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Separation and Recovery of Metals from Dilute Solids and Aqueous Phases

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ABSTRACT

Present practice for the extraction of metals from low grade dilute ore sources and the recovery of metal values from dilute solutions is reviewed. Special problems associated with extraction from both solid and aqueous phases for copper, gold and silver recovery are considered. Emphasis is directed toward the application of insitu solution mining technology for recovery of metal values in the future.

INTRODUCTION

The application of hydrometallurgy to the recovery of metals from dilute ore sources is increasing in use around the world. This results from the fact that hydrometallurgy may be used for certain metals without excessive size reduction. Such ore sources may be of sufficiently low value that extensive preparation is not economically feasible. For example the massive dump leaching of low grade open pit copper waste rock is common practice in the western United States and can only be justified economically because the waste rock can be placed on a dump in the as-mined condition and leached using recycled solutions. In addition to dump leaching, heap leaching and vat leaching of selected precious metal and oxidized copper ores is employed commercially.

LEACHING PRACTICE

In this discussion only very dilute ores, too low in value for preparation by fine grinding, will be considered. Examples of such ore sources are copper, gold and silver and uranium. Copper has been and

TABLE 1: TYPICAL ORE AND SOLUTION GRADES

OPERATION	GRADE OF SOLIDS	GRADE OF SOLUTION	METHOD OF RECOVERY
Copper Dump Leaching	<0.2% Cu <2000 g/mt <4 lb/st	∿0.1% 1000 g/mt 2 lb/st	Cementation or Solvent Extraction
Heap Leaching of Gold	$9X10^{-5}$ - $4.2X10^{-4}$%Au 0.9 - 4.2 g/mt 0.3 -0.13 oz/st	$6X10^{-5}$ - $3.8X10^{-4}$% 0.6 - 3.8 g/mt 0.02 -0.12 oz/st	Adsorption on C, Elution, Electrolysis, Precipitation
Insitu Extraction of Uranium	.05 -.5% U_3O_8 500 - 5000 g/mt 1.0 - 10 lb/st	$5X10^{-4}$ - 0.02% 5-200 g/mt .01 - .04 lb/st	Anion Exchange Resins, Elution, Precipitation

currently is recovered by dump and insitu leaching. Insitu recovery of gold and silver is not practiced but heap leaching on specially prepared pads is now common practice in the western United States. Uranium prior to the current economic down trend, enjoyed increased activity in insitu extraction from secondary roll-front formations. There has been only limited application of heap leaching for uranium recovery. Table 1 illustrates typical ore and solution grades and methods employed for metal recovery. The grade of solids in metal content in g/mt (ppm) ranges from less than 5 (0.2 oz/st) for gold to a few hundred or thousands for copper and uranium. The solution grades produced are even lower in metal content.

PRACTICE

Hydrometallurgy in general provides unique opportunities for extraction of metal values from low to medium grade ores. Applied to base metal concentrates, hydrometallurgy has not replaced modern smelting practice and will not likely do so in the near future. Compared to smelting, hydrometallurgy is more energy intensive and has its own array of environmental problems. For low grade materials hydrometallurgy is able to effect separations affordable by no other means. Figure 1 illustrates flow paths for low, medium and high grade ores. Normally ore is transported prior to leaching. Insitu leaching is practiced for copper and uranium recovery and holds great promise for the future economic extraction of several metal commodities. Examples of low grade ores are those of copper, gold, silver and uranium. Here the ore is not suitable for beneficiation and is leached in the as-mined condition. Direct leaching ores are of sufficiently high value that they can be crushed and subjected to heap, vat or agitation leaching. In some cases pretreatment by oxidative, reductive or salt roasting may be necessary for effective leaching. High grade ores have sufficient metal value that beneficiation, producing a

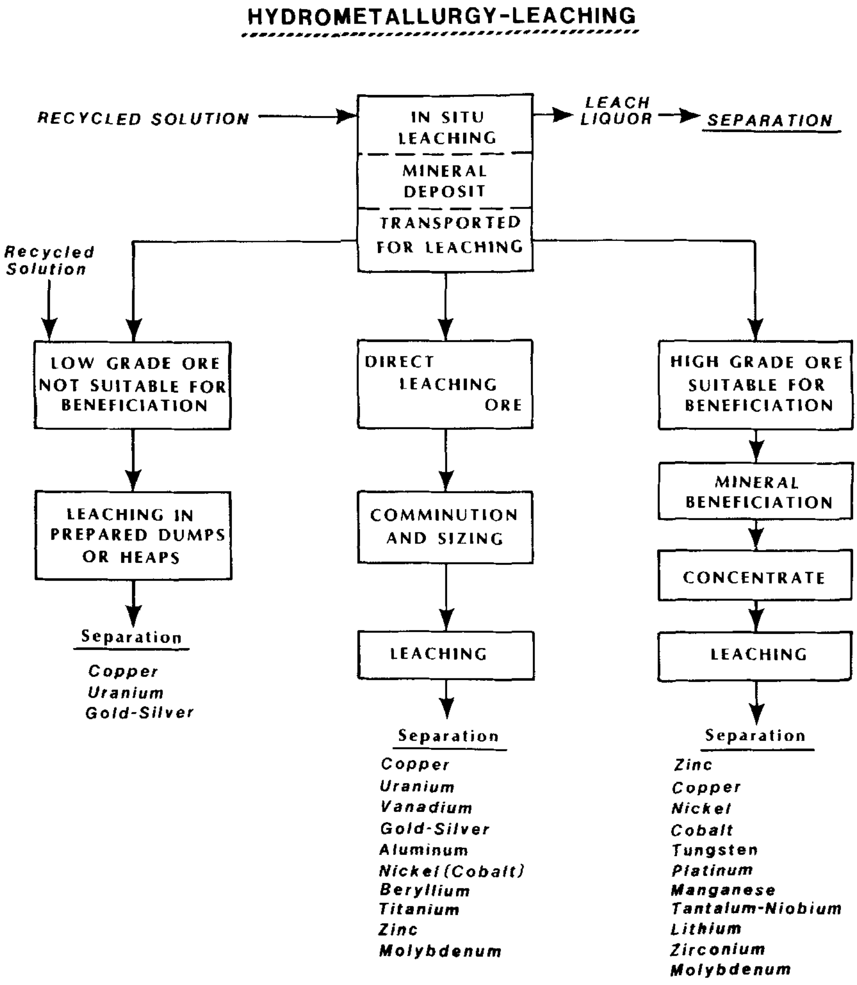


Figure 1: Ore Management in Commercial Application of Hydrometallurgy

concentrate, may be employed economically prior to leaching. The concentrates may be pretreated by roasting prior to leaching.

