RI 8236

Bureau of Mines Report of Investigations/1977

In Situ Leaching Research in a Copper Deposit at the Emerald Isle Mine

UNITED STATES DEPARTMENT OF THE INTERIOR

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By Dennis V. D'Andrea, William C. Larson, Larry R. Fletcher, Peter G. Chamberlain, and William H. Engelmann

UNITED STATES DEPARTMENT OF THE INTERIOR
Cecil D. Andrus, Secretary

BUREAU OF MINES

This publication has been cataloged as follows:

D'Andrea, Dennis V

In situ leaching research in a copper deposit at the Emerald Isle mine / by Dennis V. D'Andrea ... [et al.] [Washington] : Bureau of Mines, 1977.

43 p. : ill., maps, diagrams ; 26 cm. (Report of investigations • Bureau of Mines ; 8236)

Bibliography: p. 42-43.

1. Copper ores. 2. Leaching. I. United States. Bureau of Mines. II. Title. III. Series: United States. Bureau of Mines. Report of investigations • Bureau of Mines ; 8236.

TN23.U7 no. 8236 622.06173

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IN SITU LEACHING RESEARCH IN A COPPER DEPOSIT AT THE EMERALD ISLE MINE

by

Dennis V. D'Andrea,[1] William C. Larson,[2] Larry R. Fletcher,[3]
Peter G. Chamberlain,[1] and William H. Engelmann[4]

ABSTRACT

The Bureau of Mines and El Paso Mining and Milling Co. conducted a cooperative in situ leaching research program at the Emerald Isle mine near Kingman, Ariz. The objective of this research was to develop in situ leaching methods for 200,000 tons of ore exposed in the pit bottom and also 3,000,000 tons of ore under 200 feet of overburden adjacent to the pit. The research included core drilling for fragmentation analysis, inplace permeability testing, seismic surveys, blasting tests, blast vibration measurements, inplace leaching, and ground water monitoring.

A 15,000-ton test area in the pit bottom was blasted and leached inplace for 117 days. This successful test was followed by leaching 100,000 additional tons of unblasted pit bottom ore. Two test blasts, under 205 feet of overburden and extending to 290 feet, were detonated in an area near the open pit. The second blast was detonated because the first blast did not create adequate permeability for leaching. Pit bottom leaching was discontinued and the test area under 200 feet of overburden was not leached because the company terminated operations at the Emerald Isle mine. Based on the research completed, an expanded in situ leaching system was designed, but not implemented, to recover copper from a 1,500,000-ton area at the Emerald Isle mine.

INTRODUCTION

The Bureau of Mines is conducting research to develop technology for in situ copper leaching (3-6, 11-12).[5] Although in situ leaching is generally considered applicable to deposits that are too low in grade to be mined by conventional methods, it can also be applied to higher grade deposits as an

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

alternative to conventional mining. The advantages of in situ leaching are lower capital investment, shorter development time, reduced health and safety hazards, minimal detrimental environmental effects, and lower energy consumption during mining and beneficiation.

In situ copper leaching has been practiced at Miami, Ariz. (8); the Old Reliable mine near Mammoth, Ariz. (7); the Zonia mine near Kirkland Junction, Ariz.; the Big Mike mine near Winnemucca, Nev. (10); and the Mt. City mine in Mt. City, Nev. (2). Although initial results from these operations have been encouraging, the depressed price of copper has resulted in temporary or permanent closing of all of these mines. However, an improved copper market will encourage companies to resume some of these operations.

The major problems with in situ leaching are (1) inadequate reliable methods for evaluation of potential in situ leaching properties including estimating chemical reagent consumption costs and final copper recoveries, (2) insufficient fragmentation systems which can produce adequate permeability at reasonable costs, (3) inadequate leach solution injection and recovery systems that assure continued contact between leach solutions and broken ore, (4) final copper recoveries must be maximized, and (5) ground water contamination must be prevented. Although the basic technology exists to conduct in situ leaching, this technology must be improved and presently available equipment must be modified (9) to increase efficiency and economy.

This report presents the results of a Bureau of Mines-El Paso Mining and Milling Co. cooperative in situ leaching research program at the Emerald Isle mine near Kingman, Ariz. Both the Twin Cities (Minn.) Mining Research Center and the Salt Lake City (Utah) Metallurgy Research Center of the Bureau of Mines participated in this program. Research was conducted in two phases. In the phase I area ore was exposed on the surface of the pit bottom, and in the phase II area the ore was buried under 205 feet of overburden.

ACKNOWLEDGMENTS

The authors acknowledge the cooperation and assistance of El Paso Mining and Milling Co., El Paso, Tex., and particularly its employees--Sidney Runke, chief metallurgist; Harold Horst, mining engineer; and Lester Mead, mine superintendent. The cooperation and assistance of George M. Potter, metallurgist, and Rees D. Groves, metallurgist, Federal Bureau of Mines, Salt Lake City (Utah) Metallurgy Research Center, is also greatly appreciated.

EMERALD ISLE DEPOSIT

Geology

The cross section through the Emerald Isle deposit, figure 1, shows vertical variations in copper grade in the mineralized zone, the phase I test area in the pit bottom, and the phase II test area under 205 feet of overburden. The deepest portions of the open pit have been excavated to about 200 feet exposing the top of the ore (locally known as the Gila Conglomerate). During open pit operations El Paso mined 1,400,000 tons of ore that averaged

1.0 percent copper. Conventional open pit mining stopped because of increased stripping costs as the operation proceeded in the downdip direction.

Copper mineralization occurs in the Gila Conglomerate which averages about 70 feet thick and dips 10° to 15° to the southwest. A plan view of the mine, figure 2, shows contours of the grade in the Gila Conglomerate and again shows the phase I and phase II test areas. The copper mineralization

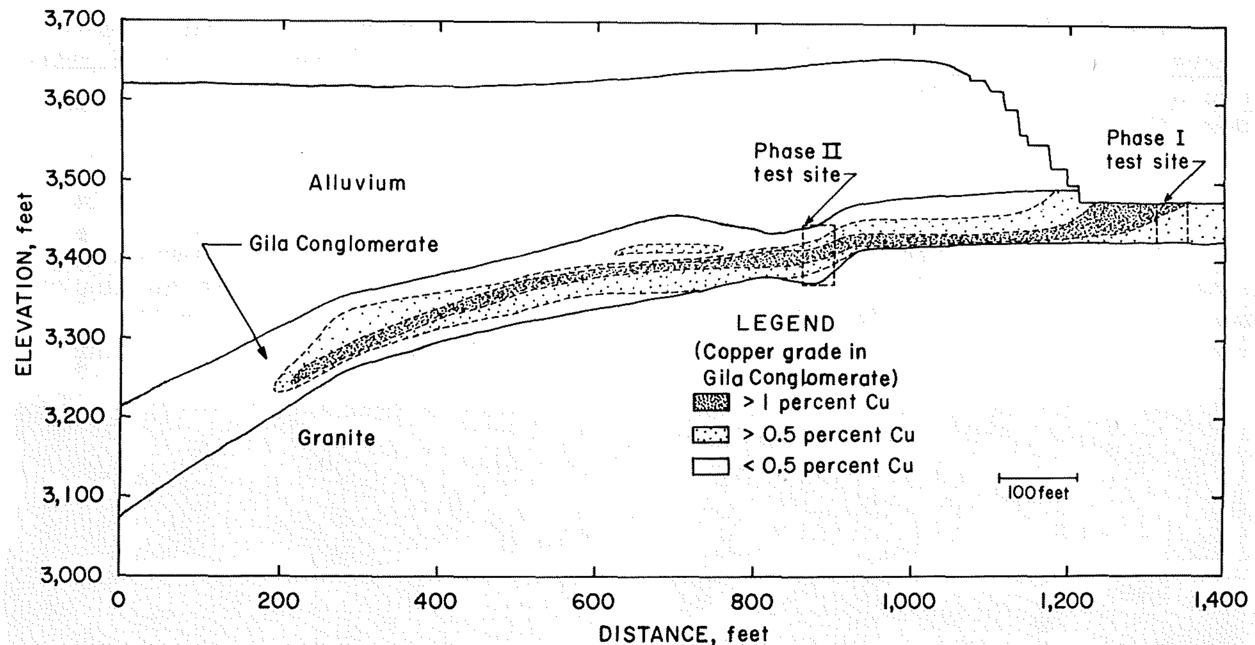

FIGURE 1. - Cross section through the Emerald Isle mine showing copper grade and phases I and II test sites.

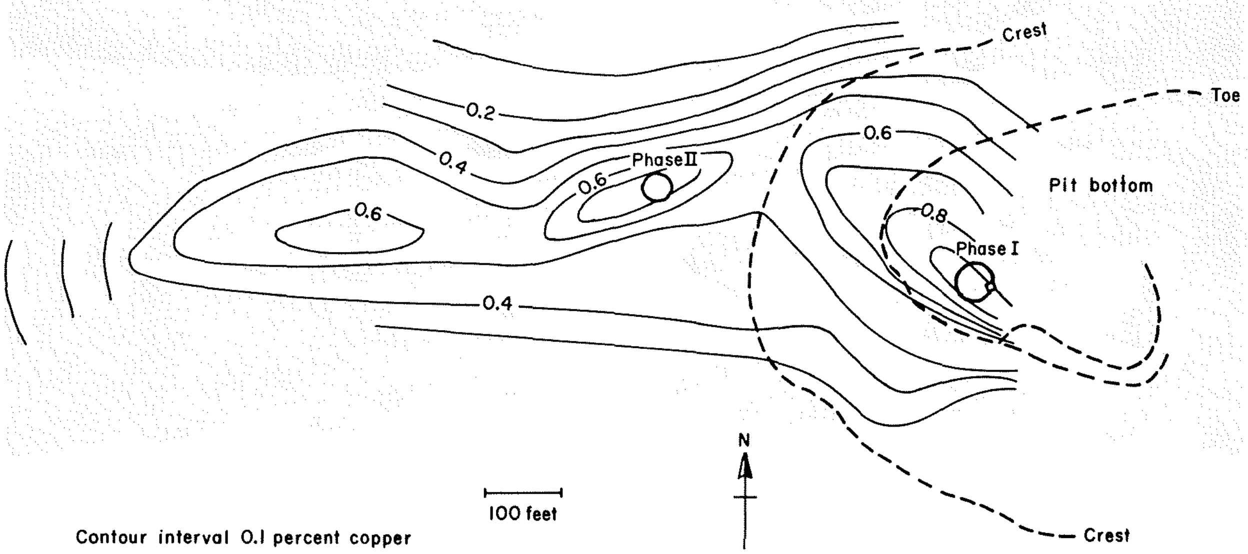

FIGURE 2. - Plan view of the Emerald Isle mine showing copper grade and phases I and II test sites.

continues beyond the pit in the downdip direction in a channel-type deposit that generally decreases in grade. At 1,000 feet west of the pit crest the grade of the ore is about 0.1 percent. Total copper resources are estimated to be about 3,000,000 tons of ore greater than 0.1 percent and 1,500,000 tons of ore at a grade above 0.4 percent. Figure 3 is a view of the pit and surface facilities at the Emerald Isle mine looking west.

Mineralization

The dominant copper mineral in the Gila Conglomerate at the mine is chrysocolla ($CuSiO_3$ 2 H_2O). Minor amounts of dioptase, tenorite, and cuprite are also present. The geology and mineralization of this deposit have been described previously (13).

Ground Water

Ground water entering the pit at the Emerald Isle mine was pumped from a sump area for about 30 hours each week at 90 gpm. The water table in the pit was maintained about 5 feet below the surface of the pit floor. In the phase II test area the water table was 235 feet below the surface. Ground

FIGURE 3. - A view of the Emerald Isle mine looking west.

water table elevations decreased in the downdip direction tending to follow the granite gneiss bedrock in the area around the open pit.

Figure 4 shows the water table elevations in the pit before leaching. Figure 5 shows these elevations during the phase I leach test when the water table was drawn down about 22 feet at the recovery well, which created a cone of depression.

An attempt was made to establish the pattern of ground water movement in the pit bottom using a red dye (Rhodamine B). The dye was injected into a drill hole in the middle of the phase I test area and water samples were taken from monitor holes around the test area. This test was unsuccessful because none of the dye was ever detected at any of the monitor holes. Another test conducted before and after blasting showed much more rapid dissipation of dye from the injection hole after blasting. The major problem with using Rhodamine B dye was its rapid absorption by the host rock.

Rock Properties

Table 1 lists the physical properties of the Gila Conglomerate at the Emerald Isle mine from phases I and II areas as measured in the laboratory using core samples. The porosity, permeability, and strength values probably reflect the characteristics of the cementing material in the conglomerate and not the actual in situ values.

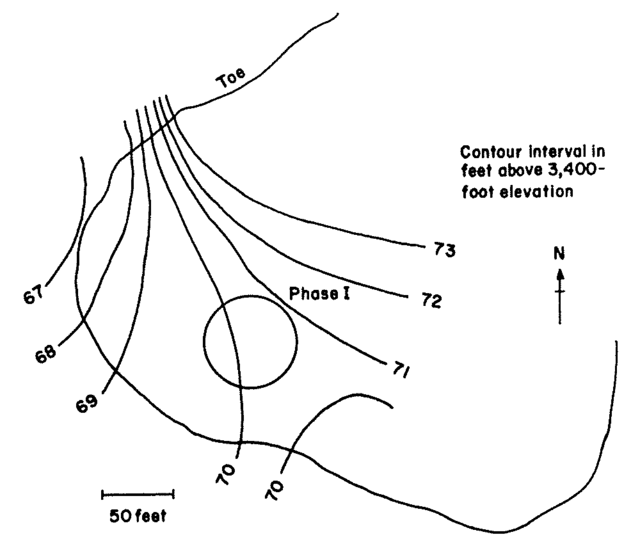

FIGURE 4. - Ground water elevations before the phase I leach test.

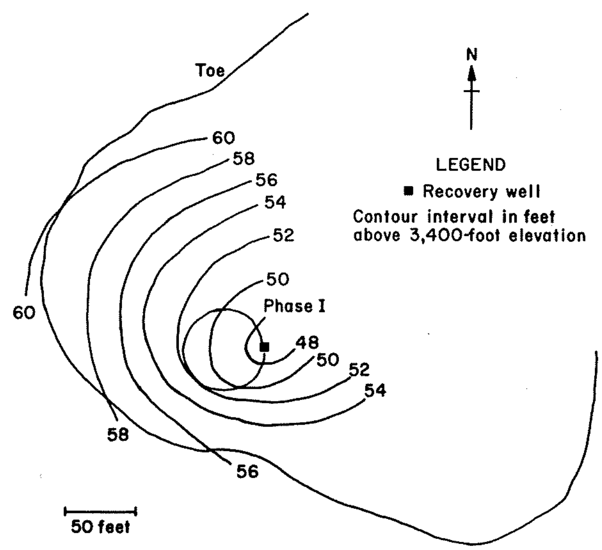

FIGURE 5. - Ground water elevations during the phase I leach test.

TABLE 1. - Physical properties of the Gila Conglomerate at the Emerald Isle Mine

Property	Phase I	Phase II
Porosity....percent..	20.6	16.3
Density....g/cu cm..	2.29	2.28
Permeability....darcies..	-	.65
Longitudinal velocity....ft/sec..	8,100	9,500
Torsional velocity....ft/sec..	4,200	5,100
Compressive strength....lb/sq in..	2,600	-
Tensile strength....lb/sq in..	72	-
Young's modulus....10^6 lb/sq in..	0.85	-

Laboratory Leaching Tests

Figure 6 shows the results of laboratory acid trickle leach tests on minus 0.5-inch ore from the phase I test area. These tests, conducted at the Salt Lake City (Utah) Metallurgy Research Center, show the effect of pH on the rate of copper recovery indicating that a pH of 1.0 would be desirable. During these tests acid consumption averaged about 6 pounds of H_2SO_4 per pound of copper, and iron consumption averaged 1.3 pounds of iron per pound of copper.

