GL01211

SUBJ
MNG
ISEF

United States Patent [19]

Coursen

[11] 3,999,803

[45] Dec. 28, 1976

[54] IN SITU LEACHING OF EXPLOSIVELY FRACTURED ORE BODIES

[76] Inventor: **David Linn Coursen**, R.D. 2, 278-T,
Mercersburg, Pa. 17236

[22] Filed: **Dec. 23, 1974**

[21] Appl. No.: **535,808**

Related U.S. Application Data

[62] Division of Ser. No. 382,845, July 26, 1973, Pat. No. 3,902,422.

[52] **U.S. Cl.** .... 299/4; 75/101 R
[51] **Int. Cl.²** E21B 43/28
[58] **Field of Search** .... 299/4, 5; 166/247, 2[illegible]9, 166/271; 75/101 R

[56] **References Cited**

UNITED STATES PATENTS

689,835	12/1901	Waterbury	75/101 R
783,600	2/1905	Weterbury	75/101 R
3,278,232	10/1966	Fitch et al.	299/4
3,278,233	10/1966	Hurd et al.	299/4
3,860,289	1/1975	Learmont	299/4

Primary Examiner—Ernest R. Purser

[57] **ABSTRACT**

Producing a fracture network in deep rock, e.g., in an ore body, by detonating explosive charges sequentially in separate cavities therein, the detonations producing a cluster of overlapping fracture zones and each detonation occurring after liquid has entered the fracture zones produced by previous adjacent detonations. High permeability is maintained in an explosively fractured segment of rock by flushing the fractured rock with liquid, i.e., by sweeping liquid through the fracture zones with high-pressure gas, between sequential detonations therein so as to entrain and remove fines therefrom. Ore bodies prepared by the blast/flush process with the blasting carried out in substantially vertical, optionally chambered, drilled shot holes can be leached in situ via a number of holes previously used as injection holes in the flushing procedure and a number of holes which are preserved upper portions of the shot holes used in the detonation process. In the leaching of ore, fines are removed from fractures therein by intermittent or continuous flushing of the ore with lixiviant and high-pressure gas, e.g., air, using, in the case of the in situ leaching of an explosively fractured ore body, a lateral and upward flow of lixiviant from zones that have been less severely, to others that have been most severely, worked by multiple detonations in the ore body.

8 Claims, 4 Drawing Figures

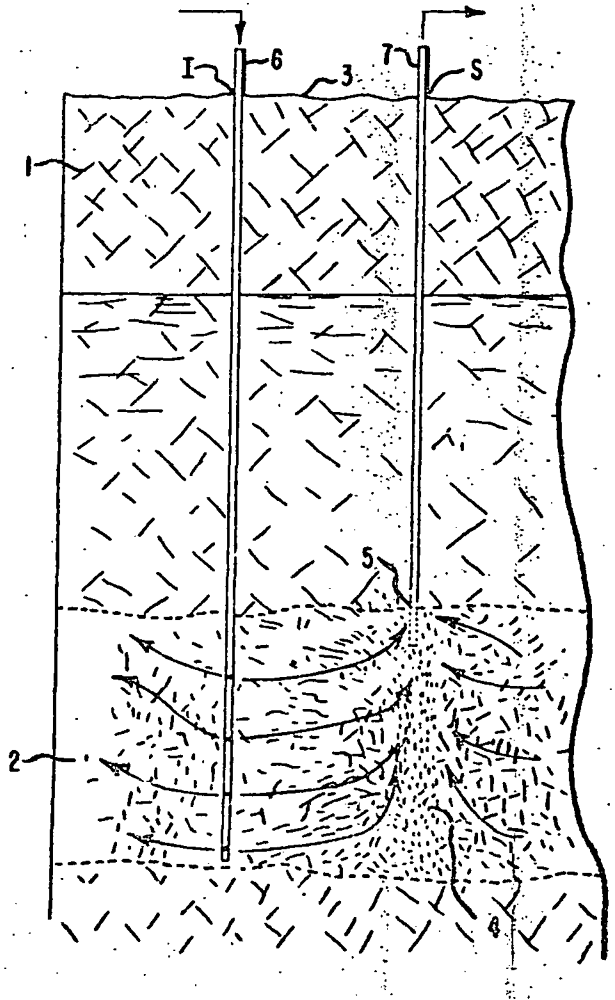


FIG. 1

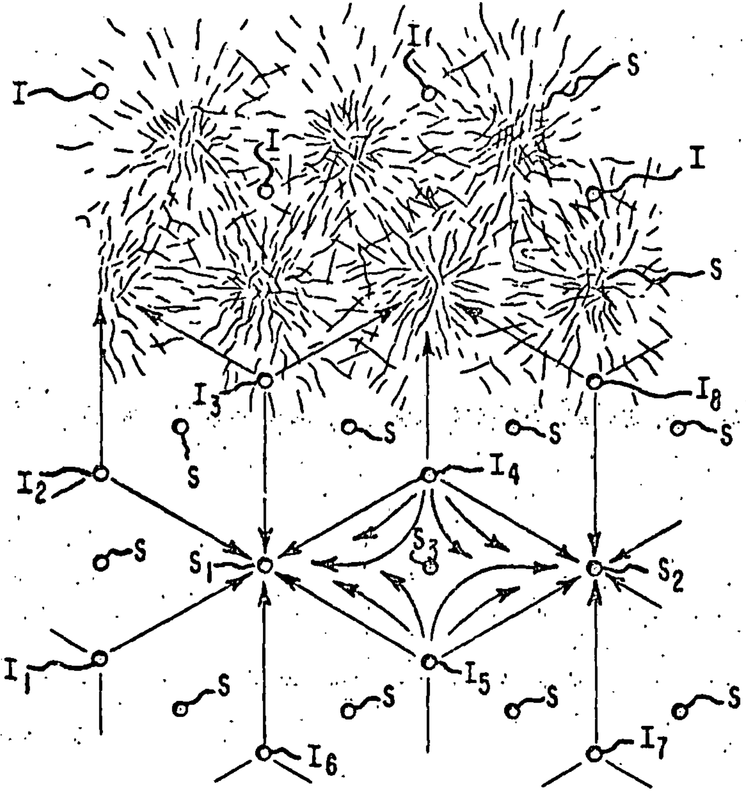


FIG. 2

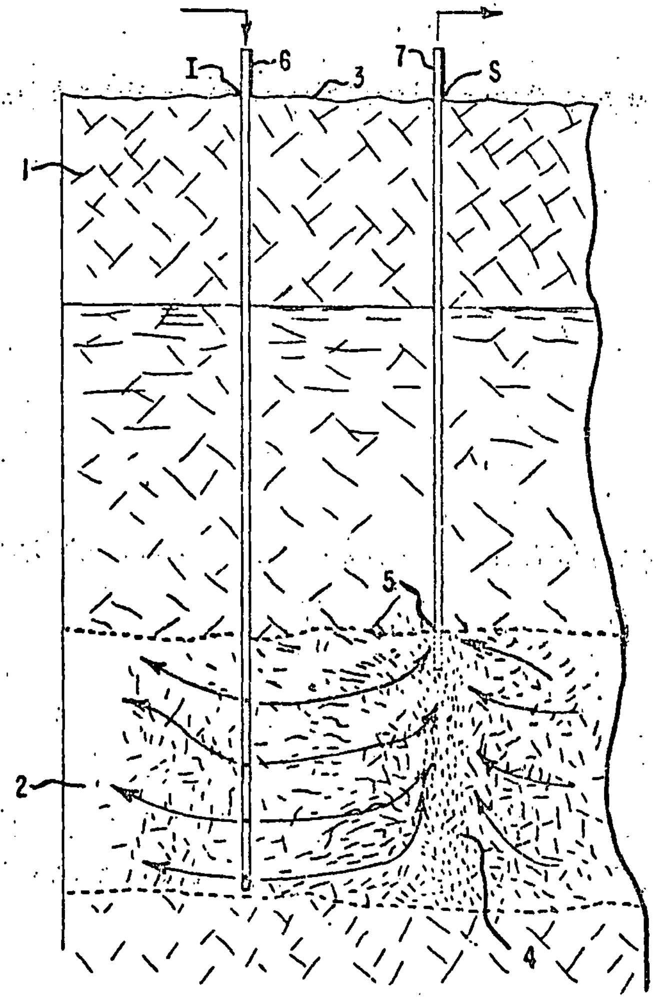
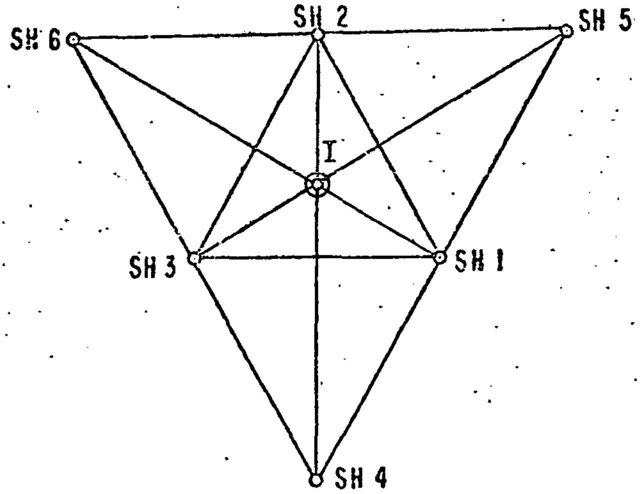
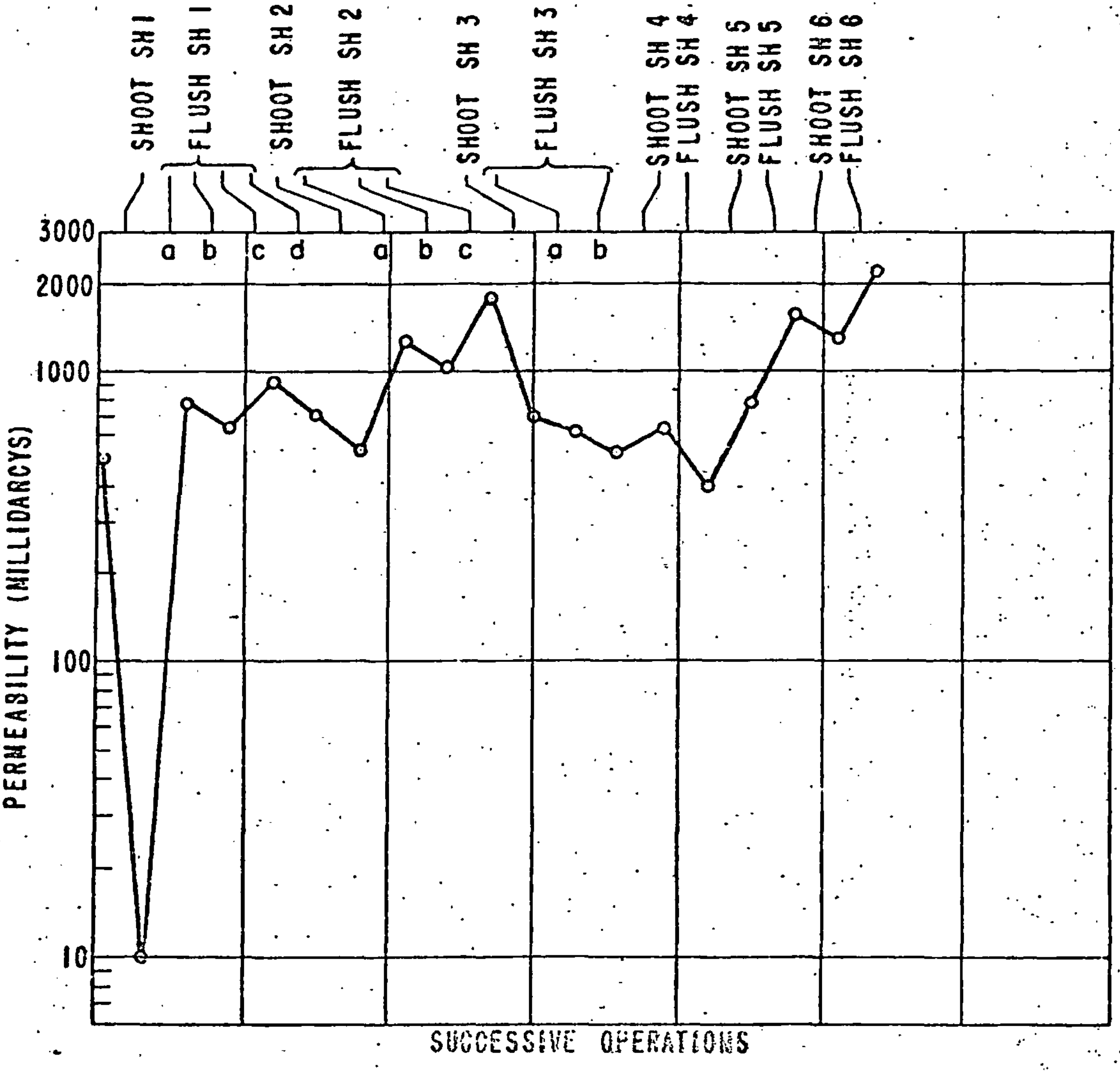


FIG. 3



SHOT HOLE PATTERN

FIG. 4



IN SITU LEACHING OF EXPLOSIVELY FRACTURED ORE BODIES

This is a Division, of application Ser. No. 382,345, filed July 26, 1973 and now U.S. Pat. No. 3,902,422.

BACKGROUND OF THE INVENTION

The present invention relates to the production of a network of fractures in a deep underground segment of rock by means of explosives, e.g., to prepare deep ore bodies for in situ leaching.

Processes for fracturing deep rock are becoming increasingly important as it becomes necessary to tap deep mineralized rock masses, e.g., ore bodies or oil or gas reservoirs located from about 100 feet to about a few thousand feet beneath the earth's surface, in order to supplement or replace dwindling energy sources and minerals supplies. Numerous deposits of ore, for example ore containing copper, nickel, or silver, lie too deep to mine by open-pit methods or are too low in grade to mine by underground methods. Open-pit methods incur both the costs and the environmental impact associated with moving large quantities of earth and rock. Underground methods incur unusually high costs per unit volume of ore mined, as well as difficult safety problems. In contrast, the leaching of ore in place circumvents these difficulties and therefore can be a preferred technique for winning values from some ores that are unsuitable, or marginally suitable, for working by traditional mining methods.

Usually however, ore that is favorably situated for leaching in place has such a large fragment size and such low permeability to leaching solutions that the leaching rate would be too low to support a commercial leaching operation. In such cases, it becomes necessary to prepare the ore for leaching, by fragmenting it in a manner such as to provide the necessary permeability and leachability. The use of explosives to fracture underground segments of mineralized rock to create areas of high permeability has often been suggested. In an oil- or gas-bearing formation the fracturing is required to increase the overall drainage area exposed to the bore of a well penetrating the formation, and thus increase the rate at which hydrocarbon fluids drain toward the well. In an ore body the fracturing is required to increase the surface area of ore accessible to an injected lixiviant, and thus increase the leachability.

The use of nuclear explosives has been proposed for fracturing large-volume, deep ore bodies for subsequent in situ leaching. Also, the use of multiple chemical explosive charges in deep reservoir rock has been described in a method for stimulating hydrocarbon-bearing rock, e.g., in U.S. Pat. 3,674,089. However, if a deep ore body, i.e., one lying at depths of about from 100 to 3000 feet from the surface, is to be effectively leached in place, and the ore prepared for leaching by blasting, i.e., blasting in the absence of a free face for the ore to swell toward, it becomes necessary to employ special blasting and associated techniques which will provide and maintain the type of fracture network required for efficient leaching.

The leachability of a fractured ore body depends on the size of the ore fragments, and on the permeability of the intact ore as well as of the fracture system separating the fragments. The permeability of the fracture system separating the fragments, which is variable and generally much higher than the permeability of a single fragment, is determined by a network of wider, open fractures (determining the permeability of the ore body as a whole), and a network of narrower, open fractures (determining the irrigability of individual particles to be leached). Therefore, in explosively fracturing a segment of an ore body to prepare it properly for in situ leaching, the objective is not simply an indiscriminate reduction in the fragment size of the ore body. Smaller-size, well-irrigated fragments have a higher leaching rate than larger-size fragments, but fragment-size reduction by means of blasting processes heretofore known to the art, when applied to deep ore, tends to leave large unbroken fragments or rock, or to create a network of fractures that are largely closed or plugged with fines. An explosive fracturing process is needed which reduces the larger fragments to a size that will leach at an economically acceptable rate, and that will result in a network of open fractures throughout the blasted ore that will permit it to be well-irrigated with leach liquid.

SUMMARY OF THE INVENTION

This invention provides a process for producing a fracture network in a deep subsurface segment of rock, e.g., in an ore body, comprising (a) forming an assemblage of cavities, e.g., drill holes or tunnels, in the segment of rock; (b) positioning explosive charges in a plurality of the cavities in the sections thereof located in the segment of rock to be fractured, e.g., in sections of drill holes which have been previously chambered, such as by an explosive springing procedure; (c) providing for the presence of liquid in the segment of rock, e.g., by virtue of the location of the segment of rock below the water table so that water naturally is present in, or flows into, fractures therein, or by introducing liquid into one or more cavities therein; and (d) detonating the charges sequentially in a manner such as to progressively produce a cluster of overlapping fracture zones, the detonation of each charge in the detonation sequence producing a fracture zone which is subject to the cumulative effect of a succession of detonations of explosive charges in a group of adjacent cavities, and the detonation of the charge in each cavity being delayed until liquid is present in fracture zones produced by the previous detonation of charges in cavities adjacent thereto, as determinable by measuring the hydraulic potential, e.g., the liquid level, in the cavity, or in a cavity adjacent thereto.

When the cavities formed are substantially vertical drill holes, some of the holes in the assemblage preferably are left uncharged with explosive, and these holes employed as a set of passageways within the fracture network from the earth's surface, generally to substantially the bottom of the blasted rock, e.g, for the introduction of liquid and/or gas to (or removal thereof from) the fracture network. The uncharged holes preferably are drilled and provided with support casing prior to the detonation of charges in adjacent holes. The sections of substantially vertical shot holes located in the overburden that overlies the rock segment to be fractured preferably survive the blasting process and serve as an additional set of passageways, leading from substantially the top of the blasted rock to the earth's surface, also for liquid and/or gas passage.

In a preferred explosive fracturing process, liquid is driven through the fracture zones produced by the sequential detonation of explosive charges in a plurality of cavities in a segment of rock, in a manner such as to entrain the fines found in the fracture zones, and the fines-laden liquid removed from the rock. This flushing of the blasted rock is achieved by sweeping or driving liquid at high velocity through the fracture zones by injecting gas into said zones at high pressure, the liquid moving laterally and upwardly through the blasted rock, passing into the fractures, for example, from the passageways formed by uncharged substantially vertical drill holes and out of the fractures into passageways formed by preserved sections of substantially vertical detonated holes located in the overburden. Best results are achieved when substantially each detonation is followed by a flushing step applied to the fracture zone thereby produced, before the next detonation in an adjacent cavity occurs, and this is preferred. In the leaching of a mass of ore, e.g., in the situ leaching of an explosively fractured ore body or in dump leaching, fines also preferably are flushed out of fractures therein by sweeping the lixiviant therethrough at high velocity by high-pressure gas.

The term "deep" as used herein to describe a subsurface segment of rock denotes a depth at which the detonation causes no significant change in the overlying topography, i.e., the surface does not swell. As a rule, deep rock as described herein lies at a depth of at least 100, and usually not more than 3000, feet. "Fracture zones" and "fractured rock" herein denote zones and rock in which new fractures have been formed, or existing fractures opened up, by the detonations. "Fracturing" denotes herein a treatment which reduces the size of, and/or misaligns, rock fragments.

BRIEF DESCRIPTION OF THE DRAWING

The explosive fracturing process of the invention will be described with reference to the attached drawing in which

FIG. 1 is a schematic representation in plan view of a subsurface segment of rock which has been fragmented by the blast/flush process of the invention, and the liquid circulation pattern between holes therein;

FIG. 2 is a schematic representation in elevation showing the surface-to-surface liquid circulation pattern through the segment of rock shown in FIG. 1;

FIG. 3 is a schematic representation of a shot hole pattern described in the example; and

FIG. 4 is a plot showing the effect of repeated blast/flush operations on the permeability of a fracture zone produced with the shot hole pattern shown in FIG. 3.

DETAILED DESCRIPTION OF THE INVENTION

In the present process, explosive charges are detonated sequentially in separate cavities in a segment of mineralized rock to be fractured, each detonation in the sequence producing a zone of fracture in the rock and being delayed until liquid is present in the fractured rock around the cavity containing the charge to be detonated, especially in fracture zones produced by the previous detonation of charges in cavities adjacent thereto. Thus, the detonations occur while fractures in the surrounding rock are filled with liquid, or the rock is in a flooded, or liquid-soaked, condition. The cavities, e.g., drill holes or tunnels, containing the explosive charges are spaced sufficiently close together, and the charges are sufficiently large, that the fracture zones produced by the detonations therein overlap one another. Thus, each fracture zone is within the region of influence of other detonations and is subject to the cumulative effect of a succession of detonations of explosive charges in a group of adjacent cavities. This cumulative effect permits the fragment size-reduction and disorientation needed to enhance leachability to be obtained readily from the available explosive energy. The degree of overlapping of the fracture zones, which are generally cylindrical in shape, is at least that required to locate all of the rock, in the segment of rock to be fractured, within the fracture zone produced by the detonation of at least one of the charges.

The cavities in the assemblage in which explosive charges are to be detonated (i.e., blast cavities) can be substantially vertical holes (shot or blast holes) drilled into the segment of rock from the surface or from a cavity in the rock, or substantially horizontal cavities such as tunnels, driven in the rock, e.g., from a hillside or shaft. Whether the cavity volume is provided by tunnel driving techniques such as are employed in coyote blasts, for example, or drilling techniques, possibly associated with chambering procedures, will be largely a question of economics, although technical practicability depending on such factors as topography, compressive strength of the rock, etc., will influence the selection of the method. Substantially vertical drill holes are preferred in many cases since the preserved sections of the shot holes can be used subsequently as passageways to or from the fractured rock, reducing the number of holes needed to be drilled solely to provide passageways for liquid injection or ejection.

Although the blast cavities need not form a regular pattern, and regularity of pattern actually may not be desirable or practical, a somewhat regular pattern is indicated in a formation of reasonably uniform contour, structure, and physical strength to assure a high degree of uniformity in the fracture network produced. In some cases, as core tests reveal unpredictable changes in the rock occurring during the sequential blasting process, it may be desirable to deviate from a regular pattern, e.g., to use one or more additional blast cavities where needed to provide the required overlapping of fracture zones. Nevertheless, substantial regularity of pattern generally will be provided in the arrangement of most of the blast cavities. It will be understood, of course, that in the case of substantially vertical drill holes the actual pattern of the holes within the segment of rock to be fractured may approach, rather than match, the hole pattern at the surface, inasmuch as the available drilling equipment may not be counted on to produce parallel holes at depths of the order considered herein.

Regardless of the blast cavity pattern employed, the distance between explosive charges (and, also therefore, between cavities) of a given composition and size is such that a cluster of overlapping fracture zones is produced by the detonation of adjacent charges. Although it may not be possible to delineate the fracture zones with precision, the extent or radius of the fracture zone that can be expected to result from the detonation of an explosive charge of a given composition, density, shape, and size under a given amount of confinement in a given geological mass can be approximated by making some experimental shots and studying the fracture zones surrounding the blast cavities by using one or more geophysical methods. Such methods include (1) coring, (2) measurements in satellite holes of compressional and shear wave propagation, of permeability, and of electrical conductivity, and (3) acoustic holography. Based on these studies, the cavities are spaced close enough together to provide the required overlapping of fracture zones.

"Adjacent" blast cavities or explosive charges, as described herein, are blast cavities or explosive charges which, although spaced from one another, are immediate or nearest neighbors to one another, as contrasted to blast cavities or explosive charges which are more distant neighbors or separated from one another by one or more other blast cavities or explosive charges.

Although I do not intend that my invention be limited by theoretical considerations, the delaying of each detonation until liquid is present in the fracture volume surrounding the cavities is believed to have two beneficial effects. First, the liquid can lubricate the fractures so that opposing faces can move suddenly in shear more easily, thereby enhancing fragmentation of the surrounding rock, which is no longer supported by the relatively high resistance of a dry fracture to transient shear. Secondly, liquid-filled fracture volume cannot be rammed shut by the suddenly applied pressure of an explosion. This incompressible behavior, together with the low resistance of the liquid-filled fractures to sudden small displacements in shear, is believed to cause disorientation of individual rock fragments and dilation and swelling of the bed of fragments as a whole. Each detonation creates a misalignment or disarrangement of fragments with an accompanying increase in void volume. Therefore, when the fracture zones produced by the successive detonations in adjacent cavities partially overlap, the fracture zone around each cavity thereby being subject to additional fracturing and/or disorientation produced by the detonations in the adjacent cavities, and previously produced fracture zones are flooded, each fracture zone will be swelled in increments, with each detonation jacking it to larger volume, and higher permeability, against the pressure of the surrounding rock. The present process makes use of the lubricating effect and incompressible behavior of the liquid in the fractures, and does not require the use of high liquid pressures, e.g., of the magnitude needed to lift the overburden and enlarge the fractures before blasting. A liquid pressure in the fractures at the time of blasting equal to the head of liquid above the blast zone is sufficient. Also, any readily available, relatively cheap liquid, e.g., water or water mixtures, can be used to flood the rock. If leaching of ore is performed in the course of the detonation sequence, a lixiviant can be used as the flooding liquid. For reasons of economy as well as because of the safety risks associated with the use of explosives which are sensitive enough to detonate in extremely small diameters, the use of explosive liquids in the fracture zones is not contemplated. Any fluid explosive which may be used in the present process will be gelled to a viscosity that will hinder any appreciable loss thereof from the blast cavities to the surrounding fracture zones, and in any case will not be sufficiently sensitive to be detonated in said zones. Thus, while small amounts of the explosive charges may escape into the fracture zones, such material will behave as a non-explosive liquid therein. Accordingly, the flooding liquid is non-explosive.

A preferred blast cavity pattern for use in the present process is one in which substatially all of the internal cavities, i.e., cavities not located at the edge of the pattern, are surrounded by at least four adjacent blast cavities, e.g., a pattern in which the blast cavities are at the corners of adjacent polygons, which are either quadrangles or triangles and which are as close to equilateral as permitted by wander of the cavities, as shown in FIG. 1.

Although all of the blast holes in a group of adjacent substantially vertial drill holes can be drilled prior to the sequential detonation of the charges, this procedure is not preferred inasmuch as it could be necessary to apply a support casing to the as-yet undetonated holes in the sections thereof located in the segment of rock to be fractured to prevent them from collapsing as a result of detonations in adjacent holes. Casing of the shot holes in these sections usually would be considered economically unsound because the casing would occupy volume that could otherwise be loaded with explosive and because casing in these sections of the holes is not needed in subsequent leaching operations. Therefore, it is preferred that in a group of adjacent drilled shot holes the detonation of each charge takes place before adjacent shot holes are drilled. In practice, one might drill and, if desired, chamber (as described later), one shot hole of a group of adjacent holes, load the hole or chamber with explosive, allow water to enter the formation surrounding the hole or chamber, and detonate the charge, and then repeat the sequence of steps with adjacent holes. In each successive sequence of steps, the entrance of water into the formation can occur prior to, or during, any of the other steps, however. The avoidance of the presence of drilled shot holes during detonations refers to holes in a group of adjacent holes, e.g., a central hole and four to six surrounding holes. However, shot holes farther removed from the detonations can be pre-drilled.

The total amount of drilling needed for vertical-cavity blasting can be reduced by drilling one or more branch or off-set holes by side-tracking from one or more points in the preserved upper portion of a trunk hole which extends to the surface. Each off-set hole is drilled after the charges in the trunk hole and other off-set holes thereof have been detonated. Such holes will be inclined at small angles to one another.

Most of the ore bodies and other mineralized formations to which the present process is expected to be primarily applicable will be located below the water table, and in such a case, unless the section to be blasted rises locally aboe the water table, or the rock surrounding this section is so impermeable that flooding of the fracture zone does not occur by natural flow, the section will be naturally flooded, or water-soaked, before the sequential blasting begins, and after a certain period of time has elapsed after each detonation to allow the water to flow naturally into the newly formed fractures. If natural flooding is incomplete or absent, water or some other liquid can be pumped into the cavity to be shot after the explosive charge has been emplaced therein, and also into any available nearby uncharged cavities, at a sufficiently high flow rate to cause the rock to be blasted to be in a flooded condition at the time of detonation.

As stated previously, liquid is present in the rock around each cavity prior to the detonation of the charge therein. This means that liquid is present in any pre-existent fractures in the zone which will become a fracture zone as a result of the detonation of the charge in that cavity, and in fractures produced by previous detonations in cavities adjacent thereto. This condition permits the above-described incremental swelling of overlapping fracture zones to take place. In the case of substantially vertical drill holes, the liquid level in the rock around the hole should be at least as high as the top of the charge in the hole, thereby assuring the presence of liquid throughout the height of the formation where fracturing will occur. With horizontal cavities, the liquid level in the rock around the cavity should be at least as high as the radius of fracture to be produced by the detonation of the charge therein. When the segment of rock to be fractured is located below the water table, the position of the water table above it will conform to the water levels in undisturbed holes, and may be inferred at other locations by interpolation between the elevations of the water levels in undisturbed holes. As a practical matter, the water table will almost always be sufficiently horizontal that the first charge can be detonated when the elevation of the liquid level in any nearby hole is at least as high as the elevation to be reached by the top of the charge (or radius of fracture in the horizontal cavity case). If the liquid level is measured in the cavity in which the charge is to be detonated, the level before loading of the explosive into the cavity should be the level measured. After the detonation, the liquid level in cavities within the resulting fracture zone drops in proportion to the new fracture volume produced, the expulsion of liquid from the immediate vicinity of the charge by the gaseous products of detonation, and the drainage of liquid into the cavity created by the detonation. The detonation of the next charge in the sequence in a cavity adjacent to the first is delayed until the liquid in the formation around the next cavity (including the new fracture volume produced by the previous detonation in an adjacent cavity) returns to its required level. It is understood, however, that explosive charges in blast cavities elsewhere in a section of the formation that is not strongly influenced by a previous detonation (i.e., where the liquid level has not dropped below the required elevation as a result of the previous detonation) can be detonated at any time after the previous detonation. The delay to allow flooding applies to detonations in cavities which are adjacent to previously detonated cavities, where the previously formed fracture zones will be subject to the effect of the next detonation.

As was stated previously, some of the holes in an assemblage of substantially vertical holes preferably are left uncharged with explosive, these holes providing passageways to the fractured rock to allow the introduction of gases and/or liquids thereto, e.g., in a subsequent leaching operation. These holes, which can thus be looked upon as injection holes (although they may serve as ejection or recovery holes depending on the required flow pattern), are also useful in preparing the ore body for leaching, as will be described more fully hereinafter, and it is preferred, on the basis of ease of drilling, that they be drilled prior to the sequential detonation process in holes surrounding them. Predrilled injection holes are provided with a support casing, e.g., unperforated pipe grouted to the upper part of the hole wall, at least in the section thereof located in the segment of rock to be fractured, and ungrouted perforated pipe or a wellscreen in the bottom section of the hole, in order to prevent hole collapse as a result of the detonations. Inasmuch as full-length casing will be required for subsequent leaching operations, however, the full length of the injection holes usually will be cased prior to blasting. Damage to the injection piping is minimized in the present blasting process owing to the sequential, long-delay character of the multiple detonations.

The location and pattern of the injection holes are selected on the basis of their intended function during the fracturing and leaching processes, which will be described in detail hereinafter. The overall purpose of these holes usually is to provide a means for introducing gases and/or liquids into the fracture network produced, or being produced, and therefore the injection holes should be distributed throughout the segment of rock among the blast cavities in a manner such that they lie within the fracture zones produced by the detonations. After the detonation of the charge in a substantially vertical shot hole, the resulting fracture zone permits communication between a neighboring injection hole and the portion of the shot hole remaining in the overburden. The shot hole remnants thereby act as passageways to complete the liquid circuit through the fractured rock.

If injection holes are present in the formation during the sequential detonation process, an injection hole lying within the fracture zone produced by a previous detonation in a cavity adjacent to a cavity to be shot can be employed to determine whether the liquid level in the rock surrounding the cavity to be shot has recovered sufficiently to flood the section to be blasted. Whenever an hydraulic potential (e.g., a liquid level) measurement is required after a blast cavity has been loaded with explosive, a nearby injection hole can be used. When the segment of rock to be blasted is at least partly above the water table, liquid is introduced into the rock in the cavity to be shot, in previously detonated cavities adjacent thereto; and/or in nearby injection holes. Flooding via multiple cavities is preferred. Liquid is run into a blast cavity after the explosive charge has been emplaced therein (if the charge is stable in the presence of water), and liquid level measurements, if required, are made in nearby injection holes. It should be understood that, in practice, hydraulic potential measurements, e.g., pressure measurements made with a piezometer, or liquid level measurements, will not be required after each detonation, inasmuch as the experience gained in determining the necessary delay times to permit recovery of hydraulic potential between a few of the early detonations in the sequence will usually allow the practitioner to select with confidence suitable delay times to be used between subsequent detonations.

Although the exact delay required depends on the size of each blast, the void volume to be filled, the elevation of the segment to be blasted relative to the water table, and the hydraulic transmissibility of the surrounding rock, delays on the order of hours or days generally will be needed. As a practical matter, the time required for a shot hole to be drilled, or a tunnel to be driven, and loaded with explosive usually will be more than sufficient for the hydraulic potential around the cavity and the previously detonated adjacent cavities to recover to the minimum required level either by natural influx of water from the surrounding rock or by introduction through cavities made in the formation. In general, delay times between detonations of at least about one hour, and typically in the range of about from 4 to 24 hours, are sufficient for flooding to take place, although much longer delays, e.g., in the range of about from 4 to 30 days, may be employed in order to prepare the next blast cavity for blasting. It will be understood that these delays refer to the time between detonations of adjacent charges, and that one or more charges whose zones of fracture are non-adjacent (i.e., whose regions of influence are mutually exclusive) can be detonated at much shorter delay times or even simultaneously.

I have found that when sequential blasting is carried out in less competent, broken, or clayey rock, the permeability of the rock may be decreased, although the fracture volume is increased, by the blasting. Lost permeability can be restored by flushing of the fractured rock, i.e., by sweeping or driving liquid through the fractures at high velocity and removing the fines-laden liquid from the rock, preferably after each detonation. The flushing procedure appears to remove from the fractures the clogging fines that prevent free irrigation around the rock fragments. Such fines are present in the form of exisiting clays and rock crushed or abraded during blasting.

The flushing can be accomplished by the pressure injection of liquid and gas into the fractured rock through one or more injection holes, and removal of the fines-laden liquid from the fractured rock by bringing it to the surface through one or more detonated shot holes, in the preserved sections of the latter which pass through the overburden to the surface. Liquid and gas, e.g., water or other aqueous liquid and air or oxygen, can both be injected; or gas alone can be injected so as to sweep ahead the liquid already present in the fractures. Alternatively, a liquefied gas, such as air, nitrogen, oxygen, can be introduced into the injection holes and allowed to vaporize therein and thereafter drive the liquid through the fractures. Inasmuch as there is a two-phase flow in a generally upward direction and laterally in the direction of the detonated shot holes, the circulation of the liquid is powered by gas lift such that the gas chases the liquid upward and outward through the broken formation, and fins are driven toward the zones of severest fracture, where their concentration is heaviest, from which zones they are ejected with the liquid. This direction of sweep is preferred inasmuch as the reverse direction drives the fines more deeply into the less severely worked zones of the formation away from their point of heaviest concentration and can cause an intensified clogging of the fractures. The surging high-velocity flow which develops with the upward two-phase flushing system removes fines that prevent free irrigation around the fragments. If necessary to achieve the required lateral circulation of liquid between injection hole and ejection hole throughout the length of the fracture zones being flushed, two or more vertically separated injection zones in a given injection hole can be employed, one substantially at the bottom of the fractured rock and one or more others above it.

The buoyancy of the pressurized gas alone can be sufficient to raise the fines-laden liquid to the surface of the ground when the water table is relatively close to the surface. When the water table is so deep that the buoyancy is insufficient, the liquid can be pumped up the collar of the shot hole.

At the start of flushing, the gas injection pressure should be higher than the ambient hydrostatic pressure at the position in the injection hole where injection occurs, and preferably higher than the lithostatic pressure at this position. The minimum gas pressure required for flushing is highest at the start of the operation and falls as gas injection proceeds.

Although there can be much variation in the number of fracture zones being flushed out at any given time, and the nature and number of other operations which can be performed during flushing, it is preferred that a detonation in any given cavity be followed by detonations in no more than two or three adjacent cavities, and most preferably by a detonation in no adjacent cavity, before the fracture zone produced by the detonation in the given cavity has been flushed out as described. In some formations, if a given fracture zone is subjected to a number of subsequent detonations without the intervention of flushing, restoration of permeability by a later flushing becomes difficult because the fractures may have become plugged up too tightly with fines. Therefore, a cyclic blast/flush/blast/flush, etc. process is preferred. One or more fracture zones can be flushed at the same time, and flushing of the same zone can be repeated, if desired. An already flushed zone can be left untreated during the flushing of adjacent zones by plugging the ejection hole in that zone. Flushing of one or more zones can be carried out while adjacent blast cavities are being drilled and loaded.

In the present process, the detonation of the charges in sequence permits the preservation of the sections of substantially vertical shot holes that pass through the overburden (the strata overlying the rock segment being worked), and these sections of the shot holes can serve as ejection holes in the flushing process, as described above. The reduced fragment size and unclogged fracture network achieved after all of the charges have been detonated, and the detonations followed by a flushing procedure, produce, in the case of an ore body, an ore which is well-prepared for in situ leaching.

The present invention also provides a leaching process wherein fines are flushed out of a mass of ore by driving lixiviant through the mass by means of high-pressure gas, e.g., in a specific circulation pattern. According to one embodiment of the present leaching process, an ore body which has been prepared for leaching by detonating explosive charges in separate cavities therein, e.g., accoding to a process of this invention, is leached in situ by introducing lixiviant for the ore into the prepared ore body through a plurality of injection holes therein and intermittently or continuously driving the lixiviant through the ore body to a plurality of recovery holes by means of high-pressure oxidizing gas, the lixiviant moving laterally and upwardly from zones that have been less severely worked, to others that have been most severely worked, by the detonations, whereby fines are removed from the ore body. When the ore body has been prepared for leaching by means of the above-described blast/flush process the lixiviant for the ore can be injected into the ore body through injection holes which have been previously been used in the flushing steps, and fines-laden pregnant leach solution recovered from the ore body through the preserved upper portions of shot holes, piping having been grouted into all holes used to circulate lixiviants and pumps provided as necessary to inject lixiviants in one set of holes and remove pregnant liquor from another set of holes. The bottom ends of the pipes and any other positions along the pipes where lixiviants are to be injected or collected are provided with perforations or wellscreens.

The lixiviant (e.g., sulfuric acid/water or sulfuric acid/nitric acid/water for ores whose acid consumption is within tolerable levels, or NH_4OH/water for ores having a high acid consumption), which is a liquid, and a gas, usually an oxidizing gas, preferably oxygen, air, NO_x, or mixtures thereof, are injected into the base of the prepared ore body at high pressure. As in the case of flushing between blasts, this type of injection gives a circulation powered by gas lift such that the gas chases the liquid through the broken rock. Even with constant flow rates of gas and liquid at the injection holes, a surging, high-velocity flow develops in the rock which is believed to be beneficial in (1) removing fines around the ore fragments (such fines being created during the leaching process in forms such as decrepitated ore slimes and precipitated iron salts), (2) increasing the leaching rate as a result of the cyclic squeezing of the ore fragments from the pressure fluctuations associated with the surging flow, and (3) working the ore gently so as to collapse wide openings among the fragments that may develop during the leaching process and can cause channelling of leaching solution. Sweeping the lixiviant laterally toward collection points in the more severely worked fracture regions of the ore body, and from injection points in the less severely worked regions reduces the chances that a more intense clogging of the ore body with fines will occur.

The circulation pattern employed in the leaching process as well as in the flushing steps of the fracturing process may be understood more clearly by reference to the accompanying drawing. In FIG. 1, the holes designated by the letter S are substantially vertical shot holes. Within the blasted segment of rock, these holes are destroyed by the detonations which have taken place therein in the fracturing process and are replaced by the adjacent, overlapping fracture zones shown in the upper half of the figure, and also denoted by the letter S, to indicate a previous shot hole. The shot holes rather than the fracture zones are shown in the lower half of the figure so that liquid circulation lines can be indicated clearly. It should be understood, however, that upon completion of the entire blast sequence all shot holes are surrounded by fracture zones (as depicted in the upper half of the figure) in the sections thereof located in the rock segment that was blasted. In the sections overlying the blasted segment, the shot holes remain substantially intact and in these sections all shot holes appear as they are shown in the lower half of the figure. The preserved upper sections of the shot holes are ejection holes in the flushing steps of the blasting process, and recovery holes in the leaching process. In the hole arrangement illustrated in FIG. 1, the shot holes are arranged in a trigonal pattern wherein lines between adjacent holes form substantially equilateral triangles.

The holes designated I are injection holes. These holes are uniformly distributed among the shot holes as shown. The arrows indicate the direction of flow of liquid from injection holes I_1, I_2, I_3, I_4, I_5 and I_6 to the preserved upper section of shot hole S_1; and from injection holes I_4, I_5, I_7, I_8, and two other undepicted injection holes to the preserved upper section of shot hole S_2. The preserved upper section of shot hole S_3 is plugged off while shot holes S_1 and S_2 are being used for flushing or as recovery holes for pregnant leach solution. At the same time, liquid injected into these injection holes is being driven to other open shot holes.

In FIG. 2, piping in injection hole I and shot hole S is shown as it passes through overburden 1 to the fractured rock segment 2. Piping 6 in injection hole I leads from the earth's surface 3 to substantially the bottom of rock segment 2. Piping 7 in shot hole S leads from the earth's surface 3 to the top of rock segment 2. Fracture zone 4 has been produced by the detonation of an explosive charge in shot hole S, which before the detonation led to substantially the bottom of rock segment 2. Piping 7 terminates in well screen 5, and piping 6 is provided with perforations vertically spaced along the length thereof located in rock segment 2. In the flushing steps of the fracturing process, and in the leaching process, liquid is injected into fractured rock segment 2 through the perforations in piping 6, then is driven by pressurized gas through the fractured rock as indicated by the arrows, and leaves the top of the rock segment through piping 7. Lateral as well as upward flow occurs from the less severely worked zone around hole I to the most-severely worked zone, i.e., fracture zone 4.

Regulation of the rate at which gas and liquid lixiviant are injected and collected at the various injection and collection holes allows a high degree of control of the in situ leaching process. By the operation of control valves, the injection and collection pressures can be regulated to obtain a relatively uniform flow through the ore body in spite of variations in permeability from place to place. Shifting the injection or collection from one set of holes to another will change the direction of flow through the ore and can be used to frustrate channelling. The regulation of pressures and flow rates at the various holes can be used to maintain a net flow of ground water toward the operation under conditions that might otherwise result in the escape of leach solution. Leakage of the leach solution is also reduced in the present process as a result of the carriage of some of the fines away from the area of gas agitation where they settle out and plug the leak. In leaching, the gas/liquid pressure injection can be intermittent or continuous, depending upon the degree to which the ore tends to plug up, and the frequency with which flow patterns are changed to obtain uniform and complete leaching throughout the ore.

When lixiviant is introduced into an injection hole simultaneously with gas, its injection pressure should be equal to that of the gas, i.e., higher than the ambient hydrostatic pressure at the injection point, and preferably higher than the lithostatic pressure at this point. In some cases, especially at greater depths, the injection of lixiviant and oxidizing gas at sufficient pressure to exceed the lithostatic pressure may be necessary in order to get sufficient flow rate through the ore. If, in some or all of the injection holes, there are periods of time when lixiviant alone is introduced into the ore, this introduction preferably is done at a pressure at least as high as the lithostatic pressure at the injection position. That is, the pumping pressure preferably is at least as high as the lithostatic pressure minus the heads of fluid in the piping leading from the pump to the injection position.

According to the present invention, permeability can be increased also in ore masses such as mine waste dumps by driving lixiviant through fractures therein by means of gas at sufficiently high pressure that the lixiviant is swept through at a rate sufficiently high to entrain fines present in the fractures, and removing the fines-laden lixiviant from the ore mass.

In a preferred embodiment of the present process, the sections of substantially vertical shot holes which are located in the segment of rock to be fractured are first chambered to larger diameter, and the explosive charges positioned in the chambered portions. In this procedure, drilling costs are reduced by drilling widely spaced-apart shot holes of smaller diameter than is required to accommodate the size of explosive charges to be employed, and enlarging or "springing" the lower parts of the shot holes to produce chambers having the volume required to hold the explosive charge. The sections of the holes in the rock segment are chambered either by drilling them out, e.g., with an expansion bit, or by detonating explosive charges therein. The chambering method is not critical, the preferred method generally being the one that results in the lowest overall cost per unit of chamber volume for the particular rock segment in question. In the present process, explosive charges used for springing may be 20 feet or more in length. If rock fragments tend to fall from the walls of an explosively sprung hole and thus to occupy some of the volume required for the explosive charge subsequently to be used in producing the fracture zone, the hole to be sprung can be drilled deeper so that the bottom of the hole is located below the bottom of the formation. In this manner, any loss in volume that is to be available for explosive loading is minimized since a portion of the chamber volume below the segment of rock to be fractured can hold the fallen rock fragments.

The advantage of chambering the shot holes before loading them with the charges which will be detonated to produce the fracture network becomes evident when it is considered that an explosively sprung hole typically will hold about ten times as much explosive as an unsprung hole. Thus, for example, a pattern of 30-inch-diameter charges on 100-foot spacings (center-to-center) typically can be achieved by drilling 9-inch-diameter holes on 100-foot spacings.

Although the blast/flush process has utility in deep underground blasting with explosives of all types, the use of chemical explosive charges is much preferred for several reasons. The many technical as well as civil (legal, political, public relations) problems associated with the undertaking of nuclear blasting are self-evident. Vibration effects and radioactivity are the two major roots of these problems. A nuclear blast which is large enough to be economically feasible must be set off at sufficient depth, e.g., preferably appreciably deeper than 1000 feet, in order to be safely contained and not release radioactivity to the atmosphere. Many potentially workable ore bodies will not be located as deep as the safe containment depth. Furthermore, the extreme magnitude and concentration of the energy produced in a nuclear blast imply that it will be difficult, if not impossible, to achieve (a) a high degree of uniformity in explosion-energy distribution and ore breakage, (b) close hydraulic control of the flow of lixiviants without an appreciable amount of additional drilling to increase the number of injection and extraction points, and (c) a close match of the broken volume with the outline of the ore body, particularly for small or irregular ore bodies, such a match resulting in economies in the use of the available explosive energy and in the consumption of lixiviants.

While single explosive charges generally will be detonated in squence to produce the fracture zones, the charges also can be multi-component charges positioned in separate cavities and detonated substantially simultaneously as a group to produce each fracture zone, each detonation in the sequence of detonations in such a case being a group of detonations.

The following example illustrates specific embodiments of the process of the invention.

The formation to be fractured was a bedded series of shales and silt stones, dipping about 45°, located at a depth of 70 to 90 feet below the surface, and therefore subjected to a lithostatic pressure of about 70 to 90 p.s.i. The water table was at a depth of about 15 feet below the surface. A 3-inch-diameter hole was drilled into the formation to a depth of 100 feet. This hole was used as a core-sampling and permeability-testing hole, and also as an injection hole for purposes of flushing surrounding shot holes. A core test revealed a competent silty shale at the 70–90 foot depth. A well screen was installed in the hole at the 70–90 foot level, and piping to the well screen was grouted to the hole. Cement filled the hole below the well screen.

The pattern of shot holes used is shown in FIG. 3. Three shot holes (SH 1, SH 2, and SH 3) were located 16.25 feet from the injection hole I, their centers lying on 120° radii from the center of hole I and the lines joining them forming an equilateral triangle. The distance between these shot holes was 28 feet. Three shot holes (SH 4, SH 5, SH 6) were located 32.5 feet from hole I, their centers also lying on 120° radii from the center of hole I, and the lines joining them (also forming an equilateral triangle) being bisected by the centers of holes SH 1, SH 2, and SH 3. The distance between holes SH 4, SH 5, and SH 6 was 56 feet. It is seen that in this arrangement the lines joining adjacent (i.e., nearest neighbor) shot holes formed equilateral triangles. SH 1, SH 2, and SH 3 each had four shot holes adjacent thereto (SH 2, SH 3, SH 4, and SH 5 for SH 1; SH 1, SH 3, SH 5, and SH 6 for SH 2; and SH 1, SH 2, SH 4, and SH 6 for SH 3), and SH 4, SH 5, and SH 6 each had two shot holes adjacent thereto (SH 1 and SH 3 for SH 4; SH 1 and SH 2 for SH 5; and SH 2 and SH 3 for SH 6).

Shot hole SH 1 was drilled first. The hole was 5 inches in diameter and 91 feet deep and was loaded with 255 pounds of an aluminized water gel explosive having the following composition: 18.9% ammonium nitrate, 10.5% sodium nitrate, 29.6% methylamine nitrate, 30% aluminum, and 11% water. The explosive column was 21.7 feet high, and was covered by a layer of water which naturally flowed into and filled the remainder of the hole and stemmed the explosive charge. The water level in the injection hole was above the level of the top of the explosive charge in the shot hole, indicating that the rock surrounding the shot hole was properly flooded. Before the explosive charge was initiated, the permeability and sound velocity of the rock surrounding the injection hole were measured. The permeability was determined by slug tests, in which the permeability is inferred from the rate at which the head of liquid subsides toward the ambient level in a hole after the rapid introduction of a slug of liquid therein (see Ferris, J. G., et al., "Theory of Aquifer Tests", U.S. Geological Survey, Water-Supply Paper 1536-E, 1962). The sound velocity, measured at depths of 74.5 feet to 85 feet between the injection hole, shot hole SH 1, and a test hole collared 13 feet on the opposite side of the injection hole, was 3970 meters per second.

The explosive charge in shot hole SH 1 in the flooded formation was detonated, whereupon the water level in the injection hole dropped to below its pre-detonation level, as a result of the formation of a new fracture volume around shot hole SH 1, the chasing of water from the rock fractures by the gaseous detonation products, and the flow of water into the cavity created by the explosive charge. After partial recovery of the water level in the injection hole, the second shot hole (shot hole SH 2) was drilled to the same size as shot hole SH 1, and the rock surrounding shot hole SH 1 was then flushed with water by (a) blowing compressed air into the bottom of the open injection hole, (b) injecting water through a packer in the injection hole, and (c) three long air injections, and then (d) 18 short air injections through a packer in the injection hole. The total flushing time was about 4 hours. Silt-laden water was ejected from shot hole SH 1 (but not shot hole SH 2) during the flushing, indicating the preservation of the stop of shot hole SH 1, the circulation of the water from the bottom of the rock segment (bottom of the injection hole) laterally and upward through the fracture network to the top of the rock segment (bottom of shot hole SH 1), and the removal of fines from the fractures. The permeability was measured in the injection hole (as described above) before and after the flushing operations.

Shot hole SH 2 and subsequently drilled shot hole SH 3 were loaded and the charges therein detonated as described for shot hole SH 1.

The water level in the injection hole returned to its pre-detonation level, above the level of the top of the explosive charge in shot hole SH 1 before detonation, in about 18 hours. Thereafter, the charge in shot hole SH 2 was detonated, whereupon the water level in the injection hole again dropped to below its pre-detonation level. The rock surrounding shot hole SH 2 was flushed with water, and silt-laden water ejected from shot hole SH 2, by sealing off shot hole SH 1 and (a) injecting air through a packer in the injection hole, (b) blowing compressed air into the bottom of the open injection hole, and (c) injecting air through a packer in the injection hole, followed by water through the packer while blowing compressed air into the bottom of shot hole SH 3. The total flushing time was about 11 hours. The permeability was again measured before and after the flushing operations.

After the water level in the injection hole had returned to its pre-detonation level, the charge in shot hole SH 3 was detonated, and the surrounding rock flushed by (a) air injection through a packer in the injection hole, followed by sealing off shot hole SH 1 and blowing air down shot hole SH 2 and shot hole SH 3 to drive water to each shot hole in turn until water was exhausted from the broken rock; and (b) two air injection flushings, each followed by water injection. The total flushing time was about 7 hours. The permeability was again measured before and after the flushing operations.

The remaining shot holes, SH 4, SH 5, and SH 6, were drilled, loaded, and detonated in the same manner as holes SH 1, SH 2, and SH 3, with the detonations occurring after the return of the water level in the injection hole to its predetonation level. Between the shooting of shot holes SH 4 and SH 5, the rock surrounding hole SH 4 was flushed by three air injections in the injection hole, each followed by water injection; between the shooting of shot holes SH 5 and SH 6, the rock surrounding hole SH 5 was flushed by injecting air into the injection hole, and blowing air down hole SH 6 (unshot), separately and simultaneously; and after hole SH 6 was shot, the rock surrounding it was flushed by alternately injecting air into the injection hole and blowing air down the surviving section of hole SH 6.

The permeabilities measured by slug tests in the injection hole before the blast/flush process began and after each blast and flush operartion at each of the six holes are plotted in FIG. 4 as a function of the operation performed, the permeabilities being presented in millidarcys on a logarithmic scale as the ordinate. Nineteen points are shown, including those obtained after the four flushing procedures (*a*, *b*, *c*, and *d*) described above after the shooting of hole SH 1; three flushing procedures (*a*, *b*, and *c*) after the shooting of hole SH 2; and two flushing procedures (*a* and *b*) after the shooting of hole SH 3. Each point denotes the average permeability measured after a given operation.

The plot shows that the permeability of the rock was increased considerably (from 500 to over 2000 millidarcys) by the total six-cycle blast/flush process, and that variations in permeability occurring during the cyclical shooting and flushing tend to decrease as the rock is broken and swelled. The plotted experimental values also show that the rapid flow of water to the remnant of a shot hole achieved by means of air injection through another hole or by strong pumping from a shot hole (by blowing air into the bottom of an open shot hole, for example) increases the permeability after blasting; best results having been achieved when both air injection at the injection hole and strong pumping at a nearby shot hole was used. While blasting was found generally to decrease the permeability, permeability which had previously been reduced by the injection of water (alone or as a final flushing step after blasting) was increased by blasting.

The degree of dilation produced in the rock by the first three of the above-described detonations in flooded rock was estimated from calculations of porosity based on sound velocity measurements. The sound velocity around the injection hole after the three blasts was 3650 meters per second at a 12 foot radius from the hole, and 2530 meters per second through paths in the blasted rock running from the shot holes in to the injection hole (compared with 3970 meters per second in the same prism of rock before blasting). The total porosity in the rock (ψ) was calculated from the following empirical equation for the sound velocity (α, in m/sec) as a function of porosity, for flooded ocean sediments having various degrees of lithification:

$$\psi = -50.748 \ln \alpha + 432.23.$$

Total porosity before blasting: 11.7%

Total porosity after blasting: (12-ft. radius) 16.0%

Total porosity after blasting: (center to shot holes) 34.6%

These porosities imply that the fracture volumes caused by the blasting were 4.3% (12-ft. radius) and 22.9% (center to shot holes).

I claim:

1. In a process for leaching a fracture-containing solid mass of ore by introducing a lixiviant for the ore and a gas into the solid mass through a plurality of holes therein, the improvement which comprises driving liquid lixiviant laterally and upwardly through the fractures in the solid mass to a plurality of recovery holes by injecting said gas, and allowing it to exit from said recovery holes with said lixiviant, at a rate sufficient to produce a surging flow of lixiviant in said recovery holes, whereby fines present in the fractures in the solid mass are entrained by said lixiviant and are removed from the solid mass of ore therewith.

2. A process of claim 1 wherein said gas is injected into said mass at a pressure in excess of the lithostatic pressure.

3. A process for the in situ leaching of an ore body which has been worked by detonating explosive charges in separate cavities therein to produce in the ore body immediately adjacent to the site of each deto-

nation a fracture zone comprised of a most severely fractured core portion surrounded by a less severely fractured outer portion, comprising introducing a liquid lixiviant for the ore into the ore body through a plurality of injection holes in the less severely fractured portions and driving said lixiviant laterally and upwardly through the ore body to a plurality of recovery holes in the most severely fractured portions by injecting gas into the ore body and allowing said gas to exit from said recovery holes with said lixiviant at a rate sufficient to produce a surging flow of lixiviant in said recovery holes, whereby fines are entrained by said lixiviant and removed from the ore body therewith.

4. A process of claim 3 wherein said lixiviant is injected into said ore body at a pressure which is at least as high as the lithostatic pressure, and said gas is injected and said fines are removed intermittently.

5. A process of claim 3 wherein the ore body has been worked by the detonation of chemical explosive charges.

6. A process of claim 5 wherein the injection holes extend from the earth's surface to substantially the bottom of the ore body, and the recovery holes extend from the earth's surface to substantially the top of the ore body.

7. A process of claim 6 wherein the recovery holes are preserved upper portions of shot holes in which explosive charges have been detonated.

8. A process of claim 3 wherein said gas is an oxidizing gas.

* * * * *

United States Patent [19] [11] **3,822,916**

Jacoby [45] **July 9, 1974**

[54] IN-SITU EXTRACTION OF MINERAL VALUES FROM ORE DEPOSITS

[75] Inventor: **Charles H. Jacoby, Dalton, Pa.**

[73] Assignee: **Akzona Incorporated, Asheville, N.C.**

[22] Filed: **Nov. 16, 1972**

[21] Appl. No.: **307,144**

[52] **U.S. Cl.** 299/4, 166/299
[51] **Int. Cl.** E21b 43/28
[58] **Field of Search** 299/4; 166/299

[56] **References Cited**

UNITED STATES PATENTS

2,847,202	8/1958	Pollen	299/4
2,952,449	9/1960	Bays	299/4
3,565,173	2/1971	Anderson	166/299 X
3,718,366	2/1973	Hill	299/4

Primary Examiner—Ernest R. Purser
Attorney, Agent, or Firm—Bean & Bean

[57] **ABSTRACT**

There is disclosed a method for the in-situ extraction of metal/mineral values from metalliferous and/or nonmetallic ores occurring in substantially impermeable ore deposits of the laminar gneiss or schist or conglomerate types, or the like. The invention is particularly adaptable for example to the mining of copper and nickle and manganese carbonate or sulphide type ores. In accordance with this invention, the geologic formation containing the ore is first penetrated by two or more, suitably spaced-apart openings, such for example bore holes extending from the earth's surface to the bottom level of the deposit. There is then established a liquid-permeable "base" fractured zone interconnecting the lower ends of the bore holes within the ore body by means of a hydraulic-fracturing technique such as described in my U.S. Pat. No. 3,064,957; the base fracture being propped open by means of any suitable agent such as described for example in U.S. Pat. No. 2,645,291. An explosive material of liquid or slurry form is then flowed into the cavitated U-tube system comprising the bore holes and the "base" fractured zone, thereby replacing the hydraulic fracturing fluid by an explosive material of fluidized powder or liquid or slurry form. This operation is conducted under only substantially the earth's-surface-ambient atmospheric pressure conditions, thereby eliminating serious prior art hazards. The explosive is then detonated so as to fracture the overlying ore body according to the petro-fabrics of the ore deposit and/or the stress patterns of the mineral constituents thereof. The ore body is thereby rendered permeable, and a suitable solvent/reactive material (such as sulphuric acid in the case of the metal carbonates) for recovery of the desired value-containing salt solution is then caused to percolate through the permeabilized ore deposit. The invention also contemplates a specific geometry for the solvent/reactive flow pattern through the ore body, featuring a vertical migration and "floating" of the solvent/reactant on top of the more metal concentrated fraction of the product flow stream; thereby maintaining an optimum reactant-to-ore status progressively upwardly within the lateral confines of the ore body. If sulphide type ores are involved oxygen or air and water may be employed as an in-situ solvent/reactant.

8 Claims, 5 Drawing Figures

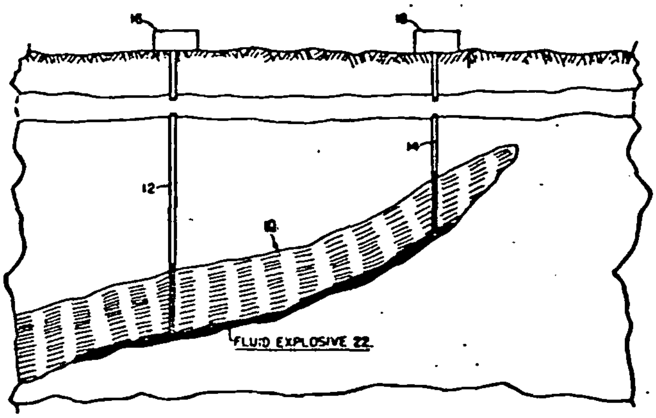


FIG. 1

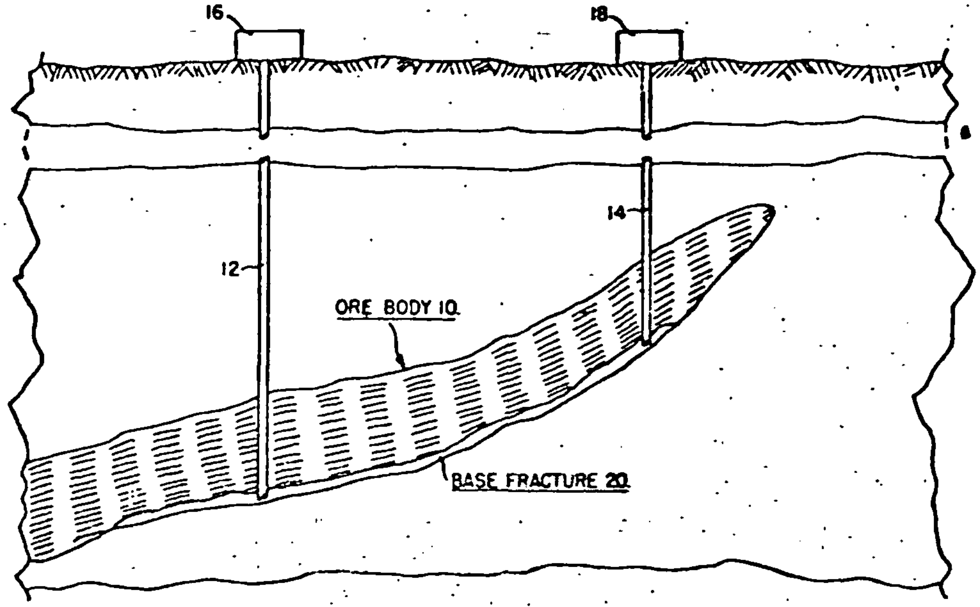

16
18
14
12
ORE BODY 10.
BASE FRACTURE 20.

FIG. 2

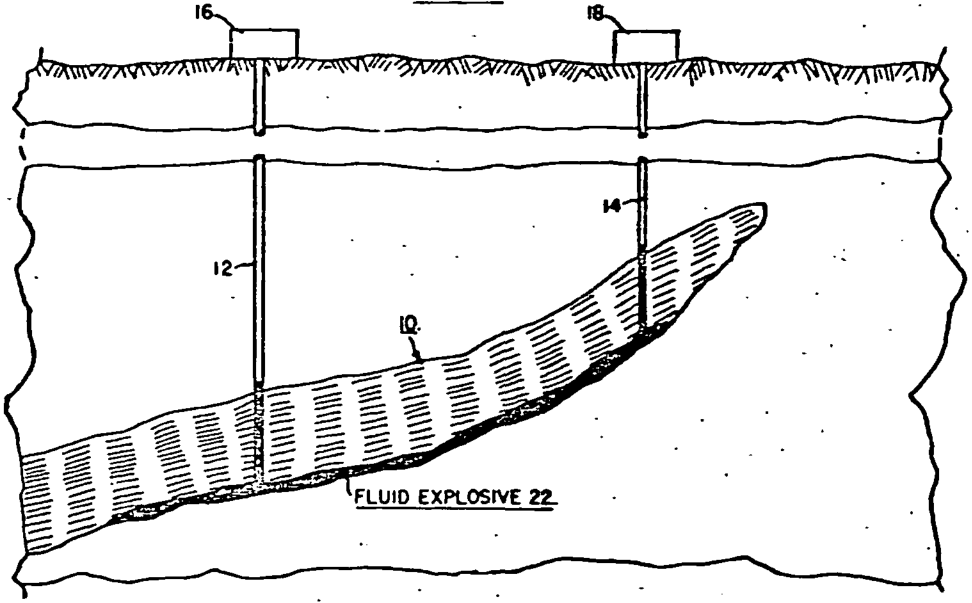

16
18
14
12
10.
FLUID EXPLOSIVE 22.

FIG. 3

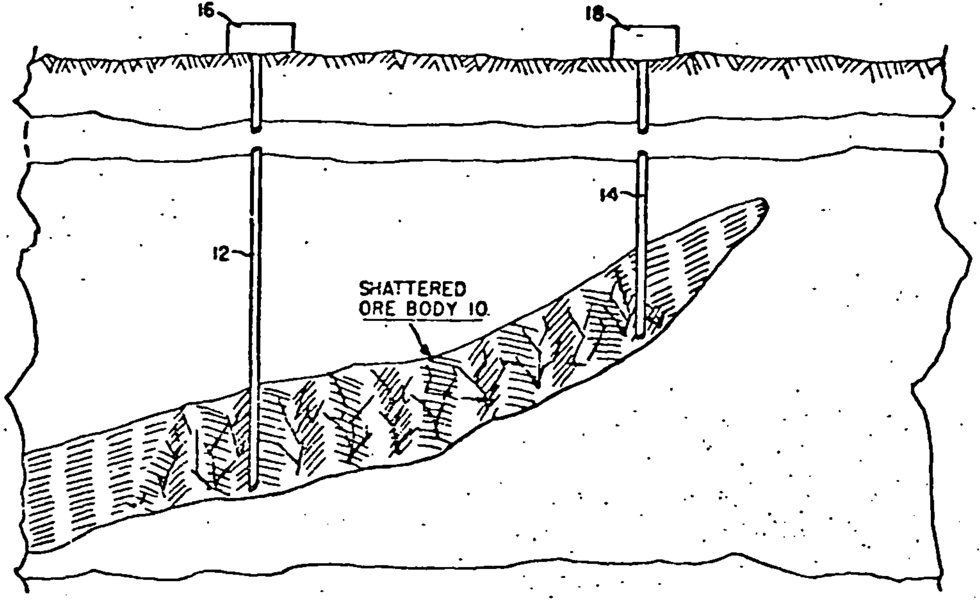


FIG. 4

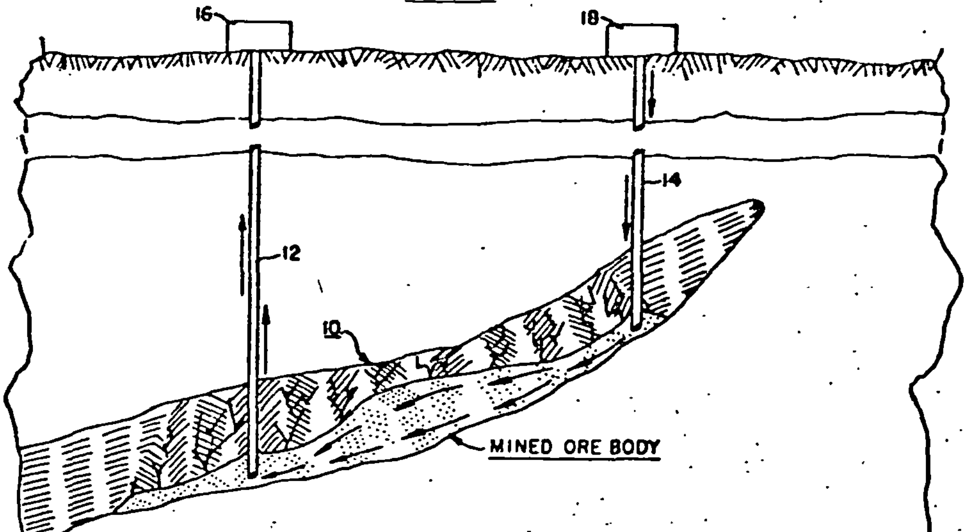
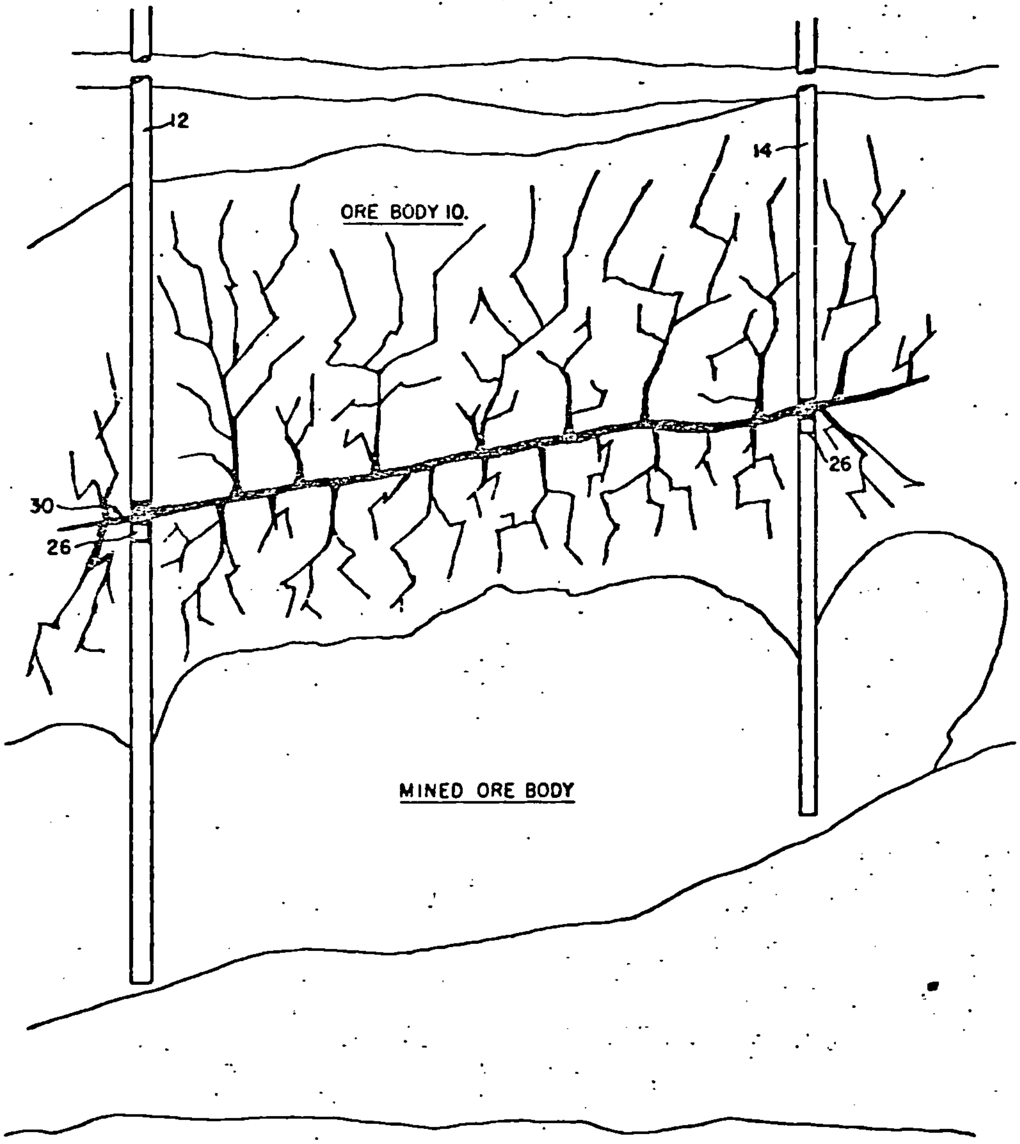


FIG. 5



IN-SITU EXTRACTION OF MINERAL VALUES FROM ORE DEPOSITS

BACKGROUND OF THE INVENTION

Economically successful methods for the in-situ mining (as by leaching) of ores have long been sought, as illustrated for example by way of U.S. Pat. No. 3,574,599; but such methods have proved to be economically and/or otherwise inadequate for various reasons. For example, previously proposed and employed methods for underground fracturing utilizing liquid explosives have necessitated supply and underground distribution of the liquid explosive under extremely high pressures (in order to overcome the hydrostatic pressures at depths below the earth's surface) thereby inherently rendering such operations highly hazardous, while at the same time being inefficient from the standpoint of optimum placement of the liquid explosives. Also, problems associated with solution-mining gallery maintenance have heretofore rendered such previously proposed methods economically impracticable; and it is an object of the present invention to provide an improved method and system whereby an ore can be reacted in-situ with a suitable solvent and/or reactant to produce a product solution of the sought-for mineral such as will flow continuously through a "production-well" to the earthsurface metal recovery plant, while leaving behind the residual insoluble gangue material. The invention also contemplates an improved solvent-/reactant solution flow pattern through the ore body which is operatively maintained in accordance with specific parameters as disclosed hereinafter, and as illustrated by way of example in the accompanying drawing, wherein:

THE DRAWING

FIG. 1 is a symbolical illustration of vertical geological section type, illustrating a typical ore-bearing schist, gneiss, dike, sheet, lode, lens, sill or the like which is of limited or varying vertical thickness; and showing diagrammatically how the initial ore-body penetration and base fracturing-propping operations are performed;

FIG. 2 is a view corresponding to FIG. 1, illustrating how the base fractured zone of the ore-body is subsequently loaded with a liquid (or slurry) type explosive;

FIG. 3 illustrates how the ore deposit is then beneficially three-dimensionally fractured so as to accommodate a dissolution/reaction flowage system in accordance with the invention;

FIG. 4 illustrates how a leach/ion-reaction mining and value-recovering operation may then be carried on in accordance with the present invention; and

FIG. 5 is a view illustrating how the system of the invention is conducted when operating upon an ore-body of substantially greater vertical extent than as illustrated at FIGS. 1–4.

SPECIFICATION

It is well known that many copper ore deposits of value occur at such depths within the earth-surface's geology as to render their exploitation by typical mechanical mining methods to be not economically feasible; and it is a primary object of the present invention to provide a practicable system whereby such deposits may be economically recovered. As illustrated by the drawing herein, the method of the present invention is applied to an inclined dike or other type copper or manganese or the like ore deposit such as is indicated generally at **10.** Such deposits occur at various depths and distances underground, and are only economically mined by conventional mechanical mining methods whenever they occur relatively close to the earth's surface and when they contain relatively high-grade recoverable mineral values.

According to the present invention as illustrated by way of example by the drawing herewith, the mineral deposit **10** may be intersected by a pair of bore holes **12, 14,** which extend substantially to the lower levels of the ore-bearing body and communicate at their upper ends with appropriate surface plant processing facilities as indicated at **16, 18.** The bore holes are preferably "cased" (as is well known in the well drilling art) by means of casings fabricated of or lined with material which is non-reactive relative to the fluids to be handled. Thus, the bore holes **12, 14,** provide means for ingress to the ore deposit; and preferably the bore hole **12** which intersects the deposit at a lower level than the bore hole **14** is used as the site to "base-fracture" the ore body as indicated at **20** (FIG. **1**) thereby utilizing the "uplift" effect of the hydraulic operation to optimum advantage. The fracturing operation may be performed in accordance with any suitable hydraulic fracturing technique; and as previously mentioned it may be performed as disclosed for example in my U.S. Pat. No. 3,064,957. As shown therein, the result of such an operation provides a solution-permeable region designated **20** extending radially from the bottom of the bore hole **12** toward the bottom of the bore hole **14.** In the case of a copper carbonate ore the fracturing operation is preferably performed by utilizing a dilute sulfuric acid solution containing in suspension a fracture "propping" agent such as sand, glass beads, or the like. Such propping devices are well known in the art; and a passageway between the lower ends of the bore holes for introduction of liquid or slurry type explosive is thereby established.

In event the "base" fracturing process referred to hereinabove does not operate to coalesce the two or more bore holes the situation may be remedied by repeated attempts to hydraulically re-fracture the zone between the bore holes, or by extending the propped cavity by means of a liquid or slurry type explosive. This may be done simply by pumping the explosive into the cavity under relatively low pressure such as is sufficient to displace the hydraulic "frac" liquid ahead of the explosive. By way of example, the fluid explosive may be fed into the system by a low pressure pump through the bore hole **12,** supplemented if preferred by use of a suction pump operatively connected to the upper end of bore hole **14.**

The explosive is then detonated, thereby propagating a "base" fracture toward the target well; but in event this operation still does not provide an open communication between the bottom ends of the wells the operations may be repeated as often as may be necessary (from the same well, or alternatively from either well) in order to complete an inter-connection between the lower ends of the wells. The system including the propped base-fractured zone may now be purged of the residual "frac" liquids and replaced by an explosive material of liquid or slurry form as shown at **22** (FIG. **2**) simply by flowing it into the system under only substantially atmospheric pressure condition. This explosive is subsequently detonated so as to vertically fracture the valuable mineral containing ore body thereabove, as illustrated at FIG. 3.

The mining process is then conducted by passing a solvent or reactant liquid and/or gas through the system so as to extract the desired mineral values from the deposit. For example, as shown at FIG. 3 the ore-reactant fluid is preferably injected into the ore body through the bore hole 14 which terminates at a higher elevation than the other bore hole (s) of the system. The fluid initially percolates through the lower level of the ore body to the bottom of the lowermost bore hole 12 for withdrawal to the surface; the operation being readily controlled by means of a suction pump or the like located in conjunction with the bore hole 12, and/or a pressure pump located in conjunction with the bore hole 14 operating to inject the solvent into the system.

The chemical content of the ore-reactant solution will of course be prescribed and/or adjusted according to the nature of the mineral content of the ore body from which it is required to extract the sought-for mineral values. As the ore-reactant operation progresses the reacting solution forms progressively enlarging voids and/or passageways through the ore body; the leaching action operating to "honeycomb" the ore body of the mineral values therein.

Note that the invention contemplates circulation of the leach/reactant liquid (or gas) through the ore body commencing from the bottom of the bore hole which stands at the highest elevation of the group of bore holes involved; the product liquid (or gas) being withdrawn from the bottom of the deepest bore hole involved. By virtue of this system advantage is taken of the phenomenon that the mineral-laden liquid (or gas) will be of higher specific gravity than the fresh solvent-/reactant and will automatically migrate down towards the bottom of the bore hole 12, while the fresh solvent-/reactant fractions will float on top so that the leach-/reaction process automatically progresses vertically from along the lower level of the mineral deposit upwardly therefrom and to the top level thereof. Otherwise stated the mineral-value-laden liquid (or gas) will automatically keep migrating toward the entrance to the product withdrawal bore hole, while the relatively fresh leach/reactant input is automatically disposed to percolate through progressively higher (still mineral-value laden) levels of the ore body.

Should the above described system operate to mine only the lower portion of the ore body (such as in the case of an ore deposit of substantial vertical extent as illustrated for example at FIG. 5 herein) the system may be repeated as many times as necessary to fracture and mine successively higher levels of the ore body. To this end the same bore holes may be employed after first mining the lower portion of the ore body by temporarily blocking off the lower end of the bore hole/-casing system as shown at 26; perforating the system at a substantial elevation above the blocking 26; hydraulically fracturing a vertically intermediate portion of the ore body as indicated at 30; and then repeating the above described fluid explosive replacement and detonating operations so as to shatter and permeablize the vertically intermediate portion of the ore body.

Note that as illustrated at FIG. 5, herein, detonation of the fluid explosive occupying the fractured zone 30 will operate to fracture the ore body both downwardly and upwardly therefrom; the downward fracturing effects being facilitated by reason of the previous honeycombing of the underlying portion of the ore body as explained hereinabove. Thus, it will be understood that in the case of an ore body of extreme vertical extent the above described progressive level fracturing and mining operations may be repeated until mining of the entire ore body is completed.

I claim:

1. The method for extracting metalliferous values from a laminar type impermeable metalliferous ore body comprising:

A. penetrating the ore body to provide two or more spaced apart clear openings extending from the earth's surface and terminating in portions of the ore body at different elevations;

B. fluid-pressure fracturing the ore body from the bottom end of the opening terminating at the lowermost elevation upwardly to the bottom end of another of said openings in the laiminar direction of the ore body to provide a permeable zone underlying a higher portion of the ore body of substantial vertical thickness, thereby establishing a substantially closed fluid circulation system including said openings and said permeable zone;

C. flowing a fluid type explosive under pressure less than the fracturing pressure into said circulation system so as to fill said permeable zone;

D. detonating said explosive so as to permeablize the ore body thereabove; and

E. then flowing a sought-for metalliferous solvent-/reactant fluid through another of said openings downwardly and through said circulation system so as to percolate through the permeabilized portions of the ore body to produce a product fluid containing the sought-for metalliferous values from the opening terminating at the lowermost elevation.

2. A method as set forth in claim 1 wherein said openings into said ore body are provided by two or more horizontally spaced apart bore holes terminating at their bottom ends at different elevations.

3. A method as set forth in claim 2 wherein said base fracturing process is repeated as many times as may be necessary to coalesce the lower ends of at least two bore holes.

4. A method as set forth in claim 1 wherein said base fracturing process is performed by means of a hydraulic fracturing technique.

5. A method as set forth in claim 4 wherein said base fracturing process is conducted initially by means of a hydraulic fracturing technique, and is supplemented by an explosive fracturing technique.

6. A method as set forth in claim 1 wherein said fluid type explosive is flowed into said circulation system by means of a low pressure pump operating to deliver explosive through the opening hole terminating at the lowest level.

7. A method as set forth in claim 6 wherein a suction pump is operably connected to another one of the openings to supplement flowage of fluid explosive into said system.

8. The method for extracting metalliferous values from a laminar type impermeable metalliferous ore body comprising:

A. penetrating the ore body to provide two or more speced apart clear openings extending from the earth's surface and terminating in lower portions of the ore body at different elevations;

B. fluid-pressure fracturing the ore body from the bottom end of the opening terminating at the lowermost elevation upwardly to the bottom end of another of said openings in the laminar direction of the ore body to provide a permeable zone underlying a higher portion of the ore body of substantial vertical thickness, thereby establishing a substantially closed fluid circulation system including said openings and said permeable zone;

C. flowing a fluid type explosive under pressure less than the fracturing pressure into said circulation system so as to fill said permeable zone;

D. detonating said explosive so as to permeabilize the ore body thereabove; and

E. then flowing a sought-for metalliferous solvent/reactant fluid through one of said openings, upwardly and through said circulation system so as to percolate through the permeabilized portions of the ore body to produce a product fluid containing the sought-for metalliferous values from an opening terminating at a higher elevation.

* * * * *

SOCIETY OF MINING ENGINEERS of AIME
540 ARAPEEN DRIVE — SALT LAKE CITY, UTAH 84108

PREPRINT NUMBER
77-AS-68

IN SITU LEACHING OF COPPER - AN ECONOMIC SIMULATION APPROACH

A. Aly Selim
Industrial Engineer
U.S. Bureau of Mines
Twin Cities Mining Research Center
Minneapolis, Minnesota

Donald H. Yardley
Professor
University of Minnesota
Minneapolis, Minnesota

For presentation at the 1977 AIME Annual Meeting
Atlanta, Georgia - March 6-10, 1977

Drive. Salt Lake City. Utah 84108.

PREPRINT AVAILABILITY LIST IS PUBLISHED PERIODICALLY IN MINING ENGINEERING.

ABSTRACT

An economic simulation model was developed to predict the amount of copper recovered from copper oxide deposits and the cost of producing it as a function of the deposit parameters. The economic simulation model includes models of fragmentation design and cost, copper extraction, solution distribution and recovery, effluent solution cutoff grade, and financial analysis. Cost and price indices were also developed to evaluate the economic feasibility of in situ leaching operations as a function of their starting date.

Application of the models to various properties having the same area and different ore thickness shows that the effluent solution cutoff grade does not vary with the ore thickness nor with its grade. The percent of copper recovery is, however, shown to increase with the increase in the ore thickness and is independent of the ore grade. It is also shown that while the profit per unit of extracted copper increases with the increase in ore grade or ore thickness, the rate of return on total investment does not show any correlation.

INTRODUCTION

The percent of copper extracted by leaching is one of the most critical factors in determining the economics of an in situ leaching operation. While the percent of copper extracted by leaching depends on many geological and physical factors, it also depends on many economic factors which affect the profitability of the operation. One of the most important economic factors is the cutoff grade of the effluent solution, and as such the total amount of copper that can be profitably extracted. The cutoff grade is defined as the amount of copper in grams per liter of effluent solution below which the leaching operation ceases to be profitable.

To calculate the cutoff grade, one has to determine the cost per unit of extracted copper and the production rate as a function of time.

An economic simulation model was developed to predict the amount of copper extracted from copper oxide deposits and the cost of producing it as a function of the deposit parameters and the time. The economic simulation model includes models to calculate the fragmentation system cost and design parameters, predict the percent of copper extracted as a function of the time of leaching and deposit parameters, calculate the cost of the solution and recovery systems as a function of the deposit parameters, calculate the optimum effluent cutoff grade as a function of the economic parameters of the operation, and analyze the cash flow from the operation and calculate the rate of return. These models were developed as part of a research project to investigate the economic and physical boundaries of in situ leaching operations.

ECONOMIC SIMULATION MODEL

The economic system simulation model is composed of a number of models which calculate the amount of ore reserve of the property being investigated, calculate the fragmentation cost of an in situ leaching operation, simulate the copper recovery process, calculate the optimum cutoff grade and amount of copper produced, and calculate the rate of return on investment based on the analysis of the cash flow resulting from the in situ leaching operation.

Ore Reserve Model

The ore reserve model was developed at the Pennsylvania State University (1)[1] and was used as a part of the economic simulation model to determine the geological parameters of the property to be investigated. These parameters are the depth to the bottom of the ore zone, the thickness of the ore, the water table elevation, the grade of the ore, and the minable area within the property.

Fragmentation Design and Cost Model

This model was developed (2) to calculate the blast design parameters and fragmentation cost for in situ leaching operations as a function of the deposit parameters. The model is based on blasting an ore zone or an ore zone plus overburden by either a coyote blast or a vertical blasthole system. The model calculates the blast design parameters and the fragmentation cost based either on a user input powder factor value or on a powder factor calculated by the model as a function of the overburden height and the ore zone thickness. Examples of blast design parameters calculated by the model are the number and spacing of blastholes for a vertical blasthole system, and the number and spacing of crosscuts for a coyote system. The model also calculates the total amount of explosives, the detonating cord length, and the number of primers. The cost of blasthole drilling can be calculated by the model based either on purchasing and operating a drill or on a user input value of the contracted cost of drilling. The fragmentation cost is based on a user input unit value for the material used in fragmenting the ore.

---

[1] Underlined numbers in parentheses refer to items in the list of references at the end of this report.

Copper Recovery Model

A copper oxide recovery simulation model was developed (3) to determine the amount of copper produced as a function of the time of leaching, acid concentration, ore grade, effective diffusivity, and particle size distribution. The model is based on a shrinking unreacted core theory (4) where the reaction is based on diffusion of the acid through the unreacted portion of the particle. The simulation approach is based on a unit volume and a unit column concept suggested by Roman (5) where a unit volume is defined as an area 1 foot square and 1 foot thick and a unit column is composed of a number of unit volumes. The leaching of the unit column is based on vertical trickle of the leach solution through the column. Fresh leach solution enters the top unit volume, and the solution leaving the unit volume at the bottom of the column goes to the extraction plant. The leaching reaction for this model is described by

$$T = \frac{1000 b \rho \nu G R^2}{aDC} \left(1 + 3\left(\frac{r_c}{R}\right)^2 + 2\left(\frac{r_c}{R}\right)^3\right) \qquad (1)$$

where T = cumulative reaction time, minutes,

b = grams of acid consumed per 1 gram of copper recovered,

ρ = density of ore, grams per cubic centimeter,

ν = correction factor for spherical particles

a = surface area correction factor for spherical particles,

D = effective diffusivity, square centimeters per minute,

G = percent copper ore grade.

R = radius of ore particle size, centimeters,

r_c = radius of unleached core, centimeters,

and C = acid concentration, grams per liter of H_2SO_4.

Equation 1 describes the leaching of a single particle. The percent recovery from a single particle is described by

$$Rec = 100(1 - (\frac{r_c}{R})^3) \qquad (2)$$

In addition to the percent recovery, the model also calculates the amount of copper recovered from each unit volume having a particle size distribution, the grade of the effluent solution, the amount of acid in this solution, and the remaining ore grade in the deposit when the leaching is terminated. The input to the computer model includes the variables shown in equation 1 and the particle size distribution of the fragmented ore. The reaction time T is based on the time it takes for the solution to flow through one unit volume. The value of r_c is determined by the solution of the polynomial in equation 1 for each unit volume and every particle size within the unit volume. Based on the value of r_c, the percent recovery for each particle size and every unit volume is determined based on equation 2. Summation of this recovery for all the particles and all the unit volumes defines the recovery from the unit column.

Cutoff Grade Model

In conventional mining operations, a cutoff grade is defined as the grade of the ore above which the mining operation is profitable. For in situ leaching operations, the definition is similar when the ore deposit is being evaluated for its economic feasibility for leaching. However, the cutoff grade at which leaching should be stopped is determined after the leaching operation has been started. This cutoff grade is defined as the grade of the effluent solution in grams of copper per liter of effluent solution at which the cost of producing a unit of copper is equal to its selling price. Any further production at lower solution grades results in an economic loss from the added production. The cutoff grade could then be alternatively defined as the grade of the effluent solution at which the profit from the operation is at its maximum. Assuming that profit maximization is the objective from the operation, the cumulative amount of copper produced up to the cutoff grade is then defined as the optimum level of production. In other words, maximization of profit is defined by

$$\pi = Pq_t - C_t \qquad (3)$$

where P = the price per pound of copper,

q_t = cumulative amount of copper produced until time t,

and C_t = total cost of production at time t.

Therefore $$\frac{d\pi}{dq} = P - \frac{dC_t}{dq} = 0 \qquad (4)$$

The total cost of an in situ operation is composed of predevelopment, development, capital, and operating costs. The latter is a variable cost (VC) which is a function of the level of production, and the other elements of costs are fixed costs (FC) which are independent of the amount of copper produced.

Therefore $$C_t = FC + VC \qquad (5)$$
$$= CC + DEV + Aq_t + Wt + Rq_t$$

where CC = capital cost (including property purchase price),

DEV = predevelopment and development cost,

A = acid, iron, and smelting cost per pound of cement copper,

W = operating cost per day such as power cost, maintenance, and wages,

t = time of production, days,

and R = royalty per pound of copper.

Iron and smelting costs per pound of copper are included in the above cost elements, assuming a cementation operation where the iron is used to cement the dissolved copper and the cement copper is then smelted. When an electrowinning operation is being analyzed, the element of cost "A" should include the acid cost, precipitation solution cost, and cathode plate cost per pound of copper produced. Analyses of solvent extraction and electrowinning operations are not included in this manuscript.

The marginal cost is defined by

$$MC = \frac{dc_t}{dq_t} = \frac{C_t - C_{t-1}}{\Delta q_t}$$

$$= A + \frac{W(t_2 - t_1)}{\Delta q_t} + R \qquad (6)$$

Equation 4 implies that profit maximization is attained at the level of production at which the marginal cost (MC)--that is, the cost of one additional unit of production--is equal to the price. Equation 6 implies that the level of production at which the profit is maximized is only a function of the variable costs, and hence of the operating cost only.

A second profit maximization condition requires that

$$\frac{d^2\pi}{dq_t^2} = -\frac{d^2C_t}{dq_t^2} < 0 \qquad (7)$$

$$\text{or} \quad \frac{d^2C_t}{dq_t^2} > 0 \qquad (8)$$

This condition implies that the marginal cost must be increasing at the profit maximization condition.

Figure 1 shows a representation of the above analysis where $\hat{q}_t$ is the optimum level of production. The average cost per pound of copper for this level of production is denoted by point C, which is the intersection of the vertical ab with the average cost curve AC. The optimum profit defined by this analysis as the difference between the total value of the copper produced and the total cost is

$$\hat{\pi} = \overline{OPab} - \overline{OdcP} = \overline{dPac} \qquad (9)$$

From the copper recovery model the amount of copper produced at time t is determined by simulation, while the optimum is defined by the preceding economic analysis. The optimum cutoff grade of the effluent solution is then defined by

$$\hat{G} = \frac{\hat{q}_t - q_{t-1}}{\nu L} \qquad (10)$$

where ν = solution flow rate in liter per minute per square foot,

and L = size of area being leached in square foot.

The above analysis pertains to a static condition where it is assumed that all costs and prices are fixed. For a static condition, it is then assumed that, if a deposit is found to be economically feasible or unfeasible for in situ leaching, it will continue in the same economic condition as a function of time.

For spatial analysis, all costs and prices are assumed to change as a function of time. Costs and price indices equations were developed to predict the values of these indices in future years as a function of their previous values and the time in years. Each element of cost is then projected in the model for any future year as a function of its current value and the current and predicted value of the index as

$$\text{Future cost} = (\text{current cost}) \cdot \frac{\text{Predicted index}}{\text{current index}} \qquad (11)$$

For spatial analysis, the problem is then to maximize

$$\pi = \sum_t Q_t P_t - \sum_t \sigma_t \qquad (12)$$

where Q_t = the amount of copper produced in period t only,

P_t = price per pound of copper,

and σ_t = total cost in period t only.

Proceeding in a similar fashion as in the static approach, the amount of copper produced, the cutoff grade, and the profit are optimized for the dynamic case.

A distinction of the spatial analysis from the static approach is that, if a deposit is economically unfeasible currently, it could become economically feasible or unfeasible at a future date depending on the rate of price increase of copper compared with the rate of total cost increase. Another distinction is that, for equal areas of leaching, the cutoff grade and the amount of copper produced are identical for all the leaching areas in the static approach and could be otherwise for the dynamic case if the areas are leached at different times. However, to attain the optimum condition by either approach, three conditions have to be satisfied. These three conditions are that the marginal cost should be

1. Equal to the price,
2. Increasing,
3. Greater than the average cost.

To satisfy these three conditions simultaneously, a computer model was developed to calculate the elements of costs for static and dynamic conditions, simulate the amount of copper extracted, and calculate the optimum cutoff grade and the time at which it occurs.

COMPUTER MODEL

The computer model is divided into a number of individual programs that are designed to run separately but could be combined by a master program. These programs are

1. FRAGMENTATION - This program calculates the blast design parameters and fragmentation cost for either a coyote system or a vertical blasthole system.
2. LEACH - This program simulates the leaching of one unit copper oxide ore column and calculates the percent of copper recovered, the grade of the effluent solution, and the grade of the ore column as a function of time, acid concentration, and geological parameters.
3. SOLDIS - This program calculates the cost of the solution distribution and recovery systems, the capital cost, and development cost for each fragmentation system as a function of the geological parameter.
4. CUTOFF - This program calculates the optimum level of cumulative production for each leaching cycle, the optimum cutoff grade of the effluent solution, the time at which this grade occurs, and the average cost per unit of copper produced.

   The interrelationships among these programs are shown in figure 2.

Programs Input

The input data for the fragmentation program consist of the ore reserve parameters of the mine property, the drilling parameters including the size of the borehole and the subdrilling depth, the explosives parameters including powder factor, and the specific gravity of the explosives. If the coyote option is required, the input data for this program will include the mining parameters, which include the cross sectional area of the crosscut, the maximum spacing between the crosscuts, the span between supports, and the length of existing drifts.

The input data for the copper recovery program consist of the acid concentration, solution flow rate, density and initial grade of ore, effective diffusivity of H^+ in leached ore, particle size distribution, and the grade of the effluent solution, and/or the number of days at which leaching simulation is terminated. A user option is to include in the input data a prespecified acid concentration in the effluent solution at which the concentration of the acid in the influent solution should change.

The input data for the solution distribution and recovery program consist of the geological parameters, the design elements for the solution distribution and recovery systems, the fragmentation cost resulting from the fragmentation program, and the various elements of costs.

The input data for the cutoff grade program consist of the output from the solution distribution and recovery systems, the effluent grade time information resulting from the copper recovery program, the elements of the operating cost, and the geological parameters.

MODEL APPLICATION

The fragmentation model was applied to the ore reserve data of an assumed mine property to be evaluated for leaching. The outline of the area of the mine is 460 meters long by 340 meters wide, and the minable area is approximately 450 meters long by 330 meters wide. The average ore thickness is 19 meters with assumed no overburden. The average values for the ore reserve parameters for this property are shown in table 1. To determine the effect of the ore zone thickness on the cutoff grade, the percent of copper recovery, and the economics of in situ leaching operations, two other ore thicknesses were assumed. These thicknesses are 25.6 and 31.7 meters, having 10,779,014 and 13,345,446 tonnes of ore respectively.

TABLE 1. - Interpolated grid value statistics

	Arithmetic mean	Standard deviation
Grade, percent	0.45	0.13
Ore thickness, meters	19.0	5.8
Area of outline, square meters	156,400	
Total ore volume, cubic meters	2,952,226	
Total tonnage, tonnes (metric tons)	7,955,939	
Area of minable reserves, square meters	148,075	
Tonnage of ore, tonnes	7,955,939	

Table 2 shows the elements of cost and total cost of fragmentation for the three assumed properties. The elements of cost in this table and throughout this manuscript are based on 1975 dollars except for the operating costs used in the cutoff program, which are based on the particular year these costs occur. As shown by table 2, the cost per tonne of ore increases with the increase in the average ore thickness.

TABLE 2. - Drill and blast cost report

Average ore thickness, meters	19.0	25.7	31.8
Total drilling cost ($6.5/meter)	556,107	825,981	1,141,380
Cost of explosives ($0.30/kilogram)	668,478	1,009,068	1,408,548
Cost of primers and delays ($0.91 each)	5,901	8,907	12,435
Detonation cord cost ($0.13/meter)	18,555	24,933	32,286
Loading supervision cost ($300/day)	7,161	10,812	15,090
Loading labor cost ($50/shift)	3,579	5,406	7,545
Stemming cost ($6.7/cubic meter)	24,642	37,197	51,924
Total cost of fragmentation	1,284,423	1,922,304	2,669,208
Cost per tonne of ore	0.16	0.18	0.20

This increase was found to be particularly due to the increase in the drilling and explosives cost associated with the greater number of holes in the cases of larger thickness. The copper recovery simulation model was applied to the three assumed properties. For all three properties it was assumed (3) that--

ore grade, G = 0.45 percent

ore density, ρ = 2.27 g/cm^3

effective diffusivity, D = 4.12×10^{-6} cm^2/min

H_2SO_4 acid consumption per gram of leached copper = 5 grams

surface factor to volume of factor ratio, $\frac{a}{V_f}$ = 2.65

time to flow through a unit volume, t = 90 minutes

The initial acid concentration was chosen at 10 grams per liter, which would change to 7 grams per liter if the acid concentration in the effluent solution exceeded 2 grams per liter; that is, around pH 1.4. The new level of influent acid concentration is further changed to 4.9 grams per liter (pH 1.0), and then to 0.49 gram per liter (pH 2.0) if the acid concentration in the effluent solution exceeds 2 and 3 grams per liter, respectively. The concentration of

0.49 gram per liter is assumed to be an acidified water rinse to extract all the dissolved copper in the leach column. The leaching simulation was set to terminate after 800 days or when the grade of copper in the effluent solution dropped below 0.1 gram per liter. A particle size distribution felt to duplicate that attained by the industry (6) and composed of six sizes having an average diameter of 21.8 centimeters was assumed.

Figures 3, 4, and 5 show the leaching simulation results for the three unit columns having the same height as the average ore thicknesses of the three properties. Figure 3 shows the amount of copper produced per day for each column height as a function of the leaching time in days. As seen by this figure, the length of time each plant is operating at a constant production level is proportional to the height of the column and as such is proportional to the amount of copper in the column. Figure 4 shows the grade of the effluent solution as a function of the time in days for the three unit columns. As seen by this figure the grade of the effluent solution--and as such, the period of higher extraction efficiency--is again proportional to the height of the ore column. The effluent solution grade is constant for the length of time ab. At point b the grade of the effluent solution starts to decrease, and the concentration of acid in the effluent starts to increase until it exceeds 2 grams per liter after 214 days at which time the influent solution acid concentration is changed to 7 grams per liter. At point c, which is around 220 days, the new extraction rate is continuously declining until point d is reached, where the acid concentration in the effluent solution is again greater than 2 grams per liter and the influent solution acid concentration is changed to 4.9 grams per liter. The effect of this concentration becomes apparent at point f at which time the new extraction rate is shown by fg. At 410 days the new effluent solution acid concentration exceeds 3 grams per liter, and the acid concentration in the influent solution is changed to 0.49 gram per liter. The effect of this new concentration is apparant at point g, where the extraction rate drops rapidly, and the simulation is terminated when an effluent solution grade of 0.1 gram per liter is reached. The variations in the curves for 25.7 and 31.8 meters are similarly explained. One can also conclude that if the strategy of acid change had not been applied, the rate of extraction at c would have continued declining at a slower rate than that showed. Figure 5 shows the final grade of each ore column after the termination of leaching as a function of the depth from the top of the unit column.

To determine the effect of the ore grade on copper recovery by leaching, the three properties were assumed to have 0.45 percent, 0.6 percent, and 0.75 percent ore grades. The simulation results for these cases are similar to those presented in figures 3 through 5, with longer duration of the constant production level and effluent solution grade and with higher final column grade than those shown in figure 5.

To calculate the optimum cutoff grade, the optimum amount of copper produced, and the maximum profit attained from each leaching operation, the amount of copper produced shown by figure 3 and the elements of costs were used to simulate the economic conditions for each leaching operation based on the cutoff grade model shown above. The elements of costs for these leaching operations are shown in table 3 as a function of the ore thickness. The predevelopment cost shown in table 3 includes the elements of exploration costs and is assumed constant for the different ore thicknesses investigated. Similarly, the operating cost per year is assumed to be independent of the thickness of the ore, but is a function of the production rate, which is identical for the three ore thicknesses. The acid and iron costs are based on 5 and 2.5 kilograms of acid and iron respectively per kilogram of copper produced. The capital cost is shown to increase slightly with the increase in the ore thickness, which is due to the larger recovery pumps required in those cases, while the development costs, which include the fragmentation and mine development costs, are shown to increase significantly with the increase in the ore thickness. These cost elements shown in table 3 are independent of the grade and are shown in detail elsewhere (3).

TABLE 3. - In situ leaching costs

Ore thickness, meters	19.0	25.7	31.8
Predevelopment cost	$314,000	$314,000	$314,000
Development cost	1,396,580	2,069,227	2,832,925
Capital cost	2,112,699	2,115,853	2,118,672
Operating cost per year (365 days per year)	815,763	815,763	815,763
Acid and iron cost (per kilogram of copper recovered) at $31.70 per tonne H_2SO_4 and $87.10 per tonne iron	0.457	0.457	0.457

In applying the model to the properties being investigated, each mine was arbitrarily divided into four equal leaching areas. The capital costs shown in table 3 were derived for each ore thickness based on a plant capacity sufficiently adequate for the amount of production from one leaching area. In addition, the operating costs shown in table 3 are based on 1975 dollars and are inflated by the model for the particular year the leaching operation is taking place.

Because of this particular feature in the model and the assumed dynamic nature of copper prices, the resulting optimum cutoff grade and the length of leaching time for each leaching area could be significantly different. Figure 6 shows the cutoff grade for each of the leaching areas of the mine property with 19-meter ore thickness and 0.6-percent grade. As shown by the figure, the cutoff grade shown in brackets and the leaching time are different for each leaching area. The average cutoff grade for the whole mine for this case is 0.37 gram of copper per liter of effluent solution at an average leaching time of 547 days per leach cycle and an overall average cost of $1.21 per kilogram (55¢/lb) of extracted copper.

Tables 4 and 5 show the results of the simulation for the preceding economic conditions for the three properties at 0.45-, 0.6-, and 0.75-percent ore grade. Table 4 shows the average cutoff grade in grams of copper per liter of effluent solution for the three properties being investigated as a function of the ore grade. As shown by this table, there is no significant variation in the average cutoff grade among these cases. Analysis of the cutoff grades for each leaching cycle for the three properties at the three ore-grade levels showed the same result. This is due to the fact that all of these cases have the same operating cost, which as indicated by equation 6 is the primary variable defining the marginal cost and the optimality condition. Table 4 also shows the optimum amount of copper produced and the percent of copper recovered. As shown by the table, while the amount of copper recovered differs significantly among the different properties, the percent of copper recovered is insensitive to the grade of ore within the range of grades investigated, but is very sensitive to changes in the ore thickness.

