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KEEP THIS ON TOP

Subject: IN-PLACE LEACHING

Gen Ex,

November 29, 1971

AIRMAIL

Mr. Elburt E. Osborne
Director, Bureau of Mines
18th & C Streets NW
Washington, D.C. 20240

Dear Mr. Osborne:

In general discussions with Mr. W. R. Hardwick of Tucson and Mr. Guy Johnson of Minneapolis, we understand that experimental in situ leaching may be given a high priority with the U.S. Bureau of Mines.

Last Tuesday, November 23rd, I reviewed with Mr. Hardwick a copper oxide mineral zone that we feel could become an excellent test site. Mr. Hardwick suggested that his metallurgical engineer inspect the core and log data. He has our permission to do so.

If your metallurgical engineer is encouraged with the diamond drill core and assay data from this copper oxide deposit, ASARCO would be interested in discussing the possibility of the U.S. Bureau of Mines using this deposit for experimental work.

Very truly yours,

R. B. Crist

RBC:lad

cc: WLKurtz, Supervisor
WRHardwick
GAJohnson

gen file.

AMERICAN SMELTING AND REFINING COMPANY
Tucson Arizona

November 11, 1971

Mr. S. Von Fay
Denver Office

Dear Steve:

I have recently talked with representatives of Dowell, who are extremely interested in the recovery of copper by in-place leaching. These people have the staff and technical know-how to perform in-place leaching tests and are interested in participating (10-25%) in the testing of such properties. It occurs to me that some of your "sedimentary" copper might be amenable to in-place leaching.

I suggest you contact a Mr. J. D. Stewart, Regional Manager, Mining Services, Dowell, 216 Security Life Building, Denver, Colorado, 303-222-3585. I believe you will be impressed with Mr. Stewart's knowledge and enthusiasm.

Best regards,

W. L. Kurtz

WLK:lad

~~F T Graybeal~~ ~~Circulate~~ → Gen. File: In-Place Leaching

THE APPLICATION OF SOLUTION-MINERAL EQUILIBRIUM CHEMISTRY TO THE LEACHING OF LOW-GRADE CHALCOCITE ORES

by

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For several decades, the minerals industry has used leaching techniques to extract metals from rocks whose grade was too low to be treated profitably using conventional metallurgical techniques, and, as of 1972, about 15% of domestic copper production was derived using leaching methods (Beall[1]). Increasing interest is currently being directed toward solution mining, particularly with respect to copper, for basically two reasons. In the first place, geological exploration has outlined several copper deposits of large tonnage and low grade which are buried at sufficient depth to preclude open-pit mining which would make such mineral deposits ore deposits. Nevertheless, these occurrences provide significant potential if they could be treated by chemical leaching in situ. Secondly, the porphyry deposits of the western United States, which provide by far the largest percentage of domestic copper, do not terminate geometrically with sharp borders. These large disseminated bodies commonly decrease in grade laterally away from volumes of rock containing ore grade metallization, and the distinction between ore and waste is strictly economic. In order to extract ore grade material from the ground, a significant amount of rock containing sub-ore-grade copper must also be removed which, when placed on dumps, is amenable to solution mining. These two considerations clearly indicate the reasons for the growing interest in solution leaching techniques--namely the ability to exploit deeply buried low grade deposits and the expansion of reserves in deposits currently under production.

Essentially, all of the operational porphyry copper deposits have undergone some enrichment in grade since their original time of formation by natural weathering processes. In this process of

supergene enrichment, relatively insoluble primary sulfide minerals are converted to iron oxide and soluble copper sulfate through exposure to oxidizing conditions at or near earth surface. Meteoric waters are turned strongly acid by oxidation of pyrite according to the reaction:

$$2FeS_2 + \frac{15}{2} O_2 + 4H_2O = Fe_2O_3 + 4SO_4^{=} + 8H^+$$

These acid waters dissolve copper and percolate downward into the subsurface where they are progressively neutralized by reaction with original rock forming silicates which have not yet been exposed to surface weathering. The neutralization is a consequence of hydrogen consumption through cation exchange as clay minerals such as kaolinite are formed from, for instance, potassium feldspar as follows:

$$2KAlSi_3O_8 + 2H^+ + H_2O = Al_2Si_2O_5(OH)_4 + 4SiO_2 + 2K^+$$

As a result of such neutralization and increasingly reducing conditions encountered with depth, dissolved copper may either precipitate directly or replace pre-existing pyrite to produce a zone in which primary copper grade is significantly enriched. The process of supergene enrichment described here is essentially the natural analogue to leaching techniques employed by the mining industry in which sulfuric acid and ferric sulfate are added to water to produce an acidic oxidizing solution which is then passed through rocks containing copper sulfides (Sheffer and Evans[2]). The ferric iron provides an oxidizing agent to produce soluble copper sulfate, and the acid prevents the reprecipitation of the dissolved copper within the rock being leached.

Several problems arise with the use of acid iron sulfate solutions as a leaching agent in addition to inherent difficulties such as reaction kinetics attending oxidation and dissolution of

sulfides. One of these is the fact that a very low pH solution must be input in order to overcome the neutralization capacity of the rocks, particularly during the early stages of the leach cycle. As in the natural enrichment process, the reaction of these acidic solutions with rock forming silicates results in formation of clay minerals which may result in reduced rock permeability, channeling of fluids, and/or coating of copper-bearing minerals. In addition, if electrolytic (or "tin-can") exchange is used to extract copper from the effluent solution by a reaction corresponding to

$$2Fe_s + 3Cu^{+2} = 2Fe^{+3} + 3Cu_s$$

a high concentration of dissolved iron will tend to inhibit reduction of dissolved copper according to the principal of le Chatelier.

The purpose of the project described in this communication was to attempt to remedy the problems described above through the application of the fundamentals of solution mineral equilibrium geochemistry to the process of solution leaching of copper. The materials utilized in this study consisted of sub-ore-grade chalcocite bearing rocks from the mine at Tyrone, New Mexico, which were generously made available to us by the Phelps Dodge Corporation. Mineral stability relations in aqueous solutions were employed to develop a leaching solution which would result in minimal reaction with rock forming silicates but would dissolve the copper-bearing sulfide minerals and maintain the copper in solution at concentration sufficient to make the process economically feasible.

Theoretical Considerations

The Tyrone rocks were considered representative of low grade chalcocitic material present in many of the operating porphyry copper operations from the southwestern United States. The Cu content was approximately 0.29 wt%, and total sulfides as

chalcocite plus pyrite made up less than 10% of the whole rock mass. The whole rock chemical analysis determined by atomic absorption is listed in Table 1, and the mineralogic analysis of the same material obtained by petrographic and x-ray analysis is given in Table 2. As would be expected from the whole rock analysis, silicate minerals make up the majority of the rock-forming materials, and these silicates consist chiefly of quartz and muscovite-sericite with trace amounts of kaolinite and relict K-feldspar.

Silicate Mineral Stabilities

The whole rock composition and mineral content of the Tyrone test material are essentially defined by the components K_2O-Al_2O_3-SiO_2-H_2O. Stability relations among common minerals in this chemical system in the presence of an aqueous phase at earth surface conditions may be displayed graphically in terms of activities (a_i) of dissolved species (Garrels and Christ[3]). Equilibrium boundaries between mineral phases are defined by chemical reactions written conserving aluminum between solids such that the appropriate equilibrium constants contain as variables: $\log \frac{a_{K^+}}{a_{H^+}}$ and $\log a_{H_4SiO_4}$. Slopes of the lines corresponding to boundaries between stability fields of various mineral pairs are defined by stoichiometric coefficients of the variables in reactions defining equilibrium between phases, and intercepts of the lines are dictated by the value of the equilibrium constant for the appropriate reaction.

For example, equilibrium between K-feldspar and kaolinite is written

$$2KAlSi_3O_8 + 2H^+ + 9H_2O = Al_2Si_2O_5(OH)_4 + 2K^+ + 4H_4SiO_4$$

with the corresponding equilibrium constant (Keq):

$$\log Keq = \log \frac{(a_{K^+})^2 \, (a_{H_4SiO_4})^4}{(a_{H^+})^2} = -5.04 \text{ (Table 3)}$$

Reactions among various phases in the system of interest along with corresponding equilibrium constants at 25°C calculated using thermodynamic data from Helgeson[4] and Ulbrich and Merino[5] are listed in Table 3. These equilibrium constants were used to calculate the mineral stability diagram shown in Figure 1. A brief comment is appropriate here concerning two polymorphs of the composition $Al_2Si_2O_5(OH)_4$: kaolinite and dickite. Both of these minerals have the same composition and layered structure, but different stacking configurations exhibited by each result in different entropies and, therefore, different free energies of formation for the two. Thus, on Figure 1, the phase boundaries between each of these two polymorphs and K-feldspar will differ in position owing to different equilibrium constants for the two governing equations, but slopes for the two equilibrium boundaries relative to K-feldspar are parallel; this is because the equilibrium constants for analogous reactions of the two polymorphs have different values but equivalent stoichiometric coefficients. A similar situation is exhibited by the two SiO_2 polymorphs: quartz and amorphous silica. These two phases also exhibit different equilibrium constants for dissolution to aqueous H_4SiO_4, but, in this case, the different free energies are a result of the contribution of grain size effects to the appropriate free energies of formation.

Mineral stability diagrams such as Figure 1 provide a useful means of determining the identity of solid phases which would coexist in equilibrium with an aqueous solution whose composition is known. Concentrations (m_i) of appropriate aqueous species must first be converted to activities (a_i) by means of stoichiometric activity coefficients (γ_i) according to the relation $a_i = m_i \times \gamma_i$.

These activity coefficients are calculated for a given aqueous solution using the Debye Huckel equation and ion association constants for solution species. For example, groundwaters normally have dissolved silica in the range 6 - 60 ppm corresponding to log $a_{H_4SiO_4}$ between -3 and -4, and log $\frac{a_{K^+}}{a_{H^+}}$ values ≤ 6 (Garrels and Christ[3]); these waters generally plot in the vicinity of the K-feldspar-muscovite boundary extending down in log $\frac{a_{K^+}}{a_{H^+}}$ to the kaolinite stability field on Figure 1.

Irreversible Reactions

The methods by which irreversible reactions are predicted between minerals and aqueous solutions with which they are not in equilibrium have been described in the geochemical literature by Helgeson and co-workers[6,7,8]. The technique described in these communications permits evaluation of changes in the composition of an aqueous phase and identification of intermediate product phases as an irreversible reaction proceeds toward equilibrium between a reactant mineral and a solution. The succession of partial equilibrium states occurring during the irreversible reaction may be conveniently plotted on mineral stability diagrams, and such a technique will be used here to demonstrate the reaction path expected to occur between two different solution compositions and a rock containing K-feldspar and kaolinite, such as might occur in low grade leach material containing a copper mineral.

Consider first a typical groundwater containing 20 ppm dissolved silica corresponding to log $a_{H_4SiO_4} = -3.5$ and log $\frac{a_{K^+}}{a_{H^+}} = 4.0$ resulting from 40 ppm dissolved potassium at pH = 7.0. This water composition would plot at point A on Figure 2, within the stability field of kaolinite. From the hydrolysis constant for

kaolinite (log K = 7.63) and the stoichiometric activity coefficient for aluminum (which accounts for Al-OH complexing), the molality of dissolved aluminum in this solution in equilibrium with kaolinite is approximately $10^{-8.8}$ molal. If this water is placed in contact with a rock containing, for example, kaolinite plus K-feldspar, a situation exhibiting partial equilibrium is established in which the aqueous phase is in equilibrium with respect to one mineral (kaolinite) but out of equilibrium with respect to another (K-feldspar). In order to attain overall equlibrium, K-feldspar reacts with solution to form kaolinite, the phase with which the solution is in equilibrium, according to the reaction:

$$2KAlSi_3O_8 + 2H^+ + 9H_2O \rightarrow Al_2Si_2O_5(OH)_4 + 2K^+ + 4H_4SiO_4$$

As this reaction proceeds, hydrogen ion is consumed as silica and potassium are delivered to the solution in mole ratios defined by the coefficients of the various species in reaction, the net effect being to raise values of log $\frac{a_{K^+}}{a_{H^+}}$ and log $a_{H_4SiO_4}$ in the solution. If pH of the solution increases from 7 to 7.5, a net mass of hydrogen equivalent to 6.8×10^{-8} moles is consumed while generating 6.8×10^{-8} and 1.4×10^{-7} moles of potassium and silica, respectively. The increase in these latter two components is trivial compared to their original concentrations in the solution, but the pH is shifted noticeably because the original concentration of hydrogen ion in the solution is of the same magnitude as that of hydrogen consumed by the reaction. The new solution composition, which plots at point B on Figure 1, lies on the equilibrium boundary between K-feldspar and kaolinite so the solution is now at equilibrium with the two solids originally present. The mass of K-feldspar necessary to produce this shift is about 2×10^{-5} gm per liter of solution, so

essentially no change in mass abundance of solids is observed.

If the same initial solution composition (point A, Figure 2) is acidified with H_2SO_4 to a pH = 2 for use as an acid-sulfate leach solution, the value of log $\frac{a_{K^+}}{a_{H^+}}$ is now -1.0 which plots at point C on Figure 2. The solution would not be in equilibrium with kaolinite, however, because according to the hydrolysis constant for kaolinite given above, the solution at pH = 2 containing 20 ppm dissolved silica would require approximately $10^{2.5}$ moles of dissolved aluminum which is far less than the $\sim 10^{-9}$ moles per liter actually present in the modified groundwater. Therefore, if this solution were to be introduced to rocks containing kaolinite and K-feldspar, neither of the two solid phases would be stable and would begin dissolving, according to the hydrolysis reaction

$$Al_2Si_2O_5(OH)_4 + 6H^+ = 2Al^{+3} + 2H_4SiO_4 + H_2O \qquad (1)$$

and

$$KAlSi_3O_8 + 4H^+ + 4H_2O = K^+ + Al^{+3} + 3H_4SiO_4 \qquad (2)$$

Such dissolution contributes potassium, aluminum, and silica to the solution while consuming hydrogen ion. Considering just kaolinite dissolution for simplicity, if pH were to rise to 2.3 corresponding to 5.1×10^{-3} moles of hydrogen consumed per liter of solution, the stoichiometric coefficients of the components in reaction (1) indicate that aluminum and silica in solution would each increase in the amounts 1.7×10^{-3} moles per liter. At this point, the solution composition would contain 2.0×10^{-3} moles per liter silica and 1.7×10^{-3} moles per liter dissolved aluminum, which corresponds to point D on Figure 2 very near saturation with respect to amorphous silica. Further dissolution of kaolinite adds aluminum to solution while consuming hydrogen, but the silica content of the fluid is

fixed by saturation with respect to solid SiO_2. Upon raising solution pH to about 3.35, at which value the aluminum content of the solution = 3.2×10^{-3} moles per liter, the solution becomes saturated with respect to pyrophyllite at point E on Figure 2. At this point, amorphous and dissolved silica combine with dissolving kaolinite to cause continued deposition of pyrophyllite according to the reaction

$$Al_2Si_2O_5(OH)_4 + 2H_4SiO_4 \rightarrow Al_2Si_4O_{10}(OH)_2 + 5H_2O$$

Thus, further dissolution of kaolinite causes depletion of dissolved silica in solution at constant pH which will eventually move the solution composition to point F on Figure 2 where sufficient silica depletion has occurred to cause kaolinite to equilibrate with the aqueous phase in the presence of pyrophyllite.

If K-feldspar had been dissolved along with kaolinite, the overall reaction path of the solution would not change significantly up to point F because the changing components are dominantly hydrogen, aluminum, and silica. At point F, the solution at pH = 3.35 would contain approximately 1.3×10^{-3} moles per liter dissolved silica and 7.1×10^{-3} moles per liter dissolved aluminum. This solution has equilibrated with respect to kaolinite, but it is still far from saturated with respect to K-feldspar (compare points G and F, Figure 2). As K-feldspar continues to dissolve according to equation 2, kaolinite and pyrophyllite will precipitate, fixing the silica content of the fluid while depleting it in aluminum and hydrogen and enriching it in potassium. During this process, the solution composition moves upward along the kaolinite-pyrophyllite boundary on Figure 2 until it reaches equilibrium with respect to

K-feldspar at point G. The final solution composition will contain masses of components approximating 1.5×10^{-3} moles of potassium, 7×10^{-4} moles of silica, and 4.8×10^{-10} moles of aluminum at pH = 6.10.

The reaction path described above may be modified by a variety of factors, such as kinetic and supersaturation effects, variations in initial solution composition, consideration of additional minerals, and mass limitations on reactant minerals. One assumption made in the previous calculations that deserves brief comment is that of permitting the aqueous phase to supersaturate with respect to quartz ($\log a_{H_4SiO_4} = -4.0$) up to amorphous silica stability ($\log a_{H_4SiO_4} = -2.7$). This allowance was made based on the observations that at low temperature the upper limit on silica solubility is controlled by amorphous silica (Siever[9]) and that many groundwaters show dissolved silica contents in excess of 6 ppm corresponding to quartz solubility (cf., Garrels and Christ[3], p. 361). If quartz saturation had been used to limit silica solubility in foregoing calculations, the solution composition would simply have increased in $\log \frac{a_{K^+}}{a_{H^+}}$ at constant dissolved silica. Because quartz saturation does not lie within the stability field of pyrophyllite, this phase would not have appeared as a reaction product, and the final solution pH at equilibrium would be slightly in excess of 7.0.

The abbreviated approach described here is employed to demonstrate the gross geochemical effects of interacting a strongly acid solution with minerals typical of rocks currently being leached to extract copper, and various modifications of the process, within reasonable limits, will produce the same general results. These may be summarized as follows:

1.) initial dissolution of rock-forming silicates at low pH to generate a solution relatively high in dissolved aluminum and silicon,

2.) equilibration and precipitation of clay minerals (chiefly aluminosilicates) as pH rises from dissolution of minerals stable at relatively high cation/hydrogen solution values, and

3.) attainment of overall equilibrium between solution and rock at pH values significantly less acid than the initial leach solution.

If this sequence of events proceeds as a solution percolates through a column of rock, the individual steps would occur along the path of fluid flow. The result would be a zone of intense dissolution of all rock components near the point of solution influx with a tendency to produce permeable channels (C-D-E, Figure 2) followed by a broad zone throughout which acid-stable aluminosilicates would be precipitated in flow channels (E-F-G, Figure 2), producing coating of original minerals and filling of flow channels with concomittant loss of rock reactivity and permeability. The initial effluent solutions would exhibit significant acid consumption, but later volumes of acid solutions passed through the rock would show less acid consumption because the original reactive minerals such as feldspars would either be converted to clay minerals near the beginning of the flow path, or be coated with such product phases further along as a result of precipitation. An important observation is that in this zone of precipitation, copper-bearing minerals may be covered in addition to reactive silicates.

Copper-Mineral Stabilities

Leach solutions are maintained oxidizing by means of dissolved oxygen, ferric iron, and sulfate. Using such agents, the common copper-bearing mineral chalcocite (Cu_2S), which is relatively insoluble, is converted to a solution containing cupric sulfate by a reaction such as

$$Cu_2S + 2H^+ + \frac{5}{2}\ O_2 \rightarrow 2Cu^{+2} + SO_4^{=} + H_2O$$

With these two oxidized species in solution, a variety of minerals have potential for saturation, such as tenorite - CuO, brochantite-$Cu_4(OH)_6SO_4$, antlerite-$Cu_3(OH)_4SO_4$, chalcanthite-$CuSO_4 \cdot 5H_2O$, and simple $CuSO_4$. All except the last of these phases have a solubility which depends wholly or in part upon solution pH in addition to dissolved copper. Figure 3 shows stability relations among these minerals as a function of dissolved sulfate (log $a_{SO_4^{=}}$) and solution pH with saturation surfaces for different activities of the cupric ion. Considering a solution containing approximately 1 gm per liter dissolved copper (log $a_{Cu^{++}} = -2$ contour, Figure 3), it is apparent that if log $a_{SO_4} = < -0.5$ (which is on the order of 30 gm per liter $SO_4^{=}$) and pH < 4, the solution will be undersaturated with respect to all phases shown on the diagram. Looking, however, at solutions of pH in the range 2-3, copper solubility is inversely related to the sulfate content of the aqueous phase. Thus, if the type of leach solution in current use is allowed to build up in dissolved sulfate, the copper content of the fluid may be limited by saturation with respect to chalcanthite.

Iron Considerations

Conventional leach solutions contain significant amounts of dissolved iron from both initial addition and later oxidation and dissolution of pyrite contained in the rocks being treated. As was

briefly mentioned at the onset, dissolved iron can inhibit retrieval of dissolved copper using solvent extraction or ion exchange techniques. Ferric iron solubility is strongly pH dependent owing to the stability of hematite-Fe_2O_3, or related iron oxides such as goethite-FeO(OH), as shown by

$$FeO(OH) + 3H^+ = Fe^{+3} + 2H_2O$$

According to the equilibrium constant for this reaction, at a pH = 1 approximately 10^{-2} moles of iron can be carried per liter of solution ($\sim$ 0.5 gm/l), and if pH is raised one unit, the solubility of iron oxide is lowered by three orders of magnitude.

Experimental Procedures

Based on the preceding theoretical considerations, several different leach solution compositions were established to examine the contribution of various intermediate processes to the leaching cycle using chalcocite-bearing rock material from Tyrone, New Mexico. In the first set of experiments, two acidified groundwaters were circulated through large columns containing about 450 kg of rock in order to compare acid consumption in a solution containing sulfuric acid alone to one containing 0.2 molal KCl in addition to sulfuric acid; pH was continually readjusted between cycles to values within the range 2 - 2.5. The basis for comparing the two solutions was to attempt to shorten the length of the aluminosilicate-depositional leg of the reaction path (segment F-G, Figure 2) by moving the starting value of log $\frac{a_{K^+}}{a_{H^+}}$ of the leach solution to a higher value by addition of potassium chloride (point C', Figure 2). Then, two small columns containing approximately 1 kg of rock were cycled with solutions containing 0.2 molal KCl and pH adjusted initially to 4 with sulfuric acid. This was done for the purpose of examining the

prolonged consequences of interaction of rock with a solution having $\log \frac{a_{K^+}}{a_{H^+}}$ adjusted to a value near equilibrium with respect to K-feldspar and kaolinite (point C", Figure 2). Finally, a set of four small-column leach tests was performed, at pH adjusted to 2, in order to evaluate the effects of potassium and chloride individually. In the first pair, two solutions containing 0.2 m NaCl and KCl were utilized; in the second case, two solutions containing 0.2 m potassium dissolved as chloride (KCl) and sulfate (K_2SO_4) were compared. In addition, a pure H_2SO_4 solution at pH = 2, a KCl-H_2SO_4 solution with pH adjusted to 4, and distilled water were run as controls to test relative leaching efficiencies of the different solutions under similar conditions.

During these experiments, leach solutions were periodically sampled and analyzed for major rock-forming components (potassium, aluminum, and silica), as well as the traditional species (copper, iron, sulfate, Eh, and pH). Total solution concentrations were then converted to activities of individual solution species, taking into account Debye-Huckel and ion association effects using a computer code employing conservation of mass and homogeneous equilibrium among solution species. Conversion of concentrations to individual ion activities permits solution compositions to be transposed to activity coordinates for representation of the aqueous phase on the mineral stability diagram shown in Figure 1. The aqueous species employed in the computer code are listed in Table 4, and the expression used to calculate individual ion activity coefficients (γ_i) in the relatively concentrated solutions was that given by Helgeson[4]:

$$\log \gamma_i = \frac{-AZ^2 \sqrt{I}}{1+a^o B\sqrt{I}} + 0.04I$$

where A and B are the Debye-Huckel solution parameters, a^o and Z

correspond to the size parameter and charge of the appropriate ion, and I is the true ionic strength of the solution. The activity coefficient of neutral aqueous silica (H_4SiO_4) was assigned a value of unity.

Two types of leach columns were used in the tests. Initially, 200 x 38 cm I.D. PVC pipe was employed, but these columns contained 450 kg of ore and proved difficult to handle. The smaller scale bench-top columns were constructed from approximately 6 cm I.D. by 150 cm long glass or Lucite and were mounted vertically. Solution was pumped to the top using Cole-Parmer tubing pumps and allowed to gravity-percolate through the ore. This solution was continually circulated in order to establish equilibrium with the rock minerals as rapidly as possible. The ore was ground to 1/4 inch and less and approximately 2 kg was used per column. Rubber stoppers with glass tubing were inserted at both ends of the columns, and glass wool was used at the bottom of the ore to keep the fines from washing from the column.

Measurement of pH and Eh were performed using a glass electrode and platinum electrode, respectively, versus a saturated calomel reference electrode. Metal ions and silica were analyzed by atomic absorption using a Perkin-Elmer 403 spectrophotometer. Sulfate was determined gravimetrically as barium sulfate, and chloride was determined volumetrically using mercuric nitrate. Whole rock chemical analysis was determined on a sample digested in hydrofluoric acid-nitric acid in a teflon-lined bomb manufactured by Parr Instrument Co.

Mineralogical analyses were performed on thin sections using standard optical procedures. Clay separations were performed on samples which were first powdered in a ball mill then stirred with

distilled water in an ultrasonic stirrer for 10 minutes. The 2 micron fraction was collected by gravity settling, and a portion was placed on a microscope slide and x-rayed.

The KCl/H_2SO_4 leach column which was equilibrated was taken apart, and 4 sections were removed for chemical analysis. The column was 15 inches long, and one inch sections were removed from the top, at the 4-5 inch level, the 7-8 inch level, and the bottom. The sections were ground, split, and digested in HF/HNO_3 in teflon-lined bombs (see above).

Experimental Results

The comparative data for successive leach cycles using H_2SO_4 and $KCl-H_2SO_4$ solutions in the large columns are given in Table 5. Beginning with cycle 6 during these tests, the leach solutions were subjected to Fe/Cu exchange with scrap iron in a process analogous to the "tin-can" reaction used in cementation plants. Both solutions contained similar dissolved copper contents, but the effluent pH was lower in the H_2SO_4 solution, and the dissolved iron content reached a level approximately an order of magnitude higher than the $KCl-H_2SO_4$ solution. Following introduction of the cementation process, copper was found to be extracted far more efficiently from the potassium bearing solution, and dissolved iron also remained significantly lower than the solution containing H_2SO_4 alone.

Progressive effluent compositions from two columns leached with solutions containing 0.2 m KCl and whose input pH values were adjusted to 4 are listed in Table 6. Complete major element analyses are listed for one of these (Table 6a), and a more abbreviated set of analyses is listed for the other (Table 6b). Also included

in this table are values of log $\frac{a_{K^+}}{a_{H^+}}$ and log $a_{H_4SiO_4}$ for the samples which are plotted on the potassium-aluminum-silicate mineral stability diagram in Figure 4. Although the two different tests had essentially identical starting compositions (point C", Figure 2), the effluent compositions after a single circulation step indicated by arrows on the figure were considerably different and lay on opposite sides of the dashed line. After two more cycles of each solution, however, the compositions of both plotted on the line. As reaction proceeded, both solutions moved along the line to lower values of log $\frac{a_{K^+}}{a_{H^+}}$, and as soon as effluent pH reached a value of 4, which corresponds to log $\frac{a_{K^+}}{a_{H^+}}$ approximating 3.15, no further acid was added to the solution. In spite of the lack of further addition of acid, the solution listed in Table 6b continued to decrease in pH. Concurrent with the shift of solution compositions along the line, the copper content of the fluid rose to values between 1 and 2 gm/liter with virtually no iron being dissolved.

The results of the comparative tests using different solution types to evaluate cation-anion effects are listed in Table 7. In this table are summarized the final copper and iron contents of the solutions at the end of the length of time indicated for operation of each of the columns, as well as the percent of original copper extracted from each. Leaching efficiencies of each of the solutions were evaluated by dividing percent Cu extracted by operating time. Solutions I(K_2SO_4) and II(KCl) can be compared to establish the anion effect, and the cation effect can be seen from solutions II (KCl) and III(NaCl) using similar alkali metal salts. Solutions V (KCl, pH = 4) and VI(H_2SO_4, pH = 2) were run under experimental

conditions different than those of solutions I through IV; progressive chemical analyses of the latter solution (VI) are presented in Table 8.

Following the leaching tests, column V, which was treated with the $KCl-H_2SO_4$ solution at pH = 4, was dissected, and whole-rock chemical analyses of intermediate sections through the column were carried out; these results are shown in Table 9. There is very little difference between each of these sections and the analysis of unleached material listed in Table 1. The copper contents in the dissected column are between 0.17 and 0.21 wt%, indicating a uniform leaching pattern down the column. The average percentage of copper remaining is 0.74%, indicating 36% extraction, which compares well with the value calculated from aqueous analyses (37%) given in Table 6. The clay separations before and after this leaching test with $KCl-H_2SO_4$ at pH = 4 show no change in amount or types of clay, suggesting that original rock forming silicates were not significantly affected by the leach solution.

Conclusions

From the experimental results obtained in this study, several observations can be made which have considerable bearing on optimization of solution leaching techniques. Perhaps the most important result is documentation of the important role played in leaching efficiency by interaction between the leach solution and rock-forming silicates. This interaction is closely related to such phenomena as neutralization, surface coating, and permeability changes within a rock column being leached.

Solution-Silicate Interaction

The general methods employed in solution leaching of copper sulfide are an artificial analogue to the natural processes of

weathering and supergene enrichment of mineralized rocks. Geologic observations at numerous porphyry copper deposits show large-scale pervasive alteration of original rock-forming silicates to clay minerals as a result of interaction with acid-sulfate solutions produced by oxidation of pyrite. A simplified calculation employing irreversible reactions and mineral stability considerations shows that comparable results will occur when strongly acidified solutions are used to leach low-grade copper from rocks containing silicate minerals. Theoretical considerations of such interaction predict an initial stage during which rock-forming silicates dissolve, followed by a period during which kaolinite and pyrophyllite are formed as a result of both neutralization of the leach solution and alteration of such minerals as K-feldspar and muscovite. Strongly-acid solutions which are low in potassium can be shown to require relatively long periods of such dissolution and deposition of kaolinite and/or pyrophyllite prior to overall equilibration with the latter-mentioned potassium aluminosilicates. The formation of these acid-stable clay minerals produces a rock column whose capacity for later acid consumption is greatly reduced, but the portion of these minerals generated by precipitation (neutralization) may serve both to inhibit leaching of copper by coating of sulfide grains and to reduce rock permeability by filling of solution channelways. Furthermore, the initial dissolution of silicates occurring immediately after introduction of the acid solution to the rock column is likely to result in production of conduits which selectively channel fluids, thus reducing the total rock surface contacted by leach solutions.

The generation of acid-stable clay minerals was demonstrated by a leach column treated with a leach solution whose pH was adjusted = 2 using sulfuric acid (Table 8). The initial effluent pH in this

column was about 4, which corresponds to 99% acid consumption, but with continued circulation the effluent pH approached that of the input solution, indicating that the neutralization capacity of the rock was essentially destroyed. The successive solution compositions plotted on Figure 5 (X's) show the aqueous phase moving first through the kaolinite stability field to the kaolinite-pyrophyllite boundary and then through the pyrophyllite stability field to amorphous silica saturation. The last point has dissolved silica exceeding amorphous silica saturation, which may correspond to the development of a silica colloid; this final solution composition is also well below saturation with respect to pyrophyllite.

The ability to equilibrate an aqueous phase with rock-forming silicates was demonstrated with a solution containing 0.2 m potassium and pH adjusted to 4. Progressive changes in effluent solution composition are listed in Table 6b, and these compositions are plotted on Figure 5 (filled circles). In contrast to the pH = 2 sulfuric acid solution, this moderately acid solution moved along a line corresponding to equilibrium between K-feldspar and dickite, the structural analogue to kaolinite. Both solutions evolved to higher concentrations of dissolved silica with progressive circulation, in an apparent attempt to equilibrate with amorphous silica. As discussed previously, the leach solution with pH = 2 moved away from the K-feldspar stability field into that of pyrophyllite as the strongly acid solution reduced the neutralization capacity of the rock by formation of acid stable clay minerals. The 0.2 m potassium solution also decreased in log $\frac{a_{K^+}}{a_{H^+}}$ with continued cycling, but it did so systematically, apparently in equilibrium with rock-forming

silicates, through the reaction of unstable quartz with the large masses of muscovite in the rock to form dickite and K-feldspar according to

$$KAl_3Si_3O_{10}(OH)_2 + 2SiO_2 + H_2O \rightarrow KAlSi_3O_8 + Al_2Si_2O_5(OH)_4$$

Dissolution of quartz also increases the concentration of dissolved silica in solution, so in order to follow the phase boundary between dickite and K-feldspar (line A-B, Figure 5), a portion of the dickite formed and/or kaolinite in the original rock, is converted to K-feldspar by the reaction:

$$Al_2Si_2O_5(OH)_4 + 2K^+ + 4SiO_2 = 2KAlSi_3O_8 + 2H^+ + H_2O$$

as the solution moved to amorphous silica saturation. During this reaction, potassium is consumed, and acid is generated, resulting in progressively lower values of log $\frac{a_{K^+}}{a_{H^+}}$; consequently, solution pH's passed below 4, and adjustment of pH was no longer necessary. Of extreme interest in this pair of tests is the fact that both leach solutions contained approximately 3 gm/l dissolved copper (3.4 - H_2SO_4, 2.65 - KCl), but the 0.2 m potassium solution contained less than 0.01 gm/l dissolved iron while the pH = 2 analogue carried over 1 gm/l.

Detection of the product phases formed during these and other reactions was, unfortunately, not possible for two reasons. In the first place, the mineral transformations proceeding occur on grain surfaces. Once a shell of product phase, e.g., kaolinite, has been formed on a reactant grain such as muscovite, reaction kinetics are reduced and are controlled by diffusion rates of material between the reactant core and the leach solution. The surface coating of product material on the grain should be very thin because the

limited duration of the experiments carried out precluded total conversion of reactant grains. Secondly, the detection of the thin reaction rinds would require very high magnification, such as provided by scanning electron microscropy. Preparation of samples for this analysis would require consolidation of the ground sample material followed by polishing. This latter procedure would more than likely destroy any thin surface films on grains. Consequently, the formation or destruction of minerals during the leaching experiments is extremely difficult if not impossible to detect, and, thus, aqueous solution analyses must be relied upon to monitor changes going on within the rock column.

Dissolution of Copper

In an earlier discussion concerning copper-mineral stabilities, it was suggested that in the pH range 2-3 concentrations of dissolved copper minerals might be limited by saturation with respect to chalcanthite ($CuSO_4 \cdot 5H_2O$). To evaluate this suggestion, all effluent leach solutions from the various tests have been plotted on Figure 6 in terms of the variables pH and log $(a_{Cu^{++}})(a_{SO_4^{=}})$ corresponding to the ion activity product for chalcanthite solubility. Line A-B on this diagram corresponds to a constant value of this latter ion activity product which is independent of solution pH. In the pH range 2-3, the product log $(a_{Cu^{++}})(a_{SO_4^{=}})$ for effluent solutions appears to approach a limiting value of -3.5, and as pH increases above 3, the maximum value of this ion activity product decreases. If a constant value is assumed for $a_{SO_4^{=}}$ above pH = 3, then a decrease in the ion activity product corresponds to a decrease in dissolved copper. A decrease in dissolved copper with decreased acidity would correspond to saturation with a copper-bearing phase such as CuO (tenorite) or $Cu_4(OH)_6SO_4$ (brochantite)

whose solubility, unlike chalcanthite, is inversely related to pH. The equilibrium ion activity product for chalcanthite is $10^{-2.93}$ (Garrels and Christ[3]), which is slightly higher than the maximum value obtained in these leaching studies ($10^{-3.5}$). This difference may indicate that 1.) all solutions below pH = 3 were undersaturated with respect to chalcanthite, or 2.) the equilibrium ion activity product of chalcanthite given by Garrels and Christ is in error, or 3.) some higher order hydrate of $CuSO_4$ than chalcanthite is forming during the experiments (e.g., boothite - $CuSO_4 \cdot 7H_2O$).

Dissolution and Transport of Iron

In copper deposits which have undergone oxidation and enrichment, leached outcrops contain significant amounts of iron oxides collectively termed limonite. The solubility product of such oxides is represented by the term log $\frac{a_{Fe^{+3}}}{a^3_{H^+}}$, the value of which varies for different phases, such as hematite and goethite. The identity of the stable iron oxide in a given situation apparently depends upon grain size, solution pH, and the mechanism operating during formation (Langmuir and Whittemore[10]). A wide range in concentrations of dissolved iron was observed during the leaching studies described here, and in order to test the hypothesis that dissolved iron concentrations are limited by saturation with respect to iron oxide, values of log $\frac{a_{Fe^{+3}}}{a^3_{H^+}}$ of effluent leach solutions are plotted in Figure 7 as a function of solution pH.* Shown along the left-hand margin of this figure are the ranges in equilibrium constants for hydrolysis

*The conversion of concentrations of dissolved iron to thermodynamic activities of Fe^{+3} requires compensation for hydroxide complexes and oxidation potential, a calculation which is readily carried out knowing solution pH and Eh.

of amorphous $Fe(OH)_3$, lepidocrosite, goethite, and hematite in terms of H^+, as given by Langmuir and Whittemore[10].

Examination of the data in Figure 7 shows near-equilibrium of solutions with respect to the most soluble phase, amorphous $Fe(OH)_3$, only in the pH range approximately 2.7 - 4.5, and beyond these limits log $\frac{a_{Fe^{+3}}}{a^3_{H^+}}$ decreases in value in both directions of pH change. Line A-B on Figure 7 corresponds to solutions having a constant value of $a_{Fe^{+3}} = 10^{-3.6}$ such that log $\frac{a_{Fe^{+3}}}{a^3_{H^+}}$ decreases by 3 units for each unit decrease in pH, and this line appears to parallel the upper limit of data points plotted on the diagram at pH < 3. If it is assumed that below pH = 3, solution pH is proportional to $a_{SO_4^=}$, then a group of lines having the same general trend as line A-B but having slightly different slopes would correspond to saturation of the solution with phases such as borgstromite or coquimbite. Because the data plotted in this pH range show significant variation in $a_{Fe^{+3}}$, it is suggested that iron concentrations in strongly acid sulfate solutions are controlled by saturation with respect to iron-sulfate compounds, but above pH = 3, the iron content is buffered by precipitation of amorphous $Fe(OH)_3$. The low concentrations of iron observed in the KCl leach columns at pH ∿ 4 are attributed to saturation with respect to $KFe_3(SO_4)_2(OH)_6$ - jarosite, a yellow-brown stain which appeared on the order of 3 inches below the top of these columns.

Anion-Cation Effects and Leaching Efficiency

Leaching efficiencies using solutions of K_2SO_4 and KCl to evaluate anion effects are essentially identical (Table 7) and indicate negligible differences between the two anions. When two solutions containing chlorides of sodium and potassium respectively were used to examine cation effects, the leaching efficiency of the

former was seen to be less than one-half as effective as the latter. These observations show that the critical factor governing leaching efficiency in these studies was the presence of the potassium ion, which controlled, to a large extent, the degree and manner of interaction of the leach solution with rock-forming silicates.

Solutions V(KCl, pH = 4) and VI(H_2SO_4 only, pH = 2) exhibited similar leaching percentages, but the KCl solution achieved that percentage in about one-half the time required for the sulfuric acid-only solution; thus, the leaching efficiency for the former was approximately 50% higher than for the latter. Progressive chemical analyses of the H_2SO_4 solution presented in Table 8 show that even though copper concentrations reach 3.4 gm/l, dissolved iron also rises rapidly as pH drops below a value of 3; the final iron content reached in this test was 1.2 gm/l, whereas in the solution containing KCl at a higher pH, dissolved iron was negligible.

The importance of the copper-iron ratio in the leach solution, as mentioned briefly above, lies in the fact that if electrolytic exchange is used to collect dissolved copper according to

$$2Fe_s + 3Cu^{+2} = 2Fe^{+3} + 3Cu_s$$

then, according to the equilibrium constant for this reaction,

$$Keq = \frac{a^2_{Fe^{+3}}}{a^3_{Cu^{+2}}}$$

If large concentrations of iron are present in solution, the solution will also retain proportionally higher concentrations of copper, and the exchange kinetics will be inhibited. As a result, the efficiency of the exchange process will be diminished. Looking at the mass ratios of Cu:Fe from the H_2SO_4 and KCl leach solutions given in Table 5, it may be seen that the latter solution maintains

a significantly higher value during the initial leaching cycles. During the later stages, however, when the effluent is analyzed after the cementation step, both solutions have similar copper: iron ratios, but the KCl solution contains less dissolved copper. These results appear to support the initial suggestions concerning exchange efficiency and indicate the importance of maintaining a low concentration of iron in the leach solution.

The Tyrone material contains large amounts of pyrite relative to chalcocite (Table 1), and the high iron contents of the sulfuric acid solution are a result of dissolution of this former sulfide mineral. Analysis of dissolution of iron described above indicates that at pH's below about 3, the concentration of this species is apparently buffered by the high sulfate contents of these solutions, whereas in the more basic leach solutions the iron contents are limited by saturation with respect to amorphous $Fe(OH)_3$ and, in the presence of large amounts of potassium below pH = 4, jarosite.

Summary

Methods currently in use for leaching low grade chalcocite ores employ strongly acid sulfate solutions to dissolve and transport copper; ferric sulfate is also present in solution to aid in oxidation. The application of solution-mineral equilibrium considerations to such leaching procedures predicts that irreversible reactions proceeding between the leach solution and rock-forming silicate minerals will result in significant acid consumption as the silicates first dissolve and then alter to clay minerals. The precipitation of clay minerals as solutions become neutralized is likely to contribute to coating of copper-bearing sulfide minerals and reduction in rock permeability. Studies of chemical

[illegible] or isomorphous replacement

interaction between leach solutions and rocks containing low-grade copper sulfide have provided the following information concerning material from Tyrone:

1.) Strongly acid solutions can be used to reduce the reaction capacity of silicate rocks by production of acid-stable clay minerals.

2.) Moderately acid solutions containing on the order of 0.2 m dissolved potassium equilibrate rapidly with rock-forming silicates, and, thus, experience minimal acid consumption.

3.) At dissolved silica concentrations approaching saturation with respect to amorphous silica in KCl solutions, acid is generated by equilibrium reaction between K-feldspar and dickite.

4.) Moderately acid potassium-bearing solutions are capable of transporting nearly the same amount of dissolved copper as strongly acid solutions, but with significantly lower concentrations of dissolved iron and higher leaching efficiency.

5.) KCl solutions containing large ratios of dissolved copper: iron undergo more efficient electrolytic exchange of copper than standard leach solutions which contain large concentrations of iron.

6.) Comparative tests using a variety of dissolved electrolytes in the leach solution (KCl, NaCl, K_2SO_4) indicate that increases in leaching efficiency using KCl solutions can be attributed to dissolved potassium.

These tests clearly indicate the importance of consideration of stability relations of gangue silicate minerals which make up the

majority of the rock mass containing leach-grade copper sulfides. Of course, copper deposits other than Tyrone, which was the source of material examined in this study, may contain a different assemblage of rock-forming silicates and, thus, will require further tests of the sort described here to determine a leach solution composition which will optimize the processes of leaching and recovery.

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Table 1. Representative Whole Rock Chemical Analysis of Tyrone Leach Material

Species	Wt.%
S	7.4
Na_2O	0.3
K_2O	4.4
MgO	0.3
CaO	0.04
SiO_2	61.2
Fe	6.9
Cu	0.29
Al_2O_3	16.2
H_2O	5.1
Total	102.1

Table 2. Optical Mineralogic Analysis of Tyrone Leach Material

Mineral	Estimated Vol%
Quartz	16-18
Muscovite-sericite	40-42
Orthoclase	32 highly altered to sericite
Kaolinite	trace
Jarosite	trace
Opaque minerals	10 chiefly pyrite

Table 3. Equilibrium Constants at 25°C and 1 atm for Reactions Used to Construct Figures 1, 2, 4, and 5. Thermodynamic data for muscovite and gibbsite from Ulbrich and Merino (5): Remaining data from Helgeson (4).

Reaction	log Keq
Kaolinite-Gibbsite	
$Al_2Si_2O_5(OH)_4 + 5H_2O = 2Al(OH)_3 + 2H_4SiO_4$	-9.89
Pyrophyllite-Kaolinite	
$Al_2Si_4O_{10}(OH)_2 + 5H_2O = Al_2Si_2O_5(OH)_4 + 2H_4SiO_4$	-5.78
Muscovite-Gibbsite	
$KAl_3Si_3O_{10}(OH)_2 + H^+ + 9H_2O = 3Al(OH)_3 + 3H_4SiO_4 + K^+$	-10.15
Muscovite-Kaolinite	
$2KAl_3Si_3O_{10}(OH)_2 + 2H^+ + 3H_2O = 3Al_2Si_2O_5(OH)_4 + 2K^+$	9.33
K-feldspar (microcline)-Muscovite	
$3KAlSi_3O_8 + 2H^+ + 12H_2O = KAl_3Si_3O_{10}(OH)_2 + 6H_4SiO_4 + 2K^+$	-12.24
K-feldspar (microcline)-Kaolinite	
$2KAlSi_3O_8 + 2H^+ + 9H_2O = Al_2Si_2O_5(OH)_4 + 4H_4SiO_4 + 2K^+$	-5.04
K-feldspar (microcline)-Dickite	
same reaction as K-feldspar-Kaolinite	-5.97
K-feldspar (microcline)-Pyrophyllite	
$2KAlSi_3O_8 + 2H^+ + 4H_2O = Al_2Si_4O_{10}(OH)_2 + 2H_4SiO_4 + 2K^+$	0.74

Table 4. Aqueous Species Afforded for in Solution Distribution Calculations. Dissociation data taken from Helgeson (4) and Langmuir (11). Solution analyses also included Eh to enable distribution between oxidized and reduced species.

Anions \ Cations	Cu^{+}	Cu^{+2}	Fe^{+2}	Fe^{+3}	Al^{+3}	K^{+}	H^{+}
OH^{-}		$Cu(OH)^{+}$	$Fe(OH)_3^{-}$	$Fe(OH)^{+2}$	$Al(OH)^{+2}$		H_2O
			$Fe(OH)_4^{=}$	$Fe(OH)_2^{+}$	$Al(OH)_4^{-}$		
				$Fe(OH)_3^{o}$			
				$Fe(OH)_4^{-}$			
Cl^{-}	$CuCl_2^{-}$	$CuCl^{+}$		$FeCl^{+2}$			
	$CuCl_3^{=}$	$CuCl_2^{o}$		$FeCl_2^{+}$			
		$CuCl_3^{-}$		$FeCl_3^{o}$			
		$CuCl_4^{=}$		$FeCl_4^{-}$			
$SO_4^{=}$		$CuSO_4^{o}$	$FeSO_4^{o}$	$FeSO_4^{+}$	$AlSO_4^{+}$	KSO_4^{-}	HSO_4^{-}
$S^{=}$							HS^{-}
							H_2S^{o}

Table 5. Effleunt Analyses from Large-Column Tests. Comparison of H_2SO_4-alone and 0.2 m KCl-H_2SO_4 solutions with input pH adjusted to 2. Copper and iron concentrations in ppm, Eh in mv.

	H_2SO_4 only					KCL-H_2SO_4				
Cycle	Cu	Fe	$\frac{Cu}{Fe}$	Eh	pH	Cu	Fe	$\frac{Cu}{Fe}$	Eh	pH
1	16	0.5	32.0	440	5.7	130	1.3	100.0	470	5.8
2	55	1.3	42.3	460	5.7	250	2.8	89.3	475	5.5
3	430	28	15.4	490	4.1	650	19	34.2	490	4.8
4	960	360	2.7	470	4.3	1025	59	17.4	510	4.4
5	1500	590	2.5	550	3.4	2000	50	40.0	550	4.1
					Cementation begun					
6	1890	3750	0.50	610	2.7	1500	3950	0.38	530	3.8
7	650	4530	0.14	620	2.4	328	2770	0.12	540	3.4
8	700	9250	0.08	610	2.7	110	2690	0.04	520	3.3

Table 6a. Effluent Analysis for First 0.2 m $KCl-H_2SO_4$ Solution with pH Adjusted to 4. Analyses in ppm except Eh in mv. Activities of aqueous species calculated as described in text.

Date	K	Cl	Cu	Fe	Al	SO_4	Mg	Eh	pH	SiO_2	$\log \frac{a_{K^+}}{a_{H^+}}$	$\log a_{H_4SiO_4}$
9/2	8000	-	170	0.7	2.4	540	-	570	5.3	20	4.5	-3.5
9/11	9000	-	120	0.0	1.4	490	-	710	4.8	18	4.0	-3.5
10/21	6900	6100	130	0.0	2.5	540	-	620	5.8	11	4.4	-3.7
10/30	6800	6240	190	0.2	2.0	700	-	640	5.3	14	4.4	-3.6
11/10	6600	6680	480	0.0	9.0	1150	10	625	4.9	18	4.0	-3.5
11/13	6700	6760	1200	0.0	28	2180	10	625	4.3	28	3.4	-3.3
11/14	6900	6800	1600	0.0	34	2350	13	630	4.4	34	3.5	-3.2
11/24	6400	6720	1900	1.5	100	3580	10	640	4.0	53	3.0	-3.0
12/1	6900	6970	2100	0.3	90	3750	10	650	4.1	50	3.2	-3.0
New Solution - 0.2 m KCl, no pH adjustment												
2/2	8000	7430	1220	0.3	48	2060	6	610	4.2	25	3.3	-3.4
2/6	8000	7410	1300	0.7	50	2100	6	615	4.1	30	3.2	-3.3

Table 6b. Partial Analyses of Effluents for Second 0.2 m $KCl-H_2SO_4$ Solution with pH Adjusted to 4. Analyses in ppm except Eh in mv. Activities of aqueous species calculated as described in text.

Date	K	Cu	Fe	Eh	pH	SiO_2	$\log \frac{a_{K^+}}{a_{H^+}}$	$\log a_{H_4SiO_4}$
3/7	7360	290	1.1	490	4.2	7	3.3	-3.93
3/9	8240	540	0	530	4.8	14	4.0	-3.63
3/11	7860	730	0.1	510	4.9	18	4.0	-3.52
3/13	8200	1000	0.3	-	4.4	22	3.55	-3.43
3/14	7990	1200	0.2	600	4.3	26	3.4	-3.36
3/18	5620	2460	0.4	610	4.6	23	3.6	-3.41
3/19	6490	1630	0.3	560	4.3	42	3.4	-3.15
3/20	5720	1840	0.4	560	4.1	48	3.1	-3.10
3/21	5700	2030	1.1	560	4.2	53	3.2	-3.05
3/24	5900	2530	1.2	580	3.9	55	2.9	-3.04
3/25	5900	2500	4.8	590	3.8	61	2.8	-2.99
3/26	5700	2650	8.3	590	3.9	67	2.9	-2.95

Table 7. Leaching Efficiency Data for Various Types of Solutions Examined. Copper and iron concentrations in ppm, time in hours, copper extracted as percent of original wt% in rock, pH adjustments with H_2SO_4.

I. 0.2 m $SO_4^{=}$ as K_2SO_4 and H_2SO_4, pH = 2

II. 0.2 m Cl^- as KCl, pH = 2

III. 0.2 m Na as NaCl, pH = 2

IV. Distilled water

V. 0.2 m KCl, pH = 4 with H_2SO_4

VI. H_2SO_4, pH = 2

	Column					
	I	II	III	IV	V	VI
% Cu extracted	70	72	33	1	37	42
Final Cu conc.	1600	1300	800	100	1300	3400
Final Fe conc.	700	1200	100	2	0.7	1200
Time operated	129	126	122	90	590	1020
$\frac{\text{\% Cu extracted}}{\text{Time operated}}$	0.54	0.57	0.27	0.01	0.063	0.041

Table 8. Selected Effluent Analyses for pH = 2 H_2SO_4 Solution. Test initiated 2/17. Analyses in ppm except Eh in mv. Activities calculated as described in text.

Date	K	Cu	Fe	Al	SO_4	Mg	Ca	Na	Eh	pH	SiO_2	$\log \frac{a_{K^+}}{a_{H^+}}$	$\log a_{H_4SiO_4}$
2/26	110	770	4.2	34	1850	39	140	30	560	4.1	35	1.56	-3.23
3/8	150	1500	6.0	100	4000	60	200	54	580	4.0	40	1.59	-3.18
4/20	114	1900	8.5	200	4160	37	26	160	770	3.0	78	0.47	-2.89
6/10	2.9	2700	380	300	9760	26	26	16	760	2.8	120	-1.34	-2.70
7/12	3.0	3400	1200	520	13800	22	5.5	120	740	2.15	150	-1.96	-2.60

Table 9. Whole Rock Chemical Analyses of Segments from Column V, Table 7, after Completion of Leaching Tests. Analyses in wt%. Length intervals represent segments measured from the top of the column of 15 inches total length.

	Wt% in Column Sections			
Species	0-1 inch	4-5 inch	7-8 inch	14-15 inch
S	6.43	5.98	6.03	6.01
Na_2O	0.64	0.67	0.86	0.67
K_2O	3.36	3.32	3.22	3.46
MgO	0.10	0.09	0.14	0.07
CaO	0.00	0.00	0.00	0.00
SiO_2	60.0	60.2	61.6	61.4
Fe	7.51	7.17	7.59	7.06
Cu	0.17	0.19	0.21	0.19
Al_2O_3	16.2	15.8	16.2	15.0
H_2O	4.73	4.33	4.29	5.41
Total	99.14	97.95	100.14	99.27

Figure 1. Stability relation among minerals in the system K_2O-Al_2O_3-SiO_2-H_2O at 25°C and 1 atm in the presence of an aqueous phase. Reactions and equilibrium constants defining phase boundaries are given in Table 3.

Figure 2. Projection of solution compositions attending calculated irreversible dissolution of K-feldspar plus kaolinite in solutions beginning with pH = 7(A) and pH = 2(C), as described in the text. The upper limit on dissolved silica is defined by amorphous silica saturation.

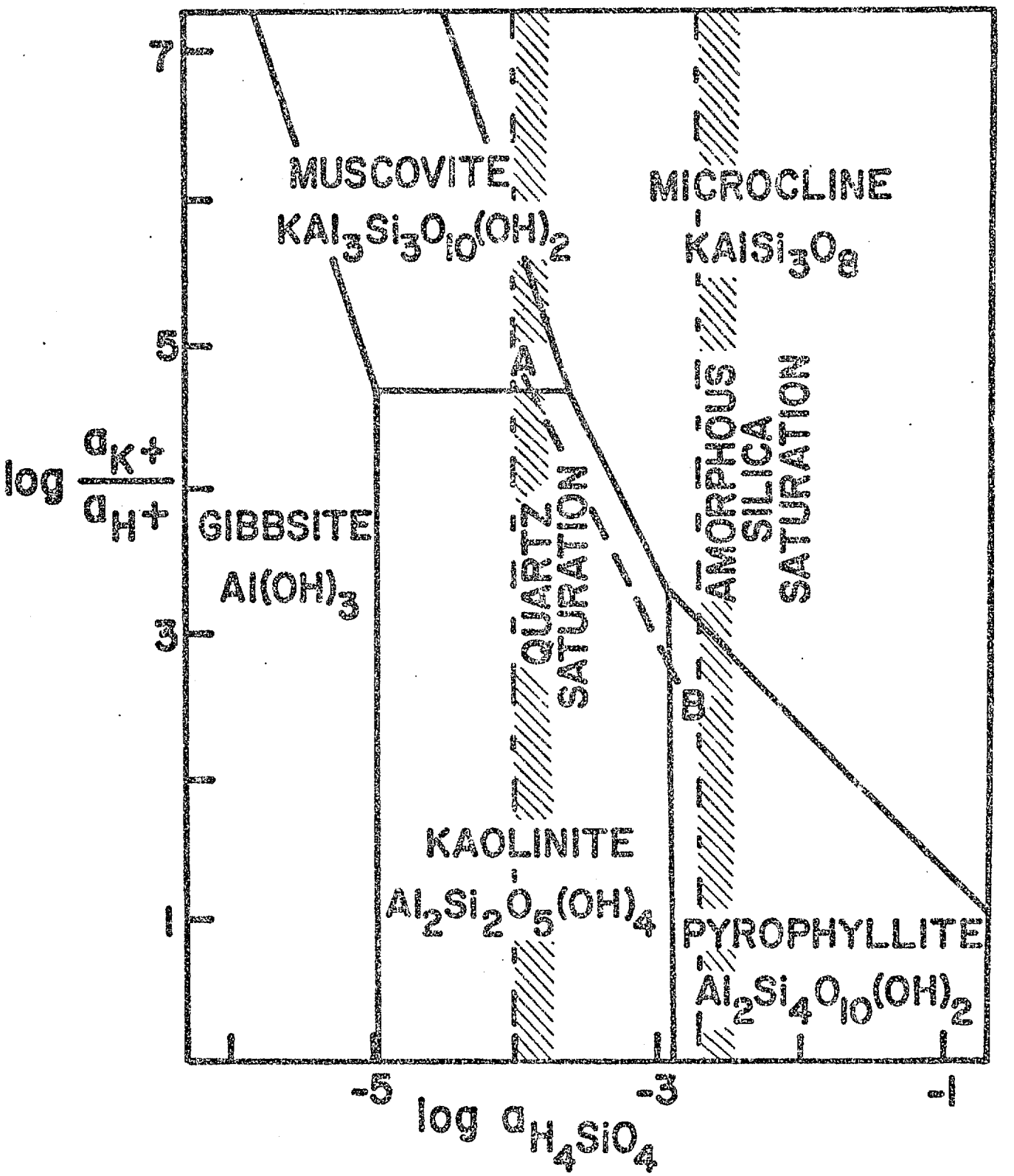
Figure 3. Stability relations among minerals in the system CuO-SO_3-H_2O at 25°C and 1 atm in the presence of an aqueous phase. Phase boundaries were calculated using thermodynamic data from Garrels and Christ.[3] Contours define saturation boundaries for solids in solutions having log $a_{Cu^{++}}$ = -1, -2, and -3.

Figure 4. Plot of log $\frac{a_{K^+}}{a_{H^+}}$ and log $a_{H_4SiO_4}$ values of effluent solutions resulting from leaching using 0.2 m KCl with pH adjusted = 4. Line A-B corresponds to equilibrium between K-feldspar and metastable dickite. Solution compositions are given in Tables 6a (filled circles) and 6b (X's).

Figure 5. Plot of log $\frac{a_{K^+}}{a_{H^+}}$ and log $a_{H_4SiO_4}$ values of effluent solutions resulting from leaching using: a) pH = 2 sulfuric acid solution, Table 8 (X's), b) pH = 4, 0.2 m KCl solution, Table 6b (filled circles).

Figure 6. Effluent values of log $a_{Cu^{++}}a_{SO_4}$ and pH for all leach solutions. X's correspond to 0.1 m KCl solution with pH adjusted = 4. Line A-B corresponds to a constant ion activity product of Cu^{++} and $SO_4^{=}$, and line C-D represents variation of $a_{Cu^{++}}$ with pH at fixed $a_{SO_4^{=}}$. The chalcanthite saturation surface is calculated using data from Garrels and Christ.[3]

Figure 7. Effluent values of log $\frac{a_{Fe^{+3}}}{a^3_{H^+}}$ and pH for all leach solutions. Line A-B corresponds to a solution having fixed log $a_{Fe^{+3}}$ = -3.6 such that the value of log $\frac{a_{Fe^{+3}}}{a^3_{H^+}}$ varies directly with pH, and line C-D represents saturation with respect to iron oxide such that log $\frac{a_{Fe^{+3}}}{a^3_{H^+}}$ is independent of pH. Iron oxide saturation surfaces are from Langmuir and Whittemore.[10]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MICROCLINE
$KAlSi_3O_8$
GIBBSITE
$Al(OH)_3$
KAOLINITE
$Al_2Si_2O_5(OH)_4$
PYROPHYLLITE
$Al_2Si_4O_{10}(OH)_2$
QUARTZ SATURATION
AMORPHOUS SILICA SATURATION
A
B
7
5
3
1
-5
-3
-1
$\log \frac{a_{K^+}}{a_{H^+}}$
$\log a_{H_4SiO_4}$

Figure 1

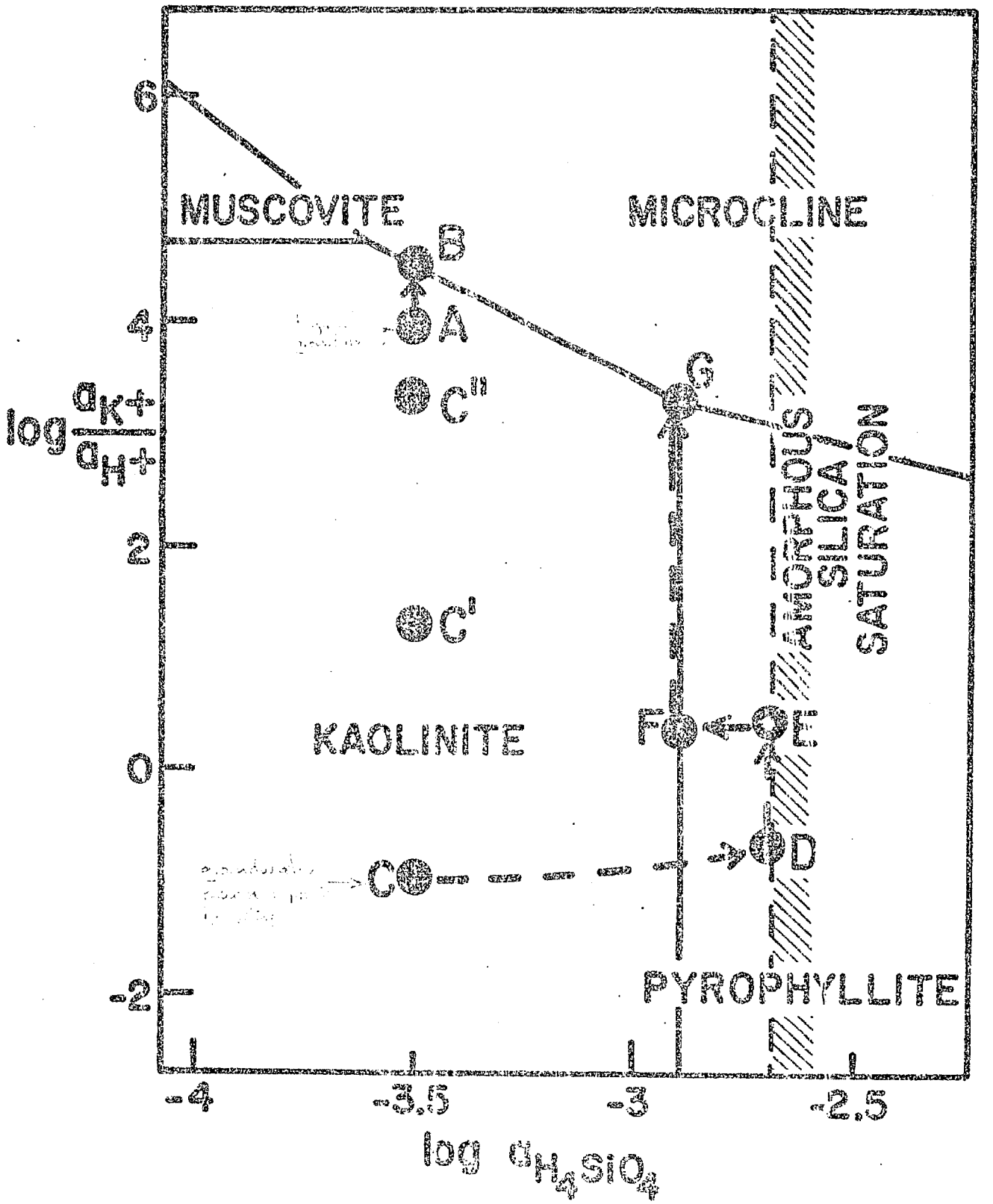
MUSCOVITE
MICROCLINE
B
A
C''
G
C'
KAOLINITE
F
E
D
C
PYROPHYLLITE
SOLID
SILICA
SATURATION
6
4
2
0
-2
-4
-3.5
-3
-2.5
log $\frac{a_{K^+}}{a_{H^+}}$
log $a_{H_4SiO_4}$

Figure 2

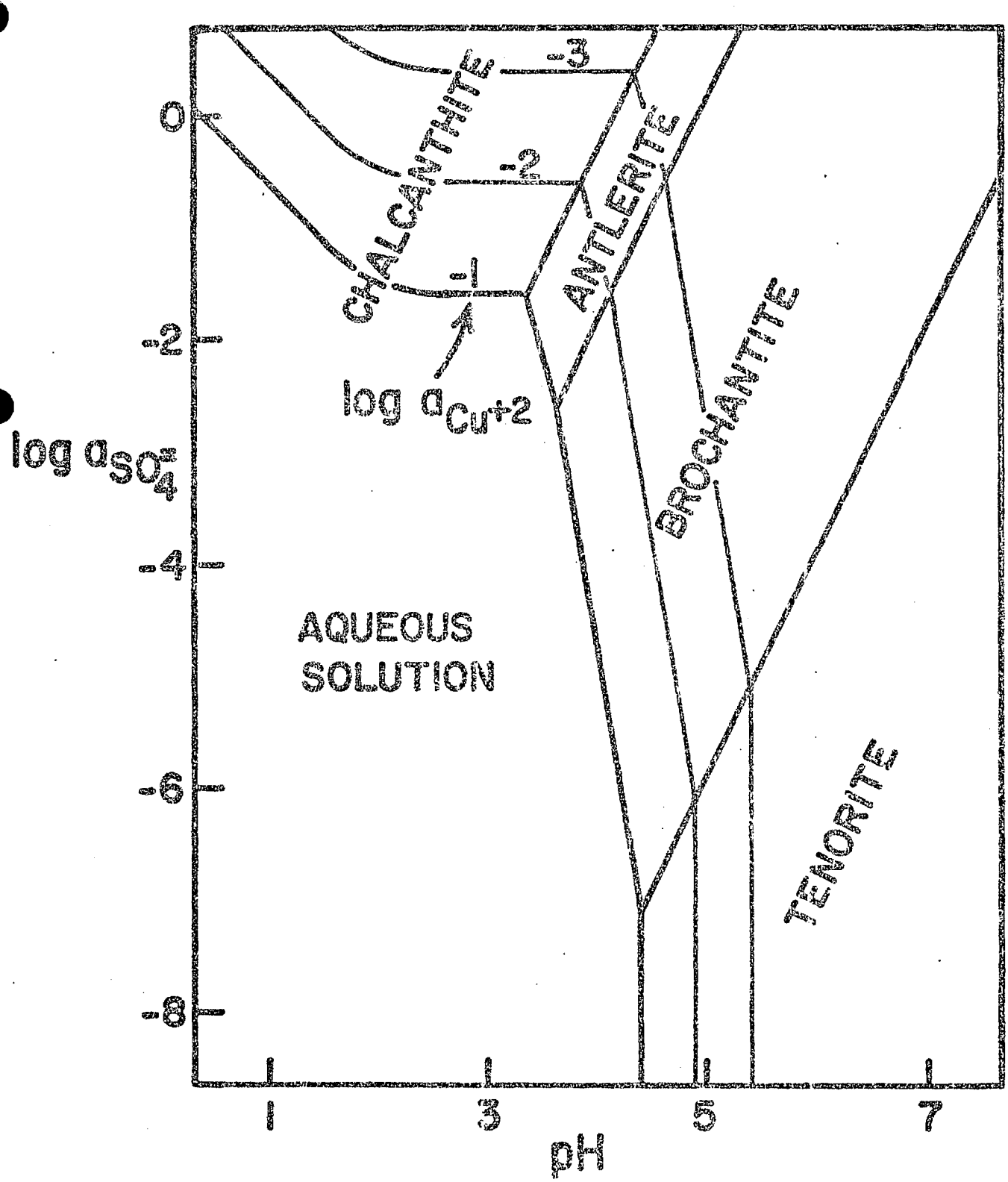
CHALCANTHITE
ANTLERITE
BROCHANTITE
TENORITE
AQUEOUS SOLUTION
-3
-2
-1
log $a_{Cu^{+2}}$
log $a_{SO_4^{=}}$
0
-2
-4
-6
-8
1
3
5
7
pH

