite consumption of $10/t for underground treated ores; also includes $5/t in narrow-vein sidewall excavation and $5/t in vat asphalt charges.
[4]Includes $8.1 million in underground vat construction costs.
[5]Ten percent dilution.

The results of this case study become more equitable if the ore body is markedly bimodal, i.e., if the hanging wall and footwall ores are identified to be high-grade narrow veins, with lower grade disseminated or vein swarm ores between the high-grade parallel veins. In this case, the full extraction of the hanging wall and footwall vein ores, necessary to form the vat sidewalls, upgrades the quality of the feed to the high-recovery surface plant. Upgrading the quality of the feedstock to the high recovery surface plant improves the ultimate metals recovery of the systems at little extra cost and yields a higher DCFROR.

CASE STUDY 4: RECHARGEABLE STOPE AUTOCLAVE-LEACH

In case study 4, the hypothesized ore body is a 13.3-m-wide, steeply dipping vein, 500 m deep, 400 m long, and under 100 m of cover. The ore grade is assumed to be 7.8 g/t Au and 90% recoverable in a fine grind, autoclave-CIL circuit, and 82.5% recoverable in a two-stage, lime-oxygen, 40 °C, 12-h residence time per stage, oxidation autoclave-CN vat leach (coarse sand-size particles).

The conventional mining system chosen for the ore body is longhole blast stoping with surface autoclave-CIL processing.

The stope leach mineral recovery system study presumes a multislice shortwall mining of the ore body, bottom to top, combined with an encapsulated underground lime-oxygen-CN processing zone. Vein ores are shortwall mined, hanging wall to footwall, face crushed to pea-gravel size, and transported to additional crushing (to coarse-sand size) in the underground processing area. Twenty percent of the ore is transported to the surface for fine grinding and high-recovery processing in an autoclave-CIL plant. Eighty percent of the ore is fed to the rechargeable stope, underground lime-oxygen-CN circuit. After a 12-h treatment in the 5,000-t stope, lime-oxygen, low-temperature autoclave (*3*) and a 12-h treatment in an ambient temperature CN solution in a stope leach vat, the underground processed ores are washed and sent to the mining area for use as backfill. The mine ventilation circuit was designed so that all stope vat gases are directly discharged to the surface through a common venting borehole.

As shown in table 4, the stope leach mineral recovery method employing the encapsulated processing zone is projected to be considerably more economic than the conventional drill-blast, longhole stoping-surface autoclave-CIL processing option. Table 4 cost estimate details are contained in appendix D.

Table 4.—Case study 4 cost comparison: underground leach plant and longhole stoping-autoclave-CIL circuit

	Underground leach plant	Longhole stoping-autoclave-CIL
Mining rate . . . . t/d . .	2,300	2,500
Mill equipment capital cost, 10^6 \$:		
Surface	10.500	37.250
Underground	7.325	NAp
Milling recovery, %:		
Surface	[1]90	90
Underground	[2]82	NAp
Mill operating cost, \$/t:		
Surface	20.64	15.50
Underground	[3]15.10	NAp
Mine equipment capital cost . . . . . 10^6 \$ . .	7.000	6.575
Mine operating cost . . . . \$/t . .	7.10	6.70
Mine preproduction cost . . . . . . 10^6 \$ . .	3.050	5.250
Mining recovery . . . . % . .	100	[4]92

NAp Not applicable.
[1]Autoclave-CIL.
[2]Lime oxidation-CN leaching.
[3]Ten dollars per metric ton.
[4]Ten percent dilution.

CONCLUSIONS

The leaching and autoclaving of ores in encapsulated, reagent-impermeable, underground processing vats are technically described and determined to be economically feasible for the treatment of most large leachable and refractory ore deposits. These conclusions are based on the efficacies of the drill-split narrow-vein and longwall mining systems presented in part 1 of this IC series. The conclusions are also based on the efficacy of asphalt or other compounds as the reagent-impermeable backfill material needed for the formation of underground in situ vat (open-topped) or autoclave (close-topped vats) walls and/or encapsulated processing zone walls. The leach process and autoclave chemistries proposed are necessarily aqueous, at ambient to moderate pressures (estimated atmospheric to 13.8 kPa) low temperatures (estimated less than 100 °C), and long residence times (days, months, or years, as required).

The proposed stope leach mineral recovery systems leave one-third or less of the ore body on the surface as tails. The remaining two-thirds of the ore is left underground as a washed backfill material. In the stope leaching and stope autoclaving options, the ores left underground as stope fill are left in open or closed-top in situ vats, and the migration of meteoric and site-transiting waters through the stope-fill ores is impeded and/or prevented.

The tenor of the economic results of the baseline technology-generic ore body case studies infers (1) that leachable and refractory ore bodies with an open-pit mine stripping ratio greater than 1.7:1 are stope leach mineral recovery system targets, (2) that increased ore body-grade modality tends to increase the competitiveness of the stope leach systems to include the targeting of ore bodies with stripping ratios less than 1.7:1, and (3) that all deeply buried sulfitic and telluridic refractory ores are stope leach mineral recovery system targets. Aqueous, whole ore chemical treatments to effect metals leaching are necessarily presumed developable in assigning these ore body targetabilities, i.e., potential reserves status.

REFERENCES

1. Idaho National Engineering Laboratory (Idaho Falls, ID). Strategic and Critical Materials Program Annual Report, FY 90 (contract J0134035). May 1990, pp. 60-71.

2. Yen, W. T., R. A. Pindred, and M. P. Lane. Hypochlorite Leaching of Gold Ore. Paper in Proceedings of the Symposium at GOLDTech 4, ed. by M. C. Fuerstenan and J. L. Hendrix (Reno, NV, Sept. 10-12, 1990). SME, 1990, pp. 67-74.

3. Lichty, L., G. Ramadorai, R. Bhappo, and R. Roman. Treatment of Refractory Ores and Concentrates. Paper in the Proceedings of the Symposium at GOLDTech 4, ed. by N. C. Fuerstenan and J. L. Hendrix (Reno, NV, Sept. 10-12, 1990). SME, 1990, pp. 193-199.

4. Camm, T. W. Simplified Cost Models for Prefeasibility Mineral Evaluations. USBM IC 9298, 1991, 35 pp.

APPENDIX A.—CASE STUDY 1 COST ESTIMATE DETAILS: STOPE LEACHING AND OPEN-PIT MINING-PAD LEACHING

The hypothesized ore body is 120 by 360 by 120 m thick and surface exposed. The ore grade is designated to be 2.6 g/t Au and 75% recoverable by CN heap leaching.

Stope leaching, featuring mechanical excavation techniques, is the mining system selected for the exploitation of the ore body. In this technique, the ore body is divided into 12 separate leach cells by the asphalt backfilling of drill-split narrow-vein sidewall and longwall undercuts. These cuts form, in effect, open-topped in situ vessels. The ores within each vessel were drill-split, spiral longwall mined, bottom to top, around a raise in the center of the leach cell. Twenty percent of the excavated ore (the swell volume) was sent to the surface; 80% of the ore was stowed behind the upward-spiralling longwall shields. All ores were crushed to a pea-gravel size. Crushed ores were transported pneumatically. The surface-delivered ores were pad leached, and the vat backfill ores were flood leached. System components and costs for the stope leach mineral recovery system are shown in table A-1.