The treatment of low grade ores is the subject of this discussion because the hydrometallurgical treatment of these ores represents an excellent example of separation from dilute solid and liquid phases. The leaching of dilute ores produces dilute process streams which are usually up-graded prior to metal recovery. Since low grade copper, gold and silver ores are currently receiving much attention, examples of dump and heap leaching of these metals will be presented. Several recent books and symposia have focused on this subject (1-6). Figure 2 illustrates the various separation steps employed in treating dilute process streams of copper, gold and silver. Direct recovery of copper may be and is achieved commercially by cementation on iron. The trend is toward the use of solvent extraction and electrowinning. Silver bearing solutions normally must be stronger than gold to compete economically. Silver and gold leach liquors may be treated by the Merrill-Crowe process where the metals are precipitated by cementation on zinc dust. This method is favored for silver recovery. In the case of heap leaching of gold, the trend is to adsorb the soluble gold cyanide on activated carbon followed by elution and electrowinning.

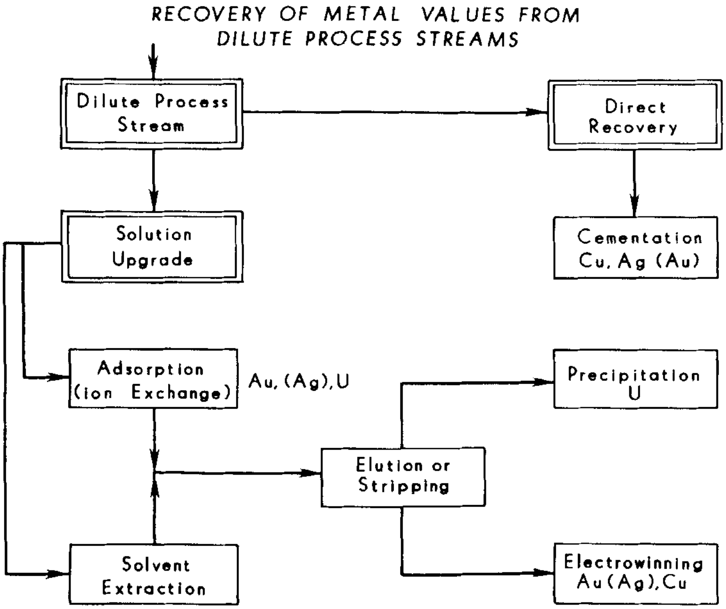


Figure 2: Management of Dilute Process Streams from Dump and Heap Leaching

The recovery of copper by dump leaching is classic in the sense that the extraction process is based upon the treatment of unusually great tonnages of low-grade copper-bearing waste material with large volumes of low-grade leaching liquors. For example, at Kennecott's Bingham mine, during operation, approximately 250,000 tons of waste rock are placed on the leaching dumps each day. Fifty million gallons of leach solution is applied daily, producing effluent solution grades of approximately 0.5 gpl. The copper waste going to leaching dumps is less than 0.2 percent copper. In a typical open-pit mining operation with a stripping raio of 3:1, an average head grade of 0.6 percent Cu, and a cut-off grade of 0.2 percent, approximately equal quantities of copper will go to the waste dump and to milling for recovery by conventional flotation technology (7).

The percentage of copper produced by leaching will probably increase in the future as the average grade of major deposits decreases. The distinction between ore and waste will require an even more definitive assessment as energy costs for milling increase (8), probably favoring a greater percentage production of copper by leaching. As an open-pit operation continues, the stripping ratio will increase to the point that underground mining or some other alternative must be considered if increased recovery is to be achieved. As a rough estimate, approximately an equal amount of copper will remain, after conventional mining, in low- to medium-grade zones including regions of "halo" mineralization, deep-seated medium-grade ores, and unleached residues in waste dumps. This is a worthy target for extraction by dump leaching and insitu solution mining.

In practice, dump leaching depends upon a sequence of processes. Within the dump three conditions are essential for leaching to occur and continue. These are:

- Effective air circulation
- Good bacterial activity
- Uniform solution contact with the particle

The major unknowns in dump leaching for any given dump are:

- A knowledge of air circulation relative to the dump configuration
- The hydrology in terms of channeling and by-pass
- The effect of fines and precipitated salts
- Effect of weathering as a function of time

Figure 3 illustrates the general flow of solution to the dump and to a holding basin. Solutions are introduced on the surface by one of several methods, most commonly ponding, trickle leaching or sprays (6). Ponding is used but the trend is to the use of trickle leaching or sprays. In ponding, channeling can cause excessive dilution of effluent. Trickle leaching is carried out by using a network of perforated PVC pipe, as shown in Figure 3, or spraying using low pressure multiple sprays or single high pressure sprays. Trickle and spray application provide uniform controlled application of leachant. Sprays suffer highest evaporation loss followed by trickle application. During winter months, ice formation may require alternate methods of

