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Pumped-Slurry Backfilling of Abandoned Coal Mine Workings for Subsidence Control at Rock Springs, Wyo.

By G. J. Colaizzi, R. H. Whaite, and D. L. Donner

UNITED STATES DEPARTMENT OF THE INTERIOR

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With an appendix on Hydraulic Model Studies for Backfilling Mine Cavities by E. J. Carlson, U.S. Bureau of Reclamation, Denver, Colo.

UNITED STATES DEPARTMENT OF THE INTERIOR
James G. Watt, Secretary
BUREAU OF MINES

As the Nation's principal conservation agency, the Department of the Interior has responsibility for most of our nationally owned public lands and natural resources. This includes fostering the wisest use of our land and water resources, protecting our fish and wildlife, preserving the environmental and cultural values of our national parks and historical places, and providing for the enjoyment of life through outdoor recreation. The Department assesses our energy and mineral resources and works to assure that their development is in the best interests of all our people. The Department also has a major responsibility for American Indian reservation communities and for people who live in Island Territories under U.S. administration.

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PUMPED-SLURRY BACKFILLING OF ABANDONED COAL MINE WORKINGS FOR SUBSIDENCE CONTROL AT ROCK SPRINGS, WYO.

by

G. J. Colaizzi,[1] R. H. Whaite,[2] and D. L. Donner[3]

ABSTRACT

The Bureau of Mines, at the request of local authorities in Rock Springs, Wyo., investigated and conducted through contracts a multistage program of exploratory drilling and pumped-slurry backfilling of 15 areas of potential subsidence in abandoned mine workings underneath that community. Initially, the Bureau in 1969 had recommended a program of gravity blind flushing of some of the inaccessible mine voids, and in 1970 a new technique, the pumped-slurry injection process, was tested for the first time in a site adjacent to the city's area of severe surface subsidence. Success of this initial testing program, and of a large-scale project in Scranton, Pa., led to further large-scale projects, funded by Congress, that resulted in the successful backfilling not only of all 15 target areas of potential subsidence in Rock Springs, but also of several areas in other States. Total cost of the projects in Rock Springs, including the original pumped-slurry test, was $3,243,993. A total of about 923,000 tons of sand was injected hydraulically into mine voids, rendering 178 acres of residential and central-downtown areas of Rock Springs less susceptible to subsidence damage. The pumped-slurry method was proved to be much superior to the gravity blind flushing method in terms of the amount of solids that could be injected underground through a single borehole. However, there are special conditions that make this technique more or less applicable in different areas or underground configurations, as noted in the report's conclusions.

INTRODUCTION

Surface subsidence is often the consequence of underground mining operations. Removal of solid material from beneath the earth's surface produces voids, and once the natural support afforded by this material is taken away, the weight of the overburden is redistributed. If pillars of material that are left unmined or timbers or other artificial support left underground are

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not sufficiently strong to support the overburden, the overlying rock breaks and falls into the voids and/or crushes the pillars. This breakage may proceed upward in overlying material as far as the surface, causing potholes, cracks, or general settling of the ground. The time between the completion of mining operations and the disturbance at the surface may be a matter of days or a period of many years, depending upon a number of factors including the nature of the overlying rocks, depth of the excavation beneath the surface, and the method of mining employed. Subsidence is postponed when correctly spaced and adequate-size pillars are used to provide overburden support or is minimized when voids are backfilled with suitable material for the same purpose.

In the United States, nearly 100,000 underground mines are in existence, of which an estimated 90,000 are closed or abandoned. The total land that has been undermined for the production of coal, metals, and nonmetallic minerals has been estimated to be about 7-1/2 million acres. The exact percentage of the undermined land that has been affected by subsidence is not known. In a recent study of land utilized by the mining industry, the surface area that had subsided or was otherwise disturbed by underground mining from 1930 through 1971 was estimated at 105,000 acres. Of this acreage, 84 percent resulted from mining coal (15).[4]

Most of the land affected by subsidence is removed from centers of population. Adverse effects in these areas consist of crop damage, altered drainage patterns, and reduced land values. The most severe damage, of course, has occurred in urban areas. Millions of dollars in damage has resulted from the differential settling of buildings, pavements, subsurface pipelines, and other facilities, compounding existing problems as the pressure to develop undermined land is increasing in many metropolitan areas in response to accelerating demands for more living space.

For the control of subsidence in built-up areas where the underlying mines have been abandoned, backfilling is believed to provide the most practical means of minimizing damage to the communities. From time to time, studies of subsidence problems have been made for the city of Scranton (7, 11, 23) and for the Commonwealth of Pennsylvania (16-17). All the studies recommended programs of hydraulic backfilling.

The first reported use of hydraulic backfilling of mine workings was in the Anthracite region over 100 years ago. The purpose was to stop the subsidence of a church, and the treatment succeeded (2, pp. 99-100). Hydraulic backfilling was developed during the late 1800's and early 1900's and was used in about one-fourth of the anthracite mines for such purposes as to extinguish mine fires, to arrest the development of progressive pillar failure known as mine squeeze, to permit the reclaiming of pillars, to dispose of unwanted mine refuse, and to protect the surface. The practice of backfilling by the coal industry in the United States decreased after World War I with the decline of the anthracite industry. In domestic bituminous coal mining operations,

[4]Underlined numbers in parentheses refer to items in the list of references preceding the appendix.

backfilling has never been common practice (4). Applications to metal mining, however, in the United States and elsewhere, have provided solutions to a variety of ground control problems resulting in greater resource recovery, safer working conditions, and reduced mining costs (12).

The principal development of hydraulic backfilling in coal mining took place in Europe in the early 1900's, where the practice had spread from the Anthracite region of the United States. Most European coalfields differ from those in the United States in that the coalbeds are much deeper, the concentration of coal within a vertical section is much greater, and longwall mining methods predominate, whereas room-and-pillar mining is the most common method in the United States (18, p. 35). Many European coal mines underlie highly developed industrial areas or commercial waterways that require protection. The purposes of backfilling are to contribute to roof control under ground pressures and to permit as nearly complete recovery of the coal as possible as well as to support the surface. Hydraulic backfilling remains a part of mining operations in some European coalfields where thick beds of coal are being removed from beneath densely populated areas such as in France and Poland.

The Bureau of Mines' interest in hydraulic backfilling is as old as the Bureau itself. The First Annual Report of the Director (8, pp. 42-43) describes an ongoing study of mine filling (hydraulic backfilling) to make the mines a safer place in which to work and to reduce settlement of the surface. The report resulting from that study covered the history, applications, methods, and costs of hydraulic backfilling (6). Other early Bureau publications reported on sources of backfill materials and applications of hydraulic backfilling to various mining problems. At the end of World War II, the Bureau looked into the backfilling problem as it related both to the conservation of anthracite and to the prevention of subsidence in order to determine what role the Federal Government might play (1). A comprehensive engineering study of the backfilling problem in the Anthracite region by the Federal Government was recommended; the work was to be done in cooperation with the Commonwealth of Pennsylvania, and with the anthracite industry, should the study lead to action.

The current participation by the Bureau of Mines in subsidence-control projects in areas of abandoned mines is provided for by two pieces of legislation that authorize Federal-State cooperation. Public Law 87-818, an amendment in 1962 to the 1955 Anthracite Mine Drainage Act (Public Law 84-162), authorized the Secretary of the Interior to participate equally with the Commonwealth of Pennsylvania in the filling of voids in abandoned anthracite mines, in those instances where such work is in the interest of the public health or safety. Under this provision, four subsidence-control projects were completed between 1962 and 1965. The Appalachian Regional Development Act of 1965 (Public Law 89-4) and its amendments of 1967 (Public Law 90-103) included authorization to fill voids in abandoned coal mines within the Appalachian region. Costs under the Appalachian Act are shared 75 percent by the Federal Government and 25 percent by the cooperating State. By the end of 1978, fifteen subsidence control projects had been completed under the Appalachian program and additional projects were in progress in the Anthracite region of

Pennsylvania. Under the same program one project in Maryland and one in West Virginia have also been completed.

Under the authority assigned to the Bureau of Mines by the Organic Act (May 16, 1910) and its succeeding amendments and pursuant to regulations (30 U.S. Code 1-11), the Bureau conducts scientific and technologic investigations concerning mining and its related problems. By the end of 1978, twenty-one subsidence-control demonstration projects had been conducted under this authority, including the Rock Springs, Wyo., projects and the demonstrations in Pennsylvania, Maryland, West Virginia, and Illinois.

In demonstration projects conducted under the Organic Act, the Bureau is the primary signatory to any contract that is entered into. Aside from any agreements with contractors, the Bureau enters into cooperative agreements with State or local authorities. Through these agreements the State or local government, referred to as the "cooperator," grants the Bureau and its contractors the right to enter onto public streets and land in order to carry out the proposed project work.

A Draft Environmental Impact Statement, DES 75-37, on "Surface Subsidence Control in Mining Regions," was used (prior to its final approval in November 1976) as a guide in conducting the subsidence control projects in Rock Springs, Wyo. (21).

Purpose and Scope of Report

This report reviews the investigation by the Bureau of Mines to determine the cause and magnitude of the subsidence problem at Rock Springs, Wyo. It discusses methods of backfilling mine voids that are used to minimize the effects of subsidence. Four subsidence control demonstration projects at Rock Springs are discussed in detail, and an assessment is made of the effectiveness of the work performed.

Acknowledgments

The contributions of many individuals to this report, and to the demonstration projects on which it is based, are gratefully acknowledged. Paul L. Russell, former Research Director of the Bureau's Denver Research Center, was responsible for Bureau of Mines activities at the demonstration projects in Rock Springs. Neil Morck, District Manager for the U.S. Bureau of Land Management at Rock Springs, provided leases for two of the borrow pits and stipulations governing their use. Robert Oster, District Engineer for the Union Pacific Railroad, provided leases for use of the railroad's right of way.

Officials of the city of Rock Springs, including Paul Wataha, Mayor, Wayne Johnson, City Engineer, and Hyram Fedji, Director of Planning, Engineering and Development, provided excellent cooperation in the performance of the work on city streets. Valuable information to the drillers on the location of utilities under streets was obtained from Mountain States Telephone and Telegraph companies and the Mountain Fuel Supply Co. Data on surface altitudes

from several bench marks in the three large-scale project areas were made available to the Bureau as needed by the firm of Johnson-Fermelia & Crank, Inc.

Injection operations at the first test of the pumped-slurry process and at the first large-scale demonstration, both at Rock Springs, were performed by the Dow Chemical Co., Dowell Division, of Tulsa, Okla. A concept that sand slurry could be transported in pipelines directly from the borrow pit to the injection boreholes, a distance of over 2 miles, demonstrated additional flexibility in the use of the pumped-slurry process. Dowell Division personnel who contributed to the success of these nonroutine operations were John D. Stewart, Milton E. Heslep, Robert Hurst, L. D. Boughton, and George Laflin; and also A. J. Meyers on the use of sonar caliper surveys.

Engineers of the WHAN Co. and of the Bober Co., contractors for the second and third large-scale projects, contributed to improving the overall operations through more efficient use of the automated controls of the slurry mixture. In this way, delivery of a more uniform mixture to the injection boreholes was gradually accomplished with a considerable saving in the use of power. As the work progressed they relocated the equipment within the confines of the plant site and altered the use of the equipment from time to time to improve performance and eliminate as much noise and dust as possible.

Charles S. Kuebler, Chief of the Bureau's Environmental Affairs Field Office at Wilkes-Barre, Pa., where similar type projects were being conducted, and E. J. Carlson, engineer, Hydraulics Branch, U.S. Bureau of Reclamation, Denver, Colo., in charge of the model studies, provided valuable insight into the operation of the pumped-slurry method and the behavior of the slurry.

Much of the description of the nature of mine subsidence, its history, and various methods of control, including discussions of the pumped-slurry technique that appear in this report, is attributed to Alice S. Allen and Ralph H. Whaite, authors of Bureau of Mines Information Circular 8667 (22).

BACKGROUND

Development of Rock Springs, Wyo., as a mining community began in the 1860's with the westward extension of the Union Pacific Railroad through southern Wyoming. The railroad's demand for coal persisted well into the 20th century, leaving about 900 acres of Rock Springs undermined. Extraction of coal by the room and pillar method under most of the city had reached an advanced stage, and according to available mine maps it appeared that many of the remaining pillars were too small to support the overburden indefinitely. Under other areas some of the pillars had been removed. Reports of surface subsidence in various parts of the city became noteworthy in the late 1960's. Damage to houses, commercial buildings, streets, gas mains, waterlines, and sewers was considerable.