The conventional drill-blast mining system selected for the extraction of the ore body was open-pit mining. In this method, one-eighth of the ore body and one-eighth of the total stripping required to maintain 45° pit walls is mined each year. The life-of-mine stripping ratio is 1.7:1. The open-pit mined ore is crushed to minus one-half inch and pad leached with CN solution. Mine systems components and costs for the open-pit mining and pad treatment of the case study ore body are shown in table A-2.

The open-pit approach to the exploitation of the ore body discards 20 Mm^3 of waste rock and tails on the surface and leaves an open-pit glory hole. The stope leach technique discards 1.5 Mm^3 of tails on the surface and leaves no glory hole. No substantial environmental permitting and reclamation bonding cost differences were developed despite the different surface disturbance characteristics of the two methods. If such costs were developed, they would be preproduction year (year 0) costs. Year 0 costs are especially significant in the calculation of time-value-of-money weighted return-on-investment values. A true allocation of environmental permitting and bonding costs, based on the magnitudes of land disturbances, will reduce the DCFROR of the open-pit option and make the stope leach approach relatively more favorable.

Most ore bodies exhibit some form of vertical or horizontal (macrogeologic) or vein and veinlet (microgeologic) grade zonation. Open-pit mining exploits the ore body from top to bottom in horizontal planes and exhibits only a minimal potential for grade zonation exploitation, i.e., "high grading." The stope leach mineral recovery system techniques proposed in this report are well adapted, as are all underground techniques, to macrogeologic high grading. Also, as a nonexplosive technique, the drill-split mining method, which underlies stope leach mineral recovery, can selectively mine veins, veinlets, and ore types at the face and/or in combination with face crush and sort systems to effect microgeologic high grading. The high grading flexibility of the drill-split systems can create larger cash flows faster, and, ultimately, recover more of the values of the ore body. This high grading capability will probably be a crucial factor in the selection of stope leach mineral recovery systems over open-pit drill-blast systems for most real-world, grade-zoned ore bodies. The case study ore body, however, was assumed to be 100% disseminated and unsortable and, in effect, totally negated the high grading advantage of the drill-split mechanical excavation technique. That the baseline costs for the proposed stope leach mineral recovery system is cost competitive with the open-pit approach for this 1.7:1 stripping ratio ore body without the incorporation of any of the drill-split methods high grading advantages infers a greater real-world competitiveness for the proposed technique than that shown in this case study.

Table A-1.—Case study 1 cost estimate details: stope leaching

Unit processes and parameter values	Capital cost, year 0	Daily operating costs: Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
MINE											
Sinking shaft, 4 m diam, 125 m deep	0	$1,820	0	0	0	0	0	0	0	0	0
Medium drifts, 6 m^2, 1,800 m long	0	1,580	$440	$440	$440	$440	$440	0	0	0	0
Large drifts, 12.5 m^2, 200 m long	0	270	160	160	160	160	160	0	0	0	0
Raise boring, 2 m diam, 500 m long	0	1,090	540	540	540	540	540	0	0	0	0
Narrow-vein mining,[1] 0.67 m wide	$1,725,000	10,710	3,210	3,210	3,210	3,210	3,210	0	0	0	0
Narrow-vein backfill[1]	0	17,140	5,000	5,000	5,000	5,000	5,000	0	0	0	0
Longwall mining,[1] 5,000 t/d	3,250,000	0	25,710	25,710	25,710	25,710	25,710	$25,430	$25,430	$25,430	0
Longwall undercut lining[1]	0	930	430	430	430	430	430	0	0	0	0
Lining raises, 2 m diam, 250 m long, steel	0	0	170	170	170	170	170	0	0	0	0
Reinforced drifts, 6 m^2, 60 m long, concrete	0	100	90	90	90	90	90	0	0	0	0
Bulkheads, 4 per year	0	60	60	60	60	60	60	0	0	0	0
Convey to surface[1]	1,000,000	285	470	470	470	470	470	410	410	410	0
Convey to shields[1]	2,000,000	0	2,060	2,060	2,060	2,060	2,060	2,060	2,060	2,060	0
Large excavations, 30 m^2, 120 m long	79,100	0	0	0	0	0	0	0	0	0	0
Electrical,[1] 17,500 kW	2,000,000	0	0	0	0	0	0	0	0	0	0
Hoisting service, double-drum, 125 m deep	1,750,000	0	900	900	900	900	900	900	900	900	0
Administrative expenses,[1] 5,000 t/d	0	0	5,500	5,500	5,500	5,500	5,500	5,500	5,500	5,500	0
Engineering fee	1,301,400	0	0	0	0	0	0	0	0	0	0
Working capital	1,342,200	0	0	0	0	0	0	0	0	0	0
Total capital	13,146,300	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	0	33,985	44,740	44,740	44,740	44,740	44,740	34,300	34,300	34,300	0
Average yearly operating cost	0	11,894,750	15,659,000	15,659,000	15,659,000	15,659,000	15,659,000	12,005,000	12,005,000	12,005,000	0
SURFACE MILL											
Agglomerating, 1,000 t/d	$40,000	0	$410	$410	$410	$410	$410	$410	$410	$410	0
Heap leaching[3]	3,000,000	0	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	0
Water supply, 100 m^3/d	52,300	$20	20	20	20	20	20	20	20	20	0
Cyanide treatment, 60 L/min	0	0	0	0	0	0	0	0	0	660	$780
Administration expenses, 1,000 t/d	0	1,590	1,590	1,590	1,590	1,590	1,590	1,590	1,590	1,590	1,000
Engineering fee	313,200	0	0	0	0	0	0	0	0	0	0
Working capital	180,600	0	0	0	0	0	0	0	0	0	0
Total capital	3,586,100	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	0	1,610	6,020	6,020	6,020	6,020	6,020	6,020	6,020	6,680	1,780
Average yearly operating cost	0	563,500	2,107,000	2,107,000	2,107,000	2,107,000	2,107,000	2,107,000	2,107,000	2,338,000	623,000

See footnotes at end of table.