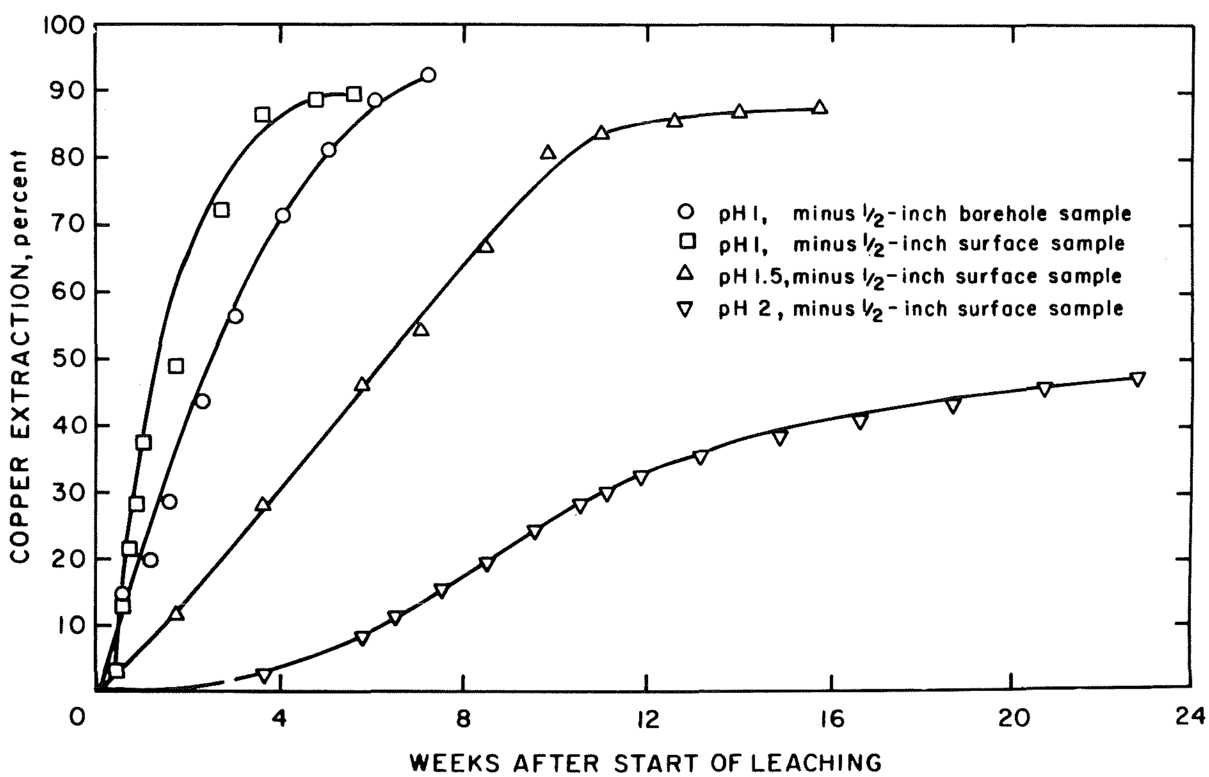

FIGURE 6. - Acid trickle leach test on ore from the Emerald Isle mine.

DRILLING PROCEDURES

Core Holes

Core holes were drilled before and after blasting in both test areas. These holes, drilled with a double-barrel wire line system using drill mud, produced core about 2 inches in diameter. In the phase II area the top 200 feet of overburden were drilled with a rotary system and cased before core drilling began in the ore zone below 200 feet (fig. 7). Drilling into ore that had been broken by blasting did not present any unusual problems and was done with return circulation of drill muds.

Blastholes

All blastholes were drilled with a rotary system using tricone bits and circulating drill mud. Blastholes in the phase I area were 8-3/8 inches wide and about 50 feet deep. In the phase II area blastholes were 9 inches wide and about 280 feet deep.

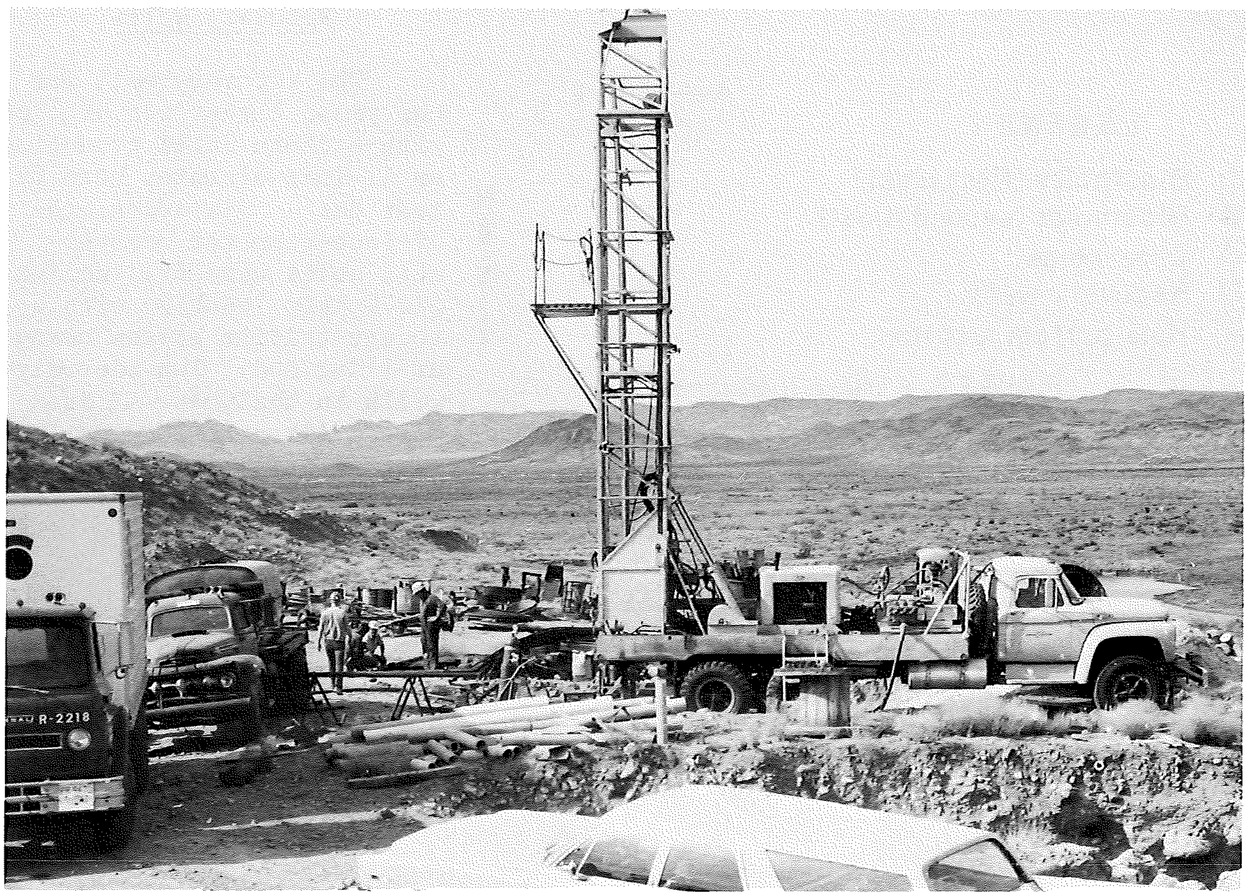

FIGURE 7. - Core drilling in the phase II area.

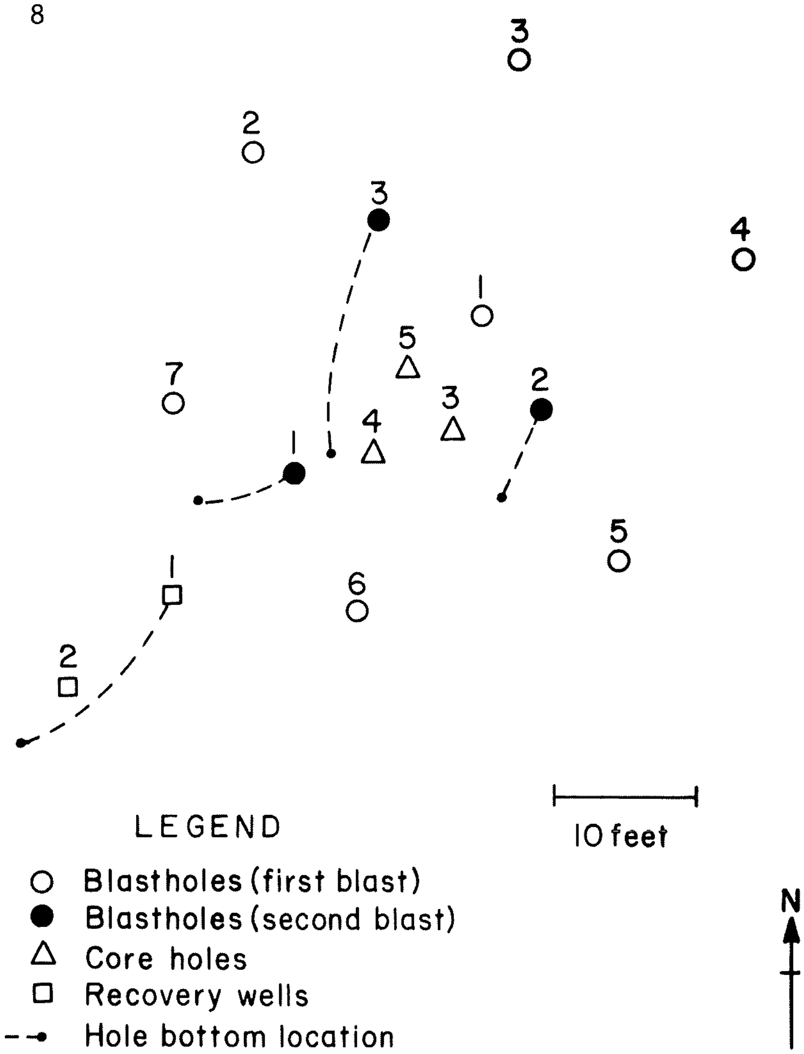

FIGURE 8. - Phase II test area.

Borehole deviation surveys were run on the three 9-inch-diameter holes for the second phase II blast and one of the 12-inch-diameter recovery wells in the phase II area. The horizontal borehole deviations are shown in figure 8. The maximum borehole deviation was 14 feet for the 280-foot-deep blastholes and 12 feet for the 300-foot recovery well. These borehole surveys point out the need for improved drilling methods that will minimize borehole deviations.

Recovery Wells

The recovery well for the phase I test was drilled with a churn drill producing an 11-inch-diameter hole 60 feet deep. Six additional 12-inch-diameter recovery well holes were drilled for pit bottom leaching with a rotary tricone system using drilling mud. Two recovery wells in the phase II area were drilled to 300-foot depths with 12-inch-diameter tricone bits and circulating drill mud. All recovery holes were cased with 10-inch diameter polyvinyl chloride (PVC) casing with the bottom 20 feet perforated.

Monitor Holes

Four-inch-diameter monitor holes in the pit bottom were drilled about 50 feet deep with an airtrack drill. These holes were not cased and remained open after the phase I test blast. Monitor holes in the phase II area were drilled with a rotary tricone system to depths of 320 to 360 feet and cased with NX (3-inch-diameter) casing. The bottom 60 feet of this casing was perforated.

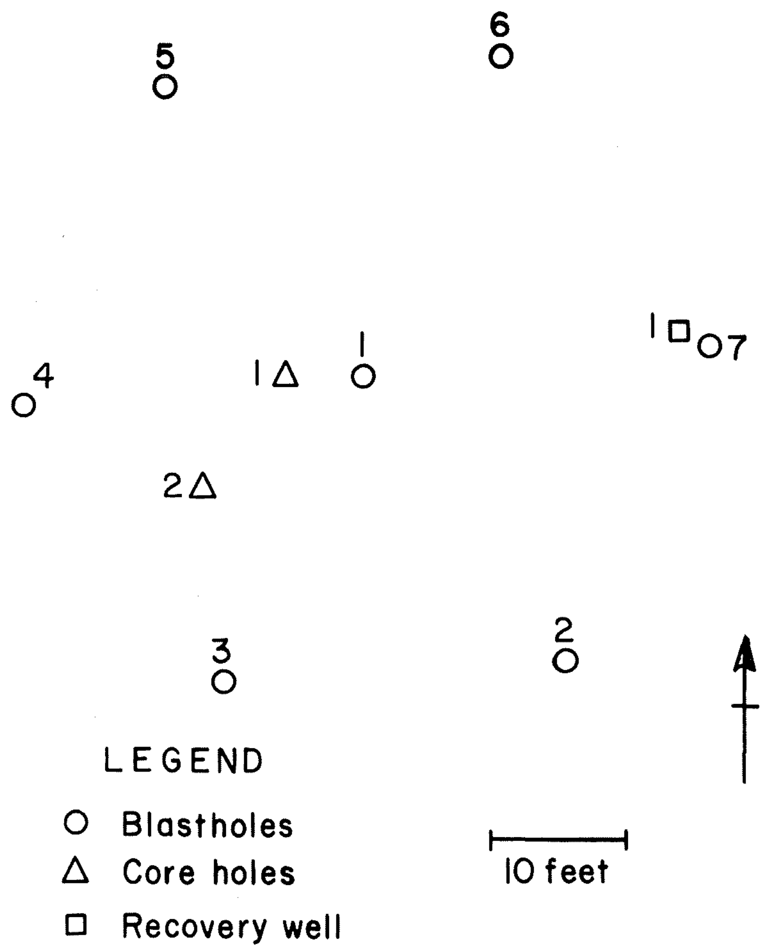

FIGURE 9. - Phase I test area.

BLASTING PROCEDURES

Phase I Blast

Figure 9 shows the phase I test blast design. Seven 8-3/8-inch-diameter blastholes, about 50 feet deep, were spaced 25 feet apart in a seven-spot pattern with one central blasthole. These holes had an average 22-foot powder column with 25 feet of stemming. A total of 4,500 pounds of slurry was detonated without delays. The slurry was bulk loaded using a slurry pump truck (fig. 10). Table 2 lists the data for the phase I blast and the two phase II blasts.

The powder factor of 0.78 lb/ton listed in table 2 for the phase I blast was calculated for an individual hole in an equilateral triangle pattern and does not include any volume of ore above or below the powder column. For this situation the powder factor (PF) in pounds per ton is given by

$$PF = 1,814 \frac{Pe}{Pr} \left(\frac{D}{S} \right)^2 ,$$

where Pe = specific gravity of explosive,

Pr = specific gravity of ore,

D = blasthole diameter, feet,

and S = blasthole spacing, feet.

TABLE 2. - Test blast design data

	Phase I	Phase II	
		1st blast	2d blast
Number of blastholes..........	7	7	3
Blasthole spacing.......feet..	25	20	18
Average hole depth.......do...	47	277	277
Average top powder column..................do...	25	205	192
Average powder column....do...	22	72	85
Average stemming.........do...	25	205	192 (90 ft gravel)
Blasthole diameter....inches..	8-3/8	9	9
Explosive diameter......do....	8-3/8	7	7-in bags cut
Explosive.....................	Nonaluminized slurry	Smokeless powder slurry	Smokeless powder slurry
Total explosives......pounds..	4,500	12,000	7,450
Loading density........lb/ft..	29.2	23.7	29.3
Powder factor[1]........lb/ton..	0.30	-	-
Powder factor[2]........lb/ton..	0.78	0.95	1.47
Delays between each hole...........milliseconds..	Instantaneous	17	25

[1] Powder factor includes ore above top of powder column.
[2] Powder factor for ore in powder column zone only.

This formula can be used to calculate powder factors in any situation where the powder column extends from the top to the bottom of an ore zone buried under overburden as with the two phase II blasts. For the phase I blast, if the ore above the powder column in the 22-foot stemming region is included, the powder factor becomes 0.30 lb/ton. This value was considerably lower than the powder factors of from 0.67 to 0.89 lb/ton used at the Old Reliable, Zonia, and Big Mike blasts (11).

Phase II Blasts

Two blasts were detonated in the phase II test area under 200 feet of overburden. The first blast had seven 9-inch-diameter blastholes drilled to an average depth of 277 feet. Blastholes were spaced 20 feet apart, again in a seven-spot pattern, with one central blasthole. Figure 8 shows the blasthole patterns for the phase II blasts. The second blast had three blastholes spaced 18 feet apart. Blast design data for the phase II blasts are listed in table 2. Powder factors were 0.95 lb/ton and 1.47 lb/ton for the first and second blasts, respectively.

FIGURE 10. - Phase I blasthole loading.

The first blast was loaded with 7-inch-diameter, 50-pound bags, of smokeless powder slurry in 9-inch-diameter blastholes. The second blast was loaded with the same explosive but the bags were cut before lowering them again into 9-inch holes as shown in figure 11. The cutting procedure, where several 3-inch slits were cut in each bag, increased the loading density from 23.7 lb/ft for the first blast to 29.3 lb/ft for the second blast. The 50-pound bags of slurry were lowered to the top of the water column in each blasthole on a lowering rope with a release hook. At the top of the water column the bags were released and allowed to freefall through the water to the bottom of the blastholes. This blasthole loading procedure was time-consuming but all blastholes were successfully loaded.

FIGURE 11. - Cutting 50-pound bags of slurry before lowering into blastholes.

Blast Swell

Detailed topographic surveys were run to determine elevation changes produced by the blasts. Figure 12 is a contour map of elevation increases produced by the phase I blast. The maximum surface rise for this blast was only about 1.4 feet. A swell factor, defined as the final volume divided by the original volume, is difficult to determine because the radius of blast damage needed for the volume calculation was not accurately known. With the assumption that the original volume was contained in a cylinder 47 feet deep and 75 feet wide, the swell factor for the phase I blast was 1.014 with a volume increase of 110 yd^3.

