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NUCLEAR CHEMICAL MINING OF PRIMARY COPPER SULFIDES

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NUCLEAR CHEMICAL MINING OF PRIMARY COPPER SULFIDES*

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REVISED ABSTRACT

A contained nuclear explosion to produce a chimney of broken ore well below the water table is proposed. After the chimney is filled with water and reaches hydrostatic equilibrium, oxygen is introduced near the bottom of the mass of broken rock. The increase in solubility of oxygen at high hydrostatic pressure is sufficient to initiate the oxidation of the primary sulfides: chalcopyrite and pyrite. The oxidation and dissolution of these sulfides produces enough heat to increase the temperature of the ore and water in the chimney as much as 100°C. The rate of dissolution of chalcopyrite is so rapid under these conditions that the rate of copper extraction becomes limited principally by the physical accessibility of the chalcopyrite within the ore fragments.

Data are presented from laboratory and pilot plant studies conducted during the past 2 yr on the rate of extraction of copper from primary sulfide ore. Results from these studies are used in developing a computer model for estimating the rate of recovery and the ultimate economic recovery of copper values in large-scale solution mining.

Economic analysis shows that nuclear solution mining is very competitive with other mining techniques. The method is broadly applicable to the low-grade, deep deposits needed for future production. Problems of ground shock and radioactivity are manageable, and many environmental problems associated with conventional mining are eliminated.

*Work performed under the auspices of the U.S. Atomic Energy Commission.

INTRODUCTION

A method for in-situ chemical mining of primary sulfide ores, which promises to have a major impact on the technology and economics of recovery of some metals from their ores, has been proposed [1-3]. If successful, the process will greatly increase the reserves of metals such as copper and nickel by reducing the cost of production of deep, disseminated deposits and will significantly reduce some of the undesirable effects on the environment that are unavoidable with conventional mining methods.

For metal to be leached from primary sulfide ores, the ore must be broken up to allow solutions capable of dissolving the metal from the ore to contact the ore. In the method proposed, contained nuclear explosives would be used to break the ore. These explosives are capable of breaking large tonnages of ore at depth; the tonnage is dependent on the explosive yield of the nuclear device and the depth for any given rock type (Fig. 1).

The chimney of broken rock is constructed well below the water table and is allowed to fill with water either by natural inflow of groundwater or by diversion of surface water (Fig. 2). Estimated chimney properties [4] are shown in Table I. Oxygen may be introduced into the bottom of the chimney through drill holes. It will be dissolved in the water as it percolates upward. The solubility of the oxygen is much greater under hydrostatic pressure than it is in water at 1 atm. The dissolved oxygen greatly increases the rate of oxidation and dissolution of the primary sulfide minerals. The oxidation of the sulfide minerals produces a large amount of heat, which is effective in raising the temperature of the chimney contents many tens of degrees centigrade because of the low thermal conductivity of rock surrounding the chimney and the low surface-to-mass ratio of a large chimney.

The net result is a rapid dissolution of primary sulfides under conditions of high temperature and high oxygen solubility at high hydrostatic pressure. The solutions would be withdrawn from the chimney through drill holes, processed in a solvent extraction plant to remove the metals, and returned to the chimney in a closed circuit.

Undesirable flow of solutions out of or into the chimney through fractures or other permeable paths may be controlled by maintaining the hydrostatic equilibrium between the chimney and the surrounding rocks. The chimney will therefore behave as a suitable container for this process even though the rocks surrounding the chimney may not be entirely impermeable.

Sulfide Dissolution

The oxidation of chalcopyrite and pyrite was studied by bubbling oxygen through a pressure vessel containing the mineral sample and distilled water. At 90°C and at oxygen pressures of 25 to 50 atm, the reaction produced sulfuric acid, and the pH of the solution rapidly decreased from that of distilled water to a constant pH plateau of about 1.9 for chalcopyrite and 1.0 for pyrite. The observed products of the decomposition of chalcopyrite were aqueous copper sulfate, sulfuric acid, hematite, and elemental sulfur. Pyrite decomposed to aqueous iron sulfate, sulfuric acid, hydrogen jarosite,

Fig. 1. Mass of broken rock as a function of explosive yield and depth of burst.