Figure 3

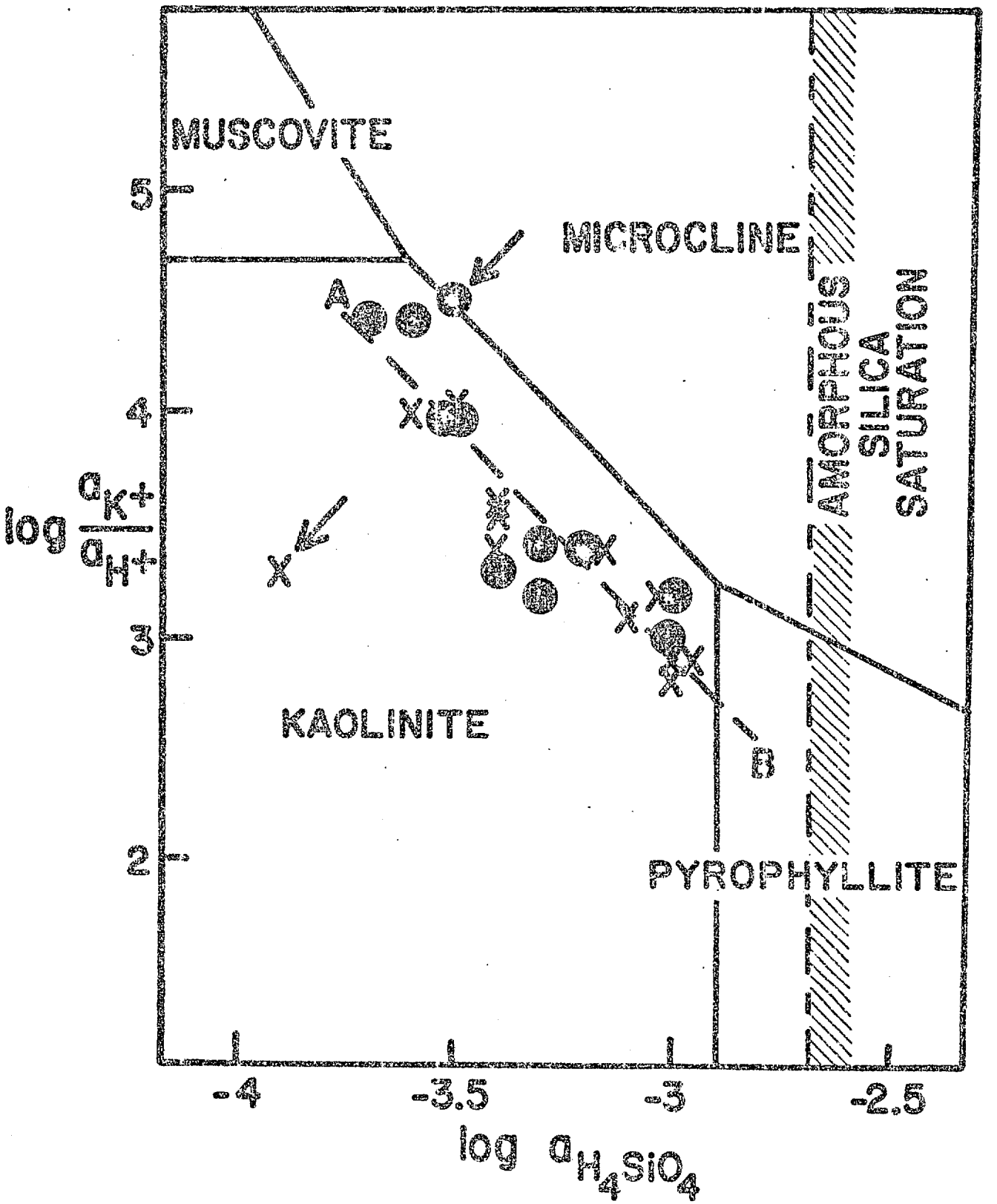
MUSCOVITE
MICROCLINE
KAOLINITE
PYROPHYLLITE
AMORPHOUS SILICA SATURATION
A
B
5
4
3
2
-4
-3.5
-3
-2.5
log $\frac{a_{K^+}}{a_{H^+}}$
log $a_{H_4SiO_4}$

Figure 4

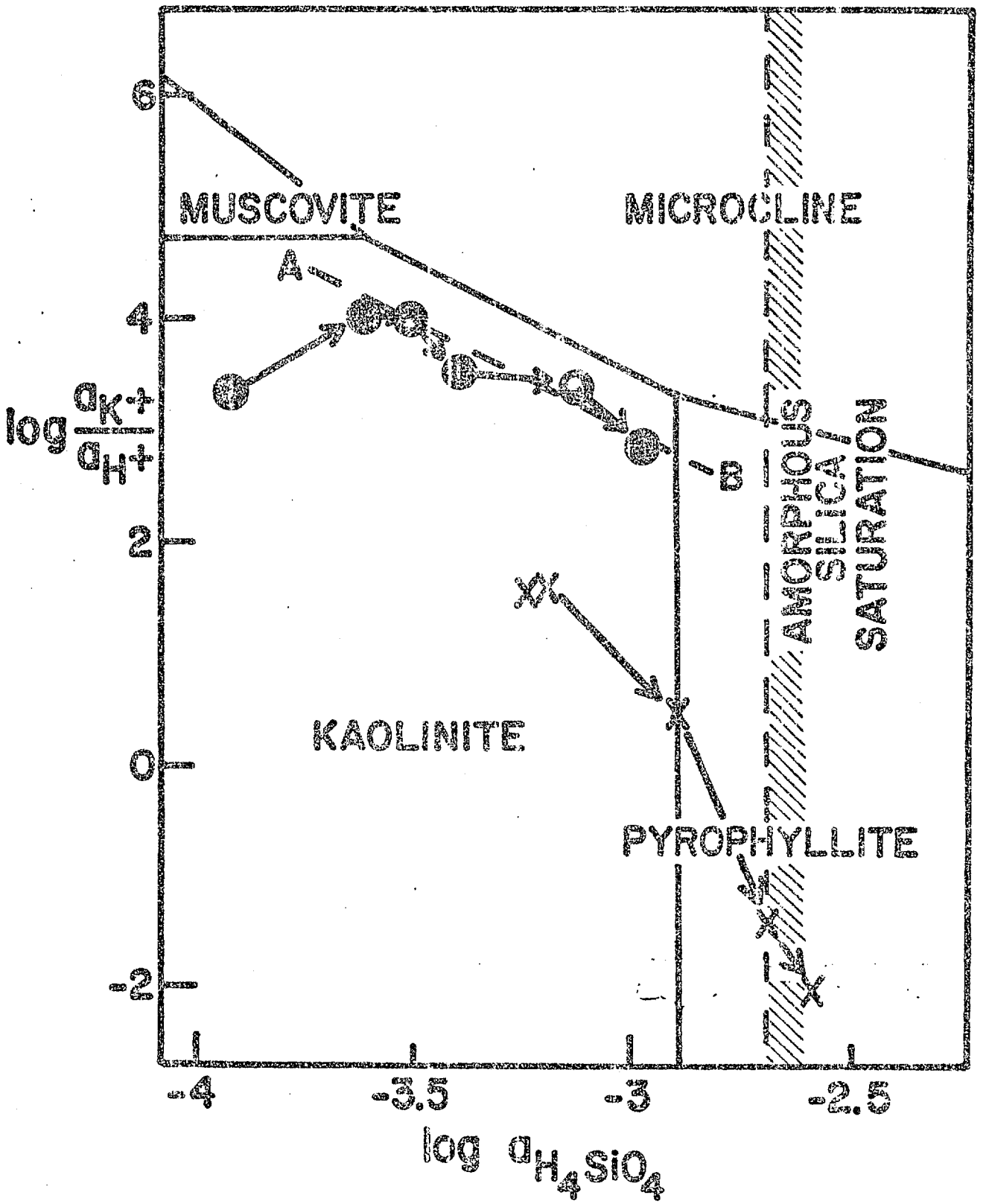
MUSCOVITE
MICROCLINE
A
B
KAOLINITE
PYROPHYLLITE
AMORPHOUS SILICA SATURATION
6
4
2
0
-2
-4
-3.5
-3
-2.5
log $\frac{a_{K^+}}{a_{H^+}}$
log $a_{H_4SiO_4}$

Figure 5

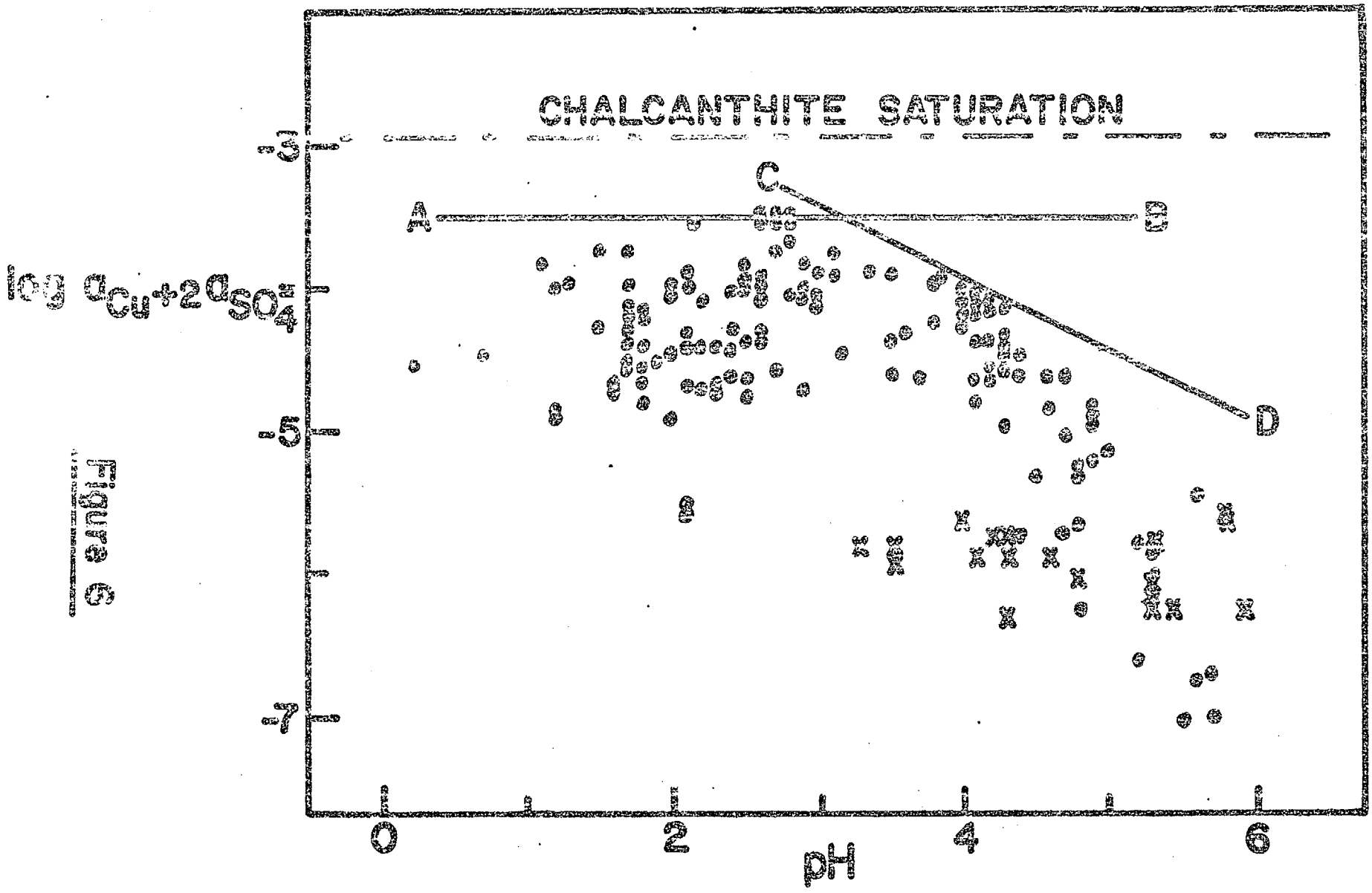

Figure 6

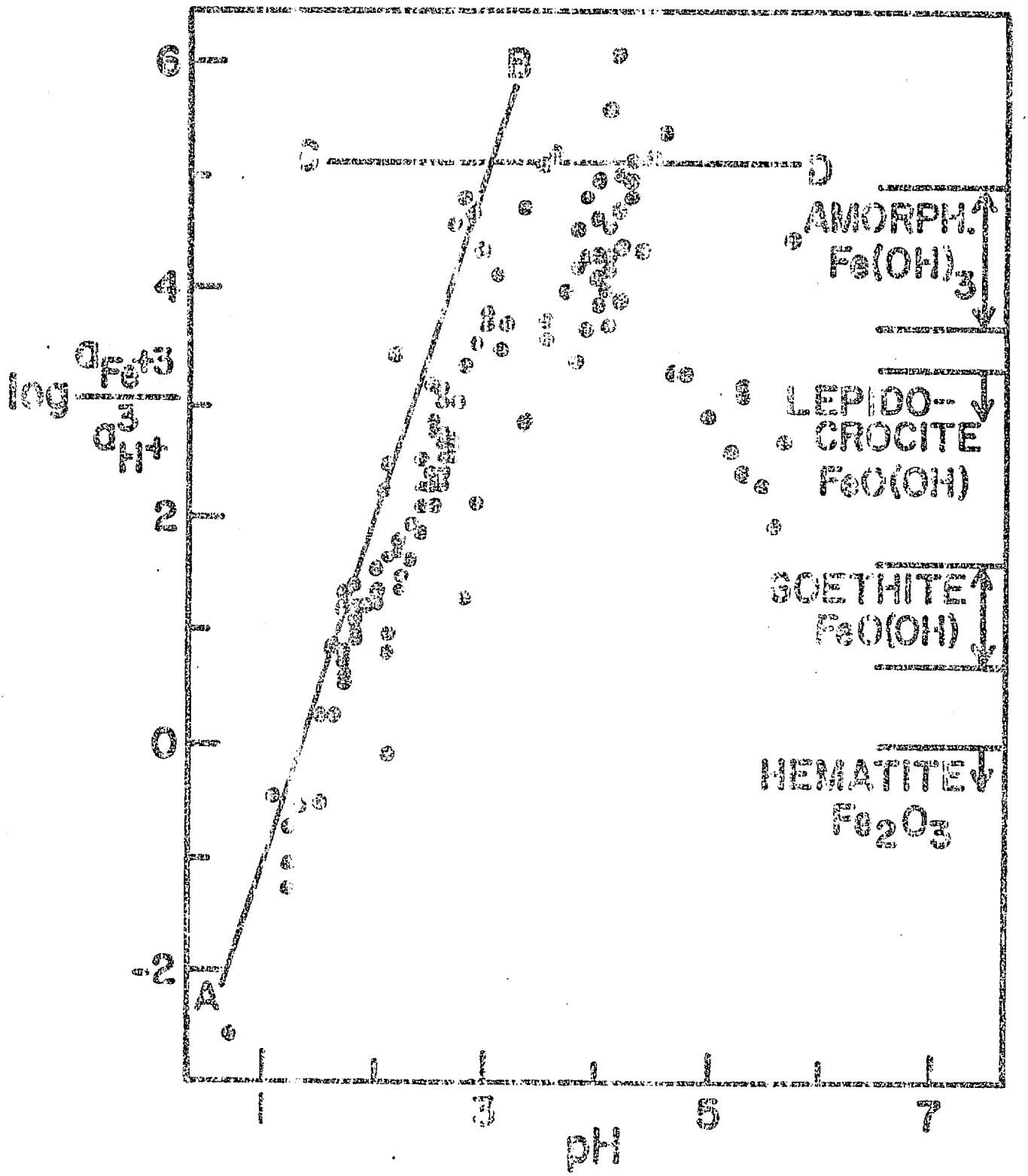
6
4
2
0
-2
log $\frac{a_{Fe^{+3}}}{a^3_{H^+}}$
A
B
C
D
AMORPH.
$Fe(OH)_3$
LEPIDO-
CROCITE
FeO(OH)
GOETHITE
FeO(OH)
HEMATITE
Fe_2O_3
1
3
5
7
pH

Figure 7

EXTRA COPY

FIELD TESTING PROGRAM AT ASARCO'S POSTON BUTTE SITE

Summer 1979

by

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INTRODUCTION

The Bureau of Mines has conducted in situ leaching research in copper deposits for several years. Although most of the work has been directed towards shallow deposits, there has been significant interest in deep leaching possibilities throughout the program.

In response to a Bureau call for show-of-interest, ASARCO expressed a desire to work jointly with the Bureau's Twin Cities Research Center in evaluating the potential of deep deposits for possible in situ leaching. A cooperative memorandum of agreement was signed by the Bureau and by ASARCO to execute an investigation of leaching at ASARCO's Poston Butte deposit. The investigation was to take place in two phases: The first phase to determine the extent of fracturing in the formation and its permeability and the second to determine pumping and leaching characteristics of the formation through a five-spot pattern of vertical wells (if the first phase tests looked promising).

This report describes results of activities conducted under phase I during the summer of 1979 at ASARCO's Poston Butte site.

GEOLOGY/HYDROLOGY

The Poston Butte vicinity west of Florence, Ariz., has supported copper exploration activity for the past several years. CONOCO has conducted most of the work, including sinking a test shaft and constructing a pilot processing plant for a possible open pit mine (1). The CONOCO operation has been mothballed until copper mining economics improves.

Adjacent to the CONOCO property, ASARCO has rights to a block of land which shows low copper values at depths below those considered suitable for open pit mining. Exploration activity at ASARCO's site has been primarily coring. The copper deposit occurs in a highly fractured granite underlying the Gila conglomerate. The portion under the ASARCO property is in the downthrown block of a graben which has placed it several hundred feet deeper than it is under the adjacent CONOCO property. At the test site approximately 1,200 feet of alluvium and Gila conglomerate overlie the granite (fig. 1). Copper mineralization in the upper 500 to 800 feet of granite is primarily oxide grading below that into sulfides. Early estimates on copper in the oxide zone ranged around 100 million tons of ore at 0.4 percent copper (2, 3).

According to ASARCO's hydrology consultants, Water Development Corp., the static water table near the site 3 or 4 years ago was around 225 feet below ground level. The water level in the test well was 232.5 feet below ground level. Ground level at the test site is approximately 1,470 feet above sea level. The Gila River bed, about 1-1/2 miles south of the site, is at elevation 1,440 feet.

TESTING PROGRAM

The tests conducted during the summer of 1979 were directed towards two objectives: determining the extent of fracturing in the ore zone and estimating the permeability of the formation. Both of these objectives could be achieved by coring with an N size bit and conducting permeability tests in the resultant core hole. Besides providing

fracture data, the drill core afforded an excellent opportunity to obtain additional grade data. Geophysical logs could also be run in the core hole if desired. Accordingly, a decision was made to drill at least one core hole to be followed up with additional holes if the data warranted them.

Under the terms of the agreement, ASARCO contracted with a driller to rotary drill three holes to a depth approximately 50 feet above the conglomerate-granite contact. Each hole was cased with 3-1/2-inch ID steel casing which was cemented over the bottom 50 feet. These holes--ASARCO numbers 36, 37, and 38--were drilled and cased in January 1979.

Hole 38 was selected for the initial coring and testing work because it was located near the center of the ore body. The Bureau of Mines hired another drilling contractor to core from the bottom of the casing to a depth beneath the oxide-sulfide contact.

The contractor drilled with a Longyear Model 44 coring rig using an NQ triple-tube (split inner tube) core barrel. The early footage was cored with Pal Mix Super X drilling mud because it is less prone to plug permeability channels in the formation (4).[1] About 4 pounds of Super X were mixed per 100 gallons to give a funnel viscosity of 40 seconds. When permeability tests were attempted, the mud could be broken down with mud breaker Pal Mix 150F.

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[1]Reference to specific equipment (or trade names or manufacturers) does not imply endorsement by the Bureau of Mines

When the hole reached 1,518 feet, drilling was halted for permeability testing and geophysical logging. Breaking down the mud and flushing with water caused sloughing so the remaining footage was drilled with Magcobar's Kwik-Thik bentonite mud. The mix was approximately 2 sacks Kwik-Thik with 4 quarts of CMC, 1 quart caustic, and 1 quart soda ash. For subsequent permeability tests the hole was simply flushed with fresh water. Water for drilling and testing was obtained from the water tanks in Florence.

TABLE 1. - Summary of core drilling

Start of drilling	June 4	10 weeks
End of drilling	August 8	
Duration of drilling, weeks	10	
Drilling 200 feet cement in casing, hours	37	
Permeability testing, hours	28	
Geophysical logging, hours	12	Total hrs
Drilling caved hole, hours	43	
Core drilling, hours	345	
Footage cored	676	
Average feet per hour	1.96	
Number of bits	7	
Average feet per bit	97	
Average footage per 11-hour day	21.6	
Maximum footage in 1 day	39	

Core Fracture Analysis

Method

Core drilling was accomplished with a Longyear Model 44 drilling rig. All coring was done with a Longyear triple-tube core barrel (split inner tube). Both 5- and 10-foot core barrels were used.

Data recorded for each core run included hole depth, run length, core recovery, and the length of all rock core pieces 1/2 inch or longer. Measurements of core recovery and core piece length were made while the core was in the split inner tube of the core barrel.

Results

A computer program was used to analyze the drill core data. For each core run the program computes length recovery, recovery and average size for all core pieces greater than 1 inch long and RQD (recovery for core pieces greater than 4 inches long). Figure 2 shows how these quantities are determined. The program also computes averages for the entire hole, averages for 10-foot drill depth intervals and a length percent passing curve.

Figure 2 shows how the drill core measurements are made and gives average results for the total hole. Core recovery and RQD versus depth (based on 10-foot interval averages) are shown in figure 3.

Drill core averages are listed in table 2 for the total hole (1,151-1,827), the ore zone (1,332-1,595), and the bottom of the hole (1,595-1,827). Length percent passing versus fragment length is shown in figure 4 for the total hole. A similar plot for the ore zone and the hole bottom is shown in figure 5.

TABLE 2. - Summary of drill core data

Quantity	Total hole	Ore zone	Hole bottom
Interval, feet	1,151-1,827	1,332-1,595	1,595-1,827
Footage, feet	676	263	232
Core recovery, percent	90	87	98
Recovery >1 inch, percent	73.3	65	90.4
RQD	50	38	72
Fifty percent length passing size, inch	3.8	2.5	6.6

The drill core fragment size analysis used here tends to underestimate the true in situ size that is important for leaching. This is because of the fact that the drilling process creates additional fractures and not all fractures will transmit leach solutions. However, based on fragment size only, it appears that the ore zone interval is sufficiently fractured for leaching, but the hole bottom interval is not adequately fractured.

Permeability Measurements

Method

Two categories of permeability tests are commonly run by the Bureau--constant head and falling head. Constant head tests are conducted by maintaining a constant head (pressure) of water over a fixed test zone in a well and measuring the rate at which water must be added to keep the head constant. The head may be controlled during an open-hole test by simply maintaining the water at a constant level in the casing. With packer tests the pressure is monitored by a pressure gauge and the total head is calculated by adding gauge pressure to the head from the water column in the test pipe (fig. 6). Since the water flow

rate through the packer arrangement in the formation was so slow during the tests at Poston Butte, no correction for friction losses was necessary.

Instrumentation for constant head tests is simple. In open holes the head is measured by lowering a pressure transducer into the well and reading out electrical voltages corresponding to the pressure. If the head is maintained at or near the well collar, the level can be monitored visually. Standard pressure gauges are used to monitor pressure during packer tests. Flow rates are determined by total volume water meters. Drilling rig mud pumps are generally used to inflate packers and pump water during packer tests. For tests at the Poston Butte site, however, the Bean 35 mud pump on the drilling rig was not able to sufficiently inflate the packers for secure seating. A piston pump was rented; rating for this pump was 5 gallons per minute at 2,000 psi. This pump was also used to inject water into the test zone at higher test pressures. For low pressures a 3 horsepower centrifugal pump was used to inject water into the formation. Packers used in the testing were Lynes Slim-Hole Production packers. These are lowered into place on the drill rod and inflated by pressure from the mud pump. When the inflating pressure exceeds 1,500-2,000 psi, a pin shears which opens a port allowing the fluids to be pumped through the packer into the test zone.

Permeability, k, was calculated from the constant head tests using the following equation (5):

$$k = \frac{Q \ln\left(\frac{L}{r}\right)}{2\pi L H}$$

where Q = flow rate pressured in gallons per minute

L = length of open test section, feet

H = hydraulic head, feet

r = radius of the well, feet

By converting gallons to cubic feet and minutes to years, the permeability is expressed as feet per year which is essentially the same as millidarcys.

Falling head tests were conducted upon conclusion of an open hole constant head test. A pressure transducer is placed in the well and the return of the water level to its original level is recorded as a function of time. A commercial transducer-recorder system called the Water Well Analyzer manufactured by Johnson Division of Universal Oil Products was used at the Poston Butte site.

Permeability is calculated from the falling head test with the following equation (6):

$$k = \frac{d^2 \ln\left[\frac{L}{D} + \sqrt{\left(1 + \frac{L}{D}\right)}\right]}{8 L t}$$

where d = inside diameter of casing, feet

D = inside diameter of the well, feet

L = length of open hole test zone, feet

t = time at which the ratio of the head divided by the original head equals 0.37.

The t factor is determined graphically from a plot of log (Head/Original Head) versus time.

Results

The first permeability tests were run on July 12 when the test hole reached 1,518 feet deep. The mud was broken down and flushed from the hole with fresh water for three hole volumes. An open hole constant head test (with a head of 71.8 feet above the original water table) was conducted followed by a falling head test. Both of these tests gave results in close agreement, 12 and 14 millidarcys respectively (see table 3 and fig. 7). Four days later the constant head test was repeated, but the hole had bridged at 1,250 feet due to sloughing. The resultant permeability of the section between the casing at 1,150 and the "bridge" was lower than that measured for the total hole earlier. When it became apparent that the sloughing was severe, permeability tests were temporarily abandoned and drilling resumed using bentonite mud.

At a hole depth of 1,713 feet, drilling was again halted for permeability tests with packers. This time a packer was lowered down the hole on the drilling rod and the hole was flushed through it before seating the packer. The first time the packer was seated at 1,475 feet

providing a test zone from 1,475 to 1,713. This test zone was immediately below a major lost circulation zone which had been encountered between 1,460 and 1,470 feet deep. Two injection pressures were used, 40 psi and 165 psi gauge pressure which corresponded to 140 psi (333 feet) and 270 psi (623 feet) pressure differential, respectively, in the test zone. Flow rates and permeability values are shown in table 3.

TABLE 3. - Constant-head permeability test results

Date	Test interval	Differential pressure, psi	Flow rate, gal/min	Permeability, millidarcy
07/12/79	1,150-1,518	24	2.8	12
07/16/79	1,150-1,250	100	0.63	2
08/01/79	1,475-1,713	140	0.7	0.8
08/01/79	1,475-1,713	270	1.9	1.1
08/01/79	1,300-1,713	140	1.0	0.7
08/01/79	1,300-1,713	240	4.6	1.8
08/01/79	1,300-1,475*	140	0.32	0.45
08/01/79	1,300-1,475*	240	2.92	2.5

*Implied from previous tests on same date.

Next the packers were seated at 1,300 feet so as to provide a test zone encompassing the entire copper-bearing portion of the formation estimated by ASARCO geologists to begin at 1,332 feet and extend down to 1,593 feet based on visual examination. Again using two pressures, the test yielded the permeability values and flow rates shown in table 1. Since the two packer tests were run at the same time, it is possible to substract the flow from the 1,475-1,713 test from that obtained in the 1,300-1,713 test to derive flow in the 1,300-1,475 section of the hole. This then permits calculation of an implied permeability for that section, also shown on table 1.

The calculated permeabilities assume uniform permeability over the entire open test section. Other core below 1,650 feet deep show less fracture than shallower core. The permeability in table 1 may be, therefore, slightly lower than the actual permeability above 1,650 feet and slightly higher than actual below 1,650 feet.

Three broad conclusions suggest themselves from these permeability data:

1. Either the formation is very tight or drilling fluids sealed the permeability channels.

2. There is a pressure effect on permeability. As the pressure increases, cracks are apparently opened increasing the formation permeability.

3. Either the permeability decreased over the life of the well or the interval between 1,250 feet and 1,300 feet is extremely permeable.

The first and third conclusions may be related. It seems fairly certain that the drilling fluid was sealing at least some of the permeability channels. Even after vigorous cleaning the return water from the hole was visually "muddy." This water apparently continually washed enough material from fault gouge zones to prevent it from running clean. As further evidence, the lost circulation zone between 1,460 and 1,470 feet eventually became sealed as drilling progressed beneath it; the good permeability that should have been present during tests spanning that zone never materialized. The permeability values presented, therefore, should be considered as minimums. The critical question is:

How much did the mud seal the permeability? Most uranium leaching operators like to see permeabilities two orders of magnitude greater than were measured at Poston Butte.

Conclusion two, the pressure effect seems real and reasonable. It offers a glimmer of hope for leaching if oil field technology on injecting fluids under pressure can be used to force leaching fluids through the formation at a profitable rate.

Geophysical Logging

A suite of geophysical logs was run in the hole from 1,713 feet up to the casing. The intent of the logging program was to see if any log could define the permeable zones in the formation. In general, no log measures permeability directly. Since several logs do respond to formation characteristics that may be related to permeability, examining a suite of logs has benefits in assessing the overall fluid flow capabilities of the formation. Birdwell, out of Tucson, Ariz., was hired by ASARCO to run a representative suite of logs; namely, temperature, density, caliper, gamma ray, neutron, 3-D acoustic velocity, and electric.

Temperature logs record the temperature of the fluid in the borehole. They are used to detect zones where gas or water enters the borehole causing an abrupt temperature change. In hole 38 the temperatue log showed only the usual linear increase in temperature with depth except for an anomaly between 1,164 and 1,190 feet. A repeat run through the section confirmed our suspicions that the tool had merely stuck at 1,164 and not broken loose until an additional 26 feet of cable had been spooled into the hole.

Caliper logs simply record the hole diameter as measured by three sping loaded arms. The caliper can detect caved zones and zones where drilling fluid entering the formation leaves behind a mud cake. Several zones had caved, but not as badly as we had feared. The caved zones generally correspond to heavily fractured sections. No appreciable mud cake was detected by the caliper. Birdwell recorded the caliper log on the same chart as the density log because the latter is extremely sensitive to borehole irregularities which mask the formation readings.

The gamma ray log records naturally emitted gamma rays. Although these logs have application in locating uranium deposits, their prime use is as a lithological correlation log. In nonuranium deposits most of the natural gamma rays come from potassium, usually contained in feldspars or the erosional derivatives. The gamma ray is the best log of the suite for lithological correlation between wells if future work is contemplated. The gamma ray count remains fairly constant down to 1,163 feet, then shows a steadily increasing count down to 1,250 feet. Whether either of these points represents the conglomerate-granite contact cannot be determined. From 1,250 to 1,590 feet the background count is high, interspersed with even higher peaks. At 1,590 significantly different lithology is encountered, characterized by a more uniform and lower gamma ray count.

For neutron logging a source of high velocity neutrons is contained in logging sonde. The neutrons travel through the formation colliding with atomic nuclei until they lose sufficient velocity to be "captured" by the atoms in the formation. Upon capture a high energy gamma ray is

emitted and this is in turn counted by a detector in the sonde. Since hydrogen atoms are many times more efficient in slowing down the neutrons than any other atom, neutron logs measure the relative amount of hydrogen in the formation. This hydrogen occurs either as water in pore spaces or interstitially in shale. The log thus is used to measure porosity of nonshaley zones. The sonde is arranged so that the count units are inversely realted to porosity--high counts represent low porosity. Neglecting the spikes on the neutron log which correpond to high natural gamma ray spikes, the neutron log for hole 38 shows realtively low count (better porosity) from 1,240-1,312, 1,378-1,400, 1,460-1,470, and 1,570-1,592. The entire interval from 1,240 to 1,625 shows at least moderate neutron counts. Accurate calculations for porosity cannot be made, however, from the neutron log in igneous rocks. The 3-D acoustic velocity logs are a continuous graphical representation of the acoustic wave train traveling through the formation from transmitter to receiver. The Birdwell logs are recorded reversed from exploration seismological records--zero time is on the right rather than left. As the log is scanned from right to left, the first dark band is the initial P wave arrival and, hence, represents the acoustic velocity. Subsequent wave arrivals show up as the alternate white and dark bands. A strong acoustic signal is characterized by sharp broad bands on the log. Fractured formations attenuate the signal and modify the wave shape so that the log in that type of rock shows a jumble of vague bands. A brief examination of the logs seems to show that a zone between 1,205 to 1,240 and one 1,595 to 1,713 exhibit the strongest signals and have the highest

velocity. The rest of the hole seems to be a mixture of alternating weak and strong signals. Since the velocity of the wave is slower when it encounters fractures or pores, acoustic velocity is commonly used to determine porosity. The velocity log correlates very well with the neutron log, which it should, because both measure porosity and are affected adversely by shale.

The principle behind density logging is a gamma ray source of a specific energy controlled so that the gamma rays are scattered as they travel through the formation towards the detector by a mechanism called the Compton effect. Briefly, the higher the electron density in the formation, the greater the scattering under the Compton effect. Since the electron density is proportional to the mass density, the log can be calibrated to read out density directly. The log is extremely sensitive to any borehole irregularities; any readings in caved zones must be ignored. If the matrix density of a formation is known, it is a simple matter to calculate porosity. Unlike the acoustic and neutron logs, the density is relatively unaffected by shale. Disregarding the caved zones, the density log for hole 38 shows several zones between 1,350 and 1,625 that have low density and which correspond to low neutron counts and slow velocity, all of which point towards porosity. Below 1,625 feet, the density log indicates a tight competent formation.

The electric log basically records the formation resistivity. Most matrix material is nonconductive. The log responds with low resistivity to water in the formation and to shaley or clayey zones. If one can

eliminate shale zones, the electric log points out water-filled pore zones. In petroleum applications the electric logs are used to differentiate between water-filled pores and oil-filled (nonconductive) pores. Logs with two differently spaced electrodes are normally run to determine influence of drilling fluid penetration into the formation. Hole 38 electric logs generally correspond to the neutron and acoustic velocity logs. A spontaneous potential log is always run in conjunction with the electric log. This measures the electrochemical voltage potential generated when the formation fluid is of different resistivity than the drilling fluid in the presence of a shale matrix which contains electrically charged particles. Since such conditions were not present at the Poston Butte site, the self-potential logs tell nothing.

To summarize logging results, there seem to be several zones between 1,240 and 1,625 feet that exhibit characteristics of a fractured porous and, hopefully, permeable formation. Below 1,625 feet the rock appears to be too competent for unstimulated leaching.

CONCLUSIONS

1. The permeability values measured are two orders of magnitude lower than those desired by most leaching operators. These values are depressed to an unknown extent, however, by the action of the drilling mud plugging the permeability channels.

2. The geophysical logs point toward a fair amount of porous and fractured rock between 1,240 and 1,625 feet.

3. Based on size analysis of drill core fragments, the ore zone interval (1,332 to 1,595 feet) is sufficiently fractured for in situ leaching. The rock below about 1,650 feet is not adequately fractured.

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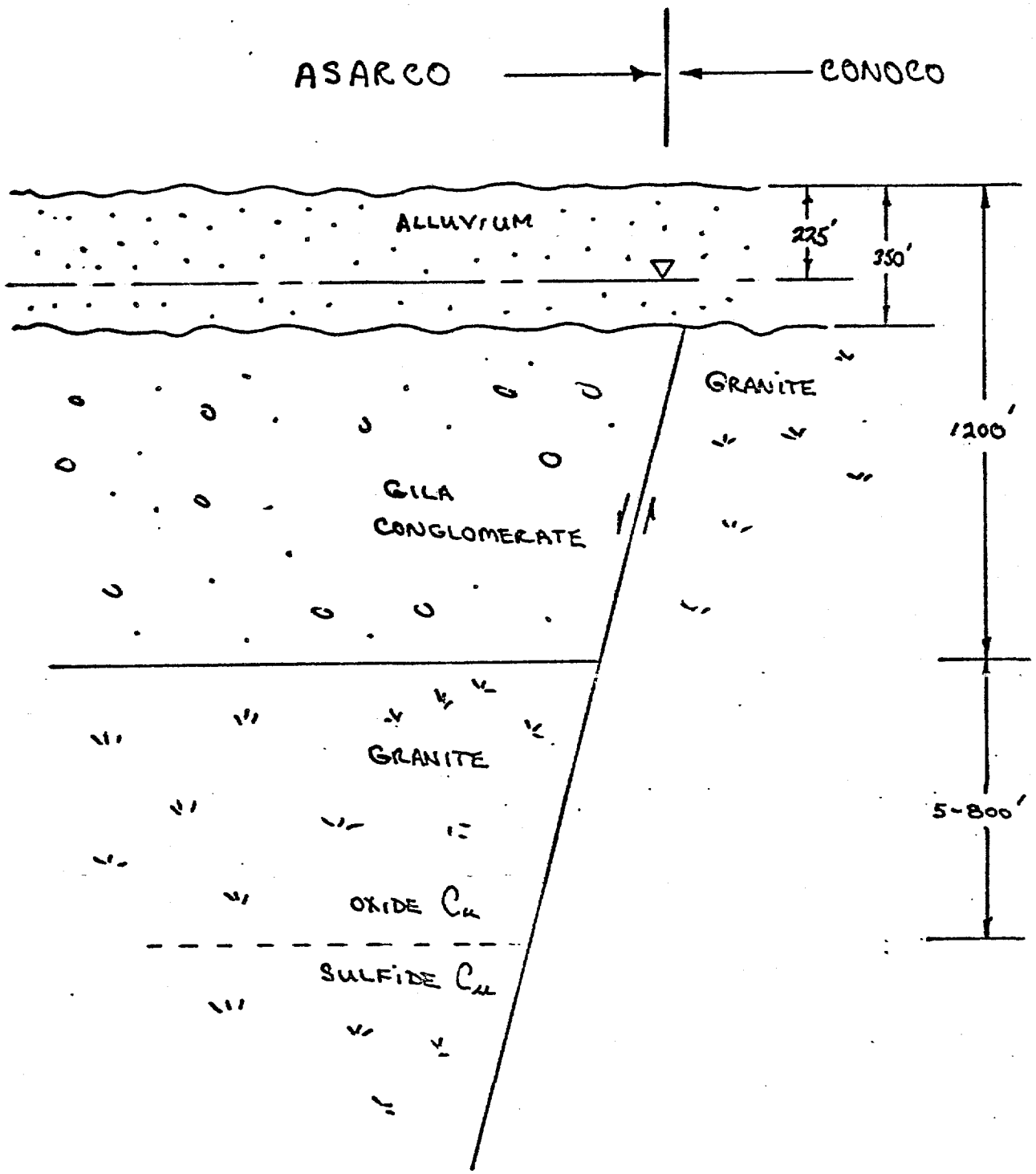


FIGURE 1. GEOLOGIC CROSSECTION (EAST-WEST)

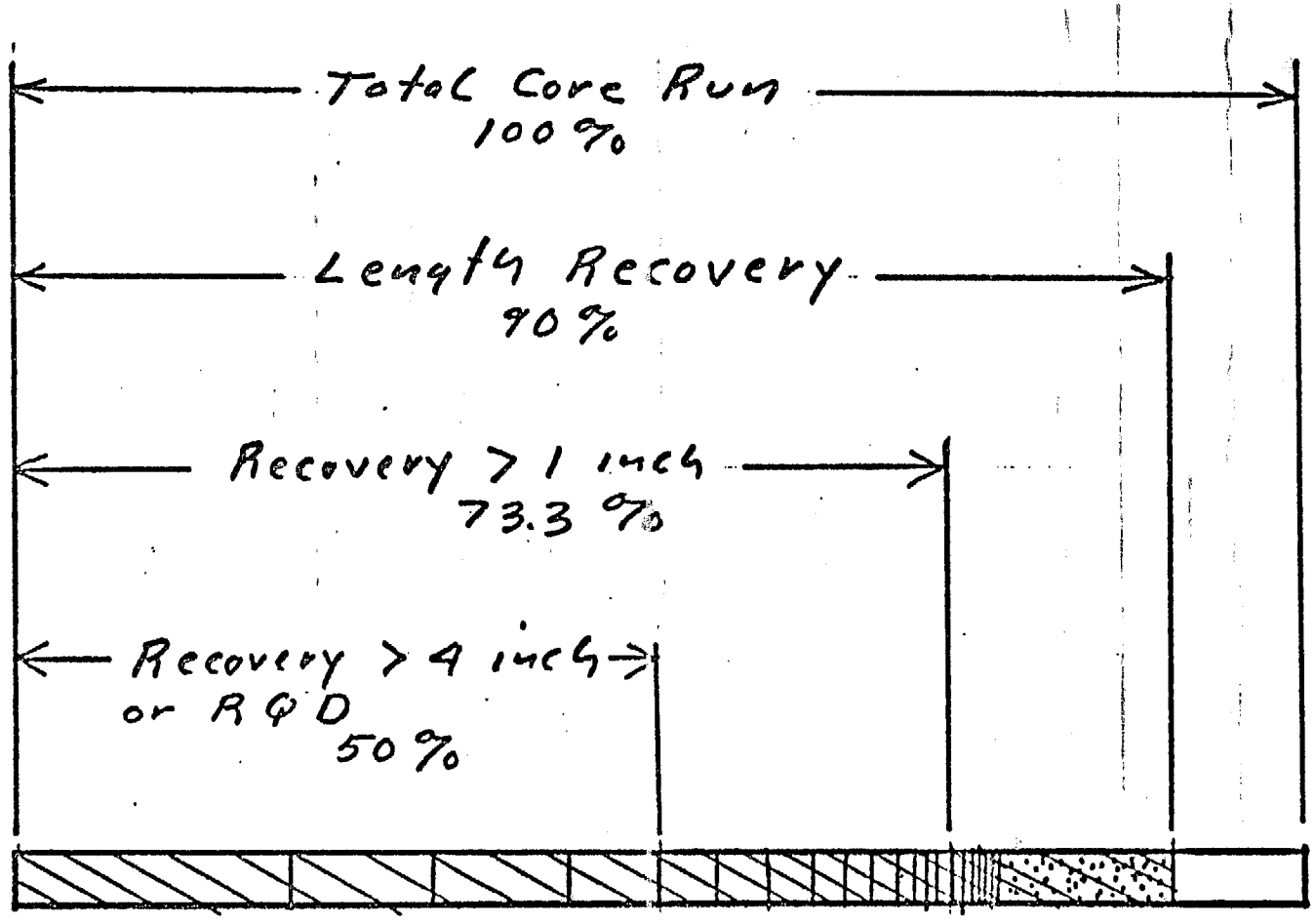


Figure 2 Drill Core Measurements - Total Hole

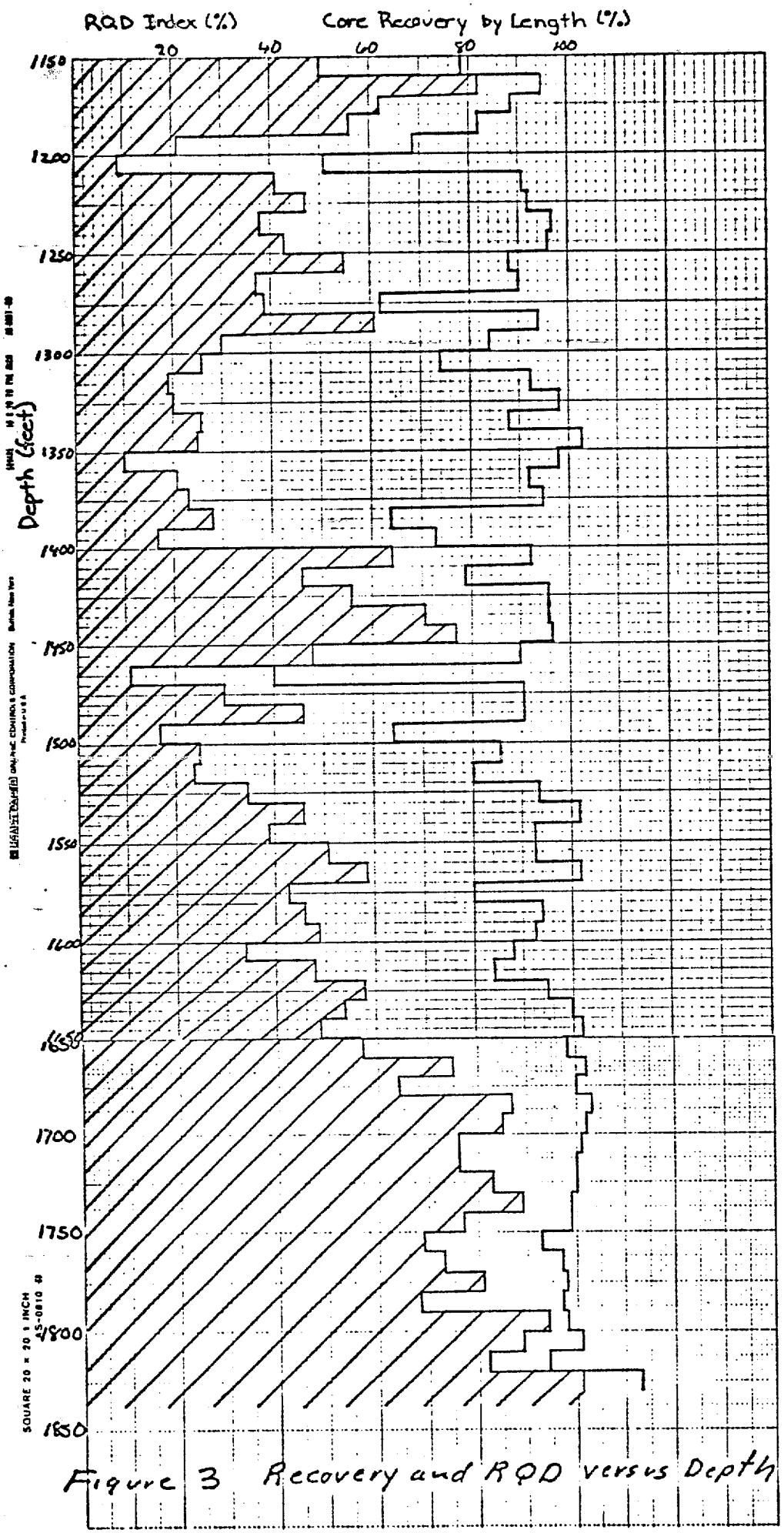


Figure 3 Recovery and RQD versus Depth

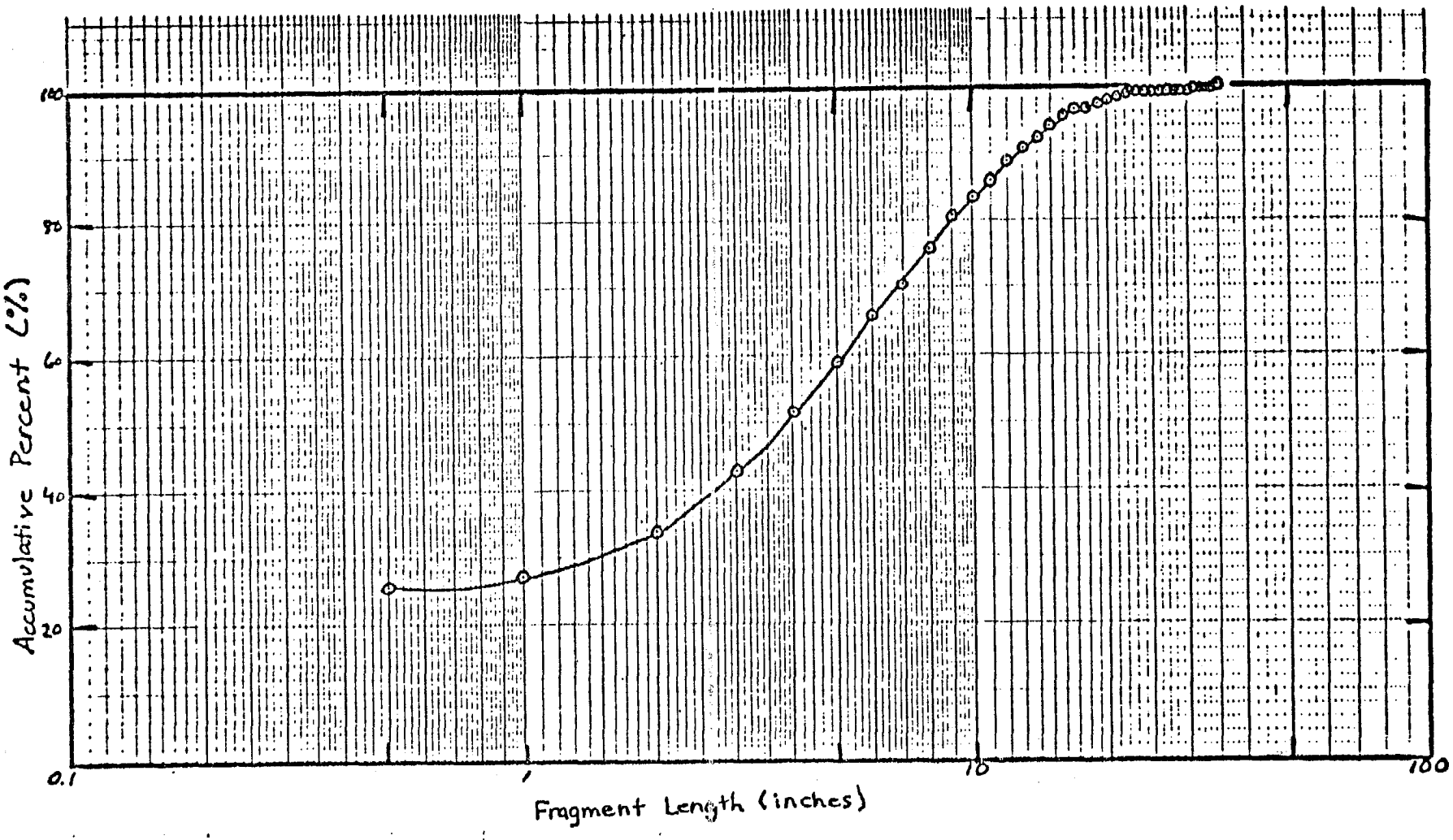


Figure 4 Size distribution - Total Hole

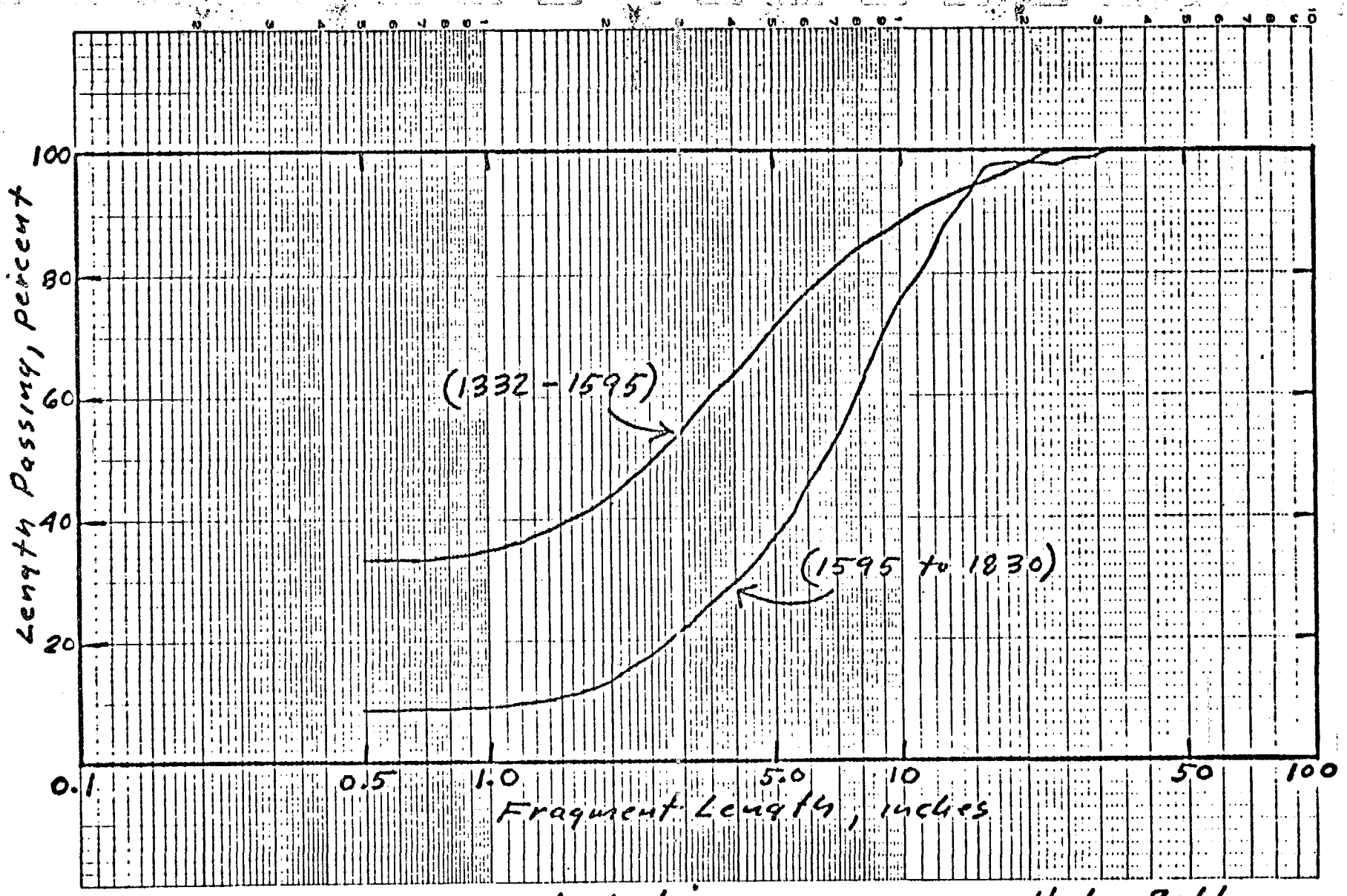


Figure 5 Size distribution - Ore zone + Hole Bottom

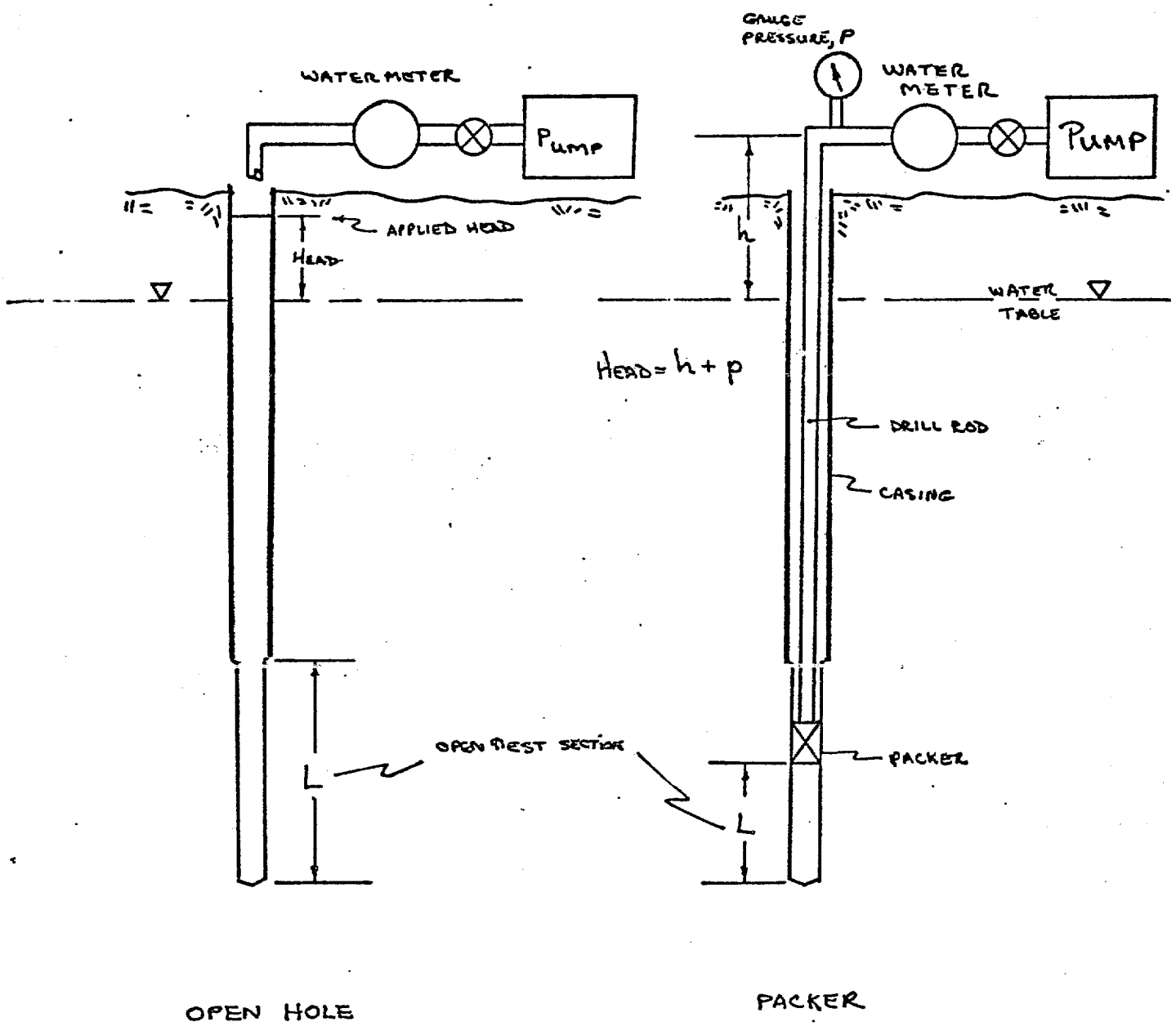


FIGURE 6. CONSTANT HEAD TEST CONFIGURATION

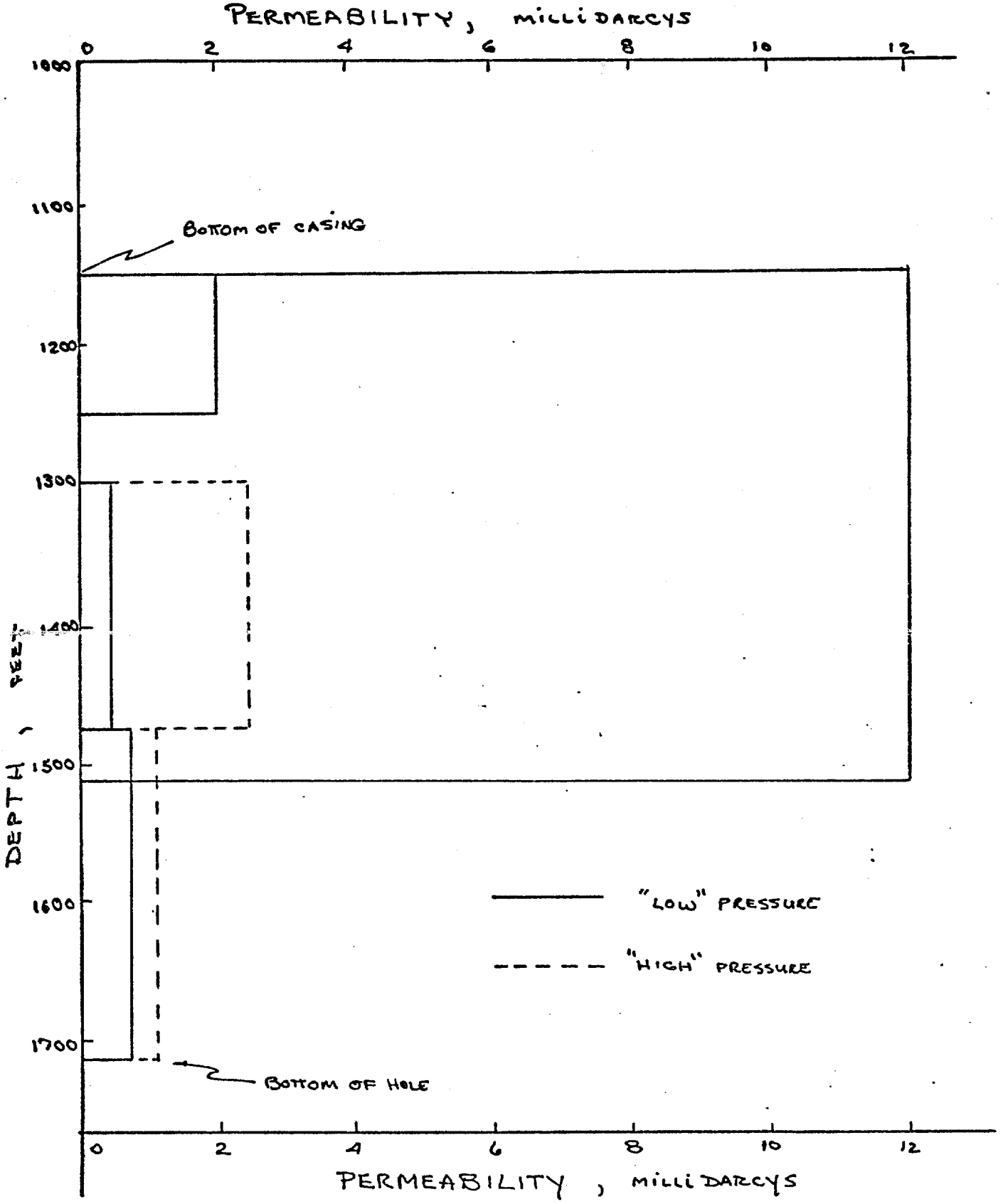


FIGURE 7. PERMEABILITY PROFILE

EQUIPMENT REQUIREMENTS FOR ASARCO COPPER LEACHING INJECTIVITY TEST

POSTON BUTTE PROJECT

9/21/79

prepared for:

Mr. Hank Kreis

ASARCO

P.O. Box 5747

Tuscon, Arizona 85703

by:

Ralph Nelms

Science Applications, Inc.
1546 Cole Blvd., Suite 210
Golden, Colorado 80401

9/22/79

Ralph L. Nelms
Science Applications, Inc
1546 Cole Blvd
Suite 210
Golden, Colorado
80401

Mr. Hank Kreis
Asarco
POB 5747
Tuscon, Arizona
85703

Dear Hank,

Enclosed is a report on the equipment options and costs anticipated for the Poston Butte injectivity tests scheduled for the second week in October. The cost estimate for the first injectivity test is anticipated to be in the $15,000-$18,000 range, depending on what equipment will be selected for the test. Cost for the second injection test should also be in the price range. Selection of equipment for each test will primarily be a function of your budget for the program.

This report is sort of a quick and dirty approach intended to identify what equipment can be used, were it can be obtained, how long will it take to get it, and how much will it cost. The report was put together with the idea of providing as quick a response as possible at a low cost. Some of the parts are hand written but should be easily understandable.

The level of effort put into the report was:

Ralph Nelms	30 hrs at $30/hr	=	$900
Typist	6 hrs at $12/hr	=	$ 72
Copies		=	$ 10
			$982

If you have any questions about the report, please contact me.

Any time I can be of service, don't hesitate to call.

Sincerely yours,

Ralph L. Nelms

Ralph L Nelms

Petroleum/Mining Eng.

Science Applications, Inc. 1546 Cole Blvd., Suite 210, Golden, CO 80401 (303) 232-7900

TABLE OF CONTENTS

	page
REPORT SUMMARY	
A List of equipment alternatives	1
B Total job cost estimates for Poston Butte injectivity tests	3
I Introduction	6
II Outline of proposed plan of action	6
III Equipment requipments for injectivity tests	
A General	7
B Optional	8
IV Service Requirements for injectivity tests	8
V USBM equipment availabe for use at Poston Butte	9
VI Discussion of Recommendations, specific equipment, options , and costs	
1 Drill rig	9
2 Drill rod	10
3 Water tanks	10
4 Water truck	11
5 Pumps	11
6 Wireline to run pressure bomb	12
7 Wireline oil saver	12
8 Downhole pressure bomb	12
9 Lynes slim hole packer	14
10 Flowmeters and pressure gages	14
11 Surface piping	16
12 Subs and crossovers	16
VII Optional equipment	
1 Blow out preventors	16
2 Flange to mate BOP to casing	16
3 Safety joint placed above packer	16
4 Wireline swab	17

page

VIII Service requirements

1 Halliburton acid job 17

2 Wireline service company to perform pressure bomb work 18

3 Caliper log 18

IX Conclusion 19

APPENDIX 20

REPORT SUMMARY

A LIST OF EQUIPMENT ALTERNATIVES

A. Use Core Rig Rods

Do

Pro: No pipe rental expense

Con: Will leak, sealing with teflon may plug packer

Rent a Work String of Tubing

Pro: Will solve leaking problem

Con: High cost $130/day
Have to rent slips and elevators @ $64/day

B. Use Rig Pump

Pro: No pump rental expense

Con: May not set packer, therefore rig time lost would pay for rental pump

Rent Pump

Do

Pro: Will set packer under any circumstance

Con: $250/day rental

C. Use USBM Flow & Pressure Monitoring Equipment - Surface

Pro: No cost

Should be checked or

Con: May be out of calibration

Buy New Equipment

Pro: Will give accurate hard copy output

Con: High cost, $3000 for both flow and pressure readings

Option: May be able to lease pressure device

D. Do Not Use BOP's

Do ?

Pro: No Cost

?

Con: If packer fails, well may kick at surface

Rent BOP's

Pro: Ultimate safety

Con: High cost $64/day rental

LIST OF EQUIPMENT ALTERNATIVES (con't.)

E. Down Hole Pressure Bomb

1.	Reservoir engineering data, Farmington	Lowest Cost
2.	Bill Bean, Farmington	
3.	Sperry Sun, Kuster	
4.	Schlumberger, Dresser	Highest Cost

B Total Job Cost Estimates for Poston Butte Injectivity Tests

First Core Hole Costs: *Optional According to Budget

A. Rig and Equipment Costs

1. Rig Costs -- assuming no hole problems

a.	Prepare rig for test hookup	4 hours
b.	Run caliper log	4 hours
c.	Trip into hole, set packer	2 hours
d.	Try first injection test	2 hours
e.	Call Halliburton - wait	8 hours
f.	Perform Halliburton stimulation	1 hour (acidize)
g.	Resume injection test	6 hours
h.	Pull out of hole	3 hours
i.	Set sand plug and cement	1 hour
j.	WOC (wait on cement?) cells drilled?	4 hours
k.	Trip into hole	2 hours
l.	Record well test	6 hours
m.	Trip out of hole	3 hours
n.	Set second sand & cement plug	1 hour
o.	WOC	4 hours
p.	Trip into hole	2 hours
q.	Perform 3rd well test	6 hours
r.	Trip out	3 hours
s.	RD & move off to second hole	4 hours
		66 hours
	safety factor	24 hours

Optimistic 66 hours x $50 = $3300
24 hours x $50 = $1200
= $4500

Pessimistic 3.75 days

2. Drill String Rental HOMCO-Farmington
1.900 Drill Pipe

$650 first 5 days + $130/day * $ 650.00

3. Subs & Crossovers JOY-Tuscon $ 200.00

4. Elevators & Slips for Drill Pipe HOMCO-Farmington

$180 first 5 days + $31/day * $ 180.00

Blow out preventer

5. BOP's HOMCO-Farmington

$640 first 5 days + $64/day * $ 640.00

6. Water Tanks HOMCO-Farmington 100 BBL

$200 first 5 days + $20/day $ 200.00

7. Pumps HOMCO-Farmington

GD PE 5 $250/day X 3 days
$250/day $ 750.00

8. Rental of Downhole Pressure Bomb Tettteller-Farmington

$175 + $7.50 X 68 hours (72 hrs total) $ 685.00

Includes lubricator and chart reader

9. Lynes Packers - Lynes Brighton, Colorado

1 new packer	$1,900.00
1 new seal (extra)	$ 500.00
balls & sets 10 X $25/set	$ 250.00
	$2,650.00

10. Flow Meters and Gauges

Use USBM Equipment "poor boy" Operation 0

11. Surface Piping, 2000 psi W.P., Check Valves, Elbows, T's, Ball Valves, etc.

$2.60/ft X 100'	$ 260.00
Valves, etc. $90.00 each X 6	$ 480.00
Check Value	$ 150.00
T's L's $25.00 each X 10	$ 250.00
	$ 1,140.00

12.	Safety Joint		
	$55/day X 1 day	$55.00	
	$30 each day X 2	$60.00	
		$115.00	$ 115.00
13.	Rental Equipment Transport		$ 800.00
	Subtotal A		$11,040.00
	* Subtotal A		$12,510.00

B. Service Company Costs

1.	Halliburton 2000 gallon treatment	$ 2,310.00
2.	Caliper Log & Temperature Log	$ 1,640.00
	Subtotal B	$ 3,950.00

C. Materials

Sand:

600' of 3" hole X .05 Ft^3/ft = 3210 # sand

or

32 sacks X $5/sack $ 160.00

Cement:

20 sacks X $5/sack $ 100.00

Subtotal C $ 260.00

D. Supervision - SAI

3 days X $280/day		$ 840.00
Per Diem - 3 days X $50		$ 150.00
Subtotal D	(approx)	$ 1,000.00

TOTAL	$16,250.00
TOTAL	*$17,720.00

Total Excludes water hauling costs

I Introduction

This report discusses the equipment requirements for the upcoming ASARCO Poston Butte copper leaching well test program. The two test holes are located near Florence, Arizona, and coring is to begin the first week in October. Plans now call for an injectivity test on the first core hole if copper oxide ore is found and then the coring and testing of the second core hole. Presently, the two core holes have 4½" casing as surface pipe. Present hole depth is in the 1100' range and coring may proceed to 2000'. Several options exist in exactly how the injectivity test will be run. It is the intent of this report to lay out each option and the associated costs so that a decision can be made on exactly what equipment is to be used.

II Outline of Proposed Plan of Action

1. Hole number one will be cored.
2. If copper oxide is found to a sufficient concentration, proceed with flow injection test program.
3. Displace mud with fresh water in an attempt to remove wall cake. Set Lynes Packer.
4. Attempt injectivity testing on lower 1/3 of wellbore. If flowrate is 10gpm or less at 300 psi, call Halliburton and inject 2000 gallons inhibited 15% HCL solution over entire ore body zone.
5. Conduct 4-hour pump test on lower 1/3 of ore zone. Vary pressures and flowrates 0-300 psi; 5-50gpm.

6. Unset packer. Pull out of hole. Sand up lower 1/3 of wellbore with 20/40 sand and cap with cement. Reset packer and test middle 1/3 of wellbore.

7. Unset packer. Pull out of hole. Sand up middle 1/3 of wellbore. Reset packer and test upper 1/3 of wellbore.

8. Unset packer and clean sand out of wellbore.

9. Core well two.

10. Repeat sequence followed for well one.

III. Equipment Requirements for Injectivity Tests

A. General

1. Drill Rig

   --2000' (+) depth capacity

2. Drill Rod

   --2000' required

   --5000 psi or greater burst capacity

3. Water tanks and mud tanks

   --minimum of 5000 gallons water storage

   --a mud pit should be installed if acid swabbing is planned

4. Water truck

   --1000 gallons minimum

5. Pumps

   --1000-1200 psi required to set packer according to Lynes

   --a second pump may be required to pump 5-50 gpm at 300psi

   -- USBM experience showed that 1500-2000 psi may be required to set Lynes packers

6. Wireline to run pressure bomb

   --2000', 5000# strength

7. Wireline oil saver to contain pressure

8. Downhole pressure bomb

9. Lynes slimhole Packer for 3" hole

10. Flow meters and pressure gauges

11. Surface piping system with manifold, checkvalves and ball valves rated to 2000 psi.

12. Various subs and crossovers for drill string.

B. Optional

1. Blow out preventor (BOP)

   --for use during acid injection or in case seal on packer breaks.