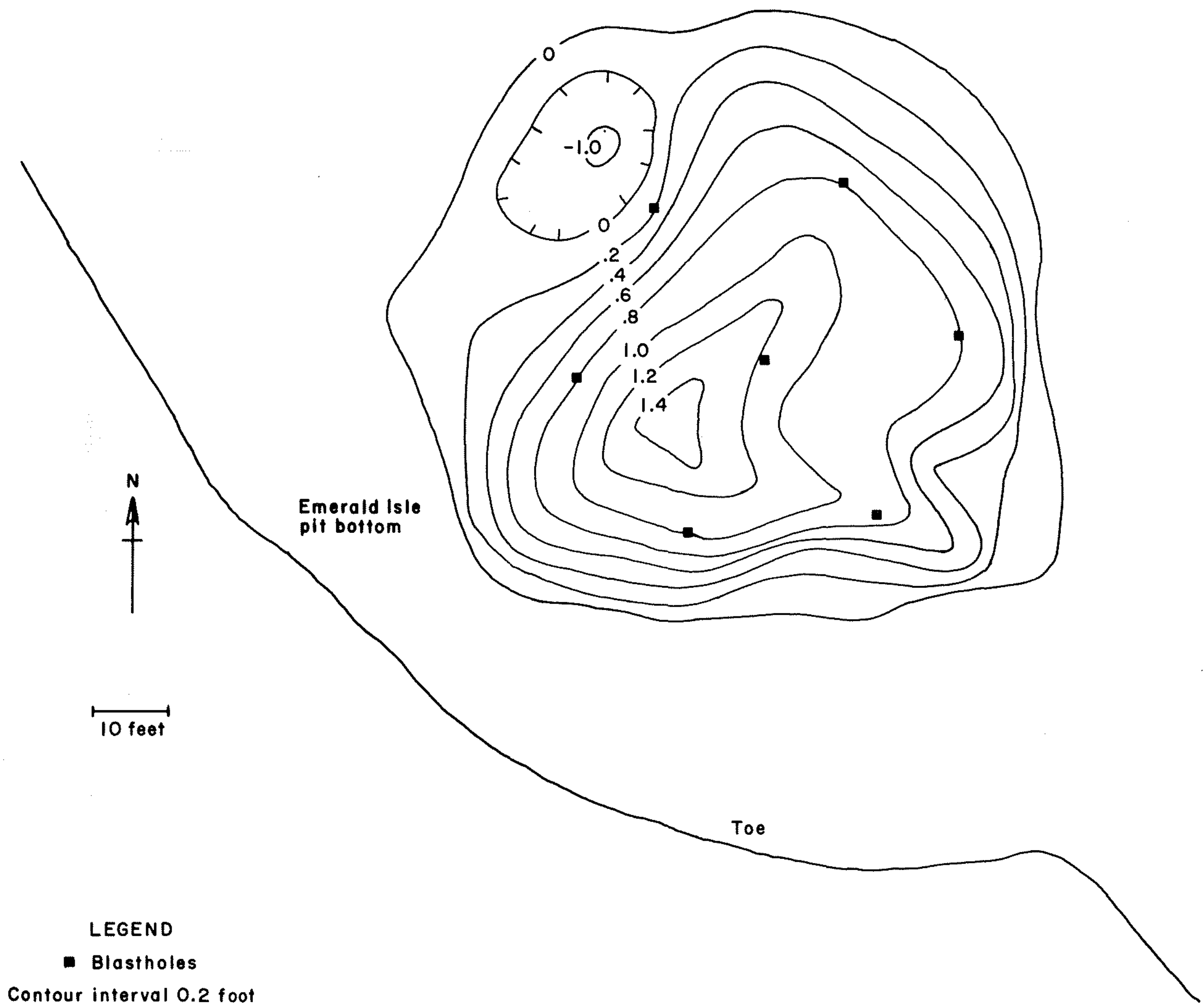

FIGURE 12. - Elevation changes produced by the phase I blast.

Elevation increases produced by the phase II blasts were generally less than 0.2 foot, which was about the estimated accuracy of the topographic surveys. Elevation increases of about 0.5 foot were observed near the collars of the blastholes that vented but these increases were considered to be a local effect.

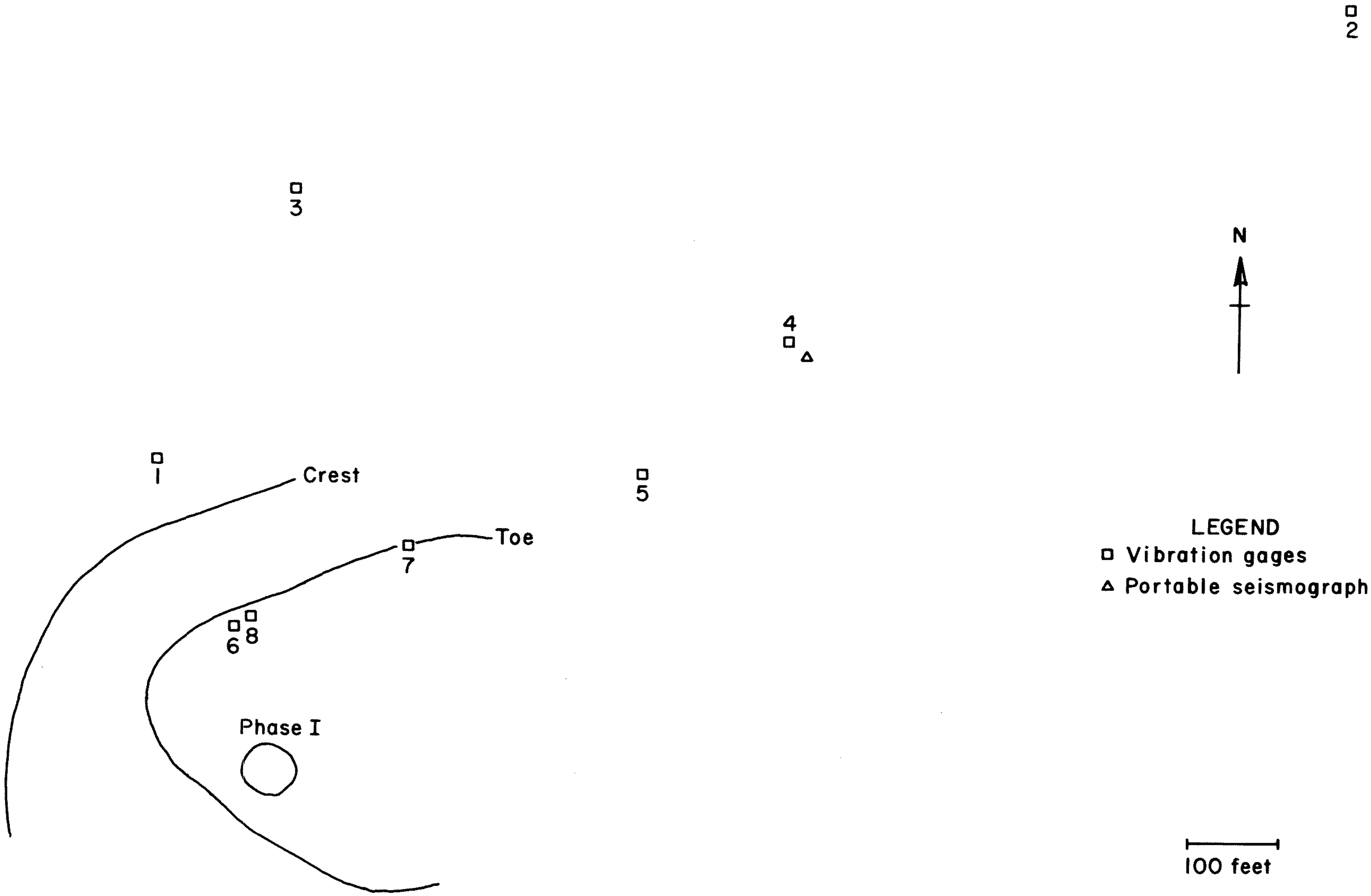

FIGURE 13. - Plan view of blast vibration gage locations for the phase I blast.

Blast Vibrations

Blast vibrations were measured for the phase I blast and the first phase II blast. Particle velocity gages and a 14-channel FM tape recorder were used to record blast vibrations. A portable three-component seismograph was also used for the phase I blast. The gage locations are shown in figures 13-15.

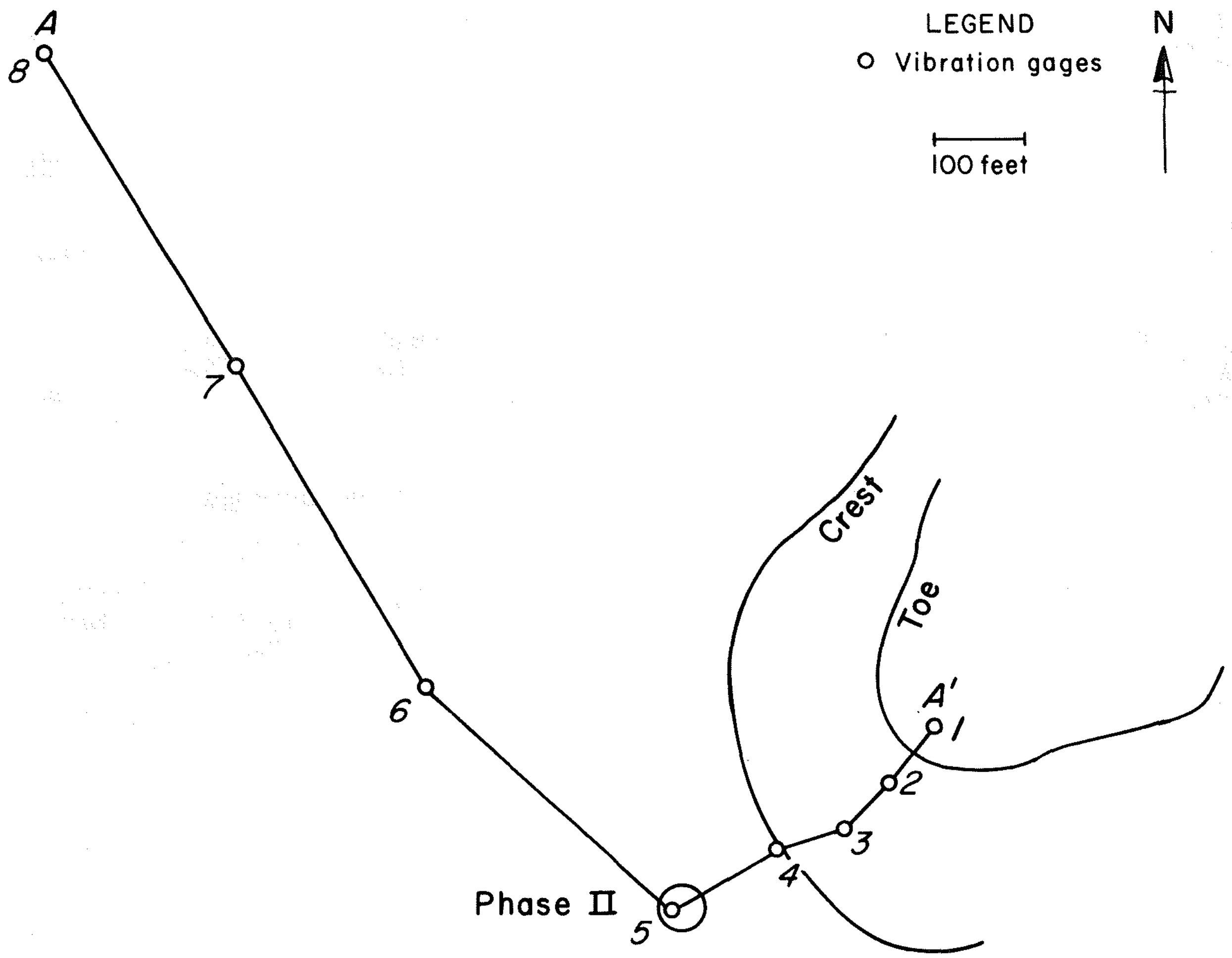

FIGURE 14. - Plan view of blast vibration gage locations for the first phase II blast.

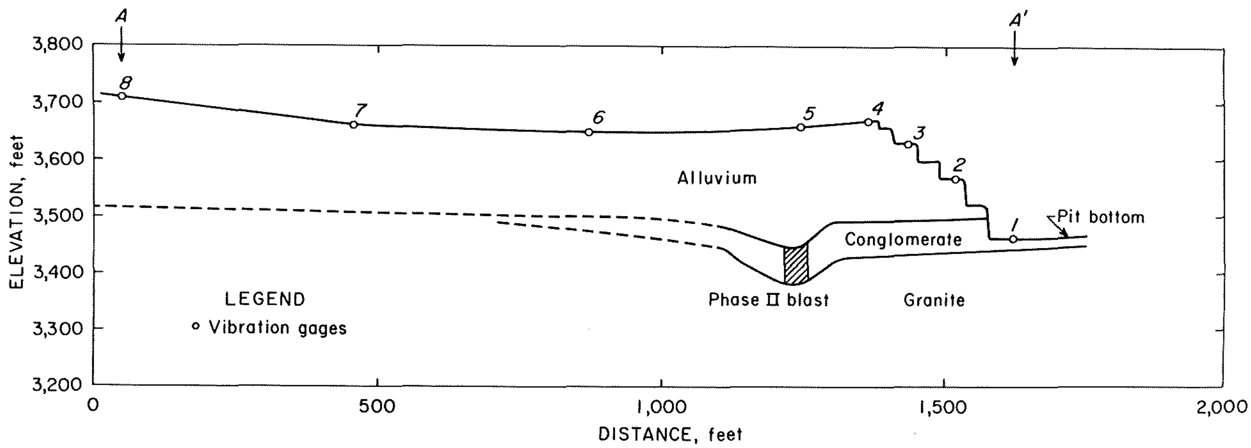

FIGURE 15. - Cross section view of blast vibration gage locations for the first phase II blast.

The peak particle velocities recorded at each gage are shown in table 3. A plot of these velocities as a function of scaled distance is shown in figure 16, along with vibration amplitude data from seismic work done in the pit bottom with 1- to 2-pound charges.

TABLE 3. - Phases I and II blast vibration measurements

Gage No.	Maximum charge weight per delay, pounds	Shot to gage distance, feet	Scaled distance, ft/lb$^{1/2}$	Peak particle velocity, in/sec	Frequency, hertz
		PHASE I			
1................	4,500	453	6.8	4.4	13
2................	4,500	1,480	22.1	.58	30
3................	4,500	682	10.2	.89	13
4................	4,500	773	11.5	.82	12
5................	4,500	547	8.2	.95	29
6................	4,500	157	2.3	2.9	22
7................	4,500	287	4.3	2.5	16
8................	4,500	166	2.5	4.9	19
Transverse........	4,500	733	10.9	1.65	9
Vertical..........	4,500	733	10.9	.60	11
Longitudinal......	4,500	733	10.9	.65	16
		PHASE II			
1................	1,858	376	8.7	1.7	67
2................	1,858	289	6.7	[1]2.2	-
3................	1,858	255	5.9	[1]3.4	28
4................	1,858	246	5.8	3.4	-
5................	1,858	200	4.6	[1]7.4	-
6................	1,858	421	9.8	[1]2.3	-
7................	1,858	809	18.8	.62	67
8................	1,858	1,223	28.8	.21	17

[1]Estimated from records that went off-scale.

