In all, 322,000 tons has been excavated in the Underground Quarry since inception of the project.

Mining Research

The mining-research program at the Oil-Shale Experimental Mine has been directed primarily toward providing data for establishing a low-cost method for mining oil shale. As this purpose has been achieved, the emphasis has been shifted from applied to fundamental research.

One phase of fundamental research has been the study of explosive action, and data were collected in cooperation with the Applied Physics Branch of the Bureau of Mines during both 1950 and 1951. These data were obtained by a series of instruments designed to record the intensity and shape of strain waves caused by detonation of an explosive; they also measured the time required for the strain wave to travel from the point of detonation to the point of measurement.

Experiments showed that the strain wave consists of a compression impulse that reflects from any free surface (such as a bench face) as a tension wave. There is good evidence that this strain wave actually produces fracturing by being reflected from a surface as a tension wave.

The peak strain developed by the compression wave was found to be directly proportional to the charge weight and is inversely proportional to the distance from the charge. The peak strain also depends upon the type and strength of explosive, type of rock, the type and amount of stemming, the charge density, and other factors.

A series of three bench rounds was blasted, using a mechanical timing device to produce the desired delay pattern. Each of the rounds consisted of a single row of eight holes spaced at 8 feet with 6-3/4-foot burden. Time measurements of fracturing and actual break were recorded for the bench face and at the hole collars. Fracturing occurred 3 to 10 milliseconds after detonation, dependent upon location with respect to the blast holes. Actual rock movement required 20 to 200 milliseconds.

The blasting-delay periods for the three rounds were 25, 12.5, and 6.25 milliseconds, respectively. The round wherein the 12.5 milliseconds delays were used had the best appearance in terms of tonnage broken and fragmentation. Differences in these rounds indicated that the optimum delay period for best break and fragmentations in bench rounds in oil shale is at or near 12.5 milliseconds. This period applies only to the delay between holes in a given row. No information was gained on the optimum delay between rows.

In addition to timing data recorded by the Applied Physics Branch, additional information was obtained by means of a high-speed motion-picture camera. However, in the first blasts photographed (two heading rounds in the Underground Quarry) smoke caused by detonation of the first holes in each round obscured the action at the blast face. Studies then were made of a bench face, using vertical blast holes to prevent smoke from obscuring the action at the face.

A camera speed of 128 frames per second was used for photographing the blasts. By analyzing the films frame-by-frame, a "timing of events" with respect to hole detonation, fracturing, and actual rock movement can be made. A frame-by-frame study of three bench blasts led to the following conclusions:

1. There is considerable variance in the rated and actual delay period for millisecond detonators. In some instances the variation was enough to cause overlapping of successive delays.

2. In two of the bench rounds where 25-millisecond delay periods were used, about 35 and 43 milliseconds elapsed after initial detonation before rock fracturing and actual rock movement, respectively. A third round, using all Instant or 0-delay detonators, showed that fracturing occurred at 8 and rock movement at 15 milliseconds.

3. Because of difficult lighting conditions, a considerable amount of detail was lost, and no information could be obtained on the possible effect of the strain wave at the bench face.

The drilling phase of the mining-research program has been directed primarily toward rotary methods. Percussion-drilling research was limited to drill-rod investigation and a study of the factors affected by a change in air pressure.

The problem of drill-rod failure has become of major concern to the mining industry. Primary causes for failure in drill rods are improper handling or misuse and improper forging or heat treatment. Failures due to improper handling or misuse can be minimized only through adequate supervision and training of employees who handle and use percussion drill rods.

Failures caused by improper heat treatment usually occur in the so-called soft zones of a drill rod, where the forging, normalizing, or hardening heat blends into the "as-rolled" section of the rod. A method for minimizing soft-zone failures has been tried with good success. It was suggested by metallurgists of the Crucible Steel Co. of America.

The technology of the soft-zone treatment, as developed by C. W. Darby and R. M. Simpson of Crucible Steel Co., can be summarized by the following comments:

> The so-called soft zone occurs where the heat zone of a drill rod grades into the unheated zone during any of the heat-treating processes. The temperature between these zones is subcritical and causes the steel to soften at this area. Maximum softness occurs when the temperature is held just under the critical point for a long enough time.
>
> This soft zone cannot be eliminated, but the amount of softening can be minimized. After the end of the rod that is being heat-treated reaches the proper temperature and just before quenching, the drill rod should be advanced into the furnace so that the portion just under the critical temperature will be momentarily heated

above critical. The rod end must then be quenched before a new equilibrium temperature gradient is reached and a new soft zone formed.

This procedure produces a more gradual drop in hardness from the treated to the untreated zones of the rod. The gradual drop in hardness minimizes the stress concentration usually experienced in sharp hardness gradient soft zones produced when the rod ends are not advanced before quenching.

Several drill-rod types were obtained from Crucible Steel Co. to test the effectiveness of soft-zone treatment. The initial test data are summarized in table 1. The success of the soft-zone treatment is exemplified by the increase in drill-rod life for Crusca carbon rods.

TABLE 1. - Summary of test data on percussion-drill rods

Rod type	Average drilling life, feet	Previous test average	Average drilling rate, inches per min.
Crusca carbon	822	530	19.6
Alva alloy	630	202	26.7
Park "A" alloy	532	298	26.4
Crusca "CA" alloy	1,271	919	24.5
Alloy "N"	958	-	24.5

Failures in the soft zone were virtually eliminated by the heat-treating procedure described. A high percentage of the failures occurred in the threads and through the lugs of the drill rods. Additional test rods were obtained, and variations in hardening temperature and method of quenching were tried. Individual rods yielded excellent performance, but the over-all averages did not exceed the averages shown in table 1. This testing is being continued on a limited scale.

An interesting result of the drill-rod investigation has been the discovery that 25 to 35 percent faster drilling rates can be obtained with any of the alloy rods over those obtained with carbon rods. No conclusive data have yet been gathered that will satisfactorily explain this marked difference.