Table A-1.—Case study 1 cost estimate details: stope leaching—Continued

Unit processes and parameter values	Capital cost, year 0	Daily operating costs									
		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
UNDERGROUND MILL											
Underground heap leaching,[2] 4,000 t/d	$3,039,000	0	$9,810	$19,620	$9,810	$19,620	$9,810	$19,620	$9,810	$19,620	0
Water supply, 150 m^3/d	75,200	0	30	30	30	30	30	30	30	30	0
Cyanide water treatment, 1,000 L/min	0	0	0	0	0	0	0	0	0	5,900	$1,600
Administration expenses	0	$680	0	0	0	0	0	0	0	0	0
Engineering fee	315,400	0	0	0	0	0	0	0	0	0	0
Working capital	295,200	0	0	0	0	0	0	0	0	0	0
Total capital	3,724,800	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	NAp	680	9,840	19,650	9,840	19,650	9,840	19,650	9,840	25,550	1,600
Average yearly operating cost	NAp	238,000	3,444,000	6,877,500	3,444,000	6,877,500	3,444,000	6,877,500	3,444,000	8,942,500	560,000

NAp Not applicable.
[1]Author estimate.
[2]Amended WFOC heap leach model.
[3]WFOC model.

Table A-2.—Case study 1 cost estimate details: open-pit mining-pad leaching

Unit processes and parameter values	Capital cost, year 0	Daily operating costs									
		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
MINE											
Clearing, 24 ha	$54,390	$330	0	0	0	0	0	0	0	0	0
Mining-pit hauls, 121 m deep, 2,850 m haul	7,077,630	0	$13,160	$14,160	$14,860	$15,380	$15,900	$16,200	$16,370	$16,830	0
Mining-drill-blast, 13,500 t/d, rotary drill	0	0	4,070	4,070	4,070	4,070	4,070	4,070	4,070	4,070	0
Offices and laboratories, 13,500 t/d	445,640	0	0	0	0	0	0	0	0	0	0
Repair shop-warehouse, 13,500 t/d	969,740	0	0	0	0	0	0	0	0	0	0
Fueling system, 13,500 t/d	52,720	0	0	0	0	0	0	0	0	0	0
Restoration, $12,000/ha	0	0	0	0	0	0	0	0	410	410	0
Administration expenses, 13,500 t/d	0	3,140	3,140	3,140	3,140	3,140	3,140	3,140	3,140	3,140	0
Engineering fee	566,160	0	0	0	0	0	0	0	0	0	0
Working capital	611,100	0	0	0	0	0	0	0	0	0	0
Total capital	9,777,380	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	NAp	3,470	20,370	21,370	22,070	22,590	23,110	23,410	23,990	24,450	0
Average yearly operating cost	NAp	1,214,500	7,129,500	7,479,500	7,724,500	7,906,500	8,088,500	8,193,500	8,396,500	8,557,500	0
SURFACE MILL											
Electrical, 1,300 kW	$144,400	0	0	0	0	0	0	0	0	0	0
Agglomerating, 5,000 t/d	200,000	0	$2,050	$2,050	$2,050	$2,050	$2,050	$2,050	$2,050	$2,050	0
Heap leaching,[1] 5,000 t/d	18,114,000	0	17,590	17,590	17,590	17,590	17,590	17,590	17,590	17,590	0
Water supply, 500 m^3/d	220,300	$70	70	70	70	70	70	70	70	70	0
Cyanide treatment, 300 L/min,	0	0	0	0	0	0	0	0	0	2,300	$1,200
Administration expenses, 5,000 t/d	0	3,400	3,400	3,400	3,400	3,400	3,400	3,400	3,400	3,400	1,000
Engineering fee	1,905,400	0	0	0	0	0	0	0	0	0	0
Working capital	693,300	0	0	0	0	0	0	0	0	0	0
Total capital	21,277,400	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	NAp	3,470	23,110	23,110	23,110	23,100	23,100	23,100	23,100	25,410	2,200
Average yearly operating cost	NAp	1,214,500	8,088,500	8,088,500	8,088,500	8,088,500	8,088,500	8,088,500	8,088,500	8,893,500	770,000

NAp Not applicable.
[1]WFOC model.

APPENDIX B.—CASE STUDY 2 COST ESTIMATE DETAILS: STOPE OXIDATION-LEACHING AND OPEN-PIT MINING-PAD OXIDATION-LEACHING TREATMENTS

The hypothesized ore body is 360 m wide, 750 m long, 30 m high, and under 120 m of cover. The ore grade was identified to be 3.9 g/t Au and 80% recoverable with sequential bio-oxidation and CN leach treatments.

In the stope leach mineral recovery system approach to the exploitation of the ore body, the ore reserve is divided into 24 separate, open-topped, in situ flood leach cells. The cells are formed by asphalt backfilled drill-split narrow-vein sidecuts and longwall undercuts. The ores within each cell are drill-split, multislice longwall mined from bottom to top, with 20% of the ore being sent to the surface and 80% retained underground in the cell as backfill. All ores are face crushed to pea-gravel size. The cell-contained ores are flooded with a bio-oxidizing solution for 3 months, washed, then CN flood leached for 3 months, and washed again. Systems components and costs for the stope leach mineral recovery system are shown in table B-1.

The conventional drill-blast mining method selected for ore extraction is open-pit mining. The life-of-mine stripping ratio is 4:1; year 0 was devoted to the strip accessing of one end of the ore body; in years 1 through 8, overburden was stripped as required to maintain production. The ores are crushed and placed on a bio-oxidation treatment pad, then removed from this pad, agglomerated, and stacked on a CN heap leach pad. Mine systems components and costs for the conventional mining and pad treatment of ores are shown in table B-2.

The open-pit approach to the exploitation of the ore body discards 50 Mm^3 of waste rock and tails on the surface and leaves an open-pit glory hole. The stope leach technique discards 2.0 Mm^3 of tails on the surface and leaves no glory hole. No substantial environmental permitting and reclamation bonding cost differences were developed despite the different surface disturbance characteristics of the two methods. If such costs were developed, they would be year 0 costs. Year 0 costs are especially significant in the calculation of time-value-of-money weighted return-on-investment values. A true allocation of environmental permitting and bonding costs, based on the magnitudes of land disturbances, will reduce the DCFROR of the open-pit option and make the stope leach approach relatively more favorable.

Most ore bodies exhibit some form of vertical or horizontal (macrogeologic) or vein, veinlet, or ore type (microgeology) grade zonation. Open-pit mining exploits the ore body from top to bottom in horizontal planes and exhibits only a minimal potential for grade zonation exploitation, i.e., "high grading." The drill-split excavation techniques proposed in this report are well adapted, as are all underground techniques, to macrogeologic high grading. Also, as nonexplosive techniques, they can selectively mine veins, veinlets, and ore types at the face and/or in combination with face crush and sort systems to effect microgeologic high grading. The high grading flexibility of the drill-split systems can create larger cash flows faster and, ultimately, recover more of the values of the ore body. This high grading capability will probably be a crucial factor in the selection of stope leach mineral recovery systems over open-pit drill-blast systems for most real-world, grade-zoned ore bodies. The case study ore body, however, was assumed to be 100% disseminated and unsortable and, in effect, totally negated the high grading advantage of the drill-split mining systems approach. That the baseline costs for the proposed stope leach mineral recovery technique is so cost competitive without the incorporation of its high grading advantages infers real-world competitiveness.

Bio-oxidation costs were extrapolated from the preliminary results of work conducted in 1989 by researchers at the Idaho National Engineering Laboratory (*1*).