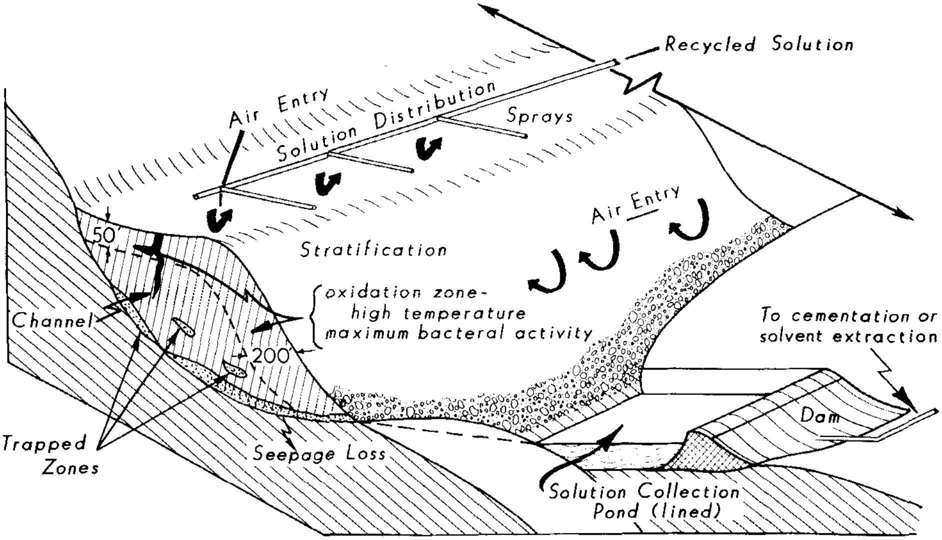


Figure 3: Cross Section of Copper Leach Dump Illustrating Solution Management

solution application. As shown in Figure 3, solutions from the toe of the dump flow down natural channels to a collection pond. Solutions are then pumped to separate circuits for copper extraction. Copper extraction is achieved either by cementation on de-tinned scrap iron or by solvent extraction using one of the LIX reagents for selective removal of cupric ion from sulfate leach liquors. The general trend is toward solvent extraction due to the high cost of iron scrap. Schlitt (7) has indicated that the operating costs for solvent extraction are less than those for cementation although the capital costs may be higher. Also, solvent extraction-electrowinning produces a marketable copper cathode.

Following extraction, solutions are recycled or enter a containment pond where some aeration occurs. It should be noted, however, that the iron balance for the greater part is achieved by precipitation of iron salts throughout the dump itself. In general, intermittent leaching with alternate leach and rest cycles is preferred to continuous leaching. This practice conserves energy consumed in pumping and is effective since pore leaching continues during the rest period, under conditions of good aeration, building up dissolved metal values in the contained liquid phase. Continuous leaching without the rest cycle, because of the large volume of water, extracts large quantities of heat from the dump (up to one-half of the exothermic heat of reaction) adversely affecting leaching rates (9).

Expected recovery from massive dumps is not known. Steady-state daily recovery, the fraction of the daily tonnage of copper added to a dump which is recovered by leaching, may fall in the range of 15 to 20 percent. For example, at Kennecott, Bingham, 250,000 tons of ore (0.17 percent copper) goes to the waste dump daily. This ore contains approximately 850,000 lbs. of copper. Under best operating conditions, 150,000 lbs. or 18 percent of the above amount is recovered daily from the total dump.

Several important features contribute to successful copper dump leaching. Leaching takes place by diffusion of an oxidizing lixiviant into the pore structure of the ore (mineral matrix) fragment. This is the pore structure which allowed penetration of metal laden magmatic waters from which mineral crystallites or veinlets precipitated during ore genesis. Subsequent changes such as cracking and fissuring, oxidation and supergene enrichment may also have occurred. These changes can alter the mineral types present, both in the gangue mineral matrix and in the metal containing minerals, and influence the leachability significantly. Oxygen solubility at dump leaching temperatures is less than 7 ppm, too low to account for observed rates of leaching. Leaching takes place by diffusion of ferric ion complexes, FeIII, into the ore matrix pore structure. Ferric ion is an effective leachant for copper and iron sulfide minerals. Bacterial oxidation of ferrous iron to ferric ion (FeIII) in solution is essential to maintain useful steady state leaching rates. The rate of oxidation of FeII to FeIII in the absence of bacteria is far too slow, by orders of magnitude. The bacterium Thiobacillus ferrooxidans, an aerobic chemoautotroph, derives its energy from the oxidation of ferrous ion to ferric and consumes oxygen from the surrounding atmosphere. Consequently, good aeration is essential to the process. Other bacteria also contribute to the oxidation steps observed. The bacterium Thiobacillus thiooxidans as well as Thiobacillus ferrooxidans oxidize elemental sulfur, thiosulfates and sulfide ions resulting in the insitu generation of sulfuric acid, an essential feature in maintaining open porosity. Pyrite (FeS_2) is a strong acid producer and also supplies ferrous iron which is subsequently oxidized to the ferric state. Pyrite greatly influences the consumption of oxygen which may often be as high as 7-20 moles of oxygen per mole of Cu^{2+} produced. The accumulation of FeIII in solution results in the hydrolytic precipitation of ferric iron salts which buffers the solution and, for a fixed oxygen potential, maintains a steady state FeIII concentration.

Figure 4 illustrates an ore mineral matrix fragment. Air circulation provides oxygen which transfers to the solution film wetting the fragment surface. At the surface bacteria oxidize ferrous iron in solution to FeIII. Under conditions of adequate insitu acid generation, porous iron salts precipitate but do not passivate the surface. The FeIII ionic species, made up of complex ferric sulfates and hydroxides plus free ferric ion, diffuse into the pore structure and react with minerals within the pore network. A steady state diffusion gradient for FeIII inward diffusion is generated. Released Cu^{2+} and Fe^{2+} similarly develop steady state diffusion gradients for outward diffusion. At the surface, ferrous iron is oxidized to ferric. During the rest period, copper accumulates in nongravitational water in capillaries, pores and interstices and iron salts continue to form.

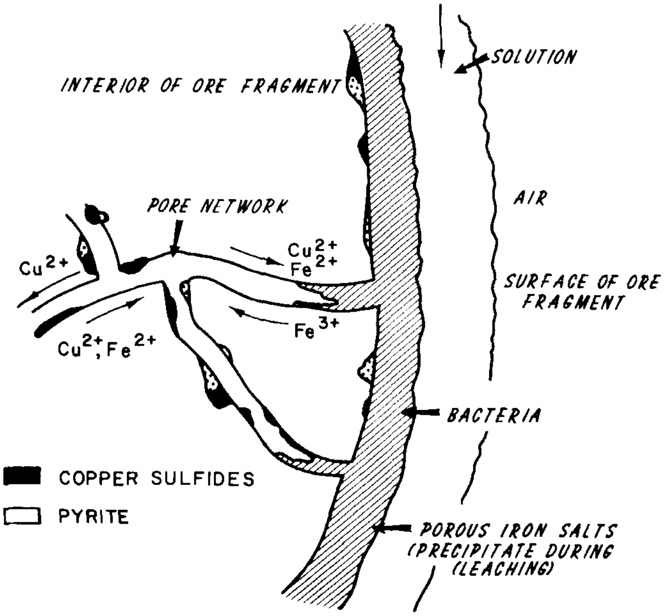


Figure 4: Ore Matrix Mineral Fragment Showing Pore Network, Diffusion of Ions and Action of Autotrophic Bacteria

During the leach (flush) cycle, when solutions are applied to the dump, the copper contained in the nongravitational water is removed with the flowing solution.