In addition to drill-rod facts, some fundamental data were collected on the relationship between drilling rate and operating pressure in percussion drilling. As a preliminary study, the blows per minute delivered by the drilling machine and the energy per unit blow at various operating pressures were tested and recorded.

Test drilling was conducted in the lower level of the Underground Quarry. Data were obtained by varying the air pressure in 5-p.s.i.g. increments from 50 to 115 p.s.i.g. Drilling rates were obtained by timing along a base distance. The number of blows per minute delivered by the drilling machine was measured with a resonant reed tachometer placed in contact with the air hose to the drilling machine. No direct means was available for measuring energy per unit blow.

Table 2 summarizes the test data recorded. In addition to the measured values of blows per minute and drilling rate, the calculated average number of blows required per inch of penetration is shown at the various pressures.

TABLE 2. - Summary of test data for air pressure versus drilling rate in percussion drilling

Test No.	Air pressure, p.s.i.g.	Ave. blows per minute	Drill rate, in. per min.	Ave. blows per in. penetration
1	50	1,135	4.7	241
2	50	1,125	5.3	212
3	55	1,150	7.2	160
4	55	1,145	6.7	171
5	60	1,200	12.2	98
6	50	1,200	11.1	108
7	60	1,240	12.1	102
8	65	1,275	12.6	101
9	65	1,255	13.3	94
10	70	1,330	16.8	79
11	70	1,325	17.0	78
12	75	1,380	18.9	73
13	75	1,400	18.7	75
14	80	1,420	20.0	71
15	80	1,410	19.8	71
16	85	1,455	21.7	67
17	85	1,440	20.8	69
18	90	1,480	22.5	66
19	90	1,480	22.2	67
20	90	1,480	22.3	66
21	95	1,515	24.0	63
22	95	1,525	27.1	56
23	100	1,535	27.3	56
24	100	1,540	27.2	57
25	100	1,525	26.4	58
26	105	1,560	29.0	54
27	105	1,565	28.8	54
28	110	1,595	29.5	54
29	110	1,610	29.0	55
30	115	1,620	31.0	52
31	115	1,600	30.8	52

The relation between air pressure and drilling rate is illustrated in figure 9. The curve shown was plotted by means of the least-squares method. As air pressure increases, the drilling rate increases. However, the rate of increase tends to level off. A substantial increase in drilling rate can still be effected by a small increase in air pressure at the upper range of the curve.

A plot of the blows per minute delivered by the drilling machine versus air pressure is shown in figure 10. As air pressure increases, blows per minute increase. The rate of increase tends to lessen as air pressure rises.

Figure 11 illustrates the relationship between the blows per minute delivered by the drilling machine and the drilling rate at corresponding air pressures. A straight-line relationship exists; as blows per minute increase, drilling rate increases.

One of the principal objectives of this experiment was to derive an expression of energy expended at the drill bit at various air pressures. One measure of energy was felt to be the number of blows required to penetrate 1 inch. This value was calculated by dividing the number of blows per minute by the average penetration rate, in inches per minute at various air pressures. The curve for these values is shown in figure 12. This curve illustrates that, at pressures above 95 p.s.i.g., there is little apparent increase in energy per unit blow. At 95 p.s.i.g., 60 blows are required to penetrate an inch. At 115 p.s.i.g., 52 blows are required to penetrate 1 inch.

At 95 p.s.i.g., about 1,510 blows per minute are delivered by the drilling machine at a drilling rate of some 25 inches per minute. At 115 p.s.i.g., 1,620 blows per minute are delivered with a penetration rate of about 30.5 inches per minute. The increase from 25 to 30.5 inches per minute between 95 and 115 p.s.i.g. apparently is due primarily to the increase in the number of blows per minute, as there is little increase in energy per blow.

It follows that percussion-drill design should be concentrated more on increasing blows per minute at given air pressures and, to a lesser extent, on the energy delivered per unit blow. More efficient machine operation would be reflected in higher drilling rates at lower operating pressures.

Because of certain inherent disadvantages of percussion drilling, rotary-drilling experiments were begun in 1948. Preliminary testing was done with coal-type auger drills and bits. These early tests were not too successful.

A rotary-air-drill test unit then was devised. It was used to test several bit types, and it was found that masonry-type bits with tungsten carbide inserts were most promising.

Throughout all these initial tests, the operating conditions, such as revolutions per minute and thrust, were either fixed or had a very limited range. To determine optimum operating conditions, the rotary-test-drill unit was designed and built in early 1950. This unit is described in an earlier report. Briefly, it consists of (1) a 50-hp. electric motor that operates hydraulic pumps to drive a drill, and (2) a feed motor. The drill motor can be controlled automatically to rotate up to 1,050 r.p.m. As much as 4,000 pounds thrust can be delivered by the feed motor. The drill unit will develop up to 2,100 inch-pounds torque. A model HD-10 tractor was used for mounting the complete assembly (see fig. 13).

Early in the program, auger and rotary-bit manufacturers showed little interest in rotary drilling of oil shale. Repeated efforts have been made to gain the cooperation of bit manufacturers, and three major companies now are taking an active interest. Several special bit types have been obtained and tested (see fig. 14).