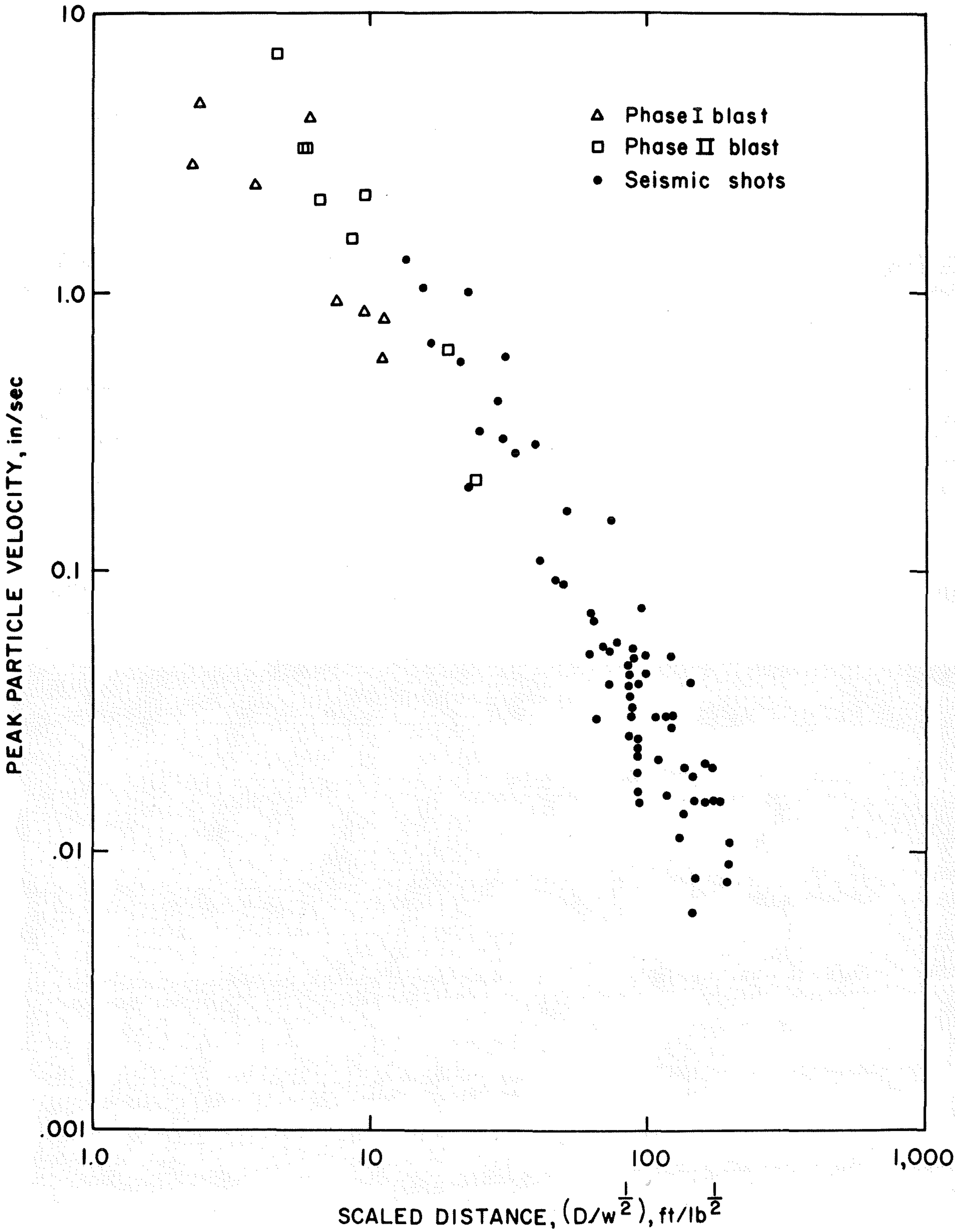

FIGURE 16. - Peak particle velocity versus scaled distance.

Concern that the blasting activities would produce slope failures in the pit walls was not warranted, nor were the vibration levels high enough to cause any damage to the plant, shop, and office facilities which were located about 300 feet from the blasts.

ANALYSIS OF BLAST DAMAGE

Drill Core Analysis

Drill core samples were obtained from the phase I test area before blasting and after leaching. Core holes were drilled in the phase II test area before blasting and after the first and second blasts. The locations of the core holes are shown in figures 8 and 9. Preshot core from the phase I test area is shown in figure 17.

All blasts had a significant effect on the fragmentation. Core recovery, RQD (rock quality designation (11)), average size, and largest piece were all reduced. The phase I postleach drill core data show that the combined effects of blasting plus leaching are significant, unfortunately, no core hole was drilled between blasting and leaching so that the specific effects of leach solutions on the ore could not be isolated.

The porosity, specific gravity, pulse velocity, core length recovery, RQD, and average size of pieces greater than 1 inch are shown as a function of depth in figures 18 and 19 for the phases I and II preshot cores. Preshot

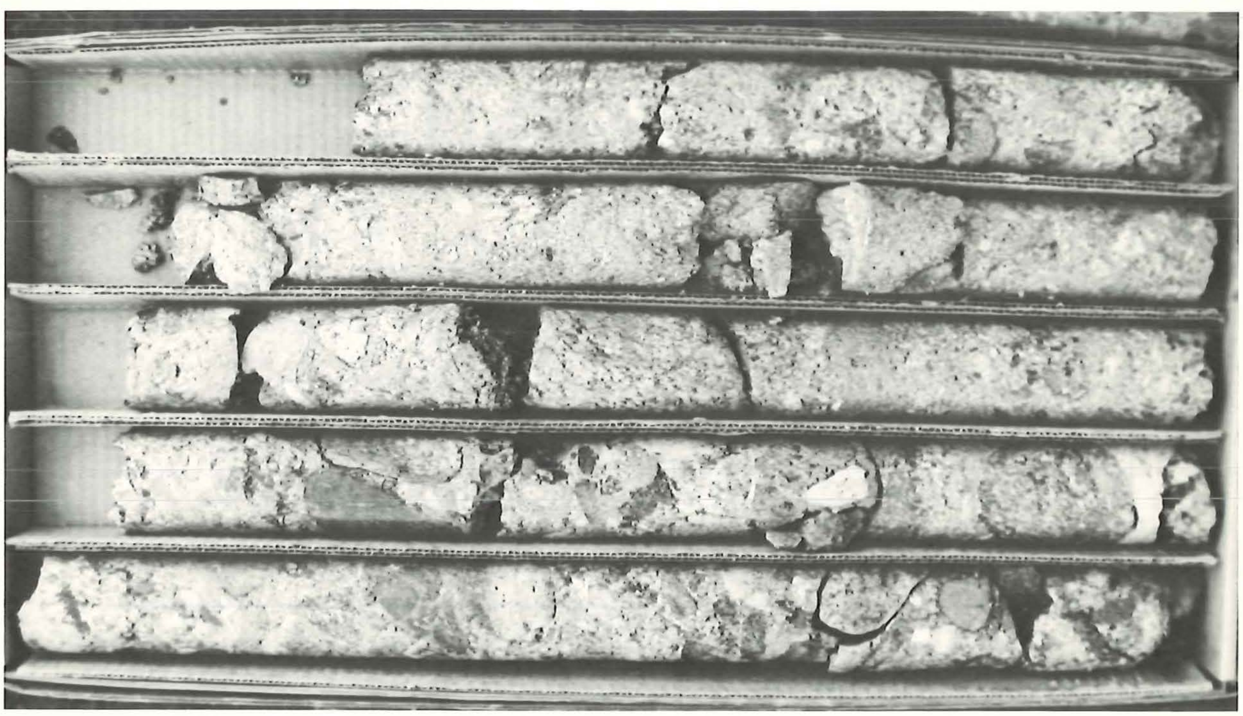

FIGURE 17. - Phase I preshot core.

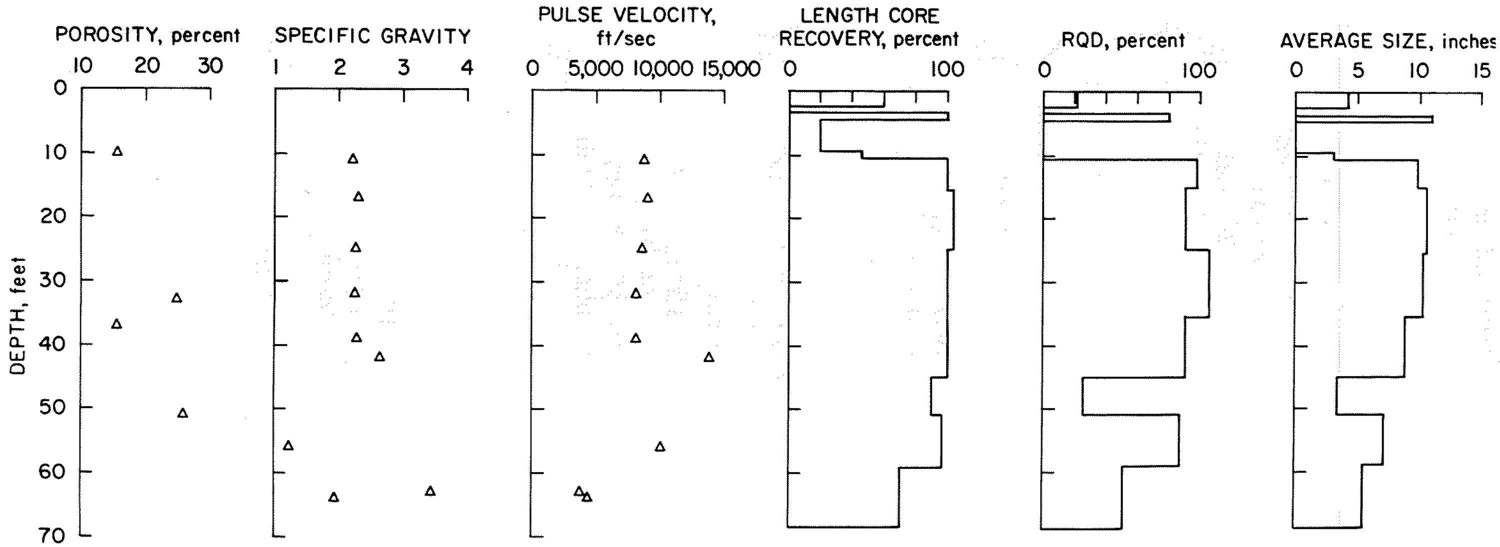

FIGURE 18. - Characteristics of phase I preshot core.

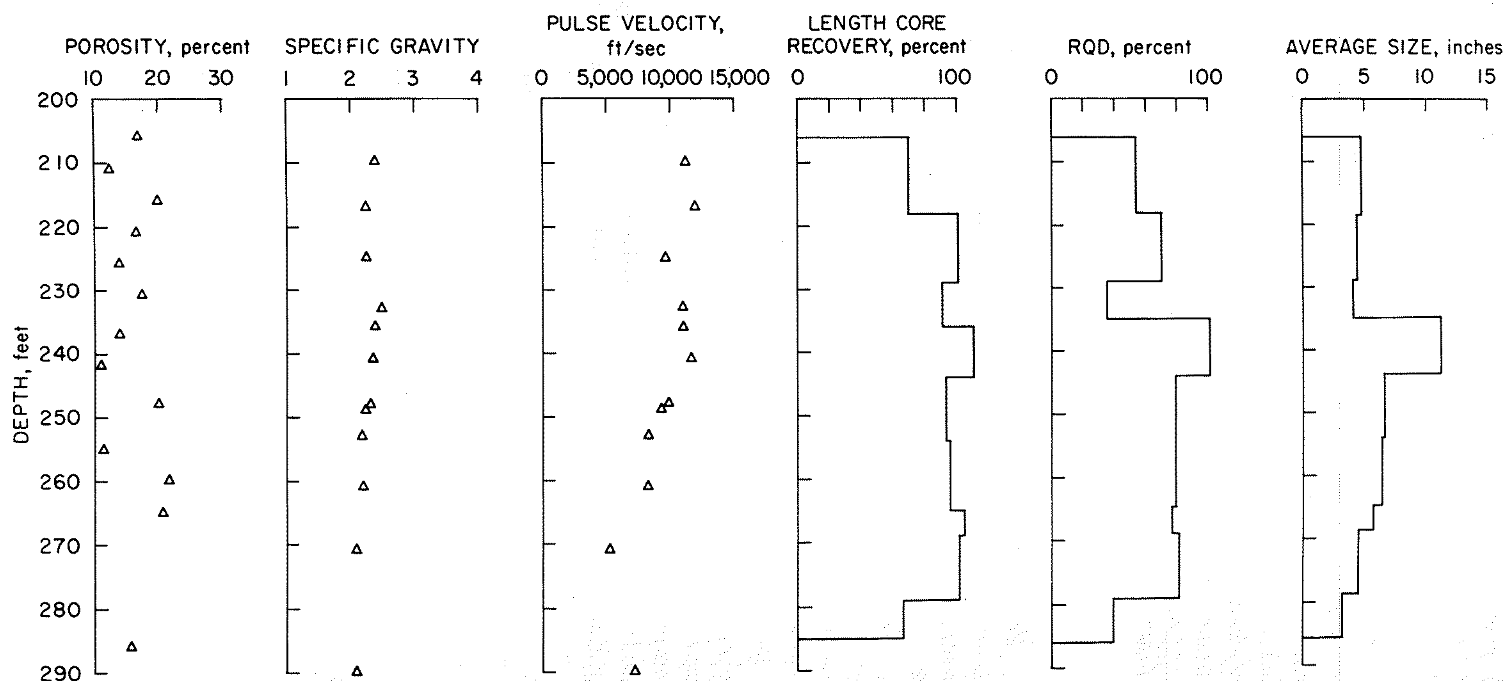

FIGURE 19. - Characteristics of phase II preshot core.

and postleach core recovery and RQD for the phase I test are compared in figures 20 and 21. The preshot and postblast data for the phase II tests appear in figures 22 and 23.

Fragmentation size distribution curves were constructed using the core length data. The length of each piece of core was assumed to represent the diameter of a rock particle that would pass through a screen with an opening equal in size to the length of the piece of core. Percent-passing versus core-length for the phases I and II drill cores are shown in figures 24 and 25 where the curves through the data were fit assuming a Weibull distribution function (16). These curves were used to determine the average size listed in table 4 from the 50-percent passing points.

Permeability Measurements

Permeability measurements were made before and after blasting in both test areas using either a constant head or a drawdown method. With the former, water is injected into a borehole to raise the water level several feet. Permeability is calculated from the dimensions of the borehole, the flow rate required to maintain a constant head, and the rise in the water level. The drawdown test requires a pump in the borehole to remove water. Permeability can be determined by pumping at a constant rate and measuring the drawdown in observation wells, or by lowering the water level in the hole, and then measuring the rise in the water level as a function of time after pumping has stopped. The measuring techniques and methods of calculating permeability are described by the U.S. Bureau of Reclamation (1, 14-15). A permeability of 0.5 darcy is considered a minimum for successful leaching of granular formations such as uranium-bearing sandstones. However, there are no guidelines for a minimum permeability for leaching formations where fluids are carried primarily through fracture networks as is the case with the Emerald Isle ore. The results of permeability measurements taken in the two test areas are listed in tables 5-6. The locations where these measurements were taken are shown in figure 26.

The permeability measurements were greatly effected by the drilling methods used. When bentonite drilling mud was used it was not possible to obtain reliable permeability values because the drill mud sealed permeable passages in the formation and generally resulted in abnormally low values. All holes in the phase I test area were drilled with air or natural ground water except corehole (CH) 1, and blastholes (BH) 2 and 5. The two blastholes were tested after blasting and yielded very large permeability values.

TABLE 4. - Drill core data

	Phase I		Phase II		
	Preshot	Postblast-leach	Preshot	Postshot	
				1st blast	2d blast
Core run...............feet..	0 to 68.6	18.0 to 72.0	206.0 to 278.5	213.0 to 256.7	200.0 to 277.5
Core recovery.......percent..	86	35	93	83	67
RQD...................do....	65	11	70	40	31
Average size (>1 inch) inches..	6.7	2.6	5.1	3.0	3.7
Average size (50 percent by weight)............inches..	6.4	<1	6.4	3.1	1.5
Largest piece..........do....	26	9	33	20	16

TABLE 5. - Permeability measurements for phase I tests

Location	Description	Preshot permeability, darcies	Postshot permeability, darcies
CH 1	Core hole	2	-
MW 1	Monitor well	-	5
MW 3	.....do	.09	27
MW 4	.....do	.02	.3
MW 5	.....do	.2	.3
MW 6	.....do	-	.2
MW 7	.....do	-	.1
BH 2	Blasthole	-	200
BH 5	.....do	-	800
RW 1	Recovery well	-	50

TABLE 6. - Permeability measurements for phase II tests

Location	Description	Preshot permeability, darcies	Postshot 1 permeability, darcies	Postshot 2 permeability, darcies
CH 3	Core hole	<0.02	-	-
CH 4	.....do	-	0.0003	-
RW 1	Recovery well	-	.002	-
RW 2	.....do	-	.0002	-
MH 1	Monitor hole	-	.0004	-
MH 2	.....do	-	6.2	-
BH 1	Blasthole	-	-	0.006
CH 5	Core hole	-	-	.020

Holes in the phase II area were all drilled with mud and the permeability values were too low for leaching. It is difficult to say whether the low permeability values are due to insufficient blast induced fracturing, overburden stresses resealing any fractures created, or drill mud clogging permeability channels. There is little doubt that the drill mud has some effect in producing low permeability values.