   --2000 psi working pressure

2. Flange to mate BOP to 4½" surface casing

3. Safety joint in drill string above packer

4. Sophisticated pressure and flowrate hard copy monitoring instrumentation.

5. Wireline swab with sinker bar and jars.

IV Service Requirements for Injectivity Test

1. Halliburton

   --to perform one or two 15% Hcl stimulation jobs --2000 gallons each

2. Possibly a wireline company to perform pressure bomb work if a bomb cannot be leased or purchased.

3. Logging

--Caliper log of cored wellbore.

--Temperature log of cored wellbore

V. USBM Equipment Available for Use at Poston Butte

1. Two Lynes packers - both in working order

--one is new - must be inspected to verify size

--the other has been used 10 - 15 times

--no extra elements

--a couple extra ball and sets with brass shear pins

2. Downhole water level indicator

3. 1" and 2" total flow meters

--domestic house type meters

How do she know

--cannot stand pressure - 200 psi maximum

4. Miscellaneous equipment

--3Hp briggs and stratton centrifugal pump --hoses are no good

--various pressure gages - in bad condition How do she know

VI. Discussion of Recommendations, Specific Equipment, Options, and Costs

1. Drilling Rig * Joy 22 Rig will work

The drilling Rig to be used at the Poston Butte coring and test project will be a Joy 22 diamond core rig. This rig should be able to perform the tasks necessary for the injection tests. Joy Equipment Company was contracted in

Tuscon and discussions were held with Mr. Jack Lloyd pertaining to what will be expected from the rig. (See Appendix) This type of rig has been used to set packers.

2. Drill Rod * NXWL Rod will work

The drill rod used will be NXWL. This well meet the test requirements for the injectivity test. Crossover subs will have to be obtained to go from the 3 thread/inch core rod to API threads. Crossover subs from Joy core rod to API threads will have to be made in Tuscon by Joy. Core rod connections will have to be teflon taped to reduce leakage. Even so, leagage of 1-2gpm will occur. Therefore, if accuracy in flow measurements is desired it may become necessary to rent a string of tubing. Homco, Farmington, New Mexico would rent a 2000' string of 1.90 drilling pipe with 1½" MT tool joint threads rated at 10,000 psi burst and 94,000# tension for $650, five day minimum and $130/day after 5 days. If this string were used, elevator and slip rental would be required at $180, 5 day minimum, plus $35/day after 5 days. Teflon tape may plug packer.

Not done by USEM

3. Water Tanks * It will be necessary to rent a 100 BBL water storage tank, and put in a 3000 gal. mud pit if swabbing out of acid is planned.

Joy Equipment, the drilling contractor, will have on site two stock tanks for drilling mud but no water tanks other than a 1000 gallon water truck. In either case, there must be a screen placed between each tank and the

pump to insure no large particles in the water. Scale or sand in the water could interfere with the Lynes packer operation. Therefore, it will be necessary to rent at least a 100 barrel storage tank. Injectivity tests call for 25 - 50 gpm to be pumped for 4 hours. This would require 6000 gallons at 25 gpm or 12,000 gallons at 50 gpm. In either case, the 1000 gallon water truck will have to be making as many trips to Florence for water as possible during water injection.

HOMCO in Farmington, N.M. will rent a 100 BBL tank for $200, 5 day minimum and $20/day. Trucking fees would probably be several hundred dollars. A 210 BBL tank can be rented for $250, 5 day minimum plus $25/day.

4. Water Truck * The 1000 gallon water truck provided by the contractor, Joy Equipment Co., will be required to stay with the rig and make several trips to Florence for water during injection tests.

5. Pumps * The Bean 435 pump on the Joy 22 core rig may not provide sufficient power to set the Lynes downhole packer. It can provide 5 - 40 gpm flow rates, to a limited extent.

This is the major equipment problem for this project. No one has for lease a high pressure low volume pump. Such a pump can be purchased for around $8-10,000 new. The main problem is

that rental companies rent commonly used items and this application is for an uncommonly used pump. Several things must be considered -- the Lynes downhole packer sets by pressuring up the drill pipe until a shear pin breaks. The USBM experience has been that this shear pin will not shear at one set pressure. Thus, 2000 psi may be needed to shear the pin and set the packer. (See Appendix)

How did they do it with Grubbs rig?

The Bean 435 has an absolute maximum pressure of 1200 psi and an operating pressure of 900 psi. Therefore: It will be risky to rely completely on the Bean pump.

Rig time to trip from 2000' will be roughly 4 hours with the core rig. At $50/hour, that's $200. If the Bean pump could not shear the pin and set the packer, and a trip had to be made to refile the shear pin, the cost would be almost equal to the rental of a more powerful pump that would do the job. SAI is proposing to set a packer at least six times to test both holes.

A Gardner Denver PE5 is available for rent out of HOMCO, Farmington, N.M. for $250/day. It may be worth it to have this pump available to set the packer because of the losses in time and therefore money if setting the packer becomes difficult because of shear pin problems (See Appendix)

Cheaper than purchase?

The low flowrate, high pressure pump which would meet our needs of 1-10 gpm at 100-400 psi could be a GASO series 3200 2" duplex or a Kobe Roto-Jet R6-30. We have been unable to find anyone with such a pump to lease today. New costs are $5000 for the Kobe without a stand or 30 hp motor and $9000 for the Gaso pump. In either case, delivery would be a minimum of 2 weeks.

6. Wireline to Run Pressure Bomb * The core rig 3/16" line used to retrieve wireline core barrel can be used. Purpose?

There will some work involved in matching the head of the downhole pressure bomb to the 3/16" wire line available on the Joy 22 core rig.

7. Wireline Oil Saver * This can be obtained through a rental company if the pressure bomb is rented or purchased if a downhole pressure bomb is purchased.

A wireline oil saver or lubricator will be necessary to contain pressures during injectivity testing if a downhole wireline pressure bomb is used. Also, if swabbing is required, it will be necessary to control swabbed fluids. Purchase price for such an item is $700. Guiberson is one source. (See Appendix) See above

8. Downhole Pressure Bomb * The only way to obtain this instrument in the short time available is to rent it. What is it used for?

Reservoir Engineering Data Co., Farmington, N.M., will lease an Amerada 1.25" OD pressure bomb, lubricator and chart reader for $175 for the first 4 hours and $7.50 for each additional hour. They will also double check readings taken with the tool. There are other people who would provide a tool, but obtaining a bomb on short time notice will be difficult. They say they need two weeks notice. Purchase of a bomb would cost $2400, but it takes 8 - 10 months to obtain one. (See Appendix)

9. Lynes Slim Hole Packer * A 2 and 1/8" Lynes slim hole packer is available off the shelf at Casper, Wyo. USBM has two at the site which may be used. USBM packer sizes must be verified. When will this be done?

The Joy NX core barrel will have an OD of 3" + 20/1000". Thus the packer used must be a Lynes 2 and 1/8" slim hole packer. The USBM used a Lynes packer on the Kennacott tests and had trouble with getting the shear pins to shear at lower pressures than 1500 psi. The tool is set by interval pipe pressure and unset by pulling up in some cases or turning to the right on a left and thread and then pulling up. Water used to set the packer must be extremely clean. Sand or pipe scale could plug the pressure release ports on the packer and make release impossible. (See Appendix) Up to SHI to provide clean water

Cost for a new 2 and 1/8" packer is $1900 plus $500 for a new element. Ball and seats sell for $25 each.

10. Flow Meters and Pressure Gages * Equipment is available at USBM trailer to "poor boy" the injection test. But it is recommended that new, more sophisticated pressure and flow measurement devices be purchased. – or rented? *

The USBM trailer has available a domestic home total flow meter gages. When used with a stop watch, flow rates could be measured. But you must consider that six 4-hour tests will be conducted and one man would have to monitor flow rates and pressure for 24 hours with much opportunity for completely overlook-

ing instantaneous events. A better approach would be to instrument flow and pressure and hardcopy the results with an automatic system. Cost for such a system would be a minimum of $1800 for an orifice meter system and $3,000 for an in-line flow turbine meter and in-line pressure sensor both with chart recorders. (See Appendix)

Perhaps justified at this stage.

Options

A. Halliburton -- Flow measurement only --one month delivery

1.	In line flow meter, 5,000 psi, 5-50 gpm	$ 600
2.	Lo II Totalizer, 120 volt	$ 400
3.	Strip chart Recorder	$ 400
	Total	$1400

Flow technology also has a system the same as Halliburton for the same cost -- two week delivery.

B. Honeywell -- pressure measurement only

1.	Pressure Transducer	$ 700
2.	Single Pin Recorder	$ 700
	Total	$1400

C. Foxboro -- Flow and pressure monitoring -- two week delivery.

1.	Orifice plate meter run	$ 400
2.	Orifice plates	$ 200
3.	Flow meter and chart recorder	$1200
	Total	$1800

D. -- A local company in Denver will lease a pressure sensor and chart recorder for $30/day which can be used to measure either subsurface or surface pressures. Flow measurement equipment cannot be leased for hard copy reading.

E. Hycalog also sells a complete system for $2,000.
Delivery time?

11. Surface Piping * All lines used for surface piping should be 2000 psi WP. This equipment is readily available.

12. Subs and Crossovers * Will have to be specially made by Joy at Tuscon. At least two days will be required to make subs.

Will SAI [illegible]? Joy letter in appendix says SAI should provide subs.

VII Optional Equipment

1. Blow out preventors * It is highly recommended that a set of BOP's be rented and used in case of packer failure. HOMCO will rent a set of 6" BOP's for this job at a cost of $600 for the first 5 days, and $60/day after five days. This will include both BOP's and hydraulic controls.

2. Flange to Mate BOP to 4½" casing *Available at a rate of $50/5 days and $4 per day.

3. Safety Joint Placed Above Packer * A safety joint should be run above the packer with bumper jars available if necessary.

A safety joint should be run above the packer under any circumstances. Thus, if the packer sticks, you can release the packer from the drill string. Homco, Farmington, will rent a safety sub, 1 and 13/16" OD for $55 first day and $30 each additional day.

In case the packer did stick, a bumper jar can flown in and used to jar up and down to free the packer. Cost of a bumper sub is $100 for the first day and $60/day each additional day. (See Appendix)

5. Wireline Swab * Wireline swab equipment is easily obtained if needed.

Guiberson sells swab units for 2" pipe or tubing for $80, cups at a cost of $25/set, sinker bars at $60/set and a wireline jar for $160.

VIII Service Requirements

1. Halliburton acid job

If the formation does not take less than 10 gpm at 300 psi, an acid job may be required to break down the bentonite in the formation. Farmington, New Mexico, would be home base for this work. Costs would be as follows:

a. Acid truck for acid transport	No Charge
b. Cost of acid, $.57/gallon	
for 4000 gallons	$2280
for 2000 gallons	$1140

c. Pump truck costs

first four hours	$ 550
each additional hour	$ 92

d. mileage for pump truck

$1.55 per mile / 400 miles	$ 620

Total	for a 4 hour 4000 gal. job	$3450
	for a 4 hour 2000 gal. job	$2310

If two jobs were planned, it would be cheaper to call Halliburton out twice and not have the acid storage truck wait several days at the site.
→ 2-3 weeks between tests.

2. Wireline Service Company to Perform Pressure Bomb Work

If it became impossible to lease a pressure bomb independently, a wireline service would be necessary. Bill Bean out of Farmington, N.M. would provide a wireline unit, a pressure bomb and a man for $200 for the first 4 hours and $35 each additional hour plus $200 mileage.

why this?

3. Caliper Log of Hole

Prior to attempting to set the packer, a caliper and probably a temperature survey should be run. Costs are as follows: (Welex, Dresser-Atlas, Go, etc.)

Service charge	$ 500
mileage ($1.55/ mile/400 miles)	$ 620
caliper log	$ 360
temperature log	$ 160
TOTAL	$1640

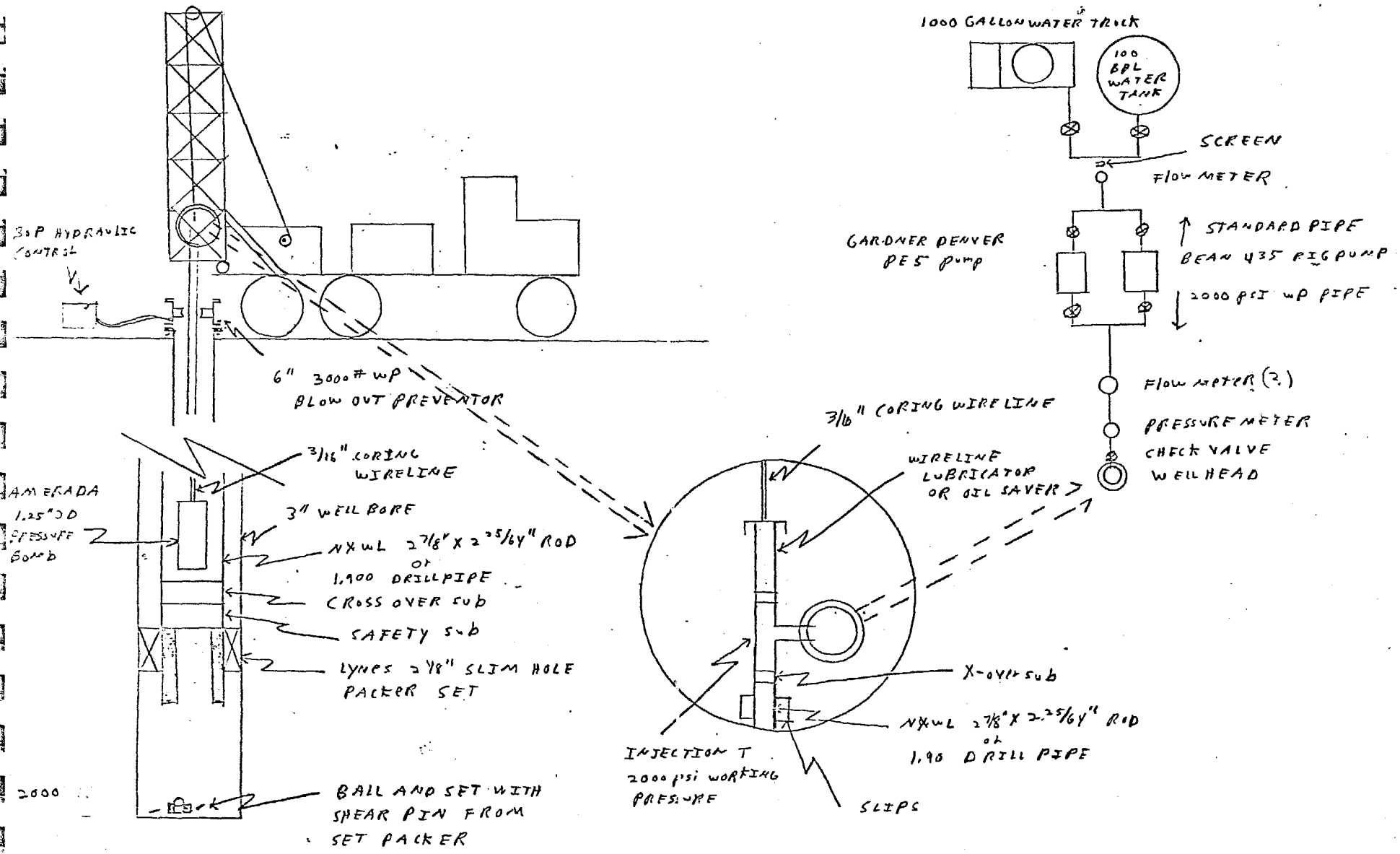
IX CONCLUSION

A sketch of how all the various equipment fits together can be seen on Figure A. Running a downhole pressure bomb is not absolutely necessary. Surface pressures could be monitored for the injectivity tests. One of the key concerns is cleaning the formation face of bentonite which may plug the fractures. One option is to use a surfactant (reduce surface tension?) in the water used to displace the mud downhole. Nalco 1111 is used in the oil industry for cleaning formation faces from oil and scale and should work well in the injectivity tests as well. Another option for cleaning the face is to take a sub for the drill string and weld a plate on the bottom with a 1/2" gap. This is a poor boy jetting device which should also work well.

The most economical operation would have to be a consideration of how the overall test is to be run. If the rental equipment is to be held over during the second coring operation, consideration should be given to obtaining a second rig. But if all the equipment rented is to be returned immediately after the first core hole injectivity test, it would be uneconomic to consider using two rigs. A second rotary rig would go for about $100-125 per hour and would cost $1600 to mobilize out of Farmington, N.M. But trip time would be about half of the core rig trip time due to making hand connections with the core rig.

FIGURE A

EQUIPMENT LAYOUT FOR POSTON BUTTE INJECTIVITY TEST



APPENDIX

LIST OF PERSONS CONTACTED

PHOTOCOPIES OF VARIOUS TYPES OF EQUIPMENT CONSIDERED FOR THIS INJECTIVITY TEST

List of Persons contacted

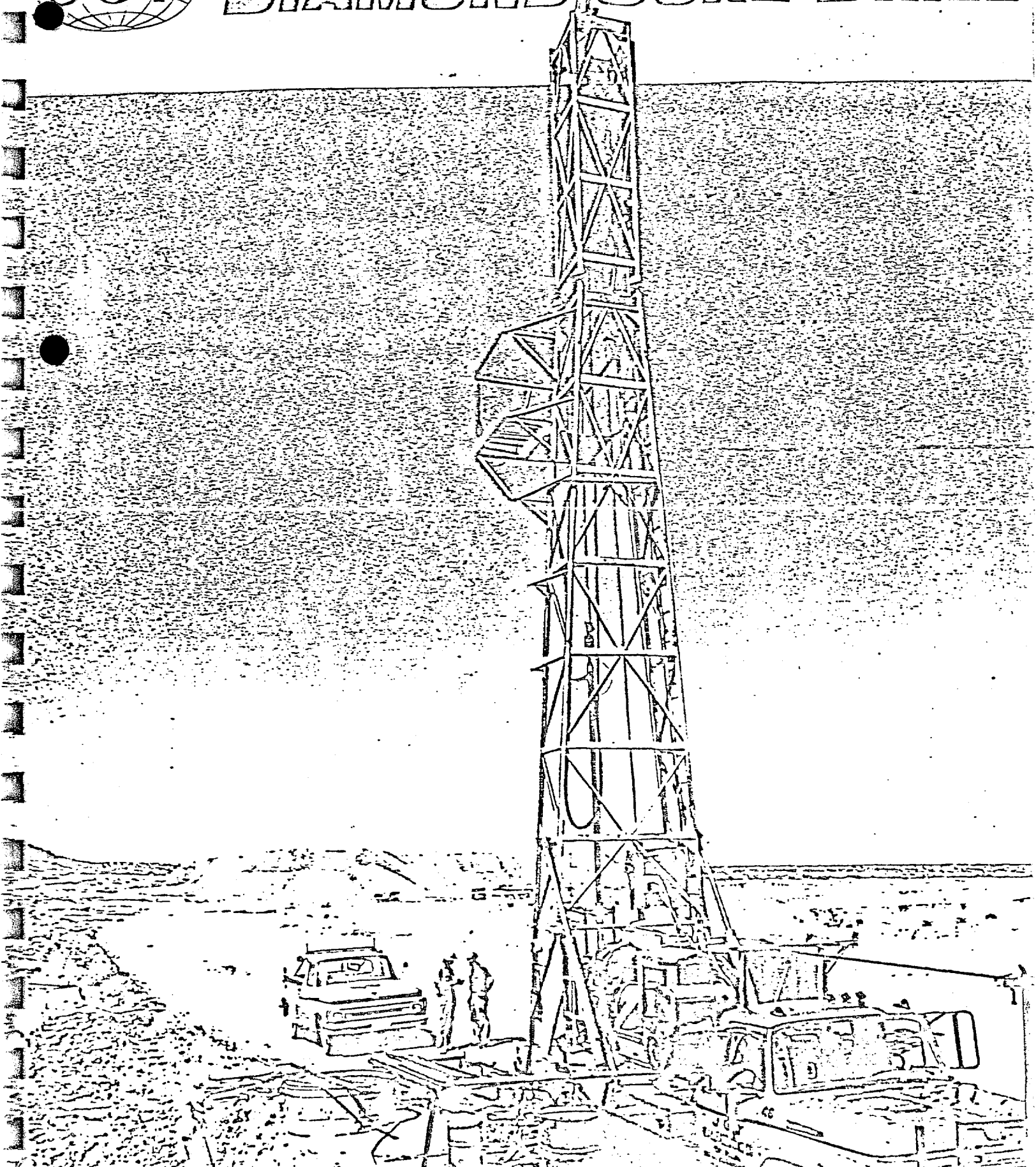
1	Hank Kreis	Asarco Tuscon, Arizona	602 - 792 - 3010
2	Norm Whaley	Asarco Tuscon, Arizona	602 - 792 - 3010
3	Peter Chamberlin	USBM Minneapolis, Minn	612 - 725 - 4547
4	Don Greene	Water Development Corp Tuscon Arizona	602 - 326 - 1133
5	Gene Larson	Larson Construction Tuscon Arizona	602 - 428 - 6940
6	Ray Huff	OXY Minerals, Inc Tuscon Arizona	602 - 473 - 4421
7	Jack Lloyd	Joy Equipment, Co Tuscon Arizona	602 - 884 - 5865
8	Ross Cooper	Joy Equipment, Co Denver, Colo	303 - 371 - 6140
9	Harry Ward	Joy Equipment, Co Laporte, Indiana	219 - 362 - 2191
10	Bill Bean	B&R Wireline Farmington, N.M	505 - 325 - 2393
11	Tom Slocum	Halliburton Farmington, N.M.	505 - 325 - 3575
12	Lyndon Harmon	Lynes Packer Denver, Colo	303 - 573 - 8027
13	Ray Wall	Foster Company Long Beach, Calif	213 - 426 - 9311
14	Bob Mathison	Crude Oil Testing Colorado Springs, Colo	303 - 473 - 6904
15	Loren McClelland	Hamco Farmington N.M.	505 - 327 - 6341
16	Jim Goss	Goss Oilfield Supply Farmington, N.M.	505 - 327 - 7227
17	Bruce Stratton	Guiberson - Presser Casper, Wyo	307 - 237 - 3568
18	Jack Riggs	Guiberson - Presser Brighton, Colo	303 - 659 - 6152
19	Jim Byrd	Kobe Pump Casper, Wyoming	307 - 265 - 6313

List of persons contacted
(continued)

	Sony	Wilson	Wilson Supply Company Farmington N.M.	505-327-2575
1	Bob	Goba	Flow technology Phoenix, Arizona	602-268-8776
2	Otto	Seeman	Foxboro Denver, Colo	303-757-0587
23	Dick	Chase	Honeywell Denver, Colo	303-779-6200
24	Larry	Cerrillo	Consultant Evergreen, Colo	303-674-5193
25	Tody	Testletter	Reservoir Engineering data Farmington, NM	505-325-6817
	Jim	Kootnz	Joy Equipment, Co Laporte, Ind	219-362-2191
	James	Spigner	Farmington well services Farmington, N.M.	505-325-5076
	Frank	Curry	Specialty Equipment Midland, Texas	915-381-0991
	Steve	Groomer	Pumps and service Farmington	505-327-6138
	Dick	Gillmore	Nalco Denver, Colo	707-237-3591

Model 22 SHD

JOY DIAMOND CORE DRILL



SPECIFICATIONS

JOY MODEL 22 SHD DIAMOND CORE DRILL

Mounting	Structural Steel Skid or Truck mounting
Power Unit*	Gasoline engine 4 cylinder water cooled 35 HP Diesel engine 4 cylinder water cooled 33 HP Air motor 5 cylinder 23 HP
Depth Capacity	Single sheave traveling block recommended on depths greater than 1500′ (457M)

English-Metric

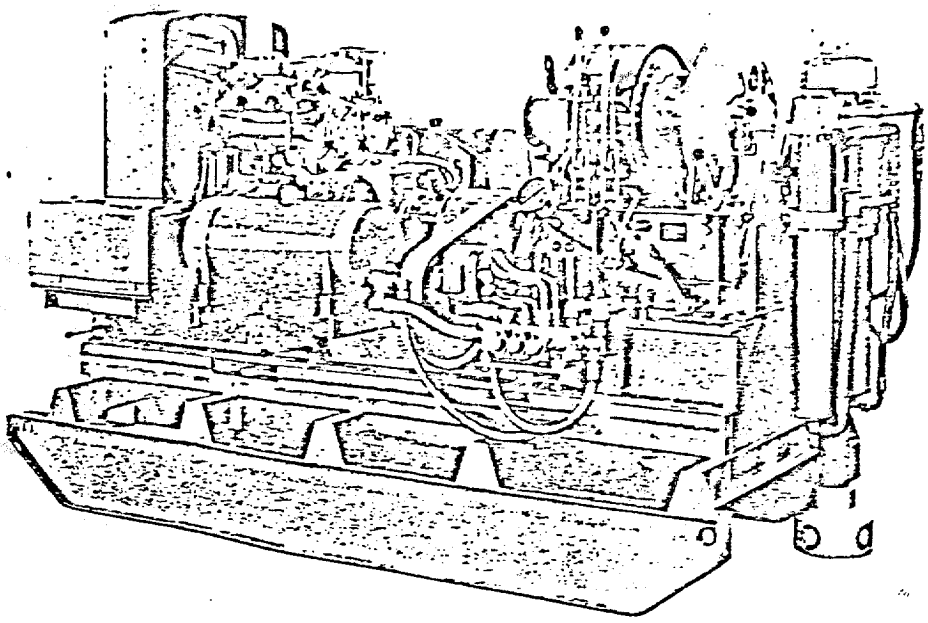
Depth	1800 Ft 550M	4000 Ft 1220M	3100 Ft 945M	2500 Ft 760M	3200 Ft 975M	2500 Ft 760M
Rods	EW	AW	BW	NW	BX Wire line	NX Wire line
Core Dia.	27/32″ 21.4MM	1 3/16″ 30.2MM	1 5/8″ 41.2MM	2 1/8″ 53.0MM	1 7/16″ 36.5MM	2″ 50.8MM
Hole Dia.	1 1/2″ 38.1MM	1 15/16″ 49.2MM	2 3/8″ 60.3MM	3″ 78.2MM	2 23/64″ 59.9MM	3″ 53.1MM

Transmission, Type	Sliding Gear
No. Speed Changes	4-Forward (standard) 4-Forward, 1 Reverse (optional)
Hoist, type	Planetary
Dia. of Drum	12 1/2″ (317.5 MM)
Length of Drum	6 3/4″ (161.4 MM)
Capacity	180′, 5/8″ Rope (54.8 M, 15.8 MM Rope)
Hoisting Speed (Per Min.)	Based on engine speed of 1500 RPM and single line operation
First Gear	50′ 15M
Second Gear	105′ 32M
Third Gear	190′ 58M
Fourth Gear	320′ 98M
Drilling Head	NW Hydraulic
Drive Rod, ID	2 13/16″ (74.6 MM)
Feed Length	24″ Standard (609.6 MM) 30″ Optional (762.0 MM)
No. Cylinders	2
Cylinder ID	3 1/2″ (88.9 MM)
Effective Area, Feed	16.78 Sq. In. (108.2 Sq. CM)
Effective Area, Retract	19.25 Sq. In. (124.1 Sq. CM)
Bit Speeds, RPM	Based on engine speed of 1500 RPM

	Std.	Slow	Med.
First Gear	190	95	142
Second Gear	390	195	295
Third Gear	715	360	535
Fourth Gear	1200	600	900
Optional with Reverse	192	96	144

Angle Range	360°	
General Dimensions	Skid Mtd.	Truck Mtd.
Width	54″ (1372MM)	7′-10″ (2388MM)
Length	140″ (3556MM)	30′-0″ (9144 MM)
Weight	4400 lbs. (1996KG)	16,000 lbs. (7257KG)

*Manufacturer's rating for continuous service of power unit 43 HP — 2400 RPM.



You get high-speed, low cost drilling plus field-proven dependability with Joy 22 SHD core drills.

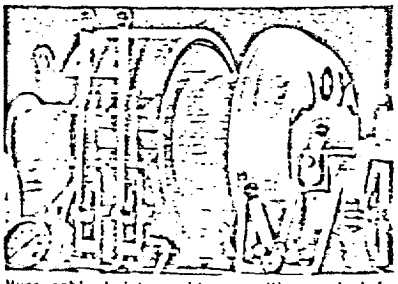
Construction features These Joy 22 SHD construction features mean top performance and long lasting endurance:

Standard:

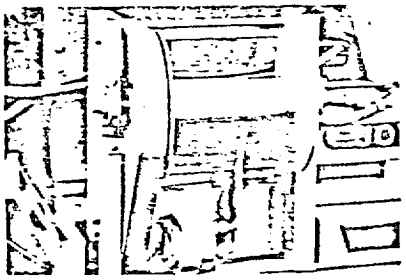
Planetary-type hoisting drum
Four-speed (forward) transmission
Heavy-duty, dry-type clutch
Power unit: 4-cylinder, water-cooled, 35 HP gasoline or 33 HP diesel engine or 5-cylinder, 23 HP air motor
NW hydraulic swivelhead, 24-inch feed with enclosed oil circulating system
Heavy-duty drive shaft to swivelhead
Structural steel skid mounting

Optional:

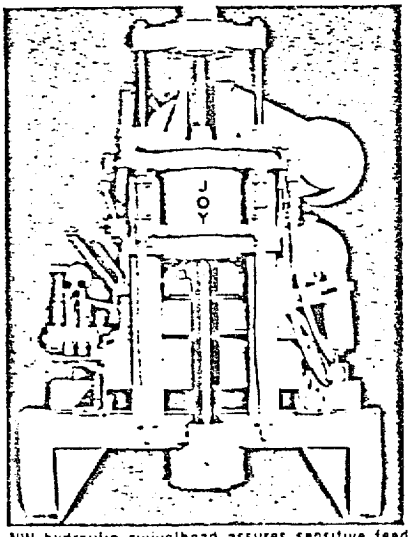
Built-in auxiliary hoist for wire line coring
Built-in mast for vertical or angle holes
Four-speed forward, one reverse transmission
Hydraulic retraction of drill unit
Shaft-mounted cathead or hydraulic motor-driven cathead
Truck mounting
Hydraulically-actuated chuck



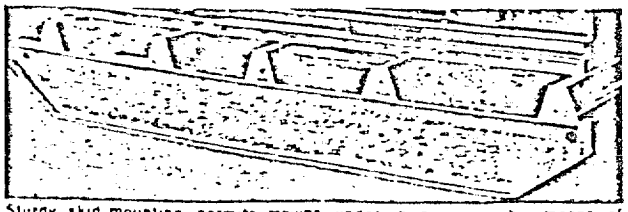
Wire cable hoist provides sensitive control for smooth handling of heavy strings.



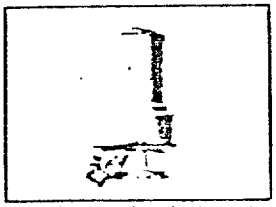
Optional wire line hoist for use with wire line coring tools is driven by a hydraulic motor.



NW hydraulic swivelhead assures sensitive feed rate control; will handle any size rod up through NX wire line or PK rod.



Sturdy skid mounting permits moving under own power by means of planetary-type wire cable hoisting drum.

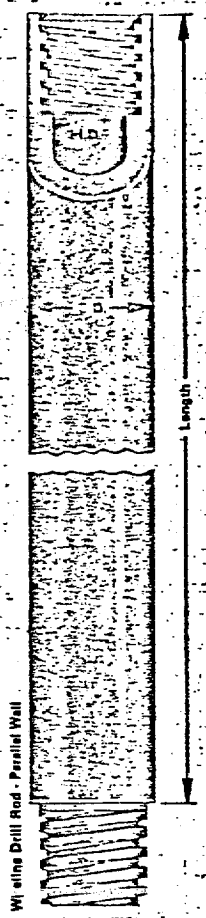


Hydraulic actuated chuck can be supplied as optional in place of standard, manual, four-jaw, safety chuck.

Wireline Drill Rods - Parallel Wall

FLUSH JOINT (MADE FROM STEEL TUBING)

		AXWL	BXWL	NXWL	NCWL
O.D.		1-13/16"	2-1/4"	2-7/8"	3-1/2"
I.D.		1-1/2"	1-29/32"	2-25/64"	3"
NET WT. PER FT. (LBS.) APPROX.		2.761	3.817	6.754	8.678
GAL PER 100 FT.		9.18	14.82	23.30	36.72
THDS. PER IN.		4	4	3	3
PART NUMBER	1 FT.	14-461	14-561	14-671	14-771
	2 FT.	14-462	14-562	14-672	14-772
	3 FT.	14-463	14-563	14-673	14-773
	5 FT.	14-465	14-565	14-675	14-775
	10 FT.	14-467	14-567	14-677	14-777
	20 FT.	14-469	14-569	14-679	14-779



Wireline Drill Rod - Parallel Wall

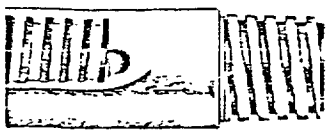
Boyle's Subs

Boyles Bros. carries a wide range of subs. The following listings are arranged for easy reference. The subs are divided into three major groups: box to pin (bushings), pin to pin (couplings), and box to box (adapters). All listed subs have right hand threads unless otherwise noted. Left hand threads are indicated with "LH". Subs are used to adapt or connect different sizes of rods, casing, rock bits, etc. A box thread is sometimes called a female end when the thread is internal and the pin thread a male when the thread is external.

Instructions: After determining the sub required, select the appropriate bushing, coupling or adapter. Box and pin ends are listed beginning with the old "X" series, followed by the "W" series, wireline rods, regular casing, drive casing and the API group.

Bushings

(Box to Pin)



BOX	PIN	PART NUMBER	NET WT. (LBS.) APPROX.
E	A	15-312	2.2
E	EW	15-310	1.3
E	AW	15-314	2.5
E	BW	15-317	4.0
E	AXWL 4 Thd	15-321	2.3
E	BXWL 4 Thd	15-322	4.6
E	EX Reg. Casing	15-321	2.3
E	AX Reg. Casing	15-322	4.6
E	EX Drive Casing	15-321	2.3
E	AX Drive Casing	15-322	4.6
A	E	15-409	1.3
A	N	15-415	4.2
A	EW	15-410	2.0
A	AW	15-411	2.3
A	BW	15-413	4.2
A	NW	15-416	6.5
A	AXWL 4 Thd	15-418	1.5
A	BXWL 4 Thd	15-419	4.5
A	EX Reg. Casing	15-418	1.5
A	AX Reg. Casing	15-419	4.5
A	BX Reg. Casing	15-405	6.7
A	EX Drive Casing	15-418	1.5
A	AX Drive Casing	15-419	4.5
N	E	15-607	3.2
N	AW	15-610	3.8
N	BW	15-612	4.8
N	NW	15-613	5.7
N	NXWL 3 Thd	15-615	7.7
N	NOWL	15-954	5.4
N	BX Reg. Casing	15-601	5.3

Bushings

BOX	PIN	PART NUMBER	NET WT. (LBS.) APPROX.
N	NX Reg. Casing	15-602	10.6
N	3-1/2" Reg. Casing	15-603	15.0
N	BX Drive Casing	15-616	7.7
N	NX Drive Casing	15-617	11.0
N	2 3/8 API IF	15-620	10.5
EW	E	15-346	1.2
EW	A	15-348	2.1
EW	N	15-352	4.5
EW	AW	15-349	2.4
EW	BW	15-351	3.8
EW	NW	15-353	6.5
EW	AXWL 4 Thd	15-355	2.5
EW	BXWL 4 Thd	15-356	4.3
EW	EX Reg. Casing	15-355	2.5
EW	AX Reg. Casing	15-356	4.3
EW	EX Drive Casing	15-355	2.5
EW	AX Drive Casing	15-356	4.3
AW	E	15-455	2.0
AW	A	15-457	2.2
AW	N	15-460	4.5
AW	EW	15-456	1.8
AW	BW	15-459	3.7
AW	NW	15-461	6.5
AW	AXWL 4 Thd	15-463	1.6
AW	AOWL	15-908	1.5
AW	BXWL 4 Thd	15-464	3.9
AW	NXWL 3 Thd	15-468	7.4
AW	NXWL 4 Thd	15-465	7.4
AW	NCWL 3 Thd	15-469	11.2
AW	EX Reg. Casing	15-463	1.6
AW	AX Reg. Casing	15-464	3.9

Bushings

BOX	PIN	PART NUMBER	NET WT. (LBS.) APPROX.
AW	BX Reg. Casing	15-453	8.2
AW	EX Drive Casing	15-463	1.6
AW	AX Drive Casing	15-464	3.9
AW	BX Drive Casing	15-465	7.4
BW	E	15-545	2.8
BW	A	15-547	3.5
BW	N	15-550	5.2
BW	AW	15-548	3.2
BW	NW	15-551	6.3
BW	BXWL 4 Thd	15-552	2.8
BW	NXWL 3 Thd	15-555	7.0
BW	NXWL 4 Thd	15-553	7.0
BW	NCWL 3 Thd	15-598	11.0
BW	NCWL 4 Thd	15-554	11.0
BW	AX Reg. Casing	15-552	2.8
BW	BX Reg. Casing	15-541	6.3
BW	NX Reg. Casing	15-542	11.0
BW	3-1/2" Reg. Casing	15-543	18.7
BW	4" Reg. Casing	15-544	20.2
BW	AX Drive Casing	15-552	2.8
BW	BX Drive Casing	15-553	7.0
BW	NX Drive Casing	15-554	11.0
BW	2-3/8 API Reg.	15-550	8.8
BW	2-3/8 API IF	15-564	11.0
BW	2-7/8 API Reg.	15-560	11.0
NW	A	15-642	4.5
NW	N	15-646	5.5
NW	EW	15-634	4.6
NW	AW	15-643	4.5
NW	BW	15-645	4.9
NW	NXWL 3 Thd	15-695	5.0
NW	NCWL 3 Thd	15-659	9.4
NW	NCWL 4 Thd	15-648	9.4
NW	NQWL	15-970	4.5
NW	BX Reg. Casing	15-635	4.4
NW	NX Reg. Casing	15-636	9.0
NW	3-1/2" Reg. Casing	15-637	14.3
NW	3-3/4" Reg. Casing	15-641	13.5
NW	4" Reg. Casing	15-638	18.3
NW	BX Drive Casing	15-647	5.0
NW	NX Drive Casing	15-648	9.4
NW	2-3/8 API Reg.	15-652	9.3
NW	2-3/8 API IF	15-650	11.4
NW	2-3/8 Mayhew	15-653	10.3
NW	2-7/8 API IF	15-651	17.0
NW	4-1/2 API Reg.	15-654	35.0
AXWL 4 Thd	E	15-490	1.7
AXWL 4 Thd	A	15-492	1.8
AXWL 4 Thd	EW	15-491	2.2
AXWL 4 Thd	AW	15-493	2.4

Bushings

BOX	PIN	PART NUMBER	NET WT. (LBS.) APPROX.
AOWL	N	15-904	5.6
BXWL 4 Thd	E	15-580	2.3
BXWL 4 Thd	A	15-582	2.8
BXWL 4 Thd	N	15-586	3.4
BXWL 4 Thd	EW	15-581	2.5
BXWL 4 Thd	AW	15-583	3.1
BXWL 4 Thd	BW	15-585	2.8
BXWL 4 Thd	NW	15-587	4.8
BXWL 4 Thd	NXWL 4 Thd	15-588	6.7
BXWL 4 Thd	BX Drive Casing	15-588	6.7
BXWL 4 Thd	2-1/2" Pipe	15-599	5.5
NXWL 3 Thd	BW	15-689	5.4
NXWL 3 Thd	NW	15-690	6.7
NXWL 3 Thd	NXWL 4 Thd	15-678	3.4
NXWL 3 Thd	BX Reg. Casing	15-682	3.5
NXWL 3 Thd	BX Drive Casing	15-678	3.4
NXWL 3 Thd	NQWL	15-967	2.6
NXWL 4 Thd	A	15-671	4.1
NXWL 4 Thd	AW	15-672	3.9
NXWL 4 Thd	NXWL 3 Thd	15-679	3.4
NXWL 4 Thd	BX Reg. Casing	15-691	3.5
NXWL 4 Thd	2-1/2" Pipe	15-686	3.4
NQWL	NW	15-960	5.3
NQWL	NXWL 3 Thd	15-965	3.2
NQWL	BX Drive Casing	15-955	3.3
NCWL 3 Thd	BW	15-702	6.5
NCWL 3 Thd	NW	15-707	9.4
NCWL 3 Thd	NCWL 4 Thd	15-709	4.3
NCWL 3 Thd	NX Drive Casing	15-709	4.3
NCWL 4 Thd	BW	15-704	7.0
NCWL 4 Thd	N	15-705	7.1
NCWL 4 Thd	NW	15-706	7.8
NCWL 4 Thd	NCWL 3 Thd	15-708	4.3
NCWL 4 Thd	NX Reg. Casing	15-717	4.5
HQWL	NCWL 3 Thd	15-975	4.4
EX Reg. Casing	E	15-490	1.7
EX Reg. Casing	A	15-492	1.8
EX Reg. Casing	EW	15-491	2.2
EX Reg. Casing	AW	15-493	2.4
AX Reg. Casing	E	15-580	2.3
AX Reg. Casing	A	15-582	2.9
AX Reg. Casing	N	15-586	3.4
AX Reg. Casing	EW	15-581	2.5
AX Reg. Casing	AW	15-583	3.1
AX Reg. Casing	BW	15-585	2.8
AX Reg. Casing	NW	15-587	4.8
AX Reg. Casing	BX Drive Casing	15-588	6.7
AX Reg. Casing	2-1/2" Pipe	15-599	5.5

Boyles Bros. DRILLING COMPANY

Bushings

BOX	PIN	PART NUMBER	NET WT. (LBS.) APPROX.
BX Reg. Casing	NXWL 3 Thd	15-576	2.2
BX Reg. Casing	BX Drive Casing	15-574	3.3
NX Reg. Casing	NCWL 3 Thd	15-665	4.5
NX Reg. Casing	NX Drive Casing	15-667	4.5
EX Drive Casing	E	15-490	1.7
EX Drive Casing	A	15-492	1.8
EX Drive Casing	EW	15-491	2.2
EX Drive Casing	AW	15-493	2.3
AX Drive Casing	E	15-580	2.3
AX Drive Casing	A	15-582	2.9
AX Drive Casing	N	15-586	3.4
AX Drive Casing	EW	15-581	2.5
AX Drive Casing	AW	15-583	3.1
AX Drive Casing	BW	15-585	2.8
AX Drive Casing	NW	15-587	4.8
AX Drive Casing	BX Drive Casing	15-588	6.7
AX Drive Casing	2-1/2" Pipe	15-599	5.5
BX Drive Casing	A	15-671	4.1
BX Drive Casing	AW	15-672	3.9
BX Drive Casing	BW	15-674	5.2
BX Drive Casing	NW	15-677	5.4
BX Drive Casing	BXWL 4 Thd	15-675	3.8
BX Drive Casing	NXWL 3 Thd	15-679	3.4
BX Drive Casing	AX Reg. Casing	15-675	3.8
BX Drive Casing	BX Reg. Casing	15-691	3.5
BX Drive Casing	AX Drive Casing	15-675	3.8
BX Drive Casing	2-1/2" Pipe	15-686	3.4
NX Drive Casing	BW	15-704	7.0
NX Drive Casing	N	15-705	7.1
NX Drive Casing	NW	15-706	7.8
NX Drive Casing	NCWL 3 Thd	15-708	4.3
NX Drive Casing	NX Reg. Casing	15-717	4.5
NX Drive Casing	4" Pipe	15-719	19.6
2-3/8 API IF	NW	15-825	10.2
2-3/8 API IF	NXWL 3 Thd	15-827	8.8
2-3/8 API IF	NCWL 3 Thd	15-830	11.0
2-3/8 API IF	2-3/8 Fail. Explor.	15-842	10.3
2-3/8 API IF	2-7/8 API IF	15-838	18.8
2-3/8 Fail. Explor.	2-3/8 API IF	15-880	9.3
2-3/8 Fail. Explor.	2-7/8 API IF	15-881	21.3
2-7/8 API IF	2-3/8 API IF	15-856	14.9
2-7/8 API IF	2-7/8 API Reg.	15-864	19.0
2-7/8 API IF	3-1/2 API IF	15-867	34.8
3-1/2 API IF	2-7/8 API IF	15-871	17.5

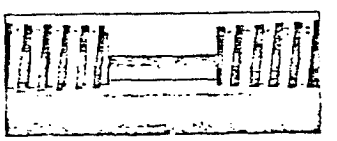
Couplings
(Pin to Pin)

PIN	PIN	PART NUMBER	NET WT. (LBS.) APPROX.
E	E	15-335	1.0
E	E (brass)	15-324	1.3
E	A	15-326	1.7
E	N	15-330	4.7
E	EW	15-325	1.5
E	AW	15-327	1.8
E	NW	15-331	5.5
A	A	15-435	1.6
A	A (alum.)	15-434	0.6
A	N	15-426	4.2
A	EW	15-358	1.5
A	AW	15-423	2.0
A	BW	15-425	2.8
A	NW	15-427	6.4
N	EW	15-361	4.4
N	AW	15-472	4.4
N	BW	15-556	4.4
N	NW	15-618	6.3
EW	EW	15-365	1.1
EW	AW	15-359	1.9
EW	BW	15-362	3.4
AW	AW	15-475	1.8
AW	BW	15-471	2.2
AW	NW	15-473	6.1
AW	NXWL 3 Thd	15-470	5.3
BW	BW	15-558	3.7
BW	NW	15-557	6.4
NW	NW	15-649	5.4
NW	BXWL 4 Thd	15-589	7.3
NW	NXWL 3 Thd	15-639	7.5
NW	NXWL 4 Thd	15-680	7.5
NW	NCWL 3 Thd	15-640	11.5
NW	NCWL 4 Thd	15-710	11.5
NW	AX Reg. Casing	15-589	7.3
NW	BX Reg. Casing	15-569	7.8
NW	NX Reg. Casing	15-692	13.0
NW	AX Drive Casing	15-589	7.3
NW	BX Drive Casing	15-680	7.5
NW	NX Drive Casing	15-710	11.5
NW	2-3/8 API IF	15-693	11.8
AXWL 4 Thd	1-1/2" Pipe	15-496	1.2
AXWL 4 Thd	2" Pipe	15-497	3.5
AXWL 4 Thd	2-1/2" Pipe	15-498	6.6
BXWL 4 Thd	1-1/2" Pipe	15-592	3.1
BXWL 4 Thd	2" Pipe	15-593	1.7
BXWL 4 Thd	2-1/2" Pipe	15-594	5.4
BXWL 4 Thd	3" Pipe	15-595	8.7
NXWL 4 Thd	1-1/2" Pipe	15-685	6.8
NXWL 4 Thd	3" Pipe	15-687	6.0

Couplings

PIN	PIN	PART NUMBER	NET WT. (LBS.) APPROX.
NCWL 3 Thd	1-1/2" Pipe	15-713	10.2
NCWL 4 Thd	3" Pipe (Heavy duty)	15-715	7.0
NCWL 4 Thd	4" Pipe	15-719	19.6
EX Reg. Casing	1-1/2" Pipe	15-496	1.2
EX Reg. Casing	2" Pipe	15-497	3.5
EX Reg. Casing	2-1/2" Pipe	15-498	6.6
AX Reg. Casing	1-1/2" Pipe	15-592	3.1
AX Reg. Casing	2" Pipe	15-593	1.7
AX Reg. Casing	2-1/2" Pipe	15-594	5.4
AX Reg. Casing	3" Pipe	15-595	8.7
BX Reg. Casing	2-1/2" Pipe	15-570	2.3
BX Reg. Casing	3" Pipe	15-571	7.5
NX Reg. Casing	3" Pipe	15-664	3.6
3-1/2" Reg. Casing	3-1/2" Pipe	15-891	4.2
4" Reg. Casing	4" Pipe	15-893	6.1
EX Drive Casing	1-1/2" Pipe	15-496	1.2
EX Drive Casing	2" Pipe	15-497	3.5
EX Drive Casing	2-1/2" Pipe	15-498	6.6
AX Drive Casing	1-1/2" Pipe	15-592	3.1
AX Drive Casing	2" Pipe	15-593	1.7
AX Drive Casing	2-1/2" Pipe	15-594	5.4
AX Drive Casing	3" Pipe	15-595	8.7
BX Drive Casing	1-1/2" Pipe	15-685	6.0
BX Drive Casing	3" Pipe	15-687	6.0
NX Drive Casing	3" Pipe (Heavy duty)	15-715	7.0
2-3/8 API IF	2-3/8 API IF	15-843	9.5
2-7/8 API IF	2-7/8 API IF	15-859	18.8
2-7/8 Fail. Explor.	2-7/8 API IF	15-877	12.0
Fail. Red Devil	2-3/8 API IF	15-885	12.0
Fail. Red Devil	2-7/8 API IF	15-886	18.5

Adapters



(Box to Box)

BOX	BOX	PART NUMBER	NET WT. (LBS.) APPROX.
E	EX Bit	15-336	0.8
A	A	15-437	1.6
A	BW	15-438	3.5
A	1/2" LH Pipe	15-444	1.6

Adapters

BOX	BOX	PART NUMBER	NET WT. (LBS.) APPROX.
A	1" LH Pipe	15-445	1.4
A	1-1/4" LH Pipe	15-447	2.0
N	N	15-624	4.3
N	BW	15-563	4.4
N	NW	15-625	5.7
N	BXWL 4 Thd	15-626	3.5
N	AX Reg. Casing	15-626	3.5
N	AX Drive Casing	15-626	3.5
N	2-3/8 API Reg.	15-630	11.0
N	3-1/2 API Reg.	15-632	23.5
N	4-1/2 API Reg.	15-633	35.4
EW	EX Bit	15-366	0.8
AW	AX Plug Bit	15-476	1.4
AW	2-3/8 API Reg.	15-478	10.3
BW	1-1/4" LH Pipe	15-540	2.3
BW	2-3/8 API Reg.	15-565	10.2
BW	2-7/8 API Reg.	15-566	15.2
NW	2-3/8 API Reg.	15-655	10.2
NW	2-7/8 API Reg.	15-656	17.0
NW	2-7/8 API IF	15-697	20.4
NW	3-1/2 API Reg.	15-657	19.0
NW	4-1/2 API Reg.	15-658	35.0
BXWL 4 Thd	2-3/8 API Reg.	15-591	9.0
NXWL 3 Thd	N	15-688	9.4
NXWL 3 Thd	NX Bevel Wall Bit	15-676	3.4
NCWL 3 Thd	NC 8 Thd Bit	15-711	3.3
AX Reg. Casing	2-3/8 API Reg.	15-591	9.0
4" Reg. Casing	4" Bevel Wall Bit	15-894	3.2
EX Drive Casing	AX Reamer Shell	15-370	1.2
AX Drive Casing	BX Bevel Wall Bit	15-480	2.0
AX Drive Casing	2-3/8 API Reg.	15-591	9.0
BX Drive Casing	NX Reamer Shell	15-567	3.9
NX Drive Casing	NC 8 Thd Bit	15-660	3.4
2-3/8" API Reg.	2-3/8" API IF	15-845	11.0
2-3/8" API Reg.	2-3/8" Mayhew	15-810	11.2
2-3/8 API IF	2-7/8 API Reg.	15-846	16.0
2-3/8 API IF	3-1/2 API Reg.	15-848	22.0
2-7/8 API IF	2-7/8 API Reg.	15-860	19.2
2-7/8 API IF	3-1/2 API Reg.	15-868	18.8
2-7/8 API IF	4-1/2 API Reg.	15-869	39.0

Boyles Bros. DRILLING COMPANY

Boyles' Diesel Driven Pump

Drilling efficiency is enhanced with the multi-volume capability of this rugged four-speed transmission pump. A diesel engine powers this unit for efficient operation in remote locations under a variety of conditions. Selection of positive pumping volumes is easily made by shifting the transmission to deliver optimum flow as drilling progresses.

The proven FMC triplex pump will deliver maximum performance with only high quality materials used in its manufacture. This fine pump features totally enclosed working parts, constant three-way oil bath lubrication, ceramic cylinders, hardened stainless steel valves and seats; many features pioneered by FMC for long service life.

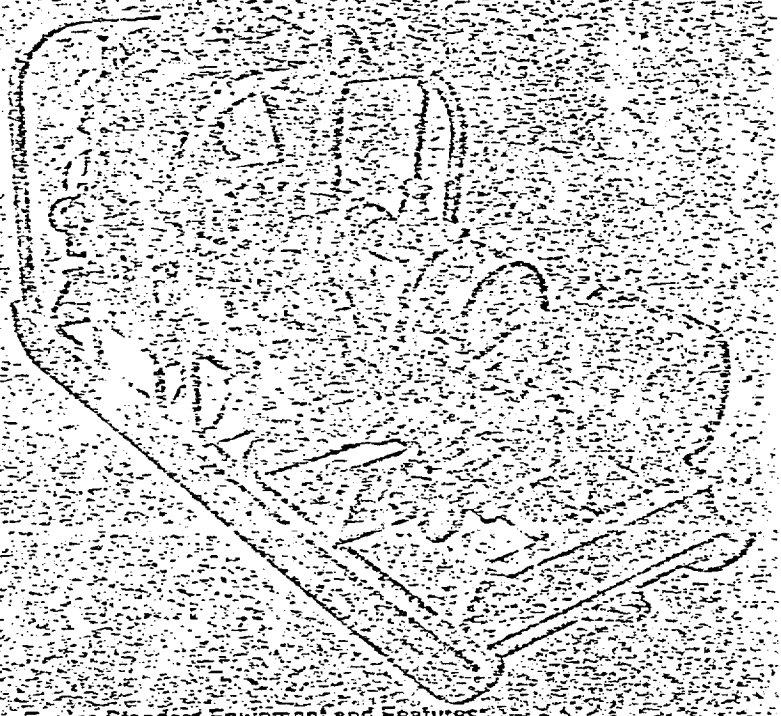
As with other Boyles Bros. equipment, this pump is designed and built for toughness and extended serviceability. The unit is mounted on a heavy-duty skid base with twin roll bars and lifting eye. The Boyles surge tank No. 18-257 and pressure relief valve No. 08-328 are standard.

Low maintenance costs are possible because of the rugged proven design of this pump unit and trouble free operation of the diesel engine.

Model 35 D 20 Pump

Price*		$7,818.00
Weight (approx.)		1,950 lbs.
Length		83 inches
Width		34 inches
Height		46 inches
Rated Volume	Speed	GPM
	1	6.8
	2	14.1
	3	25.0
	4	37.5
Pump	Boyles No.	11-903
	Make	Bean Industrial Triplex Piston Pump
	Model	W112231
	Valves	Ball
	Pressure rating inter.	1,000 p.s.i.
	Pressure rating cont.	500 p.s.i.
	Suction size	2" NPT
	Discharge size	1½" NPT
	Lubrication	5 qts SAE 30
Transmission	Boyles No.	15-978
	Make	Borg Warner
	Model	T-9
	Speeds	Four
	Oil Capacity	6 pts.
	Case	Cast Iron
	Gears	Forged, heat treated
	Bearings	Anti-friction
Engine	Boyles No.	11-971
	Make	Avco Lycoming
	Model	V-42
	Cycle	Four
	Cooling	Air Cooled
	Horsepower	20 net Continuous Operation
	RPM	3,000
	Cylinders	Two
	Valves	Overhead

FOB Salt Lake City, Utah. Subject to change without notice.



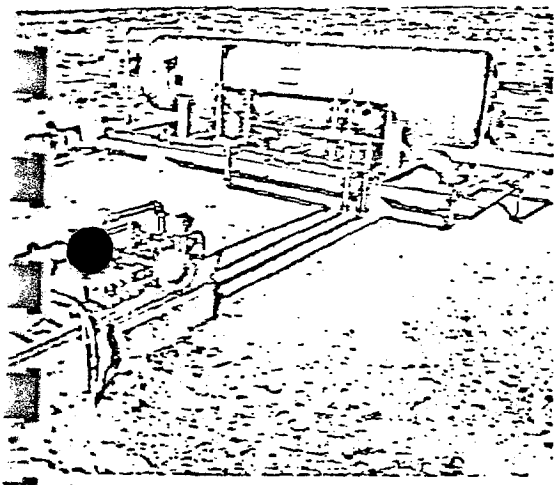
Engine Standard Equipment and Features:

- Rockford reduction gear assembly
- Rockford overcenter clutch
- Electric start
  - Starter, 12 volt
  - Starter switch
  - Battery, 12 volt with cables
  - Battery box
  - Generator, 12 volt, 13 amp
  - Voltage regulator, 12 volt, 13 amp
  - Automatic decompressor release
  - Starting crank (for optional use)
- Instrument panel
  - Amp gauge
  - Oil pressure gauge
  - Hour meter
- Filters
  - Air filter, combustion
  - Fuel filter
  - Oil filter, cleanable
- Axial cooling fan
- Pressure lubrication
- Standard oil sump - 4 qt.
- Automatic emergency low oil pressure stop
- Centrifugal governor
- Fuel injection system
- Diaphragm fuel pump with priming lever
- Fuel tank, 5 gal.
- 7½" exhaust mufflers, two each
- Engine mounting brackets

BOYLES BROS. DRILLING COMPANY

FORM C375-205

Solo incorporates an oversize 3-phase horizontal with a reserve of 50 barrels of power and twice as many cyclone centrifugal for removing abrasives from the power fluid for well-site hydraulic pumping system. Its cleaness produces a power fluid with very [illegible]ent.



C POWER PUMPS

Triplex Pumps

[illegible]put capacity of this unit makes it useful for [illegible]stallations. With a plunger diameter of 1¼" [illegible] offers up to 164 B/D displacement capacity per [illegible]27 B/D at rated speed).

Triplex Pumps

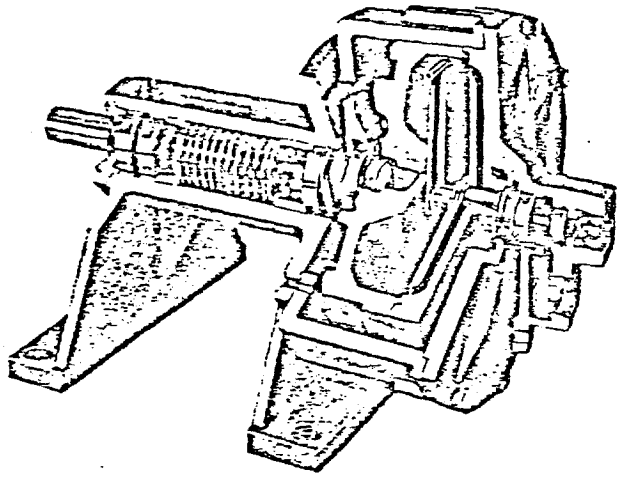
[illegible] unit is the most flexible package size for [illegible] design. Additional units can be installed as [illegible]rements increase or can be taken away and [illegible]where if requirements drop. The Size 3 with [illegible]eter of 1¾" offers a displacement capacity per 100 rpm (1850 B/D at rated speed).

[illegible]ex Pumps

[illegible]4 Triplex pump is a 180 hp. input gear drive [illegible] ratios available are 2.607, 3.407, 3.958 and [illegible] a gearless unit for slow speed engine drive. [illegible]ent is 986 B/D per 100 rpm using a 2⅜" plunger (4437 B/D at rated speed).

[illegible]umping Package

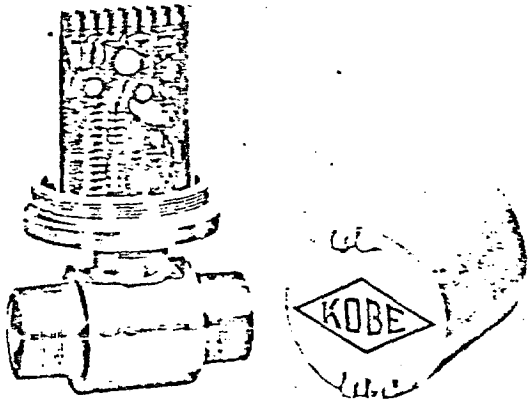
[illegible]lies a complete package for hydraulic pumping including control headers, flow and pressure and 4-way shoe-type wellhead valves.



Roto-Jet® Pump

Kobe offers five models of its new RG Roto-Jet high-pressure pump with capacities from 30 to 300 gpm. The Roto-Jet combines a rotating case with a stationary pitot tube collector arm to develop pressures up to 1000 psi. The latest versions of this pump incorporate new heavy-duty outboard bearings and offer greater efficiency and can handle greater suction pressures and higher temperature fluids. With only two basic working parts, it provides unparalleled reliability in a single-stage pump with pulsation-free flow and no need for packing glands, wear rings, throat or throttle bushings, pistons, rods, or valves.

LEASE AUTOMATION & PRODUCTION EQUIPMENT



Electronic Components

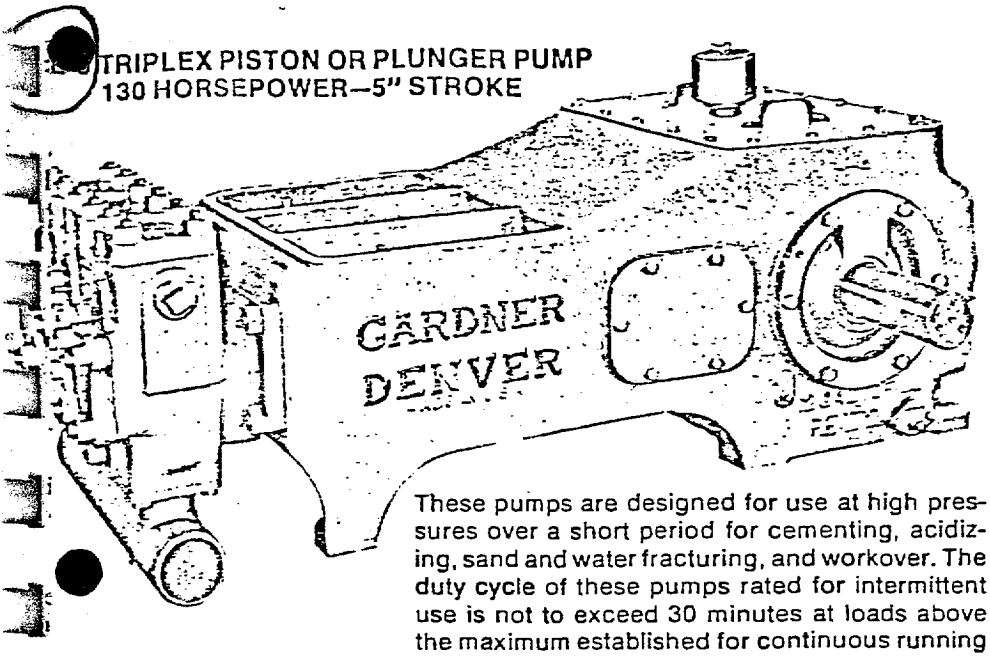
Kobe offers a complete line of end devices used in oilfield automation and fully compatible with supervisory control systems. Included in these electronic components, manufactured by Kobe Systems Division, are net oil computers, multi-point scanners, liquid vortex meters, multiple vortex meter readout panels, vortex meter totalizers, portable flow rate indicators, frequency to pulse converters, frequency to DC converters and well test panels.

WELL SERVICING PUMPS
PE-5 TRIPLEX PLUNGER / FC-FXX DUPLEX

GARDNER-DENVER

130 HP 100 HP

TRIPLEX PISTON OR PLUNGER PUMP
130 HORSEPOWER—5" STROKE



These pumps are designed for use at high pressures over a short period for cementing, acidizing, sand and water fracturing, and workover. The duty cycle of these pumps rated for intermittent use is not to exceed 30 minutes at loads above the maximum established for continuous running followed by an equal period of no load.

FEATURES:

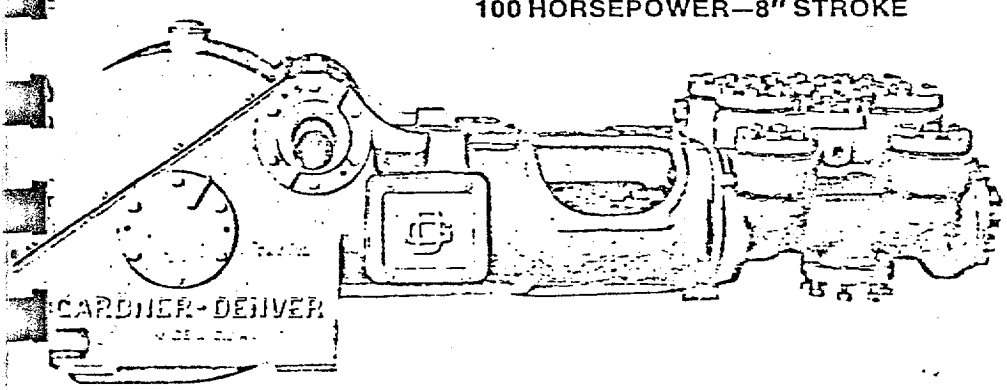
- *Eccentric construction.*
- *Left- or right-hand shaft extension.*
- *Pressure lubrication of power end.*
- *Suction and discharge on either side.*
- *Solid connecting rods.*
- *Clamp connections between plungers and crosshead extensions.*
- *Removable stuffing boxes with threaded glands.*
- *Nonadjustable-type packing.*

PE-5 SPECIFICATIONS FOR WELL SERVICING—INTERMITTENT SERVICE—130 HORSEPOWER

PUMP RPM	*PLUNGER OR PISTON DIAMETER 3"		76 mm		2¾"		70 mm		2½"		64 mm		2¼"		57 mm		2"		51 mm		1¾"		44 mm	
	GPM	PSI	LPM	Kg/cm²	GPM	PSI	LPM	Kg/cm²	GPM	PSI	LPM	Kg/cm²	GPM	PSI	LPM	Kg/cm²	GPM	PSI	LPM	Kg/cm²	GPM	PSI	LPM	Kg/cm²
[illegible]	153	1430	503	101	112	1690	423	119	92	2050	340	144	75	2540	284	178	55	3220	223	225	43	4220	170	297
[illegible]	114	1660	431	117	97	1960	367	138	80	2380	302	167	64	2950	242	207	51	3720	193	262	39	4860	147	342
200	92	2060	348	145	77	2470	291	174	64	2980	242	210	52	3580	197	259	41	4650	155	327	31	6080	117	427
150	69	2760	261	194	58	3280	219	231	48	3980	181	280	39	4900	147	245	31	6200	117	436	23	8100	87	569
	46	4150	174	292	39	4940	147	347	32	5950	121	418	26	7350	98	517	20	9300	76	654	15	10000	57	703

With Cast Steel Fluid Cylinders: 2,990 lbs. *Piston and liner construction 2½", 2¾", 3".

FC-FXX DUPLEX POWER PUMP
100 HORSEPOWER—8" STROKE



CONTRUCTION FEATURES:

- *Heavy duty roller chain — transmits power from sprocket cut in jackshaft to eccentric sprocket.*
- *Eccentric construction.*
- *Flood lubrication.*
- *Double extended jackshaft.*
- *Cast steel or alloy fluid end.*
- *Solid construction connecting rods. (No bolts)*
- *Selection of valves, pistons, liners, rods to suit service.*

Optional Equipment: Suction and Discharge Manifolds.

FC-FXXAD SPECIFICATIONS FOR WELL SERVICING—INTERMITTENT SERVICE—100 HORSEPOWER

PUMP RPM	JACKSHAFT RPM	PISTON DIAMETER 5"		127 mm		4½"		114 mm		4"		102 mm		3½"		89 mm		3"		76 mm	
		GPM	PSI	LPM	Kg/cm²	GPM	PSI	LPM	Kg/cm²	GPM	PSI	LPM	Kg/cm²	GPM	PSI	LPM	Kg/cm²	GPM	PSI	LPM	Kg/cm²
[illegible]	383	260	560	985	39	208	700	787	49	162	900	613	63	121	1200	458	84	86	1700	325	120
[illegible]	345	234	620	886	44	187	780	708	55	146	1000	553	70	109	1340	413	94	77	1870	291	133
80	308	208	700	787	49	167	870	632	61	129	1130	488	79	97	1500	367	105	69	2120	261	149
70	268	182	800	689	56	146	1000	553	70	113	1290	428	91	85	1720	322	121	60	2430	227	171
[illegible]	230	156	935	590	66	125	1160	473	82	97	1500	367	105	73	2000	276	141	51	2860	193	201
[illegible]	191	130	1120	492	79	104	1400	394	98	81	1800	307	127	60	2430	227	171	43	3390	163	236
[illegible]	153	104	1400	394	98	83	1760	314	124	65	2240	246	157	48	3040	182	214	34	4290	129	302
[illegible]	115	78	1870	295	131	62	2350	235	165	48	3040	182	214	36	4060	136	285	26	5000	98	352
20	77	52	2040	197	143	42	2500	159	176	32	3180	121	224	24	4160	91	292	17	5000	64	352

Approximately 4700 lbs Maximum piston load 40 000 lbs

GASO

SERIES 3200 — 2" STROKE TRIPLEX PLUNGER PUMPS

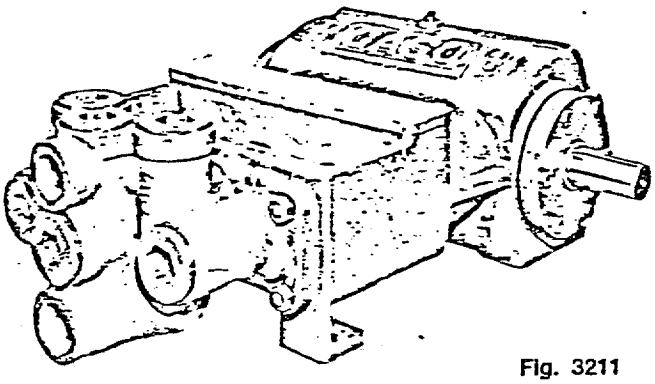
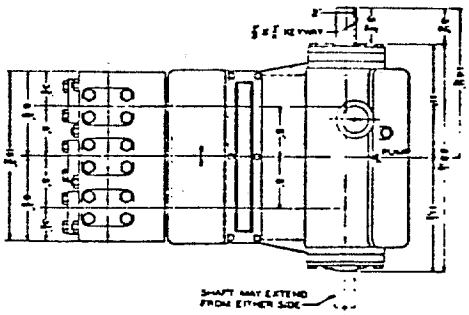


Fig. 3211
1¼" to 2¼" x 2"

Fig. 3211 with Billet Fluid End



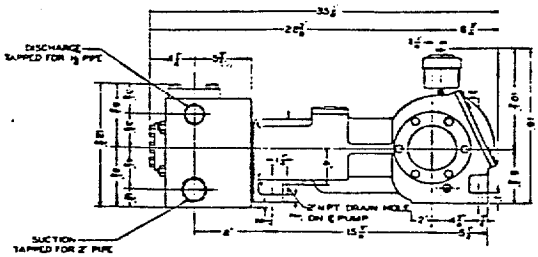
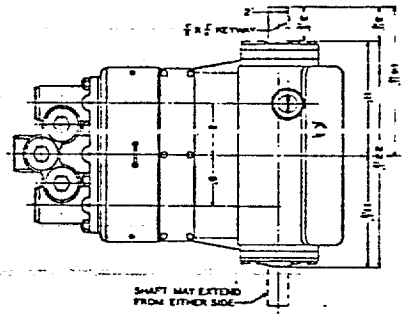
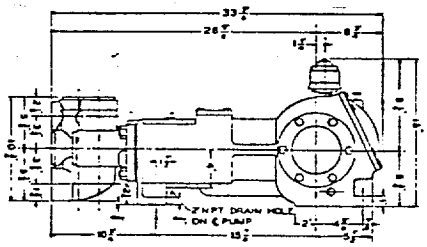


Fig. 3211 with Cast Fluid End





SPECIFICATIONS

No. of Plungers - 3

Stroke Length - 2"

Plunger Load - 2185 lbs.