Within the matrix mineral fragment, copper bearing minerals and pyrite are leached. These reactions are shown below for chalcopyrite ($CuFeS_2$) and pyrite (FeS_2),

$$CuFeS_2 + 4Fe^{3+} = Cu^{2+} + 5Fe^{2+} + 2S° \qquad (1)$$

$$FeS_2 + 14Fe^{3+} + 8H_2O = 15Fe^{2+} + 2SO_4^{2-} + 16H^+ \qquad (2)$$

The pyrite is a strong acid former and serves an important role in maintaining open porosity and providing ferrous iron. In regions of higher oxidation potential, near surfaces exposed to adequate aeration, bacterial growth is promoted by the ferrous iron and sulfur nutrients generated by the leaching process. Bacteria promoted reactions are,

$$4Fe^{2+} + O_2 + 4H^+ \xrightarrow{bac} 4Fe^{3+} + 2H_2O \qquad (3)$$

$$2S° + 3O_2 + 2H_2O \xrightarrow{bac} 2SO_4^{2-} + 16H^+ \qquad (4)$$

Precipitation of iron in regions of high oxygen potential serves to remove iron from the leach liquors continuously,

$$2Fe^{3+} + 3H_2O = Fe_2O_3 + 6H^+ \tag{5}$$

$$3Fe^{3+} + 2SO_4^{2-} + 7H_2O = H_3OFe_3(SO_4)_2(OH)_6 + 5H^+ \tag{6}$$

Reactions 5 and 6 illustrate the precipitation of hematite, Fe_2O_3, and hydronium jarosite, $H_3OFe_3(SO_4)_2(OH)_6$. In practice many oxide, hydrated oxide and jarosite iron salts are precipitated within the dump. An active leaching environment produces enhanced weathering of the matrix fragment increasing the zone of high oxidation and exposing sulfide minerals and sulfur to direct attack by bacteria. The weathering action increases the rate of leaching until decrepitation and formation of altered gangue minerals decreases porosity.

Copper values are recovered from leach liquors by cementation or solvent extraction. The cementation is carried out using de-tinned iron scrap. The reaction is

$$Cu^{2+} + Fe^{\circ} = Cu^{\circ} + Fe^{2+} \tag{7}$$

Other reactions take place which increase iron consumption by a factor of 2 to 5,

$$Cu^{\circ} + 2Fe^{3+} = Cu^{2+} + 2Fe^{2+} \tag{8}$$

$$Fe^{\circ} + 2Fe^{3+} = 3Fe^{2+} \tag{9}$$

$$Fe^{\circ} + 2H^+ = Fe^{2+} + H_2 \tag{10}$$

Recycled solutions from cementation increase the iron load which must be continuously precipitated along with the iron extracted by leaching. The cost of iron and the increased iron load contribute to the trend to solvent extraction for copper recovery.

The introduction of the LIX, hydroxyoximes, solvent extraction reagents by General Mills, Inc., (10) lead the way to successful application of solvent extraction for copper removal from low grade leach liquors. The principal extractant used commercially is LIX 64N which is a mixture of LIX 65N, 2-hydroxy-5-nonylbenzophenone-oxime with about one volume percent LIX 63, 5,8-diethyl-7-hydroxy-6-dodecanone-oxime. The LIX 63 enhances the rate of extraction. These reagents are specific for copper, forming stable 5-member ring structures which can be readily stripped. The extraction and stripping reactions are,

$$\text{extraction} \longrightarrow \qquad\qquad \longleftarrow \text{stripping}$$

$$2R\text{-}H\ (LIX)\ (org) + Cu^{2+}(aq) = R_2\text{-}Cu\ (org) + 2H^+(aq) \tag{11}$$

Spent electrolyte is used as the aqueous stripping solution providing up-graded tank house feed containing 35-60 gpl of copper (10). Electrowinning is employed for the production of copper cathodes. The reactions are,

$$Cu^{2+} + 2e = Cu^{\circ} \qquad \text{(cathode)} \qquad (12)$$

$$H_2O = 0.5O_2 + 2H^{+} + 2e \qquad \text{(anode)} \qquad (13)$$

The cathodes can be marketed whereas cement copper normally is returned to the smelting operation. Electrowinning has two major disadvantages. The cathodes are not of the purity obtainable by electrorefining and the high overpotential needed for oxygen evolution makes the operation energy intensive (8). Electrowinning requires approximately 24 million BTU per ton of cathode copper, approximately 8 times that required for electrorefining.

Heap leaching of gold is carried out on specially prepared leaching pads normally constructed of layered and packed clays or asphalt. Figure 5 (11) illustrates a heap on an impervious pad showing solution collection and management. Solutions are pumped to precious metals recovery and barren solutions are returned to the leaching step. Heap sizes vary from small heaps of the order of 50 to 60 meters on a side and depths of 3 to 4 meters to heaps sizes of the order of 150 X 450 meters with depths of the order of 6 meters (1). Usually the ore is crushed prior to placement on the leach pad and is typically -3/4

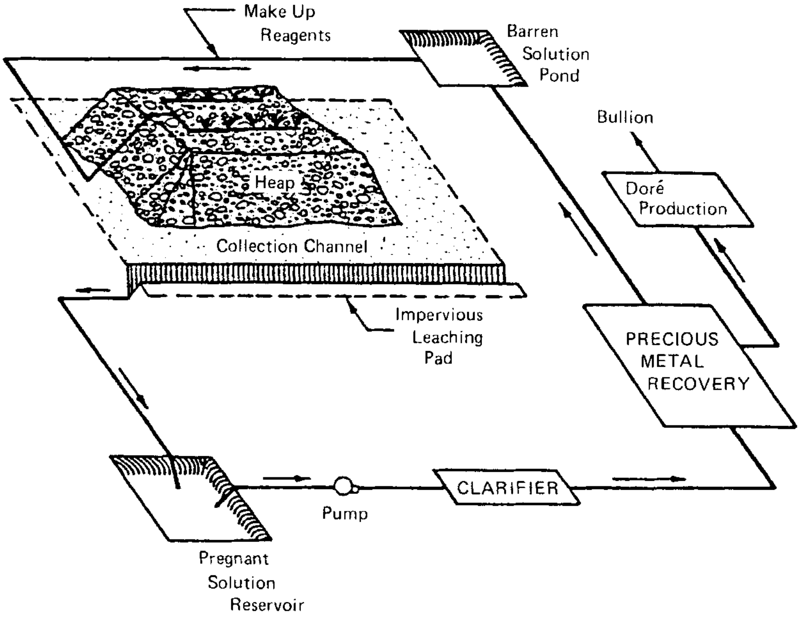


Figure 5: General Plan of Gold or Silver Heap Leaching Operation (11)