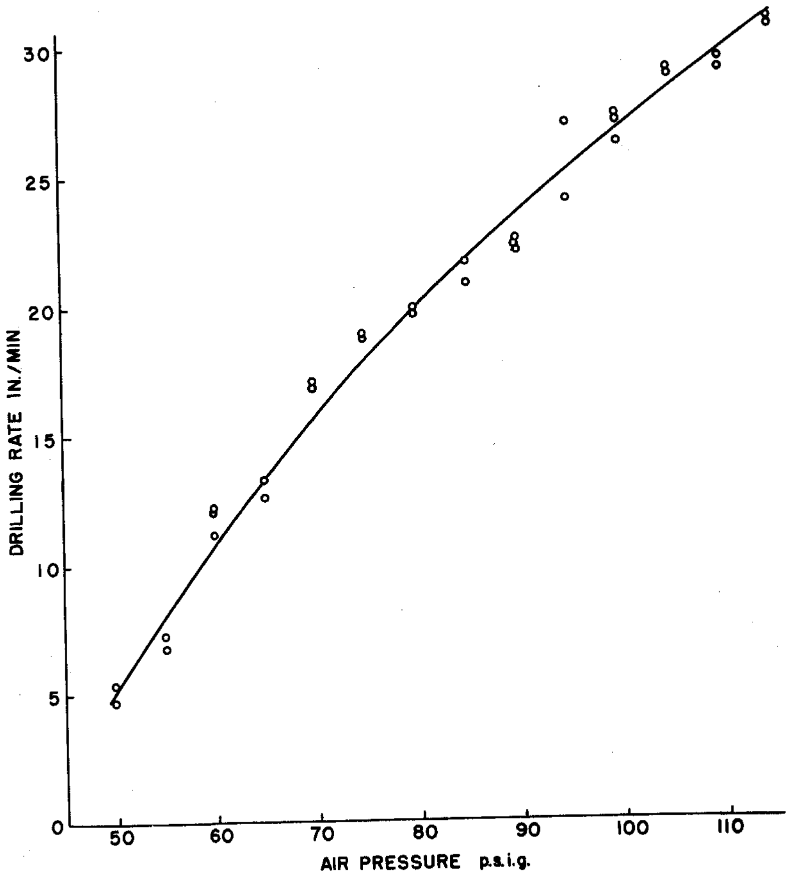

Figure 9. - Drilling rate versus air pressure (percussion drilling).

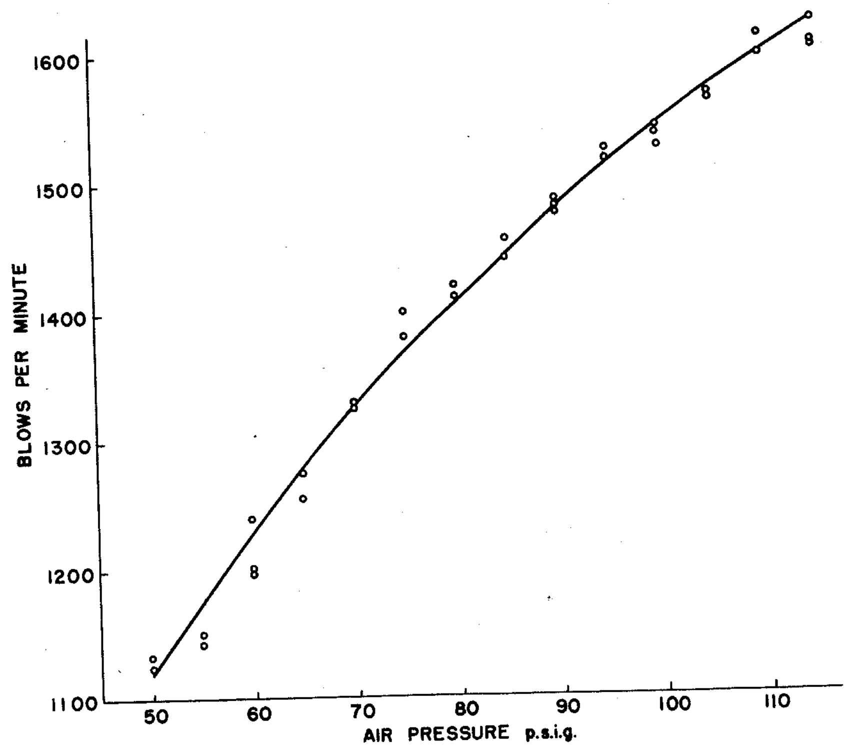

Figure 10. - Blows per minute versus air pressure (percussion drilling).

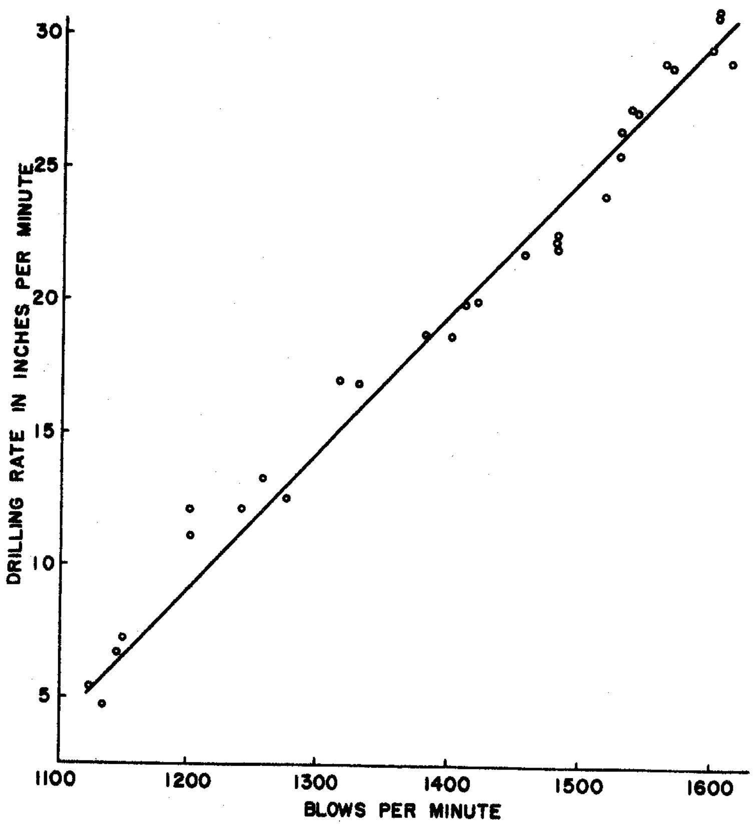

Figure 11. - Drilling rate versus blows per minute (percussion drilling).