TABLE 4. - Optimality conditions

Item	Ore thickness, meters	Ore grade, percent		
		0.75	0.6	0.45
Optimum average cut-off grade, percent	19.0	0.36	0.37	0.37
	25.7	0.35	0.36	0.36
	31.8	0.35	0.35	0.37
Optimum copper recovery, tonnes	19.0	27,408	21,814	16,321
	25.7	40,161	32.069	23.999
	31.8	52,445	41.970	31,210
Optimum copper recovery, percent	19.0	45	45	45
	25.7	49	49	49
	31.8	52	52	51

TABLE 5. - Financial analysis

Item	Ore thickness, meters	Ore grade, percent		
		0.75	0.6	0.45
Average cost, dollars per tonne	19.0	1,210	1,210	1,250
	25.7	1,210	1,210	1,230
	31.8	1,230	1,210	1,210
Rate of return, percent	19.0	40.6	36.4	29.3
	25.7	38.6	36.8	32.9
	31.8	35.2	34.1	31.2

The amount and percent of copper recovered shown in table 4 could be considered excessive when compared to recoveries obtained by the industry. This is due to conditions attained by the industry where the amount of ore that is in contact with the leaching solution could be less than that assumed in this simulation. Throughout the preceding analysis it was assumed that 80 percent of the ore is under ideal leaching conditions, and as such is in perfect contact with the leaching solution. For actual leaching conditions, the percent of the total ore deposit that is in contact with the solution could be less than 80; consequently, the recoveries shown in table 4 would be reduced. Analysis of the solution flow through the fragmented deposit would result in accurate prediction of industrial conditions. Table 5 shows the overall average production cost per tonne of extracted copper. One can conclude from this table that, for the ore grades and thicknesses investigated, the average cost is independent of both variables. The average profit per kilogram of extracted copper was, however, shown (3) to be a function of the amount of copper produced and as such a function of both ore thickness and grade. Table 5 also shows the rate of return on total investment assuming 100-percent equity for all the cases being investigated. While the rate of return is shown to decrease, for example, from 36.8 to 34.1 percent for 25.7-meter and 34.1-meter ore thicknesses, respectively, the profit per tonne for these two cases was shown (3) to increase. The decrease in the rate of return among these two cases was, however, due to the effect of the longer time of operation on the discounted cash flow associated with the second case. For all the cases investigated, the profit per tonne of extracted copper increased with the increase in the ore thickness and/or the ore grade.

SUMMARY AND CONCLUSIONS

A simulation model was developed to determine the economics of an in situ leaching operation. The model is composed of a fragmentation model which calculates the design elements and cost of a fragmentation system, a copper recovery model which simulates the leaching process of copper from an ore column, a cutoff grade model which calculates the optimum cutoff grade and amount of copper produced at the maximum profit level, and a finance model to analyze the cash flow from the mining operation and calculate the rate of return on total investment and on invested equity.

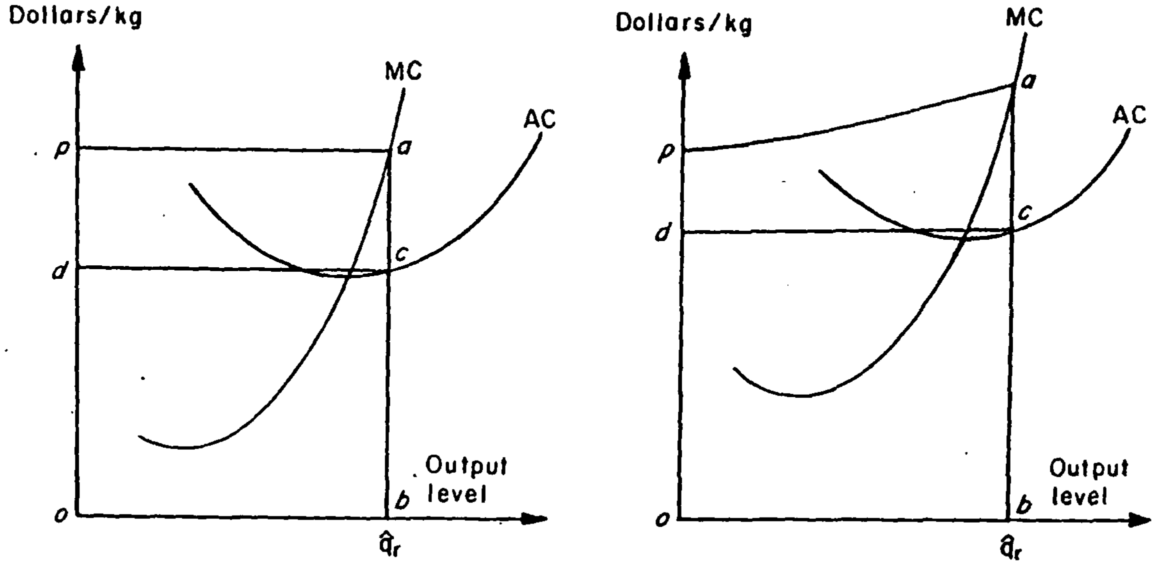
Application of these models to three assumed properties each at three assumed ore grades showed that the optimum cutoff grade does not change significantly among these conditions. This would imply that the cutoff grade is independent of the capital and development costs, which vary for these conditions. The cutoff grade is, however, shown to be a function of the operating cost as shown by the marginal cost equation. For all these cases the operating cost and, as such, the marginal costs, were identical. It can therefore be concluded that to optimize an in situ leaching operation, one has to control the level of the operating costs more than the development or capital cost. The latter costs are, however, important in controlling the average cost and therefore, the profitability of the operation. A change in the development or capital costs would result in the same optimal level of production but at a different optimal level of profits.

It was also shown that, when optimum conditions were realized, the percent of copper recovery was insignificantly different for various ore grades and significantly increased with increasing ore thickness. The calculations of the rate of return showed that, while the percent recovery and profit per unit of extracted copper increased with the increasing ore thickness, the rate of return did not have an apparent correlation and in some cases decreased with the increase of the ore thickness. This is due to the influence of the longer leaching time of the higher percent recovery on the discounted cash flow.

While the authors believe that the approach used in developing the preceding model is technically sound, the model has not been validated by comparing results from an in situ leaching operation and those derived from the model.

REFERENCES

1. White, P. M. An Application of a Weighted Moving Average Technique for Ore Reserve Calculations in Coal Deposits. M.S. Thesis, The Pennsylvania State University, University Park, Pa., March 1974, 58 pp.
2. Selim, A. Aly, and D. Yardley. The Design and Cost of a Fragmentation System for In Situ Extraction of Copper. Pres. before 14th International Annual Symposium on the Application of Computer Methods in the Mineral Industries, The Pennsylvania State University, University Park, Pa., Oct. 4-8, 1976, to be published in Proceedings; available for consultation at Bureau of Mines, Twin Cities Mining Research Center, Twin Cities, Minn., 19 pp.
3. Selim, A. Aly. A Decision Analysis Approach for In Situ Extraction of Copper. Ph.D. Thesis, University of Minnesota, Minneapolis, Minn., 659 pp.
4. Roman, R. J., B. R. Benner, and G. W. Becher. Diffusion Model for Heap-Leaching and Its Application to Scale-Up. Trans. AIME, v. 256, 1974, p. 247.
5. Roman, R. J., and C. Olsen. Theoretical Scale-Up of Heap-Leaching. Solution Mining Symposium, Proceedings. 103d AIME Annual Meeting, Dallas, Tex., Feb. 25-27, 1974, pp 211-229.
6. Rudershauser, G. G. Copper Solution at Old Reliable. Presented at National Meeting of AIChE, Salt Lake City, Utah, Aug. 18-22, 1974, 11 pp.; available from E. I. du Pont de Nemours & Co., Inc., Wilmington, Del., 19898.



Static condition

Dynamic condition

FIGURE 1.-Economic Analysis of Optimum Production Conditions

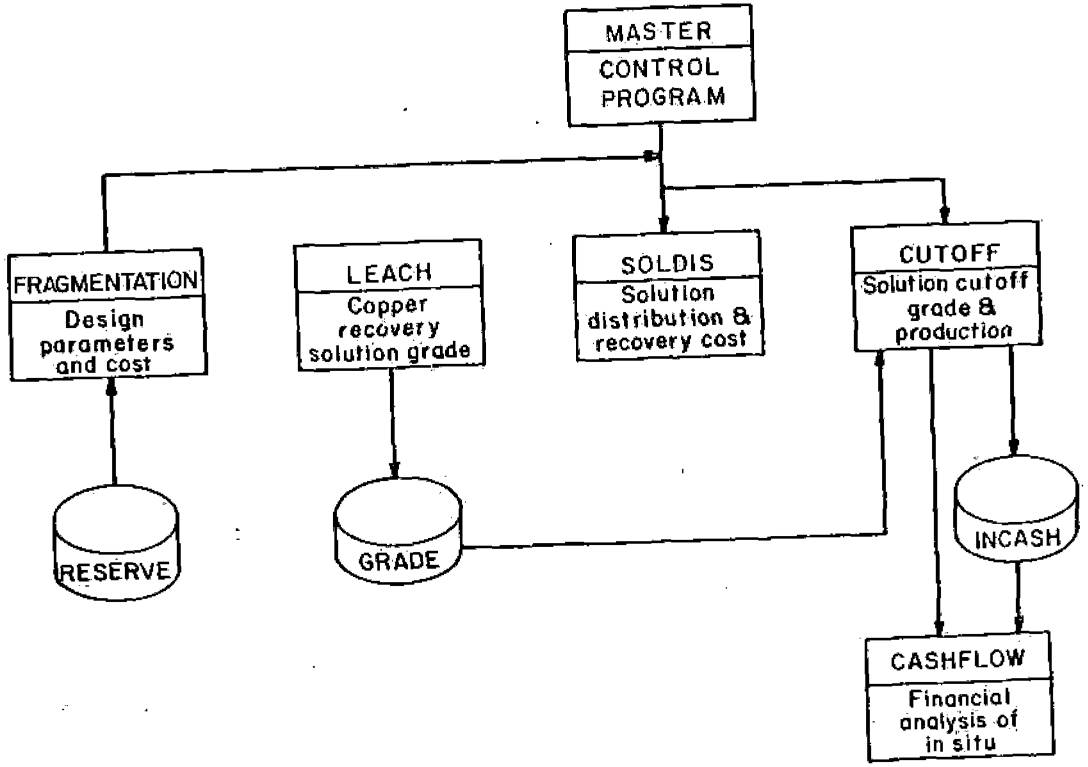


FIGURE 2.-Economic System Simulation Model, an Overview

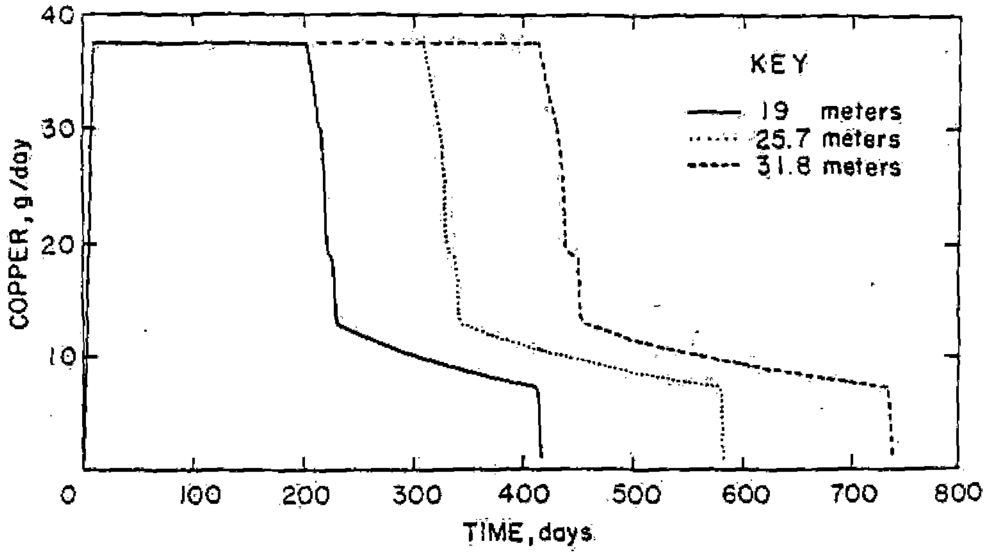


FIGURE 3.-Plant Grams Copper per Day

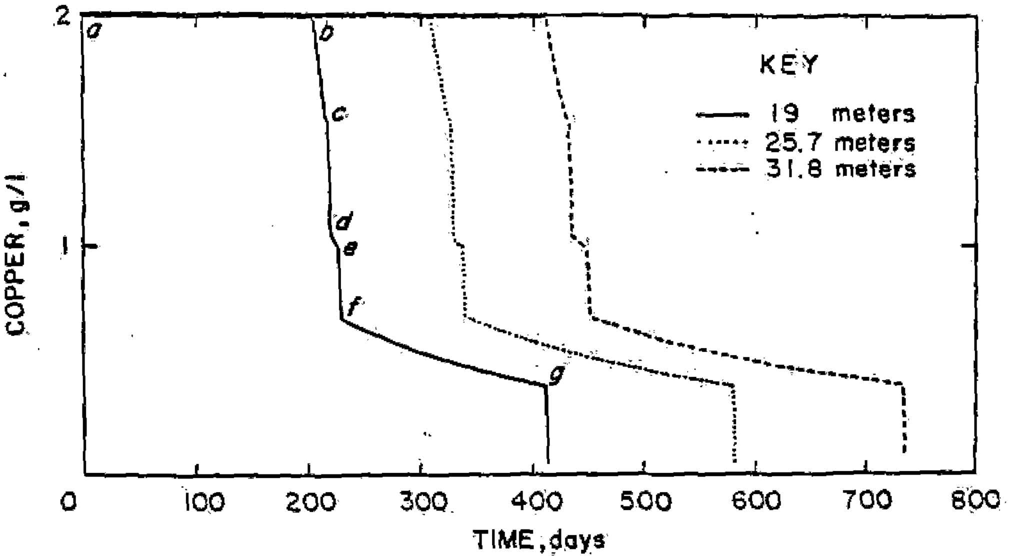


FIGURE 4.-Plant Solution Copper Concentration

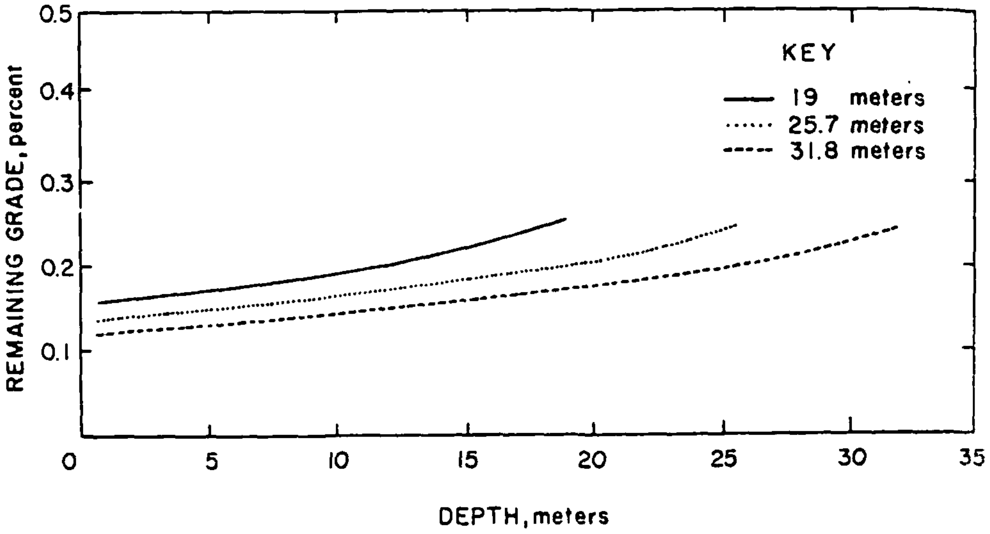


FIGURE 5.-Vertical Variation in Grade as a Function of Depth

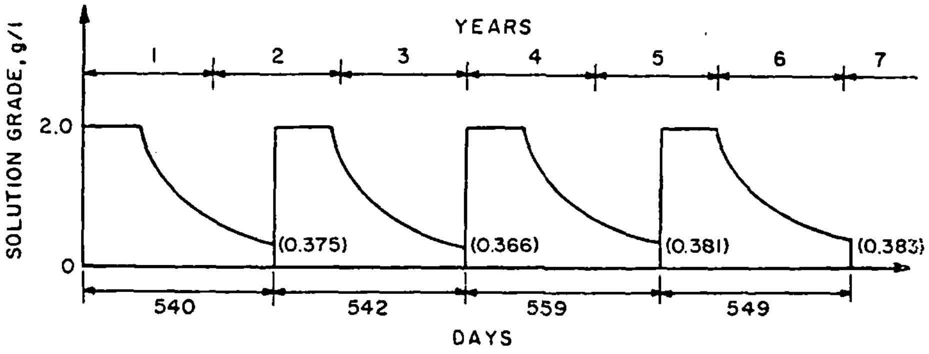


FIGURE 6.-Effluent Solution Cutoff Grade as a Function of Time

SUBJ
MNG
ISFF

IN SITU FRAGMENTATION FOR SOLUTION MINING - A RESEARCH NEED

By Richard A. Dick

* * * * * * * * * * * * * * * * * oral presentation and publication

Second International Symposium on Drilling and Blasting
Phoenix, Arizona January 22-26, 1973

CONTENTS

ILLUSTRATIONS

TABLE

IN SITU FRAGMENTATION FOR SOLUTION MINING - A RESEARCH NEED

by

Richard A. Dick[1]

ABSTRACT

A system using confined blasting with chemical explosives to fracture a mineral deposit in preparation for in situ solution mining is proposed. After a discussion of other in situ fragmentation techniques, the benefits of solution mining in environment, safety, conservation of resources, support of national interests, and the characteristics of the "ideal ore body" for solution mining are described. Blast design factors and the economics of the fragmentation system are considered, and the need for pilot scale research is pointed out. Emphasis is placed on a system designed for copper recovery from low-grade oxide deposits, but the technology is applicable to in situ recovery of such natural resources as organic fuels, uranium, and numerous other metals and nonmetals.

INTRODUCTION

The technique for leaching to extract mineral values from a host rock dates from 1752 when weathered piles of copper ore were leached at Rio Tinto, Spain (9). Leaching may have been used as early as 2500 B.C. in Cyprus. In 1965, leaching accounted for about 12 percent of the total copper production in the United States. Most of this production

---

[1] Mining engineer, Twin Cities Mining Research Center, Bureau of Mines, U.S. Department of the Interior, Minneapolis, Minn.

was from material which had been transported from its original location before leaching, such as leaching of waste dumps, heap leaching of oxide ores on prepared surfaces, vat leaching of crushed ores, and leaching as a supplement to flotation recovery.

An in situ mining system may be defined as one in which the mineral values are extracted from the ore without moving the ore from its original location. Solution mining or leaching involves separating the mineral values from the gangue by dissolution rather than by flotation, heavy media separation, or by any of the other common extractive metallurgical treatments. In situ leaching has been practiced at Ray and Miami, Arizona, but only as a secondary effort to take advantage of fragmented zones of rock created by block caving operations. This discussion is concerned with the use of chemical explosives in large diameter boreholes to fracture an ore body for the specific purpose of in situ solution mining.

ADVANTAGES OF IN SITU MINING

The mining industry has often been harshly criticized, sometimes undeservedly, for adversely affecting our environment. Figure 1 shows how in situ mining can eliminate many of the adverse environmental effects typical of conventional mining methods. In conventional mining, the excavation of large volumes of rock creates large, sometimes undesirable openings in the earth, and waste haulage creates large piles of barren rock. Neither of these problems exists with solution mining. Standard beneficiation procedures create vast piles of very fine tailings which

FIGURE 1. - Comparison of conventional and in situ mining systems

become potential sources of air, land, and water pollution. Smelter gases containing sulfur dioxide are of particular concern to the public. In situ solution mining coupled with solvent extraction recovery techniques eliminates the conventional milling and smelting processes. These factors are pointed out, not as criticism of the mining industry, but rather to illustrate some of the important environmental advantages of solution mining. Figure 2 shows what can happen to our domestic source of copper if certain proposed sulfur emission standards are imposed on the smelting industry (10). The resulting reduction of mine output could leave us with a deficit domestic supply. Any amount of copper produced without the use of smelting would help alleviate the problem.

FIGURE 2. - Possible effect of sulfur emission controls on the U.S. domestic copper supply

Because of its low development costs, in situ solution mining has its greatest advantage in small, low-grade deposits where the investment for a conventional mining system is not justified. By exploiting deposits not amenable to existing techniques, in situ mining can increase our reserves of copper, uranium, nonmetallic minerals and fuels. The resulting increased self-sufficiency in raw materials would be beneficial from a standpoint of balance of trade and national security. Reduced exposure of workers to subsurface health and safety hazards is another benefit of in situ mining.

One potential disadvantage of solution mining is the hazard of losing acid leach water to the surrounding area.

IN SITU FRAGMENTATION TECHNIQUES

In situ mining has received considerable attention recently, with most emphasis given to copper. Figure 3 illustrates four fragmentation techniques for in situ mining. The nuclear concept (A) proposed in Kennecott's Project Sloop near Safford, Ariz., is best adapted to deep, massive deposits capable of containing the tremendous nuclear energy yield. Although the technique shows both technologic and economic promise,public opposition to the commercial use of nuclear explosives and difficulties in leaching deep-lying primary ores have delayed its application.

FIGURE 3.- Fragmentation schemes proposed for in situ leaching

Coyote blasting (B), involving large concentrated charges loaded into drifts and crosscuts, has been applied in the Old Reliable project near Mammoth, Ariz. (6). A joint effort of Ranchers Exploration and Development Corp. and DuPont, this project should yield a good assessment of the compatibility between coyote blasting and solution mining. The principal shortcoming of coyote blasting in previous mining applications has been the poor fragmentation obtained because of the relatively poor powder distribution in the deposit. Ranchers' project typifies the bold, imaginative minerals research essential in maintaining an adequate domestic supply of minerals in the face of constantly dwindling ore grades. The 2,000 tons of AN-FO consumed in the Old Reliable project indicates the potential importance of in situ mining to the powder industry.

The third fragmentation system under consideration (C) involves drifting and undercutting at the base of an ore body and excavating a sufficient volume of material by block caving techniques to cause the rest of the ore body to cave. In effect, this system creates a leaching situation similar to that employed at Ray and Miami with a minimum amount of material being hoisted. This concept is being considered at the Rio Tinto property near Mountain City, Nevada, as a joint effort of the Cleveland-Cliffs Iron Co. and duPont. The system is especially applicable to depths at which conventional blasting may not give adequate swell or permeability for leaching purposes.

As this paper was in preparation, the American Smelting and Refining Co. (ASARCO) and the Dowell Division of Dow Chemical Co. announced plans to fracture and leach an ore body 1,000 ft below the surface. Water at high pressure (1,000 - 1,500 psi) will be injected through four boreholes, and the copper solution will be recovered through a fifth borehole (5). It will be interesting to see whether this technique gives adequate contact between the copper-bearing minerals and acid solution for a good copper recovery. Good recovery is especially important from a resource conservation standpoint because a low degree of copper extraction from an ore body may render it unminable, whatever the future technological advances may be. Solution losses may be difficult to control in this system,and disputes could arise between adjacent property owners about the source of the copper values.

The Ranchers, Cleveland-Cliffs, and ASARCO ventures will provide valuable information on the economics of using coyote blasting, partial block caving, and high-pressure water injection to prepare an ore body for in situ mining. However, there is a simple, very promising fragmentation technique which to date has not received adequate attention. This technique involves drilling a pattern of large-diameter (>9-in), deep, vertical boreholes and blasting with AN-FO or slurries (D). Although several companies and government agencies have indicated an interest in this approach, no significant, total-system research has yet been planned. Research is needed both to solve the technological problems and to determine the economics of using heavily confined vertical blastholes to prepare a deposit for in situ mining.

CHOOSING THE OREBODY

The proposed pilot research project consists of two phases. First, a pattern of vertical blastholes, fired without benefit of a vertical free face, is used to create permeability and surface exposure in the deposit. Second, the metal is leached from the deposit and recovered by precipitation or solvent extraction. The area of concern here is the fragmentation system. To minimize the financial risk, an ore body as nearly ideal as possible should be chosen for the initial effort. Figure 4 defines the characteristics of this "ideal ore body." Overburden should be at a minimum since barren overburden means high drilling and blasting costs which cannot be offset by mineral recovery. The initial effort should be restricted to a total depth near 200 ft. The ultimate

FIGURE 4. - Characteristics of an "ideal ore body"

practical depth limit for fracturing an ore body with heavily confined chemical explosive charges is not now known. The ore values should consist mainly of oxides and carbonates,which are most easily leached, although chalcocite would also be acceptable. The ore grade should be high enough to give the project a reasonable chance for economic success, but should be low enough to provide a realistic test. A deposit running 10 to 16 lb/ton should be about right. The deposit should be above the water table to minimize solution recovery problems and permit the use of cheaper, less water-resistant blasting agents. An impermeable underlying bed would be helpful in minimizing solution losses. Finally, to minimize capital investment, the target ore body should be near adequate maintenance facilities, a good water supply and a precipitation plant.

Realistically, since all of these characteristics will not be available in a single deposit, some practical compromises will be required.

Although leaching a high-oxide deposit may require a considerable amount of acid, a cheap, plentiful supply of sulfuric acid may be available in the future, as seen in figure 5 (10). Sulfur emission regulations imposed on smelters, coal-burning power plants, and petroleum and natural gas could result in large surpluses of available acid and elemental sulfur. Oxide leaching operations could be a convenient disposal ground for this excess acid.

FIGURE 5. - U.S. production, demand, and potential supplies of sulfur, including the effect of sulfur emission regulations

DESIGN OF THE BLAST

The blast must be designed to (1) create sufficient permeability in the deposit to enable the leaching solution to enter at given points, pass through the mineralized zone, and be collected, and (2) create adequate surface exposure to enable the leach solution to contact a sufficient proportion of the copper values to assure a satisfactory rate of recovery. The three primary blast design factors to be determined are blasthole spacing, blasthole diameter, and type of blasting agent.

In normal bench blasting, each blasthole breaks to a nearby parallel free face. In most in situ mining situations, however, the blastholes will have no free face, other than the original ground surface, to which to break. The limited amount of research that has been done on totally confined explosive charges indicates that they are less efficient in breaking rock than standard bench blasts. Figure 6 shows that zones of damage caused by firing a single confined charge at a relatively shallow depth (1). A severely crushed zone extends to a distance of about twice the charge radius. Numerous cracks are produced out to a distance of about six charge radii, and a smaller number extend to about twelve charge radii. In contrast to a single charge, the proposed in situ blasting program will have the advantage of enhanced blast effects between adjacent charges. However, there will be the disadvantage of considerably more burden to overcome. If these two factors counterbalance each other and the damage zone of twelve radii shown in figure 6 is obtained, the blasthole spacing would be about 12 times the charge diameter. If a second free face or voids from previous mining activities are present, if enhancement between adjacent charges is sufficient, or if a practical

FIGURE 6. - Zones of damage around a confined charge

system of chambering the bottoms of the boreholes were available, this spacing could be increased.

The economic factors determining the choice of a blasthole size are quite interesting. Historically, the cost per foot of borehole was partly a function of the diameter, with the cost per foot increasing as the borehole diameter increased. The advent of large-diameter rotary drilling has changed this. Bauer describes the economics of rotary drilling in a very interesting article in the January 1971 issue of the Journal of the South African Institute of Mining and Metallurgy (2). As rotary bit diameters increase, the bearing strength, bit life, and penetration rate all increase, resulting in the trends shown in figure 7, where cost per foot is plotted as a function of borehole diameter for several rock types. Only in very hard rock does the cost increase with diameter. In copper ore, costs drop progressively in the 9-, 10-, and 12-in diameter. It would be interesting to see the trend carried out to a 15-in diameter. Figure 7 indicates that the choice of blasthole diameter is not in itself an extremely important economic decision. This choice depends largely on the type of blasting agent used and the blasthole spacing.

FIGURE 7. - Cost per linear foot for rotary drilling

The blasting product used should be a bulk-loaded dry blasting agent or slurry. Although explosives experts have conflicting opinions on the relative merits of the two types of product (4, 8) and express basic disagreements about the usefulness of various techniques of estimating blasting agent strengths, the facts are, as borne out by the explosive consumption trends shown in figure 8, that dry blasting agents are heavily favored

FIGURE 8 - Explosive consumption trends

by all segments of the blasting industry (3). Figure 9 shows the reason for this preference. Dry blasting agents are cheaper per pound, or per

FIGURE 9. - Comparative costs of blasting agents

unit of theoretical energy released, than other blasting products. AN-FO has three inherent disadvantages: (1) inefficiency in very small charge diameters, (2) low density, and (3) lack of water resistance. The first disadvantage does not apply here. According to Bauer's figures on drilling costs, a larger borehole can be drilled for about the same cost to compensate for AN-FO's lower density. Wet conditions would, however, dictate the use of the more expensive water resistant slurries for part or all of the explosive charge. Blasthole dewatering and external protection of AN-FO is a possibility.

ECONOMICS OF THE DRILLING AND BLASTING PROGRAM

The cost of fracturing a deposit for in situ leaching can be estimated by making a few basic assumptions. The following formula gives the drilling and blasting costs per pound of copper recovered by in situ leaching.

$$C_t = \frac{(T+D+J)C_d + (T+D+J-Y)(B^2)(G)\ (.34)(C_e+C_l)}{(S^2)(D)(d)(R)}$$

where

C_t = total fragmentation costs, ¢/lb

T = overburden thickness, ft

D = ore body thickness, ft

J = subdrilling depth, ft

C_d = blasthole drilling cost, ¢/ft

Y = length of stemming, ft

B = borehole diameter, in

G = explosive specific gravity

C_e = explosive cost, ¢/lb

C_l = cost of loading, stemming, firing and explosive accessories, ¢/lb of explosive

S = borehole spacing, ft

d = ore density, tons/ft^3

R = recovered copper, lb/ton of ore

The left side of the numerator gives the cost of drilling a borehole; the right side gives the cost of blasting. The denominator gives the amount of copper recovered per borehole. The formula can be adapted to determine the costs for any drilling and blasting program. To calculate some sample costs for an in situ drilling and blasting program, the following assumptions are made. A 150-ft ore zone is overlain by a 50-ft barren overburden. The boreholes are subdrilled 5 ft and powder is loaded to the top of the ore zone. Drilling costs are \$2.00/ft for both the 9- and 12-1/4-in holes. Blasting agent costs are 4 cents/lb for AN-FO, 8 cents/lb for a nonaluminized slurry, and 14 cents/lb for a 10 percent aluminum slurry. Specific gravities are 0.8, 1.15, and 1.30, respectively. The cost of explosive loading, stemming, firing, and accessories is 3 cents/lb of blasting agent. The ore has a density of 12.5 ft^3/ton and yields 6 lb of recovered copper per ton.

Table 1 shows some calculated drilling and blasting costs. A 9-in blasthole with AN-FO on a 9-ft spacing gives fragmentation costs of 11.1 cent/lb of copper. This is a maximum blasting effort based on a cracked zone 12 times the blasthole diameter and gives a powder factor of 7.6 lb/yd^3, which is quite high. Using a 12-1/4-in blasthole on a proportionally expanded spacing gives a fragmentation cost of 7.9 cents/lb. This cost reduction is due to the constant drilling cost per foot of borehole, regardless of diameter, described in figure 7, and illustrates the advantage of the larger blastholes.

Using a nonaluminized slurry and a 10 percent aluminum slurry in the same 12-1/4-in borehole on a 12-1/4-ft spacing gives very high costs of 13.1 cent/lb and 20.0 cent/lb, respectively. However, the powder factors of 10.9 lb/yd^3 are unrealistically high. There is reason to believe that enhanced blast effects between adjacent charges would permit the use of spacings greater than 12 borehole diameters. Table 1 shows the striking reduction in costs obtained by expanding the spacing by 50 percent. AN-FO blasting costs are reduced to only 3.5 cents/lb,and even the higher priced slurry gives favorable costs of 8.9 cent/lb. The effect of any other changed condition on the blasting cost can easily be determined by substituting values in the cost equation. As a point of comparison, table 1 includes two cost figures reported by Hardwick (7) in 1967 for fracturing an ore body with a totally confined nuclear charge. Later sources have reported costs one-fourth to one-third higher. It should be understood, however, that chemical and nuclear blasting will not normally compete for the same orebody. Chemical blasting is better adapted to exploiting relatively shallow deposits containing oxides and secondary sulfide minerals, and confined nuclear blasting is restricted to deeper deposits where the copper is largely in the form of primary sulfides.

RESEARCH NEEDED

The fragmentation costs look promising and most of the needed leaching technology is available. What is required now is a pilot-scale research program to design an adequate fragmentation system. Among the many unknowns involved in blasting with heavily confined explosive charges are these: maximum feasible depths, optimum borehole spacings, priming techniques, delays within and between blast holes, and ground vibrations. We do not know whether uniform overall rubblization is necessary for optimum recovery or whether coarse fragmentation is sufficient, assuming that the fractures will tend to occur in zones of mineralization, which normally are planes of weakness. Although one might assume that the finest fragmentation attainable will give the best mineral recovery, some knowledgeable people believe that an excess of fines will seriously reduce ore body permeability, especially where clays are present. Inadequate fragmentation could cause solution channeling. Only an actual leaching program will properly assess the adequacy of fragmentation. A pilot blasting and leaching test in a carefully chosen piece of ground near existing facilities would not involve an unduly large capital risk, since most or all of the costs would be met by the value of the recovered copper. The initial tests could conceivably realize a profit.

SUMMARY

Most past leaching experience has been with rock which has been moved from its original location. In situ mining, where the mineral is extracted from the rock in its original location, is advantageous from the standpoint of the environment, conservation of resources, safety, and national self-sufficiency in natural resources. Fragmentation systems under study for in situ mining include coyote blasting, nuclear blasting, partial block caving, and high-pressure water injection. Blasting with large-diameter vertical blastholes offers a promising alternative. A target ore body should be chosen that will maximize the chances for a successful operation. Although little is known about blasting with heavily confined charges, the economics of a system employing dry blasting agents or slurries in large (12-1/4-in-) diameter blastholes appears favorable. Analysis of leaching and dissolution costs was beyond the scope of this paper. Research on a pilot-plant scale is needed to investigate the technological and economic aspects of the total drill-blast-leach-recover system; Such a project would provide valuable spinoff information on in situ mining of oil shale, tar sands, and numerous leachable metallic and nonmetallic minerals.

REFERENCES

1. Atchison, T.C. Fragmentation Principles. Surface Mining. AIME, New York, 1968, pp. 355-372.
2. Bauer, Alan. Open Pit Drilling and Blasting. Journal of the South African Institute of Mining and Metallurgy, v. 71, No. 6, January 1971, pp. 115-121.
3. Bureau of Mines. Mineral Industry Surveys. Washington, D.C., 1967, 1968, 1969, 1970.
4. Cook, Melvin A. How Dry Mix Explosives Can Increase Costs - Even in Dry Holes. Eng. and Min. J., v.172, No. 9, Sept. 1971, pp.140-141.
5. Engineering and Mining Journal. ASARCO and Dow Chemical to Leach Deep Copper Orebody In Situ in Arizona. V. 173, No. 6, June 1972, p. 19.
6. Engineering and Mining Journal. Ranchers' Big Blast Shatters Copper Orebody for In Situ Leaching. V. 173, No. 4, April 1972, pp. 98-100.
7. Hardwick, William R. Fracturing a Deposit with Nuclear Explosives and Recovering Copper by the In Situ Leaching method. BuMines Rept. of Inv. 6996, Nov. 1967, 48 pp.
8. Lansdale, John R. How Dry Mix Explosives Can Shave Costs. Eng. and Min. J., v. 172, No. 7, July 1971, p.77.
9. Scheffer, Herman W. and LaMar G. Evans. Copper Leaching Practices in the Western United States. BuMines Inf. Circ. 8341, 1968, 57 pp.
10. U.S. Department of the Interior. First Annual Report of the Secretary of the Interior Under the Mining and Minerals Policy Act of 1970. March 1972, 142 pp.

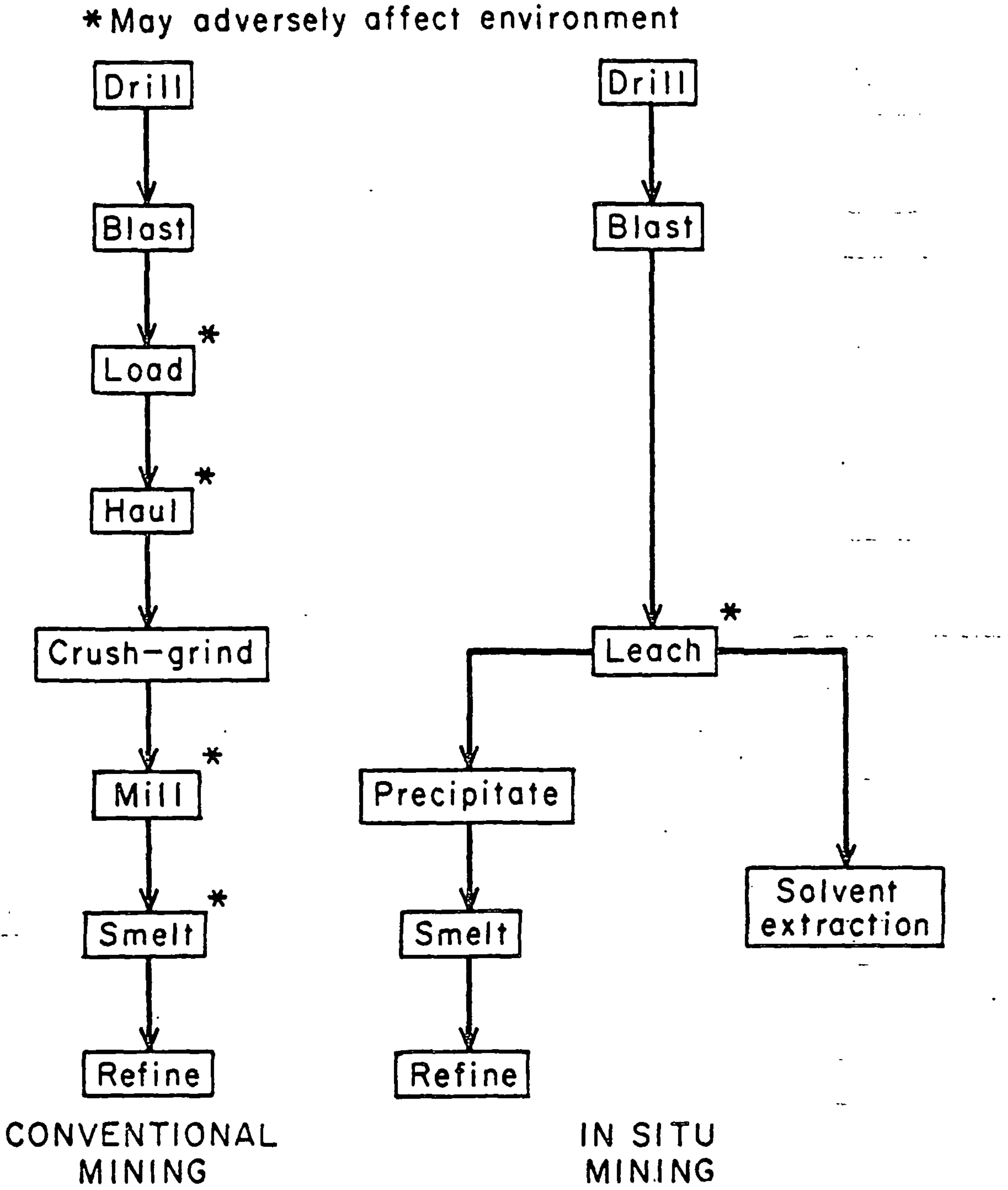


FIGURE 1.–Comparison of Conventional and In Situ Mining Systems

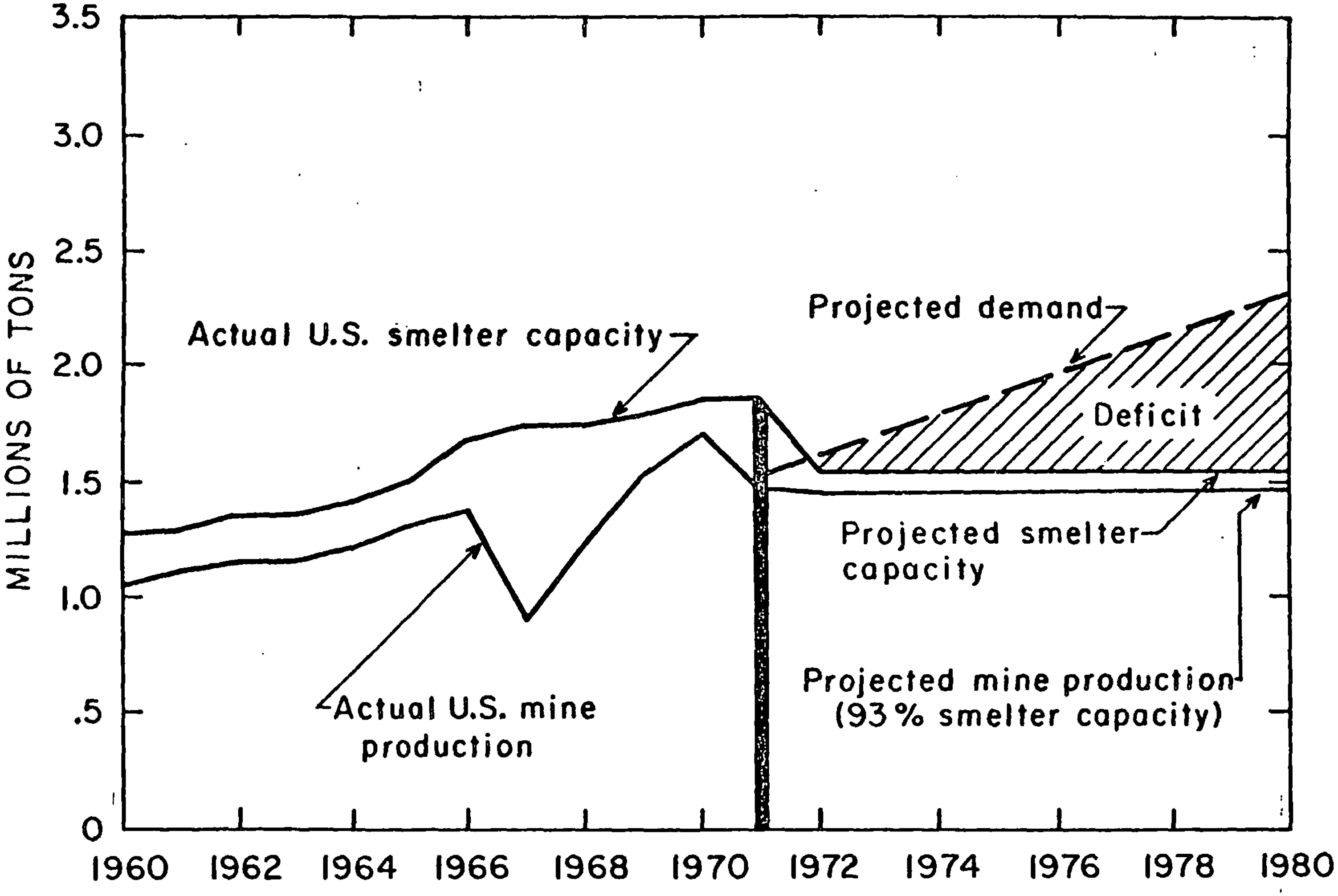


FIGURE 2.–Possible Effect of Sulfur Emission Controls on the U.S. Domestic Copper Supply.

(Source: First Annual Report–Mining and Minerals Policy Act of 1970)

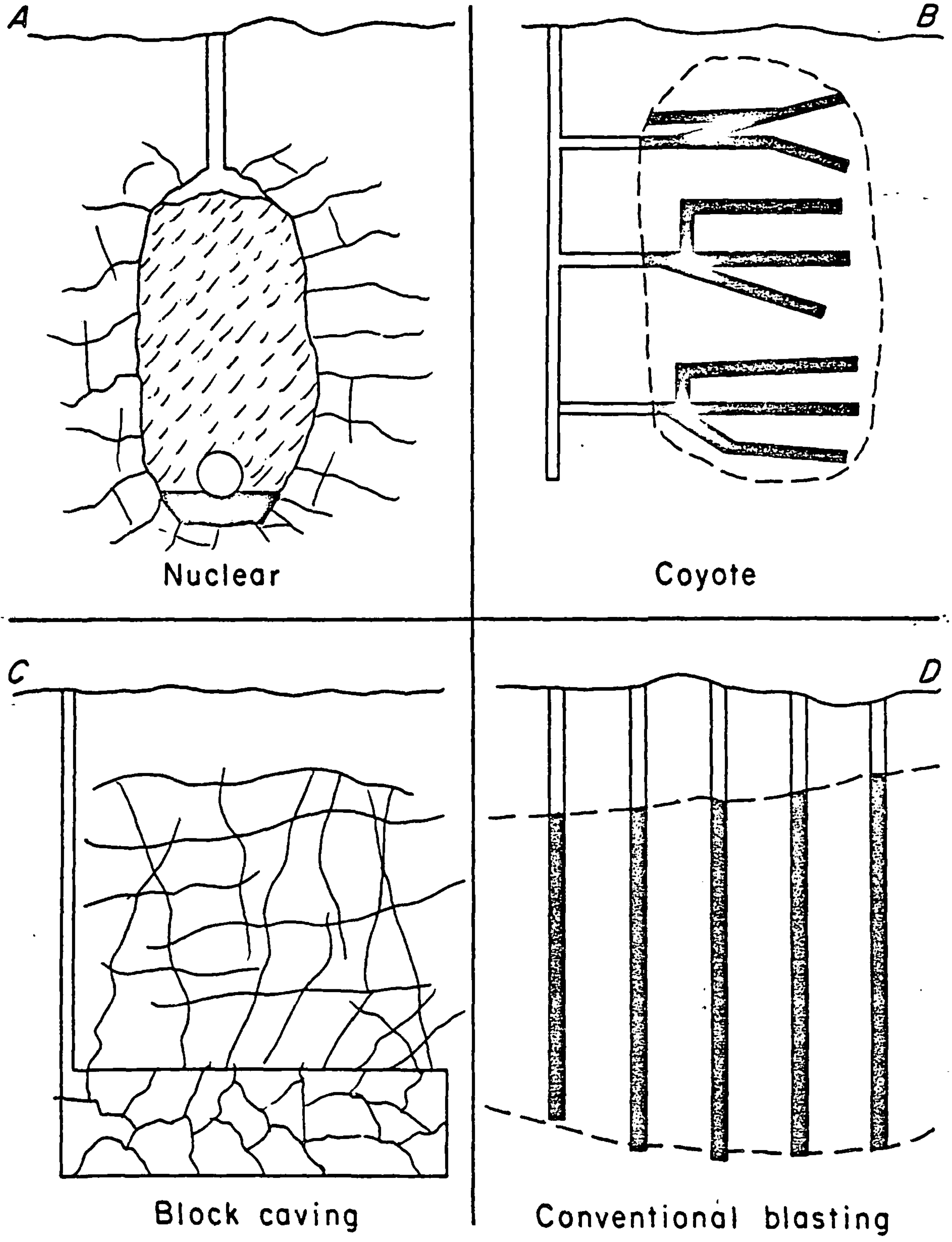


FIGURE 3.-Fragmentation Schemes Proposed for In Situ Leaching.

"IDEAL ORE BODY" CHARACTERISTICS

Minimum overburden

Total depth near 200 feet

0.5 % to 0.8 % copper

Oxide-carbonate minerals

Above water table

Impermeable underlying bed

Near existing facilities

FIGURE 4.–Characteristics of an "Ideal Ore Body."

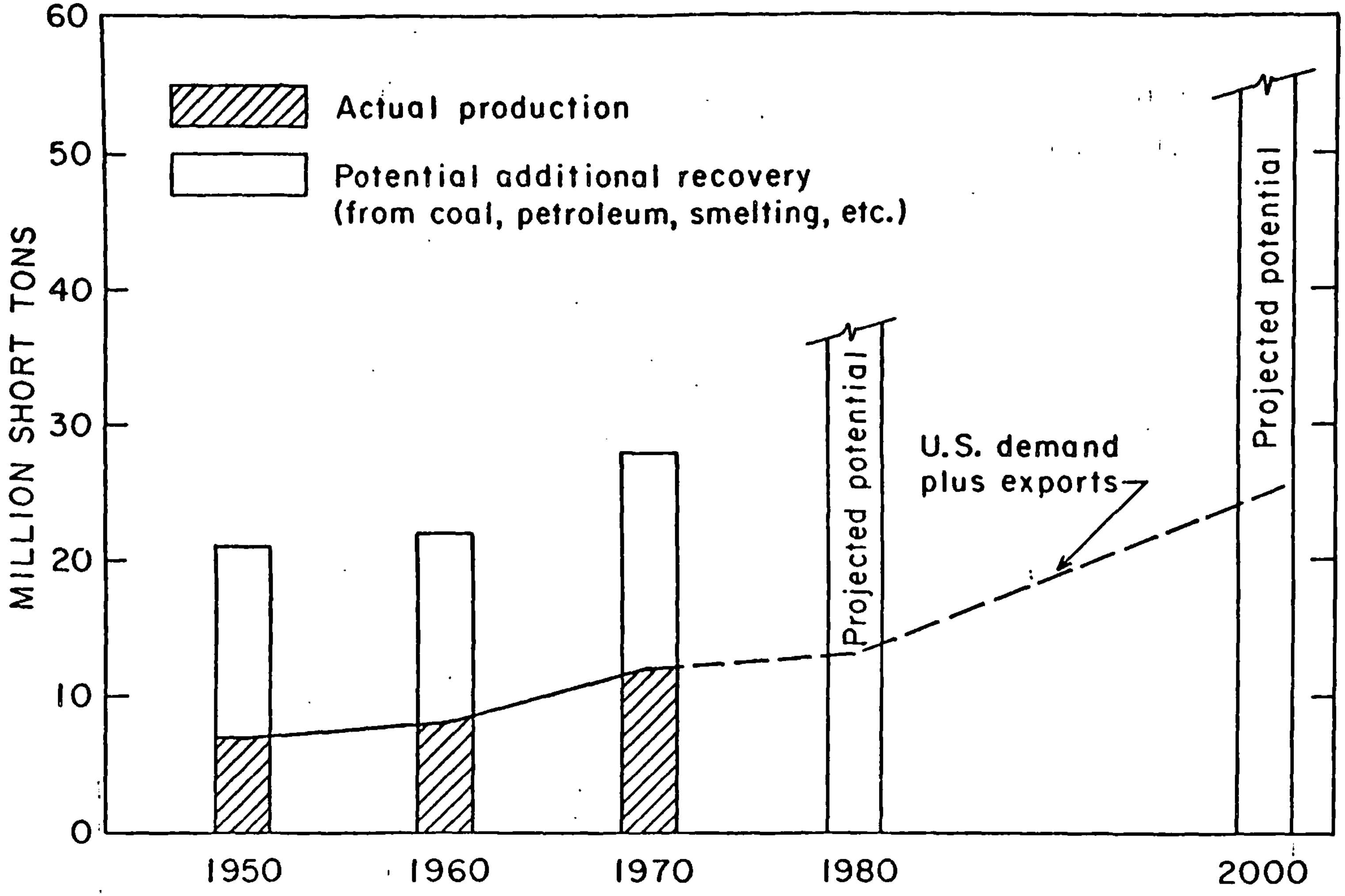
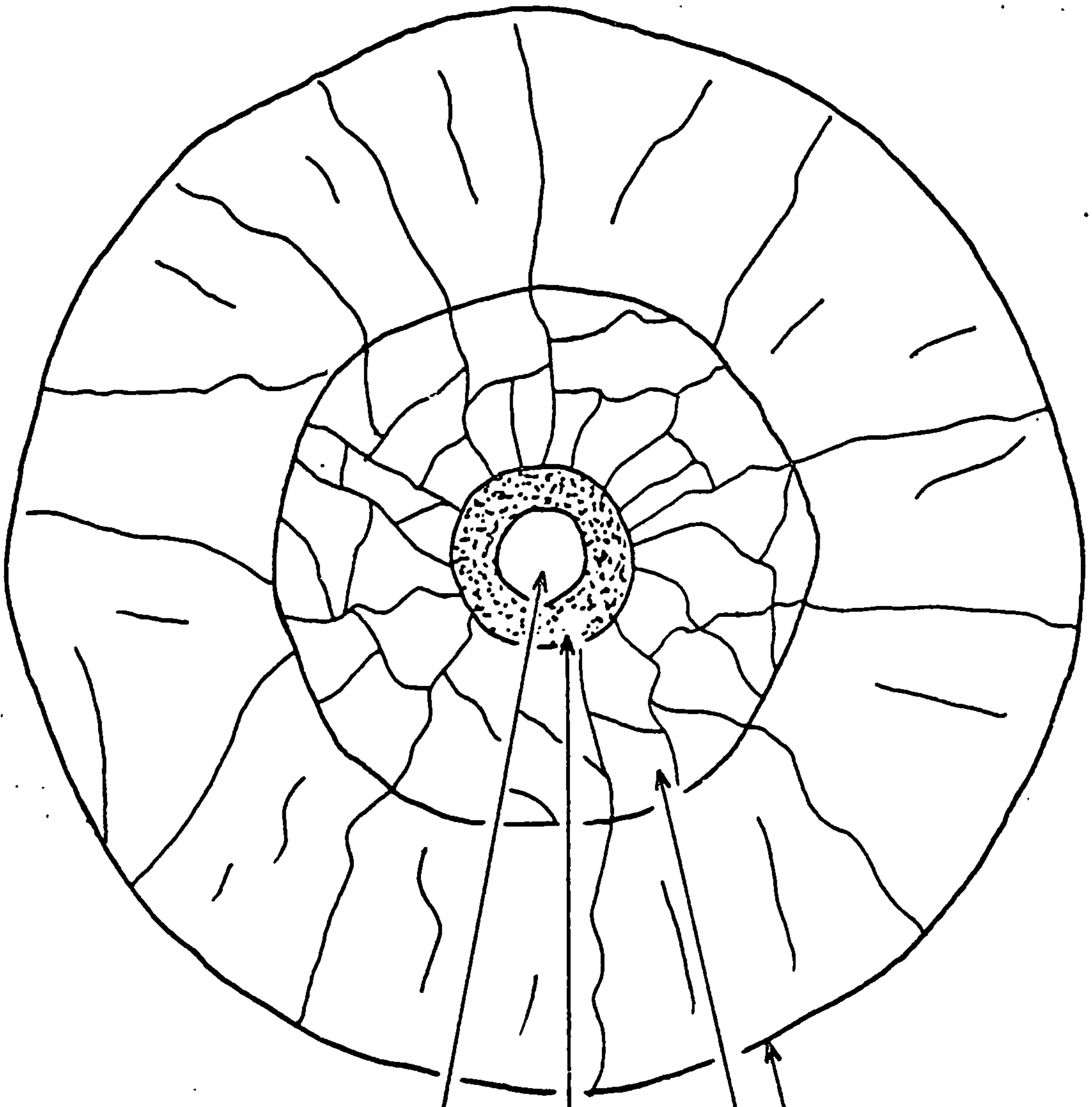


FIGURE 5.–U.S. Production, Demand, and Potential Supplies of Sulfur, Including the Effect of Sulfur Emission Regulations.

(Source: First annual report–Mining and Minerals Policy Act of 1970)



Borehole (D)
Crushed zone (≈2D)
Heavily cracked zone (≈6D)
Limit of cracking (≈12D)
D=borehole diameter

FIGURE 6.–Zones of Damage Around a Confined Charge

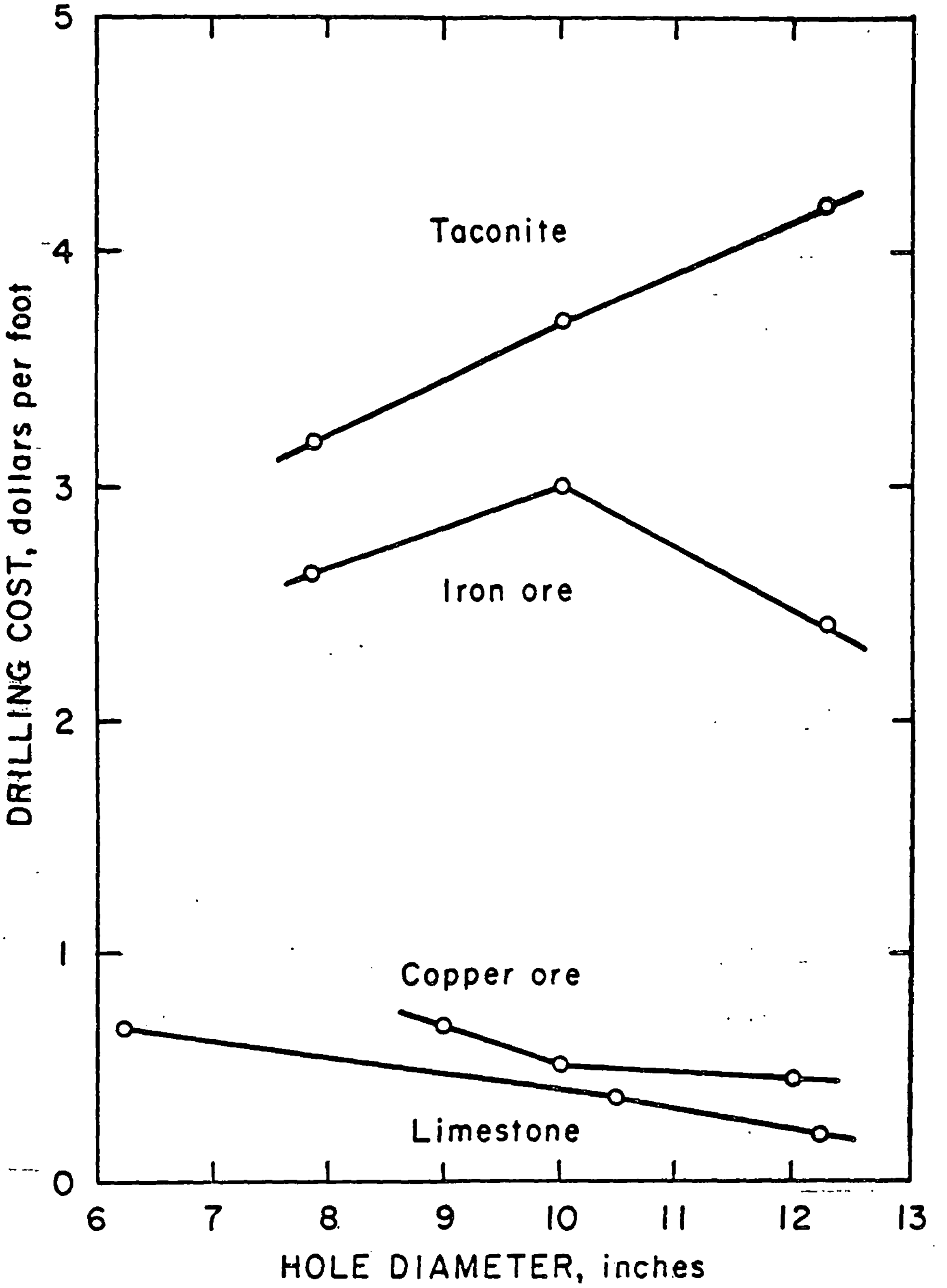


FIGURE 7.- Cost per Linear Foot for Rotary Drilling.

(After Bauer)

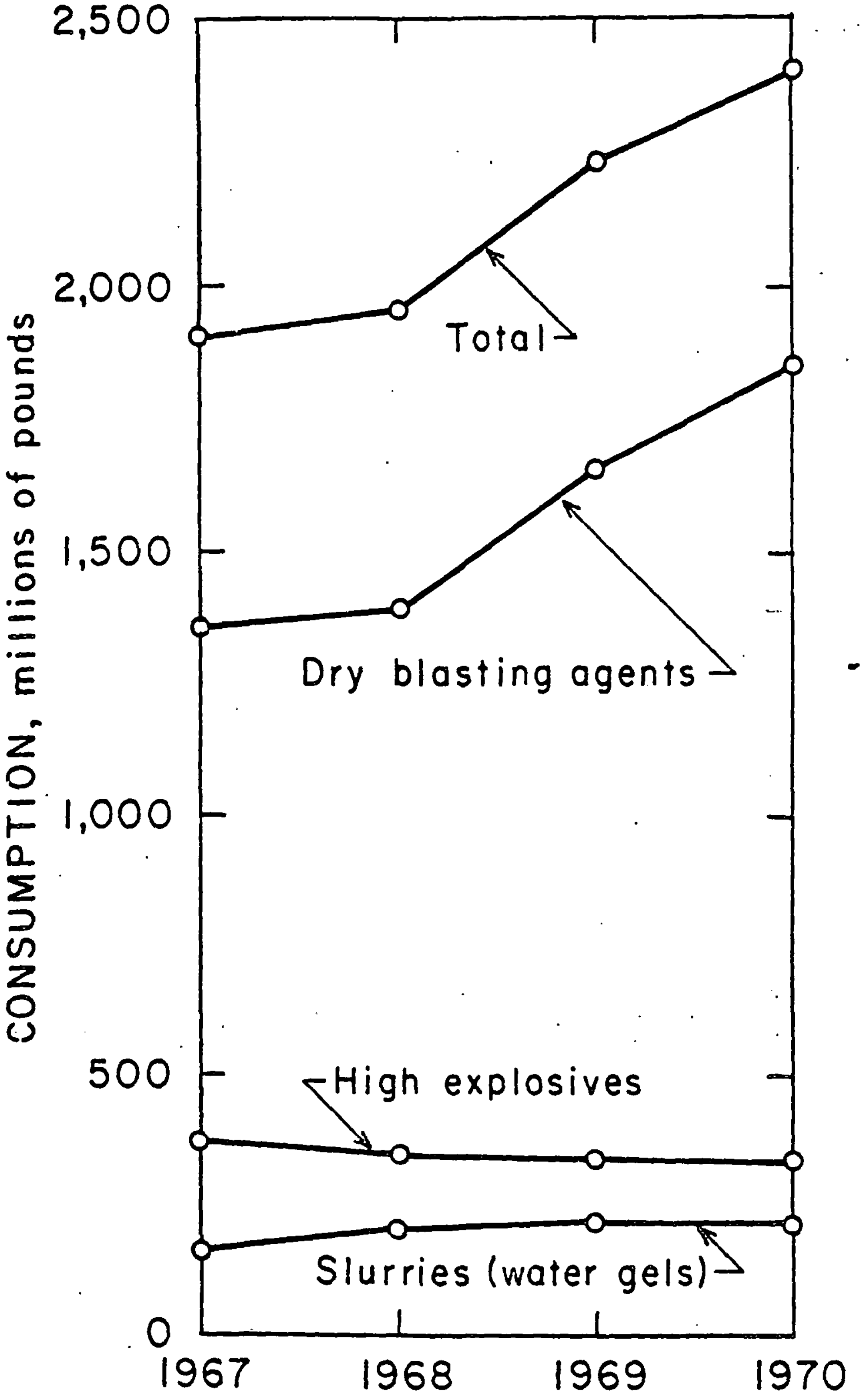


FIGURE 8.-Explosive Consumption Trends.

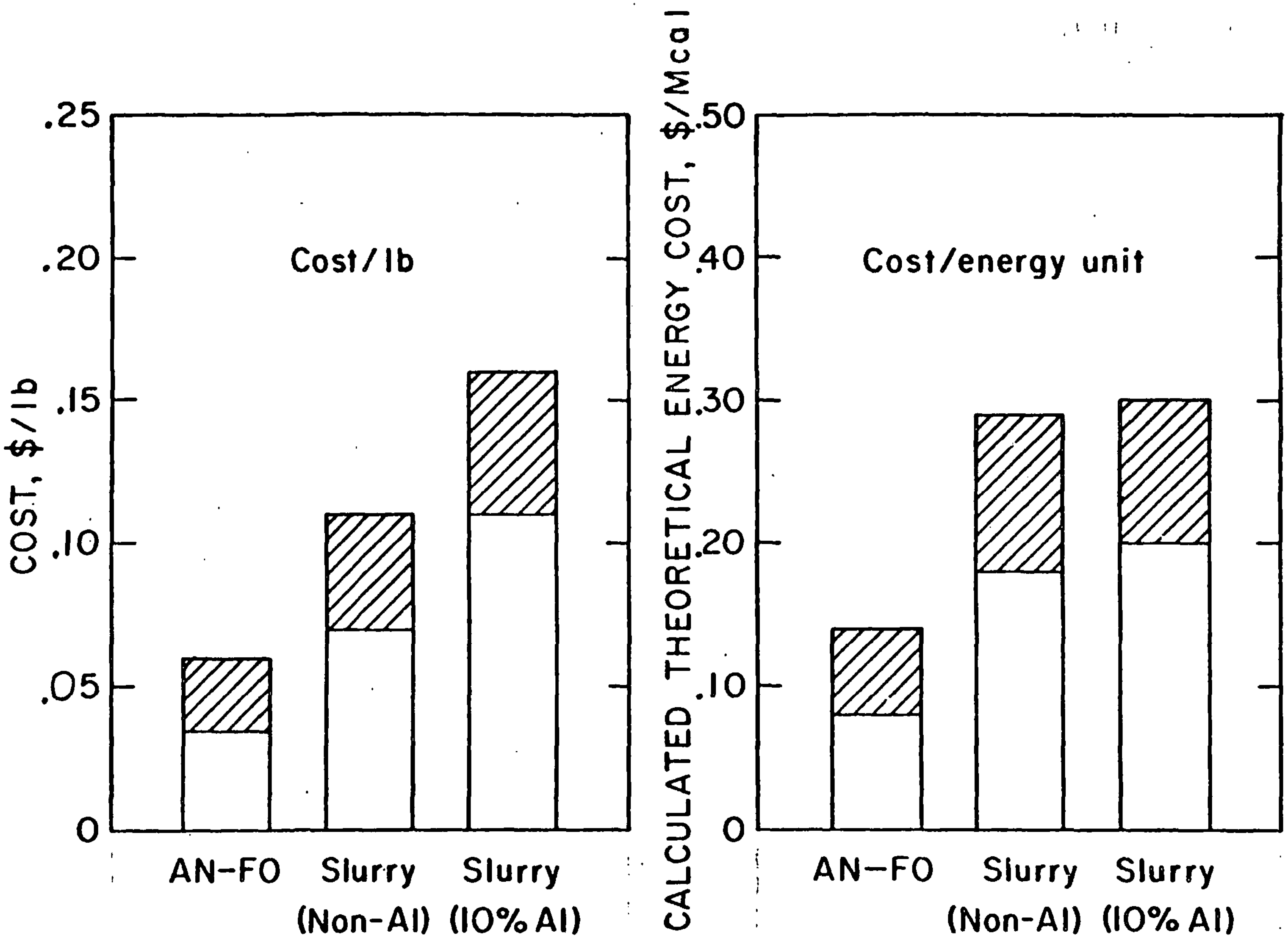


FIGURE 9.–Comparative Cost of Blasting Agents.

Charge diameter, inches	Product	Spacing, feet	Cost, cents/lb			Powder factor, lb/cu yd
			Drilling	Blasting	Total	
9	AN-FO	9	7.0	4.1	11.1	7.6
12¼	AN-FO	12¼	3.8	4.1	7.9	7.6
12¼	Non Al slurry	12¼	3.8	9.3	13.1	10.9
12¼	10% Al slurry	12¼	3.8	16.2	20.0	12.3
12¼	AN-FO	18¾	1.7	1.8	3.5	3.4
12¼	10% Al slurry	18¾	1.7	7.2	8.9	5.5

10 kton nuclear device (confined) 9.2 cents/lb

20 kton nuclear device (confined) 5.6 cents/lb

TABLE I.—In Situ Drilling and Blasting Costs.

IN SITU FRAGMENTATION FOR SOLUTION MINING - A RESEARCH NEED

By Richard A. Dick

* * * * * * * * * * * * * * * * * oral presentation and publication

Second International Symposium on Drilling and Blasting
Phoenix, Arizona January 22-26, 1973

CONTENTS

ILLUSTRATIONS

TABLE

IN SITU FRAGMENTATION FOR SOLUTION MINING - A RESEARCH NEED

by

Richard A. Dick[1]

ABSTRACT

A system using confined blasting with chemical explosives to fracture a mineral deposit in preparation for in situ solution mining is proposed. After a discussion of other in situ fragmentation techniques, the benefits of solution mining in environment, safety, conservation of resources, support of national interests, and the characteristics of the "ideal ore body" for solution mining are described. Blast design factors and the economics of the fragmentation system are considered, and the need for pilot scale research is pointed out. Emphasis is placed on a system designed for copper recovery from low-grade oxide deposits, but the technology is applicable to in situ recovery of such natural resources as organic fuels, uranium, and numerous other metals and nonmetals.

INTRODUCTION

The technique for leaching to extract mineral values from a host rock dates from 1752 when weathered piles of copper ore were leached at Rio Tinto, Spain (9). Leaching may have been used as early as 2500 B.C. in Cyprus. In 1965, leaching accounted for about 12 percent of the total copper production in the United States. Most of this production

---

[1] Mining engineer, Twin Cities Mining Research Center, Bureau of Mines, U.S. Department of the Interior, Minneapolis, Minn.

was from material which had been transported from its original location before leaching, such as leaching of waste dumps, heap leaching of oxide ores on prepared surfaces, vat leaching of crushed ores, and leaching as a supplement to flotation recovery.

An in situ mining system may be defined as one in which the mineral values are extracted from the ore without moving the ore from its original location. Solution mining or leaching involves separating the mineral values from the gangue by dissolution rather than by flotation, heavy media separation, or by any of the other common extractive metallurgical treatments. In situ leaching has been practiced at Ray and Miami, Arizona, but only as a secondary effort to take advantage of fragmented zones of rock created by block caving operations. This discussion is concerned with the use of chemical explosives in large diameter boreholes to fracture an ore body for the specific purpose of in situ solution mining.

ADVANTAGES OF IN SITU MINING

The mining industry has often been harshly criticized, sometimes undeservedly, for adversely affecting our environment. Figure 1 shows how in situ mining can eliminate many of the adverse environmental effects typical of conventional mining methods. In conventional mining, the excavation of large volumes of rock creates large, sometimes undesirable openings in the earth, and waste haulage creates large piles of barren rock. Neither of these problems exists with solution mining. Standard beneficiation procedures create vast piles of very fine tailings which

---

FIGURE 1. - Comparison of conventional and in situ mining systems

---

become potential sources of air, land, and water pollution. Smelter gases containing sulfur dioxide are of particular concern to the public. In situ solution mining coupled with solvent extraction recovery techniques eliminates the conventional milling and smelting processes. These factors are pointed out, not as criticism of the mining industry, but rather to illustrate some of the important environmental advantages of solution mining. Figure 2 shows what can happen to our domestic source of copper if certain proposed sulfur emission standards are imposed on the smelting industry (10). The resulting reduction of mine output could leave us with a deficit domestic supply. Any amount of copper produced without the use of smelting would help alleviate the problem.

FIGURE 2. - Possible effect of sulfur emission controls on the U.S. domestic copper supply

Because of its low development costs, in situ solution mining has its greatest advantage in small, low-grade deposits where the investment for a conventional mining system is not justified. By exploiting deposits not amenable to existing techniques, in situ mining can increase our reserves of copper, uranium, nonmetallic minerals and fuels. The resulting increased self-sufficiency in raw materials would be beneficial from a standpoint of balance of trade and national security. Reduced exposure of workers to subsurface health and safety hazards is another benefit of in situ mining.

One potential disadvantage of solution mining is the hazard of losing acid leach water to the surrounding area.

IN SITU FRAGMENTATION TECHNIQUES

FIGURE 3.- Fragmentation schemes proposed for in situ leaching

In situ mining has received considerable attention recently, with most emphasis given to copper. Figure 3 illustrates four fragmentation techniques for in situ mining. The nuclear concept (A) proposed in Kennecott's Project Sloop near Safford, Ariz., is best adapted to deep, massive deposits capable of containing the tremendous nuclear energy yield. Although the technique shows both technologic and economic promise,public opposition to the commercial use of nuclear explosives and difficulties in leaching deep-lying primary ores have delayed its application.

Coyote blasting (B), involving large concentrated charges loaded into drifts and crosscuts, has been applied in the Old Reliable project near Mammoth, Ariz. (6). A joint effort of Ranchers Exploration and Development Corp. and DuPont, this project should yield a good assessment of the compatibility between coyote blasting and solution mining. The principal shortcoming of coyote blasting in previous mining applications has been the poor fragmentation obtained because of the relatively poor powder distribution in the deposit. Ranchers' project typifies the bold, imaginative minerals research essential in maintaining an adequate domestic supply of minerals in the face of constantly dwindling ore grades. The 2,000 tons of AN-FO consumed in the Old Reliable project indicates the potential importance of in situ mining to the powder industry.

The third fragmentation system under consideration (C) involves drifting and undercutting at the base of an ore body and excavating a sufficient volume of material by block caving techniques to cause the rest of the ore body to cave. In effect, this system creates a leaching situation similar to that employed at Ray and Miami with a minimum amount of material being hoisted. This concept is being considered at the Rio Tinto property near Mountain City, Nevada, as a joint effort of the Cleveland-Cliffs Iron Co. and duPont. The system is especially applicable to depths at which conventional blasting may not give adequate swell or permeability for leaching purposes.

As this paper was in preparation, the American Smelting and Refining Co. (ASARCO) and the Dowell Division of Dow Chemical Co. announced plans to fracture and leach an ore body 1,000 ft below the surface. Water at high pressure (1,000 - 1,500 psi) will be injected through four boreholes, and the copper solution will be recovered through a fifth borehole (5). It will be interesting to see whether this technique gives adequate contact between the copper-bearing minerals and acid solution for a good copper recovery. Good recovery is especially important from a resource conservation standpoint because a low degree of copper extraction from an ore body may render it unminable, whatever the future technological advances may be. Solution losses may be difficult to control in this system,and disputes could arise between adjacent property owners about the source of the copper values.

The Ranchers, Cleveland-Cliffs, and ASARCO ventures will provide valuable information on the economics of using coyote blasting, partial block caving, and high-pressure water injection to prepare an ore body for in situ mining. However, there is a simple, very promising fragmentation technique which to date has not received adequate attention. This technique involves drilling a pattern of large-diameter (>9-in), deep, vertical boreholes and blasting with AN-FO or slurries (D). Although several companies and government agencies have indicated an interest in this approach, no significant, total-system research has yet been planned. Research is needed both to solve the technological problems and to determine the economics of using heavily confined vertical blastholes to prepare a deposit for in situ mining.

CHOOSING THE OREBODY

The proposed pilot research project consists of two phases. First, a pattern of vertical blastholes, fired without benefit of a vertical free face, is used to create permeability and surface exposure in the deposit. Second, the metal is leached from the deposit and recovered by precipitation or solvent extraction. The area of concern here is the fragmentation system. To minimize the financial risk, an ore body as nearly ideal as possible should be chosen for the initial effort. Figure 4 defines the characteristics of this "ideal ore body." Overburden should be at a minimum since barren overburden means high drilling and blasting costs which cannot be offset by mineral recovery. The initial effort should be restricted to a total depth near 200 ft. The ultimate

FIGURE 4. - Characteristics of an "ideal ore body"

practical depth limit for fracturing an ore body with heavily confined chemical explosive charges is not now known. The ore values should consist mainly of oxides and carbonates,which are most easily leached, although chalcocite would also be acceptable. The ore grade should be high enough to give the project a reasonable chance for economic success, but should be low enough to provide a realistic test. A deposit running 10 to 16 lb/ton should be about right. The deposit should be above the water table to minimize solution recovery problems and permit the use of cheaper, less water-resistant blasting agents. An impermeable underlying bed would be helpful in minimizing solution losses. Finally, to minimize capital investment, the target ore body should be near adequate maintenance facilities, a good water supply and a precipitation plant.

Realistically, since all of these characteristics will not be available in a single deposit, some practical compromises will be required.

Although leaching a high-oxide deposit may require a considerable amount of acid, a cheap, plentiful supply of sulfuric acid may be available in the future, as seen in figure 5 (<u>10</u>). Sulfur emission regulations imposed on smelters, coal-burning power plants, and petroleum and natural gas could result in large surpluses of available acid and elemental sulfur. Oxide leaching operations could be a convenient disposal ground for this excess acid.

FIGURE 5. - U.S. production, demand, and potential supplies of sulfur, including the effect of sulfur emission regulations

DESIGN OF THE BLAST

The blast must be designed to (1) create sufficient permeability in the deposit to enable the leaching solution to enter at given points, pass through the mineralized zone, and be collected, and (2) create adequate surface exposure to enable the leach solution to contact a sufficient proportion of the copper values to assure a satisfactory rate of recovery. The three primary blast design factors to be determined are blasthole spacing, blasthole diameter, and type of blasting agent.

In normal bench blasting, each blasthole breaks to a nearby parallel free face. In most in situ mining situations, however, the blastholes will have no free face, other than the original ground surface, to which to break. The limited amount of research that has been done on totally confined explosive charges indicates that they are less efficient in breaking rock than standard bench blasts. Figure 6 shows that zones of damage caused by firing a single confined charge at a relatively shallow depth (1). A severely crushed zone extends to a distance of about twice the charge radius. Numerous cracks are produced out to a distance of about six charge radii, and a smaller number extend to about twelve charge radii. In contrast to a single charge, the proposed in situ blasting program will have the advantage of enhanced blast effects between adjacent charges. However, there will be the disadvantage of considerably more burden to overcome. If these two factors counterbalance each other and the damage zone of twelve radii shown in figure 6 is obtained, the blasthole spacing would be about 12 times the charge diameter. If a second free face or voids from previous mining activities are present, if enhancement between adjacent charges is sufficient, or if a practical

FIGURE 6. - Zones of damage around a confined charge

system of chambering the bottoms of the boreholes were available, this spacing could be increased.

The economic factors determining the choice of a blasthole size are quite interesting. Historically, the cost per foot of borehole was partly a function of the diameter, with the cost per foot increasing as the borehole diameter increased. The advent of large-diameter rotary drilling has changed this. Bauer describes the economics of rotary drilling in a very interesting article in the January 1971 issue of the Journal of the South African Institute of Mining and Metallurgy (2). As rotary bit diameters increase, the bearing strength, bit life, and penetration rate all increase, resulting in the trends shown in figure 7, where cost per foot is plotted as a function of borehole diameter for several rock types. Only in very hard rock does the cost increase with diameter. In copper ore, costs drop progressively in the 9-, 10-, and 12-in diameter. It would be interesting to see the trend carried out to a 15-in diameter. Figure 7 indicates that the choice of blasthole diameter is not in itself an extremely important economic decision. This choice depends largely on the type of blasting agent used and the blasthole spacing.

FIGURE 7. - Cost per linear foot for rotary drilling

The blasting product used should be a bulk-loaded dry blasting agent or slurry. Although explosives experts have conflicting opinions on the relative merits of the two types of product (4, 8) and express basic disagreements about the usefulness of various techniques of estimating blasting agent strengths, the facts are, as borne out by the explosive consumption trends shown in figure 8, that dry blasting agents are heavily favored by all segments of the blasting industry (3). Figure 9 shows the reason for this preference. Dry blasting agents are cheaper per pound, or per unit of theoretical energy released, than other blasting products.

FIGURE 8 - Explosive consumption trends

FIGURE 9. - Comparative costs of blasting agents

AN-FO has three inherent disadvantages: (1) inefficiency in very small charge diameters, (2) low density, and (3) lack of water resistance. The first disadvantage does not apply here. According to Bauer's figures on drilling costs, a larger borehole can be drilled for about the same cost to compensate for AN-FO's lower density. Wet conditions would, however, dictate the use of the more expensive water resistant slurries for part or all of the explosive charge. Blasthole dewatering and external protection of AN-FO is a possibility.

ECONOMICS OF THE DRILLING AND BLASTING PROGRAM

The cost of fracturing a deposit for in situ leaching can be estimated by making a few basic assumptions. The following formula gives the drilling and blasting costs per pound of copper recovered by in situ leaching.

$$C_t = \frac{(T+D+J)C_d + (T+D+J-Y)(B^2)(G)\ (.34)(C_e+C_1)}{(S^2)(D)(d)(R)}$$

where

C_t = total fragmentation costs, ¢/lb

T = overburden thickness, ft

D = ore body thickness, ft

J = subdrilling depth, ft

C_d = blasthole drilling cost, ¢/ft

Y = length of stemming, ft

B = borehole diameter, in

G = explosive specific gravity

C_e = explosive cost, ¢/lb

C_1 = cost of loading, stemming, firing and explosive accessories, ¢/lb of explosive

S = borehole spacing, ft

d = ore density, tons/ft^3

R = recovered copper, lb/ton of ore

The left side of the numerator gives the cost of drilling a borehole; the right side gives the cost of blasting. The denominator gives the amount of copper recovered per borehole. The formula can be adapted to determine the costs for any drilling and blasting program. To calculate some sample costs for an in situ drilling and blasting program, the following assumptions are made. A 150-ft ore zone is overlain by a 50-ft barren overburden. The boreholes are subdrilled 5 ft and powder is loaded to the top of the ore zone. Drilling costs are $2.00/ft for both the 9- and 12-1/4-in holes. Blasting agent costs are 4 cents/lb for AN-FO, 8 cents/lb for a nonaluminized slurry, and 14 cents/lb for a 10 percent aluminum slurry. Specific gravities are 0.8, 1.15, and 1.30, respectively. The cost of explosive loading, stemming, firing, and accessories is 3 cents/lb of blasting agent. The ore has a density of 12.5 ft^3/ton and yields 6 lb of recovered copper per ton.

Table 1 shows some calculated drilling and blasting costs. A 9-in blasthole with AN-FO on a 9-ft spacing gives fragmentation costs of 11.1 cent/lb of copper. This is a maximum blasting effort based on a cracked zone 12 times the blasthole diameter and gives a powder factor of 7.6 lb/yd^3, which is quite high. Using a 12-1/4-in blasthole on a proportionally expanded spacing gives a fragmentation cost of 7.9 cents/lb. This cost reduction is due to the constant drilling cost per foot of borehole, regardless of diameter, described in figure 7, and illustrates the advantage of the larger blastholes.

Using a nonaluminized slurry and a 10 percent aluminum slurry in the same 12-1/4-in borehole on a 12-1/4-ft spacing gives very high costs of 13.1 cent/lb and 20.0 cent/lb, respectively. However, the powder factors of 10.9 lb/yd^3 are unrealistically high. There is reason to believe that enhanced blast effects between adjacent charges would permit the use of spacings greater than 12 borehole diameters. Table 1 shows the striking reduction in costs obtained by expanding the spacing by 50 percent. AN-FO blasting costs are reduced to only 3.5 cents/lb,and even the higher priced slurry gives favorable costs of 8.9 cent/lb. The effect of any other changed condition on the blasting cost can easily be determined by substituting values in the cost equation. As a point of comparison, table 1 includes two cost figures reported by Hardwick (7) in 1967 for fracturing an ore body with a totally confined nuclear charge. Later sources have reported costs one-fourth to one-third higher. It should be understood, however, that chemical and nuclear blasting will not normally compete for the same orebody. Chemical blasting is better adapted to exploiting relatively shallow deposits containing oxides and secondary sulfide minerals, and confined nuclear blasting is restricted to deeper deposits where the copper is largely in the form of primary sulfides.

RESEARCH NEEDED

The fragmentation costs look promising and most of the needed leaching technology is available. What is required now is a pilot-scale research program to design an adequate fragmentation system. Among the many unknowns involved in blasting with heavily confined explosive charges are these: maximum feasible depths, optimum borehole spacings, priming techniques, delays within and between blast holes, and ground vibrations. We do not know whether uniform overall rubblization is necessary for optimum recovery or whether coarse fragmentation is sufficient, assuming that the fractures will tend to occur in zones of mineralization, which normally are planes of weakness. Although one might assume that the finest fragmentation attainable will give the best mineral recovery, some knowledgeable people believe that an excess of fines will seriously reduce ore body permeability, especially where clays are present. Inadequate fragmentation could cause solution channeling. Only an actual leaching program will properly assess the adequacy of fragmentation. A pilot blasting and leaching test in a carefully chosen piece of ground near existing facilities would not involve an unduly large capital risk, since most or all of the costs would be met by the value of the recovered copper. The initial tests could conceivably realize a profit.

SUMMARY

Most past leaching experience has been with rock which has been moved from its original location. In situ mining, where the mineral is extracted from the rock in its original location, is advantageous from the standpoint of the environment, conservation of resources, safety, and national self-sufficiency in natural resources. Fragmentation systems under study for in situ mining include coyote blasting, nuclear blasting, partial block caving, and high-pressure water injection. Blasting with large-diameter vertical blastholes offers a promising alternative. A target ore body should be chosen that will maximize the chances for a successful operation. Although little is known about blasting with heavily confined charges, the economics of a system employing dry blasting agents or slurries in large (12-1/4-in-) diameter blastholes appears favorable. Analysis of leaching and dissolution costs was beyond the scope of this paper. Research on a pilot-plant scale is needed to investigate the technological and economic aspects of the total drill-blast-leach-recover system; Such a project would provide valuable spinoff information on in situ mining of oil shale, tar sands, and numerous leachable metallic and nonmetallic minerals.

REFERENCES

1. Atchison, T.C. Fragmentation Principles. Surface Mining. AIME, New York, 1968, pp. 355-372.
2. Bauer, Alan. Open Pit Drilling and Blasting. Journal of the South African Institute of Mining and Metallurgy, v. 71, No. 6, January 1971, pp. 115-121.
3. Bureau of Mines. Mineral Industry Surveys. Washington, D.C., 1967, 1968, 1969, 1970.
4. Cook, Melvin A. How Dry Mix Explosives Can Increase Costs - Even in Dry Holes. Eng. and Min. J., v.172, No. 9, Sept. 1971, pp.140-141.
5. Engineering and Mining Journal. ASARCO and Dow Chemical to Leach Deep Copper Orebody In Situ in Arizona. V. 173, No. 6, June 1972, p. 19.
6. Engineering and Mining Journal. Ranchers' Big Blast Shatters Copper Orebody for In Situ Leaching. V. 173, No. 4, April 1972, pp. 98-100.
7. Hardwick, William R. Fracturing a Deposit with Nuclear Explosives and Recovering Copper by the In Situ Leaching method. BuMines Rept. of Inv. 6996, Nov. 1967, 48 pp.
8. Lansdale, John R. How Dry Mix Explosives Can Shave Costs. Eng. and Min. J., v. 172, No. 7, July 1971, p.77.
9. Scheffer, Herman W. and LaMar G. Evans. Copper Leaching Practices in the Western United States. BuMines Inf. Circ. 8341, 1968, 57 pp.
10. U.S. Department of the Interior. First Annual Report of the Secretary of the Interior Under the Mining and Minerals Policy Act of 1970. March 1972, 142 pp.

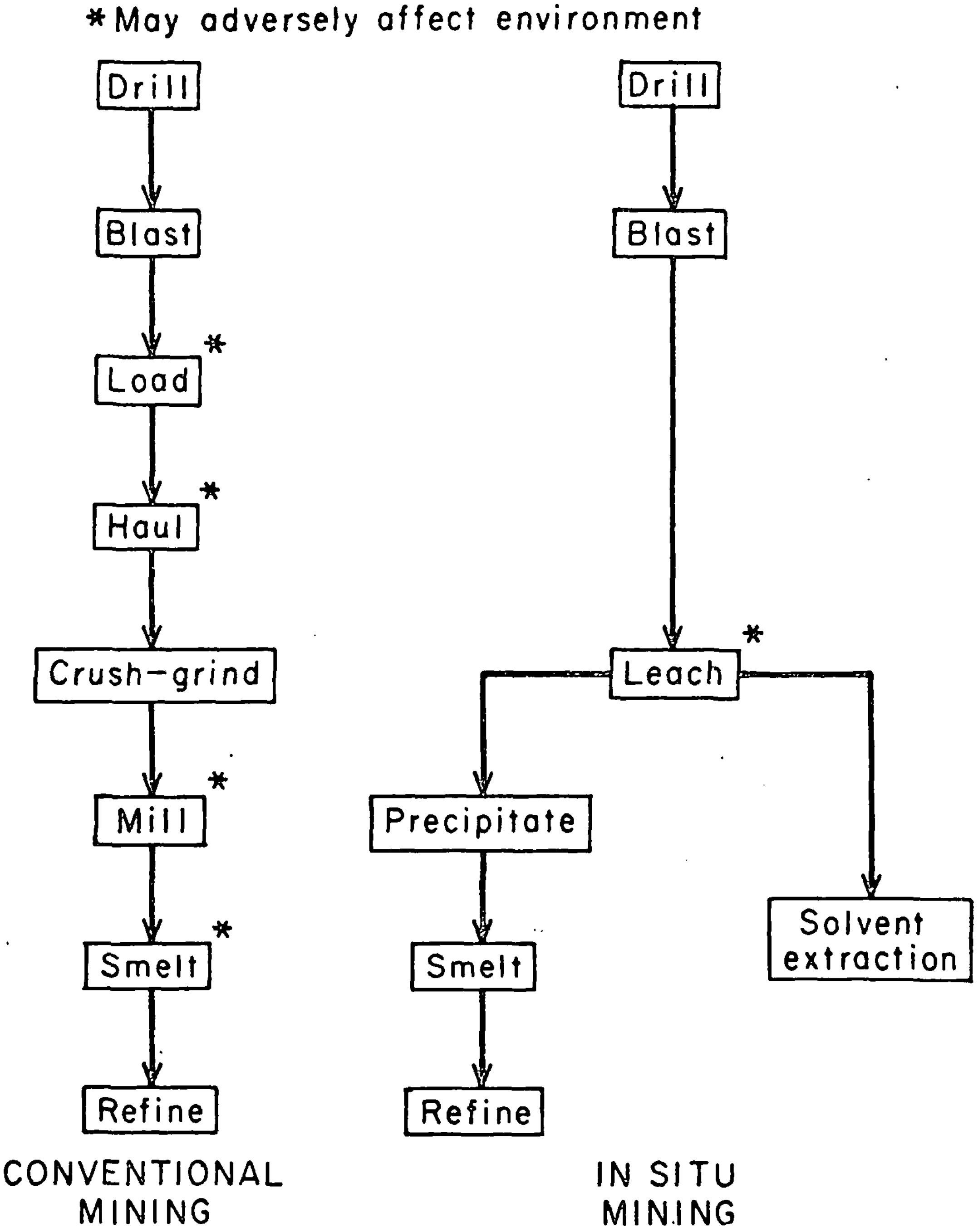


FIGURE 1.–Comparison of Conventional and In Situ Mining Systems

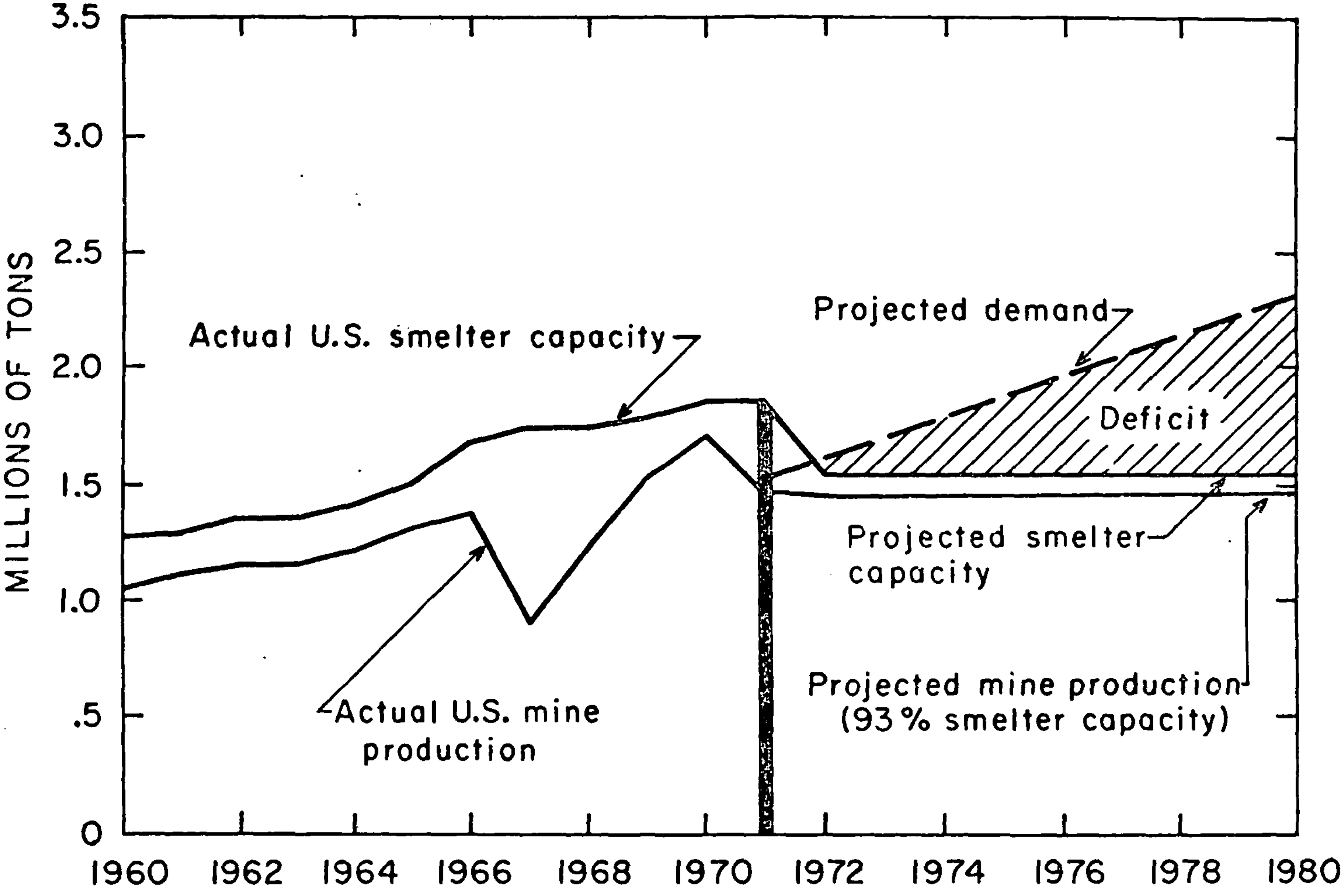


FIGURE 2.–Possible Effect of Sulfur Emission Controls on the U.S. Domestic Copper Supply.

(Source: First Annual Report–Mining and Minerals Policy Act of 1970)

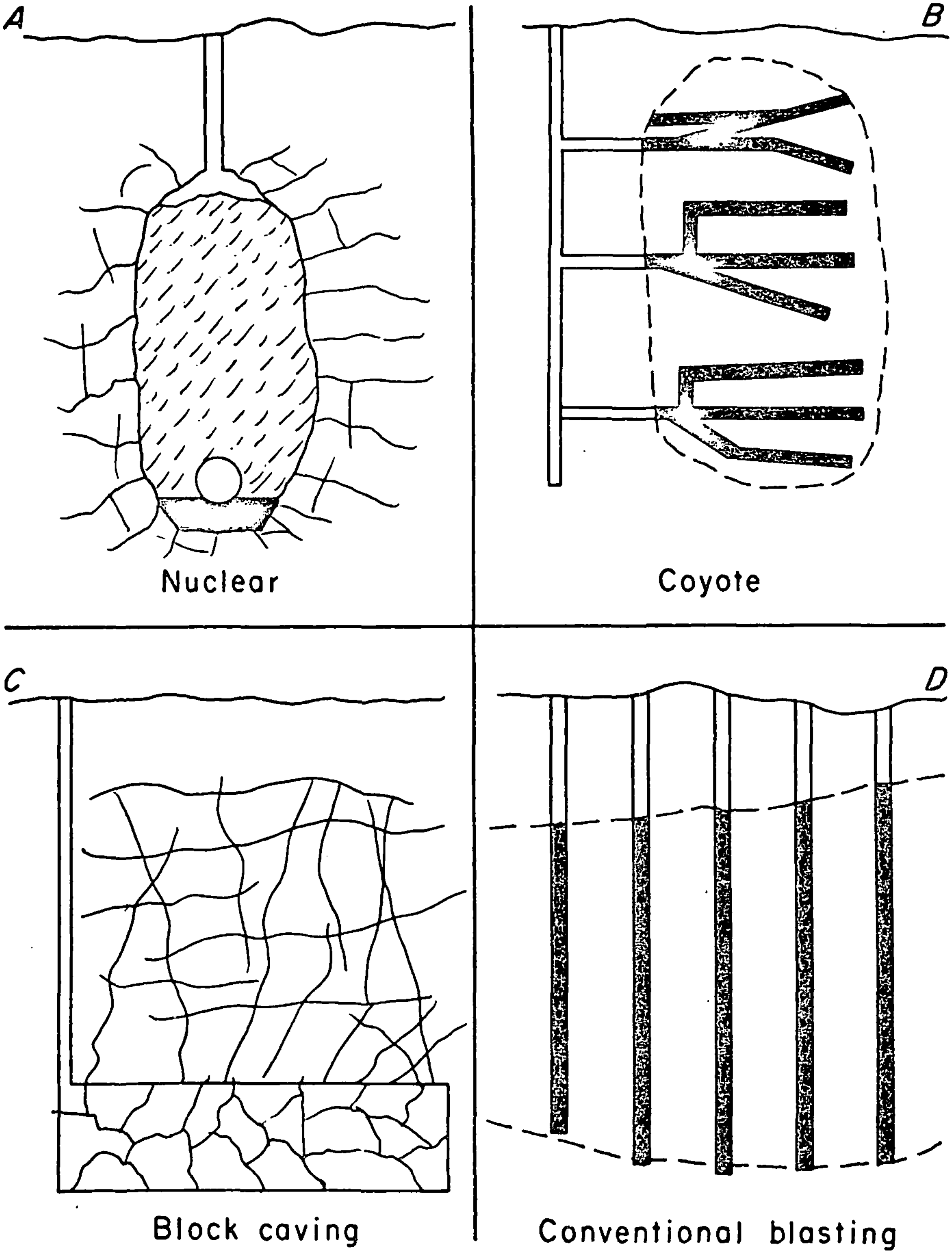


FIGURE 3.–Fragmentation Schemes Proposed for In Situ Leaching.

"IDEAL ORE BODY" CHARACTERISTICS

Minimum overburden

Total depth near 200 feet

0.5 % to 0.8 % copper

Oxide-carbonate minerals

Above water table

Impermeable underlying bed

Near existing facilities

FIGURE 4.–Characteristics of an "Ideal Ore Body."

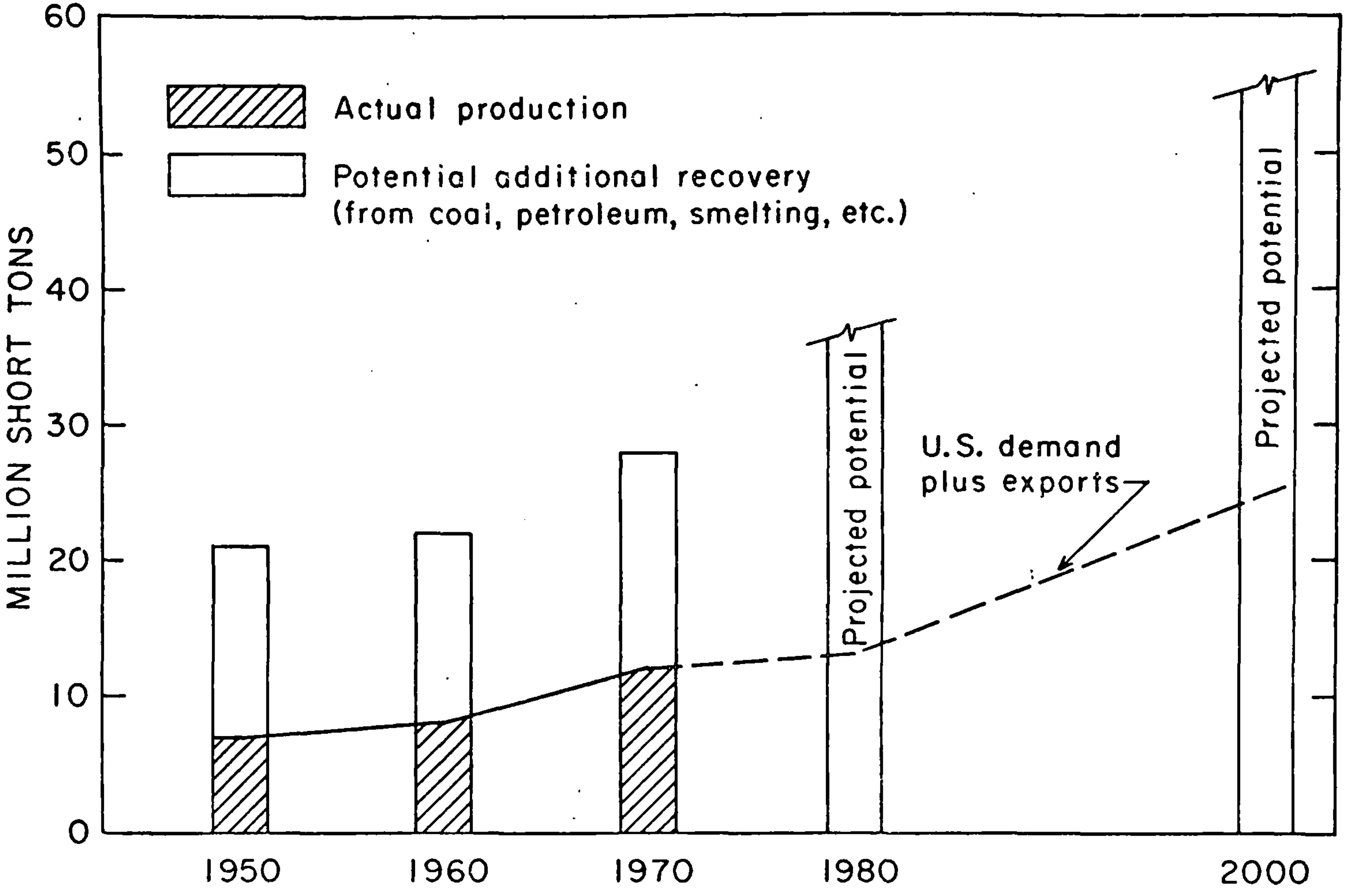


FIGURE 5.–U.S. Production, Demand, and Potential Supplies of Sulfur, Including the Effect of Sulfur Emission Regulations.