inch. Dump leaching of run-of-mine ore is carried out for some gold ores using specially constructed pads. The maximum particle size may be several inches with dump heights up to 10m or even higher in some cases. Solutions are usually added by spraying or by the use of wobblers with a rate of application of approximately 10 l/m^2-hr (0.25 gal/ft^2-hr). This is similar to the rate of application for copper dump leaching (9). Low grade gold cyanide solutions are usually treated by passing the solution through a bed of activated carbon in cones or columns where the gold cyanide complex adsorbs on the carbon. Elution is achieved by stripping with hot solutions containing approximately 1% NaOH and 0.1% NaCN. Stripping temperatures vary between 70°C and 130°C. Autoclave stripping carried out at the higher temperatures and pressures up to 25 psi provides the most rapid removal but still requires 6 to 9 hours. At lower temperatures stripping of the carbon may require as much as 50 hours (11). If alcohol (methanol or ethanol) is added to the strip solution, adequate stripping may be completed in one half the time at 75°C to 85°C. Figure 6 (1) illustrates a carbon adsorption cone. Several cones are operated in series. The carbon (∿16X30 mesh) (1) is fluidized by the solution which enters the bottom of the cone or column. Solution flows of 600 to 1000 liters per minute per square meter are required.

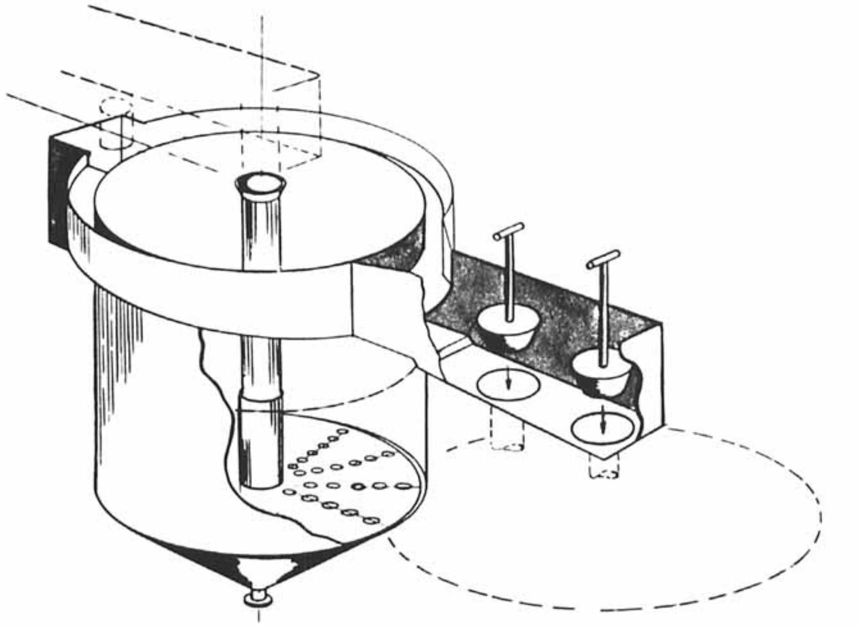


Figure 6: Carbon Adsorption Cone Showing Carbon Bypass Feature (1)

Periodically the carbon is advanced from one stage to another by eductors countercurrent to the solution flow. Maximum loading is approximately 6,800 g/mt (200 oz/st) to prevent excessive solution loss. An increment of carbon is removed from the system for stripping and reactivation before returning to the circuit. Stripped carbon is reactivated by heating in a kiln at 600-800°C. Figure 7 illustrates a flowsheet for gold recovery by this process (11).

The leaching of gold is an electrochemical process deriving energy from oxygen in air. For free leaching gold, the solubility of oxygen is sufficient to meet the stoichiometric requirements. The reaction is

$$2Au^{\circ} + 4CN^{-} + 0.5O_2 + 2H_2O = 2Au(CN)_2^{-} + 2OH^{-} \qquad (14)$$

The leaching rate is limited by the activity of oxygen. Therefore good aeration is necessary and is a major factor in limiting heap dimensions. Adsorption and stripping may be represented by the ion exchange reaction

$$\text{adsorption} \longrightarrow \qquad\qquad \longleftarrow \text{stripping}$$

$$C\text{-}OH(\text{surface}) + Au(CN)_2^{-}(aq) = C\text{-}Au(CN)_2(\text{surface}) + 2OH^{-}(aq) \qquad (15)$$

Equation 15 is idealized in assuming a simple ion exchange process. The CN^-/Au ratio on the surface is observed to be less than 2:1, probably indicating partial reduction. Thus stripping is slow and requires CN^- to reconstitute the soluble complex. Gold recovery from the stripped solution by electrowinning is carried out by employing stainless steel anodes and steel wool cathodes according to the reaction

$$Au(CN)_2^{-} + e = Au^{\circ} + 2CN^{-} \qquad (16)$$

Leaching solutions in the case of silver are of higher strength and may be treated directly by the Merrill-Crowe process. Deaerated solutions are treated with zinc dust precipitating silver and gold from solution,

$$\underset{(Au)}{2Ag(CN)_2^{-}} + Zn^{\circ} = \underset{(Au)}{2Ag^{\circ}} + Zn(CN)_4^{2-} \qquad (17)$$

SOLUTION MINING

Solution mining refers to the hydrometallurgical extraction (leaching) of selected minerals or mineral components from original matrix minerals. The original ore may be transported to a suitable site for leaching or may be leached in place. The leaching chemistry may involve nonoxidative dissolution, as in solution mining of salts, or may require oxidation, as in dump leaching of copper. Insitu solution mining refers to the leaching of an ore deposit in place. The deposit may be leached without fracturing if it has sufficient permeability. Solution mining of uranium is an example. In other cases porosity may have to be induced by hydrofracting, explosive rubblization or block caving.