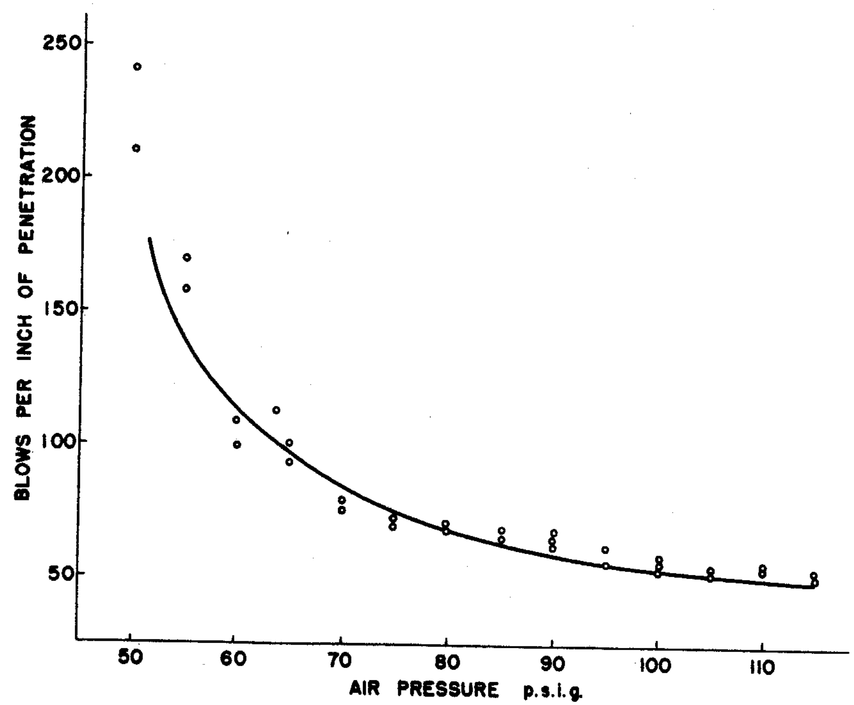

Figure 12. - Blows per inch of penetration versus air pressure (percussion drilling).

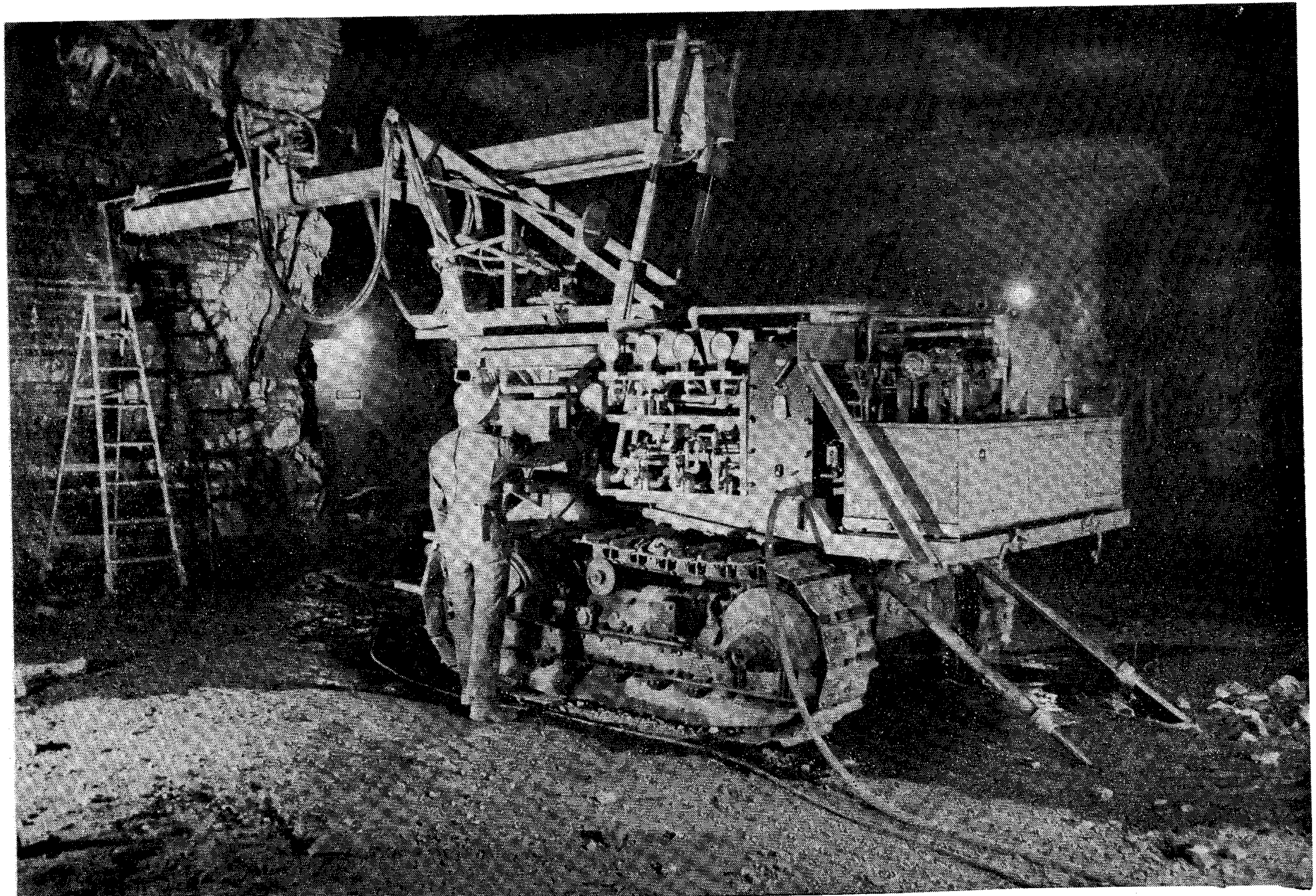

Figure 13. - Hydraulic rotary-test-drill unit positioned for drilling horizontal holes.