(Source: First annual report–Mining and Minerals Policy Act of 1970)

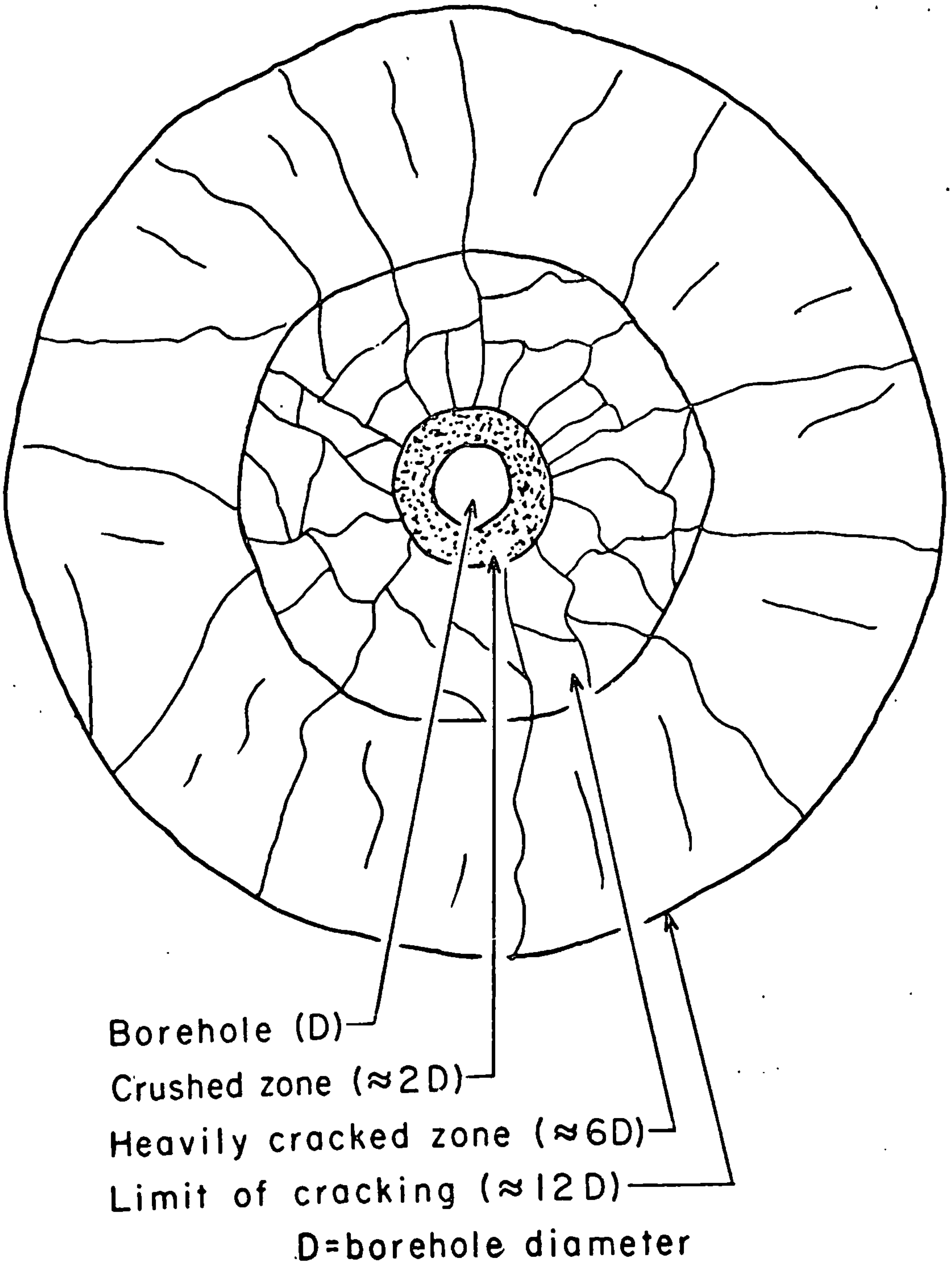


FIGURE 6.-Zones of Damage Around a Confined Charge

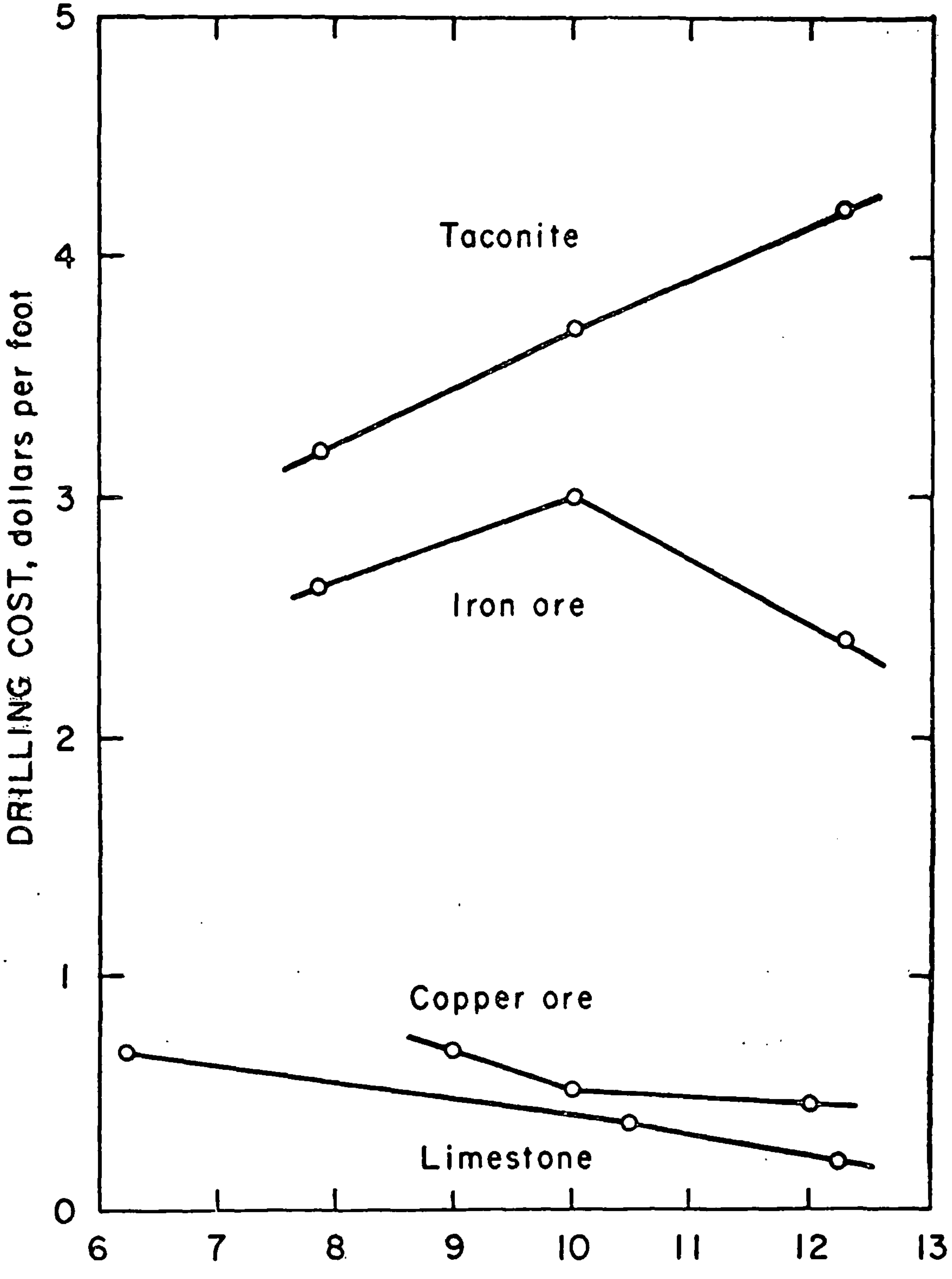


FIGURE 7.- Cost per Linear Foot for Rotary Drilling.

(After Bauer)

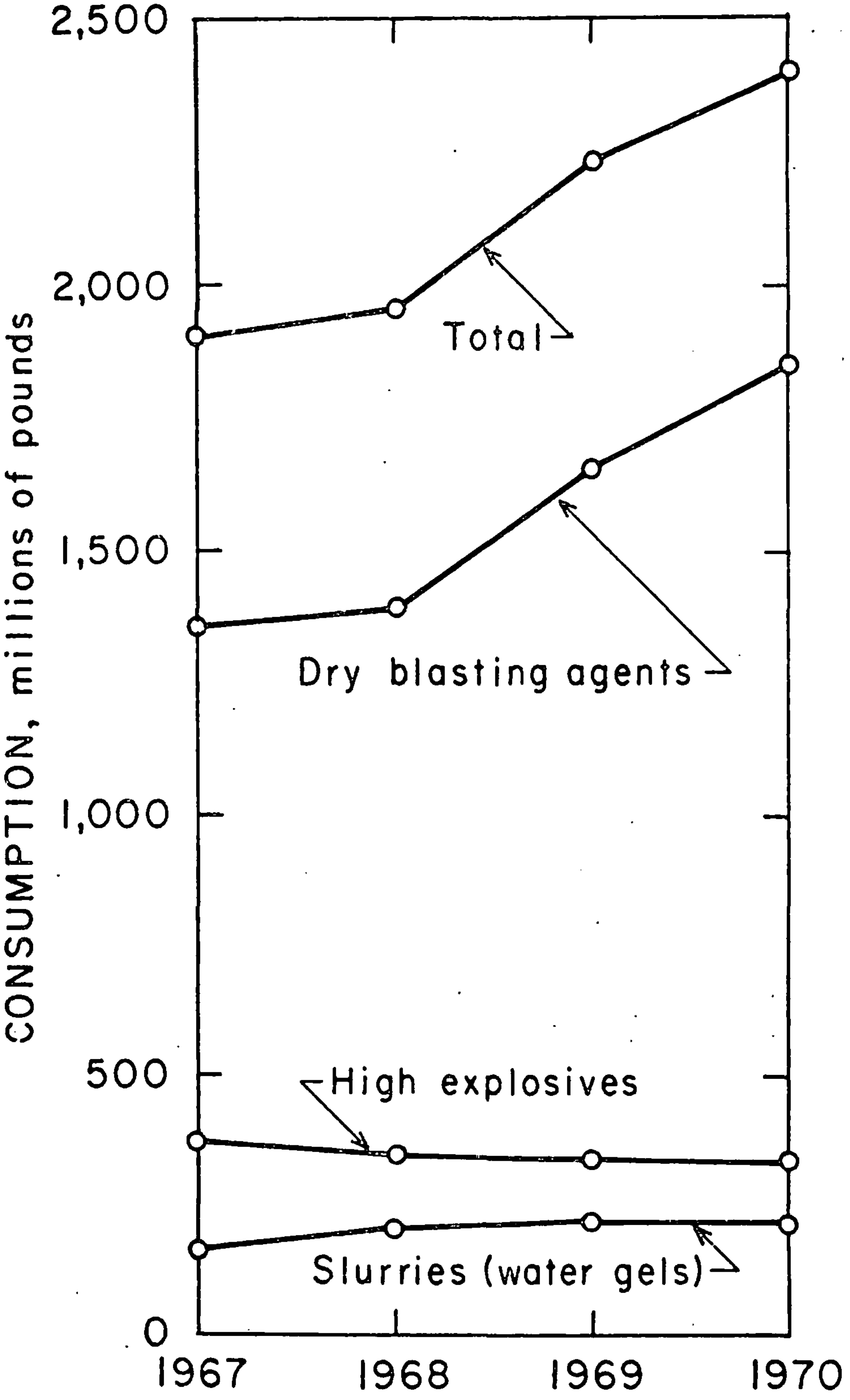


FIGURE 8.-Explosive Consumption Trends.

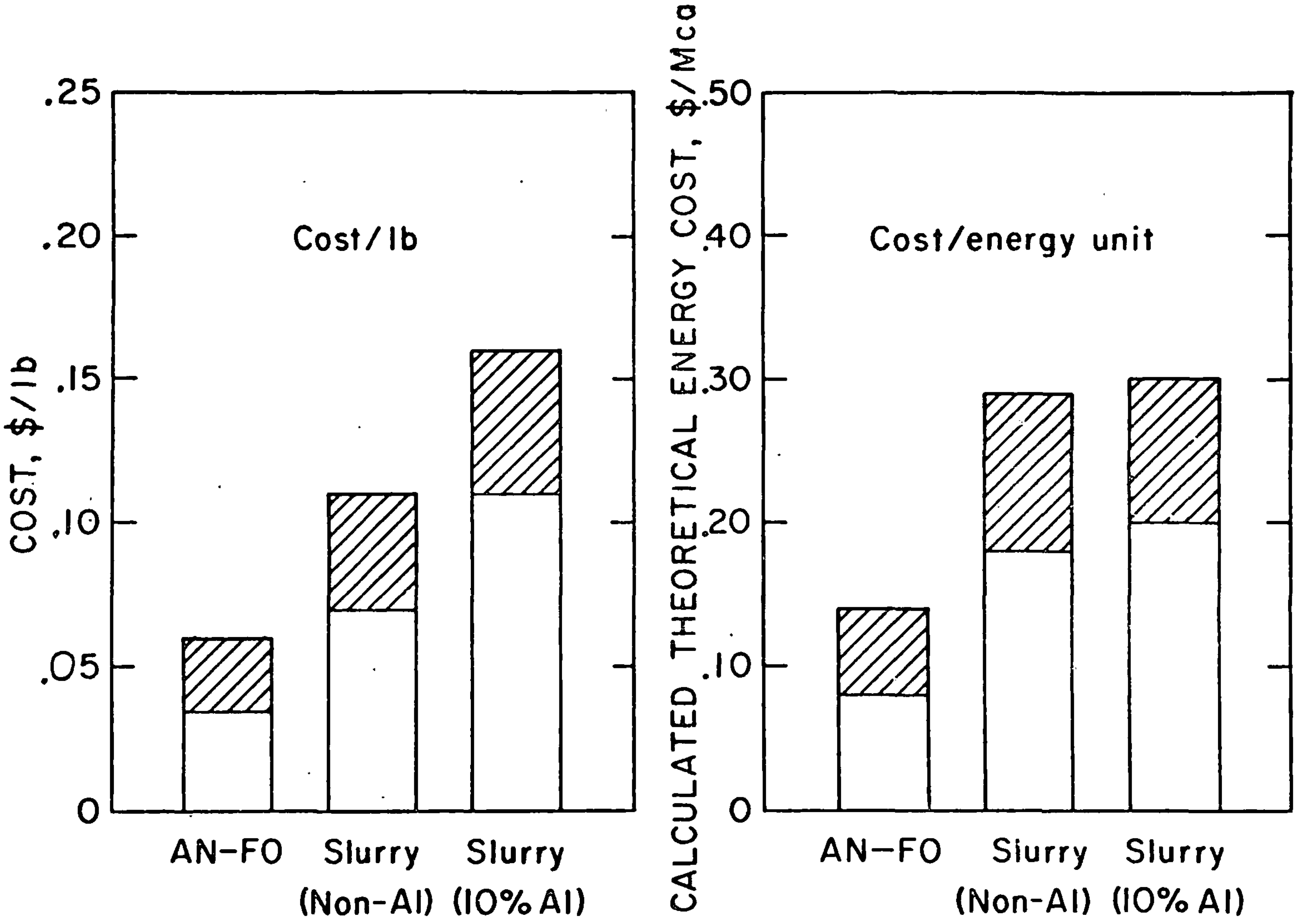


FIGURE 9.–Comparative Cost of Blasting Agents.

Charge diameter, inches	Product	Spacing, feet	Cost, cents/lb			Powder factor, lb/cu yd
			Drilling	Blasting	Total	
9	AN–FO	9	7.0	4.1	11.1	7.6
12¼	AN–FO	12¼	3.8	4.1	7.9	7.6
12¼	Non Al slurry	12¼	3.8	9.3	13.1	10.9
12¼	10% Al slurry	12¼	3.8	16.2	20.0	12.3
12¼	AN–FO	18¾	1.7	1.8	3.5	3.4
12¼	10% Al slurry	18¾	1.7	7.2	8.9	5.5

10 kton nuclear device (confined) 9.2 cents/lb

20 kton nuclear device (confined) 5.6 cents/lb

TABLE I.–In Situ Drilling and Blasting Costs.

IN SITU, 5(2), 103-131 (1981)

IN SITU GENERATION OF ACID DURING DUMP LEACH PRODUCTION OF COPPER

W. Joseph Schlitt and Jonathan S. Jackson
Hydrometallurgy Department
Kennecott Minerals Company
Research Center
P. O. Box 11248
Salt Lake City, Utah 84147

ABSTRACT

Generation of sulfuric acid while leaching copper from sulfidic material has both environmental and operational impact. The roles of acid in dump leach chemistry are defined and approaches to estimation of in situ acid generation are reviewed. These are generally indirect measurements and include an acid balance on leach solution streams, heat and oxygen balances on the dumps or heaps, mineral alteration, and leach process modeling. Of these, the acid and heat balance methods are most practical. Rates indicate that 5 to 10 times as much acid is generated in situ than would be added in acidifying cementation plant tails to prevent ferric iron precipitation in pipelines and on dump surfaces. Most of the acid generated in a dump is also consumed there. The bulk of the net generated acid is consumed in the cementation step.

INTRODUCTION

For many years systematic leaching of low-grade waste from open-pit mines has been an important source of copper. Copper sulfide waste dumps are commonly high in pyrite which, if oxidized, is a potential source of sulfuric acid. Such generation of sulfuric acid has a significant impact on various aspects of the overall leach operation. The purpose of this paper is to review the role of acid in dump leach chemistry, to identify approaches to estimating the magnitude of in situ acid generation in waste dumps, and to discuss the effects of this acid generation on the overall leach/recovery circuit.

DUMP LEACH CHEMISTRY

The basic physical and chemical processes occurring during the leaching of copper sulfides in heaps or dumps and in the copper recovery system are fairly well understood.[1-6] A diagrammatic representation of a typical dump leach-copper cementation operation is presented in Figure 1. Consider this dump as a unit operation. On a macroscale (see Figure 2), solubilization of sulfide minerals inside the dump requires that they be oxidized at least indirectly by oxygen in the air. This air generally flows upward through the dump by convection. Nonsulfide minerals are already oxidized and are simply dissolved by the acid in the leach solution. A host of reactions involving precipitation, secondary enrichment, hydrolysis, or ion exchange may also occur in solution or at the different mineral-solution interfaces. Soluble copper from both sulfide and nonsulfide sources is flushed from the dump by application of leach solution giving rise to countercurrent air/water flow through the dump. Pregnant solutions leaving the dumps typically are impounded and pumped to a cementation or solvent extraction plant for copper recovery. To avoid environmental problems and to conserve water, the barren solution is then returned to the dump surfaces to be used for additional leaching.

Typical microscopic phenomena are shown in Figure 2. Once a dump has been thoroughly wetted, individual rock fragments retain a surface film of leach water which may also fill both rock pores and interstitial voids. Oxygen from the gas phase dissolves in the liquid film and is consumed in the bacterially catalyzed oxidation of ferrous iron to the ferric state. Ferric iron is the direct oxidant for sulfides; it diffuses into the rock fragment, oxidizing both copper and iron sulfides and liberating Cu^{+2}, Fe^{+2}, SO_4^{-2}, S°, H^+, and heat. Reaction products diffuse away from reaction sites in response to the concentration and temperature gradients present. Copper ions diffusing inward may participate in secondary sulfide enrichment reactions; for example, chalcocite rimming of pyrite. Ions diffusing to rock fragment surfaces are available to be flushed from the dump and/or participate in other

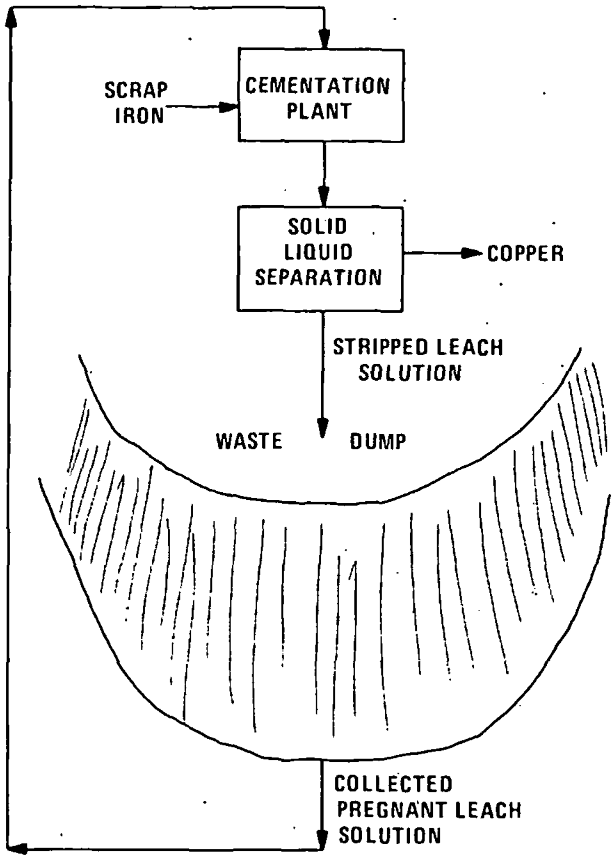


Figure 1. Simple Schematic of Dump Leach/Recovery Circuit

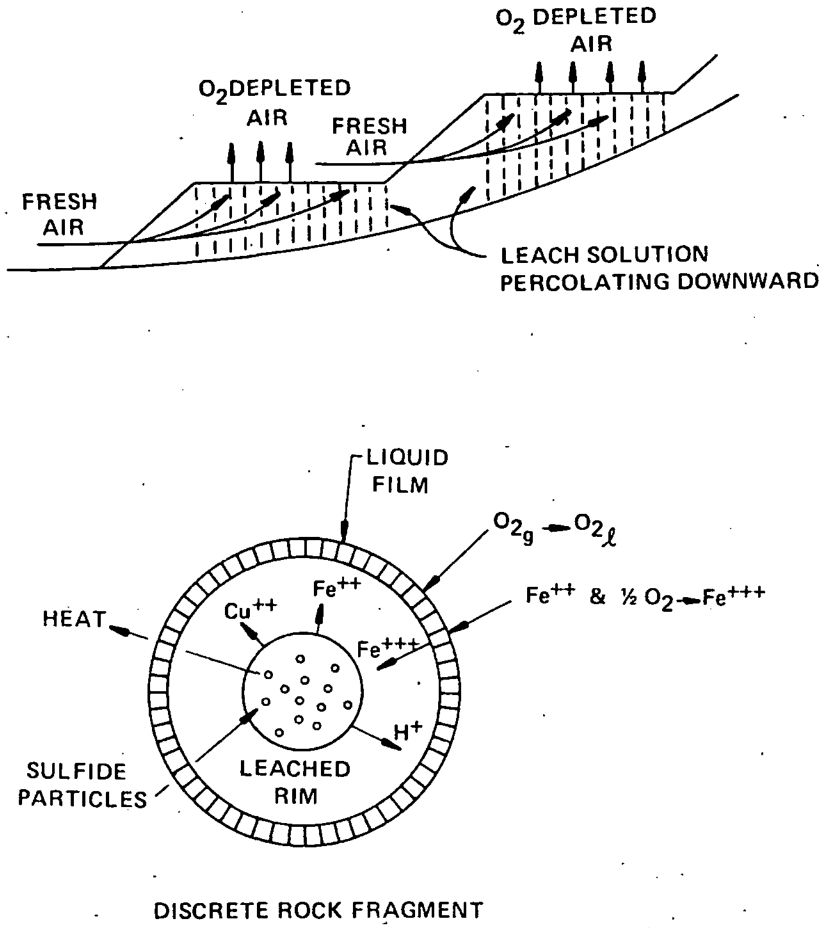


Figure 2. Cross Section of Typical Dump Showing Macro- and Micro-Scale Leaching Phenomenon

reactions. Evidence of these microscale phenomena can often be found in rock fragments which show a leached rim, a partially leached reaction zone, and an unreacted core sometimes showing secondary enrichment.

Repeated conversion of ferrous ion back to the ferric state as water percolates down through the dump is required to provide enough ferric ion to do all the leaching observed. Sulfide oxidation also liberates heat which warms both the fluid and solid phases within the dump and accelerates leach kinetics. Heating of the gas phase (air within the dump), together with oxygen depletion, provides the buoyancy causing natural convective air flow through the waste. Considerable heat can be carried out of a dump in the flowing gas and liquid phases and by conduction from dump surfaces. Some of these phenomena are depicted in Figure 2.

In the cementation step, copper in solution undergoes an exchange reaction with scrap iron. The result is metallic copper and ferrous iron in solution. Scrap iron is also consumed by reaction with ferric and hydrogen ions forming ferrous ion and evolving hydrogen gas. Similarly, in solvent extraction, copper in the aqueous phase exchanges with hydrogen ion in the organic extractant. This purifies and concentrates copper in the organic and generates an acidic raffinate for recycle to the leach step.

Stoichiometric equations representing waste leach and cementation chemistry are summarized in Table 1. For simplicity, complexed and intermediate species are not shown.

The chemical reactions in which acid participates can be separated into three general categories:

Category	Reactions*
1. Acid generating	3, 4, 5, 6, 8, 9
2. Acid consuming	2, 10, 11, 20
3. Buffering	12, 13, 14, 15, 16, 17

---

*Numbers refer to reactions listed in Table 1.

TABLE 1

Stoichiometric Equations Representing Waste Leach and Cementation Chemistry

1. $O_{2(gas)} \rightarrow O_{2(liq)}$

2. $4Fe^{+2}+O_{2(liq)}+4H^{+} \rightarrow 4Fe^{+3}+2H_2O$

3. $FeS_2+(14-6d)Fe^{+3}+(8-4d)H_2O \rightarrow (15-6d)Fe^{+2}+(2-d)SO_4^{-2}+dS^\circ +(16-8d)H^{+}$

4. $CuFeS_2+(16-6c)Fe^{+3}+(8-4c)H_2O \rightarrow Cu^{+2}+(17-6c)Fe^{+2}+cS^\circ+(2-c)SO_4^{-2} +(16-8c)H^{+}$

5. $Cu_aS+(6+2a-6b)Fe^{+3}+(4-4b)H_2O \rightarrow aCu^{+2}+(6+2a-6b)Fe^{+2}+bS^\circ +(1-b)SO_4^{-2}+(8-8b)H^{+}$

6. $S^\circ+6Fe^{+3}+4H_2O \rightarrow 6Fe^{+2}+SO_4^{-2}+8H^{+}$

7. $CuFeS_2+Cu^{+2} \rightarrow 2CuS+Fe^{+2}$

8. $5CuFeS_2+11Cu^{+2}+8H_2O \rightarrow 8Cu_2S+5Fe^{+2}+2SO_4^{-2}+16H^{+}$

9. $5FeS_2+14Cu^{+2}+12H_2O \rightarrow 7Cu_2S+5Fe^{+2}+14H^{+}+3SO_4^{-2}$

Table 1 (Continued)

10. $K_2(Mg_3Fe_3)(Si_6Al_2O_{20})(OH)_4+20H^{+} \rightarrow 2K^{+}+3Fe^{+2}+3Mg^{+2}+2Al^{+3} +6H_4SiO_4$

11. $CaCO_3+2H^{+}+SO_4^{-2}+H_2O \rightarrow CaSO_4+2H_2O+CO_2$

12. $Fe^{+3}+2H_2O \rightleftharpoons FeOOH+3H^{+}$

13. $K^{+}+3Fe^{+3}+2SO_4^{-2}+6H_2O \rightleftharpoons KFe_3(SO_4)_2(OH)_6+6H^{+}$

14. $3Fe^{+3}+2SO_4^{-2}+7H_2O \rightleftharpoons Fe_3(SO_4)_2(OH)_5 \cdot 2H_2O+5H^{+}$

15. $2Al^{+3}+2H_2O+2SO_4^{-2} \rightleftharpoons 2Al(OH)(SO_4)+2H^{+}$

16. $0.5Ca^{+2}+3.6Al^{+3}+0.7Mg^{+2}+7.7H_4SiO_4 \rightleftharpoons Ca_{0.5}(Al_{3.3}Mg_{0.7}) (Si_{7.7}Al_{0.3})O_{20}(OH)_4+13.2H^{+}+6.8H_2O$

17. $2Al^{+3}+2H_4SiO_4+H_2O \rightleftharpoons Al_2Si_2O_5(OH)_4+6H^{+}$

18. $Fe^\circ+Cu^{+2} \rightarrow Cu^\circ+Fe^{+2}$

19. $Fe^\circ+2Fe^{+3} \rightarrow 3Fe^{+2}$

20. $Fe^\circ+2H^{+} \rightarrow Fe^{+2}+H_2$

Of the acid-generating reactions, those involving the oxidation of sulfides (especially pyrite) are most important. The effect of the extent of sulfide oxidation on net acid generation is presented in Table 2. This assumes all ferric iron is generated within the dump. As the sulfide oxidation product shifts toward sulfate formation, acid production increases. However, even with eventual oxidation of all sulfide to sulfate, pyrite is the only net autogenous source of sulfuric acid in a dump. Usually the pyrite-to-copper sulfide molar ratio in porphyry copper wastes is in the range of 5 to 50 and sheer mass action dominates over relative reaction rates. As a result, sulfide oxidation as a whole is a net acid source.

Acid is consumed in a dump through net generation of ferric iron and acid attack of gangue minerals. Net generation of ferric iron (i.e., ferric iron not consumed by sulfide oxidation or removed from solution by precipitation) is a relatively small acid consumer (0.005-0.015 tpd H_2SO_4/gpm flow) when compared to consumption by acid attack of gangue minerals. Generally the princi-

TABLE 2

The Effect of the Extent of Sulfide Oxidation on Net Acid Generation

Sulfide Mineral	Sulfide Oxidation Product*	Total Moles H^+ Produced Per Mole Sulfide Oxidized	Moles H^+ Consumed by Required Fe^{+3} Production	Net Moles H^+ Produced Per Mole Sulfide Oxidized
Pyrite	$2S°$ (d=2)	0	2	-2
(FeS_2)	$S°+SO_4^{-2}$ (d=1)	8	8	0
	$2SO_4^{-2}$ (d=0)	16	14	2
Chalcopyrite	$2S°$(c=2)	0	4	-4
($CuFeS_2$)	$S°+SO_4^{-2}$(c=1)	8	10	-2
	$2SO_4^{-2}$(c=0)	16	16	0
Secondary	$S°$(b=2)	-8	2a-6	-2a-2
Sulfide	$1/2(S°+SO_4^{-2})$(b=1)	0	2a	-2a
(Cu_aS)	SO_4^{-2}(b=0)	8	2a+6	-2a+2

*See respective reaction stoichiometries in Table 1.

pal acid-consuming minerals in waste rock are calcite and micas (biotite, muscovite, chlorite, phlogopite, etc.). Acid neutralization by other rock-forming minerals is normally too slow to be of significance.

The waste dump acid-consuming and generating reactions discussed above are generally far from thermodynamic equilibrium. However, hydrolysis/precipitation reactions involving sulfide oxidation and acid/gangue reaction products may be near equilibrium, and as such, those reactions involving acid serve to "buffer" the leach solution pH at a relatively constant value. These reactions include Al and Fe hydrolysis and precipitation of goethite, jarosite, basic iron and aluminum sulfates, and kaolin.

In general, the net acid-generating/consuming nature of a particular waste dump is dependent on a balance between many factors. These include: the rates of sulfide oxidation, the relative amounts of sulfides reacting, sulfide oxidation to sulfate or elemental sulfur, the amount of acid consumed in producing the ferric oxidant, and the degree of acid neutralization by gangue minerals.

METHODS OF ESTIMATING ACID GENERATION IN THE INTERIOR OF A DUMP

There are several methods of estimating the rate and degree of in situ acid generation in a single dump or dump system. These methods are generally indirect and include an acid balance on leach solution streams, heat balance on the dumps or heaps, oxygen balance, mineral alteration, and leach process modeling. Leach solution stream acid balances provide an estimate of the net rate of acid generation by the dump or dump system; heat and oxygen balances--the gross rate of acid generation by sulfide oxidation. The degree of mineral alteration in a dump gives a qualitative indication of the extent to which acid attack has occurred and the amount of acid generated by sulfide oxidation over a given period of time. Mathematical leach models, depending on their scale and degree of complexity, may project the rate and cumulative amount of acid generated by sulfide oxidation over the useful limetime of a dump.

Estimation of the acid generating/consuming nature of a waste dump from bench scale tests on representative samples is difficult as important macro-scale phenomena cannot be duplicated on a small scale. However, Brynesteyn and Duncan[7] have developed a useful procedure for estimating the acid production potential of sulfur containing waste materials and waste waters. The maximum acid generating capacity of a waste material is first determined from its total sulfur content expressed as H_2SO_4. This maximum is attainable only for certain sulfides such as FeS_2 and requires accompanying hydrolysis of oxidation products. A -100 mesh sample is then titrated to pH 3.5 to compare acid consumption under nongenerating conditions to the maximum calculated generating capacity. If consumption exceeds calculated generating capacity, the material is judged a non-acid source. If consumption is less than calculated generating capacity, a biological oxidation test with T. ferrooxidans is run to estimate the acid production which would be expected in situ.

Acid Balance

Before discussing acid balances in dump leach solution streams, the acid content of such solutions must be defined and related to measurable parameters. According to the Bronsted-Lowry definition, an acid is a proton donor.[8] The principal acids (proton donors) in copper sulfide dump leaching solutions are thus sulfuric acid and water with very minor amounts of other acids such as phosphoric and hydrochloric. Protons liberated by sulfuric acid dissociation and water hydrolysis may be either free or complexed with other ions in solution.

The acid content of these solutions may be defined in terms of pH, free acid, hydrolytic acid, and total acid. pH is a measure of the hydrogen ion activity and only an indicator of relative acidity in typical salt-laden leach liquors like that shown in Table 3. Free acid is defined as the residual molarity of total SO_4^{-2} which must be charge balanced by H^+ after subtracting out all other cation equivalents

$$M_{free} = M_{(SO_4^{-2})} - \frac{1}{2} \sum \nu_i \left(M_i^{+\nu_i}\right) \qquad (1)$$

where M_i and $+\nu_i$ represent the individual cationic species and their corresponding charge respectively.

TABLE 3

Typical Pregnant Leach Solution Composition

Component	Concentration (gpl)
Cu	0.5
Fe (Total)	1.5
Fe^{+3}	1.3
Fe^{+2}	0.2
Al	5.0
Mg	7.0
Ca	0.6
SO_4	60.0
Cl	0.2
pH = 2.50	

timates of free acid content may be made by appropriate titra-ion techniques, the accuracy of which increases with increasing ree acid content in solution. However, in many dump leach iquors, free acid is quite low (0.01 mole/l).

Hydrolytic acid results from water dissociation or hydrolysis y cations (principally Fe^{+3} and Al^{+3}) and is defined as the total H^- molarity:

$$M_{\text{hydrolytic acid}} = M_{(OH^-)\text{free and complexed}} \qquad (2)$$

irect measurement of hydrolytic acid would be extremely difficult. otal acid in terms of proton molarity is defined as the sum of he free and hydrolytic acid:

$$M_{\text{total}} = 2\,M_{\text{free}} + M_{\text{hydrolytic}} \qquad (3)$$

r

$$M_{\text{total}} = 2M_{(SO_4^{-2})} + M_{(OH^-)} - \sum \nu_i (M_i^{+\nu_i}) \qquad (4)$$

quation (4) is essentially the solution charge balance with con-ributions of anions other than OH^- and SO_4^{-2} ignored.

Free, hydrolytic, and total acid contents of leaching solutions at various points in the leaching circuit may be estimated y measuring the pH and overall ion concentrations of these solu-ions, then theoretically calculating the ionic distribution of species, and finally applying equations (1), (2), and (4) dis-ussed above. With appropriate flow rate data, the net acid

generated or consumed by a dump can then be determined by the differences between the acid content of on- and off-flow solutions. This approach estimates only how much excess or net acid is produced by a dump, not the total actually generated and consumed; thus, the leach solution acid balance sets a lower limit on actual acid generation.

An acid balance has been completed on two dump systems (each consisting of several dumps) and the corresponding cementation plant. Individual dump on- and off-flow streams and cementation plant head and tail solutions were sampled, pH and chemical analyses run, and flow rates and temperatures measured.

The pH's, temperatures, chemical analyses, and average flow rates for the various streams are presented in Table 4. Leach solution pH's ranged from approximately 2.2 to 3.5. Chemical analyses and pH's were used as input for the initialization step of a computerized geochemical simulation system (FASTPATH).* It calculates and plots mineral creation and destruction and the molalities and activities of aqueous species in solution as the solution reacts with a mineral assemblage at a given temperature and pressure. The mathematical framework follows the theoretical approach to chemical processes developed by H. C. Helgeson and associates.[9] The initialization step solves the appropriate mass balance, charge balance and equilibrium expressions for the concentrations of all species assumed present in solution. Dissociation constants for assumed species are specified at 25°C and extended Debye-Huckel theory is used for activity coefficient determinations. Electrical neutrality is forced by balancing any charge balance error with sulfate ions, and the hydrogen ion activity is fixed by inputed room temperature (25°C) pH's. An example of the output of the initialization step for a cementation plant feed solution is presented in Table 5. Admittedly, with the near unity ionic strengths of typical dump leach solutions, Debye-

---

*FASTPATH is a Kennecott developed program.

Dump Leaching Circuit Solution Analyses, Temperature, and Flow Rates

Waste Dump or Stream	T (°C)	pH	Flow gpm	Flow l/sec	Cu	Fe(t)	Fe^{+2}	Al	Mg	Ca	SO_4
				Dump System 1							
1A	24.0	2.54	386	24.35	1.43	1.28	0.10	6.10	10.40	0.60	78.20
1B	34.5	2.38	7,300	460.51	0.62	1.90	1.58	5.75	8.85	0.58	68.70
1C	27.5	2.42	627	39.55	2.06	2.61	0.88	7.55	11.50	0.62	89.90
1D	30.3	2.13	6,292	396.92	0.54	1.48	0.05	5.60	8.15	0.57	64.80
1E	24.0	2.47	4,529	285.70	0.26	0.88	0.05	5.15	8.40	0.58	61.10
1F	15.0	3.42	1,172	73.93	0.43	1.39	1.18	4.95	8.10	0.55	58.50
				Dump System 2							
2A	24.0	2.32	6,377	402.28	0.11	1.40	0.51	4.55	7.20	0.55	54.40
2B	15.5	2.28	51	3.22	0.26	3.40	0.02	4.20	6.50	0.55	51.20
2C	18.0	2.39	284	17.92	3.73	1.15	0.03	7.75	6.40	0.55	73.50
2D	24.5	2.48	1,091	58.82	0.33	1.03	0.16	4.80	7.80	0.55	58.60
2E	26.0	2.40	7,817	493.12	0.26	1.55	1.18	4.84	7.50	0.55	60.40
2F	27.0	2.28	1,000	63.08	0.39	0.55	0.04	4.55	6.70	0.55	51.20
Cementation Plant Heads	27.0	2.45	36,926	2,329.4	0.48	1.44	0.70	5.18	8.08	0.56	61.90
Cementation Plant Tails	28.0	3.40	36,926	2,329.4	0.052	2.42	2.38	5.05	7.75	0.55	59.60

Solution Analyses (gpl): Cu, Fe(t), Fe^{+2}, Al, Mg, Ca, SO_4.

Huckel theory is inadequate. However, calculations are assumed to be at least semi-quantitative.

Distribution of species was used to calculate the free, hydrolytic, and total acid contents of the sampled streams. Using estimated or measured flow rates (Table 4), the total daily acid output of various streams and by difference, the net generation or consumption of acid for the individual units of the leaching operation were calculated. These data are presented in Table 6.

Acid balance results indicate the following:

1. All dumps in both systems are net acid producers except dump 1F. This dump contains about 8% limestone which is responsible for its net acid consumption.

2. Almost 40% more acid is produced by dump system 1 than dump system 2 (120 stpd vs 85 stpd). This is not surprising since system 2 is a much older, static (no waste currently being added) system with correspondingly lower rates of acid production.

3. The cementation plant is a net acid consumer through acid attack of scrap iron (Equation 19, Table 1). The indicated consumption of 164 stpd of sulfuric acid during cementation would result in consumption of approximately 90 stpd of scrap iron which is in good agreement with independent calculation of iron consumption.

Heat Balance

Assuming that the heat effects of acid-gangue reactions are offset by the heat effects of basic iron sulfate and jarosite precipitation, sulfide oxidation (including the heat generated in

TABLE 5

Distribution of Species in Cementation Plant Head Solution

Species	Molality	Activity Coefficient	Activity
g^{+2}	1.212×10^{-1}	0.3065	3.715×10^{-2}
l^{+3}	2.293×10^{-2}	0.0797	1.827×10^{-3}
a^{+2}	7.233×10^{-3}	0.2338	1.691×10^{-3}
e^{+2}	1.984×10^{-2}	0.2338	4.637×10^{-3}
u^{+}	5.860×10^{-12}	0.5862	3.435×10^{-12}
$l(OH)^{+2}$	2.598×10^{-4}	0.1539	3.998×10^{-5}
$l(OH)_4^{-}$	6.749×10^{-18}	0.6646	4.486×10^{-18}
aSO_4	6.467×10^{-3}	1.2220	7.903×10^{-3}
gSO_4	1.748×10^{-1}	1.2220	2.136×10^{-1}
eSO_4	1.415×10^{-2}	1.2220	1.730×10^{-2}
eSO_4^{+}	5.846×10^{-3}	0.7896	4.616×10^{-3}
$e(SO_4)_2^{-}$	1.805×10^{-2}	0.7896	1.425×10^{-2}
u^{+2}	6.589×10^{-3}	1.2220	8.052×10^{-3}
lSO_4^{+}	6.082×10^{-2}	0.7896	4.802×10^{-2}
$l(SO_4)_2^{-}$	1.299×10^{-1}	0.7896	1.025×10^{-1}
$l(OH)_2^{+}$	4.563×10^{-7}	0.6646	3.033×10^{-7}
$l_2(OH)_2^{+4}$	3.171×10^{-6}	0.0070	2.206×10^{-8}
$u_2(OH)_2^{+2}$	3.996×10^{-11}	0.1945	7.773×10^{-12}
$u_3(OH)_4^{+2}$	1.592×10^{-19}	0.2338	3.723×10^{-20}
$Fe(OH)_2^{+}$	7.449×10^{-7}	0.6646	4.951×10^{-7}
$Fe_2(OH)_2^{+4}$	1.403×10^{-4}	0.0070	9.758×10^{-7}
$Fe_3(OH)_4^{+5}$	1.970×10^{-5}	0.0004	8.017×10^{-9}
$FeHSO_4^{+}$	1.080×10^{-5}	0.6646	7.177×10^{-6}
$CuOH^{+}$	1.613×10^{-8}	0.6646	1.072×10^{-8}
HSO_4	1.371×10^{-2}	0.6646	9.112×10^{-3}
$MgOH^{+}$	5.240×10^{-11}	0.6646	3.483×10^{-11}
$FeOH^{+}$	8.467×10^{-9}	0.6646	5.628×10^{-9}
$CaOH^{+}$	1.071×10^{-13}	0.6646	7.115×10^{-14}
$FeOH^{+2}$	8.082×10^{-4}	0.1539	1.244×10^{-4}
$Cu(OH)_2$	1.525×10^{-13}	1.2220	1.853×10^{-13}
$Cu(OH)_3^{-}$	1.169×10^{-23}	0.6646	7.768×10^{-24}
$Fe(OH)_2$	5.744×10^{-16}	1.2220	7.020×10^{-16}
$Fe(OH)_3$	3.695×10^{-11}	1.2220	4.515×10^{-11}
$Fe(OH)_4$	1.668×10^{-15}	0.6646	1.108×10^{-15}
$Fe_2(OH)_3^{+3}$	6.644×10^{-6}	0.0344	2.288×10^{-7}
$AlSO_4OH$	5.948×10^{-5}	1.2220	7.269×10^{-5}
$FeSO_4OH$	8.076×10^{-5}	1.2220	9.870×10^{-5}
H^{+}	5.029×10^{-3}	0.8100	4.074×10^{-3}
OH^{-}	3.989 x 10-12	0.6153	2.455×10^{-12}

Table 5 (Continued)

Dump Leach Circuit Acid Generation and Consumption Rates

Predicted from Solution Acid Balance

Waste Dump	Acid Content (Moles H_2SO_4/l) Free	Hydrolytic	Total	Acid Flow as H_2SO_4 Tons/day	g/sec	Net H_2SO_4 Gen/Consumption Tons/day	g/sec
			Dump System 1				
1A	0.00634	0.00085	0.00719	1.64	17.17	1.42	14.90
1B	0.00984	0.00024	0.01008	43.37	455.30	39.27	412.36
1C	0.00971	0.00024	0.00995	3.68	38.58	3.32	34.89
1D	0.01631	0.00039	0.01669	61.89	649.70	58.36	612.72
1E	0.00722	0.00055	0.00767	20.46	214.80	17,92	188.16
1F	-0.00154	0.00244	0.00090	0.62	6.53	-0.03	-0.36
			Dump System 2				
2A	0.01002	0.00041	0.01043	39.18	411.30	35.61	373.86
2B	0.00916	0.00187	0.01103	0.33	3.48	0.01	0.06
2C	0.00850	0.00071	0.00921	1.54	16.19	1.38	14.52
2D	0.00687	0.00059	0.00745	4.79	50.29	4.18	43.87
2E	0.00880	0.00026	0.00906	41.74	438.20	37.36	392.22
2F	0.01100	0.000024	0.01125	6.63	69.60	6.07	63.72
Cementation Plant	-0.00070	0.00165	0.00095	20.67	217.0	-164.15	-1,723.62

producing the required ferric oxidant) is the only significant heat source in a leach dump. Heat is lost from a dump by two mechanisms--convection and conduction. Convective heat loss occurs through heating of gas and liquid phases flowing through the dump. Conductive heat loss occurs through dump surfaces.

Assuming short-term isothermicity, measured dump heat losses can be equated to the heat liberated by sulfide oxidation. With appropriate assumptions regarding relative sulfide oxidation rates, reaction products, and reaction enthalpies, the rate of acid generation by sulfide oxidation can be back calculated.

Of all heat loss mechanisms, convective heat loss by heating of leach solution is most easily estimated. This is done by measuring temperatures and flow rates of individual dump on- and off-flow streams. Respective temperature changes through the dump are then equated to the heat gained or lost by use of an appropriate heat capacity proportionality constant. Heat lost by removal of heated air and conduction through dump surfaces is much more difficult to determine and may account for as much as 50% of the total heat loss.

Temperatures and flow rates of on- and off-flow streams for the same two dump systems are summarized in Table 7. On-flow temperatures were measured at pond surfaces and off-flow temperatures at dump toes. Net convective heat losses to flowing solutions were calculated using the indicated temperature differences and a leach water heat capacity and density of 1.0 cal/gm C° and 1 gm/cc, respectively. As previously stated, these heat losses account for only a part of the total heat generated. However, when related to sulfide oxidation enthalpies, heat losses to solution do place a lower limit on the estimate of in situ acid generation by sulfide oxidation. Assuming principal sulfides of FeS_2 and $CuFeS_2$ and complete oxidation of sulfide to sulfate, the

TABLE 7

Dump Leach Circuit Solution Temperature

Waste Dump	Ambient Temp. (°C)	On-flow Temp. (°C)	Off-flow Temp. (°C)
	Dump System 1		
1A	3	18.0	33.0
1B	3	19.0	34.5
1C	3	18.0	27.5
1D	3	12.0	33.2
1E	3	23.5	32.5
1F	3	24.0	25.0
	Dump System 2		
2A	-1	21.5	24.0
2B	-1	11.0	15.5
2C	0	12.8	18.0
2D	-1	13.0	24.5
2E	-3	16.0	27.0
2F	-3	18.0	27.0

ollowing equation relates measured heat loss and reaction nthalpies to acid generation rates.

$$R_{H_2SO_4} = \frac{\frac{+\Delta H}{\Delta t}\text{ dump }(9.325)}{\Delta Hpy + \Delta Hcpy/(FPY)} \qquad (5)$$

here

$R_{H_2SO_4}$ = stpd sulfuric acid generated

$\frac{\Delta H}{\Delta t}$ dump = heat gain by leach solution in Kcal/sec

ΔHpy = net enthalpy for pyrite oxidation including heat released in providing required ferric oxidant (-335 Kcal/mole)

$\Delta Hcpy$ = net enthalpy for chalcopyrite oxidation including heat released in providing required ferric oxidant (-384 Kcal/mole)

FPY = moles pyrite oxidized per mole chalcopyrite oxidized

9.325 = units conversion factor

Calculated heat and acid generation rates for various dumps are presented in Table 8 for FPY values of 5, 10, and 50. As shown, the acid generation rate becomes less sensitive to FPY as FPY increases. The net acid generation rates determined from leach solution acid balance are repeated in Table 8 for comparison. For the most part, rates determined by the two methods are within an order of magnitude of each other. Rates calculated from acid balances are generally less than those determined from the heat balance since the former represent a net acid generation rate for the dump and the latter represent a gross lower limit on the rate of acid generation by sulfide oxidation.

Oxygen Balance

The rate of oxygen depletion from air flowing through a dump is a direct indication of the net sulfide oxidation rate. If the oxygen consumption rate is known, assumptions with respect to oxygen utilization efficiency, oxygen partitioning between sulfides, and sulfide oxidation products permit calculation of the rate of acid generation as follows:

$$R_{H_2SO_4} = \frac{(R_{O_2})(O_2\%\ EFF)(1.079 \times 10^{-6})}{3.5 + 4(FPY)} \qquad (6)$$

where

R_{O_2} = oxygen consumption rate, moles O_2/day

$O_2\%$ EFF = percentage of consumed oxygen actually contributing to sulfide oxidation

TABLE 8

Dump Leach Circuit Acid Generation and Consumption Rates Predicted from Heat Balance

Waste Dump	ΔH/Δt (Kcal/sec)	H_2SO_4 Generation Rate (stpd)			
		FPY = 5	FPY = 10	FPY = 50	Acid Balance
		Dump System 1			
1A	365.25	8.3	9.1	9.9	1.42
1B	7,137.91	161.7	178.7	194.2	39.27
1C	375.73	8.5	9.4	10.2	3.32
1D	8,414.70	190.6	210.7	229.0	58.36
1E	2,571.31	58.2	64.4	70.0	17.92
1F	73.93	1.7	1.9	2.0	-0.03
		Dump System 2			
2A	1,005.70	22.8	25.2	27.4	35.61
2B	14.49	0.3	0.4	0.4	0.01
2C	94.08	2.1	2.4	2.6	1.38
2D	791.43	17.9	19.8	21.5	4.18
2E	5,424.32	122.8	135.8	147.6	37.36
2F	567.72	12.9	14.2	15.5	6.07

Equation (6) assumes major sulfides of pyrite and chalcopyrite and complete oxidation of sulfide to sulfate. This relation is shown graphically in Figure 3 in terms of required standard air flow at 20% oxygen consumption and 90% utilization efficiency and a range of typical FPY's. Note that air flows on the order of 10,000 to 100,000 sfcm are required to account for the acid generation rates indicated by acid and heat balances previously presented.

A leach dump, of course, is not a system where the total flow and oxygen concentration of gases entering and leaving dump surfaces can be easily measured. Measurements can be made of the flow rate and oxygen concentration of gas exhausting from bore-

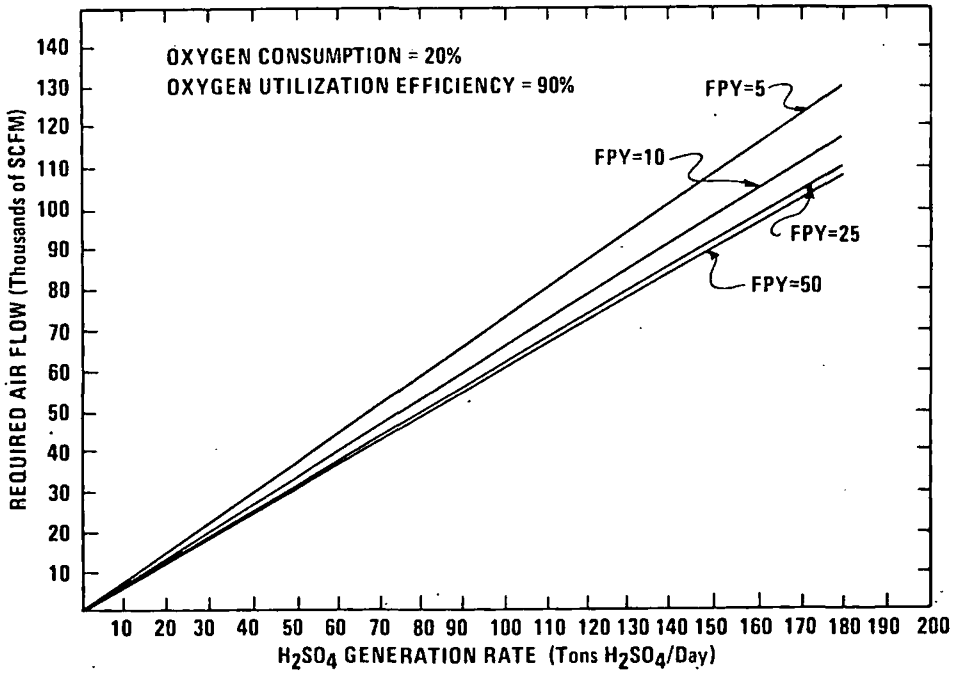


Figure 3. Required Air Flow Vs Acid Generation Rate for Various Values of FPY (Moles Pyrite Oxidized/Mole Chalcopyrite Oxidized)

holes. It may also be possible to capture gas exiting over relatively large dump surfaces under plastic sheeting with a single outlet for gas sampling and flow measurement. However, in light of the heterogeneity of dumps, such measurements probably cannot be extrapolated to the dump as a whole. Such measurement problems limit the usefulness of this technique for estimating in situ acid generation rates.

Formation of Alteration Products

The type and relative abundance of alteration products (e.g., jarosite, goethite, gypsum, kaolin, and montmorillonite) within a dump can provide at least a qualitative indication of the degree of acid attack which has occurred throughout the leaching period. These observations can be compared with the amount of acid generated by sulfide oxidation as determined by examination of residual sulfide in leached rims of remaining rock fragments. However, detailed chemical and mineralogical studies of leached waste are not often possible due to the expense of the extensive core drilling necessary to obtain a representative sample, and the number of individual chemical and mineralogical determinations which must be made.

Mathematical Modeling

Efforts have been made to develop a model of dump leaching which correlates measured dump physical, chemical, and mineralogical parameters through appropriate mass and energy balance equations to predict the dump responses to leaching.[1-3] One factor identified by such a model would be the amount of acid generated by pyrite oxidation. However, to date such models, though adequately pointing out the relative importance of dump operational parameters, have proved cumbersome and nonpredictive. As such, they are of little use in estimating in situ acid generation.

LEACH CIRCUIT IMPLICATIONS

Prior discussion has centered on the generation of sulfuric acid during the leaching of copper from sulfidic mine waste. However, methods for estimating acid generation rates within the dump have some value from an operational point of view only if they guide leach practices. Two cases must be considered since dumps may either be net acid generators or consumers.

Dumps with Net Acid Consumption

There may be several reasons why dumps are net acid consumers. These include a low pyrite content, a low sulfide oxidation rate or, frequently, the presence of acid-consuming gangue. Under any of these conditions, areas with locally high pH levels are likely to occur within these dumps. This can cause solubilized copper to precipitate and may also cause hydrolysis of the ferric ion needed to leach the sulfides; the result is low copper production.

This situation may correct itself with time as acid-consuming gangue is neutralized. External addition of acid to recycled leach solution may accelerate this neutralization process. Stimulation of sulfide oxidation can also increase acid generation rates and dump temperatures, and this may improve release of copper from the dump. However, an appropriate experimental test program would be needed to select the best approach since the cause of low acid generation and poor copper recovery may be complicated and site specific.

Dumps with Net Acid Generation

Fortunately, dumps that exhibit net acid consumption are not common, and attention has focused primarily on dumps which are net acid generators. However, even in acid-generating systems, locally high pH levels may cause problems. One place this can occur is in cementation plant effluent lines or on dump surfaces. Since the bulk of the acid in the pregnant leach solution is neutralized by reaction with scrap iron, the pH of the recycled barren solution is typically between 3.1 and 3.5. Although essentially all iron is in the ferrous state at this point, any oxidation in pipelines or on the dump surface will generate ferric iron. At the high pH levels, ferric iron is thermodynamically unstable and will precipitate as basic iron sulfate. If this occurs in pipelines, scale can build up and restrict flow capacity. If it occurs on the dumps, surface permeability will be reduced, restricting leach solution percolation and the countercurrent flow of air through the waste.

The buildup of iron salts can be controlled in several ways. Acidification of cementation plant effluent to pH 2.4 or 2.5 will effectively minimize scale formation in pipelines. Mechanical scouring of the lines with "chain balls" or the like on a periodic basis is also effective. Newer pipe materials such as high-density polyethylene, as well as stainless steel, also seem to restrict scale buildup. The choice is a matter of economics.

When solvent extraction, rather than cementation, is used for copper recovery, salt buildup will not be as serious a problem. Since leach solution is acidified during copper transfer to the organic phase, iron hydrolysis is prevented or reduced. Furthermore, no iron will be added to solution during copper recovery. Thus, iron contents in leach systems using solvent extraction are generally lower than those where cementation is used. This also minimizes iron salt buildup.

On dump surfaces, acidification again provides one way of controlling iron salt buildup. However, care must be exercised since over-acidification can accelerate rock decrepitation and create the same problem as iron salt deposition. Mechanical maintenance of dump surfaces by scraping, ripping, etc. offers an alternative to acidification. At present, economics may favor the mechanical procedures. These are especially effective when the leach solution application rate is held below the water acceptance rate at the surface. This minimizes ponding and thus the deposition of iron salts.[10]

Other considerations also favor limited use of acid for dumps that are already net acid generators. Data presented above indicate that 1 to 2 x 10^{-3} lb acid/ton waste are generated in situ each day in dumps with an FPY as low as 10. Even if recycled leach solution were acidified to pH 2.1, the amount of acid added in this way is only about 2 x 10^{-4} lb acid/ton waste/day. Furthermore, this acid is added in a concentrated manner, rather than being generated throughout the waste. Thus, acidification of leach solution will only affect the top of the dump. Both column tests and modeling indicate that this affected zone is approximately 10 ft thick for leach solution application rates of approximately 0.5 gal./ft^2/hr. After percolation to this depth, much of the acid is neutralized and the pH of the solution is that set by the "buffering" capacity of the waste. Examination of leached dumps seems to confirm this. On this basis, acidification of leach solution may be used to avoid surface salt deposition but will have little impact on internal dump chemistry.

SUMMARY AND CONCLUSIONS

Methods for estimating in situ sulfuric acid generation rates based on the physics and chemistry of pyritic copper sulfide waste leaching have been described. These include: acid balance on leach solution streams, heat balance, oxygen balance, mineral alteration, and leach process modeling. Of these, the acid and heat balance methods are most practical. The acid balance approach provides an estimate of the net rate of acid generation by the dump or dump system. The heat balance approach provides a lower limit on the gross rate of acid generation by pyrite oxidation.

In general, pyritic copper sulfide waste dumps which are producing copper and do not contain excessive acid-consuming gangue are net sulfuric acid sources in the leach-cementation circuit. This acid is produced only through pyrite oxidation. The bulk of the net generated acid is consumed in the cementation step. Data from two large dump systems indicate net rates on the order of 10^{-4} lb H_2SO_4/ton waste/day and gross rates of more than 10^{-3} lb H_2SO_4/ton waste/day. Thus, most of the acid generated in the dump is also consumed there.

Acid generation rates are useful in deciding between dump leach operational alternatives involving sulfuric acid. Rates indicate that 5 to 10 times as much acid is generated in situ than would be added in acidifying cementation plant tails to prevent ferric iron precipitation in pipelines and on dump surfaces.

ACKNOWLEDGEMENT

The authors wish to thank Kennecott Minerals Company management for permission to publish this paper. The authors also acknowledge the contribution of J. A. Apps who developed some of the approaches to estimating acid generation during his employment with Kennecott.

REFERENCES

1. Cathles, L. M., and Apps, J. A., "A Model of the Dump Leaching Process that Incorporates Oxygen Balance, Heat Balance, and Air Convection," Metall. Trans. B, Vol. 6B, No. 4, p. 617, 1975.

2. Cathles, L. M. and Schlitt, W. J., "A Model of the Dump Leaching Process that Incorporates Oxygen Balance, Heat Balance and Two-Dimensional Air Convection," in Leaching and Recovering Copper from As-Mined Materials (Schlitt, W. J., Editor) AIME, p. 9, 1980.

3. Jackson, J. S., Schlitt, W. J., and McMillan, B. B., "Forecasting Copper Production from Dump Leaching," Soc. Mining Eng. of AIME Trans., Vol. 266, p. 2009, 1979.

4. Nadkarni, R. M., et al., "A Kinetic Study of Copper Precipitation on Iron - Parts I and II," Trans. AIME, Vol 239, pp. 581 and 1066, 1967.

5. Schlitt, W. J. and Richards, K. J., "Chemical Aspects of Copper Cementation" in Solution Mining Symposium (Aplan, F. F., McKinney, W. A., and Pernickele, A. D., Editors) AIME, p. 401, 1974.

6. Warwick, C. C. I., Scuffham, J. B., and Lott, J. B., "Solvent Extraction - Today's Exciting Process for Copper and Other Metals," World Mining, Vol. 22, p. 38, 1970.

7. Bruynesteyn, A. and Duncan, D. W., "Determination of Acid Production Potential of Waste Materials," 1979 AIME Annual Meeting, The Metallurgical Society, Preprint A-79-29.

8. Wood, J. H., Keenan, C. W., and Bull, W. E., Fundamentals of College Chemistry (2nd Ed.), Harper & Row, p. 85ff, 1968.

9. Helgeson, H. C., Brown, T. H., Nigrini, A., and Jones, T., "Calculation of Mass Transfer in Geochemical Processes Involving Aqueous Solutions," Geochim. Cosmochim. Acta, Vol. 34, No. 5, p. 569, 1970.

10. Jackson, J. S. and Ream, B. P., "Solution Management in Dump Leaching" in Leaching and Recovering Copper from As-Mined Materials (Schlitt, W. J., Editor) AIME, p. 79, 1980.

IN SITU LEACHING BIBLIOGRAPHY

PART I

COMPILED BY

WILLIAM C. LARSON, GEOLOGIST

DENNIS V. DANDREA, GEOPHYSICIST

TWIN CITIES MINING RESEARCH CENTER
U.S. BUREAU OF MINES

COPIES OF BUREAU OF MINES REPORTS MAY BE ORDERED FROM:

DIVISION OF PRODUCTION AND DISTRIBUTION
U.S. BUREAU OF MINES
4800 FORBES AVE.
PITTSBURGH, PENNSYLVANIA 15213

THE AUTHORS ARE AVAILABLE FOR CONSULTATION AT, AND COPIES OF THIS BIBLIOGRAPHY MAY BE REQUESTED FROM:

TWIN CITIES MINING RESEARCH CENTER
P.O. BOX 1660
TWIN CITIES, MINNESOTA 55111

TELEPHONE: (612) 725-4500

18 NOVEMBER 1976

1. ANDERSON, A. E., AND F. K. CAMERON. RECOVERY OF COPPER BY LEACHING. OHIO COPPER COMPANY OF UTAH. TRANS. AIME, V. 73, 1926, PP. 31-57.

2. ANDERSON, J. S., AND M. I. RITCHIE. SOLUTION MINING OF URANIUM. MIN. CONG. J., V. 54, NO. 1, JANUARY 1968, PP. 20-26.

3. ANDERSON, M. L. IMPROVED PRODUCTIVITY FROM IN SITU LEACHING. PROC. ON PRODUCTIVITY IN MINING, UNIV. MISSOURI--ROLLA, ROLLA, MO., MAY 13-15, 1974, ED. BY J. J. SCOTT, PP. 82-83.

4. ANDERSON, M. L. (ASSIGNED TO MOBIL OIL CORPORATION, A CORP. OF NEW YORK). METHODS OF SELECTIVELY IMPROVING THE FLUID COMMUNICATION OF EARTH FORMATIONS. U.S. PAT. 3,565,173, FEB. 23, 1971, 6 PP.

5. ARNOLD, W. D., AND D. J. CROUSE. RADIOACTIVE CONTAMINATION OF COPPER RECOVERED FROM ORE FRACTURED WITH NUCLEAR EXPLOSIONS. OAK RIDGE NATIONAL LABORATORY, REPT. ORNL-4677, SEPTEMBER 1971, 55 PP.

6. AUSTIN, W. L. PROCESS OF EXTRACTING COPPER FROM ORE. U.S. PAT. 975,106, NOV. 8, 1910, 2 PP.

7. BALLARD, J. K. SOLUTION MINING. MINING ENGINEERING, V. 23, NO. 2, FEBRUARY 1971, P. 109.

8. BECHERKIN, S. G., V. G. BAKHUROV, AND I. K. LUTSENKO. UNDERGROUND LEACHING OF URANIUM FROM LOW-GRADE IN SITU ORES. TRANS. ATOMIC ENERGY, V. 21, NO. 2, FEBRUARY 1968, PP. 128-133.

9. BHAPPU, R. B. ECONOMIC EVALUATION OF IN PLACE LEACHING AND SOLUTION MINING SITUATIONS. PRES. AT A SHORT COURSE IN IN-PLACE LEACHING AND SOLUTION MINING, THE MACKAY SCHOOL OF MINES, UNIV. NEVADA--RENO, RENO, NEV., NOV. 10-14, 1975, 22 PP.

10. BHAPPU, R. B. IN SITU EXTRACTION AND LEACHING TECHNOLOGY. PRES. AT INTER-REGIONAL SEMINAR ON THE ECONOMICS OF MINERAL ENGINEERING, ANKARA, TURKEY, APR. 5-16, 1976, 39 PP.

11. BHAPPU, R. B. IN SITU MINING TECHNOLOGY - PRACTICAL AND ECONOMIC ASPECTS. PRES. AT THE TURKISH MINING INSTITUTE, ANKARA, TURKEY, APR. 5-16, 1976, 29 PP.

12. BRAITHWAITE, J. W. AND M. E. WADSWORTH. OXIDATION OF CHALCOPYRITE UNDER SIMULATED CONDITIONS OF DEEP SOLUTION MINING. PROC. INTERNAT. SYMP. ON COPPER EXTRACTION AND REFINING, MET. SOC., AIME, LAS VEGAS, NEV., FEB. 22-26, 1976 (PUB. AS EXTRACTIVE METALLURGY OF COPPER--HYDROMETALLURGY AND ELECTROWINNING, ED. BY J. C. YANNOPOULOS AND J. C. AGARWAL). PORT CITY PRESS, BALTIMORE, MD., V. II, 1976, PP. 752-775.

13. BRAUN, R. L., AND A. E. LEWIS. NUCLEAR SOLUTION MINING: PART VIII - OXYGEN DISTRIBUTION. LAWRENCE RADIATION LABORATORY, REPT. UCID-16008, MARCH 1972, 32 PP.

14. BRAUN, R. L., A. E. LEWIS, AND M. E. WADSWORTH. IN-PLACE LEACHING OF PRIMARY SULFIDE ORES: LABORATORY LEACHING DATA AND KINETICS MODEL. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 295-323.

15. BRAUN, R. L., AND R. G. MALLON. COMBINED LEACH-CIRCULATION CALCULATION FOR PREDICTING IN-SITU COPPER LEACHING OF PRIMARY SULFIDE ORE. TRANS. SME, AIME, V. 258ICTING IN-SITU COPPER LEACHING OF PRIMARY SULFIDE ORE. TRANS. SOC. MIN. ENG., AIME, V. 258, NO. 2, JUNE 1975, PP. 103-110.

16. BROWN, M. R. KERR GETS FIRST SALES CONTRACT FOR URANIUM FOR AGNEW LAKE MINE PRODUCTION. NORTHERN MINER, V. 62, NO. 18, JULY 15, 1976, P. 1.

17. BRUYNESTEYN, A., AND D. W. DUNCAN. EFFECT OF PARTICLE SIZE ON THE MICROBIOLOGICAL LEACHING OF CHALCOPYRITE BEARING ORE. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 324-337.

18. CARNAHAN, T. G., AND H. J. HEINEN (ASSIGNED TO THE UNITED STATES OF AMERICA AS REPRESENTED BY THE SECRETARY OF THE INTERIOR, WASHINGTON, D.C.). CHEMICAL MINING OF COPPER PORPHYRY ORES. U.S. PAT. 3,912,330, OCT. 14, 1974.

19. CARNAHAN, T. G., AND H. J. HEINEN. SIMULATED IN SITU LEACHING OF COPPER FROM A PORPHYRY ORE. BUMINES TPR 69, MAY 1973, 11 PP.

20. CARNINE, G. T., AND L. R. DARBEE (ASSIGNED TO FMC CORPORATION, NEW YORK, N.Y., A CORP. OF DEL.). STABILIZATION OF HYDROGEN PEROXIDE. U.S. PAT. 3,383,174, MAY 14, 1968, 8 PP.

21. CATANACH, C. B. DEVELOPMENT AND IN PLACE LEACHING OF MOUNTAIN CITY CHALCOCITE ORE BODY. PROC. INTERNAT. SYMP. ON COPPER EXTRACTION AND REFINING, MET. SOC., AIME, LAS VEGAS, NEV., FEB. 22-26, 1976 (PUB. AS EXTRACTIVE METALLURGY OF COPPER--HYDROMETALLURGY AND ELECTROWINNING, ED. BY J. C. YANNOPOULOS AND J. C. AGARWAL). PORT CITY PRESS, BALTIMORE, MD., V. II, 1976, PP. 849-872.

22. CLARK, D. A. STATE-OF-THE-ART. URANIUM MINING, MILLING, AND REFINING INDUSTRY. U.S. ENVIRONMENTAL PROTECTION AGENCY REPT. EPA-660/2-74-038, JUNE 1974, 113 PP.

23. CRAWFORD, E. DEVELOPERS EYE TEXAS POTENTIAL FOR IN SITU URANIUM LEACHING. ENG. AND MIN. J., V. 176, NO. 7, JULY 1975, PP. 81-82.

24. DANDREA. D. V. ZONIA MINE, NOTES FROM AUG. 16, 1973, VISIT TO ZONIA MINE, ARIZONA, 1973, 3 PP.; AVAILABLE FOR CONSULTATION AT BUREAU OF MINES, TWIN CITIES MINING RESEARCH CENTER, MINNEAPOLIS, MINNESOTA.

25. DANDREA. D. V., R. A. DICK, R. C. STECKLEY, AND W. C. LARSON. A FRAGMENTATION EXPERIMENT FOR IN SITU EXTRACTION. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEX., FEB. 25-27, 1974, PP. 148-161.

26. DANDREA, D. V., AND W. C. LARSON. PHASE I IN SITU LEACH TEST AT THE EMERALD ISLE MINE. BUMINES INTERNAL REPORT, 1975, 77 PP.; AVAILABLE FOR CONSULTATION AT BUREAU OF MINES, TWIN CITIES MINING RESEARCH CENTER, MINNEAPOLIS, MINNESOTA.

27. DANDREA. D. V., AND W. C. LARSON. PHASE II IN SITU LEACH TEST AT THE EMERALD ISLE MINE. BUMINES INTERNAL REPORT, 1975, 24 PP.; AVAILABLE FOR CONSULTATION AT BUREAU OF MINES, TWIN CITIES MINING RESEARCH CENTER, MINNEAPOLIS, MINNESOTA.

28. DANDREA. D. V., W. C. LARSON, P. G. CHAMBERLAIN, AND J. J. OLSON. SOME CONSIDERATIONS IN THE DESIGN OF BLASTS FOR IN SITU COPPER LEACHING. PRES. AT 17TH U.S. SYMP. ON ROCK MECH., SNOWBIRD, UTAH, AUG. 25-27, 1976, 15 PP.

29. DANDREA. D. V., W. C. LARSON, L. R. FLETCHER, P. G. CHAMBERLAIN, AND W. H. ENGELMANN. IN SITU LEACHING RESEARCH IN A COPPER DEPOSIT AT THE EMERALD ISLE MINE. BUMINES RI IN PREPARATION, 1976, 35 PP.

30. DANDREA. D. V., AND S. M. RUNKE. IN SITU COPPER LEACHING RESEARCH AT THE EMERALD ISLE MINE. PROC. JOINT MEETING OF THEMINING AND METALLURGICAL INSTITUTE OF JAPAN AND THE AMERICAN INSTITUTE OF MINING, METALLURGICAL, AND PETROLEUM ENGINEERS, DENVER, COLO., SEP. 1-3, 1976 (PUB. AS WORLD MINING AND METALS TECHNOLOGY, ED. BY A. WEISS). V. 1, 1976, PP. 409-419.

31. DEW, J. N., AND W. L. MARTIN (ASSIGNED TO CONTINENTAL OIL COMPANY, PONCA CITY, OKLA., A CORPORATION OF DELAWARE). IN SITU ROASTING AND LEACHING OF URANIUM ORES. U.S. PAT. 2,954,218, SEP. 27, 1960, 3 PP.

32. DICK, R. A. IN SITU FRAGMENTATION FOR SOLUTION MINING - A RESEARCH NEED. PRES. AT THE SECOND INTERNAT. SYMP. ON DRILLING AND BLASTING, PHOENIX, ARIZONA, FEB. 12-16, 1973.; AVAILABLE FOR CONSULTATION AT BUREAU OF MINES, TWIN CITIES MINING RESEARCH CENTER, MINNEAPOLIS, MINNESOTA.

33. DRAVO CORPORATION. SOLUTION MINING. SECTION 6.6.2 IN ANALYSIS OF LARGE SCALE NON-COAL UNDERGROUND MINING METHODS, USBM CONTRACT REPORT S0122059, JANUARY 1974, PP. 455-464.

34. DUDAS, L., H. MASS, AND R. BHAPPU. ROLE OF MINERALOGY IN HEAP AND IN SITU LEACHING OF COPPER ORES. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 193-209.

35. DUNCAN, D. W., AND A. BRUYNESTEYN. ENHANCING BACTERIAL ACTIVITY IN A URANIUM MINE. CAN. MIN. AND MET. BULL., MAY 1971, PP. 32-36.

36. DUNCAN, D. W., AND A. BRUYNESTEYN. MICROBIOLOGICAL LEACHING OF URANIUM. PROC. 1970 URANIUM SYMP., SOCORRO, N. MEX., COMP. BY R. J. ROMAN, 1970, PP. 55-61.

37. DUNCAN, D. W., C. C. WALDEN, P. C. TRUSSELL, AND E. A. LOWE. RECENT ADVANCES IN THE MICROBIOLOGICAL LEACHING OF SULFIDES. TRANS. SOC. MIN. ENG., V. 238, JUNE 1967, PP. 122-128.

38. ENGINEERING AND MINING JOURNAL. AEC AND KCC WILL JOINTLY STUDY POTENTIAL OF NUCLEAR BLASTING TO MINE COPPER ORE. V. 174, NO. 4, APRIL 1973, PP. 26-30.

39. ENGINEERING AND MINING JOURNAL. ASARCO AND DOW CHEMICAL TO LEACH DEEP COPPER OREBODY IN SITU IN ARIZONA. V. 173, NO. 6, JUNE 1972, P. 19.

40. ENGINEERING AND MINING JOURNAL. COPPER LEACHING WITH CYANIDE - A REVIEW OF FIVE INVENTIONS. V. 168, NO. 9, SEPTEMBER 1967, PP. 123-127.

41. ENGINEERING AND MINING JOURNAL. THE COST OF LEACHING U308 IN STOPES AND PUMPING SOLUTION TO THE SURFACE. V. 169, NO. 6, JUNE 1968, P. 186.

42. ENGINEERING AND MINING JOURNAL. THE ESTIMATED COST OF A NUCLEAR LEACHING EXPERIMENT ON COPPER ORE. V. 169, NO. 6, JUNE 1968, P. 186.

43. ENGINEERING AND MINING JOURNAL. INDIA STUDIES POSSIBILITY OF COPPER MINING BY NUCLEAR BLASTS. V. 171, NO. 7, JULY 1970, 33 PP.

44. ENGINEERING AND MINING JOURNAL. KENNECOTT INVESTIGATES SOLUTION MINING POTENTIAL OF DEEP CU DEPOSIT. V. 176, NO. 9, SEPTEMBER 1975, PP. 37, 41.

45. ENGINEERING AND MINING JOURNAL. KENNECOTT SETS SIGHTS ON NUCLEAR TEST FOR IN SITU RECOVERY OF COPPER. V. 168, NO. 11, NOVEMBER 1967, PP. 116-122.

46. ENGINEERING AND MINING JOURNAL. MINING IN-SITU BY NATURES EASY WAY. V. 168, NO. 10, OCTOBER 1967, PP. 75-80.

47. ENGINEERING AND MINING JOURNAL. NEW METHOD SUGGESTED FOR LEACHING URANIUM ORE IN PLACE. V. 168, NO. 5, MAY 1967, PP. 106-107.; DISCUSSION OF U.S. PATENT 3,309,140, LEACHING OF URANIUM ORE IN SITU, BY J. GARDNER AND M. I. RITCHIE.

48. ENGINEERING AND MINING JOURNAL. RANCHERS BIG BLAST SHATTERS COPPER OREBODY FOR IN-SITU LEACHING. V. 173, NO. 4, APRIL 1972, PP. 98-100.

49. ENGINEERING AND MINING JOURNAL. SOLUTION MINING OPENING NEW RESERVES. V. 175, NO. 7, JULY 1974, PP. 62-71.

50. ENGINEERING AND MINING JOURNAL. TEXAS-URANIUM PRODUCTION BY IN SITU LEACHING. V. 176, NO. 6, JUNE 1975, P. 304.

51. FEHLNER. F. P. ELECTROCHEMICAL METHOD OF MINING. U.S. PAT. 3,819,231, JUNE 25, 1974, 8 PP.

52. FINLAY, W. L. MOLECULAR MINING OF KEWEENAW COPPER, A PRIME CANDIDATE FOR THE ENLARGEMENT OF NON-RENEWABLE MINERAL RESERVES THROUGH TECHNOLOGY. PANEL DISCUSSION AT THE JOINT NAS-NAE MEETING ON NATIONAL POLICY, WASHINGTON D.C., OCT. 25, 1973, 15 PP.

53. FISHER, J. R. BACTERIAL LEACHING OF ELLIOT LAKE URANIUM ORE. CAN. MIN. AND MET. BULL., MAY 1966, PP. 588-592.

54. FISHER, J. R., F. C. LENDRUM, AND B. G. MACDERMID. LABORATORY AND UNDERGROUND STUDIES OF BACTERIAL LEACHING OF ELLIOT LAKE ORES. AIME ANNUAL MEETING, 1966, 9 PP.

55. FITCH. J. L., AND B. G. HURD (ASSIGNED TO MOBIL OIL CORPORATION, A CORPORATION OF NEW YORK). IN SITU LEACHING METHOD. U.S. PAT. 3,278,232, OCT. 11, 1966, 5 PP.

56. FITCH, J. L., AND B. G. HURD (ASSIGNED TO MOBIL OIL CORPORATION, A CORPORATION OF NEW YORK). METHOD OF LEACHING SUBSURFACE MINERALS IN SITU. U.S. PAT. 3,309,141, MAR. 14, 1967, 8 PP.

57. FLETCHER, D. A. (ASSIGNED TO TALLEY-FRAC CORPORATION, PRYOR, OKLA., A CORP. OF DEL.). WELL FRACTURING METHOD USING EXPLOSIVE SLURRY. U.S. PAT. 3,561,532, FEB. 9, 1971, 8 PP.

58. FLETCHER, J. B. IN PLACE LEACHING. SKILLINGS MINING REVIEW, V. 63, NO. 17, APR. 27, 1974, PP. 7-10.

59. FLETCHER, J. B. IN PLACE LEACHING AT MIAMI MINE, MIAMI, ARIZONA. PRES. AT AIME CENTENNIAL ANNUAL MEETING, SOC. OF MIN. ENG., AIME, NEW YORK, N.Y., MARCH 1971, AIME PREPRINT 71-AS-40; TRANS. SOC. OF MIN. ENG., AIME, V. 250, NO. 4, DECEMBER 1971, PP. 310-314.

60. FLETCHER, J. B. IN PLACE LEACHING MIAMI MINE. PRES. AT ARIZONA SECTION MEETING, MILLING DIV., AIME, APR. 6, 1962. 9 PP.

61. FRANK, J. N. COST MODEL FOR SOLUTION MINING OF URANIUM. PRES. AT THE URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 19-20, 1976, 25 PP.

62. FRANK, J. N. IN SITU LEACHING OF URANIUM. PRES. AT THE 1975 URANIUM AND THORIUM RESEARCH AND RESOURCES CONF., U.S. GEOLOGICAL SURVEY, DENVER, COLO., DEC. 8-10, 1975, 8 PP.

63. GARDNER, J., AND M. I. RITCHIE (ASSIGNED TO UTAH CONSTRUCTION AND MINING COMPANY, SAN FRANCISCO, CALIF., A CORPORATION OF DELAWARE). LEACHING OF URANIUM ORE IN SITU. U.S. PAT. 3,309,140, MAR. 14, 1967, 8 PP.

64. GEORGE, D. R., AND J. R. ROSS. RECOVERY OF URANIUM FROM URANIUM MINE WATERS AND COPPER ORE LEACHING SOLUTIONS, PROCESSING OF LOW-GRADE URANIUM ORES. PROC. OF A PANEL HELD IN VIENNA, JUNE 27 - JULY 1, 1966. INTERNAT. ATOMIC ENERGY AGENCY, VIENNA, 1967, PP. 227-234.

65. GIRARD, L., AND R. A. HARD. (ASSIGNED TO KENNECOTT COPPER CORPORATION, NEW YORK, N.Y.). STIMULATION OF PRODUCTION WELL FOR IN SITU METAL MINING. U.S. PAT. 3,841,705, OCT. 15, 1974, 7 PP.

66. GREENWOOD, C. C. UNDERGROUND LEACHING AT CANANEA. ENG. AND MIN. J., V. 121, NO. 13, MARCH 1926, PP. 518-521.

67. GRIMES, M. E. PREDICTION OF URANIUM EXTRACTION IN IN-SITU STOPE LEACHING. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 338-353.

68. GRISWOLD, G. B. ROCK FRACTURING TECHNIQUES FOR IN PLACE LEACHING. PRES. ANNUAL MEETING OF THE SOC. OF MIN. ENG., AIME, WASHINGTON D.C., FEB. 16-20, 1969, AIME PREPRINT 69-AS-74, 4 PP.

69. GROVES, R. D., T. H. JEFFERS, AND G. M. POTTER. LEACHING COARSE NATIVE COPPER ORE WITH DILUTE AMMONIUM CARBONATE SOLUTION. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 381-389.

70. GUBKIN, N. V., D. T. DESYATNIKOV, AND I. K. RUDNEVA. ADVANTAGES OF MINING URANIUM IN INUNDATED SHEET DEPOSITS BY UNDERGROUND LEACHING. ATOMIC ENERGY, V. 24, NO. 6, JUNE 1968, PP. 511-514.

71. HANSEN, S. M. NUCLEAR BLASTING FOR MINING AND LEACHING. WORLD MINING, V. 1, NO. 1, SEPTEMBER 1965, PP. 20-27.

72. HANSEN, S. M., AND A. R. JAGER. HOW TO MAKE ORE FROM MARGINAL DEPOSITS. ENG. AND MIN. J., V. 169, NO. 12, DECEMBER 1968, PP. 75-81.

73. HARD, R. A. (ASSIGNED TO KENNECOTT COPPER CORPORATION, NEW YORK, N.Y.). PROCESS FOR IN-SITU MINING. U.S. PAT. 3,910,636, OCT. 7, 1975, 5 PP.

74. HARD, R. A., AND R. L. RIPLEY (ASSIGNED TO UNION CARBIDE CORPORATION, NEW YORK, N.Y.). PROCESS FOR LEACHING BASE ELEMENTS, SUCH AS URANIUM ORE IN SITU. U.S. PAT. 3,708,206, JAN. 2, 1973, 10 PP.

75. HARDWICK, W. R. FRACTURING A DEPOSIT WITH NUCLEAR EXPLOSIVES AND RECOVERING COPPER BY THE IN-SITU LEACHING METHOD. BUMINES RI 6996, 1967, 48 PP.

76. HEINEN, H. J., T. G. CARNAHAN, AND J. A. EISELE (ASSIGNED TO THE UNITED STATES OF AMERICA, AS REPRESENTED BY THE SECRETARY OF THE INTERIOR, WASHINGTON, D.C.). CHEMICAL MINING OF COPPER PORPHYRY ORES. U.S. PAT. 3,890,007, JUNE 17, 1975, 6 PP.

77. HEINEN, H., AND B. PORTER. EXPERIMENTAL LEACHING OF GOLD FROM MINE WASTE. BUMINES RI 7250, APRIL 1969, 5 PP.

78. HOCKINGS, W. A., AND W. L. FREYBERGER. LABORATORY STUDIES OF IN SITU AMMONIA LEACHING OF MICHIGAN COPPER ORES. PROC. INTERNAT. SYMP. ON COPPER EXTRACTION AND REFINING, MET. SOC., AIME, LAS VEGAS, NEV., FEB. 22-26, 1976 (PUB. AS EXTRACTIVE METTALURGY OF COPPER--HYDROMETALLURGY AND ELECTROWINNING, ED. BY J. C. YANNOPOULOS AND J. C. AGARWAL). PORT CITY PRESS, BALTIMORE, MD., V. II, 1976, PP. 873-906.

79. HOGAN, D. K. IN-SITU COPPER LEACHING AT THE OLD RELIABLE MINE. MINING MAGAZINE, V. 24, NO. 5, MAY 1974, PP. 353-359.

80. HOLDERREED, F. L. COPPER EXTRACTION BY THE ACID LEACHING OF BROKEN ORE. REPORT OF PINCOCK, ALLEN AND HOLT, INC., 1975, 19 PP.

81. HUNKIN, G. G. CONCENTRATION PROCESSES FOR URANIUM IN SITU LEACH LIQUORS. PRES. AT 105TH ANN. MEETING, AIME, LAS VEGAS, NEV., FEB. 22-26, 1976, PREPRINT 76-B-77, 7 PP.

82. HUNKIN, G. G. THE ENVIRONMENTAL IMPACT OF SOLUTION MINING FOR URANIUM. MIN. CONG. J., V. 61, NO. 10, OCTOBER 1975, PP. 24-27.

83. HUNKIN, G. G. A REVIEW OF IN SITU LEACHING. PRES. AT AIME ANNUAL MEETING, SOC. OF MIN. ENG., AIME, NEW YORK, N.Y., FEB. 26 - MAR. 4, 1971, AIME PREPRINT 71-AS-88, 23 PP.

84. HURD, B. G., AND J. L. FITCH (ASSIGNED TO MOBIL OIL CORPORATION, A CORPORATION OF NEW YORK). IN SITU LEACHING OF SUBTERRANEAN DEPOSITS. U.S. PAT. 3,278,233, MAR. 14, 1967, 10 PP.

85. ITO, I. PRESENT STATUS OF PRACTICE AND RESEARCH WORKS ON IN-PLACE LEACHING IN JAPAN. PROC. JOINT MEETING OF THE MINING AND METALLURGICAL INSTITUTE OF JAPAN AND THE AMERICAN INSTITUTE OF MINING, METALLURGICAL, AND PETROLEUM ENGINEERS, DENVER, COLO., SEP. 1-3, 1976 (PUB. AS WORLD MINING AND METALS TECHNOLOGY, ED. BY A. WEISS), V. 1, 1976, PP. 349-364.

86. JOHNSON, P. H. ACID-FERRIC SULFATE SOLUTIONS FOR CHEMICAL MINING. MINING ENGINEERING, V. 17, NO. 8, AUGUST 1965, PP. 64-68.

87. JOHNSON, P. H., AND R. B. BHAPPU. CHEMICAL MINING - A STUDY OF LEACHING AGENTS. NEW MEXICO BUREAU OF MINES AND MINERAL RESOURCES, CIRC. 99, 1969, 7 PP.

88. JOHNSON, P. H., AND R. B. BHAPPU. CHEMICAL MINING - THEORETICAL AND PRACTICAL ASPECTS. PRES. AT SME FALL MEETING - ROCKY MOUNTAIN MINERALS CONF., LAS VEGAS, NEVADA, SEP. 6-8, 1967, 20 PP.

89. KACZYNSKI, D. A., G. W. LOWER, AND W. A. HOCKINGS. KINETICS OF LEACHING METALLIC COPPER IN AQUEOUS CUPRIC AMMONIUM NITRATE SOLUTIONS. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 390-400.

90. KALABIN, A. I. WINNING OF USEFUL ELEMENTS FROM MINERALS BY LEACHING UNDERGROUND. MINING MAGAZINE, V. 118, FEBRUARY 1968, PP. 129-134.

91. KALLUS, M. F. ENVIRONMENTAL ASPECTS OF URANIUM MINING AND MILLING IN SOUTH TEXAS. U.S. ENVIRONMENTAL PROTECTION AGENCY, EPA 906/9-75-004, OCTOBER 1975, 86 PP.

92. KELSEAUX, R. M. (ASSIGNED TO CITIES SERVICE OIL COMPANY, TULSA, OKLA.). STIMULATION OF RECOVERY FROM UNDERGROUND DEPOSITS. U.S. PAT. 3,593,793, FEB. 20, 1971, 8 PP.

93. KEYES, H. E. DISCUSSION OF IN PLACE LEACHING AT MIAMI MINE, MIAMI, ARIZONA, BY J. B. FLETCHER, IN TRANS. SOC. OF MIN. ENG., V. 250, NO. 4, DECEMBER 1971, PP. 310-314. TRANS. SOC. OF MIN. ENG., V. 252, NO. 2, JUNE 1972, PP. 186-187.

94. LACKEY, J. A. SOLUTION MINING (IN SITU LEACHING) A LITERATURE SURVEY. AMDEL BULL., (THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES), NO. 19, APRIL 1975, PP. 40-61.

95. LAMPARD, W. J. (ASSIGNED TO KENNECOTT COPPER CORPORATION, NEW YORK, N.Y.). CONTROLLED IN SITU LEACHING OF ORE DEPOSITS UTILIZING PRE-SPLIT BLASTING. U. S. PAT. 3,863,987, FEB. 4, 1975, 6 PP.

96. LANG, E. A., AND W. R. ARCHIBALD. INNOVATIVE SYSTEMS FOR THE RECOVERY OF URANIUM. PRES. AT THE NATIONAL WESTERN MIN. CONF., DENVER, COLO., FEB. 7, 1975, PP. 1-13.; COLORADO MINING ASSOC., 1975 MINING YEARBOOK, PP. 114-118.

97. LANG, L. C., AND W. B. MORREY. BLAST PREPARATION FOR IN-SITU LEACHING, AGNEW LAKE MINES LIMITED. PREPRINT OF PAPER TO BE PUB. BY ENG. AND MIN. J., JANUARY 1976.

98. LANG, L. C., AND W. B. MORREY. SCIENTIFIC BLASTING FOR IN-SITU LEACHING PROVES SUCCESSFUL AT AGNEW LAKE MINES. ENG. AND MIN. J., V. 177, NO. 1, JANUARY 1976, PP. 100-102.

99. LARSON, D. R. A REPORT COMPARING OPEN PIT MINING AND HEAP LEACHING WITH AN IN SITU LEACHING SYSTEM FOR A HYPOTHETICAL OXIDE COPPER DEPOSIT BASED ON THE CHARACTERISTICS OF THE CACTUS DEPOSIT, MIAMI, ARIZONA. M.S. THESIS, UNIV. MINNESOTA, DEPT. OF MINERAL AND METALLURGICAL ENG., JUNE 1973, 254 PP.

100. LASWELL, G. W. CONSIDERATIONS APPLICABLE TO UTILIZATION OF ROTARY DRILLING FOR IN-PLACE RECOVERY OF MINERALS AND HYDROCARBONS. REPORT FOR PINCOCK, ALLEN AND HOLT, INC., FEB. 1975, 7 PP.

101. LASWELL, G. W. WANTED: ROTARY DRILLING TECHNOLOGY FOR IN SITU MINING SYSTEMS. MINING ENGINEERING, V. 27, NO. 1, JANUARY 1976, PP. 22-26.

102. LEWIS, A. E. CHEMICAL MINING OF PRIMARY COPPER ORES BY USE OF NUCLEAR TECHNOLOGY. PROC. SYMP. OF ENGINEERING WITH NUCLEAR EXPLOSIVES, LAS VEGAS, NEV., JAN. 14-16, 1970, CONF. 700101, V. 21, PP. 907-917.

103. LEWIS, A. E. IN SITU PRESSURE LEACHING METHOD. U.S. PAT. 3,640,579, FEB, 8, 1972, 5 PP.

104. LEWIS, A. E. (ASSIGNED TO THE UNITED STATES OF AMERICA AS REPRESENTED BY THE U.S. ATOMIC ENERGY COMMISSION, WASHINGTON, D.C.). SITU LEACHING SOLVENT EXTRACTION-PROCESS. U.S. PAT. 3,823,981, JULY 16, 1974, 6 PP.

105. LEWIS, A. E., AND R. L. BRAUN. NUCLEAR CHEMICAL MINING OF PRIMARY COPPER SULFIDES. PRES. AT THE AIME MEETING, SAN FRANCISCO, CALIF., JAN. 27, 1972, 30 PP.

106. LEWIS, A. E., R. L. BRAUN, C. J. SISEMORE, AND R. G. MALLON. NUCLEAR SOLUTION MINING-BREAKING AND LEACHING CONSIDERATIONS. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 56-75.

107. LEWIS, F. M., AND R. B. BHAPPU. EVALUATING MINING VENTURES VIA FEASIBILITY STUDIES. MINING ENGINEERING, V. 27, NO. 10, OCTOBER 1975, PP. 48-54.

108. LEWIS, F. M., C. K. CHASE, AND R. B. BHAPPU. ECONOMIC EVALUATION OF IN SITU EXTRACTION FOR COPPER, GOLD, AND URANIUM. PROC. JOINT MEETING OF THE MINING AND METALLURGICAL INSTITUTE OF JAPAN AND THE AMERICAN INSTITUTE OF MINING, METALLURGICAL, AND PETROLEUM ENGINEERS, DENVER, COLO., SEP. 1-3, 1976 (PUB. AS WORLD MINING AND METALS TECHNOLOGY, ED. BY A. WEISS), V. 1, 1976, PP. 333-348.

109. LIVINGSTON, C. W. METHOD OF MINING ORES IN SITU BY LEACHING. U.S. PAT. 2,818,240, DEC. 31, 1957, 14 PP.

110. LONGWELL, R. L. IN PLACE LEACHING OF A MIXED COPPER ORE BODY. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 233-242.

111. MACGREGOR, R. A. THE BACTERIAL LEACHING OF URANIUM. NUCLEAR APPLICATIONS, V. 6, JANUARY 1969, PP. 68-72.

112. MACGREGOR, R. A. RECOVERY OF U308 BY UNDERGROUND LEACHING. PRES. AT ANNUAL GENERAL MEETING, MONTREAL, APRIL 1964, CAN. MIN. AND MET. BULL., MAY 1966, PP. 583-587.

113. MACGREGOR, R. A. URANIUM DIVIDENDS FROM BACTERIAL LEACHING. MINING ENGINEERING, V. 21, NO. 3, MARCH 1969, PP. 54-55.

114. MALOUF, E. E. COPPER LEACHING PRACTICES. PRES. AT ANNUAL MEETING, SOC. OF MIN. ENG., AIME, SAN FRANCISCO, CALIF., FEB. 20-24, 1972, AIME PREPRINT 72-AS-84, 7 PP.

115. MALOUF, E. E. INTRODUCTION TO DUMP LEACHING PRACTICE. PART I OF SECOND TUTORIAL SYMP. ON EXTRACTIVE METALLURGY, UNIV. UTAH, DEC. 14-16, 1972, 26 PP.

116. MALOUF, E. E. THE ROLE OF MICROORGANISMS IN CHEMICAL MINING. MINING ENGINEERING, V. 23, NO. 11, NOVEMBER 1971, PP. 43-46.

117. MARRS, L. F. UNDERGROUND LEACHING OF URANIUM AT THE PITCH MINE. PRES. AMERICAN MIN. CONG. CONVENTION, DENVER, COLO., SEP. 27-30, 1970, 14 PP.; MIN. CONG. J., V. 56, NO. 11, NOVEMBER 1970, PP. 35-43.

118. MARTIN, J. W. (ASSIGNED TO OIL RECOVERY CORPORATION, NEW YORK, N.Y., A CORPORATION OF NEW YORK). RECOVERY OF URANIUM BY CARBONATED WATER AND SURFACE ACTION AGENTS AND WETTING AGENTS. U.S. PAT. 3,130,960, APR. 28, 1964, 8 PP.

119. MCKINNEY, W. A. SOLUTION MINING. MINING ENGINEERING, V. 25, NO. 2, FEBRUARY 1973, PP. 56-57.

120. MICHIGAN TECHNOLOGICAL UNIVERSITY. A PLAN FOR DETERMINING THE FEASIBILITY OF AN IN SITU LEACHING OF NATIVE COPPER ORES. PREPARED FOR HOMESTAKE COPPER COMPANY, CALUMET, MICHIGAN, JUNE 14, 1974, 35 PP.

121. MILLER, J. D. PROCESSING OF LEACH LIQUORS PRODUCED BY NUCLEAR SOLUTION MINING. LAWRENCE LIVERMORE LABORATORY, REPT. UCRL-51350, FEB. 15, 1973, 18 PP.

122. MINING CONGRESS JOURNAL. AN EXPERIMENTAL IN-PLACE LEACHING TEST. V. 59, NO. 5, MAY 1973, P. 10.

123. MINING ENGINEERING. KENNECOTT PROPOSES NUCLEAR MINING EXPERIMENTS AT SAFFORD DEPOSIT. V. 19, NO. 11, NOVEMBER 1967, PP. 66-67.

124. MINING ENGINEERING. KERR-MCGEES AMBROSIA COMPLEX: FROM MINED ROCK TO YELLOW CAKE. V. 26, NO. 8, AUGUST 1974, PP. 28-29.

125. MINING ENGINEERING. RANCHERS DEVELOPMENT SETS OFF BLAST, WILL LEACH AT BIG MIKE. V. 25, NO. 8, AUGUST 1973, P. 10.

126. MINING MAGAZINE. IN SITU LEACHING - A REVIEW. V. 125, NO. 3, SEPTEMBER 1971, PP. 213-215.

127. MINING RECORD (DENVER, COLORADO). RANCHERS BLASTS OLD RELIABLE TO TEST NEW MINING TECHNIQUE. V. 83, NO. 12, MAR. 22, 1972, P. 1.

128. MINING RECORD (DENVER, COLORADO). SOLUTION MINING - CAN IT SPEED DEVELOPMENT OF URANIUM RESERVES. V. 87, NO. 26, JUNE 23, 1976, PP. 1, 7.

129. MURPHY, J. NEW RETURNS FROM OLD RELIABLE. SOURCE UNKNOWN, PP. 20-23.; COPY AVAILABLE FOR CONSULTATION AT BUREAU OF MINES, TWIN CITIES MINING RESEARCH CENTER, MINNEAPOLIS, MINNESOTA.

130. MYERS, D. L. MINING COPPER IN SITU. MINES MAG., V. 31, NO. 6, JUNE 1941, PP. 255-263.

131. NARAYAN, K., P. J. RAND, AND R. B. HILL. ENVIRONMENTAL CONSIDERATIONS IN URANIUM SOLUTION MINING. PRES. AT 105TH ANN. MEETING, AIME, LAS VEGAS, NEV., FEB. 22-26, 1976, PREPRINT 76-AS-70, 12 PP.

132. OLSON, J. J., W. C. LARSON, D. R. TWEETON, AND OTHERS. MINING RESEARCH FOR IMPROVED IN SITU EXTRACTION OF URANIUM. PRES. AT URANIUM IN SITU LEACHING CONF., SPONSORED BY AAPG, SME, AND SPE, VAIL, COLO., AUG. 25-27, 1976, BUMINES PREPRINT, 37 PP.

133. ORTLOFF, G. D., C. E. COOKE, JR., AND D. K. ATWOOD (ASSIGNED TO ESSO PRODUCTION RESEARCH COMPANY). MINERAL RECOVERY. U.S. PAT. 3,574,599, APR. 13, 1971, 6 PP.

134. PINGS, W. B. BACTERIAL LEACHING. MINERAL INDUSTRIES BULL., V. 2, NO. 3, MAY 1968, PP. 1-19.

135. POJAR, M. IN-PLACE LEACHING OF A COPPER SULFIDE DEPOSIT. PRES. AT DULUTH MEETING, SOC. OF MIN. ENG., AIME, DULUTH, MINN., JAN. 16-18, 1974, 9 PP.

136. POPOFF, D. R. BACTERIAL LEACHING MEANS NEW LIFE FOR MINE. CANADIAN NUCLEAR TECHNOLOGY, V. 5, NO. 4, JULY - AUGUST 1966, PP. 31-33.

137. PORTER, D. D. BLAST DESIGN FOR IN SITU LEACHING. PRES. AT SOUTHWEST MINERAL INDUSTRY CONF., PHOENIX, ARIZ., APR. 27, 1973, 7 PP.; AVAILABLE FROM E. I. DU PONT DE NEMOURS AND CO., INC., WILMINGTON, DEL. 19898.

138. PORTER, D. D., AND H. G. CARLEVATO. IN SITU LEACHING: A NEW BLASTING CHALLENGE. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 33-43.

139. POTTER, G. M. SOLUTION MINING. MINING ENGINEERING, V. 21, NO. 1, FEBRUARY 1969, PP. 68-69.

140. PRESCOTT NEWSPAPER (PRESCOTT, ARIZONA). KIRKLAND BLAST WILL OPEN NEW BODY OF COPPER. APR. 18, 1973, P. 1.

141. RABB, D. D. LEACHING OF COPPER ORES AND THE USE OF BACTERIA. LAWRENCE RADIATION LABORATORY, REPT. UCID-4958, JAN. 22, 1965, 25 PP.

142. RABB, D. D. PENETRATION OF LEACH SOLUTION INTO ROCKS FRACTURED BY A NUCLEAR EXPLOSION. ANNUAL MEETING OF SOC. OF MIN. ENG., AIME, WASHINGTON, D.C., FEB. 16-20, 1969, V. 1:A, AIME PREPRINT 69-AS-4, 14 PP.

143. RABB, D. D. SOLUTION MINING. MINING ENGINEERING, V. 24, NO. 2, FEBRUARY 1972, PP. 62-64.

144. RANCHERS EXPLORATION AND DEVELOPMENT CORP. THE OLD RELIABLE PROJECT - A TEST OF NEW TECHNOLOGY. COMPANY REPORT, 4 PP.

145. RHOADES, V. W. (ASSIGNED TO CITIES SERVICE OIL COMPANY, TULSA, OKLA.). URANIUM SOLUTION MINING PROCESS. U.S. PAT. 3,713,698, JAN. 30, 1973, 4 PP.

146. RHOADES, V. W. (ASSIGNED TO DALCO OIL COMPANY; UNITED STATES STEEL CORPORATION; PART INTEREST TO EACH). URANIUM SOLUTION MINING PROCESS. U.S. PAT. 3,792,903, FEB. 19, 1974, 4 PP.

147. RICKARD, R. S. CHEMICAL CONSTRAINTS ON IN-SITU LEACHING AND METAL RECOVERY. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 185-192.

148. ROMAN, R. J. SOLUTION MINING. MINING ENGINEERING, V. 26, NO. 2, FEBRUARY 1974, PP. 43-44.

149. ROOT, T. LEGAL ASPECTS OF MINING BY THE IN SITU LEACHING METHOD. PRES. AT 22ND ANN. ROCKY MOUNTAIN MINERAL LAW INST., SUN VALLEY, IDAHO, JULY 22-24, 1976, 18 PP.

150. ROSENBAUM, J. B., AND W. A. MCKINNEY. IN SITU RECOVERY OF COPPER FROM SULFIDE OREBODIES FOLLOWING NUCLEAR FRACTURING. PROC. SYMP. OF ENGINEERING WITH NUCLEAR EXPLOSIVES, LAS VEGAS, NEV., JAN. 14-16, 1970, CONF. 700101, V. 2, PP. 877-887.

151. ROSENZWEIG, M. D. NEW COPPER TECHNOLOGY IS WINNING THE ORE. CHEMICAL ENG., DEC. 4, 1967, PP. 88-90, 92.

152. ROUSE, J. V. ENVIRONMENTAL ASPECTS OF IN SITU MINING AND DUMP LEACHING. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 3-14.

153. RUDERSHAUSEN, C. G. COPPER SOLUTION MINING AT OLD RELIABLE. PRES. AT NATIONAL MEETING OF AMERICAN INST. OF CHEMICAL ENG., SALT LAKE CITY, NEV., AUG. 18-21, 1974, 11 PP.

154. ST. PETER, A. L. IN SITU LEACHING OF OREBODIES DESIGN AND MANAGEMENT. REPORT FOR PINCOCK, ALLEN AND HOLT, INC., 1975, 20 PP.

155. ST. PETER, A. L. IN SITU MINING -- A DESIGN FOR POSITIVE RESULTS. PRES. AT 105TH ANN. MEETING, AIME, LAS VEGAS, NEV., FEB. 22-26, 1976, PREPRINT 76-AS-36, 43 PP.

156. SAREEN, S. S., L. GIRARD, III, AND R. A. HARD. STIMULATION OF RECOVERY FROM UNDERGROUND DEPOSITS. U.S. PAT. 3,865,435, FEB. 11, 1975, 5 PP.

157. SCOTT, W. G. (ASSIGNED TO INSPIRATION CONSOLIDATED COPPER COMPANY, A CORPORATION OF MAINE). LEACHING COPPER ORES. U.S. PAT. 2,563,623, AUG. 7, 1951, 14 PP.

158. SELIM, A. A. A DECISION ANALYSIS APPROACH TO IN SITU EXTRACTION OF COPPER. PH. D. THESIS, UNIV. MINNESOTA, MINNEAPOLIS, MINN., MARCH 1976, 659 PP.

159. SHEFFER, H. W., AND L. G. EVANS. COPPER LEACHING PRACTICES IN THE WESTERN UNITED STATES. BUMINES IC 8341, 1968, 57 PP.

160. SHOCK, D. A. DEVELOPMENTS IN SOLUTION MINING PORTEND GREATER USE FOR IN SITU LEACHING. MINING ENGINEERING, V. 22, NO. 2, FEBRUARY 1970, P. 73.