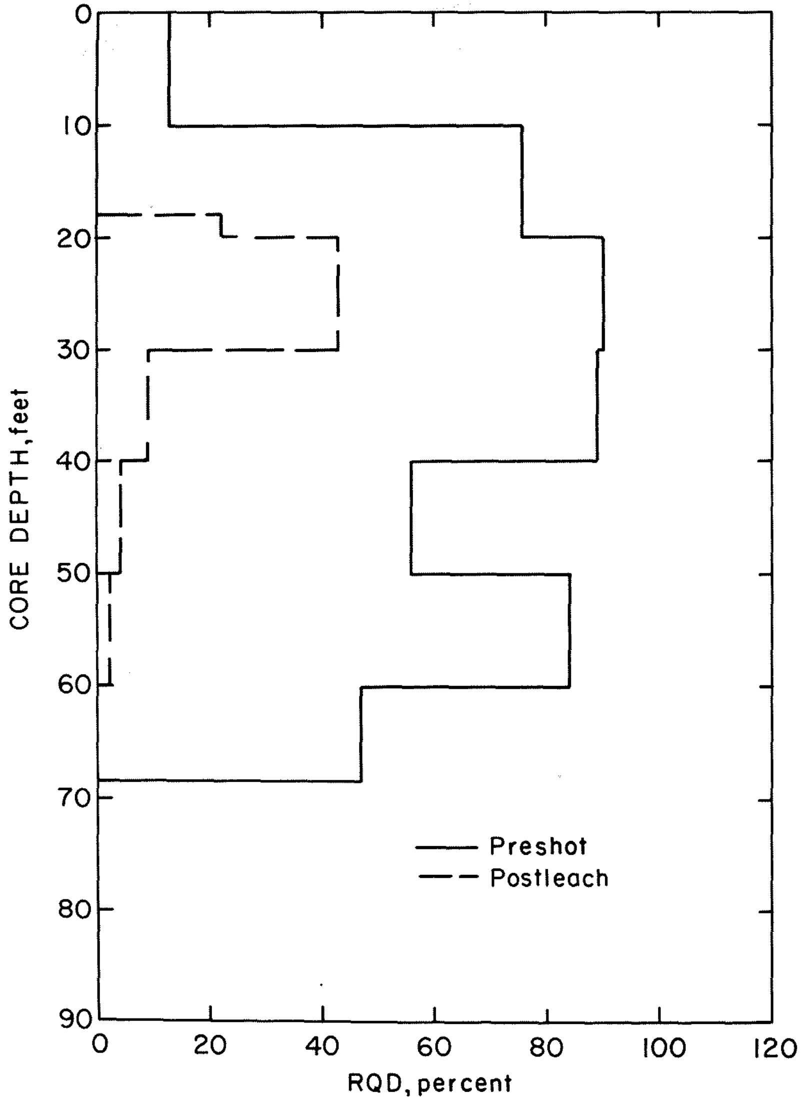

FIGURE 20. - Comparison of phase I preshot and postleach core length recovery.

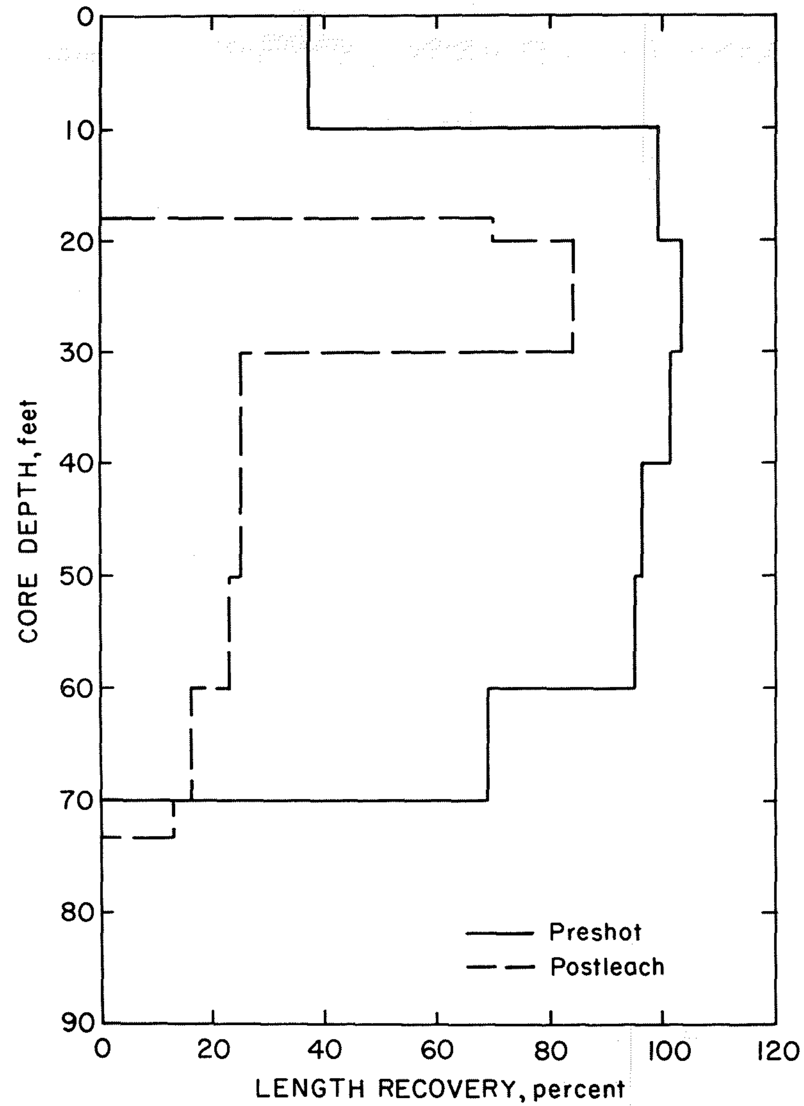

FIGURE 21. - Comparison of phase I preshot and postleach core RQD.

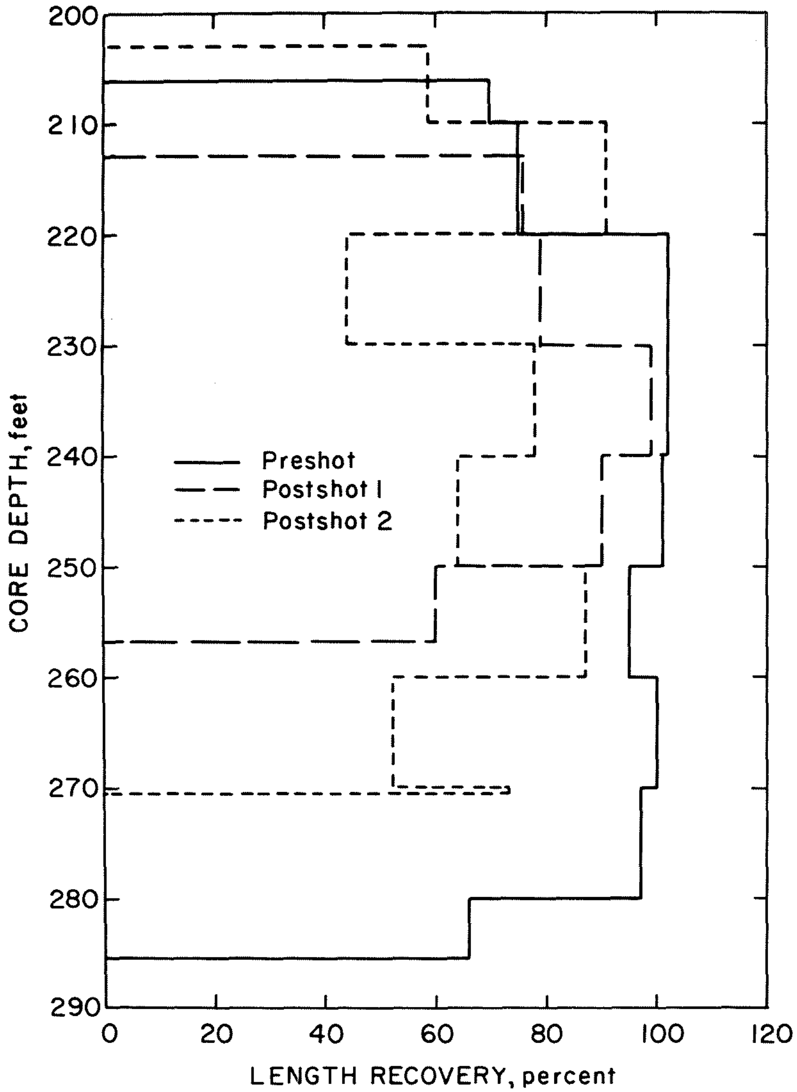

FIGURE 22. - Comparison of phase II preshot and postshot core length recovery.

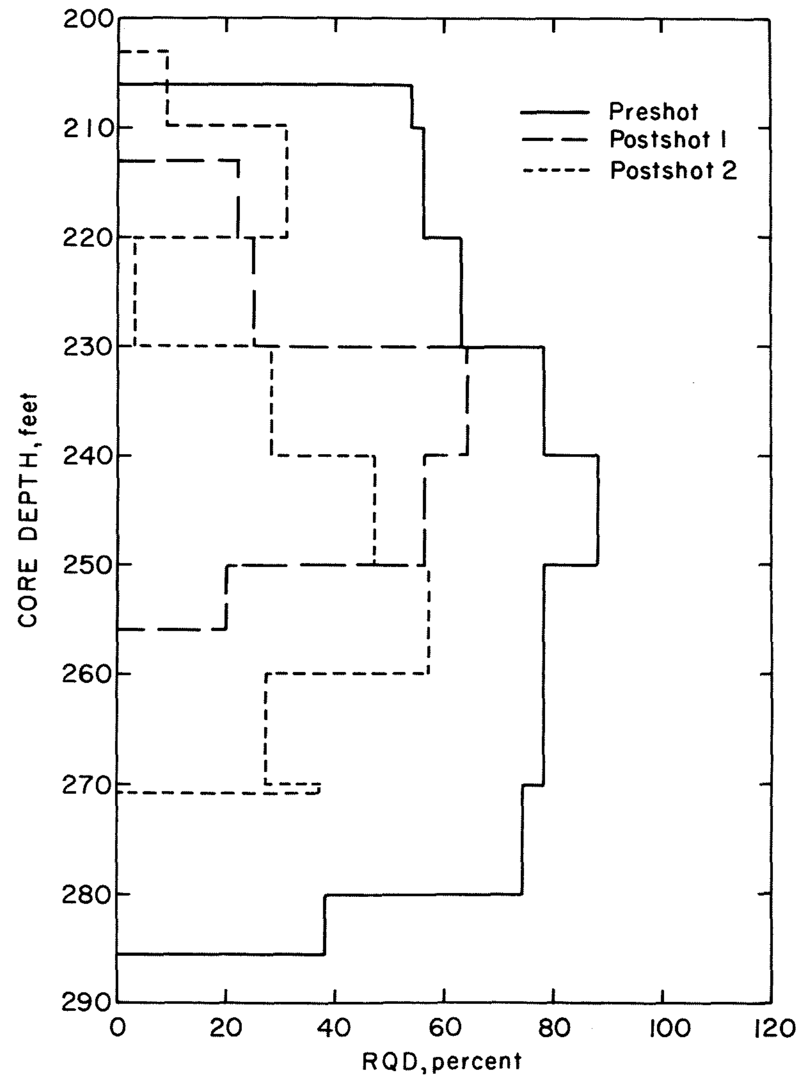

FIGURE 23. - Comparison of phase II preshot and postshot core RQD.

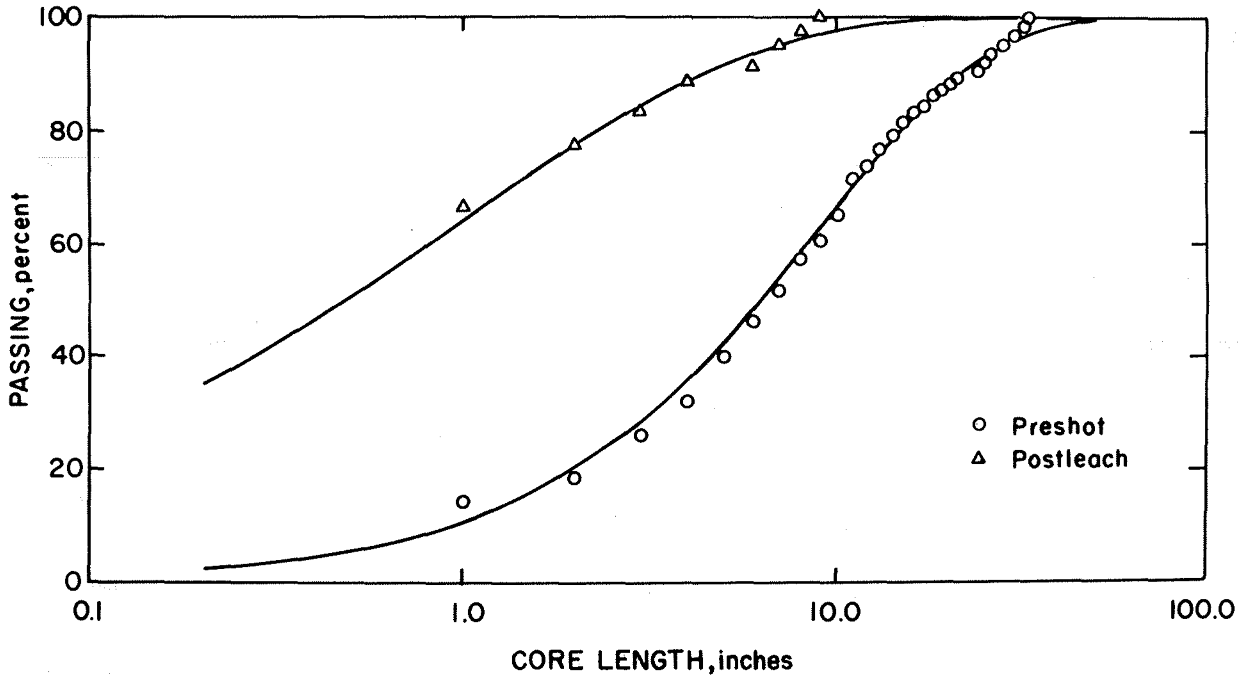

FIGURE 24. - Percent passing versus core size for phase I drill core.

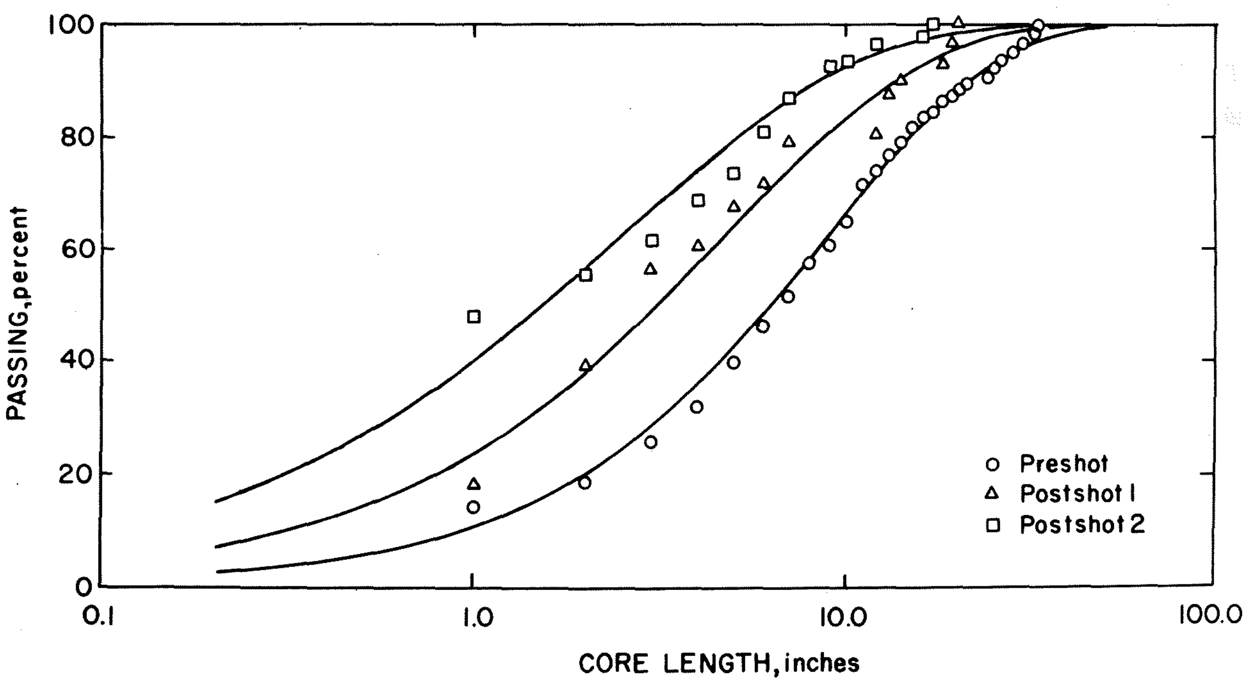

FIGURE 25. - Percent passing versus core size for phase II drill core.