Fig. 2. Chemical mining of flooded primary sulfide ore.

TABLE I. CHIMNEY PROPERTIES

Yield of nuclear device	100 kt
Depth of burial	750 m
Depth of water table	50 m
Water content of rock	3 wt. %
In-situ density of rock	2.7 g/cm ³
Bulking porosity	0.17
Radius of cavity	41 m
Height of chimney	300 m
Volume of water in chimney	$2.9 \times 10^5 \text{ m}^3$
Volume of rock in chimney	$1.4 \times 10^6 \text{ m}^3$
Mass of rock in chimney	$3.8 \times 10^{12} \text{ g}$
Hydrostatic pressure at depth of burial	70 atm
Hydrostatic pressure at top of chimney	40 atm

and elemental sulfur. The stoichiometry of each reaction at the pH plateau is shown in the following equations:

Extensive work on the kinetics and mechanism of chalcopyrite dissolution has been reported [2,5,6,7,8,9,10]. The rate of dissolution increases by a factor of approximately 2 for each 10°C rise in temperature and has first-order dependence on the oxygen partial pressure. Although chalcopyrite has a very low rate of dissolution in acid solution at ambient temperature and atmospheric pressure, it will dissolve rapidly at the high temperature and high oxygen pressure that can be readily obtained in nuclear chemical mining. The rate of dissolution is so rapid under these conditions that the rate of copper extraction from ore fragments will be primarily limited by the physical accessibility of the chalcopyrite within the ore fragment. Information on the rate of dissolution of chalcopyrite as it occurs in ore fragments is thus crucial in determining the technical feasibility of this application.

Extraction of Metal Values from Broken Ore

The extraction of copper from large fragments of ore in a large pressure vessel shown schematically in Fig. 3 is being studied. The vessel was charged with 5.8 metric tons (6.4 short tons) of primary sulfide ore and 1400 liters (370 gal) of distilled water. Operating conditions are 90°C and 27.5 atm (405 psi) oxygen. Oxygen is introduced into the bottom of the vessel at a flow rate sufficient to maintain a saturated oxygen solution. The original solution is kept in the vessel for the duration of the test with no circulation other than that induced by the oxygen.

The ore used in this experiment is a specially-selected, coarse-size fraction of block-caved primary porphyry ore of known size distribution. Approximately 50 wt.% of the ore has a diameter of more than 8 cm. Chemical analysis of the ore showed an average grade of 0.7 wt.% Cu, a pyrite/chalcopyrite mole ratio of 2, and a CO₂ content of 0.5 wt.%.

The fraction recovery of copper is shown in Fig. 4, reaching 0.27 in 1 yr. This recovery curve follows the basic pattern discussed by Harris [11] for leaching low-porosity particles of ore in which the sulfide is disseminated throughout. The initially high rate of copper extraction from the chalcopyrite exposed at the surface of the ore particles is followed by a lower rate of extraction of copper from the interior of the particles.

The pH of the leach solution (measured at 23°C) is shown in Fig. 5. It dropped from an initial value of 5.3 to a value of 1.75 at 40 d because of the production of sulfuric acid. After a slight further decrease to 1.7, the pH very slowly increased to 1.9 at 1 yr as the rate at which acid is consumed by the decomposition of host rock and gangue minerals gradually exceeded the rate at which acid is produced by the oxidation of the sulfide minerals. At this pH (1.9), the iron released from the pyrite and chalcopyrite is largely precipitated as hydrogen jarosite and hematite, with less than 80 µg/ml Fe⁺⁺⁺ and 5 µg/ml Fe⁺⁺ remaining in solution.