Subsidence Problems

The city of Rock Springs (fig. 1) is located in Sweetwater County in southwestern Wyoming on U.S. Highway 30, Interstate 80, and the Union Pacific

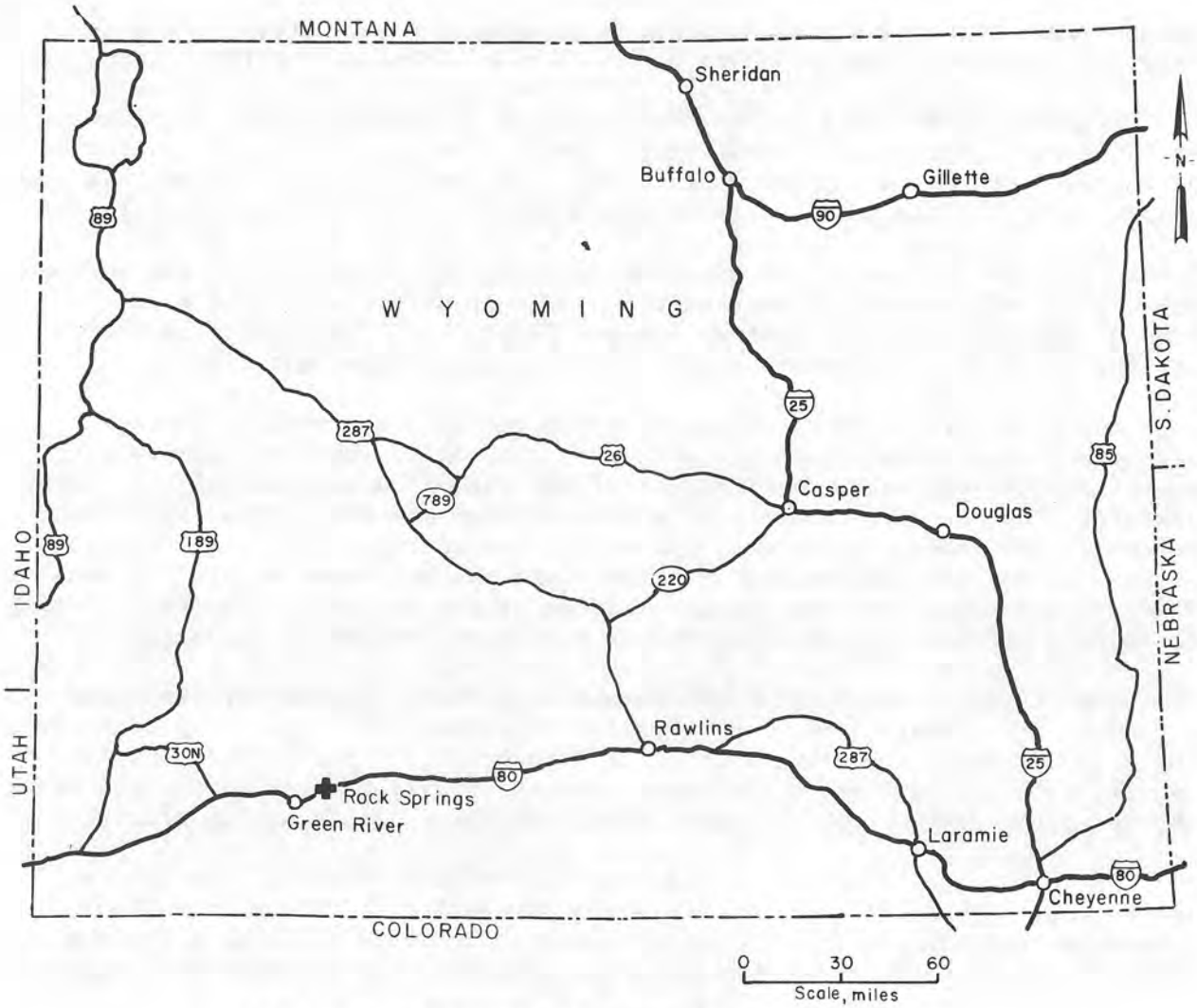

FIGURE 1. - General location map of Rock Springs, Wyo.

Railroad. The 1969 population of the city was about 12,000. Mining of bituminous coal had been the principal industry in the city from the 1860's to the 1950's, and most of the built-up area is extensively undermined, a condition typical of many of the early mining communities. As a consequence, subsidence of the surface, locally termed "sink holes," has plagued Rock Springs for many years. Both the State Highway Department and the Union Pacific Railroad have filled, in some areas, abandoned mine workings beneath their respective rights of way.

In 1969, the Bureau of Mines conducted an investigation of subsidence in Rock Springs, at the request of local authorities, to determine the cause and to recommend solutions (5). Severe subsidence damage had been experienced since 1967 within a 2-acre area in the eastern portion of the city, affecting at least 18 houses and damaging streets, sidewalks, gas mains, waterlines, and sewers. Subsidence was gradual and continuing, achieving maximum

settlement of about 30 inches, accompanied by lateral displacements and some heaving. Elsewhere in the city, localized "potholes" continued to appear on the surface of the ground from time to time.

The site of the severe damage was about 6 blocks from the center of the city near the intersection of Connecticut Avenue and D Street. About one-half of the damage at the time of the investigation was to property known as the Rock Springs Camp owned by the Mountain Fuel Supply Co. The camp consisted of seven dwellings and one office building, all on the north side of D Street. Individual private homes on the south side of D Street were also affected. Ten of the 18 affected houses were seriously damaged.

Subsurface Conditions

From a study of available mine maps it was determined that collapse over mine openings was the principal cause of the subsidence in the Connecticut Avenue-D Street area and in other parts of the city as well. Of the 900 acres of the city that was undermined, Bureau engineers reported that in 15 separate areas ranging in size up to 46 acres, the conditions in the mines with respect to supporting pillars were comparable to those under the Connecticut Avenue-D Street area. These critical areas comprising about 210 acres constituted a significant part of the city, and facilities therein served a wide range of functions. They include sections of the central business district, the city hall, and 828 dwelling units. At the time of the investigation in 1969, it was estimated that the total value of the houses alone was nearly $11 million. Figure 2 is a map of Rock Springs showing the locations of the 15 critical areas.

Geologically, the city of Rock Springs is located on the west edge of the Rock Springs Uplift (10, 20). Bedrock in the area is the Rock Springs Formation of Cretaceous age--an irregular series of coalbeds, carbonaceous shale, siltstone, claystone, and sandstone. The strike of the formations is N 36° E, and the dip is 6° to the northwest at the site of the subsidence. Downdip (northwest of the site) the dip steepens to 30°. Faults are common in the area, but under the built-up section of the city, faulting is believed to be of relatively minor consequence. Figure 3 is a typical cross section of the coal seams under the city of Rock Springs.

Depth to bedrock ranged from 6 to 52 feet in boreholes that were drilled in the Connecticut Avenue-D Street area. The overlying alluvium is silty, very fine sand. A deposit of clay beneath the silty sand was reported in one borehole. The silty sand deposit is of the type that is subject to compaction in the presence of excess water, hence a potential contributor to subsidence. However, in the Rock Springs subsidence area, the evidence indicated that collapse over mine openings was the principal cause of subsidence (13). Deeper strata within the bedrock sequence were found to have been displaced downward, and the pattern of subsidence at street level was observed to reflect the spacing of rooms and pillars in the mine below.

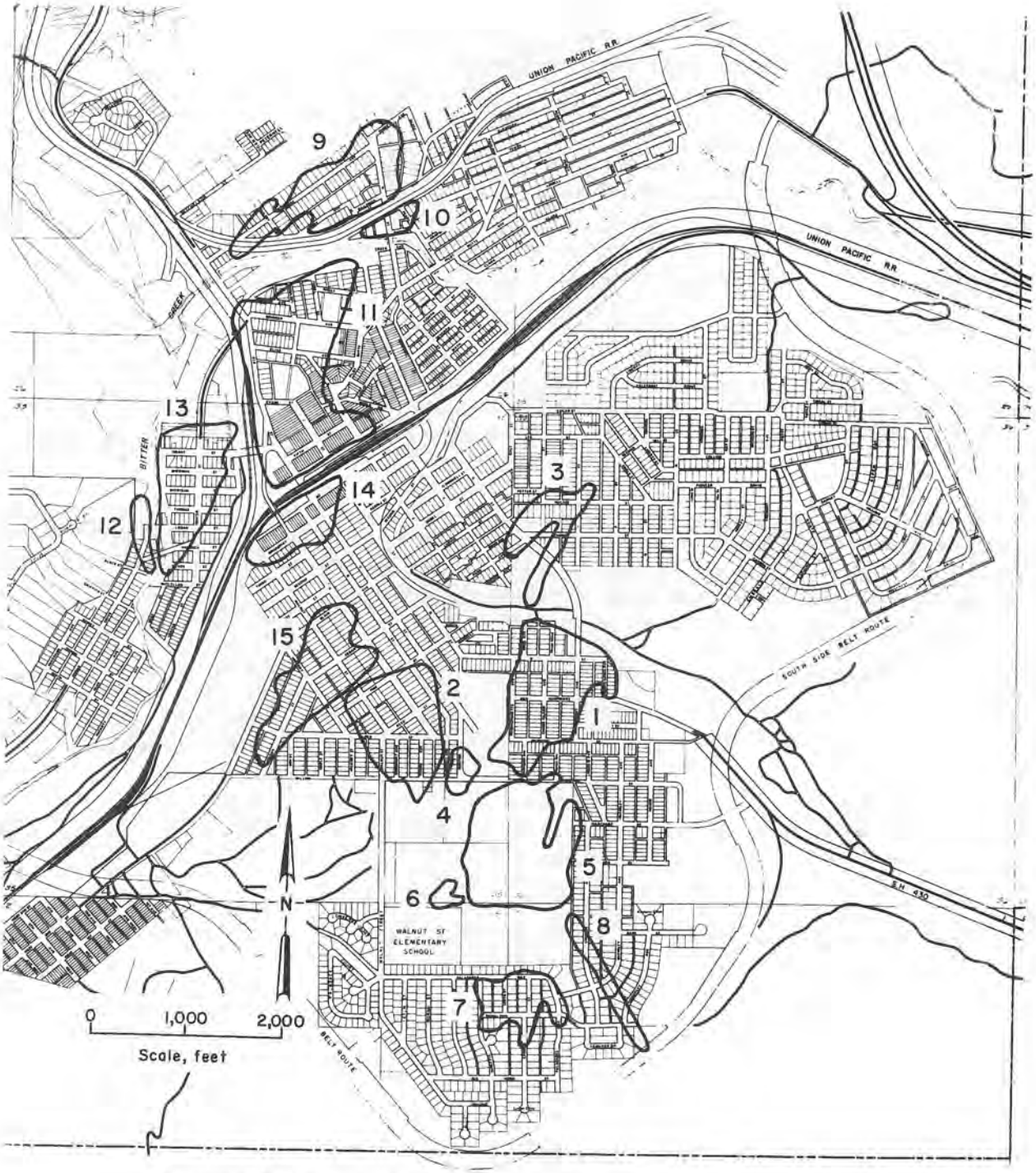

FIGURE 2. - Map of Rock Springs showing the 15 critical areas.