161. SHOCK, D. A., AND F. R. CONLEY. SOLUTION MINING-ITS PROMISE AND ITS PROBLEMS. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 79-97.

162. SHOCK, D. A., AND J. G. DAVIS, II. (ASSIGNED TO CONTINENTAL OIL COMPANY, PONCA CITY, OKLA., A CORP. OF DEL.). FRACTURING METHOD IN SOLUTION MINING. U.S. PAT. 3,387,888, JUNE 11, 1968, 4 PP.

163. SHOCK, D. A., AND J. G. DAVIS. HYDROFRACING AS A MINING TECHNIQUE. PRES. AT ANNUAL MEETING, SOC. OF MIN. ENG., AIME, WASHINGTON, D.C., FEB. 16-20, 1969, AIME PREPRINT 69-AS-63, 25 PP.

164. SHOEMAKER, R. S. AMMONIA REVIVAL FOR THE KEWEENAW. MINING ENGINEERING, V. 24, NO. 5, MAY 1972, PP. 45-47.

165. SIEVERT, J. A. (ASSIGNED TO CONTINENTAL OIL COMPANY, PONCA CITY, OKLA.). IN SITU MINING USING BACTERIA. U.S. PAT. 3,937,520, FEB. 10, 1976.

166. SIEVERT, J. A., W. L. MARTIN, AND F. R. CONLEY. IN SITU LEACHING OF URANIUM. PRES. AT SOC. MIN. ENG. FALL MEETING, ST. LOUIS, MO., 1970, 18 PP.

167. SKILLINGS MINING REVIEW. EL PASO STUDYING IN SITU LEACHING AT EMERALD ISLE. V. 63, NO. 52, DEC. 28, 1974, P. 8.

168. SKILLINGS MINING REVIEW. RANCHERS DETONATES LARGE BLAST AT BIG MIKE MINE IN NEVADA. V. 62, NO. 29, JULY 21, 1973, P. 5.

169. SPEDDEN, H. R., AND E. E. MALOUF (ASSIGNED TO KENNECOTT COPPER CORPORATION, NEW YORK, N.Y.). CONTROLLED IN-SITU LEACHING OF MINERAL VALUES. U.S. PAT. 3,815,957, JUNE 11, 1974, 3 PP.

170. SPEDDEN, H. R., AND E. E. MALOUF (ASSIGNED TO KENNECOTT COPPER CORPORATION, NEW YORK, N.Y.). IN-SITU GENERATION OF ACID FOR IN-SITU LEACHING OF COPPER. U.S. PAT. 3,834,760, SEP. 10, 1974, 4 PP.

171. SPEDDEN, H. R., E. E. MALOUF, AND J. DAVIS. IN SITU LEACHING OF COPPER--PILOT PLANT TEST. PRES. AT ANN. MEETING, AIME, NEW YORK, FEBRUARY 1971, 21 PP.

172. STECKLEY, R. C., W. C. LARSON, D. V. DANDREA. BLASTING TESTS IN A PORPHYRY COPPER DEPOSIT--IN PREPARATION FOR IN SITU EXTRACTION. BUMINES RI 8070, 1975, 57 PP.

173. STEWART, R. M., AND K. E. NIERMEYER. NUCLEAR TECHNOLOGY AND MINERAL RECOVERY. PROC. SYMP. ON ENGINEERING WITH NUCLEAR EXPLOSIVES, LAS VEGAS, NEV., JAN. 14-16, 1970, CONF. 700101, V. 2, PP. 864-876.

174. TEWES, H. A., H. B. LEVY, AND L. L. SCHWARTZ. NUCLEAR CHEMICAL COPPER MINING AND REFINING: RADIOLOGICAL CONSIDERATIONS. LAWRENCE LIVERMORE LABORATORY, REPT. UCRL-51345, REV. 1, JUNE 3, 1974, 34 PP.

175. TEXAS WATER QUALITY BOARD. IN SITU LEACHING TEST SITE NEAR BENEVIDAS, TEXAS IN DUVAL COUNTY. PERMIT ISSUED TO UNION CARBIDE

176. TEXAS WATER QUALITY BOARD. IN SITU LEACHING TEST SITE NEAR BRUNI, TEXAS IN WEBB COUNTY. PERMIT NO. 01941, ISSUED TO MOBIL OIL COMPANY, 1975.

177. TEXAS WATER QUALITY BOARD. IN SITU LEACHING TEST SITE NEAR BRUNI, TEXAS IN WEBB COUNTY. PERMIT NO. 01942, ISSUED TO WYOMING MINERALS, 1975

178. TEXAS WATER QUALITY BOARD. IN SITU LEACHING TEST SITE NEAR GEORGE WEST, TEXAS IN LIVE OAK COUNTY. PERMIT, AND AMENDMENT TO PERMIT, NO. 01890, ISSUED TO ATLANTIC RICHFIELD COMPANY, JAN. 28, 1975, 53 PP.

179. TEXAS WATER QUALITY BOARD. IN SITU LEACHING TEST SITE NEAR GEORGE WEST, TEXAS IN LIVE OAK COUNTY. PERMIT ISSUED TO DALCO

180. TEXAS WATER QUALITY BOARD. IN SITU LEACHING TEST SITE NEAR KENEDY, TEXAS IN BEE COUNTY. PERMIT ISSUED TO INTERCONTINENTAL ENERGY CORPORATION

181. TEXAS WATER QUALITY BOARD. IN SITU LEACHING TEST SITE NEAR RAY POINT, TEXAS IN LIVE OAK COUNTY. PERMIT NO. 01949, ISSUED TO WYOMING MINERALS, 1975

182. THOMAS, R. W. LEACHING COPPER FROM WORKED-OUT AREAS OF THE RAY MINES, ARIZONA. MIN. AND MET., V. 19, NOVEMBER 1938, PP. 481-485.

183. TRUSSELL, P. C., D. W. DUNCAN, AND C. C. WALDEN. BIOLOGICAL MINING. CAN. MIN. J., V. 85, NO. 3, MARCH 1964, PP. 46-49.

184. TUOVINEN, O. H. MICROBIOLOGICAL ASPECTS IN THE LEACHING OF URANIUM BY THIOBACILLUS FERROOXIDANS. ATOMIC ENERGY REVIEW, V. 10, NO. 2, 1972, PP. 251-258.

185. U.S. ATOMIC ENERGY COMMISSION. TO CONDUCT A RESEARCH IN-SITU URANIUM SOLUTION MINING PROJECT IN SWEETWATER COUNTY, WYOMING. SOURCE MATERIAL LICENSE, SUA-1223, DOCKET NO. 40-8348, ISSUED TO MINERALS EXPLORATION COMPANY, DEC. 31, 1974, 142 PP.

186. U.S. ATOMIC ENERGY COMMISSION. FOR EXPERIMENTAL INSITU URANIUM LEACHING AND REFINING ACTIVITIES IN CONVERSE COUNTY, WYOMING. SOURCE MATERIAL LICENSE, SUA-1064, DOCKET NO. 40-8064, ISSUED TO HUMBLE OIL AND REFINING COMPANY, NOV. 12, 1970, 18 PP.

187. U.S. ATOMIC ENERGY COMMISSION. INSITU LEACHING OF URANIUM, CONVERSE COUNTY, WYOMING. SOURCE MATERIAL LICENSE, SUA-1228, DOCKET NO. 40-8380, ISSUED TO ROCKY MOUNTAIN ENERGY COMPANY, FEB. 26, 1975, 39 PP.

188. U.S. ATOMIC ENERGY COMMISSION. RESEARCH AND DEVELOPMENT INSITU LEACHING ACTIVITIES IN CAMPBELL COUNTY, WYOMING. SOURCE MATERIAL LICENSE, SUA-1199, DOCKET NO. 40-8200, ISSUED TO CLEVELAND-CLIFFS IRON COMPANY, MAY 23, 1974, 43 PP.

189. U.S. ATOMIC ENERGY COMMISSION. A SOLUTION MINING TEST PROGRAM, CHARLIE PROPERTY, JOHNSON COUNTY, WYOMING. SOURCE MATERIAL LICENSE, DOCKET NO. 40-8511, ISSUED TO URANERZ U.S.A., INC., 56 PP.

190. U.S. BUREAU OF MINES. PROJECT PROPOSAL: PORPHY COPPER - IN SITU RECOVERY BY LEACHING FOLLOWING NON-NUCLEAR FRACTURING. INTERDISCIPLINARY RESEARCH TASK FORCE, COMP. BY L. W. GIBBS, NOV. 23, 1971, (REV. DEC. 6, 1971), 20 PP.; AVAILABLE FOR CONSULTATION AT BUREAU OF MINES, TWIN CITIES MINING RESEARCH CENTER, MINNEAPOLIS, MINNESOTA.

191. U.S. ENERGY RESEARCH AND DEVELOPMENT ADMINISTRATION. IN SITU LEACHING OF A NUCLEAR RUBBLIZED COPPER ORE BODY (IN TWO VOLUMES). U.S. ERDA REPORT NVO-155, JUNE 1975, 348 PP.

192. U.S. ENERGY RESEARCH AND DEVELOPMENT ADMINISTRATION. NATIONAL URANIUM RESOURCE EVALUATION PRELIMINARY REPORT. GRAND JUNCTION OFFICE, GRAND JUNCTION, COLO., REPT. GJO-111(76), JUNE 1976, 132 PP.

193. U.S. ENERGY RESEARCH AND DEVELOPMENT ADMINISTRATION. STATISTICAL DATA OF THE URANIUM INDUSTRY. GRAND JUNCTION OFFICE, GRAND JUNCTION, COLO., REPT. GJO-100(76), APRIL 1976, 87 PP.

194. WARD, M. H. ENGINEERING FOR IN SITU LEACHING. MIN. CONG. J., V. 59, NO. 1, JANUARY 1973, PP. 21-27.

195. WARD, M. H. SURFACE BLASTING FOLLOWED BY IN SITU LEACHING THE BIG MIKE MINE. PROC. SOLUTION MINING SYMP., AIME ANNUAL MEETING, DALLAS, TEXAS, FEB. 25-27, 1974, PP. 243-251.

196. WEED, R. C. CANANEAS PROGRAM FOR LEACHING IN PLACE. MINING ENGINEERING, V. 8, NO. 7, JULY 1, 1956, PP. 721-723.

197. WHITE, L. IN SITU LEACHING OPENS NEW URANIUM RESERVES IN TEXAS. ENG. AND MIN. J., V. 176, NO. 7, JULY 1975, PP. 73-80.

198. WORMSER, F. E. LEACHING A COPPER MINE. ENG. AND MIN. J., V. 116, NO. 16, OCTOBER 1923, PP. 665-670.

199. ZIMMER, P. F., AND M. A. LEKAS. (COMP. AND ED. BY). PROJECT SLOOP (NUCLEAR EXPLOSIVES-PEACEFUL APPLICATIONS). U.S. ATOMIC ENERGY COMMISSION REPORT NO. PNE 1300, JUNE 1, 1967, 44 PP.

IN SITU LEACHING BIBLIOGRAPHY

PART II

COMPILED BY

WILLIAM C. LARSON, GEOLOGIST

DENNIS V. D'ANDREA, GEOPHYSICIST

TWIN CITIES MINING RESEARCH CENTER
U.S. BUREAU OF MINES

COPIES OF BUREAU OF MINES REPORTS MAY BE ORDERED FROM:

DIVISION OF PRODUCTION AND DISTRIBUTION
U.S. BUREAU OF MINES
4800 FORBES AVE.
PITTSBURGH, PENNSYLVANIA 15213

THE AUTHORS ARE AVAILABLE FOR CONSULTATION AT:

TWIN CITIES MINING RESEARCH CENTER
P.O. BOX 1660
TWIN CITIES, MINNESOTA 55111

TELEPHONE: (612) 725-4500

[illegible]
NZ [illegible] / To B.P. [illegible]
[illegible] odd [illegible]
EBCDIC
coding [illegible]
[illegible]

10 FEBRUARY 1976

IN SITU LEACHING BIBLIOGRAPHY

THIS BIBLIOGRAPHY IS SEPARATED INTO TWO PARTS.

PART I IS A BIBLIOGRAPHY PERTAINING SPECIFICALLY TO IN SITU LEACHING AND CONTAINS ARTICLES ON BOTH THE TECHNICAL AND GENERAL ASPECTS OF IN SITU LEACHING. MAJOR EMPHASIS HAS BEEN PLACED ON COPPER AND URANIUM, HOWEVER SOME INFORMATION ON LEACHING OTHER COMMODITIES SUCH AS MANGANESE, GOLD, MOLYBDENUM, ETC. IS ALSO INCLUDED. IN ADDITION VARIOUS DOMESTIC AND FOREIGN PATENTS ARE INCLUDED.

PART II OF THIS BIBLIOGRAPHY PERTAINS TO ARTICLES THAT ARE INDIRECTLY RELATED TO IN SITU LEACHING, COVERING SUCH SUBJECTS AS LABORATORY LEACHING STUDIES, BACTERIA LEACHING, HYDROLOGY, GEOLOGY, METALLURGICAL ASPECTS OF LEACHING, AND OTHERS.

10 FEBRUARY 1976

1. ADLER, H. H. GEOLOGY AND RESOURCES OF FOREIGN URANIUM DEPOSITS. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. II, NOVEMBER 1974, PP. 39-71.

2. AGERS. D. W., J. E. HOUSE, R. R. SWANSON, AND J. L. DROBNICK. A NEW REAGENT FOR LIQUID ION EXCHANGE RECOVERY OF COPPER. MINING ENGINEERING, V. 17, NO. 12, DECEMBER 1965, PP. 76-80.

3. AGERS, D. W., J. E. HOUSE, R. R. SWANSON, AND J. L. DROBNICK. COPPER RECOVERY FROM ACID SOLUTIONS USING LIQUID ION EXCHANGE. GENERAL MILLS CHEMICALS, INC., MINNEAPOLIS, MINNESOTA, 28 PP.

4. ALLMAN, M. B., AND J. A. HARRIS. AN EXPERIMENT IN HEAP LEACHING. MIN. CONG. J., V. 55, NO. 7, JULY 1969, PP. 28-31.

5. APPELIN, C. W. CURRENT URANIUM ORE RESERVES. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. I, NOVEMBER 1974, PP. 73-82.

6. ARCHER, J. S., AND S. W. WONG. USE OF A RESERVOIR SIMULATOR TO INTEGRATE LABORATORY WATERFLOOD DATA. SOC. OF PETROL. ENG. J., DECEMBER 1973, PP. 343-347.

7. ARIZONA DEPARTMENT OF MINERAL RESOURCES. DIRECTORY OF ACTIVE MINES IN ARIZONA. PHOENIX, ARIZONA, DECEMBER 1973, 11 PP.

8. ARMSTRONG, R. E., G. C. EVANS, AND G. E. FLETCHER. TRITIATED WATER AS A TRACER IN THE DUMP LEACHING OF COPPER. BUMINES RI 7510, MAY 1971, 39 PP.

9. ARONOFSKY, J. S., H. J. RAMEY, JR. MOBILITY RATIO - ITS INFLUENCE ON INJECTION OR PRODUCTION HISTORIES IN FIVE-SPOT WATER FLOOD. PETROL. TRANS., AIME, V. 207, 1956, PP. 205-210.

10. AUCK, Y. T. LABORATORY STUDIES IN DUMP LEACHING. PART V OF SECOND TUTORIAL SYMP. ON EXTRACTIVE METALLURGY, UNIV. UTAH, DEC. 14-16, 1972, 72 PP.

11. AUDSLEY. A., AND G. R. DABORN. NATURAL LEACHING OF URANIUM ORES, PART II - A STUDY OF THE EXPERIMENTAL VARIABLES. PRES. GENERAL MEETING OF THE INSTITUTION OF MINING AND METALLURGY, FEB. 21, 1963, PP. 235-246.

12. AUDSLEY. A., AND G. R. DABORN. NATURAL LEACHING OF URANIUM ORES, PART III - APPLICATION TO SPECIFIC ORES. PRES. GENERAL MEETING OF THE INSTITUTION OF MINING AND METALLURGY, FEB. 21, 1963, PP. 247-254.

13. BALLARD. J. K. METHODS OF LEACH DUMPS SOLUTION AT ANACONDAS BUTTE OPERATION. PRES. AT FALL MEETING, SOC. OF MIN. ENG., AIME, MINNEAPOLIS, MINN., SEP. 18-20, 1968, AIME PREPRINT 68-AS-312, 4 PP.

14. BARANOWSKI, F. P. VERSATILITY OF FEED SUPPLY OF ENRICHED URANIUM. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. I, NOVEMBER 1974, PP. 72-87.

15. BARNES, C. H. DOMESTIC PRODUCTION CAPABILITY. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. I, NOVEMBER 1974, PP. 136-178.

16. BAUER, D. J., AND R. E. LINDSTROM. USE OF CHELATING AGENTS FOR RECOVERY OF COPPER FROM CARBONATE AND SILICATE ORES. J. OF METALS, MAY 1971, V. 23, PP. 31-33.

17. BECK, J. V. THE ROLE OF BACTERIA IN COPPER MINING OPERATIONS. BIOTECHNOLOGY AND BIOENGINEERING, V. 9, SEPTEMBER 1967, PP. 487-497.

18. BEVERLY, R. G., AND W. D. CHARLES. PILOT PLANT ALKALINE LEACHING OF URANIUM ORES. PROC. OF THE SECOND UNITED NATIONS INTERNAT. CONF. ON THE PEACEFUL USES OF ATOMIC ENERGY, GENEVA, SEP. 1-13, 1958 (PUB. AS PROCESSING OF RAW MATERIALS), V. 3, PP. 326-332.

19. BHAPPU, R. B., AND F. M. LEWIS. ECONOMIC EVALUATION OF AVAILABLE PROCESSES FOR TREATING OXIDE COPPER ORES. PRES. AT FALL MEETING, SOC. OF MIN. ENG., AIME, ACAPULCO, MEXICO, SEP. 22-25, 1974, AIME PREPRINT 74-AS-334, 26 PP.

20. BHAPPU, R. B., AND F. M. LEWIS. GOLD EXTRACTION FROM LOW GRADE ORES - ECONOMIC EVALUATION OF PROCESSES. MIN. CONG. J., V. 61, NO. 1, JANUARY 1975, PP. 38-41.

21. BHAPPU, R. B., D. H. REYNOLDS, AND W. S. STAHMANN. STUDIES ON HYPOCHLORITE LEACHING OF MOLYBDENITE. NEW MEXICO STATE BUREAU OF MINES AND MINERAL RESOURCES, CIRC. 66, 1963, 22 PP.

22. BLAIR, P. M., AND C. F. WEINAUG. SOLUTION OF TWO-PHASE FLOW PROBLEMS USING IMPLICIT DIFFERENCE EQUATIONS. SOC. OF PETROL. ENG. J., DECEMBER 1969, PP. 417-423.

23. BORG, I. Y. SOME SHOCK EFFECTS IN GRANODIORITE TO 270 KBAR AT THE PILEDRIVER SITE. LAWRENCE RADIATION LABORATORY, REPT. UCRL-73377, AUG. 27, 1971, 33 PP.

24. BORG, I. Y. SURVEY OF PILEDRIVER RESULTS AND PRELIMINARY INTERPRETATION OF THREE POSTSHOT CORES IN AND NEAR THE CAVITY. LAWRENCE RADIATION LABORATORY, REPT. UCRL-50865, APR. 29, 1970, 67 PP.

25. BOUGHTON, L. D. SOLUTION MINING WELL COMPLETION SERVICES AND TECHNIQUES. PRES. AT AIME CENTENNIAL ANNUAL MEETING, SOC. OF MIN. ENG., NEW YORK, FEB. 26 - MAR. 4, 1971, AIME PREPRINT 71-AS-36, 18 PP.

26. BOWYER, B. NATIONAL URANIUM RESOURCE PROGRAM. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. I, NOVEMBER 1974, PP. 53-72.

27. BREDEHOEFT, J. D., AND G. F. PINDER. MASS TRANSPORT IN FLOWING GROUNDWATER. WATER RESOURCES RESEARCH, AMERICAN GEOPHYSICAL UNION, V. 9, NO. 1, FEBRUARY 1973, PP. 194-210.

28. BRIERLEY, C. L. MOLYBDENITE - LEACHING: USE OF A HIGH-TEMPERATURE MICROBE. J. OF THE LESS-COMMON METALS, V. 36, 1974, PP. 237-147.

29. BRIERLEY, C. L., AND L. E. MURR. LEACHING: USE OF A THERMOPHILIC AND CHEMOAUTOTROPHIC MICROBE. SCIENCE, V. 179, NO. 4072, FEB. 2, 1973, PP. 488-490.

30. BRYCE, D. M., D. G. CERIGO, AND P. H. JENNINGS. PERCOLATION AND AGITATION LEACHING OF AN OXIDIZED COPPER ORE. CAN. MIN. AND MET. BULL., V. 61, NO. 673, MAY 1968, PP. 641-645.

31. BRYNER, L. C., AND R. ANDERSON. MICROORGANISMS IN LEACHING SULFIDE MINERALS. IND. ENG. CHEM., V. 49, NO. 10, OCTOBER 1957, PP. 1721-1724.

32. BRYNER, L. C., AND A. K. JAMESON. MICROORGANISMS IN LEACHING SULFIDE MINERALS. APPLIED MICROBIOLOGY, V. 6, 1958, PP. 281-287.

33. BUNUS, F., I. MATEI, C. MATYAS, S. NECIU, V. SPOREA, AND S. TATAKU. SOME QUANTITATIVE RELATIONS ON THE RADIOACTIVE MINERALS LEACHING PROCESSES. REVUE ROUMAINE DE CHIMIE, V. 14, JUNE 1969, PP. 729-736.

34. BUTLER, T. D. RECENT ADVANCES IN COMPUTATIONAL FLUID DYNAMICS. LOS ALAMOS SCIENTIFIC LABORATORY, LOS ALAMOS, N. MEX., 1973, LAUR 73-1473, 21 PP.

35. CANEER, W. T., AND N. M. SAUM. RADAR EMANOMETRY IN URANIUM EXPLORATION. MINING ENGINEERING, V. 26, NO. 5, MAY 1974, PP. 26-29.

36. CARTHEW, D. URANIUM UPDATE, EXPLODING DEMAND SPARKS A RUSH TO DEVELOP NEW RESERVES. ENG. AND MIN. J., V. 169, NO. 12, DECEMBER 1968, PP. 67-71.

37. CHAPPELEAR, J. E., AND W. L. ROGERS. SOME PRACTICAL CONSIDERATIONS IN THE CONSTRUCTION OF A SEMI-IMPLICIT SIMULATOR. SOC. OF PETROL. ENG. J., JUNE 1964, PP. 216-220.

38. CHEMICAL ENGINEERING. CHEMICAL ROUTE TO COPPER. APR. 20, 1970, PP. 64-66.

39. CHENOWETH, W. L. POTENTIAL RESOURCES OF URANIUM. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. I, NOVEMBER 1974, PP. 83-90.

40. COATES, J., AND B. S. PRESSBURG. EXAMINE LEACHING AND EXTRACTION. CHEM. ENG., V. 69, NO. 2, JAN. 22, 1962, PP. 125-130.

41. CORRICK, J. D., AND J. A. SUTTON. THREE CHEMOSYNTHETIC AUTOTROPHIC BACTERIA IMPORTANT TO LEACHING OPERATIONS AT ARIZONA COPPER MINES. BUMINES RI 5718, 1961, 8 PP.

42. CRICHLOW, H. B., AND P. J. ROOT. THE USE OF AN R-Z MODEL TO STUDY THE EFFECT OF COMPLETION TECHNIQUE ON GAS WELL DELIVERABILITY. SOC. OF PETROL. ENG. J., OCTOBER 1973, PP. 259-266.

43. CRIM, R. L. NUMERICAL METHOD FOR GROUNDWATER HYDRAULICS. U.S. ENVIRONMENTAL PROTECTION AGENCY TECH. REPT. 52, FEBRUARY 1972, 29 PP.; NTIS NO. PB 229 700.

44. CURFMAN. R. L. SOLUTION MINING PROJECT. MIN. CONG. J., V. 60, NO. 3, MARCH 1974, PP. 33-36.

45. DACOSTA. J. A., AND R. R. BENNETT. THE PATTERN OF FLOW IN THE VICINITY OF A RECHARGING AND DISCHARGING PAIR OF WELLS IN AN AQUIFER HAVING AREAL PARALLEL FLOW. INTERNAT. ASSOC. OF SCIENTIFIC HYDROLOGY, PUB. 52, 1960, PP. 524-536.

46. DALTON, R. L., JR., L. A. RAPOPORT, AND C. W. CARPENTER, JR. LABORATORY STUDIES OF PILOT WATER FLOODS. PRES. AT 34TH ANNUAL MEETING. SOC. PETROL. ENG., AIME, DALLAS, TEX., OCT. 4-7, 1959.; PETROL. TRANS., AIME, V. 219, 1960, PP. 24-30.

47. DASHER, J. CHEMICAL PROCESSING, A WAVE OF THE FUTURE. MINING ENGINEERING, V. 23, NO. 4, APRIL 1971, PP. 48-51.

48. DASHER, J. A CONCEPTUAL DESIGN FOR THE LEACHING OF SULPHIDIC URANIUM ORES WITH REAGENTS. MINERALS SCIENCE AND ENGINEERING, V. 4, NO. 4, OCTOBER 1972, PP. 28-35.

49. DAVIS. J. G., AND D. A. SHOCK. SOLUTION MINING OF THIN BEDDED POTASH. TRANS. SOC. OF MIN. ENG., AIME, V. 247, MARCH 1970. PP. 94-96.; MINING ENGINEERING, V. 22, NO. 7, JULY 1970, PP. 106-109.

50. DE CUYPER. J. A. BACTERIAL LEACHING OF LOW GRADE COPPER AND COBALT ORES. CH. IN UNIT PROCESSES IN HYDROMETALLURGY, ED. BY M. E. WADSWORTH AND F. T. DAVIS. GORDON AND BREACH, NEW YORK, 1964, PP. 126-142.

51. DE LA PENA PORTH, L., AND L. ESPINOSA DE LEON. NEW PROCESS FOR THE RECOVERY OF HIGH GRADE COPPER FROM AMMONICAL SOLUTIONS. PRES. AT FALL MEETING, SOC. OF MIN. ENG., AIME, ACAPULCO, MEXICO, SEP. 22-25, 1974, AIME PREPRINT 74-B-342, 1974, 16 PP.

52. DENAULT, K. J. A SELECTED ANNOTATED BIBLIOGRAPHY PERTINENT TO THE EXPLORATION OF URANIUM IN THE WIND RIVER BASIN, WYOMING. AUG. 8, 1970, 3 PP.; AVAILABLE FOR CONSULTATION AT BUREAU OF MINES, TWIN CITIES MINING RESEARCH CENTER, MINNEAPOLIS, MINNESOTA.

53. DESESA, M. A., P. J. MAGNO, H. E. GARDNER, AND E. DICKERMAN. CUPRIC AMMONIUM REAGENT AS AN OXIDANT IN THE CARBONATE LEACHING OF URANIFEROUS ORES. PROC. OF SECOND UNITED NATIONS INTERNAT. CONF. ON THE PEACEFUL USES OF ATOMIC ENERGY, GENEVA, SEP. 1-13, 1958 (PUB. AS PROCESSING OF RAW MATERIALS), V. 3, PP. 342-345.

54. DOYLE, R. E., AND T. M. WURL. STREAM CHANNEL CONCEPT APPLIED TO WATERFLOOD PERFORMANCE CALCULATIONS FOR MULTIWELL, MULTIZONE, THREE-COMPONENT CASES. J. OF PETROL. TECH., MARCH 1971, PP. 373-380.

55. DUMORE, J. M., AND D. MARSAL. THE USE OF INTEGRAL EQUATIONS IN THE NUMERICAL SOLUTION OF COMPLEX RESERVOIR ENGINEERING PROBLEMS. SOC. OF PETROL. ENG. J., DECEMBER 1973, PP. 321-327.

56. DUNCAN, D. W., AND A. D. DRUMMOND. MICROBIOLOGICAL LEACHING OF PORPHYRY COPPER TYPE MINERALIZATION: POST-LEACHING OBSERVATIONS. CAN. J. EARTH SCIENCE, V. 10, 1973, PP. 476-484.

57. DUNCAN, D. W., P. C. TRUSSELL, AND C. C. WALDEN. LEACHING OF CHALCOPYRITE WITH THIOBACILLUS FERROOXIDANS: EFFECT OF SURFACTANTS AND SHAKING. APPLIED MICROBIOLOGY, V. 12, 1964, PP. 122-126.

58. EDMONDS, B. P. SIGNIFICANCE OF SOLUTION MINING TO THE POTASH INDUSTRY. CAN. MIN. AND MET. BULL., SEPTEMBER 1964, PP. 953-955.

59. ELLIS, J. R. A SUBJECTIVE PROBABILITY STUDY OF URANIUM RESOURCES IN THE STATE OF NEW MEXICO. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. I, NOVEMBER 1974, PP. 91-113.

60. ENGINEERING AND MINING JOURNAL. ANACONDA AIMS AT MORE CEMENT COPPER. V. 166, NO. 4, APRIL 1965, PP. 84-85.

61. ENGINEERING AND MINING JOURNAL. CANE CREEK TO CONVERT TO SOLUTION MINING. V. 171, NO. 9, SEPTEMBER 1970, P. 171.

62. ENGINEERING AND MINING JOURNAL. CONQUISTA MINES WIDE - RANGING SOUTH TEXAS URANIUM DEPOSITS. V. 173, NO. 8, AUGUST 1972, PP. 74-77.

63. ENGINEERING AND MINING JOURNAL. IRON, COPPER AND URANIUM LEACHING FLOWSHEETS. V. 170, NO. 6, JUNE 1969, (COPPER), P. 195, (IRON), PP. 180-181, (URANIUM), P. 201.

64. ENGINEERING AND MINING JOURNAL. A NEW WAY TO TUNGSTEN. V. 171, NO. 7, JULY 1970, PP. 81-86.

65. ENGINEERING AND MINING JOURNAL. TRENDS IN CHEMICAL PROCESSING AND HYDROMETALLURGY. V. 173, NO. 6, JUNE 1972, PP. 173-177.

66. ENGINEERING AND MINING JOURNAL. URANIUM CANADA. V. 170, NO. 9, SEPTEMBER 1969, P. 143.

67. EUROPEAN NUCLEAR ENERGY AGENCY AND INTERNATIONAL ATOMIC ENERGY AGENCY. URANIUM: TWO SURVEYS OF THE SITUATION UP TO 1980. NUCLEAR ENG. INTERNAT., V. 14, NO. 156, MAY 1969, PP. 426-427.

68. EVANS, L. G., AND H. W. SHEFFER. DUMP AND HEAP LEACHING PRACTICES IN THE WESTERN UNITED STATES. MIN. CONG. J., V. 54, NO. 9, SEPTEMBER 1963, PP. 96-100.

69. FACER, J. F. PRODUCTION STATISTICS (URANIUM). AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. I, NOVEMBER 1974, PP. 132-135.

70. FATT, I., AND D. H. DAVIS. REDUCTION IN PERMEABILITY WITH OVERBURDEN PRESSURE. PETROL. TRANS., AIME, V. 195, 1952, P. 329.

71. FINCH, W. I. GEOLOGY OF THE SHINARUMP NO. 1 URANIUM MINE. SEVEN MILE CANYON AREA, GRAND COUNTY, UTAH. U.S. GEOLOGICAL SURVEY CIRC. 336, 1954, 14 PP.

72. FINCH, W. I. UNITED STATES GEOLOGICAL SURVEY URANIUM AND THORIUM RESOURCE ASSESSMENT AND EXPLORATION RESEARCH PROGRAM, FY 1975. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. I, NOVEMBER 1974, 9 PP.

73. FISHER, D. G., R. G. MCINTOSH, R. L. EAGER, AND A. B. VANCLEAVE. FACTORIAL DESIGN IN THE STUDY OF ACID LEACHING OF PEGMATITIC URANIUM ORES. CAN. J. OF CHEM. ENG., V. 39, JUNE 1961, PP. 139-144.

74. FLOH, B., A. ABRAO, AND E. C. COSTA. AMINE EXTRACTION OF URANIUM. MOLYBDENUM AND VANADIUM FROM ACID LEACH LIQUORS. PROC. INTERNAT. ATOMIC ENERGY AGENCY SYMP. ON RECOVERY OF URANIUM FROM ITS ORES AND OTHER SOURCES, SAO PAULO, BRAZIL, AUG. 17-21, 1970, IAEA-SM-135/18, 1971, PP. 267-285.

75. GORDON, E. URANIUM--PLANT EXPANSION SHOULD START NOW TO MEET PROJECTED DEMAND IN 1975. ENG. AND MIN. J., V. 174, NO. 3, MARCH 1973, PP. 125-129.

76. GORDON, E. URANIUM--PRICES FIRM UP IN 74; NEW FOREIGN POLICIES AFFECT INVESTMENT. ENG. AND MIN. J., V. 176, NO. 3, MARCH 1975, PP. 213-217.

77. GOW, W. A., H. H. MCCREEDY, G. M. RITCEY, V. M. MCNAMARA, V. F. HARRISON, AND B. H. LUCAS. BACTERIA-BASED PROCESSES FOR THE TREATMENT OF LOW-GRADE URANIUM ORES. PROC. INTERNAT. ATOMIC ENERGY AGENCY SYMP. ON RECOVERY OF URANIUM FROM ITS ORES AND OTHER SOURCES, SAO PAULO, BRAZIL, AUG. 17-21, 1970, IAEA-SM-135/3, 1971, PP. 195-211.

78. GRAVES, J. A. OPEN PIT URANIUM MINING. MINING ENGINEERING, V. 26, NO. 8, AUGUST 1974, PP. 23-25.

79. GRINGARTEN, A. C., AND H. J. RAMEY. THE USE OF SOURCE AND GREENS FUNCTIONS IN SOLVING UNSTEADY-FLOW PROBLEMS IN RESERVOIRS. SOC. OF PETROL. ENG. J., OCTOBER 1973, PP. 285-295.

80. GUAY, W. J., AND D. G. PETERSON. RECOVERY OF GOLD FROM CARBONACEOUS ORES AT CARLIN, NEVADA. TRANS. SOC. OF MIN. ENG., AIME, V. 254, NO. 1, MARCH 1973, PP. 102-104.

81. GUCCIONE, E. FUEL SHORTAGES TRIGGER A NEW URANIUM RUSH IN NEW MEXICO. MINING ENGINEERING, V. 26, NO. 8, AUGUST 1974, PP. 16-19.

82. HABASHI. F. PRESSURE HYDROMETALLURGY: KEY TO BETTER AND NONPOLLUTING PROCESS. ENG. AND MIN. J., V. 172, NO. 5, MAY 1971, PP. 89-94.

83. HANRAHAN, E. J. NUCLEAR POWER GROWTH ENRICHMENT AND URANIUM REQUIREMENTS 1974 TO YEAR 2000. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. I, NOVEMBER 1974, PP. 7-37.

84. HARSHMAN, E. N. GENETIC IMPLICATIONS OF SOME ELEMENTS ASSOCIATED WITH URANIUM DEPOSITS, SHIRLEY BASIN, WYOMING. U.S. GEOLOGICAL SURVEY PROF. PAPER, MAR. 31, 1969, PP. C167-C173.

85. HARSHMAN, E. N. GEOLOGICAL MAP OF THE SHIRLEY BASIN AREA, ALBANY, CARBON, CONVERSE AND NATRONA COUNTIES, WYOMING. U.S. GEOLOGICAL SURVEY MAP I-539, 1968.

86. HARSHMAN, E. N. PALEOTOPOGRAPHIC CONTROL OF A URANIUM MINERAL BELT, SHIRLEY BASIN, WYOMING. U.S. GEOLOGICAL SURVEY PROF. PAPER 424-C, 1961, PP. C4-C6.

87. HENRY, H. R. SALT INTRUSION INTO COASTAL AQUIFERS. INTERNAT. ASSOC. OF SCIENTIFIC HYDROLOGY, PUB. 52, 1960, PP. 478-487.

88. HERBERT. R. ANALYSING PUMPING TESTS BY RESISTANCE NETWORK ANALOGUE. GROUND WATER, V. 6, NO. 2, MARCH-APRIL 1968, PP. 12-18.

89. HILL, R. D., AND R. C. WILMOTH. LIMESTONE TREATMENT OF ACID MINE DRAINAGE. TRANS. SOC. OF MIN. ENG., AIME, V. 250, JUNE 1971, PP. 162-166.

90. HOSTETLER, P. B., AND R. M. GARRELS. TRANSPORTATION AND PRECIPITATION OF URANIUM AND VANADIUM AT LOW TEMPERATURES WITH SPECIAL REFERENCE TO SANDSTONE-TYPE URANIUM DEPOSITS. ECON. GEOL., V. 57, MARCH 1962, PP. 137-167.

91. HOUK, L. G. MONAZITE SAND. BUMINES IC 7233, FEBRUARY 1943, 19 PP.

92. HOWARD, E. V. CHINO USES RADIATION LOGGING FOR STUDYING LEACHING PROCESSES. MINING ENGINEERING, V. 20, NO. 4, APRIL 1968, PP. 70-74.

93. HURST, W. THE SOLUTION OF NONLINEAR EQUATIONS. SOC. OF PETROL. ENG. J., DECEMBER 1973, PP. 348-356.

94. JACKS, H. H., O. J. E. SMITH, AND C. C. MATTAX. THE MODELING OF A THREE-DIMENSIONAL RESERVOIR WITH A TWO-DIMENSIONAL RESERVOIR SIMULATOR-THE USE OF DYNAMIC PSEUDO FUNCTIONS. SOC. OF PETROL. ENG. J., JUNE 1973, PP. 175-185.

95. JACKSON, D., JR. SOLUTION MINING PUMPS NEW LIFE INTO CAN CREEK POTASH MINE. ENNG. AND MIN. J., V. 174, NO. 7, JULY 1973, PP. 59-69.

96. JACOBSON, R. H., C. R. MCKEE, AND R. B. BHAPPU. COMPUTER SIMULATION OF FLUID FLOW IN A LEACH DUMP OR HEAP. SOC. OF MIN. ENG., AIME, SAN FRANCISCO, CALIF., FEB. 20-24, 1972, AIME PREPRINT 72-AS-56, 15 PP.

97. JAVANDEL, I., AND P. A. WITHERSPOON. APPLICATION OF THE FINITE ELEMENT METHOD TO TRANSIENT FLOW IN POROUS MEDIA. SOC. OF PETROL. ENG. J., SEPTEMBER 1968, PP. 241-252.

98. KAISER, E. P. URANIFEROUS QUARTZITE, RED BLUFF PROJECT GILA COUNTY, ARIZONA. U.S. GEOLOGICAL SURVEY CIRC. 137, OCTOBER 1951, 10 PP.

99. KENNY, H. C., AND H. A. ABRAMSON. AMMONIACAL LEACHING PROCESS. U.S. PAT. 2,687,953, AUG. 31, 1954, 6 PP.

100. KING, J. W., AND S. R. AUSTIN. SOME CHARACTERISTICS OF ROLL-TYPE URANIUM DEPOSITS AT GAS HILL, WYOMING. MINING ENGINEERING, V. 18, NO. 5, MAY 1966, PP. 73-80.

101. KIPP, K. L. UNSTEADY FLOW TO A PARTIALLY PENETRATING, FINITE RADIUS WELL IN AN UNCONFINED AQUIFER. WATER RESOURCES RESEARCH, V. 9, NO. 2, APRIL 1973, PP. 448-462.

102. KLEMENIC, J. AN ESTIMATE OF THE ECONOMICS OF URANIUM CONCENTRATE PRODUCTION FROM LOW GRADE SOURCES. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. II, NOVEMBER 1974, PP. 5-38.

103. KUNIN, R., R. L. GUSTAFSON, E. G. ISACOFF, AND H. F. FILLING. ION EXCHANGE RESINS FOR URANIUM HYDROMETALLURGY. ENG. AND MIN. J., V. 170, NO. 7, JULY 1969, PP. 73-79.

104. LACKEY, J. A. ENVIRONMENTAL ASPECTS OF URANIUM EXTRACTION. AMDEL BULL., (THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES), NO. 19, APRIL 1975, PP. 11-20.

105. LOVE, J. D. PRELIMINARY REPORT ON URANIUM IN THE GAS HILLS AREA, FREMONT AND NATRONA COUNTIES, WYOMING. U.S. GEOLOGICAL SURVEY CIRC. 352, 1954, 11 PP.

106. LOVE, J. D. URANIUM IN THE MAYOWORTH AREA, JOHNSON COUNTY, WYOMING--A PRELIMINARY REPORT. U.S. GEOLOGICAL SURVEY CIRC. 358, 1954, 7 PP.

107. MACDONALD, R. C., AND K. H. COATS. METHODS FOR NUMERICAL SIMULATION OF WATER AND GAS CONING. SOC. OF PETROL. ENG. J., DECEMBER 1970, PP. 425-435.

108. MADSEN, B. W., AND R. D. GROVES. USING OXYGEN TO REACTIVATE A NEARLY DORMANT COPPER SULFIDE LEACH. BUMINES RI 8056, 1975, 9 PP.

109. MALOUF, E. E. BACTERIAL LEACHING. PART IV OF SECOND TUTORIAL SYMP. ON EXTRACTIVE METALLURGY, UNIV. UTAH, DEC. 14-16, 1972, 46 PP.

110. MALOUF, E. E. LEACHING AS A MINING TOOL. CH. IN INTERNATIONAL SYMP. ON HYDROMETALLURGY, CHICAGO, ILLINOIS, FEB. 25 - MAR. 1, 1973, ED. BY D. J. I. EVANS AND R. S. SHOEMAKER. THE AMERICAN INSITTUTE OF MINING, METALLURGICAL AND PETROLEUM ENGINEERS, INC., NEW YORK, 1973, PP. 615-626.

111. MALOUF, E. E., AND J. D. PRATER. NEW TECHNOLOGY OF LEACHING WASTE DUMPS. MIN. CONG. J., V. 48, NO. 11, NOVEMBER 1962, PP. 82-85.

112. MALOUF, E. E., AND J. D. PRATER. ROLE OF BACTERIA IN THE ALTERATION OF SULFIDE MINERALS. J. OF METALS, V. 13, MAY 1961, PP. 353-356.

113. MALOUF, E. E., M. E. WADSWORTH, AND Y. T. AUCK. PROC. SECOND TUTORIAL SYMPOSIUM ON EXTRACTIVE METALLURGY: LEACHING OF LOW AND HIGH GRADE ORES, DUMPS, DEPOSITS AND CONCENTRATES. UNIV. UTAH, DEC. 14-16, 1972, 233 PP.

114. MASHBIR, D. S. HEAP LEACHING OF LOW GRADE URANIUM ORE. MIN. CONG. J., V. 50, NO. 12, DECEMBER 1964, PP. 50-54.

115. MATIC, M., AND M. MROST. IN SITU LEACHING OF URANIUM FROM GOLD MINE RESIDUE DAMS. THE SOUTH AFRICAN INDUSTRIAL CHEMIST, V. 18, OCTOBER 1964, PP. 127-138.

116. MATTHEWS, C. S., AND M. J. FISCHER. EFFECT OF DIP ON FIVE-SPOT SWEEP PATTERN. PETROL. TRANS., AIME, V. 207, 1956, PP. 111-117.

117. MCCREEDY, H. H., W. A. GOW, F. J. KELLY, AND H. W. SMITH. FURTHER STUDIES OF THE LEACHING OF URANIUM ORES FROM ELLIOT LAKE, ONTARIO. MINES BRANCH, CAN. DEPT. OF ENERGY, MINES AND RESOURCES, OTTAWA, TECH. BULL. 112, SEPTEMBER 1969, 16 PP.

118. MCCREEDY, H. H., V. F. HARRISON, AND W. A. GOW. A PROPOSED METHOD, USING BACTERIA FOR THE CONTINUOUS LEACHING OF A URANIUM ORE. CAN. MIN. AND MET. BULL., FEBRUARY 1969, PP. 135-140.

119. MCGARR, H. J. LIQUID ION-EXCHANGE RECOVERS COPPER FROM WASTES AND LOW-GRADE ORES. ENG. AND MIN. J., V. 171, NO. 10, OCTOBER 1970, PP. 79-81.

120. MCGARR, H. J., N. H. BERLIN, AND W. F. A. STOLK. THE COST OF COPPER...SOLVENT EXTRACTION AND ELECTROWINNING LOOK GREAT ON PAPER. ENG. AND MIN. J., V. 170, NO. 12, DECEMBER 1969, PP. 66-67.

121. MCLEAN, D. C. CATALYTIC OXIDATION OF ORES. U.S. PAT. 3,017,241, JAN. 16, 1962, 10 PP.

122. MCMICHAEL, C. L., AND G. W. THOMAS. RESERVOIR SIMULATION BY GALERKINS METHOD. SOC. OF PETROL. ENG. J., JUNE 1973, PP. 125-136.

123. MCQUISTON, F. W., JR., AND R. HERNLUND. NEWMONTS CARLIN GOLD PROJECT. MIN. CONG. J., V. 51, NO. 11, NOVEMBER 1965, PP. 26-39.

124. MEEHAN, R. J. EXPLORATION PLANS AND TRENDS. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. I, NOVEMBER 1974, PP. 114-131.

125. MELIN, R. E. DESCRIPTION AND ORIGIN OF URANIUM DEPOSITS IN SHIRLEY BASIN, WYOMING. ECON. GEOL., V. 59, 1964, PP. 835-849.

126. MILLER, J. S., AND H. R. NICHOLLS. METHODS AND EVALUATION OF EXPLOSIVE FRACTURING IN OIL SHALE. BUMINES RI 7729, 1973, 22 PP.

127. MILLER, J. S., C. J. WALKER, AND J. L. EAKIN. FRACTURING OIL SHALES WITH EXPLOSIVES FOR IN SITU OIL RECOVERY. BUMINES RI 7874, 1974, 100 PP.

128. MILLER, R. P., E. NAPIER, AND R. A. WELLS. NATURAL LEACHING OF URANIUM ORES, PART I - PRELIMINARY TESTS ON PORTUGUESE ORES. PRES. GENERAL MEETING OF THE INSTITUTION OF MIN. AND MET., FEB. 21, 1963, PP. 217-234.

129. MILLER, R. P., E. NAPIER, R. A. WELLS, A. AUDSLEY, AND G. R. DABORN. NATURAL LEACHING OF URANIUM ORES, PARTS I, II AND III. PRES. GENERAL MEETING OF THE INSTITUTION OF MIN. AND MET., FEB. 21, 1963, PP. 217-254.

130. MINING ENGINEERING. ANACONDAS RESIN-IN-PULP PROCESS: ANOTHER ROUTE TO YELLOWCAKE. V. 26, NO. 8, AUGUST 1974, PP. 31-33.

131. MINING ENGINEERING. CONQUISTA, CONOCO-PIONEER U3O8 VENTURE ON STREAM. V. 24, NO. 8, AUGUST 1972, PP. 37-41.

132. MINING ENGINEERING. HYDROMETALLURGY IS KEY IN WINNING U3O8. V. 26, NO. 8, AUGUST 1974, PP. 26-27.

133. MINING ENGINEERING. KERR-MCGEES AMBROSIA COMPLEX: FROM MINED ROCK TO YELLOWCAKE. V. 26, NO. 8, AUGUST 1974, PP. 28-30.

134. MINING ENGINEERING. MINERALIZATION AND GEOLOGY OF THE URANIUM CAPITAL OF THE WORLD. V. 26, NO. 8, AUGUST 1974, PP. 20-21.

135. MINING ENGINEERING. UNITED NUCLEAR-HOMESTAKE PARTNERS RECOVER U3O8 VIA ALKALINE LEACHING. V. 26, NO. 8, AUGUST 1974, PP. 34-36.

136. MOORE, G. W., AND M. LEVISH. URANIUM BEARING SANDSTONE IN THE WHITE RIVER BADLANDS, PENNINGTON COUNTY, SOUTH DAKOTA. U.S. GEOLOGICAL SURVEY CIRC. 359, 7 P.

137. MOORE, G. W., AND J. G. STEPHENS. RECONNAISANCE FOR URANIUM-BEARING CARBONACEOUS ROCKS IN CALIFORNIA AND ADJACENT PARTS OF OREGON AND NEVADA. U.S. GEOLOGICAL SURVEY CIRC. 313, 8 PP.

138. MOURET, P., P. POTTIER, P. SOUDAN, AND J. LE BRIS. APPLICATION OF ALKALINE LEACHING TO THE EXTRACTION OF URANIUM FROM SHALE OF THE VOSGES. PROC. OF THE SECOND UNITED NATIONS INTERNAT. CONF. ON THE PEACEFUL USES OF ATOMIC ENERGY, GENEVA, SEP. 1-13, 1958 (PUB. AS PROCESSING OF RAW MATERIALS), V. 3, PP. 356-362.

139. MUNOZ-REBADENEIRA, F. J., AND H. J. GOMBERG. LEACHING OF CHALCOPYRITE (CUFES2) WITH SODIUM CHLORIDE SULFURIC ACID SOLUTIONS. NUCLEAR TECHNOLOGY, V. 11, JULY 1971, PP. 367-371.

140. NOLEN, J. S., AND D. W. BERRY. TESTS OF THE STABILITY AND TIME-STEP SENSITIVITY OF SEMI-IMPLICIT RESERVOIR SIMULATION TECHNIQUES. SOC. OF PETROL. ENG. J., JUNE 1972, PP. 253-266.

141. NOVIKOV. E. N. REV. OF MINING BY UNDERGROUND LEACHING. BY A. I. KALABIN. SOVIET MINING SCIENCE, V. 7, NO. 5, MAY-JUNE 1971, PP. 380-381.

142. OIL AND GAS JOURNAL. WORLDWIDE ENERGY OUTLOOK TO 1985. NOV. 12, 1973, PP. 119-132.

143. PARKS, G. A., AND R. E. BAKER. MERCURY PROCESS. U.S. PAT. 3,476,552, NOV. 4, 1969, 6 PP.

144. PARKS, G. A., AND N. A. FITTINGHOFF. MERCURY EXTRACTIONS NOW POSSIBLE VIA HYPOCHLORITE LEACHING. ENG. AND MIN. J., V. 171, NO. 6, JUNE 1970, PP. 107-109.

145. PARSONS, R. W. DIRECTIONAL PERMEABILITY IN DEVELOPED AND UNCONFINED FIVE-SPOTS. J. OF PETROL. TECHNOLOGY, APRIL 1972, PP. 487-494.

146. PATTERSON, J. A. FOREIGN PRODUCTION CAPABILITY AND SUPPLY. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. II, NOVEMBER 1974, PP. 63-71.

147. PATTERSON, J. A. URANIUM SALES AND COMMITMENTS. AEC URANIUM INDUSTRY SEMINAR, GRAND JUNCTION, COLO., OCT. 22-23, 1974, V. I, NOVEMBER 1974, PP. 38-52.

148. PERKINS, E. C., AND F. NOVIELLI. BACTERIAL LEACHING OF MANGANESE ORES. BUMINES RI 6102, 1962, 11 PP.

149. PHILIP, J. R. FLOW IN POROUS MEDIA. ANN. REV. FLUID MECH., V. II, 1970, PP. 177-204.

150. PIRSON, S. J. BETTER PHOSPHATE MINING SUGGESTED. ENG. AND MIN. J., V. 160, NO.6, JUNE 1959, P. 208.

151. POLLARA, F. Z., N. LEVINE, J. R. KILLELEA, R. C. MUSA, AND M. D. HASSIALIS. RECOVERY OF URANIUM FROM CHATTANOOGA SHALE. SOURCE UNKNOWN, PP. 229-233.; FILE COPY AVAILABLE FOR CONSULTATION AT BUREAU OF MINES, TWIN CITIES MINING RESEARCH CENTER, MINNEAPOLIS, MINNESOTA.

152. POTTER, G. M. RECOVERING GOLD FROM STRIPPING WASTE AND ORE BY PERCOLATION CYANIDE LEACHING. BUMINES TPR 20, DECEMBER 1969, 5 PP.

153. PRATS, M., W. R. STRICKLER, AND C. S. MATTHEWS. SINGLE-FLUID FIVE-SPOT FLOODS IN DIPPING RESERVOIRS. PETROL. TRANS., AIME, V. 204, 1955, PP. 160-174.

154. PRATS, M. A METHOD FOR DETERMINING THE NET VERTICAL PERMEABILITY NEAR A WELL FROM IN SITU MEASUREMENTS. J. OF PETROL. TECHNOLOGY, MAY 1970, PP. 637-643.

155. PRATS, M., C. S. MATTHEWS, R. L. JEWETT, AND J. D. BAKER. PREDICTION OF INJECTION RATE AND PRODUCTION HISTORY FOR MULTIFLUID FIVE-SPOT FLOODS. PETROL. TRANS., AIME, V. 216, 1959, PP. 98-105.

156. PRICE, H. S., AND K. H. COATS. DIRECT METHODS IN RESERVOIR SIMULATION. SOC. OF PETROL. ENG. J., JUNE 1974, PP. 295-308.

157. RACKLEY, R. I., P. N. SHOCKEY, AND M. P. CAHILL. CONCEPTS AND METHODS OF URANIUM EXPLORATION. 20TH FIELD CONF., BLACK HILLS AREA, SEP. 23-25, 1968, WYOMING GEOL. ASSOC. GUIDEBOOK, PP. 115-124.

158. RAWLING, K. R. COMMERCIAL SOLVENT EXTRACTION PLANT RECOVERS COPPER FROM LEACH LIQUORS. WORLD MINING, V. 22, DECEMBER 1969, PP. 34-37.

159. RAZZELL, W. E. BACTERIAL LEACHING OF METALLIC SULPHIDES. TRANS. OF THE CAN. INST. MIN. MET., V. 65, 1962, PP. 136-137.

160. RAZZELL, W. E., AND P. C. TRUSSELL. MICROBIOLOGICAL LEACHING OF METALLIC SULFIDES. APPLIED MICROBIOLOGY, V. 11, 1963, PP. 105-110.

161. REYNOLDS, D. H., W. V. LONG, AND R. B. BHAPPU. WINNING METALLIC VALUES FROM LEACH SOLUTIONS BY SORPTION PROCESSES. TRANS. SOC. OF MIN. ENG., AIME, V. 235, DECEMBER 1966, PP. 355-360.

162. RITCEY, G. M., M. J. SLATER, AND B. H. LUCAS. A COMPARISON OF THE PROCESSING AND ECONOMICS OF URANIUM RECOVERY FROM LEACH SLURRIES BY CONTINUOUS ION EXCHANGE OR SOLVENT EXTRACTION. CH. IN INTERNAT. SYMP. ON HYDROMETALLURGY, CHICAGO, ILL., FEB. 25 - MAR. 1, 1973, ED. BY D. J. I. EVANS, AND R. S. SHOEMAKER. THE AMERICAN INSTITUTE OF MINING, METALLURGICAL AND PETROLEUM ENGINEERS, INC., NEW YORK, 1973, PP. 419-474.

163. ROBINSON, C. S., AND J. N. ROSHOLT, JR. URANIUM MIGRATION AND GEOCHEMISTRY OF URANIUM DEPOSITS IN SANDSTONE ABOVE, AT, AND BELOW THE WATER TABLE. PART II, RELATIONSHIP OF URANIUM MIGRATION DATES, GEOLOGY, AND CHEMISTRY OF THE URANIUM DEPOSITS. ECON. GEOL., V. 56, NO. 8, DECEMBER 1961, PP. 1404-1420.

164. ROBINSON, W. J. FINGER DUMP PRELIMINARIES PROMISE IMPROVED COPPER LEACHING AT BUTTE. MINING ENGINEERING, V. 24, NO. 9, SEPTEMBER 1972, PP. 47-49.

165. ROOTS, E. F. CERIUM AND THORIUM. WESTERN MINER, AUGUST 1946, PP. 50-56.

166. ROSENBAUM, J. B., AND D. R. GEORGE. COST REDUCTIONS IN ION EXCHANGE PROCESSING OF URANIUM ORES. PROC. INTERNAT. ATOMIC ENERGY AGENCY SYMP. ON RECOVERY OF URANIUM FROM ITS ORES AND OTHER SOURCES, SAO PAULO, BRAZIL, AUG. 17-21, 1970, IAEA-SM-135/24, 1971, PP. 297-309.

167. ROSENBAUM, M. J. F., AND C. S. MATTHEWS. STUDIES ON PILOT WATER FLOODING. PETROL. TRANS., AIME, V. 216, 1959, PP. 316-323.

168. ROSHOLT. J. N., JR. URANIUM MIGRATION AND GEOCHEMISTRY OF URANIUM DEPOSITS IN SANDSTONE ABOVE, AT, AND BELOW THE WATER TABLE. PART I, CALCULATION OF APPARENT DATES OF URANIUM MIGRATION IN DEPOSITS ABOVE AND AT THE WATER TABLE. ECON. GEOL., V. 56, NO. 8, DECEMBER 1961, PP. 1392-1403.

169. ROSHOLT. J. N., E. N. HARSHMAN, W. R. SHIELDS, AND E. L. GARNER. ISOTOPIC FRACTIONATION OF URANIUM RELATED TO ROLL FEATURES IN SANDSTONE, SHIRLEY BASIN, WYOMING. ECON. GEOL., V. 59, 1964, PP. 570-585.

170. SCHEIDEGGER, A. E. STATISTICAL HYDRODYNAMICS IN POROUS MEDIA. J. APPL. PHYS., V. 25, NO. 8, AUGUST 1974, PP. 994-1001.

171. SCHEIDEGGER, A. E. UNDERGROUND DISPERSION OF MISCIBLE LIQUIDS. INTERNAT. ASSOC. OF SCIENTIFIC HYDROLOGY, PUB. 52, 1960, PP. 462-469.

172. SCHICHT, R. J. GROUND WATER DEVELOPMENT IN EAST ST. LOUIS AREA, ILLINOIS. ILLINOIS STATE WATER SURVEY, 1965, 70 PP.

173. SCHORTMANN, W. E., AND M. A. DE SESA. KINETICS OF THE DISSOLUTION OF URANIUM DIOXIDE IN CARBONATE-BICARBONATE SOLUTIONS. PROC. OF SECOND UNITED NATIONS INTERNAT. CONF. ON THE PEACEFUL USES OF ATOMIC ENERGY, GENEVA, SEP. 1-13, 1958 (PUB. AS PROCESSING OF RAW MATERIALS), V. 3, PP. 333-341.

174. SCUFFHAM, J. B., AND G. A. ROWDEN. SOLVENT EXTRACTION OF METALS FROM AMMONIACAL SOLUTIONS. MINING ENGINEERING, V. 25, NO. 12, DECEMBER 1973, PP. 33-34.

175. SETTARI, A., AND K. AZIZ. A COMPUTER MODEL FOR TWO-PHASE CONING SIMULATION. SOC. OF PETROL. ENG. J., JUNE 1974, PP. 221-236.

176. SHERMAN. J. T. URANIUM. ENG. AND MIN. J., V. 170, NO. 3, MARCH 1969, PP. 104-108.

177. SHERMAN. J. T. URANIUM. ENG. AND MIN. J., V. 171, NO. 3, MARCH 1970, PP. 92-96.

178. SHERMAN. J. T. URANIUM. ENG. AND MIN. J., V. 172, NO. 3, MARCH 1971, PP. 108-111.

179. SHERMAN. J. T. URANIUM. ENG. AND MIN. J., V. 173, NO. 3, MARCH 1972, PP. 140-143.

180. SHOCK, D. A. USE OF HYDRAULIC FRACTURING TO MAKE A HORIZONTAL STORAGE CAVITY IN SALT. SECOND SYMP. ON SALT, CLEVELAND, OHIO, 1965, V. 2, PP. 406-411.

181. SHOCK, D. A., AND J. G. DAVIS. SOLUTION MINING TEST SITE - CARLSBAD BASIN, NEW MEXICO. THIRD SYMP. ON SALT, THE NORTHERN OHIO GEOLOGICAL SOCIETY INC., CLEVELAND, OHIO, 1970, PP. 433-438.

182. SHOEMAKER, R. S., AND R. M. DARRAH. THE ECONOMICS OF HEAP LEACHING. MINING ENGINEERING, V. 20, NO. 12, DECEMBER 1968, PP. 68-70.

183. SHUM, Y. M. USE OF THE FINITE ELEMENT METHOD IN THE SOLUTION OF DIFFUSION - CONVECTION EQUATIONS. SOC. OF PETROL. ENG. J., JUNE 1971, PP. 139-144.

184. SIERZPUTOWSKI, A. E. OXIDATIVE TREATMENT OF URANIUM ORE PRIOR TO ACID LEACH. U.S. PAT. 3,488,162, JAN. 6, 1970, 24 PP.

185. SILO, R. S. A NEW PROCESS FOR THE TREATMENT OF OXIDIZED COPPER ORE BY LEACHING. TRANS. SOC. OF MIN. ENG., AIME, V. 235, SEPTEMBER 1966, PP. 225-230.

186. SIMON, R., AND F. J. KELSEY. THE USE OF CAPILLARY TUBE NETWORKS IN RESERVOIR PERFORMANCE STUDIES: I. EQUAL-VISCOSITY MISCIBLE DISPLACEMENTS. SOC. OF PETROL. ENG. J., JUNE 1971, PP. 99-112.

187. SKIBITZKE, H. E. ELECTRONIC COMPUTERS AS AN AID TO THE ANALYSIS OF HYDROLOGIC PROBLEMS. INTERNAT. ASSOC. OF SCIENTIFIC HYDROLOGY, PUB. 52, 1960, PP. 347-358.

188. SMITH, S. E., AND P. A. WHITE. SOME RECENT DEVELOPMENTS IN URANIUM ORE PROCESSING RESEARCH IN THE UNITED KINGDOM. PROC. INTERNAT. ATOMIC ENERGY AGENCY SYMP. ON RECOVERY OF URANIUM FROM ITS ORES AND OTHER SOURCES, SAO PAULO, BRAZIL, AUG. 17-21, 1970, IAEA-SM-135/21, 1971, PP. 3-16.

189. SMITH, S. E., AND P. A. WHITE. URANIUM ORE PROCESSING. ENG. AND MIN. J., V. 170, NO. 6, JUNE 1969, PP. 113-115.

190. SONIER, F., P. BESSET, AND O. OMBRET. A NUMERICAL MODEL OF MULTIPHASE FLOW AROUND A WELL. SOC. OF PETROL. ENG. J., DECEMBER 1973, PP. 311-320.

191. SOONAWALA, N. M. DATA PROCESSING TECHNIQUES FOR THE RADON METHOD OF URANIUM EXPLORATION. CAN. MIN. AND MET. BULL., APRIL 1974, PP. 110-116.

192. SPEDDEN. H. R. THE IMPACT OF ENVIRONMENTAL CONTROLS ON NONFERROUS METALS EXTRACTION. PRES. AT AMERICAN MIN. CONG., DENVER, COLO., SEP. 29, 1970, 20 PP.

193. STEPHENS, D. R. FRENCH-AMERICAN TECHNICAL EXCHANGE OF GEOLOGIC INFORMATION RELATED TO PEACEFUL USES OF NUCLEAR EXPLOSIVES. LAWRENCE LIVERMORE LABORATORY, REPT. UCRL-51171, JAN. 11, 1972, 14 PP.

194. SULLIVAN, J. D., AND K. O. BAYARD. EXTRACTION OF SOLUBLE COPPER FROM ORES IN LEACHING BY PERCOLATION. BUMINES RI 3073, APRIL 1931, 43 PP.

195. SUTTON, J. A., AND J. D. CORRICK. MICROBIAL LEACHING OF COPPER MINERALS. MINING ENGINEERING, V. 15, NO. 6, JUNE 1963, PP. 37-40.

196. TODD, M. R., P. M. O DELL, AND G. J. HIRASAKI. METHODS FOR INCREASED ACCURACY IN NUMERICAL RESERVOIR SIMULATORS. SOC. OF PETROL. ENG. J., DECEMBER 1972, PP. 515-530.

197. TROYER, M. L., E. J. MCKAY, P. E. SOISTER, AND S. R. WALLACE. SUMMARY OF INVESTIGATIONS OF URANIUM DEPOSITS IN THE PUMPKIN BUTTE AREA, JOHNSON AND CAMPBELL COUNTIES, WYOMING. U.S. GEOLOGICAL SURVEY CIRC. 338, 17 PP.

198. TRUMP, E. N. MINING SOLUBLE SALINES BY WELLS. MINING TECHNOLOGY, V. 8, NO. 3, MAY 1944, 7 PP.

199. U.S. ATOMIC ENERGY COMMISSION. NUCLEAR FUEL SUPPLY. AEC REPT. WASH-1242, MAY 1973, 11 PP.

200. U.S. ENVIRONMENTAL PROTECTION AGENCY. METHODS FOR CHEMICAL ANALYSIS OF WATER AND WASTES (16020-07/71), 1971. WATER QUALITY OFFICE, ANALYTICAL QUALITY CONTROL LABORATORY, CINCINNATI, OHIO, PP. 1-11, 78-130, 275-285.

201. VAN WAMBEKE, L. APPLICATION OF X RAYS TO THE INVESTIGATION OF RADIOACTIVE MINERALIZATIONS. PROC. OF SECOND INTERNAT. CONF. ON THE PEACEFUL USES OF ATOMIC ENERGY, GENEVA, SEP. 1-13, 1958 (PUB. AS PROCESSING OF RAW MATERIALS), V. 3, PP. 541-549.

202. VICKERS, R. C. OCCURRENCES OF RADIOACTIVE MINERALS IN THE BALD MOUNTAIN GOLD-MINING AREA, NORTHERN BLACK HILLS, SOUTH DAKOTA. U.S. GEOLOGICAL SURVEY CIRC. 351, 1954, 8 PP.

203. WADSWORTH, M. E. ADVANCES IN THE LEACHING OF SULPHIDE MINERALS. MINERALS SCIENCE AND ENGINEERING, V. 4, NO. 4, OCTOBER 1972, PP. 36-47.

204. WADSWORTH, M. E. ADVANCES IN THE LEACHING OF SULFIDE AND OXIDE MINERALS. PART III OF SECOND TUTORIAL SYMP. ON EXTRACTIVE METALLURGY, UNIV. UTAH, DEC. 14-16, 1972, 69 PP.

205. WADSWORTH, M. E. RATE PROCESSES IN HYDROMETALLURGY. PART II OF SECOND TUTORIAL SYMP. ON EXTRACTIVE METALLURGY, UNIV. UTAH, DEC. 14-16, 1972, 119 PP.

206. WALTON, W. C., AND T. A. PRICKETT. HYDROGEOLOGIC ELECTRIC ANALOG COMPUTERS. J. OF THE HYDRAULIC DIVISION, AMER. SOC. OF CIVIL ENG., V. 89, NO. HY6, NOVEMBER 1963, PP. 67-91.

207. WARREN, I. H., A. VIZSOLYI, AND F. A. FORWARD. THE PRETREATMENT AND LEACHING OF CHALCOPYRITE. CAN. MIN. AND MET. BULL., V. 61, NO. 673, MAY 1968, PP. 637-640.

208. WILMARTH, V. R. YELLOW CANARY URANIUM DEPOSITS, DAGGET COUNTY, UTAH. U.S. GEOLOGICAL SURVEY CIRC. 312, 1953, 8 PP.

209. WILSEY, L. E., AND W. G. BEARDEN. RESERVOIR FRACTURING - A METHOD OF OIL RECOVERY FROM EXTREMELY LOW PERMEABILITY FORMATIONS. TRANS. AIME, V. 201, 1954, PP. 169-175.

210. WILSON, J. W., AND H. D. S. MILLER. DRILLING AND BLASTING IN THIN TABULAR STOPES AT DEPTH IN HARD ROCK. TRANS. AIME, SECTION A, JULY 1973, PP. A101-A114.

211. WORLD MINING. RANCHERS RECOVERS BLUEBIRD OXIDE COPPER BY SOLVENT EXTRACTION. APRIL 1969, PP. 63, 79.

212. WORLD MINING. URANIUM ORE DEPOSITS: HOW THEY WERE FORMED, WHERE THEY ARE FOUND. JULY 1974, PP. 40-47.

213. WYMAN, R. V. DISTRIBUTION OF GROUNDWATER U308 ON THE COLORADO PLATEAU. MINING ENGINEERING, V. 22, NO. 12, DECEMBER 1970, PP. 39-40.

214. ZELLER, H. D. RECONNAISSANCE FOR URANIUM-BEARING CARBONACEOUS MATERIALS IN SOUTHERN UTAH. U.S. GEOLOGICAL SURVEY CIRC. 349, 1955, 9 PP.

215. ZELLER, H. D. URANIUM-BEARING COPPER DEPOSITS IN THE COYOTE DISTRICT, MORA COUNTY, NEW MEXICO. U.S. GEOLOGICAL SURVEY CIRC. 334, 1954, 11 PP.

by DONALD H. DAVIDSON
Assistant Manager, Project Development, Energy Systems Planning Division, TRW Inc

In-Situ Leaching of Nonferrous Metals

In-situ leaching (ISL) of minerals has the potential of opening up resources which currently are uneconomical to mine by conventional methods with less disturbance to the local environment. The basic principle involves the recovery of metal values by transporting fluids through rock, as contrasted with mining and processing rock by conventional mining. This type of process is best suited for minerals that are deep-lying, lower grade and water saturated, with high flow conductivity and mineral contact with water.

Significant progress has been made over the past decade in the development of ISL technology for recovery of uranium and copper. Specific activities over this time period include:

- Commercialization of uranium production in south Texas.[1]
- Pilot operations for uranium in Wyoming and Colorado.[1]
- Kennecott's development project for copper sulfide leaching in Arizona.[2]
- Occidental Minerals development of copper oxide leaching in Arizona.[3]

The purpose of these remarks is to review technical and economic aspects of the ISL process and briefly discuss three of the key engineering tasks: selection of metal concentrations, oxygen injection (uranium and sulfide operations), and well pattern design considerations.

An ISL operation consists of surface and sub-surface facilities (fig. 1). Summarizing the major activities:

- Chemicals required to dissolve and maintain metals in solution are first processed through the surface facility.
- A set of injection wells is used to force the solvent into the pores or fractures of the rock by using a pressure in excess of the hydrostatic pressure in the deposit.
- Solvent travels through the rock and reacts with the solid mineral, transferring the metal value to the liquid phase.
- A set of production wells is used to create a low pressure sump where the metal enriched solution can be collected for transport to the surface.
- The produced solutions are processed for both metal recovery and solution make-up for an additional trip through the pores in the rock.

Successful implementation of these five activities requires an integration of mineral geology, and oil field and extractive mineral technologies. Although specific technical aspects of ISL can vary widely, uranium versus copper leaching, or wellfield operations ranging from massive rubblization of rock to water flooding deposits with natural permeability, most ISL systems have in common the following (fig. 2):

- The majority of the capital investment is tied up in the surface facility, where technology is somewhat standard and controlled, unlike the sub-surface component.
- A number of sub-surface mining strategies are likely to achieve a given production capacity.
- Limited or no prior industrial experience exists.

Therefore, it is important to develop mining strategies which minimize risk, and this requires understanding the interactions of key technical and economic parameters.

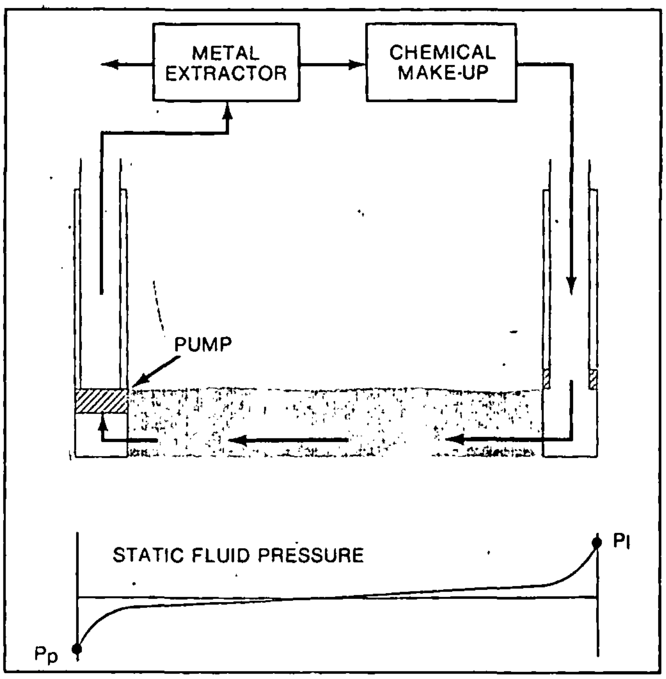


Fig. 1. Overview of in situ leaching

Solution volumes, flow rates determine performance

The economic performance of an in-situ operation is governed by the following:

- The volume of solution processed through the system.
- Rates of fluid injection and production from wells.
- The rate of mineral solubilization in the liquid phase.
- The volume of rock swept by the fluid.

Relating the above to site-specific capital investments and operating costs requires gathering, evaluating, and integrating technical data on deposit and solvent characteristics.

- Deposit characterization
  - —Depth
  - —Ore grade
  - —Total contained metal
  - —Ore thickness
  - —Flow conductivity (permeability)
  - —Void space in rock (porosity)
  - —Mineral distribution in rock
  - —Natural groundwater flow.
- Solvent characterization
  - —Composition
  - —Rate of metal solubilization
  - —Interaction with gangue minerals
  - —Impact on rock permeability
  - —Impact on materials of construction used in wells and surface facilities
  - —Trace metal solubilization and impact on surface processing and environmental requirements.

A final design requires specification of the following sub-surface parameters: effluent metal concentration, number of wells, well spacing and frequency of future well pattern additions.

For a given operation, it is likely that several combinations of the above, involving trade-offs of initial capital versus cash flow, will provide an acceptable economic performance.

Metal concentrations impact process economics

As would be expected, the level of the metal concentration in the produced solutions has a major impact on the economics of ISL. The four major capital investments that are impacted by the level of produced metal concentration are the surface facility, wells, pumps, and solution inventory for well pattern start-up.

Surface facility investment. This has two components, one related to total production capacity (tons per year), the other to the total rate (gpm) at which solution is circulated through the system. At a fixed level of production capacity, the gpm related investment declines as metal concentration increases, as smaller volumes of solution can be processed to obtain a fixed tonnage per year of metal. The product of gpm and metal concentration is proportional to tons per year production capacity. At a fixed level of production capacity, the volume of solution continually being handled (total gpm) decreases as the concentration of metal in solution increases. Thus, from the standpoint of surface facility capital, it is desirable to achieve a high effluent metal concentration.

Wells. Total well costs are related to individual well costs, which increase with depth, and the number of wells. Since the individual production capacity of a well is proportional to the product of flow rate and metal concentration in solution, it is desirable to have a high metal concentration to achieve a high level of individual well capacity, which in turn keeps the number of wells at a minimum level.

Pumps. Pump investments usually increase with both the volume of solution processed and the pressure head against which the fluid must be pumped. High effluent metal concentration results in low solution volumes, while the pressure head is proportional to both well depth and rock permeability. If the rock is tight, high pressure differentials in the wells must be used to move fluids between wells.

BASIS 250,000 LB/YEAR 0.1% ORE-GRADE,
50% RECOVERY, 30% POROSITY, 50 GPM PER WELL
PLANNED WELL LIFE 1 YEAR

50 PPM URANIUM	**300 PPM URANIUM**
• 1250 GPM FLOW	• 210 GPM FLOW
• 10 MILLION GALLONS FLUID IN ROCK PORES BETWEEN WELLS	• 10 MILLION GALLONS FLUID IN ROCK PORES BETWEEN WELLS
• 5 1/2 DAYS TO DISPLACE PORE FLUID	• 33 DAYS TO DISPLACE PORE FLUID
+ 1 1/2% OF OPERATING WELL LIFE START-UP TIME NEGATIVE CASH FLOW	– 9% OF OPERATING WELL LIFE START-UP TIME NEGATIVE CASH FLOW
– 25 WELL UNITS	+ 4 WELL UNITS
– MOVE FLUID AROUND 66 TIMES	+ MOVE FLUID AROUND 11 TIMES

Table 1. Opposing effects in systems handling 50 and 300 ppm uranium concentrations

Start-up. The pores of the rock in the well pattern are initially filled with water. Each time a well pattern operation is initiated the solubilized metal is diluted by the pore fluid, requiring at least one exchange of the volume of water in a well pattern (one pore-volume) to attain full production capacity. Prior to this time, operating costs are likely to exceed revenues resulting in a negative cash flow. Since the time required to achieve this one-pore volume displacement increases as the total flow rate of the system is reduced, the negative cash flow of start-up will be higher at increased levels of effluent metal concentration, as the product of total gpm and metal concentration is a constant for a given production capacity.

Table 2. Design parameters affecting major capital investments for in situ leaching

- **SURFACE FACILITIES**
  - PRODUCTION CAPACITY
  - TOTAL PLANT GPM
- **WELLS**
  - DEPTH
  - FLOW RATE PER WELL
  - METAL CONCENTRATION
- **PUMPS**
  - TOTAL PLANT GPM
  - DEPTH }
  - PERMEABILITY } PUMPING HEAD
- **START-UP**
  - OPERATING COST
  - TIME TO EXCHANGE PORE FLUID
    - WELL LIFE }
    - POROSITY } VOLUME OF PORE FLUID
    - GRADE }
    - TOTAL PLANT GPM

Three of the four major investments (surface facility, wells, and pumps) decrease with increasing metal concentration. However, since the negative cash flow associated with start-up is higher for a system designed for high metal concentration (low gpm), it is likely that an optimum concentration exists at which the initial total cash outlay is minimized. Putting it simply, even if it is possible to technically achieve very high metal concentrations, it may not be economically desirable. Table 1 illustrates these opposing effects, and table 2 summarizes the design parameters affecting each of the four major investments.

Sub-surface engineering considerations

In-situ leaching of uranium and copper sulfide deposits requires an oxidizing agent for metal solubilization, with direct oxygen injection being the least expensive. Oxidants can be grouped into two categories; those that are totally liquid soluble, such as chlorate, and those that are oxygen based, such as hydrogen peroxide and oxygen. Peroxide probably decomposes to oxygen and water on contact with rock.

Economically speaking, hydrogen peroxide is five to 10 times more expensive than gaseous oxygen. Although it probably provides equivalent leaching performance in the rock, as it decomposes to oxygen and water on contact with rock, peroxide injection avoids the difficulties of mixing and transporting downhole a gas-liquid mixture. The added cost of using peroxide in uranium operations is tolerable, as peroxide operating costs are in the $1 to $2 per lb uranium range. In leaching copper sulfides, gaseous oxygen is the only choice, as even at minimum consumption oxygen costs are 5¢ per pound of copper and use of peroxide would approach the selling price of the metal. The handling and transport of gaseous oxygen-liquid mixtures in injection wells is described in two patents,[4,5] and has been demonstrated to be a safe and stable operation.

Free oxygen gas impacts injection and leaching

When the back pressure of the water table is not sufficient to maintain oxygen in solution, free oxygen gas will form in the rock. This leads to the possibility of reducing the efficiency of well pattern operations by reducing permeability at the point of injection, and/or venting gas in an unproductive manner in the vertical direction, or fingering through the solvent in the horizontal direction. In a uranium operation, free gas formation will be related to a combination of the ore-reducing capacity and water table (oxygen requirements versus solubility). To achieve copper loadings above 4 lb per 1000 gal ($^1/_2$ gpl) in a sulfide deposit, well pattern operations must be designed to handle free gas flow, as free gas will exist even at depths of 3000 ft. The presence of free gas in the rock affects both the rate of liquid injection and the operation of the well pattern.

As the oxygen concentration in the injected liquid is increased to achieve higher metal concentrations, some level is reached at which free oxygen gas forms. This free gas increases the resistance of liquid flow in the rock (reduces permeability). The productivity of the well unit (lb per day of metal), being the product of flow rate and concentration, will continue to increase, although at a lower rate than with a soluble oxidant, until a point is reached at which the free gas caused reductions of flow rate exceed the added oxidizing power. Beyond this point additional wells must be used to achieve higher metal concentrations.

Uneven flows of oxygen and solvent will occur in both the vertical and horizontal directions. Well separations and injected oxygen concentrations have to be balanced when free gas is present in the deposit to insure that an adequate oxidizing condition can be maintained within the well pattern.

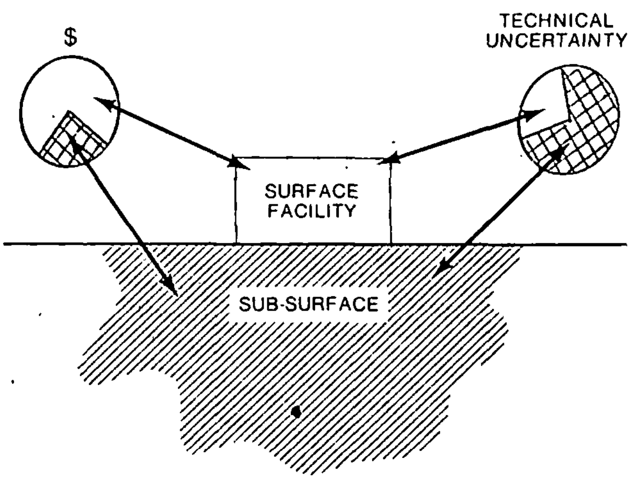


Fig. 2. Although the technical aspects of in situ leaching can vary widely, most ISL systems have in common high capital investment in the surface facilities, where the technology is standard and controlled, unlike the sub-surface component

Free gas can vertically segregate in an unproductive manner in the leach zone. The vented oxygen is not available to oxidize minerals in the bulk of the rock between wells. Two actions can be taken to compensate for gas venting.

- Increase the level of oxygen injection. This will reduce injection permeability and increase chemical and operating costs. Too much excess free gas could lead to mineral blinding, reducing the rate of leaching.
- Reduce well spacing. This will increase the number of wells and their total cost and may not provide sufficient time for oxygen to react with the mineral of interest.

The frequency at which well patterns are brought into production have significant economic and technical impact on operation of the process. Frequent replacement or addition of wells relative to the life of the operation increases the present worth of future investment, which in turn reduces the initial investment that can be allocated for wells. This places a significant burden on the technical performance of the well pattern, requiring the production capacity associated with an operation using a minimum of wells in service at any one time.

The selection of well spacing also has both economic and technical significance. Should the well spacing be made too large, then the time required to displace the water from the well pattern with enriched metal solution is high, resulting in negative cash flow impact on overall economic performance. Short well spacings provide only a limited time for the solvent to react with mineral

Donald H. Davidson is deputy manager of project development at TRW's Resource Development Organization. He was employed from 1971 until 1978 by Kennecott Copper Corp., where he worked on the development of in-situ leaching, and from 1967 to 1971 he was affiliated with Shell Development Co. Davidson holds a PhD in chemical engineering from New York University.

and enrich the metal value in solution to the desired level.

The local geology and its relationship to structural implications must be accounted for to insure high flow capture. In many cases fluid moves more rapidly in one direction versus another. Adequate hydrological testing of the deposit can provide sufficient data such that well separations can be adjusted to insure equal travel times of fluids in the deposit. This avoids uneven mining and premature loading declines.

In conclusion, in-situ leaching of non-ferrous metals has developed to a considerable extent over the last decade. As experience is gained in dealing with subsurface operations, it is likely that additional quantities of the large U.S. resource of low grade non-ferrous ore can be brought into economic production. ◆

References

[1] Bureau of Mines Information Circular/1978, IC-8777.

[2] Tucson Daily Citizen. July 22, 1975, "*Kennecott Trying Mining by Liquid.*"

[3] Simpson J. Wall Street Journal, "*New Ways to Extract Metals Are Found as Deposits Thin Out and Prices Stay Up.*" June 7, 1979.

[4] Hsueh, L: R. A. Hard, D. H. Davidson, and R. V. Huff, U.S. Patent 4,045,084 (Aug. 30, 1977), "*In-Situ Mining of Copper and Nickel.*"

[5] Hsueh, L., R. A. Hard, D. H. Davidson, and R. V. Huff. U.S. Patent 4,116,488, "*In-Situ Mining Method and Apparatus.*"

continued from p. 25

er that we're paying no attention to the debate. One just has to learn how to listen to two conversations at the same time. While the legislative chamber is an interesting place, it is also chaotic. Remember that the people there are remarkably representative.

When you're looking at a legislature, the first thing to figure out is whether you're dealing with a professional legislature or a part time legislature. Most of your Western mining states have part-timers, but California, for instance, has a full-time legislature. Again, you're going to get an incredibly different kind of person running in those two kinds of states.

Do's and don'ts of communication

First of all, try to be informed, and if you are asked a question when you don't know the answer, say you don't know. Above all, try not to give misinformation. We're tremendously dependent on other people. I don't know very much about insurance and have to ask other people about it; if I'm told the wrong thing, I remember that. Don't be defensive about saying you don't know something. Just make sure that what you're saying is correct.

Another thing is, be honest. When I'm dealing with lobbyists, I will often ask them about other bills to see from where they are coming. Legislators want someone with whom they can communicate, so if there are some negative things about a bill you want a legislator to sponsor, for instance, explain those negative aspects, too. Someone is bound to know them and ask the lawmaker questions about them, so the bill's sponsor must be well prepared. It's like being honest with your lawyer; you've got to be honest with legislators.

Tact and common sense essential

Another point to remember is that courtesy really counts. There are a lot of unwritten conventions in any legislature. In Utah, one of them is smoking. If you're going up to a group of people who don't smoke at all, or at least aren't smoking, be careful about lighting up a cigar. You may find that they vote against your bill just because they don't like the smell of your cigar.

Another thing is that legislators are very careful about the power they have. Don't try to be a legislator when you're lobbying one. Concerning access to the legislative floor, lobbyists sometimes do themselves more harm by speaking on the floor than just staying in the wings and being a resource. So be sensitive to that, because you may think you're helping your cause by standing up and giving a speech, while it may be just the opposite.

It is particularly important for a lobbyist to try to like the folks he is dealing with. You can't be very effective if legislators think you're talking down to them, if they think you don't respect them, if they think that you don't like them. If you really don't like the people that you're being asked to talk to, don't talk to them. You'll be making a negative impression.

Remember, different strokes for different folks. It might be difficult for lobbyists to approach each person differently when there are a lot of new legislators, but it is necessary. We approach every mining property differently; you've got to approach every legislator differently. At the same time, never lay on a heavy concept the first time you meet somebody. It would be best to be introduced by someone who knows the new legislator and wait at least until the second meeting to introduce whatever concept you want them to understand or act on.

In terms of what it costs, mostly what getting to know your legislature costs is your time. And that's sometimes the most precious thing we have.

The other thing I'd warn you about is shows of force. A legislator doesn't like to be cornered into doing something. He is likely to become resentful and take an opposite stand. When people feel that they "own a legislator," the lawmaker almost has to turn around and show his constituency that he isn't owned.

As for when to communicate with your legislator, there are two things to keep in mind. One is access to the legislator; and two, you want to influence what he's doing. Access to a legislator you can get any time. Contributing to a campaign gives almost automatic access. It doesn't have to be a lot, either—ten, fifteen dollars. My whole campaign costs $1000 a year. It's not expensive in dollars; it is expensive in time, but get to know your legislator if you can in the off-season. Then, when you have to influence the legislator, at least you have access. Then, use positive reinforcement in dealing with the official.

The last thing I would suggest is that if you're really unsuccessful in dealing with legislators and you really want to change your local legislative policy, run for the legislature. You may be surprised if you win, but you won't be nearly as surprised as your opponent. ◆

Genevieve Atwood has been a senior geologist for Ford, Bacon and Davis Utah Inc. since 1975. Earlier experience includes two years as staff officer, Energy Studies Board at the National Academy of Sciences, and a year as field geologist for the government of Honduras. Atwood has been a member of the Utah House of Representatives for five years.

United States Patent [19] | [11] 4,239,286
Coursen | [45] Dec. 16, 1980

[54] IN SITU LEACHING OF ORE BODIES

[75] Inventor: David L. Coursen, Mercersburg, Pa.

[73] Assignee: E. I. du Pont de Nemours and Company, Wilmington, Del.

[21] Appl. No.: 547,750

[22] Filed: Feb. 6, 1975

Related U.S. Application Data

[62] Division of Ser. No. 382,845, Jul. 26, 1973, abandoned.

[51] Int. Cl.^{3} **E21B 43/28**; E21C 41/14
[52] U.S. Cl. 299/4; 299/13
[58] Field of Search 299/4, 5, 13; 166/247, 166/271, 299

[56] **References Cited**

U.S. PATENT DOCUMENTS

3,630,278	12/1971	Parker	166/247
3,822,916	7/1974	Jacoby	299/4
3,823,981	7/1974	Lewis	299/4
3,841,705	10/1974	Girard et al.	299/4

Primary Examiner—Ernest R. Purser

[57] ABSTRACT

Producing a fracture network in deep rock, e.g., in an ore body, by detonating explosive charges sequentially in separate cavities therein, the detonations producing a cluster of overlapping fracture zones and each detonation occurring after liquid has entered the fraction zones produced by previous adjacent detonations. High permeability is maintained in an explosively fractured segment of rock by flushing the fractured rock with liquid, i.e., by sweeping liquid through the fracture zones with high-pressure gas, between sequential detonations therein so as to entrain and remove fines therefrom. Ore bodies prepared by the blast/flush process with the blasting carried out in substantially vertical, optionally chambered, drilled shot holes can be leached in situ via a number of holes previously used as injection holes in the flushing procedure and a number of holes which are preserved upper portions of the shot holes used in the detonation process. In the leaching of ore, fines are removed from fractures therein by intermittent or continuous flushing of the ore with lixiviant and high-pressure gas, e.g., air, using, in the case of the in situ leaching of an explosively fractured ore body, a lateral and upward flow of lixiviant from zones that have been less severely, to others that have been most severely, worked by multiple detonations in the ore body.

23 Claims, 4 Drawing Figures

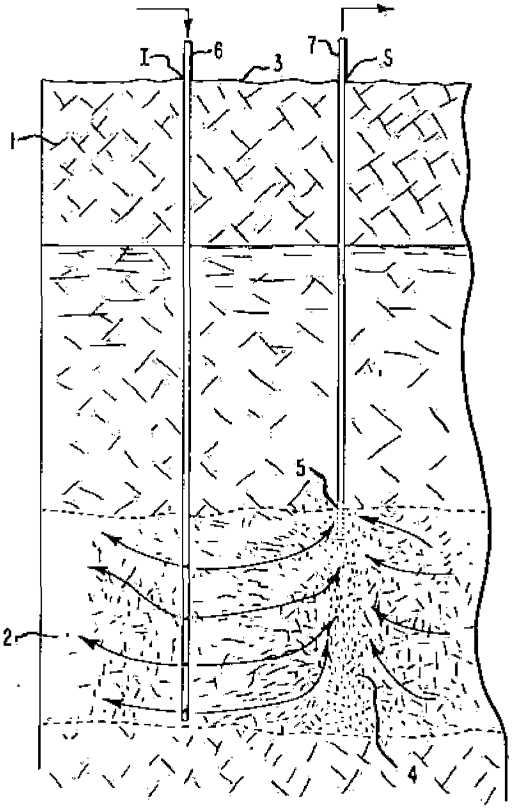


FIG. 2

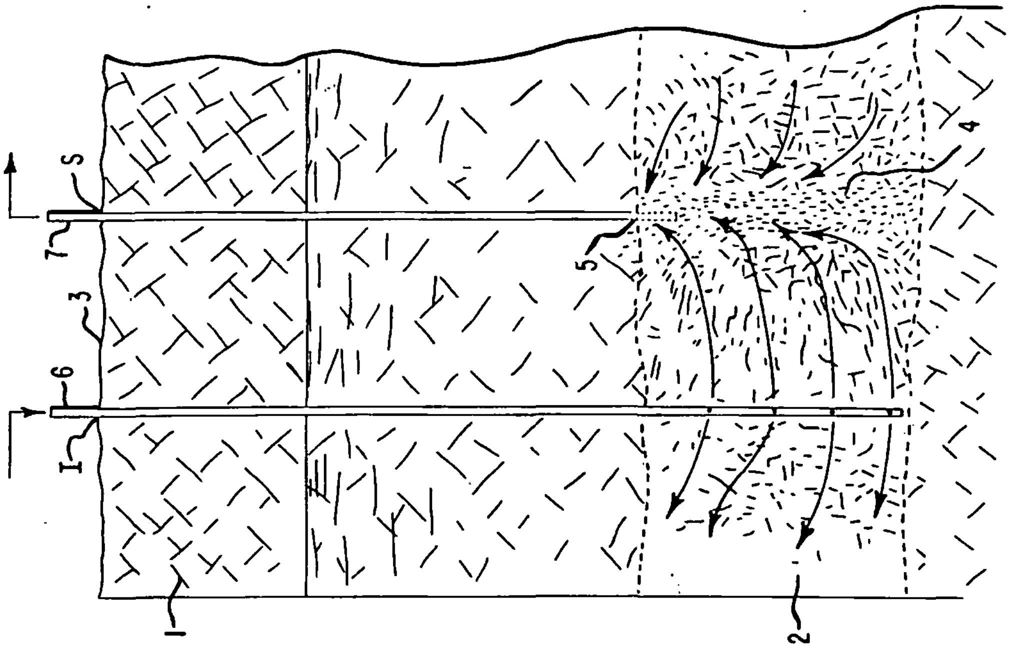


FIG. 1

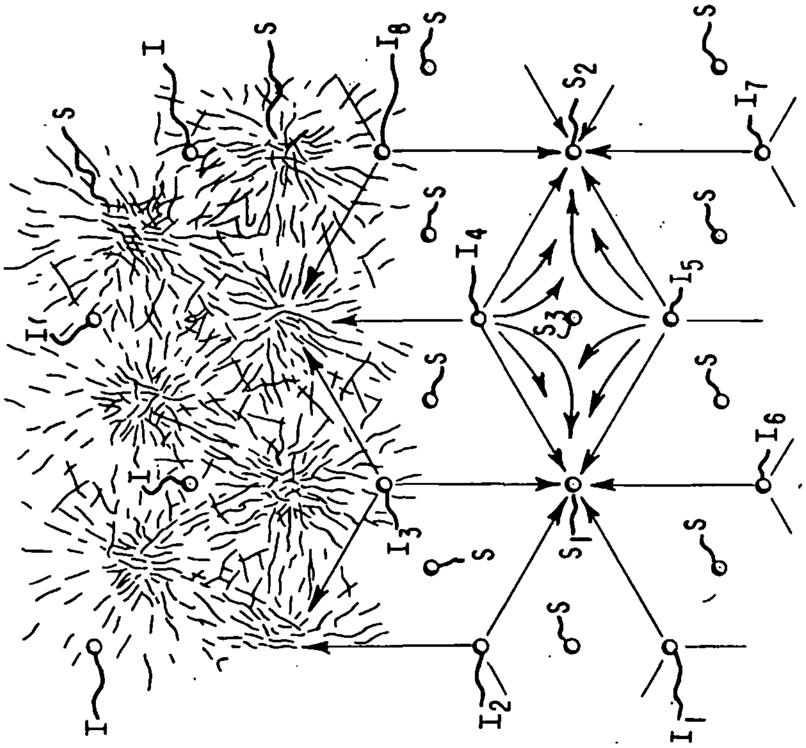
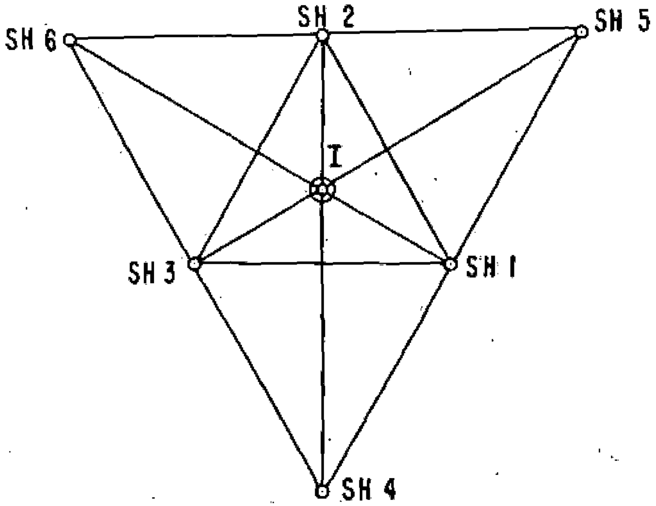
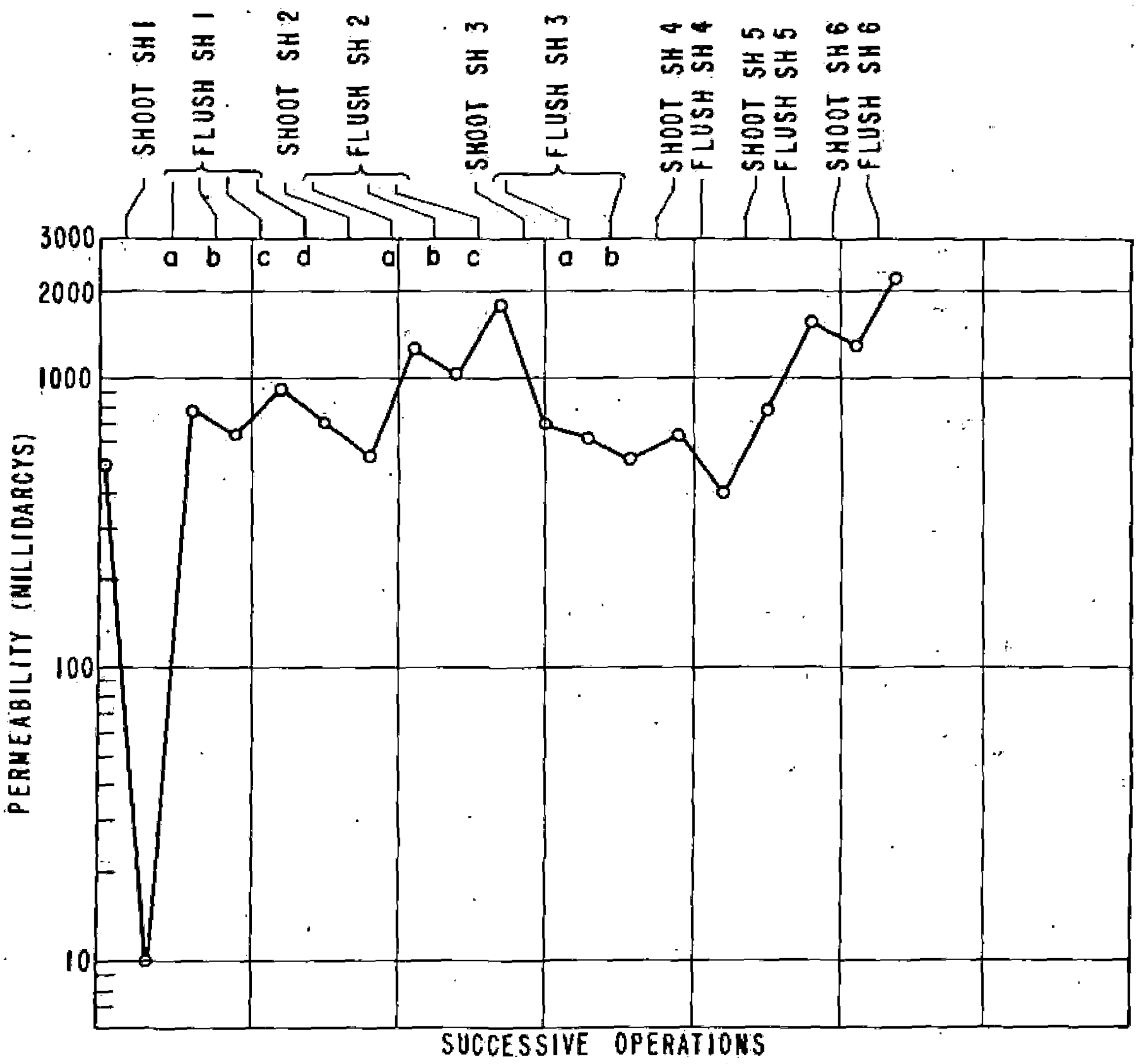


FIG. 3



SHOT HOLE PATTERN

FIG. 4



UNITED STATES PATENT OFFICE
CERTIFICATE OF CORRECTION

PATENT NO. : 4,239,286
DATED : December 16, 1980
INVENTOR(S) : David L. Coursen

It is certified that error appears in the above-identified patent and that said Letters Patent are hereby corrected as shown below:

Title page, item designated as [62] under "Related U.S. Application Data: delete ", abandoned". Col. 1, line 5: delete "abandoned" and substitute -- U.S. Patent 3,902,422 --.

Signed and Sealed this

Twenty-fourth **Day of** *March 1981*

[SEAL]

Attest:

RENE D. TEGTMEYER

Attesting Officer ***Acting Commissioner of Patents and Trademarks***

IN SITU LEACHING OF ORE BODIES

This is a continuation, of application Ser. No. 382,845, filed July 26, 1973 now abandoned.

BACKGROUND OF THE INVENTION

The present invention relates to the production of a network of fractures in a deep underground segment of rock by means of explosives, e.g., to prepare deep ore bodies for in situ leaching.

Processes for fracturing deep rock are becoming increasingly important as it becomes necessary to tap deep mineralized rock masses, e.g., ore bodies or oil or gas reservoirs located from about 100 feet to about a few thousand feet beneath the earth's surface, in order to supplement or replace dwindling energy sources and minerals supplies. Numerous deposits of ore, for example ore containing copper, nickel, or silver, lie too deep to mine by open-pit methods or are too low in grade to mine by underground methods. Open-pit methods incur both the costs and the environmental impact associated with moving large quantities of earth and rock. Underground methods incur unusually high costs per unit volume of ore mined, as well as difficult safety problems. In contrast, the leaching of ore in place circumvents these difficulties and therefore can be a preferred technique for winning values from some ores that are unsuitable, or marginally suitable, for working by traditional mining methods.

Usually however, ore that is favorably situated for leaching in place has such a large fragment size and such low permeability to leaching solutions that the leaching rate would be too low to support a commercial leaching operation. In such cases, it becomes necessary to prepare the ore for leaching, by fragmenting it in a manner such as to provide the necessary permeability and leachability. The use of explosives to fracture underground segments of mineralized rock to create areas of high permeability has often been suggested. In an oil- or gas-bearing formation the fracturing is required to increase the overall drainage area exposed to the bore of a well penetrating the formation, and thus increase the rate at which hydrocarbon fluids drain toward the well. In an ore body the fracturing is required to increase the surface area of ore accessible to an injected lixiviant, and thus increase the leachability.

The use of nuclear explosives has been proposed for fracturing large-volume, deep ore bodies for subsequent in situ leaching. Also, the use of multiple chemical explosive charges in deep reservoir rock has been described in a method for stimulating hydrocarbon-bearing rock, e.g., in U.S. Pat. No. 3,674,089. However, if a deep ore body, i.e., one lying at depths of about from 100 to 3000 feet from the surface, is to be effectively leached in place, and the ore prepared for leaching by blasting, i.e., blasting in the absence of a free face for the ore to swell toward, it becomes necessary to employ special blasting and associated techniques which will provide and maintain the type of fracture network required for efficient leaching.

The leachability of a fractured ore body depends on the size of the ore fragments, and on the permeability of the intact ore as well as of the fracture system separating the fragments. The permeability of the fracture system separating the fragments, which is variable and generally much higher than the permeability of a single fragment, is determined by a network of wider, open fractures (determining the permeability of the ore body as a whole), and a network of narrower, open fractures (determining the irrigability of individual particles to be leached). Therefore, in explosively fracturing a segment of an ore body to prepare it properly for in situ leaching, the objective is not simply an indiscriminate reduction in the fragment size of the ore body. Smaller-size, well-irrigated fragments have a higher leaching rate than larger-size fragments, but fragment-size reduction by means of blasting processes heretofore known to the art, when applied to deep ore, tends to leave large unbroken fragments of rock, or to create a network of fractures that are largely closed or plugged with fines. An explosive fracturing process is needed which reduces the larger fragments to a size that will leach at an economically acceptable rate, and that will result in a network of open fractures throughout the blasted ore that will permit it to be well-irrigated with leach liquid.

SUMMARY OF THE INVENTION

This invention provides a process for producing a fracture network in a deep subsurface segment of rock, e.g., in an ore body, comprising (a) forming an assemblage of cavities, e.g., drill holes or tunnels, in the segment of rock; (b) positioning explosive charges in a plurality of the cavities in the sections thereof located in the segment of rock to be fractured, e.g., in sections of drill holes which have been previously chambered, such as by an explosive springing procedure; (c) providing for the presence of liquid in the segment of rock, e.g., by virtue of the location of the segment of rock below the water table so that water naturally is present in, or flows into, fractures therein, or by introducing liquid into one or more cavities therein; and (d) detonating the charges sequentially in a manner such as to progressively produce a cluster of overlapping fracture zones, the detonation of each charge in the detonation sequence producing a fracture zone which is subject to the cumulative effect of a succession of detonations of explosive charges in a group of adjacent cavities, and the detonation of the charge in each cavity being delayed until liquid is present in fracture zones produced by the previous detonation of charges in cavities adjacent thereto, as determinable by measuring the hydraulic potential, e.g., the liquid level, in the cavity, or in a cavity adjacent thereto.