Fig. 3. Diagram of pilot-plant pressure vessel for experimental leaching of primary sulfide ore.

Fig. 4. Fraction recovery of copper vs time, for 5.8 metric tons of San Manuel ore.

Fig. 5. CO₂ evolution and pH vs time.

The rate of CO_2 evolution from the decomposition of calcite decreased from an initially high value of nearly 600 g/d to a gradually declining value of less than 50 g/d after 60 d. The net amount of CO_2 evolved in 1 yr corresponds to approximately 70% of the initial CO_2 content of the ore. No trace of SO_2 or H_2S has been detected in the effluent gas by gas chromatographic analysis.

In the extraction of copper from an ore particle, the rate of recovery of copper is initially limited by the rate of oxidation of the chalcopyrite grains located at or near the surface of the ore. As this outer, accessible chalcopyrite becomes depleted, however, the rate of recovery becomes increasingly limited by diffusion processes. Examination of leached ore particles from small-scale leaching experiments shows that the oxidation of the sulfide grains occurs in a reaction zone that moves progressively into the interior of the ore particle. This suggests an extraction model based on mixed kinetics in which the rate of oxygen consumption in the reaction zone just balances its rate of diffusion through the pores in the leached shell outside the reaction zone. A calculational model based on an approximate solution to this mixed-kinetics problem has been proposed [2] for quantitatively describing the overall extraction kinetics. This model is calibrated with copper recovery data from the large-scale leaching experiment and from other leaching experiments involving smaller quantities of ore ranging from 0.15 to 8 cm in diameter.

Model of Recovery of Metal Values in a Nuclear Chimney

To apply the preceding extraction model in predicting the rate of copper recovery in a nuclear chimney, it is necessary to utilize many different properties of the chimney and the broken ore contained in it. For present purposes, we use the values summarized in Table I for a 100-kt chimney.

The main factors, in addition to dissolved oxygen, influencing the rate of recovery and the ultimate limit of recovery of copper are 1) the size distribution of the broken ore and 2) the temperature within the chimney.

The rock-size distribution for a given nuclear chimney is grossly dependent upon the type of rock and its preexisting fracture pattern. The most accurate measurement of chimney rubble size was done for the Piledriver nuclear explosion (61 kt in granitic rock); the Piledriver rock-size distribution [12] is shown in Fig. 6. The more extensive preexisting fractures in a copper ore deposit should favor the fragmentation of the ore into smaller sizes, which in turn would allow a higher fraction of copper recovery before the extraction process becomes seriously diffusion-limited. The rock-size for monzonite-type open pit mine waste [13], for example, is indeed much smaller than for the Piledriver event. Unless otherwise specified, the curve shown in Fig. 6 for size-distribution finer than the Piledriver sizes and coarser than the mine-waste sizes was used in the calculations that follow.

The temperature within a flooded chimney has a profound influence on the rate of copper recovery, because the specific rate constant for the oxidation of chalcopyrite approximately doubles for each 10°C rise in temperature. A valid model for the in-situ mining of primary sulfide ore must, therefore, take into account the initial chimney temperature, the

Fig. 6. Rock size distribution in Piledriver chimney.

heat generated within the chimney by the exothermic oxidation reactions, and the heat lost from the chimney to the surrounding rock by thermal conductivity.

The initial chimney temperature is determined primarily from the ambient temperature of the earth at the particular depth of interest and from the energy deposited within or near the chimney by the nuclear explosion. For a 100-kt explosion, approximately 2×10^{13} cal (20% of the device energy) is deposited within the chimney. Assuming that the preshot ambient temperature is 50°C and that water at 25°C is used in flooding the chimney, the resultant temperature of the chimney contents is about 60°C upon reaching equilibrium. If a higher chimney temperature is desired at the start of the oxygen injection (~9 ms after detonation), then the chimney can be further heated by injecting high-pressure steam.