Oil Capacity - 1½ U. S. gallons

Connections - Fig. 3211-H
- Standard - 2" - 1500# ANSI suction
  - 1½" - 1500# ANSI discharge

Connections - Fig. 3211
- Cast Fluid End
  - Standard - 2" NPT suction
    - 1½" NPT discharge
- Billet Fluid End
  - Standard - 2" NPT suction
    - 1½" NPT discharge
  - Optional - 2" - 600# ANSI suction
    - 1½" - 600# ANSI discharge

Pump Weight
- Cast Fluid End - 530 lbs.
- Billet Fluid End - 830 lbs.

Ratings published below are intended to be used only for preliminary planning purposes, and as such carry no warranty whatsoever. All applications for Gaso pumps must be approved by Gaso in writing.

PUMP FIG. NO.	PLGR. DIA. IN.	MAX. PRESS PSI	DISP. GAL. PER REV.	VOLUMES							
				100 RPM		200 RPM		300 RPM		400 RPM	
				GPM	BPD	GPM	BPD	GPM	BPD	GPM	BPD
3211-H	⅞	3300	.0156	1.6	55	3.1	105	4.7	160	6.2	215
	1	2780	.0204	2.0	70	4.1	140	6.1	210	8.2	280
	1 ¼	1780	.0319	3.2	110	6.4	220	9.6	330	12.8	440
3211	1 ¼	*1780	.0319	3.2	110	6.4	220	9.6	330	12.8	440
	1 ½	1235	.0459	4.6	160	9.2	315	13.8	475	18.4	630
	1 ¾	910	.0625	6.3	215	12.5	430	18.8	645	25.0	860
	2	695	.0816	8.2	280	16.3	560	24.5	840	32.6	1120
	2 ¼	550	.1033	10.3	355	20.7	710	31.0	1065	41.3	1415
Brake Horsepower Required				3.6		7.3		11.0		14.6	

ireline Oil Savers

es H
lraulic Wire-
ne Oil Savers

Type H With Guard

iberson hydraulic avers strip wire- clean of fluids; r costs by saving aluable oil; pro- e safety by imizing danger f explosion or fire the well site.

ilable in several types to meet every stripping re- ement, Guiberson hydraulic oil savers use a special, il, gas and abrasion resistant, one piece rubber for effici low cost oil removal.

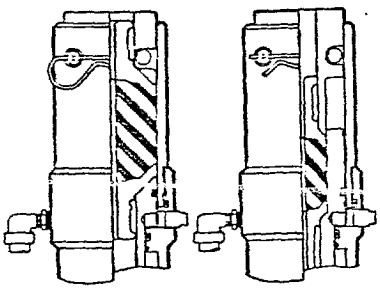
eries H savers are operated by the field-proved piston iciple; there are no troublesome bladders or compli- eated mechanisms. All provide an extra large rubber and enerous bushings for smooth, constant seal and imum utilization of each rubber.

H savers are available with 2⅜" (60mm) EUE or 3½ (89mm) O.D. LP 8V field connections, and all savers may be adapted from one connection to another simply by nging the base. Each complete saver includes a lraulic-fluid-filled hose with quick change connections and a check valve at each end. A large capacity pump is also included. The accurate pressure gauge of this pump is efully calibrated. The condition of the rubbers and the nt at which they should be changed is clearly indicated on the dial. The pump is also filled with fluid at the factory, making the entire oil saver assembly ready to operate as soon as it is installed.

Type H Rubber

Type H Oil Saver Guiberson's Type H is a 3000 psi (211 kg/cm²) test version of our single stripper hydraulic oil saver. It has proven its reliability in thousands of installations.

Type HD Oil Saver The Type HD saver offers the added safety and convenience of double line stripping. Basically similiar in design to the Type H.



Types H & HS5 Type HS10

Types HS5 and HS10 Oil Savers These hydraulic oil savers are designed for the higher pressures now commonly found in wireline operations. The Type HS5 is rated at 5000 psi (352kg/cm²) test pressure; the Type HS10 at 10,000 psi (703kg/cm²).

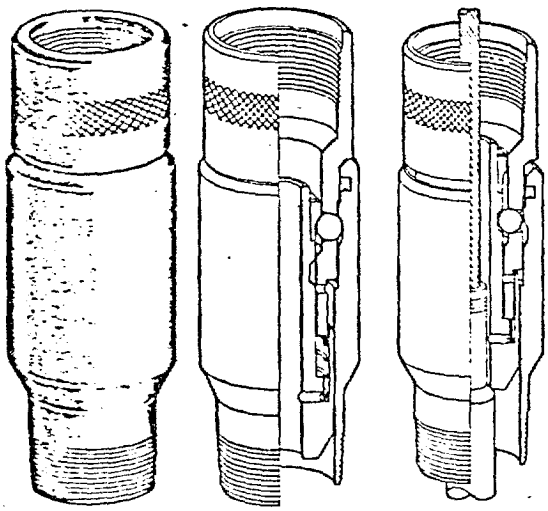
ype C Releasing Attachment

e Type C releasing attachment provides a safe, simple, automatic release for Guiberson wireline oil savers. The Type C locks the saver securely to the tubing, lubricator or rotating rculation head while tools are in the hole and needs no justment in operation. It releases automatically when the rope socket contacts the locking dog. The Type C will not trip under ressure, nor will it leak while pulling swabs, core barrels or rveying instruments.

Engineering Data — Type C Releasing Attachment

Part Numbers	55834	59218*	33276	45272	53763
Tubing Top Connection-Box	2⅜" (60mm) O.D.	2⅞" (73mm) O.D.	2⅞" (73mm) O.D.	3½" (89mm) O.D.	4½" (114mm) O.D.
Thread Top Connection	EUE 8 Rd.	EUE 8 Rd.	EUE 8 Rd.	8V	EUE 8 Rd.
Bottom Connection Pin	2⅜" (60mm) O.D.	4½" (114mm) O.D.	3½" (89mm) O.D.	3½" (89mm) O.D.	4½" (114mm) O.D.
Thread Bottom Connection	EUE 8 Rd.	8V	8V	8V	EUE 8 Rd.
Hose Size	¼" (6mm)-1" (25mm)	¼" (6mm)-1" (25mm)	¼" (6mm)-1" (25mm)	¼" (6mm)-1" (25mm)	¼" (6mm)-1" (25mm)

Note: When ordering specify part number, type and connections
*Full opening for 4½" (114mm) swab



Type C Engaged Disengaged

Releasing Attachment

KPG
Subsurface Recording Gauges
Amerada RPG-3 Type

GRC

The Kuster KPG Subsurface Recording Gauge uses the time-proven system of a multiple coil bourdon tube and sliding chart tube perfected in the early 1930's. Exclusive Kuster developed improvements have been added for rugged, dependable operation. These improvements have been field proven to minimize repair and maintenance. The KPG is completely interchangeable with the Amerada RPG-3 gauge.

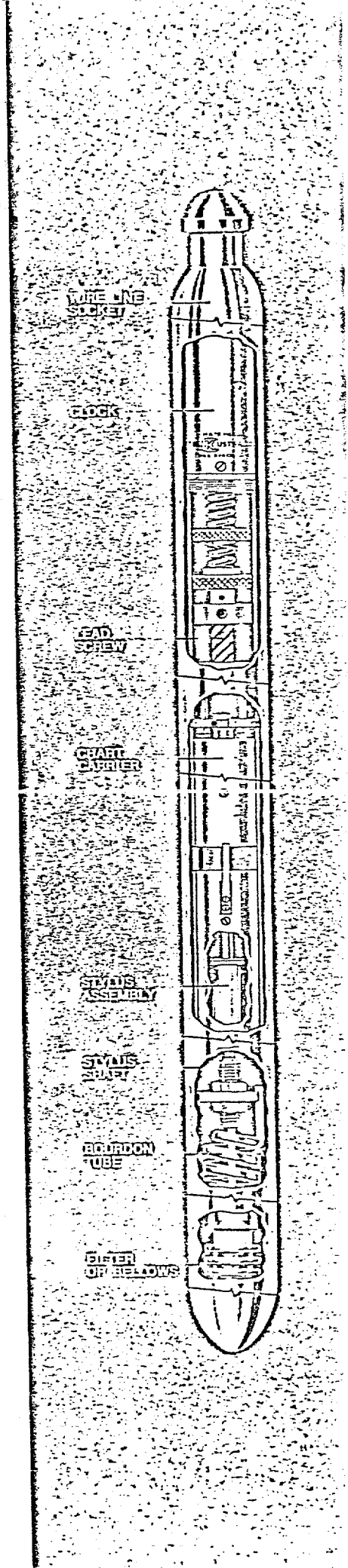
Features

The KPG clock is designed, manufactured and assembled by Kuster personnel. This same design with improvements in material and operation has been used world-wide in the Amerada type gauge. The range is from 2 to 360 hours. The clock hours can be reduced one-half by using a fast lead screw. By custom assembly and lubrication operation to 700°F is obtainable. The Kuster designed lead screw and nut assembly has a sealed ball bearing and solid bronze nut. The chart carrier is of hardened stainless steel to withstand any accidental deformation. Bourdon tubes are of Ni-Span C temperature corrected to about 220°F. A calibration table furnished with each pressure element provides fast, accurate (to 0.1 psi) conversions from deflection to pressure. The KPG Gauge is calibrated by direct intercomparison against a National Bureau of Standards calibrated dead weight piston gauge. Bellows are plastic coated for years of trouble free service.

Operation

Pressure from the well is sensed by the bellow-bourdon tube fluid filled system. Pressure increase causes the bourdon tube to rotate. This rotation is transmitted to the stylus assembly by means of the stylus shaft. The chart in the chart carrier is moved past the stylus at a controlled rate by a spring driven clock. A recording of pressure deflection vs. time is made in one passage of the stylus across the face of the chart with no overlapping. The recording by the stylus is a very lightly burnished mark on the black or grey coated chart. The mark is read with a chart reader (page 8) to the nearest .0005 inch. Conversions to pressure are made with the Kuster digital printed calibration table supplied with each element.

Note: All instruments available in metric and SI units.



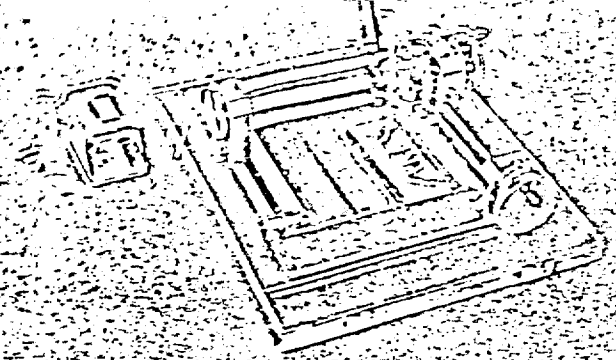
Specifications

Accuracy	±0.2% of full scale
Sensitivity	1 part in 2000
Length	73⅜ inches (186 cm)
Diameter	1¼ inches (32 mm)
Weight	15 pounds (6.8 kg)
Pressure Ranges	800 to 30,000 psi (56 to 2100 kg/sq cm)
Temperature Operating Range	25° to 700°F (0°-370°C)
Chart Size	2" x 5"
Stylus Travel	5" (15 turns of lead screw)
Bourdon Tube	Ni-Span C, temperature compensated
Clocks	Kuster jeweled, operation to 370°C (3 hr)
Clock Hours	2, 3, 12, 24, 48, 72, 120, 144, 168, 180, 360

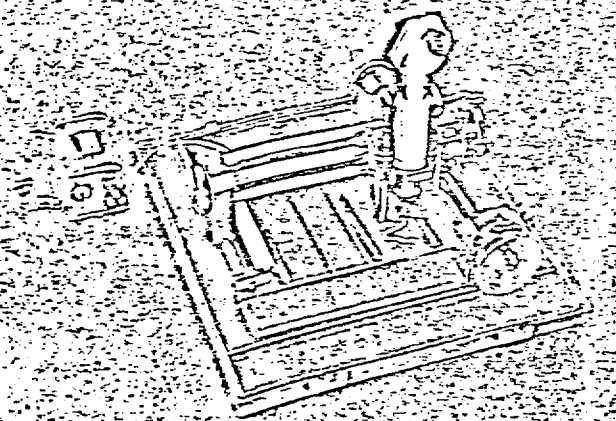
KPG Gauge, complete includes standard clock, pressure element, thermometer well, tools, o-rings, 25 charts, steel stylus (sapphire or diamond extra), calibration table.*	Part No. 10000-1
KPG Gauge, complete as above with 360° hr clock	Part No. 10000-3
KPG Gauge, complete for Temperature Recording includes standard clock, KT Temperature Element ranges from 0° to 370°C, tools, o-rings, 25 charts, steel stylus, calibration table.	Part No. 10000-2
KPG Gauge Recorder only less pressure element and clock, but includes thermometer well, wire line socket tools, o-rings, 25 charts, with steel stylus (sapphire or diamond extra). Net weight 9 lbs. Interchangeable with the Amerada RPG-3.	Part No. 10100
KPG Pressure Element, complete with calibration ranges 800 to 30,000 psi (56 to 2100 kg/sq cm)	Part No. 10201-1
KT Temperature Element, complete with calibration ranges 30-700°F (0-370°C)	Part No. 600-1/601-1
KPG Clocks, ranges 3, 12, 24, 48, 72, 120, 168, 180, 240, 360 hours specify service temperature, hour	Part No. 1201
KPG Carrying Case, holds 1 - recorder, 2 - clocks, 3 - elements, tools and accessories	Part No. 10254
KPG Shop Tool Kit, complete set for major maintenance	Part No. 10236

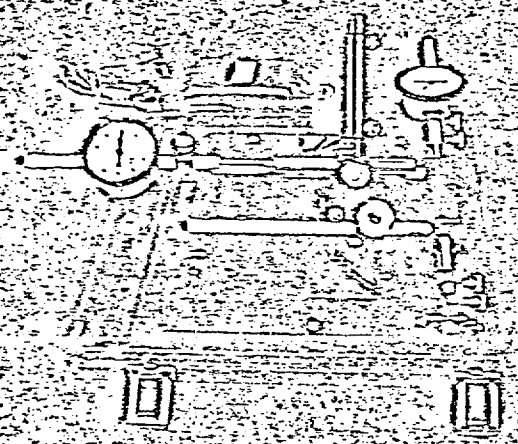
*Specify range of pressure or temperature element

Chart Readers

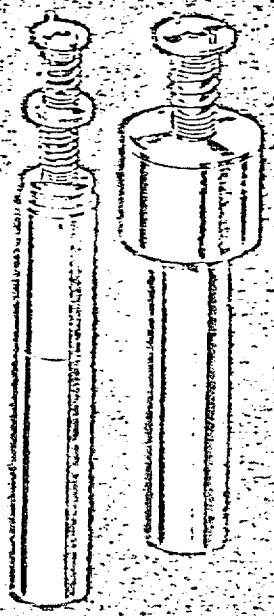


2-Way Chart Reader

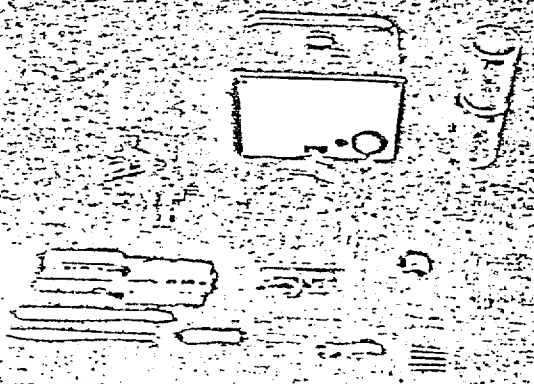




Field Reader



Subsurface Gauge Blocks



Clock Maintenance Equipment

Production — Injection Packer

LYNES

OPERATION

The PIP is inflated by applying pressure to the tubing. High pressure fluid is trapped in the packer element by a triple seal check valve, which is the ONLY moving part in the packer. A shear plug, temporarily plugging the bottom of the mandrel, is removed by tubing pressure.

Right hand rotation and slight upward movement of the tubing releases the entrapped fluid to the annulus to deflate and retrieve the packer.

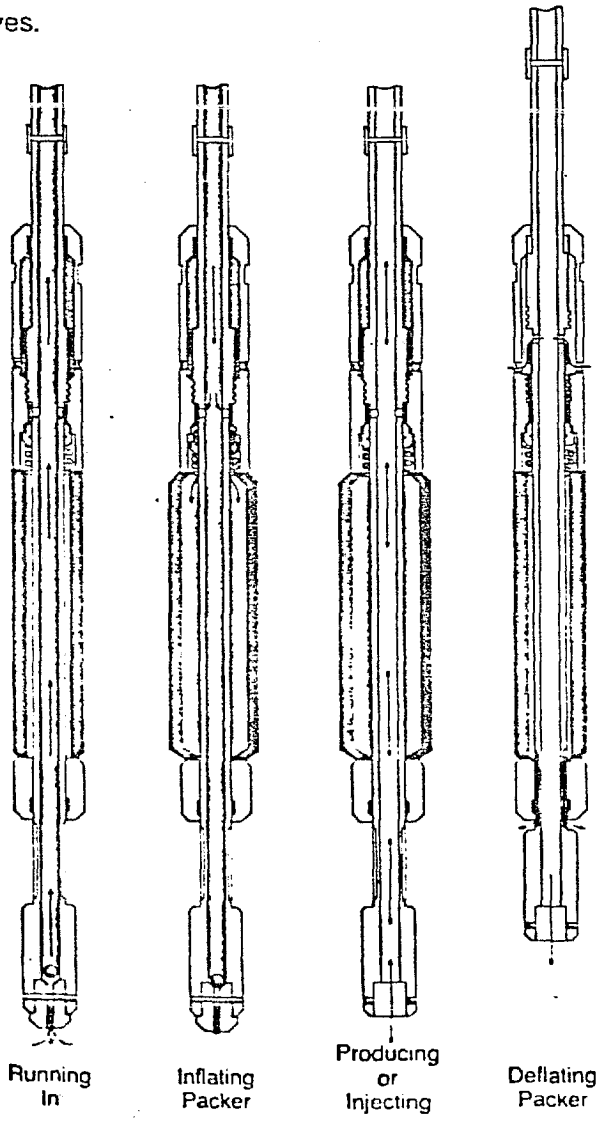
Pressure capability of the packer varies with hole or casing size and is approximated in the Differential Pressure Curves.

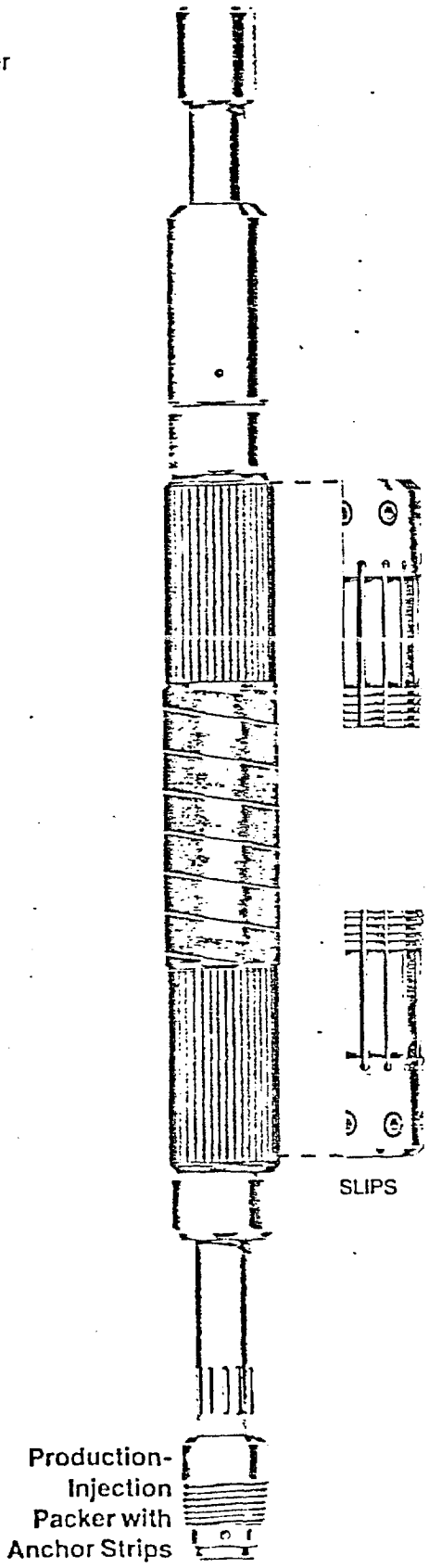
SPECIAL FOR CASING

A Special PIP packer is available with ANCHOR STRIPS in the outer cover for EXTRA anchoring ability in casing.

It has particular application in highly deviated wells where it is difficult to operate mechanical packers.

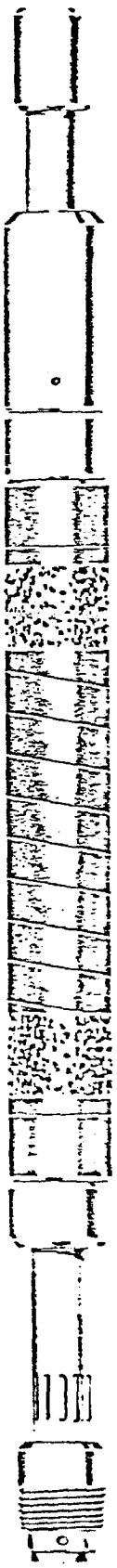
*For maximum anchoring ability, a PIP packer is available with collet slips completely surrounding the element.





LYNES

Production — Injection Packer



Product No. 300-01

Product No. 300-01
The PIP (Production-Injection Packer) is a single set high-pressure inflatable packer for permanent or temporary use in OPEN HOLE or casing.

APPLICATIONS/BENEFITS
The PIP packer is the most versatile packer available ANYWHERE. It is used for . . .

- Producing Wells
- Injection Wells
- Zone Isolation
- Treating, Acid or Frac
- Snubbing Under Pressure
- Side Wall Impressions
- Will even seal in a casing collar!!
- Squeeze Cementing
- Bridge Plug
- Setting thru Restrictions
- Test B.O.P.'s
- Sealing in Corroded, Split, Perforated, or Slotted Casing

FEATURES:
- Seals in Irregular/Out of Round Holes or Casing
- Self Anchoring (No weight, tension, or hold downs required)
- Long Continuous Seal (52" Wellbore contact is standard . . . 10 ft. seals are available)
- GREATER EXPANSION
  . . . for Running Clearance
  . . . Setting below restrictions
  . . . One size fits many holes
  i.e. 4¼" will seal in 5", 5½" 6⅝", 7" and 7⅝" casing.

AVAILABLE SIZES

SINGLE SET PRODUCTION — INJECTION PACKER
Product No. 300-01
Outside diameters from 3½" thru 7⅞". Elements available in three lengths: 42" (27" actual seal); 66" (52" actual seal) and 136" (122" actual seal)

SLIM HOLE PRODUCTION — INJECTION PACKER
Product No. 300-06
Outside diameters from 1 7/16 thru 2¾". Elements available in two lengths: 23" (12" actual seal) and 48" (36" actual seal)

HALLIBURTON SERVICES

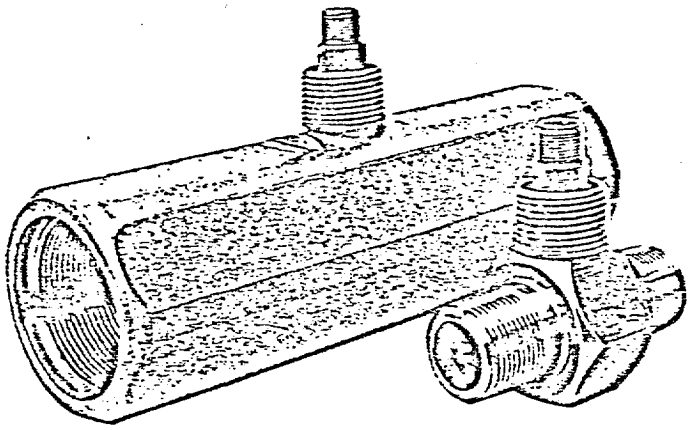
Measurement and Sensing Instruments

Industrial Turbine Flow Meters

In 1953 Halliburton developed the first successful turbine flow meter that incorporated tungsten carbide shaft and bearings. Since then, these meters have built a record that's unsurpassed for taking rugged punishment in stride while providing outstanding accuracy in a wide range of industries.

Every Halliburton meter is backed-up with service, engineering assistance and a replacement parts availability that can assist in both initial proper application and continued long-term operation with minimum maintenance.

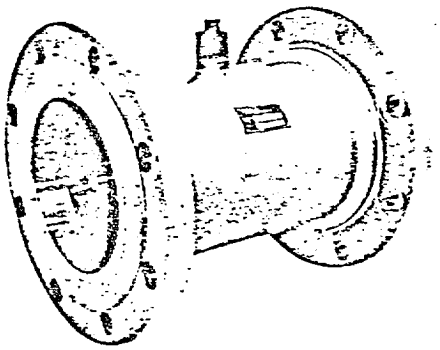
THREADED TURBINE FLOW METERS



FLOW METER SIZE	GPM	LINEAR FLOW RANGE* BPH	BPD	NOMINAL CALIBRATION FACTOR (PULSES/GAL)	MAXIMUM OUTPUT FREQUENCY (PULSES/SEC)
½"†	.75-7.5	1-10	25-250	13,000	1,625
¾"†	2-15	3-20	66-515	3,000	750
1"	5-50	7-70	170-1,700	920	765
1½"	15-180	21-250	515-6,000	330	990
2"	40-400	54-540	1,300-13,000	55	365

*The linear flow range of liquids with non-lubricating characteristics is limited to the upper 60% of rating.
†Recommended only for viscosities less than 5 centipoise.

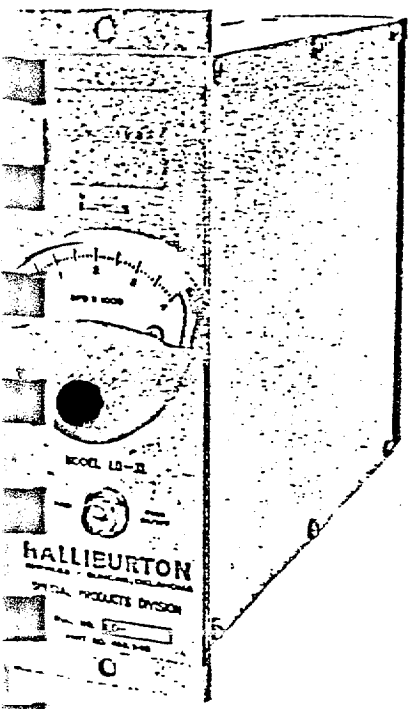
FLANGED TURBINE FLOW METERS



FLOW METER SIZE	GPM	LINEAR FLOW RANGE* BPH	BPD	NOMINAL CALIBRATION FACTOR (PULSES/GAL)	MAXIMUM OUTPUT FREQUENCY (PULSES/SEC)
1"	5-50	7-70	170-1,700	920	765
1½"	15-180	21-250	515-6,000	330	990
2"	40-400	54-540	1,300-13,000	55	365
3"	60-600	88-880	2,100-21,000	57	570
4"	100-1,200	142-1,708	3,400-41,000	29	580
6"	200-3,000	283-4,250	6,800-102,00	7	350
8"	350-4,000	500-5,708	12,000-137,000	3	200

*The linear flow range of liquids with non-lubricating characteristics is limited to the upper 60% of rating

O-II Flow Totalizer



A flow totalizer should offer rugged construction, operational omy and maintenance-free ures, and the Halliburton LO- Flow totalizer, designed for use with Halliburton's Turbine Flow r, provides economical perfor- ce, ruggedness and easy peration and maintenance, plus e advantage of compactness.

e LO-II totalizer conserves l space because of its com- actness. A 10½ by 19 inch panel ill accommodate up to six izers. The LO-II instrument significantly lower cost than mparable units. Its solid state, tegrated circuit card means lified operation. All the elec- ic circuitry, including the ower supply transformer, ounts on a single plug-in board, h means simplified operation maintenance. Plug-in connec- rs accomplish calibration adjust- ent.

Installation requires no specialized training or equipment.

This versatile instrument can be operated on a table, in a panel, on a rack mount, or can be furnished in a weatherproof housing for outdoor installation. All materials in the unit are primarily non-corrosive, another maintenance advantage.

Single units are in stock. A companion rate meter can be used with as many as 16 totalizers.

Accurate to within ±1 count, the LO- II totalizer registers total flow directly in barrel increments. Calibration adjustments can be made for all integers up to 16,000 pulses per count simply by plugging a pin in a socket.

Every LO-II instrument is factory-adjusted to match its companion turbine flow meter. The turbine flow meter transmits an AC signal with a frequency proportional to the fluid velocity. The LO-II totalizer receives the signal, amplifies it and shapes it. The frequency is reduced by a predetermined adjustable divisor established by the flow meter calibration factor. This signal drives the counter, which registers the totalized flow in barrels.

Besides offering economy, rugged construction, compactness and ease of maintenance, the LO-II instrument has these optional features:

PULSE OUTPUT - A dry contact closure to actuate a counter located remotely from the flow totalizer, or a remote terminal unit.

FLOW RATE - The rate of flow can be displayed on a meter located on the front of the instrument.

SPECIFICATIONS:

- Power Supply—110 volt, 60 Hz, 30 watts
- Temperature Range— −20° to +130°F
- Totalizer—8-digit non-reset (standard)
- Totalizer Units—See Table 1
- Accuracy—±1 count
- Input Frequency—0 to 1000 HZ
- Input Amplitude—30 to 2000 mv peak to peak
- Signal Cable Length—50′ (standard)
- Power Cable Length—10′
- Pulse Output Option—Normally open contacts, 10 VA contact rating—½ amp max or 200 vdc max, 60 ms closures at same increments as totalizer
- Flow Rate Option—Instantaneous flow rate in desired units displayed on meter located on front of instrument
- Test Switch Option—Built-in 120 Hz test frequency to check instrument operation
- 12 Volt DC Power Option—Instrument wired for 12 volt DC supply

Table 1

Turbine Flow Meter Size	Totalizer Units
⅜″	0.01 bbl
½″	0.01 bbl
¾″	0.10 bbl
1″	0.10 bbl
1½″	bbl
2″	bbl
3″	bbl
4″	bbl
6″	bbl
8″	bbl

General Specifications

GS
3-1A1 G
March 1977

MODEL 12 FLOW RECORDER AND INDICATOR

The Model 12 with a Type 9, 27, 28, or 37 Differential Pressure Element continuously records or indicates measurement of one or more flow variables. Used whenever accurate chart records or precision indication are required for efficient flow supervision. Provides high accuracy, local reading, and fast response to flow change.

Broad Application Used throughout the process industries on flows of liquids, gases, and steam.

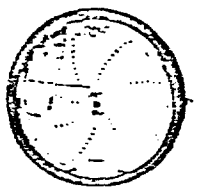
Wide Choice of Differential Pressure (ΔP) Ranges Twenty-four ranges from 0 to 0.25 up to 0 to 50 kPa, 0 to 1 up to 0 to 200 inH_2O, 0 to 2.5 up to 0 to 500 mbars. Suppressed-zero and elevated-zero ranges are also available.

Precision Motion Transfer Motion is transferred from measurement element or receiver bellows to pen movement with friction-free, ball-joint type linkage with no backlash. A micrometer adjustment on the pen arm assembly enables precise zero setting.

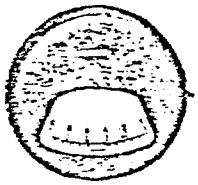
Durable Case Construction Drawn steel with corrosion-resistant vinyl finish. Fully gasketed, dust-tight door. Meets NEMA 3 weatherproof classification.

Versatile Mounting Choice of yoke, flush panel, portable, or wall mounting.

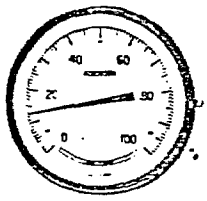
Dimensionally-Stable Charts Precision printed in a humidity-controlled atmosphere on special paper. HUMITEX™ charts maintain the accuracy of the record under widely varying humidity.



Circular Chart Recorder



Eccentric Scale Indicator



Concentric Scale Indicator

PERFORMANCE

(In % of span not including primary device)

Element	ΔP Spans	Accuracy	Repeatability	Dead Band
T/9	0.25 up to 0.375 kPa 1.0 up to 1.5 inH_2O 2.5 up to 3.75 mbar	±1.0	0.1	0.2
	0.375 kPa and above 1.5 inH_2O and above 3.75 mbar and above	±0.5	0.1	0.2
T/27	All	±0.5	0.1	0.2
T/28	All	±0.25	0.1	0.1
T/37	5 up to 12.5 kPa 20 up to 49 inH_2O 50 up to 122.5 mbar	±0.5	0.2	0.02
	12.5 up to 25 kPa 50 up to 99 inH_2O 125 up to 250 mbar	±0.25	0.15	0.02
	25 kPa and above 100 inH_2O and above 250 mbar and above	±0.25	0.1	0.02

STANDARD SPECIFICATIONS

Case Circular, drawn steel with gasketed dust-tight door. Meets NEMA 3 weatherproof classification.

Finish Black, textured vinyl

Mounting Yoke or flush panel, as specified.

Element Mounted on rear of case. Maximum of two per instrument.

Connections
Element Refer to individual element specifications.
Electrical 22 mm (0.9 in) hole for PG16 or 1/2 inch conduit.

Indicator Pointer Black

Indicator Scale Black markings on white background. Refer to Chart and Dial Catalog 600 for available ranges.
Eccentric 140 mm (5.5 in) effective length
Concentric 595 mm (23.4 in) effective length

Recorder Chart Drive
Electric 24 hour rotation, 120 or 240 volts, 50 or 60 Hz, as specified.

Mechanical 24 hour rotation, 24 hour wind.

Recorder Pens 1 or 2, box type or V-type, as specified.

Recorder Supplies
Charts 100 HUMITEX nominal 300 mm (12 in) circular charts per instrument.

Ink 29.6 ml (1 fl oz) per pen in plastic dispenser.

Mass Approximate net mass less fluid but including element.

Type 9, Type 37 34 kg (75 lb)

Type 27, Type 28 43 kg (95 lb)

TM Trademark of The Foxboro Company

FOXBORO

Printed in U.S.A.

ORIFICE PLATES, FLANGES, AND CONNECTIONS

The orifice plate is by far the most commonly used primary element. The plate is installed in the pipe between flanges; since the orifice is smaller than the pipe, the pressure below the orifice is less than that above the orifice; this difference in pressure is a measure of the rate of flow. The size of the orifice is critical. Calculation of orifice size is keyed to the method of installation, so installation procedures must be rigidly followed.

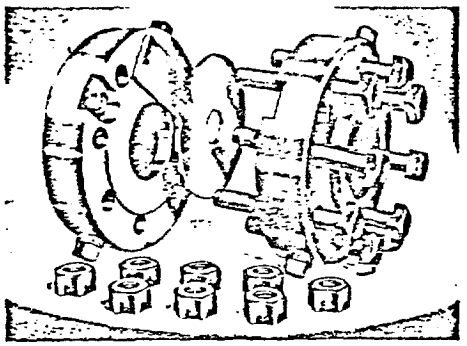


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Fig. 1

— PRESSURE CONNECTIONS —

Except when special instructions to the contrary are issued, pressure connections for the piping from the primary element to the instrument should be made as follows: (a) on a gas installation, to the top of the pipe or flange, (b) on a liquid installation, to the side of the pipe or flange, and (c) on a steam installation, to the top of the pipe or flange when the instrument is above the line, and to the side when it is below the line. When the connections are made to vertical piping, location can be anywhere around the circumference of the pipe. The more common types of connections are:

1. *Flange Taps.* Pressure connection holes are located in the flanges, 1 inch upstream and 1 inch downstream from the corresponding orifice plate faces. See Figs. 3, 4, and 5.

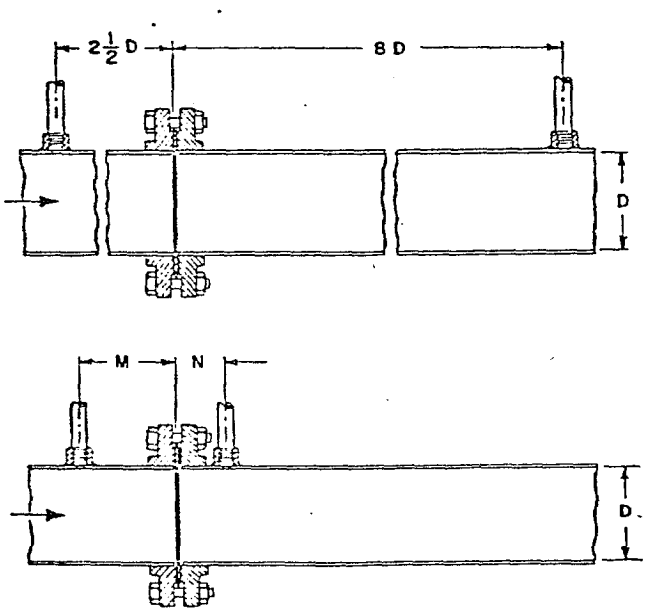


Fig. 2

2. *Corner Taps.* On lines smaller than 2 inches, the orifice flange union shown in Fig. 6 is used. The effect is that the pressure taps are right at the orifice plate.

3. *2½ D and 8 D Taps,* also called Full-Flow Pipe Taps. Connections are installed 2½ pipe-diameters upstream from the upstream face of the orifice plate and 8 pipe-diameters downstream from the upstream face of the orifice plate. See upper illustration in Fig. 2.

4. *1 D and ½ D Taps.* This type of connection (also called Radius Taps) is a modification of, and has largely replaced, the Vena Contracta taps. It yields equally accurate results and has the advantage that the downstream connection is easier to locate. The upstream connection (dimension M, Fig. 2) should be at 1 D above the upstream face of the orifice plate, but location of the upstream connection anywhere between ½ D and 2 D introduces only negligible error. The downstream connection (dimension N, Fig. 2) is located ½ D downstream from the upstream face of the orifice plate.

5. *"Vena Contracta" Taps.* The upstream connection (dimension M, Fig. 2) may be located anywhere between ½ and 2 pipe-diameters above the upstream face of the orifice plate. The downstream connection (dimension N, Fig. 2) is made at the point of maximum flow constriction and is given by the table below:

Orifice to Pipe Diameter Ratio, d/D	Location of Downstream Pressure Tap (N) (Pipe-Diameters)		
	Minimum	Mean	Maximum
0.2	.37	.85	1.30
0.3	.44	.80	1.15
0.4	.47	.73	1.00
0.5	.47	.66	.84
0.6	.42	.57	.70
0.7	.35	.45	.55
0.8	.25	.33	.41

NL Hycalog

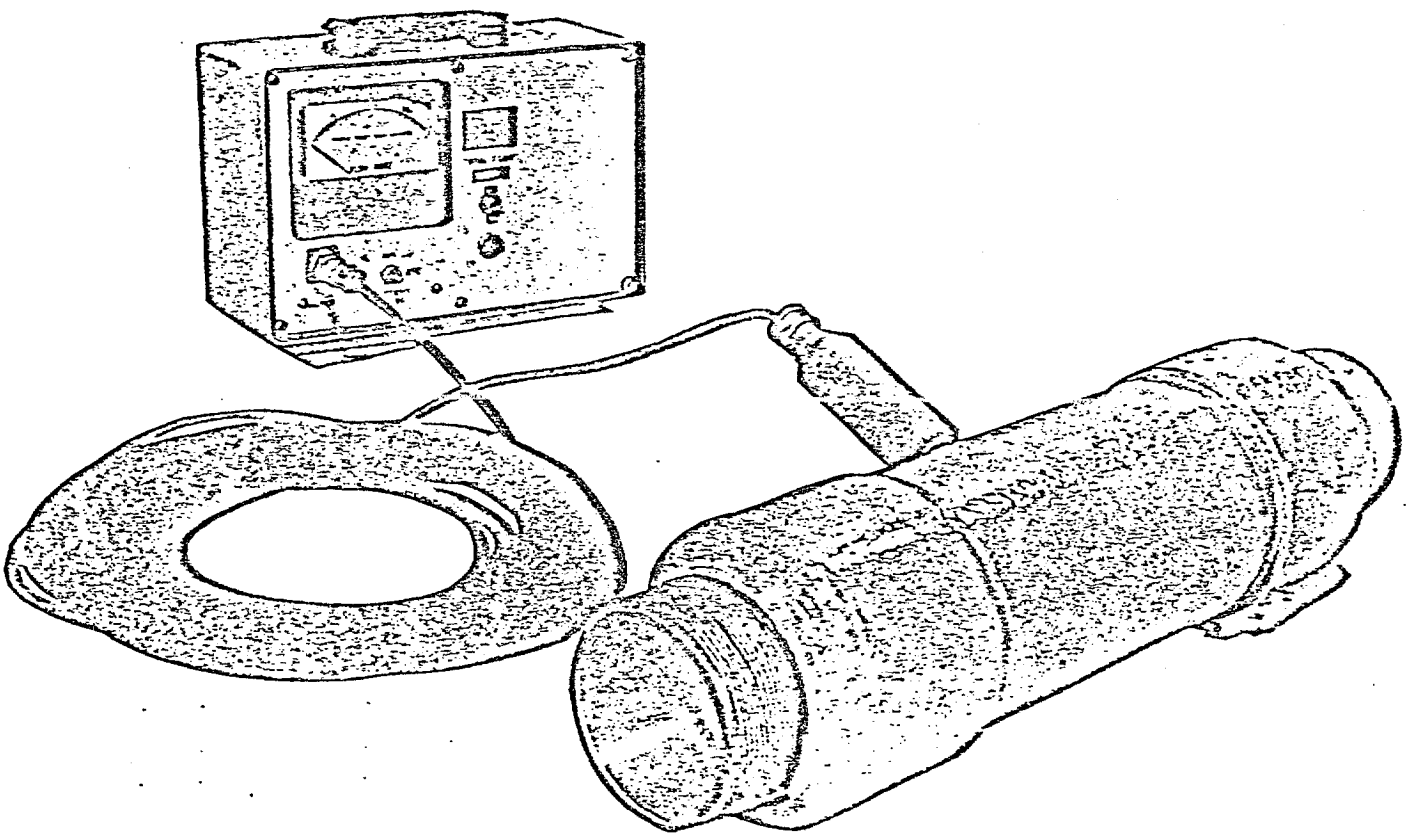
VOLUME METER SURVEY

A patented* service for determining cement volumes—without lost rig time.

NL Hycalog's Volume Meter Survey is the ideal method for predetermining the volume requirements for any selective cementing job. It gives accurate information, already computed into cubic feet—without lost rig time.

It's a one man, one meter Survey—exclusive with NL Hycalog. Use the service to figure lag time on samples—or to measure the precise volume of cement required to:

- Circulate cement behind casing.
- Bring cement volume to any desired height between casing and well bore.
- Circulate cement between stage collars.
- Circulate a liner.



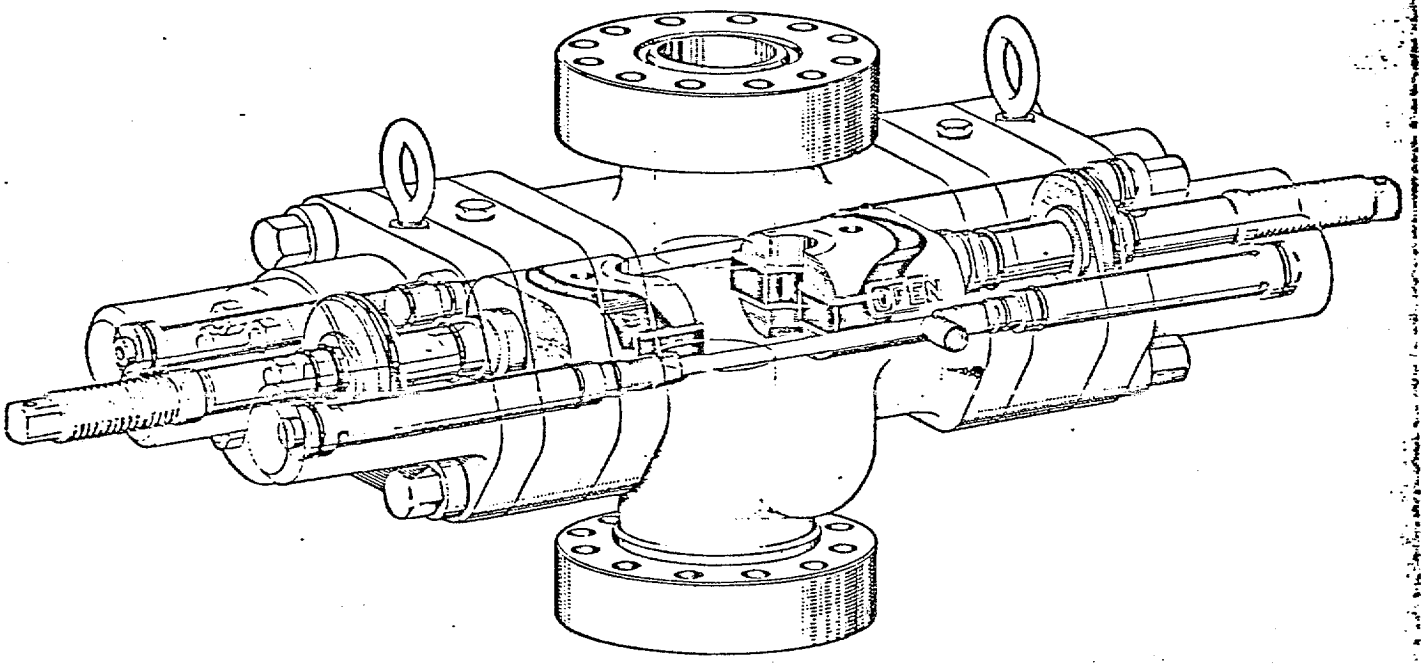
Performance Features

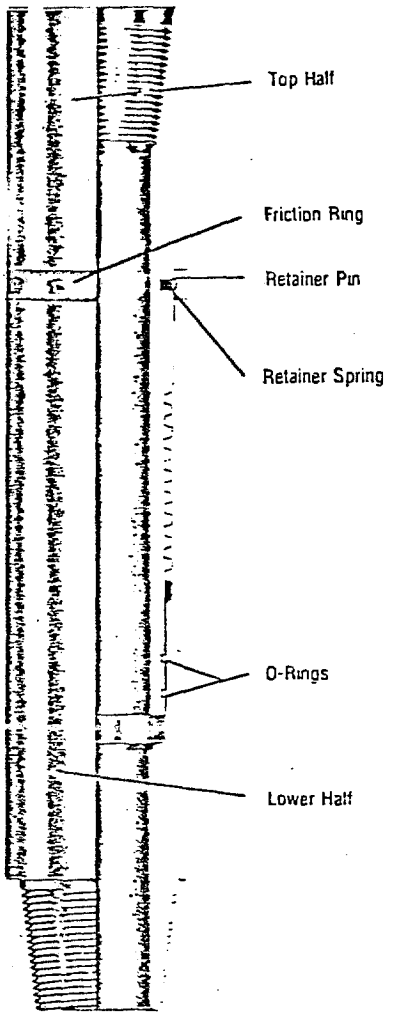
- NO LOST RIG TIME—NL Hycalog simply breaks the union and installs a Volume Meter on the discharge end of your mud pump . . . survey runs while you drill ahead or circulate to condition the hole before running casing.
- NO GUESSWORK—Highly-accurate electronic recorder gives you positive readings, already converted into cubic feet. Unlike wireline caliper surveys, there's no chance of mathematical error in converting hole sizes to cubic feet.
- NO FOUL-UPS—Nothing mechanical is run in the hole. No wireline units required. No fishing problems. No skipped-over fissures, cavities, or thief-zones. Best of all, better cementing results without the necessity for extra temperature logs, "1-inch jobs," or perforations and squeeze jobs.
- FULLY PROVEN—NL Hycalog's Volume Meter Survey has been consistently reliable in all parts of the U.S., on literally thousands of successful jobs, including wells to 23,000 feet and below.
- LOW COST—Users tell NL Hycalog the Volume Meter Survey cost virtually adds up to zero when they balance the savings in rig time and wireline surveys against the price—including travel to remote locations.

*U.S. Patent No. 3,020,952

CAMERON TYPE "U" BLOWOUT PREVENTERS

- Major components forged for greater strength
- Generous, self-feeding ram packing
- Moderate operating pressure
- Bonnets are opened and closed hydraulically
- Easy ram change
- Type "U" Preventers are compact
- All working parts are enclosed
- Equally suitable for conventional or underwater use
- Optional hydraulic ram locks are interchangeable with manual screw ram locks
- Fluid passages are an integral part of forged body
- Wellbore pressure assists in ram closing
- Well pressure cannot reach operating cylinders
- Ram operating and bonnet operating cylinders replaceable on location
- Small number of parts
- Low replacement parts inventory
- Choke and kill line outlets in preventer body, if desired, eliminate need for drilling spool





Drilling Safety Joint

TRI-STATE DRILLING SAFETY JOINT

Tri-State's drill pipe safety joint is installed in the drill string, usually a few stands above the drill collars, to provide a readily releasable back-off point in the event of a stuck string. Some operators prefer to run several safety joints spaced at intervals in the drill pipe string. Drill pipe safety joints are also used as an integral part of fishing tool strings to provide readily releasable connections.

Tri-State drilling safety joints are made from heat treated alloy steel and are stronger than the pipe they are designed to run on.

OPERATION

To release the safety joint, slack off so that the neutral point of the string is at the safety joint location. Apply left hand torque and the joint will unscrew. Maintain the neutral point location while unscrewing. When separated, the friction ring will remain with the upper half of the safety joint.

To reengage in the hole, lower the string until the two sections of the joint make contact. Apply a small amount of weight, make one revolution to the left, then rotate slowly to the right, maintaining a small amount of weight. A sudden torque increase indicates that the joint is fully made up.

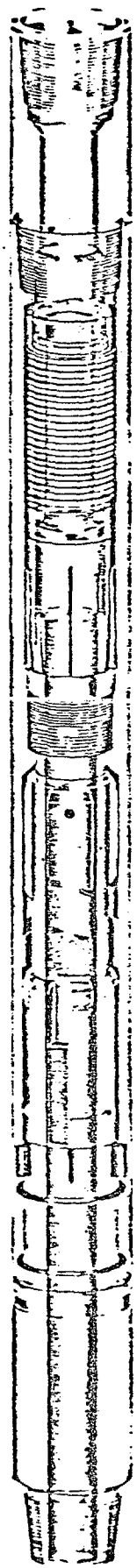
TRI-STATE DRILLING SAFETY JOINT

To order, specify symbol number.

SIZE	1¼" F.J.	1½" W.T.L.H.	1½" F.J.	2⅜" API Reg.	2⅜" API I.F.	2⅜" F.J.	2⅜" E.U.E.	2⅞" API Reg.	2⅞" API I.F.	2⅞" X-Hole	2⅞" E.U.E.	3" E.U.E.	3½" API Reg.	3½" API F.H.	3½" API I.F.	3½" D.S.L.	4" API F.H.	4" H.90	4½" API I.F.	4½" API F.H.	4½" X-Hole	4½" H.90	5" X-Hole	5½" API Reg.
O.D.	2⅝"	2"	1⅝"	3⅜"	3⅜"	2⅞"	2⅝"	3⅜"	4⅜"	3⅞"	3⅜"	4⅜"	4⅜"	4⅜"	4⅜"	3⅞"	5⅜"	5⅜"	6⅜"	5⅜"	6"	6⅜"	6⅜"	6⅜"
I.D.	⅝"	9/16"	⅝"	1"	1⅜"	1"	1"	1⅜"	2⅜"	1⅞"	2 1/16"	2"	1½"	2 1/16"	2 13/16"	1⅞"	2 13/16"	2⅞"	3⅜"	3"	3⅜"	3"	3⅜"	3⅜"
LENGTH	15⅜"	13⅜"	13"	25⅜"	27⅜"	14"	18¼"	28⅜"	29⅜"	27"	31⅜"	31⅜"	32"	35⅜"	34⅜"	32"	34⅜"	35"	36"	35"	36⅜"	36"	36"	35"
APPROXIMATE WEIGHT (LBS.)	17	11	8	47	58	35	37	68	78	73	76	91	84	108	102	78	131	158	175	158	212	190	175	211
SYMBOL NO.	25-100	25-110	25-120	25-130	25-140	25-150	25-160	25-170	25-180	25-190	25-200	25-210	25-220	25-230	25-240	25-250	25-260	25-270	25-280	25-290	25-300	25-310	25-320	25-330
		Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd	Req'd
TOP HALF — Part No. / Weight (LBS.)	1-637 1 / 6	2-110 1 / 5	1-636 1 / 3	1-1363 1 / 23	1-637 1 / 25	1-638 1 / 7	1-663 1 / 7	1-651 1 / 33	1-654 1 / 41	1-668 1 / 32	1-650 1 / 41	1-663 1 / 23	1-672 1 / 44	1-675 1 / 47	1-687 1 / 49	1-646-1 1 / 32	1-679 1 / 53	1-633-1 1 / 80	1-1362 1 / 80	1-633 1 / 80	1-684 1 / 105	1-730 1 / 103	1-1363-4 1 / 80	1-[illegible] 1 / 132
LOWER HALF — Part No. / Weight (LBS.)	1-637-2 1 / 5	2-110-1 1 / 6	1-636-2 1 / 4	1-1363 1 / 23	1-639 1 / 29¼	1-636-2 1 / 7¼	1-663 1 / 9	1-653 1 / 33¼	1-656 1 / 37	1-663 1 / 39	1-667 1 / 44	1-671 1 / 27	1-674 1 / 38	1-677 1 / 60	1-689 1 / 51	1-646-2 1 / 38	1-680 1 / 64	1-633-1 1 / 76	1-1362-2 1 / 80	1-633 1 / 76	1-686 1 / 105	1-732 1 / 92	1-1362-3 1 / 92	[illegible] 1 / 7
O-RINGS — Part No.	6277-33 2	6277-21 2	6277-17 2	6730-3 3	6730-8 2	6730-1 2	6730-5 2	6730-4 2	6277-30 2	6730-7 2	6730-9 2	6730-15 2	6277-37 2	6277-37 2	6277-48 2	6730-2 2	6277-44 2	6277-46 2	6277-60 2	6277-46 2	6277-45 2	6730-21 2	6277-50 2	6277-44 2
FRICTION RING — Part No. / Weight (LBS.)	1-652-1 1 / 8	2-110-2 1 / 6	1-636-1 1 / 6	1-1364 1 / 1	1-638 1 / 1¼	1-638-1 1 / 8	1-664 1 / 1¼	1-652 1 / 1¼	1-655 1 / 1¼	1-667 1 / 1¼	1-661 1 / 2	1-670 1 / 3	1-673 1 / 1¼	1-676 1 / 1¼	1-688 1 / 2	1-667 1 / 1¼	1-678 1 / 2	1-687 1 / 2	1-1362-1 1 / 6	1-687 1 / 2	1-685 1 / 2¼	1-731 1 / 2¼	1-1362-2 1 / 3	1-[illegible] 1 / 3

Recommended Spare Parts:

4 Sets O-Rings
2 Friction Rings



NL HYCALOG BUMPER SAFETY JOINT

A Bumper Sub/Safety Joint that protects the drill stem against the torque and vibration of today's high-speed, deep-hole drilling.

The NL Hycalog Bumper Safety Joint combines the best double-action protection against pipe stuck downhole in a single tool. It's a bumper sub that jars downward automatically and a quick-release safety joint. And it's made only by NL Hycalog.

The Bumper Safety Joint runs directly in the drill string and is sturdy enough to take all the vibration and torque of modern, high-speed, deep-hole driling operations. In fact, it has been used to protect the drill string on many of the world's deepest wells, and has been backed off repeatedly at depths greater than 13,000 feet with only 2½ turns of back torque at the table.

The Bumper Safety Joint offers these advantages over single-action tools in both applications:

- Three heavy-duty drive keys, not threads, transmit torque loads in either direction to prevent wobble and rotational backlash.
- Large through-bore assures full circulation and permits easy passage of wireline tools.
- Special disc-type spring permits high tonnage loads without tripping.
- Tripping mechanism holds solidly until preset tripping weight is placed on tool.
- Bumper Safety Joint maintains unusually high efficiency at maximum drilling depths.

Jars Downward Automatically

An NL Hycalog Bumper Safety Joint run above the drill collar provides immediate action for jarring through bridges, tight spots, or key seats; for jarring loose when drill pipe freezes; or for pulling off bottom with a balled-up bit.

The driller simply slacks off weight equivalent to about 2,000 feet of drill pipe for instant jar-down. A foolproof tripping mechanism automatically trips to allow the weight of the pipe to strike a sharp downward blow.

Raising the string automatically resets the tool for another blow, and jars can be struck as fast as the pipe can be raised and lowered. Frequently, when bridges are stuck while going down the hole, the tool jars the string on through automatically and is instantly reset by the weight of the drill collars—all without any drill pipe manipulation by the driller.

Tripping Weight Can Vary

The weight of 2,000 feet of pipe seems optimum for tripping in most drilling opertions. However, the tripping weight can easily be varied as needed during assembly of the Bumper Safety Joint before it is run in the well.

The remarkable new disc-spring mechanism of the Bumper Safety Joint allows it to be assembled with a preloaded spring compression that holds the entire tool in a set position. It stays locked against any movement while drilling until a downward blow is necessary and the predetermined tripping weight is applied. Thus, when drilling with drill collar weight — as most operators do — or even with substantial added drill pipe weight, there is no movement in the Bumper Safety Joint and no wear on packing or other parts.

Drilling weight, in addition to the drill collars, can be added to the bit whenever needed. If the added weight is enough to trip the tool, the string is simply raised to reset the tool and then the added weight is kept at a point less than the actual tripping load.

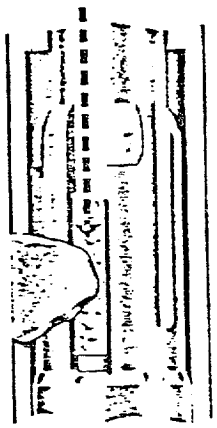
Drilling with the Bumper Safety Joint in the tripped position causes no disconnection or damage to the drill string. But since it does increase wear on the tool, drilling in the tripped position is not advisable.

The Bumper Safety Joint drives through three heavy keys, not through threads, so there is no tightening of the tool under any drilling or torque loads. No amount of drilling torque can affect the

THE BUMPER SAFETY JOINT

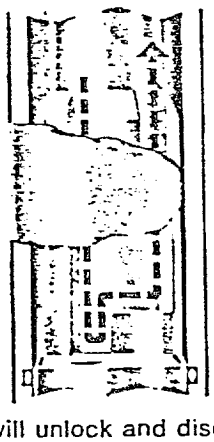
operation of the tool or its positive releasing feature.

Joint Releases Quickly



When jar-down action fails to free stuck drill collars, the NL Hycalog Bumper Safety Joint can be quickly released, recovering all the drill pipe. Release is obtained by tripping the tool and applying reverse torque as the string is slowly raised. One-sixth turn of the drive keys rotates them into an unlocking slot similar to that of the "J" tool so that complete disengagement is obtained as the string is raised. Release and re-engagement can be effected as often as desired.

Only the prescribed sequential action of first tripping the tool and then applying torque as the string



is raised will unlock and disengage the Bumper Safety Joint. After the drive keys have entered the unlocking slot as shown, complete release is effected simply by continuing to raise the drill string.

The upper part of the tool disengages from the lower part, thereby releasing the drill pipe from the stuck drill collars and bit to permit full recovery of the drill pipe. This feature permits not only right or left continuous rotation, but permits rotation in either direction with applied bit pressure up to that of the tripping load.

In recovering stuck drill collars, the upper part of the Bumper Safety Joint can be removed from the recovered drill pipe and made up on the bottom of the fishing string. This permits easy re-engagement with the lower part of the Bumper Safety Joint attached to the top of the fish. This is desirable when operating in tight holes with limited clearances and also assures a tight, leakproof pack-off for any necessary circulating operations.

Aids Twist-Off Recovery

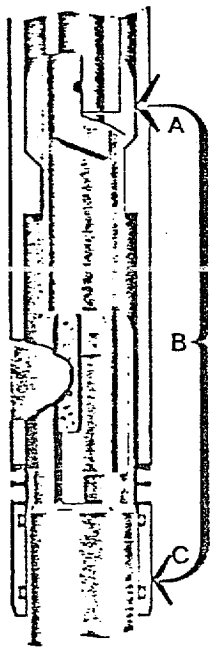
The Bumper Safety Joint simplifies recovery operations when a twist-off occurs at any point above the tool while drilling. If normal fishing procedures fail, weight can be set down on the stuck pipe, the safety joint tripped, and the drill pipe recovered down to and including the upper part of the tool. The drill pipe all recovered, fishing jars can be installed and run down to and immediately above the stuck drill collars for maximum jarring effectiveness.

Provides Strongest Joint

The NL Hycalog Bumper Safety Joint is the strongest joint in the string—not the weakest as in conventional safety joints. In fact, until the definite releasing operations are performed in proper sequence, the Bumper Safety Joint will remain the strongest joint in the entire drill string.

The Bumper Safety Joint prevents wobble and misalignment and maintains close concentricity tolerance at all rotating speeds, under all normal drilling conditions. It is designed so that its two sections, the mandrel and the barrel, triple lock in true alignment at three widely spaced points for maximum stabilization of the tool.

At the upper point of stabilization (A) the taper of the mandrel and barrel centralizes the mandrel within the barrel, and the helically cut angle on the face of the taper causes a slight rotation of the mandrel with respect to the barrel.



Triple stabilization points eliminate wobble and misalignment.

This action wedges the locking keys (B) into tight engagement, preventing any rotational movement between mandrel and barrel. The snug, hardened-and-ground fit between the mandrel's stabilizing shoulder and the barrel sleeve (C) provides a third stabilization point near the bottom of the tool that eliminates any "rocking" action between the barrel and the mandrel. Spaced well apart, they provide ample leverage to hold the tool in precise alignment and rotational concentricity during rough or high-speed drilling operations.

Tubing Swabs Less Cups

Engineering Data-Knuckle Joint Tubing Swab Less Cups

Tubing Size			Part Number	Maximum O.D. Inches	Approximate		Field Thread Size
O.D. Inches	Weight lb/ft	I.D. Inches			Weight Pounds	Length Inches	
2⅜ (60mm) 2⅞ (73mm) 2⅞ (73mm)	4.6 (6.9kg/m)-4.7 (7.0kg/m) 7.9 (11.8kg/m) XLN 7.9 (11.8kg/m) XLN 10.4 (15.5kg/m) DP	1.995 (51mm) 2.323 (59mm) 2.151 (55mm)	33429	1.906 (48mm)	7 (3.2kg)	20½ (521mm)	¾" (19mm) S.R.
2⅞ (73mm) 3½ (89mm) 3½ (89mm)	6.4 (9.5kg/m)-6.5 (9.7kg/m) 9.3 (13.8kg/m) 13.3 (19.8kg/m) DP	2.441 (62mm) 2.992 (76mm) 2.764 (70mm)	34084	2.360 (60mm)	13 (5.9kg)	26 (660mm)	

NOTE: When ordering specify part number, type and tubing size.

Engineering Data-Standard Tubing Swab Less Cups

Tubing Size			Part Number	Maximum Inches			Approximate		Field Thread Size
O.D. Inches	Weight lb/ft	I.D. Inches		With No-Go	Less No-Go	Thimble	Weight Pounds	Length Inches	
2⅜ (60mm)	4.6 (6.9kg/m)-4.7 (7.0kg/m)	1.995 (51mm)	43040	1.906 (48mm)	1.750 (44mm)	1.875 (48mm)	8 (3.6kg)	24 (610mm)	¾" (19mm) S.R.
2⅞ (73mm) 2⅞ (73mm)	10.4 (15.5kg/m) DP 8.8 (13.1kg/m)	2.151 (55mm) 2.250 (57mm)	43045	1.906 (48mm)	1.750 (44mm)	1.875 (48mm)	11 (5.0kg)	28 (711mm)	
2⅞ (73mm)	6.4 (9.5kg/m)-6.5 (9.7kg/m)	2.441 (62mm)	43041	2.375 (60mm)	1.875 (48mm)	2.250 (57mm)			
3½ (89mm) 3½ (89mm)	15.5 (23.1kg/m) DP 13.3 (19.8kg/m) DP	2.602 (66mm) 2.764 (70mm)	43041 43047	2.375 (60mm) 2.375 (60mm)	1.875 (48mm) 1.875 (48mm)	2.260 (57mm) 2.438 (62mm)	12 (5.4kg)	29 (737mm)	
3½ (89mm)	8.5 (12.7kg/m)-10.2 (15.2kg/m)	3.018 (77mm)-2.922 (74mm)	43042	2.375 (60mm)	1.875 (48mm)	2.750 (70mm)	13 (5.9kg)	29 (737mm)	
4 (102mm)	9.5 (14.1kg/m)-11.6 (17.3kg/m)	3.548 (90mm)-3.428 (87mm)	43043	3.250 (83mm)	.....	3.250 (83mm)	23 (10.4kg)	37 (940mm)	⅞" (22mm) S.R.

NOTE: When ordering specify part number, type and tubing size.

Engineering Data — UF Swab Less Cups

Tubing Size			Part Number	Maximum O.D. Inches	Approximate		Field Thread Size
O.D. Inches	Weight lb/ft	I.D. Inches			Weight Pounds	Length Inches	
1.315 (33mm)	1.7 (2.5kg/m)-1.8 (2.7kg/m)	1.049 (27mm)	41329	.875 (22mm)	1¼ (.6kg)	17½ (444mm)	⅜" (10mm) NPT Box
1.660 (42mm)	2.3 (3.4kg/m)-2.4 (3.6kg/m)	1.380 (35mm)	42628	1.125 (29mm)	2½ (1.1kg)	18¼ (464mm)	
1.900 (48mm) 2.000 (51mm) 2.063 (52mm)	2.75 (4.1kg/m)-2.9 (4.3kg/m) 3.4 (5.1kg/m) 3.4 (5.1kg/m)	1.610 (41mm) 1.670 (42mm) 1.750 (44mm)	43539	1.375 (35mm)	4½ (2.0kg)	23 (584mm)	
2⅜ (60mm) 2⅞ (73mm) 2⅞ (73mm)	4.6 (6.9kg/m)-4.7 (7.0kg/m) 7.9 (11.8kg/m) XLN 10.4 (15.5kg/m) DP	1.995 (51mm) 2.323 (59mm) 2.151 (55mm)	42856	1.906 (48mm)	8 (3.6kg)	22 (559mm)	¾" (19mm) S.R.
2⅞ (73mm) 3½ (89mm) 3½ (89mm)	6.4 (9.4kg/m)-6.5 (9.7kg/m) 9.3 (13.8kg/m) 13.3 (19.8kg/m) DP	2.441 (62mm) 2.992 (76mm) 2.764 (70mm)	43007	2.360 (60mm)	10¼ (4.6kg)	27 (686mm)	

NOTE: When ordering specify part number, type and tubing size.

Engineering Data — TA Aluminum Tubing Swab Less Cups

Tubing Size			Sheer Value Pounds	Part Number	Max O.D. Inches		Approximate		Field Thread Size
O.D. Inches	Weight lb/ft	I.D. Inches			With No-Go	Less No-Go	Weight Pounds	Length Inches	
2⅜ (60mm) 2⅞ (73mm)	4.6 (6.9kg/m)-4.7 (7.0kg/m) 10.4 (15.5kg/m) DP	1.995 (51mm) 2.151 (55mm)	8,000 10,000 12,000	55218 55217 55201	1.906 (48mm)	1.719 (44mm)	3½ (1.6kg)	22½ (572mm)	¾" (19mm) S.R.
2⅞ (73mm) 3½ (89mm)	6.4 (9.5kg/m)-6.5 (9.7kg/m) 9.3 (13.8kg/m)	2.441 (62mm) 2.992 (76mm)	8,000 10,000 12,000	55155 55154 55126	2.375 (60mm)	2.125 (54mm)	4½ (2.0kg)	24 (610mm)	

NOTE: When ordering specify part number, type, tubing size and shear value.