Table B-1.—Case study 2 cost estimate details: stope oxidation-leaching

Unit processes and parameter values	Capital cost, year 0	Daily operating costs									
		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
MINE											
Sinking shaft: 4 m diam, 160 m deep	0	$2,300	0	0	0	0	0	0	0	0	0
Raise boring: 2.5 m diam, 152 m long	0	440	0	0	$440	0	0	0	0	0	0
Medium drifts: 7.5 m^2, 3,150 m long	0	3,140	$300	$300	300	$300	$300	$300	$300	$300	0
Large drifts: 15 m^2, 56 m long	0	80	800	800	800	800	800	800	800	740	0
Small drifts: 8 m^2, 5,013 m long	0	7,190	3,900	3,900	3,900	3,900	3,900	3,900	0	0	0
Large excavation: 30 m^2, 100 m long	0	190	0	0	0	0	0	0	0	0	0
Small raises: 6 m^2, 213 m long	0	360	250	250	250	250	250	250	250	250	0
Narrow-vein mining[1]	$2,000,000	7,140	5,000	5,000	5,000	5,000	5,000	5,000	0	0	0
Longwall mining:[1] 8,000 t/d	5,500,000	0	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	0
Convey to surface:[1] 1,600 t/d	1,500,000	340	1,270	1,270	1,270	1,270	1,270	1,270	1,120	1,120	0
Shield backfill:[1] 6,400 t/d	500,000	0	750	750	750	750	750	750	750	750	0
Convey to shields:[1] 6,400 t/d	200,000	0	360	360	360	360	360	360	360	360	0
Narrow-vein lining[1]	0	10,700	10,700	10,700	10,700	10,700	10,700	10,700	0	0	0
Longwall base lining[1]	0	0	3,570	2,860	2,860	2,860	2,860	2,860	0	0	0
Bulkheads: 54 total	0	0	770	770	770	770	770	770	770	770	0
Offices and laboratories: 850 m^2	772,400	0	0	0	0	0	0	0	0	0	0
Repair shops-warehouse: 2,000 m^2	729,100	0	0	0	0	0	0	0	0	0	0
Electrical: 30,000 kW	3,400,000	0	0	0	0	0	0	0	0	0	0
Hoisting service: double drum, 125 m deep	2,250,000	0	900	900	900	900	900	900	900	900	0
Water pumping: 12 L/s, 500 ft head	0	0	0	0	0	0	0	0	0	30	0
Administration expenses[1]	0	0	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	0
Engineering fee	2,129,200	0	0	0	0	0	0	0	0	0	0
Working capital	1,817,100	0	0	0	0	0	0	0	0	0	0
Total capital	24,547,800	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	NAp	31,894	60,570	59,860	63,090	59,860	59,860	59,860	37,250	37,220	0
Average yearly operating cost	NAp	11,163,000	21,199,500	20,951,000	22,081,500	20,951,000	20,951,000	20,951,000	13,037,500	13,027,000	0
SURFACE MILL											
Bioleaching:[1] 1,600 t/d	0	$6,400	$6,400	$6,400	$6,400	$6,400	$6,400	$6,400	$6,400	$6,400	0
Agglomerating:[1] 1,600 t/d	$64,000	610	610	61	610	610	610	610	610	610	0
Heap leaching: 1,600 t/d	4,800,000	0	6,400	6,400	6,400	6,400	6,400	6,400	6,400	6,400	0
Water supply: 200 m^3/d	97,200	30	30	30	30	30	30	30	30	30	0
Cyanide treatment: 100 L/min	0	0	0	0	0	0	0	0	0	980	$860
Administration expenses: 1,600 t/d	0	1,980	1,980	1,980	1,980	1,980	1,980	1,980	1,980	1,980	1,000
Engineering fee	817,337	0	0	0	0	0	0	0	0	0	0
Working capital	471,300	0	0	0	0	0	0	0	0	0	0
Total capital	6,249,837	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	NAp	2,010	15,420	15,420	15,420	15,420	15,420	15,420	15,420	16,400	1,860
Average yearly operating cost	NAp	703,500	5,397,000	5,397,000	5,397,000	5,397,000	5,397,000	5,397,000	5,397,000	5,740,000	651,000

See footnotes at end of table.

Table B-1.—Case study 2 cost estimate details: stope oxidation-leaching—Continued

Unit processes and parameter values	Capital cost, year 0	Daily operating costs									
		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
UNDERGROUND MILL											
Bioleaching:[1] 6,400 t/d, $1/t	0	0	$6,400	$6,400	$6,400	$6,400	$6,400	$6,400	$6,400	$6,400	0
Underground heap leaching[1]	$3,302,800	$23,800	23,800	23,800	23,800	23,800	23,800	23,800	23,800	23,800	0
Water supply makeup: 500 m^3/d	220,300	0	70	70	70	70	70	70	70	70	0
Cyanide water treatment: 1,000 L/min	0	0	0	0	0	0	0	0	0	5,900	$1,600
Engineering fee	357,000	0	0	0	0	0	0	0	0	0	0
Working capital	716,100	0	0	0	0	0	0	0	0	0	0
Total capital	4,596,200	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	NAp	0	30,270	30,270	30,270	30,270	30,270	30,270	30,270	36,170	1,600
Average yearly operating cost	NAp	0	10,594,500	10,594,500	10,594,500	10,594,500	10,594,500	10,594,500	10,594,500	12,659,500	560,000

NAp Not applicable.
[1]Author estimate.

Table B-2.—Case study 2 cost estimate details: open-pit mining-pad oxidation-leaching