Summary of Computer Model

A finite-difference computer code, based on the preceding kinetic and thermal considerations, has been developed to calculate the temperature in the chimney and the fraction of copper recovered as a function of time. In summary, the calculation takes into account the following factors:

- Dimensions of chimney
- Amount of broken ore
- Particle size distribution of broken ore
- Grade of ore
- Mole ratio of pyrite to chalcopyrite
- Initial temperature of flooded chimney
- Degree of oxygen saturation
- Heat generated in chimney by oxidation of sulfide minerals
- Heat lost from chimney by thermal conductivity
- Dissolution of chalcopyrite particles within ore fragments
- Change in rate of dissolution of chalcopyrite with temperature

Computational Results

The following calculational results were obtained for the nuclear chimney described in Table I, using a degree of oxygen saturation of 0.5 and an estimate (see Fig. 6) for the rock-size distribution for a 100-kt nuclear explosion in copper ore. Figure 7 shows the chimney temperature as a function of time for three grades of ore for an initial temperature of 60°C and a pyrite/chalcopyrite mole ratio of 1. Figure 8 shows the fraction of copper recovered for the same set of conditions. These two figures illustrate that an appreciably greater fraction of copper can be recovered from the higher-grade ore as a result of the higher temperatures produced when more sulfide fuel is available. The irregularity in the curves is a

Fig. 7. Chimney temperature vs time for various grades of copper ore.

Fig. 8. Fraction recovery vs time for various grades of copper ore.

consequence of two effects: 1) decrease in the rate of extraction as the unreacted chalcopyrite becomes less accessible to the leach solution, and 2) change in the rate of extraction as the temperature of the system changes.

For a given grade of ore, the rate of copper recovery can be increased by preheating the chimney with high-pressure steam. The effects of preheating the chimney to initial temperatures of 60, 80, and 100°C are shown in Fig. 9 (temperature vs time) and Fig. 10 (fraction recovery vs time).

For a given grade of ore, the rate of copper recovery is higher for ores containing more pyrite. The increase results from the additional temperature rise produced by the oxidation of pyrite. The effects of varying the pyrite/chalcopyrite ratio in 0.50 wt.% Cu ore are shown in Fig. 11 (temperature vs time) and Fig. 12 (fraction recovery vs time).

Although the rate of copper recovery is appreciably influenced by the thermal considerations presented in the preceding examples, the ultimate economic limit of copper recovery is primarily influenced by the particle size of the broken ore. For example, if the particle size were actually as small as that of the monzonite-type open-pit mine waste (from Fig. 6), then the fraction of copper recovery would be as depicted in Fig. 13. For comparison, the recovery curve for the particle size used in the preceding examples is shown, as well as the recovery curve for the Piledriver particle size. In this example, the grade of ore is 0.5 wt.% copper, the pyrite/chalcopyrite mole ratio is 1, and the initial chimney temperature is 60°C.

The poor rate of recovery for the Piledriver particle-size distribution, reaching only 0.25 recovery in 5 yr, can be considerably improved by preheating the chimney, as shown in Fig. 14 for initial chimney temperatures of 60, 80, and 100°C. Thus, even for this coarse ore size, a recovery exceeding 0.5 can be obtained in less than 5 yr.

We suspect that the projections shown for copper recovery in a nuclear chimney are conservative. The projections are based on a conservative calibration of the calculational model and neglect the very important possibility of breakdown of the large ore fragments during leaching. The question of this possibility will best be answered by longer leaching experiments. Also, the initial rock breakage in an actual ore deposit may be significantly better than in the deposit that we have used as an example.

Nickel sulfide deposits containing pentlandite and pyrrhotite will be easier to leach than the copper deposits containing chalcopyrite and pyrite. Both pentlandite and pyrrhotite dissolve at much faster rates than chalcopyrite, and the rock types in which these nickel minerals occur (basic igneous rocks) are more easily decomposed than the porphyry copper ores under the accelerated weathering conditions produced in a chimney. Molybdenite in molybdenum ores is soluble enough, but a base such as NaOH or NH_4OH would have to be added to maintain a pH above 7 in order to keep the molybdenum in solution.