Engineering Data — 1.315" (33mm — 4" (102mm) Tubing Swab Cups

Tubing O.D. Inches	1.315 (33mm)	1.660 (42mm)	1.900 (48mm)	2.000 (51mm)	2.063 (52mm)	2⅜ (60mm)	
Weight, lb/ft	1.8 (2.7kg/m)	2.4 (3.6kg/m)	2.9 (4.3kg/m)	3.4 (5.1kg/m)	3.4 (5.1kg/m)	4.7 (7.0kg/m)	4.7* (7.0kg/m)
Tubing I.D., Inches	1.049 (27mm)	1.380 (35mm)	1.610 (41mm)	1.670 (42mm)	1.750 (44mm)	1.995 (51mm)	1.995 (51mm)
Min. R. I.D., Inches	.970 (25mm)	1.300 (33mm)	1.610 (41mm)	1.670 (42mm)	1.700 (43mm)	1.995 (51mm)	1.945 (49mm)
Drift Dia., Inches	.955 (24mm)	1.286 (33mm)	1.516 (39mm)	1.576 (40mm)	1.656 (42mm)	1.901 (48mm)	1.901 (48mm)
Fluid Cap. Bbl./1000'	1.07 (.6 l/m)	1.85 (1.0 l/m)	2.52 (1.3 l/m)	2.71 (1.4 l/m)	2.98 (1.6 l/m)	3.86 (2.0 l/m)	3.86 (2.0 l/m)
TYPE UF CUP Part Number O.D. Inches I.D. Inches	38831 .970 (25mm) .437 (11mm)	38869 1.300 (33mm) .578 (15mm)	29641 1.520 (39mm) .688 (17mm)	45307† 1.650 (42mm) .688 (17mm)	31594 1.694 (43mm) .688 (17mm)	29966‡ 1.975 (50mm) 1.078 (27mm)	37679 1.940 (49mm) 1.078 (27mm)
TYPE MV CUP** Part Number O.D. Inches I.D. Inches				57842 1.735 (44mm) .759 (19mm)	57842 1.735 (44mm) .759 (19mm)	57468 1.992 (51mm) 1.084 (28mm)	
TYPE TA CUP Part Number O.D. Inches I.D. Inches	54510 1.000 (25mm) .430 (11mm)	54511 1.350 (34mm) .569 (14mm)	53669 1.600 (41mm) .759 (19mm)	53690 1.660 (42mm) .759 (19mm)	53691 1.742 (44mm) .759 (19mm)	53409 1.990 (51mm) 1.084 (28mm)	54508 1.950 (50mm) 1.084 (28mm)
TYPE I CUP Part Number O.D. Inches I.D. Inches						14669 1.985 (50mm) 1.094 (28mm)	
TYPE GW CUP Part Number O.D. Inches I.D. Inches						10716 2.000 (51mm) 1.094 (28mm)	
TYPE HPR CUP Steel Bushing Part Number O.D. Inches I.D. Inches						64173 1.940 (49mm) 1.095 (28mm)	65530 1.923 (49mm) 1.095 (28mm)
TYPE HPR CUP Aluminum Bush. Part Number O.D. Inches I.D. Inches						64174 1.940 (49mm) 1.095 (28mm)	65531 1.923 (49mm) 1.095 (28mm)

Tubing O.D. Inches	2⅞ (73mm)				3½ (89mm)				4 (102mm)	
Weight, lb/ft	6.5 (9.7kg/m)	6.5* (9.7kg/m)	8.8 (13.1kg/m)	10.4 (15.4kg/m) DP	8.5 (12.7kg/m)-10.2 (15.2kg/m)	9.3 (13.8kg/m)	13.3 (19.8kg/m) DP	15.5 (23.1kg/m) DP	9.5 (14.1kg/m)	11 (16.4kg/m)-11.6 (17.3kg/m)
Tubing I.D., Inches	2.441 (62mm)	2.441 (62mm)	2.250 (57mm)	3.018 (77mm)-(55mm)	2.922 (74mm)	2.922 (74mm)	2.764 (70mm)	2.602 (66mm)	3.548 (90mm)	3.476 (88mm)-3.428 (87mm)
Min. R. I.D., Inches	2.441 (62mm)	2.375 (60mm)	2.250 (57mm)	2.151 (55mm)	3.018 (77mm)-2.922 (74mm)	2.922 (74mm)	2.764 (70mm)	2.602 (66mm)	3.548 (90mm)	(56mm)
Drift Dia., Inches	2.347 (60mm)	2.347 (60mm)	2.156 (55mm)	1.963 (50mm)	2.893 (73mm)-2.797 (71mm)	2.867 (73mm)	2.375 (60mm)	2.375 (60mm)	3.423 (87mm)	2.303 (84mm
Fluid Cap. Bbl./1000'	5.80 (3.0 l/m)	5.80 (3.0 l/m)	4.92 (2.6 l/m)	4.50 (2.3 l/m)	8.84 (4.6 l/m)-8.29 (4.3 l/m)	8.70 (4.5 l/m)	7.42 (3.9 l/m)	6.59 (3.4 l/m)	12.25 (6.4 l/m)	11.40 (5.9 l/m)
TYPE UF CUP Part Number O.D. Inches I.D. Inches	32582 2.378 (60mm) 1.350 (34mm)	37723 2.370 (60mm) 1.350 (34mm)		34950 2.090 (53mm) 1.078 (27mm)	36353 2.935 (75mm) 1.350 (34mm)	1.350 (34mm)	36357 2.720 (69mm) 1.350 (34mm)			
TYPE MV CUP** Part Number O.D. Inches I.D. Inches	57469 2.431 (62mm) 1.375 (35mm)				57839 2.985 (76mm) 1.375 (35mm)					
TYPE TA CUP Part Number O.D. Inches I.D. Inches	53494 2.431 (62mm) 1.375 (35mm)	54507 2.385 (61mm) 1.375 (35mm)		55235 2.111 (54mm) 1.084 (28mm)	53692 2.940 (75mm) 1.375 (35mm)	53692 2.940 (75mm) 1.375 (35mm)				
TYPE I CUP Part Number O.D. Inches I.D. Inches	14772 2.420 (61mm) 1.344 (34mm)		16711 2.215 (56mm) 1.094 (28mm)	17643 2.125 (54mm) 1.094 (28mm)		15684 2.972 (75mm) 1.344 (34mm)	16435 2.724 (69mm) 1.344 (34mm)	16781 2.552 (65mm) 1.344 (34mm)		
TYPE GW CUP Part Number O.D. Inches I.D. Inches	10047 2.490 (63mm) 1.344 (34mm)		12993 2.285 (58mm) 1.094 (28mm)		14960 3.020 (77mm) 1.344 (34mm)		14082 2.770 (70mm) 1.344 (34mm)		14034 3.590 (91mm) 1.938 (49mm)	14036 3.526 (90mm) 1.938 (49mm)
TYPE HPR CUP Steel Bushing Part Number O.D. Inches I.D. Inches	64176 2.395 (61mm) 1.370 (35mm)	65532 2.355 (60mm) 1.370 (35mm)								
TYPE HPR CUP Aluminum Bush. Part Number O.D. Inches I.D. Inches	64175 2.395 (61mm) 1.370 (35mm)	65533 2.355 (60mm) 1.370 (35mm)								

The Slim Hole Production-Injection Packer is an inflatable packer designed to be run in casing or open hole. It can be used as a permanent or temporary packer for treating, testing, production, injection or numerous other zone isolation applications. Tool sizes are available for 2-7/8" – 3-1/2" pipe and 2-1/4" – 4-3/4" open hole. Tools for 1-1/2" – 2-3/8" pipe and 1-5/8" – 2-1/4" open hole are available on special order.

Pressure ratings of the packer vary with open hole or casing size and are shown in the Packer Selection Curves.

DESCRIPTION

This packer utilizes the inflatable packer element. This element securely anchors in the well bore of either open hole or casing, conforming to the most irregular of open holes within its expansion limits. External pressure against the end area of the inflated element transmits to the smaller end area inside the element so that the pressure of the inflating fluid always exceeds the external pressure. This prevents communication by the packer even though the external, or differential pressure, exceeds the inflation pressure. The PIP withstands high differential pressures from above or below without the use of tubing weight, tension, or auxiliary aids.

The inflation fluid under pressure is maintained inside the element by a spring-loaded, triple seal check valve. This valve, besides having a superior seal, affords an increased passage area for the inflation of the packer element, thus reducing the possibility of plugging caused by scale, corrosion, or sand. Straight circulation, reverse circulation, or "spotting" of fluid can be performed before setting the packer. The greater expansion of the inflatable element make the Production-Injection Packer ideally suited for numerous difficult jobs.

CONSTRUCTION

1. Poppet Type Check Valve – Spring and hydraulic actuated check valve to retain packer inflation.
2. Releasing Mechanism – Left hand Acme thread sealed from well fluids. Tubing turns to right to unthread and release packer.
3. Inflation of packer takes place when pressure is applied to tubing. Packer element deflates when releasing thread is disengaged and upstrain is applied. End of mandrel moves inside bottom sub.
4. Shear plug in bottom shears out at predetermined pressure and opens the tool below the packer.
5. Element sizes 1-7/16" thru 1-7/8" are run on tool body of 1-7/16" OD and .459 ID. Element sizes 2-1/8" thru 2-3/4" are run on tool body of 2-1/8" OD and .747 ID.

OPTIONS

Packer Element: The inflatable packer element is available in either a plain rubber or grit impregnated cover. The plain cover is used where the anchoring effect of the grit type element is not needed. Where numerous settings are to be made the plain cover affords a longer life. The grit impregnated cover is used for casing and hard rock cored hole applications.

Shear Plug: Two types of shear plugs are available for the PIP tool – solid and ball-and-seat. The solid plug is used where it is necessary to blank the tubing off completely when running in and the ball-and-seat plug is used where it is necessary to circulate or spot fluid prior to setting the packer. The ball and seat plug is standard on this tool.

Adapters: By the use of adapters several size elements can be adapted to the same tool body. As an example the same size tool body will accept the 1-7/16", 1-5/8", 1-3/4" and 1-7/8" elements with the appropriate adapters.

OPERATION

The Production-Injection Packer is set by applying pressure to the tubing. Fluid enters the packer element through a check valve and inflates the element against the well bore. A shear plug at the bottom of the mandrel is sheared by tubing pressure or a bar run on wireline to open the tool below. Right hand rotation releases the packer for retrieving.

Running

1. Check releasing thread for free turning to right. This is accomplished by holding a back-up wrench on the Releasing Sub (18) and turning the Top Sub (19) to the right, then turn back to left until thread is fully engaged.
2. Check shear plug in bottom for proper shear pin size.
3. When the .459 ID mandrel is run with ball seat shear plug, the 5/8" ball must be in place between the shear plug and end of mandrel.
4. If tubing has loose scale or dirt inside, it should be cleaned before running tool, otherwise, mandrel could plug.
5. Fill first two tubing joints above packer with clean fluid (free of solids), if solid shear plug is used.
6. Run tool in well at moderate rate and do not allow to rotate.
7. Do not do any filling of tubing until setting depth is reached.

Setting Packer

A. Solid Shear Plug and Tools with .459 ID mandrel with ball-and-seat shear plug.
   1. Fill tubing at moderate pump rate. Stop pump at first indication of pressure.
   2. Resume pumping slowly to shear pin pressure.
   3. When shear pin shears, shear plug is blown out and tool is open below packer. Pull against packer to check packer set.

B. Ball-and-Seat Shear Plug
   1. The ball-and-seat shear plug is run when it is desireable to spot fluids or to be able to wash around packer.
   2. Drop ball in tubing when packer is at setting depth. Since ball is made of drillable material and is light, it may be desireable to circulate the ball down. Caution should be taken to avoid pressuring to more than 500 psi when ball seats. When ball seats, use same setting procedure as described above for solid shear plug.

Unsetting Packer

1. Pull up about 2000 lbs. above weight of tubing and turn to right. Six (6) rounds at tool should unthread releasing connection.
2. Tool has 4" unsetting stroke. Packer deflates when mandrel end is moved inside bottom sub.

DRESSING INSTRUCTIONS

Remove Plug Retaining Bottom Sub (13). (When putting wrench or tong on inflatable packer element, place wrench or tong about 2" from end of element.) Remove Set Screws (7). Remove O-Rings (1), (5) and (3). Clean all parts

thoroughly. Check O-Rings for cuts or flatness and replace with new O-Rings where needed. Replace O-Rings in Plug Retaining Bottom Sub (13). Insert Ball Seat Shear Plug (20)* or Solid Shear Plug (21) being sure to align shear pin holes in sub with holes in shear plug. Place the desired Shear Pin (6) in the shear pin hole and replace Set Screws (7).

Remove Inflatable Packer Element (22) from Packer Adapter Sub (8). Clean inside of packer element and inspect innertube for cuts. Check that tube is properly secured at each end with brass expansion rings. Inspect outer cover. If innertube and outer cover are in good condition the packer element can usually be run again. Small cuts or breaks in the outer cover usually do not impair the performance of the element. If outer cover is damaged to the extent that wire reinforcing is exposed, the packer element should be replaced.

Remove Top Sub (19) from Upper Mandrel (16). Dress smooth any wrench marks left on Upper Mandrel (16). Remove O-Rings (2) and check for cuts or flatness. Clean Top Sub (19). Replace O-Rings (2).

Remove Bumper Sub Housing (17) from Check Valve Sub (14). This leaves Bumper Sub Housing (17) and Releasing Sub (18) together. Remove these parts from Upper Mandrel (16). Upper Mandrel (16), Mandrel Connector (15) and Mandrel (12) can now be removed. Remove Check Valve Sub (14) from Packer Adapter Sub (8). Remove Poppet (9) from Check Valve Sub (14) taking care not to damage Poppet (9). Remove O-Ring (3) from Poppet, inspect and replace O-Ring (3) after cleaning Poppet (9). Inspect rubber seal end of Poppet (9). If damaged, replace poppet. Remove O-Rings (3) and (4) from Check Valve Sub (14). Check for cuts and flatness, clean and replace in check valve sub. Remove Spring (10). Check length of Spring (10)**.

Remove O-Rings (1) from Packer Adapter Sub (8). Check for cuts or flatness and replace after cleaning Packer Adapter Sub (8).

Check all parts for burrs and file smooth.

ASSEMBLY INSTRUCTIONS

Wipe a thin coat of oil on Mandrel (12) and slide mandrel thru Check Valve Sub (14). Recoat Mandrel (12) and slide Poppet (9) up on mandrel. Slide Spring (10) on Mandrel (12) behind Poppet (9). Put Packer Adapter Sub (8) on Mandrel (12) and make up on Check Valve Sub (14). Pressure of spring must be overcome in order to thread these two parts together. Slide Element (22) on Mandrel (12) and make up on Packer Adapter Sub (8) being sure O-Rings (1) do not squeeze out and pinch between packer end and sub.

Slide the Bumper Sub Housing (17) and Releasing Sub (18) over the Upper Mandrel (16) and make up on the Check Valve Sub (14). Make up Top Sub (19) on the Upper Mandrel (16). Make up Top Sub (19) in Releasing Sub (18). NOTE: This is a left hand thread and must be made up by turning to the left. Since this is the releasing thread for releasing the packer in the well, this thread should be shouldered and barely tightened to avoid difficulty in unthreading down hole.

Turn tool in vertical bottom up position and fill area between Mandrel (12) and inside of Element (22) with fluid to bleed element of air. With tool still in vertical bottom up position, proceed to next step.

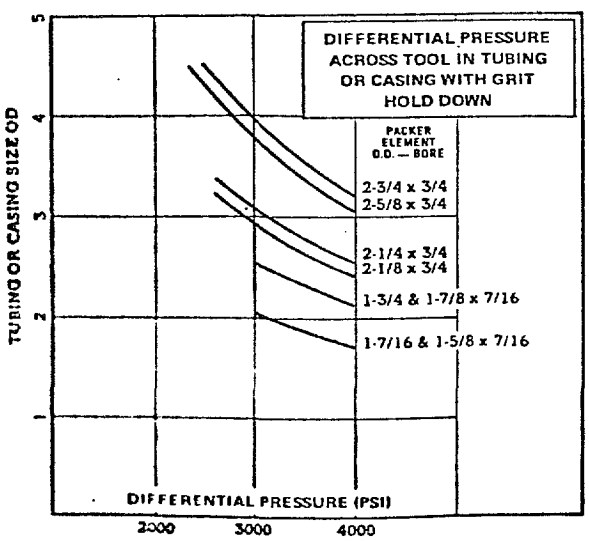
Coat portion of Mandrel (12) extending out of Element (22) with oil and slide Plug Retaining Bottom Sub (13) on mandrel and make up in element being sure again that O-Rings (1) do not squeeze out and pinch between packer end and sub.

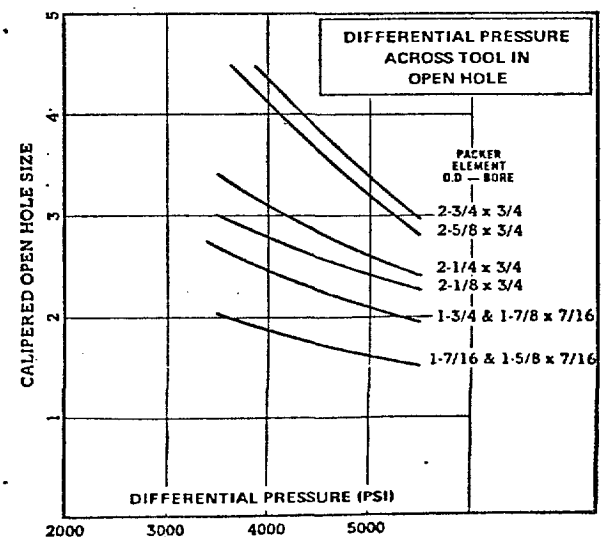
* When using Ball Seat Shear Plug (20) in the 1-1/2" to 1-7/8" packer sizes, the 5/8" ball should be placed in the Plug Retaining Bottom Sub (13) before the shear plug is inserted. Also, the ball cannot be dropped from the surface since it will not pass thru the .459 inside diameter of the mandrel.

** Minimum length of Spring (10) should be as below. Shorter spring length should be replaced.

1-1/2" thru 1-7/8" – 2-1/16"
2-1/8" thru 2-3/4" – 3-1/4"

PACKER SELECTION CURVES





INFLATING PRESSURES

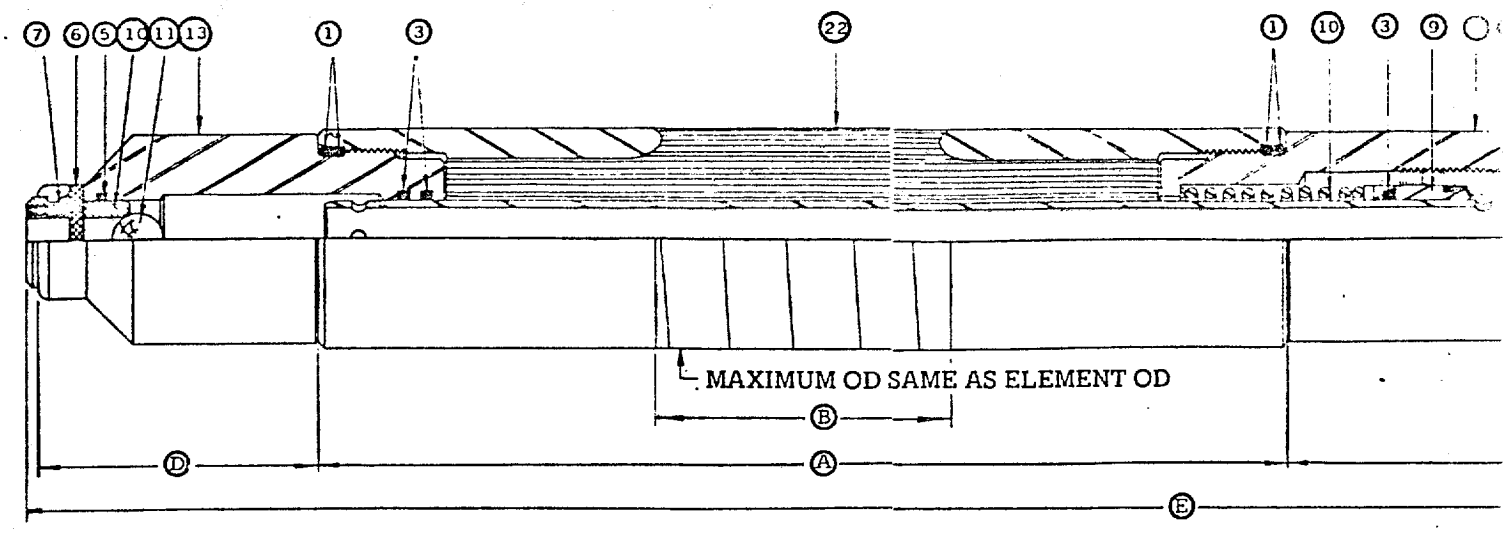
Inflating pressure should be approximately 1500 psi when setting packers in open hole and approximately 2000 psi when setting packers in casing. Inflating pressure is the difference between the tubing pressure at the packer and the annulus (casing) pressure at the packer while the packer is being set. Lower inflation pressures than those shown can be used under certain conditions.

DIFFERENTIAL PRESSURES

Differential pressure is the difference between the pressure below the packer and the pressure above the packer while injecting, producing, treating, etc. This limiting pressure varies with packer expansion, as shown in the above graphs for open hole and casing, and should be calculated prior to selecting a packer size.

These limiting pressures should be used as a guide only. Applications calling for higher differential pressure than those shown should be discussed with a Lynes representative.

LYNES SLIM HOLE PRODUCTION-INJECTION PACKER
PRODUCT NO. 300-06



BASED ON T/M DWG.

SHEAR PIN SELECTION CHART

COMMODITY NUMBER	DIA.	SHEAR PRESSURE
04-00471-10	.093	815 PSI
04-00471-12	.125	1450 PSI
04-00471-13	.140	1835 PSI
04-00471-14	.156	2270 PSI (Std)
04-00471-15	.187	3260 PSI

Shear Pin Length is 1-7/16 + 0, −1/32

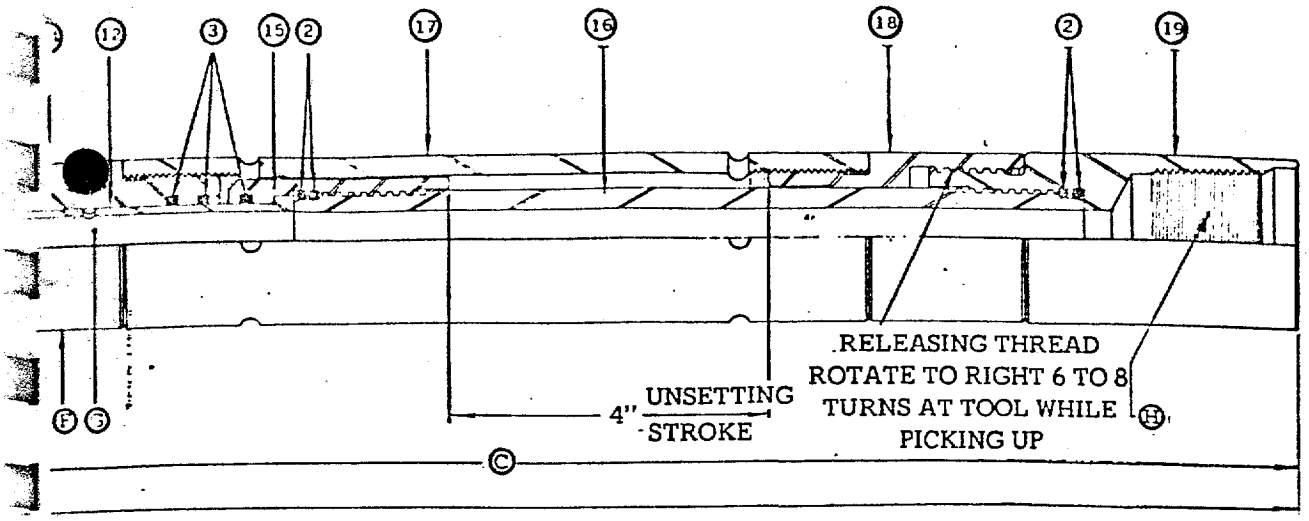
SPECIFICATIONS

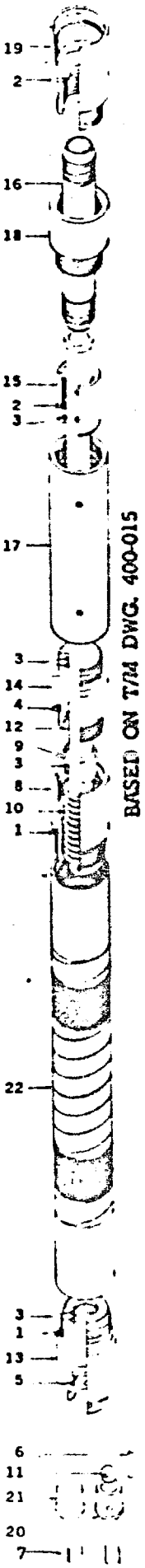
TOOL SIZE					
Element OD (Inches)	Element Length (Inches)	Mandrel Bore (Inches)	Seal Length (Inches)	Tool Length (Inches)	For Calipered Open Hole, Tubing, or Casing Sizes*
1-7/16	17	0.459	6	40	1.900
1-7/16	36	0.459	25	59	1-5/8 – 2-1/2
1-5/8	17	0.459	6	40	2-1/16
1-5/8	48	0.459	36	71	1-3/4 – 2-1/2
1-3/4	17	0.459	6	40	2-3/8
1-3/4	48	0.459	36	71	1-7/8 – 2-1/2
1-7/8	17	0.459	6	40	2-3/8
1-7/8	48	0.459	36	71	2 – 2-3/4
2-1/8	23	3/4	12	47	2-7/8
2-1/8	48	3/4	36	72	2-1/4 – 3-1/2
2-1/4	23	3/4	12	47	2-7/8
2-1/4	48	3/4	36	72	2-1/2 – 3-3/4
2-5/8	23	3/4	12	47	3-1/2
2-5/8	48	3/4	36	72	3 – 4-1/2
2-3/4	23	3/4	12	47	3-1/2 – 4
2-3/4	48	3/4	36	72	3-1/2 – 4-3/4

NOTE: 1-7/16" thru 1-7/8" tools available only on special order.

* 17" and 23" show tubing and casing size. 36" and 48" show open hole range.

ITEM NO.	PART NAME	NO. REQ'D.	1-7/16 x 7/16	1-5/8
1	O-Ring – Packer End Seal	See Size	WW-B120-H40 [2]	WW-B21
2	O-Ring – Upper Mandrel End Seal	See Size		
3	O-Ring – Mandrel OD Seal	6		
4	O-Ring – on Check Valve Sub	2		
5	O-Ring – Shear Plug OD Seal	1		
6	Shear Pin – See Selection Chart for Sizes	1		
7	No. 6-32 NC Setscrew x 5/16 Long	1		
8	Packer Adapter Sub	1	04-00134-00	04-00
9	Poppet	1		
10	Spring	1		
11	5/8 Dia. Aluminum Ball	1		
12	Mandrel – PIP 17 or 23	1		
	Mandrel – PIP 36 or 48		04-00103-00	
13	Plug Retaining Bottom Sub	1	04-00117-00	04-0
14	Check Valve Sub	1		
15	Mandrel Connector	1		
16	Upper Mandrel	1		
17	Bumper Sub Housing	1		
18	Releasing Sub	1		
19	Top Sub	1		Item 16
20	Ball Seat Shear Plug – 1" Dia. (Std)	1		
21	Solid Shear Plug – 1" Dia. (Optional)	1		
22	Inflat. Pkr. Element – 17 or 23" Plain	1	04-04001-00	04-0
	Inflat. Pkr. Element – 36 or 48" Plain		04-04002-00	04-5
	Inflat. Pkr. Element – 17 or 23" Grit		04-04231-00	04-0
	Inflat. Pkr. Element – 36 or 48" Grit		04-04232-00	04-0





BASED ON T/M DWG. 400-015

0-006

DIMENSIONAL DATA

SIZES ELEM. OD LONG ID	A ELEMENT LENGTH	B SEAL LENGTH	C ABOVE ELEMENT	D BELOW ELEMENT	E LENGTH OVERALL
1-7/16 x 17 x 7/16	17-1/4	6	19-9/16 1.050 OD Tbg. – 10 Rd. Non Upset	3-1/16	40-1/8
1-7/16 x 36 x 7/16	36	25			58-7/8
1-5/8 x 17 x 7/16	17-1/4	6			40-1/8
1-5/8 x 48 x 7/16	48	37			70-7/8
1-3/4 x 17 x 7/16	17-1/4	6			40-1/8
1-3/4 x 48 x 7/16	48	37			70-7/8
1-7/8 x 17 x 7/16	17-1/4	6			40-1/8
1-7/8 x 48 x 7/16	48	37			70-7/8
2-1/8 x 23 x 3/4	23-1/4	12	19-1/2 1.900 OD Tbg. – 10 Rd. Non Upset	3-5/8	46-5/8
2-1/8 x 48 x 3/4	48	36-3/4			71-3/8
2-1/4 x 23 x 3/4	23-1/4	12			46-5/8
2-1/4 x 48 x 3/4	48	36-3/4			71-3/8
2-5/8 x 23 x 3/4	23-1/4	12			46-5/8
2-5/8 x 48 x 3/4	48	36-3/4			71-3/8
2-3/4 x 23 x 3/4	23-1/4	12			46-5/8
2-3/4 x 48 x 3/4	48	36-3/4			71-3/8

H = 1.9 OD NON UPSET 10 ROUND

PARTS LIST

SIZE (PIP OD x ID) & COMMODITY NUMBER						
7/16	1-3/4 x 7/16	1-7/8 x 7/16	2-1/8 x 3/4	2-1/4 x 3/4	2-5/8 x 3/4	2-3/4 x 3/4
[2]	WW-B216-H40 [2]	WW-B217-H40 [2]	WW-B129-H40 [4]	WW-B131-H40 [4]	WW-B226-H40 [4]	
B116-H40 [2]				WW-B215-H40 [4]		
W-B114-H40			WW-B213-H40			
WW-B120-H40			WW-B129-H40			
WW-B120-H40						
04-00471-14						
WW-G506-0B0						
5-00	04-00136-00	04-00137-00	04-00138-00	04-00139-00	04-00140-00	
04-00093-00			04-00092-00			
4-00098-00			04-00097-00			
04-00252-00						
04-00102-00			04-00105-00			
04-00104-00			04-00106-00			
04-0019-00		04-00120-00	04-00121-00	04-00122-00	04-00123-00	
4-00058-00			04-00087-00			
04-00074-00			04-00073-00			
04-00070-00			04-00069-00			
4-00063-00			04-00062-00			
4-00080-00			04-00079-00			
Includes Top Sub on these sizes			04-00066-00			
04-00052-00						
04-00048-00						
0	04-04011-00	04-04016-00	04-04021-00	04-04026-00	04-04031-00	04-04036-00
2-00	04-04012-00	04-04017-00	04-04022-00	04-04027-00	04-04032-00	04-04037-00
6-00	04-04241-00	04-04246-00	04-04251-00	04-04256-00	04-04261-00	04-04266-00
0	04-04242-00	04-04247-00	04-04252-00	04-04257-00	04-04262-00	04-04267-00

JOY MANUFACTURING COMPANY
707 BOYD BOULEVARD
P. O. DRAWER 489
LaPORTE, INDIANA 46350
Phone: (219) 362-2191

September 28, 1979

Mr. Ralph Nelms
Science Applications
16625 West 51st Ave.
Golden, Colorado 80401

Dear Mr. Nelms:

This letter will confirm our recent conversations on the permeability testing for Asarco for their Poston Butte Project near Florence, Arizona.

1. Our water pump, which we call a "John Bean 435" is actually Model W1122B-1. It has been rerated recently for intermittent pressures to 1000 p.s.i. FMC advises, because of the ball valve characteristics, that you would probably get no more than 40 g.p.m. (rated @ 37 g.p.m.) by running above the rated r.p.m.

2. I suggested you may want to consider setting your packer at 1000 p.s.i. rather than 1200 p.s.i. Also, to accommodate our piping and hoses, we have not pressured above 1000 p.s.i. Higher pressure may begin to give us trouble.

3. As I understand it, your packer probably has a two inch pipe thread on the bottom and discharges a plug from the bottom each time you set it. You should make a device to catch the plug so we don't leave three plugs in the bottom of the hole after the three tests. This would make any additional drilling possible, if desired, without having to drill out the plugs.

4. You should plan to provide a sub for us to thread in Tucson to connect your packer to our drill rods.

This sub SAI provides

5. We have run packers in the Southeast on both the 2-3/8 EUE oilfield tubing provided for us and our own wire line rods. If using our own rods, we would plan to put teflon tape on the joints to ensure the best seal. The taping may be good for more than one packer test.

6. Water leakage at the joints would be checked by (a) pouring the open rods full of water to the top to look for a fall in level, and (b) pressuring to 700 to 800 p.s.i. and observing the flow rate (hopefully none) on the flowmeter.

7. As I understand it, each of the three tests in each of two holes requires a round trip into the drill hole.

8. By way of additional information, the diesel engine driving the pump produces 37 h.p. @ 1800 r.p.m. It drives through a transmission with 6.4:1, 3.09:1, 1.69:1 and 1.00:1 reduction ratios. There is a further chain drive reduction of 2.647 between the transmission and the input shaft to the pump.

This covers everything we can think of at this time. If Joy is awarded this work, we will need to collaborate further in more detail to ensure preparedness prior to testing. *

Yours truly,

JOY MANUFACTURING COMPANY

J. H. Koontz, General Manager
Drill Division
Joy Machinery Company

JHK:js

cc: Norm Whaley
Asarco Incorporated

Bean®

Triplex piston pumps give you...

More than eighty-five years experience in designing and building pumps for a broad spectrum of applications assures you of field tested, proven and dependable performance. New material designs are constantly tested to provide better and longer lasting components. FMC has been an innovator, introducing and pioneering many of the features that are standard on competitive pumps today.

long life

Durable, high quality materials; totally enclosed working parts; constant three way oil bath lubrication; ceramic cylinders; hardened stainless steel valves and seats; precision bearing alignment are but a few of the major features that provide the longer service life of FMC Triplex pumps.

low maintenance costs

Standard, interchangeable parts used in FMC Triplex pumps reduce the number of parts to be stocked, saves downtime on equipment, reduces installation costs and total parts investment. Easy servicing with ordinary tools helps to reduce costs. Packing can be quickly changed without disturbing suction and discharge piping.

compact design

Space saving horizontal design with internal gear reduction requires minimum space for installation. Small, easily obtained, stock sheaves permit use with standard rpm motors. Fluid end parts easily accessible for servicing as required.

Specifications

Pump model	Gal. per pinion rev.	gpm	at pinion rpm	crankshaft rpm	pressure rating int.	pressure rating cont.
L0914B-1	.0195	12	616	171	1000	900
L0918B	.0322	20	621	172	700	600
L0914D-1	.0195	17	872	242	1000	900
L0918D	.0322	28	870	241	700	600
W1118B-1	.0394	25	635	176	1000	1000
W1122B-1	.0589	37	628	174	1000	800
W1118D-1	.0394	35	890	247	1000	1000
W1122D-1	.0589	50	850	236	1000	800
[illegible]	[illegible]	57	[illegible]	[illegible]	[illegible]	1000
[illegible]	[illegible]	40	870	[illegible]	1000	800
[illegible]	[illegible]	50	1270	[illegible]	1000	1000
[illegible]	.0589	75	1273	354	1000	800

Ratings are for pinion shaft as shown turning counter-clockwise, either direct drive or with belt pull directly away from fluid end in 'direction X' as shown in the drawing. For other drive arrangements, consult factory.

B - ball valves

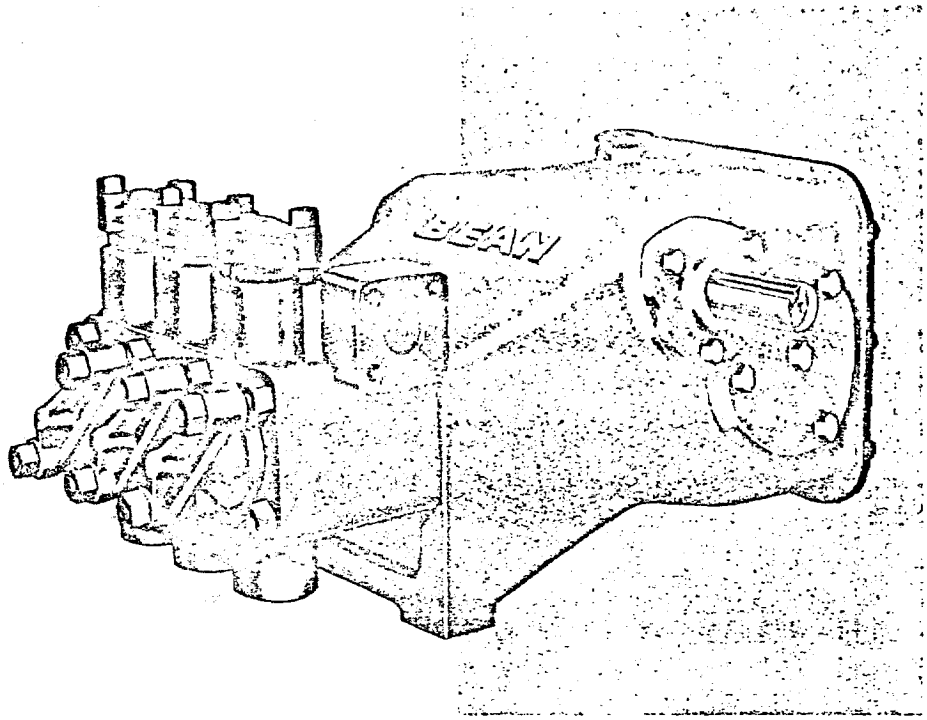
D - disc valves

GPM - based upon 100% volumetric efficiency. Use 95% of gpm shown to obtain the actual delivered gpm.

Triplex piston pumps

Models L-09, W-11

deliver up to
50 gpm
at pressures
up to 1000 psi



Special drive end features insure longer, dependable performance of FMC pumps

Enclosed design, alloy grey iron, internally ribbed for added strength. Crosshead guides machined as integral part of crankcase for precision alignment. Crankcase designed with reservoirs for positive lubrication. Large end cover for easy access. Oil drains on both sides of crankcase.

Crankshaft

One piece, drop-forged steel gives rugged strength. Has a close grain structure with induction hardened bearing throws for long, uniform wear.

Connecting rods

Cupped and channelled oil reservoirs within the forged steel rods provide extra lubrication to the bearing surfaces. Split-cap design permits easy installation on crankshaft.

Crossheads

Cylindrical design with a large surface area floats piston cup in cylinder for proper alignment and extended packing life. Cast iron crossheads and stainless steel crosshead rods are ground as a unit to eliminate assembly problems. Crosshead rods operate through spring-loaded oil seals to prevent contamination of crankcase lubricant.

Bearings

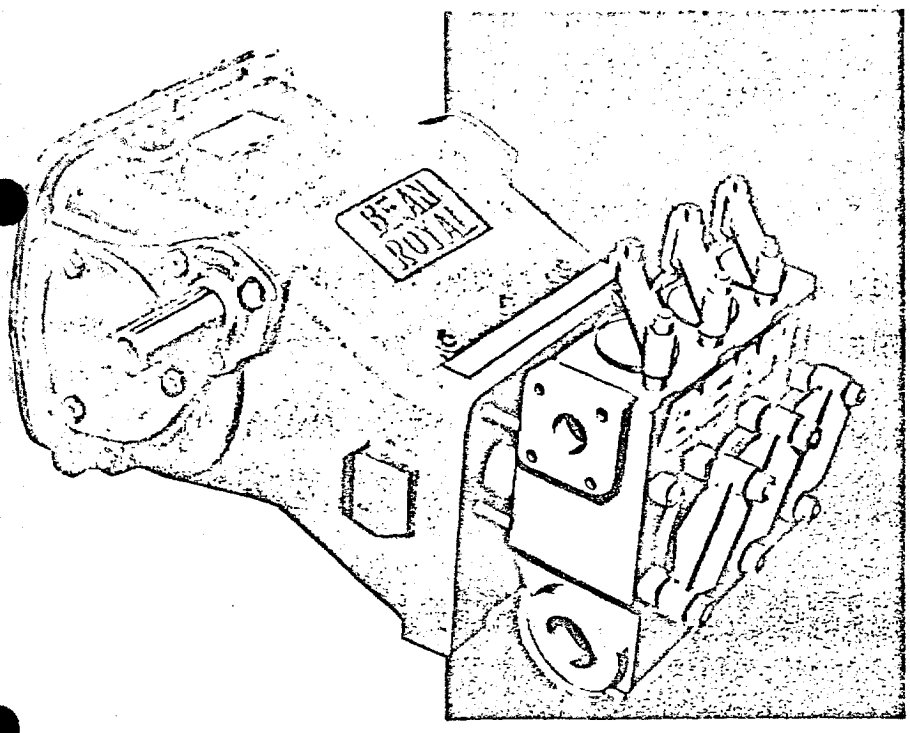
High load capacity, tapered roller bearings on crankshaft, precision ball bearings on countershaft and steel backed babbitt automotive type bearings on the connecting rods ride on film of oil giving long trouble-free life.

Crankshaft ring gear

Forged steel, precision machined shrink-fit for positive alignment and concentricity. Induction hardened gear teeth for uniform hardness and maximum wear resistance. Helical design offers more contact surface, better load distribution and quieter operation.

One-piece drop forged steel for added strength and minimum deflection. Induction hardened gear teeth are integral with the shaft. Can be reversed with the crankshaft for either right or lefthand drive.

Built-in gear reduction produces constant oil spray over all moving parts and at the point of gear contact. A three way lubrication system - spray, dip and splash - thoroughly lubricates the drive end. (For very low rpm operation, auxillary lubrication is required and available.)



Triplex
Piston pumps

Model L-11

deliver up to
75 gpm
at pressures
up to 1200 psi

Special fluid end features

Alloy cast iron construction with two discharge outlets, threaded and flanged for maximum connection flexibility. In-line flow reduces channel erosion and increases volumetric efficiency due to lower friction loss. Packing can be changed without disturbing suction or discharge piping.

Piston cups

Self-expanding piston cups seal firmly on each pressure stroke and maintain maximum suction lift. Easily replaced when worn. Reinforced neoprene is standard, Buna-N, natural rubber and thiokol cups available for special applications.

Valves

Two types of valves are available. Stainless steel, spring-loaded disc valves give maximum volume and quiet operation with free flowing liquid. When viscous materials or liquids with small suspended solids are pumped, the stainless ball valves provide the system sealing for those difficult-to-pump fluids.

Easily removed bar clamps release slip-in type valve and cylinder covers for quick inspection or maintenance. Pump can be repacked in minutes without removing valve chamber or disconnecting piping. Parts are standardized and interchangeable with various models.

Cylinders

Ceramic cylinders are standard on FMC pumps. The tough cylinders are diamond honed for uniform concentricity and exacting smoothness. Where thermal shock or mechanical damage is a problem, steel backed ceramic cylinders are available for most models.

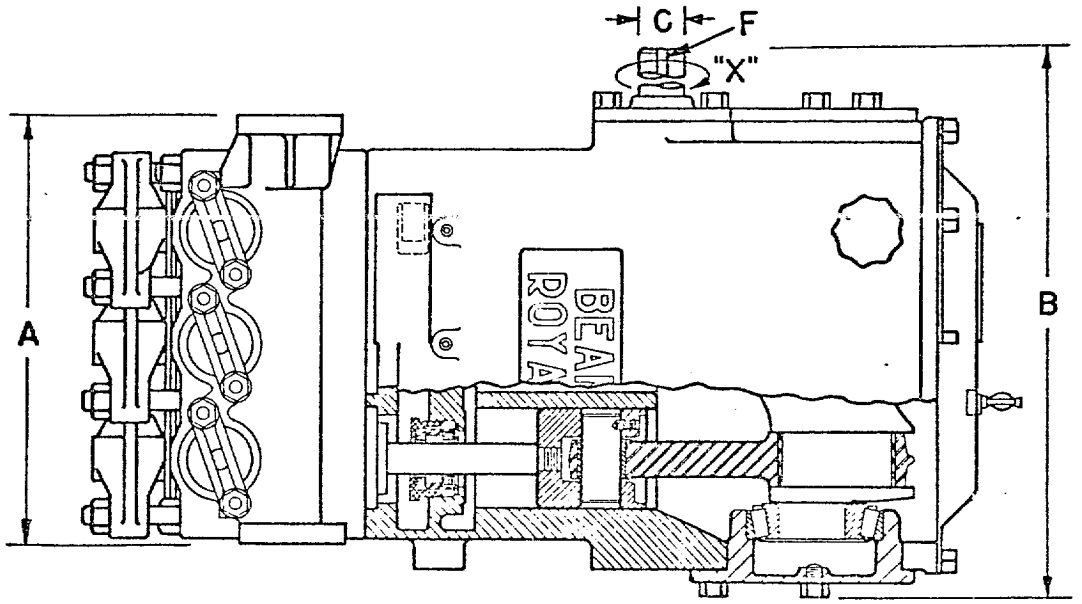
FMC reserves the right to incorporate changes in design and materials, affecting product improvements, without notice or obligation.

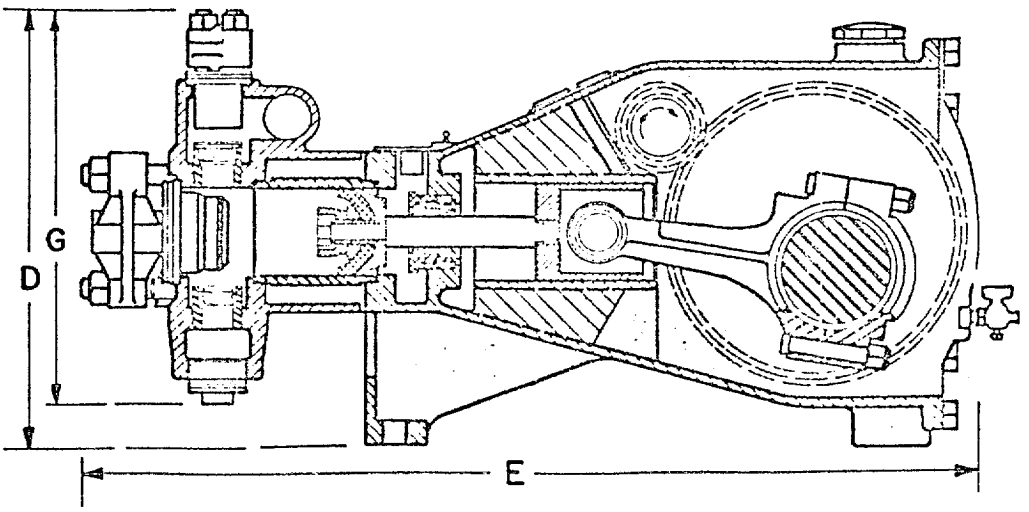
Triplex piston pumps

Models L-09, W-11 and L-11

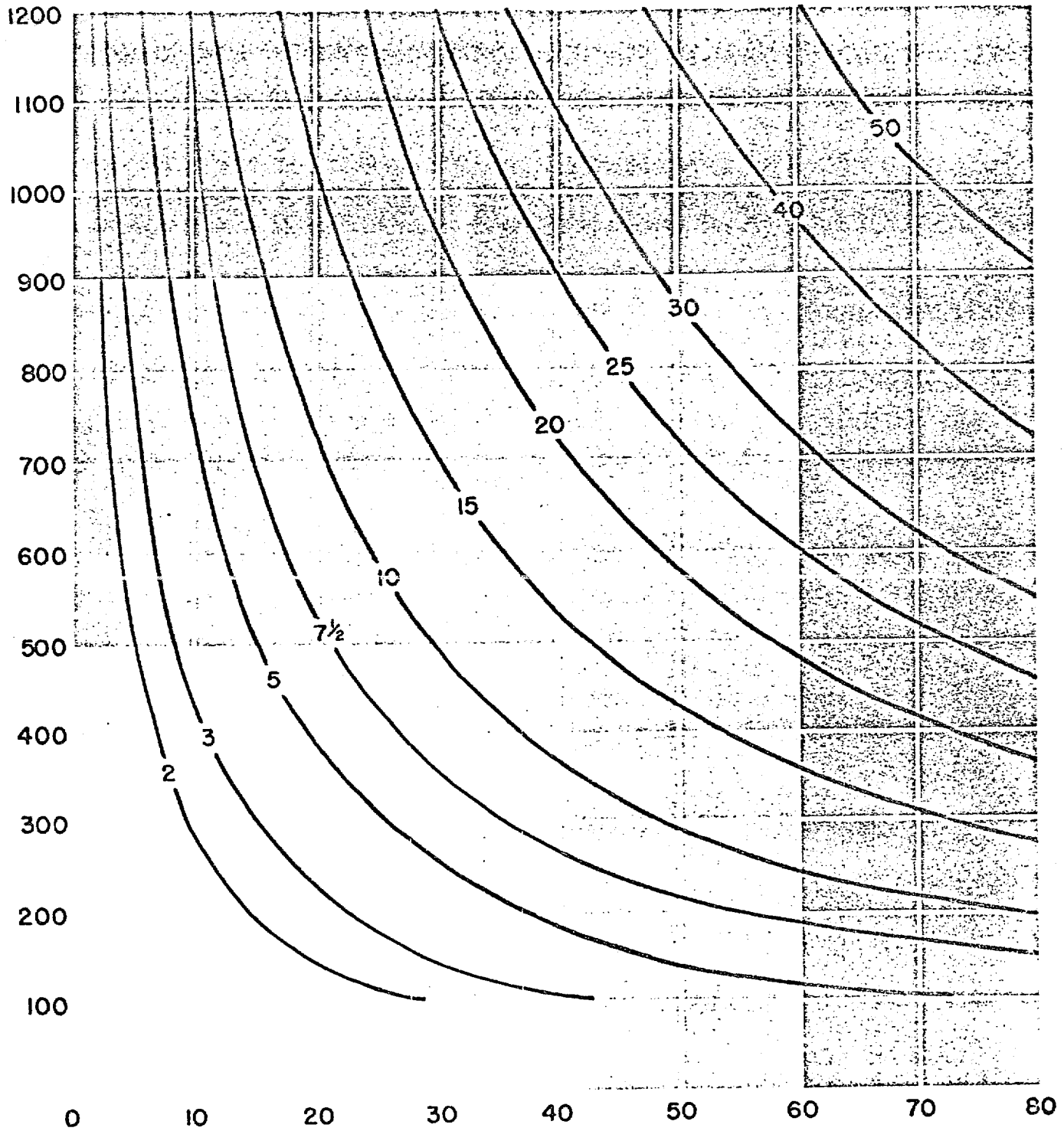
Dimensions

Code	L0914/L0918	L1118/L1122	W1118/W1122
A	12	14.12	13.59
B	15	20.25	20.25
C	1.12	1.50	1.50
D	11.30	14.12	14.00
E	23.12	30.94	29.06
F	.250	.375	.375
G	11.75	15.56	13.12
Suction	1 1/2 NPT	2 NPT bottom 1 1/4 NPT side	2 NPT
Discharge	1 NPT	1 1/4 NPT	1 NPT
Oil capacity	SAE 30	5 qts. SAE 30	SAE 30
Weight	200 lbs.	385 lbs.	365 lbs.





Horsepower selection curves



This chart is for electric motor selection - for gasoline engine horsepower requirements, multiply the value from the chart by 140%.

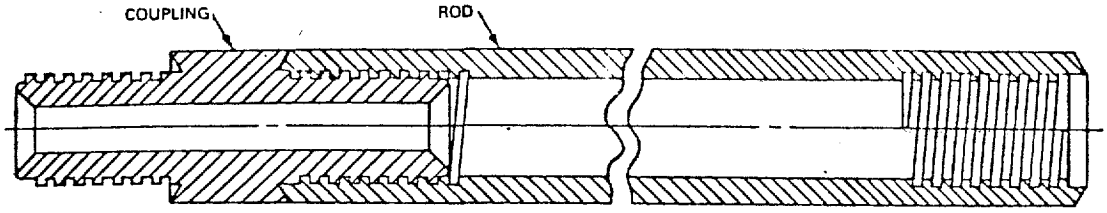
If gpm, psi intersection point is between two horsepower curves, select the higher horsepower.

For further information, contact:
FMC Corporation
Industrial Pumps
Agricultural Machinery Division
Jonesboro, Ark. 72401

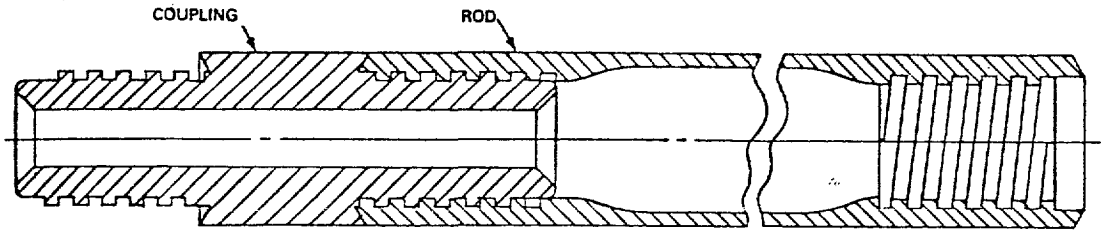
FMC Fluid Control Equipment

Printed in USA J1041-1

DRILL RODS



TYPICAL STRAIGHT WALL DRILL ROD



TYPICAL UPSET DRILL ROD

W Group Drill Rods

ROD SIZE		EW	AW	BW	NW	HW
SPECIFICATIONS	Rod O. D., Ins.	1⅜	1¾	2⅛	2⅝	3½
	Rod I. D., Ins.	27/32	1⅜	1¾	2¼	3 1/16
	Coupling I. D., Ins.	7/16	⅝	¾	1⅜	2⅜
	Threads Per Inch	3	3	3	3	3
	Wt. Per Ft., Lbs.	3.2	3.4	4.3	5.5	
	Joints Packed Per Box	20	20	16	9	
	Gross Wt. (Boxed), Lbs. Per Ft. For Export Shipment	3.4	3.6	4.6	5.9	

DESCRIPTION		EW PART NO.	EW WGT. LBS.	AW PART NO.	AW WGT. LBS.	BW PART NO.	BW WGT. LBS.	NW PART NO.	NW WGT. LBS.	HW PART NO.	HW WGT. LBS.
Rod and Coupling Assembly	1 Ft.	515230-11	3.2	515230-21	4.6	515230-51	5.0	515230-61	5.5		
	2 Ft.	515230-12	6.4	515230-22	9.2	515230-52	10.0	515230-62	11.0		
	5 Ft.	515230-13	16.0	515230-23	23.0	515230-53	21.5	515230-63	27.5	515230-9	45.3
	10 Ft.	515230-14	32.0	515230-24	34.0	515230-54	43.0	515230-64	55.0	515230-10	83.7
	20 Ft.	515230-15	64.0	515230-25	68.0	515230-55	86.0	515230-65	110.0		

W Group Rod Couplings

Coupling Only	259908	1.2	259914	2.0	259920	3.8	259926	5.3	1291112	8.4

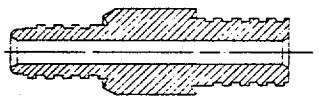
NOTES: 1. All lengths of EW rod are made from straight wall tubing.
All AW rods 1 ft., 2 ft., and 5 ft. are made from straight wall tubing.
BW and NW rods 1 ft. and 2 ft. are also made from straight wall tubing.
2. Left hand drill rods in the above sizes will be furnished on special order.
3. O. D and I. D. dimensions listed apply to 10 ft. rod lengths.

SUBS

Subs are used as bushings to connect drill rods, casing, pipe, etc. that have a different size or type of thread. The types of subs listed on the following pages are the most commonly used. Others to meet your needs can be supplied on special order. Be sure to order all the subs necessary to make up rod connections to core barrels, casing, standpipe, waterswivels, rock bits, fishtails, fishing tools, clinometers, hole testers, etc.

PIN TO PIN

For connecting right-hand drill rod to right-hand drill rod.



R.H. rod pin to
R.H. rod pin

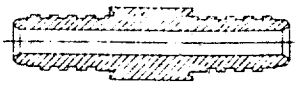
R.H. Rod Pin	R.H. Rod Pin	Part No.	Wgt. Lbs.
EW	AW	290000-5	2.0
EW	BW	290000-6	2.4
EW	NW	290000-7	5.0
AW	BW	290001-5	3.0
AW	NW	290001-6	5.1
BW	NW	290002-5	5.3

R. H. Rod Pin	R. H. Rod Pin	Part No.	Wgt. Lbs.
E	A	6361 AC	1.5
E	B	6362 AC	2.3
E	N	6363 AC	4.5
A	B	6364 AC	2.3
A	N	6365 AC	4.5
B	N	6366 AC	4.4
BX Wire Line	N	1291658	
NX Wire Line	N	1291659	
AX Wire Line	N	1291989	
NC Wire Line	N	1291660	

R H Rod Pin	R.H. Rod Pin	Part No.	Wgt. Lbs
EW	E	290000-1	1.1
EW	A	290000-2	1.5
AW	E	290001-1	2.1
AW	A	290001-2	2.3
AW		290001-4	5.0
BW	E	290002-1	2.3
BW	A	290002-2	2.5
BW	B	290002-3	3.1
BW		290002-4	5.1
NW	A	290003-2	3.5
NW	B	290003-3	4.1
NW	N	290003-4	6.0
NW	NX Wire Line	1291999	6.0

PIN TO PIN

For connecting right-hand drill rod to left-hand rod.



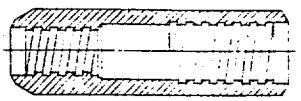
R.H. rod pin to
L.H. rod pin

R.H. Rod Pin	L.H. Rod Pin	Part No.	Wgt. Lbs.
EW	EW	290000-8	1.1
AW	AW	290001-7	2.3
BW	BW	290002-7	3.1
NW	NW	290003-6	6.0

R.H. Rod Pin	L.H. Rod Pin	Part No.	Wgt. Lbs.
E	E	2165 AC	0.6
A	A	1049 AC	1.4
B	B	262936	2.2
N	N	1927 AC	3.5

BOX TO BOX

For connecting right-hand rod coupling to right-hand rod coupling.



R.H. rod box to
R.H. rod box

R.H. Rod Box	R.H. Rod Box	Part No.	Wgt. Lbs.
EW	EW	290029-2	1.3
EW	AW	290029-4	2.1
EW	BW	290029-6	3.0
EW	NW	290029-8	4.2
AW	AW	290030-3	2.1
AW	BW	290030-5	3.0
AW	NW	290030-7	4.0
BW	BW	290031-4	3.0
BW	NW	290031-6	4.0
NW	NW	290032-5	4.0

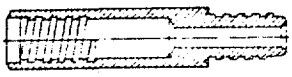
R.H. Rod Box	R.H. Rod Box	Part No.	Wgt. Lbs.
E	E	342 AC	1.1
E	A	343 AC	1.6
E	B	344 AC	2.4
E	N	345 AC	4.1
A	A	346 AC	1.4
A	B	347 AC	2.1
A	N	348 AC	3.5
B	B	349 AC	2.1
B	N	350 AC	3.4
N	N	351 AC	3.3
N	BX Wire Line	1291997	
N	NX Wire Line	1291998	

R.H. Rod Box	R.H. Rod Box	Part No.	Wgt. Lbs.
EW	E	290029-1	1.3
AW	E	290030-1	2.1
AW	A	290030-2	2.1
BW	E	290031-1	3.0
BW	A	290031-2	3.0
BW	B	290031-3	3.0
NW	E	290032-1	4.2
NW	A	290032-2	4.0
NW	B	290032-3	4.0
NW	N	290032-4	4.0

SUBS

BOX TO PIN

For connecting right-hand rod coupling to right-hand drill rod.



R.H. rod box to
R.H. rod pin

R.H. Rod Box	R.H. Rod Pin	Part No.	Wgt. Lbs.
EW	AW	290004-5	2.0
EW	BW	290004-6	2.5
EW	NW	290004-7	4.3
AW	EW	290005-5	1.5
AW	BW	290005-6	2.3
AW	NW	290005-7	5.0
BW	EW	290006-5	2.1
BW	AW	290006-6	2.3
BW	NW	290006-7	4.5
NW	EW	290007-5	3.3
NW	AW	290007-6	3.5
NW	BW	290007-7	4.0

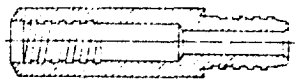
R.H. Rod Box	R.H. Rod Pin	Part No.	Wgt. Lbs.
E	EW	290004-9	1.5
E	AW	290746	1.5
A	AW	290005-9	1.5
B	BW	290006-9	2.4
N	AW	290007-17	3.5
N	NW	290007-9	4.3

R.H. Rod Box	R.H. Rod Pin	Part No.	Wgt. Lbs.
E	A	6149 AC	1.5
E	B	6150 AC	2.3
E	N	6152 AC	4.2
A	E	6160 AC	1.2
A	B	6151 AC	2.2
A	N	6153 AC	4.5
B	E	6158 AC	1.5
B	A	6159 AC	2.0
B	N	6154 AC	4.4
N	E	6155 AC	3.3
N	A	6156 AC	3.2
N	B	6157 AC	3.4

R.H. Rod Box	R.H. Rod Pin	Part No.	Wgt. Lbs.
EW	E	290004-1	1.5
EW	A	290004-2	1.5
EW	B	290004-3	2.3
EW	N	290004-4	4.2
AW	A	290005-2	1.5
AW	B	290005-3	2.2
AW	N	290005-4	4.5
BW	B	290006 3	2.3
BW	N	290006-4	4.4
NW	N	290007-4	4.3
AW	BX Wire Line	1292000	
B	BX Wire Line	1292001	
N	NX Wire Line	1292002	
NW	NX Wire Line	1292003	
BX Wire Line	A	1292004	
BX Wire Line	N	1292005	
NX Wire Line	N	1292006	
NX Wire Line	NW	1292007	
NX Wire Line	B	1292008	

BOX TO PIN

For connecting right-hand rod coupling to left-hand drill rod.



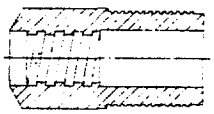
R.H. rod box to
L.H. rod pin

R.H. Rod Box	L.H. Rod Pin	Part No.	Wgt. Lbs.
EW	EW	290004-8	1.5
AW	AW	290005-8	1.5
BW	BW	290006-8	2.5
NW	NW	290007-8	4.5

R.H. Rod Box	L.H. Rod Pin	Part No.	Wgt. Lbs.
E	E	1165 AC	1.1
A	A	1211 AC	1.4
B	B	1212 AC	2.0
N	N	1213 AC	4.0

BOX TO PIN

For connecting right-hand rod coupling to right-hand casing.



R.H. rod box to
R.H. casing pin

R.H. Rod Box	R.H. Casing Pin	Part No.	Wgt. Lbs.
EW	EX	290004-11	1.5
EW	AX	290004-12	1.5
EW	BX	290004-13	2.1
EW	NX	290004-14	3.3
AW	AX	290005-11	6.1
AW	BX	290005-12	6.3
AW	NX	290005-13	6.5
BW	AX	290006-10	9.0
BW	BX	290006-11	9.3
BW	NX	290006-12	9.5
NW	BX	290007-10	11.1
NW	NX	290007-11	11.5
EW	EW	01291532	
AW	AW	01291535	
BW	BW	01291538	
NW	NW	01291540	
HW	HW	01291544	

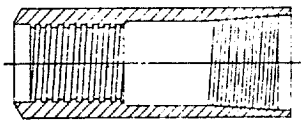
R.H. Rod Box	R.H. Casing Pin	Part No.	Wgt. Lbs.
E	EX	6124 AC	1.5
E	AX	6125 AC	1.5
E	BX	6126 AC	2.5
E	NX	6127 AC	3.3
A	EX	6128 AC	1.5
A	AX	6129 AC	6.5
A	BX	6130 AC	6.3
A	NX	6131 AC	6.5
B	AX	6132 AC	6.0
B	BX	6133 AC	6.3
B	NX	6134 AC	9.5
N	BX	6135 AC	11.5
N	NX	6136 AC	11.5
BX Wire Line	BX	1292010	
NX Wire Line	NX	1292011	
AX Wire Line	AW	01291945	
AX Wire Line	BW	01291946	
BX Wire Line	BW	01291947	
BX Wire Line	NW	01291948	
NX Wire Line	NW	01291943	
NX Wire Line	HW	01291944	

R.H. A.P.I. Box	R.H. Rod Pin	Part No.	Wgt. Lbs.
2 3/8	NX Wire Line	1292009	

SUBS

BOX TO BOX

For connecting right-hand coupling to right-hand A.P.I. pin.



R.H. rod box to R.H. A.P.I. box

R.H. Rod Box	R.H. A.P.I Box	Part No.	Wgt. Lbs.
NW	2⅜	290032-6	11.3
NW	2⅞	290032-7	21.5
NW	3½	290032-8	28.5
R.H. Rod Box	**R.H. A.P.I. Box**	**Part No.**	**Wgt. Lbs.**
N	2⅜	4282 AC	11.8
N	2⅞	4401 AC	18.0
N	3½	4327 AC	27.0
BX Wire Line	2⅜	1292012	
NX Wire Line	2⅜	1292013	

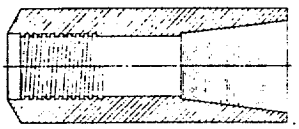
BOX TO PIN

For connecting right-hand rod coupling to right-hand A.P.I. box.

R.H. Rod Box	R.H. A.P.I. Pin	Part No.	Wgt. Lbs.
N	2⅜	3639 AC	
N	2⅞	7498 AC	
N	3½	7496 AC	

BOX TO BOX

For connecting right-hand rod coupling to right-hand pipe.

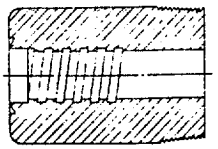


R.H. rod box to R.H. pipe box

R.H. Rod Box	R. H. Pipe Box	Part No.	Wgt. Lbs.
NW	2	290032-12	8.0
NW	2½	290032-11	7.3
NW	4	290032-10	26.0
R.H. Rod Box	**R.H. Pipe Box**	**Part No.**	**Wgt. Lbs.**
N	2	1899 AC	7.5
N	2½	1949 AC	6.6
N	4	1898 AC	25.3

BOX TO PIN

For connecting right-hand rod coupling to right-hand pipe coupling.



R.H. rod box to R.H. pipe pin

R.H. Rod Box	R.H. Pipe Pin	Part No.	Wgt. Lbs.
EW	2	290004-15	3.2
AW	2	290005-14	3.0
AW	2½	290005-15	6.2
AW	3	290005-16	10.5
BW	2½	290006-13	5.3
BW	3	290006-14	10.1
NW	2½	290007-12	6.0
NW	3	290007-13	10.0
NW	3½	290007-14	14.3
NW	4	290007-15	22.2
NW	4½	290007-16	30.0
R.H. Rod Box	**R. H. Pipe Pin**	**Part No.**	**Wgt. Lbs.**
E	2	2859 AC	3.1
A	2	1528 AC	2.5
A	2½	1529 AC	6.2
A	3	1530 AC	10.5
B	2½	1765 AC	5.0
B	3	2549 AC	10.1
N	2½	1645 AC	5.3
N	3	1616 AC	9.5
N	3½	1595 AC	14.3
N	4	1646 AC	22.2

ROD DOPE

Rod dope, or rod grease, is applied to the outside of the rods and serves the main purpose of reducing vibration. This in turn cuts down unnecessary abrasive and concussive wear on drill rods, bits, core barrels and the machine itself.

Rod dope adheres well to the rods and resists the washing action of the pumping fluid. Optimum results are obtained by doping the entire string of drill rods. *Do not dope the core barrel* as there is only a small clearance between the barrel and the wall of the hole, and the dope and cuttings will plug this area and stop circulation.

DESCRIPTION	PART NO
20 Lb. Pail	90068 AK

ROD WICKING

Rod wicking serves the dual purpose of sealing rod joints, minimizing loss of circulation, and easing the uncoupling of the drill string. The joints are tightened by applying high torque and are kept exceedingly tight by friction. Before each rod is completely coupled, a few wraps are put on the thread shoulders. The rods are then coupled tight and "bucked up." *Do not wick bits, reaming shells, or core barrel heads,* since tightening up on wicking at these points will swell and distort the thread connections. The composition of Joy wicking is soft cotton wrap.

DESCRIPTION	PART NO
8 Balls Per Lb	1270638

WIRE LINE EQUIPMENT

Wire line core barrels are double tube, swivel type with an end assembly similar to the M design in which an inner tube extension carries the flushing fluid very close to the crown of the bit. Wire line tools differ from other standard tools in that the inner tube containing the core can be pulled without need of removing the drill string from the hole. The advantages of this barrel are quickly apparent to any operator familiar with round trip time required in recovering filled core barrels from a coring operation. Rods need be pulled only on completion of hole or to change a dulled bit.

Joy wire line tools feature a new patented ball latching device which attaches the inner tube upper head to the outer tube coupling of the core barrel during drilling, and permits positive unlatching with the use of the overshot when the barrel is filled. A water shut-off valve insures instant indication of a filled barrel or a core block by showing an increase in pump pressure, and permits the operator to obtain maximum core recovery even in previously known "hard to core" ground.