When the cavities formed are substantially vertical drill holes, some of the holes in the assemblage preferably are left uncharged with explosive, and these holes employed as a set of passageways within the fracture network from the earth's surface, generally to substantially the bottom of the blasted rock, e.g., for the introduction of liquid and/or gas to (or removal thereof from) the fracture network. The uncharged holes preferably are drilled and provided with support casing prior to the detonation of charges in adjacent holes. The sections of substantially vertical shot holes located in the overburden that overlies the rock segment to be fractured preferably survive the blasting process and serve as an additional set of passageways, leading from substantially the top of the blasted rock to the earth's surface, also for liquid and/or gas passage.

In a preferred explosive fracturing process, liquid is driven through the fracture zones produced by the sequential detonation of explosive charges in a plurality of cavities in a segment of rock, in a manner such as to entrain the fines found in the fracture zones, and the fines-laden liquid removed from the rock. This flushing of the blasted rock is achieved by sweeping or driving liquid at high velocity through the fracture zones by injecting gas into said zones at high pressure, the liquid moving laterally and upwardly through the blasted rock, passing into the fractures, for example, from the passageways formed by uncharged substantially vertical drill holes and out of the fractures into passageways formed by preserved sections of substantially vertical detonated holes located in the overburden. Best results are achieved when substantially each detonation is followed by a flushing step applied to the fracture zone thereby produced, before the next detonation in an adjacent cavity occurs, and this is preferred. In the leaching of a mass of ore, e.g., in the in situ leaching of an explosively fractured ore body or in dump leaching, fines also preferably are flushed out of fractures therein by sweeping the lixiviant therethrough at high velocity by high-pressure gas.

The term "deep" as used herein to describe a subsurface segment of rock denotes a depth at which the detonation causes no significant change in the overlying topography, i.e., the surface does not swell. As a rule, "deep" rock as described herein lies at a depth of at least 100, and usually not more than 3000, feet. "Fracture zones" and "fractured rock" herein denote zones and rock in which new fractures have been formed, or existing fractures opened up, by the detonations. "Fracturing" denotes herein a treatment which reduces the size of, and/or misaligns, rock fragments.

BRIEF DESCRIPTION OF THE DRAWING

The explosive fracturing process of the invention will be described with reference to the attached drawing in which

FIG. 1 is a schematic representation in plan view of a subsurface segment of rock which has been fragmented by the blast/flush process of the invention, and the liquid circulation pattern between holes therein;

FIG. 2 is a schematic representation in elevation showing the surface-to-surface liquid circulation pattern through the segment of rock shown in FIG. 1;

FIG. 3 is a schematic representation of a shot hole pattern described in the example; and

FIG. 4 is a plot showing the effect of repeated blast/flush operations on the permeability of a fracture zone produced with the shot hole pattern shown in FIG. 3.

DETAILED DESCRIPTION OF THE INVENTION

In the present process, explosive charges are detonated sequentially in separate cavities in a segment of mineralized rock to be fractured, each detonation in the sequence producing a zone of fracture in the rock and being delayed until liquid is present in the fractured rock around the cavity containing the charge to be detonated, especially in fracture zones produced by the previous detonation of charges in cavities adjacent thereto. Thus, the detonations occur while fractures in the surrounding rock are filled with liquid, or the rock is in a flooded, or liquid-soaked, condition. The cavities, e.g., drill holes or tunnels, containing the explosive charges are spaced sufficiently close together, and the charges are sufficiently large, that the fracture zones produced by the detonations therein overlap one another. Thus, each fracture zone is within the region of influence of other detonations and is subject to the cumulative effect of a succession of detonations of explosive charges in a group of adjacent cavities. This cumulative effect permits the fragment size-reduction and disorientation needed to enhance leachability to be obtained readily from the available explosive energy. The degree of overlapping of the fracture zones, which are generally cylindrical in shape, is at least that required to locate all of the rock, in the segment of rock to be fractured, within the fracture zone produced by the detonation of at least one of the charges.

The cavities in the assemblage in which explosive charges are to be detonated (i.e., blast cavities) can be substantially vertical holes (shot or blast holes) drilled into the segment of rock from the surface or from a cavity in the rock, or substantially horizontal cavities such as tunnels, driven in the rock, e.g., from a hillside or shaft. Whether the cavity volume is provided by tunnel driving techniques such as are employed in coyote blasts, for example, or drilling techniques, possibly associated with chambering procedures, will be largely a question of economics, although technical practicability depending on such factors as topography, compressive strength of the rock, etc., will influence the selection of the method. Substantially vertical drill holes are preferred in many cases since the preserved sections of the shot holes can be used subsequently as passageways to or from the fractured rock, reducing the number of holes needed to be drilled solely to provide passageways for liquid injection or ejection.

Although the blast cavities need not form a regular pattern, and regularity of pattern actually may not be desirable or practical, a somewhat regular pattern is indicated in a formation of reasonably uniform contour, structure, and physical strength to assure a high degree of uniformity in the fracture network produced. In some cases, as core tests reveal unpredictable changes in the rock occurring during the sequential blasting process, it may be desirable to deviate from a regular pattern, e.g., to use one or more additional blast cavities where needed to provide the required overlapping of fracture zones. Nevertheless, substantial regularity of pattern generally will be provided in the arrangement of most of the blast cavities. It will be understood, of course, that in the case of substantially vertical drill holes the actual pattern of the holes within the segment of rock to be fractured may approach, rather than match, the hole pattern at the surface, inasmuch as the available drilling equipment may not be counted on to produce parallel holes at depths of the order considered herein.

Regardless of the blast cavity pattern employed, the distance between explosive charges (and, also therefore, between cavities) of a given composition and size is such that a cluster of overlapping fracture zones is produced by the detonation of adjacent charges. Although it may not be possible to delineate the fracture zones with precision, the extent or radius of the fracture zone that can be expected to result from the detonation of an explosive charge of a given composition, density, shape, and size under a given amount of confinement in a given geological mass can be approximated by making some experimental shots and studying the fracture zones surrounding the blast cavities by using one or more geophysical methods. Such methods include (1) coring, (2) measurements in satellite holes of compressional and shear wave propagation, of permeability, and of electrical conductivity, and (3) acoustic holography. Based on these studies, the cavities are spaced close enough together to provide the required overlapping of fracture zones.

"Adjacent" blast cavities or explosive charges, as described herein, are blast cavities or explosive charges which, although spaced from one another, are immediate or nearest neighbors to one another, as contrasted to blast cavities or explosive charges which are more distant neighbors or separated from one another by one or more other blast cavities or explosive charges.

Although I do not intend that my invention be limited by theoretical considerations, the delaying of each detonation until liquid is present in the fracture volume surrounding the cavities is believed to have two beneficial effects. First, the liquid can lubricate the fractures so that opposing faces can move suddenly in shear more easily, thereby enhancing fragmentation of the surrounding rock, which is no longer supported by the relatively high resistance of a dry fracture to transient shear. Secondly, liquid-filled fracture volume cannot be rammed shut by the suddenly applied pressure of an explosion. This incompressible behavior, together with the low resistance of the liquid-filled fractures to sudden small displacements in shear, is believed to cause disorientation of individual rock fragments and dilation and swelling of the bed of fragments as a whole. Each detonation creates a misalignment or disarrangement of fragments with an accompanying increase in void volume. Therefore, when the fracture zones produced by the successive detonations in adjacent cavities partially overlap, the fracture zone around each cavity thereby being subject to additional fracturing and/or disorientation produced by the detonations in the adjacent cavities, and previously produced fracture zones are flooded, each fracture zone will be swelled in increments, with each detonation jacking it to larger volume, and higher permeability, against the pressure of the surrounding rock. The present process makes use of the lubricating effect and incompressible behavior of the liquid in the fractures, and does not require the use of high liquid pressures, e.g., of the magnitude needed to lift the overburden and enlarge the fractures before blasting. A liquid pressure in the fractures at the time of blasting equal to the head of liquid above the blast zone is sufficient. Also, any readily available, relatively cheap liquid, e.g., water or water mixtures, can be used to flood the rock. If leaching of ore is performed in the course of the detonation sequence, a lixiviant can be used as the flooding liquid. For reasons of economy as well as because of the safety risks associated with the use of explosives which are sensitive enough to detonate in extremely small diameters, the use of explosive liquids in the fracture zones is not contemplated. Any fluid explosive which may be used in the present process will be gelled to a viscosity that will hinder any appreciable loss thereof from the blast cavities to the surrounding fracture zones, and in any case will not be sufficiently sensitive to be detonated in said zones. Thus, while small amounts of the explosive charges may escape into the fracture zones, such material will behave as a non-explosive liquid therein. Accordingly, the flooding liquid is non-explosive.

A preferred blast cavity pattern for use in the present process is one in which substantially all of the internal cavities, i.e., cavities not located at the edge of the pattern, are surrounded by at least four adjacent blast cavities, e.g., a pattern in which the blast cavities are at the corners of adjacent polygons, which are either quadrangles or triangles and which are as close to equilateral as permitted by wander of the cavities, as shown in FIG. 1.

Although all of the blast holes in a group of adjacent substantially vertical drill holes can be drilled prior to the sequential detonation of the charges, this procedure is not preferred inasmuch as it could be necessary to apply a support casing to the as-yet undetonated holes in the sections thereof located in the segment of rock to be fractured to prevent them from collapsing as a result of detonations in adjacent holes. Casing of the shot holes in these sections usually would be considered economically unsound because the casing would occupy volume that could otherwise be loaded with explosive and because casing in these sections of the holes is not needed in subsequent leaching operations. Therefore, it is preferred that in a group of adjacent drilled shot holes the detonation of each charge takes place before adjacent shot holes are drilled. In practice, one might drill and, if desired, chamber (as described later), one shot hole of a group of adjacent holes, load the hole or chamber with explosive, allow water to enter the formation surrounding the hole or chamber, and detonate the charge, and then repeat the sequence of steps with adjacent holes. In each successive sequence of steps, the entrance of water into the formation can occur prior to, or during, any of the other steps, however. The avoidance of the presence of drilled shot holes during detonations refers to holes in a group of adjacent holes, e.g., a central hole and four to six surrounding holes. However, shot holes farther removed from the detonations can be pre-drilled.

The total amount of drilling needed for vertical-cavity blasting can be reduced by drilling one or more branch or off-set holes by side-tracking from one or more points in the preserved upper portion of a trunk hole which extends to the surface. Each off-set hole is drilled after the charges in the trunk hole and other off-set holes thereof have been detonated. Such holes will be inclined at small angles to one another.

Most of the ore bodies and other mineralized formations to which the present process is expected to be primarily applicable will be located below the water table, and in such a case, unless the section to be blasted rises locally above the water table, or the rock surrounding this section is so impermeable that flooding of the fracture zone does not occur by natural flow, the section will be naturally flooded, or water-soaked, before the sequential blasting begins, and after a certain period of time has elapsed after each detonation to allow the water to flow naturally into the newly formed fractures. If natural flooding is incomplete or absent, water or some other liquid can be pumped into the cavity to be shot after the explosive charge has been emplaced therein, and also into any available nearby uncharged cavities, at a sufficiently high flow rate to cause the rock to be blasted to be in a flooded condition at the time of detonation.

As stated previously, liquid is present in the rock around each cavity prior to the detonation of the charge therein. This means that liquid is present in any pre-existent fractures in the zone which will become a fracture zone as a result of the detonation of the charge in that cavity, and in fractures produced by previous detonations in cavities adjacent thereto. This condition permits the above-described incremental swelling of overlapping fracture zones to take place. In the case of substantially vertical drill holes, the liquid level in the rock around the hole should be at least as high as the top of the charge in the hole, thereby assuring the presence of liquid throughout the height of the formation where fracturing will occur. With horizontal cavities, the liquid level in the rock around the cavity should be at least as high as the radius of fracture to be produced by the detonation of the charge therein. When the segment of rock to be fractured is located below the water table, the position of the water table above it will conform to the water levels in undisturbed holes, and may be inferred at other locations by interpolation between the elevations of the water levels in undisturbed holes. As a practical matter, the water table will almost always be sufficiently horizontal that the first charge can be detonated when the elevation of the liquid level in any nearby hole is at least as high as the elevation to be reached by the top of the charge (or radius of fracture in the horizontal cavity case). If the liquid level is measured in the cavity in which the charge is to be detonated, the level before loading of the explosive into the cavity should be the level measured. After the detonation, the liquid level in cavities within the resulting fracture zone drops in proportion to the new fracture volume produced, the expulsion of liquid from the immediate vicinity of the charge by the gaseous products of detonation, and the drainage of liquid into the cavity created by the detonation. The detonation of the next charge in the sequence in a cavity adjacent to the first is delayed until the liquid in the formation around the next cavity (including the new fracture volume produced by the previous detonation in an adjacent cavity) returns to its required level. It is understood, however, that explosive charges in blast cavities elsewhere in a section of the formation that is not strongly influenced by a previous detonation (i.e., where the liquid level has not dropped below the required elevation as a result of the previous detonation) can be detonated at any time after the previous detonation. The delay to allow flooding applies to detonations in cavities which are adjacent to previously detonated cavities, where the previously formed fracture zones will be subject to the effect of the next detonation.

As was stated previously, some of the holes in an assemblage of substantially vertical holes preferably are left uncharged with explosive, these holes providing passageways to the fractured rock to allow the introduction of gases and/or liquids thereto, e.g., in a subsequent leaching operation. These holes, which can thus be looked upon as injection holes (although they may serve as ejection or recovery holes depending on the required flow pattern), are also useful in preparing the ore body for leaching, as will be described more fully hereinafter, and it is preferred, on the basis of ease of drilling, that they be drilled prior to the sequential detonation process in holes surrounding them. Pre-drilled injection holes are provided with a support casing, e.g., unperforated pipe grouted to the upper part of the hole wall, at least in the section thereof located in the segment of rock to be fractured, and ungrouted perforated pipe or a wellscreen in the bottom section of the hole, in order to prevent hole collapse as a result of the detonations. Inasmuch as full-length casing will be required for subsequent leaching operations, however, the full length of the injection holes usually will be cased prior to blasting. Damage to the injection piping is minimized in the present blasting process owing to the sequential, long-delay character of the multiple detonations.

The location and pattern of the injection holes are selected on the basis of their intended function during the fracturing and leaching processes, which will be described in detail hereinafter. The overall purpose of these holes usually is to provide a means for introducing gases and/or liquids into the fracture network produced, or being produced, and therefore the injection holes should be distributed throughout the segment of rock among the blast cavities in a manner such that they lie within the fracture zones produced by the detonations. After the detonation of the charge in a substantially vertical shot hole, the resulting fracture zone permits communication between a neighboring injection hole and the portion of the shot hole remaining in the overburden. The shot hole remnants thereby act as passageways to complete the liquid circuit through the fractured rock.

If injection holes are present in the formation during the sequential detonation process, an injection hole lying within the fracture zone produced by a previous detonation in a cavity adjacent to a cavity to be shot can be employed to determine whether the liquid level in the rock surrounding the cavity to be shot has recovered sufficiently to flood the section to be blasted. Whenever an hydraulic potential (e.g., a liquid level) measurement is required after a blast cavity has been loaded with explosive, a nearby injection hole can be used. When the segment of rock to be blasted is at least partly above the water table, liquid is introduced into the rock in the cavity to be shot, in previously detonated cavities adjacent thereto, and/or in nearby injection holes. Flooding via multiple cavities is preferred. Liquid is run into a blast cavity after the explosive charge has been emplaced therein (if the charge is stable in the presence of water), and liquid level measurements, if required, are made in nearby injection holes. It should be understood that, in practice, hydraulic potential measurements, e.g., pressure measurements made with a piezometer, or liquid level measurements, will not be required after each detonation, inasmuch as the experience gained in determining the necessary delay times to permit recovery of hydraulic potential between a few of the early detonations in the sequence will usually allow the practitioner to select with confidence suitable delay times to be used between subsequent detonations.

Although the exact delay required depends on the size of each blast, the void volume to be filled, the elevation of the segment to be blasted relative to the water table, and the hydraulic transmissibility of the surrounding rock, delays on the order of hours or days generally will be needed. As a practical matter, the time required for a shot hole to be drilled, or a tunnel to be driven, and loaded with explosive usually will be more than sufficient for the hydraulic potential around the cavity and the previously detonated adjacent cavities to recover to the minimum required level either by natural influx of water from the surrounding rock or by introduction through cavities made in the formation. In general, delay times between detonations of at least about one hour, and typically in the range of about from 4 to 24 hours, are sufficient for flooding to take place, although much longer delays, e.g., in the range of about from 4 to 30 days, may be employed in order to prepare the next blast cavity for blasting. It will be understood that these delays refer to the time between detonations of adjacent charges, and that one or more charges whose zones of fracture are non-adjacent (i.e., whose regions of influence are mutually exclusive) can be detonated at much shorter delay times or even simultaneously.

I have found that when sequential blasting is carried out in less competent, broken, or clayey rock, the permeability of the rock may be decreased, although the fracture volume is increased, by the blasting. Lost permeability can be restored by flushing of the fractured rock, i.e., by sweeping or driving liquid through the fractures at high velocity and removing the fines-laden liquid from the rock, preferably after each detonation. The flushing procedure appears to remove from the fractures the clogging fines that prevent free irrigation around the rock fragments. Such fines are present in the form of existing clays and rock crushed or abraded during blasting.

The flush can be accomplished by the pressure injection of liquid and gas into the fractured rock through one or more injection holes, and removal of the fines-laden liquid from the fractured rock by bringing it to the surface through one or more detonated shot holes, in the preserved sections of the latter which pass through the overburden to the surface. Liquid and gas, e.g., water or other aqueous liquid and air or oxygen, can both be injected; or gas alone can be injected so as to sweep ahead the liquid already present in the fractures. Alternatively, a liquefied gas, such as air, nitrogen, oxygen, can be introduced into the injection holes and allowed to vaporize therein and thereafter drive the liquid through the fractures. Inasmuch as there is a two-phase flow in a generally upward direction and laterally in the direction of the detonated shot holes, the circulation of the liquid is powered by gas lift such that the gas chases the liquid upward and outward through the broken formation, and fines are driven toward the zones of severest fracture, where their concentration is heaviest, from which zones they are ejected with the liquid. This direction of sweep is preferred inasmuch as the reverse direction drives the fines more deeply into the less severely worked zones of the formation away from their point of heaviest concentration and can cause an intensified clogging of the fractures. The surging high-velocity flow which develops with the upward two-phase flushing system removes fines that prevent free irrigation around the fragments. If necessary to achieve the required lateral circulation of liquid between injection hole and ejection hole throughout the length of the fracture zones being flushed, two or more vertically separated injection zones in a given injection hole can be employed, one substantially at the bottom of the fractured rock and one or more others above it.

The buoyancy of the pressurized gas alone can be sufficient to raise the fines-laden liquid to the surface of the ground when the water table is relatively close to the surface. When the water table is so deep that the buoyancy is insufficient, the liquid can be pumped up the collar of the shot hole.

At the start of flushing, the gas injection pressure should be higher than the ambient hydrostatic pressure at the position in the injection hole where injection occurs, and preferably higher than the lithostatic pressure at this position. The minimum gas pressure required for flushing is highest at the start of the operation and falls as gas injection proceeds.

Although there can be much variation in the number of fracture zones being flushed out at any given time, and the nature and number of other operations which can be performed during flushing, it is preferred that a detonation in any given cavity be followed by detonations in no more than two or three adjacent cavities, and most preferably by a detonation in no adjacent cavity, before the fracture zone produced by the detonation in the given cavity has been flushed out as described. In some formations, if a given fracture zone is subjected to a number of subsequent detonations without the intervention of flushing, restoration of permeability by a later flushing becomes difficult because the fractures may have become plugged up too tightly with fines. Therefore, a cyclic blast/flush/blast/flush, etc. process is preferred. One or more fracture zones can be flushed at the same time, and flushing of the same zone can be repeated, if desired. An already flushed zone can be left untreated during the flushing of adjacent zones by plugging the ejection hole in that zone. Flushing of one or more zones can be carried out while adjacent blast cavities are being drilled and loaded.

In the present process, the detonation of the charges in sequence permits the preservation of the sections of substantially vertical shot holes that pass through the overburden (the strata overlying the rock segment being worked), and these sections of the shot holes can serve as ejection holes in the flushing process, as described above. The reduced fragment size and unclogged fracture network achieved after all of the charges have been detonated, and the detonations followed by a flushing procedure, produce, in the case of an ore body, an ore which is well-prepared for in situ leaching.

The present invention also provides a leaching process wherein fines are flushed out of a mass of ore by driving lixiviant through the mass by means of high-pressure gas, e.g., in a specific circulation pattern. According to one embodiment of the present leaching process, an ore body which has been prepared for leaching by detonating explosive charges in separate cavities therein, e.g., according to a process of this invention, is leached in situ by introducing lixiviant for the ore into the prepared ore body through a plurality of injection holes therein and intermittently or continuously driving the lixiviant through the ore body to a plurality of recovery holes by means of high-pressure oxidizing gas, the lixiviant moving laterally and upwardly from zones that have been less severely worked, to others that have been most severely worked, by the detonations, whereby fines are removed from the ore body. When the ore body has been prepared for leaching by means of the abovedescribed blast/flush process the lixiviant for the ore can be injected into the ore body through injection holes which have previously been used in the flushing steps, and fines-laden pregnant leach solution recovered from the ore body through the preserved upper portions of shot holes, piping having been grouted into all holes used to circulate lixiviants and pumps provided as necessary to inject lixiviants in one set of holes and remove pregnant liquor from another set of holes. The bottom ends of the pipes and any other positions along the pipes where lixiviants are to be injected or collected are provided with perforations or wellscreens.

The lixiviant (e.g., sulfuric acid/water or sulfuric acid/nitric acid/water for ores whose acid consumption is within tolerable levels, or NH_4OH/water for ores having a high acid consumption), which is a liquid, and a gas, usually an oxidizing gas, preferably oxygen, air, NO_x, or mixtures thereof, are injected into the base of the prepared ore body at high pressure. As in the case of flushing between blasts, this type of injection gives a circulation powered by gas lift such that the gas chases the liquid through the broken rock. Even with constant flow rates of gas and liquid at the injection holes, a surging, high-velocity flow develops in the rock which is believed to be beneficial in (1) removing fines around the ore fragments (such fines being created during the leaching process in forms such as decrepitated ore slimes and precipitated iron salts), (2) increasing the leaching rate as a result of the cyclic squeezing of the ore fragments from the pressure fluctuations associated with the surging flow, and (3) working the ore gently so as to collapse wide openings among the fragments that may develop during the leaching process and can cause channelling of leaching solution. Sweeping the lixiviant laterally toward collection points in the more severely worked fracture regions of the ore body, and from injection points in the less severely worked regions reduces the chances that a more intense clogging of the ore body with fines will occur.

The circulation pattern employed in the leaching process as well as in the flushing steps of the fracturing process may be understood more clearly by reference to the accompanying drawing. In FIG. 1, the holes designated by the letter S are substantially vertical shot holes. Within the blasted segment of rock, these holes are destroyed by the detonations which have taken place therein in the fracturing process and are replaced by the adjacent, overlapping fracture zones shown in the upper half of the figure, and also denoted by the letter S, to indicate a previous shot hole. The shot holes rather than the fracture zones are shown in the lower half of the figure so that liquid circulation lines can be indicated clearly. It should be understood, however, that upon completion of the entire blast sequence all shot holes are surrounded by fracture zones (as depicted in the upper half of the figure) in the sections thereof located in the rock segment that was blasted. In the sections overlying the blasted segment, the shot holes remain substantially intact and in these sections all shot holes appear as they are shown in the lower half of the figure. The preserved upper sections of the shot holes are ejection holes in the flushing steps of the blasting process, and recovery holes in the leaching process. In the hole arrangement illustrated in FIG. 1, the shot holes are arranged in a trigonal pattern wherein lines between adjacent holes form substantially equilateral triangles.

The holes designated I are injection holes. These holes are uniformly distributed among the shot holes as shown. The arrows indicate the direction of flow of liquid from injection holes I_1, I_2, I_3, I_4, I_5 and I_6 to the preserved upper section of shot hole S_1; and from injection holes I_4, I_5, I_7, I_8, and two other undepicted injection holes to the preserved upper section of shot hole S_2. The preserved upper section of shot hole S_3 is plugged off while shot holes S_1 and S_2 are being used for flushing or as recovery holes for pregnant leach solution. At the same time, liquid injected into these injection holes is being driven to other open shot holes.

In FIG. 2, piping in injection hole I and shot hole S is shown as it passes through overburden 1 to the fractured rock segment 2. Piping 6 in injection hole I leads from the earth's surface 3 to substantially the bottom of rock segment 2. Piping 7 in shot hole S leads from the earth's surface 3 to the top of rock segment 2. Fracture zone 4 has been produced by the detonation of an explosive charge in shot hole S, which before the detonation led to substantially the bottom of rock segment 2. Piping 7 terminates in well screen 5, and piping 6 is provided with perforations vertically spaced along the length thereof located in rock segment 2. In the flushing steps of the fracturing process, and in the leaching process, liquid is injected into fractured rock segment 2 through the perforations in piping 6, then is driven by pressurized gas through the fractured rock as indicated by the arrows, and leaves the top of the rock segment through piping 7. Lateral as well as upward flow occurs from the less severely worked zone around hole I to the most severely worked zone, i.e., fracture zone 4.

Regulation of the rate at which gas and liquid lixiviant are injected and collected at the various injection and collection holes allows a high degree of control of the in situ leaching process. By the operation of control valves, the injection and collection pressures can be regulated to obtain a relatively uniform flow through the ore body in spite of variations in permeability from place to place. Shifting the injection or collection from one set of holes to another will change the direction of flow through the ore and can be used to frustrate channelling. The regulation of pressures and flow rates at the various holes can be used to maintain a net flow of ground water toward the operation under conditions that might otherwise result in the escape of leach solution. Leakage of the leach solution is also reduced in the present process as a result of the carriage of some of the fines away from the area of gas agitation where they settle out and plug the leak. In leaching, the gas/liquid pressure injection can be intermittent or continuous, depending upon the degree to which the ore tends to plug up, and the frequency with which flow patterns are changed to obtain uniform and complete leaching throughout the ore.

When lixiviant is introduced into an injection hole simultaneously with gas, its injection pressure should be equal to that of the gas, i.e., higher than the ambient hydrostatic pressure at the injection point, and preferably higher than the lithostatic pressure at this point. In some cases, especially at greater depths, the injection of lixiviant and oxidizing gas at sufficient pressure to exceed the lithostatic pressure may be necessary in order to get sufficient flow rate through the ore. If, in some or all of the injection holes, there are periods of time when lixiviant alone is introduced into the ore, this introduction preferably is done at a pressure at least as high as the lithostatic pressure at the injection position. That is, the pumping pressure preferably is at least as high as the lithostatic pressure minus the heads of fluid in the piping leading from the pump to the injection position.

According to the present invention, permeability can be increased also in ore masses such as mine waste dumps by driving lixiviant through fractures therein by means of gas at sufficiently high pressure that the lixiviant is swept through at a rate sufficiently high to entrain fines present in the fractures, and removing the fines-laden lixiviant from the ore mass.

In a preferred embodiment of the present process, the sections of substantially vertical shot holes which are located in the segment of rock to be fractured are first chambered to larger diameter, and the explosive charges positioned in the chambered portions. In this procedure, drilling costs are reduced by drilling widely spaced-apart shot holes of smaller diameter than is required to accommodate the size of explosive charges to be employed, and enlarging or "springing" the lower parts of the shot holes to produce chambers having the volume required to hold the explosive charge. The sections of the holes in the rock segment are chambered either by drilling them out, e.g., with an expansion bit, or by detonating explosive charges therein. The chambering method is not critical, the preferred method generally being the one that results in the lowest overall cost per unit of chamber volume for the particular rock segment in question. In the present process, explosive charges used for springing may be 20 feet or more in length. If rock fragments tend to fall from the walls of an explosively sprung hole and thus to occupy some of the volume required for the explosive charge subsequently to be used in producing the fracture zone, the hole to be sprung can be drilled deeper so that the bottom of the hole is located below the bottom of the formation. In this manner, any loss in volume that is to be available for explosive loading is minimized since a portion of the chamber volume below the segment of rock to be fractured can hold the fallen rock fragments.

The advantage of chambering the shot holes before loading them with the charges which will be detonated to produce the fracture network becomes evident when it is considered that an explosively sprung hole typically will hold about ten times as much explosive as an unsprung hole. Thus, for example, a pattern of 30-inch-diameter charges on 100-foot spacings (center-to-center) typically can be achieved by drilling 9-inch-diameter holes on 100-foot spacings.

Although the blast/flush process has utility in deep underground blasting with explosives of all types, the use of chemical explosive charges is much preferred for several reasons. The many technical as well as civil (legal, political, public relations) problems associated with the undertaking of nuclear blasting are self-evident. Vibration effects and radioactivity are the two major roots of these problems. A nuclear blast which is large enough to be economically feasible must be set off at sufficient depth, e.g., preferably appreciably deeper than 1000 feet, in order to be safely contained and not release radioactivity to the atmosphere. Many potentially workable ore bodies will not be located as deep as the safe containment depth. Furthermore, the extreme magnitude and concentration of the energy produced in a nuclear blast imply that it will be difficult, if not impossible, to achieve (a) a high degree of uniformity in explosion-energy distribution and ore breakage, (b) close hydraulic control of the flow of lixiviants without an appreciable amount of additional drilling to increase the number of injection and extraction points, and (c) a close match of the broken volume with the outline of the ore body, particularly for small or irregular ore bodies, such a match resulting in economies in the use of the available explosive energy and in the consumption of lixiviants.

While single explosive charges generally will be detonated in sequence to produce the fracture zones, the charges also can be multi-component charges positioned in separate cavities and detonated substantially simultaneously as a group to produce each fracture zone, each detonation in the sequence of detonations in such a case being a group of detonations.

The following example illustrates specific embodiments of the process of the invention.

The formation to be fractured was a bedded series of shales and silt stones, dipping about 45°, located at a depth of 70 to 90 feet below the surface, and therefore subjected to a lithostatic pressure of about 70 to 90 p.s.i. The water table was at a depth of about 15 feet below the surface. A 3-inch-diameter hole was drilled into the formation to a depth of 100 feet. This hole was used as a coresampling and permeability-testing hole, and also as an injection hole for purposes of flushing surrounding shot holes. A core test revealed a competent silty shale at the 70-90 foot depth. A well screen was installed in the hole at the 70-90 foot level, and piping to the well screen was grouted to the hole. Cement filled the hole below the well screen.

The pattern of shot holes used is shown in FIG. 3. Three shot holes (SH 1, SH 2, and SH 3) were located 16.25 feet from the injection hole I, their centers lying on 120° radii from the center of hole I and the lines joining them forming an equilateral triangle. The distance between these shot holes was 28 feet. Three shot holes (SH 4, SH 5, and SH 6) were located 32.5 feet from hole I, their centers also lying on 120° radii from the center of hole I, and the lines joining them (also forming an equilateral triangle) being bisected by the centers of holes SH 1, SH 2, and SH 3. The distance between holes SH 4, SH 5, and SH 6 was 56 feet. It is seen that in this arrangement the lines joining adjacent (i.e., nearest neighbor) shot holes formed equilateral triangles. SH 1, SH 2, and SH 3 each had four shot holes adjacent thereto (SH 2, SH 3, SH 4, and SH 5 for SH 1; SH 1, SH 3, SH 5, and SH 6 for SH 2; and SH 1, SH 2, SH 4, and SH 6 for SH 3), and SH 4, SH 5, and SH 6 each had two shot holes adjacent thereto (SH 1 and SH 3 for SH 4; SH 1 and SH 2 for SH 5; and SH 2 and SH 3 for SH 6).

Shot hole SH 1 was drilled first. The hole was 5 inches in diameter and 91 feet deep and was located with 255 pounds of an aluminized water gel explosive having the following composition: 18.9% ammonium nitrate, 10.5% sodium nitrate, 29.6% methylamine nitrate, 30% aluminum, and 11% water. The explosive column was 21.7 feet high, and was covered by a layer of water which naturally flowed into and filled the remainder of the hole and stemmed the explosive charge. The water level in the injection hole was above the level of the top of the explosive charge in the shot hole, indicating that the rock surrounding the shot hole was properly flooded. Before the explosive charge was initiated, the permeability and sound velocity of the rock surrounding the injection hole were measured. The permeability was determined by slug tests, in which the permeability is inferred from the rate at which the head of liquid subsides toward the ambient level in a hole after the rapid introduction of a slug of liquid therein (see Ferris, J. G., et al., "Theory of Aquifer Tests", U.S. Geological Survey, Water-Supply Paper 1536-E, 1962). The sound velocity, measured at depths of 74.5 feet to 85 feet between the injection hole, shot hole SH 1, and a test hole collared 13 feet on the opposite side of the injection hole, was 3970 meters per second.

The explosive charge in shot hole SH 1 in the flooded formation was detonated, whereupon the water level in the injection hole dropped to below its pre-detonation level, as a result of the formation of a new fracture volume around shot hole SH 1, the chasing of water from the rock fractures by the gaseous detonation products, and the flow of water into the cavity created by the explosive charge. After partial recovery of the water level in the injection hole, the second shot hole (shot hole SH 2) was drilled to the same size as shot hole SH 1, and the rock surrounding shot hole SH 1 was then flushed with water by (a) blowing compressed air into the bottom of the open injection hole, (b) injecting water through a packer in the injection hole, and (c) three long air injections, and then (d) 18 short air injections through a packer in the injection hole. The total flushing time was about 4 hours. Silt-laden water was ejected from shot hole SH 1 (but not shot hole SH 2) during the flushing, indicating the preservation of the top of shot hole SH 1, the circulation of the water from the bottom of the rock segment (bottom of the injection hole) laterally and upward through the fracture network to the top of the rock segment (bottom of shot hole SH 1), and the removal of fines from the fractures. The permeability was measured in the injection hole (as described above) before and after the flushing operations.

Shot hole SH 2 and subsequently drilled shot hole SH 3 were loaded and the charges therein detonated as described for shot hole SH 1.

The water level in the injection hole returned to its pre-detonation level, above the level of the top of the explosive charge in shot hole SH 1 before detonation, in about 18 hours. Thereafter, the charge in shot hole SH 2 was detonated, whereupon the water level in the injection hole again dropped to below its pre-detonation level. The rock surrounding shot hole SH 2 was flushed with water, and silt-laden water ejected from shot hole SH 2, by sealing off shot hole SH 1 and (a) injecting air through a packer in the injection hole, (b) blowing compressed air into the bottom of the open injection hole, and (c) injecting air through a packer in the injection hole, followed by water through the packer while blowing compressed air into the bottom of shot hole SH 3. The total flushing time was about 11 hours. The permeability was again measured before and after the flushing operations.

After the water level in the injection hole had returned to its pre-detonation level, the charge in shot hole SH 3 was detonated, and the surrounding rock flushed by (a) air injection through a packer in the injection hole, followed by sealing off shot hole SH 1 and blowing air down shot hole SH 2 and shot hole SH 3 to drive water to each shot hole in turn until water was exhausted from the broken rock; and (b) two air injection flushings, each followed by water injection. The total flushing time was about 7 hours. The permeability was again measured before and after the flushing operations.

The remaining shot holes, SH 4, SH 5, and SH 6, were drilled, loaded, and detonated in the same manner as holes SH 1, SH 2, and SH 3, with the detonations occurring after the return of the water level in the injection hole to its predetonation level. Between the shooting of shot holes SH 4 and SH 5, the rock surrounding hole SH 4 was flushed by three air injections in the injection hole, each followed by water injection; between the shooting of shot holes SH 5 and SH 6, the rock surrounding hole SH 5 was flushed by injecting air into the injection hole, and blowing air down hole SH 6 (unshot), separately and simultaneously; and after hole SH 6 was shot, the rock surrounding it was flushed by alternately injecting air into the injection hole and blowing air down the surviving section of hole SH 6.

The permeabilities measured by slug tests in the injection hole before the blast/flush process began and after each blast and flush operation at each of the six holes are plotted in FIG. 4 as a function of the operation performed, the permeabilities being presented in millidarcys on a logarithmic scale as the ordinate. Nineteen points are shown, including those obtained after the four flushing procedures (a, b, c, and d) described above after the shooting of hole SH 1; three flushing procedures (a, b, and c) after the shooting of hole SH 2; and two flushing procedures (a and b) after the shooting of hole SH 3. Each point denotes the average permeability measured after a given operation.

The plot shows that the permeability of the rock was increased considerably (from 500 to over 2000 millidarcys) by the total six-cycle blast/flush process, and that variations in permeability occurring during the cyclical shooting and flushing tend to decrease as the rock is broken and swelled. The plotted experimental values also show that the rapid flow of water to the remnant of a shot hole achieved by means of air injection through another hole or by strong pumping from a shot hole (by blowing air into the bottom of an open shot hole, for example) increases the permeability after blasting, best results having been achieved when both air injection at the injection hole and strong pumping at a nearby shot hole was used. While blasting was found generally to decrease the permeability, permeability which had previously been reduced by the injection of water (alone or as a final flushing step after blasting) was increased by blasting.

The degree of dilation produced in the rock by the first three of the above-described detonations in flooded rock was estimated from calculations of porosity based on sound velocity measurements. The sound velocity around the injection hole after the three blasts was 3650 meters per second at a 12 foot radius from the hole, and 2530 meters per second through paths in the blasted rock running from the shot holes in to the injection hole (compared with 3970 meters per second in the same prism of rock before blasting). The total porosity in the rock (ψ) was calculated from the following empirical equation for the sound velocity (α, in m/sec) as a function of porosity, for flooded ocean sediments having various degrees of lithification:

$$\psi = -50.748 \ln\alpha + 432.23.$$

Total porosity before blasting: 11.7%

Total porosity after blasting: (12-ft. radius) 16.0%

Total porosity after blasting: (center to shot holes) 34.6%

These porosities imply that the fracture volumes caused by the blasting were 4.3% (12-ft. radius) and 22.9% (center to shot holes).

I claim:

1. A process for the in situ leaching of an ore body which has been worked by detonating explosive charges in separate cavities therein to produce in the ore body immediately adjacent to the site of each detonation a fracture zone comprised of a most severely fractured core portion surrounded by a less severely fractured outer portion, comprising introducing lixiviant for the ore into the ore body through a plurality of injection holes in the less severely fractured portions and recovering pregnant leach solution from the ore body through a plurality of recovery holes in the most severely fractured portions.

2. A process of claim 1 wherein the ore body has been worked by the detonation of chemical explosive charges.

3. A process of claim 2 wherein the injection holes extend from the earth's surface to substantially the bottom of the ore body, and the recovery holes extend from the earth's surface to substantially the top of the ore body.

4. A process of claim 3 wherein the recovery holes are preserved upper portions of shot holes in which explosive charges have been detonated.

5. A process of claim 1 wherein the lixiviant is injected at a pressure at least as high as the lithostatic pressure at the injection position.

6. A process of claim 5 wherein gas is introduced into the injection holes with the lixiviant.

7. A process of claim 6 wherein the gas is injected at a pressure in excess of the lithostatic pressure at the injection position.

8. An improved process for recovering metal values by in situ leaching an ore body located below the water table which comprises:

(a) forming in an ore body a fracture zone comprised of a most severely fractured core portion surrounded by a less severely fractured outer portion whereby the fracture zone contains fractured metal-bearing ore particles;

(b) injecting a leach solution through one or more injection wells located in the ore body adjacent to but outside the most severely fractured portion, the leach solution solubilizing metal values in the ore body and in the most severely fractured portion; and

(c) recovering a metal-containing leach solution through one or more production wells located in the most severely fractured portion.

9. The process of claim 8 wherein the fracture zone is produced by detonating one or more strategically placed explosives in the ore body, said explosive selected from nuclear and chemical explosives.

10. The process of claim 8 wherein the ore body contains a copper-bearing ore.

11. The process of claim 10 wherein the leach solution is injected through the one or more injection wells at a pressure less than the formation fracture pressure.

12. The process of claim 11 wherein the leach solution contains a dispersion of an oxygen-bearing gas.

13. The method of leaching a metal-bearing ore in place which comprises:

(a) injecting a leach solution at a pressure below the formation fracturing pressure through at least one injection well located in a less severely fractured outer portion which surrounds a most severely fractured core portion of a fracture zone of a metal-bearing ore body located below the water table;

(b) allowing the leach solution to remain in the ore body to solubilize metallic ions present in the ore body; and

(c) recovering metallic-ion-containing leach solution from at least one production well located in a most severely fractured portion of a fracture zone in the ore body.

14. The process of claim 13 wherein the ore body contains copper-bearing ore.

15. The process of claim 14 wherein the leach solution is aqueous sulfuric acid containing an oxygen-bearing gas.

16. An improved process for recovering metal values by in-situ leaching an ore body located below the water table which comprises:

a. forming a rubblized zone in an ore body whereby the rubblized zone contains fractured metal bearing ore particles;

b. injecting a leach solution through one or more injection wells located in the ore body adjacent to but outside the rubblized zone, the leach solution solubilizing metal values in the ore body and in the rubblized zone; and

c. recovering a metal containing leach solution through one or more production wells located in the rubblized zone.

17. The process of claim 16 wherein the rubblized zone is produced by detonating one or more strategically placed explosives in the ore body, said explosive selected from nuclear and chemical explosives.

18. The process of claim 16 wherein the ore body contains a copper bearing ore.

19. The process of claim 18 wherein the leach solution is injected through the one or more injection wells at a pressure less than the formation fracture pressure.

20. The process of claim 19 wherein the leach solution contains a dispersion of an oxygen bearing gas.

21. The method of leaching a metal bearing ore in place which comprises:

a. injecting a leach solution at a pressure below the formation fracturing pressure through at least one injection well located in a non-rubblized zone of a metal bearing ore body located below the water table;

b. allowing the leach solution to remain in the ore body to solubilize metallic ions present in the ore body, and

c. recovering metallic ion containing leach solution from at least one production well located in a rubblized zone in the ore body.

22. The process of claim 21 wherein the ore body contains copper bearing ore.

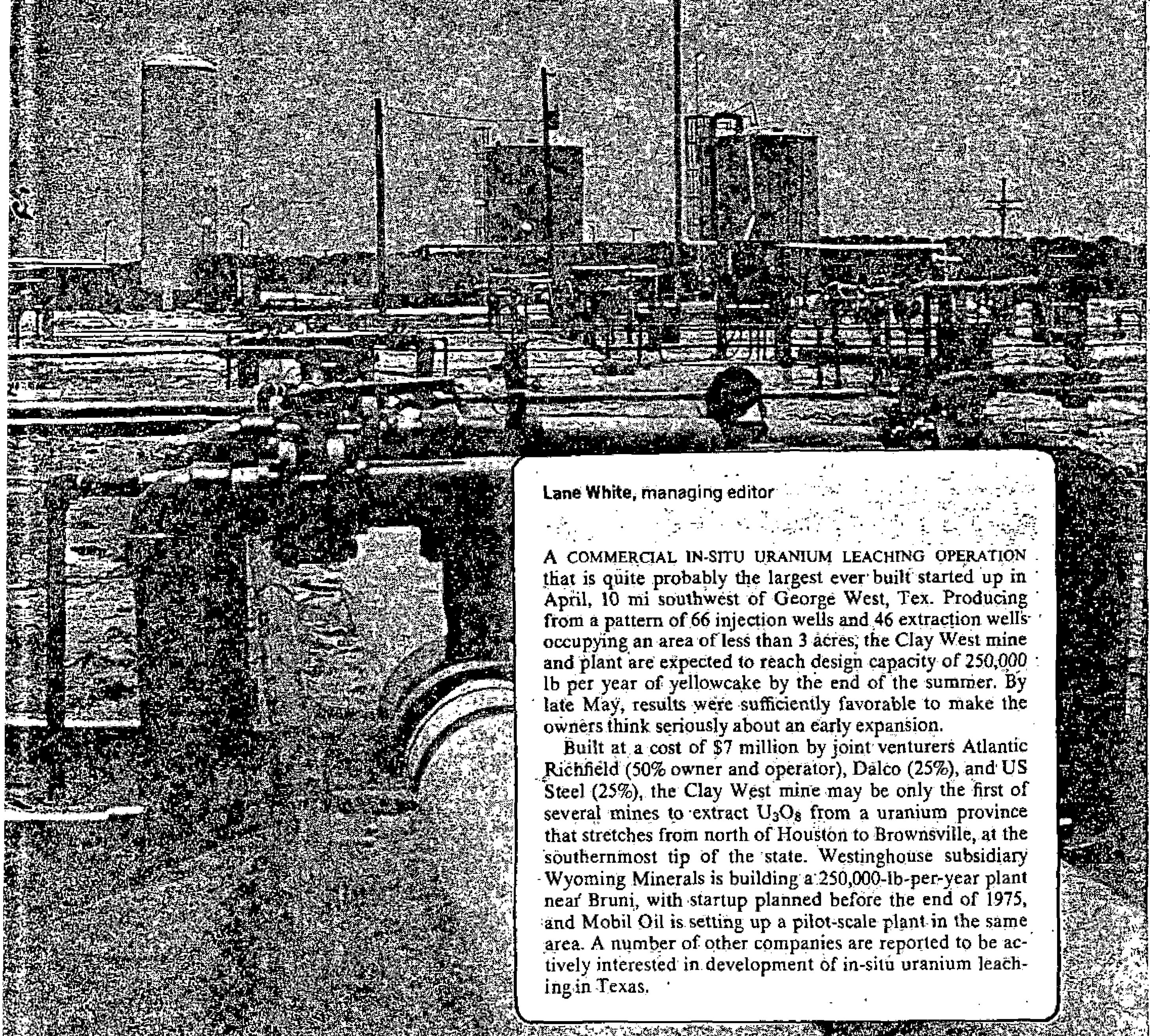
23. The process of claim 22 wherein the leach solution is aqueous sulfuric acid containing an oxygen bearing gas.

* * * * *

E/MJ
ENGINEERING AND MINING JOURNAL

In-situ leaching opens new uranium reserves in Texas

This complex of pipes, valves, and wells—part of a less-than-3-acre pattern near George West, Tex—does not look like a uranium mine. But since April it has been extracting U_3O_8 from sandstones at depths to 550 ft by in-situ leaching. The Atlantic Richfield-US Steel-Dalco joint venture may be only the first of several commercially scaled, in-situ leaching operations to successfully produce uranium concentrate from Texas ores.

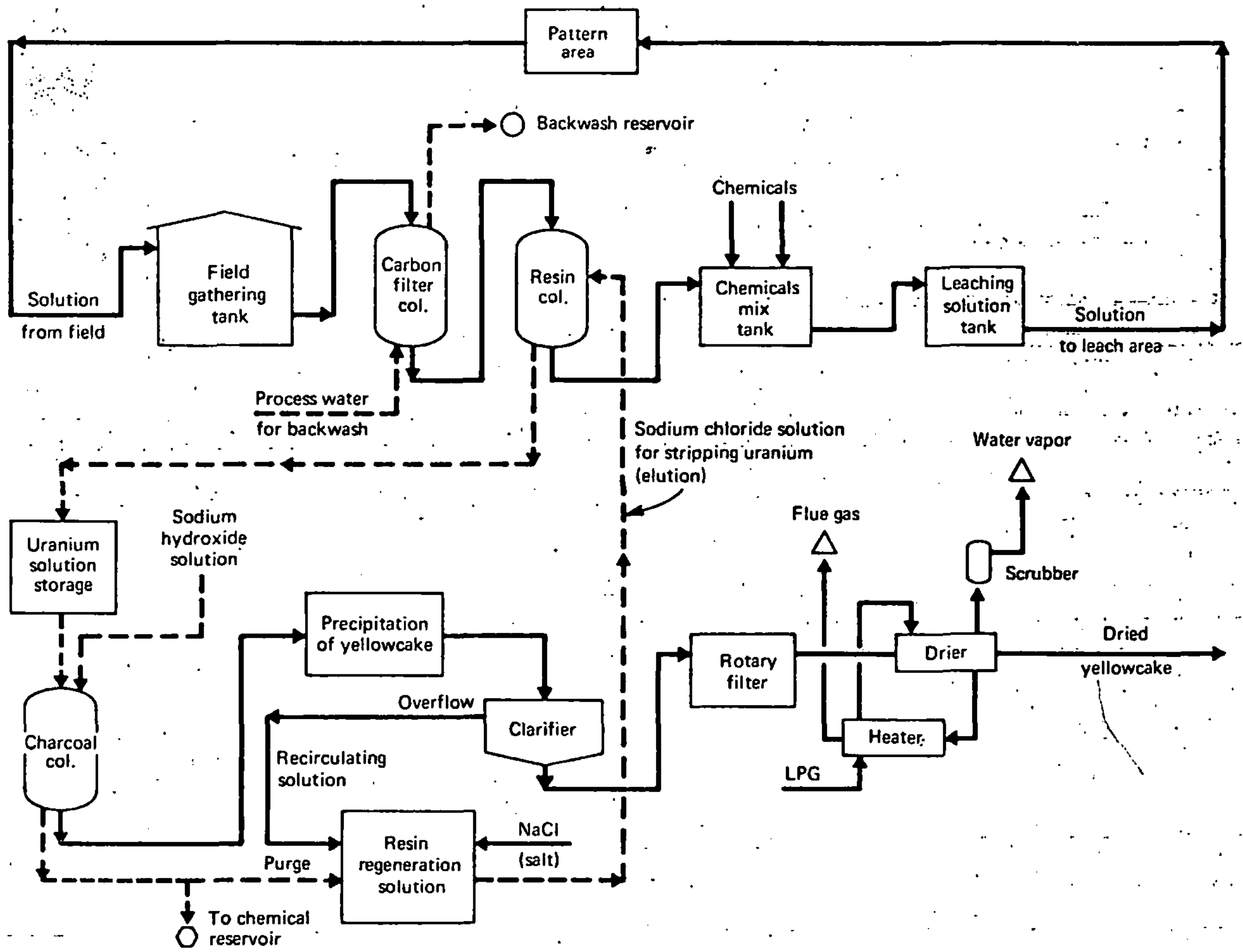


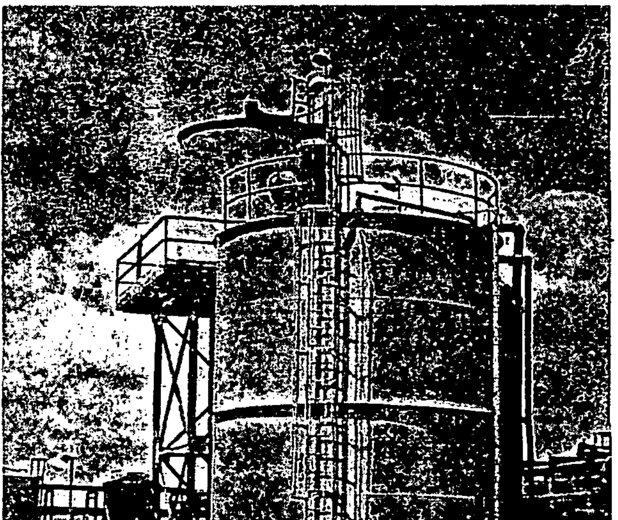
Lane White, managing editor

A COMMERCIAL IN-SITU URANIUM LEACHING OPERATION that is quite probably the largest ever built started up in April, 10 mi southwest of George West, Tex. Producing from a pattern of 66 injection wells and 46 extraction wells occupying an area of less than 3 acres, the Clay West mine and plant are expected to reach design capacity of 250,000 lb per year of yellowcake by the end of the summer. By late May, results were sufficiently favorable to make the owners think seriously about an early expansion.

Built at a cost of $7 million by joint venturers Atlantic Richfield (50% owner and operator), Dalco (25%), and US Steel (25%), the Clay West mine may be only the first of several mines to extract U_3O_8 from a uranium province that stretches from north of Houston to Brownsville, at the southernmost tip of the state. Westinghouse subsidiary Wyoming Minerals is building a 250,000-lb-per-year plant near Bruni, with startup planned before the end of 1975, and Mobil Oil is setting up a pilot-scale plant in the same area. A number of other companies are reported to be actively interested in development of in-situ uranium leaching in Texas.

Fig. 1—Uranium extraction flowsheet, Clay West, Tex.





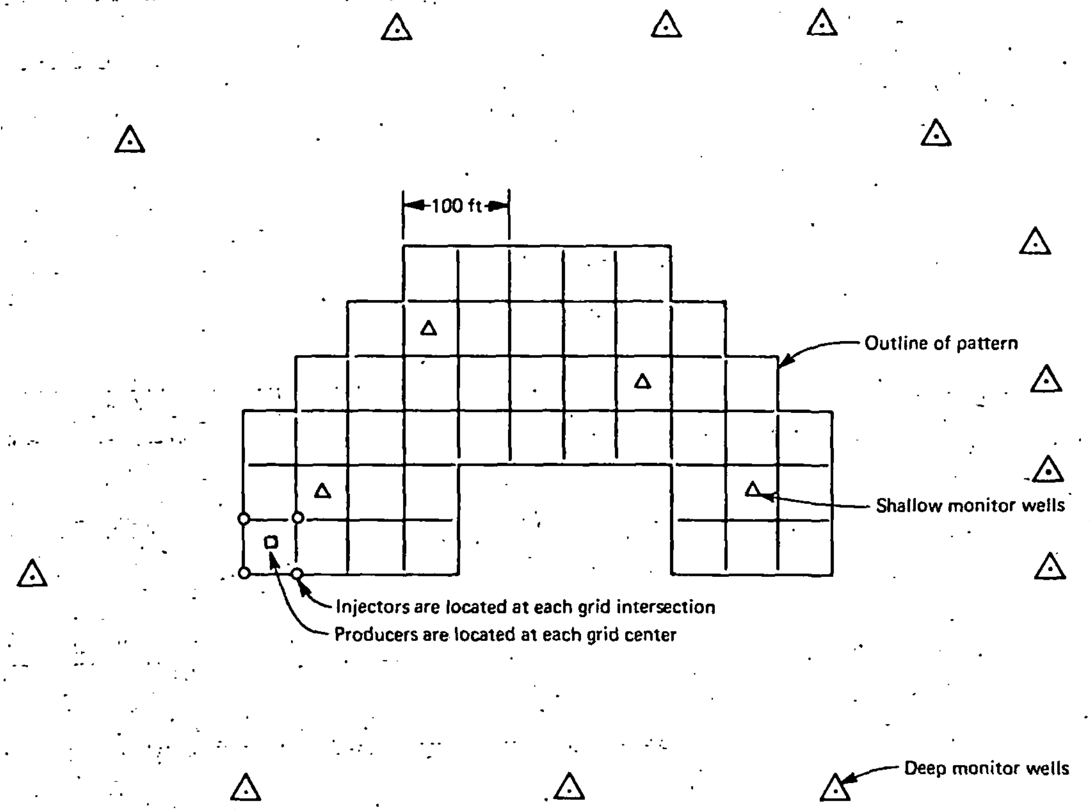
Chemicals mix tank in the Clay West plant area feeds leaching solution to a surge tank located at the in-situ mining pattern.

The Atlantic Richfield (Arco) Clay West operation will employ 45 people, supported by a staff of 12 at Arco uranium operations headquarters in Corpus Christi. Reserves are thought to be sufficient for at least 20 years' production, uranium operations manager Glen R. Davis said during E/MJ's visit to the property. The Arco-Dalco-US Steel partnership has access to 35-40 sections, and patterns of injection and production wells will eventually be operated up to 2 mi from the central processing plant.

The Clay West operation uses a proprietary alkaline leach solution developed during pilot testing by Dalco and US Steel. Solution is pumped into and out of uranium-bearing ore zones at equal rates to prevent loss of solution in the sandstone formation. The pregnant solution is collected at an 800-bbl field gathering tank and then pumped to the plant area. There the solution passes through one of six parallel trains, each consisting of a carbon column and an ion exchange column in series (Fig. 1). A commercial resin extracts U_3O_8 in the ion exchange column. Solution circulates through the leach pattern at 2,000 gpm.

New environmental control procedures were formulated by the Texas Water Quality Board to govern Clay West operations, as there was no precedent for the Arco request to pump a solution into and out of an aquifer. The company will be required to monitor the zones around producing patterns to assure that no leaching solution escapes the producing area, and when a pattern is exhausted, Arco

Fig. 2—Clay West uranium leaching pattern



must restore the aquifer to the chemical condition that prevailed prior to operation.

A deep disposal well will be used to dispose of solution pumped from the patterns during aquifer renovation.

Oakville sandstone is source horizon of U_3O_8

The Clay West mine extracts uranium from a maximum depth of 550 ft in the Miocene Oakville sandstone—about 350 ft of interbedded sands, silts, and bentonitic clays. The ore occurs in zones 10 to 60 ft thick and grades from a low 0.05% U_3O_8 to as high as 0.5%—the latter being uncommon. Regionally, rock formations dip toward the Gulf of Mexico at about 79 ft per mi.

The Oakville is underlain by Catahoula clays—about 2,100 ft of clays and thin beds of sand that have a higher-than-normal background count of U_3O_8. The U_3O_8 in the Catahoula is thought to derive from volcanic ash deposited during Catahoula time, according to Arco geologist Charles Trimble. Percolating ground waters dissolve the U_3O_8 and transport it to neighboring rocks, where, in the presence of H_2S derived from oil-bearing formations, the U_3O_8 precipitates, sometimes in commercial concentrations.

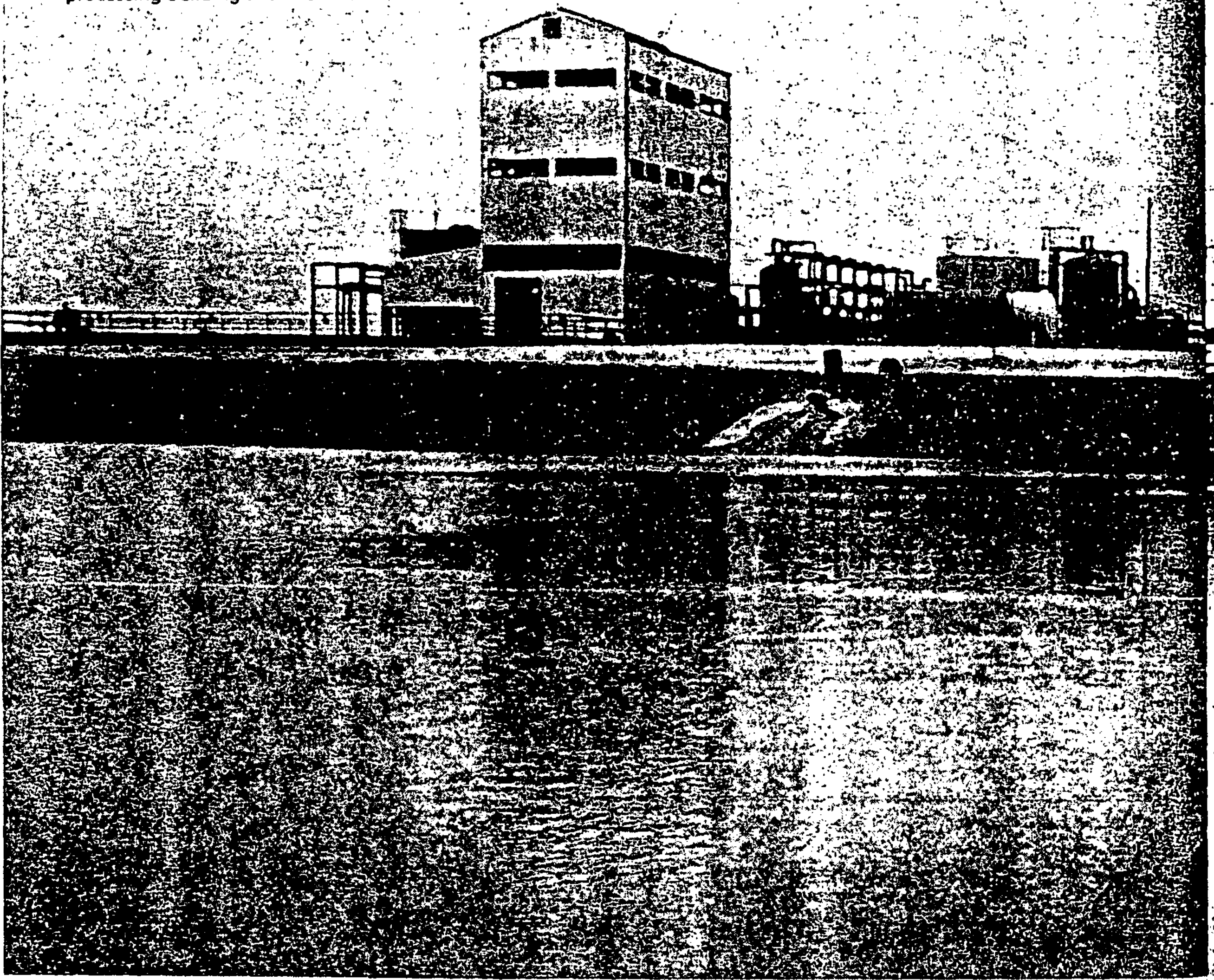
The Clay West mine taps the Oakville deposits through a pattern of production wells spaced at 50-ft intervals (Fig. 2). The 46 extraction wells are interspersed at regular intervals with a grid of injection wells and are surrounded by monitor wells.

The Oakville sandstone is a regional aquifer having a slow rate of flow—about 12 ft per year. Should the monitor wells detect leaching solution escaping from the production zone, the rate of production can be increased or the rate of injection decreased to pull the solution back.

A Midway drill of the type used for seismic blasthole drilling sinks the 4-in.-dia injection wells and the 6-in.-dia extraction wells (Fig. 3), the latter being larger to accommodate a submersible pump at the bottom of the well. Open screened sections of pipe in the ore zones permit production of leach solution with minimal extraction of entrained particulates.

Well piping, along with most other piping at the Clay West mine, is of PVC, to minimize corrosion and to keep corrosion products out of the system, Davis said. This protection will prevent plugging of injection wells and contamination of the yellowcake product. Two 12-in. mainline PVC pipes connect the plant and production areas.

The pregnant leach solution arrives at the processing plant carrying dissolved U_3O_8. It first passes through a carbon column, which removes any sand present in the solution, and then through an ion exchange column, which extracts the uranium. The carbon columns are backwashed periodically to remove trapped sand. A backwash reservoir provides clean water and acts as a settling pond. The water



View of the Clay West plant across one of two solution storage ponds: chemicals mix tank is shown at the right, storage and process facilities at center, and yellowcake processing building to the left.

is recirculated and evaporation losses are made up by water from a fresh water well.

A sodium chloride solution is used to extract U_3O_8 from the ion exchange column. The rich uranium-bearing solution (about 1% U_3O_8) passes through a charcoal column for removal of impurities, including molybdenum, and then reports to precipitation tanks, where yellowcake is precipitated with ammonia. Recovery of moly as a byproduct is a possibility in the future.

Chemical wastes from the plant are disposed of in reservoirs lined with highly resistant chlorinated polyethylene lining. The design of the reservoirs keeps net annual evaporation rate equal to the volume of chemical wastes generated each year. Reservoir capacity is sufficient to accommodate periods of high rainfall and low evaporation. During the first year of operation, a critical examination of waste streams is seeking opportunities for waste reduction.

After precipitation, yellowcake is allowed to settle in a clarifier, filtered, dried, and packed for shipping to the Allied Chemical uranium hexafluoride plant at Metropolis, Ill. The dryer, the only source of air emissions in the plant, is a screw-conveyor unit based on indirect heat transfer between a recirculating heated oil and yellowcake solids. The heat-transfer oil is heated externally in a unit fueled with liquefied petroleum gas. The quantity of flue gas from the dryer is insufficient to require a permit from the Texas Air Control Board.

One of the advantages of the dryer is minimal dust problems, because of the absence of air contact. For added protection, however, exhaust gases will be scrubbed by a two-stage water scrubber. The exhaust from this scrubber will be virtually dust-free, releasing only water vapor to the air.

The Clay West operation is not energy-intensive, as circulating solutions all flow at ambient temperatures. Power draw for the entire operation is about 1,500 kw from San Patricio Electric, a small co-op producer at Sinton, Tex. The major costs of operation are labor and chemicals.

Mine manager Walter Ely noted that creation of jobs at the Clay West mine has provided a welcome injection of

cash
ant,
and
The
Geo
near
port
A
that
area
crea
ploy
for

Co

sou
lev
ron

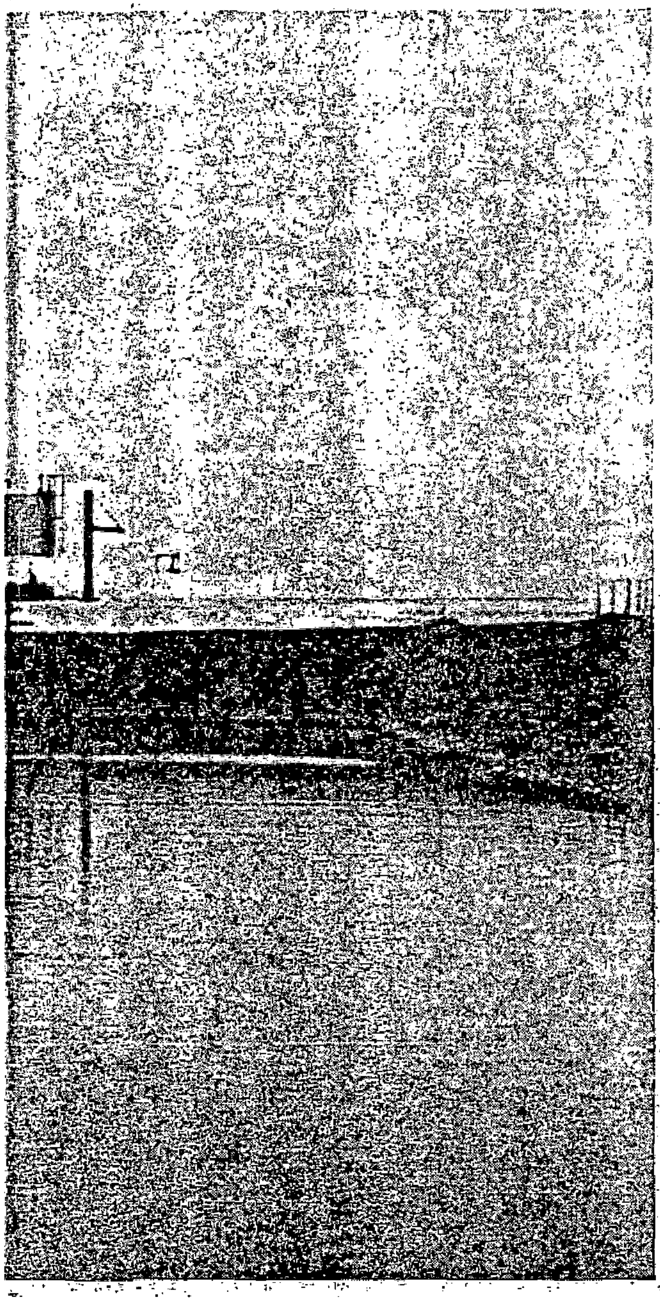
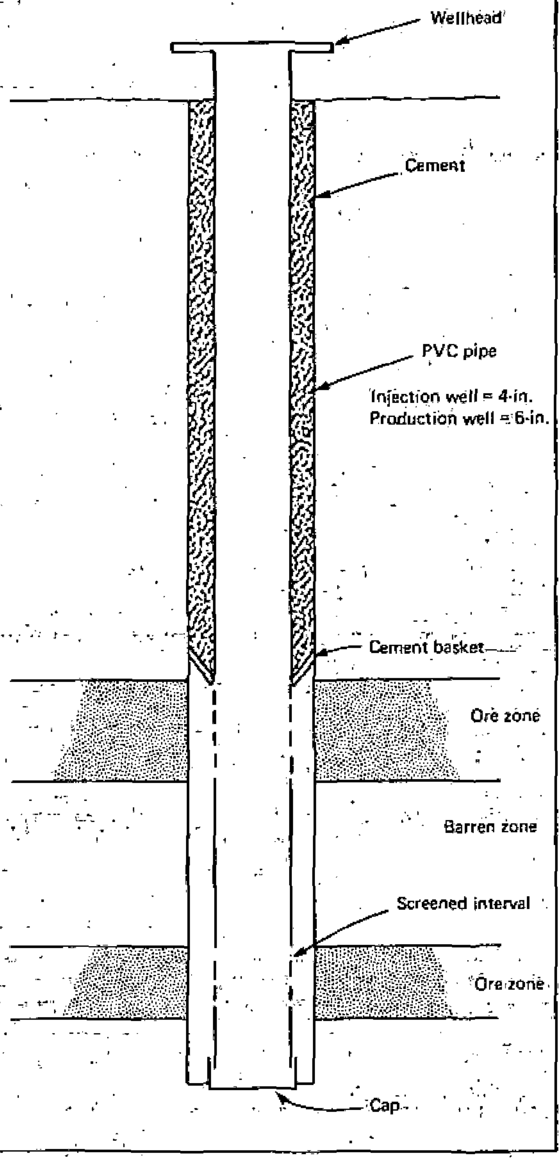


Fig. 3—Generalized section of Clay West injection and production wells



cash into the local economy. The mine is located in pleasant, rolling countryside that has produced well for farmers and ranchers but has provided few non-agricultural jobs. The Clay West mine offers an opportunity for some George West natives to secure long-term employment near home, and the $500,000 annual payroll will help support additional jobs in the George West community.

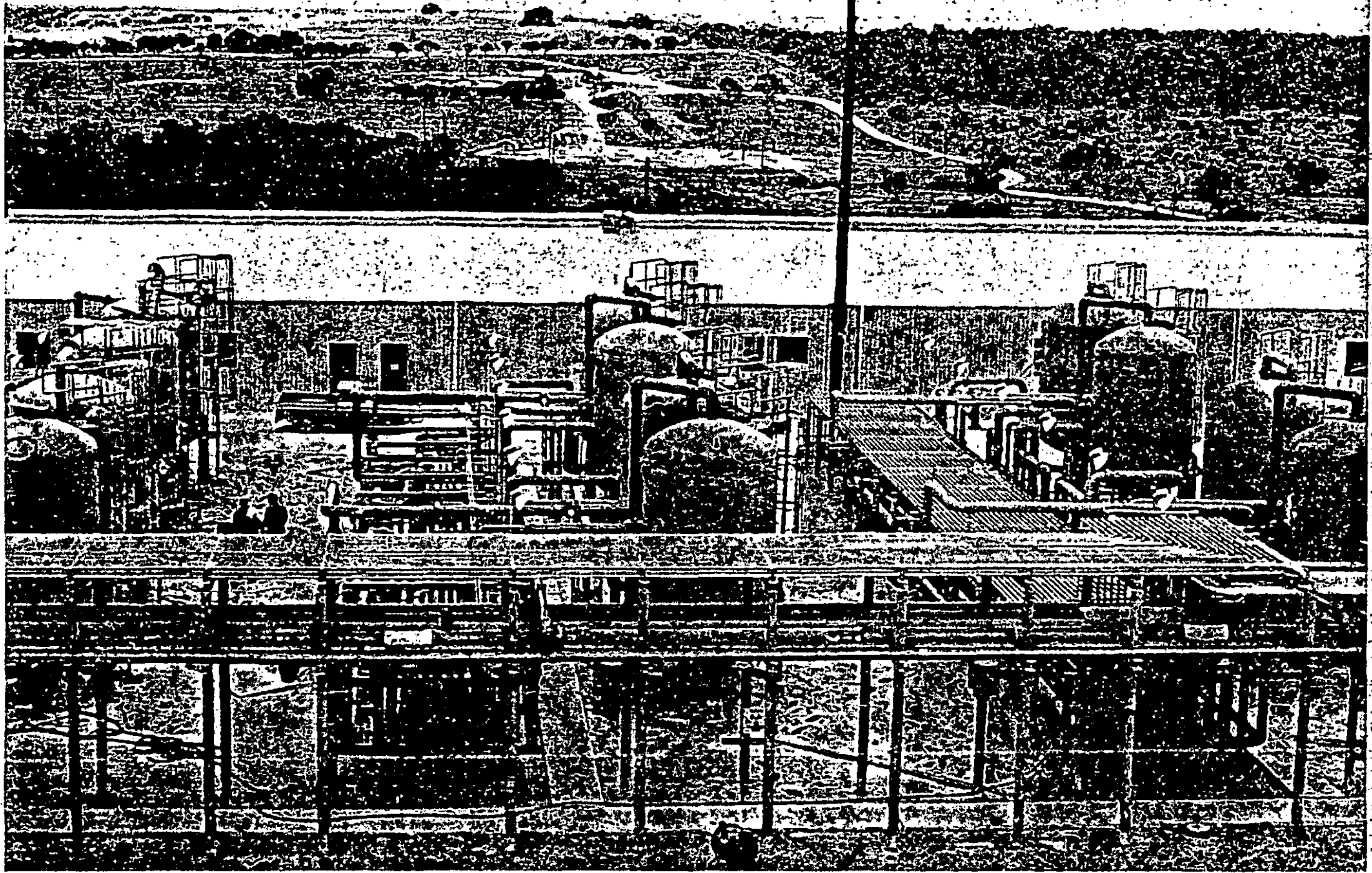
Arco has adopted a system of rotation for its employees that will regularly move each into a different job in various areas of mine and plant activity. The system is expected to create a more interesting work environment for the employees and more highly trained and valuable employees for Arco.

Controlling solution in the leaching zone

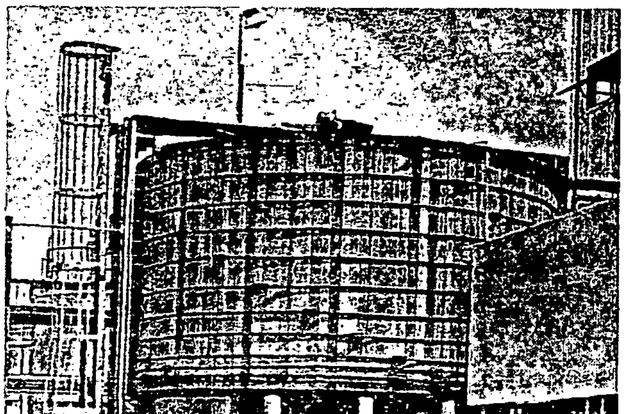
Waters present in the Clay West orebody are not now a source of potable water due to naturally-occurring high levels of radium. As noted above, the two primary environmental considerations will be to prevent contaminants from leaking out of the ore zone during leaching and to restore the water in the orebody after conclusion of leaching. During leaching, injection and production rates will be balanced, with each injection and production line equipped with direct-reading flow meters. A central instrument panel in the pattern area monitors the flow meters, and a daily check is made to balance total 24-hr production against total 24-hr injection.

All injection and producing wells in the leach area are cased and cemented to the surface, preventing upward migration of injected solutions behind the casing. Four shallow monitor wells drilled in the pattern area are sampled and analyzed routinely to determine whether the leach solution moves upward.

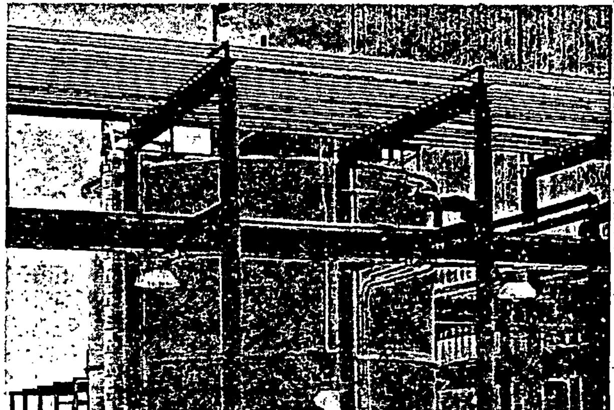
A detailed program of water well sampling was formulated by Arco in consultation with the Texas Water Quality Board, and all water wells within a 5-mi radius of the pattern were sampled and analyzed prior to production. During production, one-eighth of these wells will be examined every three months to detect any change in the



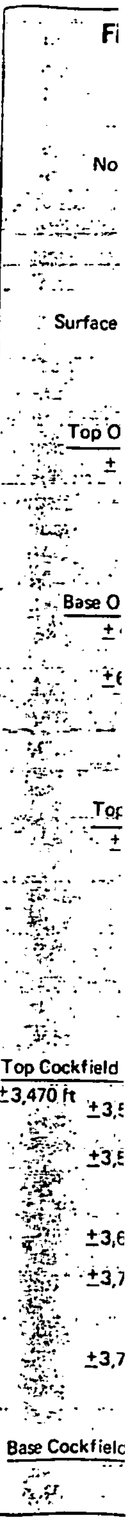
Six pairs of carbon column and ion exchange column tanks are the heart of the Clay West plant site. Building in the background houses offices and a well-equipped laboratory. Farm country beyond the building is typical of the George West area.



Clay West clarifier is a wood-stave thickener-type unit.

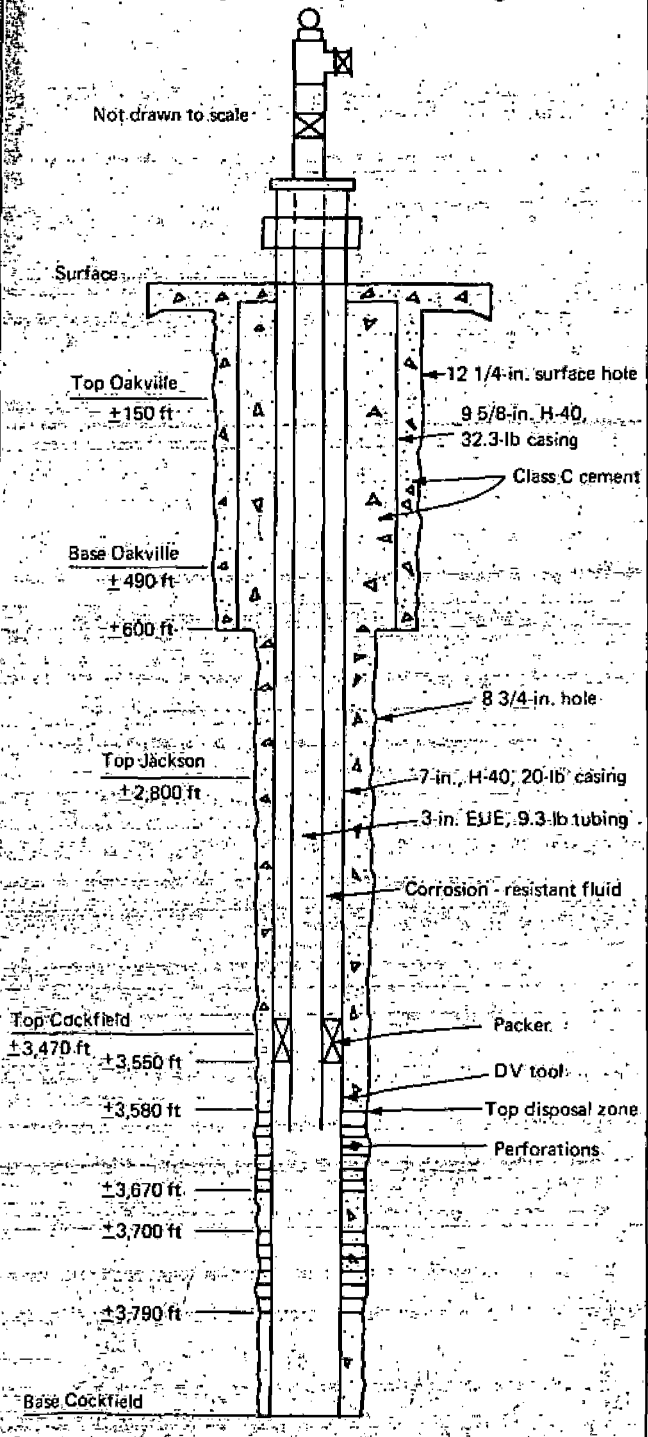


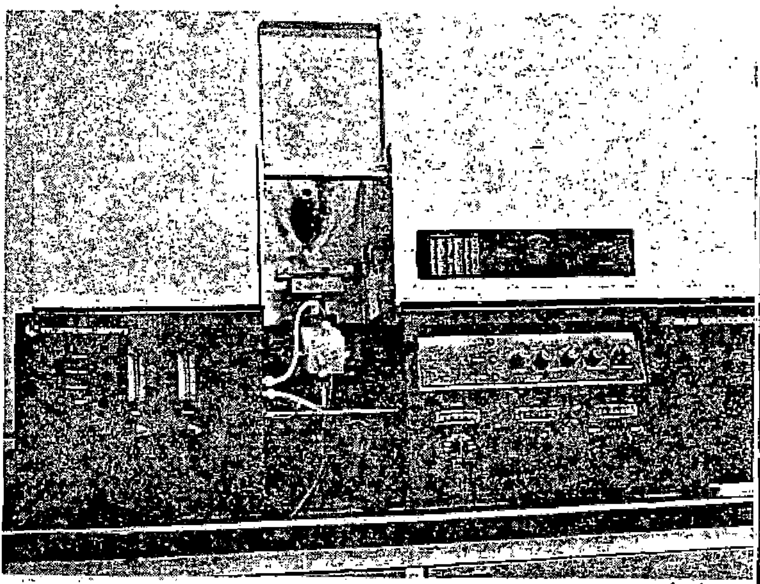
Rich eluate tank provides surge capacity ahead of charcoal column.



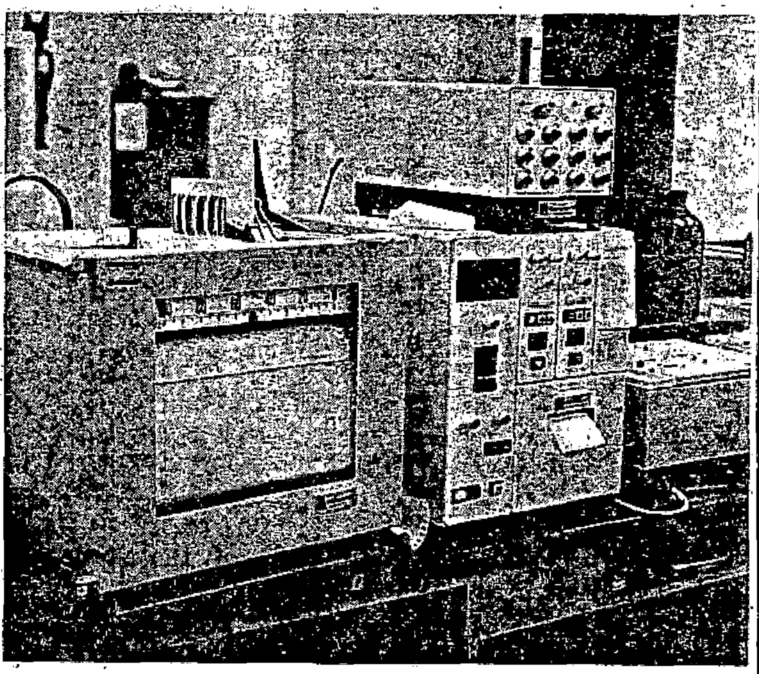
chemical ana
for calcium,
phate, chlori
dissolved sol
for Gross Al
exceeds thre
tation for rad
ceeds 50 pico
be performed
for drinking
List A analys

Fig. 4—Clay West disposal well diagram





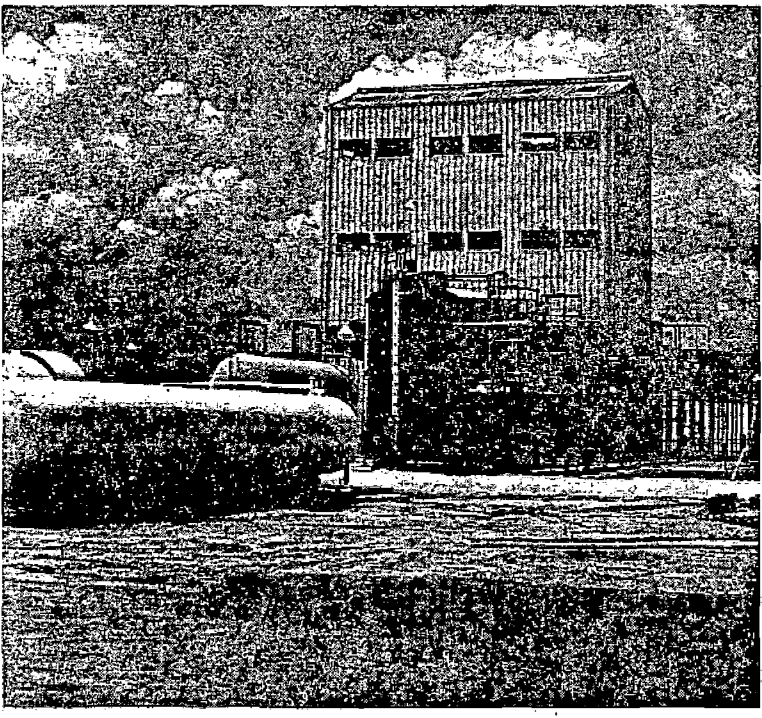
The Clay West laboratory is equipped with a spectrophotometer, a Technicon, and an atomic adsorption apparatus for analyzing the many samples required both for process solutions and for regional water sampling. Monitor wells in the leach area are sampled regularly to detect any migration of injected solutions. If migration occurs, the material balance of reservoir fluids can be controlled, resulting in a net flow of native water into the leaching area, preventing any migration outside the leach area.



chemical analysis of the water. The wells will be analyzed for calcium, sodium, iron, molybdenum, bicarbonate, sulphate, chloride, nitrate, nitrite, ammonium, pH, and total dissolved solids (List A). The same wells will be analyzed for Gross Alpha and Gross Beta radiation. If Gross Alpha exceeds three picocuries per liter, a barium co-precipitation for radium 226 will be performed. If Gross Beta exceeds 50 picocuries per liter, a lead 210 determination will be performed. Every six months, the nearest six wells used for drinking water will be examined radiometrically and List A analyses will be run.

During development of the Clay West leach pattern, 28 operating wells were sampled and analyzed for List A ions. In addition, 10 of these wells were analyzed for radiation and heavy metals—magnesium, fluoride, arsenic, barium, boron, cadmium, copper, chromium, lead, manganese, mercury, nickel, selenium, silver, uranium, zinc, simple cyanide and P-alkalinity as $CaCO_3$, and total alkalinity as $CaCO_3$ (List B). As part of this program, some wells were sampled and analyzed as many as four times prior to operation.

To monitor surface drainage, Spring Creek is sampled



Clay West process building houses precipitation tanks, rotary filter, heater, dryer, and packaging and shipping facilities.

upstream of the Clay West plant site and downstream of the pattern area. The Nueces River, which provides drinking water for Corpus Christi, is sampled above and below its confluence with Spring Creek. Samples are analyzed radiometrically and chemically.

Arco also organized an in-house environmental task force to perform a baseline study of the Clay West environment prior to moving into the area. The task force included biologists, radiologists, chemical engineers, petroleum engineers, geologists, hydrologists, and zoologists. Among its subjects of study were fish, amphibia, birds, insects, bacteria, shrubs, grasses, roots, stems, soils, and streams.

The present plan for orebody restoration is to pump contaminated solutions out of the leached zone into a deep disposal well until ion concentrations are down to acceptable levels. Pilot tests conducted by Dalco and US Steel demonstrated that when injection is stopped but production maintained, there is a gradual decrease in ionic species in the produced solutions. These data are proprietary but were released on a confidential basis to the Texas Water Quality Board. Other alternatives to pumping the orebody will also be studied, and techniques for converting the aquifer back from an oxidized state to a reduced state are under investigation.

The Clay West disposal wells—two are planned—will be drilled to a total depth of 4,500 ft, down through the Cockfield sandstone of Upper Claiborne (Oligocene?) age. The

On a smaller scale, other mines have recovered U_3O_8 by in-situ leaching

While the new Clay West in-situ uranium leaching mine in Texas is the first of its type to produce on a commercial scale, limited quantities of uranium have been produced by other in-situ mines operating now or in the past. A number of these operations are described briefly in "Analysis of Non-coal Underground Mining Methods," prepared by Dravo Corp. for the US Bureau of Mines and published in 1974.

Mine water near Grants, N. Mex., was found to carry 2-10 ppm of dissolved uranium, the report states. The slightly basic water (7 to 8 pH) takes the uranium into solution as the tricarbonate ion, which is readily extracted by ion exchange. Oxidation of the insoluble tetravalent uranium to a soluble hexavalent uranium is accomplished by exposure to air and possibly by bacterial action. After passing through ion exchange, the water is returned to the mine and sprayed on drift and stope walls. It then drains back to a sump for return to the ion exchange columns. Production of 5,000 to 15,000 lb per month is dependent on the flow of water and the ground area being wetted.

The same procedure is used in the Elliot Lake district in Canada, the report states, except that an acid environment is generated by the oxidation of iron sulphides in the ore.

"The *Pitch mine in Colorado* was water leached by drilling injection wells from the surface into the abandoned and caved stoping areas," the Dravo report notes. "The water percolated through the old mine workings, collected in the main haulage tunnel, and passed through an ion exchange column. The stripped water was then returned to the injection wells. Later in the program, soda ash and bicarbonate were added to the leach solution to improve extraction. This operation is now closed, but has potential for producing additional uranium by in-situ leaching."

Utah Mining and Construction Co. (now Utah International) experimented with a series of injection wells, usually three, around a production well. Based on groundwater flow, the injection wells were drilled upstream from the production well. Tracer dyes were used to study solution flow between injection wells and the production well. After flow patterns were established, an acid leach solution was injected. Withdrawal rate was slightly more than the injection rate. The pregnant solution was passed through ion exchange columns for uranium extraction, and then it was rejected because it contained soluble gypsum that would blind the leach area if reused.

"The area under leach from each set of wells was very small," according to the Dravo report, "seldom larger than 25-ft dia. The project never produced more than 7,000 or 8,000 lb per month of U_3O_8, even with as many as five-well systems operating. Leaching was discontinued in 1968, when the operation was converted to an open pit."

Exxon (Humble Oil) is experimenting with the same leach method in the Powder River Basin in Wyoming, *the report states, except that a carbonate system is used,* permitting recirculation of the leach solution.

"With the depletion of higher grade deposits of uranium," the report concludes, "much of the future production may depend on successful in-situ leach extraction. This is particularly important for small, isolated deposits of lower grade values that will not economically support conventional . . . extraction."

Cockfield is the proposed disposal zone.

During aquifer renovation following leaching, Arco expects to pump an estimated 150 gpm of slightly saline water from the leach pattern, with a probable maximum of 2,000 mg per liter total dissolved solids. This water will be pumped to the disposal well after most of the uranium has been removed. The native water in Cockfield sands is much more saline than the disposal stream.

The surface hole will be drilled 12½-in. dia and 600 ft deep, through the Oakville aquifer and into the underlying Catahoula clays. H-40, 32.3-lb pipe with a diameter of 9⅝ in. will be set as casing to 600 ft, and Class C cement circulated to the surface. An 8¾-in. hole will then be drilled to 4,500 ft. Upon completion of drilling, electrical and bulk density logs will be run in the hole.

Casing—7-in.-dia, 20-lb, H-40 pipe—will be set on the bottom and cemented in two stages. The first stage will bring cement to the top of the Cockfield sand, a depth of about 3,500 ft. Utilizing a D-V tool, sufficient additional cement will be pumped to circulate to the surface.

Injection intervals will be determined from electric and gamma-ray logs. Proposed intervals are from 3,580 ft to 3,670 ft and from 3,700 ft to 3,790 ft. The balance of the Cockfield sand between the depths of about 3,790 ft and 4,500 ft is considered a reserve for future disposal, if volumes or pressure gradients exceed present estimates.

When the ore zone of a leaching pattern has been restored, Arco will turn the surface back to normal use. The area will need reseeding, but reclamation in the usually understood sense will be unnecessary. The leaching operations will leave the surface essentially undisturbed. Within a few years after leaching stops, it may be difficult to identify a leach pattern area as ever having been different from the surrounding countryside. ☐

Developers eye Texas potential for in-situ uranium leaching

Emmy Crawford, McGraw-Hill World News, Houston

In addition to the Arco-US Steel-Dalco plant now in production near George West, Tex., several other in-situ uranium leaching projects are in various stages of planning, design, and construction in south Texas.

The most advanced project belongs to Westinghouse's wholly owned subsidiary, Wyoming Minerals, which is building a 250,000-lb-per-year plant near Bruni, Tex., about 40 mi east of Laredo. While Wyoming Minerals has not revealed any details of its process, a source at the Texas Water Quality Board states that the operation will use a weak ammonia leach and a proprietary ion exchange extraction system.

Wyoming Minerals also plans a second commercial uranium in-situ leach plant near Ray Point, north of George West in Live Oak County. The Texas Water Quality Board has received an application for this operation, but as of early June, no date had been set for a hearing.

In Duval County, Union Carbide is considering a pilot-scale in-situ leach operation, but for it too, no hearing date had been set by early June.

(While not of direct interest to Texas uranium production, a Burlington Northern-Wyoming Minerals joint venture calls for Wyoming Minerals to conduct minerals exploration on 8.4 million acres of Burlington Northern railroad property in Oregon, Washington, Idaho, Wyoming, Montana, North Dakota, Minnesota and Wisconsin. The agreement, which includes gas, oil, and uranium as well as other minerals, will run five to eight years, and Wyoming Minerals will get up to 50% interest in discovered properties, depending on how much money it invests in the project. The joint venture company—called Bur-West—is headquartered in Billings, Mont.)

Mobil pilot plant is under construction

Southeast of Bruni, in Webb County, Tex., Mobil Oil Corp. is building a pilot plant to test the feasibility of a proprietary in-situ uranium leaching method. The plant is expected to start up around October 1 to leach U_3O_8 in place, at depths of 410 to 430 ft.

Present plans call for an initial 18-month test program, with an overlapping 15-month test at a nearby site if the first program is successful. The capacity of the pilot scale operation will not be determined until solution is actually put into the ground, and Mobil has no present timetable for commercial development.

The Mobil pilot operation will inject a leach solution into the uranium-bearing formation through 15 injection wells and extract it through seven production wells. The solution is identified as a dilute ammonium carbonate made by mixing gaseous ammonia and carbon dioxide with formation water treated with ordinary water softener. An unidentified oxidant is also added. Solution produced will be collected in a surge tank and then pumped to a holding tank at the recovery plant site, about 700 ft away.

A sand filter will remove suspended solids, and an ion exchange column will extract U_3O_8. The barren effluent from the ion exchange column will be pumped to a holding tank where an oxidant and leach chemicals are added, and the solution will be recycled to the injection wells.

Uranium will be extracted from the ion exchange resin by an unidentified chemical, and the U_3O_8-rich eluate will be pumped to a holding tank and then to a precipitation tank for uranium precipitation. Ammonia and carbon dioxide gases generated in the precipitation process will be vented to a 20-ft-high fume scrubber. Water spray in the scrubber will absorb the vapors, and the resulting solution will be recycled to the injection well stream.

Filters, ion exchange, and water softener tanks will be backwashed periodically and the water held in a 500-bbl tank. If backwash water is found to contain excess solid or radioactive material, it will be pumped to an evaporation pit.

A system of eight monitoring wells will determine if there is leakage of chemicals outside the planned injection production pattern.

SOCIETY OF
MINING ENGINEERS
of
AIME

540 ARAPEEN DRIVE – SALT LAKE CITY, UTAH 84108

SUBJ
MNG
ISLS

PREPRINT
NUMBER

78-AS-102

178

UNIVERSITY OF UTAH
RESEARCH INSTITUTE
EARTH SCIENCE LAB.

IN-SITU LEACHING SIMULATION STUDIES OF URANIUM ORES

P. S. Sundar
Senior Engineer

Westinghouse Electric Corporation
Research and Development Center
Pittsburgh, Pennsylvania

ENGINEERING SOCIETIES
APR 18 1978
LIBRARY

For presentation at the 1978 AIME Annual Meeting
Denver, Colorado - February 28 - March 2, 1978

NOTICE: THIS MATERIAL MAY BE PROTECTED BY COPYRIGHT LAW (TITLE 17 U.S. CODE).

Permission is hereby given to publish with appropriate acknowledgments, excerpts or summaries not to exceed one-fourth of the entire text of the paper. Permission to print in more extended form subsequent to publication by the Institute must be obtained from the Secretary of the Society of Mining Engineers of AIME.

If and when this paper is published by the Institute, it may embody certain changes made by agreement between the Technical Publications Committee and the author, so that the form in which it appears here is not necessarily that in which it may be published later.

These preprints are available only on a coupon basis. The coupon books may be obtained from SME headquarters for $10.00 a book (10 coupons) for SME members, $15.00 for members of other AIME societies, or $25.00 for nonmembers. Each coupon entitles the purchaser to one preprint. Mail completed coupons to PREPRINTS, Society of Mining Engineers, 540 Arapeen Drive, Salt Lake City, Utah 84108.

PREPRINT AVAILABILITY LIST IS PUBLISHED PERIODICALLY IN
MINING ENGINEERING.

Abstract. A laboratory method for the simulation of in-situ leaching of uranium was developed under contract to the Interior Department's Bureau of Mines. Simulation tests using sulfuric acid, ammonium bicarbonate and sodium carbonate as reagents, and hydrogen peroxide as oxidant were carried out with representative ores from Texas and Wyoming. The tests not only yielded recovery rates of uranium and permeability changes, but also qualitative inferences on the chemical and physical phenomena which occur underground. Leaching with basic solutions was found to be kinetic constrained and with acid solutions, equilibrium constrained. The rapid permeability changes observed when leach solutions were introduced, were probably caused by gas blockages in the packing. Precipitation of calcite or gypsum and the reaction of clay to leach solutions had slow and gradual effect on the packing permeabilities.

Introduction

In-situ leaching is becoming increasingly attractive as a method for recovering uranium from secondary ore deposits, since it offers the advantages of (a) low capital and operating costs, (b) minimum disturbance of the environment, and (c) reduction in the time required to develop the mine. To understand in-situ leaching phenomena, investigators have generally resorted to experimentation at three levels: (a) Batch agitation tests, (b) Tests on short lengths of undisturbed ore, and (c) Field tests on a typical portion of the ore body.

The batch agitation leach tests use a small quantity of ore (10 to 500 g) which is agitated with a known amount of leach solution. Samples are withdrawn with time to identify the kinetics and the equilibrium extraction of leaching. Agitation leach tests at best provide information on the chemistry of leaching and should, as such, be used mainly to identify potential leach reagents. The tests may also provide a limited amount of information on the leach reaction mechanism. The agitation leach tests, however, do not reproduce the physical ore body characteristics (such as permeability, bed density, mineralogical orientations, particle size, etc.) or the hydro-dynamics of underground leaching. Therefore, it is not an effective method of studying the physical phenomena of in-situ leaching.

Although the leach tests using short (2 to 4 foot) lengths of core samples approximate in-situ leaching better than the agitation leach tests, these tests have several limitations: (a) the distribution and the amount of uranium in the core is not known precisely, (b) because of the heterogeneous nature of the deposit, the core may not contain the desired uranium levels, and (c) the length of the cores are, in general, perpendicular to the direction in which the leach solution would flow during in-situ leaching. Because of the anisotropic nature of the ore deposit, the permeabilities observed during the tests are not representative of those in the field.

The principal limitation, however, is the short geometry of this system. Many chemical phenomena (such as chromatographic effects, reprecipitation or recrystallization of minerals, gas blockage, depletion front movement, etc.) cannot be detected in such a system.

Field leaching tests reproduce actual leaching conditions in all respects, but their high cost limits their number. Furthermore, there is limited knowledge and little control over the conditions existing underground. Although complex theoretical mathematical models have been developed to predict flow patterns of the leach solution and the volume of ore subjected to leaching, the inhomogeneities and the changes in ore body characteristics during the leach operation render the selection of correct parameters for such models very difficult and verification of observations and analyses impractical. No visual observations of the underground body during leaching is feasible and it is difficult to locate precisely the sample withdrawal points. Furthermore, most ore bodies are remote enough to require detailed advance planning and long delays if equipment breaks down. In general, field testing is a good verification method but a very inefficient investigative technique for understanding in-situ leaching phenomena.

A less expensive and more convenient method of testing is laboratory simulation of in-situ leaching. This is an acceptable method of investigation when the ore body is represented by a geometry and packing which reasonably reproduce the hydraulics of in-situ leaching. Advantages of this method include the ability to visually observe leaching, the ability to at least partially control some of the ore body characteristics, the ability to determine characteristics of the uncontrolled variables and the ability to conduct reproducible experiments. In addition, the experiments take less time than field tests because of the smaller sample.

Disadvantages or limitations of the laboratory simulation tests lie in the difficulty of obtaining a sample which represents the underground ore body (in terms of permeability, uranium content, oxidation state, reaction surface area and mineralogy) and in understanding quantitatively the differences between the controlled test conditions and the non-ideal conditions encountered in the underground deposit. Although prediction of field behavior from the results of laboratory simulation tests is not quantitative, these tests do provide a common denominator for studying the leach behavior of various types of ores.

This paper summarizes some of the results and conclusions arrived at during the first three phases of a program to develop a technique for the laboratory simulation of in-situ leaching of uranium and the associated environmental restoration of the post-leach ore body.

The program was performed under contract to the U.S. Department of Interior's Bureau of Mines, Washington, D.C., with D. C. Seidel of the Bureau of Mines Salt Lake City Metallurgy Research Center as Technical Project Officer, and John Arnold of the Bureau of Mines, Denver, Colorado, as the Contracting Officer.

The principle objectives of the first three phases of the program were: (a) Acquisition and characterization of two 3-ton samples of uranium ore, one each from Texas and Wyoming, and similar to those being leached in-situ; (b) Determination of the agitation leach behavior of each ore sample using sulfuric acid, ammonium bicarbonate and sodium carbonate as leach reagents and hydrogen peroxide and oxygen as oxidants; (c) Design and construction of three similar in-situ leach simulation hardware systems and (d) Determination of the effects of leaching variables upon permeability and uranium recovery using sulfuric acid, ammonium bicarbonate and sodium carbonate as leach reagents, and hydrogen peroxide as oxidant.

The observations, results and conclusions from the first three phases of the program are available in the Bureau of Mines Open File Report No. 140-77.

The In-Situ Simulation System

In designing a system to simulate the in-situ leaching of underground ore bodies, it is important to take into consideration such aspects as the flow characteristics of leach solution, the physical characteristics of the ore body, and the leaching conditions which control the kinetics. The following features were considered in arriving at an acceptable design:

- Ore sample container of representative geometry (cylindrical, rectangular slab, etc.) capable of operating at temperature and pressure constraints occurring in in-situ leaching.
- Adequate sampling ports for obtaining solution and ore samples for analysis and characterization.
- Sufficient instrumentation to monitor all operating parameters (flow, pressure, temperature, etc.) and to allow identification of the state of the system.
- A leach solution(s) feed system to the ore sample container that does not superimpose any additional changes in the bed characteristics such as packing density, voidage, homogeneity, etc.
- Provision for use of oxidant injection into the feed.
- Visibility of the leach process in progress.
- Safe operating conditions.
- Ease of maintenance and assembly.

A schematic diagram of the in-situ simulation system designed and constructed is given in Figure 1. The system consists of a pressurized feed section, from which the leach solution is fed to the ore container through flow metering and monitoring devices. The pregnant leach solution is collected in large tanks with provision for mixing. A pressurized feed section was selected to prevent any movement or re-orientation of the ore packing that may be induced if a positive displacement or other pulsing pump is used. Furthermore, absence of any moving parts in the pressurized mode of feed makes this method more reliable than pumping. A pressurized system also minimizes the loss of volatile reagents such as ammonium carbonate from the feed solution.

The feed section consists of two primary Type 316 stainless steel captive gas tanks (one each for the leach reagent and the oxidant solutions) of 16 gal. capacity with PVC liners and two similar reserve captive gas tanks of 2 gal. capacity each. The PVC liners separating the solution and the pressurizing gas were introduced in order to prevent nucleation of the dissolved pressurizing gas in the ore packing as the static pressure decreases. This is particularly true of inert gases (argon or nitrogen), if used. However, the solubility of gas can be exploited in tests with gaseous oxidants such as oxygen, to supply the oxidant to the ore. The reserve tanks serve as feed tanks during recharging of the primary feed tanks. The feed section is capable of delivering solution up to a maximum working pressure of 120 psig.