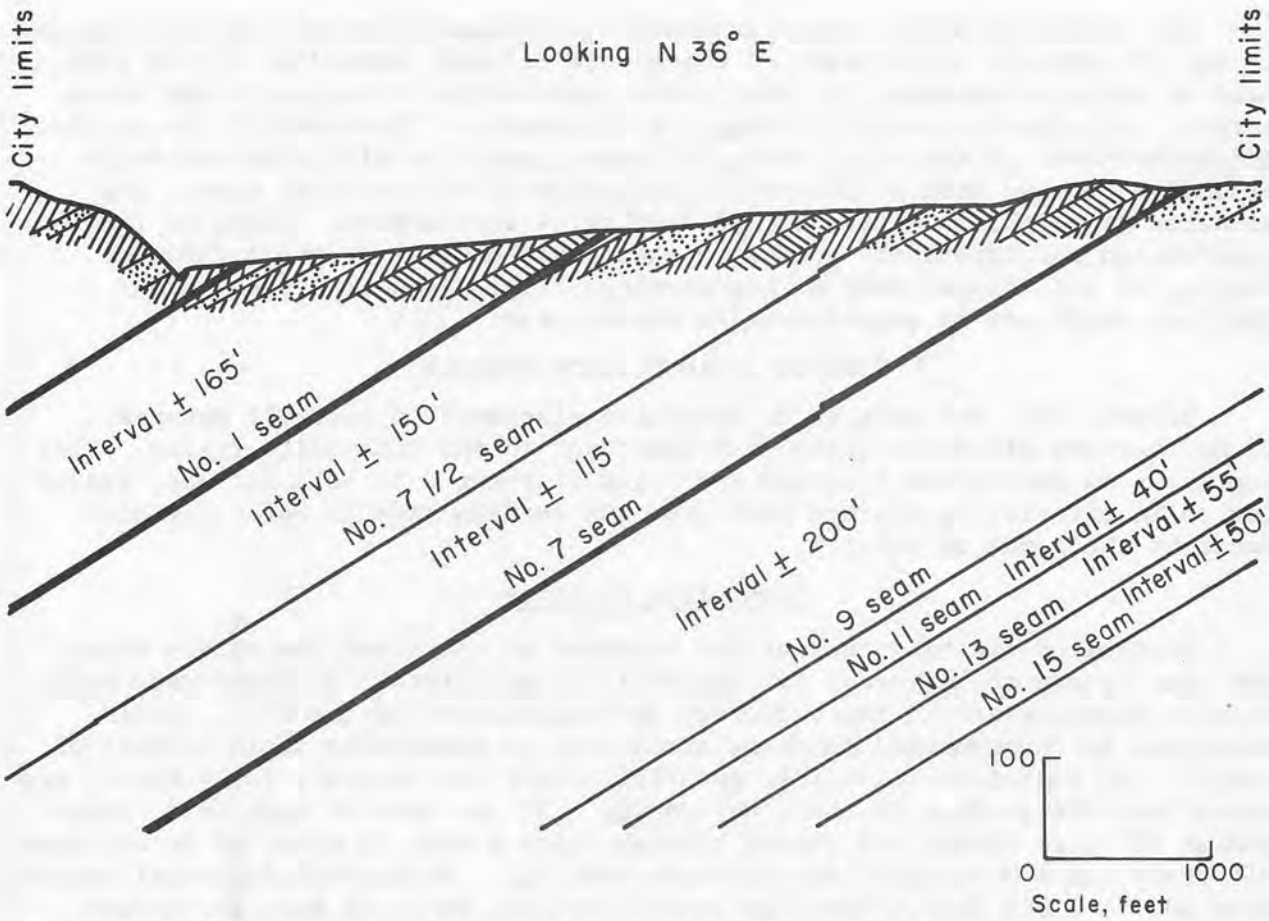

FIGURE 3. - Typical section of the coal seams under the city (note distorted scale; actual dip of seams is 6°).

In the vicinity of Rock Springs, the aggregate thickness of coal exceeds 90 feet. Under the built-up part of the city, two coalbeds were mined extensively, but their structural position is such that one bed has been mined beneath another mined bed in only one section of the city. Two other coalbeds were mined to a lesser degree. The room-and-pillar system of mining was used exclusively. Patterns of extraction, however, were irregular owing to minor faulting and changes in mining techniques over the years.

Water presented a problem during mining operations. Reportedly, one mine pumped 500,000 gallons per day when operating, and water-related problems were a factor in the closing of another mine in the demonstration area. Water levels measured in drill holes indicated that about 75 percent of the workings are now flooded.

A key element in planning successful backfill operations is the availability of accurate and complete mine maps. The information available from mine maps on coalbeds beneath Rock Springs was not complete, and data obtained from drilling indicated that the maps would require adjustment to correlate with surface maps. About a third of the exploratory boreholes, which were drilled to intersect openings according to the maps, terminated in solid pillars of coal.

The Bureau of Mines report included a recommendation that the mine voids in the 15 critical areas, most of which were flooded, should be filled with sand to support remaining pillars, reduce rock strata breakage in the overburden, and minimize possible damage at the surface. To backfill the entire undermined area of the city, about 900 acres, would be difficult and quite expensive because most of the voids, exclusive of the critical areas, are situated at depths in excess of 300 feet below the surface. Based on local observation and experience gained in subsidence control in the Anthracite region, it was assumed that strata breakage originating at depths below 300 feet would not be significant at the surface.

HYDRAULIC BACKFILLING METHODS

Before 1970, two methods of hydraulic placement of backfill material in underground mined-out spaces had been used in the Anthracite region. They are known as controlled flushing and blind flushing. In both methods, granular solid material is sluiced down from the surface through boreholes with water by the force of gravity.

Controlled Flushing

Controlled flushing is possible in mines in which men can safely enter and gain access to key areas for the filling operations. Bulkheads are built in mine passages around the periphery for containment of the fill. Drain boxes may be incorporated in these structures to facilitate rapid removal of water. The injection boreholes, generally about one borehole for 4 acres, are cased from the surface to the mine opening. At the base of each hole, long-radius 90° pipe elbows are placed through which slurry is diverted to horizontal pipes and distributed into the mine workings. Horizontal dispersal ranges from 300 to 1,000 feet, depending on the vertical distance from the ground surface to the mine opening and the solids concentration of the slurry. Controlled flushing provides the best support and is used where conditions permit.

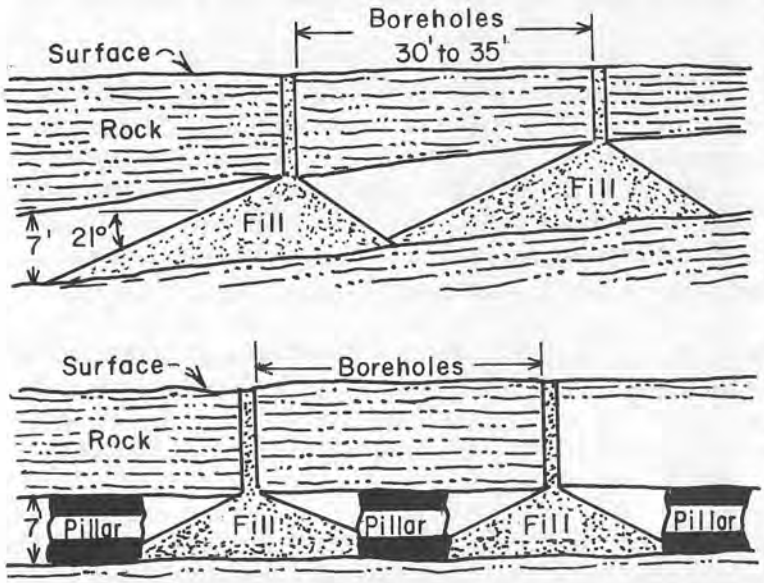

FIGURE 4. - Typical sections showing boreholes and configuration of fill material placed in mine voids by the gravity blind flushing method.

Blind Flushing

Many abandoned mine openings are inaccessible because of flooding or extensive caving. Such openings were flushed blindly in the past. The gravity-feed method simply builds a conical pile beneath the underground opening of the flushing hole. When the apex of the cone builds up to the mine roof, no more fill will enter the mine opening. Further injection must use other boreholes, as shown in figure 4.

Depending on conditions underground, such as the dip of the bed, its height, and the proximity of pillars or occasional caved roof strata, the volume of material that can be injected in the this manner from each borehole ranges from 20 to 1,000 cubic yards. In a 6-foot bed that is relatively flat, for example, in which about 45 percent of the bed remains in pillars, only about 100 cubic yards can be injected from a single hole. Therefore, injection holes must be closely spaced, but at best only about a third of the underground open space is filled by this blind flushing technique. Most blind flushing projects have required hundreds of flushing holes. In built-up areas, it may not be possible to drill boreholes in critical areas where buildings or other structures interfere or where easements cannot be obtained from property owners. Therefore, most of the backfilling under built-up areas is done through boreholes drilled in streets and alleys, and the support given is of only indirect benefit to adjoining buildings (fig. 5).

Pumped-Slurry Injection

A new technique for the blind flushing of inaccessible mine workings, termed "the pumped-slurry injection process," was first demonstrated in 1970 at Rock Springs, Wyo., in a test site adjacent to the Connecticut Avenue-D Street area where severe subsidence problems had developed. This technique differs from the open gravity-feed methods previously described in that pumping energy is used to achieve a dynamic suspension of solid particles in water and the system is completely closed from the point of slurry mixing to the bottom of the injection borehole.

In this process, granular material is blended with water, and the suspension (slurry) is pumped to the point of deposition. Water from an inundated mine may be used and recirculated, or water from an external source may be used without being recirculated. During mixing, each solid particle becomes enclosed by fluid so that friction during transit is reduced. The slurry is pumped continuously from the mixing tank through a pipeline on the surface and thence down through a borehole to the mine opening. The energy provided by the pump and the static head in the borehole give the velocity required to keep the solid particles in suspension and to transport them.

The completeness of filling in the open spaces is responsive to changes in the velocity of flow, which changes with the growth of the mound of deposited solids. As the slurry first enters the open space from the injection hole, its velocity drops rapidly, and solid particles settle out near the borehole, forming a doughnut-shaped mound on the mine floor. As the height of the mound approaches the mine roof, the velocity of the slurry increases through the narrowing channel, and solid particles are transported to the outer limit of the mound. Here the velocity decreases abruptly and solids are deposited. This type of deposition continues and the mound of deposited fill builds outward. Stages in the filling of a mine void are shown diagrammatically in figure 6.

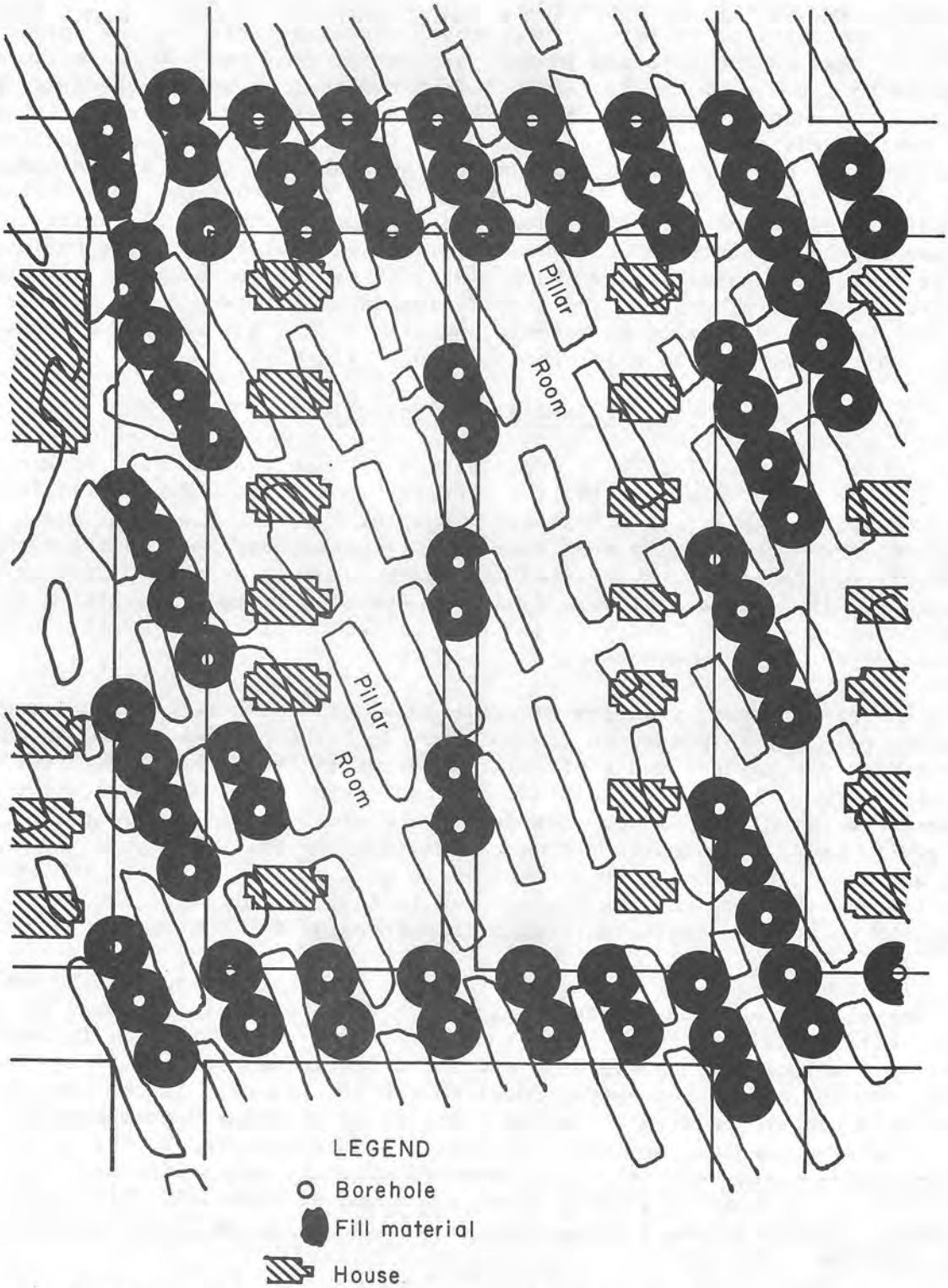

FIGURE 5. - Typical residential block showing the pattern of boreholes in streets, alleys, and vacant lots proposed for blind flushing. Circular areas around boreholes represent backfill material to be placed in the mine voids.