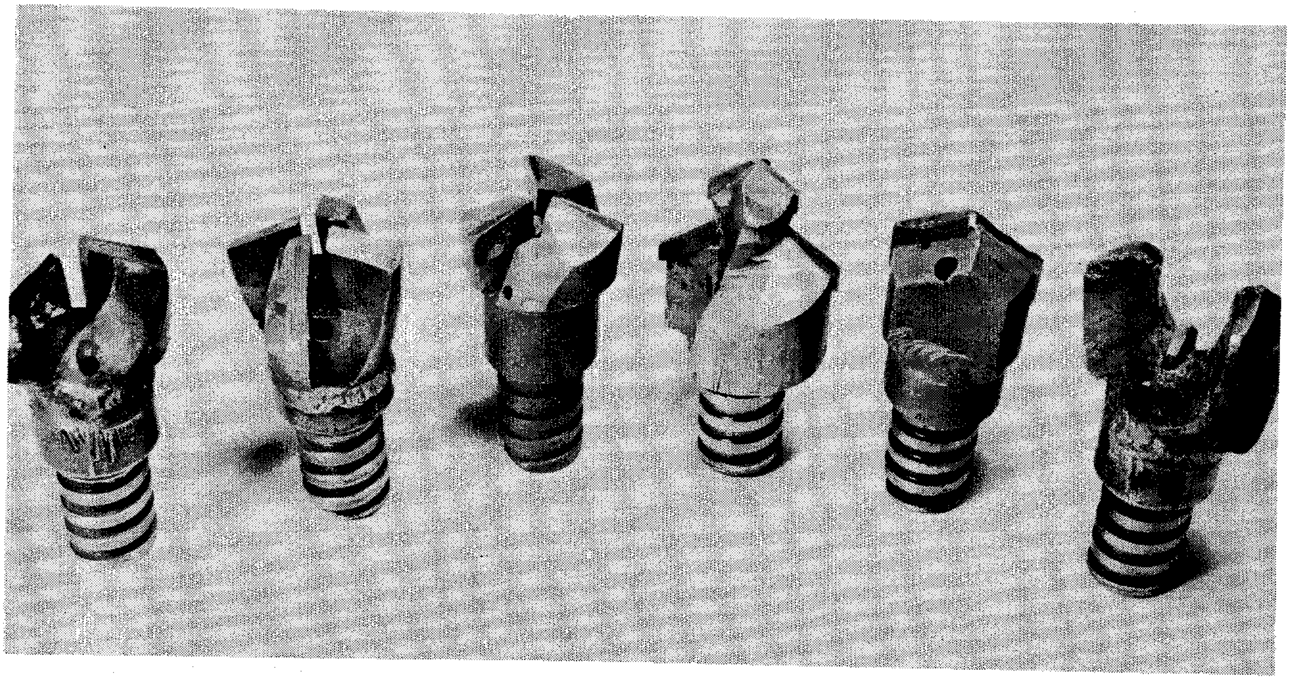

Figure 14. - A few of many bits tested with rotary drill.

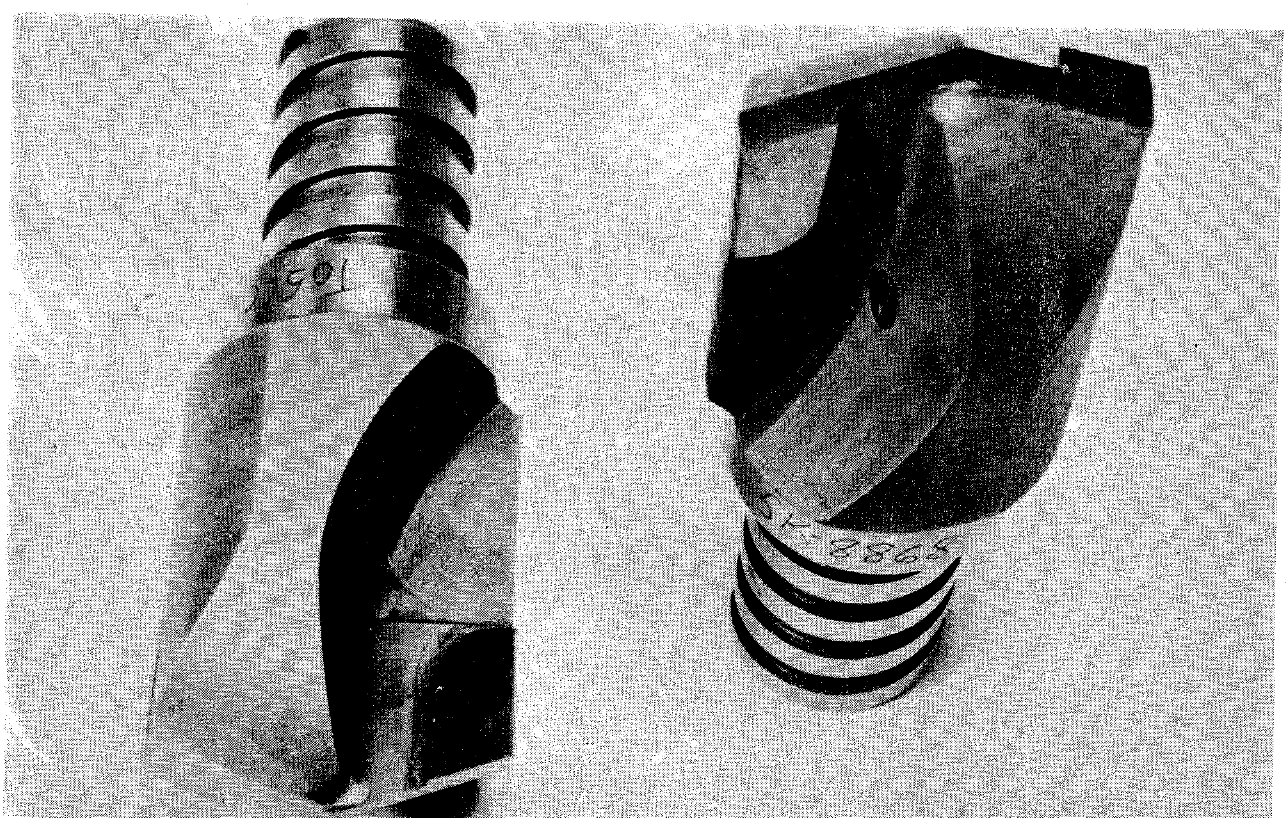

Figure 15. - Rotary-drill test bit, designed by Tool Specialty Co., that employs two grades of tungsten carbide inserts.