Wire line tools required for coring, which differ from standard equipment are as follows:

- Drill Rods
- Core Barrel
- Overshot
- Wire Line Hoist
- 3/16" Wire Cable
- Parmalee Wrenches
- Subs (Hoisting Plug and Water Swivel to Wire Line Rods)
- Set Reaming Shell
- Set Core Bit
- Chuck Jaws and Clamp Jaws

Refer to Drillers Check List (Page 4 for other required standard items such as, water swivels, hoses, pumps, casing, etc.

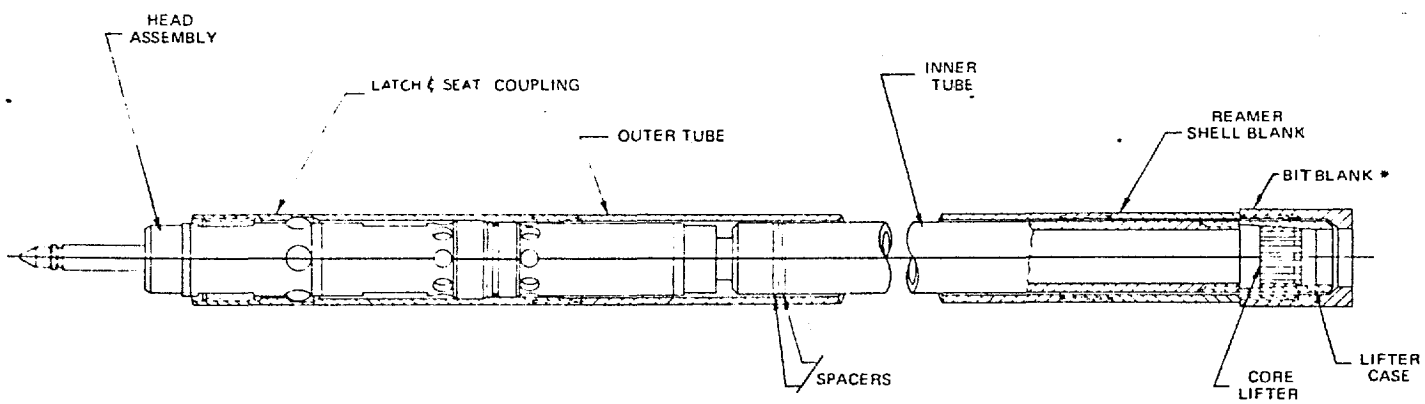
Core Barrel Specifications

SIZE	AX		BX		NX		NC	
	English	Metric	English	Metric	English	Metric	English	Metric
Hole Diameter (Approx)	1 7/8	47.6	2 3/8	60.3	3	76.2	3 11/16	93.7
Core Diameter (Approx)	1 1/64	25.8	1 7/16	36.5	2	50.8	2 13/32	61.1
Reaming Shell—Set O.D.	1 55/64	47.2	2 3/8	60.3	2 63/64	75.8	3 43/64	93.3
Bit—Set O.D.	1 27/32	46.8	2 23/64	59.9	2 31/32	75.4	3 21/32	92.9
Bit—Set I.D.	1 1/64	25.8	1 7/16	36.5	2	50.8	2 13/32	61.1
Outer Tube O.D.	1 3/4	44.5	2 9/32	58.0	2 27/32	72.2	3 1/2	88.9
Outer Tube I.D.	1 27/64	36.1	1 29/32	48.4	2 15/32	62.7	3 1/16	77.8
Inner Tube O.D.	1 1/4	31.8	1 3/4	44.5	2 5/16	58.7	2 29/32	73.8
Inner Tube I.D.	1 1/16	27.0	1 1/2	38.1	2 5/64	52.8	2 9/16	65.0

Drill Rod Specifications

Rod—O.D.	1 3/4	44.5	2 1/4	57.2	2 13/16	71.4	3 1/2	88.9
Rod—I.D.	1 11/32	34.1	1 7/8	47.6	2 7/16	61.9	3 1/16	77.8
Threads Per inch	4		4		4		3	

Wire Line Core Barrel Assembly



*BIT BLANK IS SHOWN FOR REFERENCE ONLY THD. PROTECTOR WILL BE FURNISHED WITH CORE BARREL ASSY.

EFTELLER, INC.

reservoir engineering data

Phone (915) 682-5574 Midland, Texas 79701
P. O. Box 5247

Phone (505) 325-6817 Farmington, New Mexico 87401
P. O. Box 1198

Phone (817) 559-8531 Breckenridge, Texas 76024
P. O. Box 1206

EFFECTIVE MARCH 15, 1979

RESERVOIR DATA

B.H.P.	**BOTTOM HOLE PRESSURE TESTS**		
	Shut in and flowing traverses	Per Test	$ 45.00
	Additional charge for wells below 7000'	Per 1000	7.00
B.U.	**BOTTOM HOLE PRESSURE BUILD UP TESTS**		
F.L.	**PRODUCTIVITY INDEX TESTS**		
D.D.	**BOTTOM HOLE PRESSURE DRAWDOWN TEST**		
	First four hours (Equipment Time)	Per Test	175.00
	Additional equipment time	Per Hour	7.50
	Operator's time (Shop to Shop)	Per Hour	17.50
G.W.T.	**GAS WELL TESTS**		
	OPEN FLOW POTENTIAL DETERMINATIONS		
	RECOMBINATION RATIO DETERMINATION (INCLUDES SAMPLING)		
	DELIVERABILITY DETERMINATION		
	GAS/OIL RATIO TEST		
	First four hours (Equipment Time)	Per Test	175.00
	Additional equipment time	Per Hour	7.50
	Operator's time (Shop to Shop)	Per Hour	17.50
R.F.S.	**RESERVOIR FLUID SAMPLE**		
	Bottom hole sampling (8-hour Minimum)	Per Hour	35.00
	Surface sampling (8-hour Minimum)	Per Hour	25.00
F.L.	**FLUID LEVEL ACOUSTICAL SOUNDING**		
	Shut in and pumping fluid levels	Per Test	25.00
T.S.	**TEMPERATURE SURVEYS (INJECTION PROFILE AND CEMENT TOP)**		
	To 3000 feet (Minimum Charge)		$175.00
	Below 3000 feet	Per Foot	0.015

Special Tests, Production Problems, Interference Tests, Lease Evaluation Data, Surface Pressure Exceeding 4000 PSI, etc., by quotation.

RENTAL: Deadweight testers, critical flow prover, single pen mercury meter & orifice well tester, 2 pen pressure recorders.

First day		35.00
Additional days	Per Day	17.50

Plus any necessary repairs upon return.

WIRELINE SERVICES

WIRELINE UNIT (and operator) FOUR CORNERS AREA ONLY

First four hours (Minimum Charge)		175.00
Additional time	Per Hour	30.00
Service requiring extra man	Per Hour	15.00

FULL INSURANCE COVERAGE — CERTIFICATES FURNISHED UPON REQUEST
PRICES SUBJECT TO CHANGE WITHOUT NOTICE

WIRELINE SERVICES (Continued)

WIRELINE RENTAL TOOLS FOUR CORNERS AREA ONLY

Pulling, running & setting tools (RB, JDC, JUG, etc.)	Per Day	$ 40.00

Special tools such as shifting tools, selector tools, separation tools and special running and pulling tools, will be charged subject to manufacturing company's price schedule.

Standard wireline tool string consists of rope socket, stem and wireline jars. Any tool used below the jars will be charged for as rental tool, such as knuckle joints, running and pulling tools, overshots, etc. Exception: Paraffin cutters will be part of tool string.

Camco hydrostatic tubing bailer & sand bailer	Per Day	90.00

Blanking Plugs, Circulating Plugs, Tubing Stops, etc.*

*Rental rates are based on 20% of the retail price for the first day (plus any parts). Each additional day at 10%.

TERMS

All services subject to a four-hour minimum charge		175.00
Mileage charge (round trip) Car or Pickup	Per Mile	0.50
Mileage charge (round trip) One-Ton Truck	Per Mile	0.70
Stainless line where required	Per Day	100.00

Tandem instruments add 50% to regular charge.

Hazardous conditions (H_2S, High Temperature, Surface Equipment, etc.) by quotation.

Time applies shop to shop.

Jobs requiring personnel layover as a customer convenience will be charged at the rate of 8 hours per day.

Distant Base Points may be set up for extensive testing.

Unrecoverable equipment will be billed at regular replacement cost.

DRILL STEM TEST

D.S.T. FORMATION EVALUATION BY OPEN HOLE DRILL STEM TEST

Base Charge	Per Test	310.00

For hole sizes 10" or less, first 12 hours or fraction for minimum equipment and operator to perform the service of an open-hole drill stem test (successful or unsuccessful).

Depth Charge, to 3000 feet (Minimum Charge)		120.00
Below 3000 feet	Per Foot	.04

On successful test depth charge is made to total depth of well and to depth of lower packer on a straddle test.

Straddle Test Assembly	Per Test	210.00

Includes minimum equipment plus one additional packer.

Positive Control Flow Test (PoCoFlo)	Per Test	300.00

Includes surface measuring equipment to provide complete surface pressure log, calculations of formation production before opening to surface flow, reservoir analysis and final report.

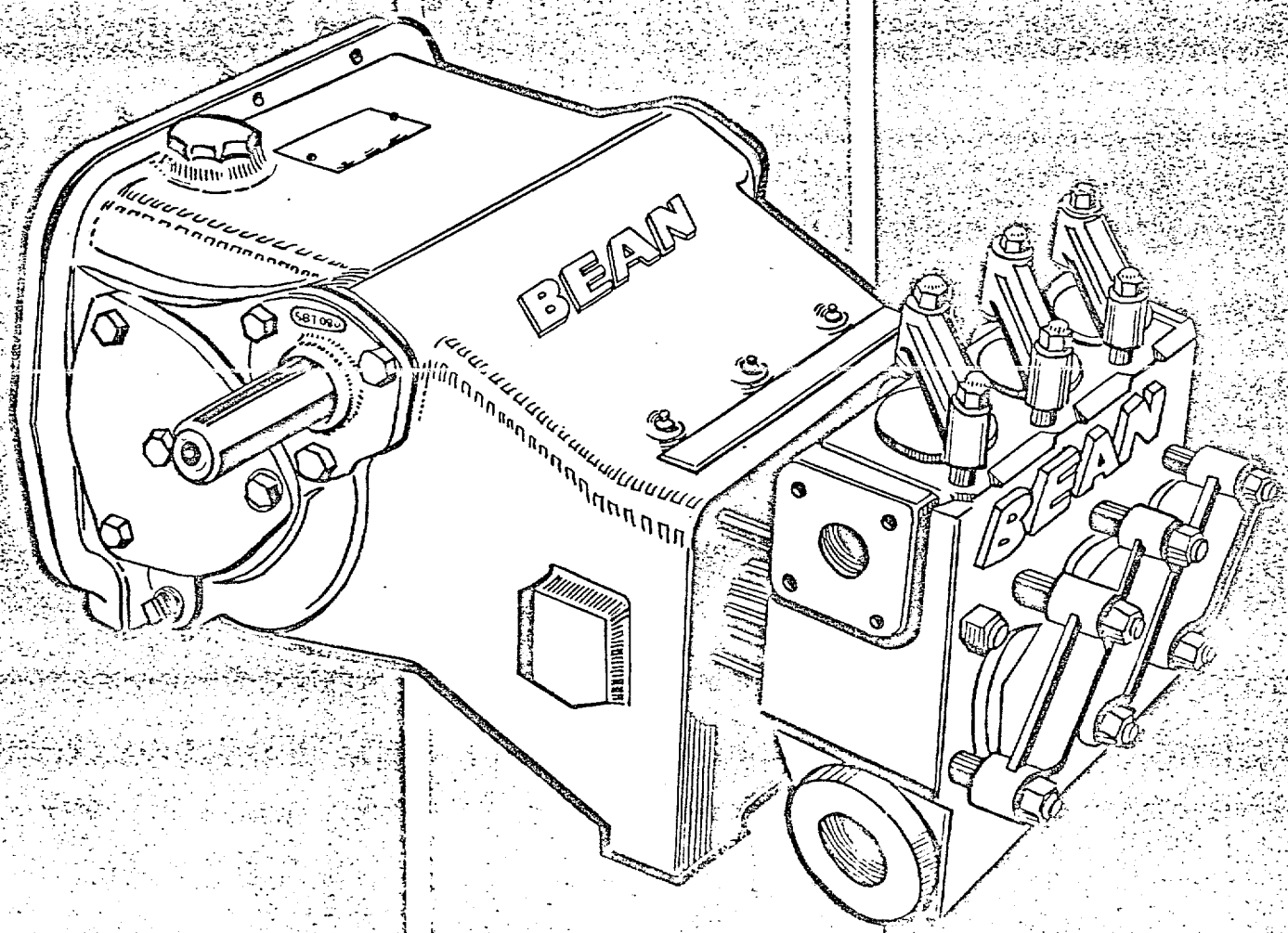
DRILL STEM TEST (Continued)

ADDITIONAL EQUIPMENT AND SERVICES

1. Packer Assembly ..... Per Run $200.00
   For hole sizes 10" or less, larger sizes on quotation, in excess of one packer on conventional test and two on a straddle test.
2. Jars ..... Per Run 210.00
   Through 5" O.D., larger sizes on quotation.
3. Safety Joint ..... Per Run 35.00
   Through 5" O.D., larger sizes on quotation.
4. Selective Zone Anchor ..... Per Setting 240.00
   For hole sizes through 9", larger sizes on quotation.
5. Reverse Circulating Valve
   If not used ..... Per Run 15.00
   If used ..... Per Run 60.00
6. Formation Fluid Sampler ..... Per Run 125.00
   Approximately 2000 c.c. fluid sample caught at final flow time. If chamber is sent to laboratory, a rental of $62.00 per day will be made on sampler.
7. Pressure Recorders ..... Per Run 125.00
   In excess of two recorders on a conventional test, and in excess of three recorders on a straddle test.
8. Operator's Time ..... Per Hour 15.00
   In excess of 12 hours, time starts when operator arrives at location until operator leaves location.
9. Additional Equipment Time ..... Per Hour 6.50
   In excess of 12 hours, time starts when equipment arrives on location until equipment leaves location.
10. Mileage Charge ..... Per Mile .60
    In excess of 100-mile round trip from shop.
11. Canceled Service Charge ..... Each 310.00
    In the event the service is canceled after operator arrives on location a service charge plus mileage (in excess of 100 miles, round trip) will be made.
12. Reservoir Analysis ..... Per Chart 125.00
    Analysis from drill stem test pressure chart giving permeability, reservoir pressure, wellbore damage, radius of investigation, flow rate, etc., and a letter of discussion for values calculated. Reservoir analysis from other pressure data (build-up, drawdown, and d.s.t.'s other than Tefteller's) will be charged at a rate of $35.00 per hour.
13. Incremental Pressure Readings, per point $0.25 ..... Minimum Charge 20.00
    Incremental pressure readings presented in tabular and graphical form.
14. Extra Test Reports ..... Per Copy 7.00
    In excess of 5 reports which are included with test.

Bean®

Triplex piston pumps



Pressures
up to 1200 psi

GASO PUMPS, INC.
Box 2069, Tulsa, Oklahoma 74101

Oct 1, 1979 Gaso file #4-100179-1

PROPOSAL

Science Application
16625 West 51st. Ave.
Golden, Col. 80401
Attn: Mr. Ralph Nelms

CONDITIONS: Unit designed to deliver 30 GPM of FRESHWATER at 400 PSIG discharge, flooded suction. Pump cal at 95% Ev. and 90% Em. with a input of 8.1 BHP. SPGR is 1.0 at ambient pumping temp. Pump speed is 305 RPM from a 2200 RPM Gas engine.

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1 Gaso figure 3211, 2-1/4" x 2", single acting horizontal triplex plunger pump with totally enclosed, self-lubricated power end with crankshaft journaled in single row tappered roller bearings, right or left hand crankshaft extension when standing behind power frame facing the fluid end. Equipped with a Cast Iron (Meehanite GA-50) fluid end with Ductile Iron Valve covers and Cylinder heads. Stainless Steel Wing Guided Valves and Seats. Colmonoy surface on Steel plungers with style 838-LG Fluid End stuffing box packing. Style double lip U-Cup power end packing. Integral Cast Iron stuffing boxes with bolt-on glands. Single 2" NPT suction and Single 1-1/2" NPT discharge connections.

1 3-feed pressure lubricator installed on the pump and piped to the fluid end stuffing boxes with copper tubing.

1 Wisconsin air cooled gasoline engine model TJD, with all standard equipment to include electric starter with fly wheel alternator, High temp./low oil press safety shut down, Clutch Power Take Off running at 2200 RPM.

1 V-Belt drive to consist of a 30.0" PD, 4-B groove, QD, pump sheave, a 4.2" PD, 4-B groove, QD, engine sheave, and four B-112, OHS, V-belts, matched set of four belts.

1 Totally enclosed sheet metal V-Drive guard with Gaso's interpretation of OSHA regulations.

1 5" channel pump skid and 5" channel motor subbase with built in V-Belt adjustment.

NET PRICE EACH UNIT, F.O.B. GASO PLANT, TULSA, OKLA..... $5985.00

CONTINUED

SUBJECT TO DELAYS OCCASIONED BY FIRE, STRIKES OR OTHER CAUSES BEYOND OUR REASONABLE CONTROL
AND TO DOMESTIC PROPOSAL STANDARD PROVISIONS ON REVERSE SIDE

GASO PUMPS, INC.
Box 2069, Tulsa, Oklahoma 74101

Oct 1, 1979 Gaso File #4-100179-1

PROPOSAL

Science Application

DELIVERY - 4 - 5 Wks. ARO (subject to prior sale)
WEIGHT - 1520# approx.

RECOMMENDED EQUIPMENT

1 - Safety relief valve (shipped loose)part #7601-1-LP, ADD $146.00 NET.
1 - Spare set fluid end plunger packing, style 838-LG, part #1111-2-1/4, Non-Adjustable ADD $70.00 NET(spare set to consist of 3 unit packing)

PRICES QUOTED HEREIN ARE SUBJECT TO CHANGE AFTER A PERIOD OF 30 DAYS FROM PROPOSAL DATE, AFTER WHICH PRICES WILL BE SUBJECT TO CONFIRMATION.

SUBJECT TO DELAYS OCCASIONED BY FIRE, STRIKES OR OTHER CAUSES BEYOND OUR REASONABLE CONTROL
AND TO DOMESTIC PROPOSAL STANDARD PROVISIONS ON REVERSE SIDE

Date OCT 1, 1971

GASO PUMPS, INC.

PUMP APPLICATION & SPECIFICATION SHEET (PASS)

File Ref: _____ Date Received 9/21/79 Received Via. TELEPHONE

Received From SCIENCE APPLICATION User SAME

Order Will Be Placed By SCIENCE APPLICATION

Final Location of Pump (City, County, State, Country) _____

SPECIFICATION FROM CUSTOMER

Item No. _____

Liquid Pumped FRESHWATER

Liquid Analysis:

Solids: _____ %, Size & Type

Pumping Temperature: AMB °F Suction

Gravity @ Pumping Temperature 1.0 API/Specific

PH _____, Viscosity @ Inlet _____

_____ % CO_2 _____ % O_2 _____ % H_2S

_____ % Methane, _____ % Ethane

_____ % Propane, _____ % Butane

_____ % Pentane, _____ % Hexane

_____ % Entrained Gases

_____ % (other)

Special Materials Req'd: _____

PUMP OPERATING CONDITIONS:

Req'd Delivery Rate 30 GPM MAX

Pressure: Suction FLOODED PSIG

Discharge 400 PSIG

Differential 400 PSI

Vapor Pressure at Suction Temp. _____ PSIG

Static Head at Suction Inlet _____

System: Open / Closed _____

Limitations on Rpm _____

Duty: Continuous/Intermittent

Prime Mover Specifications: GAS ENGINE

Furnished By: GASO Mounted By: GASO

Type of Drive Required V-BELT

Guard OSHA

Base STD

Special Painting Required —

Special Equipment Required —

Data Required with Proposal —

GASO RECOMMENDATIONS TO MEET CUSTOMERS SPECIFICATIONS

Pump Figure No. 3211 Size 2 1/4 x 2

Type of Pump TRIPLEX

Number of Pumps Required ONE

Displacement of Each Pump @ 100% Volumetric Efficiency 31.5 GPM

Delivery Rate of Each Pump 30 GPM

@ 95 % Ev, and 90 % Em

Pump RPM @ Specified Conditions 305

Brake Horsepower @ Specified Conditions 8.1

* * Net Positive Suction Head Required at the Pump Inlet _____ (PSI); _____ (Ft. Water)

MATERIALS:

Liquid Cylinder MECHANITE

~~Liner~~ _____

~~Piston~~ _____ ~~Piston Rings~~ _____

~~Rod or~~ Plunger COLMONOY SURFACED

Valve Type WING GUIDED

Valve & Seat Material STAINLESS STEEL

Liquid End Packing 838-LG

Stuffing Box: replaceable / (integral) IRON

Packing Gland: Threaded / (Bolt-on) IRON

Connections:

Suction: SINGLE 2" NPT

Discharge: SINGLE 1 1/2" NPT

GENERAL PUMP SPECIFICATIONS:

Power End Lubrication SPLASH

Maximum Working Pressure 235 PSIG when Fitted with Smallest Allowable Pistons or Plungers.

Maximum Discharge Pressure of the Liquid Cylinder when Fitted as Described Above 400 PSIG

Hydrostatic Test Pressure of Liquid Cylinder:

Suction: 2000 PSIG

Discharge: 2000 PSIG

Suction Valve Seat Area .95 SQ-IN

Velocity thru Suction Valve Seat 7.1 FT/SEC

Liquid End Packing Lubrication Required: (Yes) / No

Other Specifications: _____

Data Being Furnished with Proposal _____

* * Net Positive Suction Head Required (NPSHR) is calculated from theoretical values and is that absolute dynamic pressure required at the center line of the pump inlet during operation of the pump, based on water, and is not supported by experimental results.

No responsibility is assumed for pump installation, piping, or conversion of NPSH in terms of product.

NOTE: THIS PASS BECOMES AN INTEGRAL PART OF ANY GASO PROPOSAL MADE FROM SPECIFICATIONS CONTAINED HEREIN.

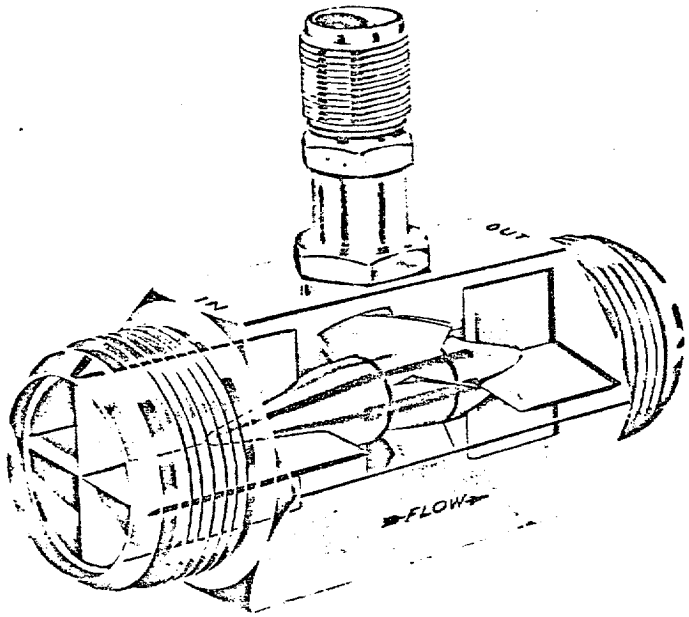
G-2/78

PRODUCT DATA
from
FLOW TECHNOLOGY, INC.

4250 EAST BROADWAY ROAD ▫ P.O. BOX 21346 ▫ PHOENIX, ARIZONA 85036 ▫ PHONE (602) 268-8776 ▫ TELEX 668-344

STANDARD LINE TURBINE FLOWMETERS

- FLOW RATES FROM 0.03 TO 20,000 GALLONS PER MINUTE
- LIQUID OR GAS MEASUREMENT
- TEMPERATURES FROM —430°F. TO +750°F.
- HIGH ACCURACY
- DYNAMIC FLUID THRUST BEARING
- HIGH OVERSPEED CAPABILITY
- LOW MASS ROTOR FOR HIGH DYNAMIC RESPONSE, LONG BEARING LIFE
- WIDE CHOICE OF MATERIALS
- ADDED SAFETY OF BOTH UPSTREAM AND DOWNSTREAM ROTOR SUPPORTS
- LOW PRESSURE DROP



OVER 25 YEARS OF TURBINE FLOWMETER EXPERIENCE GOES INTO EVERY FTI TURBINE FLOWMETER

FLOW TECHNOLOGY, INC. engineers helped develop the first axial turbine flowmeters over 25 years ago. Because of continual development and improvement, FTI's turbine flowmeters have paced the state of the art. Fluid flow measurement is FTI's only business; we have to be very good at it.

GENERAL DESCRIPTION

Basically the FLOW TECHNOLOGY, INC. Standard Line Turbine Flowmeter is a miniature propeller suspended in a pipe. This freely-suspended axial turbine is rotated by the flow of fluid — gas or liquid — through the flowmeter. The rotational speed of the turbine if proportional to the velocity of the fluid. Since the flow passage is fixed, the turbine's rotational speed is also a true representation of the volume of fluid flowing through the flowmeter. This volume can be expressed as gallons per minute, liters per minute, cubic feet per minute, or various other engineering units.

This ideo is very old. Flowmeters based on the water wheel, a similar principle, have been used for centuries. However, modern technology has developed the turbine flowmeter to an outstanding level of accuracy, linearity, durability and reliability.

There is no direct physical connection other than the turbine bearings between the turbine and its housing. The rotation of the turbine is sensed through the flowmeter body by an externally mounted pickoff on the surface directly above the flowmeter rotor. The rotation of this turbine rotor produces a train of electrical pulses in the pickoff. The frequency of these pulses is directly proportional to the volume flowrate. The pulses can then be transmitted to appropriate read-out electronics near the flowmeter or at a remote location. They can be amplified, counted, interfaced with computer terminals and used to measure and control fluid flow. The pulse train can be processed in any digital system.

It is necessary to translate the pulses into meaningful information. Flow Technology, Inc. manufactures a complete line of electronic devices for that purpose. These units can display flow rate in any engineering units, either analog or digital. There are also digital totalizers with LED or mechanical counters as well as batch controllers and blind converters which change signals into a format required to interface a customer's built in system.

GENERAL SPECIFICATIONS

(* Terminology per ANSI C85.1 and ISA S37.1)

ACCURACY:*

±0.05% at all points in the linear flow range (1 centistoke liquid)

± ½ % (gas)

LINEARITY:*

±0.5% of reading over the normal 10:1 range (liquid)

± 1% special premium linearity for special pre-selected ranges. (liquid)

±1% of full scale (gas)

ELECTRICAL OUTPUT:

The output level conforms with ISA RP 31.1 and is a minimum of [illegible] peak-to-peak for frequencies at the bottom of the nominal flow range.

PRESSURE DROP:

Ask for Technical Data Report TD-019.

CALIBRATION:

Each turbine flowmeter is furnished with a calibration with the standard reference fluid, MIL-C-7024B. Special calibrations available for applications where viscosity varies considerably.

OPERATING PRESSURE:

In general, the limiting factor governing the operating pressure of an FTI Standard Line turbine flowmeter is the rating of the end connectors. Because there is no porting of the flowmeter body, the flowmeter can be constructed to handle exceptionally high pressures if desired.

DYNAMIC RESPONSE:

3 millseconds or better response to step input change of flow rate for meters smaller than 1 ½ inches, increasingly longer response times as the size of the meter and the mass of the rotor increases.

END CONNECTIONS:

Flowmeters FT-8 through FT-32 available with either AN Series 37° flared tube (MS-33656) or NPT end connections. Sizes FT-8 through FT-224 available with ASA B 16.5 flanges, 150 lb. through 2500 lb. ratings. Other end connections available on request.

ELECTRICAL CONNECTIONS:

AN3102A-10SL-4P with mating connector supplied. Flowmeters with explosion-proof pickoffs terminate in ½" conduit union.

MATERIALS:

All portions of FTI Standard Line turbine flowmeters that come into contact with the fluid are fabricated of 300-series and 400-series stainless steels. An extremely wide choice of materials is available to satisfy even the most severe specifications. See Options.

OPERATING TEMPERATURE RANGE:

FTI Standard Line turbine flowmeters can be fabricated to measure fluids within a temperature range of —430°F. to +750°F. Nominal temperature range determined by pickoff selected. See bulletin No. PO-762.

APPLICATIONS OF THE TURBINE FLOWMETER

ENGINE R & D:

Extreme precision flow rates of fuels and oxidizers for rocket and jet engine testing and today's urgent requirements for the same measurements for internal combustion and Diesel engine pollution testing.

POLLUTION CONTROL:

Precise measuring and control of fuel oils and natural gas to combustion turbine electric generating units allows conservation of energy resources and more efficient operation.

ON-LINE BLENDING:

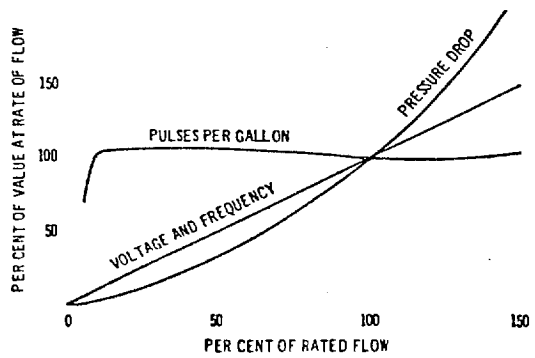
In the petrochemical and chemical industries, turbine flowmeters are precisely measuring catalysts and other fluids to insure the highest quality control and the lowest possible waste in automated systems.

BATCH CONTROL:

Turbine flowmeters are accurately measuring exact batches of liquids with a consistency and reliability vastly superior to weighing and other techniques.

FLOW MONITORING:

The output of turbine flowmeters is being used to monitor critical flow rates of both liquids and gases in a number of different industrial processes, protecting pumps and other equipment as well as insuring consistent end products.



SIZES & SPECIFICATIONS
LIQUID

Model No.	Nominal End Fitting Size (In.)	Normal Flow Range (U.S. GPM)		Extended Flow Range (U.S. GPM)			Approximate** Frequency Output (CPS)	Approximate "K" Factor Pulses Per Gallon
		Minimum	Maximum	Minimum*		Maximum		
				Journal	Ball			
FT-4-8	½	0.25	2.5	N/A	0.03	3	2300	55000
FT-6-8	½	0.5	5.0	0.1	0.05	5	2100	25000
FT-8-8	½	0.75	7.5	0.16	0.08	8	2000	16000
FT-8	½	1.0	10.0	0.2	0.1	10	2000	12000
FT-10	⅝	1.25	12.5	0.2	0.15	15	1700	8300
FT-12	¾	2.0	20	0.33	0.25	25	2000	6000
FT-16	1	5.0	50	0.6	0.6	60	2000	2400
FT-20	1¼	9.0	90	0.9	0.9	90	1950	1300
FT-24	1½	15	150	1.5	1.5	150	1500	600
FT-32	2	22	225	2.5	2.5	250	1300	350
FT-40	2½	40	400	4.5	4.5	450	650	100
FT-48	3	65	650	7.5	7.5	750	812	75
FT-64	4	125	1250	15	15	1500	625	30
FT-80	5	200	2000	25	25	2500	300	9
FT-96	6	300	3000	35	35	3500	***	***
FT-128	8	550	5500	60	60	6000	***	***
FT-160	10	850	8500	100	100	10000	***	***
FT-192	12	1200	12000	150	150	15000	***	***
FT-224	14	1600	16000	200	200	20000	***	***

Other sizes available; check with factory.
The above data is based on a liquid with a S.G. of 1 and a viscosity of 1 centistoke.
Flow Rates and Frequencies other than shown available upon request.

* The extended range requires an active (RF) pickoff and a Range Extending Amplifier Model LFA-300 for meters 2" and smaller.
** At maximum of normal flow range. *** Consult factory.

SIZES & SPECIFICATIONS
GAS

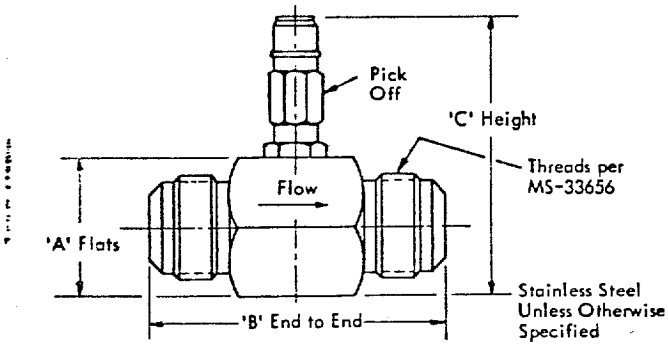
Model No.	Nominal End Fitting Size (In.)	Normal Flow Range (ACFM)		Extended Flow Range (ACFM)		Approximate** Frequency Output (CPS)	Approximate "K" Factor Pulses Per Cu. Ft.
		Minimum	Maximum	Minimum	Maximum		
FT-4-8	½	0.25	2.5	0.2	3	2300	55000
FT-6-8	½	0.5	5.0	0.25	5	2100	25000
FT-8-8	½	0.75	7.5	0.4	8	2000	16000
FT-8	½	1.0	10.0	0.5	10	2000	12000
FT-10	⅝	1.25	12.5	0.6	15	1700	8300
FT-12	¾	2	20	1.0	25	2000	6000
FT-16	1	5	50	1.5	60	2000	2400
FT-20	1¼	9	90	2.25	90	1950	1300
FT-24	1½	15	150	3.75	150	1500	600
FT-32	2	22	225	5	250	1300	350
FT-40	2½	40	400	9	450	650	100
FT-48	3	65	650	15	750	812	75
FT-64	4	125	1250	30	1500	625	30
FT-80	5	200	2000	50	2500	300	9
FT-96	6	300	3000	70	3500	***	***

The above data is based on air at 60°F. and 14.7 psi for meters with ball bearings. Gases with less density will have a more limited range.
** At maximum of normal flow range. *** Consult factory.

DIMENSIONS

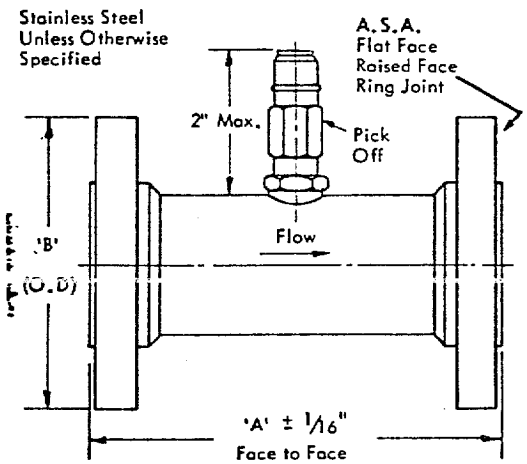
All dimensions in inches — certified dimensions available on request.

AN (MS-33656) or NPT
NPT (consult factory for larger size NPT meter)



SIZE & MODEL		A	B	C
¼" — ⅜" — ½"	FT-8M	1" sq.	2.45	3.00
⅝"*	FT-10M	1.312	2.72	3.187
¾"	FT-12M	1.375	3.25	3.250
1"	FT-16M	1.625	3.56	3.5
1½"	FT-24M	2.125	4.59	4.375
2"	FT-32M	2.750	6.06	4.750

END FLANGED (ASA B16.5):



SIZES (½" - 14")

LINE SIZE	150# ANSI		300# ANSI		400# ANSI		600# ANSI		900# ANSI		1500# ANSI		2500# ANSI	
	A	B	A	B	A	B	A	B	A	B	A	B	A	B
.5	5.0	3.5	5.0	3.75	5.0	3.75	5.0	3.75	7.0	4.75	7.0	4.75	7.0	5.25
*.63	5.5	3.88	5.5	4.63	5.5	4.63	5.5	4.63	7.0	5.13	7.0	5.13	7.0	5.5
.75	5.5	3.88	5.5	4.63	5.5	4.63	5.5	4.63	7.0	5.13	7.0	5.13	7.0	5.5
1.0	5.5	4.25	5.5	4.88	5.5	4.88	5.5	4.88	8.0	5.88	8.0	5.88	8.0	6.25
1.25	6.0	4.63	6.0	5.25	6.0	5.25	6.0	5.25	8.0	6.25	8.0	6.25	8.0	7.25
1.5	6.0	5.0	6.0	6.13	6.0	6.13	6.0	6.13	9.0	7.0	9.0	7.0	9.0	8.0
2.0	6.5	6.0	6.5	6.5	6.5	6.5	6.5	6.5	9.0	8.5	9.0	8.5	9.0	9.25
2.5	7.0	7.0	7.0	7.5	7.0	7.5	7.0	7.5	10.0	9.63	10.0	9.63	10.0	10.5
3.0	10.0	7.5	10.0	8.25	10.0	8.25	10.0	8.25	10.0	9.5	10.0	10.5	10.0	12.0
—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
4.0	12.0	9.0	12.0	10.0	12.0	10.0	12.0	10.75	12.0	11.5	12.0	12.25	12.0	14.0
5.0	14.0	10.0	14.0	11.0	14.0	11.0	14.0	13.0	14.0	13.75	14.0	14.75	14.0	16.50
6.0	14.0	11.5	14.0	12.5	14.0	12.5	14.0	14.0	14.0	15.0	—	—	—	—

Consult Factory for dimensions if meter is over 6".

Optional 1" NPT pipe connection available for Explosion-Proof electrical hook-up (FT-24 & larger).

* NPT & Flange meter has ¾" end fittings.

Refer to PMNS-772 for complete model no. information.

OPTIONS

- Special end fittings as required.
- Bi-directional flow.
- Special materials — aluminum, 316 stainless steel, 347 stainless steel, hastalloy, K-Monel, Titanium, and PVC material available.
- Special bearings — carbide, graphite, or Rulon journals.
- Pickoff coils — Inductive, reluctance, high temperature (to +750°F.), modulated rf 400°F. and explosion-proof.
- Special and additional calibrations available.

FOR ADDITIONAL INFORMATION CONTACT FTI OR YOUR FTI REPRESENTATIVE:

4250 EAST BROADWAY ROAD ■ PHOENIX, ARIZONA 85040 ■ TELEPHONE (602) 268-8776 ■ TELEX 668-344

1 JANUARY 1979

BASIC PRICES

The following prices are published for informational purposes only and should be considered as basic starting prices for FTI flowmeters and equipment, subject to change without notice. Every individual flowmeter made by FTI must be specifically tailored to its particular application -- i.e.: fluid type, fluid viscosity, flow rate, temperature, and pressure. This may (and usually does) require special bearings, pickoffs, rotors, materials, calibration, and other options that are not included in these basic prices. Only a precise, written quotation from FTI or one of our sales representatives should be considered to be binding.

STANDARD LINE TURBINE FLOWMETERS
(303 SS, ball bearings, MS/NPT ends)

¼ - inch diameter..... $ 600 up
½ - inch diameter..... $ 600 up
1 - inch diameter..... $ 570 up
1½ - inch diameter..... $ 605 up
2 - inch diameter..... $ 710 up

TURBO-PROBE INSERTION FLOWMETERS

MiniProbe..... $ 550 up
AirVelometer..... $ 575 up
Stack Sniffer..... $ 995 up
TR-1..... $1415 up
TR-3..... $1835 up
TR-5 "Low Profile"..... $3675 up

OMNIFLO TURBINE FLOWMETERS
(303 stainless steel)
$630 up

FLANGED OMNIFLO TURBINE FLOWMETERS
$790 up

UNIPROBE NO-FLO SWITCH FTU-101
$195 up

FLOCAPSULE TURBINE FLOWMETER

Model FTC-8..... $575 each
Model FTC-16..... $630 each

PICK-A-BACK TURBINE FLOWMETER

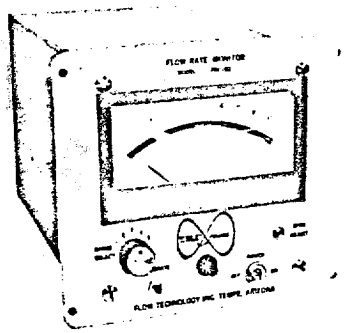
Model PB-1F..... $630 each

ELECTRONICS

Flow Rate Monitor PRI-102A..... $500 up
Pre-amplifier PRA-103..... $ 50 up
Portable Indicator PRI-1000.... $470 up
Flow Rate Monitor PRI-703..... $445 each
Range Extending Amplifier LFA-307 $155 up
Pulse Rate Converter PRC-101..... $290 each
Pulse Rate Converter PRC-407..... $350 up
Pulse Rate Converter PRC-408..... $240 up
Model 7000 Series MSP's..... $735 up

For options, special materials, or special enclosures, please consult the factory. Minimum order for spare parts or documentation is $50.00.

READOUT EQUIPMENT AND ELECTRONIC ACCESSORIES for use with TURBINE FLOWMETERS

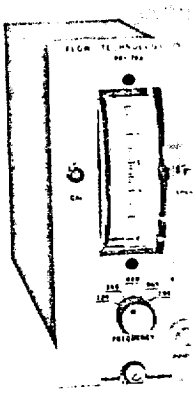


FLOW RATE MONITOR MODEL PRI-102

This electronic unit accepts the pulsed output of FTI turbine flowmeters and displays the pulse rate on a visual indicator and/or as an analog electrical signal. This pulse rate is equivalent to fluid flow rate. The PRI-102 series is available with a panel meter or with a digital readout, with analog signal output, and with power source requirements of 120v 60 hz. The unit is normally supplied in a 6" x 6" panel mount chassis. (see Bulletin PRI-102 or PRI-102FR for bi-directional capability)

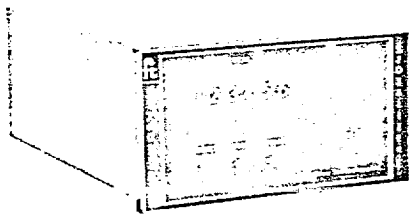
FLOW RATE MONITOR MODEL PRI-703

A rack panel mount flowmeter readout device, the PRI-703 utilizes the 60-hertz line frequency for internal calibration of its various measurement ranges. The vertical indicator meter with 2% f.s. accuracy is supplemented by a 0-5 volt 0.2% f.s. analog electrical signal output. The maximum internal calibration frequency is 1200 hz. (see Bulletin PRI-703)



SERIES 7000 MSP's MULTI-PURPOSE SIGNAL PROCESSORS

The Series 7000 MSP family feature five different models tailored to a wide variety of counting, indicating, signal processing, and signal management tasks. All models accept the pulse inputs from any turbine flowmeter or most pulse outputs from instrument sensors and control systems. They will also accept contact closure inputs. All models provide a front panel LED digital display of total pulse count or pulse rate, depending on the model.

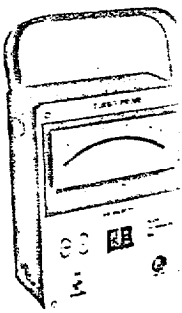


When used with turbine flowmeters, the Series 7000 Signal Processors display total fluid volume or fluid flow rate. Internal multiplier and divider circuits permit the display and output signals to be expressed in engineering units — gallons, liters, barrels, etc.

Some models feature control switching functions that can be used in various ways in modern industrial process control applications. (see Bulletin No. MSP)

FLOW RATE MONITOR MODEL PRI-1000

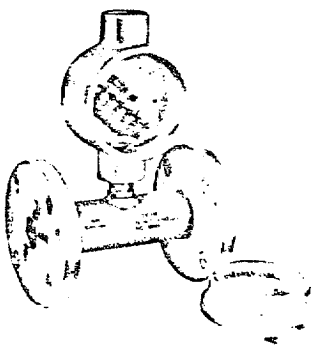
This is a portable, battery-operated electronic unit designed to accept the output of FTI "active" r-f turbine flowmeter pickoff and to present the flow rate data in analog form on a front panel meter. The PRI-1000 operates from an internal, rechargeable Ni-Cd battery pack or from a 115 VAC line. Batteries provide enough power for eight hours of operation and may be recharged from a 115 VAC line using the internal charging circuit. The PRI-1000 contains all necessary electronics in solid-state and integrated circuit form. (see Bulletin PRI-1000)



SIGNAL PRE-AMPLIFIER MODEL PRA 103X & 104

Pre-amplification of the normal low level turbine flowmeter output signal is necessary if the data is to be transmitted more than 1000 feet to the read-out equipment or if the environment is saturated with high levels of electromagnetic noise generated by electrical devices.

The Model PRA-103X/104 are solid-state, modular, electronic packages designed for the express purpose of transmitting these output signals from any Flow Technology, Inc. turbine flowmeter. The PRA-103X can transmit for a distance of one mile over land lines to remotely located read-out devices.



The PRA-103X contains all of the electronic components within an explosion-proof enclosure that can be mounted directly on the flowmeter pickoff.

The PRA-104 electronics are packaged in heavy gauge JIC enclosure with military type MS connectors or conduit fittings with terminal strip connections. This unit has adjustable gain and AC or DC power source. (see Bulletin No. PRE-AMP)

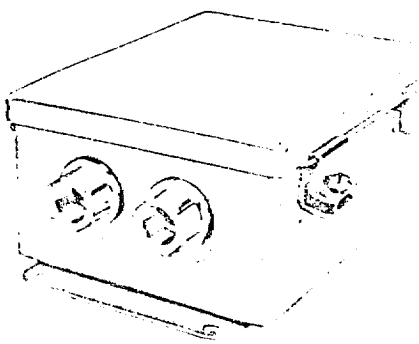
RANGE EXTENDING AMPLIFIER MODEL LFA-303/307 SERIES

This amplifier extends the operating range of the FTI t bine flowmeters to very low flow ranges — down to 0.0 gpm for the Omniflo flowmeter. It must be used with an tive" pickoff coil on the flowmeter. This combina eliminates the pickoff drag normally encountered with turb flowmeters at very low flow rates. By using the LFA-303 3 the flow range span of an Omniflo flowmeter can be extend to as much as 100:1 while the standard line turbine flowme range can be extended to as much as 250:1. The LFA-303 is available in a heavy-gauge JIC box, or an explosion-pr enclosure mounted on the flowmeter. (see Bulletin REA)

LIN-256 LINEARIZER

The LIN-256 is designed to compensate for the non-linear relation between frequency and flow rate when a flowmeter is operating at low flow rates or in fluids of high viscosity. Flowmeters operating at these low flow rates produce a K-factor that varies over the flow range for a particular meter. This K-factor can change as much as 10 to 1 over the range of the flowmeter and must be linearized by a device such as the LIN-256 before accurate totalizing can be accomplished.

The LIN-256 will correct the K-factor for a given flowmeter at 256 different points along the K-factor curve, thus producing a linear flow rate curve with a minimum amount of error over the range of the flowmeter. (see Bulletin LIN-256-781).



Recommended Equipment for testing of Copperhead well

Pressure transducer (Bell & Howell)
installed at surface
with connection to drill rod.
CEC-1000 optional (495.00)
4-424-0010 295.00

Rockwell 1" totalizing
water meter placed
in discharge line between
tank & well (From Dave Kepner) 95.00

2" valve between pump
& well —
(Probably on site)

Esterline Angus mini
servo recorder &
accessories 908.95

$ 1298.95
10% contingencies. 129.89
$ 1428.84

* will need some hard wire
btwn transducer & recorder.

~~WLK Copy~~

IN-PLACE LEACH TEST PROGRAM
SANTA CRUZ JOINT VENTURE
PINAL COUNTY, ARIZONA

by

D. H. Davidson
Science Applications, Inc.
La Jolla, California

T.D.H.
MAR 11 1980

ASARCO

IN-PLACE LEACH TEST PROGRAM
SANTA CRUZ JOINT VENTURE
PINAL COUNTY, ARIZONA

by

D. H. Davidson
Science Applications, Inc.
La Jolla, California

RECEIVED

JAN 14 1980

S. W. U. S. EXPL. DIV.

27 December 1979

Mr. F. T. Graybeal
Asarco Incorporated
1150 North 7th Avenue
Tucson, Arizona 85703

Dear Mr. Graybeal:

Enclosed is my report outlining a stepwise program to assess the feasibility of in-place leaching the Santa Cruz property.

Very truly yours,

Donald H. Davidson
Manager, Petroleum Engineering and
Solution Mining of Minerals

DHD:j

Encl. (4)

27 December 1979

To: F. T. Graybeal, Asarco Incorporated

From: D. H. Davidson, Science Applications, Inc.

Subject: Step-Wise Program to Assess In-Place Leaching of Santa Cruz Joint Venture, Pinal County, Arizona

INTRODUCTION

This report outlines a stepwise program to determine whether the Santa Cruz Joint Venture property can be in-place leached, as requested by your letter of December 10, 1979. Costs and timing contained in this report should be used for planning purposes only, as development of a detailed project plan, with alternatives, was not within the time frame of this report.

SUMMARY

1. It is estimated that a 4-year/$11 million pilot project would be associated with determining the feasibility of in-place leaching the Santa Cruz property (see Table 1 for Summary). Contrasting this with other reported in-place leach projects and future commercial expenditures:

- Occidental Minerals has reported (Silver Belt, December 13, 1979) spending $10 million to date, and is planning to spend $16 million over the next two years for in-place leaching at the Van Dyke property.

- Kennecott reported (Tucson Daily Citizen, July 22, 1975) spending $8 million over 5 years, with $4-6 million for addition of a 5-spot operation at Safford, Arizona, to develop in-place leaching.

- At $3000/AT initial investment, a 10,000 ton per year copper operation would require $30 million initial expenditure.

2. Description of 4-year pilot operation

- The first two years are concerned with:

  - Demonstrating that adequate commercial injection and production well rates can be achieved.

  - Demonstration of inter-well fluid communication and leaching.

TABLE 1

SUMMARY OF PROGRAM

FUNCTION	MONTHS									
	5	10	15	20	25	30	35	40	45	50
PHASE I ASSESS PERMEABILITY	$690K									
PHASE II INTER-WELL FLOW AND LEACHING			$690K							
PHASE III 2-SPOT LEACHING				$850K						
PHASE IV 5-SPOT LONG TERM LEACHING						$6.95 MILLION				
ENGINEERING SUPPORT	$155K 1.8 M-Yr		$143K 1.7M-Yr	$333K 5.3 M-Yr		$1.21 MILLION 16.7 MAN YR				
CUMULATIVE	$845K		$1.7M	$2.9M		$11 MILLION				
	1.8 M-Yr		3.5M-Yr	5.8 M-Yr		25.5 MAN YR				

- Obtaining operating experience with acid injection and pregnant solution recovery from large diameter wells (10")

- Design of 5-spot operation with surface facility copper recovery.

- The second two years involves the integrated operation of two 5-spot well patterns with a surface facility (Ion-exchange, electrowinning, and solution make-up). The objectives are:

  - Demonstrate long term well pattern leaching and performance.

  - Obtain operating experience and cost information.

  - Determine quality of cathode copper obtained from in-place leach solutions.

3. The 4-year program consists of four phases, with an engineering support function associated with each phase.

- Phase I - 8 months, $690K field tests, $155K engineering support. Major activities include:

  - Determine injection and production rates from 3 wells.

  - Determine permeability variations in one well; high-low grade, and percambrian vs. porphyry rock types.

  - Conduct economic sensitivity analysis to define milestones and objectives for subsequent testing.

  - Plan for subsequent testing.

- Phase II - 7 months, $690K field test, $143K engineering support. Major activities include:

  - Determine relationship of permeability to geologic structure.

  - Demonstrate inter-well fluid communication and leaching.

  - Plan for subsequent testing.

- Phase III - 8 months, $850K field tests, $335K engineering support:

  - Demonstrate 2-hole acid injection and copper recovery.

  - Obtain operating experience with large hole operation.

- Obtain laboratory data on effects of long term leaching on permeability and effluent solution chemistry.

- Plan for 5-spot operation.

- Phase IV - At least 24 months, $6.95 million field testing, $1.21 million engineering support. Major activities include:

  - Operation of two 5-spot patterns.

  - Recycle of solutions, with ion-exchange and electrowinning.

  - Demonstrate sustained leaching and flow in sub-surface.

  - Plan for commercial operation.

4. Following the two-year operation of the five-spots, continuous scale-up to either a semi-commercial operation (sale of copper offsets operating cost) or full commercial. This scale-up can be achieved by addition of well pattern and surface facility modules.

5. Is is recommended that a detail pre-test plan be developed for phase I and II prior to initiation of any test. It is estimated that this would cost between $25 to $50K and take two months.

DISCUSSION

Engineering Support (See Table 2 for Schedule)

1. Economic Analysis

- Cash flow analysis incorporating integration of geologic, well-field and surface facility process parameters.

- Define milestones for field testing based on economic requirements of commercial operation.

- Continually up-date projection of commercial performance as field data becomes available.

2. Laboratory Core Leaching

- Measure effects of acid reactivity on copper and gangue leaching and permeability over time.

- Assess variations in core leachability at different sections of the deposit.

- Assess effect of recyling pregnant liquor through rock.

TABLE 2

ENGINEERING SUPPORT

1.84 MILLION 4-YEARS and 25.5 MAN-YEARS

ACTIVITY	MONTHS																							
	PHASE I			PHASE II				PHASE III				PHASE IV												
	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30	32	34	36	38	40	42	44	46	48
1. CASH FLOW ANALYSIS																								
• DEVELOP MODEL	$20K																							
• MILESTONE ANALYSIS		$20K																						
• UP-DATING			$10K .2 MAN YR		$17.5K .3 MAN YR				$17.5K .3MAN YR								$120K 2 MAN YR							
2. CORE TESTING																								
• CONSTRUCT FACILITY							$25K																	
• PROVIDE DATA									$40K - .7 MAN YR $25K SERVICES								$240K - 4 MAN YR $100K - EQUIPMENT AND SERVICES							
3. PLANNING AND ANALYSIS FIELD TESTS	$40K - 0.8 MAN YR				$35K - 0.6 MAN YR				$65K - 1.1 MAN YR								$240K - 4 MAN YR							
4. DESIGN PILOT PLANT									$80K - 1.6 MAN YR				$25K - .4 MAN YR											
5. GEOLOGY AND ENVIRONMENTAL	$40K - 0.8 MAN YR $25K - EQUIPMENT				$40K - 0.8 MAN YR $25K - EQUIPMENT				$80K - 1.6 MAN YR $25K - EQUIPMENT								$240K - 4 MAN YR $100K - EQUIPMENT							
6. COMMERICAL PLANT DESIGN																	$140K - 2.3 MAN YR							

1 Man Year Equivalent to $60K

3. Geological and Environmental Analysis

- Develop structural and mineralogical maps.
- Develop copper reserve information in well pattern areas.
- Core and analyze leached areas.
- Assess environmental requirements.
- Collect hydrological and chemical data and samples for environmental reporting.

4. Planning and Analysis of Field Tests

- Develop detail plan for field tests.
- Specify requirements.
- Supervise wellfield operations.
- Gather and analyze all data.
- Report results of field tests with economic significance.

5. Design Pilot Plant Facilities

- Provide engineering design for 5-spot.
- Develop schedule and order equipment.
- Supervise installation of all equipment.
- Operate surface facilities.

6. Commercial Plant Design

- Develop schedules and designs for commercial operations based on all available information.

PHASE I - ASSESSMENT OF DEPOSIT PERMEABILITY (Table 3)

1. Objectives

- Demonstrate that well rates are above minimum levels associated with commercial requirements (probably 50 gpm minimum level).
- Assess permeability variability: location, grade and rock type.
- Determine effect of acid strength on well productivity.
- Assess effect of hydraulic-fracturing on well injection and production rates, if necessary.
- Develop correlations between geologic logging and permeability.

2. Plan

- Drill 3 core holes using bentonite mud (see Figure 1). These holes should be located within or near an existing core hole. In phase II, all core holes will be used as pressure and chemical monitor holes during injection in large 10" well.
- Tests in holes 1 and 2 -
  - Measure gpm vs. pressure drop for injection and production.
  - Acidize with 15% HCl
  - Repeat injection tests.
- Test in hole 3 -
  - Measure gpm vs. pressure drop gross interval on injection.
  - Measure gpm vs. pressure drop for zones of:
    * high grade
    * low grade
    * Porphyry rock type
    * Precambrian rock type
  - Acidize gross interval and measure permeability.

TABLE 3

PHASE I - ASSESSMENT OF DEPOSIT PERMEABILITY

$690K.

8 MONTHS

ACTIVITY	MONTHS							
	1	2	3	4	5	6	7	8
1. ORDER AND ASSEMBLE ALL FLOW TEST EQUIPMENT	$150K							
2. DRILL 3 CORE HOLES	$90-$150K							
3. TEST HOLES 1 AND 2 GROSS INTERVAL gpm vs. PRODUCTION AND INJECTION			$120K					
4. TEST HOLE 3 gpm vs. • HIGH GRADE • LOW GRADE • PRECAMBRIAN • PORPHYRY					$60K			
5. TEST HOLE 3 gpm PRODUCTION vs. ACID STRENGTH						$60K		
6. HYDRAULIC FRACTURING AND PROPPING 3 HOLES							$150K	

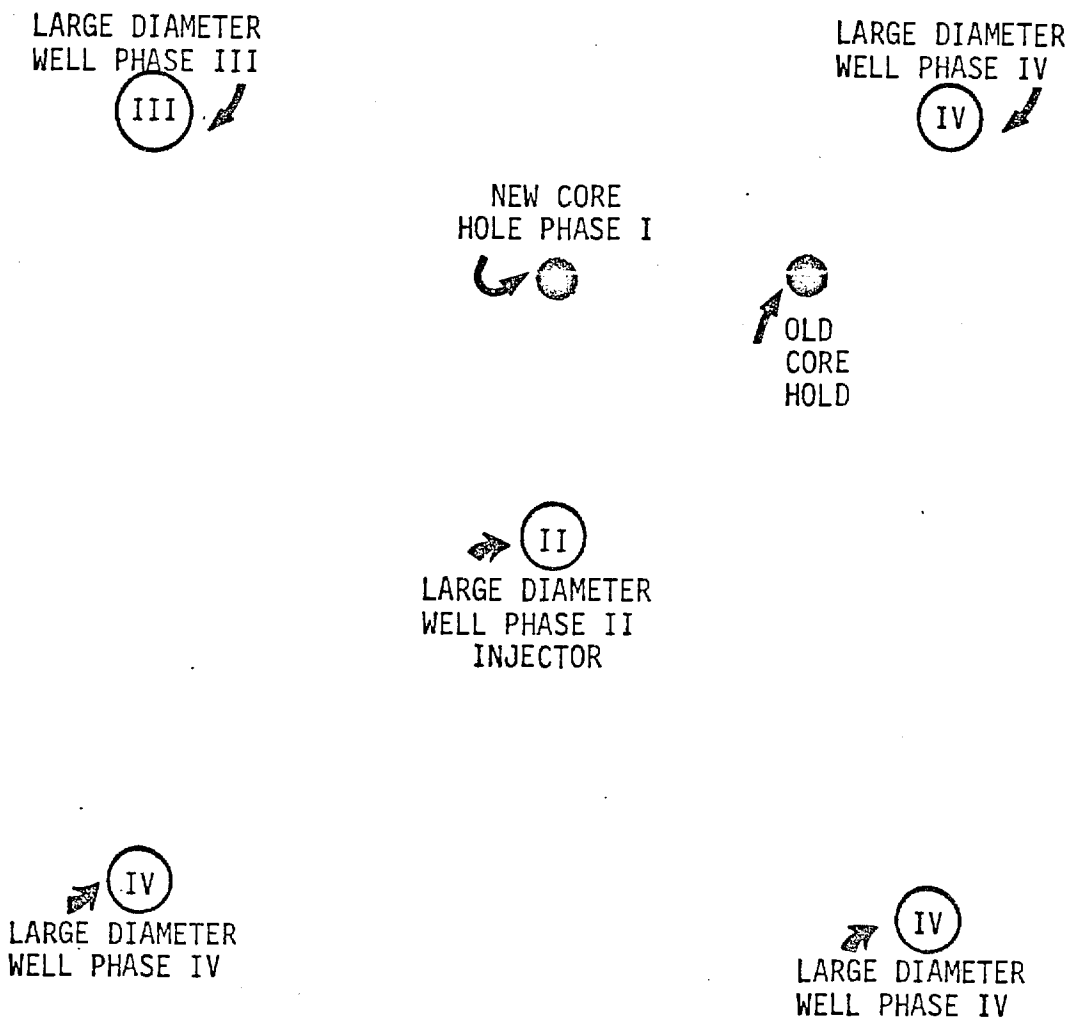


FIGURE 1 - DEVELOPMENT OF 5-SPOT FOR PILOT OPERATION

- Test in hole 3 - Production gpm versus acid strength. $CaSO_4$ may limit well productivity when leaching with H_2SO_4. In a production well the pressure drop around the wellbore can induce gypsum precipitation in the rock, as gypsum is less soluble at low pressure. This effect is more likely at higher acid strengths (high sulfate). To determine whether production well impairment by gypsum is likely, and if so, at what sulfate level, the following test sequence is recommended:

  - Test production rate of formation water.

  - Inject 10 gpl H_2SO_4 and test production rate.

  - Inject 20 gpl H_2SO_4 and test production rate.

  - Inject 40 gpl H_2SO_4 and test production rate.

- Hydraulic fracturing 3 holes. It may be desirable to evaluate the increase in flow obtained from wells that are hydraulically fractured and propped.

3. Cost Estimates

   - Core holes $30K-$50K each.

   - Testing cost $2000 per day.

PHASE II - ASSESSMENT OF INTER-WELL FLOW COMMUNICATION AND LEACHING (Table 4)

1. Objectives

   - Obtain data on the variation of permeability with structure or direction.

   - Demonstrate chemical movement and copper leaching between wells.

2. Plan

   - Drill and complete two large diameter wells, 10 inch (see Figure 1). The wells will serve as injectors, if the patterns are subsequently expanded to 2-hole then 5-spot patterns.

   - Directional permeability measurements

     - Inject in 10" well, measure fluid rise in both core holes.

TABLE 4

PHASE II - ASSESSMENT OF INTER-WELL FLOW COMMUNICATION AND LEACHING

$690 K

7 MONTHS

ACTIVITY	MONTHS						
	1	2	3	4	5	6	7
1. ORDER EQUIPMENT AND ASSEMBLE	$100K						
2. DRILL AND COMPLETE TWO 10" DIAMETER WELLS	$300K						
3. WATER INJECTION DIRECTIONAL PERMEABILITY TESTING				$60K			
4. ACID AND TRACER INJECTION AND SAMPLING DOWNHOLE IN CORE HOLES					$180K		
5. CHEMICAL ANALYSIS AND OTHER SERVICES					$50K		

- Inject in new core hole, measure fluid rise in old core hole and 10" well.

- Demonstrate inter-well and chemical communication and leaching of copper.

  - Inject H_2SO_4 and NaCl into 10" well, collect downhole samples in core holes.

  - By not producing, preclude necessity of handling pregnant liquor.

3. Cost estimates

   - 10" wells, $150K @ 2000 feet

   - Testing costs @ $2000 per day.

PHASE III - DEMONSTRATION OF COPPER RECOVERY (Table 5)

1. Objectives

   - Demonstrate that with continuous acid injection and fluid production that copper loadings can be achieved and maintained.

   - Obtain assessment of solution recovery.

   - Obtain operating experience with corrosive fluids in large diameter wells.

2. Plan

   - Drill and complete two additional large diameter wells (see Figure 1). These wells will serve as production wells if each pattern is expanded to a five-spot.

   - Operate the 10" wells as two 2-spot patterns, monitoring solutions in the core holes. The system should be operated at equal rates of injection and production. Tracer recovery will provide data on solution recovery.

   - A water well must be provided for water source for acid make-up, as produced solutions will not be reprocessed. Technical effort should be concentrated on sub-surface operation, and not on surface processing of pregnant liquor.

   - Two additional exploration holes should be drilled to acquire geologic data required for future well pattern drilling.

TABLE 5

PHASE III - DEMONSTRATION OF COPPER RECOVERY FROM SUB-SURFACE

$850 K

8 MONTHS

ACTIVITY	MONTHS							
	1	2	3	4	5	6	7	8
1. DRILL TWO ADDITIONAL 10" WELLS AND COMPLETE	$300K							
2. 2-SPOT LEACHING IN TWO PATTERNS				$450K				
3. DRILL TWO ADDITIONAL EXPLORATION HOLES FOR FUTURE WELL PATTERN DEVELOPMENT	$100K							

3. Cost Estimates

- 10" wells, $150K @ 2000 feet
- Core holes, $50K @ 2000 feet
- Testing costs @ $3000 per day.

PHASE IV - DEMONSTRATION OF LONG-TERM WELL PATTERN PERFORMANCE (Table 6)

1. Objectives

- Demonstrate that copper leaching and flow rates in five-spot pattern can be sustained over 2-year time frame.
- Obtain data on operating costs and operability for design of commercial operation.
- Demonstrate integrated performance of sub-surface and surface extraction facilities.
- Assess the quality of cathode copper obtainable from pregnant liquors obtained by in-space leaching.
- Develop plans for commercial operation.

2. Plan

- Install surface pilot facility to process 50-100 gpm flow
  - Solution make-up
  - Ion-exchange
  - Electrowinning
- Drill six additional 10" holes to complete two 5-spot patterns,(see Figure 1). Table 6 indicates alternatives involving 5-spot operations.
- Drill three additional core holes to develop geologic data necessary for further well pattern development.
- Operate system for up to two years.

3. Cost Estimates

- 10" wells, $150K @ 2000 feet.
- Core holes, $50K @ 2000 feet.
- Testing costs @ $4000 per day.

TABLE 6

PHASE IV - DEMONSTRATION OF LONG TERM WELL PATTERN PERFORMANCE

$6.95 MILLION

2-1/2 YEARS

ACTIVITY	MONTHS														
	2	4	6	8	10	12	14	16	18	20	22	24	26	28	30
1. INSTALL SURFACE FACILITY	$2 MILLION														
2. DRILL AND COMPLETE 3-10" WELLS 1st FIVE SPOT	$450K														
3. 5-SPOT LEACHING				$2.9 MILLION											
4. DRILL AND COMPLETE 3-10" WELLS 2nd FIVE SPOT					$450K										
5. EQUIPMENT	$1 MILLION														
6. ADDITIONAL EXPLORATION DRILLING	$150K														

TABLE 7

COMPARISON OF DIFFERENT MODES OF 5-SPOT OPERATION

SPACING OF SQUARE FEET	INTERVAL OF INJECTION FEET	FLUID* GALLONS IN 5-SPOT MILLIONS	TONS** COPPER IN 5-SPOT	TIME*** TO DISPLACE FLUID IN 5-SPOT	TIME**** TO LEACH OUT COPPER
150	650	5.5	5900	76 days	10.7 yr
150	100	0.85	908	11.7 days	1.65 yr
100	650	2.4	2622	33.8 days	4.8 yr
100	100	0.37	403	5.2 days	0.74 yr

* 5% porosity

** 0.5% copper grade

*** 50 gpm

**** 5 gpl copper

~~WLK Copy~~

ASSESSMENT OF FEASIBILITY IN-PLACE SOLUTION MINING SANTA CRUZ JOINT VENTURE PINAL COUNTY, ARIZONA

by

D. H. Davidson
Science Applications, Inc.
La Jolla, California

ASARCO

ASSESSMENT OF FEASIBILITY IN-PLACE SOLUTION MINING SANTA CRUZ JOINT VENTURE PINAL COUNTY, ARIZONA

by

D. H. Davidson
Science Applications, Inc.
La Jolla, California

21 December 1979

Mr. F. T. Graybeal
Asarco Incorporated
1150 North 7th Avenue
Tucson, Arizona 85703

Dear Mr. Graybeal:

Enclosed is my report assessing the feasibility of in-place leaching the Santa Cruz Joint Venture Property, as per your letter request of December 10, 1979. Based on the current data, I recommend this property as a candidate for in-situ leaching. A second report outlining a stepwise program to determine whether this deposit can be leached in place is currently being developed.

Very truly yours,

Donald H. Davidson
Manager, Petroleum Engineering and
Solution Mining of Minerals

DHD:j

Encl.

19 December 1979

To: F. T. Graybeal

From: D. H. Davidson

Subject: Assessment of Feasibility of In-place Solution Mining Santa Cruz Joint Venture, Pinal County, Arizona

INTRODUCTION

This report discusses the feasibility of in-place leaching of the Santa Cruz Joint Venture property, per your request in letter dated December 10, 1979. A second report outlining a step-wise program to determine whether the deposit can be commercial in-place leached is being prepared.

SUMMARY

At this time I would recommend that the Santa Cruz Joint Venture be considered as a candidate for in-place leaching. This recommendation is based on a review of the current Santa Cruz data base, my prior experience with technical-economic evaluations of in-situ copper operations, and available literature on other in-situ copper projects. A significant short term milestone for consideration would be demonstrating that well flow rates in excess of 45 gpm can be achieved. It is very likely that a minimum expenditure of $7 million over two years will be required to demonstrate that a commercial operation can be justified. As you requested, I will be providing a second report outlining a step-wise approach to determining commercial feasibility of in-place leaching.

Summary of rationale used in this evaluation:

1. Size of Ore-Body

   A. Estimates of copper reserves indicate that a commercial operation in excess of 10,000 tons per year copper for 20 years could be easily supported. This is important because it is not likely that a return in investment exceeding 15% could be achieved at production capacities below 10,000 TPY of copper, no matter how favorable the sub-surface performance would be.

B. The Santa Cruz property appears to have six times more copper than Occidental's Van Dyke project (100 million tons of ore at 0.5%). Oxy has indicated it would spend an additional $16 million[1] over the next two years if a permit to operate is granted by the town of Miami.

2. Leachability of Ore with Acid

A. Leaching tests[2] on representative ore samples yielded recoveries of copper over 75% using H_2SO4. This demonstrates that no fundamental chemical problems exist with respect to solubilizing the non-sulphide minerals.