Unit processes and parameter values	Capital cost, year 0	Daily operating costs									
		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
MINE											
Clearing: 40 ha	$86,250	$330	$80	$80	$80	$80	$80	$80	$80	$80	0
Pit hauls: 70 m deep, 2,970 m haul	10,890,100	32,600	29,280	29,280	29,280	29,280	29,280	29,280	29,280	29,280	0
Drill blast: 35,500 t/d, rotary drill	0	10,560	9,380	9,380	9,380	9,380	9,380	9,380	9,380	9,380	0
Offices and laboratories: 35,500 t/d	836,880	0	0	0	0	0	0	0	0	0	0
Repair shops-warehouse: 35,500 t/d	1,562,810	0	0	0	0	0	0	0	0	0	0
Fueling system: 35,500 t/d	120,500	0	0	0	0	0	0	0	0	0	0
Restoration: $12,000/ha	0	0	0	0	0	0	0	0	1,530	1,530	0
Administration expenses: 35,500 t/d	0	5,570	5,570	5,570	5,570	5,570	5,570	5,570	5,570	5,570	0
Engineering fee	849,300	0	0	0	0	0	0	0	0	0	0
Working capital	1,329,300	0	0	0	0	0	0	0	0	0	0
Total capital	15,675,140	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	0	49,060	44,310	44,310	44,310	44,310	44,310	44,310	45,840	45,840	0
Average yearly operating cost	0	17,171,000	15,508,500	15,508,500	15,508,500	15,508,500	15,508,500	15,508,500	16,044,000	16,044,000	0
SURFACE MILL											
Electrical: 2,100 kW	$215,920	0	0	0	0	0	0	0	0	0	0
Bioleaching:[1] 8,000 t/d	0	0	$32,000	$32,000	$32,000	$32,000	$32,000	$32,000	$32,000	$32,000	0
Agglomerating:[1] 8,000 t/d	320,000	0	3,040	3,040	3,040	3,040	3,040	3,040	3,040	3,040	0
Heap leaching[2]	22,710,700	0	25,800	25,800	25,800	25,800	25,800	25,800	25,800	25,800	0
Water supply: 1,000 m^3/d	409,160	0	110	110	110	110	110	110	110	110	0
Cyanide treatment: 500 L/min	0	0	0	0	0	0	0	0	0	3,440	$1,280
Administration expenses: 8,000 t/d	0	$4,270	4,270	4,270	4,270	4,270	4,270	4,270	4,270	4,270	1,000
Engineering fee	2,415,400	0	0	0	0	0	0	0	0	0	0
Working capital	1,956,600	0	0	0	0	0	0	0	0	0	0
Total capital	28,027,780	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	0	4,270	65,220	65,220	65,220	65,220	65,220	65,220	65,220	68,660	2,280
Average yearly operating cost	0	1,494,500	22,827,000	22,827,000	22,827,000	22,827,000	22,827,000	22,827,000	22,827,000	24,031,000	798,000

[1]Author estimate.
[2]Camm (*4*).

APPENDIX C.—CASE STUDY 3 COST ESTIMATE DETAILS: STOPE OXIDATION-LEACHING AND LONGHOLE STOPING-AUTOCLAVE-CARBON-IN-LEACH CIRCUIT

The hypothesized ore body is a 13.3-m-wide steeply dipping vein, 500 m deep, 400 m long, and under 100 m of cover. The ore grade was assumed to be 7.8 g/t Au, 90% recoverable in an autoclave-CIL circuit, and 70% recoverable in a hypochlorite oxidation-CN leaching circuit.

Stope leaching featuring mechanical excavation is the mining method selected for the exploitation of the ore body. The ore body was divided into 16 separate leach cells, 2 cells on each of 8 mining levels. Cells were located on either side of a central pillar dividing the ore body. Cells were also separated vertically by 2.5-m sill pillars. Hanging wall and footwall narrow-vein channel cuts were made and connected on the ends, then backfilled with asphalt to form the vat sidewall enclosures. A 2-m-thick longwall slice was made at the base of each leach cell to connect the sidewalls and complete the vat construction. The ores within the vat constructions were multislice shortwall mined from top to bottom using a 9-m-length shield line. The shortwall-mined ores were face crushed to pea-gravel size. Eighty percent of the crushed ore was returned behind the shields; 20% of the ore was sent to the surface for fine grind, autoclave-CIL, high-recovery processing. The backfill ores were flooded with an ore oxidizing hypochlorite solution, washed, then flooded with a CN leaching solution, and washed again. The mine systems components and costs for the stope leach mineral recovery system are shown in table C-1.

The conventional mining and milling methods chosen for the ore body are longhole blast stoping and surface autoclave-CIL processing. The life of the mine is 8 years. Mine systems components and costs for the conventional mining of the ore body are shown in table C-2.

The longhole stoping of the medium-vein ore body dilutes the ore so that a 2,500 t/d mining rate is required. The more selective drill-split mining of the ore body reduces the mining rate to 2,300 t/d.

All of the longhole stoped ore, i.e., 2,500 t/d, is hoisted to the surface and treated in the autoclave-CIL circuit. For the stope autoclave mineral recovery system, the narrow-vein excavated sidewall ores and 20% of the shortwall-mined ores that lie between the hanging wall and footwall side slices, i.e., 675 t/d, are surface autoclave-CIL treated.

In the case study, the medium-width vein ores are assumed to be disseminated. This chosen, disseminated ore scenario is not particularly realistic in that most ore bodies exhibit hanging wall, footwall, and/or veined grade zonations. The exploitation of ore body grade zonation is easily effected in drill-split mining by either selective mining or face sorting procedures. High grading improves the quality of the ore stream sent to high-recovery surface processing, which tends to improve the rate-of-return of the drill-split option.

Hypochlorite was charged at \$10/t of treated ores in the stope autoclave option; this value was chosen from literature examples that ranged between \$2/t to \$30/t.

The total charge for the construction of ore zone vats over the 8 years mine life was \$47 million. These vat construction expenditures enabled the stope oxidation-leaching of some 4.5 Mt of ore. If the vein had been 40 m wide and not 13.3 m wide, a \$56.5 million vat construction charge would have enabled the underground treatment of 13.5 Mt of ore. The vat construction charge per unit of ore treated drops from \$10.44/t in the medium-width scenario to \$4.18/t in the wide-vein scenario. The \$6.26/t savings accruing to the changed vein geometry represents a 25% reduction in underground mill unit operating costs. The competitiveness of the ore zone vat construction technique is seen from this case study to be vein-width sensitive.

In the stope-leaching scenario, some \$8.1 million in vat construction work was done in year 0. This expenditure built both the required year 1 vats and the required year 2 narrow-vein vat sidewalls. These expenditures could have been reduced to \$2.5 million expended in year 0 if only a single 6-month production vat had been built.

Asphalt may not be the lowest cost vat wall material. Asphalt was chosen because of its historical use as pad material beneath heap leach piles. It is reasoned that if it is environmentally acceptable as a pad liner and as a permanent addition to the geologic sequence of the property, it will also be acceptable as a nonrecoverable component in underground constructions. New, lower cost wall construction materials have recently been developed (specifically, sand-sodium silicate and sand-polyurethane mixtures). The sand-sodium silicate mixture is touted to be both impermeable and autogenously healing (self-healing; i.e., cracks rebond). The sand-polyurethane mixture is cellular, i.e., crushes, but does not crack. If these materials are made using process plant sands, vat wall costs would be reduced by more than 50% over the \$50/t in-place asphalt used in the case study.