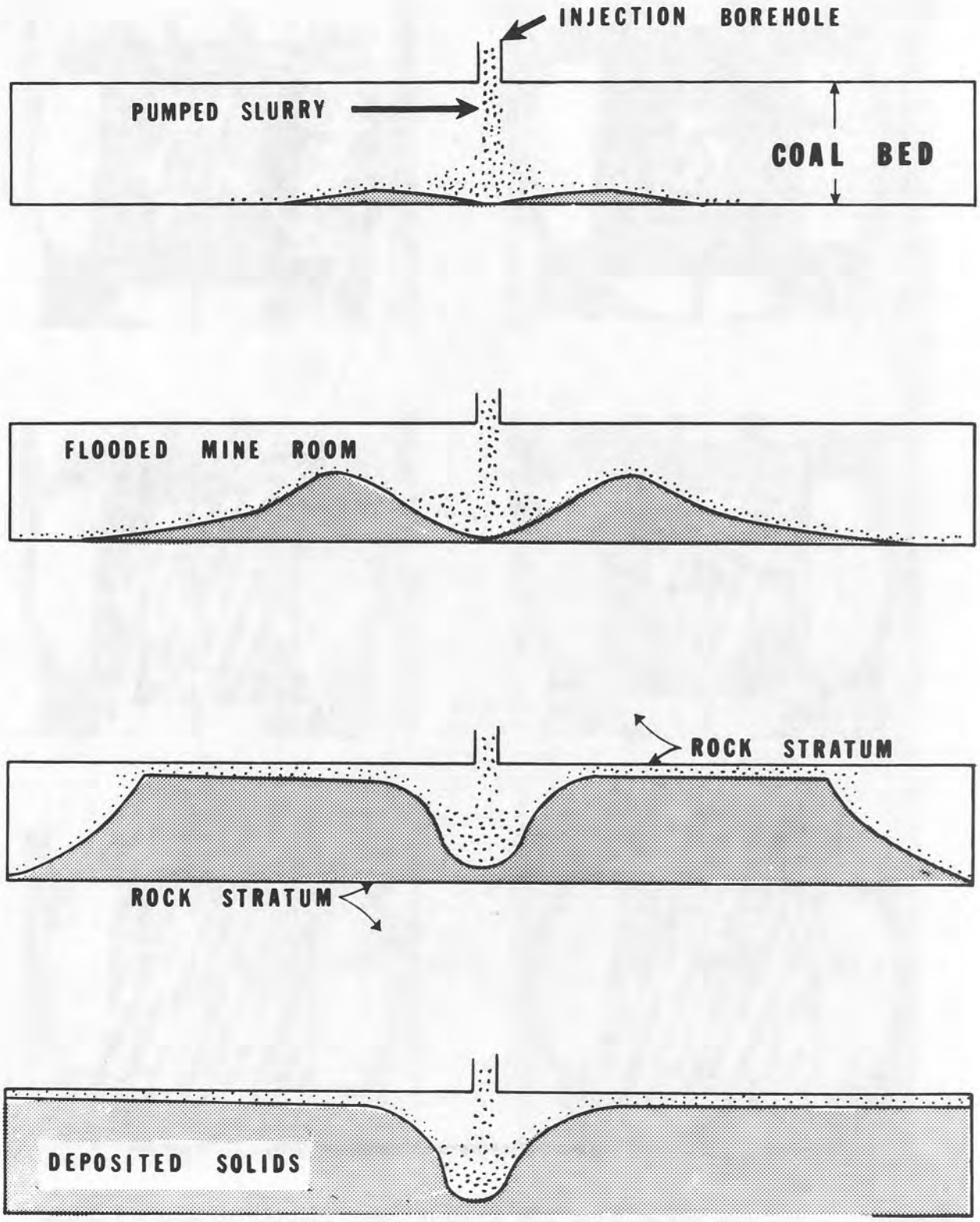

FIGURE 6. - Sectional views through a flooded mine room at the point of slurry injection showing movement of particles and growth of deposit.

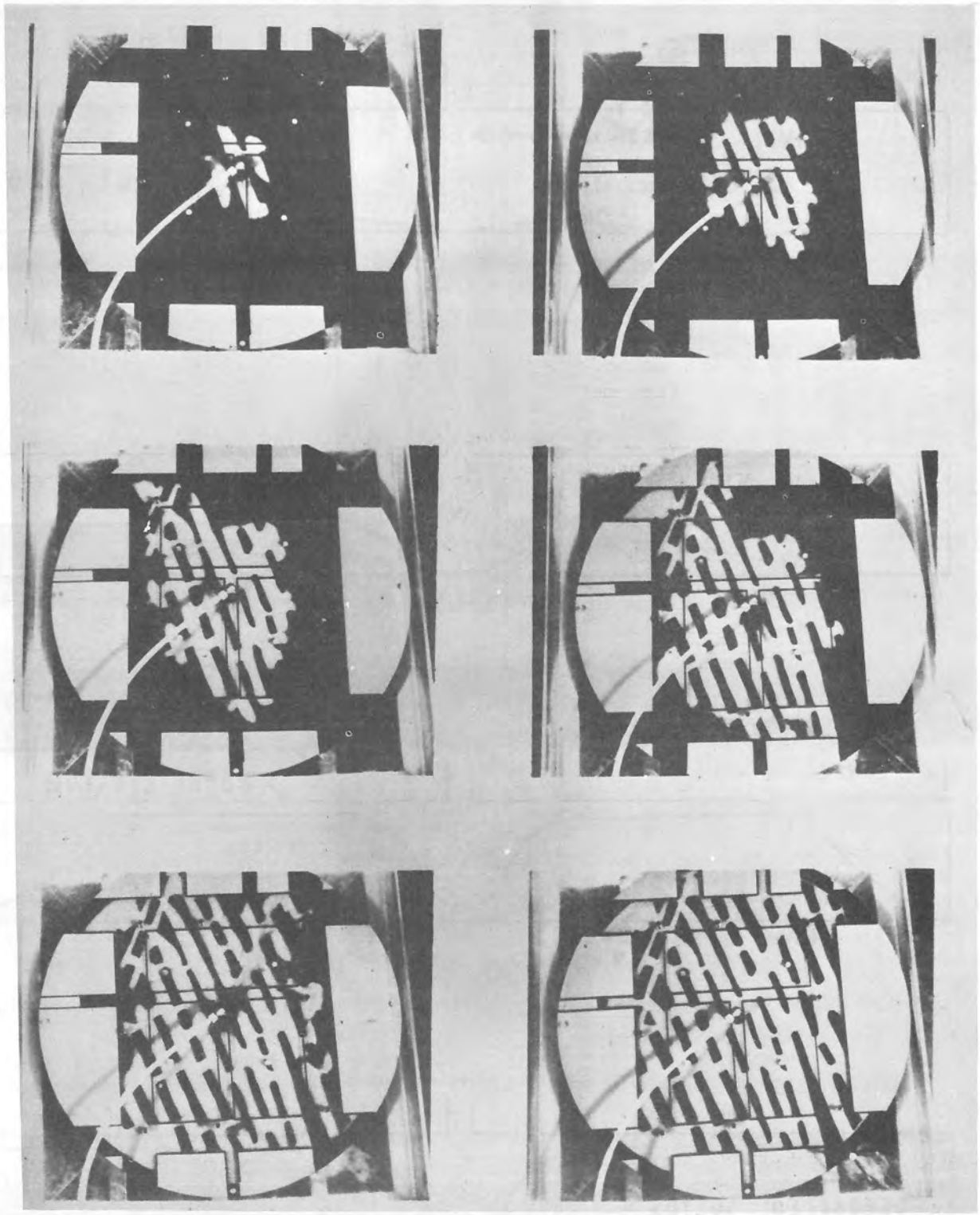

FIGURE 7. - Top view of mine model with transparent roof, showing stages of radial distribution of fill material by the pumped-slurry process.
(Courtesy of the Dowell Division of the Dow Chemical Co.)

In a table-size mine model simulating the arrangement of rooms and pillars, deposition of fill material is shown in figure 7. As resistance to flow of the slurry develops in one direction, a new channel is formed in another direction along a line of less resistance. Eventually, nearly all mine openings are filled. The lateral extent of the fill is determined largely by the available energy in the system. As the mound of fill material builds outward in the mine, the flow channels between the mound and the mine roof become longer and resistance to flow increases. When this resistance, combined with resistance in the pipe, becomes great enough to reduce the velocity of the slurry below that required to transport the solid particles, transportation of the particles ceases. The particles then settle out, and the passage becomes filled.

The pumped-slurry method of blind flushing has the following advantages over the open gravity-feed method previously used:

1. Great reduction in the number of injection boreholes. A single injection hole serves the purpose of many injection holes in the gravity-feed method.

2. More complete vertical filling of mine openings.

3. More complete areal coverage. Areas inaccessible because of surface improvements can be filled.

4. Less disruption of the community in the form of noise, dust, and traffic interference by drilling operations and trucking of fill material.

Disadvantages to the use of the pumped-slurry method would be apparent under the following conditions:

1. Where it is safe for men to enter the mine and control the backfilling operation.

2. Where the mine voids to be filled are close to the surface with little or no rock strata cover, in which case the slurry water could invade surface areas.

3. Where the amount of mine void to be filled is too small to warrant the cost of acquiring and installing slurry pumping equipment.

PUMPED-SLURRY TEST AT ROCK SPRINGS

The test at Rock Springs in 1970 was carried out in cooperation with the city of Rock Springs, the Department of Housing and Urban Development, and the Bureau of Mines. The Bureau's financial participation in the project to the extent of $55,000 was covered by a contribution contract with the city. The city of Rock Springs entered into a contract with Dowell Division of the Dow Chemical Co. of Tulsa, Okla., to perform the injection operation. The total contract cost of the demonstration was $173,140.

A site adjacent to the Connecticut Avenue-D Street area was selected for the test because it was believed to have a high subsidence potential. Exploratory drilling in conjunction with sonar scanning indicated the mine workings under the site were flooded and included both collapsed areas and open spaces in the coalbed. Open passages usable both for water supply and receiving a large quantity of fill material were identified by the detection equipment.

Injection Operation

The objective of the Rock Springs test project was to place 20,000 cubic yards of sand in underground voids through a single injection hole. This should constitute a convincing demonstration inasmuch as the quantity of fill that could be injected under the existing method of blind flushing was estimated at an average of 100 cubic yards per hole.

It was estimated that 20,000 cubic yards of sand would fill mine voids within an average radius of 210 feet from the injection hole, based on an average 6-foot thickness of the coalbed and 65-percent extraction of the coal. The corresponding surface area overlying the mine openings to be backfilled was calculated to be 3.2 acres.

The material used for backfill was fine sand of wind-blown origin available from a nearby deposit. The sand was screened at the pit to reject particles larger than one-fourth inch in diameter, and pieces of debris. The sand was transported by truck to the injection site and stockpiled.

Water to form the slurry was obtained from the flooded mine by means of two wells located about 325 feet downdip from the injection well. The mine water contained about 13,500 ppm dissolved solids but had no disagreeable odor and was not highly corrosive (3). Two submersible water pumps, each with

FIGURE 8. - Photograph of the plant site at the first test of the pumped-slurry mine backfilling technique.

capacity of 4,000 gpm, pumped water to the mixing tank. A reserve supply of water for purging the system was maintained in four storage tanks adjacent to the mixing tank. Figure 8 is a view of the plant site, and figure 9 shows the equipment installation in relation to streets.

Water entered the mixing tank at an average rate of 5,500 gpm and mixed with the sand, which was supplied at the rate of 120 cubic yards per hour, to form a slurry. A slurry pump impelled the slurry through the 13-3/8-inch-ID injection pipeline at an average velocity of 17 fps. The injection borehole was cased with 13-3/8-inch-ID pipe to within 5 feet of the mine roof, which was 116 feet below the surface.