B. Acid consumption of ore containing 0.5 copper grade was less than 6 lbs H_2SO4 per lb. copper. This is equivalent to 9¢/lb. copper raw materials operating cost. A detail USERDA[3] study on in-situ copper leaching (1975) indicated a return on investment of 18% was possible at raw materials costs of 8¢/lb. copper.

C. The 6 lb. H_2SO_4/lb. Cu value corresponds to three times more acid attack of gangue than copper. This is not excessively high, considering that ore was crushed to 10 mesh, exposing more matrix gangue than would likely see acid under in-situ conditions.

3. Core Data

A. An analysis of core[4] from the exploration holes indicates that over 50% of the copper lies within 2-1/2 inches of a broken fracture (4.8 B.F.F. per foot). This suggests that copper is near flow fractures and could even be recovered by diffusion leaching. Copper recoveries of 50% in one year from 2-inch rock fragments containing chalcopyrite was reported in USERDA[3] NVO-155. Acid leachable copper can be expected to be recovered more rapidly at comparable fracture separations.

B. Although a correlation has been established between lower grades and the degree to which core is broken, my experience with field and lab evaluation of permeability does not indicate that it correlates with core integrity or recovery.

4. Ore-Body Geometry

A. The fact that the mineralization exists over distances in excess of several hundred feet[5], and is 950 feet below the water table, signifies a high probability that fluid can be pumped at adequate rates from rock that is likely to be low in permeability. The long interval reduces the flow velocity, the 950 feet of water head provides the driving force to force fluid into the production well.

B. Preliminary calculations indicate that well rates in excess of 45 gpm are required for a commercial operation. Permeabilities in the 5 md range are required to achieve this flow for the geometry of the Santa Cruz mineralized zone. Kennecott patents 3,917,345 and 4,116,488 indicate that permeabilities ranging from 2.5 to 25 md were obtained at Safford in zones containing sulphide copper. It is not likely that permeability of rock containing acid leachable non-sulphide copper mineralization will be less than the data reported by Kennecott.

As you requested in your December 10, 1979 letter (copy attached), I am preparing a second report outlining a stepwise program to determine whether the Santa Cruz property can be commercially in-place leached. At this time I would anticipate that an absolute minimum program would require expenditures of $7 million over a two year period. The reason for this is rooted in the economic demands placed on a sub-surface commercial operation. The cash flow is such that extra well pattern additions within five years of start-up significantly reduce economic profitability. Before proceeding with large expenditures associated with a commercial surface facility, it is well advised that well pattern performance with respect to loading and flow rate be demonstrated for at least two years. A single 5-spot operation at 50 gpm

and 5 gpl copper (500 tons per year copper production), with surface copper recovery and solution make-up, is likely to require up to $7 million, including:

- Five wells - $1 million total
- Surface facility - $1-2 million
- Extra equipment - $1 million
- $4000/day operating - $3 million for two years expense.

If the copper could be sold for 50¢/lb., conceivably $1 million could be recovered. Clearly, even at $1 per lb, this feasibility evaluation project could not pay for itself.

DISCUSSION

The analysis of the feasibility of in-lace leaching the Santa Cruz Joint Venture Property is divided into the following sections for discussion:

- Magnitude of copper resource
- Acid leach data
- Core analysis
- Ore-body geometry and relationship to well production potential
- Literature information concerning other in-situ copper operations.

Magnitude of Copper Resource

Having a large copper resource that can supply production capacities in excess of 10,000 tons per year (TPY) copper for 20 years is important. As in most commercial operations economy of scale-up must be considered and this is especially pertinent for in-situ copper leaching. The surface facilities (ion-exchange or solvent extraction, electrowinning, offsites, solution make-up) are likely to represent over 60% of the total capital investment. The unit cost of this investment declines with increasing production capacity, leaving additional funds for the more uncertain sub-surface operation. In addition, once a surface facility has been built, copper reserves should be large enough to insure that the facility operates at design capacity.

Table 1 compares the estimated copper resource in a number of locations at Santa Cruz with production rates above 10,000 TPY at 50% copper recovery. Clearly, the Santa Cruz deposit has the potential of supplying very high production capacities.

- Even the minimum tonnage associated with the triangular area in block B could sustain a 10,000 TPY operation for over 30 years at 50% copper recovery. Production capacities as high as 100,000 TPY are a possibility.

- The 640,000 tons of copper exceeds the reported reserve for the Van Dyke Occidental Project, where 500,000 tons of copper has been reported, and Oxy Min appears willing to spend an additional $16 million over the next 2 years for commercial development. The Oxy sub-surface costs are likely to exceed that of an operation at Santa Cruz since the drilling and well operations will be conducted from underground workings due to constraints associated with the deposit being located under the town of Miami.

To illustrate quantitatively the sensitivity of production capacity to profitability, reference is made to the USERDA Report NVO-155 (Table 2 for summary of conditions), "In-Situ Leaching of a Nuclear Rubblized Copper Ore Body," June 1975. This report represents a co-operative study between Kennecott Copper, AEC and Lawrence Livermore Laboratory. The report provides a detail economic sensitivity analysis for combined surface facility and sub-surface operation. Although the ore-body geometry and grade is hypothetical, it is similar to the Santa Cruz property. Costs of nuclear components are minor relative to well and surface facilities. A base case of 40,000 TPY copper production results in an 18% return on investment. Reducing the capacity to 12,700 TPY lowers the ROI to below 9%. Other aspects of this USERDA report will be cited at other parts of this discussion.

Another illustration of the impact of production capacity on unit investment costs is obtained by comparing the electrowinning capital investment at Cities Service with that reported in USERDA study (Table 3). Correcting for inflation (10% per year), an apparent 75% reduction in investment per unit production is likely in going from 6,000 TPY to 42,300 TPY. Don Crowell of Asarco indicated that he was aware of an electrowinning installation in 1974 with 35,000 TPY copper at $15 million ($428/AT) which is comparable to the USERDA value.

Table 1.

Comparison of Copper Resource with Production Capacity Demand

Area	Tons Rock (Million)	Grade (% Copper)	Tons Cu	TPY Cu Production	% Recovery	Years
Triangle Apex Block	125	0.51	640,000	10,000	50	34
B holes	125	0.51	640,000	25,000	50	13 1/2
SC-51, -52, -39	125	0.51	640,000	40,000	50	8 1/2
All B	380	0.42	1,600,000	40,000	50	20
All C	250	0.53	1,330,000	40,000	50	16 1/2
All reserves	880	0.43	3,800,000	100,000	50	19

Table 2.

Summary of USERDA REPORT

TECHNICAL AND ECONOMIC PARAMETERS

1. Geology

   - Depth to bottom of mineralization, 2770 feet
   - Depth to top of ore-zone, 1500 feet
   - Area of mineralization - 180 acres
   - Average ore-grade - 0.5% copper
   - Water table - 300 feet below surface

2. Base case design criteria

   - 42,300 7PY copper production
   - 25 year mine life
   - Average copper loading 4.6 g pl

3. Economic Parameters (1974 dollars)

   - Surface Facilities - $943/AT
     - o Offoites (non-oxygen) - $177/AT
     - o Liquid storage and pumping - $123/AT
     - o Liquid IX - $156/AT
     - o Organic Inventory - $95/AT
     - o Electro winning - $392/AT
   - Wells (non-nuclear related) - initial investment $240/AT, with 7 additions every 3 years total present worth of sub-surface investment $590/AT
   - Drilling and completion for igneous rock $63/foot of total depth
   - Operating cost, 31¢/lb copper including 8¢/lb raw materials
   - Copper selling price, 70¢/lb copper

4. 18% ROI base case

Table 3.

Comparison of Electrowinning Capitol Investment

Source	Copper Production Capacity TPY	$/AT	$/AT Adjusted in Time
USERDA REPORT	42,300	392 (1974 Dollars)	474 (1976 Dollars 10% Inflation)
Cities Service	6,000	833 (1976 Dollars)	833 (1976 Dollars)

Acid Leach Data

Table 4 summarizes Asarco's data relating to H_2SO_4 leach of Santa Cruz ore. Sample variations included: ore grade (.5 to 2.6%), rock type (Precambrian Granite and Porphyry), type of copper mineralization (copper oxide and chalcocite). The significant results are:

- Aside from the chalcocite ore, copper recoveries using H_2SO_4 exceed 75%. Thus, there is no fundamental chemical limitation to achieving copper recoveries above 50%.
- The amount of acid required to leach copper is less than 6 lb H_2SO_4/lb of copper. At $30/ton for sulfuric acid, this translates into less than 9¢/lb copper raw materials cost. The raw materials cost in the USERDA report was 8¢/lb (1974 $), for the base case 18% return on investment.
- At 6 lb acid/lb copper, a copper loading of 5 gpl can be obtained with 30 gpl H_2SO_4. Attaining high copper loadings with low an acid strength is important because of the presence of sulfate. Calcium release from gangue will eventually lead to $CaSO_4$ saturation. Surface facilities will have to accommodate

some means of removing this saturation to prevent undue gypsum formation in the sub-surface. The higher the initial acid strength, the more pronounced this operation becomes. In addition, due to continual acid make-up, which involves both H^+ and $SO_4^=$ addition, a sulfate level could be reached where copper sulfate saturation is a controlling factor in copper recovery. Keeping the initial sulfate level down is an important design consideration.

- Actual in-situ acid consumptions could be lower, as the 10 mesh fragment size exposes more gangue relative to copper than is likely to occur in the in-situ leach.

- Acid consumption appears to correlate with grade (see Figure 1), increasing with decreasing grade. At 2.7%, 1.5 lb acid/lb copper is consistent with the minimum stoichiometry required to leach copper oxide with almost no acid consumption associated with gangue. At grades in the 0.25% range, acid consumption in excess of 10 lb/lb is not anticipated. At acid consumptions above 10 lb/lb copper, significant reduction in profitability would result. Since the Asarco data does not indicate that 10 lb/lb copper is likely at grades in the 0.25% range, recovery of copper from zones below 0.5% average grade can be expected at reasonable raw materials cost.

Table 4.

Summary of Acid Consumption

Sample	Hole and Interval	gpl H_2SO_4 Used	lb H_2SO_4 Ton of Rock	%Cu Initially	%Cu Leached With Acid	gH_2SO_4 Used gCu Leached	% Initial Assay Leached	Comments:
1	SC-19	30	37.8	0.58	0.481	3.1	82.9	63%Chrysocolla
	1205'-1460'	50	44.4	0.58	0.491	4.5	84.7	37%Cuo
2	SC-19	30	79.2	2.70	2.62	1.51	97.0	6% Chrysocolla
	1620'-1770'	50	79.8	2.67	2.60	1.53	97.4	94%Cuo
3	SC-24	30	46.8	0.75	0.632	3.71	84.2	77%Chrysocolla 14%Cuo
	1550'-1820'	50	55.8	0.75	0.642	4.34	85.6	9% Chalcocite
4	SC-46	30	55.2	1.38	1.26	2.19	91.3	26%Chrysocolla
	1570'-1706'	50	61.8	1.53	1.42	2.18	92.8	74%Cuo
5	SC-46	30	34.8	0.52	0.385	4.52	74.0	92%Chrysocolla
	1706'-1900'	50	44.4	0.51	0.383	5.80	75.1	8%Cuo
6	SC-46	30	28.8	0.80	0.230	6.26	28.8	98%Chalcocite
	1900'-2040'	50	29.4	0.79	0.260	5.65	32.9	

Samples from SC-46 1570'-1706' and 1706'-1900' reported to have produced a slime.

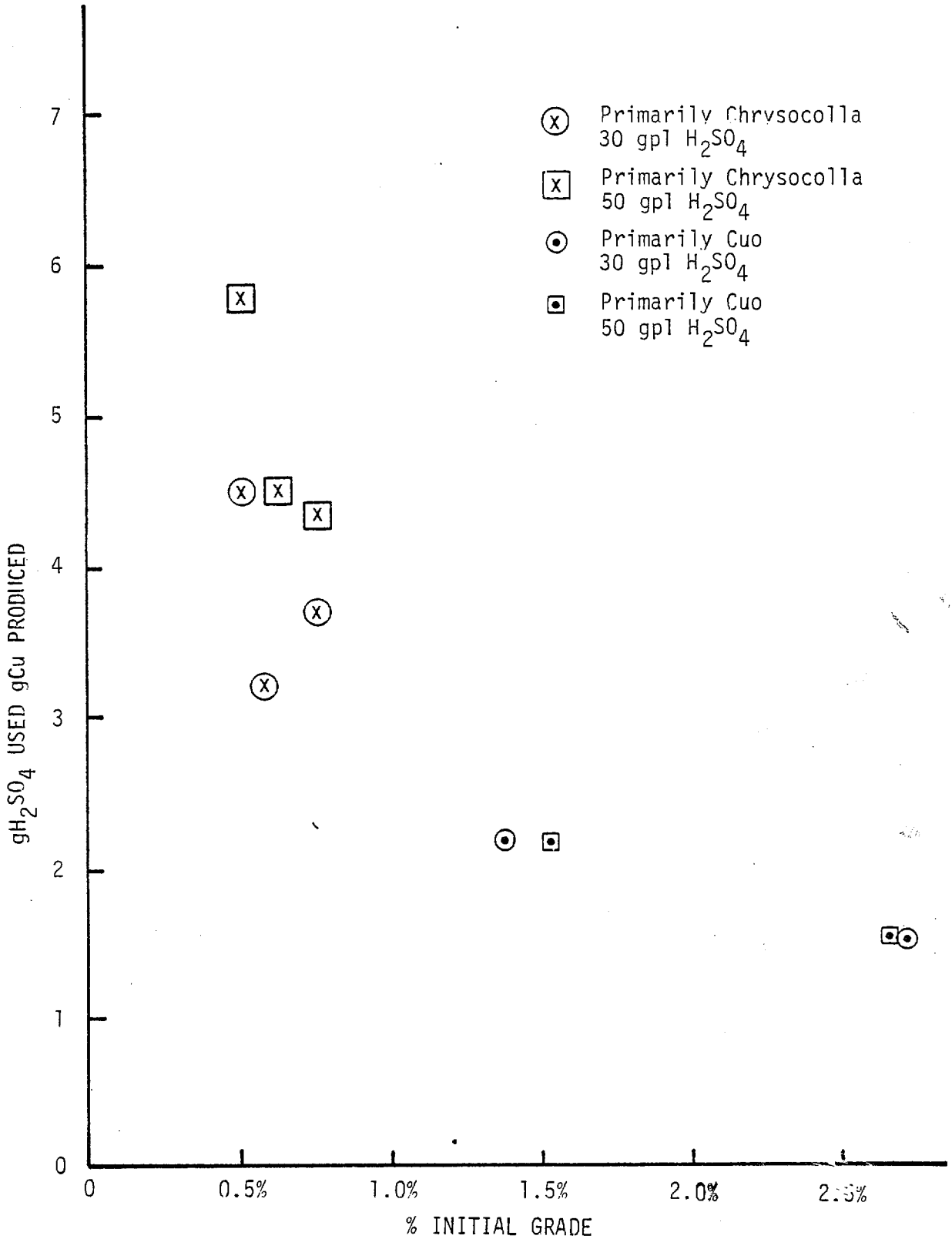


Figure 1. Acid Consumption Vs. Grade

Core Analysis

Kreis's analysis of drill core indicates that over 50% of the copper is associated with more than 4.8 broken fractures per foot. One interpretation of this is that copper mineralization lies within 2 1/2 inches of a fracture. If flow is principally in the fractures, leaching of copper could be limited by diffusion of acid from the fractures to copper mineralization. Laboratory data on copper recovery from fragments of sulphide ore leach, indicates that over 50% of the more difficult to leach sulphide mineral is obtained in less than 1 year with fragment sizes under 2 inches. Thus, even if leaching is diffusion controlled the close proximity of Santa Cruz oxide copper to fractures is likely to lead to efficient copper recovery.

Kreis's analysis also indicated that rock integrity, as represented by broken fractures and size of recovered core, is less at locations of low grade. This suggests that fracture intensity is more intense in the lower grade zones, and raises concern regarding the possibility that zones of high flow capacity may be those of low grade. Prior efforts that I am aware of to correlate the quality of core recovery with in-situ permeability measurements have not resulted in substantiation of a positive correlation. The conduit sizes in rock that result in flow resistances comparable to the 10md are of the order of 10 microns, less than 1/5 the size of a human hair. A rule of thumb is that aperatures in the rock that can be seen by the eye are not likely to correspond to the location resisting flow at the md level of permeability. Even for sandstone's of several hundred md pore-sizes are the size of human hairs.

Ore-body Geometry and Relationship to Well Production Potential

The Santa Cruz deposit has mineralization distributed over long intervals (in excess of several hundred feet), with the top of the ore zone lying over 500 feet below the water table. Both of these conditions increase the probability of achieving production well flow rates required for a commercial operation in a low permeability hard rock deposit.

Three factors that control the rate a fluid movement into and out of porous rock are: the permeability (conductivity of the rock, K or md), the interval of flow (in this case mineralization, H feet), and the pressure drop driving the flow ($\Delta\rho$). The gpm per well is proportional to the product of [KH Δp], and can be calculated using equation (1):

$$gpm = 0.00003 \times KxHx \Delta p \quad (1)$$

The rate of flow obtained from a production well will probably be the limiting flow parameter affecting well pattern operations. High rates of injection can always be achieved by applying high surface pressures and fracturing the formation. This may not be desirable from a flow control and operating cost standpoint, but it is attainable. A production well relies on the hydrostatic pressure in the mineralized zone to force fluid to the sump created around a production well by a down hole pump. This pump reduces the well bore pressure and lifts the fluid to the surface. On the average, the 640,000 tons of copper mapped out by Kreis lie 1450 feet below the surface. Assuming a water table of 500 feet, 950 feet of fluid head (411 psi) are available for Δp to the production well, when the fluid level in the bottom of the well is drawn down to atmospheric pressure. In contrast, downhole fracturing pressures ranging from 1450 psi (1 psi per foot) to 1015 psi (0.7 psi per foot) can be used to inject fluid. The corresponding injection pressure drops are 994 psi and 604 psi, 2.4 to 1.5 times greater than the Δp for a production well.

Data contained in the economics section USERDA Report NVO-155 can be used to estimate a target gpm well flow rate for the Santa Cruz deposit. Referring to Table 2, at the base case of 18% ROI, the present worth of all injection and production wells is $590/AT. The gpm per well is computed from the total sub-surface investment and the cost per foot of depth of a completed well, $63/foot in this case (equation 2). Although

$$gpm = \frac{(a)(C_D)(D)}{(2)(Cu)(\$/AT \text{ for wells})} \qquad (2)$$

a = average number of wells per pattern

C_D= \$/foot completed well cost

Cu= gpl copper loading

C_D and \$/AT are in 1974 dollars, the ratio is not anticipated to be sensitive to inflation. Assuming 2 wells per pattern and a depth of 2150 feet (average depth for 640,000 tons identified by Kreis), a gpm per well of 45 is calculated. A KH value obtained from equation (1) for 45 gpm and 411 psi Δp is 3650 md-feet. Assuming an average mineralized zone of 695 feet (as reported by Kreis) 5 md permeability is required.

Evidence of permeabilities in the 1 to 25 md range for igneous rock formations can be found in the patents assigned to Kennecott Copper:

- U.S. patent 3,917,345 - column 5 between lines 50 and 60. It is stated that 25 md permeability has been measured in laboratory tests on cores as well as in field tests in holes in igneous rock containing chalcopyrite. Additional data of interest; void space (porosity) is 2%, and fracture spacings are of the order of 6 inches. The later is in line with some of Kreis's observations of broken core at Santa Cruz.

- U.S. patent 4,116,488
  - Column 16, bottom of page. Field test over 70 feet of hole at Kennecott's Safford property yielded 15 gpm, which translates into 2.5 md for the particular test conditions (1050 md-feet).
  - Column 14, Table I identifies flow rates of 40-240 gpm, permeabilities 0.6 to 8.4 md.
  - Column 17, example I states that a communication test involving copper production was carried out successfully over a 70 foot well separation using a 100 foot long zone of mineralization at the Safford, Arizona deposit in 1975.

The Kennecott data was obtained in sulphide ore, and it is very likely that the Santa Cruz oxide ore would have comparable or higher permeabilities and inter-well flow communication. Visual observations of the Santa Cruz core would be in line with this assumption.

Literature Information Concerning Other In-situ Copper Operations

- o Occidental Minerals Van Dyke Project
  - Reported in Silver Belt, December 13, 1979
    - + $10 million expanded since 1968
    - + $16 million planned for next 2 years
    - + $360 million in capital planned for 15 year commercial operation
  - Report to Town Council of Miami, Arizona, September 12, 1977
    - + 100 million tons of ore at 0.5% grade, or 500,000 tons of copper.

Assuming a 50% recovery, a 15 year operation translates into 17,000 TPY copper production capacity.

- o Kennecott Copper Safford Project, reported in the Tucson Daily Citizen, Tuesday, July 22, 1975. A copy of the article is attached to the report. Significant data:
  - $1 million per well at 5000 feet (This is high by a factor of 2)
  - 2 billion tons of ore at 0.4%, 8 million tons of copper
  - $8 million spent prior to July 1975
  - $4 to $6 million cost of next step, a pilot five-spot
  - Technology could be used for both copper oxides and sulphides
  - Solution is expected to penetrate the same cracks, fissues and faults where copper mineral was deposited millions of years ago.
- o Also attached are copies of the Kennecott patents.

REFERENCES

1. Silver Belt, Thursday, December 13, 1979.
2. "Acid Leach Tests on Santa Cruz Project Copper Ores - July 1978," by T. D. Henderson, December 11, 1979.
3. USERDA Report NVO-155, June 1975, "In-Situ Leaching of a Nuclear Rubblized Copper Ore Body," Vol I & II.
4. "Fracturing of the Possible Copper Reserve, The Lands, Santa Cruz Project," by H. G. Kreis, August 21, 1978.
5. "Possible Copper Reserve for Bulk Mining Consideration, the Lands, Santa Cruz Project," by H. G. Kreis, July 17, 1978.

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PRELIMINARY FEASIBILITY REPORT

ON

IN SITU MINING METHODS

ASARCO'S SANTA CRUZ PROJECT

PINAL COUNTY, ARIZONA

PRELIMINARY FEASIBILITY REPORT

ON

IN SITU MINING METHODS

ASARCO'S SANTA CRUZ PROJECT

PINAL COUNTY, ARIZONA

For

ASARCO, INCORPORATED
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TABLE OF CONTENTS

SECTION

1. INTRODUCTION

2. CRITERIA AND DESIGN BASIS

3. SUMMARY AND RECOMMENDATIONS

3.1 GENERAL

3.2 CAPITAL COST

3.3 OPERATING COST

3.4 RECOMMENDATIONS

4. VOID HOLE FRAGMENTATION IN SITU LEACH SYSTEM

4.1 DESCRIPTION

4.2 CAPITAL COSTS

4.3 OPERATING COSTS

5. BLOCK CAVE IN SITU LEACH SYSTEM

5.1 DESCRIPTION

5.2 CAPITAL COSTS

5.3 OPERATING COSTS

6. PROCESS PLANT

6.1 DESCRIPTION

6.2 CAPITAL COSTS

6.3 OPERATING COSTS

7. APPENDIX

SECTION 1

INTRODUCTION

Mountain States Research and Development was authorized by Asarco, Incorporated to evaluate in situ methods of copper recovery for application to their Santa Cruz Project, Pinal County, Arizona.

As a basis for the study, Asarco provided geologic, mineralogic and preliminary leach test data. From this data, MSRD formulated study parameters, developed the concept and order of magnitude capital cost estimates for two mining methods namely, Void-hole Fragmentation Technique and Block Cave In Situ Leach process which lend themselves to in situ leach operations.

This report contains descriptions, capital cost estimates and operating cost estimates for each mining system and the process plant.

A summary of the report is found in Section 3.

SECTION 2

CRITERIA AND DESIGN BASIS

This study was based upon information provided by ASARCO on the size and nature of the ore body to be developed. A series of assumptions were then made to establish design criteria against which the study was developed as described in the following sections of this report. Figure 2.1 shows the area under consideration and this study is based on mining Block B.

ASARCO Data

1. The ore body consists of Block B only.

2. Block B has the following statistics:

   Surface dimensions - 2,500' x 2,500'

   Top of ore body - 1,300' depth

   Ore zone thickness - 800'

   Copper content - 0.42% copper

3. Mineralization is predominently oxide copper. Approximately 35 percent of the oxide copper is green vitreous minerals comprised of brochantite, atacamite and dioptase. Atacamite is a chloride mineral which will release chlorine to the leach solutions.

4. Acceptable copper recoveries can be attained with sulfuric acid leach.

Assumptions

1. The ore zone formation must be fractured or broken to allow a reasonable solution flow rate for leaching in place.

2. The ore body is too deep to allow conventional mining techniques and treatment of the ore in a surface plant.

3. Leaching in place, or in situ leaching, will recover 50 percent of the total copper. Acid consumption averages 4.0 lbs. H_2SO_4 per lb. copper recovered.

4. The ore body will be mined at a rate that will provide a 25 year life for the project.

5. Copper will be recovered from solutions by a solvent extraction plant and electrodeposition to produce marketable cathodes.

6. The project is close enough to urban areas such that no housing will be provided.

7. Leaching solution strengths of 2.9 grams per liter copper can be attained initially.

8. The capital cost estimates will not include:

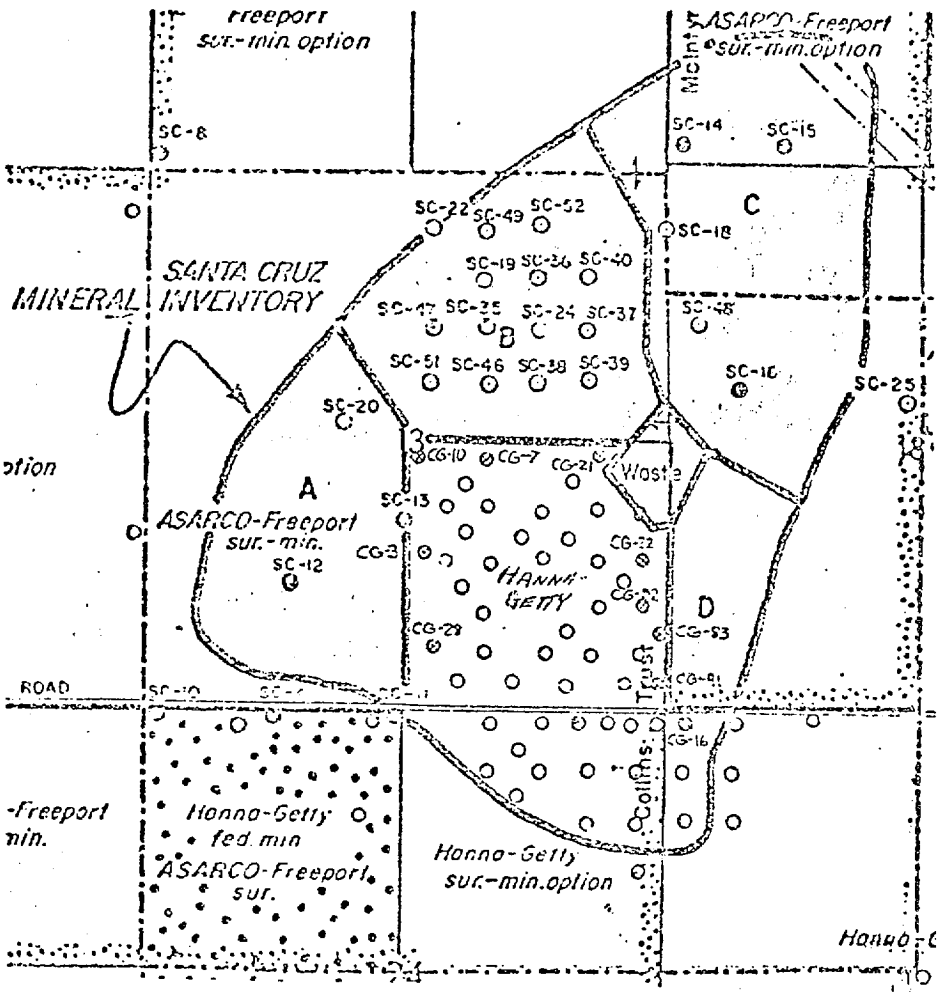
   Property purchase cost

   Access roads to the project

   Power generation or transmission

   Water development beyond the plant area.

PRELIMINARY
ROUGH DRAFT



OUTLINE OF AREA OF COPPER INVENTORY
IN AND ADJACENT TO SEC. 13 (T6S, R4E)
Pinal County, Ariz.

CG-22
⊗ Hanna-Getty holes used to estimate
ASARCO-Freeport Inventory

Outline of Hanna-Getty/ASARCO-Freeport
area of exchange

H.G.K-F.T.G. December 12, 1979

1" = 2000'

FIGURE 2.1

SECTION 3

SUMMARY AND RECOMMENDATIONS

3.1 GENERAL

Two mining methods were examined as an approach to fracturing the ore body such that solutions could be circulated at a reasonable rate to recover an average of 174,904 pounds of copper as cathode each day. These methods are:

1. Void Hole Fragmentation In Situ Leach System

2. Block Cave In Situ Leach System

They are described respectively in Sections 4 and 5.

As leach solutions from in situ and heap leaching operations are generally too low in copper for direct deposition in an electrolytic cell, a solvent extraction (S-X) plant is included in the study. Pregnant solutions from the mine containing up to 2.9 grams per liter of copper initially will be treated in the S-X plant to produce a decopperized acid solution for return to the leach area and a strong copper solution (25-30 gpl copper) for feed to the electrolytic tankhouse. It will be necessary to expand the S-X plant after the fourth year to accommodate larger volumes with lower copper content. The process plant is covered in Section 6.

The scope of the study did not include such infrastructure as access roads, property acquisition, power supply and water supply. Additionally, environmental permit programs were not addressed.

3.2 CAPITAL COST

Order of Magnitude capital costs were developed for each mining system studied. As each system will produce the same solution flowrates and copper production rate, the same process plant will serve each system. It is also assumed that warehouse and office facilities are equivalent for all systems. A summary comparison of capital costs is presented in Table 3.1.

TABLE 3.1

SUMMARY CAPITAL COST COMPARISON
($000)

	Void Hole System Case 1	Void Hole System Case 2	Block Cave System
Exploration	--	--	$ 3,205
Mine Development - Preproduction	$ 75,620	$ 71,498	22,422
Surface Facilities	1,750	1,750	3,300
Process Plant - Preproduction	12,646	12,646	12,646
Total Direct Costs	$ 90,016	$ 85,894	$41,573
Contingency	22,066	21,036	7,500
	$112,082	$106,930	$49,073
S-X Plant Expansion - Year 5*	3,370	3,370	3,370

*Note: Costs are as of 12/31/79.

3.3 OPERATING COSTS

Operating costs are based upon cost per pound of copper produced. The annual dollar costs may be calculated using a production rate of 63,840,000 pounds copper per year.

A summary comparison of operating costs is presented in Table 3.2.

TABLE 3.2

SUMMARY OPERATING COST COMPARISON
(Cents per pound copper)

	Void Hole System Case 1	Void Hole System Case 2	Block Cave System
Mining and Well Install.	56.3	52.2	12.8
Underground Leaching	7.5	7.5	10.5
Surface Leaching	--	--	2.0
Process Plant	11.3	11.3	11.3
Administration & Indirects	13.0	13.0	13.0
Total Operating	88.1	84.0	49.6
Depreciation (Straight Line over 14 years)	12.5	12.0	5.5
	100.6	96.0	55.1

Of the three systems, the block cave system is the only one which appears economically attractive.

3.4 RECOMMENDATIONS

The block cave mining system with in situ leaching followed by solvent extraction and electrowinning appears economically attractive on the basis of a 50¢ per pound copper cost. However, the capital costs do not reflect such influencing factors as interest and escalation and will require a study in greater detail.

It is recommended that an exploration program be implemented to include a shaft and a trial block to determine block caving characteristics. This program will provide bulk samples which can be column leach treated to establish the design criteria for the solution circuits as well as resolve the question of chlorine removal or recovery.

SECTION 4

VOID HOLE FRAGMENTATION IN SITU LEACH SYSTEM

4.1 DESCRIPTION

Discussion

The Void Hole Fragmentation technique is a conceptual approach (patent pending) to the fracturing of a large mineralized body such that solutions may be freely circulated throughout the body for recovery of the valuable minerals in solution.

The technique is based upon the principle that when rock is broken, voids are created throughout the mass and therefore the total volume expands. Openings, or voids, must be made into which the expanded mass can move. This patent-pending system is essentially an explosives column vertically (or angled) through the ore zone under its hundreds of feet of barren overburden. Each explosive column is surrounded by satellite large diameter drill holes that are left void. When the explosives are detonated the subsequent expansion can take place by moving into the void holes provided.

The percentage of voids (or expansion of volume) can be designed into the project by the number of void-holes drilled.

This technique appeared to be a viable system to apply to the Santa Cruz project and was developed as a potential alternate to the Block Cave system covered in Section 5 of this study.

Due to the potential high cost of drilling through the 1,300 of overburden, two cases were developed.

Case 1. Ground Level Drilling

In Case 1 all drilling will be from the surface as shown in Figure 4.1 with a pattern of void and explosive holes. The void holes will be drilled at 12-1/4-inch diameter to the top of the ore zone and from that point drilled and reamed to 26-inch diameter through the ore zone. The largest under-reamer now available is 26-inch diameter and the cost estimate is based upon this factor. However, special 30-inch bits are feasible on a special order basis and the extra cost involved may be more than offset by a 33 percent reduction in the number of void holes required.

The explosion holes will be drilled from the surface to the bottom of the ore zone at 12-1/4-inch diameter.

Leach blocks will be drilled and blasted to provide a fragmented zone measuring 250 feet by 250 feet by 800 feet deep. After blasting, a series of injection and production wells will be installed to form a conventional well field. Each leach block will have the following statistics for purposes of this study:

Surface Area	62,500	square feet
Volume	50,000,000	cubic feet
Tonnage @ 2.56 S.G.	3,993,600	tons
Total Copper @ 0.42% Cu	33,546,240	pounds
Recoverable Copper @ 50%	16,773,120	pounds
Voids, after Fragmentation	4	%
Void Holes Required	660	
Explosive Holes Required	100	
Production Wells Required - 6" Dia.	13	
Injection Holes Required - 4" Dia.	15	

The production-injection well pattern is a basic 5-spot with 50-foot spacing between the injection and production wells. Eight blocks will be developed during a two-year preproduction period to provide for initial operations at 5,000 GPM to 8 blocks, or a rate of 625 GPM per block. At 13 production wells per block, relatively small pumps of 50 GPM rated capacity will provide the needed capacity.

Case 2. Sub level Well Field

This concept was developed as an alternative to the extremely heavy drill schedule required to penetrate the 1,300 foot thick overburden. In this case, development shafts will be sunk to a depth about 80 feet over the top of the orebody and from that level drive rooms out over the orebody. From these rooms the vertical explosives and void holes can be drilled and shot. The 80-foot sill over the ore will maintain the room so that after the blast re-entry will be possible and leachants can be applied and pumped from the same room.

If the cost of underground preparation is less than drilling from the surface, these advantages will still obtain:

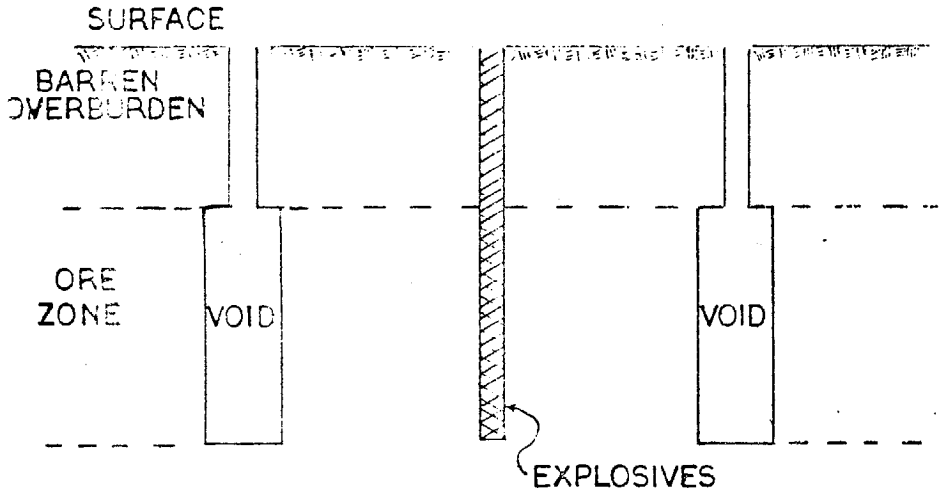
1. The full thickness of the ore can be blasted at once, which is safer and cheaper than the many drill, blast, muck cycles required to take it out in lifts.

2. Miner exposure is minimized and the environment can be controlled.

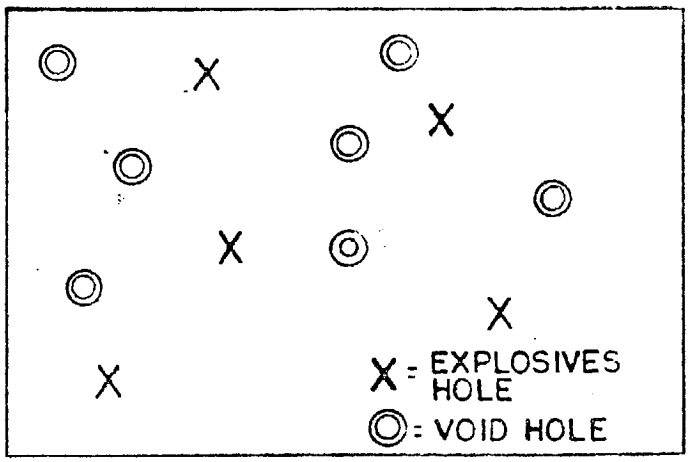
3. Underground development work will be minimal.

This concept is shown in Figure 4.2.

The block statistics are the same as previously given under Case 1.



CROSS-SECTION



PLAN AT TOP OF ORE ZONE

FIGURE 4.1 VOID HOLE FRAGMENTATION

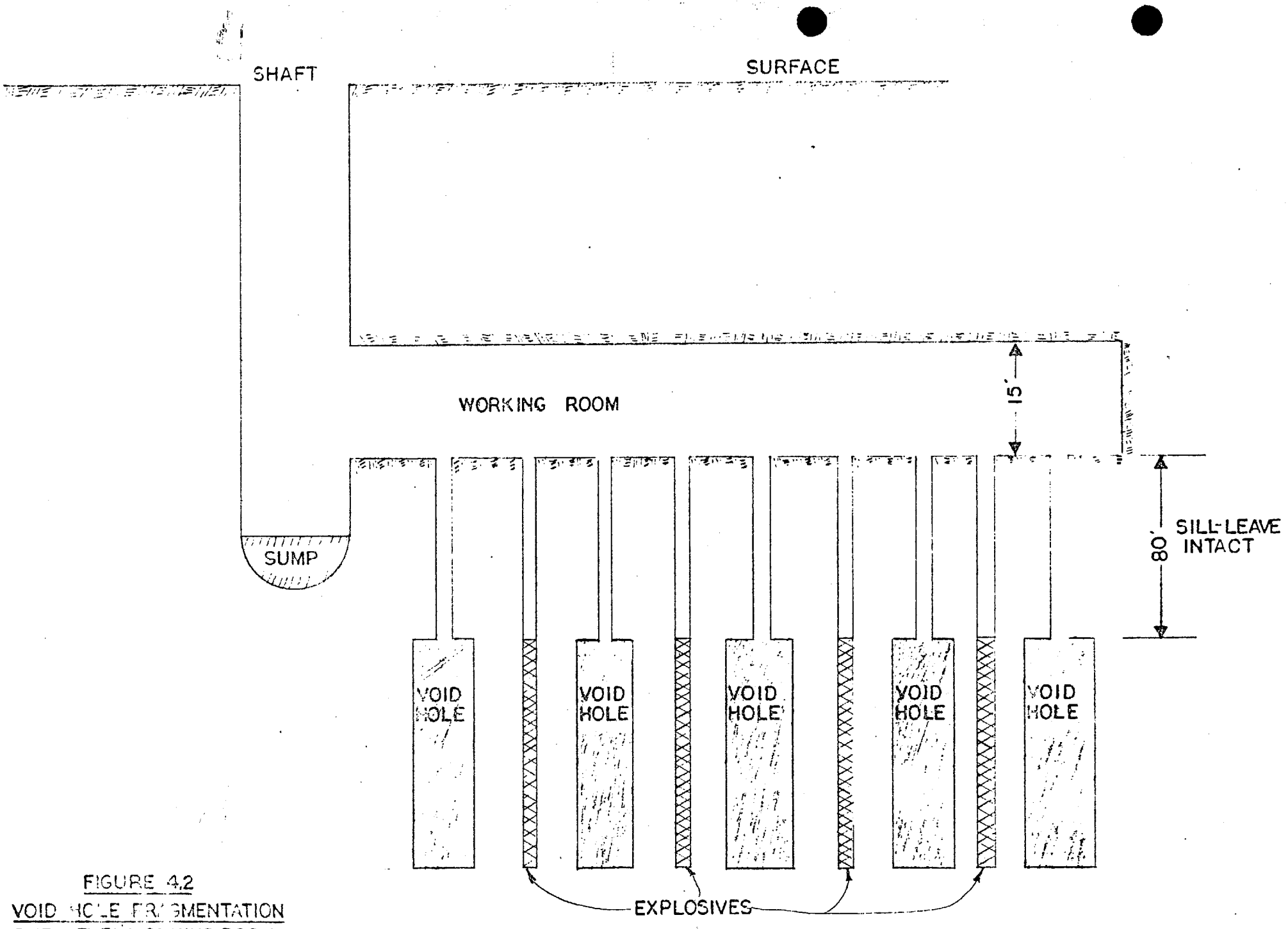


FIGURE 4.2
VOID HOLE FRAGMENTATION
SUB-LEVEL WORKING ROOM

4.2 CAPITAL COSTS

Estimated capital costs were developed for Cases 1 and 2 to provide a comparison between the two concepts. They are based upon the use of unit costs derived from similar operations and from qualified contractors. As both cases are conceptual in nature, the estimates are considered to be order of magnitude only. All costs are in 1980 dollars.

CASE 1. GROUND LEVEL DRILLING.

1. DRILLING AND BLASTING (CONTRACT)

	Cost ($000)	
Drilling		
660 (12-1/4") Holes x 1350 ft. each @ $2.78/ft.	$2,477	
100 (12-1/4") Holes x 2150 ft. each @ $2.78/ft.	598	
Total Drilling	$3,075	
Reaming Void Holes		
660 (26") Holes x 800 ft. @ $3.57/ft.	$1,885	
Explosives - 3.3#/cu. yd.		
6,111,112# @ $50/100#	$3,056	
Detonation & Loading - $12.50/100#	764	
Total Explosives	$3,820	
Total Cost - Drill & Blast	$8,780	
Total Cost 8 Blocks - Preproduction		$70,240

2. PRODUCTION AND INJECTION WELLS (8 BLOCKS)

Injection Wells (Ream, Case, Cement)		
15 x 8 x $10,000	$1,200	
Production Wells (Ream, Case, Cement)		
13 x 8 @ $12,000	$1,248	
Pumps, Installed 13 x 8 @ $23,000	$2,392	
Surface Piping, Storage, Instrumentation		
27,000 ft. @ $20.00	540	
Total Cost Production and Injection Wells	$5,380	$ 5,380
Total Preproduction Direct Costs		$75,620
Contingency (25%)		18,905
Total Estimated Cost		$94,525

CASE 2. SUB LEVEL WELL FIELD.

		Cost ($000)	
1.	SHAFT - 1,300 ft. @ $1,200/ft.		$ 1,560
2.	WORKING ROOM - 250 x 250 x 15' WITH PILLARS	$ 2,082	
	Total - 8 Rooms	$16,656	$16,656
3.	DRILLING AND BLASTING (CONTRACT)		
	Drilling		
	660 (12-1/4") Holes x 80 @ $2.78/ft.	$ 147	
	100 (12-1/4") Holes x 880 @ $2.78/ft.	245	
	Total Drilling	$ 392	
	Reaming		
	660 (26") Holes x 800 @ $3.51/ft.	$ 1,885	
	Explosives - 3.3#/cu. yd.		
	6,111,112# @ $50/100#	$ 3,056	
	Detonation & Loading - $12.50/100#	764	
	Total Explosives	$ 3,820	
	Total Cost - Drill & Blast	$ 6,097	
	Total Cost 8 Blocks - Preproduction		$48,776
4.	PRODUCTION AND INJECTION WELLS (8 BLOCKS)		
	Injection Wells (Ream, Case, Cement)		
	15 x 8 x $8,000	$ 960	
	Production Wells (Ream, Case, Cement)		
	13 x 8 x $9,600	998	
	Pumps, Installed, 13 x 8 @ $21,000	2,184	
	Booster Pumps/Pipe to Storage	100	
	Distribution Piping, 22,000 ft. @ $12.00	264	
	Total Cost Production & Injection Wells	$ 4,506	$ 4,506
	Total Preproduction Direct Costs		$71,498
	Contingency (25%)		17,875
	Total Estimated Costs		$89,373

4.3 OPERATING COSTS

Operating costs for the void hole system are considered to be the cost of developing new blocks each year plus the cost of solution circulation. The operating cost estimates were calculated and presented herein on the cost per pound of copper produced. Separate costs are given for Case 1 and Case 2.

Mining

Block Size - 250' x 250' x 800'	
Block Area, Square Feet	62,500
Total Tons @ 2.56 S.G.	3,993,600
Copper Recovery - %	50
Copper Recoverable/Ton @ 0.42% Cu	4.2
Copper Recoverable - Total Pounds	16,773,120
Copper Recoverable/Square Foot	268

CASE 1

Cost per Block	$9,452,500
Cost per Square Foot	$ 151
Cost/lb Copper - $151/268	56.3¢

CASE 2

Cost per Block	$8,742,000
Cost per Square Foot	$ 140
Cost/lb Copper - $140/268	52.2¢

		Cost (¢/lb. Copper)
CASE 1		
1. Mining and Well Installation		56.3¢
2. Leaching		
Acid - 2.5 lb. @ 1.0¢	2.5¢	
Pumping	5.0¢	
Total Leaching	7.5¢	7.5¢
Total Direct Operating Costs		63.8¢
Administration & Indirects		13.0¢
Total (Excluding Process Plant)		76.8¢
CASE 2		
1. Mining and Well Installation		52.2¢
2. Leaching		
Acid - 2.5 lb. @ 1.0¢	2.5¢	
Pumping	5.0¢	
Total Leaching	7.5¢	7.5¢
Total Direct Operating Costs		59.7¢
Administration & Indirects		13.0¢
Total (Excluding Process Plant)		72.7¢

It should be noted that the above capital and operating costs are based on assuming that 4 percent void space would be sufficient to provide the required permeability in the fractured ore. Previous studies have indicated that about 8 percent void space is necessary to obtain the desirable percolation rates. Accordingly, the cost may be substantially higher if the higher void space is required.

SECTION 5

BLOCK CAVE IN SITU LEACH SYSTEM

5.1 DESCRIPTION

The Block Cave In Situ Leach System described herein was conceptually developed for this study as a potentially viable approach to solution mining of the deep-lying Block B oxide ore zone. At first inspection, the 1300' thick overburden represented a sizeable drilling cost to install a typical drilled well field and so an alternate solution was sought. In developing this system the following criteria were considered:

1. The method would provide maximum permeability for solution flow.

2. Operations would be restricted to the ore zone.

3. Maximum solution control could be maintained using presently developed technology.

4. Known mining technology could be employed.

The Block Cave In Situ Leach System meets all of these conditions.

In concept, the system is based upon sinking shafts to the bottom level of the ore zone (approximate depth 2,300 feet) and driving a series of haulage and crosscut drifts as in a conventional block caving system. The ore body will then be undercut with approximately 10 percent of the ore body hoisted to the surface, stockpiled and heap leached. Concurrently, a series of solution drifts will be driven at the 1,300 foot level to provide access to the top of the ore body for solution injection.

As mining progresses, blocks 150' x 150' by 800' deep will be induced to cave such that the entire block is fractured. Solutions will then be circulated through the block until the copper extraction rate decreases to an uneconomic level.

Mining will continue on the two levels throughout the 25 year life of the project at such a rate as to provide new blocks for leaching as earlier blocks are exhausted.

Figures 5.1 and 5.2 show respectively plan and cross section views of the mine development.

Shafts

Access to the ore body will be provided by a main shaft containing both hoisting and service facilities with a ventilation shaft provided at the opposite side of the ore body. The main haulage drift at the 2,300 foot level will connect the two shafts. The shaft will be capable of hoisting 7,000 tons per day on a 5-day work week.

Mining

Based upon results obtained at Miami Copper, San Manuel Copper and the Lawrence Radiation Lab curve developed from nuclear underground explosions, it is expected that 10 percent of the ore column must be removed to break the column. This, however, will have to be confirmed or revised by underground testing to assure block cave arch failure to the top of the ore body.

The mining system will incorporate lateral transfer by slushers on the control level with rail haulage on the main level. The slusher drifts will be on 30 foot centers with draw raises on 17 foot centers with a 15 foot pillar between panels. These pillars will be crushed and broken for leaching.

A main drift will connect the main and ventilation shafts on the 1,300 foot level to provide for ventilation and the main solution distribution headers. Lateral crosscuts will be driven over each block to provide access for distribution of leaching solutions to the blocks.

Each 150 x 150 x 800 leach block has the following statistics:

Area for solution injection	22,500 square feet
Solution flow @ 0.0045 gpm/ft^2	100 GPM
Total volume	18,000,000 cubic feet
Volume to be mined	1,800,000 cubic feet
Tonnage to be mined @ 2.56 SG	144,000 tons
Mining days at 7,000 TPD	20.6 days
Blocks per year @ 250 days/yr	12.1

As the entire Block B contains 6,250,000 square feet, a total of 40 leach blocks will be developed during preproduction and 238 blocks over a period of 19.7 years and leaching completed in 25 years.

Solution Operations

Block B contains an estimated 380,000,000 tons of ore averaging 0.42 percent copper. The block is approximately 2,500 x 2,500 feet with an average thickness of 800 feet. At 50 percent recovery, total recoverable copper is:

$380 \times 10^6 \times 0.42\% \times 50\% = 1,596 \times 10^6$ pounds copper.
At a projected 25 year mine life average production will be:
$1,596 \times 10^6/25 = 63.84 \times 10^6$ pounds copper per year.

At 365 days per year:
$63.84 \times 10^6/365 = 174,904$ pounds copper per day.

A strong solution of 2.9 grams per liter (gpl) flowing at 5,000 gallon per minute (GPM) will produce copper at this rate using the first solvent extraction module; however, as the leach blocks decrease in copper grade the total flow must be incresed to compensate for lower extraction rates and weaker solutions produced. The projected production schedule giving consideration to block

development time and leaching times are given in Table I. At approximately year 5, the second phase S-X plant will be placed in service to provide capacity for the increased flows.

TABLE I

PRODUCTION SCHEDULE

Production Year	Flow GPM	Assay G/L Cu	Copper/Year Lbs x 10^6
1	4,000	2.6	45.6
2	5,000	2.5	54.7
3	6,000	2.4	63.2
4	7,000	2.3	70.6
5	8,000	2.1	73.7
6	9,000	1.8	71.1
7-25	10,000	1.46	1,217.1
Total			1,596.0

Table II shows the number of blocks in service at various total flow rates if a constant solution distribution rate of 0.0045 GPM per square foot of leach area is maintained.

TABLE II

TOTAL FLOW/BLOCK RELATIONSHIP

Total Flow GPM	Blocks in Service
4,000	40
5,000	50
6,000	60
7,000	70
8,000	80
9,000	90
10,000	100

As individual blocks will vary in copper content and total leaching time, the actual operating schedule will be developed to maintain a balance of new and almost depleted blocks such that overall copper production will be at a relatively constant level.

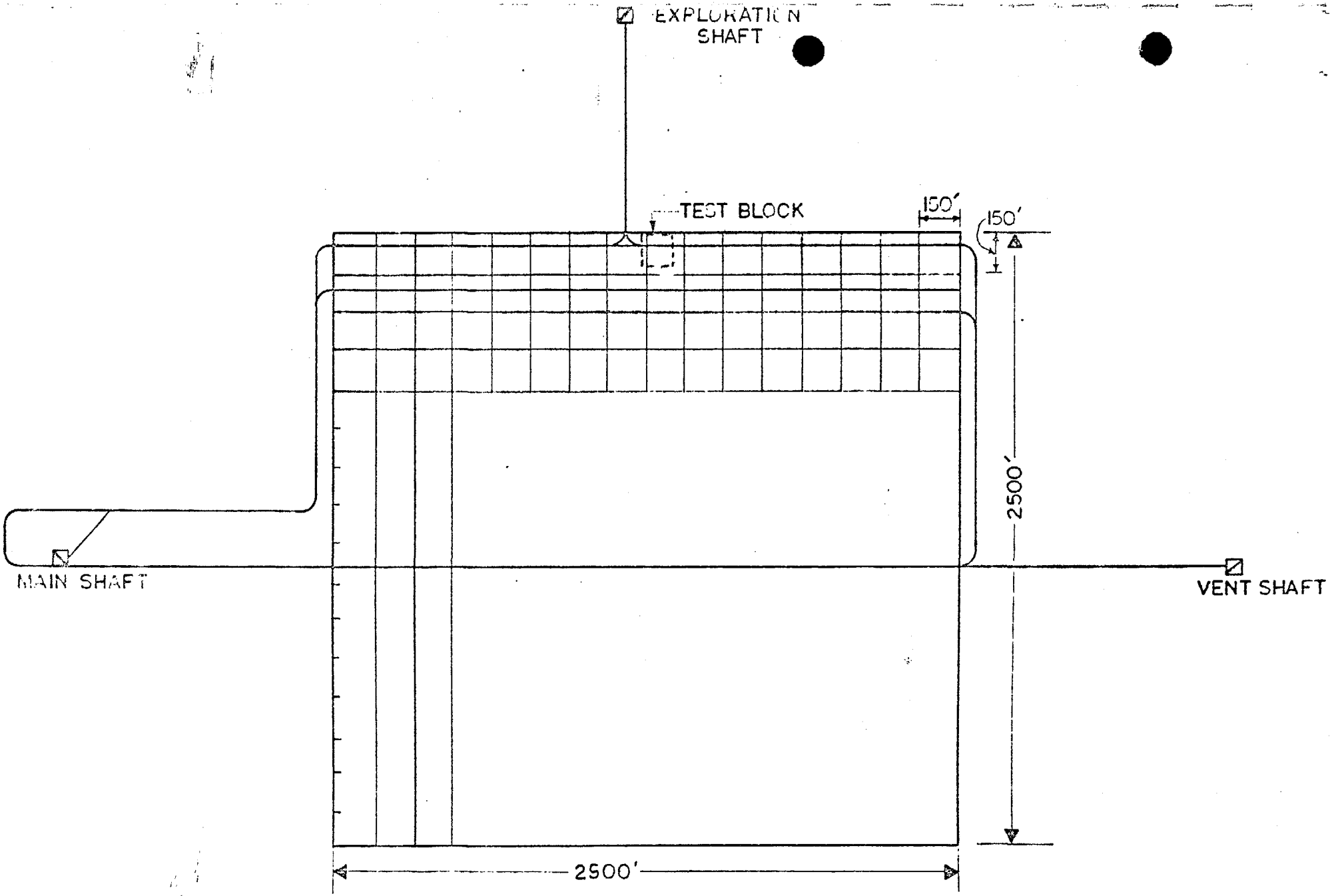


FIGURE 5.1

BLOCK CAVE LEACH IN PLACE

PLAN VIEW

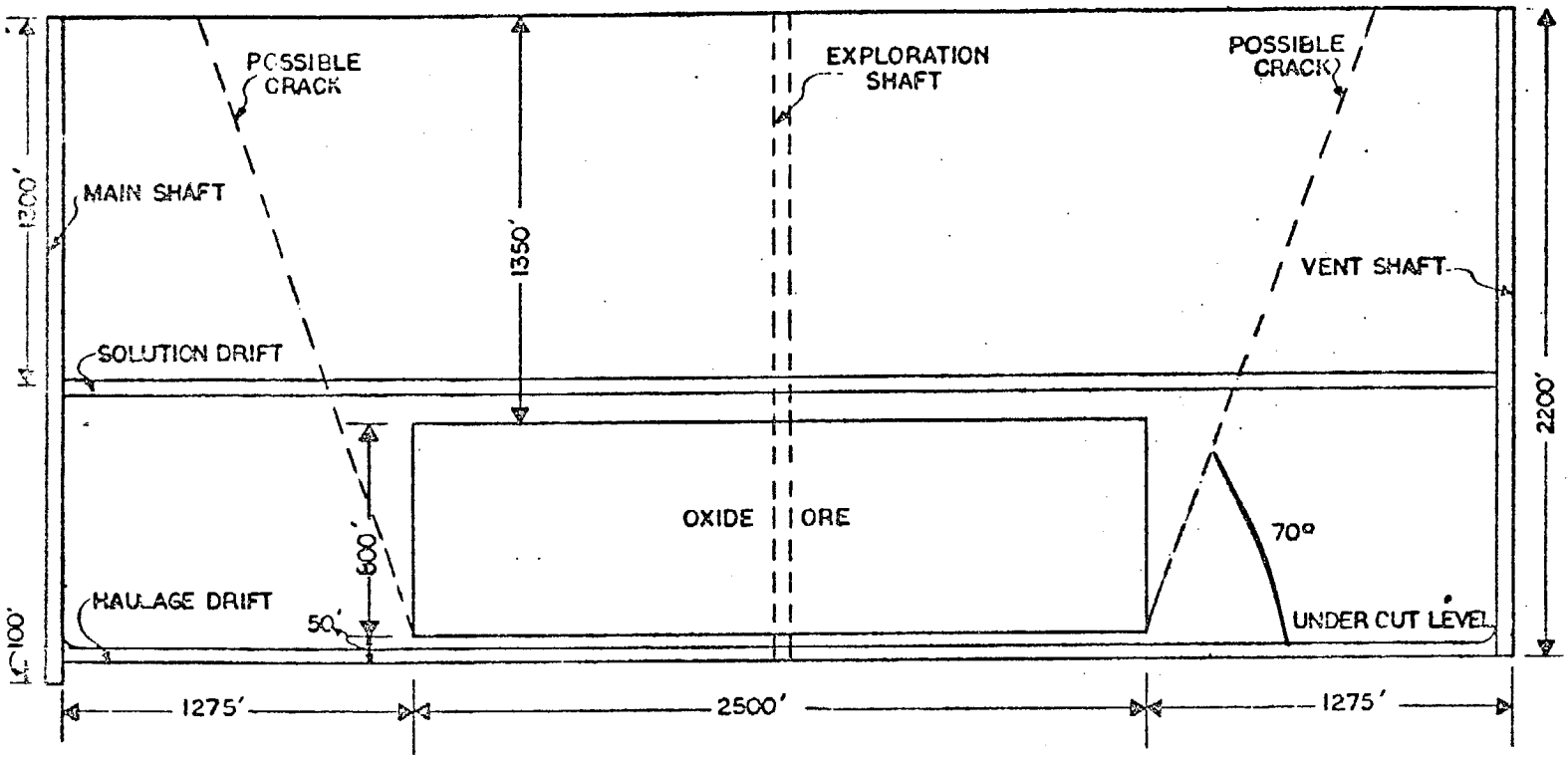


FIGURE 5.2
BLOCK CAVE LEACH IN PLACE
TYPICAL CROSS SECTION

NOTE:

SHAFTS OFFSET 1275' FROM BLOCK CAVE AREA TO ALLOW FOR 70° VERTICAL SHAFT SUPPORT COLUMN.

5.2 CAPITAL COSTS

Estimated capital costs were developed for the Block Cave System to include:

1. Exploration Costs

2. Mine Development (Preproduction)

3. Surface Facilities (EXCLUDING PROCESS PLANT)

The estimates are based upon unit cost factors and cost factors derived from similar type operations. All costs are in 1980 dollars.

1.	Exploration Costs	$ 3,205
2.	Mine Development Costs (Preproduction)	22,422
3.	Surface Facilities (Excluding Process Plant)	3,300
	Contingency (15%)	4,339
	Total	$33,266

		Cost ($000)	
1. EXPLORATION COSTS			
1.1	Development		Shaft
	Design	$ 40	
	Development Setup	400	
	Shaft Sinking & Equipping - 2,200 ft. @ $1,000		2,200
	Total Development Shaft	$2,640	$2,640
1.2	Exploration - Underground		
	Development Drifts - 1,200 ft. @ $300	$ 360	
	Drill Cut-Outs - 20 @ $2,000	40	
	Drilling BX Core - 6,000 @ $15	90	
	Exp. Leach Block Launders	40	
	Total Exploration - Underground	$ 530	$ 530
1.3	Surface Leach Pad and Dump	$ 35	$ 35
	Total Direct Exploration Costs		$3,205

	Cost ($000)	
2. MINE DEVELOPMENT - PREPRODUCTION		
2.1 Hoisting Shaft		
Design	$ 120	
Sinking & Equipping - 2,300 ft. @ $1,600	3,680	
Haulage Level Station	200	
Solution Level Station	120	
Pocket	400	
Total Hoisting Shaft	$4,520	$ 4,520
2.2 Underground Facilities		
Pump Station	$ 800	
Electrical Sub Station	200	
Ventilation Drifts - 3,000 ft. @ $300	900	
Haulage Drift - 12,000 ft. @ $300	3,600	
Supply Elevator Raises - 1,000 ft. @ $450	450	
Standby Power	150	
Total Underground Facilities	$6,100	$ 6,100
2.3 Developed Blocks (4 Total)	$2,102	$ 2,102
2.4 Shaft Equipment		
Hoist	$2,000	
Cage & Skips	200	
Elevator	600	
Hoist & Cage - Vent Shaft	600	
Total Shaft Equipment	$3,400	$ 3,400
2.5 Mine Equipment		
Mining Equipment	$1,500	
Haulage Equipment	3,000	
Fans	400	
Pipe	600	
Pumps	800	
Total Mine Equipment	$6,300	$ 6,300
Total Direct Mine Development Costs		$22,422

		Cost ($000)	
3.	Surface Facilities (Excluding Process Plant)		
3.1	Hoist House, Change Room Office, Shops	$1,000	$1,000
3.2	Head Frame and Bins	500	500
3.3	Electrical Distribution, Transformer Stations	800	800
3.4	Water Development	1,000	$1,000
	Total Surface Facilities		$3,300

	Cost ($000)
UNIT MINING COST (150 x 150 FOOT BLOCK)	
Haulage Drift - 150 ft. @ $300	$ 45
Chutes - 5 @ $2,200	11
Transfer Raise - 100 ft. @ $70	7
Slusher Drift - 750 ft. @ $240	180
Fringe Drift - 150 ft. @ $240	36
Draw Raises - 1,200 ft. @ $30	36
Undercutting - 18,000 square feet @ $7	126
Sub-Total	$ 441
Solution Distribution Drifts - 150 ft. @ $450	67.5
Drill Holes - 1,600 ft. @ $10	16
Sub-Total	83.5
Total Cost per Block	$525.5

At 22,500 Square Feet

Mining: $441,000/22,500 = $19.60 per square foot

Solution: $ 83,500/22,500 = $ 3.71 per square foot

Total $23.31

5.3 OPERATING COSTS

Operating costs for mining and solution operations were estimated on the basis of cost per pound of copper produced. The estimates are in 1980 dollars and based on the following factors:

Mining Cost

The average head of ore to be broken is 800 feet with 10 percent of the column to be mined and leached in dumps on the surface. The 80 feet mined represents 80/12.5 or 6.4 tons per square foot of leach block. The cost per ton is estimated as:

Drawing	-	70¢
Haulage	-	20¢
Hoisting	-	35¢
Dumping	-	20¢
Total		$1.45

$1.45 x 6.4 = $9.28 per square foot leach area.

$9.28 + $23.32 = $32.59 total mining cost per square foot of leach block developed.

Recoverable Copper

Recoverable copper per square foot of leach area:

380×10^6 tons ore at 4.2 lbs/ton = $1,596 \times 10^6$ pounds copper.

2500' x 2500' = 6.25×10^6 square feet.

1,596/6.25 = 255 lbs. recoverable copper/square foot.

Acid Consumption

Leach test results indicate 4.0 pounds sulfuric acid required per pound copper recovered.

The S-X plant returns 1.5 pounds in the raffinate for a net consumption of 2.5 pounds.

			Cost (¢/lb. Copper)
1.	Mining		
	$32.59/255 =		12.8¢
2.	Underground Leaching		
	Acid - 2.5 lb. @ 1.0¢	2.5¢	
	Solution Distribution	3.0¢	
	Underground Pumping	5.0¢	
	Total Leaching	10.5¢	10.5¢
3.	Surface Leaching		
	Distribution & Pumping		2.0¢
	Total Direct Operating Costs		25.3¢
	Administration and Indirects		13.0¢
	Total (Excluding Process Plant)		38.3¢

SECTION 6

PROCESS PLANT

6.1 DESCRIPTION

This study is based upon a solvent extraction-electrowinning plant for recovery of copper from strong leach solutions and the production of marketable cathode copper. The strong leach solutions from the mine and dump heap leach operations will be treated in the solvent extraction (S-X) section for removal of copper by an organic solvent, and then returned to the mine and dump solution circuits as new lixiviant. The organic solvent will be treated with strong sulfuric acid solution to strip the copper from the organic and produce a high copper content aqueous solution suitable for electrolysis. This solution will then be sent to the tankhouse for electro-deposition of the copper as cathodes. The spent electrolyte from the tankhouse is returned to the S-X section for stripping of loaded organic.

Mining Solutions

A dilute sulfuric acid solution will be used in the mining operation. An initial acid charge will be required to acidulize the ground water in the ore zone and produce the first strong leach solutions. As operations progress, decopperized acid solutions (raffinate) from the S-X plant will be reinjected to the leach zone. Make up acid and water will be added to the mining solution circuit to maintain the desired acid and volume conditions.

The process plant will include storage tanks for receipt of strong leach solution ahead of the S-X section, raffinate storage tanks, acid storage tanks, fresh water storage, organic storage, kerosene storage and necessary pumps.

Solvent Extraction Section

The solvent extraction section consists of a series of mixer settler tanks which provide for the mixing of the aqueous organic phases followed by a quiescent settling zone for separation of the two phases. To provide for 95 percent extraction of the copper to the organic phase from the aqueous phase, a typical three stage circuit is described.

The strong leach solution is fed to the mixer of No. 1 mixer-settler where it is mixed with the organic overflow of No. 2 mixer-settler. The mixed phases are pumped by the mixer impeller to the settler portions of the cell where the two phases are allowed to settle and separate. The lighter organic overflows the top of the settler and is pumped to the stripper circuit. The aqueous phase discharges over a bottom weir and flows to No. 2 mixer where it is mixed with the organic overflow from No. 3 settler. After separating in the No. 2 settler the aqueous progresses to the No. 3 mixer where it is mixed with barren organic solvent from the stripper circuit. After separating in the No. 3 settler, the aqueous phase, or raffinate, is pumped to storage and subsequent recycling to the mining area. As can be seen, the barren organic enters the extraction circuit at No. 3 mixer-settler and progresses countercurrent to the aqueous phase.

The copper loaded organic from No. 1 mixer is fed to a series of mixer-settlers in the stripper circuit where the copper is stripped into a strong acid solution. As just described, the organic and aqueous phases progress countercurrent to provide maximum stripping efficiency.

The entire S-X Section is designed as an all weather installation and does not require any building or enclosures other than those provided on the tanks and mixer-settlers.

Electrolytic Tankhouse

The tankhouse comprises a single building with a full basement area under the cell floor to provide adequate headroom for gravity return of solutions from the cells to the appropriate pump sumps. The cells will be monolithic concrete tanks with preformed paraliners. A number of the cells will be dedicated to starting sheet production with stainless steel cathodes and lead-antimony anodes. Rolls, shears and looping machines are provided for preparation of the starting sheets on a daily basis. These starting sheets become the core cathode in the production cells. A rectifier station will provide the required electrolytic current.

A cathode storage and shipping area is provided adjacent to the main building as well as solution storage tanks.

Evaporation Ponds

Due to the presence of Atacamite ($Cu_2(OH)_3Cl$), the pregnant solutions will gradually increase in chlorine content to the point that adverse reactions will occur in the solvent extraction circuit. With an assumed tolerance of 20 grams per liter of chlorine by the LIX 64N solvent used in the S-X circuit, the pregnant solutions will be allowed to increase in chlorine to this level and then controlled by a bleed stream discard of raffinate (low copper) to evaporation ponds. An average discard rate of approximately 40 gallon per minute is required to maintain the solution in balance. Under this system copper losses will be minimal but will involve an acid loss.

Two evaporation ponds 660 feet x 660 feet x 5 feet deep will be provided to meet the evaporation requirements. Each pond will be membrane lined and provide a four foot evaporation depth with one foot of freeboard.

Sizing was based upon the following information and assumptions:

1. 31% of the copper in Block B occurs in zones containing Brochantite, Atacamite, and Dioptase. These are all designated as vitreous green oxides. (H.G. Kreis memorandum of July 28, 1978).

2. 65% of the vitreous green oxide minerals occur as the chlorine containing Atacamite.

3. Atacamite contains 59.5% Cu and 16.6% Cl.

4. Production solutions average 2.9 grams per liter copper.

5. Net evaporation rate is 60 inches per year.

6. Maximum LIX64N tolerance for chlorine in low copper-acid solutions is 20 grams per liter.

6.2 CAPITAL COSTS

Basis of the Estimate

The capital cost estimate for the process plant is a factored estimate determined through application of an exponent factor of 0.7 to the capacity ratio between this plant and a similar project for which reliable data is available.

It is based upon the following general conditions and assumptions:

1. The project site is located within reasonable commuting distance of an adequate supply of craft labor.

2. Project is to be on a "turn key" basis.

3. The estimate includes facilities within the battery limits for the solvent extraction plant, electrolytic plant, and evaporation ponds only.