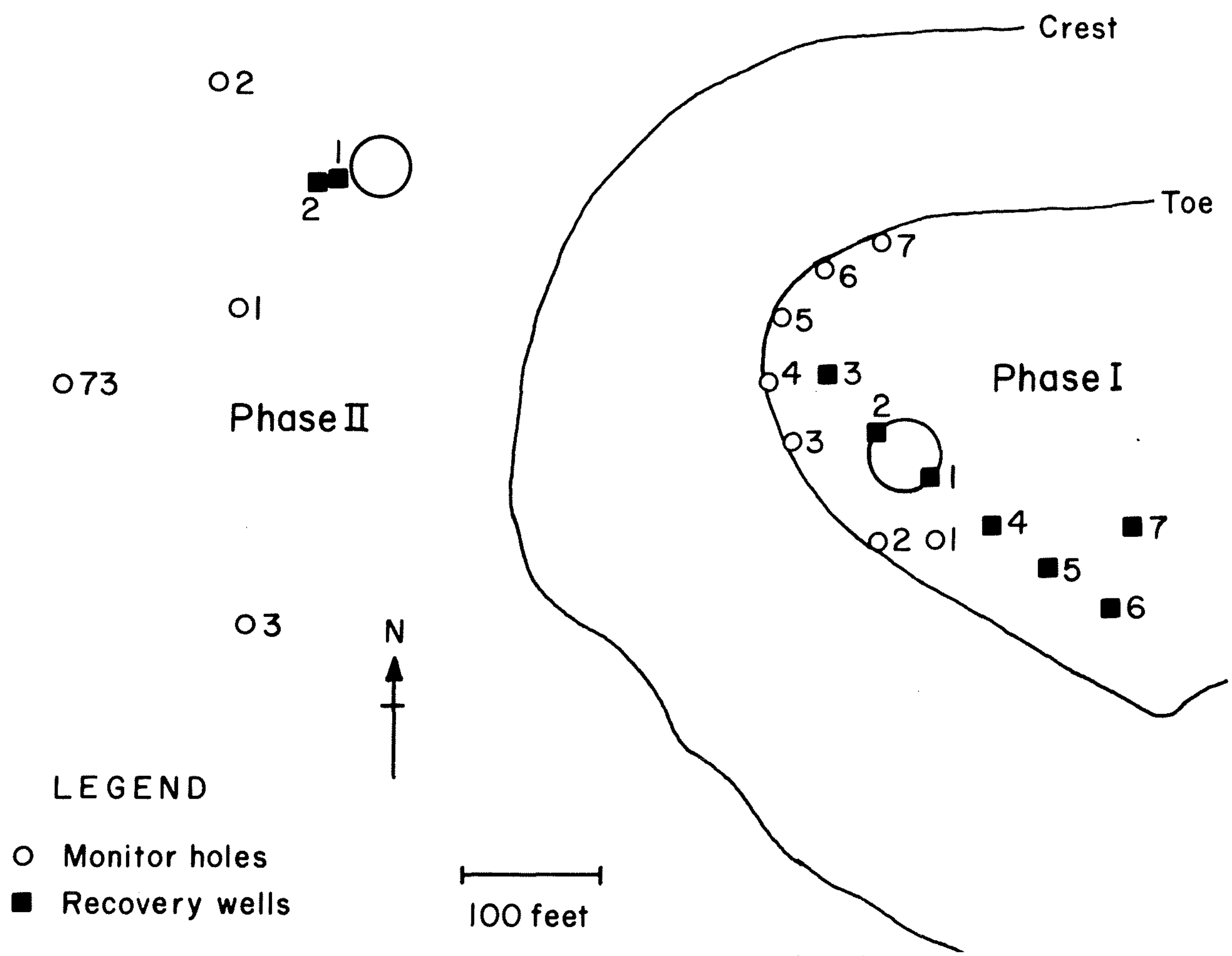

FIGURE 26. - Recovery well and monitor hole locations.

Seismic Evaluation

Seismic studies were conducted in the phase I test area before and after blasting. Seismic signals were generated with 1- to 2-pound charges of dynamite in 2- to 3-foot-deep blastholes. These signals were measured with particle velocity gages and recorded with a 14-channel FM tape recorder. Seismic refraction profiles were run with the shot point to the east and geophones to the west, and then the shot point-geophone relationship was reversed. Eight geophones were spaced about 20 feet apart.

Table 7 lists the results of the seismic refraction surveys. The refraction study showed that a low-velocity layer near the surface extended 3 to 8 feet deep. This low-velocity layer was probably the result of fractures created by previous mining activity. The preshot seismic velocity in the conglomerate ore was 15,000 ft/sec and dropped to 5,000 ft/sec after blasting. The seismic records revealed that the phase I blast did not produce a fractured zone that reduced the amplitude or changed the frequency of the transmitted seismic signals.

TABLE 7. - Results of seismic surveys

Location 1:	
Surface velocity..........ft/sec..	7,500
Lower velocity..........do....	15,200
Surface layer depth..........feet..	7.8
Location 2:	
Surface velocity..........ft/sec..	5,000
Lower velocity..........do....	15,200
Surface layer depth..........feet..	3.1
Postshot:[1] Velocity broken zone..........ft/sec..	5,000

[1]No vibration amplitude nor frequency change.

LEACHING

Recovery Well Pumps

Recovery well pumps used for the phase I leach test and the pit bottom leaching were of the walking beam-plunger type (fig. 27). Recovery well

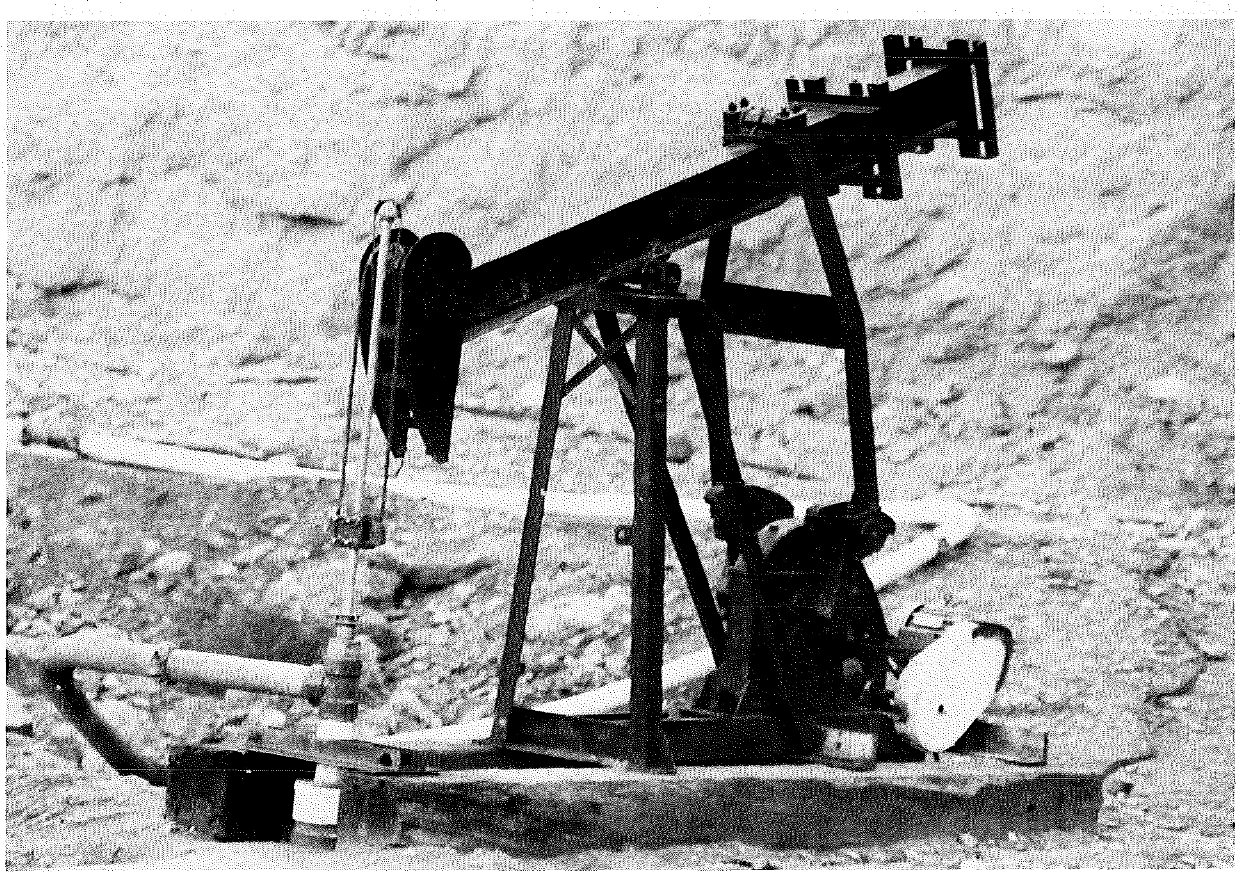

FIGURE 27. - Recovery well pump used during pit bottom leaching.

holes in the pit area were 12 inches wide and 50 to 60 feet deep, and were cased with 10-inch OD PVC casing, of which the lower 20 feet was perforated. The same hole and PVC casing diameters were used in the phase II area but the perforated casing was on the bottom 60 feet. A 20-hp submersible turbine pump was used in the phase II recovery well, which was positioned 290 feet down on a 4-inch ID stainless steel pipe inside the PVC casing.

Phase I Leach Test

The phase I leach test began in March 1974, and continued for 114 days. Leach solutions were distributed over the surface of the broken ore through perforated pipes and recovered in a well located on the east side of the blasted zone. The solutions were then pumped out of the pit to a recovery plant where cement copper was produced by precipitating the copper from the pregnant liquor with shredded iron in a cementation drum. Figure 28 shows the phase I test area during leaching. The 11-inch-diameter, 50-foot-deep recovery well hole was drilled with a churn drill. This hole was cased with a 10-inch-diameter PVC casing with the bottom 20 feet perforated.

During the leaching period flow rates averaged 63.5 gpm. The average flow rate was affected by significant shutdown time during the second and last weeks of operation. Because of ground water dilution and the desirability of drawing down the water table in the leach area, a bleed of 9 gpm was established in the flow circuit. Figure 29 shows the water table elevations at the

FIGURE 28. - Phase I leach test area.

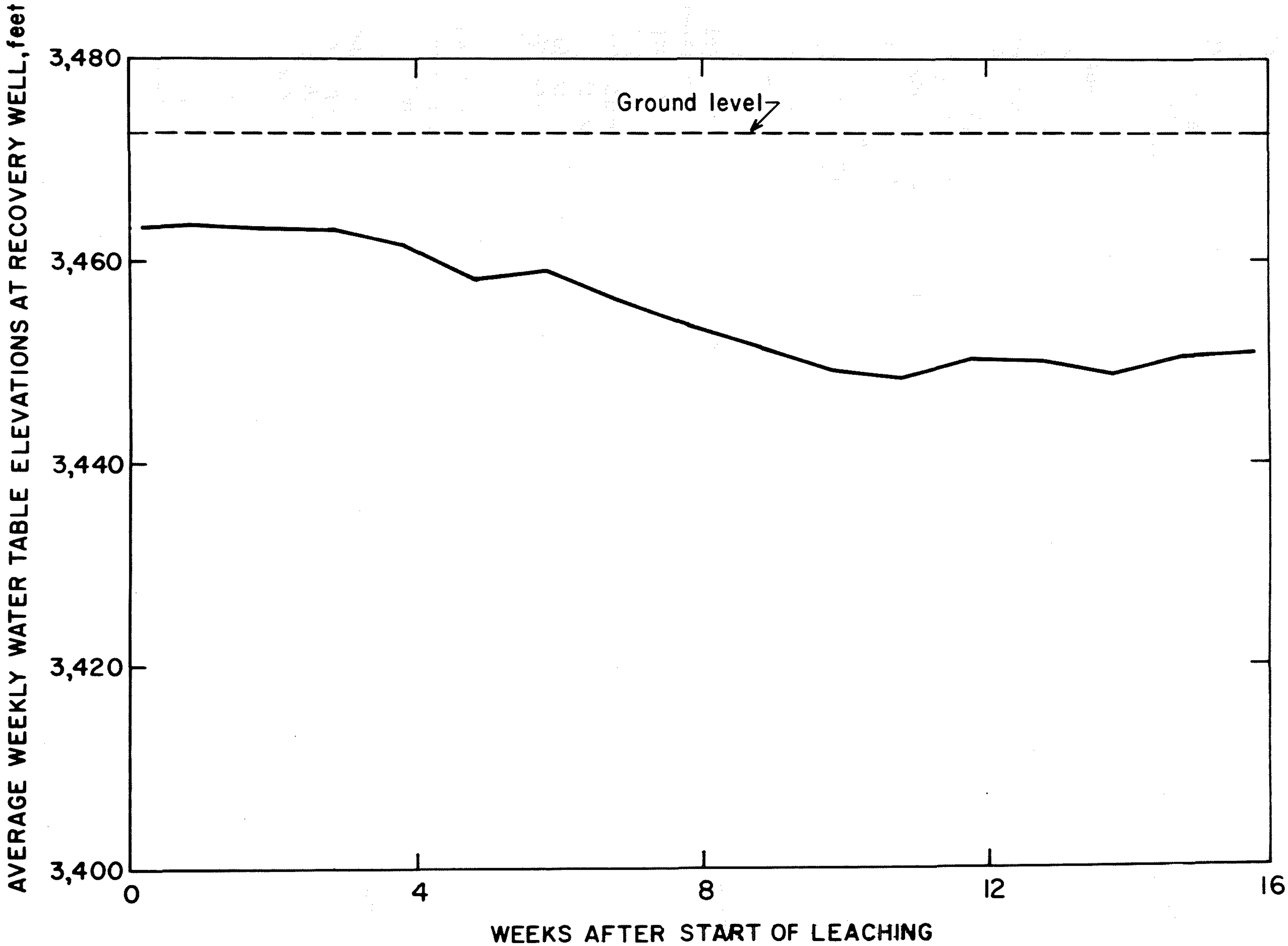

FIGURE 29. - Water table elevations versus time at the phase I recovery well.

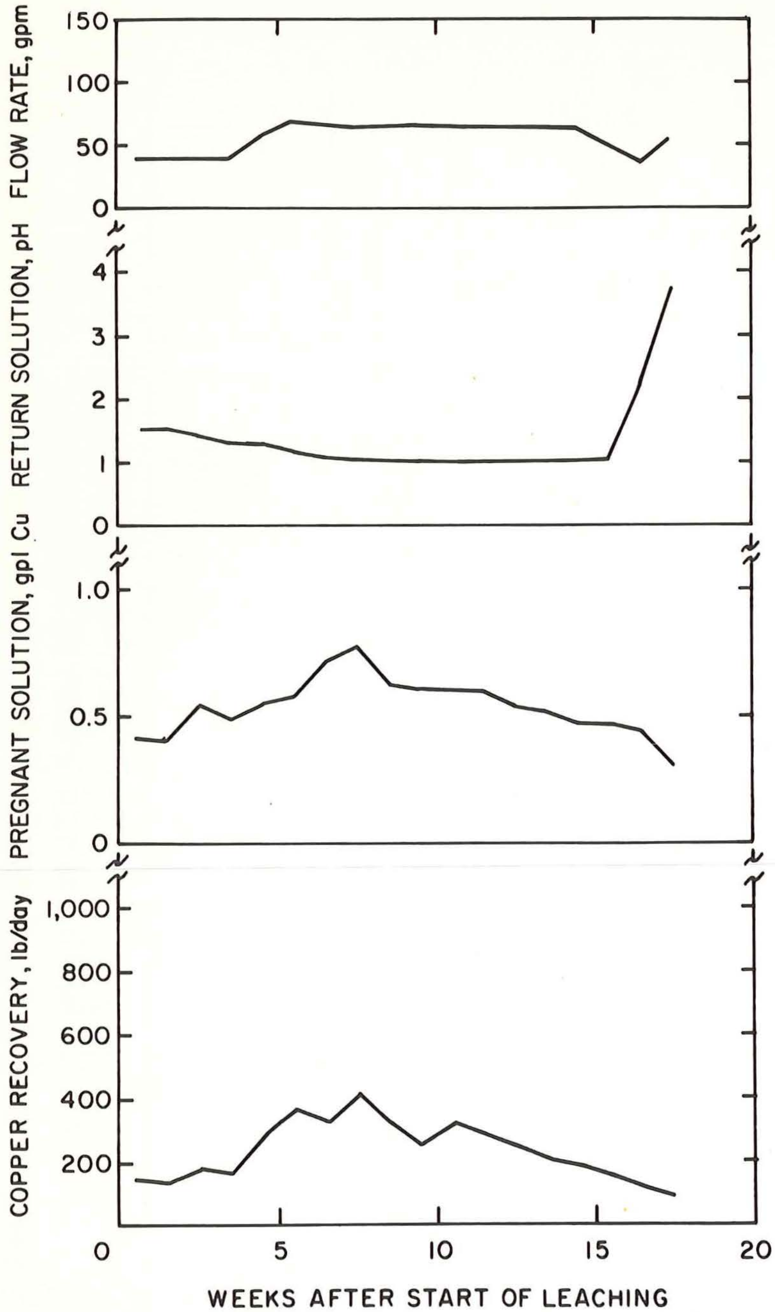

FIGURE 30. - Flow rate, leach solution analysis, and daily copper recovery for the phase I leach test.

recovery well as a function of time. The maximum water table drawdown was 23 feet, and the maximum depth of the water table below the surface was 25 feet.