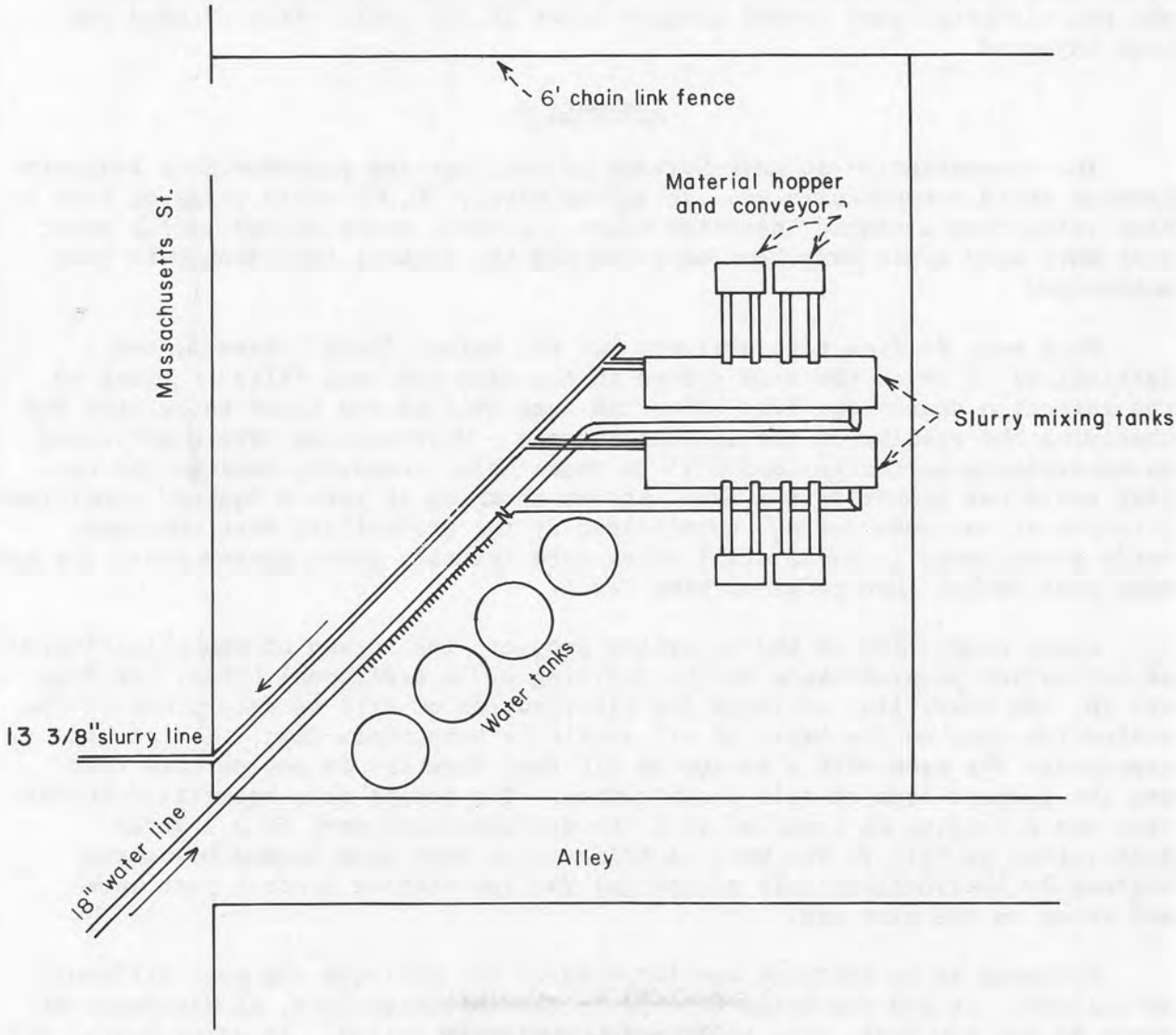

FIGURE 9. - Plan view of installed equipment at the plant site and adjoining streets.

The sand slurry was successfully injected into the mine workings over a 10-day period. Operations were scheduled for 24 hours per day, but actual daily injection time ranged up to a maximum of 21 hours because of electrical and mechanical problems. The solids concentration in the slurry averaged 0.8 pound per gallon of water. No resistance to injection was encountered, and the pressure measurements made at the top of the injection borehole were below atmospheric (vacuum) throughout most of the period.

During the last 12 hours of injection, it was planned to increase the sand concentration and decrease the discharge rate in order to achieve complete filling up to the mine roof. When the flow rate decreased to about 3,000 gpm, however, the sand in the mixing tank was not kept in suspension and the discharge pump became plugged after 19,500 cubic yards of sand had been injected.

Evaluation

The demonstration at Rock Springs proved that the pumped-slurry injection process could successfully emplace approximately 20,000 cubic yards of sand in mine voids from a single injection hole. In fact, there seemed little doubt that more sand could have been emplaced had the initial injection rate been maintained.

Much more difficult to evaluate was the extent (both laterally and vertically) to which the open spaces in the mine had been filled. Prior to the injection operation, four holes had been drilled and cased to be used for observing the results of the filling process. Difficulties were experienced in determining depths to sand fill in these holes, however, because the sampler could not penetrate the sand without churning it into a "quick" condition. Interpretations were further complicated by the probability that the open voids encountered in these drill holes were actually caved spaces above the mine roof rather than rooms at mine level.

After completion of the injection project, the Bureau of Mines instituted an evaluation program based on the drilling of 36 additional holes. In figure 10, the heavy line encloses the distribution of fill as determined by the evaluation team on the basis of all available subsurface data. The circle represents the area with a radius of 210 feet from the injection hole that was the planned area of fill distribution. The actual area backfilled in the mine was 2.8 acres as compared with the predetermined area of 3.2 acres. Restriction of fill to the west is believed to have been caused by human-engineered obstructions (air stoppings) for ventilation control that were not shown on the mine map.

Evidence as to vertical completeness of the fill was the most difficult to collect. In the two holes closest to the injection hole, at distances of about 60 and 100 feet, mine voids were completely filled. In other holes, the degree of vertical filling was hard to interpret, in part because the original mine roof had caved, extending the opening upward from 7 to 10 feet, and the caved material formed mounds on the original mine floor. In some drill holes, sand was found interbedded with caved rubble; in others, the sand was in the

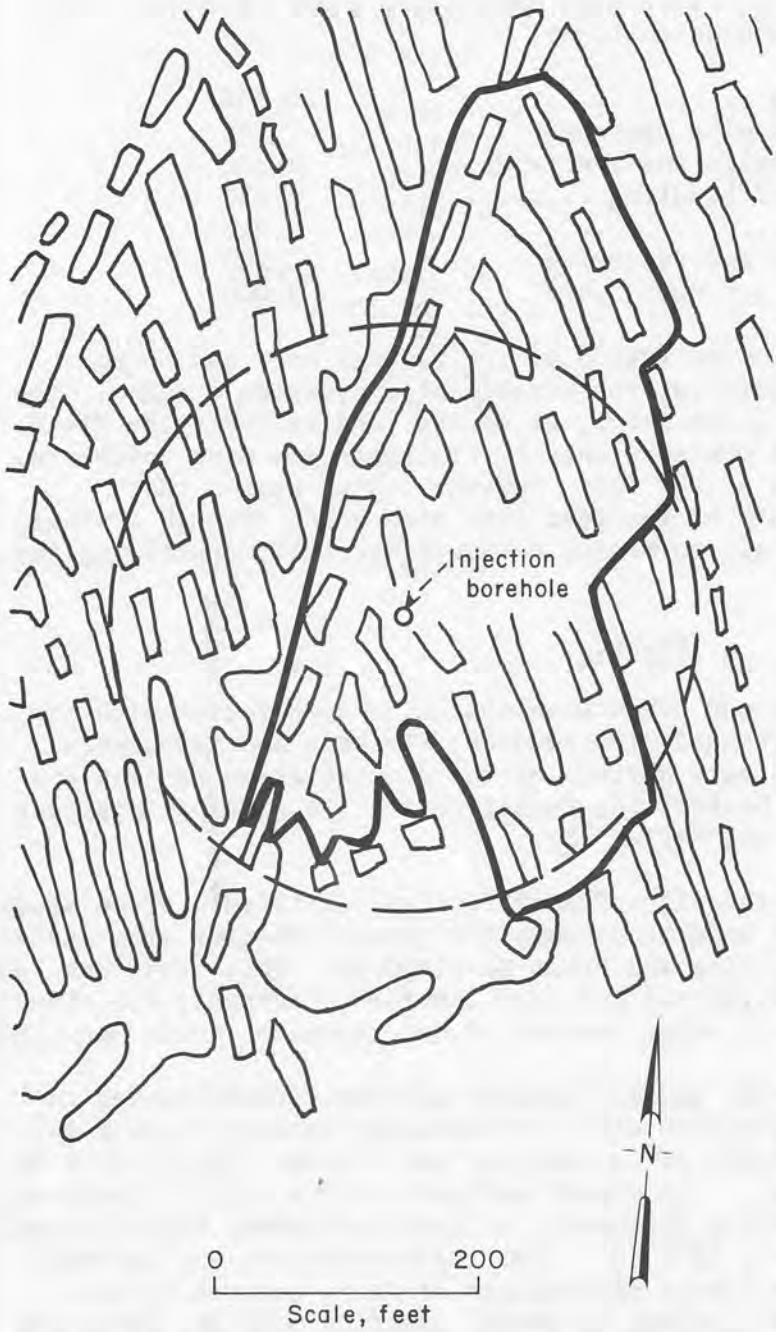

FIGURE 10. - Map of mine workings at the site of the first demonstration project in Rock Springs, Wyo., showing pattern of sand deposition in the mine voids. Dashed circle shows area of planned backfill.

caved space above the rubble. In a few holes, sand was blown up the hole when the drill encountered a cavity, indicating air pockets trapped at the top of the caved spaces. In a hydraulic filling operation, such air-filled voids would remain unfilled.

The relative density of the fill was found to decrease from the injection point. Tests by the Wyoming Highway Department indicated average in-place density of about 127 pounds per cubic foot with relative density values ranging from 36 to 81 percent. This range included mixtures of sand and shale rubble (14).

The degree of subsidence control effected by this project is difficult to evaluate. The emplaced fill in such a relatively small area of the mine would not be expected to completely prevent subsidence throughout the area because of apparent decreasing height and density of the fill away from the injection hole. Any further subsidence that might take place in peripheral areas, however, would tend to be somewhat less than in comparable areas that had not been backfilled. The process was considered to hold sufficient promise to justify further experimentation.

Cost

The cost of the demonstration project, including

all the extras associated with a first-time application, was $9.00 per cubic yard ($6.75 per ton) broken down as follows:

Project planning.....	$0.158
Investigation of mine openings.....	.542
Preparation of wells and manifolds.....	2.350
Fill material and handling.....	2.832
Injection.....	2.818
Site restoration and reporting.....	.300
Total cost per cubic yard.....	9.000

These costs reflect the demonstration of the process only and do not include the preceding site studies or the subsequent evaluation program. The cost per acre, about $62,500, is excessive, of course, and reflects the disadvantage of applying the method wherever only a relatively few mine voids are to be backfilled. The success of the test, however, which proved that 19,500 cubic yards of sand could be inserted into mine voids through a single borehole, promised invaluable aid to mining communities, fully justifying the cost of the experiment.

Reports

The Department of Housing and Urban Development, in cooperation with the city of Rock Springs, Wyo., arranged with Candeub, Fleissig and Associates, Consultants, Newark, N. J., to make a study of the demonstration project and a comprehensive report on the backfilling technique and its possible application in other mining areas of the Nation (3).

After the demonstration, the city of Rock Springs initiated a broad study of its overall community needs under the Community Renewal Program with assistance from the Department of Housing and Urban Development. This study has led to the establishment of priorities not only for subsidence control, but also for the treatment and renewal of other sources of deterioration and blight (19).

A comprehensive study of the general geology and underground mining in the Rock Springs area, in conjunction with the Community Renewal Program for the city, was made for the purpose of determining the economic feasibility of backfilling the mined-out areas. The report was made by the firm of Johnson-Fermelia & Crank, Inc., Consulting Engineers and Land Surveyors, Rock Springs, Wyo., and was completed March 1, 1972 (9). Two recommendations are included in the report: (1) That an extensive exploratory drilling program be conducted in various parts of Rock Springs to verify existing mine map data; and (2) that Federal funding be obtained for a large-scale mine backfilling project under the Belmont-Kerback area to assess the effectiveness of the process in arresting the continuing subsidence in that area.