The most successful bit tested to date has been one manufactured by Tool Specialty Co. of Los Angeles, Calif. Early rotary-bit testing showed that the hardest-grade tungsten carbide inserts were the most resistant to abrasion but were subject to failure by compression in the center portion. Softer grades resisted failure by compression but yielded poor bit life because of low abrasion resistance. A bit combining the abrasion resistance of harder-grade inserts and compression strength of softer-grade inserts was designed by the Tool Specialty Co. (see fig. 15).

A 91.0-Rockwell-A-hardness tungsten carbide is used for the main insert, and harder-grade tungsten carbide blanks are used at the gage points where maximum abrasion occurs. One 2-inch-diameter bit of this type yielded a total footage of 1,123 at an average penetration rate of 61 inches per minute in the lower bench of the Underground Quarry. The bit was sharpened six times for an average of 129 feet per sharpening.

Two 1-5/8-inch-diameter bits of the same design have yielded 600 feet at an average penetration rate of 65 inches per minute and 765 feet at about 75 inches per minute before the first sharpening was required. Estimated bit and drill-rod cost with bits of this type is about $0.01 per foot of hole compared to about $0.028 per foot of hole for percussion drilling.

In addition to the direct saving in bit and drill-rod costs, the higher drilling rates achieved by rotary methods would make possible additional savings, such as lower labor costs, lower power consumption, etc.

All the test drilling to date has been in the bench level of the Underground Quarry. Enough operating and bit-testing data have been gathered to show that the bench level of a commercial or prototype oil-shale mine could be drilled by rotary methods at a saving over the present percussion methods.

Only limited testing has been done on the drilling of headings by rotary methods. This testing has indicated that the headings can be drilled provided a drilling and blasting pattern can be worked out wherein the blast holes are in the medium- to high-grade layers of the shale. The lower-grade sections are extremely abrasive, and bit wear is excessive.

An important phase of the mining-research program has been the study of mine structure and stress analysis. Preliminary studies of the physical properties of the Green River oil-shale formation were made in the Barodynamics Laboratory at Columbia University during the latter part of 1945 and the early part of 1946. The results of this investigation indicated that underground caverns 60 feet wide supported by 60-foot-square pillars could be opened safely in the Underground Quarry.

As a check of the laboratory data, an experimental test room 50 feet wide by 100 feet long was excavated directly under the selected roofstone. This excavation was completed in December 1946. Specialized equipment and apparatus were installed in early 1947 to permit a study of the various physical properties of oil shale as a formation in place and to demonstrate the practicability and use of scientific instruments in mining research.

After installation of the specialized equipment, the test room was widened to 80 feet. The procedure was to widen in 10-foot increments, with periods of inactivity between widenings. Daily observations of roofstone behavior were made. By November 1948 the room had been widened to 80 feet. During the entire period of widening from 50 to 80 feet, no indications of excessive strain had been observed.

To reduce the effect of end support, an extension to the test room was excavated. This extension was 100 feet long, making the total length of the test room 200 feet. In the extension, the procedure also was to widen in 10-foot increments, with periods of inactivity between widenings. The extension was widened to 80 feet by June 1951 (see fig. 16).

Special equipment for observing the behavior of roofstone included microseismic recorder units, subsidence and differential sag stations, differential water-level gage, and SR-4 strain gages. The microseismic units detect and record the subaudible noises caused by cracking or rearrangement of rock crystals in the roofstone as it becomes subject to increasing stress. By means of geophones, these noises are converted into electrical impulses which, after amplification, are automatically recorded on strip charts. Six geophones have been installed in the test-room roofstone (see fig. 16).

To measure the subsidence of the immediate roofstone and to detect separations within the roofstone, three subsidence and differential sag stations were installed (fig. 16). Specially designed equipment was used to record subsidence of the immediate roofstone and at the 4-, 6-, 8-, 12-, and 16-foot levels in the roofstone. The differential sag obtained by differences between readings at any two levels indicated separations that occurred as the room was widened.

Measurements taken at the subsidence stations actually measure convergence. To determine the relative amounts of floor rise and roof sag, a water-level gage was installed early in 1950. The data obtained with this instrument have indicated no rise of the test-room floor. The subsidence measurements taken have been designated as roofstone sag measurements.

The separations found by sag differential measurements were checked visually by an instrument termed a boroscope. This instrument, sometimes called a stratascope, was obtained from the Roof Control Section, Coal Mine Inspection Division, Bureau to Mines, College Park, Md. Seven vertical NX-size diamond-drill holes, spaced as shown in figure 16, were drilled and examined to the 20-foot thickness level. The results logged in December 1949 placed a consistent separation at 8-1/2 feet above the roofstone surface. The next major separation was at 13 feet. The only other separations were at 6 feet and about 18 to 20 inches, but neither was totally reproducible throughout the area checked.