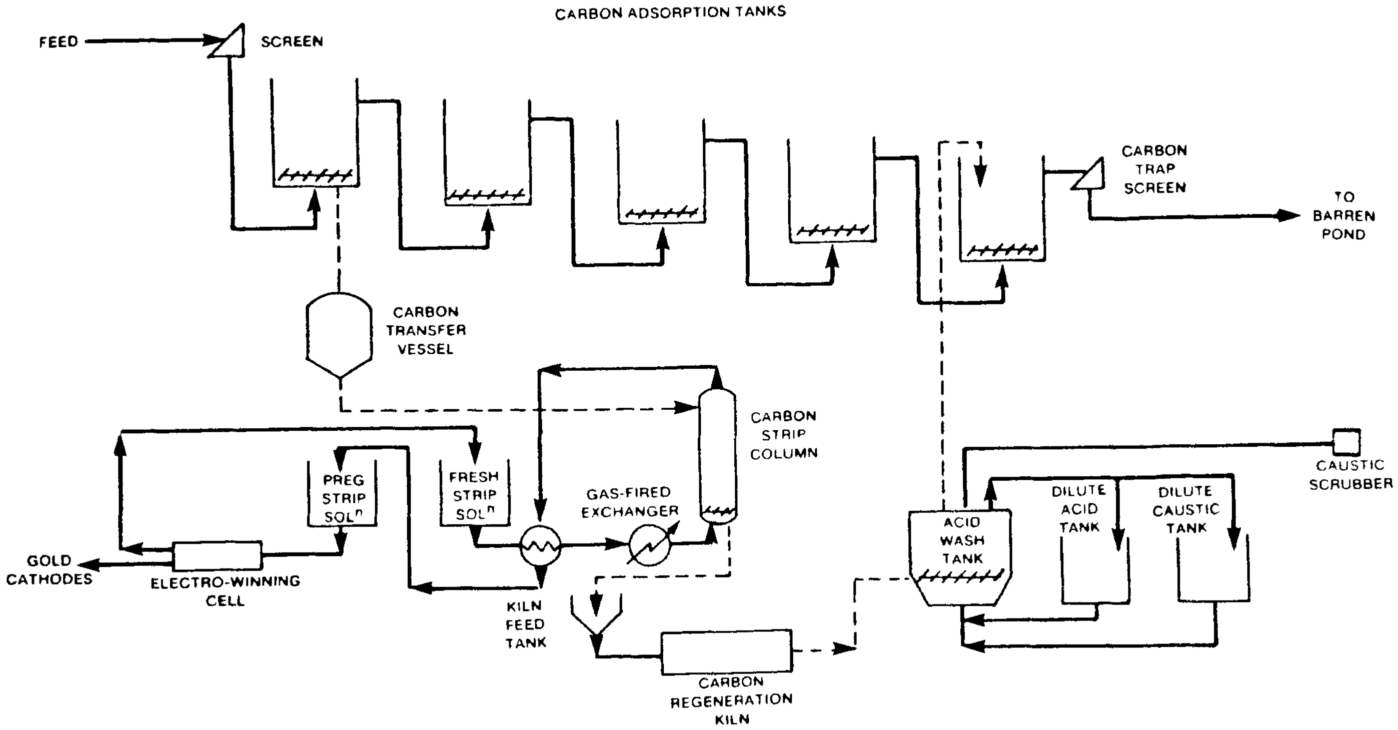


Figure 7: Flowsheet for Gold Recovery from Heap or Dump Leach Solutions (11)

In recent years, insitu soltuion mining has received increased emphasis and appears to be an important new technology for the future. Bhappu (12) lists several advantages and disadvantages.

Advantages:

- Less surface disturbance and environmental impact than conventional mining, beneficiation and smelting.
- Lower capital and operating costs.
- Potential economic advantage for recovery of metals from materials that could not be treated by conventional methods.
- Increased ore sources and reserves.
- Solutions lend themselves to a variety of metal recovery processes.

Disadvantages:

- Complex technology relative to chemical and physical features.
- Testing short of field operation is difficult.
- Ground water contaminants may result.
- Detailed data base has not been established commercially.

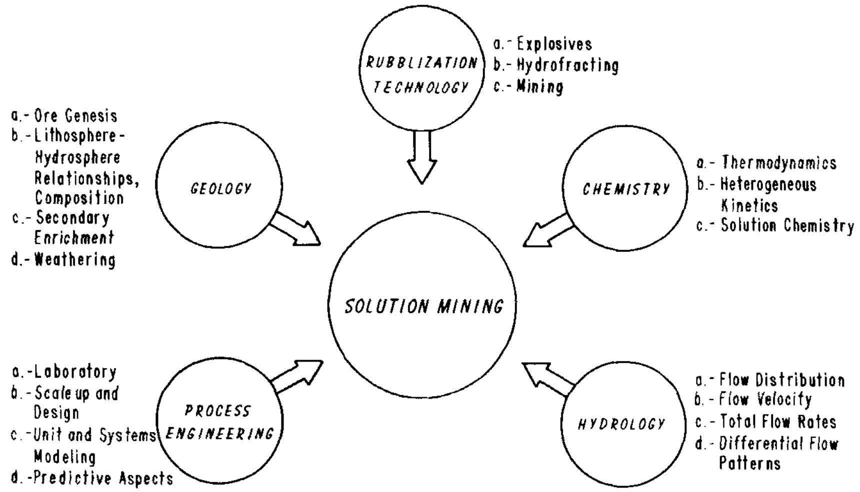
Ground water contamination, for example, is a serious problem in uranium insitu extraction because of the attenuation of ammonium ion.

Because of its complexity, the development of a solution mining system requires the interfacing of several technologies. This is illustrated in Figure 8. The mining plan, associated geochemistry, chemical reaction kinetics, solution chemistry, hydrology, and systems design are required components which must be combined. The application of secondary and tertiary oil recovery technology to uranium solution mining was a logical development since such interfacing was readily achieveable by oil-producing companies. As a result major developments occurred rapidly leading to commercial production of uranium by solution mining. Recently the University of Utah Research Institute (13) prepared a four volume assessment of the State of the Art for solution mining, outlining research needs for the future.

Several attempts have been made, with variable success (12) for insitu extraction of copper. Greatest success has been achieved for leaching of previously mined copper deposits (14). The cost and complexity of solution mining applied to recovery of values from base metal sulfides mitigate against rapid developments in this area. Dump leaching is a close and successful parallel but does not correspond to the complexity of a true insitu extraction system.