FIRST LARGE-SCALE TEST

Scranton, Pa., was selected by the Bureau as the locale for the full-scale pumped-slurry demonstration project in 1972-73 because of its subsidence history and the active local interest in subsidence control. Population

centers in the Anthracite region of northeastern Pennsylvania have had a history of subsidence problems as a result of multiple-bed mining over a period of 150 years. Scranton is the largest of the cities in the Anthracite region. As many as 11 different coalbeds have been mined under Scranton, and most of the central part of the city overlies 6 mined beds. The demonstration project site is in an area of potential subsidence due to past mining in five abandoned coalbeds. Subsidence had not yet become apparent at the ground surface in this area, and caving below ground was not believed to be sufficient to block effective movement of slurry.

A 30-acre residential area was stabilized by injecting about 450,000 cubic yards of crushed mine refuse into two coalbeds through five injection boreholes. Nearly 200,000 cubic yards was injected through one borehole from which the material moved into the mine workings on all sides; the injected material reached a maximum lateral distance of 640 feet and filled mine openings from floor to roof. The new method, designed for inundated mines, proved successful also in mine workings above mine-water pool level (22).

HYDRAULIC MODEL STUDIES FOR BACKFILLING MINE CAVITIES

In October 1973, the Bureau of Mines arranged for model studies of the pumped-slurry method of backfilling mine cavities to be conducted by the U.S. Bureau of Reclamation at their hydraulic laboratory in Denver, Colo. The purpose of the studies was to obtain qualitative and quantitative data on the deposit pattern of fine sand, such as that at the Rock Springs projects, when used for backfill material and injected into cavities of a simulated coal mine. The results of the first series of 18 tests are given in a report entitled, "Hydraulic Model Studies for Backfilling Mine Cavities," by E. J. Carlson of the U.S. Bureau of Reclamation, which is included as an appendix in Bureau of Mines Information Circular 8667 (22).

A second series of model studies of the pumped-slurry method using the same fill material and simulating conditions not covered in the first series was made by the Bureau of Reclamation for the Bureau of Mines in 1974. That report was also by E. J. Carlson, U.S. Bureau of Reclamation; dated March 1975, it is included as an appendix to this report.

THREE LARGE-SCALE PROJECTS AT ROCK SPRINGS

The success achieved with the pumped-slurry technique at Rock Springs and at Scranton, where for the first time extensive mine voids in two beds under about 30 acres were filled from only five injection boreholes, prompted the Bureau to initiate several more large-scale subsidence control demonstration projects in both the Anthracite region and Rock Springs. Experiments were needed to determine the various types and sizes of material as well as the optimum range of conditions under which the pumped-slurry method could be used more effectively in minimizing adverse environmental problems associated with underground coal mining. In addition to the original 20,000-cubic-yard test, this report covers the three large-scale demonstration projects that were conducted in Rock Springs during the period 1973-76.

It became apparent, however, that the Bureau's proposed subsidence control efforts at Rock Springs would proceed in an atmosphere of unprecedented industrial expansion. Areas near the city saw the construction of a $500 million coal-fired steam-electric powerplant; development and expansion of trona operations by four chemical companies; oil and gas exploration; and stepped-up activities relating to the extraction of coal and uranium. The population of Rock Springs rose from a more or less stable 12,000 in 1970 to 26,000 in January 1974. In addition to subsidence problems, therefore, the city was obliged to cope with crowded schools, traffic jams, lack of housing, rising crime rates, overloaded public facilities, and insufficient tax revenues.

Under the authority of the Organic Act, the Bureau of Mines entered into a cooperative agreement with the city of Rock Springs to conduct large-scale demonstrations of the pumped-slurry method that would also provide surface support for the subsidence-prone critical areas of the city. The agreement provided that the Bureau design, direct, and supply the funds to accomplish the work by contracts obtained through Federal rules and regulations governing competitive procurements. The Bureau, through its contractors, was required to conduct all phases of the work with a minimum of inconvenience to residents and with as little adverse effect upon the environment as possible. The city was required to grant the Bureau, its officials, contractors, or employees, the right to enter streets, roads, and any other land owned or controlled by the city overlying or adjacent to the project areas for the purpose of conducting the injection operations. The city also furnished without charge all licenses, permits, or other authorizations required in the conduct of project work, which included detours for local traffic and police surveillance of Bureau property. City engineers and utility personnel also provided accurate information regarding locations of utility pipelines, wires, cables, and sewer lines together with maps showing ownership of property within and adjacent to the project areas.

Drilling Operations

All drilling and reaming of boreholes that were used for water wells, slurry injection, exploration, and monitoring was done with rotary drills equipped with efficient dust control mechanisms. While drilling was done with air, insofar as possible, water (mist) injection was also used to reduce the amount of dust discharged into the atmosphere. After drilling through unconsolidated overburden, all boreholes that were used for water supply and injection were cased and cemented to a point 4 feet into the rock stratum. From here drilling was continued at sufficient diameter to accommodate the required casing to the mine opening. Such boreholes were collared, sealed to the standpipe, and fitted with removable pressure-tight caps at the surface. For the monitoring boreholes, casing was not installed below the point 4 feet into the rock stratum. Logs of all the boreholes, and particularly the depth at which the bit broke through to the mine opening as well as the height of the mine opening, were furnished to the Bureau of Mines. Figures 11 and 12 are typical cross sections of the various boreholes used in the projects.

Boreholes that penetrated into solid coal and therefore not usable were filled with drill cuttings to within 3 feet of the top of solid rock, then

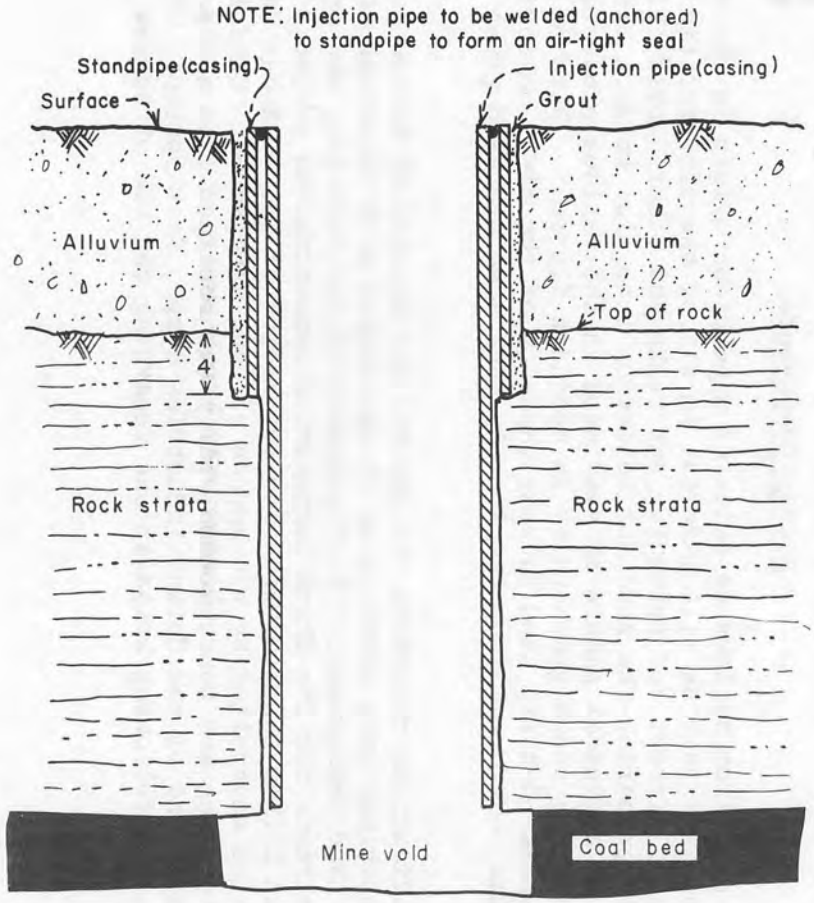

FIGURE 11. - Typical sections of water supply and injection boreholes.

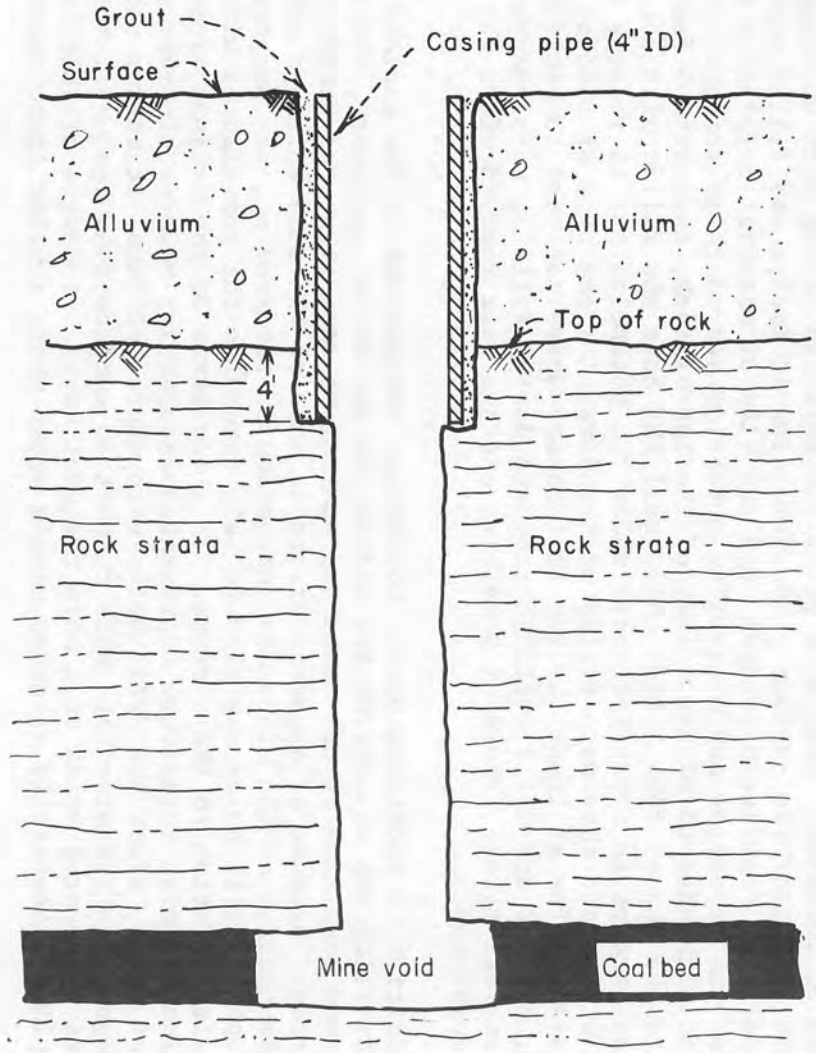

FIGURE 12. - Typical section of a monitoring borehole.

with 3 feet of concrete. From this point the hole was filled with cuttings or soil within 1 foot of the surface. The remaining 12 inches was filled with dirt, asphalt, or concrete depending upon the kind of material originally present. After the injection and monitoring boreholes served their purpose, they were filled with sand to within 4 feet of the top of rock, then with 4 feet of concrete to the top of rock. From this level the hole was filled with sand to the surface, except in those instances where the boreholes are in roadway surfaces, in which case they were filled with concrete or the type of roadway material present for at least the top 12 inches. Where not used to backfill boreholes, the mud and drill cuttings produced during drilling were removed from the various sites. Wherever possible, casings were removed after the work was completed.

In addition to providing usable boreholes, the purpose of the exploratory borehole drilling was to outline and define, to the extent necessary, the size, location, and condition of the abandoned coal mines underlying the proposed project areas. Because of apparent errors in maps of the coal workings and the inaccessibility of the mines for inspection or resurveying, the purpose of the proposed drill holes was also to help establish the position of mine workings with relation to the surface. In a few parts of the project areas the locations of mine voids were represented accurately; however, in most of the project areas, after the drill holes encountered pillars, it became necessary to reposition the mine maps with respect to surface features until the drill holes penetrated voids on a more frequent basis. A device of this kind may be applied successfully in other mining areas where similar inaccuracies exist.

Slurry Components

Fine sand from various deposits of wind-blown origin in the vicinity of Rock Springs was used as fill material in all of the demonstration projects at Rock Springs. Except for screening out rocks and other debris, little else was done in preparing the sand for injection into mine voids. The grain-size gradation of a typical sample of sand used in the various projects and in the model studies is shown graphically in appendix figure 3. Data on settlement tests, in-place density, and bearing capacity of the fine, uniform, blow sand (median size, ~ 0.14 mm) from the Rock Springs area are also given in the appendix.