The roofstone in the test room remained intact until August 21, 1951, when an 18-inch layer in section 1, figure 17, dropped to the floor. The area in section 2 fell about 10 days later. In both sections the fallen material

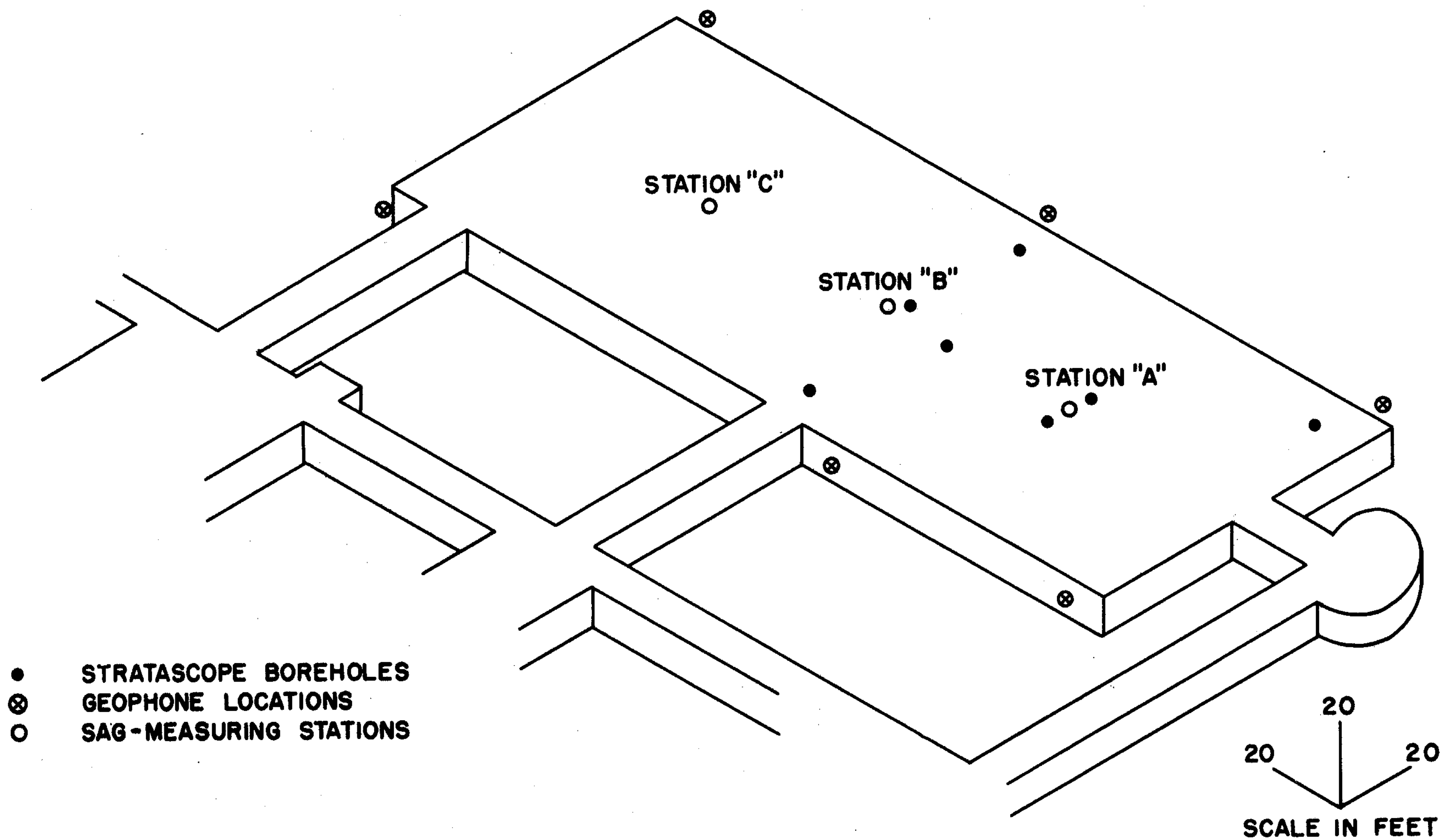

Figure 16. - Isometric drawing of test room, showing stratascope holes, geophones, and sag-measuring stations.