The solution flow from each tank (both primary and reserve) is metered through a needle valve and monitored by a 0 to 50 ml/min flowmeter. Solution check valves downstream from the flowmeter prevent backflow of solutions into the tanks. The solutions mix downstream before entering the pressure switch. Toggle valves shut off the feed from either the primary or the reserve tanks. By providing individual metering and monitoring capability on the flow from each tank, it is possible to control closely the desired oxidant and reagent compositions.

Each of the three in-situ simulation systems constructed consists of two clear glass cylinders of 4 in. inside diameter and 13 ft. long to allow at least a 12-ft. long ore packing to be leached. These glass cylindrical containers are rated for a maximum allowable working pressure of 50 psig. The component pieces and the lengths of packing to be employed for leaching are illustrated in Figure 2. The entrance to the column, the first 7 inches, is to be filled with fine casting sand (silica) of permeability less than that of the ore sample packing to obtain a uniform distribution of leach solution prior to contacting the ore. This fine sand packing is followed by a 12-ft. packing of the ore sample for leaching. The last 5 inches of the cylinder is packed with coarse casting sand. The packing in the cylinder is held in place during leaching by Type 316 stainless steel coarse mesh, double-weave wire cloths at the inlet and the outlet ends.

To monitor and obtain solution samples from the ore packing during the test, the ore packing was divided into three sections. The tee-sections allow insertion of pressure tap and facilitate sample withdrawal. The permeability of the sections of ore was determined from the differential pressure readings across these sections. The concentration profiles and the rate of exhaustion front movement were estimated based on the solution samples collected.

The Simulation Test Method

The development of a packing method that would consistently reproduce packing permeability was paramount to the success of the simulation tests. Based on limited data on the field permeabilities of uranium mineralizations, a water permeability in the range of 0.5 to 2.5 darcy was preferred in developing a packing method.

The correlation (Darcy equation) used in the estimation of the permeability during the simulation tests is given by

$$\frac{\Delta P}{L} = \alpha \cdot \mu \cdot V \qquad (1)$$

where $\frac{\Delta P}{L}$ = Flow pressure gradient, atm/cm
V = Liquid superficial velocity, cm^3/cm^2 sec.
u = Solution viscosity, centipose
$\frac{1}{\alpha} \triangleq$ Permeability, Darcy (by definition)

The equation (1) could be rewritten as

$$\frac{1}{\alpha}\,[\mathrm{Darcy}] = \frac{\mu(V')/60\ A}{(\frac{\Delta P'}{14.7})/(L' \times 2.54)} \qquad (2)$$

where V' = Solution Volumetric flow, cm^3/min

A = Packing area of cross-section, cm^2

$\frac{\Delta P'}{L'}$ = Flow pressure gradient, psi/in.

and L' = Packing length, inches.

For a 4-in. inside diameter column, the area of cross-section $A = 81.06\ cm^2$. Assuming solution viscosity = 1.0 cp, the permeability is given by the equation

$$\frac{1}{\alpha}\,[\mathrm{Darcy}] = 7.677 \times 10^{-3}\ (\frac{V'L'}{\Delta P'}) \qquad (3)$$

This correlation has been consistently used to estimate the permeability of the 4-in. inside diameter column packing.

The Texas ore sample used in the tests represented fine grain sandstone ore of medium to dark gray color which had been recently mined from the Sickenius Pit. This ore with a moisture content of 18.5 percent was predominantly unconsolidated, and equilibrated in deionized water to a pH of approximately 4.0. Since possible preoxidation could be a cause of such low equilibrium pH, tests were carried out to estimate the degree of ore preoxidation. Approximately 20 percent of the uranium in the ore was extractable by water in the presence of sufficient sulfate and carbonates during these tests.

The Wyoming ore sample acquired for the tests was very poorly consolidated, coarse grained, light gray to tan in color, and had a moisture content of 5.8 percent. Each sample was spread over the floor and air dried. After screening the entire sample at 10 mesh (Tyler), it was stage crushed and ground to -10 mesh. The entire -10 mesh ore thus obtained was blended together by coning and quartering using a front-end loader. The size distribution of each sample was determined by both dry and wet screening. The results of this determination are summarized in Table 1 for the Texas ore sample and in Table 2 for the Wyoming Ore Sample used in the simulation tests. The chemical and mineralogical characterization results for the two samples are summarized in Table 3.

The ore samples were packed into the glass columns according to the methods developed for each ore sample. It was opted to pack the ore dry to minimize preoxidation during the packing operation. The columns were packed in a vertical-position and were leached in a horizontal-position to eliminate any gravity effects on solution flow, fines migration and packing density variations due to its own weight. The packings were examined visually for voids, faults and gross segregation prior to acceptance for water flow test. The packings were flow tested using deionized water over a period of 125-200 hours to ensure correct permeability and stability. To obtain maximum stability, the packings were allowed to consolidate prior to accepting the full test flow rate. The dry ore packing was wetted very slowly (over a period of 24 to 48 hours) till solution breakthrough was observed at the outlet. The flow was maintained at approximately 2 to 5 ml/min. in the first 24 hours after breakthrough. At the end of this period, it was gradually increased to its full value over a period of approximately 18 hours. A solution migration rate of 1 ft/hour in the packing was considered as full test flow rate. This corresponds to approximately 10-12.5 ml/min solution volumetric flow rate.

TABLE 1

Summary of the Screen Size Analysis Results for the Texas Ore Sample

Sample Size: Dry Screen Size Analysis = 2000 g
Wet Screen Size Analysis = 825 g

Screen Product				Cumulative Passing		
(Tyler) Mesh		Dry Weight %	Wet Weight %	(Tyler) Mesh	Dry Weight %	Wet Weight %
-10	+14	0.2	0.8	10	100.0	100.0
-14	+20	2.9		14	99.8	—
-20	+28	3.5		20	96.9	—
-28	+35	2.7		28	93.4	—
-35	+48	5.5	1.1	35	90.7	99.2
-48	+65	10.0	3.5	48	85.2	98.1
-65	+100	47.9	33.9	65	75.2	94.6
-100	+150	17.6	34.7	100	27.3	60.7
-150	+200	4.2	7.7	150	9.7	26.0
-200	+270	1.6	1.4	200	5.5	18.3
-270	+325	0.5	0.7	270	3.9	16.9
-325		3.4	16.2	325	3.4	16.2

The feed was switched to the chemical leach solution from deionized water, if the packing was found acceptable. The criteria for rejection could be (a) permeability of any one section outside the range of 0.5 to 2.5 darcy and (b) development of a fault such as a crack or flow compaction of the packing. The feed solutions were stored in separate feed tanks to prevent premature decomposition of the peroxide and were mixed prior to the packing inlet. The

TABLE 2

Summary of the Screen Size Analysis Results for the Wyoming Ore Sample

Sample Size: Dry Screen Size Analysis = 2000 g
Wet Screen Size Analysis = 903 g

Screen Product			Cumulative Passing		
(Tyler) Mesh	Dry Weight %	Wet Weight %	(Tyler) Mesh	Dry Weight %	Wet Weight %
-10 +14	9.3	8.5	10	100.0	100.0
-14 +20	25.1	22.6	14	90.7	91.5
-20 +28	16.0	14.1	20	65.6	68.9
-28 +35	9.6	8.2	28	49.6	54.8
-35 +48	8.4	7.2	35	40.0	46.6
-48 +65	7.7	6.3	48	31.6	39.4
-65 +100	7.4	5.4	100	23.9	33.1
-100 +150	5.0	4.4	150	16.5	27.7
-150 +200	4.0	3.8	200	11.5	23.3
-200 +270	1.9	1.8	270	7.5	19.5
-270 +325	0.7	1.1	325	5.6	17.7
-325	4.9	16.6		4.9	16.6

TABLE 3

Ore Characterization Summary for Texas and Wyoming Ore Samples

Fraction Description	Texas Ore Sample Weight %	Wyoming Ore Sample Weight %
U_3O_8	0.07%* 0.056%	0.119%* 0.091%
CaO	1.63%	1.12%
MoO_3	0.0015%	0.001%
V_2O_5	0.041%	0.014%
Fe	0.58%	0.94%
CO_2	0.45%	0.84%
SO_4	0.16%	0.005%
Total S	0.24%	0.33%
-325 mesh (dry screen)	3.4%	4.9%
-325 mesh (wet screen)	16.2%	16.6%
Clay fraction	6.9%	9.1%
U-distribution (dry screen)	27% in -100M (27%)	54% in -48M (32%)
U-distribution (wet screen)	70% in -150M (26%)	81% in -48M (39%)
U-minerals	Not identified	Not identified
Clay minerals	Montmorillonite	Montmorillonite
Heavy minerals (2.8)	Mica, pyrite, magnetite	Clorite, clinozoisite, pyrite, magnetite
Bulk sample mineralogy	Quartz, montmorillonite, feldspar	Quartz, feldspar, montmorillonite, mica
Feldspar/quartz ratio	1÷5	1÷2
Volcanic rock fragments	yes	no
Rock classification	Feldspathic Litharenite	Lithic Arcose

---

[*] Mean values determined at Westinghouse R&D Center, based on analyses in triplicate.

adjustments to the desired flow were made by increasing (or decreasing) the inlet pressure. It was, therefore, constrained by the maximum allowable operating pressure for the glass cylinder of 50 psig. For the simulation tests, this limit was lowered to 45 psi for added safety.

At the termination of the leach test, the packing section was disconnected from the system and allowed to drain vertically. After draining, the packing was returned to a horizontal position for approximately 72 hours to allow the remaining moisture to distribute uniformly. The pressure profile along the length of the column was recorded every 6 hours to permit estimation of permeability changes. The cumulative product solution volume was measured every 48 hours and a solution sample was collected for uranium analysis. Solution samples along the length of the packing were collected approximately every 48 hours to determine uranium concentration profiles. A chemical leach period of a minimum of 750 hours was observed during the tests.

Simulation Test Results

In-situ leach simulation tests were carried out with each of the ore samples acquired using deionized water, 1.0 g/l ammonium bicarbonate -0.5 g/l peroxide, 1.0 g/l sodium carbonate -0.5 g/l peroxide, and sulfuric acid solution of pH = 1.0.

In addition to the above, simulation tests were carried out with the Wyoming ore sample only using 1.0 g/l ammonium bicarbonate, 0.5 g/l peroxide, and 1.0 g/l ammonium bicarbonate -0.5 g/l peroxide.

The latter three tests were carried out to study the effect of each reagent on the leaching process and to verify the reproducibility of the in-situ simulation test method. The specific objectives of each simulation test were to determine:

- The maximum overall uranium recovery over the test duration.
- Uranium recovery as a function of the volume of leach solution required at a fixed solution flux through the ore.
- Number of bed displacements required to obtain the maximum recovery.
- Permeability change with time.

The simulation leach studies also sought to derive qualitative inferences on the possible leach controlling mechanism, effect of solution flux upon recovery rate, and the physical phenomena occurring in the simulated ore body (packing) during the test.

The observations and results from all the above tests are given in their entirety in the Bureau of Mines Open File Report No. 140-77. In the following only the permeability variations and the uranium recovery rates observed during the simulation leach tests with the Wyoming ore sample using deionized water, 1 g/l ammonium bicarbonate -0.5 g/l peroxide, 1 g/l sodium carbonate -0.5 g/l peroxide and sulfuric acid of pH = 1.0, and that with the Texas ore sample using deionized water and sulfuric acid (pH = 1.0) are summarized.

Water Leach Tests

The simulation leach tests using deionized water as leach solution was carried out as a control test to establish the baseline for the permeability changes and the uranium recovery rate during the chemical leaching tests. The permeability variations for the three sections of the packing observed during the deionized water flow test with the two ore samples were similar. The observations are illustrated in Figure 3 for the Wyoming ore sample and in Figure 4 for the Texas ore sample. In the case of both the ore samples, the permeability of the front and the middle section remained near initial values throughout the test. However, the end section permeability decreased to a minimum in the first approximately 350 hours, followed by a gradual recovery to near initial conditions over the remainder of the test duration. This trough shape variation of the end section permeability was attributed to the migration of the fines fraction (decreasing permeability period) followed by reorientation of the packing to the flow (permeability recovery period).

The uranium recovery rate observed, and the total carbonate and bicarbonate concentration of the product solutions (expressed as bicarbonate ion concentration) are illustrated in Figure 5 for the Wyoming ore sample. The uranium recovery rate and the sulfate ion concentrations of the product solution observed during the test with the Texas ore sample are given in Figure 6. Because of the low product solution pH (< 4.8) in the case of the Texas ore, the sulfate ion concentrations (principal anion) instead of the carbonate-bicarbonate levels, were determined. In both the tests, the uranium recovery rate was directly proportional to the principal anion concentration in the solution. The overall uranium recoveries estimated based on the solution values (approximately 63% for the Wyoming ore and 58% for the Texas ore) were high relative to that estimated on the post leach ore residue analysis (approximately 30% for the Wyoming ore and 48% for the Texas ore).

The product solution pH indicated no significant change with time over the duration of the tests. The solution equilibriated to pH approximately 8.25 with the Wyoming ore and to approximately 3.7 with the Texas ore sample. The product solution redox potential (measured on a Pt-Ag/Ag^+Cl^- electrode system) in both tests indicated the reducing nature of the ore packings. The redox potential observed were in the range of +190 mV to +250 mV with the Wyoming ore sample and +220 mV to +280 mV with the Texas ore sample.

Ammonium Bicarbonate Leaching

The permeability and flow variations observed during the simulation test with the Wyoming ore sample using ammonium bicarbonate-peroxide leach solution is given in Figure 7. Upon introduction of the ammonium bicarbonate-peroxide leach solution, the front ore section permeability decreased from approximately 0.75 darcy to 0.25 darcy in the first 40 hours, and displayed a gradual decrease during the remainder of the test. The permeability of the middle and end section decreased to a lesser degree than that of the front section upon chemical introduction. However, these recovered to near original values during the test.

This test is an example of shift in the system operation mode from constant flow to constant inlet pressure. Adjustments to maintain constant flow were made only during the first approximately 50 hours, as the maximum acceptable inlet pressure (45 psig) was attained at the end of this period. The leach system was allowed to drift with respect to volumetric flow rate over the remainder of the test. During this test, it was difficult to maintain an equal volume flow of each solution (reagent and oxidant). The actual oxidant-to-reagent flow ratio varied from 0.82 to 2.5 based on average flow rates calculated using the volume changes in the feed tanks.

The uranium recovery observed and the bicarbonate ion concentration of the product solution with time for the Wyoming ore sample are illustrated in Figure 8 as a function of the product solution volume collected. Based on the product solution uranium values, 86.7% of the uranium in the packing was recovered. Based on the analysis of the ore before and after the leaching operation, the overall uranium recovery achieved was 78.8%. Even though the bicarbonate concentration varied over the duration of the test, the recovery rate seemed unaffected by it. This is due to the presence of sufficient anion to complex the available uranium, even at the minimum reagent concentration of 570 ppm obtained at oxidant to reagent flow ratio of 2.5. The uranium concentration levels observed in solution were typical of in-situ leach solution levels. The equilibrium pH observed was nearly constant in the range of 7.80 to 8.20. The redox potential of the solution samples were, in general, lower than those observed with deionized water by approximately 50 to 100 mV. The decreasing redox potential from the inlet to the outlet was still observed. A duplicate leach test was carried out to verify the reproducibility of the experimental method. Similar permeability variations as observed in the first test were observed. An overall uranium recovery of 81.2% based on solution concentrations and 73.9% based on the residue concentrations was obtained during the duplicate test. The results indicated that the simulation test method was reproducible within the margin of experimental variance.

The leach test with the Texas ore sample using 1.0 g/l ammonium bicarbonate -0.5 g/l peroxide leach solution was aborted because of loss of control of the feed reagent to oxidant composition ratio. The poor permeability of the packing to the leach solution caused flow rates too low to monitor leading to loss of control.

Sodium Carbonate Leaching

In packing the column for the leaching of the Wyoming ore using sodium carbonate-peroxide solution, a 50-50 mixture of sand and ore was used at the inlet instead of the sand only. This was done to give the inlet section additional mechanical stability. Upon introduction of the 1.0 g/l sodium carbonate -0.5 g/l peroxide leach solution (pH $\simeq$ 9.0), the permeability of the column decreased very rapidly, leading to loss of control over the feed composition. Near plugging conditions were reached in approximately 50 hours from the time of chemical introduction. The permeability and the flow changes observed during this test are shown in Figure 9. Similar ore behavior was observed during the test with the Texas ore sample using sodium carbonate-peroxide solution. The rate of permeability loss was more rapid in the case of the Texas ore, than that observed with the Wyoming ore sample. These tests were aborted due to loss of control on feed, before the carbonate breakthrough at the outlet could be obtained. Therefore, no uranium recovery rates for the sodium carbonate-peroxide leach solution were obtained.

Sulfuric Acid Leaching

The permeability and flow variations observed during sulfuric acid leaching of the Wyoming ore sample are given in Figure 10. Upon acid injection the front ore section permeability decreased rapidly in the first 50 hours (from 2.0 to 0.25 darcy), followed by a gradual decrease at a very slow rate. A similar decrease but of lower magnitude was observed of the middle and the end section permeabilities. However, gradually recovered (in approximately 300 hours) to near their original values.

The uranium recovery obtained and the sulfate ion concentration in the product solutions collected as a function of the total volume of solution processed, for the Wyoming ore sample, are illustrated in Figure 11. The uranium concentration in the product solution exhibited a very sharp breakthrough and cut-off with minimum tailing. Approximately 89.2% of the uranium in the ore was recovered based on the solution concentrations and approximately 83.7% based on the solid residue analysis. Straight line recovery data generally indicate an equilibrium constrained system, and the recovery rate insensitive to leach solution flow variations. The sulfuric acid leaching of the Wyoming ore sample seemed to indicate this behavior. Although the recovery data indicate equilibrium, the identity of the mechanism or reaction at equilibrium is not understood. The redox potentials observed were in the range of 170 to 200 mV during the water flow test period (increasing from the inlet to the outlet), and indicated a breakthrough similar to the pH breakthrough. The redox potential decreased from +440 mV at the inlet to +385 mV at the outlet, indicating the reducing capacity of the ore.

A rapid decrease in the permeability of all three sections of the ore packing upon sulfuric acid (pH = 1.0) introduction was also observed with the Texas ore sample. However, unlike in the case of the Wyoming ore sample, the middle and the end section permeabilities did not recover after the initial decrease. The permeability and flow variations observed during this test are shown in Figure 12. Because of the poor permeability of the packing only two packing volumes (or approximately 6.5 void volumes) of solution were passed through the Texas ore packing during this test. An overall recovery of 75.9% based on the solution uranium value (and 87.2% based on the residue value) was observed. However, the contribution to the overall recovery (solution value based) during the water flow test period was approximately 47.3%. The uranium recovery and the sulfate ion concentrations observed in the product solution for the Texas ore sample are illustrated in Figure 13. The slow response of the uranium in the product solution to acid introduction was attributed to uranium reprecipitation within the packing. The reprecipitation occurred in the neutralization reaction zone between the acid and the ore. This conclusion was based on the solution uranium concentration profiles obtained along the length of the packing. The pH profiles indicated a slow neutralization rate of the acid by the ore (with a reaction zone length of 50.5 in. at a solution flow rate of 1 to 2 ml/min). The delay in sulfate breakthrough could probably be attributed to the long retention time of solution in ore, leading to total relaxation of gypsum supersaturation within the column. This would explain the observed behavior of the middle and the end section permeabilities not recovering to their original values.

Permeability Change Mechanism

Based on the observations with the three leach solutions and the two ore samples, the following mechanisms are proposed for the permeability variations observed:

Ammonium Bicarbonate/Sodium Carbonate-Peroxide Leach System

The rapid decay of the permeability of the packings during leaching by ammonium bicarbonate or sodium carbonate solutions with peroxide as oxidant may be caused by (a) calcium carbonate precipitation within the packing, (b) clay reaction to the chemical reagent, and (c) gas blockage of the packing caused by the rapid decomposition of the peroxide upon contact with the reduced ore.

The principal source of calcium ion for precipitation is the montmorillonite clay fraction of the ore. It can be estimated that if all the calcium present in the clay be precipitated, it amounts to only 0.5 percent increase in fines fraction of the packing and could be significant to the permeability change. The two mechanism by which clay fraction affects the permeability are [1]: (a) structural expansion of clay particles in which water adsorbs in the interlayer spacing between the silicate units. Structural expansion consumes pore space in the packing to decrease the permeability; (b) deflocculation and dispersion of individual clay particles followed by migration downstream until they lodge at the pore inlets to plug the smaller pores and constrict the larger pores. The deflocculation is caused by the dissociation of the surface cations (ammonium and sodium) in dilute or low ionic strength solutions leading to increased electrostatic repulsion forces between the particles.

The contribution to the reduction in permeability by structural expansion is considered small since Na^+ or NH_4^+ as exchangeable cation is generally associated with a single layer of water and calcium ion is generally associated with two molecular layers of water [2]. It is reasonable to assume that the clay is predominantly in calcium form prior to chemical leaching. Therefore, sodium or ammonium ions should not cause any additional structural expansion.

Although it is difficult to estimate the effect of deflocculation and migration of clay particles, it is considered a plausible mechanism that could lead to the types of permeability changes observed.

Oxygen blockage of the packing is considered a likely mechanism causing the permeability decrease observed upon chemical introduction. The decomposition of peroxide is autocatalytic and once initiated can proceed rapidly to completion. The decomposition is initiated by the presence of reduced ore. The oxygen released nucleates on the solid particles within the void space. The net effect of this nucleation is to (a) reduce the area available for flow, (b) increase tortuosity of the packing, and (c) increase the contribution of gas-liquid surface tension forces. A simple model based on the first order effect that the area available for the flow is inversely proportional to the volume of gas entrained was applied to the leaching of the Wyoming ore sample using ammonium bicarbonate-peroxide solution. The estimated permeability based on this simple model was in good agreement with the observed value.

Sulfuric Acid Leaching

Permeability changes during acid leaching of ore samples could be caused by (a) increase in the fraction of fines, changes in the shape of the particles and in the viscosity of the solution caused by the dissolution of ore, (b) precipitation of heavy metals in the leading edge of the neutralization reaction zone, (c) precipitation of gypsum, and (d) carbon dioxide blockage caused by the action of acid on the calcium carbonate. Among these, only the precipitation of gypsum and carbon dioxide blockage of the packing were considered to be significant.

The post precipitation of gypsum is slow. This is considered a viable mechanism for the gradual drop in permeability observed during acid leach. The sensitivity of a given packing to any of the mechanism will be a function of the particle size distribution, shape factor of the particles and the packing mean pore diameter. The precipitation of gypsum downstream could aid in the consolidation of the packing, leading to a more rigid packing to pressure drop variations. Furthermore, precipitation of gypsum could reduce the permeability by the gradual decrease in the bed voidage of the ore. This may explain the lack of recovery of the middle and end section permeabilities during the acid leaching of the Texas ore sample.

The physical resemblance and the rapidity of the change in permeability of the front section observed to that during bicarbonate-peroxide leaching, indicates that carbon-dioxide may be the dominant cause of the permeability drop observed. In the case of the Wyoming ore, the carbonate content of the front section (0.84 wt% CO_2) alone is equivalent to 25.29 liters of carbon-dioxide at a mean pressure of 28.85 psig (pressure corresponding to the mean pressure of the front section). This is approximately eight times the void volume in this section. Calculations similar to those carried out in the case bicarbonate-peroxide leach solutions, would indicate definite rapid permeability change of the front section of the packing from the carbon dioxide evolution.

In summary, the most likely mechanisms causing the rapid change in the permeability of the front section are (a) the deflocculation and migration of clay particles and oxygen blockage in the case of bicarbonate or carbonate solutions with peroxide as oxidant, and (b) carbon dioxide blockage in the case of acid leach solutions. The gradual decrease in the permeability, in the case of bicarbonate-carbonate solutions may be caused by (a) clay swelling from hydration of the clay and (b) calcite precipitation. The gradual decrease in the permeability in the case of acid leach may be caused by (a) change in packing and solution characteristics due to acid dissolution of ore and (b) gypsum post precipitation.

Discussion of the Simulation Test Findings

Because of the capability to obtain representative ore packings consistently and the capability to produce the physical phenomena of in-situ leaching, the simulation test is a superior laboratory method to the existing methods of studying in-situ leaching. The phenomena such as reprecipitation of uranium at the leading edge of a neutralization (pH) front, uranium exhaustion front movement rate, etc., are nearly impossible to detect in short packings or undisturbed core tests. In addition, the simulation test could be of assistance in qualitatively estimating the impact of various mechanisms that cause permeability changes.

Even though the tests exposed the complex nature of in-situ leaching, effects unique to the low pressure operation, in the tests performed, might have masked the true nature of the permeability changes occurring in the ore packing. The manifestation of gas blockage of an ore body could not occur when the absolute pressure at any point along the flow line is less than the partial pressure of the gas corresponding to its solution concentration at that point. Therefore, to prevent any gas liberation in the ore body, the only sufficient condition is that the packing be operated at a minimum pressure greater than the partial pressure of the gas corresponding to its solution concentration at all points in the packing.

The simulation tests exposed the importance of correct size distribution in achieving successful testing. Texas ore with finer particles and near bi-modal distribution represented the undesirable end of the spectrum. The sensitivity of a packing to plugging by precipitation clay swelling or gas blockage is a strong function of the mean pore diameter, particle shape factor and surface tension characteristics. Selection of the correct size reduction unit operation may be critical to this effect.

An area of concern was the potential for the oxidation of uranium by the dissolved oxygen content of the water during the water flow test period. Calculations indicated that in 125 hours at 10 ml/min flow of water enough oxygen at 8 ppm level is available to oxidize 8.925 g of uranium. This is approximately 40 percent of the total uranium in a simulation test with the Texas ore. This, in addition to 20 percent preoxidation during ore processing, allows 60 percent extractions if sufficient complexing agent is made available.

The high recoveries obtained with the Wyoming ore sample (86.7 percent with NH_4HCO_3-H_2O_2 leach and approximately 89.2 percent with acid) may be indicative of high accessibility of the uranium to leach solution. Since the effort is directed at simulation of in-situ leaching, it is essential to identify the differences between uranium occurrence in the formation underground and in the processed ore. This is considered important to understand the kinetics of leach phenomena. For instance, leaching of very accessible uranium in the ore may be controlled by dissolution rate whereas a very inaccessible uranium in the ore may be controlled by the oxidation rate. The effect of dissolved oxygen (8 ppm) in the solution in the two cases would be vastly different. The accessible uranium will be readily preoxidized, thereby distorting the leach responses observed.

The Phase IV of this continuing development program has been addressed to obtain quantitative answers to some of these questions.

Conclusions

A method to simulate in-situ leaching reproducibility in the laboratory was developed. Although the conclusions summarized are specific to the particular ore samples used in the tests and the low pressure nature of the system, the qualitative and relative conclusions do provide generalities of interest in understanding in-situ leaching.

In the tests carried out using representative samples of ores from Texas and Wyoming, the sulfuric acid solution gave higher solution uranium concentrations, and shorter rise and fall times for the uranium solubilization than the ammonium bicarbonate-peroxide leach solution. Furthermore, only 75-85% of the uranium in the ores was extractable by either leach solution. The remaining uranium is believed to be inaccessible to the leach solution. During acid leaching of the ore samples, precipitation of the extracted uranium does occur at the high pH edge of the neutralization zone. The precipitated uranium is redissolved at the low pH edge of the reaction zone. The leaching of uranium by basic ($pH \geq 7$) leach solutions was leaching rate (kinetic) constrained in the 12 ft. long ore packing length and solution retention times in the range of 12-32 hours. The leaching by acid leach solution under similar conditions was equilibrium constrained.

Sulfuric acid and ammonium bicarbonate leach solutions gave higher overall uranium recovery rates and ore packing permeabilities than the sodium carbonate solution. Comparable permeabilities were obtained using the acid and the ammonium bicarbonate solutions. Deflocculation and migration of montmorillonite clay and oxygen blockage of the packing were considered the significant mechanisms leading to rapid decay in the packing permeability with alkaline leach solutions. With acid leach solution, the carbon-dioxide blockage was considered the significant mechanism. The gas blockage of the packing may be unique to the low pressure nature of the tests and may not occur if the tests are carried out at near formation pressures.

Acknowledgment

The author acknowledges the many significant suggestions and the cooperation given to him by D. C. Seidel of U.S. Department of Interior, Bureau of Mines, Salt Lake City, Utah, and the cooperation in the administration of the program by John Arnold of U.S. Bureau of Mines, Denver, Colorado.

References

1. Reed, M. G., "Water and Steam Injectivity Retention with Calcium Chloride Pretreatments," API Prod. Div. Pacific Coast Dist. Spring Meeting, May 14-16, 1968, Preprint No. 801-441.

2. Grim, R. E., "Clay Mineralogy," McGraw-Hill Book Co., New York, NY, 1968, pp. 77-92.

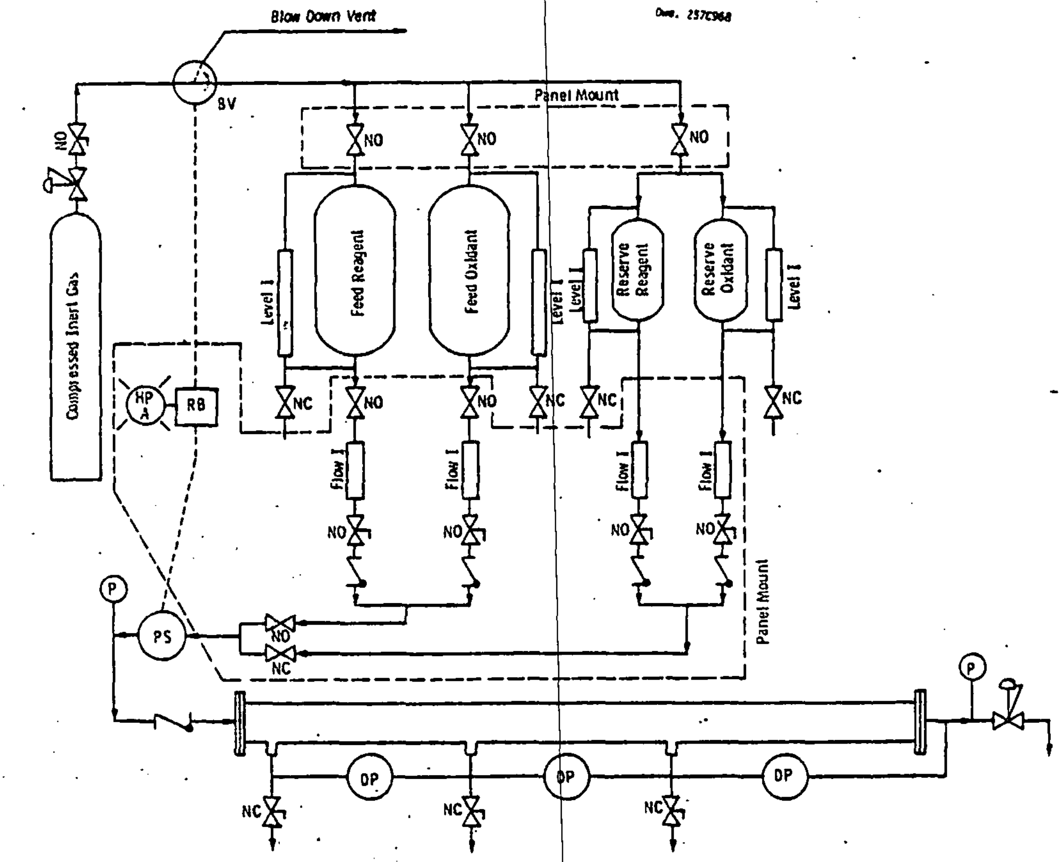


Fig. 1 — Schematic diagram of insitu leaching system

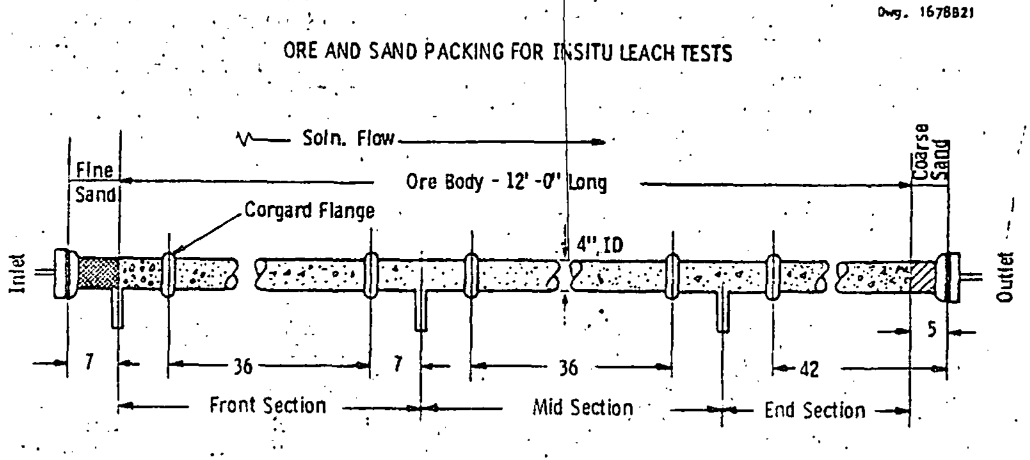


Fig. 2 — Lengths of cylinder pieces and ore and sand packing for insitu leach tests

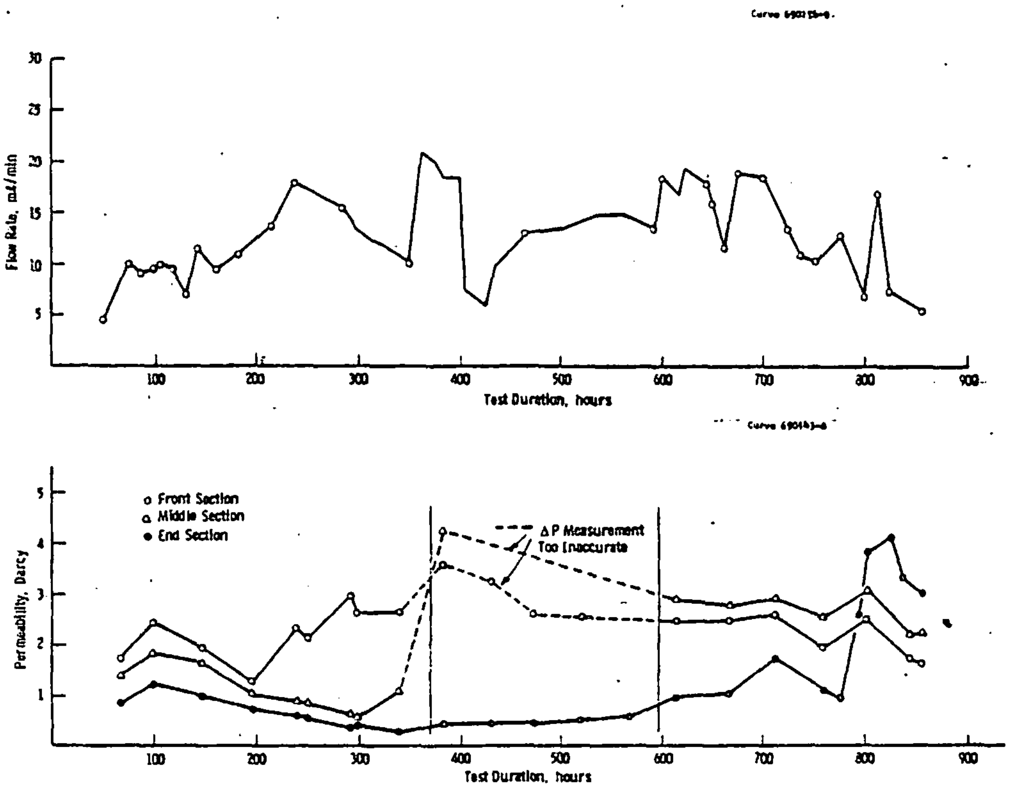


Fig. 3 – Permeability and solution flow variations with time during the Leach test with Wyoming ore using deionized water

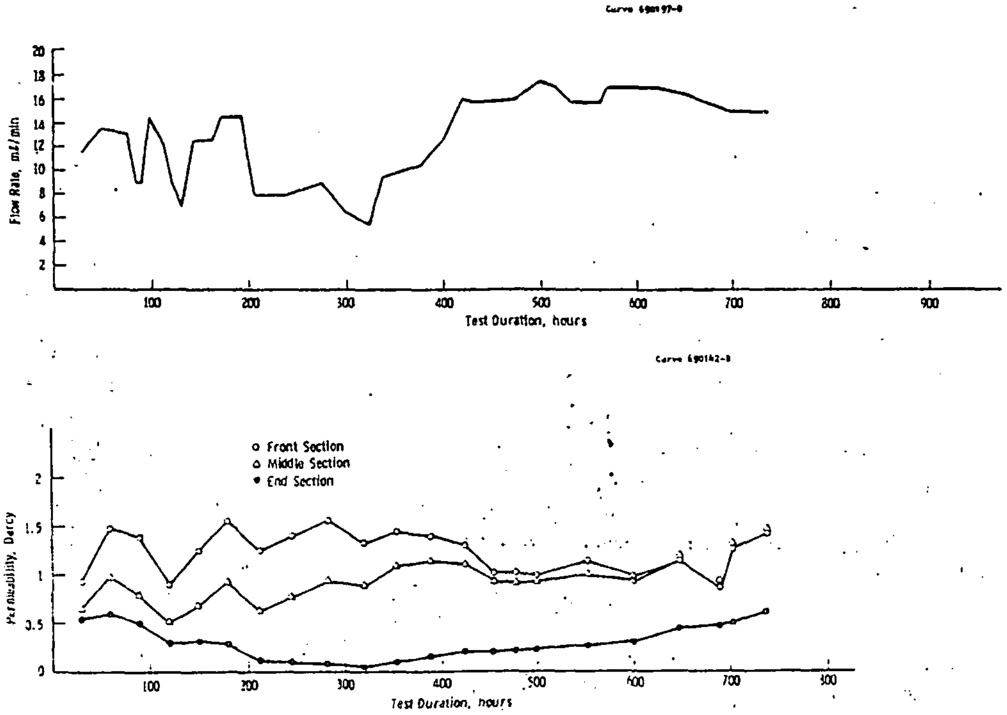


Fig. 4 – Ore packing permeability and solution flow variations observed during leaching of the Texas ore sample using deionized water

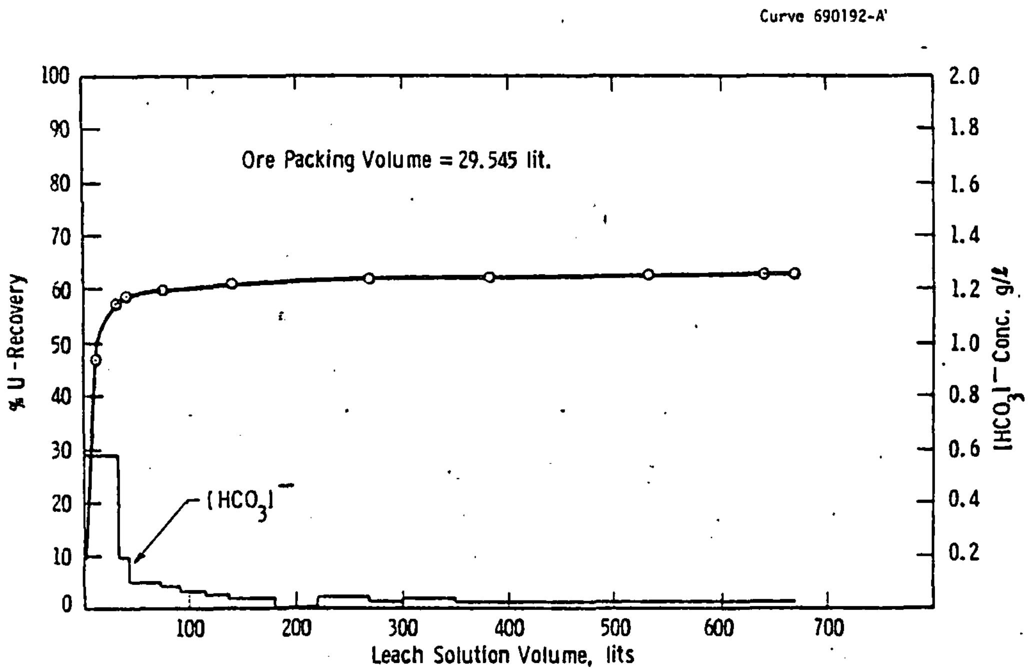


Fig. 5 – Uranium recovery rate observed during leaching of the Wyoming ore sample using de-ionized water

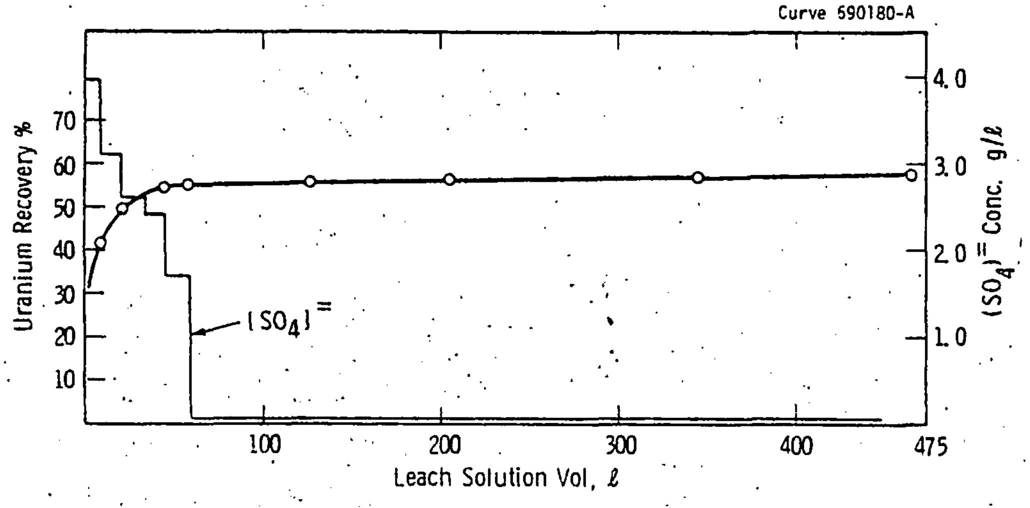


Fig. 6 – Uranium recovery rate observed during leaching of the Texas Ore using water. Ore packing volume = 29.545 ℓit

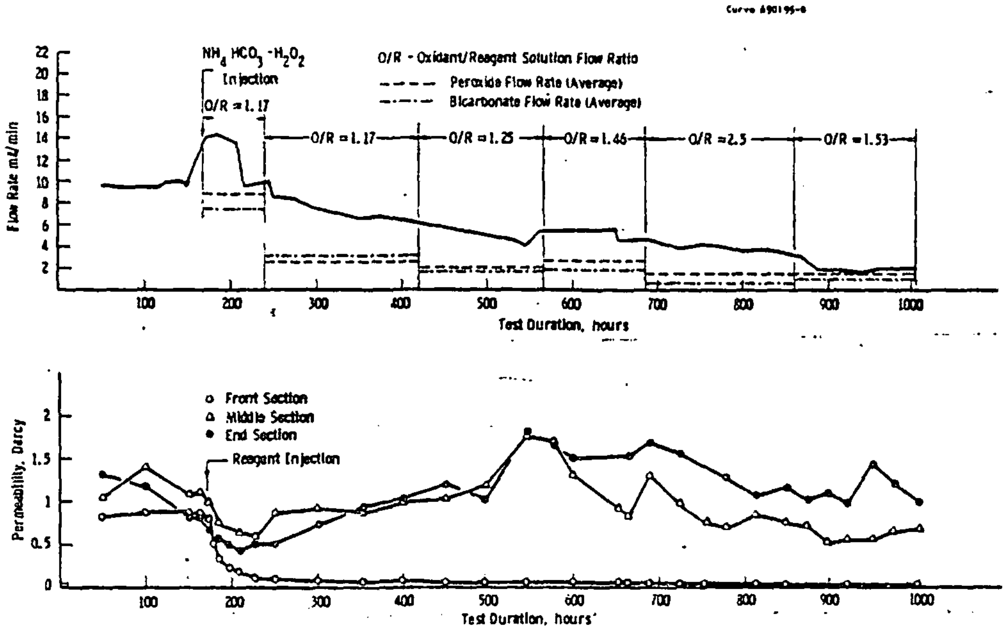


Fig. 7 — Ore packing permeability and solution flow variations observed during leaching of the Wyoming ore sample using 1.0 g/ℓ NH_4 HCO_3 - 0.5 g/ℓ H_2O_2

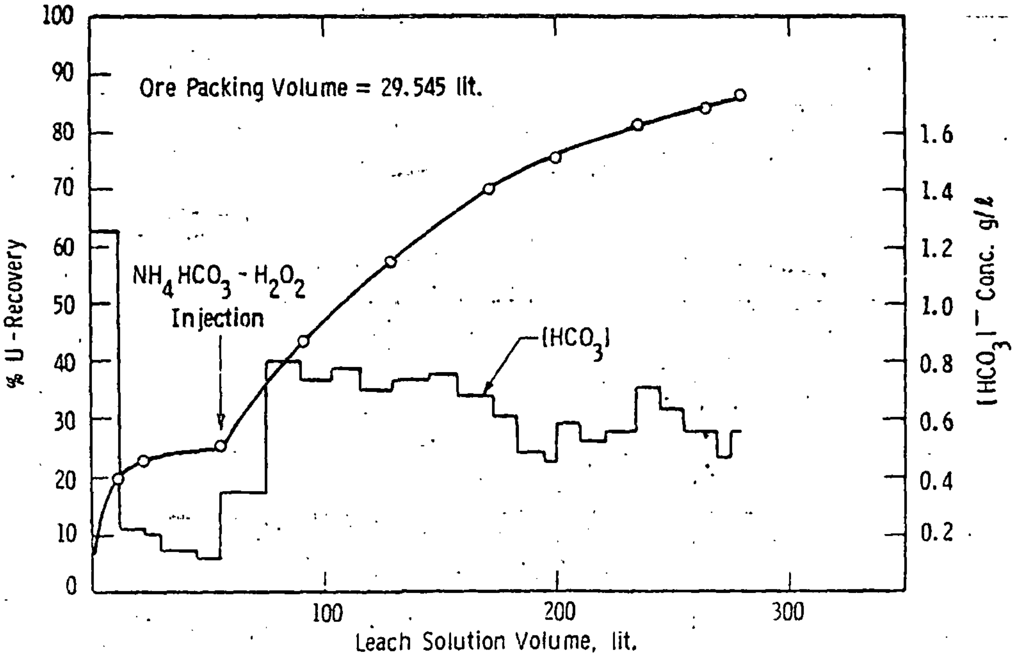


Fig. 8 — Uranium recovery rate observed during leaching of the Wyoming ore using 1.0 g/ℓ NH_4HCO_3 - 0.5 g/ℓ H_2O_2

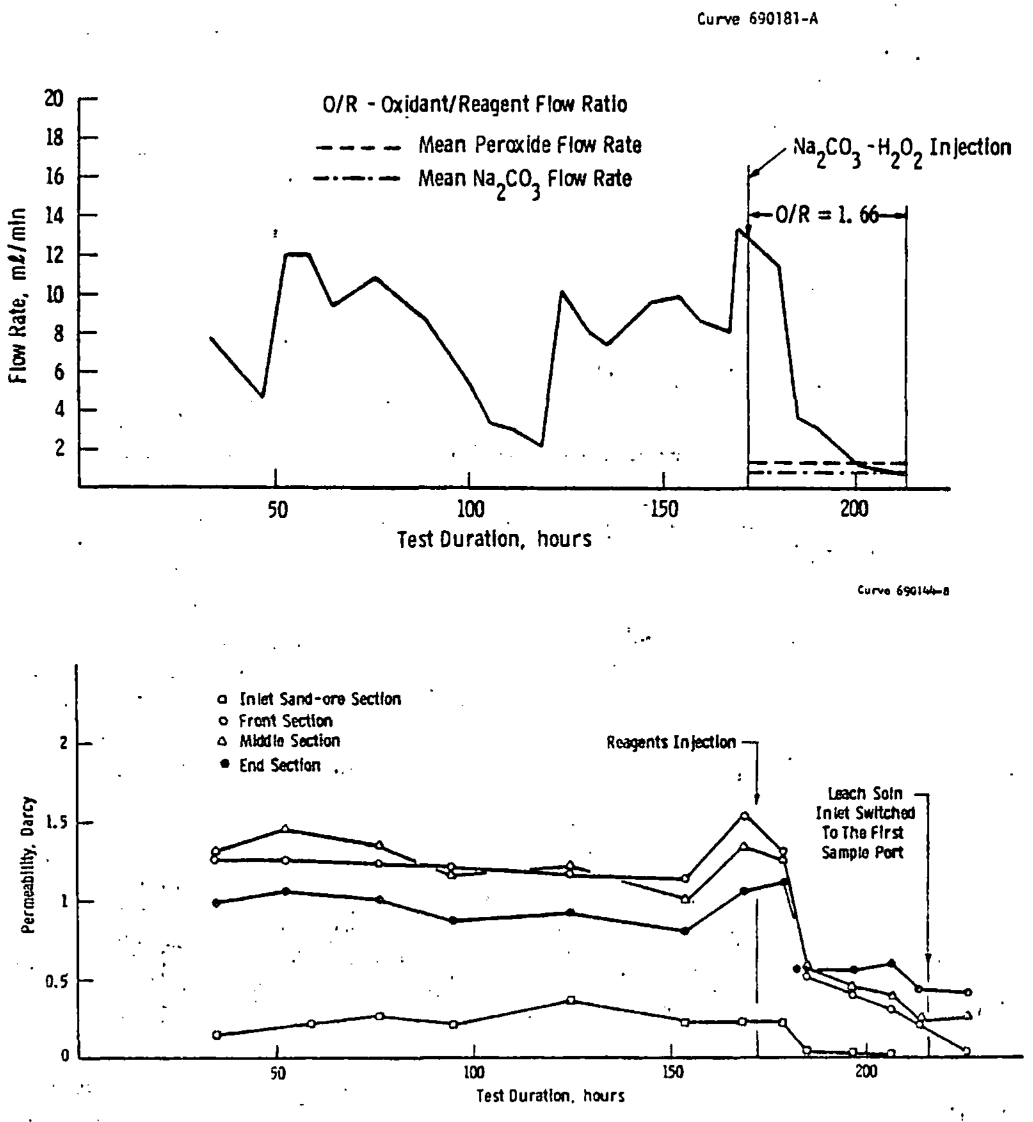


Fig. 9 — Permeability and flow variations observed during the Leach test with Wyoming ore using 1.0 g/ℓ Na_2CO_3 - 0.5 g/ℓ H_2O_2 Leach solution

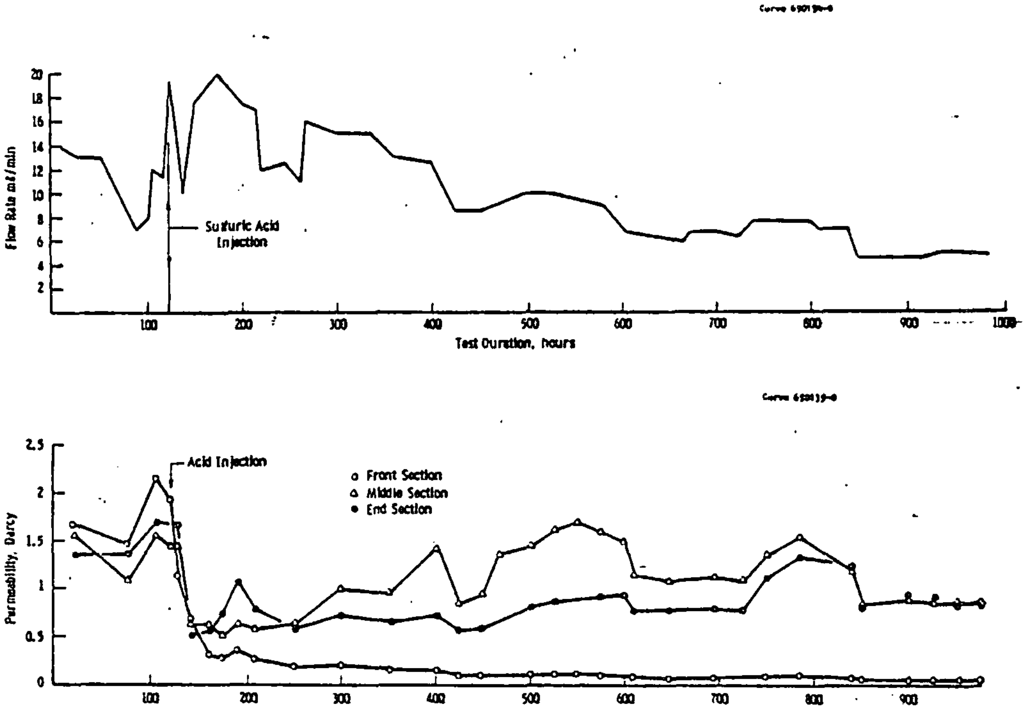


Fig. 10 – Packing permeability and solution flow variations observed during leaching of the Wyoming ore sample using sulfuric acid of pH = 1.0

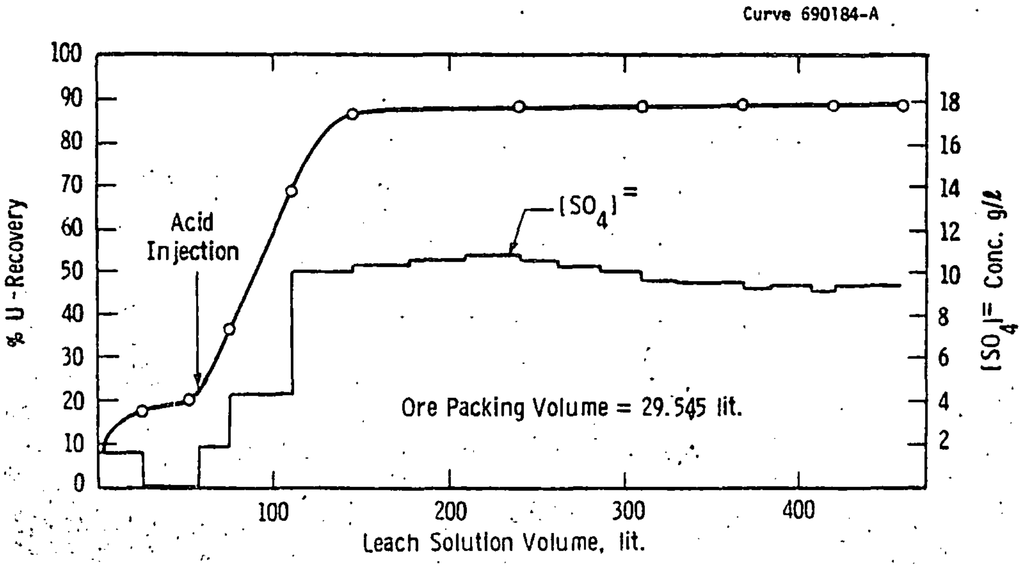


Fig. 11 – Uranium recovery rate observed during leaching of the Wyoming ore using sulfuric acid

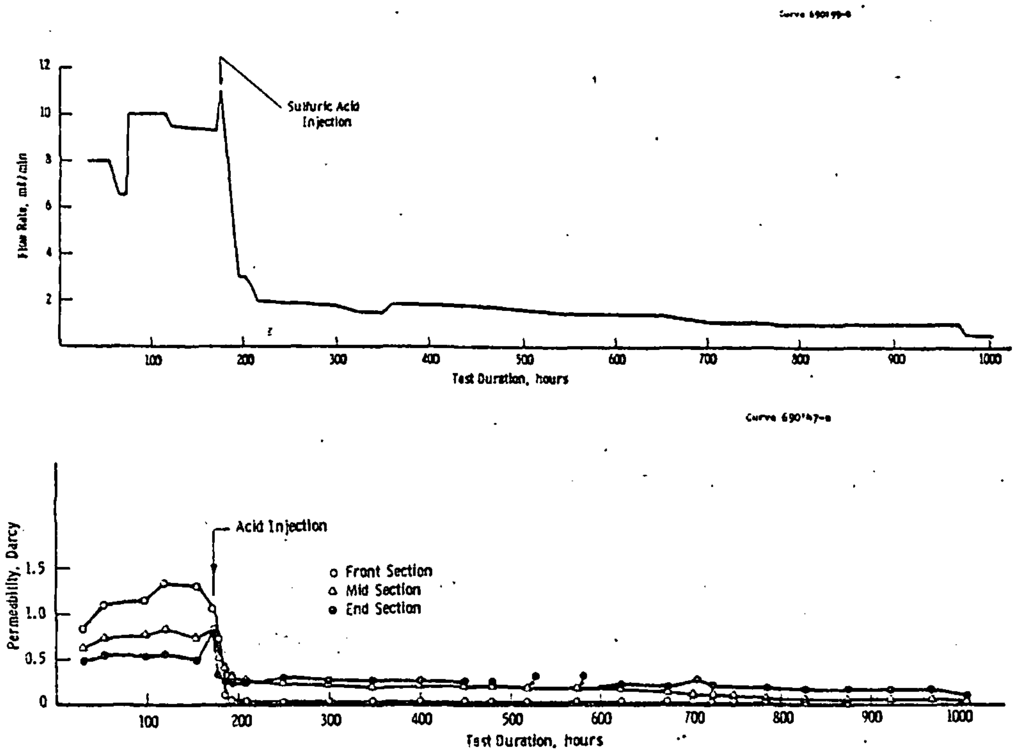


Fig. 12 — Ore packing permeability and solution flow variations observed during Leach test with Texas ore sample using sulfuric acid

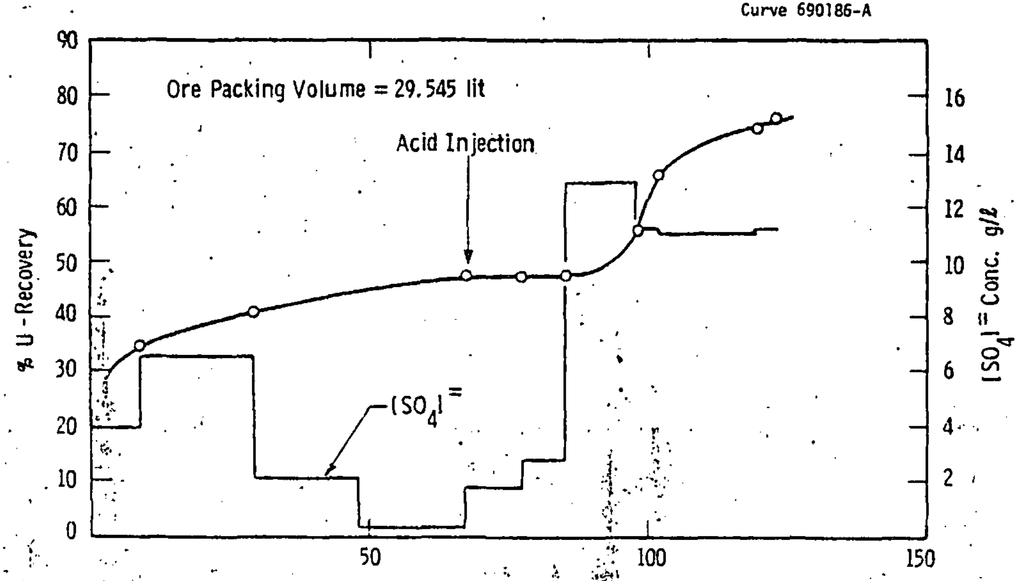


Fig. 13 — Uranium recovery rate observed during leaching of the Texas ore using sulfuric acid

United States Patent [19] [11] **4,045,084**
Hsueh et al. [45] **Aug. 30, 1977**

[54] IN-SITU MINING OF COPPER AND NICKEL

[75] Inventors: **Limin Hsueh**, Bedford; **Robert A. Hard**, Still River; **Donald H. Davidson**, Bedford; **Ray V. Huff**, Acton, all of Mass.

[73] Assignee: **Kennecott Copper Corporation**, New York, N.Y.

[21] Appl. No.: **724,547**

[22] Filed: **Sept. 20, 1976**

[51] **Int. Cl.**^{2} **E21B 43/28**
[52] **U.S. Cl.** **299/4**; 175/69
[58] **Field of Search** .... 299/4, 5; 175/69; 75/101 R

[56] **References Cited**

U.S. PATENT DOCUMENTS

3,498,674	3/1970	Matthews	299/5 X
3,708,206	1/1973	Hard et al.	299/5
3,937,520	2/1976	Sievert	299/4
3,999,803	12/1976	Coursen	299/4

Primary Examiner—Ernest R. Purser
Attorney, Agent, or Firm—Anthony M. Lorusso; John L. Sniado

[57] ABSTRACT

A process is disclosed for in-situ mining of copper from a subterranean ore body characterized, at least in part, by the presence of a sulfidic ore and by natural, microscopic fracture openings. The process comprises forcing a stable, two-phase lixiviant comprising an aqueous phase, a multiplicity of gaseous, oxygen-containing bubbles having a size sufficient to pass through the natural fracture openings in the ore body, and a surfactant for enhancing the formation of the bubbles and for minimizing bubble coalescence through the ore body to leach copper.

The aqueous and gaseous phases of the lixiviant are mixed at the surface and injected into the leaching interval through an injection hole, or preferably, are mixed in a subterranean sparger within the hole above the leaching interval. The pregnant liquor is recovered through one or more production holes and, after the copper is recovered, the lixiviant may be reconstituted and recirculated.

11 Claims, 7 Drawing Figures

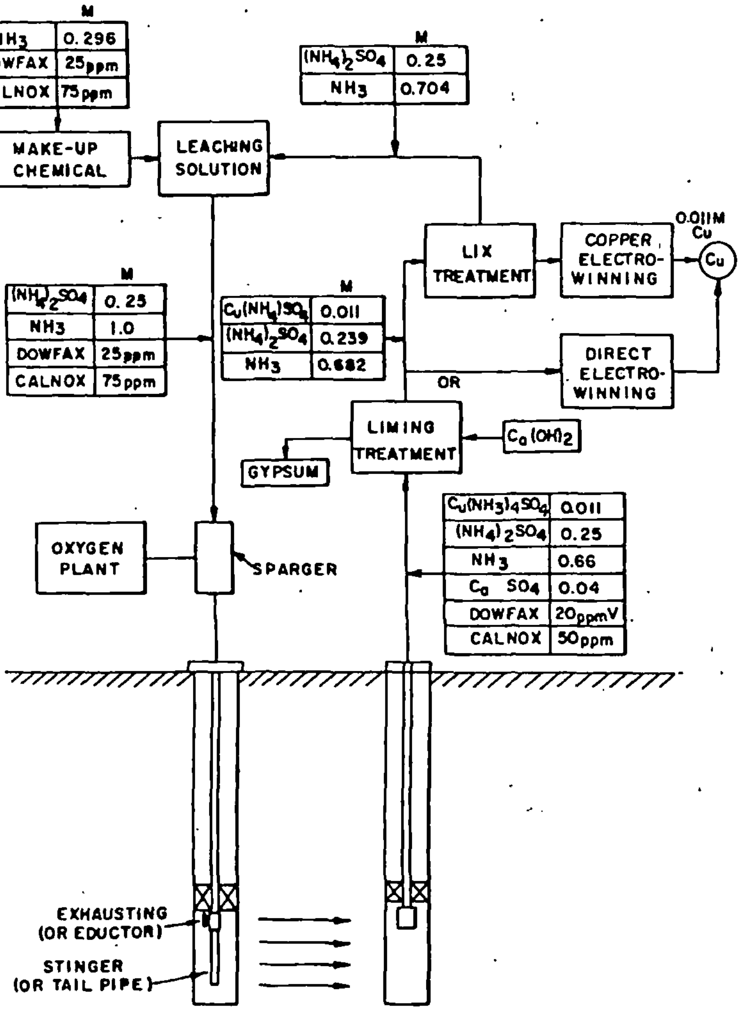


FIG. 1.

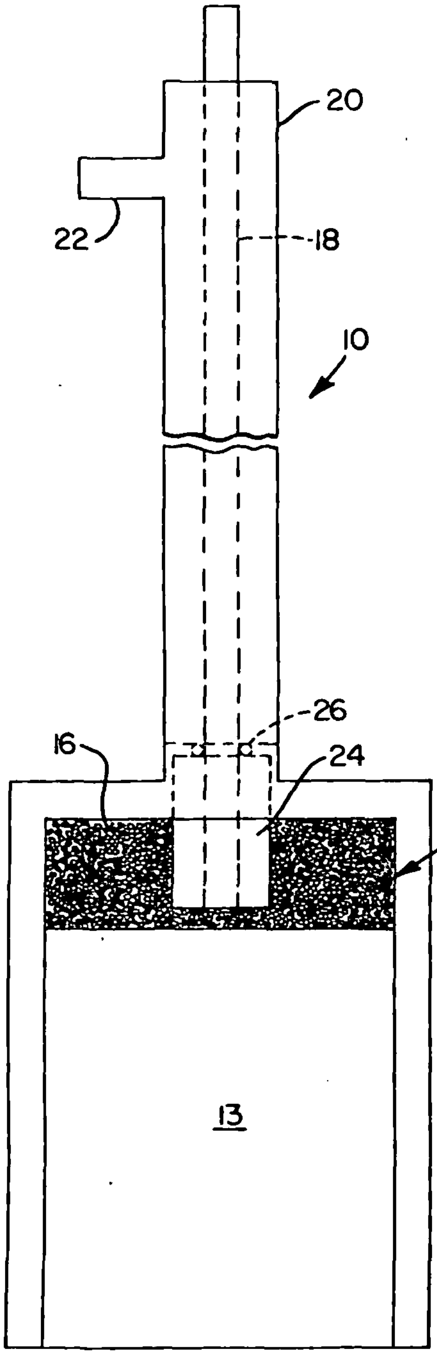


FIG. 2.

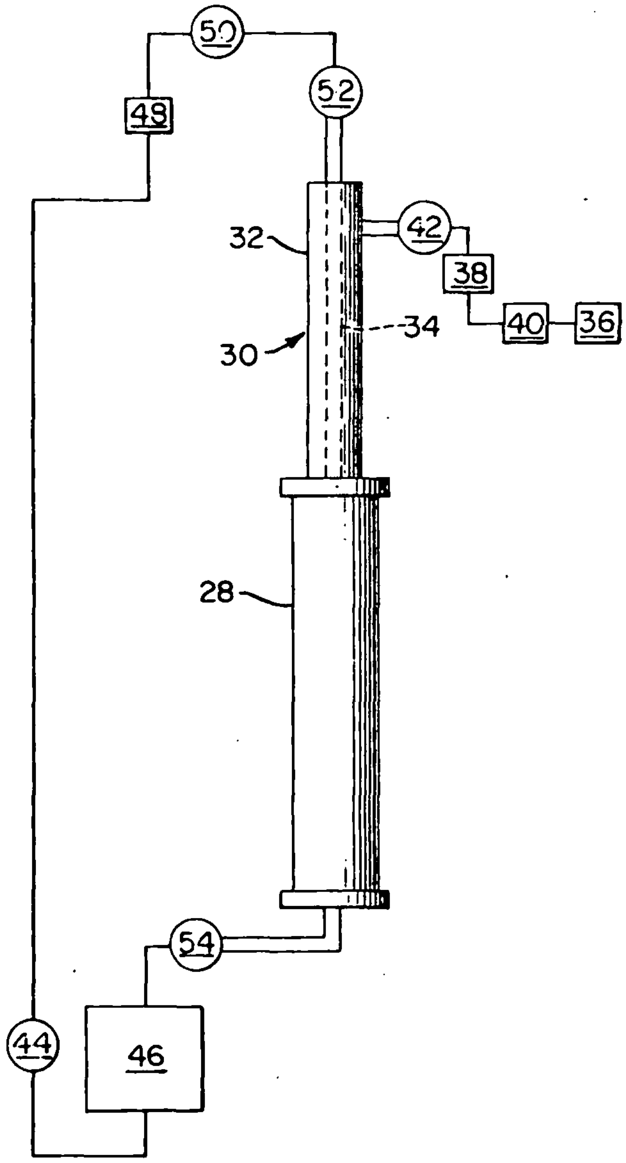


FIG. 3.

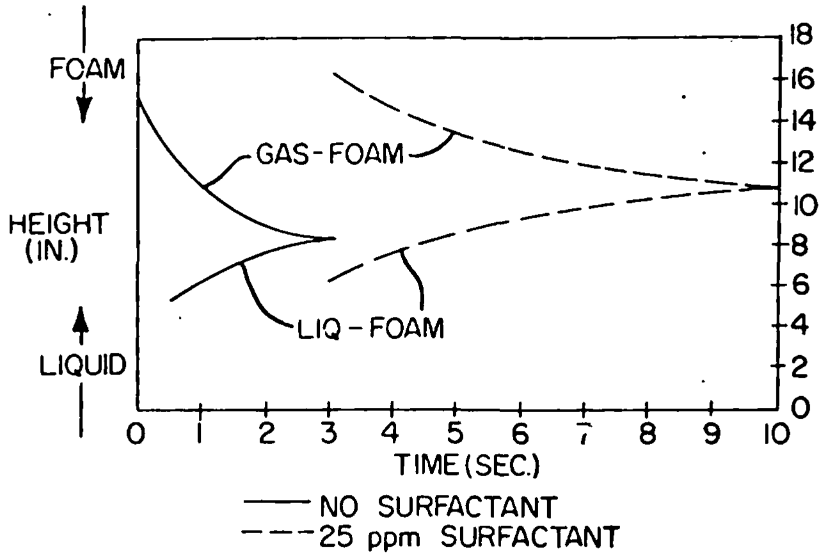

FOAM
HEIGHT (IN.)
LIQUID
GAS-FOAM
LIQ-FOAM
18
16
14
12
10
8
6
4
2
0
0
1
2
3
4
5
6
7
8
9
10
TIME (SEC.)
NO SURFACTANT
25 ppm SURFACTANT

FIG. 4.

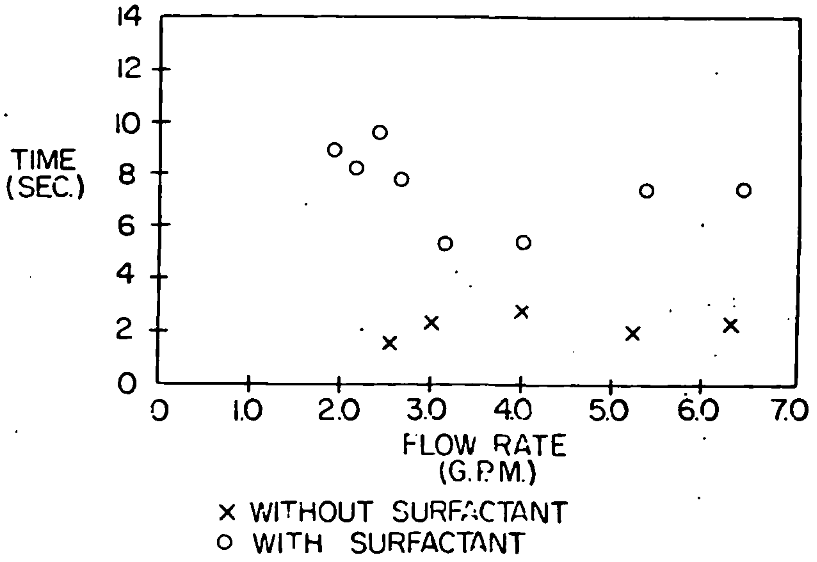
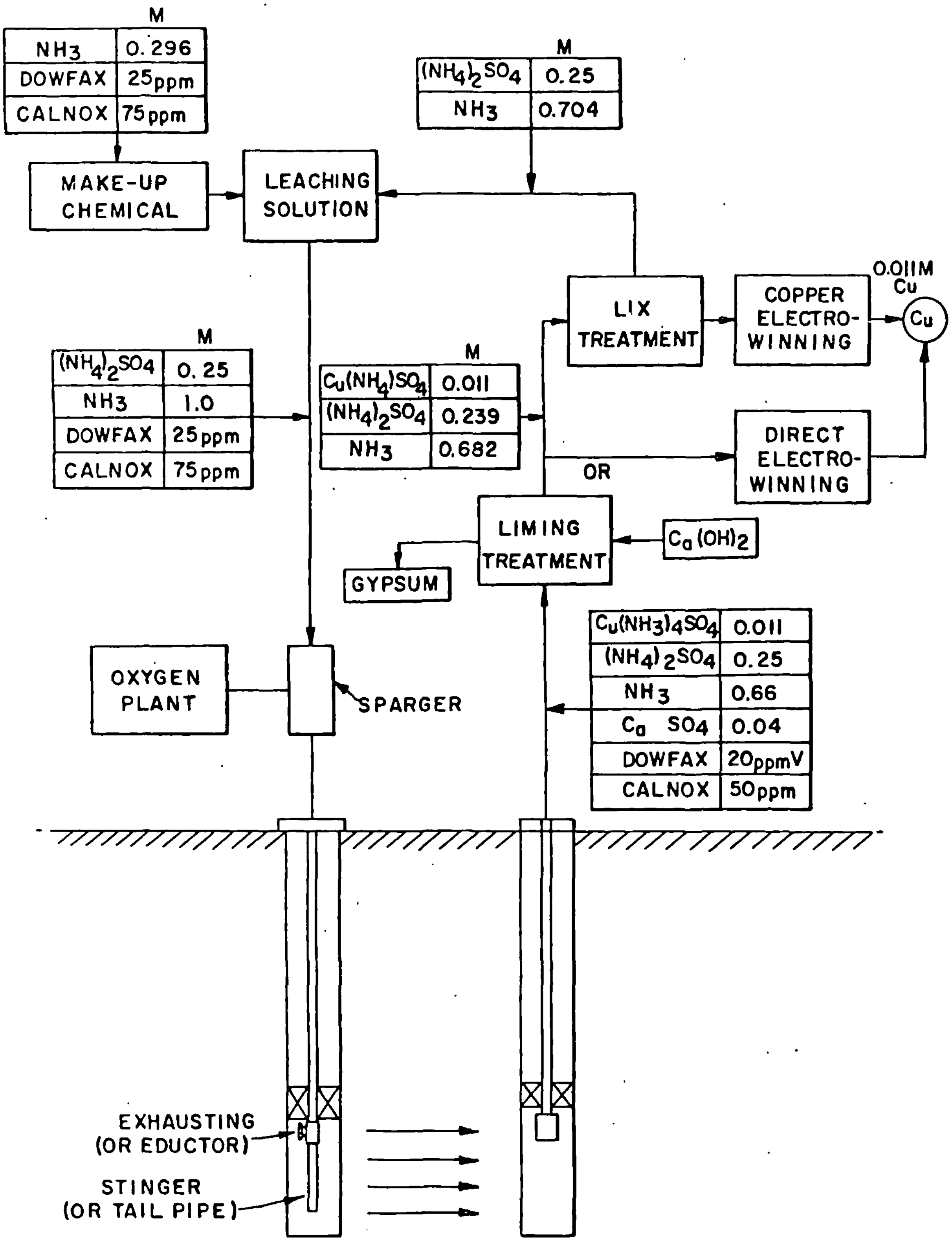

14
12
10
8
6
4
2
0
TIME (SEC.)
0
1.0
2.0
3.0
4.0
5.0
6.0
7.0
FLOW RATE (G.P.M.)
× WITHOUT SURFACTANT
○ WITH SURFACTANT

FIG. 7


M
NH_3 0.296
DOWFAX 25ppm
CALNOX 75ppm
MAKE-UP CHEMICAL
LEACHING SOLUTION
M
$(NH_4)_2SO_4$ 0.25
NH_3 0.704
LIX TREATMENT
COPPER ELECTRO-WINNING
0.011M Cu
Cu
M
$(NH_4)_2SO_4$ 0.25
NH_3 1.0
DOWFAX 25ppm
CALNOX 75ppm
M
$Cu(NH_4)SO_4$ 0.011
$(NH_4)_2SO_4$ 0.239
NH_3 0.682
DIRECT ELECTRO-WINNING
OR
LIMING TREATMENT
$Ca(OH)_2$
GYPSUM
$Cu(NH_3)_4SO_4$ 0.011
$(NH_4)_2SO_4$ 0.25
NH_3 0.66
Ca SO_4 0.04
DOWFAX 20ppmV
CALNOX 50ppm
OXYGEN PLANT
SPARGER
EXHAUSTING (OR EDUCTOR)
STINGER (OR TAIL PIPE)

IN-SITU MINING OF COPPER AND NICKEL

BACKGROUND OF THE INVENTION

This invention relates to a process for in-situ mining of copper and nickel from a deep subterranean ore body using a stable, two phase lixiviant. More specificially, it relates to a modification of the process set forth in copending patent application Ser. No. 724,548 entitled *In-Situ Mining Method and Apparatus* filed on even date herewith, the teachings of which are incorporated herein by reference.

There is known to be located through out various regions of the globe, large, deep lying deposits of copper in the form of low grade porphyry ores. A porphyry deposit is one in which the copper bearing minerals occur in disseminated grain and/or in veinlets through a large volume of rock such as schist, silicated limestone, and volcanic rock. Acid igenous intrusive rocks are usually in close association. The deposits are typically large tonnage but low grade, and have an average copper concentration of less than about 1 percent. Copper minerals found in these deposits are usually sulfides and most commonly are chalcopyrite. There are also massive sulfide deposits treatable by the present invention which are deep seated and contain discrete blebs of nickel sulfide, copper sulfide, or copper-nickel sulfide in association with iron sulfide. A representative list of minerals which can be treated to recover copper using the process of this invention includes chalcocite, digenite, covellite, pentlandite, heazlewoodite, vaesite, and violarite.

When a deposit of the type described is of sufficiently high grade and outcrops on the surface or is sufficiently close to the surface, the ore may be mined by open pit methods, and the metal values separated from the gangue constituents by techniques such as floatation.

Deeply buried or very low grade copper deposits cannot be easily exploited. Conventional open pit mining is not available because the costs involved are prohibitive and because landscape destructive open pit mining techniques have been restricted in many areas.

It has been proposed to extract the copper from the deeply buried porphyry deposits by in-situ leaching techniques. With in-situ mining, a hole is drilled and a leach liquor is pumped down the hole into the ore containing the metal to be recovered. After the liquor has leached the metal values, it is brought back to the surface and the values are recovered.

There are many prior art procedures for in-situ mining. Most of these procedures, however, involve rubblizing the ore which is to be leached by explosive methods. In contrast, the process and apparatus disclosed in the aforementioned copending Ser. No. 724,548 and the instant invention involve leaching the copper in-situ, without rubblizing the ore by employing a two-phase lixiviant comprising very small oxygen containing gas bubbles admixed with leach liquor. For this method to be successful, the oxygen containing bubbles must be small enough to penetrate the natural fracture openings within the rock so that the sulfidic minerals may be oxidized. Thereafter, the leach liquor solubilizes the metal values.

Prior to the present invention and prior to the process disclosed in the aforementioned copending U.S. Ser. No. 724,548 two-phase lixiviants useful for such purposes, although theoretically desirable, were thought to be unattractive for a number of reasons. The two primary problems were the size of the bubbles and the difficulties stemming from phase separation, i.e., formation of gas pockets. So severe were the problems associated with such two-phase in-situ mining procedures, that research in this area has been discouraged.

The types of copper and nickel bearing ores with which the present invention is concerned generally have a porosity of about 3% and are found on the order of 2,000 feet below the surface. The cracks, pores, and other fracture openings in these rocks usually have dimensions on the order of 10 to 300 microns. Since these openings contain the metal values of interest, and since the sulfidic minerals in which the metal values are contained must be oxidized before the leach liquor can take effect, it is necessary that the lixiviant contain a high concentration of oxygen containing bubbles small enough to move freely through the openings. The ideal lixiviant would comprise a leach liquor containing a high concentration of oxygen in the form of stable bubbles having a diameter less than about 10 microns. It is also desirable that the lixiviant have a viscosity at the temperature of use cf close to 1.0 centipose.

In practice, as mentioned above, such a two-phase lixiviant has been difficult to produce and even more difficult to maintain. Methods are known for forming small bubbles in a liquid, but in the known two-phase systems, the bubbles tend to coalesce, form larger bubbles, and ultimately form large pockets of gas. The higher the viscosity of the liquid phase, the easier it is to form bubbles of the size described and to maintain them. In systems using low viscosity compounds with the high volume fraction of gas, i.e. higher than 15-30%, production and maintenance of small gas bubbles becomes very difficult. Another variable which affects the stability of gas bubbles in a two-phase system is the flow rate of the lixiviant. Generally, it has been observed that the higher the flow rate, the easier it is to maintain the gas-liquid dispersion. However, in use, there is an upper limit in the flow rate which limit makes unavailable any advantage which might theoretically be gained by employing a high flow rate. When deterioration of the two-phase system occurs in the in-situ mining procedure such as that disclosed in the aforementioned copending application, a two-phase lixiviant is rendered inoperative or unacceptably inefficient.

SUMMARY OF THE INVENTION

The present invention provides an in-situ mining process which utilizes a stable, two-phase lixiviant suitable for mining copper and nickel contained in sulfidic ore deposits characterized by natural microscopic fracture openings. The lixiviant comprises an aqueous leach liquor phase, a multiplicity of oxygen containing bubbles having a size, when subjected to the pressure employed during the in-situ mining, sufficient to pass through the fracture openings in the ore formation and a surfactant for maintaining the size and individuality of the bubbles and for enhancing the mass transfer rate of oxygen to the metal values of interest. In preferred embodiments of the process of the invention, the bubbles in the lixiviant are less than 10 microns in diameter.

Surprisingly, it has been discovered that many surfactants, when added to the liquor phase prior to introduction of the bubbles, greatly increases the stability of bubbles formed, enables smaller bubbles to be formed, and tends to maintain the size of the smaller bubbles for a substantial time during the mining procedure. In addition to the surfactant, the lixiviant may include aluminum ions for further minimizing coalescence of the bubbles and an agent to increase solubilization of Ca++ ions which are often present in ore formations.

The term "surfactant" as used herein is an abbreviated term for "surface active agent." A surfactant is any compound that usually reduces surface tension when dissolved in water or water solutions, or which similarly affects interfacial tension between two liquids. Soap is such a material but the term is more frequently applied to organic derivatives such as sodium salts of high molecular weight alkyl sulfates, alkyl sulfonates and other organic derived surfactants as further described herein.

The addition of one or more surfactants to the lixiviant also imparts increased wetting ability of the mineral surfaces as well as reducing bubble size and increasing the intersurface area. Still further, the surfactant's known ability to reduce surface tension tends to enhance the mass transfer rate of oxygen with the ore of interest. The preferred aqueous phase comprises an ammoniacal leach liquor, although, as will be set forth below, other aqueous phases are contemplated for certain applications.

In accordance with the invention, an injection hole and at least one recovery hole are drilled to provide communication with the ore body. The stable, two phase lixiviant is then forced through the ore body under pressure, the oxygen bubbles passing through the fracture openings and oxidizing the sulfidic minerals. The copper and/or nickel ions are solubilized in the aqueous phase and recovered through the recovery hole. Copper and/or nickel metal may then be recovered from the pregnant liquor by conventional techniques. The two phases of the lixiviant may be put together either at the well head or, preferably, at a point well below ground level close to the leaching interval. In this latter regard, see copending U.S. application Ser. No. 724,549 entitled "In Situ Method and Apparatus for Sparging Gas Bubbles" filed on even date herewith, the disclosure of which is incorporated herein by reference.

In the process disclosed in U.S. application Ser. No. 724,548, apparatus is employed adjacent the bottom of the injection hole at the leaching interval to minimize separation of the phases of the two phase lixiviant. Briefly explained, the apparatus comprises an exhauster and tail pipe located below a cemented off portion of the injection pipe. Two phase lixiviant passing through the exhauster creates a reduced pressure zone in the exhauster by aspiration and then passes out into the injection interval through the tail pipe. Gas which separates from the lixiviant collects in the hole adjacent the exhauster and is drawn into the lixiviant stream. In the process of the present invention, the provision of the surfactant stabilized lixiviant substantially reduces separation of the phases and hence improves the efficiency of the leaching process.

Accordingly, it is an object of the invention to provide a process for in-situ mining of copper and nickel bearing ores of the type which have naturally occurring microscopic fractures therein and which contain sulfidic minerals which must be oxidized before being leached.

Another object of the invention is to provide a process utilizing a lixiviant having gaseous oxygen containing bubbles small enough to penetrate microscopie fractures found within ores of the type described.

Still another object of the invention is to provide a process utilizing a lixiviant containing oxygen bubbles less than about **10** microns in size.

Another object of the invention is to provide an in-situ mining procedure which omits the step of rubblizing the ore in the leaching interval.

Another object of the invention is to provide a process utilizing an ammoniated leach liquor containing small bubbles of oxygen containing gas which resist coalescence to an improved degree and which thereby minimizes the formation of gas pockets in the leaching zone or interval.

Another object of the invention is to provide an economical process for obtaining copper and/or nickel from deep, subterranean or submerged deposits.

Yet another object of the invention is to leach copper using a two phase lixiviant containing a surfactant which inhibits coalescence of the phases.

BRIEF DESCRIPTION OF THE DRAWING

FIG. **1** is a schematic representation of laboratory apparatus for measuring the bubble size of two phase lixiviant formulations;

FIG. **2** is a schematic diagram showing laboratory apparatus for testing the stability, under various conditions, of two phase lixiviants;

FIGS. **3–6** are graphs made from data derived from experiments conducted using the apparatus of FIG. **2**; and

FIG. **7** is a schematic diagram illustrating one embodiment of the process of the invention.

DESCRIPTION OF THE PREFERRED EMBODIMENT

At the outset, the invention is described in its broadest overall aspects with a more detailed description following. According to an important aspect of the invention, it has been found that in-situ mining of copper and nickel values is possible in deep ore formations of the type characterized by the presence of sulfidic mineral and by natural microscopic fracture openings, without rubblizing, if a stabilized, two-phase lixiviant comprising an aqueous ammoniacal leach liquor and a multiplicity of oxygen containing bubbles having a size sufficient to pass through the fracture openings in the formation is employed. According to another aspect of the invention, it has been discovered that a large number of surfactants which are available commercially, when added in a effective amount (usually about 5 to 200 parts per million) to the aqueous phase of the two phase lixiviant described above, enable the production of oxygen containing bubbles of the necessary size and greatly reduce the tendency of the two phases to separate into liquid and gas phases, i.e., greatly reduces the tendency of the bubbles to coalesce.

The improved stability of the lixiviant greatly enhances the distribution of oxygen within the leaching interval and ensures delivery of the gaseous phase to the ore even at relatively low lixiviant downward linear velocity in the pipe. Accordingly, the stabilized lixiviants make possible a relatively efficient and inexpensive in-situ mining process which heretofore was believed impossible to accomplish. Furthermore, the presence of a surfactant in the lixiviant has been observed to enhance the mass transfer rate of oxygen to such ores.

Referring to FIG. **1**, laboratory apparatus for producing two phase lixiviants and for measuring the size of the bubbles dispersed therein is shown. The apparatus consists of sparger **10** and a bubble viewer **12**. The viewer **12** comprising a clear plastic case, 0.25 inch in depth, 2.5 inches wide, and 6 inches high. The outlet of the viewer **12** (not shown) is partially submerged in a beaker full of water which keeps the viewer full of fluid during experimentation. The upper portion **14** of the viewer **12** contains a layer **16** of glass beads which reduces vortex formation while the viewer of the apparatus is filled with a lixiviant.

The sparger comprises a ¼ inch inside diameter stainless steel sintered, porous tube **18**, enclosed by a pipe **20**, which may be filled with pressurized gas through gas inlet **22**. A PVC plug **24** sealed to the bottom of the pipe **20** by an O-ring **26** serves as an air-tight connection between sparger **10** and viewer **12**.

As a modification of the apparatus of FIG. **1**, a twisted stainless steel strap (not shown) having one spiral per inch may be inserted into the porous tube **18**. The spiral is designed to create an angular velocity component in addition to the longitudinal velocity component, both components being the same order of magnitude.

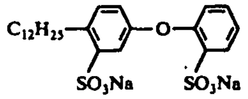
In use, the apparatus is filled with liquid, and water or ammoniated water is introduced through the top of porous tube **18** at a given flow rate. Gas (nitrogen, air, oxygen, or oxygen enriched air) is introduced through gas inlet **22** under pressure and thereby forced through the porous walls of tube **18**. The gas may also include various gaseous oxidants comprising acid forming gases such as SO_2, SO_3, or NO_2. The gas bubbles produced within the tube **18** are then sheared from the interior walls of tube **18** and carried through plug **24** and glass beads **16** into the viewing area **13** of viewer **12** by the liquid flow. Using this procedure and apparatus, it is possible to study the effects of various parameters on the bubble size and stability of lixiviants produced, e.g., the effect of the gas flow rate, liquid flow rate, inclusion of the spiral, and the inclusion of various additives combined with the liquid phase of the lixiviants. The object of the experiments was to produce a stable, two-phase lixiviant which could be delivered to the leaching interval of an in-situ mine at a reasonable flow rate without phase separation. In this regard, it has been discovered that the success of such lixiviants in in-situ mining techniques depends on the size of the gas bubbles being generally about 10–100 microns.

The size of a single gas bubble, in general, can be determined quite easily from its ascending velocity in a fluid of known viscosity. However, the apparatus of FIG. **1** was developed since there was no established method for measuring the size of large numbers of gas bubbles in the fluid. The size of bubbles present in the viewing area **13** of the apparatus of FIG. **1** may be easily determined if a photograph is taken of viewing area **13**. The photographic method was employed because it was both direct and simple.

From a series of experiments on various lixiviants produced using the apparatus of FIG. **1**, a lixiviant has been developed which makes the in situ mining of the type described above more efficient. Specifically, the addition of an effective amount of one or more surfactants to the aqueous phase of the two-phase lixiviants produced enabled the production of bubbles of the desired size range and substantially reduced coalescence of bubbles. With a surfactant, the size of the bubbles are within the range of 0.1 to 0.5 mm (lixiviant at atmospheric pressure). Without a surfactant, two-phase lixiviants produced under identical conditions have a bubble size range of 1.0 to 1.5 mm. In short, the addition of the surfactant yields a 3 to 15 times improvement in gas bubble size distribution in the low pressure simulator of FIG. **1**. These results have been observed to be reproducible and even improved upon in commercial embodiments of the lixiviant production apparatus of the type described in the aforementioned U.S. applications Ser. Nos. 724,548 and 724,549. Further, some attempts at in-situ mining operations which failed when conventional two-phase lixiviants were employed because of the formation of debilitating gas pockets, were rendered successful by using the modified process of the instant invention employing the surfactant stabilized lixiviant.

With conventional two phase lixiviants, the minimum liquid downward velocity of the lixiviant is about one foot per second. At slower velocities, pockets of gas tend to form and the upward rise of the bubbles exceeds the downward flow of the liquid. However, the addition of as little as 25 parts per million surfactant in the lixiviant reduces the minimum downward velocity of the liquid phase by a factor between about 3 and 5.

The exact nature of the surfactant useful in the lixiviant compositions of the invention is not particularly critical. A surfactant sold under the tradename Dowfax 2A1 by Dow Chemical Company has been found satisfactory. Dowfax 2A1 surfactant is an anionic surface-active agent of the sulfonate type. In the dry state, it is a light-colored, free-flowing powder containing a minimum of 90 percent active ingredient. The solution form is a clear, light yellow liquid having a minimum active concentration of 45 percent. The product is the sodium salt of dodecylated oxydibenzene disulfonate having the formula



The selected surfactant should have good wetting ability on mineral surfaces. However, it should not impair the mass transfer rate of the oxygen (or other oxidants) to the mineral surface. Furthermore, the surfactant should not interfere with the reactions which take place. Non-limiting examples of useful surfactants include anionic surfactants such as carboxylates, sulfonates, sulfates, and phosphates, cationic surfactants such as primary, secondary, and tertiary amines and quaternary ammoniums, and nonionic surfactants such as the type having hydroxyl solubilizing groups. Polyoxyethylene chaigs are also useful with the invention as are amphoteric surfactants made from combinations of anionic and cationic moieties.

A broad disclosure of surface active agents is found in U.S. Pat. Nos. 3,249,465 and 3,318,817. Nonlimiting examples of carboxylates include N-acrylsarcosinates and aminocaboxylates. Useful sulfonates include alkylbenzenesulfonates, petroleum sulfonates, sulfosuccinates, naphthalenesulfonates, N-acyl-N-alkyl taurates, 2-sulfoethyl esters of fatty acids, and olefin sulfonates. Sulfates and sulfated product type anionic surfactants include sulfated alcohols, natural fats and oils, oleic acids, alkanolamides, and esters, and further include ethoxylated and sulfated alkylphenols, and ethoxylated alcohols. Phosphate esters are also useful.

Anionic sulfates include alkylbenzenesulfonates and LAS, a sodium salt of linear dodecylbenzenesulfonate.

Further, petroleum sulfonates produced as byproducts of refining certain petroleum products such as substances known as green soap and mahogany soap are also useful. Useful dialkyl sulfosuccinates include sodium di(2 ethyl hexyl) sulfosuccinates, available commercially from American Cyanamide. Useful naphthalenesulfonates include salts of alkylnaphalene sulfonates, sulfonated formaldehyde, naphthalene condensates, naphthlenesulfonates, and tetrahydronaphthalenesulfonates. Further, 2-sulfoethylesters of fatty acid are useful. Useful nonionic surfactants include such compounds as ethoxylated alkyl phenols, ethoxylated aliphatic alcohols, carboxylic esters, glycerol esters, polyethylene glycol esters, and anhydrosorbital esters.

Suitable surfactants are available commercially under the following tradenames.

Anionic	
Union carbide	
Tergitol 7	Sodium Heptadecyl Sulfate
Tergitol 8	Sodium 2-Ethylhexyl Sulfate
Arco Chemical Co.	
Ultrawet 60L	Triethanolamine Linear Alkylate-Sulfonate
Ultrawet 30 DS	Sodium-Linear Decyl-Benzene Sulfonate
Dow Chemical Co.	
Dowfax 3B2	Disodium 4 Decylated Oxy-Di-Benzene Di Sufonate
Nonionic	
Union Carbide	
Tergitol NP44	Alkylphenyl-Hydroxpoly-oxyethylene
GAF	
Alipal Co-436	Alkylphenoxypoly(ethyleneoxy) Ethanol

It is preferred that the lixiviant also contain an agent which enhances the stabilization of Ca++ ions in solution since these are often found together with the metal values of interest. The use of such an agent, however, is conventional in the oil drilling art to inhibit scale formation. A suitable Ca++ ion stabilizer is Calnox® which is a sodium polyacrylate sold by the Atlas Powder Co. In general, other sodium polyacrylates can be used as scale inhibitors. It should be noted that the inclusion of an agent such as Calnox® in the lixiviant is not considered to be a part of the invention.

The amount of surfactant added per volume of liquid phase of the lixiviant will vary with the particular surfactant used. Successful lixiviant compositions have been made which include only 25 ppm Dowfax and mixtures of 25 ppm Dowfax with 75 ppm Calnox®. As an additional bubble coalescent inhibitor, aluminum ions in the form of 1-2 grams per liter $Al_2(SO_4)_3$ has been found to be effective.

The advantages and features of the process of the invention will be further understood from the following examples, which in no event should be construed as limiting.

EXAMPLE 1

Effect of Liquid Velocity in Porous Tube

Experiments were carried out with a constant gas flow rate of 700 standard cubic centimeters per minute (SCCM) for various liquid flow rates. At high flow rate-low gas volume fraction, the gas is well dispersed in solution. As the liquid flow rate decreases, the volume fraction of gas increases and the flow gradually changes from bubbly flow to slug flow, i.e., large amoebalike bubbles are formed. As the liquid flow rate is decreased, there is a transition where large agglomerated gas bubbles are formed having a non-spherical shape in excess of 5 ml in size.

The estimated bubble size, as a function of linear velocity in the porous tube and as measured by photographic methods using the apparatus of FIG. 1, is given in Tables 1 through 3. It is abundantly clear that the greater the liquid velocity, the smaller the gas bubble size range.

Table 1

Estimated Bubble Size, No Surfactant, With Spiral

Run #	Liq flow rate (GPM)	linear velocity (ft/sec)	gas volume fraction %	Bubble Sizes (mm)
424-1	4.8	31.4	3.7	0.5-1.5
-2	3.15	20.6	5.5	1.0-1.5
-3	2.2	14.4	7.7	2.0-5.0 Transition
427-16	3.75	24.5	4.7	0.5-1.5
-17	3.15	20.6	5.5	1.0-2.5
-18	2.48	16.2	6.9	1.0-3.0
-19	2.2	14.4	7.7	2.0-5.0 Transition

Table 2

Estimated Bubble Size, With Dowfax, No Spiral In Porous Tube

Run #	Liq flow rate (GPM)	linear velocity (ft/sec)	gas volume fraction %	Bubble Sizes (mm)
419-1	3.65	23.9	4.8	0.1-0.5
-2	4.05	26.5	4.4	0.1-0.5
-3	3.15	20.5	5.5	0.1-0.5
-4	2.65	17.3	3.8	0.2-0.6
-5	2.15	14.1	7.9	0.25-1.0
-6	1.8	11.8	9.3	0.2-2.0
-7	1.6	10.5	10.4	0.2-3.0
-8	1.5	9.8	11.0	0.2-4.0 Transition

Table 3

Estimated Bubble Size, With Dowfax, With Spiral In Porous Tube

Run#	Liq flow rate (GPM)	linear velocity (ft/sec)	gas volume fraction %	Bubble Sizes (mm)
425-4	4.8	31.4	3.7	0.1-0.5
-5	4.1	26.0	4.3	0.1-0.5
-6	3.15	20.6	5.5	0.2-0.6
-7	2.2	13.7	7.7	0.2-0.75
-8	1.25	8.2	12.9	1-2.5 Transition
-9	0.60	3.9	23.5	2-5.0
426-11	3.75	24.5	4.7	0.1-0.5
-13	3.15	20.6	5.5	0.2-0.6
-14	2.2	14.4	7.7	0.3-0.75
-15	1.25	8.17	12.9	0.5-2.0 Transition

EXAMPLE 2

The Effect of Surfactant Addition

As can be seen from a comparison of Tables 1 and 2 above, the bubble size range is significantly smaller in lixiviant containing a surfactant versus a lixiviant without a surfactant. The bubble size range in lixiviants containing a surfactant, as exemplified by a comparison between the 3.5 gpm flow rate, are in the range of 0.1 to 0.5 mm, whereas, without a surfactant, the range is between 1.0 and 1.5 mm. It is estimated that the addition of an effective amount of surfactant reduces bubble size by a factor of 15.

EXAMPLE 3

The Effect of Inclusion of a Spiral Strap

A twisted stainless steel strap having one spiral per inch was inserted into the interior of porous tube **18** to create an angular velocity component in addition to the longitudinal velocity component. By comparing the results disclosed in Table 2 with those of Table 3, it can be seen that the spiral can reduce the transition flow rate from 1.5 gpm to 1.25 gpm. However, no noticeable effect on bubble size was observed.

The separation of the gaseous and liquid phases in the field at the injectin interval seriously reduces the leaching efficiency and oxygen utilization. The following examples disclose laboratory data which demonstrate the improvement in stability of the lixiviant useful in the process of the invention as compared to conventional two-phase lixiviants containing no surfactant.

FIG. **2** shows laboratory apparatus designed to measure and compare the stability of various lixiviant formulations. The apparatus comprises a 2-inch inside diameter clear plastic tube **28**, **18** inches long, the upper end of which is attached to a sparger unit **30**. The sparger unit **30** comprises a pair of coaxial pipes **32**, **34** for producing the two-phase lixiviants. Pipe **34** is porous (sintered metal) and carries the liquid phase of the lixiviant which, within sparger **30**, is filled with gas bubbles formed when gas passes from reservoir **36** into pipe **32**, under pressure, and through the wall of pipe **34**. The rate of gas flow, in ml/min., is monitored by flow meter **38**. In addition, the mass of gas entering tube **32** per unit time may be monitored by mass flowmeter **40**. A solenoid valve **42** is provided in the gas feed line.

The liquid phase of the lixiviant is pumped at various flow rates by a variable speed gear pump **44** from reservoir **46**. Its flow rate is monitored by flowmeter **48**. A pressure gage **50** serves to display the pressure of the liquid phase during operation of the apparatus and at a time after shut down. Solenoid valves **52**, **54** are designed to operate synchronously together with valve **42**. A T.V. camera, monitor, and video tape deck equipped with a slow motion and frame stopping feature were used during all experiments so that precise measurements could be taken after performance of the experiments.

A typical run started with mixing gas and liquid at the sparger **30**. The mixture was allowed to flow through tube **28** for 15 minutes to achieve steady state, and thereafter, solenoid valves **42**, **52**, and **54** were closed simultaneously. At the same time, a precision timer (not shown) was actuated.

Upon shut off of the valves, gas bubbles begin to rise in the tube **28**. Larger bubbles rise faster than small ones. When the gas bubbles reach the gas-liquid interface faster than they break away, a third phase, the foam phase, is created. As time goes on, the foam-liquid interface builds up and the foam-gas interface moves down. FIG. **3** is a plot of fluid interface changes as a function of time comparing a run using a lixiviant containing no surfactant to one using 25 ppm surfactant. As can be seen from FIG. **3**, the inclusion of a surfactant increases the time necessary for separation of the phases by a factor of three, and, in addition, reduces the total amount of gas accumulated in the flow tube as evidenced by the higher final liquid-gas interface level.

Because of the asymptotic nature of the curves, separation time was defined as the time when the liquid foam interface reached 95% of its final height. FIG. **4** is a plot of separation time as a function of liquid flow rate which compares the stability of lixiviants with and without added surfactant. The results indicate that the separation time is not a simple function of flowrate, but that, in the presence of a surfactant, the separation time is generally 4–5 times longer than in its absence. This means the average bubble size is 16–25 times smaller in the surfactant solutions.

After the gas is totally separated from the liquid, the volume fraction of gas in the tube **28** is calculated from the height of the interface and the tube pressure. This volume fraction is compared with the sparger volume fraction, as calculated from the gas and liquid flow rate, in the graph of FIG. **5**. As can be seen from FIG. **5**, the addition of a surfactant to the lixiviant effectively reduces gas accumulation in the tube. Apparently accumulation takes place only during the first few minutes of the run, since prolonging the flow of the two phase mixture from 15 minutes to 2 hours has no effect on the total gas accumulation.

Experiments were also conducted to study the effect of reducing the void space within tube **28** to simulate underground leaching interval conditions. Tube **28** was filled with 7/16 in. diameter Al_2O_3 spheres to reduce the void space and thereby increase creeping velocity. Results of experiments using the apparatus of FIG. **2** with this modification show that the separation time increases by a factor of **10** (see FIG. **6** for interface-time curves) and that gas accumulation is further reduced in the tubes.

To test the effect of lengthening of tube **28**, runs were conducted using the apparatus of FIG. **2** except that a 50 inch, 1.75 I.D. tube was used in place of tube **28**. Based on these experiments, the separation time was found to be proportional to the tube length. The volume fraction of gas accumulation was not affected by tube length for lixiviants embodying the invention (see open triangles of FIG. **5**). However, for lixiviants containing no surfactant, the flow was very unstable and large gas accumulations resulted (see solid triangles of FIG. **5**).

EXAMPLE 4

To illustrate the effectiveness of the process of the invention employing a lixiviant of the type set forth above, a leaching test was carried out in a copper mineral deposit, consisting chiefly of chalcopyrite, having an average grade of copper of 0.45% and an ore body porosity of approximately 3%. No rubblizing was conducted. As set forth in the flow sheet illustrating the process (FIG. **7**), the aqueous phase of the lixiviant was made from 1 M NH_3, 0.25 M $(NH_4)_2SO_4$, 25 parts per million Dowfax,® and 75 parts per million Calnox®. This phase was injected into the leaching interval of the ore body at a rate of 10 gallons per minute after being mixed with 12 standard cubic feet per minute gaseous oxygen utilizing a stainless steel sparger of the type set forth in the aforementioned U.S. application Ser. No. 724,549, i.e., a commercial embodiment of a sparger comprising a sintered metal porous tube such as that briefly described herein. The two-phase fluid was injected under pressure into the leaching interval through a tailpipe extending to 3160 feet. Downward fluid velocity in the 1.5 inch pipe was 1.8 feet per second.

Pregnant solution was recovered at a rate of 10 gallons per minute from a production well located 70 feet from the injection well. Forty-five days later, the recovered solution contained 0.71 g/l copper, 0.66 M NH_3, 0.25 M $(NH_4)_2SO_4$, 0.04 M $CaSO_4$, 20 ppm Dowfax,® and 50 ppm Calnox.®

As part of the $(NH_4)_2SO_4$ was treated with lime to regenerate the ammonia and also to remove the $CaSO_4$ from the solution. The copper was then extracted by electrowinning and the raffinate of the extraction was recycled with addition of make up chemicals as indicated in FIG. 7.