Specifications regarding the use and reclamation of borrow areas where sand was obtained were established in accordance with requirements of the Bureau of Land Management, U.S. Department of the Interior, and incorporated in the contracts for the three large-scale demonstration projects. Sandfill for Project 1 (Dow Chemical Co.) and for Project 3 (M. J. Bober Co.) was taken from deposits controlled by the Bureau of Land Management. Sand for Project 2 (WHAN Engineering and Construction, Inc.) was obtained from a deposit on privately owned land of the Upland Industries Corp., Rock Springs, Wyo. In all instances, the following stipulations regarding the use of borrow pits applied:

1. Topsoil, if present, will be removed and stockpiled.

2. At such time when materials (blow sand) has been removed and rehabilitation is underway, the floor of the pit shall be leveled to coincide with the existing watershed slope. If no other use is made of the screening rejects they will be scattered over the pit floor. Topsoil will be replaced. Pit slopes will be 3:1 ratio. Haulage roads will be leveled and berms smoothed. Haulage roads and materials pit will be seeded with crested wheat at 5 lb per acre and thickspike wheatgrass at 3 lb per acre. Seeding will be accomplished by using a grass seed or grain drill with grass seed attachment. Seeding shall be at 1/2-inch depth and will continue until a satisfactory stand is established and approved by the Bureau's representative.

3. Seeding will be done in the fall after September 15 and before freeze-up.

4. Leveling, grading, and seeding shall be completed within 1 year after the completion or termination of any operation hereunder.

5. Road crossings shall not impede drainage; either culverts should be installed or approaches cut down to drainage bottom level.

6. In the event an archeological site is unearthed, the Bureau of Mines shall be notified immediately.

7. Signs shall be properly posted to warn the public of the extraction area.

8. Proper precaution will be taken at all times to prevent and suppress fires. The contractor will be held responsible for suppression costs for the fires caused through negligence of his employees or subcontractors.

9. Should the material be slurried at the site, water shall be controlled.

10. Upon termination of the pit, the contractor shall submit a final accounting of the total material removed.

The source of water, which is needed in large quantities for the pumped-slurry injection technique, was the local mine-water pool beneath the test site and beneath the three large-scale projects. Advantages of using mine water include its availability, low cost, and the probability that withdrawal and injection of large quantities of water from, and into, the same body of water would create minimum disturbance of the subsurface equilibrium. The mine water contained about 13,500 parts per million (ppm) dissolved solids but had no disagreeable odor and was not highly corrosive (9).

Probe boreholes that intersected the mine-water pool at required pump drawdown depths at or near the various project sites were enlarged to accommodate large casings in which deep-well or submersible pumps were installed. Specifications for the first and second large-scale projects required delivery

of a minimum of 4,000 gallons per minute (gpm) to the mixing tank. This was increased to a minimum of 5,000 gpm for the third large-scale project to allow for an increased volume of slurry injection. Data on solids, water, and slurry covering the range of quantities utilized at the various projects are given in table 1.

TABLE 1. - Data on solids, water, and slurry

Slurry water to solids ratios	Water pumped				
	4,000 gpm (16.66 tons/min)	5,000 gpm (20.82 tons/min)	6,000 gpm (24.99 tons/min)	7,000 gpm (29.15 tons/min)	8,000 gpm (33.32 tons/min)
SOLIDS INJECTED: 200 TONS/HR, 3.33 TONS/MIN, 307 GPM					
Weight.....	5.0	6.3	7.5	8.7	10.0
Volume.....	13.0	16.3	19.5	22.8	26.0
SOLIDS INJECTED: 350 TONS/HR, 5.83 TONS/MIN, 538 GPM					
Weight.....	2.9	3.6	4.3	5.0	5.7
Volume.....	7.4	9.3	11.2	13.0	14.9
SOLIDS INJECTED: 450 TONS/HR, 7.50 TONS/MIN, 692 GPM					
Weight.....	2.2	2.8	3.3	3.9	4.4
Volume.....	5.8	7.2	8.7	10.1	11.6

Sand:

Bulk density 100 pcf (loose) 38.3 pct voids; 61.7 pct solids
2.6 (specific gravity) × 62.4 lb = 162.2 pcf (solid)
162.2 ÷ 7.48 gal/ft^3 = 21.68 lb/gal (solid)

Water:

4,000 gpm × 8.33 lb/gal = 33,320 lb/min = 16.66 tons/min
5,000 gpm × 8.33 lb/gal = 41,650 lb/min = 20.82 tons/min
6,000 gpm × 8.33 lb/gal = 49,980 lb/min = 24.99 tons/min
7,000 gpm × 8.33 lb/gal = 58,310 lb/min = 29.15 tons/min
8,000 gpm × 8.33 lb/gal = 66,640 lb/min = 33.32 tons/min

Injection rate of solids:

200 tons/hr = 3.33 tons/min = 307 gal/min
350 tons/hr = 5.83 tons/min = 538 gal/min
450 tons/hr = 7.50 tons/min = 692 gal/min

Injection rate of slurry by volume (two examples):

Water	+	Solids	=	Slurry
5,000 gpm		3.33 tons/min		5,307 gpm
7,000 gpm		5.83 tons/min		7,538 gpm

Mixing Plants and Slurry Pumps

Equipment at a typical mixing plant site at Rock Springs included a hopper, variable-rate feeder, belt weighing scale, mixing tank, diesel-powered slurry pump, piping and valves, and other auxiliary apparatus necessary to operate, control, maintain, and monitor the backfilling operation. Each

mixing tank was constructed and arranged in such a manner that it was capable of receiving measured and weighed sand at a rate of at least 1,600 tons per shift and mine water ranging from 3,000 to 6,000 gpm from electric-powered deep-well pumps. Mixing tanks were fitted with manifolds connecting the incoming water supply with a series of nozzles, producing sufficient turbulence to maintain the solids in suspension. In each tank the suction line was fitted with a snorepiece that picked up the suspended material (slurry) and conveyed it to the impellers in the slurry pumps. From here the slurry was pumped through distribution lines to the various injection boreholes.

In each case all operations involved in the mixing, pumping, and injection processes were controlled from a conveniently located instrument panel. This included the rate of waterflow, the rate at which sand was introduced, the amount of slurry discharged, cumulative weight of solids injected, and the pressure both at the slurry pump and injection borehole.

Because of the experimental nature of the work and varying conditions to be encountered at the three projects, the specifications allowed a certain amount of flexibility in conducting the work. Each contractor and/or his engineer was encouraged to select and operate the equipment so as to discover the optimum range of such variables as volume, velocity, and density of slurry flow in relation to changes in pump head, pressure at the injection borehole, length of discharge line, and depth to mine voids.

PROJECT 1

Subsequent to the completion of the first successful large-scale application of the pumped-slurry mine backfilling process at Scranton, Pa., Congress in fiscal year 1973 appropriated $700,000 for the Bureau to conduct a similar large-scale demonstration at Rock Springs, Wyo.

In designing the project, the Bureau selected critical areas 1, 7, and 8 because the estimated cost of the work and material to fill the mine voids would approximate available funds. Moreover, it was hoped that providing support in the mine workings under area 1 might arrest the continuing subsidence occurrences in and near the Connecticut Avenue-D Street section. The three critical areas were located in the southeast portion of Rock Springs above both flooded and dry mine voids in the No. 7 seam of the abandoned No. 2 Mine, Central Coal and Coke Co. The old mine workings were believed to be under less than 300 feet of overburden, and this was to be the first time that sand fill would be used in the pumped-slurry process to fill both inundated and dry coal mine voids. It was proposed to demonstrate that approximately 150,000 cubic yards (200,000 tons) of screened sand could be introduced into the designated mine voids under critical areas 1, 7, and 8, a total area of about 33 acres (see fig. 13).

Prior to preparing specifications for the backfilling project, exploratory drilling was conducted in the three critical areas. It was necessary to know the nature and depth of overburden; condition of the mine workings; and whether or not the available maps of the mine workings show a true relationship with overlying surface features. The Bureau, therefore, through Federal

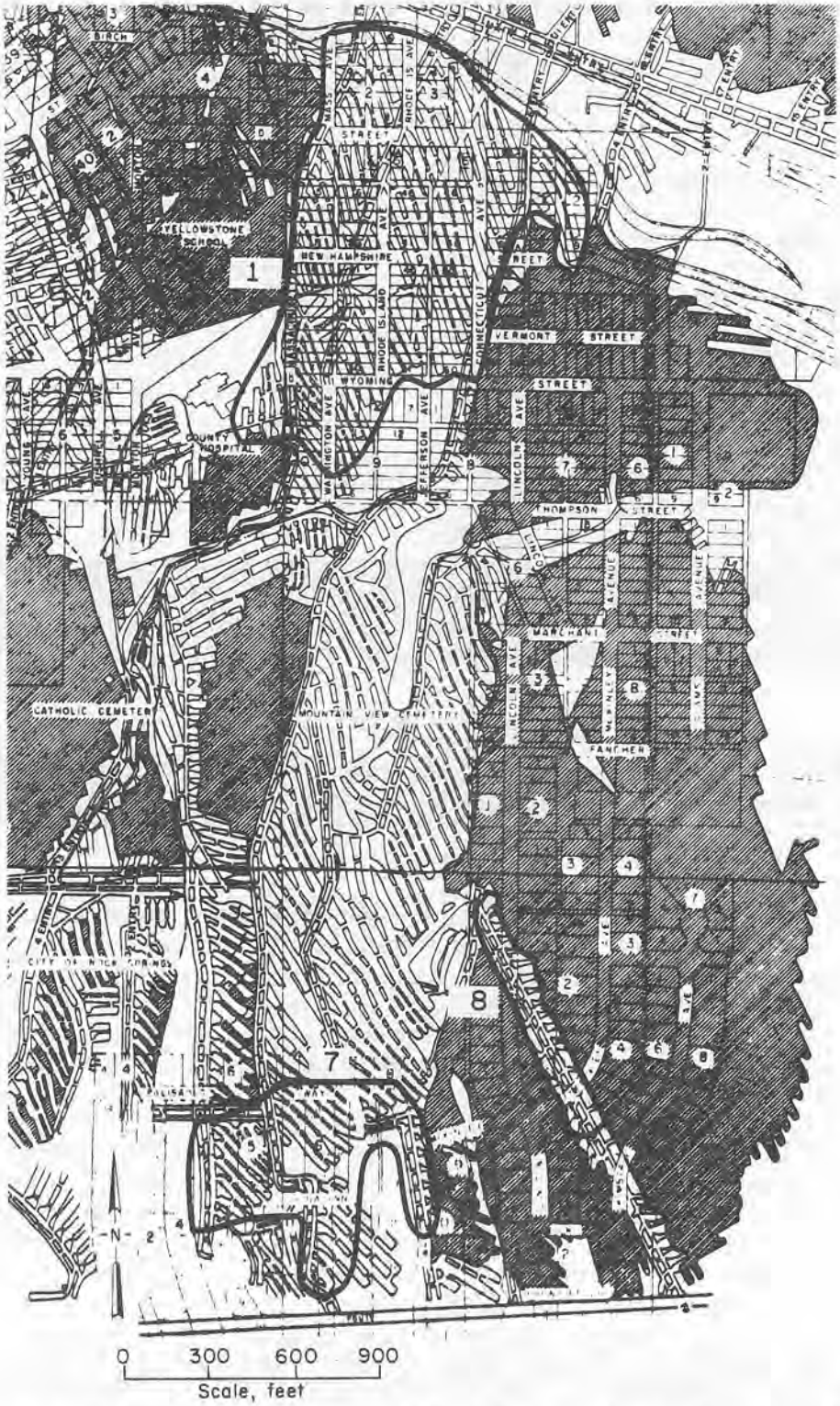

FIGURE 13. - Map of a portion of Rock Springs showing locations of critical areas 1, 7, and 8 in relation to mine workings of the No. 7 Seam.

Government competitive contracting and bidding procedures, awarded a drilling contract on January 26, 1973, to the Delta Drilling and Development Co., Dana, Ind., in the amount of $39,636.00. Forty usable boreholes, large enough to accommodate 4-inch-ID plastic casing, were eventually drilled in the project areas to provide the necessary information. Depths to the mine voids ranged from 85 to 272 feet, and alluvium ranged between 10 and 45 feet. Actual cost of the work was $37,707.02. Under the ensuing backfilling contract, eight of the boreholes were reamed to larger sizes to provide water-supply wells and injection boreholes. The remaining 32 boreholes were used to monitor the backfilling progress.