Table C-1.—Case study 3 cost estimate details: stope oxidation-leaching

Unit processes and parameter values	Capital cost, year 0	Daily operating costs									
		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
MINE											
Clearing: 15 ha	0	$100	0	0	0	0	0	0	0	0	0
Shaft sinking: 4 m diam	0	7,570	0	0	0	0	0	0	0	0	0
Raise boring: 3 and 4 in diam, 1,040 m long	0	9,530	0	0	0	0	0	0	0	0	0
Small drifts: 6 m^2, 720 m long	0	840	$840	$840	$840	$840	$840	$840	$840	$840	0
Medium drifts: 13 m^2, 2,585 m long	0	3,600	570	1,750	570	1,750	570	1,750	570	570	0
Large drifts: 15 m^2, 300 m/yr	0	230	440	440	440	440	440	440	440	440	0
Large excavations: 24 m^2, 30 m long	0	50	0	0	0	0	0	0	0	0	0
Longwall mining:[1] 2,100 t/d	$2,200,000	650	6,600	6,600	6,600	6,600	6,600	6,600	6,600	5,940	0
Pneumatic hoisting:[1] 675 t/d to surface	500,000	420	460	500	460	500	460	500	300	260	0
Pneumatic backfill:[1] 930 t/d	250,000	20	210	210	210	210	210	210	210	210	0
Backfill rehandle:[2] 750 t/d	100,000	10	170	170	170	170	170	170	170	140	0
Raise driving: 2 m^2, 250 m long	0	240	240	0	240	0	240	0	0	0	0
Drainage: 1,000 m^3/d, 520 m head	0	360	200	200	200	200	200	200	200	200	0
Electrical: 1,000 kW	716,000	0	0	0	0	0	0	0	0	0	0
Offices and laboratories: 550 m^2	587,300	0	0	0	0	0	0	0	0	0	0
Repair shops-warehouse: 1,000 m^2	406,200	0	0	0	0	0	0	0	0	0	0
Ventilation: 13,000 m^3/min	471,700	0	1,440	1,440	1,440	1,440	1,440	1,440	1,440	1,440	0
Bulkheads[3]	0	0	1,370	1,370	1,370	1,370	1,370	1,370	1,370	1,370	0
Service hoisting: 675 mt/d, 520 m deep	0	510	910	910	910	910	910	910	910	910	0
Administration expenses[1]	0	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	0
Engineering fee	936,700	0	0	0	0	0	0	0	0	0	0
Working capital	919,800	0	0	0	0	0	0	0	0	0	0
Total capital	7,493,900	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	0	27,630	16,950	18,170	16,950	18,170	16,950	18,170	16,950	16,220	0
Average yearly operating cost	0	9,670,500	5,932,500	6,359,500	5,932,500	6,359,500	5,932,500	6,359,500	5,932,500	5,677,000	0
SURFACE MILL											
Autoclave and CIL: 675 t/d	$12,354,400	0	$12,000	$12,000	$12,000	$12,000	$12,000	$12,000	$12,000	$12,000	0
Tails placement: 675 t/d	869,500	0	0	0	0	0	0	0	0	0	0
Administration expenses: 675 t/d	0	0	1,320	1,320	1,320	1,320	1,320	1,320	1,320	1,320	$1,320
Engineering fee	1,347,100	0	0	0	0	0	0	0	0	0	0
Working capital	399,600	0	0	0	0	0	0	0	0	0	0
Total capital	14,970,600	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	0	0	13,320	13,320	13,320	13,320	13,320	13,320	13,320	13,320	1,320
Average yearly operating cost	0	0	4,662,000	4,662,000	4,662,000	4,662,000	4,662,000	4,662,000	4,662,000	4,662,000	462,000

See footnotes at end of table.

Table C-1.—Case study 3 cost estimate details: stope oxidation-leaching—Continued

Unit processes and parameter values	Capital cost, year 0	Daily operating costs									
		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
UNDERGROUND MILL											
Narrow-vein mining[1]	$2,500,000	$12,860	$6,430	$6,430	$6,430	$6,430	$6,430	$6,430	0	0	0
Narrow-vein fill (asphalt)[1]	0	10,000	10,000	10,000	10,000	10,000	10,000	10,000	$10,000	0	0
Longwall fill (asphalt)[1]	0	280	280	280	280	280	280	280	280	0	0
Hypochlorite-cyanide leach:[4]											
1,625 t/d	2,837,000	0	22,480	22,480	22,480	22,480	22,480	22,480	22,480	$22,480	0
Solution ponds: 4 ha	233,200	0	30	30	30	30	30	30	30	30	0
Cyanide destruction: 78 L/min	0	0	0	0	0	0	0	0	0	810	$810
Water neutralization: 78 L/min	0	0	0	0	0	0	0	0	0	360	120
Administration expenses: 1,625 t/d	0	0	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000
Engineering fee	329,300	0	0	0	0	0	0	0	0	0	0
Working capital	735,300	0	0	0	0	0	0	0	0	0	0
Total capital	6,634,800	0	0	0	0	0	0	0	0	0	0
Average daily operating cost	0	23,140	41,220	41,220	41,220	41,220	41,220	41,220	34,790	24,510	2,930
Average yearly operating cost	0	8,099,000	14,427,000	14,427,000	14,427,000	14,427,000	14,427,000	14,427,000	12,176,599	8,578,500	1,025,500

[1]Author estimate.
[2]Pneumatic storage.
[3]To close stope penetrations.
[4]Camm (*4*).

Table C-2.—Case study 3 cost estimate details: longhole stoping-autoclave-CIL circuit

Unit processes and parameter values	Capital cost, year 0	Daily operating costs								
		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
					MINE					
Clearing: 42 ha	0	$260	0	0	0	0	0	0	0	0
Shaft sinking: 32 m^2 diam	0	12,060	0	0	0	0	0	0	0	0
Raise boring: 3 m diam, 520 m in year 1	0	0	$1,940	0	0	0	0	0	0	0
Medium drifts: 12 m^2	0	690	0	0	0	0	0	0	0	0
Ore pockets: 2 each per year	0	130	60	$60	$60	$60	$60	$60	$60	$60
Load-haul-dump drill: 2,500 t/d	$2,000,000	0	0	0	0	0	0	0	0	0
Stope preparation:[1] 7,700 m^2	0	1,000	4,960	2,980	2,980	2,980	2,980	2,980	2,980	0
Longhole stoping: 3 stopes per year: 2,500 t/d	0	0	6,900	6,900	6,900	6,900	6,900	6,900	6,900	6,900
Hoisting: double-drum	2,282,500	900	1,450	1,380	1,310	1,230	1,150	1,080	1,000	900
Electrical: 7,500 kW	582,100	0	0	0	0	0	0	0	0	0
Fueling system	18,700	0	0	0	0	0	0	0	0	0
Offices and laboratories: 700 m^2	645,200	0	0	0	0	0	0	0	0	0
Repair shops-warehouse: 1,000 m^2	406,300	0	0	0	0	0	0	0	0	0
Ventilation: 10,000 m^3/min, 4,171 Pa	418,400	0	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100
Compressed air: 75 m^3/min	200,200	0	150	150	150	150	150	150	150	150
Water drainage: 600 m^3/d, 520 m per head	88,600	0	120	120	120	120	120	120	120	120
Administration expenses[2]	0	4,100	4,100	4,100	4,100	4,100	4,100	4,100	4,100	4,100
Engineering fee	570,300	0	0	0	0	0	0	0	0	0
Working capital	623,400	0	0	0	0	0	0	0	0	0
Total capital	7,835,700	0	0	0	0	0	0	0	0	0
Average daily operating expense	NAp	19,140	20,780	16,790	16,720	16,640	16,560	16,490	16,410	13,330
Average yearly operating cost	NAp	6,669,000	7,273,000	5,876,500	5,852,000	5,824,000	5,796,000	5,771,500	5,743,500	4,665,500
					SURFACE MILL					
Autoclave and CIL: 2,500 t/d	$36,495,000	$38,700	$38,700	$38,700	$38,700	$38,700	$38,700	$38,700	$38,700	$38,700
Tails placement:[3] 2,500 t/d	822,200	0	0	0	0	0	0	0	0	0
Engineering fee	3,817,200	0	0	0	0	0	0	0	0	0
Working capital	1,161,000	0	0	0	0	0	0	0	0	0
Total capital	42,295,400	0	0	0	0	0	0	0	0	0
Average daily operating cost	NAp	38,700	38,700	38,700	38,700	38,700	38,700	38,700	38,700	38,700
Average yearly operating cost	NAp	13,545,000	13,545,000	13,545,000	13,545,000	13,545,000	13,545,000	13,545,000	13,545,000	13,545,000