In one important embodiment of the process of the invention, as set forth in the example above, the aqueous phase of the lixiviant comprises an ammoniacal solution and the gaseous phase comprises oxygen. During leaching, the following reactions are believed to occur:

$$4\ CuFeS_2 + 17\ O_2 + 6H_2O + 24\ NH_3 \rightarrow 4\ Cu(NH_3)_4SO_4 + 4FeOOH + 4\ (NH_4)_2SO_4 \quad (1)$$

$$4FeS_2 + 15O_2 + 10H_2O + 16NH_3 \rightarrow 4FeOOH + 8(NH_4)_2\ SO_4 \text{ (a possible undesirable side reaction)} \quad (2)$$

Of course, nickel, cobalt and molybdenum, if present as sulfides in the ore will also be leached in accordance with known chemistry. At this point, it should be noted that the primary purpose of the oxygen is to break the chemical bonds holding the copper in the chalcopyrite by oxidizing the sulfide and iron components. Once the chalcopyrite is oxidized, the aqueous ammonia is able to dissolve the copper values. It makes no difference whether or not the copper is oxidized. Indeed it is believed that the $CuFeS_2$ contains copper as cupric copper and iron as ferrous iron. Thus, during oxidation in accordance with the foregoing reaction, the oxidation state of copper remains unchanged while the iron is oxidized from Fe^{+2} to Fe^{+3}. Of course, if Cu^+ copper is present in the ore, it would also be leached by the lixiviant. Since both forms of copper ions are leachable, it is not necessary to oxidize cuprous ions to cupric ions in order to leach copper.

A sufficient excess of aqueous ammonia is used to keep the pregnant solution alkaline. Under these conditions, dissolution of gangue materials is negligible and the pregnant solution contains essentially only ammonia, ammonium sulfate, and cupric ammine sulfate.

It is to be understood that the process of the invention can be employed with aqueous phases other than ammoniacal solution. The oxygen-ammonia lixiviant is preferred where there is a lot of acid-consuming minerals in the ore body. However, a representative example of another two-phase system that can be used to leach copper and nickel from a sulfate deposit includes the so-called oxygen-water lixiviant.

The chemistry for the oxygen-water lixiviant appears below.

$$4CuFeS_2 + 17\ O_2 + 6\ H_2O \rightarrow 4\ CuSO_4 + 4\ FeOOH + 4\ H_2SO_4$$

$$4\ FeS_2 + 15\ O_2 + 10\ H_2O \rightarrow 4\ FeOOH + 8\ H_2SO_4$$

When an oxygen-water lixiviant is used, cupric sulfate and sulfuric acid are generated in the leaching process or added on the surface. The cupric sulfate and sulfuric acid dissolve gangue metal oxides (Fe, Mg, Al, Ca, etc.) as sulfates. Much of the iron and aluminum precipitates in-situ as jarosite and alunite. In the surface plant, copper is extracted, and if necessary, the pH is adjusted to the desired level. The resulting leach solution is reinjected together with make-up oxygen. Another name for the oxygen-water lixiviant is the oxygen-acid lixiviant.

The process of the present invention is used to great advantage for deep-lying ore bodies, that is, ore bodies located at a depth of 1,000 feet or more below the surface. Although, the surface is normally a land surface, it should be noted that there is no reason why this process cannot be used to recover copper from deposits located below the bottom of the continental shelf or a lake bed. Thus, when reference is made to the depth of deposit below the surface, the surface can either be land surface or water surface.

The invention may be embodied in other specific forms without departing from the spirit or essential characteristics thereof. The present embodiments are therefore to be considered in all respects as illustrative and not restrictive, the scope of the invention being indicated by the appended claims rather than by the foregoing description, and all changes which come within the meaning and range of equivalency of the claims are therefore intended to be embraced therein.

What is claimed is:

1. A process for the in-situ mining of metal values selected from copper, nickel or mixtures thereof from a subterranean ore body characterized by natural microscopic fracture openings, a portion of said body being a sulfidic ore, said process being characterized by the steps of:

a. drilling at least one injection hole and at least one production hole communicating with said ore body;

b. forming a stable, two-phase lixiviant comprising an aqueous phase,

a multiplicity of gaseous, oxygen containing bubbles having a size, when subjected to the pressure employed during in-situ mining, sufficient to pass through the fracture openings in said ore body, and

a surfactant for maintaining the size and individuality of said bubbles;

c. forcing the lixiviant formed in step (b), under pressure, through said ore body to cause the lixiviant to penetrate the ore body through the natural microscopic fracture openings, the oxygen in said lixiviant reacting with the sulfide in the ore to enable metal ions to be solubilized by said aqueous phase;

d. withdrawing the resultant metal bearing aqueous phase to the surface through said production hole; and

e. recovering metal values from said aqueous phase.

2. The process as set forth in claim 1 wherein the metal barren aqueous phase produced in step (e) is returned to the ore body through an injection hole.

3. The process as set forth in claim 1 wherein the ore body is a porphyry copper ore body in which copper bearing minerals occur in disseminated grains.

4. The process as set forth in claim 1 wherein the ore body comprises chalcopyrites.

5. The process set forth in claim 1 wherein said aqueous phase comprises an ammoniated leach liquor.

6. The process as set forth in claim 1 wherein said forming step is effected as follows:

1. supplying said aqueous phase containing said surfactant to the interior of a porous tube formed of sintered powdered metal;

2. supplying said oxygen containing gas to the exterior of said tube under pressure sufficient to cause said gas to penetrate to the interior of said tube; and

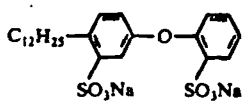
3. passing the aqueous phase supplied in step 1 through said tube to cause said aqueous phase to shear gas bubbles from the interior thereof.

7. The process as set forth in claim 6 wherein said forming step is effected above ground.

8. The process as set forth in claim 6 wherein said surfactant is selected from nonionic surface active agents and anionic surface active agents.

9. The process as set forth in claim 6 wherein said surfactant is the sodium salt of dodecylated oxydibenzene disulfonate.

10. The process as set forth in claim 9 wherein said sodium salt of dodecylated oxydibenzene disulfonate has the following formula



11. The process as set forth in claim 10 wherein said surfactant is present in the lixiviant in an amount between the range of 5–200 ppm.

* * * * *

which demonstrates the inadequacy of the second-order model. With the use of trial 7 by calculation of one coefficient B_{123} it was possible to expand the quadratic model into an incomplete cubic model:

$$y = 1,194X_1 + 44,044X_2 + 288,438X_3 + 177,856X_1X_2 +$$
$$+ 247,244X_1X_3 - 116,236X_2X_3 - 960,189X_1X_2X_3$$

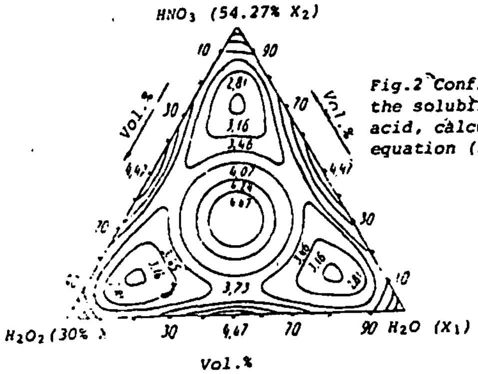


Fig.2 Confidence interval for the solubilities of molybdic acid, calculated by means of equation (2).

A check by means of trials 8 and 9 showed that the incomplete cubic model (2) adequately describes the experimental results (calculated Student criterion -2.48, tabular value -3.36, significance level 0.1). This equation was used to construct contour curves for the solubility of molybdic acid in nitric acid-peroxide solutions (fig. 1).

To determine the error in the solubility value predicted by Eq. (2) we calculated the confidence interval, the values of which are given on the diagram (fig. 2).

References

1) A N Zelikman et alia: Izv. Vuz. Tsvetnaya Metallurgiya 1966, No. 6
2) Yu N Yurkevich and K Ya Shapiro in: Metallurgy of tung molybdenum, and niobium, Nauka, Moscow, 1967 p. 33
3) A N Zelikman et alia: Zh. Neorgan. Khim. 1972, 17, N
4) V V Nalimov and N A Chernova: Statistical methods of planning extremal experiments, Nauka, Moscow, 1965.
5) V V Nalimov (editor): New ideas in experimental desig Nauka, Moscow, 1969.

Sov. Non-Fe
1975 v.3 N6

UDC 669.283:661.183.12

Investigation of the sorption of molybdenum ions by Furan anion-exchange resins

N A Gabrielyan, V Abdilakhadov, R A Nazirova and M A Askarov (Tashkent Polytechnical Institute - Department of the Technology of the Processing of Plastics)

It is known that only the use of anion-exchange resins is effective for the isolation of molybdenum from industrial solutions. In the present work the sorption characteristics of weakly basic furan anion-exchange resins, synthesized by the Tashkent Polytechnical Institute by polycondensation of furfural with polyethylenepolyamine (PEPA) in the presence of hydrofuramide (FAN-3 resin), phenol sulphides (FA-S), melamine (FA-M), thiuram-D (FA-T), were investigated to establish the possibility of using them in the hydrometallurgy of molybdenum[1-3]. The industrial polycondensation anion-exchange resin AN-2F was used for comparison.

The resins were prepared by the previously described method[4]. The working solutions were prepared from ammonium concentration was determined by a thiocyanate method on an SF-14 spectrophotometer[5]. The principal characteristics of the investigated samples of resins in the OH form are given in table 1, where SEC represents the statistical exchange capacity, V is the specific volume of the swollen resin, G is the bulk density, and φ is the moisture content.

Table 1: Characteristics of the furan anion-exchange resins

Resin grade	SEC in 0.1N HCl mg-eq/g (dry)	V ml/g	G g/ml	φ %
FAN-3	4.32	2.61	0.65	13.3
FA-S	3.60	2.10	0.70	26.0
FA-T	5.00	3.50	0.55	12.5
FA-M_1	4.70	3.44	0.65	16.7
FA-M_2	4.30	3.00	0.68	10.5
AN-2F	7.50	2.70	0.80	5.0

The FA-M_1 and FA-M_2 resins contain different amounts of melamine for the weight of furfural: in the first the amount of melamine is 1 and in the second it is 0.1 parts by weight to 0.3 part by weight of furfural. The experiments on the sorption of molybdenum were carried out with the anion-exchange resins in the Cl and SO_4 forms under static conditions with a resin-solution ratioof 1:1000 and a length of contact between the phases of three days. The molybdenum content of the resin was calculated from the difference between the initial and equilibrium concentrations in the solution.

Table 2 gives the results for the sorption of molybdenum by the resins from solutions of ammonium molybdate with (B) and without (A) competing ions (48 g/l SO_4). The pH of the initial solutions amounted to 4.5-5. Compared with AN-2F, the resins investigated, except FA-S, possess increased sorption capacity for molybdenum ions from pure solutions. Under the above-mentioned conditions high sorption parameters were obtained on FAN-3 resin (350 mg/g).

A different pattern was observed in solutions with competing sulphate ions. Under these conditions the sorption of molybdenum by the AN-2F resin decreases by 71%, and greater selectivity towards molybdenum is given by resins FA-T and FA-M_1, the sorption capacity of which remains as high as before - 277-254 mg/g (i.e., 181-214 mg/g higher than with AN-2F) and decreases by only 5-10% against 49-50% for the others.

The anion-exchange resin synthesized in the presence of phenol sulphides FA-S, being somewhat inferior to AN-2F in its sorption of molybdenum from pure solutions (table 2), possesses identical sorption capacity (63-64 mg/g) in the background solutions.

Table 2: Sorption of molybdenum by various anion-exchange resins

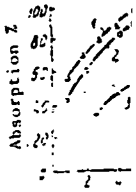
Resin grade	A mg/g (dry)	B mg/g (dry)	Decrease in the sorption of Mo %
FAN-3	350	174	50.0
FA-S	116	60	49.0
FA-T	292	277	5.5
FA-M_1	283	254	10.3
FA-M_2	306	151	50.0
AN-2F	220	63	71.4

On the basis of data on the sorption of molybdenum the tested anion-exchange resins can be arranged in the following series.

From pure solutions of ammonium molybdate: FAN-3 >

FA-M_2 > FA-
From backgr
FA-M_2> AN-2
resin AN-2F,
traction of mo
except the res
in its sorption
ground solutio

On the basis
to study the e
ting ions on t
the FA-T res
towards moly
on the sorptio
etween 1 and
ddition of su
ium sorption
omparison
ne FA-T res
esin in the
that suppres
the chloride
sent in the S



The compe
form of amr
centrations
accompany
of poor mol
composition

**Table 3: Th
of**

NO₃ conten g/l
5
10
15
25
50

The pres
solution su
the presen
concentrat
creases by

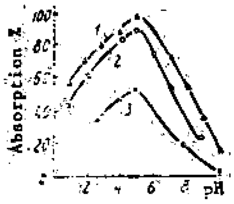
Of practi
the sorptio
the resins
and 7 days
tion of mo
and the ex
equations

Table 4
periods in
of saturat
0.5. At th
constants
constant.
consider

FA-M$_2$ > FA-T$_1$ > FA-M$_1$ > AN-2F > FA-S.

From background solutions: FA-T$_1$ > FA-M$_1$ > FAN-3 > FA-M$_2$ > AN-2F = FA-S. Thus, the industrial anion-exchange resin AN-2F, which is used and recommended for the extraction of molybdenum^1), is inferior to furan resins, except the resin obtained in the presence of phenol sulphides, in its sorption capacity from both pure solutions and background solutions.

On the basis of the results obtained it seemed of interest to study the effect of pH and of the concentration of competing ions on the sorption of molybdenum. For this purpose the FA-T resin was selected as being the most selective towards molybdate ions. The effect of the pH of the medium on the sorption of molybdenum was investigated at values between 1 and 10. The required pH value was obtained by addition of sulphuric acid and ammonia solutions, and maximum sorption was observed in the range of pH 3-5 (fig.). Comparison of the data on the sorption of molybdenum by the FA-T resin in the Cl and SO$_4$ forms shows that on the resin in the Cl form the sorption of molybdenum is somewhat suppressed on account of anionic exchange between the chloride and sulphide ions, whereas this effect is absent in the SO$_4$ form.



Effect of the pH of the medium on the sorption of molybdenum by anion-exchange resins: 1 - FAT - SO_4^{2-}; 2 - FAT-Cl form; 3 - AN-2F-Cl form.

The competing ions were sulphate and nitrate ions in the form of ammonium sulphate and ammonium nitrate at concentrations between 2 and 50 g/l (table 3): they frequently accompany molybdenum in the solutions from the leaching of poor molybdenum-containing products and from the decomposition of molybdenite with nitric acid^7).

Table 3: The effect of NO_3^- and SO_4^{2-} ions on the sorption of molybdenum on FA-T anion-exchange resin

NO_3^- content g/l	Mo absorbed, % of initial	SO_4^{2-} content g/l	Mo absorbed, % of initial
5	75	2	98
10	75	4	-
15	68.5	8	97
25	68.5	15	-
50	62.0	25	95
		50	95

The presence of increasing amounts of nitrate ions in the solution suppresses the sorption of molybdenum more than the presence of sulphate ions. Within the limits of SO_4^{2-} ion concentrations of 2-50 g/l the sorption of molybdenum decreases by only 2-5%.

Of practical interest also is investigation of the kinetics of the sorption of molybdenum, which were investigated on all the resins under static conditions for times between 20 min and 7 days. It is known that the controlling stage in the sorption of molybdenum is usually diffusion in the resin grain^{8,9}), and the experimental data were therefore treated by the usual equations for internal diffusion^{10}

Table 4 gives the diffusion coefficients ($\bar{D}$) for the initial periods in the sorption of molybdenum, where the degree of saturation of the anion-exchange resin (F) did not exceed 0.5. At the initial moments of sorption the exchange rate constants and diffusion coefficients remained practically constant. Under identical conditions molybdenum is sorbed considerably more quickly by the FA-S, FA-M, and FAN-3 resins, where the diffusion coefficients are 2-4 times higher than in the AN-2F resin. The maximum $\bar{D}$ values are obtained in the FA-S resin.

Table 4: Kinetics characteristics of the sorption of molybdenum on anion-exchange resins

Resin grade	F	$D_{\frac{1}{2}}$/sec cm^2/sec	τ, min
FAN-3	0.16 0.23 0.32 0.51	0.89 1.12 1.05 1.48	20 30 60 120
FA-S	0.22 0.37 0.51	3.36 3.00 2.86	20 60 120
FA-T	0.055 0.097 0.18 0.24 0.30 0.35	0.60 0.50 0.83 0.73 0.70 0.64	20 30 60 120 180 240
FA-M$_1$	0.25 0.34 0.46	1.97 2.28 2.16	20 30 60
FA-M$_2$	0.14 0.18 0.35 0.45	0.59 0.65 0.56 0.47	20 30 120 240
AN-2F	0.25 0.28 0.36	0.85 0.81 0.62	30 60 120

An increase of the amount of melamine in the FA-M$_1$ resin raises the sorption rate. The diffusion coefficients on FA-M$_1$ are 3.3-3.5 times greater than on FA-M$_2$. In their kinetic characteristics the FAN-3, FA-M$_1$, and FA-S resins are superior to the industrial AN-2F resin.

Thus, furan ion-exchange resins, which possess a set of valuable characteristics (increased thermal, chemical, and radiation stability, mechanical strength) are capable of sorbing anionic complexes of molybdenum both from pure solutions and from mixtures. The high sorption capacity, selectivity and good kinetic characteristics make it possible to consider that these resins can be used in the hydrometallurgy of molybdenum for the purposes of extraction and separation and are therefore of practical interest for further investigation.

References

1) Author's certificate No.204580, 1967, SSSR.
2) Author's certificate SSSR, No.384374, 1973.
3) Author's certificate SSSR, No. 339169, 1970.
4) K M Saldadze et alia: Ion-exchange high-molecular compounds. Goskhimizdat, Moscow 1960.
5) F M Shemyakin and V V Stepin: Ion-exchange chromatographic analysis of metals. Metallurgiya, Moscow, Metallurgiya 1970.
6) A N Zelikman: Molybdenum. Metallurgiya, Moscow 1970.
7) E I Kazantsev et alia: Izv. Vuz Tsvetnaya Metallurgiya 1972, (2), 68.
8) N M Sobinyakova et alia: Ion-exchange sorbents in industry. Izd. Akad. Nauk SSSR, Moscow 1963.
9) B N Laskorin et alia: Ion-exchange sorbents in industry. Izd. Akad. Nauk SSSR, Moscow 1963.
10) Boyd et alia: Chromatographic method for separation of ions. IL, Moscow 1949.

United States Patent [19] | [11] 4,116,488
Hsueh et al. | [45] Sep. 26, 1978

[54] **IN-SITU MINING METHOD AND APPARATUS**

[75] Inventors: **Limin Hsueh**, Bedford; **Robert A. Hard**, Still River; **Donald H. Davidson**, Bedford; **Ray V. Huff**, Acton, all of Mass.

[73] Assignee: **Kennecott Copper Corporation**, New York, N.Y.

[21] Appl. No.: **724,548**

[22] Filed: **Sep. 20, 1976**

[51] **Int. Cl.**^{2} **E21C 43/28**
[52] **U.S. Cl.** **299/4**; 261/122
[58] **Field of Search** .... 299/4, 5; 261/77, 122, 261/94, 100, 103, 104, 107, DIG. 75; 423/18

[56] **References Cited**

U.S. PATENT DOCUMENTS

3,307,567	3/1967	Gogarty et al.	137/13
3,545,731	12/1970	McManus	261/122
3,708,206	1/1973	Hard et al.	299/5
3,958,945	5/1976	Takahashi	261/122

FOREIGN PATENT DOCUMENTS

694,918	7/1953	United Kingdom	261/DIG. 75

Primary Examiner—Ernest R. Purser
Attorney, Agent, or Firm—John L. Sniado; Anthony M. Lorusso

[57] **ABSTRACT**

An environmentally compatible, industrially safe, and potentially economic means for recovering copper and/or nickel from deep-seated deposits without resorting to extensive underground development. A two-phase ammoniacal leach solution containing oxygen bubbles is forced under high pressure through an injection hole several thousand feet deep into a leaching interval of a deep lying deposit containing copper or nickel or copper and nickel. The two-phase leach solution travels through the leaching interval of the deep lying deposit and is pumped out of withdrawal holes spaced apart from the injection hole.

The two-phase leach solution under high pressure (more than 500 psi) penetrates the deposit through cracks, fissures and fractures, leaching copper and/or nickel along the way. Under a controlled pressure gradient, the leaching solution migrates over a period of time to receiving holes from which the pregnant leaching solution is withdrawn. The pregnant solution is then processed for recovery of copper or nickel or copper and nickel before it is returned to an injection hole.

A method and apparatus for producing the two-phase leach solution and for maintaining a system in which the gas bubbles are able to penetrate the deposit is disclosed.

Also disclosed are various parameters, such as hole spacing techniques, hole completion techniques and stimulation techniques.

98 Claims, 8 Drawing Figures

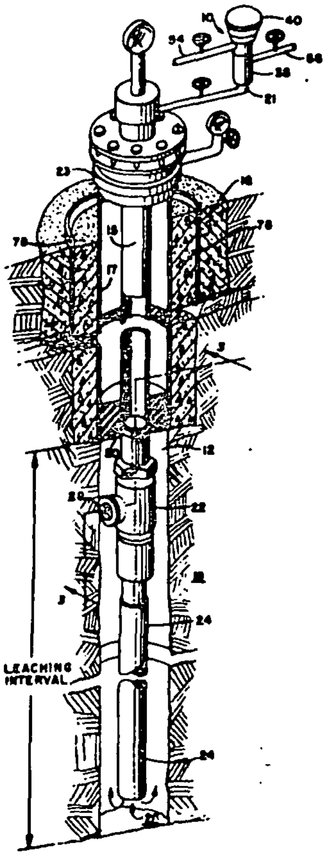


FIG. 1.

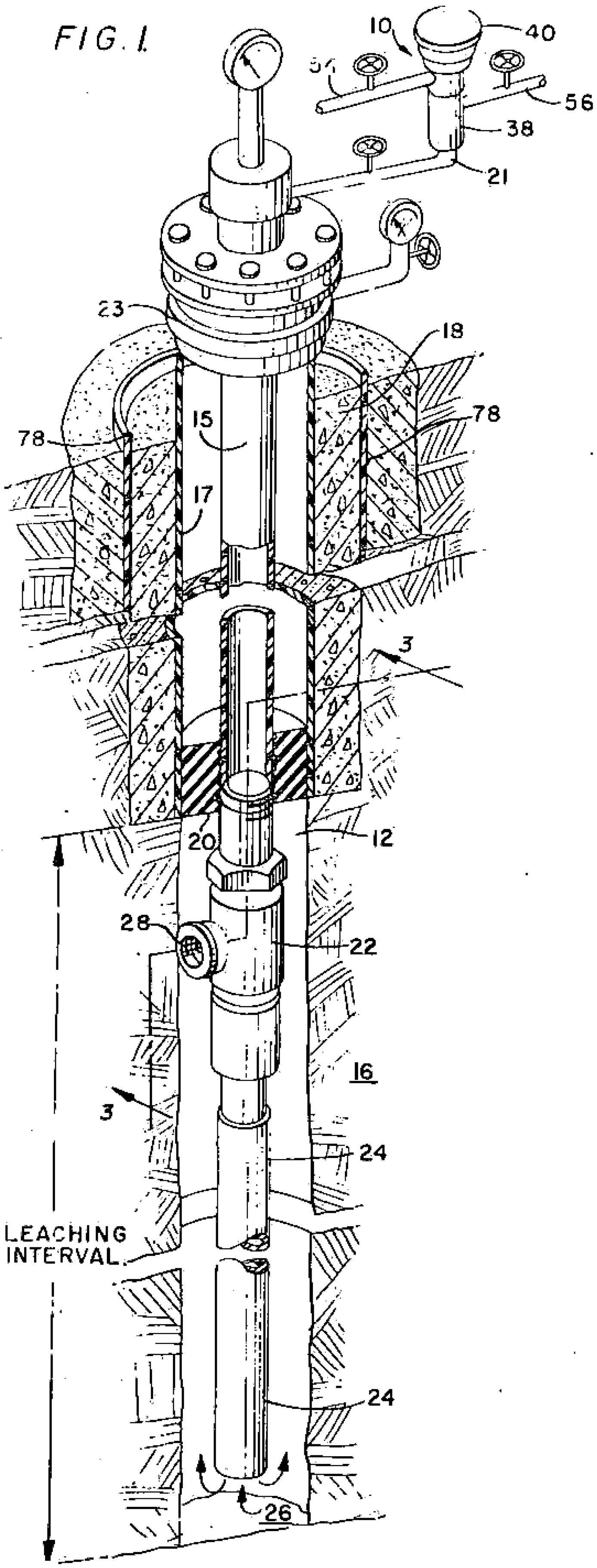

10
40
54
56
38
21
23
18
78
15
78
17
3
20
12
28
22
16
3
24
LEACHING INTERVAL
24
26

FIG. 2.

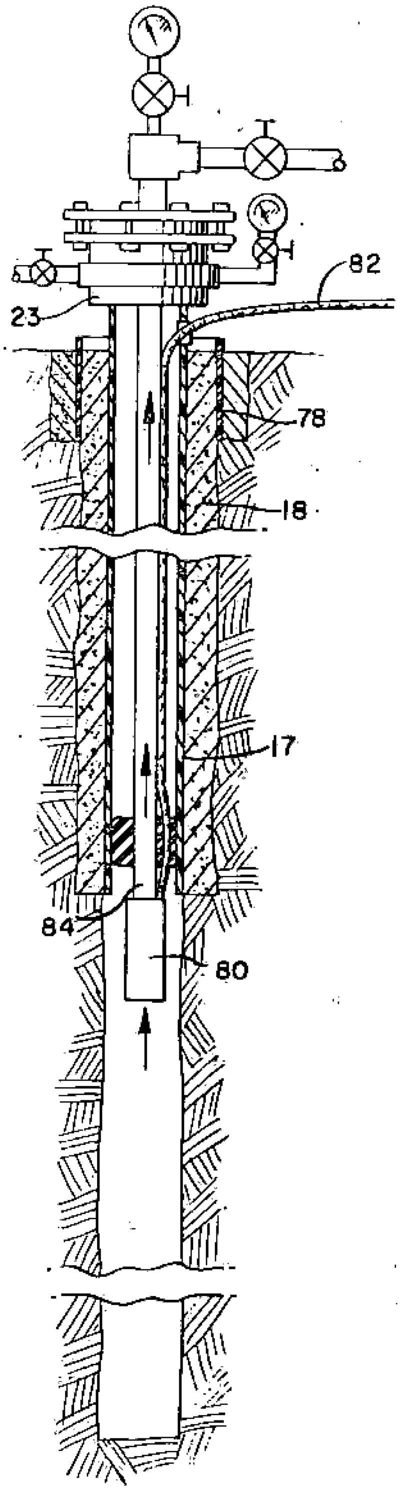

82
23
78
18
17
84
80

FIG. 3.

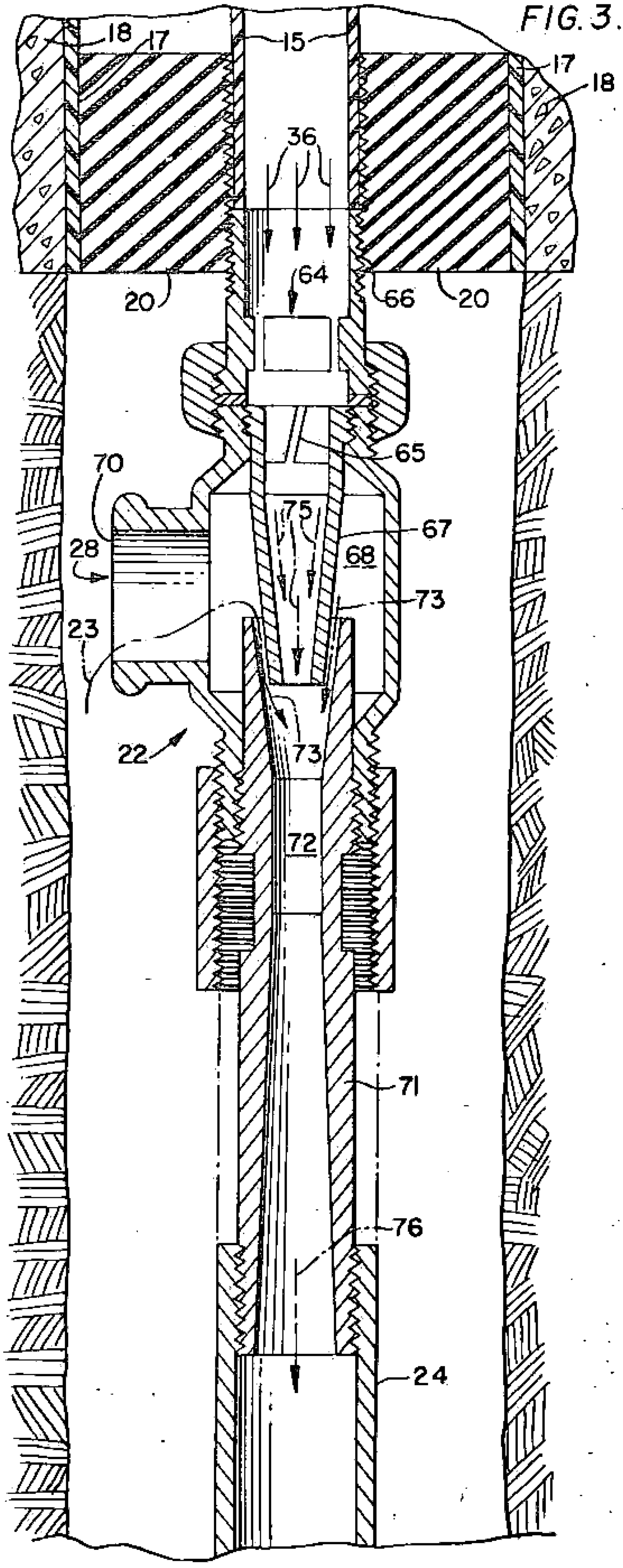

18
17
15
17
18
36
64
20
66
20
65
70
75
67
28
68
73
23
73
22
72
71
76
24

FIG. 4.

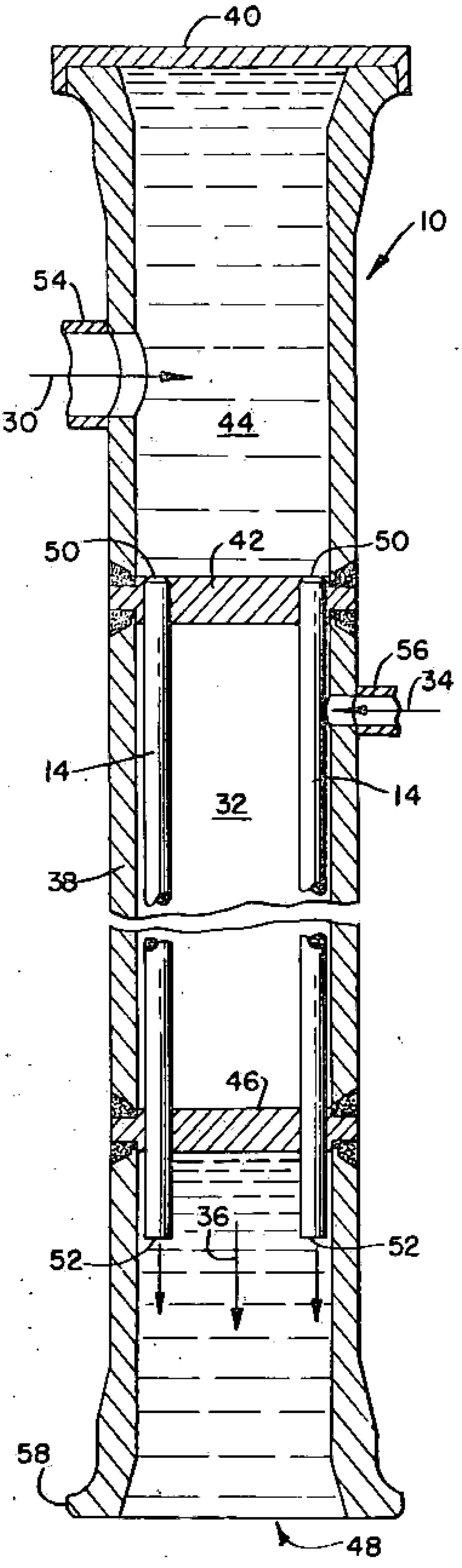

40
10
54
30
44
50
42
50
56
34
14
32
14
38
46
36
52
52
58
48

FIG. 5.

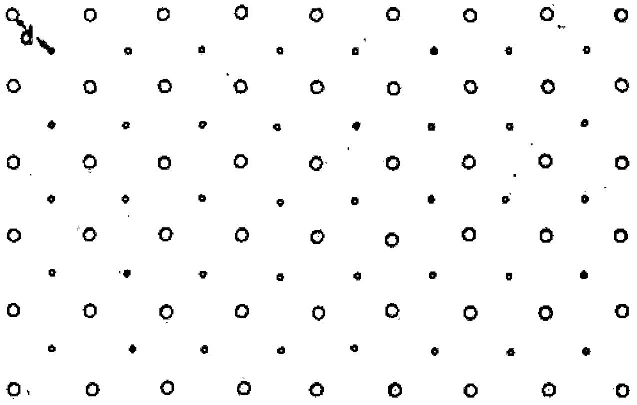


FIG. 6a.

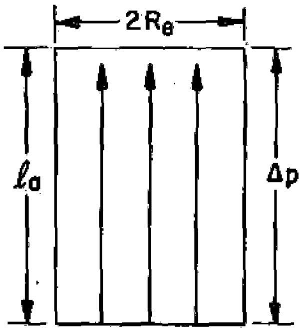


FIG. 6b.

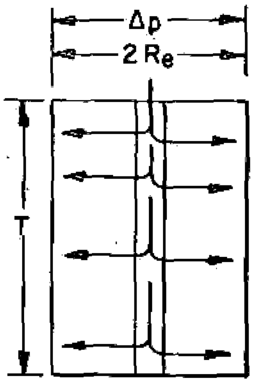


FIG. 6C.

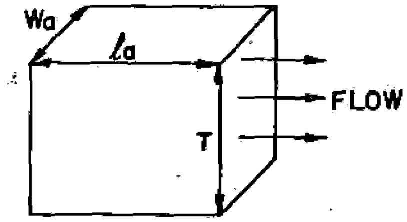


FIG. 7.

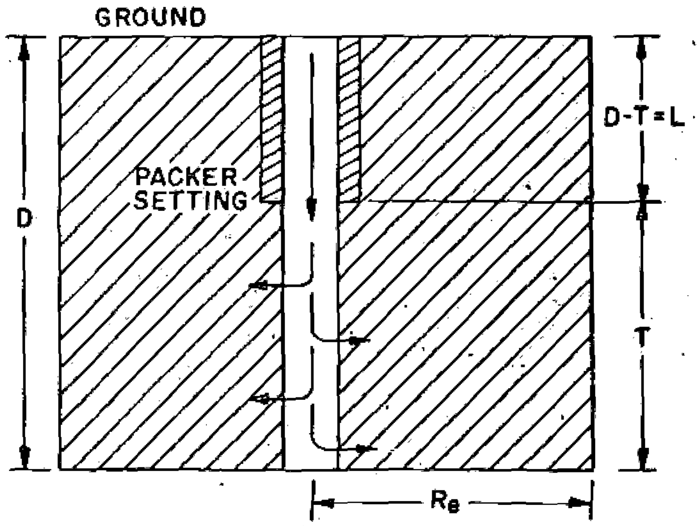
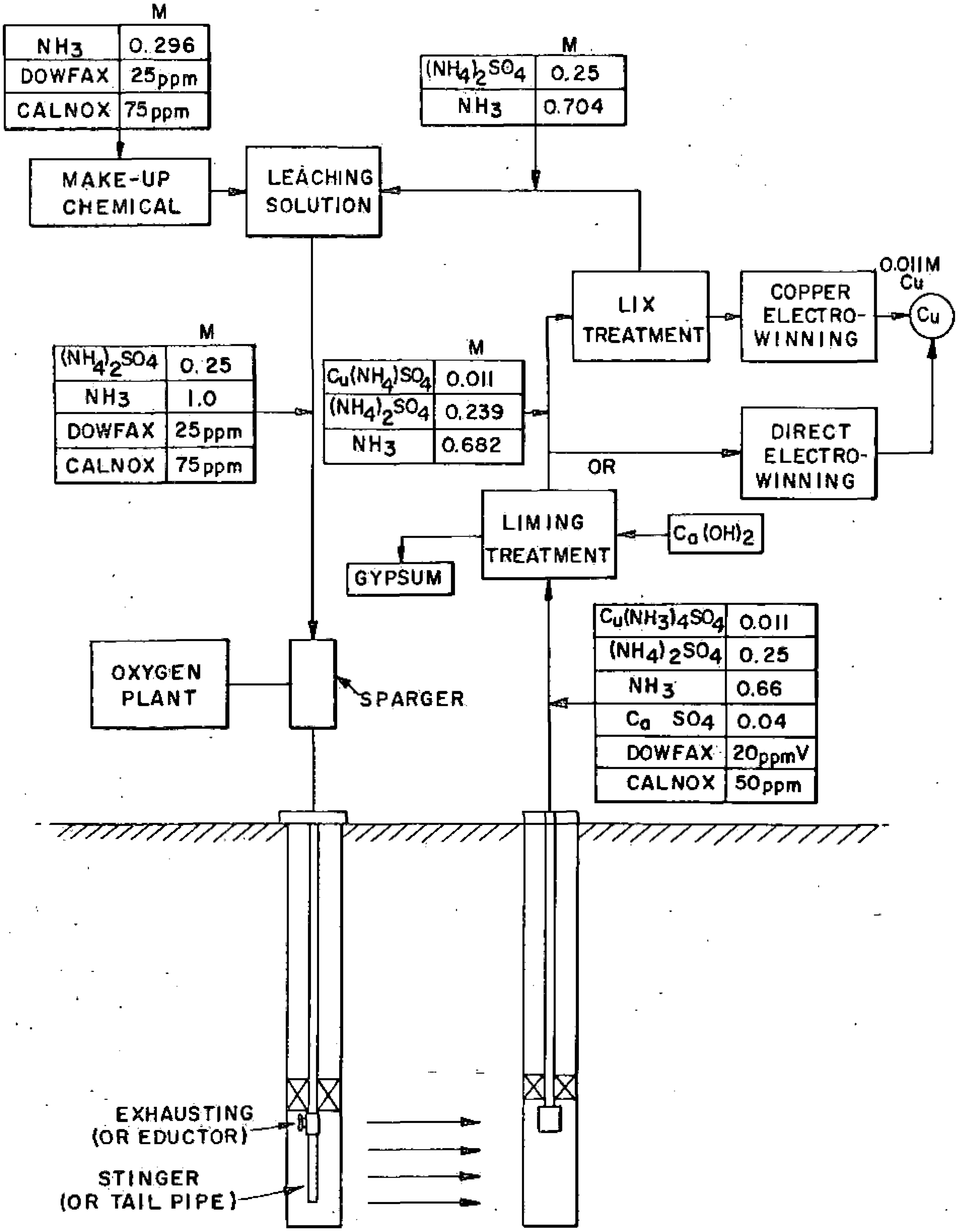


FIG. 8.


M
NH3 0.296
DOWFAX 25ppm
CALNOX 75ppm
M
(NH4)2SO4 0.25
NH3 0.704
MAKE-UP CHEMICAL
LEACHING SOLUTION
LIX TREATMENT
COPPER ELECTRO-WINNING
0.011M Cu
Cu
M
(NH4)2SO4 0.25
NH3 1.0
DOWFAX 25ppm
CALNOX 75ppm
M
Cu(NH4)SO4 0.011
(NH4)2SO4 0.239
NH3 0.682
DIRECT ELECTRO-WINNING
OR
LIMING TREATMENT
Ca(OH)2
GYPSUM
Cu(NH3)4SO4 0.011
(NH4)2SO4 0.25
NH3 0.66
Ca SO4 0.04
DOWFAX 20ppmV
CALNOX 50ppm
OXYGEN PLANT
SPARGER
EXHAUSTING (OR EDUCTOR)
STINGER (OR TAIL PIPE)

IN-SITU MINING METHOD AND APPARATUS

BACKGROUND OF THE INVENTION

With the world's known sources of high grade copper and nickel diminishing rapidly, great emphasis has been placed on discovering new sources of these metals. There is known to be located throughout various regions on the globe, large, deep lying deposits of copper in the form of low grade porphyry ores. A porphyry copper ore deposit is a copper deposit in which the copper-bearing minerals occur in disseminated grains and/or in veinlets through a large volume of rock. The term was introduced because some of the first large copper deposits that were mined in the western United States occurred in porphyritic granodiorite and quartz monzonite. Today, the term implies a large low-grade disseminated copper deposit in various host rocks such as schist, silicated limestone, and volcanic rocks; but, acid igneous intrusive rocks are usually in close association.

The deposits are typically large tonnage but low grade, having an average copper concentration of less than about 1 percent. Copper minerals found in these deposits usually are sulfides and most commonly are chalcopyrite. When such a deposit is of sufficiently high grade, and either outcrops on the surface or is sufficiently close to the surface, then the ore is mined by open pit methods and the copper minerals are separated from the gangue constituents by techniques such as flotation.

Deeply buried or very low grade copper porphyry deposits cannot be easily exploited. Recovering the copper values from such deposits presents many challenges. For example, conventional open pit mining is not available for such recovery for a number of reasons. First of all, the cost would be prohibitive. Secondly, because open pit mining scars the landscape, restrictions have been placed on the recovery of ores by such techniques.

It has been proposed to extract the copper from deeply buried porphyry deposits by in-situ leaching techniques. In-situ leaching is a well-known technique which has long been practiced; its origin can be traced as far back as the 15th century. With in-situ mining, a hole is drilled and a leach liquor is pumped down the hole into the ore containing the metal to be recovered. After the liquor has leached the metal values, the pregnant leach liquor is extracted to enable metal values to be recovered.

There are also massive sulfide deposits treatable by the present invention which are deep seated and contain discrete blebs of nickel sulfide, or copper sulfide or copper-nickel sulfide in association with iron sulfide. A representative list of minerals which can be treated to recover copper or nickel or both by the present invention includes: native copper, chalcocite, digenite, covellite, pentlandite, heazlewoodite, vaesite and violarite.

There are many prior art procedures for in-situ mining. Most of these procedures, however, involve rubblizing the ore which is to be leached by explosive methods.

The present invention involves leaching the copper in-situ (without rubblizing it) with a lixiviant containing very small oxygen bubbles admixed with an ammoniated leach liquor. The oxygen bubbles are produced by a sparger or mixing device. To be effective, the oxygen bubbles should be smaller than the fractures in the ore.

A two-phase lixiviant containing small sparger-produced oxygen bubbles is an important aspect of the invention. However, the broad concept of utilizing an ammoniated lixiviant containing small bubbles of oxygen to leach copper from ore formations in-situ is disclosed in U.S. Pat. No. 3,708,206 to Hard et al. However, prior to the present invention, a two-phase introduction of oxygen was unattractive for a number of reasons. Problems such as phase separation of oxygen prevented a two-phase system from being used efficiently. In attempting to bring a suitable dispersion of oxygen into a bore when employing an aqueous solution, numerous adverse conditions apply. For example, elaborate methods and/or equipment was thought to be necessary to obtain a stable dispersion of oxygen as a gas in an aqueous fluid.

Indeed, so severe were the problems associated with two-phase in-situ mining procedures that research in this area was discouraged. The problems associated with two-phase in-situ mining are severe because the dispersion of oxygen must be sufficiently well distributed and the bubbles of oxygen must be sufficiently small so that these may enter the pores or fracture apertures in the rock before phase disengagement can occur. Still further, the quantity of oxygen should be evenly distributed throughout an entire ore column which is being worked by the in-situ method. Prior to the present invention, from the standpoint of complexity, economics and utilization of oxygen these considerations have made it almost intolerable to use oxygen as a gas dispersed in a liquid.

At this point it should be noted that the system disclosed in Hard et al. U.S. Pat. No. 3,708,206 involves recovering metals from porous rock such as sandstone located at depths close to the surface (300–500 ft.). The present process on the other hand is directed toward recovering metals from deep, hard rock formations of low porosity.

With a section of core material taken from a leaching interval of a typical deep lying porphyry copper ore, the copper is found primarily within the fractures. The fractures from which the copper is leached may be very small in size. Indeed, with the process of the present invention, copper can be recovered from fractures that are only 30 microns to 300 microns in width.

When the present invention is practiced, it is not necessary to disturb the deposit by blasting. Indeed, it is believed that the present invention is the only practical process presently known in which copper can be leached economically from deep lying deposits by in-situ mining techniques without rubblizing the ore. That copper can be leached from deep-lying porphyry ores without disturbing the ore is remarkable when the nature of the ore being leached is considered.

Another significant advantage of the process of the present invention is that the copper can be mined economically from deep lying porphyry deposits without any significant environmental impact. For example, with the present invention, there are no subsidence problems. Furthermore, the only alteration on the land surface is the presence of a few buildings and pumps which can be removed after the copper and/or nickel has been mined.

SUMMARY OF THE INVENTION

Economical recovery of copper from deep lying porphyry deposits in accordance with the present invention is accomplished by the use of a two-phase lixiviant. The two-phase lixiviant includes an aqueous leach solution which carries finely divided bubbles of oxygen gas. In order for the two-phase system to be effective, the gas must remain dispersed as finely divided bubbles in liquid and the bubble size must be small enough to penetrate into the extremely small fractures of the porphyry deposits. In one preferred embodiment of the invention in order to introduce the finely divided gas bubbles into the liquid, the liquid phase is supplied to a plurality of porous tubes formed of sintered powdered metal while the gas is supplied under pressure about the tubes to cause the gas to penetrate into the interior of the tubes in the form of fine bubbles which are then wiped from the interior of the tube by the lixiviant passing therethrough. This mixing may be affected by a sparger located at the surface of the injection hole. The two-phase lixiviant can also be produced by mixing oxygen and liquid and maintaining a high linear velocity in the tubing which carries the solution to the leaching interval.

The two-phase lixiviant thus produced is passed down an injection hole to the leaching interval of a deep lying ore body located beneath a cemented and packed off portion of the injection hole. The two-phase lixiviant is injected into this hole through a venturi-type exhauster.

The exhauster unit has an extended ejection nozzle (stinger) which is downwardly directed and terminated within the injection interval. The exhauster and stinger prevent coalescence of the oxygen bubbles by enabling continuous vertical circulation of the lixiviant between the outlet of the injecting nozzle, which is located in the lower portion of the leaching interval, and an aspirator passage inlet which is located in the upper portion of the leaching interval.

The cooperative interaction between the sparger and exhauster yields an oxygenated lixiviant or leach liquor containing well dispersed minute oxygen bubbles. This unique two-phase lixiviant is able to effectively penetrate the fractures of the ore body and effect dissolution of the copper due to the minute bubble characteristics of the oxygen phase of the leach solution. The dissolved copper is removed through output holes and is recovered from the pregnant liquor by conventional technology.

Accordingly, an object of the present invention is to provide a new and improved method and apparatus for economically recovering metals such as copper or nickel from deep lying ores without resorting to open pit mining or extensive underground development.

Another object of the present invention is to provide a new improved method for in-situ mining of deep lying deposits of metal bearing ore that does not involve rubblizing the ore.

A further object of the present invention is to provide a process for recovering metals from deep lying deposits in a manner that is environmentally acceptable.

Another object of the present invention is to provide a method and apparatus for admixing very small oxygen bubbles into an ammoniated or acidic leach liquor to produce a two-phase lixiviant which is capable of penetrating the fractures of a deep lying ore deposit to oxidize the ore and thereby enable the metals in the ore to be leached in-situ.

Another object of the invention is to provide a process for recovering copper from deep lying porphyry ores by admixing very small oxygen bubbles into an ammoniated or acidic leach liquor to produce a two-phase lixiviant which is capable of penetrating the ore to leach copper in-situ.

BRIEF DESCRIPTION OF THE DRAWING

FIG. 1 is a perspective view of an assembly used in accordance with the present invention for admixing small oxygen bubbles into a liquid for injection into the leaching interval of an injection hole;

FIG. 2 is a view of a production hole for withdrawing pregnant leach liquor;

FIG. 3 is a cross-sectional view taken along line 3—3 of FIG. 1;

FIG. 4 is a cross-sectional view of the sparger of FIG. 1;

FIG. 5 is a diagram of a "five-spot" drilling pattern;

FIG. 6a is a diagram showing axial flow through horizontal fractures;

FIG. 6b is a diagram showing radial flow through vertical fractures;

FIG. 6c is a diagram showing axial flow through vertical fractures;

FIG. 7 is a diagram showing the flow pattern within the injection hole such as that shown in FIG. 1; and,

FIG. 8 is a diagram illustrating a process for in-situ mining in accordance with the present invention.

DESCRIPTION OF THE PREFERRED EMBODIMENTS

At the outset, the process and apparatus of the present invention is described in its broadest overall aspects with a more detailed description following.

In accordance with one important embodiment of the present invention, an ammoniacal lixiviant containing oxygen bubbles is employed to mine copper in-situ from ores lying thousands of feet below the ground.

In practicing the invention, very small oxygen bubbles are admixed with an ammoniated leach liquor to obtain a "two-phase" lixiviant. This mixing may be effected by a sparger **10** located at the surface of an injection hole **12**. The sparger includes a sintered metal tube **14** arrangement (See FIG. 4) for injecting oxygen bubbles into the ammoniated leach liquor. The oxygen bubbles are minute and smaller than the fractures, and/or interstices of the ore body **16**.

The two-phase lixiviant thus produced is passed down the tubing **15** of an injection hole **12** to the leaching interval of a deep lying ore body **16** located beneath a cased **17**, cemented **18** and packed-off portion **20** of the injection hole **12**. The two-phase lixiviant is injected into this zone through a venturi-type exhauster **22**. This exhauster unit has an extended ejecting nozzle **24** (stinger) which is downwardly directed and terminated near the bottom of the injection interval. The purpose of the exhauster **22** and the stinger **24** is to prevent coalescence of the oxygen bubbles before they enter the ore by enabling continuous vertical circulation of the lixiviant between the outlet **26** of the ejecting nozzle, which is located in a lower portion of the leaching interval, and an aspirator passage inlet **28**, which is located in an upper portion of the leaching interval.

The cooperative interaction between the sparger **10** and exhauster **22** yields an oxygenated lixiviant or leach liquor containing well dispersed, minute oxygen bubbles. This unique two-phase lixiviant is able to effectively penetrate the fractures of the ore body and effect dissolution of the copper due to the minute bubble characteristics of the oxygen phase of the leach solution.

As is shown in FIGS. 1 and 4, the apparatus for use in in-situ mining in accordance with one embodiment of the present invention includes a sparger 10. In its overall aspect, the sparger 10 includes a plurality of sintered metal tubes 14 which serve to convey ammoniated lixiviant 30 through a pressurized oxygen containing chamber 32. Pressurized oxygen 34 passes through the porous walls of the sintered tubes and enters the ammoniated lixiviant flow as is shown by arrows 36. The two-phase flow thus produced passes from the sparger down into the hole to an injection zone or leaching interval beneath a packer 20. The porosity of the sintered metal tubes is such as to provide an oxygen bubble size which is of the same order of magnitude as the fracture openings in the ore formation 16. A sparger which can be used in accordance with the present invention is disclosed in U.S. Pat. application Ser. No. 724,549 entitled "In-Situ Mining Method and Apparatus for Sparging Gas Bubbles" by Hsueh et al filed on even date herewith and commonly assigned, the teachings of which are incorporated herein by reference.

Referring to FIG. 4, a gas sparging unit 10, constructed in accordance with the present invention, is illustrated in detail and consists of a generally cylindrical casing 38 formed from a plurality of annular members which are secured together to form an elongated cylindrical sleeve. The sleeve is closed at one end in any convenient manner, as for example by a flanged cap 40 or the like, and has a first partition plate 42 welded therein in order to define a first chamber 44 within the sleeve. The partition plate 42 is a generally circular member. A second similar partition member 46 is located adjacent the opposite end 48 of the sleeve (which defines an outlet end for the sparging unit) so as to define a second chamber 32 within the sparging unit.

A plurality of hollow elongated tubes 14 are mounted in the partitions 42 and 46 with one end 50 of each of the tubes being located in communication with the interior of chamber 44. The other ends 52 of the tubes 14 extend through the partition 46, adjacent the outlet opening 48 of the sleeve. These tubes are preferably formed of a sintered metal powder porous material having micro pores of a diameter of, for example, 50 microns, to permit small gas bubbles to be diffused therethrough. A general useful range of pore diameter is from 2 microns to 1,000 microns. A preferred range is from 10 to 100 microns. Such tubes may be formed of stainless steel powder or similar porous corrosion resistant material.

The porous sintered tubes are commercially available from Newmet, Inc., Pequabuck, Connecticut. The size of the pores in a tube is controlled by selecting proper particle size distribution of stainless steel powder and by sintering at a temperature slightly below the melting point of the stainless steel powder.

The number of such tubes used in a particular gas sparger unit can be varied as desired in accordance with the amount of gas bubbles required to be introduced into the lixiviant solution and the type of ore formation being treated as described hereinafter.

The first chamber 44 of the gas sparging unit 10 includes an inlet opening 54 through which an ammoniated lixiviant under pressure, is supplied from a source as is shown by arrow 30. Any convenient pumping apparatus can be utilized to supply the lixiviant from the source to the first chamber 44 of the gas sparging unit.

The second chamber 32 formed in the sleeve 38, includes an inlet opening 56. The gas to be introduced into the lixiviant solution is supplied through the inlet 56 under pressure from a source as is shown by arrow 34, in any convenient manner.

In accordance with the present invention, the gas supplied will be an oxidizing gas such as air, oxygen, oxygen enriched air, or a combination of oxygen and some catalyst, such as for example, SO_2, SO_3 or NO_2 as an acid forming gas. By supplying gas under pressure in this manner to the chamber 32, the gas is forced to penetrate through the porous tubes 14 in order to form small bubbles on the interior surfaces of the tubes. Since the upper ends 50 of the tubes are in communication with the chamber 44, the liquid lixiviant supplied to that chamber will flow through the tubes into contact with the small bubbles formed therein. The movement of the lixiviant through the tubes towards the discharge ends 52 thereof will wipe the bubbles from the interior surfaces of the tube and cause the bubbles to be intermixed within the lixiviant.

It has been found that the greater the velocity at which the barren lixiviant moves through the tubes, the smaller the bubbles introduced into the lixiviant will be. Generally, the proper velocity of lixiviant in a tube can be calculated from the amount and pressure of introduced lixiviant. Fluid velocity ranges from 2 ft./sec. to 50 ft./sec. have been found satisfactory when porous tubes of ¼ inch inside diameter are used. The size of the bubbles can also be varied or controlled by using porous tubes of varying diameters at a fixed flow. In this connection tubes having inside diameters of between about ⅛ inch and ½ inch have been found satisfactory when the tubes have pores with diameters ranging between 10 to 100 microns and with lixiviant velocities between 2 ft/sec. to 50 ft/sec.

The lixiviant solution thus mixed with the fine gas bubbles passes through the discharge openings 52 of the tubes to the discharge end 48 of the gas sparger unit.

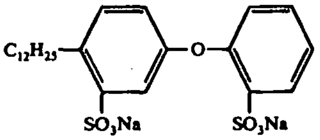
In the embodiment of the invention shown in FIGS. 1 and 4, the gas sparging unit is adapted to be used above the ground. Accordingly, the end 58 may be connected in any convenient manner, as for example by an elbow joint 21, to the well head 23 and tubing string 15 which extends down the bore hole. In this embodiment, lixiviant mixed with gas bubbles passes down the tubing string 15 to the ore formation 16 to treat the metal values in the ore formation and create a pregnant liquor.

At this point, it should be noted that the preferred embodiments of the invention involves a sparger unit which is located on the surface of the ground. The invention, however, is not intended to be limited to this mode of operation. In fact, in some situations it may be desirable to place the sparger unit down the bore hole. Indeed, the patent application Ser. No. 742,549 referred to above entitled In-Situ Mining Method and Apparatus for Sparging Gas Bubbles, discloses a sparger unit which can be placed within the bore hole.

It should also be noted that the process of the present invention is not intended to be limited to the production of oxygen bubbles in a leach liquor by forcing the oxygen gas through the porous tubes in the manner discussed above. There are many methods for producing a two-phase lixiviant. In this regard, any mixer that is capable of mixing oxygen bubbles into a leach liquor and produce a two-phase lixiviant from which the oxygen bubbles can be forced into the fractures of the ore can be used in the present invention. Merely by way of example, the sparger or mixer can be a simple pipe with a T junction in which oxygen gas is supplied on one side of the T and the leach liquor is supplied to the other side of the T. The gas and liquid are mixed at the junction.

In any embodiment of the invention, the velocity and diameter of the tubing string are important. It will be apparent to those skilled in this art that if oxygen and liquid are passed down a narrow tube at high velocity, a two-phase lixiviant will be produced. Thus, an important concept to consider in conjunction with any mixing device or sparger is that when the sparger is combined with a tubing string of a particular diameter, with a gas and liquid being subjected to a particular high velocity down the tube string, turbulence would result which would homogenize the gas and the liquid to produce a two-phase lixiviant.

Another factor which influences the two-phase lixiviant is the including in the leach liquor of a surfactant. In accordance with the invention set forth in U.S. Pat. No. 4,045,084 entitled Improvements in In-Situ Mining of Copper and Nickel, filed on even date herewith, the teachings of which are incorporated herein by reference, it has been discovered that many surfactants, when added to the liquid phase prior to the introduction of the gas which form the bubbles greatly enhances the stability of bubbles formed. The use of a surfactant enables smaller bubbles to be formed and tends to maintain the size of the smaller bubbles for a substantial time during the mining procedure. In addition to the surfactant, the lixiviant may include an agent (Calnox ®) to stabilize calcium sulfate which results from a combination of calcium ions from certain orebody minerals and sulfate from chalcopyrite oxidation. A surfactant sold under the tradename Dowfax 2A1 by the Dow Chemical Co. has been found satisfactory. Dowfax 2A1 is the sodium salt of dodecylated oxydibenzene disulphonate having the formula:



Immediately below the tubing string 15 is the exhauster (or eductor) 22. In one preferred embodiment of the invention the exhauster 22 is an aspirator or suction device that operates on well-known principles. Such exhausters (aspirators) are commercially available. Indeed, the exhauster 22 shown in the drawing is a Penberthy Series 180A exhauster which is manufactured by Penberthy-Division of Houdaille Industries, Inc., Prophetstown, Illinois 61277. This exhauster is available through plumbing contractors.

During the mining operation in accordance with the present invention, the two-phase lixiviant represented by arrow 36 (see FIG. 3) enters inlet 64 through nozzle 66 and travels past a whirler 65 into a nozzle 67 in suction chamber 68. Nozzle 67 converts the pressure head of this motive fluid into a high velocity stream which passes from the discharge side of the inlet nozzle.

Pumping action or suction begins when the vapor, liquid or gas in the suction chamber 68 is entrained by the jet stream emerging from the nozzle 67. This jet stream lowers the pressure in the suction chamber 68. The resulting action causes the fluid in the suction system to flow to the delivery jet 71 as is shown by arrow 73. The foregoing action creates suction through suction nozzle 70. The entrained material 73 from the suction system mixes with the motive fluid represented by arrow 75 and acquires a part of its energy in a diffuser 72. The velocity of the mixture leaving delivery jet 71 represented by arrow 76 is reconverted to a pressure greater than the suction pressure but lower than the motive pressure.

At this point it should be noted that a Penberthy steam, liquid, or air-operated jet pump, used for pumping, handling slurries or granular solids is called an ejector. However, it performs the same function as an aspirator. When a liquid-operated jet pump is used for pumping gases or vapor it is often called an exhauster. Regardless of the name, the principles of operation remain the same. Such pumps have many inherent advantages: they have no moving parts; therefore, there is nothing to break or wear; and, no lubrication is required.

The Penberthy series 183A exhauster is made of 316 stainless steel and has a nozzle size of 0.157 inch. To minimize the possibility of plug up, a course filter (not shown) is installed before the exhauster.

For a Penberthy series 183A exhauster with an injection rate (motive fluid) of 5 GPM, the suction fluid (entrained fluid) rate is 8.5 GPM and total discharge from the stinger 24 is 13.5 GPM. The pressure drop across the exhauster is 60 psi, the friction loss in stinger 24 is estimated to be 0.06 psi/ft which necessitates an additional 60–65 psi pressure increase at the surface sparger 10.

It is most advantageous to place the exhauster at the upper most point in the leaching interval where the gas may accumulate. Thus, in a preferred embodiment of the invention it is placed 8 inches or less below the packer 20. With this placement, the pressure differential within suction chamber 68 will draw the gas phase into suction nozzle 70 and deplete the gas pocket.

At this point, it should be noted that the purpose of the exhauster 22 is to prevent the formation of gas pockets beneath the packer. There are many devices which can perform this function, indeed an electrical pump can be used. It should be apparent, however, that a device such as exhauster 22 is preferred. Without any device, a gas pocket would form due to the disengagement of the oxygen bubbles from the two-phase lixiviant.

Of course, the selection of a particular exhauster is controlled by the flow rate of the lixiviant. The Penberthy series 183A exhauster is applied to systems in which there is a maximum flow rate of 10 gallons per minute. When the flow rate is increased, a larger exhauster is necessary. The selection of a suitable exhauster for a commercial mining operation is well within the skill of those in this art.

Immediately below the exhauster is a tail pipe or stinger 24 which extends to a point near the bottom of the leaching interval.

A factor which affects the efficiency of the process is the extent of the dispersion of oxygen in the leach liquor. It has been found that a turbulent flow can be used to maintain a uniform dispersion of gas and liquor from the surface to the bottom of the injection interval. To obtain copper loadings in excess of ½ gpl using oxygen as a lixiviant, a two-phase mixture must be uniformly injected into the ground. When gas and liquid are mixed at the surface, a uniform dispersion must be maintained as the mixture is transported downhole to insure uniform injection of both phases into the rock or ore.

There are at least three parameters which can be controlled to effect a high dispersion of oxygen in the liquid. These include the velocity of the fluid, the diameter of the tubing string and stinger and the point at which the mixture is injected in the injection interval. It is preferred to utilize a tubing string and stinger having an inside diameter of 3 inches or less. With tubing strings of this diameter it is advantageous to maintain the flow rate of the two-phase lixiviant at a velocity of one foot per second or greater. It is also advantageous to inject the lixiviant at the bottom of the injection of leaching interval.

When the leaching interval extends to the bottom of the bore, the stinger is positioned so the end of the stinger is about two feet from the bottom of the bore. The two-foot spacing is used to provide for the possibility of particulates and debris depositing beneath the stinger. At this point, it should be noted in the commercial operation, the length of the leaching interval can be between 1,000 and 3,000 feet in length. Of course, leaching intervals in the order of 200 feet are also possible. The length of the leaching interval is controlled by the nature of the ore formation being mined. Factors to be considered are concentration, ore grade, depth, etc. The size of the eductor or exhauster is influenced by the pressure drop in the tail pipe or stinger which is influenced by the length and the diameter of the tail pipe.

Although the drilling of injection holes shown in FIG. 1 is conventional, a brief description of the procedures for constructing injection holes appears below.

Prior to drilling an injection hole, a drill pad must be constructed for the drilling site. The size of the pad will depend on the size and type of the rig to be employed and the number of holes to be drilled from the pad. Many rotary-drill rigs will require a pad 200 ft. square.

To bore a large diameter, (in excess of 5 inches) hole into the deposit to a depth of 5000–6000 feet requires a moderate size rotary drilling rig. These rigs are commonly used in the exploration and exploitation of oil reserves.

The casing is a tubular form of steel or fiber glass reinforced plastic that is screwed or welded together as it is lowered into the hole to a desired depth. The function of the casing is to control fluid movement. One or more strings of casing of different diameters may be required during the drilling or completion of the hole. The conductor casing string (not shown) is the largest diameter string used in the hole and is required to control erosion of the soil at the surface by the return flow of drilling fluid. So called surface casing 78 has the next largest diameter. Surface casing 78 is fitted inside the conductor casing and is used to isolate the near-surface formation to protect fresh water zones, if any, and prevent weathered rock from falling into the hole during subsequent operation. The smallest diameter string of casing, the long string 17 is set above or through the injection or production interval. This string of casing is placed within the surface casing. If the long string 17 is set above the leaching interval, the hole is said to have an open-hole completion. Such a completion is shown in FIG. 1. If the long string casing is set through the interval, the casing must be perforated to gain access to the formation. This results in a perforated casing completion. Perforated casing completion may be used in the present invention; but, open hole completion is preferred.

Each string of casing is cemented into the hole using known techniques. Cementing is necessary to bond the casing firmly to the rock to prevent fluid from moving up or down the annulus behind the casing and to provide support for any subsequent casing string to be run into the hole. The cement may also protect the casing from corrosive fluids.

After the hole has been drilled, casing landed and cemented, additional equipment is placed in the hole. It is preferred to install injection equipment as follows. A tail pipe 24 or stinger long enough to extend from the top to near the bottom of the injection interval is run first. Attached to the top of the tail pipe is the exhauster 22. The exhauster 22 is attached onto the bottom of a packer 20. The packer 20 is screwed onto the bottom of the tubing string 15 which is hung from the well head 23.

As shown in the drawing, the tubing string 15 is within the casing string 17 and the casing string 17 is surrounded by a cement wall 18. The packer 20 is between the tubing string 15 and the casing string 17. The purpose of the packer 20 is to prevent the lixiviant from rising in the annular space between the tubing string 15 and the casing string 17. The tubing string 15 itself is formed from sections of fiberglass or other tubing of a single diameter which are screwed together. In a commercial operation tubing string inside diameter may be between 2½ and 3½ inches. The wall thickness is approximately ½ inch.

The packer 20 is a standard instrument used in the oil industry which is composed of central mandrel with an expandable rubber element which can be expanded either hydraulically or mechanically. Once positioned, the packer 20 is expanded so that the sealing element engages the inside wall of the casing 17.

The seal effected by the packer prevents subsequently injected fluids from rising up the annulus between the tubing and the casing. This forces all injected fluids to flow into the injection interval.

It is preferred to withdraw the pregnant metal bearing liquor from production holes (see FIG. 2) that are separated from the injection holes. Production holes are drilled in the same manner as injection holes.

As is shown in FIG. 2, an electrical, submersible pump 80 is lowered into the hole by use of a reinforced power cable 82. Suitable well-head equipment 23 is installed to control the movement of produced fluid and provide a suitable seal where the power cable 82 exits. After the power cable is energized, the pump can be activated and fluid pumped to the surface.

A preferred surface layout is a so-called 5-spot pattern which is shown in FIG. 5 of the drawing. In FIG. 5, a dot (.) indicates the location of an injection hole, a circle (o) indicates the location of a production hole, and (d) indicates the distance between injection and production holes. For environmental reasons there are no injection holes on the perimeter of the layout. Further details on the significance of the 5-spot pattern appear below.

In accordance with one important embodiment of the present invention, copper is leached from a sulfide deposit such as chalcopyrite with a two-phase lixiviant. The two-phase lixiviant includes ammonia and oxygen. During the leaching, the following reaction is believed to occur.

$$\begin{aligned}
&Cu^{+2}Fe^{+2}S_2 + 4.25\ O_2 + 1.5\ H_2O + 6NH_3 = \\
&\quad Cu^{+2}(NH_3)_4SO_4 + Fe^{+3}OOH + (NH_4)_2SO_4 \\
&\quad Fe^{+2}S_2 + 3.75\ O_2 + 2.5\ H_2O + 4NH_3 = \\
&\quad Fe^{+3}OOH + 2(NH_4)_2SO_4
\end{aligned}$$

Of course, nickel, cobalt and molybdenum, if present as sulfides in the ore will also be leached in accordance with known chemistry. At this point, it should be noted that the primary purpose of the oxygen is to break the chemical bonds holding the copper in the chalcopyrite by oxidizing the sulfide and iron component. Once the chalcopyrite is oxidized, the aqueous ammonia is able to dissolve the copper values. It makes no difference whether or not the copper is oxidized. Indeed it is believed that the $CuFeS_2$ contains copper as cupric copper and iron as ferrous iron. Thus, during oxidation in accordance with the foregoing reaction, the oxidation state of copper remains unchanged while the iron is oxidized from Fe^{+2} to Fe^{+3}. Of course, if Cu^+ copper is present in the ore, it would also be leached by the lixiviant. Since both forms of copper ions are leachable, it is not necessary to oxidize cuprous ions to cupric ions in order to leach copper.

A sufficient excess of aqueous ammonia is used to keep the pregnant solution alkaline. Under these conditions, dissolution of gangue materials is negligible and the pregnant solution contains virtually only ammonia, ammonium sulfate, and cupric ammine sulfate.

The foregoing system in which oxygen is admixed with an ammoniacal leach liquor is referred to as an oxygen-ammonia lixiviant. It is to be understood, however, that other two-phase lixiviants can be used in accordance with the present invention. The oxygen-ammonia lixiviant is preferred where there is a lot of acid-consuming minerals in the ore body. However, a representative example of another two-phase system that can be used to leach copper and nickel from a sulfate deposit includes the so-called oxygen-water lixiviant.

The chemistry for the oxygen-water lixiviant appears below.

$$CuFeS_2 + 4.25\,O_2 + 1.5\,H_2O \rightarrow CuSO_4 + FeOOH + H_2SO_4$$

$$FeS_2 + 3.75\,O_2 + 2.5\,H_2O \rightarrow FeOOH + 2H_2SO_4$$

When an oxygen-water lixiviant is used, cupric sulfate and sulfuric acid are generated in the leaching process or added on the surface. The cupric sulfate and sulfuric acid dissolve gangue metal oxides (Fe, Mg, Al, Ca, etc.) as sulfates. Much of the iron and aluminum precipitates in-situ as jarosite and alunite. In the surface plant, copper is extracted, and if necessary, the pH is adjusted to the desired level. The resulting leach solution is reinjected together with make-up oxygen. Another name for the oxygen-water lixiviant is the oxygen-acid lixiviant.

The process of the present invention is used to great advantage for deep-lying ore bodies, that is, ore bodies located at a depth of 1,000 feet or more below the surface. Although, the surface is normally a land surface, it should be noted that there is no reason why this process cannot be used to recover copper from deposits located below the bottom of the continental shelf or a lake bed. Thus, when reference is made to the depth of deposit below the surface, the surface can either be land surface or the surface of a body of water underneath which the deposit is located. The real significance of the fact that the process is used to treat deep-lying deposits is that in order for the process to be used practically, the lixiviant must be injected into the ore body under a head of pressure which is just below the fracturing pressure of the ore body. The process of the present invention takes advantage of the fact that the depth at which the ore body is located provides a lid for the pressurized lixiviant. It is preferred to locate the injection interval below the water table because it acts as the lid for this pressure head. In short, the process of the present invention could not be used to great advantage to leach ore bodies that are located close to the surface, that is 200 feet or less. The correct maximum down hole injection pressure of the lixiviant is limited by the fracture pressure at the top of the leaching interval.

The two-phase flow injection system for the commercial operation is divided into two modes of operation; downhole, and surface sparging. In the former case the surface pressures of gas and liquid must be controlled separately, because the pressure difference between the surface and the top of the leaching interval is related to the individual phase densities. In the surface sparging mode the surface pressures of gas and liquid are the same, and must exceed a minimum level to insure that a stable gas-liquid dispersion is transported downhole. In both cases, a second control point requires that the *pressure at the top of the injection interval be less than* the rock fracturing pressure.

In downhole sparging, the liquid surface pressure is equal to the pressure at the top of the injection interval, less the hydrostatic head from the surface to the top of the injection interval, plus friction drops through the sparger, eductor, and tubing string. At commercial flow rates, the friction drop in the tubing string for the liquid is less than 10% of the hydrostatic gradient for tubing diameters greater than 2.5 inches. The surface pressure for the liquid, ^{P}SL, is approximated as:

$$p_{SL} = (A_f - 0.433)\,(D-H) + \Delta p_{fe} + \Delta p_{fs} \qquad (1)$$

A_f = fracture gradient, $0.7 \leqq A_f \leq 1$, psi/ft

Δp_{fE} = pressure drop across eductor, psi

Δp_{fS} = pressure drop across sparger, psi

D = distance from surface to bottom of leaching interval

H = distance from bottom of leaching interval to bottom of packer

$D-H$ = distance from surface to bottom of packer

Example

$A_f = 0.7$ psi/ft

$D = 5000$ feet

$H = 2500$ feet

$\Delta p_{fe} = \Delta p_{fs} = 200$ psi

$p_{SL} = 0.267 \times 2500 + 400 = 1067.5$ psig (2)

If a downhole pressure measurement is available, and the surface pressure of the liquid should be adjusted so that the pressure at depth (D−H) does not exceed A_f (D−H), in the example case 1750 psi.

The gas surface pressure, Pgs, is equal to the pressure drop at the top of the injection interval, less the hydrostatic variation, plus the friction drops through the eductor and sparger. Since the gas is compressible, the hydrostatic pressure variation must be corrected for pressure and temperature variations in the tubing string.

$$P_g = [A_f(D - H) + p_{fe} + p_{fs})\left[\frac{T_g}{T_g + B(D - H)}\right]^{0.625} \qquad (3)$$

T_{gs} = surface temperature of gas in degree of Rankine, ° R

B = geothermal gradient, ° R per foot — 0.0325

Example

$A_f = 0.7$ psi/foot, $D = 5000$, $H = 2500$ ft, $T_{gs} = 535°$ R

$$P_{gs} = [(.7)(2500) + 200 + 200]\left[\frac{535}{535 + 2500(.0325}\right]^{0.625} \qquad (4)$$

$$P_{gs} = (2150)\,(0.915) = 1968 \text{ psi} \qquad (5)$$

In surface sparging, the gas and liquid surface pressures are the same and must be controlled above a minimum level such that the gas volume fraction in the tubing string does not exceed a critical value between 20% and 25%. The minimum surface pressure, P_{SM}, is also related to the copper loading and efficiency of oxygen usage, assuming sulfate as the oxidation product.

$$P_{SM} \geqq (23.7)\,(1-f_{gc}/f_{gc})\,(\mathrm{Cu}/E), \text{ in psi} \qquad (6)$$

Cu = copper loading, gpl

E = overall efficiency of oxygen utilization

f_{gc} = critical gas volume fraction associated with bubbly flow, $0.2 \leqq f_{gc} \leqq 0.25$

Example

$$f_{gc} = 0.25,\ \mathrm{Cu} = 6.\ gpl,\ E = 0.40 \qquad (7)$$

$$P_{SM} \geqq 1067 \text{ psi}$$

The constant 23.7 is good for chalcopyrite only. In the case of Pentlandite, $NiFe_2S_3$, a main nickel sulfide ore, the constant would be 39.2 and for the case of molybdenite, MoS_2, a major molybdenum ore, the constant would be 16.6

In general, for a sulfide ore body having the formula of MFe_yS_x where the oxidized product are M^{+z} and sulfate, the constant can be calculated from

$$P_{SM} \geqq \left(\frac{354.7\,[1.5x + 0.75y + 0.25z]}{MW}\right)\left(\frac{1 - f_{gc}}{f_{gc}}\right)(M/E.)$$

where M is the metal loading in gpl of the ore metal values to be recovered, E is the overall efficiency of oxygen utilization, MW is the molecular weight of the metal ion to be recovered, z is the valance of the metal ion to be recovered in solution, and y and x are the subscripts for Fe and S, respectively, in the sulfide mineral structure MFe_yS_x. For example during leaching -

$$MFe_yS_x + \ldots \rightarrow M^{+z} + SO_4^{=} + FeOOH + \ldots$$

so for;

CuFeS: $x = 1, y = 1, z = 2$

$NiFe_2S_3$: $x = 3, y = 2, z = 2$

MoS_2: $x = 2, y = 0, z = 6$

To insure that gas does not segregate and rise to the surface as it is being transported downhole the tubing size must be maintained below some maximum size, d_{TM}.

$$d_{TM} \leqq (0.69)\,(Q_L)^{.4} \text{ for water} \qquad (8)$$

d_{TM} = tubing diameter, inches

Q_L = flow rate of liquid in gallons per minutes

For example at 190 gpm the maximum tubing size that can be used is 5.6 inches. Since the inside diameter of the cased portion of the hole above the injection interval is 6 inches, *all tubing sizes that can be used* are less than the critical diameter. When 25 ppm of the surfactant Dowfax is used in the lixiviant, d_{TM} can be increased by a factor of 1.5.

The surface pressure can be controlled by: developing a friction drop in the tubing string, using a downhole choke or developing a large friction drop across the eductor. It is not necessary to have the diameter of stinger tubing **24** (d_2) the same as the diameter of tubing string **18** which runs from the surface to the eductor (d_1). The eductor operates at maximum efficiency when the friction drop in the stinger is a fraction of eductor suction, 40 psi. The minimum tubing diameter associated with a friction drop of 4 psi namely 1/10 oz. suction pressure is given by (9).

$$d_2 > (0.069)\,(H)^{.2}\,(Q_L)^{.4} \qquad (9)$$

Example

$H = 2500$ feet, $Q_L = 380$ gpm w/double recirculation

$d_2 > 3.6$ inches

Thus, if a 3.6 inch I.D. stinger **24** is used with a total circulation of 380 gpm in a 2500 foot interval less than a 4 psi friction drop will result, but stable flow will be insured as d_2 is less than the d_{TM} of 5.6 inches computed from (8) with 190 gpm.

The surface pressure can be calculated once the following process parameters are fixed:

1. The liquid flow rate, Q_L as gallons per minutes, gpm.
2. The gas flow rate Q_G, as standard cubic feet per minutes, SCFM.
3. The formation permeability, K as millidarcy, md.
4. The distance from the surface to the top of the injection interval, (D−H) as feet.
5. The injection interval, H as feet.
6. The tubing inside diameter, d as inches.

The surface pressure is related to the above parameters in the following manner:

$$p_S = C_1 \frac{Q_L^{\,a}\,Q_g^{\,b}}{(K)^c\,(H)^d\,(D - H)^e\,d_1^{\,f}} \qquad (10)$$

The constants and ranges of the independent variables are listed in Table I and II. If p_S as calculated from (10) is less than the value of p_{SM} as calculated from (6), either a downhole choke must be used to increase p_S or the tubing string diameter d_1 decreased.

Table I

The Range of Parameters that Equation (10) is Applicable

Q_L, gpm	Q_g, SCFM	K(md)	(D-H) ft.	H, feet	d_1, inches
40–240	120–360	0.6–8.4	2500–5000	875–3205	1.6–4.0

Table II

Range of Constants

C_1	a	b	c	d	e	f
31,031	0.206	0.673	0.661	0.657	0.316	0.623
25,5076	0.148	0.654	0.707	0.616	0.564	0.385

Prior to the present invention, the only deposits from which metal values were recovered by a two-phase lixiviant in-situ were sandstones or rubblized deposits. Indeed, the Hard et al patent discussed above is directed to recovering uranium from sandstone or rubblized deposits located about 200 feet below the surface of the land. The present invention, on the other hand, is directed to the recovery of metal values from rock or ore that has a permeability such that those skilled in this art would have been discouraged from attempting to recover metal values therefrom. For example, the permeability of a typical uranium containing sandstone is on the order of 100 to 1000 md. On the other hand the present invention is directed to recovering metal values from porphyry ores which have permeabilities of 50 md or lower. Thus, the present invention is applicable to treating hard rock located at depths of 1000 ft. or greater, which rock has a permeability of 50 md or less.

Another parameter which is conveniently dealt with by the present process is the leaching temperature. In order for the lixiviant to be able to extract metal values from the hard rock, the temperature of the lixiviant should be 40° C. or greater. If the lixiviant had to be heated to this temperature, that fact would increase processing cost. However, because the present invention is directed to recovering metal values from deep-lying deposits, the geothermal properties of the earth are used to heat the lixiviant to the required temperature. It is known that the thermal gradient is approximately 2 ½° F. per 100 feet in areas such as Safford, Arizona. Thus, at 1,000 feet below the surface, the temperature of the two-phase lixiviant would be 25° F. above the ambient temperature at the surface. Accordingly, an important aspect of the invention is to actually leach the metal values with a lixiviant that is maintained at a temperature of 40° C. In the present process this is accomplished without the necessity of any means for heating the lixiviant.

The lixiviant can be forced through the ground in either linear or radial flow (see FIGS. 6A, 6B and 6C). Linear flow is obtained when the surface area normal to flow is constant between equipotential surfaces, that is, the pressure gradient is uniform between the injection and withdrawal points. In radial flow, the pressure gradient is inversely proportional to the distance from the point of injection.

The fluid flow analysis that follows is based on radial flow in a five-spot pattern (see FIG. 5). A vertical view of the hole is shown in FIG. 7. The hole is drilled to a total depth, D, and fluid is injected over some interval, T. In the production hole, a downhole pump, air-lift or swab is used to reduce the pressure at height, T, to the level at which the production and injection rates are comparable.

The flow rate for each hole in a five-spot pattern is calculated from equation (11) when the permeability, fluid viscosity, pressure drop, injection interval, and well spacing are specified. A consistent set of units must be used.

$$Q = \frac{\pi K T \Delta p}{\mu (\ln d/R_w - 0.619)} \qquad (11)$$

If Q is expressed in gpm, K is in md, μ in centipoise, Δp in psi, and T in feet, equation (11) becomes:

$$Q = 1.05 \times 10^{-4} \frac{kT \Delta p_T}{\mu} \; \frac{1}{\ln d/R_w - 0.619} \qquad (12)$$

Δp corresponds to the pressure drop between the injection and production holes, the maximum injection pressure is equal to the fracturing pressure. When the production hole is operated by drawing down the pressure at the top of the interval to atmospheric pressure, the maximum pressure drop is obtained.

For example, when:

$K = 3$ md

$\Delta p_T = 700$ psi

$\mu = 0.5$ centipoise

$d = 180$ feet

$R_w = 0.25$ feet

$T = 4000$ feet

then

Q = 312 gpm

An injectivity test is used to measure the pressure drop that is required to inject fluid at a fixed rate into the deposit. Equation (13) is used to compute the deposit permeability.

$$K = (4760\, Q\, \mu \ln R_e/R_w)/(T\Delta p) \qquad (13)$$

Δp corresponds to the pressure drop between the top of the injection interval and the fluid in the deposit. When the water table is at ground level, the pressure drop is equal to the surface injection pressure. R_e is the drainage radius, i.e., the distance from the injection well at which the fluid pressure is equal to the hydrostatic pressure at depth, L. The exact location of R_e is ambiguous. In a pressure injection test, less than two hours are required to obtain steady-state conditions. In this period of time, the reservoir pressure will not change by more than 10% at distance 100 feet away from the injection hole. The value of ln (R_e/R_w) is approximately equal to six. Equation 13 becomes:

$$K = (28{,}600)\,(Q\mu/T\Delta p) \qquad (14)$$

The viscosity of a fluid is a function of temperature. When the fluid is a liquid, the viscosity decreases as the temperature increases, thus the flow rate will increase at fixed pressure drop and permeability as the temperature increases. The converse is true of gas flow, because the viscosity of a gas increases with a rise in temperature. Table III lists values of the viscosity of water between 70° F. and 200° F.

Table III

Viscosity of Water as a Function of Temperature

Temperature, ° F	Viscosity, centipoise
70	1.00
100	0.75
140	0.50
200	0.30

An injectivity test performed in the bottom 70 feet of hole DDH-147 at Kennecott Copper Corporation's mine at Safford, Arizona gives the following results:

$Q = 15$ gpm

$T = 70$ feet

$\Delta p = 783$ psi

$\mu = 0.3$ centipoise

The permeability is computed from Equation (14).

$D = 28{,}600\ (15 \times 0.3/70 \times 783) = 2.4$ md

CONDITIONS SELECTED FOR BASE CASE

The base calculations assumed that fluid is injected over a 2,500 foot interval, with the top of the injection interval 2,500 feet below the surface. The maximum injection pressure is 1750 psi when the fracture gradient is taken as 0.7 psi per foot of depth. The maximum pressure drop between the injection and production wells in the five-spot pattern is 1750 psi when the injection well is drawn down to atmospheric pressure at the 2500 foot level. The flow rate is computed from Equation (12) for

d = 180 foot well spacing

$R_w = 0.25$ feet

$K = 2.4$ md

$\Delta p_T = 1750$ psi

$T = 2500$ feet

$\mu = 0.5$ centipoise

$Q = (1.05\times10^{-4})\ (2.4\times2500\times1750/0.5)\ (1/5.96) =$ 370 gpm

The flow rate per well computes to be 370 gallons per minute, which is equivalent to 532,000 gallons per day. The base case study used 400,000 gallons per day as a conservative estimate.

EXAMPLE I

On May 29, 1975, an ammoniacal sulfate leaching test was carried out at the Kennecott Copper Corporation in-situ mine in Safford, Arizona. The injection hole was equipped in accordance with the procedure outlined above and shown in FIG. 1 of the drawing. The various material balances are shown in FIG. 8 of the drawing. The main copper mineral deposit was chalcopyrite. The average grade of copper was 0.45% and the porosity of the ore body was 3%.

At the start of the in-situ mining operation, the effluent copper concentration is diluted with the deposit water that is stored in the pores of the rock. It is expected that after a volume of lixiviant equal to the volume of deposit water stored in the rock between holes is pumped, i.e., one pore volume, the copper concentration will attain the design level. A similar dilution will be obtained at the end of the mining venture in order to recover the copper that is in solution in the pores of the rock.

When the reaction between oxidant and chalcopyrite is rapid, all of the oxidant is consumed in one pass of the fluid through the deposit. When the reaction is slow, oxidant remains in the lixiviant at the production hole and the effluent copper concentration will decrease.

The composition of the lixiviant was 1 M NH_3, 0.25M $(NH_4)_2SO_4$ with 25 ppmV Dowfax ®surfactant and 75 ppm Calnox ® additive. The solution was injected into a hole (ID: 149A) at a rate of 10 gallons per minute and mixed with 12 SCFM (standard cubic feet per minute) of gaseous oxygen. The packer was set at 3060 ft. and the two phase fluid was injected into the leaching interval with a tailpipe extended to 3160 feet. The downward fluid velocity in the 1 ½ inches pipe was 1.8 ft./sec.

The solution was recovered from a hole (ID:SA-2) which was located 70 ft. away from the injection hole. It was produced at 10 gallons per minute. On July 11, 1975, the produced solution had 0.71 g/l of copper, 0.66 M NH_3, 0.04M $CaSO_4$.

A part of $(NH_4)_2SO_4$ was treated with lime to regenerate the ammonia and also to remove $CaSO_4$ in solution. After the pregnant liquor from the production hole was treated with lime, it was contacted with a liquid ion exchange extractant to extract the copper values. The extractant used was LIX-64N which is an oxime extractant sold by General Mills. At this point it should be noted that recovering the copper values from the pregnant solution is a step which is well known to those in the art and does not constitute a part of the invention. After the organic extractant is loaded with copper, it is stripped with a sulfuric acid (H_2SO_4). The stripped solution containing the copper values is then sent to an electrowinning circuit where the copper is electrowon.

EXAMPLE II

Details of a typical commercial process appear as follows. The ore body to be leached is a block lying between the levels 2500 feet and 5000 feet below the surface and having an aerial extent associated with 18 contiguous 5-spots, each having a producer to producer spacing of 330 feet. An example pattern is 18 5-spots contained within the area with dimensions 1650 feet by 1320 feet. The ore block is completely below the water table which lies 1000 feet below the surface.

The leaching process is initiated by pumping fluid from producer wells (28 in number), adding to that fluid: ammonia, sulfuric acid (to generate ammonium sulfate), an oxygen as a second phase; and pumping the fluid into injection wells in a continuous fashion. A concentration of 1.6M NH_3 and 0.4M NH_4 at the injection well is maintained. M indicates moles per liter.

Although the ultimate leaching interval is 2500 feet to 5000 feet, the initial interval exposed to leach solution contact is 3750 feet to 5000 feet ("half interval"). This is done to decrease the initial pore volume to be primed and thus speed breakthrough of copper, ammonia, and ammonium ion at producer wells. About 9600 tons of copper are produced in the first year of pumping with 88% of full production (40,000 TPY) being achieved in the second year of pumping. The rest of the interval is assumed to be opened up (by perforation of casing) in two stages occurring in pumping years 6–7 and years 14–15. Overall recovery as cathode copper over the life of the project is 45%.

Pregnant solution is pumped from producer wells by submersible pumps through a gas-liquid separator where gas entrainment occurs. The gas, which may contain some hydrogen, is diluted by an air blower before being vented. Total flow is 3450 gpm with a final copper concentration of 6 gpl.

Following gas separation, pregnant solution is pumped to the calcium treatment area of the main processing plant. Here, lime (in a crystallizer) is used to convert a portion of ammonium ion in the solution to ammonia. Under normal circumstances, ammonium ion builds up in the circuit due to chemical reactions associated with copper leaching and copper extraction. Lime treatment allows a savings in ammonia makeup. Calcium treatment also serves to control calcium supersaturation of pregnant solution.

As a result of the pressure of sulfate in the pregnant solution, gypsum is precipitated in the crystallizer. These solids are removed from the pregnant solution by a thickener followed by a rotary drum filter. Copper losses are kept to a minimum by washing the solids twice: first with a portion of raffinate from the liquid ion exchange section, and second, with water on the filter.

Copper is removed from solution by liquid ion exchange and electrowinning. Liquid ion exchange is operated with LIX-64N at 40° C. Aqueous feed solution must be cooled from about 70° to 40° C. prior to copper extraction. Activated carbon adsorbers are used to treat raffinate in order to remove most of any entrained or dissolved organic.

Makeup ammonia and two additives, Dowfax and Calnox ®, are then added (by in-line mixer) to the leach solution. The purpose of Dowfax (25 ppm level) is to improve oxygen dispersion characteristics of the solution. The purpose of Calnox ® (20 ppm level) is to inhibit scale formation on production well equipment. Calnox is removed by lime treatment in the gypsum crystallizer.

Leach solution, following reconstitution, is pumped back to the well field. An injection pump (up to 1200 psi pressure) pumps solution into a surface sparger where oxygen is dispersed. A guard filter precedes the injection pump to remove solids down to 20 ppm.

From the foregoing, one skilled in the art is taught how to remove base metals such as copper, nickel, molybdenum and mixtures thereof from igneous rocks located at a depth of 800 feet or more. The invention is particularly applicable to treating ore bodies that lie at a depth in feet and have a permeability in md that is twenty thousand md-ft or less. Prior to present invention there was no acceptable way of treating such ores in situ. The metal values are removed from the minerals in the minute fractures in the ore by forcing a two-phase lixiviant containing small oxygen bubbles into the fractures of the ore. To recover a metal, M, from a mineral in the ore having the general formula, MFe_yS_x, the minimum surface pressure, P_{SM}, of the two-phase lixiviant, in psi, is controlled at the surface in accordance with the following generalized formula:

$$P_{SM} \geqq \left(\frac{354.7\,[1.5x + 0.75y + 0.25z]}{MW}\right)\left(\frac{1 - f_{gc}}{f_{gc}}\right)(M/E),\ \text{in psi}$$

where M is the metal loading in gpl of the ore metal value to be recovered, E is the overall efficiency of oxygen utilization, and MW is the molecular weight of the metal ion to be recovered, z is the valance of the metal ion to be recovered in solution, and y and x are the subscripts for Fe and S, respectively, in the mineral.

It is also desirable to design the porosity of the sintered metal tubes so as to provide gas bubbles (O_2) which have diameters in the same order of magnitude or smaller than as the fracture openings in the ore. The diameters of the bubble therefore would be within the range of 2–1000 micron, but preferable, 10–100 microns.

The invention may be embodied in other specific forms without departing from the spirit or essential characteristics thereof. The present embodiments are therefore to be considered in all respects as illustrative and not restrictive, the scope of the invention being indicated by the appended claims rather than by the foregoing description, and all changes which come within the meaning and range of equivalency of the claims are therefore intended to be embraced therein.

We claim:

1. A process for the in-situ mining of a metal value selected from the group consisting of copper, nickel, molybdenum and mixtures thereof from an underground igneous ore body located at a depth of 800 ft. or more, said ore body having a permeability such that the product of the ore body thickness in ft. and permeability in md is a value of 20,000 md-ft or less, said ore also having minute fractures in which the metal values to be recovered are located in chemical combination with sulfur in a mineral of the generel formula MFe_yS_x, said process comprising:

(a) drilling at least one injection hole and at least one production hole into said ore body;

(b) introducing a two-phase lixiviant down said injection hole and into a leaching interval in said ore body, said leaching interval being beneath the water table, said two-phase lixiviant being formed from

  (1) an aqueous leach liquor capable of solubilizing the metal values, and,

  (2) minute oxygen bubbles of a size small enough to enter the fractures in the ore body from which the metal values are to be recovered;

(c) forcing the two-phase lixiviant through the leaching interval of the underground ore body to enable the two-phase lixiviant to penetrate the ore body through the fractures in the ore body and to enable the oxygen bubbles in the two-phase lixiviant to react with the sulfur to which the metal values are chemically bonded to enable the metal values to be solubilized by the aqueous leach liquor to produce a pregnant solution of metal values, said two-phase lixiviant being forced through said leaching interval by controlling the surface pressure of the two-phase lixiviant so that the minimum surface pressure, P_{SM}, in psi, is in accordance with the equation

$$P_{SM} \geqq \left(\frac{354.7\,[1.5x + 0.75y + 0.25z]}{MW}\right)\left(\frac{1 - f_{gc}}{f_{gc}}\right)M/E,\ \text{in psi}$$

where M is the metal loading in gpl of the metal value to be recovered, E is the overall efficiency of oxygen utilization, MW is the molecular weight of the metal ion to be recovered, z is the valance of the metal ion to be recovered, y and x are the subscripts for Fe and S respectively in a mineral of the general formula MFe_yS_x, where M is the metal to be recovered, and f_{gc} is the gas volume fraction associated with bubbly flow, the pressure of the two-phase lixiviant also being controlled so that the pressure of the two-phase lixiviant at the top of the leaching interval is less than the fracture pressure of the ore;

(d) withdrawing the pregnant solution to the surface through a production hole; and,

(e) recovering metal values from the pregnant solution.

2. The process as set forth in claim 1, wherein the mineral is chalcopyrite and copper is leached from the chalcopyrite and wherein the minimum surface pressure P_{SM}, in psi is in accordance with the equation

$$P_{SM} \geqq (23.7)\,(1\text{-}f_{gc}/f_{gc})\,(Cu/E)$$

where

Cu = copper loading, gpl

E = overall efficiency of oxygen utilization

f_{gc} = critical gas volume fraction associated with bubbly flow.

3. The process as set forth in claim 2 wherein $0.15 \leqq f_{gc} \leqq 0.25$.

4. The process as set forth in claim 3 wherein the igneous ore body has an average permeability of 10 md or less.

5. The process as set forth in claim 4 wherein the two-phase lixiviant is produced by forcing oxygen bubbles having a size between the range of 30 to 300 microns into the leach liquor.

6. The process as set forth in claim 5 wherein the surface pressure, P_{SM}, is greater than 800 psi.

7. The process as set forth in claim 6 wherein the pressure at the top of the leaching interval is 560 psi or more.

8. The process as set forth in claim 7 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq.in. for each foot of depth from the surface to the top of the leaching interval.

9. The process as set forth in claim 4 wherein the surface pressure, P_{SM}, is greater than 800 psi.

10. The process as set forth in claim 9 wherein the pressure at the top of the leaching interval is 560 psi or more.

11. The process as set forth in claim 10 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq.in. for each foot of depth from the surface to the top of the leaching interval.

12. The process as set forth in claim 3 wherein the surface pressure, P_{SM}, is greater than 800 psi.

13. The process as set forth in claim 12 wherein the pressure at the top of the leaching interval is 560 psi or more.

14. The process as set forth in claim 13 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq.in. for each foot of depth from the surface to the top of the leaching interval.

15. The process as set forth in claim 2 wherein the igneous ore body has an average permeability of 10 md or less.

16. The process as set forth in claim 15 wherein the two-phase lixiviant is produced by forcing oxygen bubbles having a size between the range of 30 to 300 microns into the leach liquor.

17. The process as set forth in claim 16 wherein the surface pressure, P_{SM}, is greater than 800 psi.

18. The process as set forth in claim 17 wherein the pressure at the top of the leaching interval is 560 psi or more.

19. The process as set forth in claim 18 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq.in. for each foot of depth from the surface to the top of the leaching interval.

20. The process as set forth in claim 15 wherein the surface pressure, P_{SM}, is greater than 800 psi.

21. The process as set forth in claim 20 wherein the pressure at the top of the leaching interval is 560 psi or more.

22. The process as set forth in claim 21 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

23. The process as set forth in claim 2 wherein the surface pressure, P_{SM}, is greater than 800 psi.

24. The process as set forth in claim 23 wherein the pressure at the top of the leaching interval is 560 psi or more.

25. The process as set forth in claim 24 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq.in. for each foot of depth from the surface to the top of the leaching interval.

26. The process as set forth in claim 1, wherein the mineral is pentlandite and nickel is leached from the pentlandite and wherein the minimum surface pressure, P_{SM} in psi, is in accordance with the equation

$$P_{SM} \geqq (39.2)\,(1\text{-}f_{gc}/f_{gc})\,(\mathrm{Ni}/E)$$

where,

Ni = nickel loading, gpl
E = overall efficiency of oxygen utilization
f_{gc} = critical gas volume fraction associated with bubbly flow.

27. The process as set forth in claim 26 wherein the igneous ore body has an average permeability of 10 md or less.

28. The process as set forth in claim 27 wherein the two-phase lixiviant is produced by forcing oxygen bubbles having a size between the range of 30 to 300 microns into the leach liquor.

29. The process as set forth in claim 28 wherein the surface pressure, P_{SM}, is greater than 800 psi.

30. The process as set forth in claim 29 wherein the pressure at the top of the leaching interval is 560 psi or more.

31. The process as set forth in claim 30 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq.in. for each foot of depth from the surface to the top of the leaching interval.

32. The process as set forth in claim 31 wherein the surface pressure, P_{SM}, is greater than 800 psi.

33. The process as set forth in claim 32 wherein the pressure at the top of the leaching interval is 560 psi or more.

34. The process as set forth in claim 33 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

35. The process as set forth in claim 26 wherein the surface pressure, P_{SM}, is greater than 800 psi.

36. The process as set forth in claim 35 wherein the pressure at the top of the leaching interval is 560 psi or more.

37. The process as set forth in claim 36 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

38. The process as set forth in claim 26 wherein $0.15 \leqq f_{gc} \leqq 0.25$.

39. The process as set forth in claim 1 wherein the mineral is molybdenite and molybdenum is recovered from the ore and wherein the minimum surface pressure, P_{SM} in psi, is in accordance with the equation

$$P_{SM} \geqq (16.6)\,(1\text{-}f_{gc}/f_{gc})\,(\mathrm{Mo}/E)$$

*M*o = molybdenum loading, gpl
E = overall efficiency of oxygen utilization
f_{gc} = critical gas volume fraction associated with bubbly flow.

40. The process as set forth in claim 39 wherein the igneous ore body has an average permeability of 10 md or less.

41. The process as set forth in claim 40 wherein the two-phase lixiviant is produced by forcing oxygen bubbles having a size between the range of 30 to 300 microns into the leach liquor.

42. The process as set forth in claim 41 wherein the surface pressure, P_{SM}, is greater than 800 psi.

43. The process as set forth in claim 42 wherein the pressure at the top of the leaching interval is 560 psi or more.

44. The process as set forth in claim 43 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

45. The process as set forth in claim 40 wherein the surface pressure, P_{SM}, is greater than 800 psi.

46. The process as set forth in claim 45 wherein the pressure at the top of the leaching interval is 560 psi or more.

47. The process as set forth in claim 46 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

48. The process as set forth in claim 39 wherein the surface pressure, P_{SM}, is greater than 800 psi.

49. The process as set forth in claim 48 wherein the pressure at the top of the leaching interval is 560 psi or more.

50. The process as set forth in claim 49 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

51. The process as set forth in claim 39 wherein $0.15 \leqq f_g \leqq 0.25$.

52. The process as set forth in claim 1 wherein the igneous ore body has an average permeability of 10 md or less.

53. The process as set forth in claim 52 wherein the two-phase lixiviant is produced by forcing oxygen bubbles having a size between the range of 30 to 300 microns into the leach liquor.

54. The process as set forth in claim 53 wherein the surface pressure, P_{SM}, is greater than 800 psi.

55. The process as set forth in claim 54 wherein the pressure at the top of the leaching interval is 560 psi or more.

56. The process as set forth in claim 55 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

57. The process as set forth in claim 52 wherein the surface pressure, P_{SM}, is greater than 800 psi.

58. The process as set forth in claim 57 wherein the pressure at the top of the leaching interval is 560 psi or more.

59. The process as set forth in claim 58 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

60. The process as set forth in claim 1 wherein the solution produced after values are recovered from the pregnant solution is returned back to an injection hole to be used as the liquid phase of the two-phase lixiviant.

61. The process as set forth in claim 60 wherein the surface pressure, P_{SM}, is greater than 800 psi.

62. The process as set forth in claim 61 wherein the pressure at the top of the leaching interval is 560 psi or more.

63. The process as set forth in claim 62 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

64. The process as set forth in claim 1 wherein the ore body is a porphyry copper ore in which copper bearing sulfide minerals occur in desseminated grains or veinlets.

65. The process as set forth in claim 64 wherein the copper bearing mineral is chalcopyrite.

66. The process as set forth in claim 65 wherein the surface pressure, P_{SM}, is greater than 800 psi.

67. The process as set forth in claim 66 wherein the pressure at the top of the leaching interval is 560 psi or more.

68. The process as set forth in claim 67 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

69. The process as set forth in claim 64 wherein the surface pressure, P_{SM}, is greater than 800 psi.

70. The process as set forth in claim 69 wherein the pressure at the top of the leaching interval is 560 psi or more.

71. The process as set forth in claim 70 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

72. The process as set forth in claim 1 wherein the two-phase lixiviant is produced by forcing oxygen bubbles having a size between the range of 30 to 300 microns into the leach liquor.

73. The process as set forth in claim 72 wherein the surface pressure, P_{SM}, is greater than 800 psi.

74. The process as set forth in claim 73 wherein the pressure at the top of the leaching interval is 560 psi or more.

75. The process as set forth in claim 74 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

76. The process as set forth in claim 1 wherein the surface pressure, P_{SM}, is greater than 800 psi.

77. The process as set forth in claim 76 wherein the pressure at the top of the leaching interval is 560 psi or more.

78. The process as set forth in claim 77 wherein the pressure at the top of the leaching interval is approximately 0.7–1.0 lbs. per sq. in. for each foot of depth from the surface to the top of the leaching interval.

79. The process as set forth in claim 1 wherein the pressure of the two-phase lixiviant at the top of the leaching interval is lowered from the surface pressure if the pressure of the lixiviant at the top of the leaching interval would exceed the fracture pressure of the ore.

80. The process as set forth in claim 79 wherein the pressure at the top of the leaching interval is lowered by the use of a choke.

81. The process as set forth in claim 1 wherein the aqueous leach liquor of step (b) (1) is an acidic leach liquor.

82. The process as set forth in claim 1 wherein the aqueous leach liquor of step (b) (1) is an ammoniacal leach liquor.

83. A process for the in-situ mining of a metal value selected from the group consisting of copper, nickel, molybdenum and mixtures thereof from an underground igneous ore body located beneath the water table comprising the following steps:

(a) selecting an ore body located at a depth of 800 feet or more below the surface and having a permeability of 10 md or less and having minute fractures 30–300 microns wide in which the metal values to be recovered are located in a mineral which contains sulfur;

(b) drilling at least one injection hole and at least one production hole into said ore body;

(c) introducing a two-phase lixiviant down said injection hole and into a leaching interval in said ore body, said leaching interval being beneath the water table, said two-phase lixiviant being formed from -

  (1) an aqueous leach liquor capable of solubilizing the metal values, and,

  (2) minute oxygen bubbles of a size small enough to enter the fractures in the ore body from which the metal values are to be recovered;

(d) forcing the two-phase lixiviant through the leaching interval of the underground ore body at a pressure greater than 800 lbs. per square inch but less than the fracture pressure of the ore to enable the oxygen bubbles in the two-phase lixiviant to react with the sulfur to which the metal values are chemically bonded to enable the metal values to be solubilized by the aqueous leach liquor to produce a pregnant solution of metal values;

(e) withdrawing the pregnant solution to the surface through one or more production holes; and,

(f) recovering metal values from the pregnant solution.