Table 8 lists weekly averages for running time, flow rate, leach solution analysis, and computed copper recovery for the phase I leach test. The flow rates ranged from 40 to 68 gpm because of shutdown time and attempts to control the water table elevation in the leach area. The flow rate, pH of leach solutions, and daily copper recovery are shown in figure 30. Daily copper recovery fell off markedly toward the end of the leaching program because downtime increased and copper content in the solutions was lower. The cumulative copper recovery for the phase I leach test and for the pit bottom leaching are shown in figure 31.

Table 9 summarizes the phase I leach test results. The average pH of influent leach solutions was 1.04, the copper-grade of effluent solutions was 0.59 gpl, and the daily copper production was 248 pounds. During the test, 29,000 pounds of copper were produced. Acid consumption averaged 15.0 pounds of H_2SO_4 per pound of copper and iron consumption averaged 4.7 pounds of iron per pound of copper. The acid consumption was high; but this was attributed to the 9-gpm bleed in the leach solution flow circuit, the small size of the leach area with its important fringe effects, the short duration of the test, and the small difference between the pH of influent and effluent solutions.

TABLE 8. - Phase I leach test data

Week	Operating days/wk	Average operating time, hr/day	Total flow/wk, 1,000 gallons	Average flow, gpm	Copper content, gpl			Copper recovery, lb/day	pH levels	
					Pregnant solution	Precipitator	Return solution		Precipitator	Pit return
1.....	4	23.8	228.0	40.0	0.41	0.070	0.1	147.4	3.64	1.52
2.....	7	23.5	394.8	40.0	.40	.039	.1	141.2	4.02	1.53
3.....	7	20.4	343.2	40.0	.55	.033	.1	184.1	2.54	1.41
4.....	7	20.9	350.4	40.0	.49	.040	.1	162.9	1.85	1.28
5.....	7	21.9	529.7	57.5	.55	.059	.094	289.2	1.73	1.28
6.....	7	22.4	638.2	68.0	.58	.049	.099	365.9	1.65	1.16
7.....	7	17.4	488.5	66.7	.71	.058	.156	322.6	1.44	1.05
8.....	7	21.2	583.8	65.5	.77	.056	.171	416.8	1.41	1.03
9.....	7	23.7	655.2	65.8	.63	.064	.165	363.2	1.37	1.02
10.....	7	17.2	476.0	65.8	.61	.060	.157	257.0	1.36	1.02
11.....	7	21.7	588.0	64.5	.61	.072	.158	316.8	1.32	1.01
12.....	7	20.6	551.3	63.8	.60	.078	.171	281.9	1.30	1.00
13.....	7	21.7	581.3	63.7	.54	.098	.189	243.2	1.29	1.00
14.....	7	18.3	494.9	64.4	.52	.084	.177	202.3	1.33	1.00
15.....	7	20.3	528.2	62.0	.46	.070	.166	185.1	1.27	1.00
16.....	7	19.5	400.7	48.9	.47	.105	.160	148.1	1.26	1.00
17.....	7	16.1	373.5	35.3	.43	.093	.173	114.1	2.37	2.22
18.....	1	18.0	59.4	55.0	.29	.071	.110	89.2	3.15	3.75

TABLE 9. - Summary of leaching results

	Phase I leach test	Pit bottom leaching
Ore leached............................tons..	15,000	100,000
Grade of ore........................percent..	1.0	1.0
Duration of leaching...................days..	117	190
Average running time.................hr/day..	20.5	23.2
Leach influent:		
pH...	1.18	1.10
Copper................................gpl..	0.147	0.091
Iron..................................gpl..	8.5	-
Leach effluent:		
Flow rate.............................gpm..	57.4	115.9
pH...	1.3	-
Copper................................gpl..	0.562	0.646
Iron..................................gpl..	6.2	-
Precipitation effluent:		
pH...	1.71	1.62
Copper................................gpl..	0.067	0.076
Acid consumption.............lb H_2SO_4/lb Cu..	15.0	10.0
Iron consumption...............lb Fe/lb Cu..	4.7	2.75
Copper production...................lb/day..	245	748
Total copper production.............pounds..	29,000	142,000

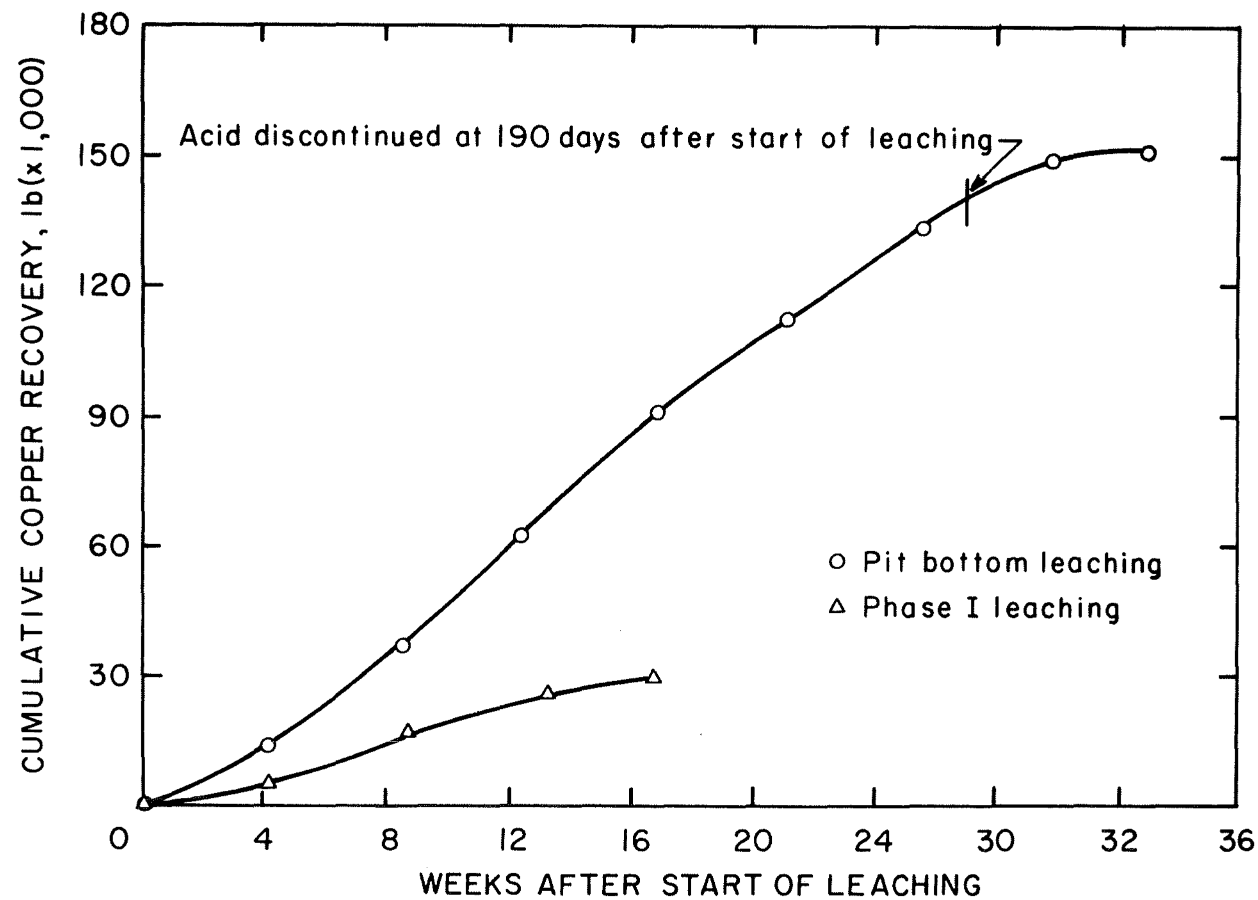

FIGURE 31. - Cumulative copper recovery for phase I leach test and pit bottom leaching.

Pit Bottom Leaching

Pit bottom leaching of about 100,000 tons of ore began in December 1974, and continued for 190 days. The system of seven 50-foot-deep recovery wells, spaced about 50 feet apart is shown in figures 26 and 32. The ore in the pit bottom was not blasted before this leaching in the hope that natural permeability would allow successful leaching.

The flow rate, pH of return leach solutions, copper content of pregnant leach solutions, and daily copper recovery are graphed in figure 33 and listed with other data in table 10. The copper content in the pregnant solution began to fall off only after acid was no longer added to the return leach solution after 190 days. Solutions were circulated for 6 weeks longer to bring up the pH of solutions remaining in the pit bottom, and an additional 9,000 pounds of copper were produced. The cumulative copper recovery for pit bottom leaching is shown in figure 31.

TABLE 10. - Pit bottom leaching data

Week	Operating days/wk	Average operating time, hr/day	Total flow/wk, 1,000 gallons	Average flow, gpm	Copper content, gpl			Copper recovery, lb/day	pH levels	
					Pregnant solution	Precip-itator	Return solution		Precip-itator	Pit return
1....	7	23.7	1,110.0	111.4	0.491	0.058	0.078	546.4	2.09	1.13
2....	7	22.7	920.4	96.5	.424	.029	.039	422.4	2.11	1.13
3....	7	21.1	838.1	94.4	.512	.033	.056	455.6	2.31	1.06
4....	7	21.8	1,005.4	109.9	.561	.066	.071	587.2	2.01	1.08
5....	7	24.0	1,390.7	138.0	.558	.055	.071	807.3	1.45	1.00
6....	7	23.1	1,465.5	150.8	.502	.062	.074	747.7	1.94	1.00
7....	7	24.0	1,599.6	158.7	.448	.048	.057	745.5	2.01	1.00
8....	7	24.0	1,321.9	131.1	.497	.055	.061	687.0	1.90	1.05
9....	7	23.4	1,254.3	127.9	.620	.064	.060	837.3	1.72	1.16
10....	7	19.9	916.2	109.9	.798	.091	.103	759.0	1.67	1.14
11....	7	24.0	1,178.3	116.9	.851	.089	.078	1,085.7	1.53	1.13
12....	7	24.0	923.9	91.7	.884	.888	.086	878.8	1.58	1.15
13....	7	24.0	1,221.6	121.2	.724	.107	.074	946.5	1.37	1.09
14....	7	22.4	1,219.4	129.4	.661	.091	.072	856.1	1.37	1.04
15....	7	23.3	1,392.5	142.4	.624	.059	.065	927.9	1.39	1.02
16....	7	21.3	1,202.6	134.5	.674	.080	.075	858.7	1.32	1.01
17....	7	24.0	1,155.7	114.7	.708	.053	.075	872.0	1.31	0.99
18....	7	24.0	1,027.5	101.9	.744	.089	.076	818.2	1.33	1.01
19....	7	24.0	1,150.1	114.1	.643	.085	.072	782.8	1.32	1.01
20....	7	23.7	797.8	80.1	.607	.083	.071	509.7	1.31	1.01
21....	7	23.9	1,022.9	101.8	.762	.154	.147	749.9	1.40	1.01
22....	7	23.0	881.1	91.2	.807	.134	.140	700.5	1.33	1.01
23....	7	23.6	861.0	86.7	.711	.101	.149	576.8	1.42	1.00
24....	7	23.9	1,172.8	117.0	.702	.074	.184	724.2	1.42	1.14
25....	7	23.4	1,240.2	126.2	.692	.097	.190	742.1	1.32	1.02
26....	7	24.0	1,146.1	113.7	.764	.073	.136	857.9	1.44	1.21
27....	7	22.6	1,067.7	112.6	.687	.056	.132	706.3	2.26	2.04
28....	1	24.0	187.6	130.3	.69	.089	.14	860.9	2.17	1.98
28....	7	17.9	793.9	105.4	.718	.077	.141	546.0	3.48	2.05
29....	7	22.0	1,002.7	108.5	.342	.039	.089	302.4	4.39	-
30....	7	23.9	1,043.2	104.1	.313	.054	.091	276.1	4.48	2.91
31....	7	24.0	1,094.2	108.6	.215	.060	.100	150.0	4.40	3.00
32....	7	21.4	821.7	91.3	.198	.059	.117	79.3	4.64	3.09
33....	7	19.6	830.8	100.7	.205	.068	.132	72.3	4.72	3.24

FIGURE 32. - Pit area during pit bottom leaching.

Table 9 summarizes the pit bottom leaching results. Average values are for the 190-day period only and not the 41-day period during which acid was not added. The average flow rate was 115.9 gpm, pH of return leach solution was 1.10, copper content of pregnant leach solution was 0.65 gpl, and copper recovery was 748 pounds per day. Acid consumption averaged 10 pounds of H_2SO_4 per pound of copper and iron consumption averaged 2.75 pounds of iron per pound of copper. The acid and iron consumption were lower than the phase I leach test but were still high. However, acid consumption was dropping as the test continued.

Pit bottom leaching was stopped because the flow rates of leach solutions were not as high as desired. With a total flow of 116 gpm and seven recovery wells, the average was only 17 gpm per well. Highest production was from wells near the blast-fractured phase I test area so the flows ranged from 45 to 5 gpm from individual wells. Pit bottom leaching was stopped and a plan to drill and blast the ore to improve permeabilities and flow rates was started. The drilling and blasting program began, but the Emerald Isle operation was closed before any further leaching of the pit bottom ore was done.

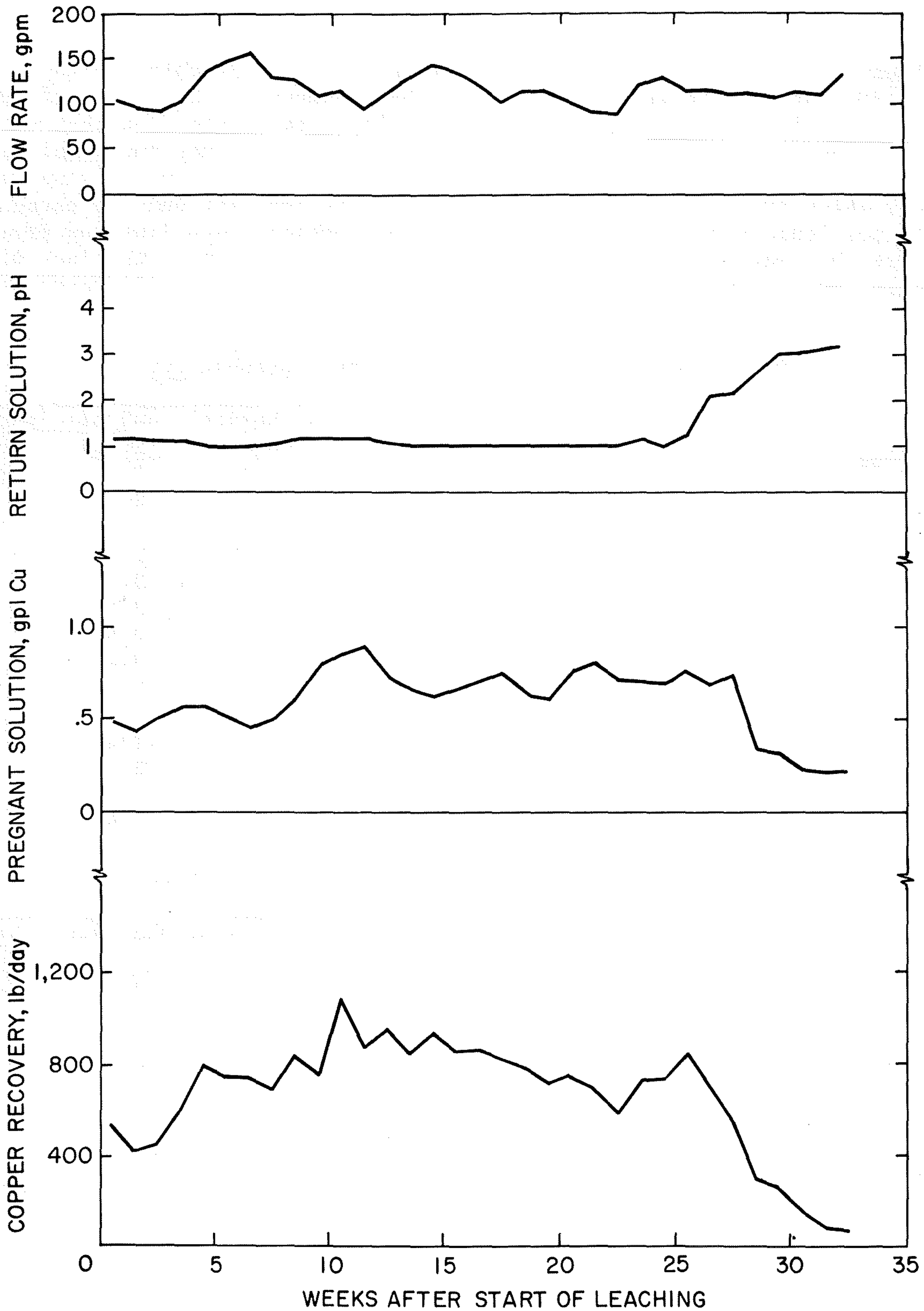

FIGURE 33. - Flow rate, leach bottom solution analysis, and daily copper recovery for pit bottom leaching.