OXYGEN DISTRIBUTION IN CHIMNEY

A key requirement in the successful application of this mining method is that a relatively high dissolved oxygen content must be achieved in a

Fig. 9. Chimney temperature vs time for various initial chimney temperatures.

Fig.10. Fraction recovery vs time for various initial chimney temperatures.

Fig. 11. Chimney temperature vs time for various ratios of pyrite to chalcopyrite.

Fig. 12. Fraction recovery vs time for various ratios of pyrite to chalcopyrite.

Fig. 13. Fraction recovery vs time for ore of different size distributions.

Fig. 14. Fraction recovery vs time for various initial chimney temperatures for Piledriver particle-size distribution.

much of the chimney solution as possible. This is important, because regions of the chimney having a low content of dissolved oxygen not only will have a low rate of dissolution of chalcopyrite, but also may have undesirable side effects such as depletion of the copper already in solution by precipitation of covellite. Although the maximum achievable oxygen content (i.e., the solubility limit of oxygen) will be dictated by the temperature and hydrostatic pressure within the chimney, the degree of oxygen saturation (i.e., the fractional part of the solubility limit that is actually reached) will be determined by factors related to the flow of oxygen gas and oxygenated solution through a fixed bed of solid particles. These factors are 1) the geometric dispersion of the input oxygen gas, 2) the rate of dissolution of oxygen, and 3) the rate of internal circulation of the chimney solution.

We have investigated these factors in detail both experimentally and theoretically [14]. The results of this study indicate that adequate dissolution of oxygen and circulation of the oxygenated solution can be obtained even if only one perforated oxygen line across the bottom of the chimney is used. This is accomplished, however, at the expense of venting a considerable fraction of the input oxygen as undissolved gas passing out of the chimney. The use of more input gas lines not only decreases this vent fraction, but gives added assurance that, if solution circulation problems arise, a large fraction of the chimney solution will still be adequately oxygenated.

When three input oxygen lines are used, 35% of the chimney solution is directly supplied with oxygen by contact with rising gas bubbles. The remaining 65% can be supplied with oxygen by the circulation induced by the column of rising gas bubbles. The faster the gas is introduced into the chimney the faster the circulation, but at the expense of a lower utilization of oxygen. Enough circulation to provide a saturation level of 61% throughout the chimney can be induced by a gas flow rate that utilizes 50% of the gas. At this rate, 50% of the oxygen is thrown away unless it is collected at the top of the chimney and recirculated.

ECONOMIC ANALYSIS

An economic analysis of the potential profitability of nuclear chemical mining of copper ores has been made [3]. A hypothetical 70-million-ton ore body with a copper content of 0.45% between 400 and 750 m deep would require eleven 100-kt nuclear explosions. Environmental studies, assay, drilling, and site preparation would require 1 yr to complete. Plant construction and the first detonation would be done in the second year. Thereafter, one detonation would be fired each year. Copper production would start in the third year and continue until 4 yr after the last explosion. Thus, a total of 16 yr would be required to complete the project. Major elements in relation to capital and operating cost are the oxygen plant, storage tanks, liquid extraction plant, electrowinning facility, and plumbing. The gross capital investment and reserve required is estimated to be \$13,140,000 excluding land acquisition. Average annual sales are estimated to be \$10,326,000 at \$0.43/lb copper sales price. Operating costs are estimated to be \$3,872,000 per year, and the capital investment, including working capital, would be returned from net cash income at \$4,157,000 per year in 3.16 yr. Over the 16-year project life, the internal rate of return on the investment would be 31%.