NAp Not applicable.
[1]Profile view.
[2]Author estimate.
[3]Camm (*4*).

APPENDIX D.—CASE STUDY 4 COST ESTIMATE DETAILS: UNDERGROUND LEACH PLANT AND LONGHOLE STOPING-AUTOCLAVE-CARBON-IN-LEACH CIRCUIT

The hypothesized ore body is a 13.3-m-wide steeply dipping vein, 500 m deep, 400 m long, and under 100 m of cover. The ore grade is assumed to be 7.8 g/t Au, 90% recoverable in an autoclave-CIL circuit, and 82.5% recoverable in a lime-oxygen oxidation-CN leach circuit.

The stope leach mineral recovery system chosen for the exploitation of the ore body is shortwall multislice mining. The bottom-to-top mining of the ore body removes 100% of the ore. The ore is face crushed to pea-gravel size, and 20% of it is transported to the surface for fine grind, high-recovery autoclave-CIL treatment. Eighty percent of the ore is sent to ore body-adjacent oxidation and leach processing stopes. In the underground processing zone, the ores are roll crushed to coarse sand size before being charged into 5,000-t stope lime-oxygen oxidation autoclaves (a 12-h, 2.8 kPa, 40 °C oxidation step). These ores are then washed and discharged into a 12-h, ambient pressure-and-temperature CN flood leach. After leaching, the ores are washed and hydraulically transported back to the mining area for use as backfill. System components and costs for the stope leach mineral recovery system using encapsulated, rechargeable processing stopes are shown in table D-1.

The conventional mining and milling methods chosen for the ore body are longhole blast stoping and surface autoclave-CIL processing. The life of the mine is 8 years. Mine systems components and costs for the conventional mining of the ore body are shown in table D-2.

In the stope leaching scenario, six each 2-m-wide, 30-m-high, 50-m-long stopes were cut. The stopes were constructed on two levels, three stopes per level, and vertically aligned. The vertically coincident stopes were connected by a bored raise with a flap valve. All stopes were fitted with extensible belt ore charging systems and Pachuca (air-lift) ore discharging systems. The stopes were individually asphalt encapsulated on all six sides. All encapsulated stopes were connected and flap valved to a central, raise bored, exhaust ventilation shaft. (The exhaust ventilation shaft had no purpose other than ventilation, i.e., it was not part of the mine emergency egress system.) Each of the top-level encapsulated stopes, by virtue of valving, could be pressurized to 2.8 kPa by oxygen sparging. The spargings take place after the charging of ore and are part of the lime-oxygen ore oxidation process. An aqueous lime-oxygen reagent recirculation system was used to heat solutions and maintain a 40 °C reaction vessel (stope) temperature.

The capital cost of the underground facility per metric ton of ore treated per day is $3,700; values recovery is 82.5%, as shown in table 4. The capital cost of the 90% recovery autoclave-CIL circuit on the surface was $14,900 t/d. This 75% reduction in plant construction costs with less than a 10% loss in recovery, combined with nominally equal operating costs ($15/t for both systems), accounts in large part for the cost superiority of the stope leaching system over the drill-blast longhole stoping system.

The pressure and heat retention characteristics of stope vats, and the ease with which these stope-vats can be excavated, encapsulated, and piped and valved to transfer reagents and solids, when compared with the costs of construction of pressurized vessels, pipe networks, etc., on the surface, implies the underground space has a value that has heretofore been unrecognized, hence unexploited. The underground processing zones are inexpensive to construct relative to conventional (high temperature-high pressure-high throughput-high recovery) surface autoclave-CIL systems. This infers a new genre of cost effective low temperature-low pressure-long residence time in situ treatment vessels is imminent. New, aqueous whole ore treatment chemistries will need to be developed for use in these low cost, in situ vessels.

Table D-1.—Case study 4 cost estimate details: underground leach plant

Unit processes and parameter values	Capital cost, year 0	Daily operating costs								
		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
MINE										
Clearing: 15 ha	0	$93	0	0	0	0	0	0	0	0
Shaft sinking: 3 m diam	0	3,760	0	0	0	0	0	0	0	0
Raise boring: 3 m diam, 520 m long	0	1,940	0	0	0	0	0	0	0	0
Small drifts: 6.25 m^2, 135 m long	0	160	$90	$90	$90	$90	$90	$90	$90	$90
Medium drifts: 12.25 m^2, 200 m long	0	670	350	350	350	350	350	350	350	350
Large drifts: 12.50 m^2, 200 m long	0	270	180	180	180	180	180	180	180	180
Longwall mine:[1] 2,500 t/d	$2,500,000	0	8,200	8,200	8,200	8,200	8,200	8,200	8,200	8,200
Convey to surface:[1] 500 t/d, pneumatic	350,000	220	320	320	320	320	320	320	320	320
Convey to underground vats:[1] 2,000 t/d	1,200,000	0	1,010	1,010	1,010	1,010	1,010	1,010	1,010	1,010
Hydraulic backfill:[1] 2,000 t/d	500,000	0	1,510	1,510	1,510	1,510	1,510	1,510	1,510	1,510
Electrical: 10,000 kW	716,000	0	0	0	0	0	0	0	0	0
Offices and laboratories: 2,500 t/d	467,200	0	0	0	0	0	0	0	0	0
Repair shops-warehouse: 2,500 t/d	423,600	0	0	0	0	0	0	0	0	0
Ventilation: 10,000 m^3/min	418,400	0	1,060	1,060	1,060	1,060	1,060	1,060	1,060	1,060
Water drainage: 480 m^3/d (average)	138,000	120	170	170	170	170	170	170	170	170
Service hoists:[2] 520 m deep	274,100	0	900	900	900	900	900	900	900	900
Administrative expenses: 2,500 t/d	0	0	4,020	4,020	4,020	4,020	4,020	4,020	4,020	4,020
Engineering fee	818,600	0	0	0	0	0	0	0	0	0
Working capital	534,300	0	0	0	0	0	0	0	0	0
Total capital	8,340,200	0	0	0	0	0	0	0	0	0
Average daily operating cost	0	7,233	17,810	17,810	17,810	17,810	17,810	17,810	17,810	17,810
Average yearly operating cost	0	2,531,550	6,233,500	6,233,500	6,233,500	6,233,500	6,233,500	6,233,500	6,233,500	6,233,500
SURFACE MILL										
Autoclave and CIL:[3] 500 t/d	$9,811,700	0	$9,170	$9,170	$9,170	$9,170	$9,170	$9,170	$9,170	$9,170
Tails placement: 500 t/d	680,600	0	0	0	0	0	0	0	0	0
Administrative expenses: 500 t/d	0	0	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150
Engineering fee	1,067,800	0	0	0	0	0	0	0	0	0
Working capital	309,600	0	0	0	0	0	0	0	0	0
Total capital	11,869,700	0	0	0	0	0	0	0	0	0
Average daily operating cost	NAp	0	10,320	10,320	10,320	10,320	10,320	10,320	10,320	10,320
Average yearly operating cost	NAp	0	3,612,000	3,612,000	3,612,000	3,612,000	3,612,000	3,612,000	3,612,000	3,612,000
UNDERGROUND MILL										
Raise boring: 2 m diam, 125 m long	0	$380	0	0	0	0	0	0	0	0
Raise boring: 0.67 m diam, 125 m long	0	70	0	0	0	0	0	0	0	0
Inclines: 9 m^2, 300 m	0	370	0	0	0	0	0	0	0	0
Small drifts: 6.25 m^2, 1,550 m	0	1,860	0	0	0	0	0	0	0	0
Narrow-vein mining[1]	$500,000	2,500	0	0	0	0	0	0	0	0