84. An apparatus for enabling metal values to be leached below the surface comprising:

(a) an injection hole for introducing a leach liquor into the mineral to be leached;

(b) a first means for introducing bubbles of an oxidizing gas having diameters within the range of 2-1,000 microns into said leach liquor to produce a two-phase lixiviant in which one phase is minute bubbles of an oxidizing gas which bubbles have diameters within the range of 2-1,000 microns; and,

(c) a second means for enabling continuous vertical circulation of the two-phase lixiviant within the leaching interval of said injection hole comprising a suction device with an outlet located in a lower portion of the leaching interval and an aspirator passage inlet located in the upper portion of the leaching interval, said first and second means together enabling continuous circulation of the two-phase lixiviant and also reducing coalescence of the bubbles of oxidizing gas.

85. Apparatus for the in-situ mining of minerals comprising:

(a) a gas sparging unit for use in introducing finely divided gas bubbles into a lixiviant used for in-situ mining of minerals, said device comprising a hollow casing having a first chamber formed therein into which liquid lixiviant is supplied and a second chamber isolated from said first chamber; a plurality of porous tubes formed of sintered powdered metal extending into said second chamber with said tubes having one end in fluid communication with said first chamber; and, means for introducing a pressurized gas about the portion of said tubes in said second chamber to enable the gas to penetrate into said tubes so that the gas can be wiped from the interior of the tubes by the lixiviant flowing through the tubes to form a lixiviant containing finely divided bubbles; and,

(b) a venturi-type exhauster in fluid communication with the sparging unit.

86. The apparatus as set forth in claim 85 wherein said exhauster has a tailpipe forming an outlet and an aspirator.

87. The apparatus as set forth in claim 86 wherein the exhauster and tailpipe prevent coalescence of the oxygen bubbles by enabling continuous vertical circulation of the lixiviant between the outlet of the injection nozzle and the aspirator passage inlet.

88. The apparatus as set forth in claim 87 wherein said casing has an outlet end with said first chamber being isolated from said outlet end and with the down stream ends of said tubes being positioned so that lixiviant containing gas bubbles can pass through said outlet end.

89. The apparatus as set forth in claim 85 wherein said means for introducing pressurized gas to said chamber comprises an inlet opening into said second chamber.

90. The apparatus as set forth in claim 89 wherein said inlet opening is formed through said casing.

91. The apparatus as set forth in claim 89 wherein said inlet opening is formed in a partition isolating said first and second chambers and wherein a gas supply tube positioned within said first chamber delivers pressurized gas through said inlet opening into said second chamber.

92. The apparatus as set forth in claim 88 wherein said gas supply tube, said first chamber and said second chamber are located in axial alignment to enable said unit to be inserted down a well bore.

93. The apparatus as set forth in claim 88 including means for removing gas bubbles trapped upstream of said outlet, said means comprising a conduit for providing communication between the interior of said casing adjacent the outlet end and the exterior of said casing.

94. The apparatus as set forth in claim 93 wherein said casing comprises a generally vertically extending cylindrical sleeve with said first chamber and second chamber being located in axial alignment in said sleeve and with said conduit extending axially within said sleeve through said first and second chambers.

95. The apparatus as set forth in claim 94 including a generally funnel shaped guide surface adjacent the outlet for guiding bubbles into the conduit.

96. The apparatus as set forth in claim 82 wherein said venturi type exhauster is positioned in the leaching interval of an in-situ injection hole beneath a cemented and packed off position of the injection hole.

97. A method for recovering metal values in-situ comprising:

(a) drilling at least one injection hole and at least one production hole into an ore body;

(b) introducing a two-phase lixiviant down said injection hole into a leaching interval in said ore body, said two-phase lixiviant being made by supplying a leach liquor into a porous tube of sintered metal while an oxidizing gas is supplied under pressure around the tube, said pressure being sufficient to cause the oxidizing gas to penetrate into the interior of the tubes as fine bubbles which are wiped off by the leach liquor passing therethrough; and,

(c) injecting the two-phase lixiviant into the leaching interval of the ore body through an aspirator which reduces coalescence of the bubbles of oxidizing gas by maintaining continuous vertical circulation within the leaching interval.

98. The process as set forth in claim 97 wherein the porosity of the sintered metal tubes are controlled so as to provide bubbles of oxidizing gas which are of the same order of magnitude or smaller than the fracture openings in the ore.

* * * * *

②

①

Turkish Mining Institute; Ankara, Turkey
Apr 5-16, 1976

IN SITU MINING TECHNOLOGY

Practical and Economic Aspects

1976

by

Roshan B. Bhappu

Mountain States Research and Development
Post Office Box 17960
Tucson, Arizona 85731

ABSTRACT

This paper reviews the current technical and economic aspects of in situ mining technology which is emerging into a major extractive process. Discussion under practical aspects includes in situ leaching techniques, controlling parameters in the leaching and metal recovery phases, and research needs. Topics under economic aspects cover role of feasibility studies, capital and operating costs. Finally, two case histories, one for copper extraction and the other for uranium recovery, are presented to illustrate the economic viability of this versatile mining and extraction process.

INTRODUCTION

The ever increasing demand for today's metals, the necessity for treating complex and lower grade ores, with higher operating costs, the need for conserving our dwindling minerals resources, and the public awareness of environmental pollution factors, make it mandatory to develop new techniques for extracting metals. In this endeavor, it is inevitable that in situ mining and in-place extraction technologies will be increasingly employed in metal production to satisfy the insatiable demands of our mineral based societies.

In situ, in-place, chemical or solution mining is the in-place extraction of metals from ores located within the confines of a mine (unfractured or fractured ore, stope fill, caved material, and ores in permeable zones) or in dumps, ore heaps, slag piles, and tailing ponds. These materials represent an enormous, untapped, potential source of all types of metals. The field of in situ mining encompasses the preparation of ore for subsequent in-place leaching, the flow of solutions and ionic species through rock masses and within rock pores, the leaching of minerals with inexpensive and regenerable leaching reagents under conditions prevailing in-place, the generation and regeneration of such solutions, and the recovery of metals or metal compounds from the metal-bearing solutions. Accordingly, the overall scope of this potential mining method embraces interdisciplinary science and technology requiring application of the principles of basic sciences, mineral technology, hydrology, and economics.

It is not inconceivable that eventually our ore reserves will consist largely of low-grade, refractory and inaccessible new deposits and low-grade zones near previously worked deposits, caved ore and stopes filled with submarginal ore, waste dumps, tailing ponds, and slag heaps. In situ mining promises economic recovery of metals from such types of deposits and sources.

Heretofore, this kind of mining has been more or less limited to the extraction of copper from low-grade materials; however, it has a much greater potential than this. Practically all metals are susceptible to leaching in the in situ environment. Processes will soon be developed for the in-place extraction and recovery of metals such as lead, zinc, nickel, manganese, uranium, silver, gold, molybdenum, and mercury.

Advantages of in situ mining include environmental attractiveness due to less land disturbance, improved mineral utilization due to working on lower and submarginal ores, and favorable economics due to earlier return on investment. Some of the disadvantages of the process are: possible contamination of ground water, potential land subsidence, and lack of field and operational experience.

A detailed bibliography on in situ leaching technology has been compiled by Twin Cities Mining Research Center of the United States Bureau of Mines (1).

TECHNOLOGICAL CONSIDERATIONS

In Situ Leaching Techniques

There are basically three distinct applications and associated leaching systems encountered in the field of in situ leaching, depending mainly in the physical location of the ore deposit. These situations are illustrated in Figure 1.

Typical examples of Type 1 are the numerous dump leaching operations practiced by the open pit copper mines in the Southwestern United States (2). Dump leaching, in general, is a convenience for open pit mining operations to dispose of lower grade (submarginal) stripped material in a remote area with the intention of recovering the valuable metals contained therein. Because of limited available area and easy placement, such dumps tend to be very high and the leaching of such piles, either through spraying or ponding of solution, is relatively slow and inefficient. Nevertheless, the production of copper from such dump leaching operations at the present amounts to about 200,000 tons of copper per year in the United States or approximately 15 percent of the total production. It should be noted that the effectiveness of copper extraction in these dump leaching operations has been partially attributed to bacterial oxidation of sulfide minerals.

Several successful heap leaching operations for extraction of copper, gold, and uranium as carried out in USA (Arizona, Nevada, Texas, New Mexico and Wyoming) also belong to Type 1. Heap leaching is a system in which low grade or mixed oxide-sulfide ores are leached in relatively small heaps on prepared pads or surfaces with positive drainage over a shorter leaching period under controlled conditions. In some cases the ore is brought to a designated site for treatment and removal, followed by piling of fresh ore on the same site for treatment.

Other viable in-place leaching situations belonging to Type 1 are the truly in situ leaching systems in which the ore is broken in-place by some type of blasting technique followed by percolation leaching of the fragmented ore mass. Appropriate collection galleries are provided to collect the leach solution for subsequent recovery of metal and recirculation of the barren solution for additional leaching. Since the ore is broken and left in place, such leaching systems are economically attractive. Ranchers Exploration and Development Corporation's Old Reliable Mine operation in Arizona and the Big Mike Mine in Nevada are fine examples of this in situ leaching method (3,4).

Also, belonging to Type 1 are the classic in situ leaching operations of the Miami and Ray Mines, Arizona USA in which the leaching has been carried out on fractured submarginal ore zones resulting from block caving operations.

In the case of the Miami Mine operation, it has been estimated that 85,000 tons of copper metal have been produced over a 15 year period since completion of the underground mining operation in 1959, with an estimated recovery of about 60-70 percent of the metal left in place. The leaching operation continues with no decrease in recovery rate and no immediate end in sight (5).

The second (Type II) situation reflects the leaching of relatively shallow deposits located less than 300 meters from the surface and which are under the water table. Such deposits may need to be fractured in-place and the solution drained in order to commence the alternate leaching and drying cycles. On the other hand, metal values from such deposits may be extracted by utilizing the bore hole mining technology as is currently practiced by uranium mining operation in the United States. The Bore Hole Mining technique usually consists of a properly designed pattern of wells suitable for injection of the leaching solution and recovery of metal-bearing liquors. The wells are like water wells except that the casings are perforated for optimum leaching conditions (6, 7, 8).

The final Type III situations are characterized by relatively deep-seated deposits in excess of 300 meters and under the water table. Leaching of such deposits through chemical mining has been proposed by Lawrence Livermore Laboratory, University of California, USA, after fracturing the ore body by conventional or nuclear devices. Leaching at such depths is enhanced by direct oxidation of sulfide minerals due to increased solubility of oxygen resulting from hydrostatic head (9-10).

A listing of various USA commercial and pilot plant operations employing the above three types of in-place leaching techniques in the extraction of copper, gold, and uranium are given in Table I. These examples of practical and economic operations clearly indicate that the in situ extraction technology has developed into a matured and a viable metal extraction process.

CONTROLLING PARAMETERS IN IN SITU LEACHING

From a practical application view point, the successful leaching of metals and minerals under in situ leaching environments is governed by several factors including geological and mineralogical considerations, elements of the leaching phase, solubility of minerals in various leaching reagents, effect of particle size, effective fragmentation, solution application and regeneration, and metal recovery methods.

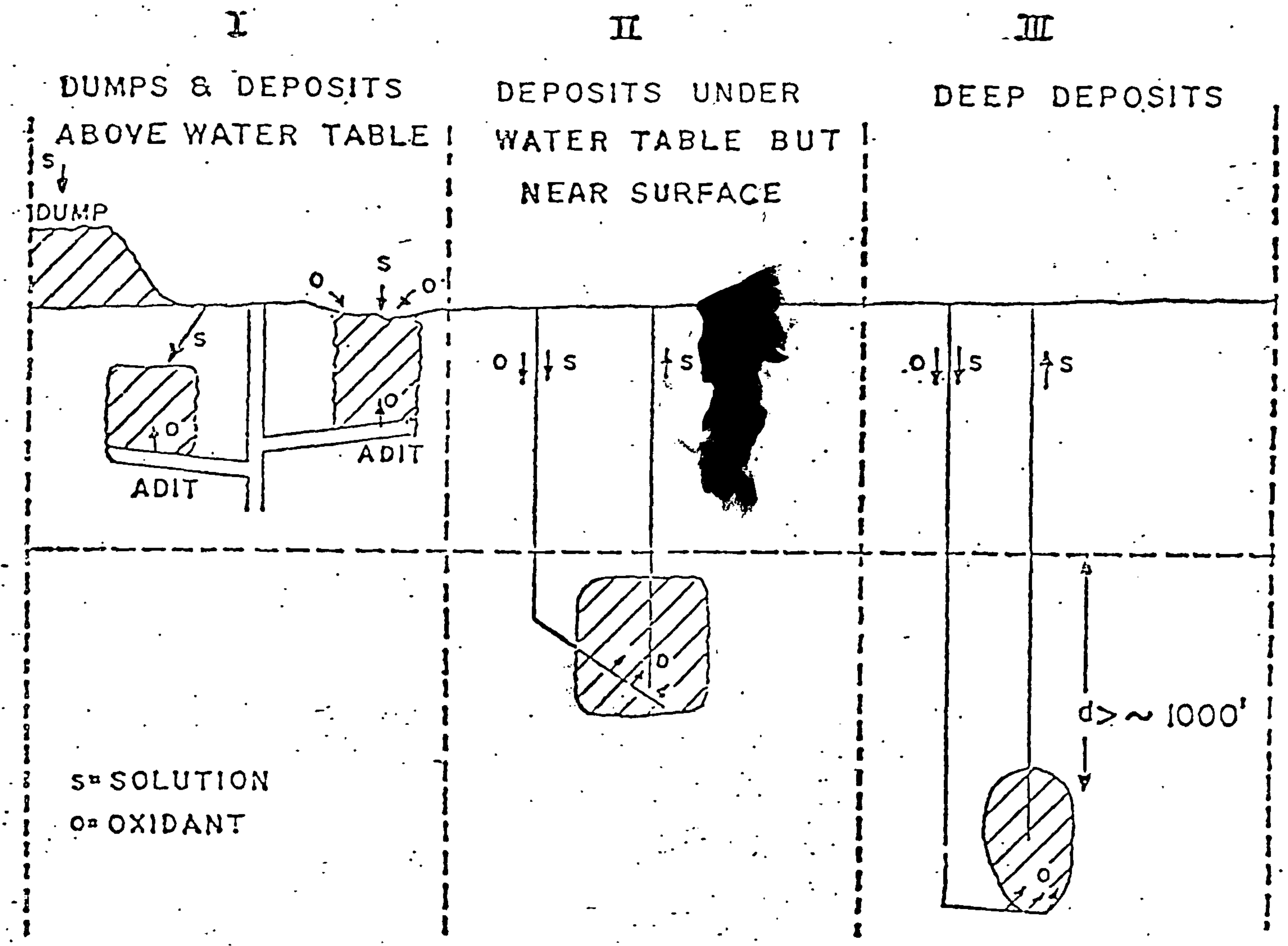


Figure 1

THREE TYPES OF IN SITU LEACHING SITUATIONS

(According to Dr. Milton Wadsworth)

TABLE 1

IN SITU LEACHING PRACTICES IN USA

Dump Leaching

1. Several Operations in Southwest USA (copper)

Heap Leaching

1. Bluebird Mine, Arizona (copper)
2. Inspiration Mine, Arizona (copper)
3. Johnson Camp Mine, Arizona (copper)
4. San Juan Mine, Arizona (copper)
5. Carlin and Cortez Gold Mines, Nevada (gold)
6. Round Mountain Mine, Nevada (gold)

In Situ Leaching (Copper)

1. Miami, Arizona (block caving operation)
2. Ray, Arizona (block caving operation)
3. Mountain City, Nevada (block caving operation)
4. Old Reliable, Arizona (4 MM Lbs. AN-FO coyote blast)
5. Big Mike, Nevada (4.14 M Lbs. AN-FO pit wall blast)
6. Zonia, Arizona (4.14, 0.9 and 1.5 MM Lbs. AN-FO blasts)
7. Nuclear Blast (AEC - Kennecott concept for deep-seated deposits)
8. Kennecott's Experiment (near Safford, Arizona)

In Situ Leaching (Uranium)

1. Utah Mining and Construction Company, Shirley Basin, Wyoming
2. Mine Water Leaching at Grants, New Mexico
3. Bacterial Leaching at Elliot Lake, Canada
4. Wyoming Minerals, Bruni, Texas
5. Wyoming Minerals, Ray Point, Texas
6. Arco-US Steel-Dalco, George West, Texas
7. Union Carbide, Duval County, Texas
8. Exxon, Powder River Basin, Wyoming
9. Mobil Oil Corporation, Webb County, Texas

Role of Mineralogy

In the past, very little attention has been devoted by both the geologist and mineral processing engineer to the geological and mineralogical occurrences which may be potential sources of ores. On the one hand, geologists have devoted less attention to lower grade, complex mineral deposits, and therefore, may have inadvertently curtailed an exploration program. On the other hand, the metallurgists have neglected to study in detail the geological environments and mineralization of potential ores with the view of determining the critical parameters that could assist in economically processing complex sub-marginal deposits as would be encountered in in situ leaching situations. In the final analysis, a chemical processing technique converts the valuable metals to a mobile state under conditions similar to those which deposited them originally in the host rock. Thus, the relationship between ore-genesis and chemical processing is close when the process of mineral or metal deposition in a particular ore deposit is clearly understood. Phase equilibria studies, Eh-pH diagrams, solubility products, mineral chemistry, surface chemistry and thermodynamics are utilized more and more by processing engineers to selectively separate minerals, dissolve minerals, and recover valuable metals and by-product metals. Doubtless, many of the future advances in processing technology will be derived from improved understanding of ore-genesis and related geological sciences.

It is interesting to note that the effectiveness of in situ leaching of copper, uranium, and gold by appropriate reagents is economically feasible because of favorable mineralization of the metal values in the host rock. This is especially true for successful leaching of oxidized gold ores at coarser size since these ores contain submicron size gold particles in fracture fillings and cavities in the host rock consisting of either limey or silicified siltstone, sandy dolomite, or schist. These host rocks also tend to be porous and remain permeable throughout the leaching cycle. In the case of oxide-copper ores, the mineralization also occurs along macro and micro fractures in the host rock, while in the case of uranium, the values are in the form of secondary-enriched precipitates coating the sand grains of the sandstone host.

Elements of the Leaching Phase

Accessibility, physicochemical interaction, and transport constitute the elements of the leaching phase involved in in-situ extractions. Limitations imposed on any of these factors restrict the leaching process. ✓

Accessibility is essential because interaction between the desired constituents and the lixiviant cannot take place in the absence of contacts, which depend on exposure and penetrability. The factors to consider are locations of the values, their volume and shape distribution, exposure area, specific surface, particle size, porosity, capillary pressure, viscosity, solubility of gases in leach solution, and surface roughness.

gas
[illegible]
gas wel
gas liq
also di

Physicochemical interaction converts the desired constituents from a fixed to a mobile condition and is governed by the solubility of the solid in leach solutions, vapor pressure in gases, and the presence of bacteria. Knowledge of free energies of reactants and products helps to determine whether a reaction is possible. The kinetic factors involved include time, concentration, diffusivity, specific rate constants, wettability, and oxidizing conditions, especially due to bacteria activity.

The first two elements by themselves do not ensure successful leaching without transport of products away from and reactants to the zone of interection through diffusion and convection. Diffusion is governed by concentration gradient and diffusivity, which in turn are influenced by particle size, micropore radius, temperature, and molecular mass. On the other hand, convective flow concerns interparticle penetration and is restricted by pressure gradient, permeability, viscosity, and surface roughness.

Solubility of Minerals

In general, nearly all the minerals are more or less soluble in either acids, alkalies, or other specific reagents such as cyanide. Most of the oxide-copper minerals such as azurite, malachite and chrysocolla are quite soluble in sulfuric acid. For this reason, sulfuric acid is the preferred leaching reagent for in situ extraction of oxide copper ores. However, in cases when the oxide copper ore contains considerable amounts of calcite ($CaCO_3$), the acid consumption is very high and it may not be economical to use sulfuric acid for leaching. Thus, the solubility of gangue (worthless mineral) contained in the ore may have profound influence in the selection of the appropriate reagent for leaching of valuable minerals.

As far as the sulfide copper minerals are concerned, simple sulfuric acid leaching is not very effective and the presence of $Fe_2 (SO_4)_3$ is necessary to dissolve minerals such as chalcocite and chalcopyrite under in situ leaching conditions. The strong oxidizing conditions required for their dissolution are satisfied by the presence of ferric iron, and oxygen (air). Many commercial operations have failed because the required oxidizing conditions were not maintained during the leaching process. This is also the reason why the presence of oxidizing bacteria of the Thiobacillus thiooxidans is necessary in dump, heap, and in situ leaching operations. It should be noted that the required iron is either available in the natural ore or that it is added to the leach system through the cementation step in which copper from the leach solution is precipitated as elemental copper when the liquid is contacted with scrap iron.

In the case of in-place leaching of uranium, both dilute sulfuric acid and carbonate-bicarbonate solutions are effective reagents for dissolving uranium minerals. However, both the reagents require a chemical oxidant to oxidize uranium from the insoluble +4 oxidation state to the soluble +6 state. Typical acid leach oxidants are $NaClO_3$ or MnO_2 and typical carbonate leach oxidants are hydrogen peroxide or NaOCl.

For gold and silver leaching, cyanide has been found to be the most selective reagent and is the universal solvent used. Here again, oxygen is required in the reaction and efforts must be made to maintain oxidizing conditions during the in-place leaching of gold (11).

Effect of Particle Size

Since leaching of minerals by various chemical reagents involves effective contact between the two, it is obvious that the mineral must be liberated or freed from the host rock. The only other way the reagent can contact the valuable mineral is by effective penetration or diffusion through fractures, cracks, capillary action, and other physical factors considered under the "elements of the leaching phase". Regardless of the process of contact, it is quite evident that the reduction in particle size would be of considerable benefit in in situ leaching of metals and minerals. With finer ore size it would be possible to obtain higher extraction rates in shorter leaching periods.

Effective Fragmentation

The desired particle size in in situ leaching situations can be obtained through proper blasting and fracturing techniques. The blast design for in situ leaching closely follows the principles of blast design used in more conventional mining practices. However, in case of in situ blasting unusually large dimensions are involved and these pose inherent problems associated with large blasting situations. To date, several unusually large blasts have been made for leaching with results that are very encouraging (12, 13, 14, 15).

The important factors that need to be considered in engineering a blast for in situ leaching are (a) shape of deposit, (b) characteristics of ore body, (c) seismic activity encountered, (d) missile consideration, (e) type of explosive, (f) powder factor and distribution, (g) initiating and delay system, (h) loading logistics, (i) economics, and (j) safety. All these factors should be considered in the decision-making procedure for the blast.

Besides conventional fracturing by blasting compounds, efforts have been made to explore the possibility of using liquid explosives, hydrofracting, and nuclear explosives. No doubt these and many other new concepts for fracturing will be considered as in situ leaching becomes a more conventional method in the future.

Solution Application and Regeneration

The primary goals in in situ leaching are: (1) to recover maximum metal content in the minimum solution volume and (2) to recover the major portion (+90 percent) of the specified volume of leach solution fed to the system with a given economic metal content over the life of the operation. This minimum content should be such that the value of recovered metal will offset the cost of operation, amortization, overhead and profit. It is also important from environmental considerations that the leach solution be confined in the leaching system and not report as a pollutant in ground water. This is especially important in gold leaching using cyanide as the specific reagent. Both the toxic nature of cyanide and preciousness of leach solution containing gold necessitates in situ leaching conditions approaching a "zero discharge" objective.

Inasmuch as the chemical reagents used in in situ leaching generally constitute a major cost item and greatly influence the economics of leaching, reagent generation and regeneration play a very important part in any in-place leaching process. Some reagents can be generated and regenerated by natural processes in the leaching cycle, whereas others require chemical processing. The only known example of auto-regeneration of reagent is the natural reactions involved in the production of sulfuric acid and ferric sulfate from pyrite and spent ferrous sulfate solution under natural air or biological oxidation conditions. Most of the other reagents are generated and regenerated by various industrial methods such as contact process for H_2SO_4, electrolysis for generating hypochlorite, and catalysis autoclaving to obtain $Fe_2(SO_4)_3$. In some cases, part of the reagent is regenerated by the metal extraction process incorporated in the flowsheet, such as the regeneration of H_2SO_4 through solvent extraction and electrowinning as practiced in the extraction of copper from leach solutions. In other cases, though the regeneration of reagent from spent solutions is possible, it is not economically feasible or environmentally attractive such as the regeneration of cyanide from spent solution in gold leaching.

The Recovery of Metals from Leach Solution

The last phase of any hydrometallurgical process, including in situ leaching, is the recovery of metals from leach liquors. Conventional purification of a metal-containing solution followed by recovery of metals or compounds from the solution by either chemical or electrolytic precipitation is employed to obtain the marketable product. These recovery techniques are adequately covered in the literature and their effectiveness is clearly demonstrated in several successful plant practices.

In connection with in situ leaching applications, however, the recovery phase poses certain technical problems that may influence the overall effectiveness of the process. One such difficulty concerns treating a large volume of very dilute metal-bearing solution. This may require recirculation of the leach solution to build up the metal content and then bleeding off of a small part of the concentrated leach stream for metal recovery.

Newer techniques of ion exchange, solvent extraction, and charcoal adsorption are being used for effective concentration of metals from leach solution. Moreover, these procedures have proved very effective for processing large volumes of leach solutions containing more than one valuable metal. In recent years, the recovery of copper from leach solutions by solvent extraction - electrowinning has become more attractive than the copper cementation process used conventionally all over the world. Similarly, ion exchange has been exclusively used for recovery of low concentrations of uranium from in situ leaching operations. Finally, the use of activated carbon for the recovery of gold and silver values (as low as one part per million or less) from in-place leaching operations has become a favorite process. In this case, the gold is subsequently extracted by desorbing gold values from carbon by hot caustic solution followed by electrolysis to obtain the gold.

Research Needs

Although in situ leaching in recent years has matured into a viable metal extraction process, considerable additional research is needed to bring it in competition with other high-confidence and universally acceptable processes.

The principle areas of research required to accomplish the above goal are:

1. Improved techniques for measuring physical properties of ores in-place.
2. Improved techniques for drilling and blasting to produce desired fragmentation.
3. Improvement in injection and recovery systems.
4. Improved understanding of hydrology in broken ore masses.
5. Design of leaching systems for maximum oxidation.
6. Basic studies on leaching of dispersed mineral particles.
7. Improved methods of scaling up from laboratory tests.
8. Optimum parameters for leaching of massive deep-seated deposits.
9. Improved environmental controls.

ECONOMIC CONSIDERATIONS

Feasibility Studies

A well organized and realistic feasibility study should include: confirmation of ore reserves and grades, an effective mine plan, leaching and metal recovery flowsheet, preliminary capital and operating cost estimates, determination of rate and cost of production and selling price of end products, and a cash flow analysis and return on investment (ROI) study. The profit potential and the viability of the project can then be accurately assessed by all the parties concerned with the development of the mine. Such a feasibility study also assists in screening and evaluating various choices and options involved in mining and processing. This is especially true for the development of lower grade deposits, since the paramount question here is the selection of the best method of extracting the metal values from a particular deposit.

In the case of in situ leaching, especially, the feasibility study plays a very significant role since the percentage recovery of metal under in situ leaching environment is very difficult to predict. Even though the laboratory or pilot plant tests indicate a certain recovery, say 60 or 70 percent, there is no guarantee that in actual practice it would be possible to attain the specified recoveries. In such cases, it behooves the mine management to carry out the above feasibility study and cash flow analyses at 60, 50, 40, 30, 20, and 10 percent recovery levels. If the cash flow and thus the economic viability of the project does not appear to be favorable for 50 or 40 percent recovery levels, then it would be preferable to cancel the project or undertake additional laboratory and field studies in order to increase the confidence level of the economically acceptable recovery figure. On the other hand, if the feasibility study appears favorable for 30 to 20 percent recovery levels, all efforts should be made to initiate the in situ leaching operation as soon as possible.

As a matter of fact, the economic viability of in situ leaching operations through realistic feasibility studies have been confirmed by recent in situ leaching experiences at Old Reliable Mine, Arizona and Big Mike Mine, Nevada (by Ranchers Exploration and Development Company), both of which proved economically successful at 15 to 20 percent overall copper extractions.

Since the cash flows and thus the overall economic feasibility of an in situ leaching venture may be influenced significantly by the price of metal, it is customary to carry out the above feasibility studies at selected metal prices below and above the prevailing price. In this way, the mine management is in a better position to assess the economics of the venture under different economic climates.

Capital and Operating Cost Estimates

From the viewpoint of economics, the net return from a mining venture using a specific process, such as in situ leaching, will depend upon the quantity and quality of metal extracted from the ore. The overall economics will also be influenced by the capital cost of unit processes (cementation, solvent extraction, electrowinning, etc.), the operating cost of unit operations (drilling, blasting, leaching, etc.), the cost of reagents, supplies, and utilities, and the prevailing prices of metals and by-products. For these reasons, the economic feasibility of any new venture and process must be considered in light of all these and many more potential factors.

Table 2 shows the estimated capital costs for unit processes encountered in typical in-situ leaching operations for extraction of specified quantities of copper, gold and uranium. It should be noted that in some cases the capital cost is based on quantity of metal produced per day while for others, it is possible to estimate the capital cost of similar processes for other production schedules (pounds or ounces per day) and leach solution volumes (gpm).

Tables 3 and 4 show the estimated costs for various unit operations encountered in in situ and heap leaching extraction of copper, gold and uranium. As can be seen, the cost of solvent extraction - electrowinning of copper is considerably less than for cementation. The primary reason for this difference is the additional cost of smelting and refining ($0.20 to 0.28 per lb.) required for impure copper in comparison to high purity cathode copper produced in the solvent extraction - electrowinning process.

It is also interesting to note that in the processing of gold ores either by in situ or heap leaching, the overall operating costs vary from $1.48 to $1.89 per ton. These figures indicate that using these leaching techniques it may be profitable to extract gold values from ores averaging 0.03 to 0.05 ounce per ton when the price of gold is above $100 per ounce gold.

Finally, in the case of extracting uranium by "bore hole mining" technology, the overall operating costs vary from $5.00 to $17.00 per pound U_3O_8 (yellow cake) produced. Since the current price of uranium in the USA is in excess of $25 per pound U_3O_8, it may be possible to treat uranium ores containing grades as low as 0.02 percent U308.

TABLE 2

ESTIMATED CAPITAL COSTS FOR UNIT PROCESSES

UNIT PROCESSES	BASIS	ESTIMATED COST
Copper (40,000 lbs./D; 4,000 gpm at 1.0 gpl Cu)		
Cementation	\$70,000 - 80,000/T	\$1,400,000 to 1,600,000
Solvent Extraction	\$150-200/ ft.^{2} settling area	\$3,000,000 to 4,000,000
Electrowinning	\$60 to 70/lb. cathode Cu	\$2,400,000 to 2,800,000
Total SX-EW	\$135 to 170/lb. Cu/D	\$5,400,000 to 6,800,000
Gold (200 oz/d; 2,000 gpm; 0.016 oz Au/T solution)		
Carbon Adsorption	\$250 - 300/gpm	\$500,000 to 600,000
Desorption-Electrowinning	\$1,750 - 1,800/oz Au	\$350,000 to 360,000
Total ADS - DES - EW		\$850,000 to 960,000
Uranium (2,750 lbs./D; 2,000 gpm at 0.125 gpl U_3O_8)		
Ion Exchange	\$1,750 - 2,000/gpm	\$3,500,000 to 4,000,000
Precipitation	\$450 - 500/lb. U_3O_8 D	\$1,237,500 to 1,375,000
Total IX-PPT		\$4,737,500 to 5,375,000

TABLE 3

ESTIMATED DIRECT OPERATING COSTS FOR IN-PLACE EXTRACTION

UNIT OPERATION	ESTIMATED COST
Fracturing in place (with development)	\$0.30 to 0.40/T
Mining (drilling, blasting, hauling and dumping)	0.40 to 0.50/T
Crushing (primary, secondary, tertiary)	0.11 - 0.22* - 0.33/T*

COPPER ORE (AVERAGING 0.5 TO 1.0% Cu)

Leaching (heap; pads; 4.0 lbs. H_2SO_4/lb. Cu)	\$0.35 to 0.45/T
Leaching (in situ, 4.0 lbs. H_2SO_4/lb. Cu)	0.22 to 0.32/T
Cementation	0.10 to 0.12/lb.
Smelting, refining, and transportation	0.20 to 0.28/lb.
Solvent extraction - electrowinning	0.07 to 0.09/lb.
Cost/lb. Cu (in situ - cementation)	0.39 to 0.46
Cost/lb. Cu (in situ - SX - EW)	0.15 to 0.22
Cost/lb. Cu (heap-cementation)	0.50 to 0.66
Cost/lb. Cu (heap - SX - EW)	0.26 to 0.42

GOLD ORE (AVERAGING 0.03 TO 0.06 OZ./T)

Leaching (in situ; 1.0 lb. NaCN; 3.0 lbs. CaO)	\$0.45 to 0.55/T
Leaching (heap with pads; 1.5 lbs. NaCN; 3.0 lbs. CaO)	0.55 to 0.65/T
Carbon adsorption, desorption & electrowinning	0.16 to 0.20/T
Cost/T (in situ - carbon adsorption - EW)	1.48 to 1.58
Cost/T (heap with pads - carbon adsorption - EW)	1.79 to 1.89

URANIUM ORE (AVERAGING 0.05 TO 0.10% U_3O_8)

Bore hole drilling (with casing)	\$13.00 to 18.00/Ft.
Leaching (drilling and carbonate leaching)	2.00 to 3.00/Lb.
Cost/lb. U_3O_8 (in situ leach - IX-PPT)	5.00 to 7.00

* Cumulative at each stage.

TABLE 4

COST COMPARISON (0.66% Cu = 8.0 lbs./ton recovered)
DIRECT OPERATING COSTS PER TON ORE

UNIT OPERATIONS	CONVENTIONAL	IN SITU
	(strip ratio = 1:1)	
Drilling Blasting Hauling Dumping	$0.70 - 0.90	Drilling Blasting $0.35 - 0.45
Crushing	0.15 - 0.30	
Ore Handling	0.10 - 0.15	
Leaching	0.40 - 0.50 (with pad)	0.20 - 0.30
Metal Recovery (SX-EW)	0.56 - 0.64	0.56 - 0.64
Supervision and Administration	0.35 - 0.40	0.25 - 0.30
Total	$2.26 - 2.89	$1.36 - 1.69
Cost/lb.	$0.28 - 0.36	$0.17 - 0.21

TABLE 5

PRELIMINARY FEASIBILITY STUDY IN SITU LEACHING FOR COPPER

Ore Reserve	5,000,000 tons
Grade	0.66% Cu
Mine Life	7 years at 715,000 TPY
Estimated recovery	60% or 8.0 lbs/ton
Daily production	16,000 lbs.

CAPITAL COST		
Leaching facilities	$ 200,000	
SX-EW (with LIX inventory)	3,300,000	
Supporting facilities	100,000	
Cost for processing		$ 3,600,000
Cost for fracturing (5MM tons at 0.40/T)		2,000,000
Total capital cost		$ 5,600,000
Operating cost for processing		$1.06 per ton

TABLE 6

PRELIMINARY CASH FLOW ANALYSIS (IN THOUSANDS OF DOLLARS)

Capital (3,300,000 + 2,000,000)	$5,600
Sales (5,760,000 lbs. at $0.60)	3,456
Operating (715,000 at $1.06/ton)	758
Depletion (15% of sales)	518
Depreciation (straight line)	757
Local tax and insurance (8% of sales)	276
Total operating cost	2,309
Operating income	1,147
Income tax (48%)	551
Net operating income	596
Investment credit (10% capital or ½ net income)	298
Net income	894
Working capital (20% of total operating cost)	462
Cash flow	1,707
Payout time	3.1 years
ROI (discounted to present worth)	30%

Case Histories

In order to demonstrate the economic viability of the in situ leaching systems and to demonstrate the usefulness of preliminary feasibility studies, two case histories, one of in situ leaching of copper and the other pertaining to bore hole mining of uranium, will be reviewed.

(1) In Situ Leaching for Copper

The hydrometallurgical processes currently available for treating oxide-copper ore include dump leaching, heap leaching, vat leaching, agitation leaching, and in situ leaching in combination with the conventional cementation (with scrap iron) or the newly developed solvent extraction-electrowinning system for copper production.

In the case history illustrated here, we are concerned with a 5 million ton oxide copper deposit with a grade of 0.66 percent copper. The in situ leaching system employed will consist of blasting ore in-place and leaching the broken ore with sulfuric acid under the in situ environment followed by recovery of copper from leach solution by solvent extraction and electrowinning. The leach system and recovery plants are designed to produce 16,000 pounds of copper per day over a 7-year mine life.

The capital and operating costs as shown in Table 5 are taken from the pertinent data provided in previous tables. Table 6 shows the preliminary cash flow analysis for this particular in situ leaching operation. As can be seen, the payout time (time required to pay back the capital expenditure) is about two to three years with a return on investment of 30 percent. (ROI is the average compound interest rate at which the expected net cash flow, discounted to present worth, would return cash outlay in full.) Obviously, this study indicates an economically attractive venture.

(2) Bore Hole Mining for Uranium

The in situ mining of lower grade uranium ores has received increasing attention in recent years due to the favorable price of uranium in the free market. An in-place extraction technique of considerable promise is the so-called "bore hole" mining in which the uranium is recovered by drilling into the ore body, circulating a lixiviant fluid to dissolve the mineral, extracting the uranium values from the pregnant solution, regenerating and recycling the lixiviant. Such a technique is economically and environmentally attractive in extracting uranium values from deeper, lower grade reserves.

In the evaluation of a uranium-mineralized block as a potential producer by in-place leaching using the bore hole mining technique, it is first necessary to establish a grade-thickness product which will cover development costs and operating costs with something left over for recovery of investment in plant and equipment and for profit. This grade-thickness product is then a cutoff parameter for determining whether another hole is included in reserves or excluded.

Once the holes have been classified as potential ore or waste, the reserves in each potential block can be estimated, for example, by the ore outline method.

Development costs for determination of whether a hole is or is not a potential contributor to ore include:

1. Costs of drilling and casing injection and production wells.
2. Pumps.
3. Surface piping and electrical requirements to bring an ore block into production.

Operating costs in the field include:

1. Operating and repair costs of injection of solvent and of reagents required to convert uranium minerals to soluble form.
2. Operating and repair costs of pumping the pregnant solution from collecting point to the ion exchange plant and costs for recovering the uranium minerals.

Now an injection and production well pattern can be finalized and any necessary adjustments to development costs can be made. The capital cost for the piping to carry the solution from the ore blocks to the ion exchange plant, the capital cost of the ion exchange and precipitation plant, and other ancillary equipment can be estimated.

A cash flow analysis is now made to determine whether the project has satisfactory profit potential. At this stage, it is necessary to be sure that a realistic estimate of the true thickness of strata which will be penetrated by solution has been made. Large amounts of reagents will often be consumed in reactions in sub-ore grade zones above and below the "ore interval." Also, there will often be one or more sub-ore zones within an ore interval which will consume reagents.

Table 7 gives the details of the capital and operating costs for the second case-history involving the feasibility of extracting uranium values from a given sandstone deposit. Table 8 shows the financial analysis for the project under consideration.

As can be noted, the bore hole mining technique appears to be a very attractive method for extracting uranium from lower grade ores. At the old price of $8.00 per pound U_3O_8 the project is not economically favorable. However, at uranium price of over $16.00 per pound, the project appears to be very attractive with ROI in excess of 85 percent.

TABLE 7

IN SITU URANIUM LEACHING OPERATION

Type of Operation	Bore Hole Mining
Deposit	30-foot Ore Zone at 400-foot Depth
Reserves	4,000,000-Ton Deposit at 0.1 Percent U_3O_8
Contained U_3O_8	8,000,000 Pounds
Recoverable U_3O_8	5,600,000 Pounds at 70 Percent Recovery
Plant Capacity	1,000,000 Pounds U_3O_8 per Year 2,750 Pounds per Day
Production Life	5-6 Years
Type Process	Ion Exchange-Precipitation
Type Leaching	Acid at 100 pounds H_2SO_4 per Ton of Ore
Leaching Rate	2,000 GPM
Solution Grade	0.125 gpl U_3O_8
Capital Cost, Leaching	\$3,400,000
Capital Cost, IX-PPT	\$5,000,000 (IX = \$4,000,000; PPT = \$1,000,000)
Total Capital Cost	\$8,400,000
Operating Cost, Leaching	\$3.00 per Pound U_3O_8
Operating Cost IX-PPT	\$4.00 per Pound U_3O_8
Total Operating Cost	\$7.00 per Pound U_3O_8

Cut-off Grade Based on:

Operating Cost at \$8.00 per Lb. - 0.062 Percent U_3O_8

Operating Cost at \$16.00 per Lb. - 0.032 Percent U_3O_8

Operating Cost at \$24.00 per Lb. - 0.020 Percent U_3O_8

TABLE 8

IN SITU URANIUM LEACHING ECONOMICS

(BORE HOLE MINING - IX - PPT.)

CASH FLOW (In Thousands of Dollars)

	Price of U_3O_8			
	$8/Lb.	$16/Lb.		$24/Lb.
Capital Cost	$8,400	$8,400	$8,400	$8,400
Net Sales (at 70% Extraction)	8,000		16,000	24,000
(at 35% Extraction)		8,000		
Cost and Expenses:				
Total Operating	7,000	7,000	7,000	7,000
Depreciation (1)	1,500	1,500	1,500	1,500
Depletion (2)	1,760	1,760	3,520	5,280
Indirect Cost (3)	350	350	350	350
Total	10,610	10,610	12,370	14,130
Operating Income	<1,210>	<1,210>	3,630	9,870
Income Tax (4)	-	-	1,742	4,738
Net Operating Income	<1,210>	<1,210>	1,888	5,132
Investment Credit	-	-	840	840
Net Income	<1,210>	<1,210>	2,728	5,972
Cash Flow	2,050	2,050	7,748	12,752
Pay-Out in Years	4.1	4.1	1.1	0.7
Return on Investment (6)	12%	12%	86%	117%

Notes:

(1) Depreciation - Straight -line, 5.6 years

(2) Depletion - 22% of Net Sales

(3) Indirect Cost - 5% of Total Operating Cost

(4) Income Tax - 48%

(5) Investment Credit- 10% of Capital Cost for the First Year, Not to Exceed 50% of Net Operating Income in One Year

(6) Rate of Return Discounted to Present Worth

BIBLIOGRAPHY

1. Larson, W. C. and Dandrea D. V. In Situ Leaching Bibliography, Parts I and II, U.S.B.M., February 1976.

2. Sheffer, H. W. and Evans, L. G. Copper Leaching Practices in the Western United States. U.S.B.M. IC 8341, 1968, 57 PP.

3. Rudershausen, C. G. Copper Solution Mining at Old Reliable. Pres. at National Meeting of American Inst. of Chemical Eng., Salt Lake City, Nev., August 18-21, 1974, 11 PP.

4. Ward, M. H. Surface Blasting Followed by In Situ Leaching the Big Mike Mine. Proc. Solution Mining Symp., AIME Annual Meeting, Dallas, Texas, February 25-27, 1974, 243-251 PP.

5. Fletcher, J. B. In Place Leaching-Miami Mine, Miami, Arizona. Pres. at AIME Centennial Annual Meeting, Soc. of Min. Eng., AIME, New York, N.Y., March 1971, AIME Preprint 71-AS-40; Trans. Soc. of Min. Eng., AIME, V. 250, No. 4, December 1971, 324-337 PP.

6. Crawford, E. Developers Eye Texas Potential for In Situ Uranium Leaching. Eng. and Min. J., V. 176, No. 7, July 1975. 81-82 PP.

7. Shock, D. A., and Conley, F. R. Solution Mining - Its Promise and Its Problems. Proc. Solution Mining Symp., AIME Annual Meeting, Dallas, Texas, February 25-27, 1974, 79-97 PP.

8. Steckley, R. C., Larson, W. C. and Dandrea, D. V. Blasting Tests in a Porphyry Copper Deposit -- In Preparation for In Situ Extraction. U.S.B.M. RI 8070, 1975, 57 PP.

9. Lewis, A. E. Chemical Mining of Primary Copper Ores by Use of Nuclear Technology. Proc. Symp of Engineering with Nuclear Explosives, Las Vegas, Nev., January 14-16, 1970, Conf. 700101, V. 21, 907-917 PP.

10. Lewis, A. E., Braun, R. L., Sisemore, C. J. and Mallon, R. G. Nuclear Solution Mining - Breaking and Leaching Considerations. Proc. Solution Mining Symp., AIME Annual Meeting, Dallas, Texas, February 25-27, 1974 56-75 PP.

11. Heinen, H., and Porter, B. Experimental Leaching of Gold from Mine Waste. U.S.B.M. RI 7250, April 1969, 5 PP.

12. Dandrea, D. V., Dick, R. A., Steckley, R. C. and Larson, W. C. A Fragmentation Experiment for In Situ Extraction. Proc. Solution Mining Symp., AIME Annual Meeting, Dallas, Texas, February 25-27, 1974, 148-161 PP.

13. Dick, R. A. In Situ Fragmentation for Solution Mining - A Research Need. Pres. at the Second Internat. Symp. on Drilling and Blasting, Phoenix, Arizona, February 12-16, 1973.; Available for consultation at U.S.B.M., Twin Cities Mining Research Center, Minneapolis, Minnesota.

14. Porter, D. D. Blast Design for In Situ Leaching. Pres. at Southwest Mineral Industry Conf., Phoenix, Arizona., April 27, 1973, 7 PP.; Available from E. I. duPont de Nemours and Co., Inc., Wilmington, Del. 19898.

15. Porter, D. D., and Carlevato, H. G. In Situ Leaching: A New Blasting Challenge. Proc. Solution Mining Symp., AIME Annual Meeting, Dallas, Texas, February 25-27, 1974, 33-43 PP.

INTERACTION OF SILICATE-CALCIUM MATERIALS WITH METAL IONS IN AQUEOUS SOLUTIONS

UDC 669.2:628:334

V. A. Fedorov, I. P. Alekseeva, A. P. Dushina, and V. B. Aleskovskii

Inorganic ion-exchange materials, particularly slags, can be used when purifying the industrial effluents from the ions of heavy ion-exchange metals (1-3).

Dissolution and hydrolysis of the slag precedes dissolution and hydrolysis of the slag (4). Depending on the chemical composition of the slag, which is determinedby its basicity, even the composition of the dissolved part of the sorbent should change. The literature contains no information on the connection between the solubility of a slag and its sorption capacity, which has led to this work.

Table 1
Chemical Composition (%) and Basicity of Metallurgical Slags and Slimes

Spec.	Slag and slime used	CaO	SiO_2	MgO	Al_2O_3	Fe_2O_3	Misc.	$\frac{CaO}{SiO_2}$
1	Blast furnace slag of Novo-Lipetsk Metallurgical Combine	41.5	38.9	6.5	11.5	0.33	0.95 MnO	1.1
2	Blast furnace slag of Kommunarsk Metallurgical Plant	45.0	36.0	5.0	6.0	0.5	1.2-1.5 MnO	1.2
3	Red mud of Pavlodar Aluminum Plant	37.6	18.6	-	7.8	22.7	2.6	2.0
4	Electric furnace slag of Kirovsk Plant in Leningrad	55-65	15-20	10-20	1-5	1-5	9-10 MnO	3.2-3.7
5	Electric furnace slag of Izhorsky Plant in Leningrad	49-59	24.5	19	-	1.0	1.6 MnO	2.0-2.4
6	Slag of carbon free ferrochrome of Aktyubinsk Ferroalloys Plant	48-52	25-27	9.0	6.0	-	4-5 Cr_2O_3	1.9
7	Blast furnace slag of Cherepovets Metallurgical Plant	38.7	38.3	10.6	7.9	0.4	-	1.1
8	Slag of Serovo Metallurgical Combine	43	36.9	-	12.6	1.2	-	1.2
9	Slag of Makeevka Metallurgical Combine	47.9	36.1	2.7	9.9	0.46	-	1.3
10	Slime of Ural'sk Aluminum Plant	11.2	8.2	-	-	-	-	1.4

For the research, use was made of a series of metallurgical slags and slimes (Table 1), differing in their chemical composition and basicity. Samples 3-7* were selected to study the solubility. The dissolution kinetics were studied under static conditions at a s:l ratio of 1:1000 and a pH value of 6.0. After contact in the filtrate a determination was made of the concentration of silicic acid and of calcium ions, and the pH value. The sorption activity of the slags was checked under static conditions at an s:l ratio of 1:1000 from artificially prepared solutions, corresponding in composition to the effluents from the Ust-Kamenogorsk Lead-Zinc Combine, containing (mg/l): 1-5 Cu^{2+}, 1-6 Cd^{2+}, 1-5 Pb^{2+}, and 5-40 Zn^{2+}, with the pH at 3-7. Slag activity was judged from the residual concentration of metal ions in filtrate following a 30 minute mixing of the slag with the solutions.

Dissolution of the slags (see Table 2), similarly to the situation with individual

---

* Slags were used with particle sizes < 0.1 mm. Specimens 4-6 are self-crumbling. 87% of their content were fractions < 0.1 mm and about 4% were fractions > 0.2 mm. These slags were used without preliminary separation into fractions. The granulometric composition has almost no effect on their sorption capacity.

Table 2
Solubility of Slags

Contact period	Aktyubinsk slag			Kirovsk slag			Izhorsky slag			Cherepovets slag			Pavlodar slag		
	$[Ca]\cdot10^4$ mg·ion/ml	$[SiO_2]\cdot10^5$ mmole/ml	pH	$[Ca]\cdot10^4$ mg·ion/ml	$[SiO_2]\cdot10^6$ mmole/ml	pH	$[Ca]\cdot10^4$ mg·ion/ml	$[SiO_2]\cdot10^6$ mmole/ml	pH	$[Ca]\cdot10^4$ mg·ion/ml	$[SiO_2]\cdot10$ mmole/ml	pH	$[Ca]\cdot10^4$ mg·ion/ml	$[SiO_2]\cdot10$ mmole/ml	pH
5 min	4.0	2.1	10.51	4.3	1.4	10.45	2.6	1.2	10.17	0.7	0.6	8.55	1.4	0.9	9.45
30 min	6.9	2.4	10.64	4.9	1.4	10.45	2.9	1.3	10.21	0.9	0.6	8.80	1.7	1.0	9.63
1 hr	7.9	2.3	10.65	7.0	1.6	10.46	4.2	1.4	10.24	1.4	1.0	9.08	2.4	1.3	9.94
3 hr	8.1	2.4	10.65	7.2	1.8	10.55	5.2	1.4	10.26	1.5	1.2	9.45	2.6	1.8	10.08
1 day	8.4	2.3	10.55	7.5	1.6	10.77	8.6	1.3	10.66	2.6	1.5	9.94	4.0	3.2	10.50
5 days	8.6	1.8	10.78	9.3	2.0	10.72	9.6	1.2	10.61	4.2	2.3	9.98	8.8	4.7	10.96

silicates[1], during the initial period is accompanied by the passage to solution of equimolar concentrations of all slag constituents. After 5 minutes of dissolution, the CaO/SiO_2 ratio in the solution and in the slag composition is equal. However, for the high-basis slags (Aktyubinsk, Izhorsky, Kirovsk), in which the CaO/SiO_2 ratio ≥ 2, this relationship changes substantially in the solution. Calcium silicates in slag with a high CaO/SiO_2 ratio pass into low-basis silicates and Ca^{2+} accumulates in the solution.

Low-basis ($CaO/SiO_2 \approx 1$) slags do not undergo any substantial changes when coming into contact with water. The sorption activity of the slags increases with an increase in the total calcium silicate content in their composition, and increases in their basicity (Table 3) and solubility. However, increases in the sorption activity of the slag are noted only

Table 3
Absorption of Zn, Cd, Cu, and Pb Ions by Various Slags and Slimes (Starting Concentrations, mg/l: 40 Zn^{2+}, 5.9 Cd^{2+}, 4.9 Cu^{2+}, 5.0 Pb^{2+}, s:l = 1.5:1000, τ = 30 min)

Slags and slimes	pH_{orig}	$[Zn^{2+}]_{con}$ mg/l	$[Cd^{2+}]_{con}$ mg/l	$[Cu^{2+}]_{con}$ mg/l	$[Pb^{2+}]_{con}$ mg/l	pH_{con}
Novo-Lipetsk slag	3.0	40	5.6	1.1	0.92	5.95
	5.3	35	5.0	2.8	1.4	6.23
	7.0	32	3.8	0.98	0.4	6.87
Kommunarsk slag	3.0	25	5.0	2.6	0.72	6.95
	5.3	25	3.4	0.45	0.14	6.85
	7.0	17	2.9	0.01	0.10	7.
Pavlodar slime	3.0	38	5.5	0.3	0.09	6.75
	5.3	32	3.8	0.3	0.14	6.62
	7.0	22	3.4	0.16	0.20	6.9
Kirovsk slag	3.0	21	3.4	0.16	0.18	6.72
	5.3	4.2	0.9	0.2	0.06	7.8
	7.0	0.1	0.1	0.01	0.05	8.6
Izhorsky slag	3.0	17	3.1	0.1	0.18	6.75
	5.3	3.8	1.0	0.1	0.10	7.7
	7.0	0.1	0.1	0.05	0.05	9.0
Aktyubinsk slag	3.0	18	3.3	0.1	0.19	6.72
	5.3	0.1	0.1	0.01	0.05	9.45
	7.0	0.1	0.1	0.01	0.05	9.55
Cherepovets slag	3.0	38	5.3	2.5	0.98	6.1
	5.3	37	48	2.1	1.30	6.30
	7.0	34	3.9	0.11	0.38	6.85
Serovsk slag	3.0	31	4.3	1.9	1.6	5.25
	5.3	30	4.2	1.8	1.2	6.28
	7.0	26	3.5	0.13	0.31	6.70
Makeevka Slag	3.0	3.0	3.9	1.3	1.4	6.35
	5.3	17	3.4	0.28	1.3	6.37
	7.0	8.0	1.4	0.03	0.05	7.0
Ural'sk slime	3.0	32	4.1	3.1	1.2	5.40
	5.3	32	4.0	1.1	0.6	6.37
	7.0	31	3.3	0.1	0.1	6.8

---

[1] K. I. Sasnaukas, Studies into Processes of Interaction and Transformation of Phases, and Questions Related to Intensification in the Production of Autoclave Silicate Articles, Abstract of Doctoral Dissertation, L., 1972.

where the CaO/SiO_2 ratio is no higher than 2. Among the examined specimens, the most active sorbent was slag of carbon-free ferrochrome (specimen 6) with a basicity of about 2 and containing about 50% CaO.

However, the presence in the solid phase of free CaO obstructs the dissolution of calcium silicate, which can be judged from the SiO_3^{2-} concentration in the solution. In this connection, there are deteriorations in the sorption characteristics of the high-basis slags from the Kirovsk and the Izhorsky plants.

A comparison of the obtained results with slag solubility shows (see Tables 2 and 3) that over a given time period (30 min) the solution has the greatest concentration of active silicic acid capable of participating in metal silicate formation. The concentration of the latter was calculated by the formula:

$$[SiO_3^{2-}] = \alpha_{SiO_3^{2-}} [H_2SiO_3],$$

where $\alpha_{SiO_3^{2-}}$ represents the share of silicate-ions at the given pH value (5, p. 50).

Table 4

Repeated Use of Slags to Purify Solutions of Mixtures of Zn, Cd, Cu, and Pb Ions (original concentrations, mg/l: 5.0 Zn^{2+}, 0.9 Cd^{2+}, 1.1 Cu^{2+}, and 0.9 Pb^{2+}; s:l = 1:1000; τ = 30 min)

Specimen	pH	No. of solution changes	Metal ion content (mg/l) in last solution change after contact with slime*				
			Zn^{2+}	Cd^{2+}	Cu^{2+}	Pb^{2+}	pH_{cont}
No. 4	3.08	3	1.2	0.4	0.13	<0.05	7.20
	4.95	6	1.9	0.4	0.1	<0.05	6.25
	7.02	6	0.4	0.2	0.2	<0.05	6.80
No. 5	3.08	2	1.9	0.3	0.1	0.05	7.75
	4.95	3	0.15	0.2	0.05	<0.05	6.25
	7.02	3	0.15	0.1	0.2	0.05	7.70
No. 6	3.08	4	2.2	0.95	0.1	0.05	7.79
	7.02	8	0.15	0.1	0.12	<0.05	7.28

*In previous shifts there was an almost complete purification of the solution: the concentration of Zn^{2+}, Cd^{2+}, and Cu^{2+} ions is to 0.01 mg/l, and of Pb^{2+} up to 0.05 mg/l.

For the Aktyubinsk, Kirovsk, Izhorsky, Pavlodar, and Cherepovets slags, these concentrations are equal to $1.6 \cdot 10^{-5}$, $4.2 \cdot 10^{-6}$, $2.1 \cdot 10^{-6}$, $2.4 \cdot 10^{-7}$, and $3.8 \cdot 10^{-9}$ mg-ion/ml, respectively.

Of the sorbents studied, the least active are the Cherepovets and Novo-Lipetsk slags with basicities of 1.0 and 1.1, and the slime from the Ural'sk Aluminum Plant with a low content of calcium oxide (about 11%). However, their use makes it possible to attain a significant reduction in the content of lead and copper ions within the entire pH range and a rather marked absorption of zinc ions at pH_{orig} = 5–7 (see Table 3).

Of the number investigated (specimens 4–6), for the most active slags the possibility was examined for repeated use (rotation) of sorbents at various pH levels for the original solution. The slag batch was mixed for 30 min with a specific amount of solution, following which the solution was filtered; a new portion of solution was then added to the slag and remixed for another 30 min. The content of metal ions in the filtrate was determined as well as the pH.

All of the examined slags can be reused repeatedly (4–8 times) to purify water from metal ions (Table 4). With repeated use there is an additional dissolution of the slag batch, and calcium ions and silicic acid again pass into the solution, governing the sorption activity. However, formation of less soluble metal silicates on the surface leads to a reduction in the sorption activity of the slags. Because of repeated slag use in the solid phase, there is a metals accumulation amounting to 4–8% of the slag weight.

Conclusions

1. The solubility and sorption activity of slags increase with an increase in their content of CaO and SiO_2 and with a CaO/SiO_2 ratio no higher than 2.
2. The sorption capacity of slags, when used repeatedly, is 4–8% of the slag weight when solutions are thoroughly purified of metal ions.

References

1. A. P. Dushina and V. B. Aleskovskii, Silica-Gel - An Inorganic Cationite, L., Goskhimizdat, 1963, 92 pp., ill.
2. V. A. Fedorov, I. P. Alekseeva, and A. P. Dushina, ZhPKh, 1973, 7, pp. 1443–1447.
3. I. N. Signalov, A. P. Dushina, and V. B. Aleskovskii, Tsvetne Metally, 1972, No. 6, pp. 27–30.
4. I. P. Alekseeva, A. P. Dushina, V. B. Aleskovskii, and V. A. Fedorov, ZhPKh, No. 7, 1973, pp. 1447–1451.
5. G. A. Laitinen, Chemical Analysis, M., Khimiya, 1966, 655 pp., ill.

Name of parameter	Variance	Scattering coefficient	Mean-square deviation	Correlation coefficient	Regression equation	
					Linear	Non-linear
Shrinkage of sinter at 1000°C, Y	38.4	41.9	6.19	0.69	$y = 7.66 - 5.93C$	$y = 17.23 - 10.56C + 51.85C^2$
Sulphur content of sinter S_s, %	0.44	20.83	0.66	0.68	$S_s = 2.46 - 0.626C$	$S_s = 2.91 - 0.1528C - 0.24C^2$
Amount of sulphur removed from the charge S_c, kg/100 kg	2.15	41.58	1.47	0.86	$S_c = 1.44 - 11.74C$	$S_c = -0.57 + 5.21C - 1.09C^2$
Desulphurisation Q^s, %	175.11	25.28	13.23	0.56	$Q^s = 40.21 - 10.09C$	$Q^s = -14.54 - 54.34 - 13.92C^2$
Maximum temperature in bed T_b	19889.1	15.11	141.03	-0.84 0.83	$T_b = 1128.2 - 162.31C$	$T_b = 1131.46 - 168.01C + - 1.77C^2$
Productivity in sinter Q_s	49.33	45.81	7.02	-0.34	$Q_s = 19.29 - 3.30C$	$Q_s = 9.44 + 13.68C - 5.34C^2$
Relative magnetism of sinter μ, rel. unit	32.44	28.01	5.69	0.89	$\mu = 11.97 - 6.96C$	$\mu = 7.04 - 15.47C - 2.68C^2$

of the results close relationships were established between the charge-recycle ratio (C) and the main parameters of the sinter roasting. The existence of a correlation was checked by the condition $r_{i,j} \neq 0$, i.e., a correlation exists if

$$|r_{i,j}| \sqrt{N - 1} \geq 3$$

and $r_{i,j} = 0$, i.e., a correlation does not exist if

$$|r_{i,j}| \sqrt{N - 1} < 3$$

where $r_{i,j}$ = the paired correlation coefficient
N = the number of data.

Table 1: The dependence of the principal characteristics of the sinter roasting of lead sulphide concentrates and ores on the charge-recycle ratio

Name of parameter	Paired correlation coefficient	Paired regression equation
Temperature in bed	-0.82	$T_b = 1138.61 + 115.43C$
Total sulphur content of sinter	0.95	$S_{tot} = 1.0 + 1.21C$
Sintering time	0.74	$t_s = 601.44 + 322.11C$
Air consumption	-0.88	$Q_a = 10.99 - 4.54C$
Magnetic permeability of sinter	-0.89	$\mu = 70.36 - 15.87C$
Specific productivity of sintering machine	0.96	$Q = 5.99 + 47C$

The data from treatment of the results of the investigation are given in table 1. The statistical model of the process has the following form

$$C = 8.937 - 0.004\,T - 0.637\,S_{tot} - 0.001\,t_s - 0.042Q_a - 1.555\mu$$

where C is the charge-sinter ratio. It allows the principal sintering characteristics to be forecast as a function of the charge-recycle ratio.

The best charge-recycle ratio (1:2.65) was determined (for existing industrial charge preparation conditions) by linear programming on a computer on the basis of the statistical model of the sintering process and the optimisation criterion (the production of maximum productivity in the sintering machines with respect to the sinter) with the use of the semiannual amount of lead production charge stacks (approximately 100,000 t. of charge), having approximately the same granulometric composition in the ore part of the charge and the recovery (+6mm, +8mm). The mathematical criterion of optimisation was determined by the requirements of the industrial conditions and is represented mathematically in the parameters:

$$Q \to Q^s_{max},\ S_{tot} \leq Q^g_{tot},\ \mu \geq \mu^g,\ T_b \geq T^g_b,\ t_s \leq t^g_s,\ Q_s \geq Q^g_a$$

where Q^s_{max} = the maximum productivity in suitable sinter
S^g_{tot} = the given sulphur content of the sinter
μ = the given magnetic permeability of the sinter
T^g_b = the given temperature in the bed on the belt of the machine
Q^g_a = the given air consumption.

The data given in table 2 were obtained during treatment of the results from investigations with back depletion of the charge. An optimum ratio of 1:1.76 was obtained by linear programming methods on an electronic computer on the basis of the statistical model of the sintering process and of the criterion of optimisation, i.e., maximum productivity with respect to the sinter in the sintering machine, with charges prepared for laboratory investigations (about 50 charges) and having a constant granulometric composition in the ore part of the charge and recycled sinter with an average grain size of +6mm (in various charge-recycle ratios).

From the obtained data it is seen that the difference between the obtained optimum values in the charge-recycled ratio amounts to 1.3 and is explained by difference in the preparation both in the ore part and in the recovery of the charge for sinter roasting. In order to reduce the amount of circulating recovered material in the sintering sections of lead plants and to obtain the optimum productivity for the sintering machines it is necessary to pay special attention to the quality of preparation of the components of the charge and its granulation.

References

1) A A Ivanov et alia: Tsvetnye Metally 1973, (2).
2) T D Sautiev et alia: Byull. TsIIN, Tsvetnaya Metallurgiya 1972, (8).

Sov. Non-Fe ~
1978 v. 6 N1

UDC 661.183

Investigation of the sorption of nickel from sulphuric acid solutions by an aminohydrazine ion-exchange resin

I P Tulenko, E I Kazantsev, V A Leitsin, I F Khudyakov and I V Samborskii (Urals Wood-Technology Institute. Department of Physicochemical Technology of the Protection of the Biosphere. Chelyabinsk Zinc Works)

Nickel, which is one of the harmful elements in the hydrometallurgy of zinc, accumulates in the cadmium cycle. To secure the production of metallic cadmium meeting the standard in respect of nickel content the concentration of the latter in the cadmium electrolyte must not be greater than 0.4 - 0.6 g/l. The normally employed cementation schemes do not finally resolve the problem of the removal of nickel from the system. Recently ion-exchange methods have been proposed for the extraction of nickel[1-4]. In the present work we investigated the sorption of nickel on a newly synthesized ampholyte containing aminohydrazine groups. (T V Pilipchuk and O P Myakisheva took part in inear

56C + 51.85C²
28C - 0.24C³

1C - 1.09C³
1.34 - 13.92C²
68.01C +
1.77C³
8C - 5.34C²

7C - 2.68C³

aving approxi-
n the ore part
). The mathe-
ned by the
is represented

Q_s^g, $Q_s \geq Q_a^g$

suitable sinter
e sinter
ty of the sinter
ed on the belt

ing treatment
lepletion of
obtained by
c computer
ntering pro-
., maximum
sintering
ry investiga-
t granulometri
l recycled
n various

ifference be-
rge-recycled
rence in the
covery of
educe the
the sintering
mum produc-
ary to pay
of the com-

, (2).
ya Metallurgiya

UDC 661.183

te.

of the removal
hange methods
kel[1-4]. In the
nickel on a
nohydrazine
took part in

---

the work). Sorption on the aminocarboxylic ampholyte ANKB-1, recommended for the purification of cadmium sulphate solutions from nickel[1]), was investigated at the same time for comparison. The ion-exchange resins were prepared by the standard procedure[5]).

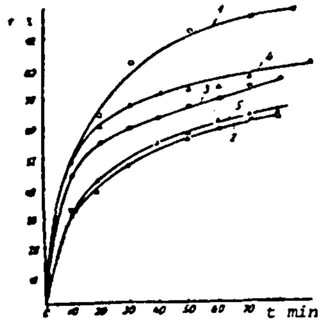


Fig.1 The degree of saturation of the ion-exchange resin (F) during the sorption of nickel by aminohydrazine resin in grains with various sizes cm: 1 - 0.0145; 2 to 4 - 0.0194; 5 - 0.0225. Temperature °C: 2 - 2; 1,3 and 5 - 20; 4 - 60.

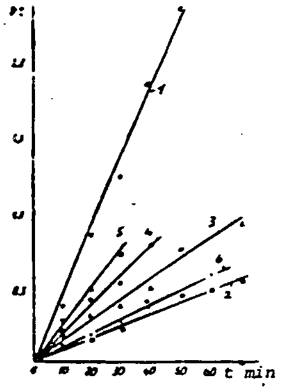


Fig.2 The dependence of Bt on time for the aminohyrazine resin with grains of various sizes cm: 1 - 0.0145; 2 to 5 - 0.0194; 6 - 0.0225; Temperature °C: 2 -2; 1,3 and 6 - 20; 4 - 40, 5 - 60.

A nickel sulphate solution containing about 0.5 g/l of nickel at pH = 4.5 was used to investigate the sorption kinetics. During sorption the solution became acid, and the final pH value was 3. The experiments were carried out in the following way. A 1-g sample of the air-dry resin was first swollen in water for 24h and then brought into contact with 100 ml of the solution. During investigation of the sorption rate the solution and the resin were agitated by means of a mechanical stirrer. At specific intervals of time the solution was analysed for nickel content. The sorption of nickel by the aminohydrazine resin in grains of various sizes is shown in fig.1. The dependence of the degree of saturation on the grain size and the practical absence of an effect from the stirrer rotation rate between 10 and 45 rpm make it possible to consider that the determining stage of the exchange is diffusion of the nickel ions inside the grain of the resin. The obtained data on the sorption rate were treated by the method described in the literature[6]). The Bt:t ratio (where B is the kinetic coefficient and t is the sorption time) proved constant up to a degree of coverage of 0.6-0.7 (fig.2). This is further evidence for the preferential significance of gel-type diffusion.

The calculated diffusion coefficients are given in table 1. The effect of temperature on the kinetics can be assessed from the variation in the diffusion coefficients (D_{av}): with increase in temperature from 2 to 60°C the value of D_{av} increased by 2.27 times. During investigation of the equilibrium we studied the dependence of the sorption of nickel on the pH and on the temperature of the solution, the nickel concentration, and the zinc and cadmium contents, i.e., the elements which are present at macroconcentrations in the industrial solutions. The results from the sorption of nickel from a solution of sulphate (1.20 and 0.48 g/l Ni) and from an industrial solution (0.74 g/l Ni, 240 g/l Cd, 40 g/l Zn) for various equilibrium pH values are shown in fig.3. In all cases the equilibrium concentration of nickel was lower and the static exchange capacity was higher for the aminohydrazine resin than for the ANKB-1 resin. With decrease in the pH value the sorption decreases, approaching a certain constant value, and this is in all probability due to nonexchange sorption of the electrolyte.

Table 1: The diffusion coefficients (D_{av}) for the exchange reactions of nickel ions on the aminohydrazine resin in the H-SO$_4$ form

Radius of grain, cm	Temperature °C	$D_{av} \cdot 10^8$ cm^2/sec
0.0225	20	0.73
0.0194	2	0.55
0.0194	20	0.91
0.0194	40	1.06
0.0194	60	1.25
0.0145	20	1.88

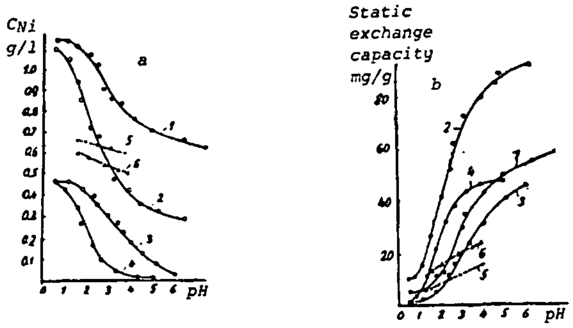


Fig.3 The residual concentration of nickel (a) and the static exchange capacity of the resin with respect to nickel (b) as functions of the equilibrium pH value of the solution during sorption by the ANKB-1 (1,3,5) and the aminohydrazine ampholyte (2,4,6). Type of solution: 1-4 - synthetic; 5,6 - industrial. Initial nickel concentration g/l: 1,2 - 1.20; 3,4 - 0.48; 5,6 - 0.74.

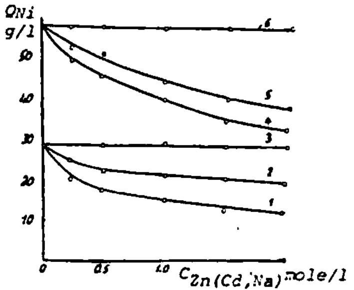


Fig.4 The amount of absorbed nickel as a function of the sulphate concentration during sorption by the ampholytes ANKB-1 (1-3) and aminohdrazine resin (4-6). Initial nickel concentration 1.2g/l, pH = 3,4. Added salt: 1,4 - zinc sulphate; 2,5 - cadmium sulphate; 3,6 - sodium sulphate.

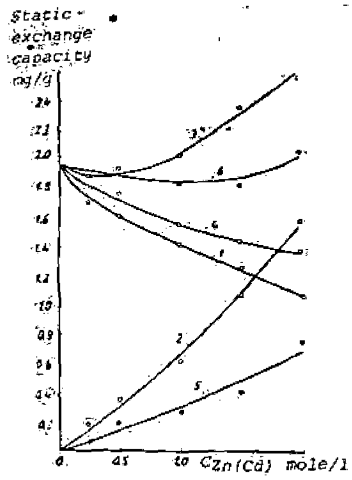


Fig.5 The static exchange capacity with respect of nickel, zinc and cadmium (mg-eq/g) on the aminohydrazine resin during sorption from zinc (1-3) and cadmium (4-6) sulphate solutions. Initial nickel concentration 1.2g/l: 1 – Ni; 2 – Zn; 3 – Zn + Ni; 4 – Ni; 5 – Cd; 6 – Ni + Cd.

Table 2: The sorption of nickel and zinc (cadmium) in their joint presence by the ion-exchange resins in the H-SO_4 form. Initial nickel concentration 1.2 g/l; Initial pH = 3.4; Amount of sorbent 1.0 g; Volume of solution, 0.1 l, Temperature, 20°C.

Resin	Zinc (cadmium) concentration mole l	Zinc solution			Cadmium solution		
		Sorbed mg-eq g			Sorbed mg-eq g		
		Nickel	Zinc	λ	Nickel	Cadmium	λ
ANKB-1	0	0.99	0	-	1	0	-
	0.25	0.83	0.19	68.4	0.85	0.07	185
	0.50	0.60	0.41	41.4	0.78	0.11	209
	1.0	0.55	0.68	44.1	0.78	0.24	190
	1.5	0.40	1.08	33.3	0.73	0.42	157
	2.0	0.40	1.57	30.5	0.73	0.56	157
Amino-hydrazine	0	1.97	0	-	1.97	0	-
	0.25	1.69	0.18	197	1.79	0.07	654
	0.5	1.57	0.33	176	1.74	0.19	389
	1.0	1.39	0.68	176	1.53	0.35	481
	1.5	1.23	1.12	113	1.49	0.38	400
	2.0	1.09	1.52	84.2	1.34	0.72	274

Increase in temperature affects the equilibrium state to a lesser degree than the process rate, slightly increasing the capacity of the resin. With increase in temperature from 20 to 70°C the static exchange capacity of the aminohydrazine resin increased from 57.6 to 64.6 mg/g, while that of the aminocarboxylic resin increased from 29.2 to 34.4 mg/g. During investigation of ion exchange from solutions containing various ions it was found that increase in the concentration of sodium ions has practically no effect on the sorption of nickel (figs.4 and 5). At the same time, a increase in the zinc and cadmium content leads to a decrease inthe sorbability of the nickel. However, in spite of the reduced capacity, the absolute value of the sorptio of nickel remains high, and the investigated resins (partic larly the aminohydrazine resin) can be used for the purification of zinc and cadmium solutions. The supposition that zinc and cadmium are also sorbed by ampholytes to some degree was confirmed by treatment of the resin aft sorption (washing with water and desorption with 4N sulphuric acid, followed by analysis of the eluate). The amo of zinc and cadmium sorbed from 1 m and 2 m solutions o their salts proved commensurable with the amount of sor nickel, while the overall static exchange capacity of the resins increased. It should be noted that zinc ions, which are sorbed in larger amounts than cadmium ions, suppres the sorbtion of nickel ions to a greater degree.

The separation factors for nickel and zinc (and cadmium during sorption from the investigated solutions, calculated by means of the equation

$$\lambda = \frac{K_{pNi}}{K_{pZn(Cd)}}$$

where K_{pNi} and $K_{pZn(Cd)}$ are the distribution coefficients of nickel, and zinc (cadmium) respectively, are given in table 2. The λ values are greater in the cadmium solution than in the zinc solutions. Compared with the aminocarboxylic resin the aminohydrazine resin is distinguished by the greater sorbability of nickel and by the selectivity of its sorption from the investigated solutions.

Conclusions

1. The aminohydrazine ampholyte possesses better sorp characteristics for nickel (a greater capacity and selectiv than the ampholyte ANKB-1.
2. The obtained data make it possible to recommend the resin with aminohydrazine groups in place of ANKB-1 for the purification of zinc and cadmium sulphate solutions fr nickel.

References

1) V N Startsev et alia: Tsvetnye Metally 1972, (11).
2) R Zh. Aidarov et alia: Author's Certificate No. 346228 Byull. Otkr. Izobr. i Tovarnykh Znakov 1972, (23).
3) B N Laskorin et alia: Author's Certificate No.457749. Byull. Otkr. Izobr. i Tovarnykh Znakov 1975, (3).
4) I P Tulenkov et alia: Tsvetnye Metally 1975, (1).
5) E I Kazantsev et alia: Ion-exchange materials, their synthesis and properties. Sverdlovsk, Izd. UPI, 1969
6) H E Boyd et alia, Collection: Chromatographic method for separation of ions. IL, Moscow 1949, p.333.

Sov. Non-Fe
1979 v. 2 N1

UDC 661.48

Reaction of various forms of aluminium hydroxides and oxides with hydro-fluorsilicic acid

O S Ignat'ev, S D Vishnevskaya and N N Korobov (Moscow Institute of Steels and Alloys, Department of the Metallurgy of Light Metals. Institute of Inorganic Materials)

Silicofluoride methods are extremely promising for the solution of problems in the metallurgy of light metals¹). On their basis an effective method has been developed for the production of aluminium fluorides for the electrolytic production of aluminium, employing byproduct fluorine from the manufacture of phosphorus fertilisers. Progressive methods have been proposed for the treatment of various forms of aluminosilicate material, and some have been successfully realised in practice. The restraining factor in the further development of silicofluoride methods for the production of aluminium fluorides is the inadequate investigation of their physicochemical princoples and technological characteristics.

The liquid-phase silicofluoride methods for the productio of aluminium fluorides and for the treatment of aluminosilicate raw material are based on the little-investigated reactions of the various forms of aluminium hydroxide and oxide with hydrofluorosilicic acid (HFSA). Published data on these questions²)³) only concern the reaction of HFSA with hydrargillite in a narrow range of temperatures (40-9 with the amounts of the initial compounds in F and Al ratio

Soo. Non-Fe... 1975 v. 3 N5

Investigation of some quaternary phosphonium compounds for the extraction of gold from cyanide media by "foamed plastic" liquid extraction

M D Ivanovskii, M A Meretukov and V D Potekhin (Moscow Institute of Steel and Alloys. Department of the Metallurgy of Heavy Nonferrous Metals)

The method of liquid extraction with the use of a porous support for the organic phase (1) secures satisfactory extraction of gold from cyanide solutions and pulps with quaternary ammonium compounds (QAC) and, in particular, solutions of trialkylbenzylammonium chloride (TABAC) in kerosene with additions of decyl alcohol[3]). Insofar as it has been demonstrated[3]) that quaternary phosphonium compounds (QPC) possess better extraction characteristics than quaternary ammonium compounds in the extraction of gold from cyanide media by normal liquid extraction, an investigation was carried out into certain representatives of this class of compound (table 1).

Table 1: Some characteristics of QPC

Extractant	Formula of compound	Number of carbon atoms in radical R	Content of main compound, %	Molecular weight
Tetraalkylphosphonium bromide (TAPB)	(R_4P) Br	5-7	98	395-507
Tetranonylphosphonium bromide (TNPB)	$((C_9H_{19})_4P)$	9	97-98	619
Tetradecylphosphonium bromide (TDPB)	$((C_{10}H_{21})_4P)$Br	10	99	675
Tetraoctadecylphosphonium bromide (TODPB)	$((C_{18}H_{37})_4P$ Br	18	97-98	1123
TABAC (for comparison)	$(R_3CH_2C_6H_5N)$ Cl	7-9	70-85	438-522

It was not possible to use TODPB on account of its limited solubility in the normally used solvents or their mixtures. The remaining compounds extract gold almost completely under the following conditions: aqueous phase, solutions of gold in KCN containing 0.05% of free cyanide and 0.01% of protective alkali (NaOH); organic phase, solutions of the extractants in purified kerosene with additions of decyl alcohol. During the investigation the support for the organic phase was granules (~5 mm) of foamed polyethylene (poropolyethylene, PPE), having 70-80% of open pores. Samples of foamed polyethylenes PPE-200 (pore diameter 5-10 μ), PPE-9 (15-20 μ), and PPE-187 (200-700 μ) were tested. On the basis of preliminary tests a sample of PPE-9 was selected for the granules, which were impregnated with the organic solutions of TDPB with various concentrations.

The full exchange capacity of TDPB and TABAC was determined under normal extraction conditions and during extraction with the porous support by successively bringing the extractant into contact with portions of fresh gold-containing cyanide solution.

The conditions for the experiments on normal extraction were as follows: Concentration of extractants, 0.01 M solution in kerosene with addition of 10 vol.% of decyl alcohol; volume ratio of organic and aqueous phases, 1:2; agitation time for each contact, 15 min.

For extraction with the porous support 0.2M solutions of TABAC and TDBP in kerosene were used. (The decyl alcohol content of the organic phase amounted to 30 and 5% respectively). The ratio between the organic and aqueous phases was 1:90. The contact time was the same.

The molar ratios (table 2) and the form of the saturation curves (fig.1) show that the best output characteristics are given by TDPB. As expected, this correlates with the theories about the differences in the chemical properties of nitrogen and phosphorus. It is due to the presence of free d orbitals with low energies at the phosphorus atom, and this makes a substantial contribution to the bonding in the formation of complexes with the transition elements.

The rates of "foamed plastic" extraction of gold with various quaternary phosphonium compounds (fig.2) were compared for volumes of the organic and aqueous phases in a ratio of 1:70 and with the addition of 5-10% of decyl alcohol. As a result of the comparison it is seen that even under conditions of limited mass exchange surface, restricted by the surface area of the granules, the use of solutions of TNPB with a moderate molar concentration, for example, makes it possible to extract more than 95% of the gold in 10 min. This is comparable with the best results obtained with quaternary ammonium compounds[3]).

Table 2: Capacity of TDPB and TABAC in respect of gold

Solution of extractant in kerosene M	Capacity in gold, g/g of extractant		(S):(Au)
	Before breakthrough	Total	
0.01 solution of TABAC	0.199	0.290	1.52
0.01 solution TDPB	0.183	0.249	1.18
0.2 solution TABAC	-	0.275	1.60
0.2 solution TDPB	-	0.258	1.23

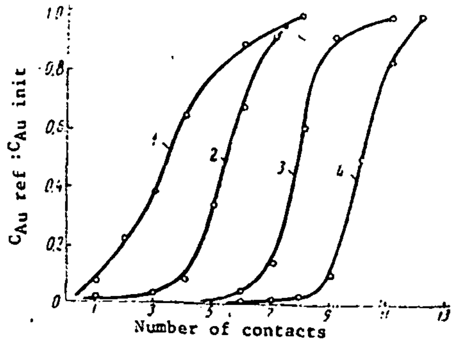


Fig.1 Output curves during saturation of extractants with gold in "foamed plastic"[1,2]) and normal [3,4]) extraction: 1) 0.02M TABAC; 2) 0.2M TDPB; 3) 0.01M TABAC; 4) 0.01M TDPB.

To i
plasti
comp
tainir
tory
was a
FeO,
0.14
was f
with

Ext
under
100%
0.135
tratio
the s
and t
The
meth
meth
Most
agita

Deco

L N
Depa

Ma
of us
How
ism
exar
have

Fe

Acc
crib

Fe

The
the
sulp

Table 3: Results from extraction leaching of a sample of the gold among the phases

	Organic phase		Aqueous phase		Solid phase		Au balance	
Time h	mg	%	mg	%	mg	%	mg	%
0.5	0.37	59.6	0.12	19.3	0.13	21.0	0.62	100
1	0.42	67.7	0.06	9.65	0.12	19.3	0.60	96.7
2	0.48	77.5	0.03	4.8	0.12	19.3	0.63	101.6
4	0.52	84.0	0.01	1.6	0.08	12.9	0.61	98.4

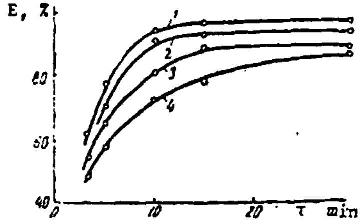


Fig.2 Rate of extraction of gold by some quaternary phosphonium compounds under conditions of "foamed plastic" extraction (M): 1 - 0.14 TNPB; 2 - 0.2 TAPB; 3 - 0.12 TDPB; 4 - 0.22 TABP; organic-aqueous = 1:90.

To investigate the possibility of the practical use of "foamed plastic" extraction of gold with quaternary phosphonium compounds the method of extraction leaching[4]) of gold-containing ore (gold content 6.2 g/ton) was tested under laboratory conditions. The chemical composition of the ore sample was as follows, %: 73.02 SiO_2, 7.89 Al_2O_3, 8.51 Fe_2O_3, 0.55 FeO, 0.45 CaO, 0.56 MgO, 0.37 TiO_2, 0.08 Na_2O, 2.93 K_2O, 0.14 S_{tot}, 0.09 MnO, 0.11 Cu, 0.02 Pb. The gold in the ore was finely impregnated ($<150\mu$) and mechanically combined with the clay fraction and iron hydroxides.

Extraction leaching was realised in 0.35-litre glass beakers under the following conditions: Weight of ore, 100 g; grinding, 100% -74μ; liquid-solid ratio 2:1; organic phase 3 ml of 0.135 M TNPB in kerosene + 10% decyl alcohol; KCN concentration 0.04%; CaO 0.02%. After the second hour of leaching the solution was strengthened in KCN, and after the first and third hours it was strengthened in CaO.

The extraction tailings were analysed for gold by an assay method (for the solid phase) and by an atomic-absorption method (for the liquid phase) on a Spektr-1 instrument. Most of the gold (~60%) is cyanided in the first 30 min of agitation (table 3). Subsequently, the gold content of the cyaniding tailings decreases much more slowly than in the aqueous phase. The final content of gold in the solid (0.8 g/ton) can be reduced by increasing the contact time (after 4 h equilibrium is not established in the system) or by increasing the number of extraction stages.

After 3 consecutive stages of extractive leaching lasting 1 h each the gold content was reduced to 0.3 - 0.4 g/ton in the solid phase and to trace quantities in the liquid phase of the tailings, and this is comparable with the data obtained for quaternary ammonium compounds[5]).

Conclusions

1. During "foamed plastic" liquid extraction of gold with quaternary phosphonium compounds a better molar ratio of extractant and metal complex (with a satisfactory extraction rate) is observed than with quaternary ammonium compounds.
2. Quaternary phosphonium compounds have greater stability against the formation of emulsions than quaternary ammonium compounds.

References

1) M D Ivanovskii et alia: Authors' Certificate No.240245, 11 May 1967. Byul. Izobretenii, 1969, (12).
2) M D Ivanovskii et alia: Izv. Vuz. Tsvetnaya Metallurgiya 1974, (2), 36.
3) B N Laskorin et alia: Authors' Certificate No.285763, Class 40a, 11/04, August 1970.
4) M D Ivanovskii et alia: Authors' Certificate No.451756, Otkrytiya Izobreteniya. Promyshlennye Obraztsy. Tovarnye Znaki, 1974, (44), 62.
5) M D Ivanovskii et alia: Tsvetnye Metally 1975, (1), 81.

UDC 669.213

United St...s Patent [19]
Larson et al.

[11] **4,222,611**
[45] **Sep. 16, 1980**

[54] IN-SITU LEACH MINING METHOD USING BRANCHED SINGLE WELL FOR INPUT AND OUTPUT

[75] Inventors: **William C. Larson; Roger J. Morrell,** both of Bloomington, Minn.

[73] Assignee: **United States of America as represented by the Secretary of the Interior,** Washington, D.C.

[21] Appl. No.: **66,998**

[22] Filed: **Aug. 16, 1979**

[51] **Int. Cl.**^{3} **E21B 43/28**
[52] **U.S. Cl.** **299/4**; 299/5; 166/268
[58] **Field of Search** 299/2, 4, 5; 166/268-272; 175/61

[56] **References Cited**

U.S. PATENT DOCUMENTS

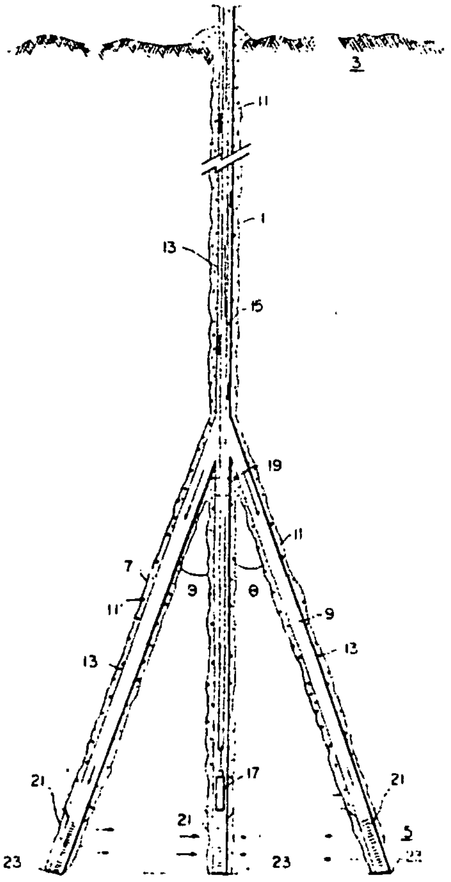
2,404,341	7/1946	Zublin	175/61 X
3,587,743	6/1971	Howard	175/61 X
4,022,279	5/1977	Driver	166/271

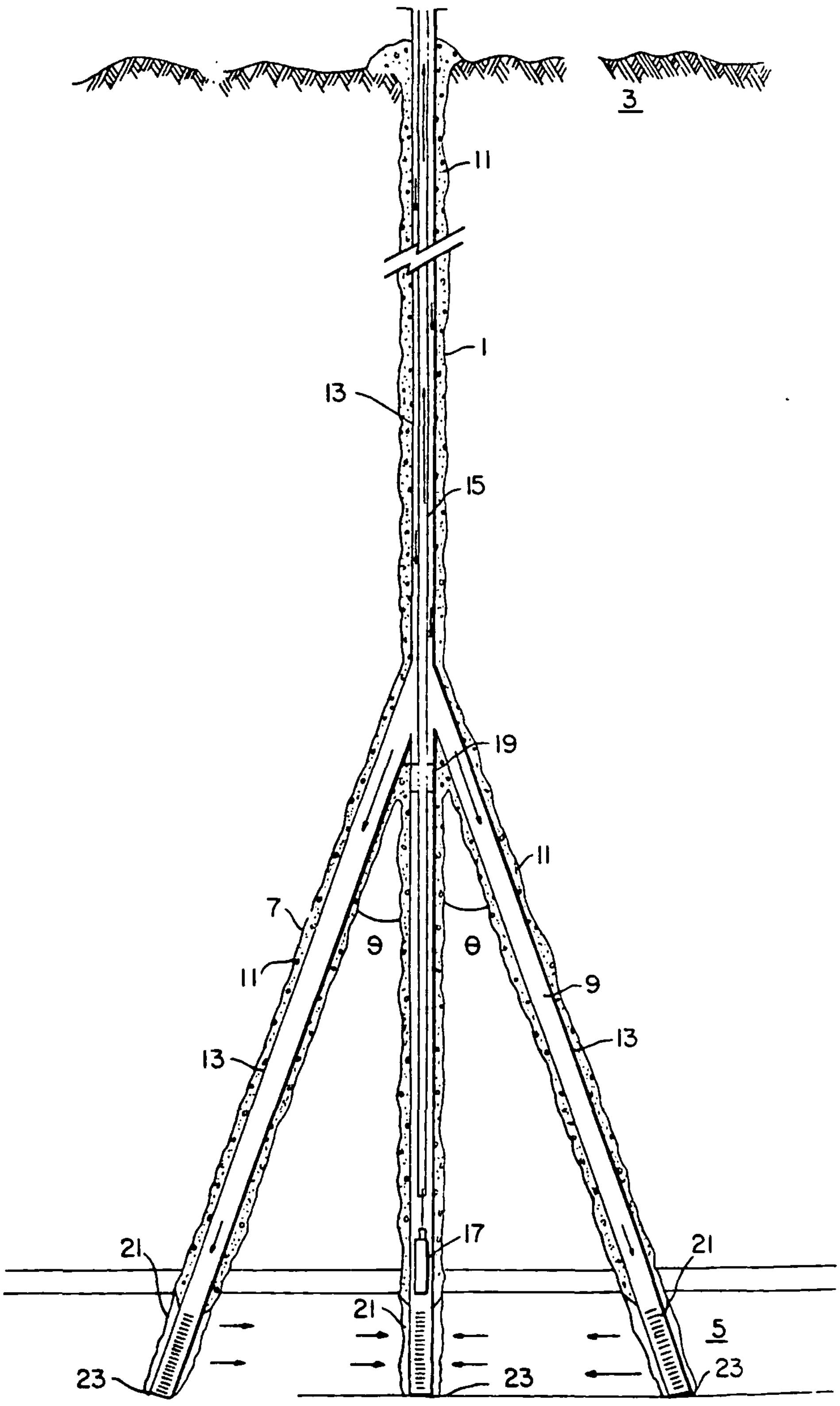
Primary Examiner—Ernest R. Purser
Attorney, Agent, or Firm—Thomas Zack; Donald A. Gardiner

[57] ABSTRACT

A method and apparatus for in-situ leach mining which utilizes multiple downhole branch wells whipstocked off a single main generally vertical well. Initially the vertical well is drilled into the earth so that it may intersect the ore zone of material to be recovered. The vertical well may be cased and cemented after drilling. Thereafter from this same well a plurality of branch wells are whipstocked or drilled which intersect the vertical well at an angle from about 2 degrees to 60 degrees. These branch wells extend into the ore zone. A submersible pump and drop pipe are placed in the main well and at some point below the intersection of the main well with the branch wells a packer is placed around the drop pipe. The drop pipe carries solutions to the surface from the pump. Leaching solutions are forced or injected in the main well in the volume between the exterior surface of the drop pipe and well hole or casing so that upon encountering the branch wells and packer their solutions are forced into the branches and out into the ore body. After exiting near the opened ends (or well screens) of the branch wells into the ore zone, the leaching solution dissolves the mineral values and the solution flow with its entrapped minerals is forced by the action of the pump through the ore zone towards the vertical well, where the solution is pumped up to the surface, via the inner drop pipe.

2 Claims, 1 Drawing Figure




3
11
1
13
15
19
11
7
11
θ
θ
9
13
13
17
21
21
21
5
23
23
23

IN-SITU LEACH MINING METHOD USING BRANCHED SINGLE WELL FOR INPUT AND OUTPUT

BACKGROUND OF THE INVENTION

1 Field of the Invention

The invention described herein relates to a method of mining which employs leach solutions to recover materials by using a single injection/recovery well.

2 Description of the Prior Art

The current basic practice in situ leach mining field which utilizes injection solutions requires the drilling of a separate hole for each of several wells. Most of these wells are usually injection wells and one or more are recovery wells. Normally these holes are drilled to depths greater than 300 feet and then each cemented and cased to the surface. For the typical inverted 5-spot or 7-spot pattern used in situ leach mining, one recovery well is surrounded by four or six, as the case may be, injection wells. Other patterns of injection/recovery wells are possible such as those described in the commonly assigned copending U.S. patent application bearing Ser. No. 60101 filed July 24, 1979, and entitled "Method of In Situ Mining," or those described in Bureau of Mines Information Circular (IC)-8777, titled "Uranium in Situ Leach Mining in the United States."

With these prior art practices there is considerable time and expense involved in the drilling, moving, and setting up of the drill rig and other associated equipment for drilling each well hole. In addition, when deep holes (more than 1,000 feet) are drilled vertical deviations become a serious problem particularly where the injection and recovery wells are less than 50 feet apart. Not only are the above problems greatly reduced or practically eliminated by using a single injection/recovery well, as we propose herein, the total footage of drilled material would be considerably less than the present state of the art 5 spot pattern since all of the separate injection wells are eliminated.

None of the known patent or other prior art is concerned with multiple completions for in situ leach mining. U.S. Pat. No. 2,171,416 (R. E. Lee) is concerned with a method of drilling wells by using angular (approximately 90 degrees) drilling at the bottom of the hole to enlarge the productive area of the well. Within these branch channels acid, other chemicals or explosive charges are used to stimulate oil and gas production by introducing them thereto. A pump may be used (column 2, lines 30-38) to introduce the acid or other chemical into the central tube placed in the main vertical drill hole. Nothing is said about using leaching solutions in an injection/recovery type of well to obtain underground minerals from an ore zone.

Other United States patents of interest include U.S. Pat. Nos. 3,223,158 (Baker), 3,941,422 (Henderson), 3,978,926 (Allen), and 4,022,279 (Driver). None are believed any more relevant than the mentioned R. E. Lee patent nor do they relate to mining using leaching solutions that employ a single injection/recovery well.

SUMMARY OF THE INVENTION

Our invention is a method with its associated apparatus of in situ leach mining in which a single well head is used to inject the leaching solution and also to recover the minerals contained in the recoverable leaching solution. The well head has a main generally vertical well and a plurality of interacting branch wells that are drilled to extend outwardly therefrom at an angle of say 2 to 60 degrees to the main well. Extending down the main well is a central conduit which receives the recoverable leaching solution and minerals. A submersible pump at the lower end of the conduit may be placed within the conduit near the mineral ore zone. A packer is placed between the pump and the branch wells around and outside of the main conduit to prevent the flow of solution pass it. The outer ends of each of the branch wells extend into the ore zone and provide for the injection of the leaching solution thereto from the surface down the main well outside of its conduit. After the solution has been forced into the ore zone, a negative pressure is built up by the pump to cause the recoverable solution and minerals to be forced to the vertical well, and finally the solution is pumped to the surface via the conduit (drop pipe).

The primary object of our invention is an improved method of leach mining.

BRIEF DESCRIPTION OF THE DRAWINGS

The FIGURE schematically illustrates in cross-section how the preferred embodiment of the invention would operate.

DESCRIPTION OF THE PREFERRED EMBODIMENT

The basic set up for the preferred embodiment is illustrated in the drawings. Initially a main drill hole 1 is drilled into the earth in a generally vertical direction down from the surface 3. Since we are herein concerned with leach mining this main drill hole could easily be 1,000 feet or more. It would normally extend into the ore zone 5 of mineral sought to be recovered by a leaching solution mining method. At some point above the ore zone, through the main first hole, an angle (θ) branch hole 7 (or holes 7 and 9) is then drilled or whipstocked to intersect the ore zone near its lowest end. Depending on the location of the ore zone and the desired operating parameters, the angle θ between the main vertical drill hole and the downwardly and outwardly extending branch hole would typically be in the range of 2 to 60 degrees from the vertical. And the branch drill hole segment would normally extend 25 to 200 feet from its intersection with the main well depending, of course, on the location of the ore zone. The lower end of each branch hole is so constructed that leaching solution injected from the surface, at about 100 psi., will exit therefrom into the ore zone. As is conventional in the drilling art each of the holes (three shown) are each cemented 11 and cased 13 to the surface. A slotted screen 21 or pipe with a cap for the well's end may be utilized near the ends of the branch wells and the main well to allow the solution to exit.

After the main drill hole and the branch hole (or holes) are drilled to complete the well head, a hollow drop pipe 15 with an electrically operated submersible pump 17 at its lower end are placed in the main vertical drill hole so that the pump extends near or into the ore zone. Within the main drill hole below the intersection of the branch wells a packer 19 is placed around the hollow drop pipe to act as a barrier to prevent the flow of leaching solution with or without mineral in it from moving past that point from below or above, respectively.

Several of the components disclosed in the drawings are conventional off the shelve items in common usage.

These would include the drop pipe, casing, cement for the casing, pump, packer, and the screened or slotted capped end for each well. Further, explanation as to their exact embodiment is believed unnecessary as their composition and function(s) are well known to those skilled in the leach mining art. A good example of the type of components that could be used can be found in the Bureau of Mines IC-8777 titled Uranium In Situ Leach Mining in the U.S., by W. C. Larson 1978. The operation is self-evident. The particular leaching solution selected for the particular mineral sought to be recovered is injected down the main hole from the surface. This is done in the annular volume located between the exterior surface of the drop pipe and the well casing. Pressure ranges from 0 to 100 psi. would normally be employed so that the solution moves in the direction of the arrows as shown in the drawings. Upon exiting from the screened or slotted sections near the ends of the branch wells, the solution would permeate the ore zone. Concurrent or shortly after the injection step takes place, the electric pump 17 is made operative to cause a negative pressure zone to develop around the end of the main drill hole. This pressure differential causes solution with minerals to move from the branch wells towards the slotted screen and pump and to the pipe. Eventually it moves up the inside of the drop pipe to the surface in the direction of the arrows. Thereafter further conventional processing is employed to extract the uranium, copper, gold, silver or other minerals from the recoverable solution depending on the type of mineral(s) being leached.

It is important to note that our invention is specifically adopted for use with leach mining and the various patterns of injection/recovery wells commonly being used. These would include the five spot pattern, the seven spot pattern, the Utah Construction and Mining Co. pattern, the Ore body configuration pattern, the Multiple five-spot and the Multiple seven-spot pattern. The United States Bureau of Mines Information Circular (IC) 8777, dated in 1978, on page 68 authored by co-inventor William C. Larson entitled "Uranium In Situ Leach Mining in the United States" illustrates these common injection-recovery well patterns. Our single injection/recovery well heretofore disclosed can be used in these same field patterns to provide for the injection and recovering of leach solutions at each well. This would allow the same ore body well patterns used by industry to be retained. Commonly each of our wells would have a spacing between themselves of 15–200 feet that would include both the main vertical well and the branch wells.

Drilling or whipstocking the branch well at the angle from the main drill hole may be accomplished by using the well known down-hole deviation and/or conventional whipstock drilling techniques. These type of drilled angled branch wells are per se well known and by themselves do not consitute novelty. However, the prior art is silent as to this type of drilling technique as applied to leaching solutions. Further the prior art does not disclose these techniques being employed in a single injection/recovery leaching solution mining well.

One of the most important benefits or our invention over the prior art is the reduction of materials and related costs to accomplish the same objectives as the present state-of-the art techniques. For example, assuming a typical 5-spot pattern is used employing four injection wells placed on four corners of a square with the center well (of the square) being the single recovery well. If each well were 2,00 feet deep then the total footage drilled for this mine leaching solution pattern would be 10,000 feet at a cost of X dollars per foot times 10,000 feet at a cost of X dollars per foot times 10,000 or $10,000 X. With our present invention we propose one well (the counterpart to the center well) would be drilled to a total depth of 2,000 feet. Four angled holes would then be drilled therefrom at about 250 feet each through the ore zone making a total branch footage of 1,000 feet or 3,000 feet for the total well head. This would be 7,000 feet less of drilling than the 5 spot pattern and a cost saving of about $7,000 X for drilling. Some of these apparent drilling cost savings would be lost due to the more sophisticated drilling equipment required, nevertherless, the savings would be very real and substantial.

It would be apparent that our invention allows deep leaching wells to be effective and more closely spaced together than the prior art. Thus it would find particular application to those situations where closely spaced bore holes are required (Low Permeability Rocks). Also it could be used for a variety of mineral commodities in for example small isolated ore bodies. Besides the mentioned mining uses, our invention could also be used for methane drainage where the requirement is to maximize the amount of drill hole in the coal seams itself. Placing 4 or 5 holes in the coal seam would greatly improve the drainage of methane gas with only a small increase in drilling costs.

Depending on the type of minerals sought to be recovered and the compositions of the ore zone, many different types of leaching solutions may be used. Typically for uranium these solutions are made weak acids or alkali solutions made up of 0.5 to 5 grams of an additive, such a acid or alkali, per liter of water. However, many types of leach solution can be used and as such is not a limitation of the proposed invention. Normally about 30–2000 parts per million of recoverable minerals are in the solution pumped to the surface. The method disclosed herein is contemplated as being practiced continuously, once started, until the concentration of the recoverable minerals in the leaching solution falls below some preestablished level. Other variations are, of course, possible. None should be used to limit the scope and extent of our invention which is to be measured only be the claims which follow.

We claim:

1. An in situ leaching mining method employing a combined injection/recovery well for obtaining materials from an ore zone comprising the steps of:

(1) drilling a generally vertical main injection and recovery well so that it is in communication with the ore zone of recoverable minerals;

(2) drilling at least one outwardly extending branch well from said main well at an angle between 2 to 60 degrees from the vertical, the lower ends of said at least one branch well being in fluid communication with and extending into the recoverable ore zone;

(3) inserting a conduit having a lower pump unit into the main well, said pump being capable of recovering a leaching solution that passes through the ore zone and forcing it to the surface;

(4) blocking off the volume between the conduit and the main well below where the at least one branch well intersects it; and

(5) injecting a leaching solution down into said vertical well outside of its conduit to force solution to and through said at least one branch well to the ore zone whereby the pump will cause at least part of the solution with its recoverable minerals to be moved to the surface.

2. The method of claim 1 whereby drilling step (1) is at least 300 feet and drilling step (2) at least 25 feet.

* * * * *