The effects of grade, tonnage, and depth of the ore deposit and of explosive size were examined by varying each in the same hypothetical mine. Changing the grade of copper ore has the effect shown in Fig. 15. Both

Fig. 15. Influence of grade of ore on the economics of nuclear chemical mining of copper. Explosive yield is held constant at 100 kt, depth of burst at 750 m (2460 ft), and ore body size at 70 million tons.

internal rate of return and capital investment decrease smoothly with decreasing copper content. The rate of return is 45% for an ore grade of 0.75 wt.% copper and decreases to 7% for an ore grade of 0.25 wt.% copper. Ore grades lower than 0.25 wt.% copper could be more profitably mined by detonating several explosions as capital investments prior to production, then simultaneously producing copper from several chimneys. For ore grades greater than 1 wt.% copper, the capital investment tends to become constant at about \$16 million. The direct operating cost, however, increases as the ore grade decreases, because the unit cost of the oxygen plant operation dominates the operating cost at low production rates.

The effects illustrated in Fig. 15 are for a 70-million-ton ore deposit. Variation of the tonnage of the deposit has little effect on either capital investment, rate of return, or direct operating cost until the ore body becomes so small that only a few explosives are required to shatter it. The minimum size is dependent on ore grade. For 0.5 wt.% copper, the return rate drops sharply when the ore body is reduced to 25 million tons. For 2 wt.% copper, the return rate is still 20% for 4 million tons of ore, requiring only one detonation.

The effect of depth of burst on the economic factors is shown in Fig. 16. As depth increases, the amount of broken rock decreases (see Fig. 1), causing a sharp increase in the unit operating costs for the drilling, casing, and coring. The result is that the internal rate of return drops sharply as depth increases, falling to 7% at about 1100 m. Larger explosive yield would extend the profitable working depth.

Varying the size of explosive affects the economics as shown in Fig. 17. Capital investment increases almost linearly with yield, due largely to increased oxygen plant costs. The rate of return increases rapidly between 50 and 100 kt, then more slowly. Return falls below 7% at 60 kt because not enough copper is produced from any one chimney to offset the fixed expense of drilling and casing and the increased oxygen costs. Thus, there is great incentive to use the largest yield allowed by seismic damage limitations.

ENVIRONMENTAL IMPACT

Detailed studies of the environmental impact of nuclear chemical mining have also been performed [3]. Potential risks to the immediate community out to about 80 km include risks from radioactivity and from ground motion due to the blast. In contrast to present conventional mining methods, however, there are no open pits, mine waste dumps, tailing ponds, or oxides of sulfur produced and therefore less risk of surface water contamination with this proposed mining plan. Furthermore, there is no radioactive isotope of copper in the product and, according to laboratory studies [15], no radioactivity from other elements in the copper when liquid ion exchange and electro-winning are used for processing, as proposed.

Radioactivity can potentially create hazards in four ways: 1) uncontrolled venting or seepage at the time of detonation, 2) atmospheric contamination with gaseous radioactivity carried with the off-gas oxygen and CO₂ during leaching, 3) personnel exposure to contaminated leach solutions, and 4) contamination of ground water. The probability of a release of radioactivity during the explosions is estimated to be less than 0.001, based on that part of the past nuclear test experience that is judged to be

Fig. 16. Influence of depth of burst on the economics of nuclear chemical mining. Ore body size is held constant at 70 million tons, explosive energy at 100 kt, and ore grade at 0.45%.

Fig. 17. Influence of size of explosive on the economics of nuclear chemical mining of copper. Ore body size is held constant at 70 million tons, ore grade at 0.45% copper, and depth of burst at 750 m (2460 ft).

comparable. How much less is not known, since no venting has accompanied the 80 test explosions of comparable yield, even though they were generally buried at shallower depths and in weaker rock than proposed for this application. Although the probability of an accidental release of radiation is very low, the consequences of such a release are always considered. Methods of ensuring the safety of the population who may be affected must always be guaranteed. These methods are well understood and vary with the location and characteristics of any particular site.