4. The estimate is based on 1980 dollars.

5. The accuracy of this estimate is assumed to be within ± 35%.

Estimates

Estimates for the process plant are given on Sheets S-1 and S-2 following.

Phase I covers the process plant facilities to treat strong leach solutions at a rate of 5,000 gallon per minute with a copper content of 2.9 grams per liter and produce electrolytic cathode at a rate of 200,000 pounds per day.

Phase II covers an additional solvent exchange section for the treatment of an additional 5,000 gallon per minute of leach solution. This addition will be required to handle increased solution flow rates to compensate for a decrease in solution grade as a result of ore body depletion. There will be no expansion of the tankhouse due to the offsetting effects of increased solution flow with decreased copper content.

mountain states engineers

REV. DATE

PROJECT: IN SITU COPPER LEACH PROCESS — PREPARED BY: JDK — FIRST ISSUE

CLIENT: ASARCO — APPROVED BY — SHEET S-1 of S-2

A C NO.	AREA / PHASE I PROCESS	QUANTITY	UNIT	MANHOURS PER UNIT	MANHOURS TOTAL	COST UNITS L M E	COST UNITS TOT. or SUB	COSTS LABOR	MATERIAL	EQUIPMENT	SUB CONTRACTOR	TOTAL
	FACILITY											
	STAGE I SOLVENT EXTRACTION (5000) GPM											1,975,000
	ELECTROWINNING PROCESS (200,000 LB/DAY)											6,350,000
	EVAPORATION PONDS - NO ROCK ASSUMED											1,131,000
	SUBTOTAL - DIRECT COSTS											9,456,000
	SUPPLEMENTAL DIRECT COSTS											1,417,000
	FIELD INDIRECT COSTS											424,000
	TOTAL FIELD COSTS											11,297,000
	HOME OFFICE SUPPORT											220,000
	ENGINEERING											1,129,000
	SUBTOTAL											12,646,000
	CONTINGENCY (+25%)											3,161,000
	ESCALATION - NONE: ALL COSTS ARE THROUGH											
	1/31/80											
	TOTAL EXCLUDING FEE*											15,807,000
	IF ROCK ENCOUNTERED IN EXCAVATING EVAP. PONDS CAPITAL COST MIGHT INCREASE $260,000											
	* THE ACCURACY OF THIS ESTIMATE IS ASSUMED TO BE WITHIN ± 35%										PROJECT NUMBER	2 1 4 0

FORM 5-301

mountain states engineers

REV. DATE

PROJECT: IN SITU COPPER LEACH PROCESS
PREPARED BY: RHK 12/26/79
FIRST ISSUE
CLIENT: ASARCO
APPROVED BY
SHEET S-2 of S-2

A/C NO.	AREA PHASE II PROCESS	QUANTITY	UNIT	MANHOURS PER UNIT	MANHOURS TOTAL	COST UNITS L M E	COST UNITS TOT. or SUB	COSTS LABOR	MATERIAL	EQUIPMENT	SUB CONTRACTOR	TOTAL
	FACILITY											
	STAGE II SOLVENT EXTRACTION											
	(5000 GPM)											1,975,000
	SUPPLEMENTAL DIRECT COSTS											300,000
	FIELD INDIRECT COSTS											120,000
	TOTAL FIELD											2,395,000
	HOME OFFICE SUPPORT											50,000
	ENGINEERING											250,000
	SUBTOTAL											2,695,000
	CONTINGENCY (+25%)											675,000
	ESCALATION - NONE: ALL COSTS ARE THROUGH											
	12/31/79											
	TOTAL EXCLUDING FEE*											3,370,000

*THE ACCURACY OF THIS ESTIMATE IS ASSUMED TO BE WITHIN ± 35%

PROJECT NUMBER 2 1 4 0

FORM S 301

6.3 OPERATING COSTS

Direct operating costs for the process plant were developed using general experience factors from similar type plants and operations. Repair labor and supply costs were taken as a percentage of the plant capital cost. The estimates are presented in cost per pound copper recovered in 1980 dollars.

			Cost (¢/lb. Copper)
1.	Labor		
	Supervision	0.427¢	
	Operating Labor	1.548¢	
	Fringe Benefits - 45%	.889¢	
	Total Labor	2.864¢	2.864¢
2.	Repair Labor and Supplies		2.256¢
3.	LIX and Kerosene		0.912¢
4.	Sulfuric Acid		2.500¢
5.	Power		2.800¢
	Total Direct Operating Costs		11.332¢

Note: Administration and indirect overhead costs are included in mine operating costs - Subsection 5.3.

RECENT ADVANCES IN MINING OF LOWER GRADE ORES

Part I

IN SITU MINING - THEORETICAL AND PRACTICAL ASPECTS

Part II

BLOCK CAVE - IN PLACE LEACHING

by

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Introduction

The conventional approach to the mining and processing of low-grade ore deposits is "giantism". The economic advantages of large-scale and mass-production methods have been utilized for lowering overall production costs, thus compensating for lower grade of ore. The large, open-pit mines in the Southwestern United States, Chile, Peru and the USSR are classic examples of reducing unit cost by spreading the cost of highly mechanized automated facilities over the largest possible volume of output.

However, large capital investment and considerable risk are involved in these larger mineral development projects as the following figures for two new copper mining projects will show.

Project Name	Reserves (million tons)	Grade (%)	Capital Required (million $)	Mill Capacity (tons/day)
Sierrita - Arizona, USA	414	0.35 Cu 0.036 Mo	163	80,000
Bethlehem Copper Co., Highland Valley British Columbia, Canada	1,000	0.48	300	75,000

Technological advances in mining have been the subject of several recent technical meetings and are well documented; the topics covered include larger trucks and conveyors used for transporting mined ores, larger milling equipment, studies of back slopes of open pits, construction of large tailing dams, recent advances in underground mining and ground supports, rapid excavation, mechanized mining, raise boring, shaft sinking and many other relevant items.

Rather than a representation of published information on the above techniques, the present paper discusses newer developments in mining technology which are readily applicable to lower grade and submarginal ore deposits. These include in situ mining technology and the block cave - in place leaching concept, two classical examples of which are the economically successful in place leaching operation at Miami, Arizona, and the more recent application of the process at the Old Reliable Copper deposit near Mammoth, Arizona. In the latter, 4 million pounds of explosives were employed to shatter a near-surface deposit of copper containing about 4 million tons of ore, to a size that will permit satisfactory in place leaching operations.

PART I

IN SITU MINING - THEORETICAL AND PRACTICAL ASPECTS[1]

In situ mining may be defined as the extraction of metals from ores located within the confines of a mine (broken or fractured ore, stope fill, caved material, ores in permeable zones) or in dumps, prepared ore heaps,

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[1]This contribution was prepared in cooperation with Dr. Paul Johnson former Metallurgist with N.M. Bureau of Mines and Mineral Resources, Socorro, N.M. The authors gratefully acknowledge the cooperation of Mr. Don H. Baker, Jr., Director of the Bureau for permitting us to present this contribution.

slag heaps, and tailing ponds on the surface. These materials represent an enormous, untapped, potential source of all types of metals. The field of in situ mining, now in its infancy, 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 lixiviants under prevailing conditions of the in place environment, the generation and regeneration of such solutions, and the recovery of metals or metal compounds from the metal-bearing liquors.

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 and gob-filled stopes, waste dumps, tailing ponds, and slag heaps. In situ mining promises economic recovery from these types of deposits, but full appreciation of its potential needs a much better understanding of its chemical and physical aspects.

This kind of mining has previously largely been limited to the extraction of copper from low-grade materials. The potential is, however, much greater as practically all metals are susceptible to leaching in the in situ environment. Processes will soon be developed for the in situ extraction and recovery of metals such as copper, lead, zinc, nickel, manganese, uranium, silver, gold, molybdenum, and mercury.

Any process used in mining or mineral processing has certain advantages and disadvantages. A few for chemical mining are listed below:

Advantages

1. In situ mining can often be used to recover metals economically from materials that could not be so treated by more conventional mining, milling, and smelting processes.

2. An in situ mining plant usually requires less capital investment than a conventional mine and mill plant.

3. An in situ mining process usually increases a mine's ore sources and reserves. Low-grade or inaccessible ore zones, gob and caved fill, and dumps and tailings may become ores.

4. The leach liquors obtained through in situ mining usually lend themselves to a variety of metal recovery processes. The pure metal or metal compounds so obtained may be of greater value than the sulphide or oxide products normally obtained by conventional milling processes.

5. In situ mining may prove applicable for ores that are too refractory for conventional recovery processes.

6. In situ mining can often be used in conjunction with a conventional mining or milling process to boost metal recoveries and increase ore reserves.

Disadvantages

1. Both physical and chemical restraints may limit the usefulness of a chemical mining process. The effectiveness of contacting ore with solutions and the recovery of leach solutions from the system without appreciable loss are two important physical factors. Dissolution or dissolution rates, metal precipitation, and solution regeneration are major chemical factors.

2. Testing an in situ mining process short of actual field operation sometimes proves difficult.

3. Groundwater contamination may result from some chemical mining operations.

4. Basic information on the physical and chemical factors involved is presently lacking.

In Situ Mining Technology

The field of in situ mining may be considered under the headings of (1) mining economics and ore evaluation, (2) elements of the leaching phase, (3) preparation of ores, (4) practical aspects of in situ leaching, (5) reagent generation and regeneration, and (6) recovery of metals from leach liquors.

Mining Economics and Ore Evaluation

In considering the economic exploitation of a deposit through in situ mining, one must determine the size of the deposit, tonnage of ore in place, and amount of metal contained therein. In past as well as present mining operations, the cut-off grade has been governed by the total operating cost, including mining, which usually constitutes a significant portion of the overall cost. In chemical mining considerations, however, the cost of mining would be minor and the cut-off grade can be lowered correspondingly. This would inevitably increase the tonnage as well as the metal content of the deposit, which in turn would influence the overall economics of the venture.

Unfortunately, information concerning the relationship between tonnage and grade is largely lacking in the literature. No doubt records of mining companies may contain such valuable information, and some attempts should be made to obtain pertinent data from these sources.

Lasky (1950), Musgrove (1965) and a few others have studied this relationship through a statistical analysis of known deposits and perusal of past records of some mining companies. These studies reveal that there is an exponential relationship between grade and tonnage of ore reserves. Especially for deposits in which there is a gradation from relatively rich to relatively lean material, there appears to be a consistent mathematical relation between tonnage and grade, according to the equation $G = K_1 - K_2 \log T$, (1)

where T is the tonnage produced to a given time plus the estimated reserves, G is the weighted average grade of this tonnage, and K_1 and K_2 are constants to be determined for each deposit. Using equation (1), Lasky (1950) showed that for a typical porphyry copper deposit, the tonnage increases at a compound rate of 14.9 per cent for each 0.1 per cent decrease in grade.

Another important aspect of in situ mining on which hardly any data are available is determining a minimum reserve and grade for profitable exploitation. The only operational data available are from copper dump leaching and in place leaching practices in which the grade of material treated is above 0.16 per cent.

The important factor in chemical mining, as in dump leaching, is making sure that the major portion (+90%) of the specified volume of leach solution fed to the deposit or dump is recovered with a given minimum amount of metal content in solution over the life of the economic operation. This minimum metal content in the specified volume is such that the value of the recovered metal will provide for the cost of operation, amortization and profits.

Naturally, the metal content and its value in leach solution differs from metal to metal. From a hydrometallurgical recovery viewpoint alone, it is estimated that at the current prices of metals and operating conditions, the break-even contents for a minimum operation of 200,000 gallons a day are 250 ppm (0.25 g/l) copper, 50 ppm (0.05 g/l) molybdenum, and 10 ppm (0.01 g/l) uranium. If the mining and development cost, overhead, and profit amount to 200 per cent of the metallurgical treatment cost, then the metal contents in leach solution must be 750 ppm (0.75 g/l) copper, 150 ppm (0.15 g/l) molybdenum, and 30 ppm (0.03 g/l) uranium for an economic operation.

In general, it may be safe to assume that because of lower treatment and capital costs incurred in in situ mining, a sufficiently large deposit containing half the grade of deposits currently mined and milled could be treated economically. Thus, deposits containing 0.25 per cent copper, 0.12 per cent molybdenum, and 0.1 per cent uranium could profitably be mined with this technique. In actual practice, it may well be possible to treat even lower-grade deposits than these.

It may be emphasized, however, that utilizing in situ mining schemes would require a new approach on all phases of the mining operation, especially in exploration, reserve estimation and overall evaluation of leaching and metal recovery parameters.

Elements of the Leaching Phase

Accessibility, physicochemical interaction and transport constitute the elements involved in the leaching phase of in situ mining. 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 depends on exposure and penetrability. The factors to consider are locations of the metal values, their volume and shape distribution, exposure area, specific surface, particle size, porosity, capillary pressure, viscosity pressure, solubility of gases in the lixiviant and surface roughness.

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 and vapour pressure in gases. 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 and wettability.

The first two elements by themselves do not ensure successful leaching without transport of products away from and reactants to the zone of interaction, 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.

Broadly speaking, the factors governing leaching can be grouped as either physical or chemical. The majority of leaching studies in the past have emphasized chemical factors; it is, however, essential that we also consider the physical factors since they definitely influence the leaching process. We must develop new techniques for physical and chemical testing of ore samples and for establishing the limitations and optimum parameters for successful extraction of values from the broken ore.

Preparation of Ores

To be processed by *in situ* mining techniques, ores may be:

(a) in place but requiring fragmentation prior to leaching;

(b) in place and permeable enough to permit flow of solutions through them; or

(c) previously mined or fragmented.

Waste dumps, tailings, filled stopes and caved ground fall into the last group.

Several means have been proposed for fragmenting an ore body prior to *in situ* leaching. In recent years, various authors[2] have proposed the use of nuclear explosives. Griswold (1967) suggested using hydrofracturing techniques to break ore for subsequent leaching; liquid explosives would be injected into ore bodies along planes of weakness and detonated at a slow rate.

Although conventional mining methods have seldom been used to prepare ore for *in situ* leaching, there is no reason why they could not be used. Present methods such as caving techniques and shrinkage stoping are well adapted to breaking ore for subsequent underground leaching.

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[2]Johnson (1959), Smith and Young (1960), and Hansen and Lombard (1964).

In some instances, it may be advantageous to use a conventional mining method for selectively removing the higher-grade ores from an ore body prior to in situ leaching of the lower-grade materials. If, for example, in the mining of an ore body by a shrinkage method, a lower-grade zone adjacent to the higher-grade body were to be drilled as the stope progressed upward and the holes were loaded as the stope was drained, the low-grade material could be broken into the stope cavity and then leached.

Varying degrees of preparation may be required when the ore is already broken. When it is located underground as stope fill, very little ore preparation is required. Ores to be treated by in situ mining techniques on the surface, however, may or may not require some preparation prior to leaching. Waste dumps, slag heaps, and the like may require crushing and stacking on prepared pads, whereas fine materials like tailings may require rebedding, slime removal, or placement on an impervious pad.

Figure 1 illustrates how fine tailings might be leached by downward percolation techniques. In this system, an impermeable pad of plastic, asphalt, or concrete would prevent solution losses through ground seepage. Alternating layers of coarse rock and fine mill tailings would then be laid down over the pond area. Leaching, either concurrent with tailings deposition or following deposition, would be by downward percolation through the tailings beds of limited thickness. Percolation rates would be considerably higher through beds of limited thickness than unlimited ones. Solutions could be fed on the top bed or injected to selected beds through wells. After percolating through the tailings, the pregnant leach solutions would flow to a central recovery well. Possibly, gases could be injected to displace solutions and to react with the metal-bearing minerals.

Practical Aspects of In Situ Leaching

In in situ mining, large volumes of ore are in contact with relatively large volumes of leaching solution over a period of time. The mechanics or pattern of solution flow varies according to the chemistry involved, the means available for solution containment and recovery, and the need to prevent groundwater contamination.

Two principal types of solution flow through a porous ore bed are

(a) downward percolation under the influence of gravity and

(b) flow within an immersed system.

In a downward percolation system, leach liquor is usually distributed over the top of a pile of ore and allowed to flow through the pile to a liquor collection system. This type of solution-ore contact has the advantage that only the floor of the ore bed need be impervious to the leach solutions. This type of flow allows some circulation of air within the ore bed, possibly an important factor in the oxidation of ore minerals. Use of downward percolation in an underground ore bed could prevent solution seepage into the groundwater strata. The chief disadvantage of this type is that incomplete solution-ore contact can result from localized impermeable zones and from channelling.

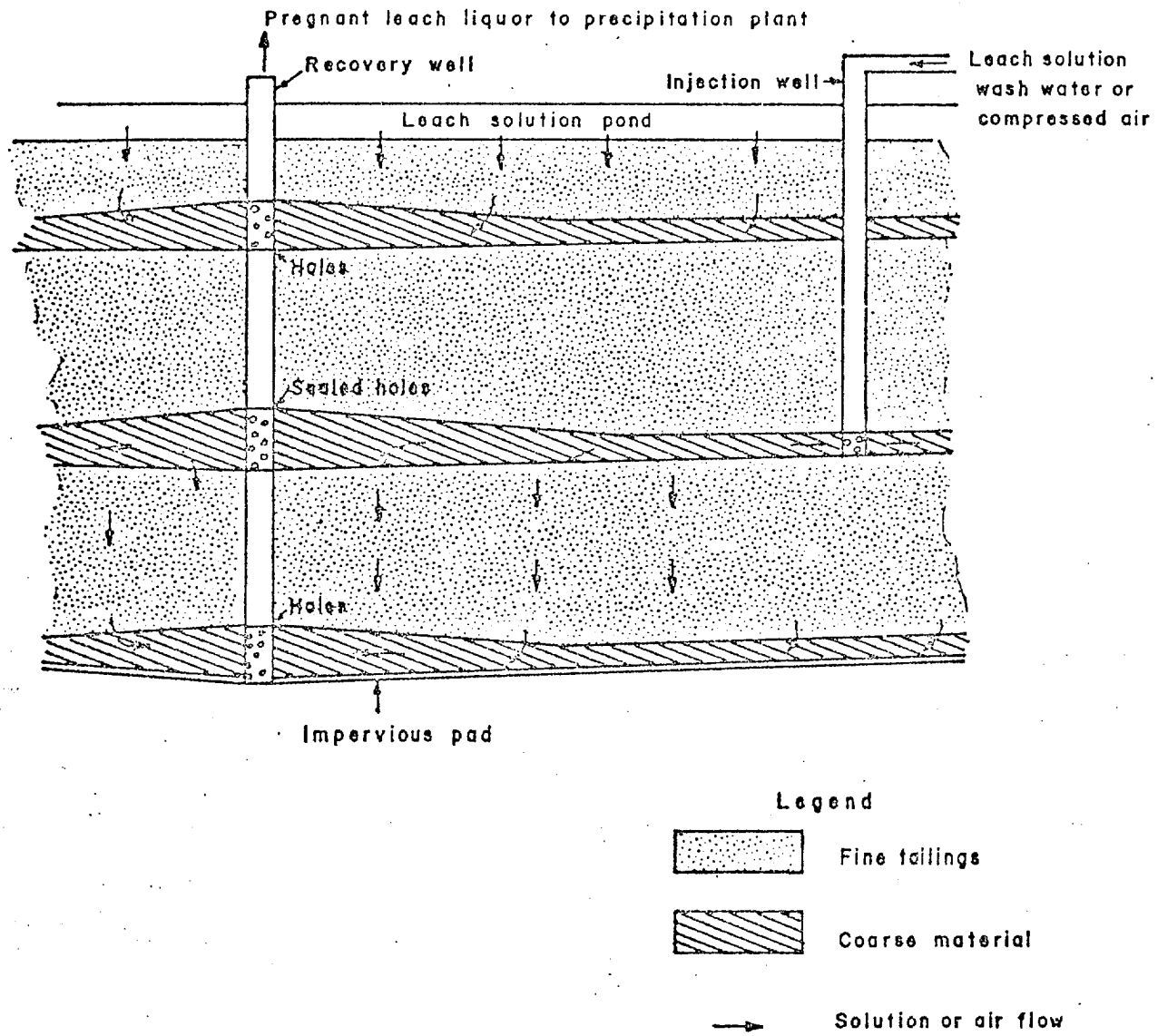

Figure 1
Bedding of Fine Tailings for Subsequent In Situ Leaching.

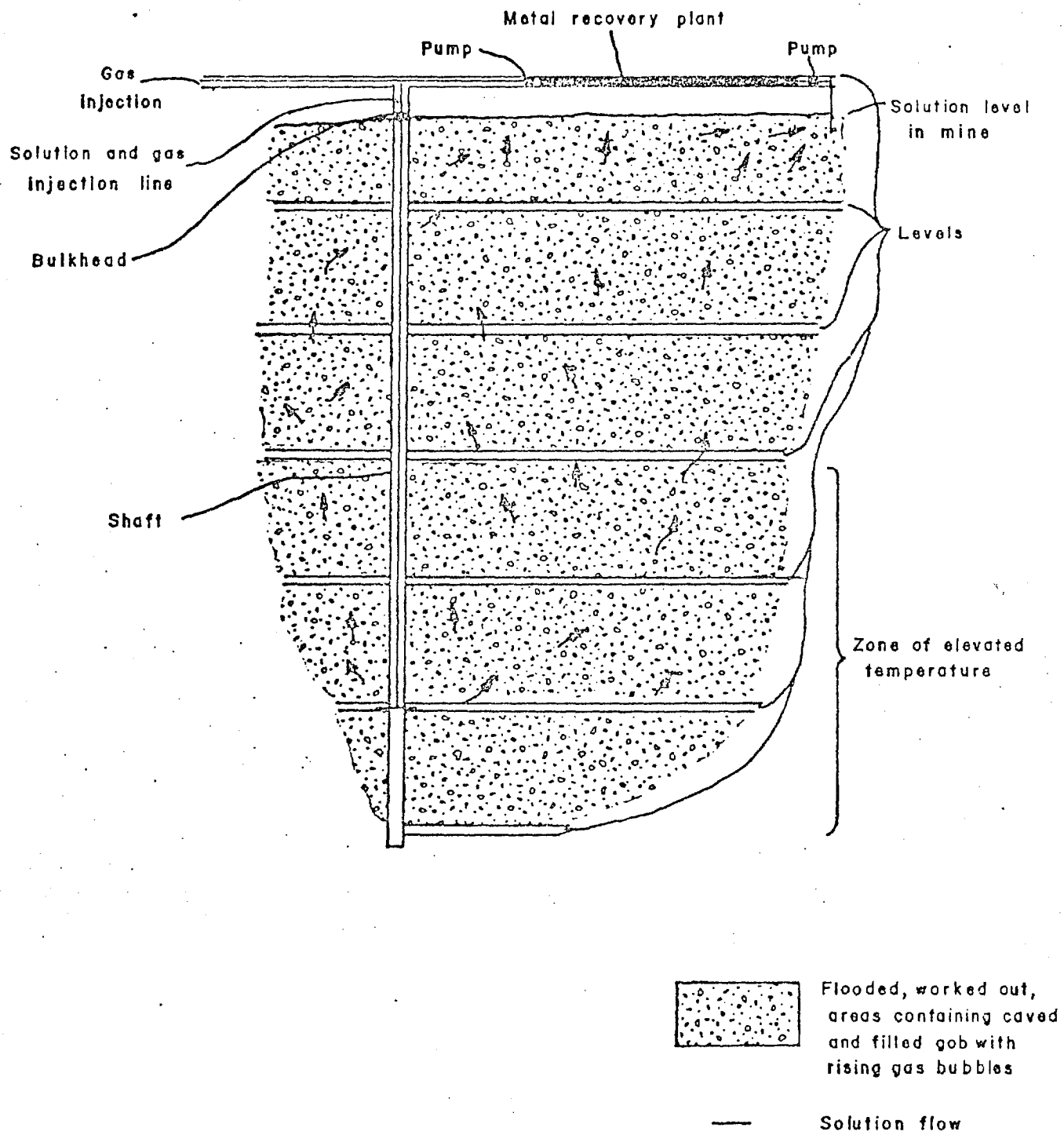

Figure 2
The Leaching of an Immersed Mine Under Autoclave Conditions

Downward percolation has been, and probably will continue to be, the principal method of leaching ore beds. The technique, which has been discussed by several authors[3], has been used in the leaching of waste dumps[4], crushed and uncrushed ores on prepared pads[5], mill tailings[6], and filled and caved workings[7]. Solution transfer may be either by convection or by diffusion. Convection may be caused by mechanical means or by differences in the density of the solution at different points within the system. This type of flow offers positive movement of solutions through an ore bed at a desired flow rate and complete contact of the solution with the bed. The main disadvantages of this type of flow are the necessity of having the ore in an impermeable container to prevent solution loss and contamination of groundwater, the restriction of natural oxidation by air circulation, the necessity of pumping solutions, and the large amount of solution involved in the leaching system.

Immersion techniques have been used in in situ mining. Copper oxide-sulphide ores have been leached in concrete tanks by upward percolation techniques at Inspiration - Robie (1928) - for many years. Utah Construction Company - Robie (1967) - recently conducted leaching tests on an unmined uranium ore body by using a series of injection, monitor, and recovery wells to force a leaching solvent through the permeable uranium ore body and to recover the pregnant leach solutions. Pirson (1959) proposed similar techniques for the in situ leaching of phosphate beds. Uranium is being recovered from mine waters at Grants, New Mexico. Copper has recently been recovered from water in the flooded Rio Tinto mine, Mountain City, Nevada.

Certain features inherent in an underground environment can assist leaching. One is the hydrostatic pressure imposed on an immersed deposit at depth and another is the natural increase in rock temperatures with depth. If gas(es) were introduced into an inflowing stream of solution entering a deposit under a hydrostatic pressure and/or the rock temperatures were above the normal temperatures at the surface, one could possibly use this system as a huge, low temperature-high pressure autoclave. Leach reaction rates can usually be increased manyfold when temperatures and gas pressures are increased.

Figure 2 illustrates how autoclave conditions might be imposed on an immersed mine. This system consists of an abandoned mine flooded with a leaching solution. A pipe introduced into the bottom of the mine via a

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[3] Seidel; Levine and Hassialis (1962); Sullivan and Bayard (1931).

[4] Irving (1921); Power (K.L.), Sawyer (1929); Argall (1963); Bogert (1961).

[5] Thompson (1948) and Nashbir (1964).

[6] Greenawalt (1912).

[7] Greenawalt (1912); Wormser (1923); Thomas (1938).

shaft would carry a gas (usually air) and/or spent leach liquor into the bottom of the system. These conditions would greatly enhance the rates of dissolution of most oxide and sulphide ore minerals. Solution circulation within the system would result from the air-lift and convection effects of rising gas bubbles, convection currents caused by the variation in temperature between the top and bottom of the system, and pumping of solution in closed circuit within the system. Metal would be recovered from the pregnant solution in its circuit from the top to the bottom of the mine.

A similar technique could leach a particular area underground; only the leaching zone would need immersion. If the area constituted a worked-out part of the mine, bulkheads placed in appropriate drifts or openings would seal it off. If it were a fragmented or permeable area underground reached by boreholes (Fig. 3), no seal would be required. Solution and/or gases would enter the leaching zone under pressure and either would be forced back to the surface by the internal pressure in the system or would require pumping. If the volume of the cavity leached were not too large, heating the influent leach solution could prove advantageous.

If the effluent stream of gas and solution discharges from the leached zone at a lower pressure than the influent stream, one could introduce a low-pressure gaseous stream at the top of the influent pipe and allow the downward-flowing liquid to compress the gases during their flow. The downward-flowing stream would act as a hydraulic compressor and as an autoclave. Possibly, a hydraulic compressor-autoclave of this type would serve to oxidize or chemically change the leaching solution prior to its use in either a percolation or immersion type of system.

Another idea that may warrant consideration is that the flow of a low-amperage current could be directed from an electrode on one side of a broken ore zone undergoing leaching to another electrode on the other side. Under given conditions of voltage and amperage, one might change the leaching solutions chemically, accelerate physicochemical reactions, and cause an increased rate of ion migration to a local area of solution recovery.

Solution Generation and Regeneration

Inasmuch as the chemical reagents used in in situ mining greatly influence leaching and generally constitute a major cost item, reagent generation and regeneration is a very important part of any chemical mining process. Some reagents can be generated and regenerated by natural processes in the leaching cycle, whereas others require various chemical processes.

Practically, the only time a leach solution is generated and regenerated by natural processes is in the production of sulphuric acid and ferric sulphates from pyrites and spent ferrous sulphate leach liquors under air oxidation conditions. Although this process is used in leaching great quantities of copper from both sulphide and sulphide-oxide copper ores, very little is really known about the reaction mechanisms; reaction rates and factors influencing them, such as oxygen availability, temperature,

bacterial activity, and degree or extent of solution generation or regeneration; location of chemical processes within the dump; or means of enhancing this natural process. Kennecott Copper Corp. and others have attempted in recent years to solve some of these problems.

Various industrial methods are used for producing and regenerating leaching solvents. Sulphuric acid is produced by the contact process, ferric sulphate - Thomas and Ingraham (1963) - can be made by the air oxidation of ferrous sulphate solutions in the presence of sulphuric acid, and all of the salts, acids, or bases used in leaching - NaOH, Na_2CO_3, $NaHCO_3$, $NaHClO_3$ and NaCN - are prepared commercially and can be regenerated from spent leach liquors. Unfortunately, the use and regeneration of these reagents are uneconomical in many instances.

Johnson (1965) described a sulphuric acid-ferric sulphate solution generation and regeneration process for chemical mining. A specially designed, air-agitated autoclave generates sulphuric acid and ferric sulphate leach solutions from pyrite and concurrently oxidizes and hydrolyzes a spent ferrous sulphate leach liquor to sulphuric acid and ferric sulphate.

The key to low-cost lixiviants for in situ mining purposes probably lies in the efficient use of either low-cost materials found in the ores to be leached, such as pyrite, or raw or low-cost prepared materials readily available, like pyrite, sodium chloride, trona, or liquid ammonia. Much research work is required to determine how these materials can be used efficiently.

The authors' current studies will determine the effectiveness of common leaching reagents, such as acids, acid-iron salts, and salts like NaCl, Na_2CO_3, $NaHCO_3$, in leaching ores of copper, lead, zinc, nickel, silver, gold, uranium, and molybdenum under chemical mining conditions. These tests include percolation leach conditions, static leach conditions, and high gas pressures-low temperature (below 100° C) conditions. A large column-type autoclave may be used soon to leach coarse materials under each of these conditions.

The Recovery of Metals from Leach Liquors

The last phase of any hydrometallurgical process, including in situ mining, 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. This recovery technique is adequately covered in the literature and its effectiveness is clearly demonstrated in several successful plant practices.

In connexion with in situ mining 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 solutions that may contain more than one valuable metal. Unlike copper, not all metals easily precipitate

on scrap iron. Recirculation of the leach solution may be required to build up the metal content, with bleeding off of a small part of the concentrated leach stream for metal recovery.

Newer techniques of ion exchange, solvent extraction, and charcoal sorption may effectively concentrate dilute leach liquors. These procedures have proved very effective for processing large volumes of leach solutions containing more than one valuable metal. The Climax process - Johnson (1966) - for recovering oxide molybdenum values by charcoal sorption and the New Mexico Bureau of Mines procedure - Reynolds, Long and Bhappu (1966) - recently developed for recovery and selective separation of molybdenum, tungsten, and rhenium by sorption processes are typical of techniques that metal recovery systems will increasingly employ.

Since the crux of the in situ mining process is the particular lixiviant in leach solution, any recovery phase that regenerates the solvent or provides an essential component of the leach solution would be the preferred procedure. Also, since in situ mining depends on the continuous circulation of the leach solution at its peak volume, it is imperative that the retention time in the metal recovery step be as short as possible. This requirement necessitates a metal recovery procedure that is relatively quick and capable of handling large volumes effectively.

Discussion

This paper outlines a novel approach for extracting the metal values contained in low-grade deposits, worked-out mines, dumps, and tailing piles. With the ever increasing demand for today's metals, the necessity of treating complex and low-grade ores, increasing operational costs, and the public awareness of environmental pollution factors, future metal production will inevitably employ chemical mining on an increasing scale. The scope of this mining method encompasses interdisciplinary science and technology, requiring application of the principles of basic sciences, economics, mining, metallurgy, hydrology, and allied disciplines. Although some technological information is available from several dump leaching and a few in place leaching operations for copper and uranium, not enough is known about this technique.

Some researchers have tried to develop newer metal recovery techniques, but a much more concentrated endeavour would take fullest advantage of the in situ mining process for extracting metal values from marginal, low-grade deposits, dumps, and tailings.

The dearth of information illustrates the need for broad and imaginative research. Inasmuch as in place leaching occurs on coarse materials on a large scale over a period of time and under conditions considerably different from most other leaching practices, study of the process from both basic research and practical application standpoints is of great importance.

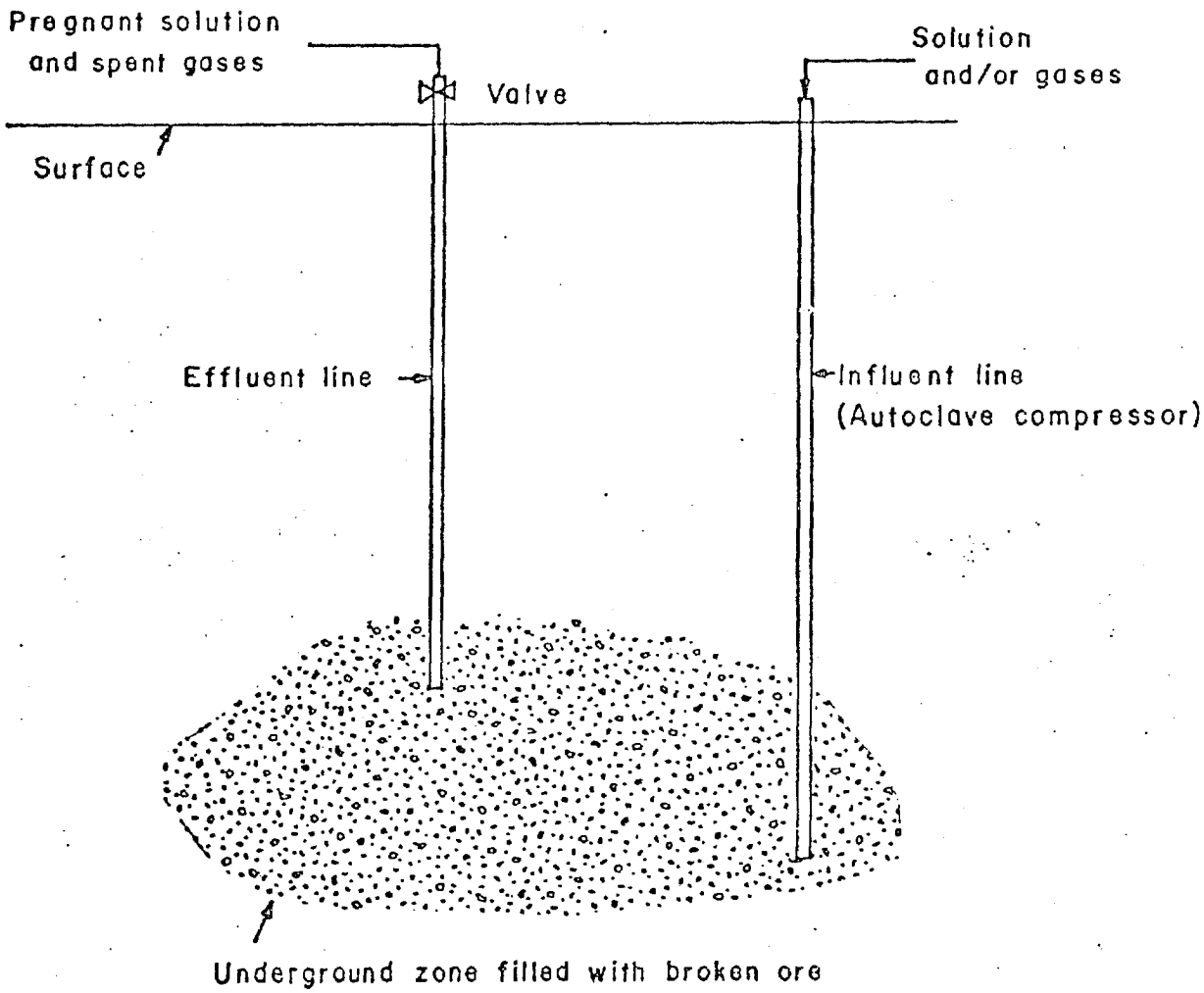

Figure 3
The Leaching of a Local Underground Area Under Autoclave Conditions

PART II

BLOCK CAVE - IN PLACE LEACHING

To introduce the concept of Block Cave-In Place Leaching, a mixed oxide-sulphide (chalcocite) copper deposit has been selected. It comprises 102,000,000 tons, assaying 0.50 per cent total copper (0.25 per cent oxide and 0.25 per cent sulphide). The ore body averages 425 feet in thickness with 2,000 feet of overburden. This hypothetical ore body can be used for the study of deposits of different types, shapes and contained minerals.

Porphyry ore bodies are usually highly altered and fractured with much of the mineralization occurring in the seams or fractures. This type of ore body caves readily and breaks into small pieces (5% + 10", 25% + 4" and 70% - 4") which would be ideal for leaching.

An ore body of this type must be proven by drilling. The normal block cave mine, before development, is usually drilled on 200 foot centres. However, due to the extreme depth of the hypothetical ore body, a surface drilling on 600 foot centres would probably justify shaft sinking and additional drilling from underground. All such holes should be saved for future use.

Mining

From studies relating to block cave mining and underground nuclear explosion (chimneys), it is apparent that 10 per cent to 15 per cent of the ore column must be removed to break the column. This will have to be tested and monitored underground. It is necessary to get the typical block cave arch failure to the top of the ore in order to have the entire ore column broken for leaching. The exploratory drill holes can be used for monitoring the progress of the cave.

Using the drill hole data, a contour map of the bottom and top of the ore body can be drawn. With this information, a block cave-in place leach plan can be formulated. The most suitable and economical plan would be lateral transfer by scrapers on the control level with belt conveyor haulage. This system allows the undercut to follow the slope of the ore body. For this study, a 140 foot wide panel system, with 20 foot pillars between panels was selected. Slusher drifts on 30 foot centres and draw raises on 17 1/2 foot centres along the slusher drifts were selected. With this system a good draw control is obtained and additional pillars can easily be left. Blocks of 150' x 140' comprise a unit within the panel. Figures 1, 2 and 3 give details concerning the operation.

From a block cave layout a $12.54 per sq. ft. mining cost was calculated (Appendix 1).

Leaching

Test leaching of all the exploratory drill holes, cores and leaching tests of underground work should be made. From observations of results

from numerous leaching operations, a 70% recovery of 0.50% copper ore is considered conservative. This is the point where the grade of the pregnant solution drops below 1 g/l. This average recovery is applied to the entire ore body and gives 7.0 lbs. copper recovery per ton of ore. With 425 ft. of ore at 7 lbs. per ton this gives

$$\frac{425}{12.5} = 34 \text{ tons per sq. ft.} \tag{2}$$

$$34 \times 7 = 238 \text{ lbs. Cu/sq. ft.} \tag{3}$$

Operating Cost

(a) Mining: $\frac{12.54}{238} = 5.3$ cents per lb. Cu (4)

(b) Underground leaching:

Acid at $20/ton and net consumption of 2 lbs/lb Cu (3.5 lbs minus 1.5 lbs recovered from electrowinning)	- 2.0¢	
Solution distribution	- 2.0¢	
Underground pumping	- 3.0¢	= 7.0¢ per lb Cu
(c) Surface leaching		= 1.0¢ per lb Cu
(d) SX[8] electrowinning		= 8.0¢ per lb Cu
(e) General		= 5.0¢ per lb Cu
	Total	26.3¢ per lb Cu

[8] SX = solvent extraction.

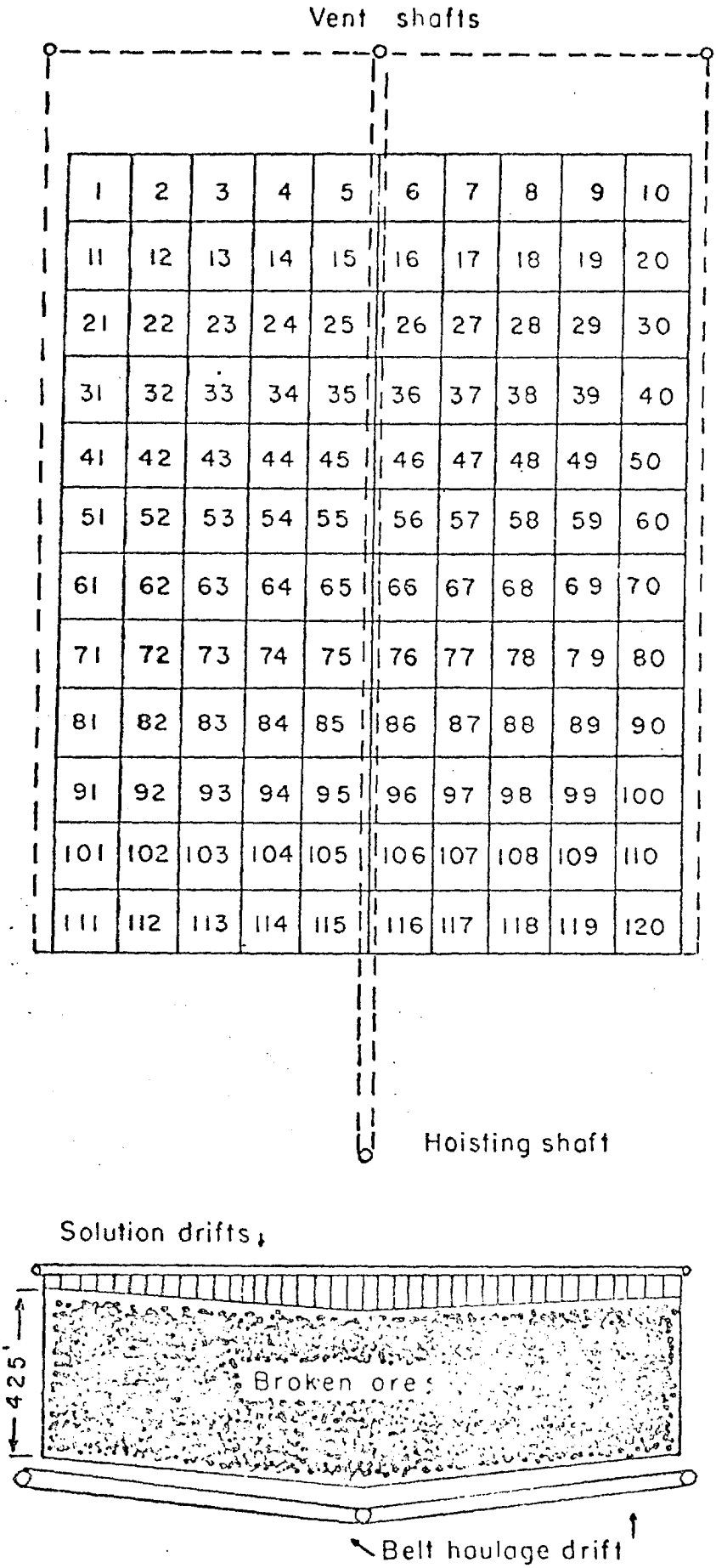

Figure 1 A Typical Block Cave System

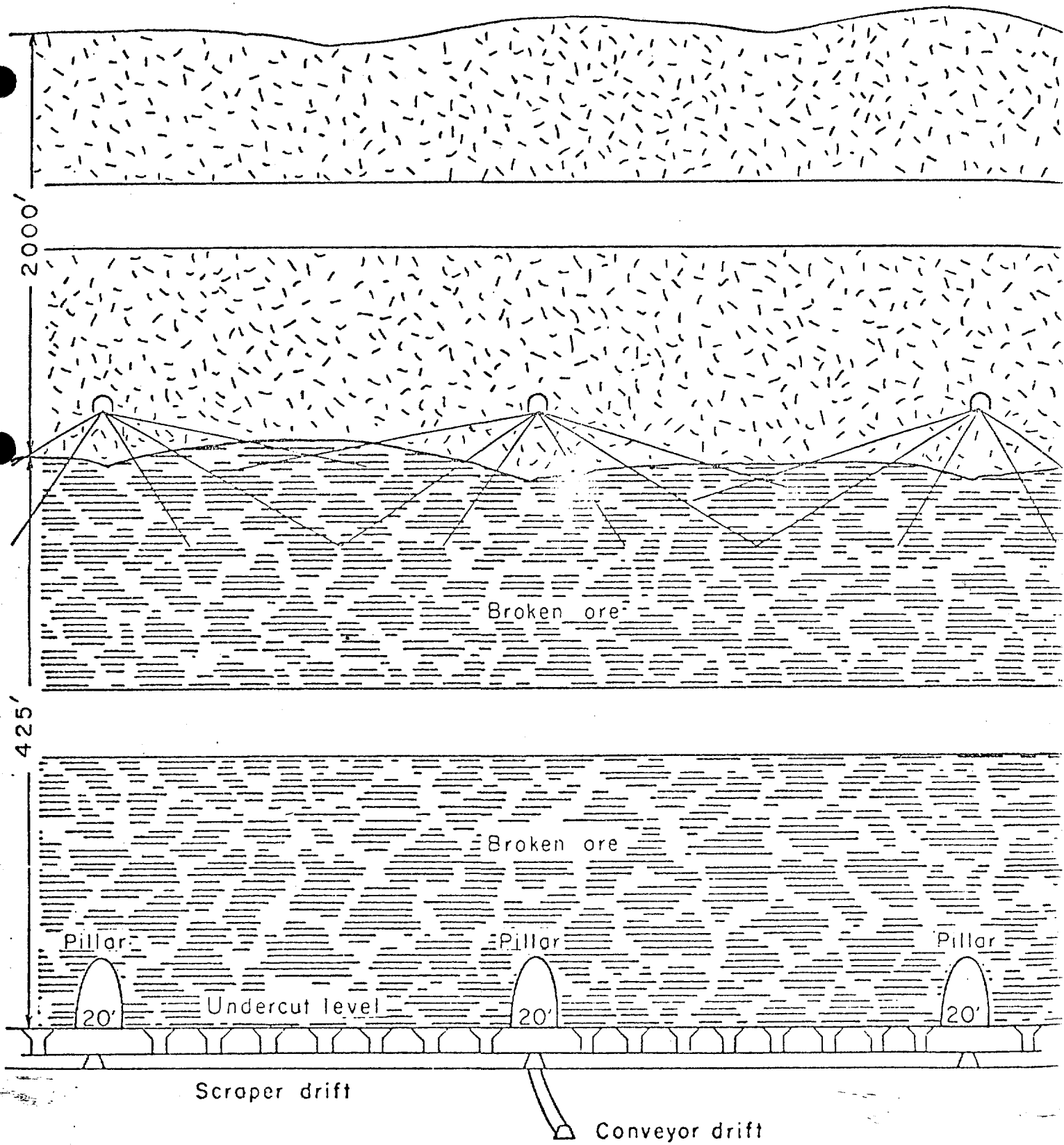

Figure 2 A Cross Section of the Block Cave System

With a sale price of 50¢ per lb. Cu this gives an indicated operating margin of 23.7¢ per lb. Cu.

Capital Cost

The capital cost estimated to bring the mine into production is estimated at $30,000,000 in Appendix 2. The $1,154,000 cost listed therein for developing four blocks is not properly capital investment but the money will be needed prior to production. Block preparation should go into the deferred development account and is actually part of the mining cost.

Plant

A 4,000 g.p.m. plant was selected for this study. With an average of 2 g/l copper this gives a production of 96,000 lbs. Cu/day or 35,000,000 lbs. Cu per year.

$$\frac{96{,}000 \text{ lbs. Cu/day}}{238 \text{ lbs. Cu/sq. ft.}} = 404 \text{ sq. ft. of development required/day} \quad (6)$$

$$\frac{714{,}000{,}000 \text{ lbs. Cu}}{35{,}000{,}000 \text{ lbs. Cu/yr}} = 20.4 \text{ years life} \quad (7)$$

This indicates a mining crew of 120 men per day on a five-day week basis, or 80 man shifts per leaching day (30 day month). This gives 5 sq. ft./man/day. With 425 ft. of ore to break and 10% to hoist gives 17 tons/man shift or 2,000 tons per day, using a five-day week.

A 4,000 tons per day hoisting system should be installed to take care of the initial development and any possible expansion.

Operations

A completely developed block must be left between the leaching and development blocks to prevent any breakthrough of solution into the working areas. On the block lay out plan (fig. 1) blocks 2, 11, and 12 would have to be completely developed before leaching of block no. 1 started. Block 3 and 13 before leaching of no. 2, blocks 4 and 14 before leaching of no. 3, etc.

A system of water doors will be needed to store solution in case of power failures. Or a stand-by power system could be used.

Four hundred g.p.m. of new water will be needed to maintain the 4,000 g.p.m. flow.

Ecology

This process should have environmental advantages because there would be no smelting of concentrates nor disposal of tailings, and a minimum disturbance of surface is envisioned. The solution will all be re-cycled with no seepage loss.

The heap leach dumps would be placed in terraces which would be re-planted.

Basis for Conclusion and General Remarks

Mining

One of the problems in block caving is maintaining the openings while drawing the large tonnages of ore. The mining method proposed in this report surmounts this problem, as only enough ore is drawn to break the column. No repair work will be required on the extraction levels. The flow of solution can be maintained by a system of pipes (French drain) in the caved areas.

Guarding against caving at the surface is a serious problem. The mining must be watched very carefully so as not to overdraw, but the draw must break the ore column in order to successfully leach the ore. Due to compaction there may be a small sway in the surface (which can be detected by surveying) with no damage to building or structures. There could, however, possibly be a small crack at the 70° angle circle outside the caved line, which would damage pipe lines, railroads, etc. A fund should therefore be set aside for possible damage. A pillar study should be made as mining progresses.

Leaching

The following are characteristics of leaching:

1. The grade of the pregnant solution varies with the amount of copper in the ore. As leaching progresses, the assay of the pregnant solutions drops.

2. The rate of recovery varies with the height of the column to be leached and the size of the material.

3. The percent recovery varies with the grade of the ore.

With computer programming the proposed mining system is flexible enough to give a reasonably steady production.

Several block cave mines have successfully leached the copper left after mining (dilution and pillars). Notable examples are the Ray Mines (Kennecott), Inspiration and Miami Copper. Miami Copper block caved in order to break the ore to leach in place.

Construction of Leaching Rate Working Curves

In order to optimize dump or heap leaching operations under a given set of operating conditions such as, number of dumps or heaps, their heights, time required for preparing each dump or heap, rate of solution

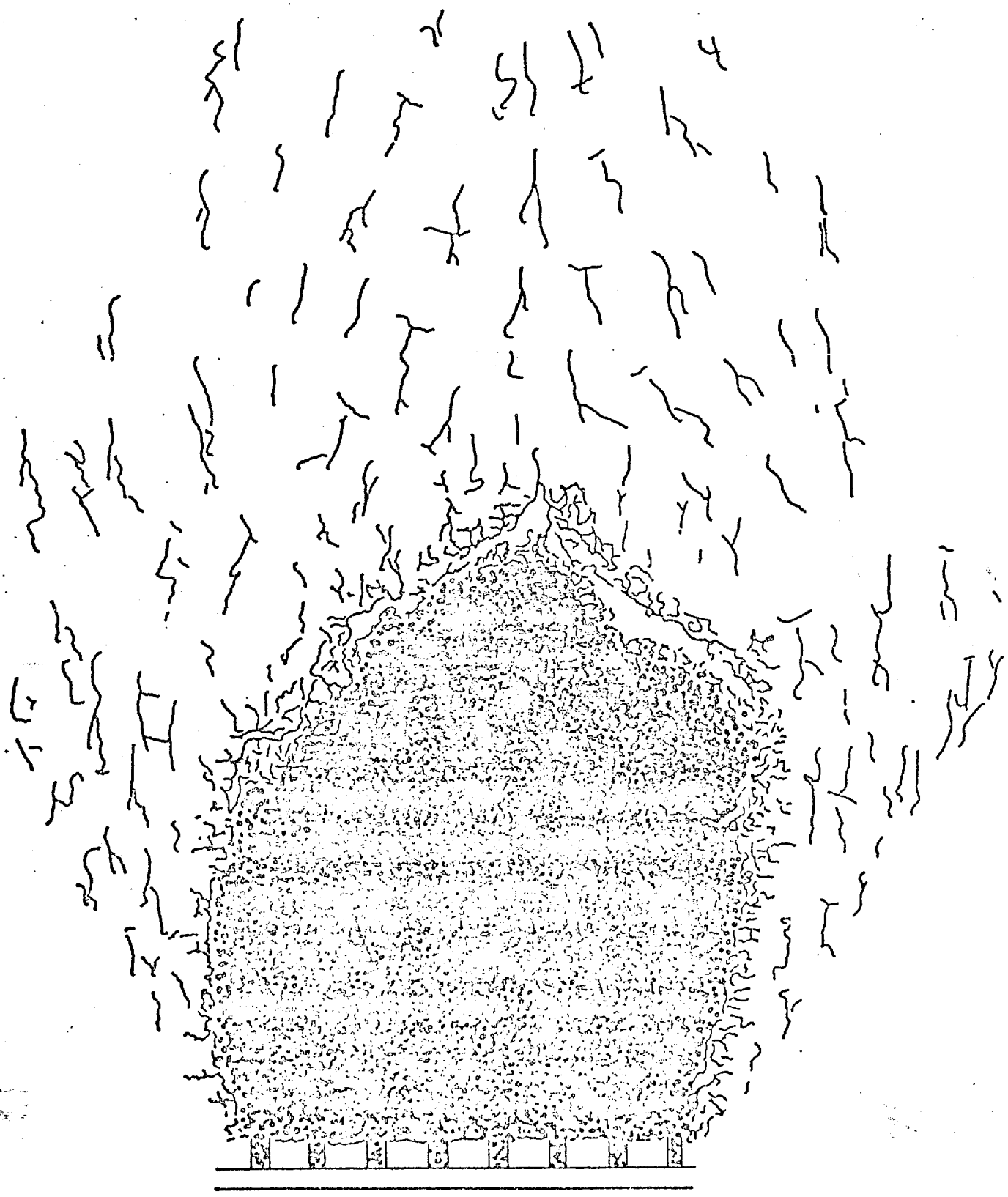

Figure 3 A Section Through Caving Block

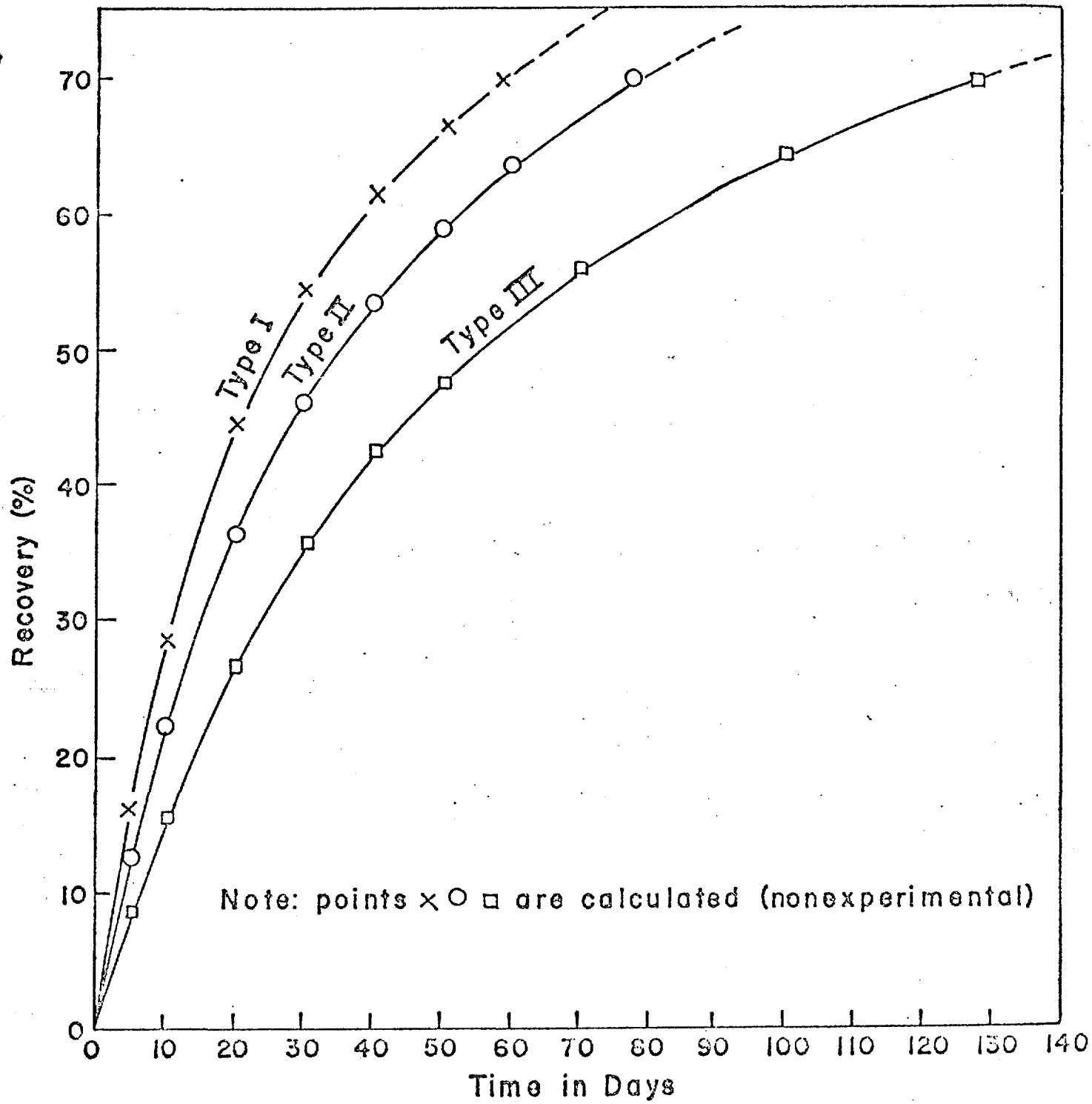

Figure 4 Recovery as a Function of Time, Heap Leaching Situation
(Ore Size = 2½"; Heap Height = 50')

flow, and holding the metal concentration in leach solution at a given value (gpl), it is imperative to have a fair idea of the extent of metal recovery at a given time for a particular dump or heap. In order to achieve this goal, leaching rate working curves must be constructed as follows:

1. Recovery as a function of time for different types of ores (from the same deposit) encountered during the leaching operation under dump or heap leaching situations.

Such working curves for the different types of ores under consideration may be constructed by establishing the recovery at any one time for each type of ore by column leaching tests and by utilizing the following general mathematical expression (Pade's approximation):

$$\text{Recovery} = \frac{at}{(b + t)} \qquad (8)$$

where,

a = maximum recovery possible,

t = time for obtaining specified recovery, and

b = a constant depending on the type of ore under consideration and depth of the dump or heap and numerically equal to the time required to obtain 50% of the maximum recovery.

It should be noted that because of the empirical nature of these working curves, they are more reliable for recovery values above 30%.

Experimental Data

In order to illustrate the construction of working curves, let us examine the experimental data (Table 1) obtained for a typical oxide copper ore containing malachite, azurite, and chrysocolla as the major oxide copper minerals along with lesser amounts of sulfide copper mineralization (the sulfide copper content being less than 25% of the total copper content).

TABLE 1

LEACHING RATES TO OBTAIN 70% RECOVERY

(day or months per given depth)

Type Ore	Size: Depth:	Minus 2 1/2 inch 4-feet	Minus 2 1/2 inch 50-feet	Minus 12-inch 4-feet	Minus 12-inch 200-feet
Type I	(+90% oxide)	4.6 days	58 days	13.0 days	21 months (1.8 years)
Type II	(+75% oxide)	6.2 days	78 days	17.5 days	29 months (2.4 years)
Type III	(+50% oxide)	10.2 days	128 days	28.6 days	48 months (4.0 years)

These results are obtained from actual laboratory tests using 4-foot columns of appropriate diameter. Let us assume that experimental data for minus 2 1/2 inch material at 50-feet depth represents a heap leaching situation while data for minus 12-inch material signifies a dump leaching operation. Also, let us assume that the leaching time is directly proportional to depth.

Working Curves

Since the times required for 70 per cent recovery for minus 2 1/2-inch material at 50-feet depth and minus 12-inch ore at 200-feet depth are known, we can now obtain the values for "b" as follows:

Type Ore	Values of "b" 50-feet depth	200-feet depth
Type I	24.9	9.0
Type II	34.4	12.4
Type III	55.0	20.3

Using the above values of "b" and for given time "t" it is now possible to construct the curves for the three types of ores for the heap leaching situation (Fig. 4) and the dump leaching operation (Fig. 5).

Using the above leaching rate working curves, it is now possible to develop a tentative production schedule for predicting the length of time each dump or heap would be leached, the grade of the pregnant solution from each heap or dump (since the quantity of leach solution is fixed the concentration of copper in the leach solution at a known time is also fixed) and the daily copper production throughout the life of the mine. Such information is invaluable in predicting the net cash flow as well as the return on the investment (ROI), the criteria on which the modern mining business is based.

It is recognized that, in the construction of the leaching rate working curves and in the consequent development of the production schedule, the data and the techniques used are at best engineering approximations. Consequently, the results are not absolute, but are nevertheless, thought to be of value in predicting trends and possible problems in trying to achieve the optimum production rate as well as operation conditions.

Efforts are currently under way to correlate the usefulness of such working curves with actual plant operations. Moreover, additional laboratory experiments are being conducted to develop techniques for scale up from laboratory data to actual plant practice. Scale up rules are being developed which take into account variations in: (a) size distribution of the material being leached, (b) changes in the rate controlling step during leaching operation, and (c) fluctuations in solution flow rate through the material being leached. It is hoped that the results of these studies may enable predictions of leach solution grades and recoveries

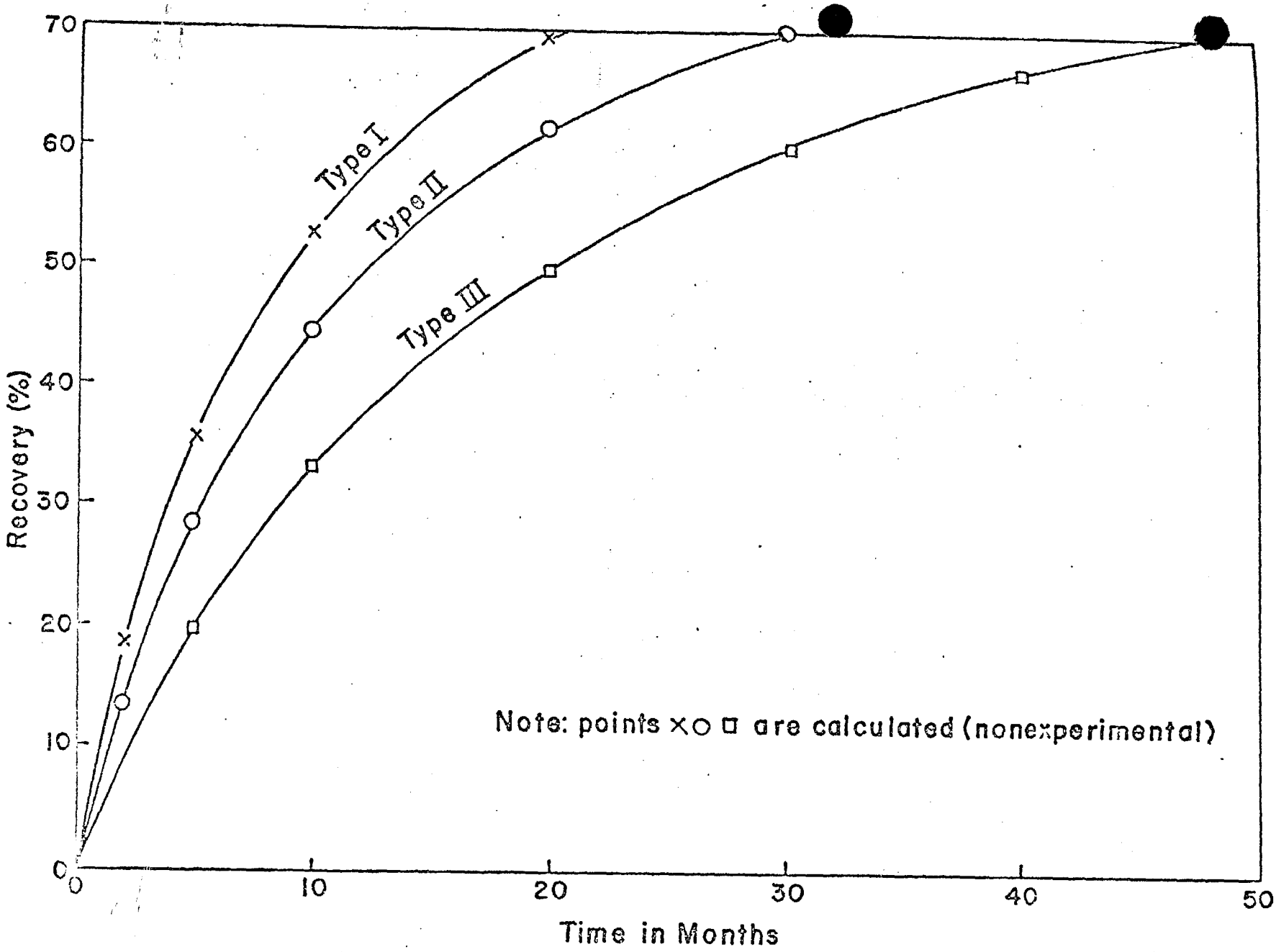

Figure 5 Recovery as a Function of Time, Dump Leaching Situation
(Ore Size = ~12"; Dump Height = 200')

in a large dump or heap to be made on the basis of three simple laboratory tests.

Computer Programme for Estimating Production

A computer algorithm may be developed to predict copper production and acid consumption under different options of mine operation, which should be based on drill core data, metallurgical test data, and a mine plan. In order to ensure maximum usefulness from such a programme, the following factors should be considered in its development.

First, each stope should be assigned the mineralogy, metallurgy, and physical characteristics of the four or more surrounding drill holes weighted by the distance from each of the drill holes.

Second, detailed laboratory testing should be conducted to determine the rate at which acid is consumed by the rock (as distinct from the copper minerals) and the total amount of acid one ton of rock will consume. This should be done for all samples or until it is proven that both rate and total consumption are independent of location. This data should then be incorporated into a form compatible with the model of acid consumption.

Third, the decision to add 0, 1, or 2 new stopes to the system should be based on results of calculations for each of the three parameters (mineralogy, metallurgy and physical characteristics) rather than on the results of a past operational period. This would eliminate some overshooting and undershooting taking place in a production schedule.

The computer programme has two major areas of usefulness. The first is to predict production levels from different operating options. The second is in conducting a sensitivity analysis on the variables of ore grade, ore tonnage, copper recovery, acid consumption and average solution grade. The sensitivity analysis would be valuable in suggesting areas in which the data should be firmed up.

Appendix 1

PRESENT DAY UNIT MINING COST IN UNITED STATES

150 Ft. x 140 Ft. Block
24,000 Sq. Ft. (includes pillar)

150 ft. Conveyor Drift at $135	$ 22,250
5 Chutes at $1,000	5,000
100 ft. Conveyor Raise at $30	3,000
800 ft. Slusher Drift at $110	88,000
150 ft. Fringe Drift at $110	16,500
1,600 ft. Draw Raises at $10	16,000
Undercutting 21,000 sq. ft. at $3/sq. ft.	63,000
Sub-Total $8.40/sq. ft.	$213,750

To Introduce Solution

150 ft. Distribution Drift at $200	$ 30,000
1,600 ft. Drill Holes at $5	8,000
Sub-Total $1.58/sq. ft.	$ 38,000
TOTAL	$251,750

$$\frac{\$251,750}{24,000} = \$10.50 \text{ per sq. ft.} \qquad (9)$$

The average head to be broken is 425 ft. with 10% of the column to be mined, this gives 42.5 ft. to be mined 15 ft. of this tonnage will be mined in the undercutting. This gives 42.5 ft. or 3.4 tons per sq. ft. to draw to break the ore column.

Per Ton		
Drawing	=	30¢
Conveying	=	5¢
Hoisting	=	15¢
Dumping	=	10¢
TOTAL		60¢

$$3.4 \times 60¢ = \$2.04 \text{ per sq. ft.} \qquad (10)$$

TOTAL $2.04 + $10.50 = $12.54/sq. ft.

Appendix 2

CAPITAL COST

Hoisting shaft 16' dia 2425' at 800	$ 1,940,000
Design	50,000
Exploration and ventilation station	50,000
Haulage station	90,000
Pocket	150,000
Contingency at 15%	342,000
Sub-Total	$ 2,622,000
Three 40" dia ventilation holes 2425 at 150	$ 1,091,000
Pump station	300,000
Electrical sub-station	100,000
Explorator entry and ventilation drift 2650 ft. at 200	530,000
Ventilation drift 5,000 ft. at 150	750,000
Haulage drift 2,600 ft. at 200	520,000
Supply elevator 500 ft. at 200	100,000
Transfer raises 2,000 ft. at 100	200,000
Contingency at 15%	539,000
Sub-Total	$ 4,130,000
Hoist house, change room, office and shops	$ 500,000
Hoist	900,000
Head frame to bins	200,000
Cage and skips	80,000
Mine equipment	500,000
Electrical equipment, transformers, etc.	400,000
Pumps	300,000
Pipe	250,000
Belt conveyors 10,000 ft. at 100	1,000,000
Elevator	250,000
Water development	1,000,000
4 Blocks developed	1,154,000
Sub-Total	$ 6,534,000
Surface plant, SX, etc.	$14,000,000
Working Capital	2,000,000
TOTAL	$29,286,000

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