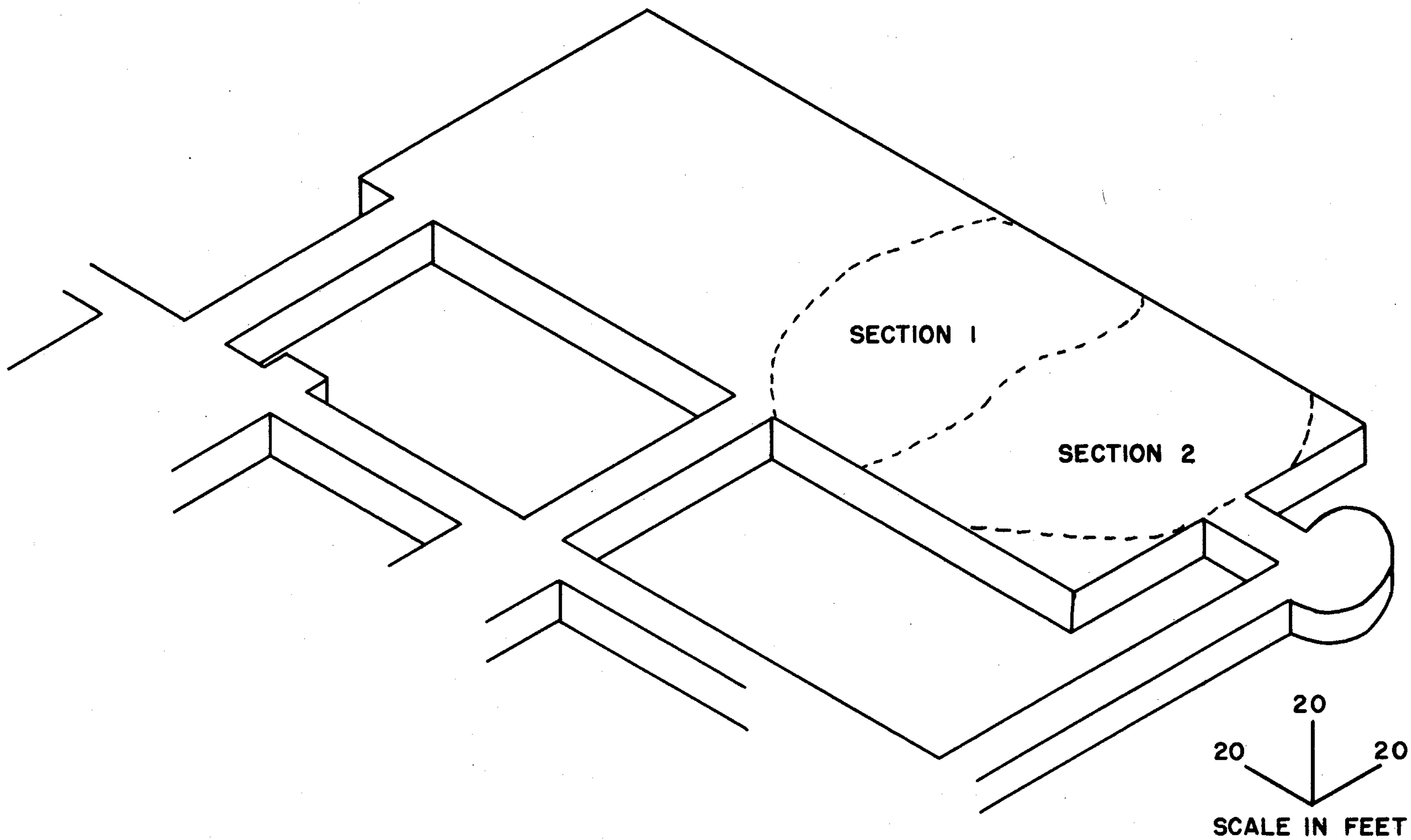

Figure 17. - Isometric drawing of test room, showing caved sections.

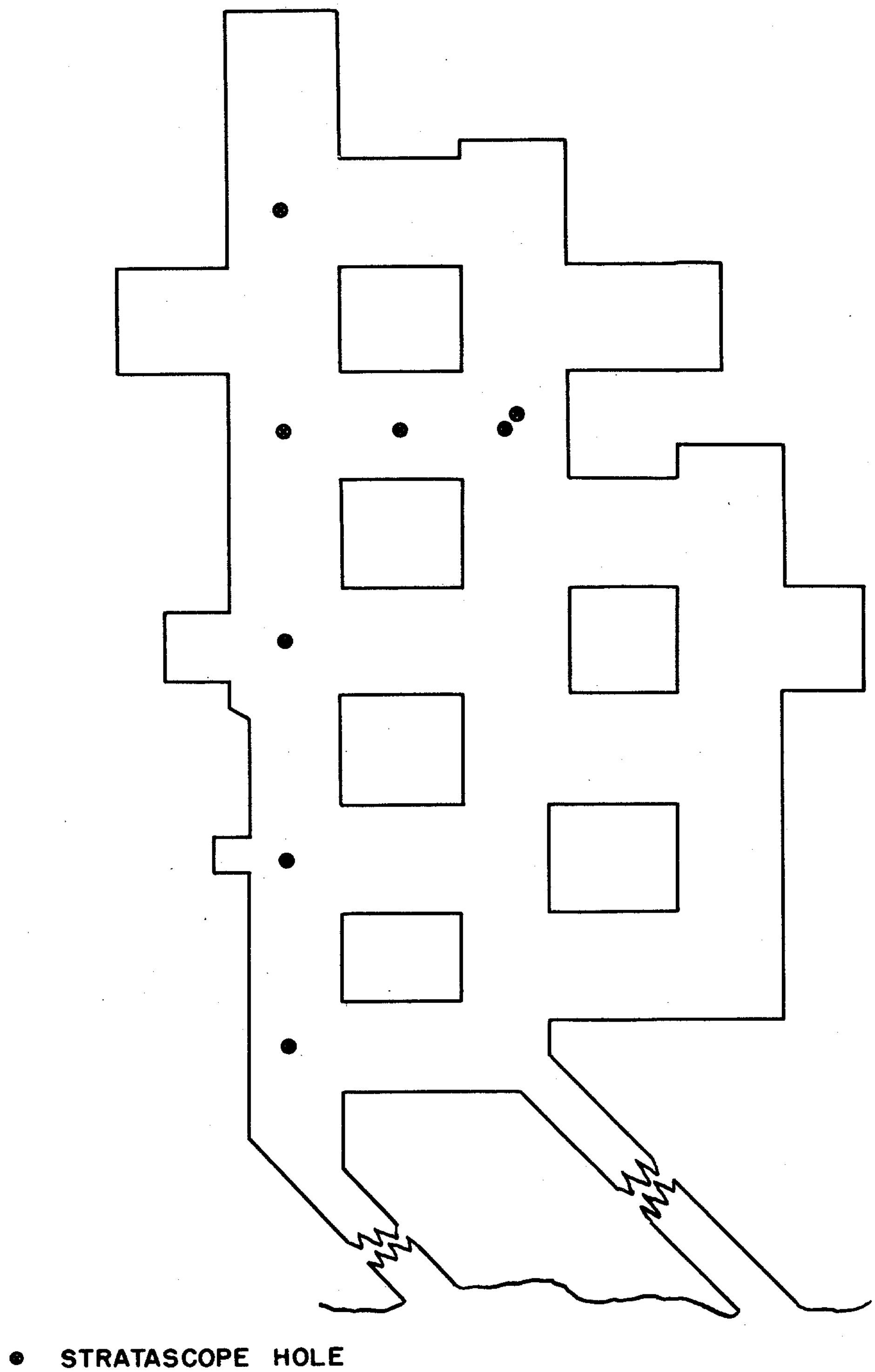

Figure 18. - Plan of Underground Quarry, showing stratascope holes.