See footnotes at end of table.

Table D-1.—Case study 4 cost estimate details: underground leach plant—Continued

Unit processes and parameter values	Capital cost, year 0	Daily operating costs								
		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
UNDERGROUND MILL—Continued										
Longwall mining[1]	0	$120	0	0	0	0	0	0	0	0
Narrow-vein asphalt fill[1]	0	860	0	0	0	0	0	0	0	0
Longwall asphalt fill[1]	0	460	0	0	0	0	0	0	0	0
Bulkheads[4]	0	310	0	0	0	0	0	0	0	0
Surface ponds[5]	$10,300	0	0	0	0	0	0	0	0	0
Underground heap leach[6]	3,233,300	0	$27,570	$27,570	$27,570	$27,570	$27,570	$27,570	$27,570	$27,570
Solution heating: 2,000 t/d, 40 °C	250,000	0	860	860	860	860	860	860	860	860
Grinding: 2,000 t/d (to coarse sand size)	915,900	0	1,730	1,730	1,730	1,730	1,730	1,730	1,730	1,730
Engineering fee	498,100	0	0	0	0	0	0	0	0	0
Working capital	904,800	0	0	0	0	0	0	0	0	0
Total capital	6,312,400	0	0	0	0	0	0	0	0	0
Average daily operating cost	NAp	6,930	30,160	30,160	30,160	30,160	30,160	30,160	30,160	30,160
Average yearly operating cost	NAp	2,425,500	10,556,000	10,556,000	10,556,000	10,556,000	10,556,000	10,556,000	10,556,000	10,556,000

NAp Not applicable.
[1]Author estimate.
[2]Two shafts.
[3]Camm (*4*).
[4]Ore in-out.
[5]Cost-estimating-system amended.
[6]Camm (*4*); NAOH (primary-secondary crush included).

Table D-2.—Case study 4 cost estimate details: longhole stoping-autoclave-CIL circuit

Unit processes and parameter values	Capital cost, year 0	Daily operating costs								
		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
MINE										
Clearing: 42 ha	0	$260	0	0	0	0	0	0	0	0
Shaft sinking: 33 m^2 diam	0	12,060	0	0	0	0	0	0	0	0
Raise boring:[1] 3 m diam, 520 m/yr	0	0	$1,940	0	0	0	0	0	0	0
Medium drifts: 12 m^2	0	690	0	0	0	0	0	0	0	0
Ore pockets: 2 each year	0	130	60	$60	$60	$60	$60	$60	$60	$60
Load-haul-dump drill: 2,500 t/d	$2,000,000	0	0	0	0	0	0	0	0	0
Stope preparation:[1] 7,700 m^2	0	1,000	4,960	2,980	2,980	2,980	2,980	2,980	2,980	0
Longhole stoping: 2,500 t/d, 3 stopes per year	0	0	6,900	6,900	6,900	6,900	6,900	6,900	6,900	6,900
Hoisting: double-drum	2,282,500	900	1,450	1,380	1,310	1,230	1,150	1,080	1,000	900
Electrical: 7,500 kW	582,100	0	0	0	0	0	0	0	0	0
Fueling system: 2,600 t/d	18,700	0	0	0	0	0	0	0	0	0
Offices and laboratories: 700 m^2	645,200	0	0	0	0	0	0	0	0	0
Repair shops-warehouse: 1,000 m^2 (surface)	406,300	0	0	0	0	0	0	0	0	0
Ventilation: 10,000 m^3/min, 4,171 Pa	418,400	0	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100
Compressed air: 75 m^3/min	200,200	0	150	150	150	150	150	150	150	150
Water drainage: 600 m^3/d, 520 m/head	88,600	0	120	120	120	120	120	120	120	120
Administrative expenses[2]	0	4,100	4,100	4,100	4,100	4,100	4,100	4,100	4,100	4,100
Engineering fee	570,300	0	0	0	0	0	0	0	0	0
Working capital	623,400	0	0	0	0	0	0	0	0	0
Total capital	7,835,700	0	0	0	0	0	0	0	0	0
Average daily operating expense	NAp	19,140	20,780	16,790	16,720	16,640	16,560	16,490	16,410	13,330
Average yearly operating cost	NAp	6,669,000	7,273,000	5,876,500	5,852,000	5,824,000	5,796,000	5,771,500	5,743,500	4,665,500
SURFACE MILL										
Autoclave and CIL: 2,500 t/d	$36,495,000	$38,700	$38,700	$38,700	$38,700	$38,700	$38,700	$38,700	$38,700	$38,700
Tails placement:[3] 2,500 t/d	822,200	0	0	0	0	0	0	0	0	0
Engineering fee	3,817,200	0	0	0	0	0	0	0	0	0
Working capital	1,161,000	0	0	0	0	0	0	0	0	0
Total capital	42,295,400	0	0	0	0	0	0	0	0	0
Average daily operating cost	NAp	38,700	38,700	38,700	38,700	38,700	38,700	38,700	38,700	38,700
Average yearly operating cost	NAp	13,545,000	13,545,000	13,545,000	13,545,000	13,545,000	13,545,000	13,545,000	13,545,000	13,545,000

NAp Not applicable.
[1]Profile view.
[2]Author estimate.
[3]Camm (4).