DRILL CORE ASSAYS

Assays run on drill core before blasting and after leaching in the phase I area are listed in tables 11-12. The preshot core hole was located about 11 feet from the postleach core hole. The data listed show that the copper values did not change while the iron and sulfur values increased as a result of leaching. Interpretation of these data must take into account core recovery which averaged 86 percent for the preshot core and only 35 percent for the postleach core. Core recovered during postleach drilling was from solid ore that may not have been attacked by leach solutions. The lack of change in copper content may also have been due to statistical differences in copper grade at the two drill hole locations.

TABLE 11. - Assay values for phase I preshot core

Depth, feet	Chemical analysis, percent		
	Copper	Iron	Sulfur
0.0 to 10.0	0.34	2.7	0.023
10.0 to 15.0	.65	2.7	.041
15.0 to 20.0	.62	2.7	.056
20.0 to 25.0	.68	2.6	.110
25.0 to 30.0	.91	3.7	.039
30.0 to 35.0	.95	3.0	.090
35.0 to 40.0	.87	2.6	.092
40.0 to 45.0	.60	3.4	.052
45.0 to 50.0	.32	3.3	.049
50.0 to 55.0	.25	3.4	.120
55.0 to 60.0	.16	2.4	.017
60.0 to 65.0	.31	2.4	.026
65.0 to 68.7	.13	2.3	.017
Average	.52	2.9	.056

TABLE 12. - Assay values for phase I postleach core

Depth, feet	Chemical analysis, percent		
	Copper	Iron	Sulfur
15.1 to 20.0	1.19	4.6	0.140
20.0 to 25.0	1.03	4.2	.140
25.0 to 35.0	1.17	4.1	.110
35.0 to 40.0	.29	4.5	.100
40.0 to 50.0	.19	4.0	.029
50.0 to 55.0	.16	3.9	.037
55.0 to 62.0	.24	4.4	.210
62.0 to 70.3	.12	5.1	.056
Average	.53	4.3	.098

Assay values for the preshot core in the phase II area are listed in table 13. This core had a higher copper content than that in the phase I area with a high-grade zone between 230 and 250 feet which averaged 1.5 percent copper.

TABLE 13. - Assay values for phase II preshot core

Depth, feet	Chemical analysis, percent		
	Copper	Iron	Sulfur
206.0 to 210.0	0.69	3.08	0.160
210.0 to 216.0	.46	3.03	.080
216.0 to 220.0	.54	3.30	.088
220.0 to 225.0	.37	2.81	.055
225.0 to 230.0	.34	3.39	.062
230.0 to 235.0	1.69	2.90	.054
235.0 to 240.0	1.47	2.76	.084
240.0 to 245.0	1.70	2.90	.130
245.0 to 250.0	1.07	2.53	.150
250.0 to 255.0	.40	2.85	.064
255.0 to 260.0	.59	2.22	.022
260.0 to 265.0	.46	3.62	.110
265.0 to 270.0	.40	2.85	.024
270.0 to 275.0	.41	3.30	.064
275.0 to 280.0	.48	3.03	.058
280.0 to 285.0	.21	2.17	.045
285.0 to 290.0	.44	4.62	.026
290.0 to 294.0	.32	5.70	.028
Average	.67	3.17	.072

Ground Water Monitoring

During the phase I leach test ground water samples were taken at seven monitor holes in the pit bottom, at a sump pond in the pit, and at hole No. 73 (drilled through the overburden into the ore zone downdip from the pit area). During pit bottom leaching, ground water samples were taken at six monitor locations in the phase II area (fig. 26). The monitor locations sampled during pit bottom leaching were holes drilled through the overburden, into the ore zone, and below the water table.

Ground water samples were taken each day during the phase I leach test and about once per week during pit bottom leaching. These samples were analyzed for pH, copper, iron, manganese, and nickel during the phase I test. Tables 14 and 15 list average chemical analysis values and figure 34 shows changes in pH at four monitor holes during the phase I leach test.

This ground water monitoring program showed that leach solutions were contained and that ground water contamination did not occur. Leach solutions were detected in monitor holes 1 and 2 about 8 weeks after the start of the phase I test. However, these holes were only about 50 feet from the phase I test area.

TABLE 14. - Chemical analysis of ground water samples during the phase I leach test

Element	Days sampled	Location								
		MW 1	MW 2	MW 3	MW 4	MW 5	MW 6	MW 7	MW 73	Sump
Copper............ppm..	93	23.37	4.06	1.16	0.11	0.10	0.09	0.10	0.01	0.12
Iron..............ppm..	84	3.23	0.11	0.09	0.10	0.09	0.08	0.09	0.06	0.09
Manganese.........ppm..	93	33.92	6.68	9.20	0.12	0.08	0.04	0.07	0.02	0.21
Nickel............ppm..	84	11.31	0.29	0.46	0.14	0.12	0.12	0.12	0.05	0.24
Cadmium...........ppm..	18	0.36	0.08	0.05	0.04	0.04	0.04	0.04	0.03	0.04
Molybdenum........ppm..	18	0.07	0.04	0.05	0.05	0.05	0.06	0.06	0.04	0.04
Calcium...........ppm..	18	360	489	548	391	452	479	500	222	581
Magnesium.........ppm..	18	233	209	218	176	197	235	254	66	213
Aluminum..........ppm..	18	67.5	4.8	5.3	4.9	4.7	5.2	5.6	3.5	6.2
Zinc..............ppm..	18	27.64	3.05	1.06	0.10	0.09	0.09	0.08	0.10	1.91
Cobalt............ppm..	18	0.73	0.10	0.05	0.04	0.04	0.04	0.04	0.02	0.03
pH....................	36	6.3	6.6	6.8	7.2	7.2	7.0	7.0	7.6	7.6

TABLE 15. - Chemical analysis of ground water samples during pit bottom leaching

Element	Days sampled	Location in phase II area					
		MH 1	MH 2	MH 3	RW 1	RW 2	MW 73
Copper....................ppm..	37	0.04	0.04	0.06	0.05	0.05	0.02
Iron......................ppm..	37	0.06	0.08	0.04	0.05	0.06	0.04
Manganese.................ppm..	37	6.2	0.13	0.16	1.19	0.39	0.05
Nickel....................ppm..	37	0.08	0.09	0.04	0.09	0.10	0.06
Cadmium...................ppm..	37	0.03	0.04	0.03	0.03	0.04	0.03
Molybdenum................ppm..	37	0.08	0.08	0.16	0.13	0.13	0.08
Calcium...................ppm..	37	179	288	10.1	202	229	172
Magnesium.................ppm..	37	97.3	182	4.66	100	110	59.0
Aluminum..................ppm..	37	1.6	1.81	0.44	1.59	1.66	0.97
Zinc......................ppm..	37	0.73	0.07	0.14	0.18	0.09	0.11
Cobalt....................ppm..	37	0.05	0.07	0.04	0.06	0.06	0.06
pH...........................	37	7.14	7.45	8.05	7.64	7.73	7.58

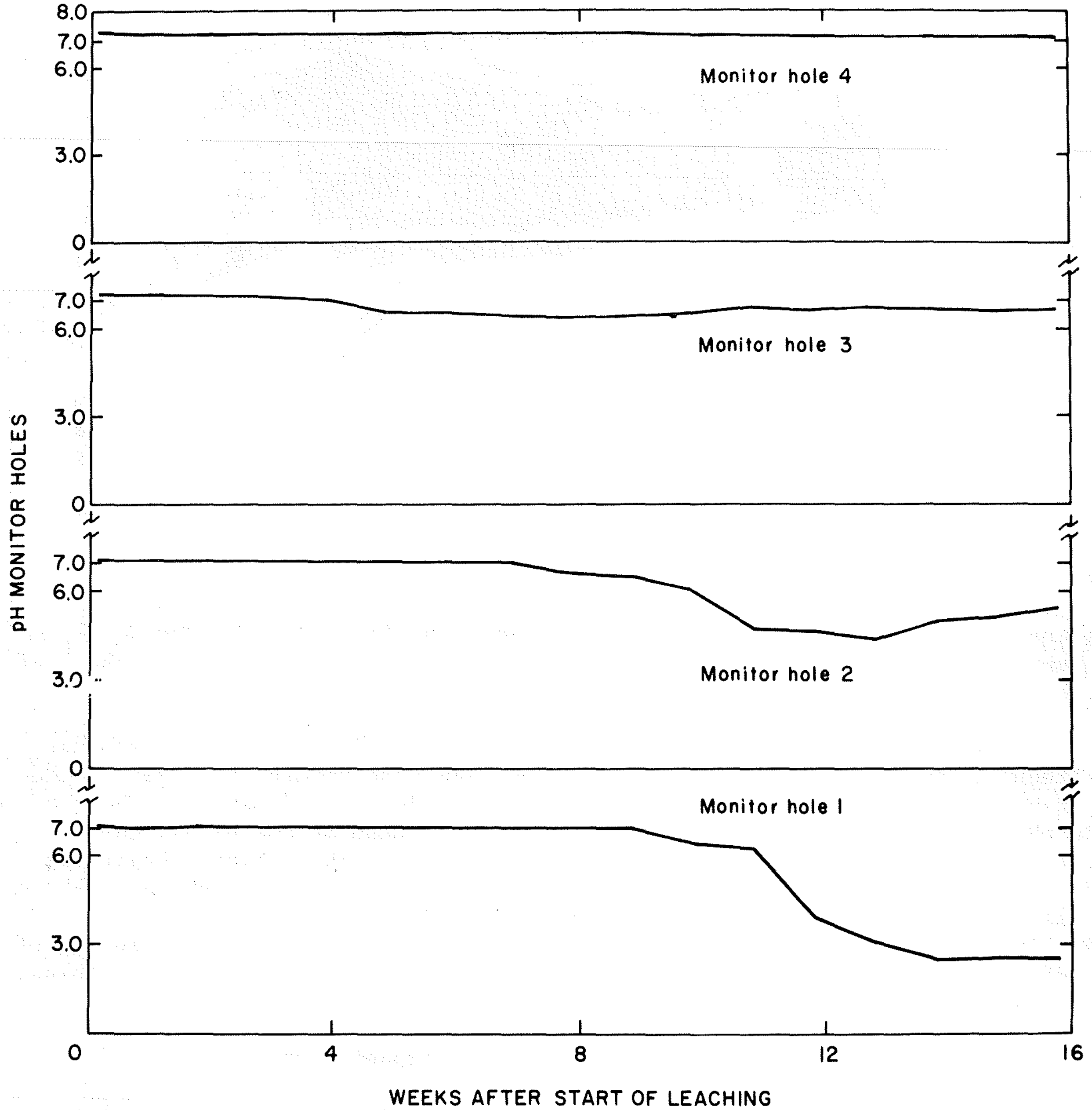

FIGURE 34. - Changes in pH at monitor holes 1-4 during the phase I leach test.

DESIGN OF EXPANDED IN SITU LEACHING SYSTEM

Based on the blasting and leaching test results, a full-scale in situ leaching system was designed to recover the majority of the remaining copper. This plan involved blasting and leaching a higher grade area first, then working on a lower grade area. The first area would include ore in the pit bottom, under the pit walls, and under 180 to 250 feet of overburden along a channel extending 700 feet from the crest of the pit. The channel would follow the high-grade copper mineralization as shown in figure 35. The

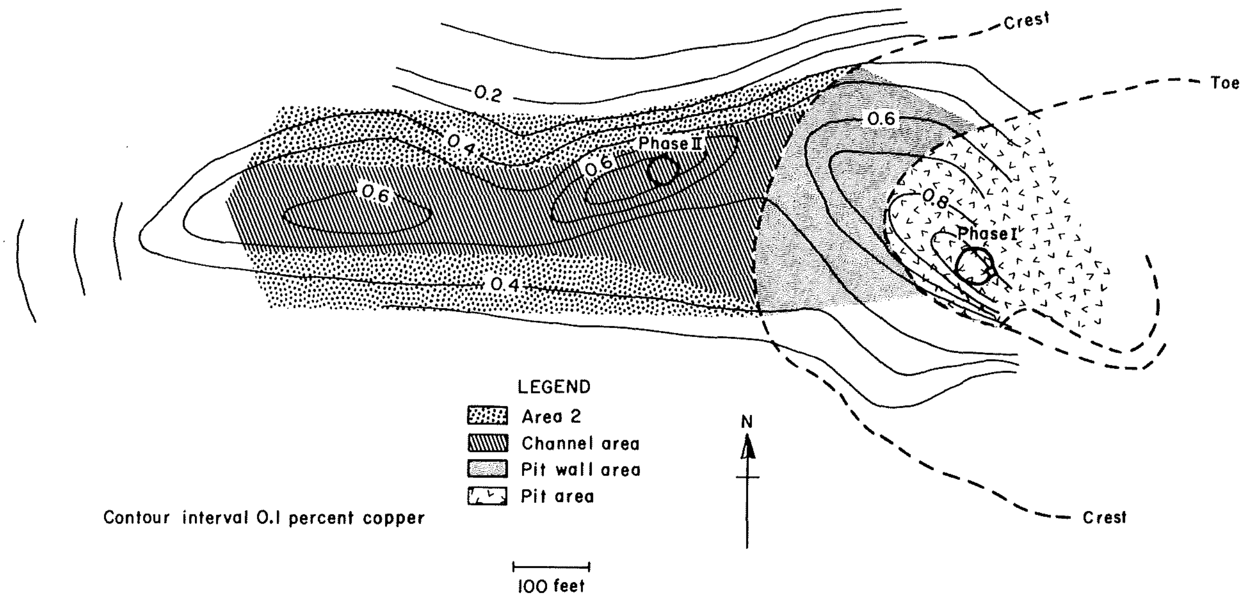

FIGURE 35. - Proposed blast design for expanded operation.

second area would leach lower grade ore along the flanks of the 700-foot-channel. A third area, parallel to the second, could be added if leaching of the initial areas were successful.

Fragmentation would be accomplished primarily with vertical blastholes although some inclined holes from the crest and some horizontal holes in the pit bottom would be required to break the ore under the pit walls. The ore in the pit bottom and under the walls would be broken in one blast, while the channel area would be broken in a series of seven-hole blasts beginning at the crest and proceeding in the downdip direction. Each of the seven-hole blasts would be detonated as soon as drilling was completed so that this drilling and blasting program would continue for several months. A series of blasts would also break the second area. Table 16 lists the full-scale blast design data.

TABLE 16. - Blast design data for expanded operation

	Area 1			Area 2
	Pit	Walls	Channel	
Area....................sq ft..	59,000	56,000	96,000	88,000
Thickness.................feet..	40	69	78	76
Volume....................cu yd..	87,000	143,000	276,000	246,000
Weight....................tons..	167,000	274,000	530,000	472,000
Grade..................percent..	0.70	0.58	0.55	0.41
Total copper.............pounds..	2,300,000	3,200,000	5,800,000	3,900,000
Drilling depth.............feet..	40	125	287	297
Powder factor............lb/ton..	0.5	1.0	1.5	.75
Total drilling.............feet..	2,200	12,900	76,300	36,000
Total explosives..........pounds..	83,000	274,000	795,000	354,000

Leach solution injection would occur primarily in the pit bottom with some vertical injection holes near the pit crest. Solutions would be recovered downdip with a series of wells. It is estimated that four wells would be required at about 700 feet from the pit crest. The first area would be leached for about 2 years depending on the grade of solutions before the second area would be added. Both areas would then continue to be leached for another 2 years.

The major advantages of this in situ leaching system are:

1. Initial leaching is in the higher grade ore. The second lower grade area does not need to be added unless the first area had yielded good results.

2. Leach solutions are in contact with the ore for a longer time since the solutions migrate downdip from the pit bottom to the recovery wells. This should result in higher grade solutions if the pH could be maintained along the 700-foot channel.

3. The second blast which fragments lower grade ore could be detonated at a lower powder factor than the first, because the initial channel would be weakened by the action of leach solutions and provide a volume into which material from the second blast could expand.

4. Blasting the second area should shake up and restimulate copper production from the first area.

This full-scale leaching is feasible and would be profitable if the price of copper were to increase.

SUMMARY

An in situ leaching test was run for 117 days in a phase I test area in the pit bottom where blasted ore was exposed from the surface to a depth of about 50 feet. This successful test was followed by inplace leaching another 100,000-ton area, which was not blasted before leaching and where flow rates were not as high as desired. Pit bottom leaching was halted and a drilling and blasting program initiated but not completed before the Emerald Isle mine closed.

A phase II test area under 205 feet of overburden and extending to 290 feet was blasted. Water circulation revealed that the first blast did not create adequate breakage and permeability for leaching. A second blast did improve the fragmentation, but water circulation tests were not conducted because the mine closed.

An expanded in situ leaching system was designed to recover copper from 1,500,000 tons of ore in the pit bottom, under the pit walls, and under overburden extending 700 feet along a channel from the pit crest. This program was not implemented.

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U.S. GOVERNMENT PRINTING OFFICE: 1977-703-001/92

INT.-BU.OF MINES,PGH.,PA. 22338