It is useful to consider an ore deposit relative to its position in the lithosphere. Deposits amenable to solution mining may be classified into the three general groupings shown in Figure 9:

- Type I. Surface dumps or deposits having one or more sides exposed, and deposits within the earth's crust but above the natural water table.



SCIENCE AND TECHNOLOGY INTERFACES IN HYDROMETALLURGY

Figure 8: Science and Engineering Contributions Required for Successful Development of Solution Mining

- Type II. Deposits located below the natural water table but accessible by conventional mining or well-flooding techniques.
- Type III. Deposits below the natural water table and too deep for economic mining by conventional methods.

Type I would be the leaching of a fractured ore body near the surface above the natural water table in the surrounding area. This would apply to mined-out regions of old mines such as a block-caved portion of copper mine or regions which have been fractured by hydrofracturing or by use of explosives. The chemistry and physical requirements would be essentially the same as in dump leaching.

Type II refers to the leaching of deposits which exist at relatively shallow depths, less than approximately 500 feet, and which are under the water table. Deposits of low permeability will have to be fractured in place and dewatered so they may be subjected to alternate oxidation and leach cycles or percolation leaching; although the use of special oxidants may eliminate the drainage cycle. This is a special problem requiring a complete knowledge of the hydrology of the region. Water in the deposit, if removed during the oxidation cycle, must be processed, stored and returned under carefully controlled conditions. An alternate method of leaching is by flooding as described for Type II below. The application of flooding using

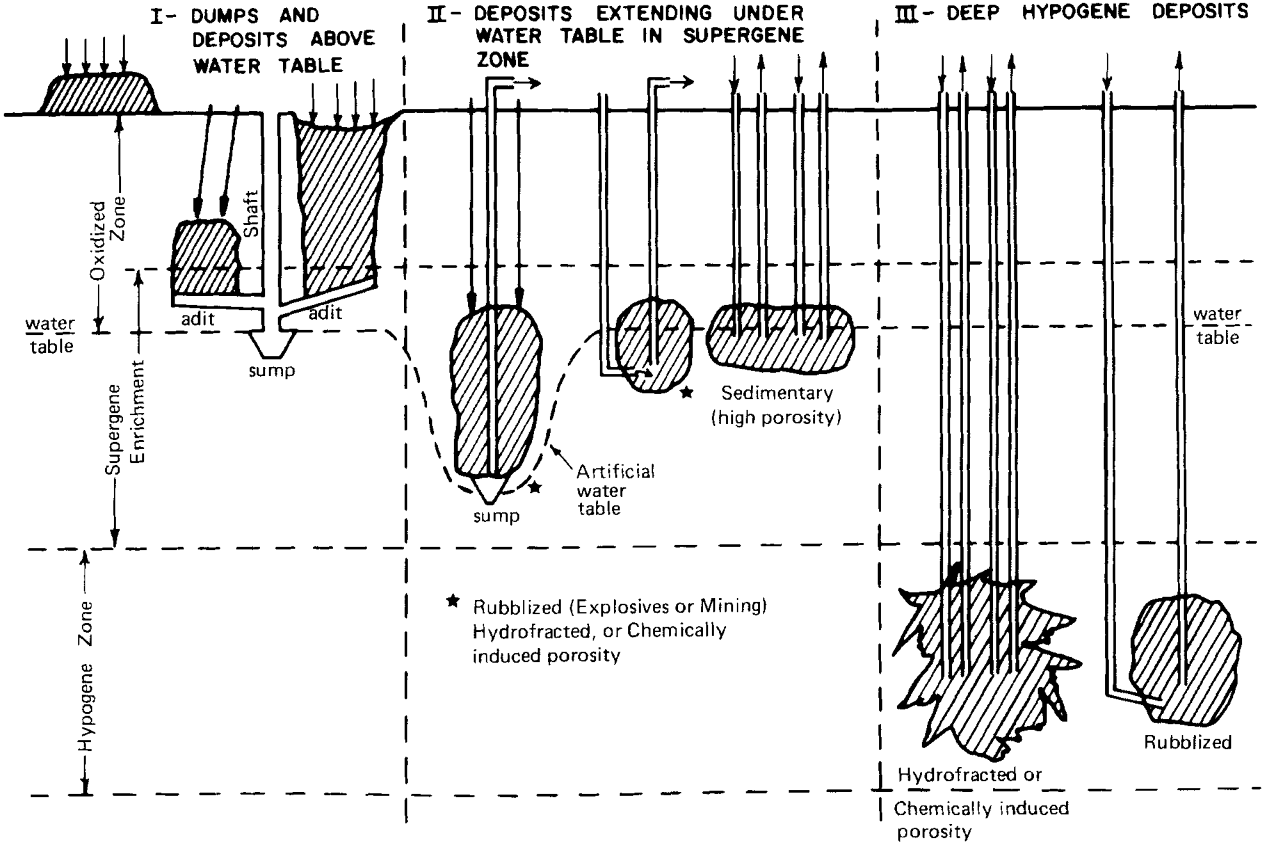


Figure 9: Solution Mining Systems

wells distributed on a grid, such has been used for uranium insitu extraction (12), is an example of Type II.

The third general type (Type III) of solution mining is represented by deep deposits below the water table and below approximately 500 ft. in depth. An ore body of low permeability will have to be rubblized or fractured for solution penetration. The hydrology of the region must be well known for proper containment of solutions. This represents a unique situation in that the hydrostatic head will increase the oxygen solubility to the point that the direct oxygen oxidation of sulfide minerals and rapid release of metal values becomes possible (15).

The combination of mining and insitu leaching provides interesting possibilities for treating massive quantities of ore under conditions of maximum solution and aeration control. Roman (16) proposed a plan for the sequential leaching of block-caved mixed oxide and sulfide copper ore. Figure 10 illustrates a single caved stope. It was proposed that 10 percent of the ore be removed by undercutting and removal to the surface for heap leaching. Multiple stopes would be leached to maintain constant metal production. A similar analysis was made by Bhappu (8) in which it was proposed that lateral drifts from a large central shaft provide access for room and pillar mining in multiple levels. A minimum ore draw of 8% would be required. The withdrawn ore would be heap leached at the surface. Using this method the deposit could be drilled for optimal placement of explosives. The deposit would be mined in a flooded condition using injection and effluent wells.