The second kind of radiation exposure is caused by release of radioactive krypton and tritium, which are released with the off-gases. This leads to very small downwind radiation exposure, calculated to be less than 0.01 of the natural radioactive background 1 km downwind from the release stack.

The third kind of radiation hazard is involved in the surface processing of the pregnant leach solution. Small amounts of radioactivity are leached from the radionuclides deposited on the fractured ore surfaces and in the ore melted by the explosion [15,16]. The amounts contained in these solutions produces external radiation fields in the surface processing plant that are comparable to those observed in uranium mills and that could easily be controlled with minimal shielding. These solutions are recirculated into the chimney after the copper is extracted. Small amounts of tritium and krypton gas in the plant can be held to safe levels by good solvent-control practice and adequate ventilation. Operating costs in the economic analysis have included costs for continued monitoring of air levels and personnel exposures to ensure safety in the processing plant.

Finally, hazard due to radioactive contamination of ground-water supplies is considered. In-situ leaching will not be carried out in regions having high ground water movement. Furthermore, solution loss from a nuclear chimney during the active period of leaching is controlled by maintaining hydrostatic equilibrium between the chimney and the surroundings, to prevent migration of solutions containing copper and radionuclides. Following shutdown of a site, even if some migration of solution from the chimney occurs, the hazard due to the small amount of radionuclides in solution would be small because of sorption on rock surfaces during underground movement. The cost for periodic monitoring of hydrologic test wells has been included in the economic analysis.

From experience with ground-shock effects from nuclear explosions conducted by the U.S. Atomic Energy Commission [17,18,19,20], shock effects on a given structure at a given distance from an explosion can be predicted quite accurately. Ground shock from 100-kt detonations will cause architectural damage out to a range of a few kilometers. The probability of masonry cracking at a range of 25 km is about 0.06 per structure, and the chance of damage at a range of 65 km and beyond is very small. Therefore, ore bodies selected for nuclear chemical mining should be in sparsely populated regions more than 65 km from urban areas. The plant facilities can be designed to withstand the ground shock if they are located at least 400 m from the surface over the ore body. Cost estimates cited above were prepared with this factor included.

Risk from both radioactivity and ground shock are very small, providing the ore bodies are selected with some care. Estimates of the environmental cost to society when this mining method is used are about one

tenth of the environmental cost of the conventional open-pit-flotation-roasting and fire-refining copper-recovery process.

CONCLUSION

We believe that the method of mining proposed here is technically feasible and economically very promising. Large increases in reserves of low-cost copper and nickel will be obtained if the process is successful. Production could be started rapidly, compared with conventional underground or open-pit mining, and with a relatively small capital investment.

Problems of ground shock and radioactivity are manageable, and environmental problems associated with conventional mining, such as atmospheric SO_2 pollution, open pits, waste dumps, and tailing ponds, are eliminated.

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FIGURE LIST

- Fig. 1. Mass of broken rock as a function of explosive yield and depth of burst.
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- Fig. 12. Fraction recovery vs time for various ratios of pyrite to chalcopyrite.
- Fig. 13. Fraction recovery vs time for ore of different size distributions.
- Fig. 14. Fraction recovery vs time for various initial chimney temperatures for Piledriver particle-size distribution.
- Fig. 15. Influence of grade of ore on the economics of nuclear chemical mining of copper. Explosive yield is held constant at 100 kt, depth of burst at 750 m (2460 ft), and ore body size at 70 million tons.
- Fig. 16. Influence of depth of burst on the economics of nuclear chemical mining. Ore body size is held constant at 70 million tons, explosive energy at 100 kt, and ore grade at 0.45%.
- Fig. 17. Influence of size of explosive on the economics of nuclear chemical mining of copper. Ore body size is held constant at 70 million tons, ore grade at 0.45% copper, and depth of burst at 750 m (2460 ft).