Wells (17) has proposed a plan similar to that of Roman for deposits containing predominantly all sulfide mineralization. Multiple cells to provide continuous production would be used. Cell sizes of approximately 30 X 30 X 60 meters (100 X 100 X 200 ft.), are proposed contiguous to underground mining, but isolated with separate ventilation control, Figure 11. The cells would be developed using approximately a 20% withdraw of ore. The withdrawn ore, as before, would be leached at the surface. Explosive rubblization would be employed to give as near as possible optimal size distribution. All solution treatment plant components would be located underground to minimize pumping costs. The separate ventilation system is important because enriched oxygen atmospheres would be used to accelerate leaching. Leaching could be carried out using intermittent cycles of drainage and gaseous phase oxidation. Alternatively, two phase (gas-liquid) continuous leaching could be carried out with internal cycling of gas and liquid. Heat, in addition to that released by the reaction, would be supplied to provide leaching temperatures in the range of 70-90°C.

SUMMARY AND CONCLUSIONS

No attempt has been made to review the available literature (13) on kinetics and mathematical modeling of solution mining systems. Current practice applied to copper, gold, silver and uranium clearly illustrates many innovative and successful developments in the field.

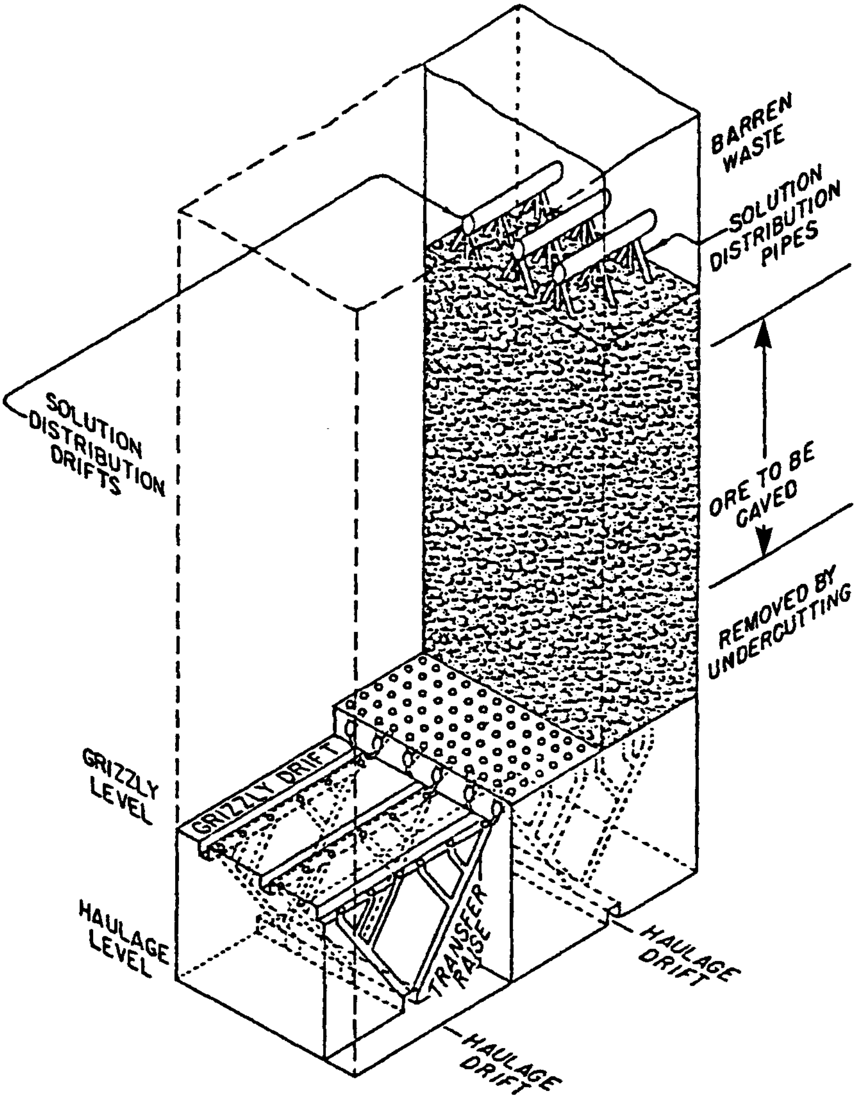


Figure 10: Schematic of Block-Caved Zone for Insitu Leaching (16)

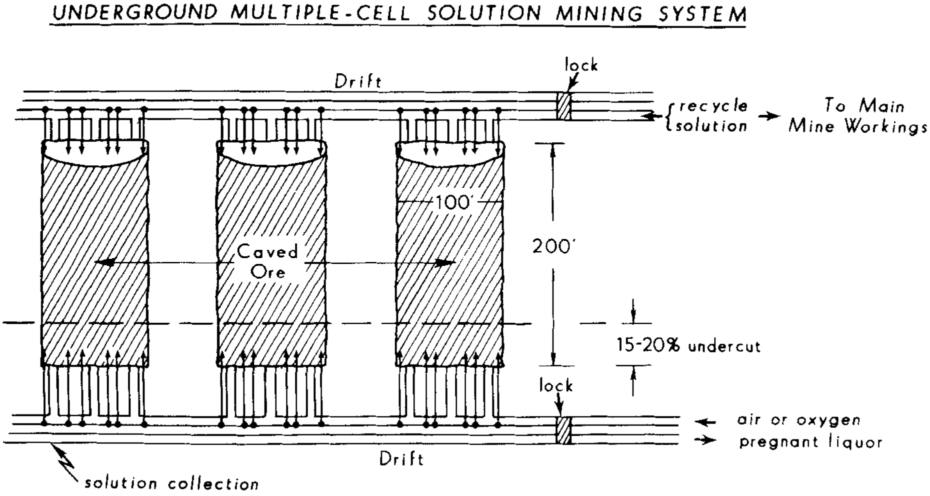


Figure 11: Solution Mining of Multiple Leaching Cells Illustrating Solution Management and Separate Ventilation (17)

These must be considered as only the beginning when potential future applications are considered. The expansion and improvement of technology for the extraction of metal values from low grade ore sources is an area of vital national importance. Extensive research is needed to better understand the geology, hydrology, solution and mineral chemistry and rate limiting processes needed for engineering design and economic analysis. The combined resources of mining and process engineers, earth scientists, chemists and hydrologists are required. Existing low grade sources have and are receiving attention but the technology must be extended to a much broader array of mining plans and metal commodities for the future.

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