As in the earlier test project, it was believed that while the exploratory drilling provided approximate vertical depth of the void at each borehole, the lateral extent of the voids was not apparent. The Bureau, therefore, employed the Dowell Division of the Dow Chemical Co. to explore the extent of mine passageways between boreholes with "Sonar Caliper" surveys. Eleven boreholes were probed with the instrument, which confirmed to some extent the continuity of the voids. The cost of the work was $5,135.15.

Through competitive contracting procedures, the Bureau, on June 4, 1973, awarded a pumped-slurry backfilling contract to the Dowell Division of the Dow Chemical Co. in the amount of $721,777.50 to provide surface support under areas 1, 7, and 8. The contract was modified on June 29, 1973, to allow for the drilling, casing, and cementing of a necessary alternate injection borehole, increasing the total contract amount by $7,924.00 to a new total of $729,701.50.

During the mobilization and installation of equipment, the contractor elected to locate the injection plant at the borrow site rather than use trucks to haul the sand about 1-1/2 miles to the Bureau's suggested plant site location. This of course involved pumping the slurry directly from the borrow site through a network of pipelines an average distance of 2.1 miles to the injection boreholes. Other work included under mobilization involved preparation of the borrow pit, installation of the water-supply system, erection of the injection plant facilities, and construction of the 14-inch-ID slurry pipeline. At this time, the injection boreholes were also provided and suitably capped to await being used. The slurry injection was started August 24, 1973.

Two submersible water pumps were installed, one in each of the two water wells just north of critical area 1. The pumps were 137 feet below the surface and about 35 feet below the average mine-water pool level. The pumps were rated at 4,000 gpm each, at a dynamic head of 150 feet, and were driven by 200-hp electric motors. The water was pumped from the pool in the No. 7 Seam and impelled with the aid of a 1,200-hp, diesel-powered booster pump through 1-1/2 miles of 14-inch-ID steel pipeline to the mixing plant at the borrow site. Because of the friction head in the relatively long pipeline, only an average of 3,600 gpm was delivered to the mixing plant.

The sandfill material was obtained from a nearby deposit on Federal land controlled by the Bureau of Land Management (BLM). This designated site was

made available to the contractor through a "Free Use Permit" between the BLM and the Bureau of Mines. Removal of the fill material and subsequent reclamation of the borrow site was conducted in accordance with the stipulations required by the BLM, which were made part of the demonstration project specifications by an amendment. Adequate screening equipment was installed to reject rock particles larger than one-half inch as well as other foreign debris.

At the borrow pit sand was loaded by bulldozers and scrapers into a hopper and fed through a reciprocating plate onto a conveyor, then weighed as it was being delivered to the steel mixing tank. Water from the submersible pumps entered the tank through a manifold and a series of nozzles producing sufficient turbulence to maintain the solids in suspension. The slurry pump, of high-chrome cast iron, was a 1,200-hp, diesel-powered, centrifugal dredge service pump, capable of pumping slurry at a rate of 8,400 gpm at a head of 175 feet. Figure 14 is a view of the slurry injection plant at the site of the borrow pit. Although the slurry was transported slightly down-grade, the friction head that developed in the long pipelines reduced the volume of slurry delivered to the injection holes to an average of about 4,000 gpm.

The slurry distribution pipelines were buried beneath the busier streets and intersections to minimize the impact on traffic. On less-traveled streets they were laid along curbs where arrangements with property owners permitted construction of ramps over pipelines at entrances to driveways. The locations of the water-supply wells, the water-supply pipeline complete with booster pump, the borrow pit, injection plant, housing for recording and control equipment, slurry distribution pipelines, and injection boreholes are shown in figure 15.

FIGURE 14. - View of the slurry mixing plant installation at the site of the borrow pit about 2 miles from the built-up areas of the city. Note natural rock outcropping in left background off project limits and slurry pipeline in left foreground.

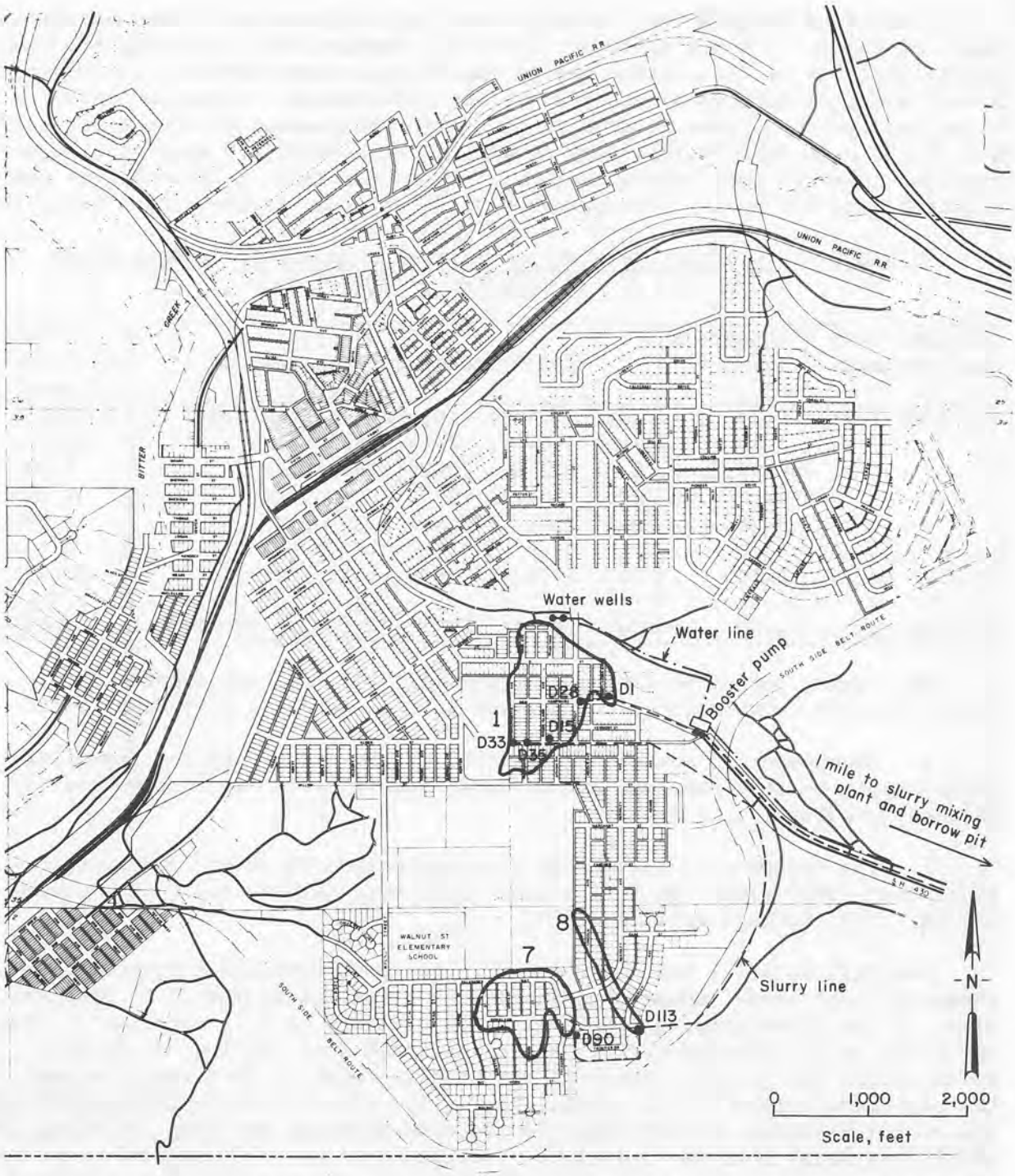

FIGURE 15. - Map showing location of water-supply wells and pipeline, slurry pipelines, and injection boreholes in relation to critical areas 1, 7, and 8. (The distance to the injection plant at borrow pit is indicated.)

A total of 152,467 tons of solids was injected into the three separate areas of the No. 7 mined coalbed. The total surface area overlying the backfilled portions of the coalbed was estimated to be approximately 33.2 acres. Seven injection boreholes were used in the backfilling. Injection boreholes D-33 and D-35 were located in inundated mined areas and boreholes D-15, D-1, D-28, D-90, and D-113 in dry areas. A total of 62,119 tons of fill material was injected into inundated mine voids, and a total of 90,348 tons was injected into dry voids. The progress of backfilling is summarized in table 2.

TABLE 2. - Injection data on the initial large-scale demonstration project, 1973-74

Project area and acreage	Injection borehole	Approximate injection periods	Total injections, tons
1, 21.10 acres	D-33.....	Aug. 24-Sept. 10, Sept. 17-19, 1973...	54,726
	D-15.....	Sept. 10-17, 22-24, 1973.....	15,914
	D-35.....	Sept. 19-22, 25-26, 1973.....	7,393
	D-1.....	Sept. 24-25, 1973.....	1,105
	D-28.....	Sept. 25, 1973.....	1,579
7, 3.90 acres	D-113....	Oct. 3-13 and Nov. 11-12, 1973.....	26,054
8, 8.30 acres	D-90.....	Nov. 12-18, 1973, and Apr. 2-28, 1974.	45,696

The slurry operation began on August 24, 1973, and was completed on April 28, 1974. Two major interruptions in the operation were experienced:

1. Breakdown and subsequent overhaul of the diesel engines operating the water-supply booster pump and slurry pump, resulting in downtime October 14 through November 11, 1973.

2. The shutdown of operations in anticipation of severe winter conditions that would cause the freezing of pipelines, valves, etc., from November 18, 1973, through April 1, 1974.

Backfill material was distributed in the mine workings throughout the three critical areas, comprising about 33.2 acres, as indicated by observations of the 32 monitoring boreholes. The presence of fill material in monitor holes at or above mine-roof levels indicated that filling of the mine voids within the project areas was essentially complete from floor to roof. The estimated extent of the backfilling in the three areas is consistent with the computed volume of void space in the mine workings and with the quantity of fill material that was injected. Undoubtedly some fill material extends beyond the limits of the completely filled voids where it would be expected to conform to the submerged angle of repose of the sand, about 30°.

In the three project areas, the dip of the No. 7 Seam ranged up to 5°. The monitoring results, which were later confirmed by laboratory model studies (see appendix), indicated that in inundated voids the water serves as a flow medium yielding a balanced updip-downdip distribution pattern. When it became

apparent that injected slurry in dry mine voids traveled more readily downdip, subsequent injection boreholes in dry areas were placed at higher seam altitudes to take advantage of the downdip flow tendency. Data on the behavior of slurry deposition in a dipping bed in both flooded and dry mine cavities are described in more detail in the appendix.

Only two significant operational design problems were encountered during the demonstration. One was the detection of slurry through the water supply wells. This indicated that the mine water was in fact returning to the mine pool as planned, and that recirculation of the mine water was occurring. Because excessive slurry recirculation through the water supply wells could cause damage to well pumps, it was concluded that in future demonstration projects the water supply wells should be located in a more remote area so as to prevent any recirculation of slurry.

On one occasion some entrained air forced slurry to the surface through a plastic-cased monitoring borehole. No property damage was reported; however, slurry was discharged into the street and required a cleanup. It was concluded that henceforth all monitoring boreholes would be cased with steel pipe to a point 4 feet into the top of rock and that accessible steel caps would be welded on to provide for a pressure-tight covering.

The cost of the hydraulic backfill demonstration project, in which 152,467 tons of sandfill material was injected, was $729,000. Reductions in the unit cost of the pumped-slurry technique were anticipated as additional experience was gained in applying the method. Total cost of the project, including exploratory drilling and sonar surveys, was $772,543.67--a unit cost of $5.07 per ton.

The demobilization of the operation, including cleanup and restoration of all project work areas, was conducted in accordance with the contract specifications. Restoration of the borrow pit was completed in accordance with the stipulations set forth by the Bureau of Land Management and included seeding of the borrow area in fall 1974 and a verification inspection of the satisfactory growth after one complete growing season in spring 1976.

PROJECT 2

In fiscal year 1974 Congress provided $700,000 for the Bureau of Mines to continue utilizing the pumped-slurry backfilling technique in areas of Rock Springs where subsidence control would be beneficial to the city.

The demonstration project site was selected in an area of potential subsidence due to extensive mining in No. 1 Seam of the abandoned No. 1 Mine, Union Pacific Coal Co. This would be the first time that the pumped-slurry method of backfilling mine voids would be demonstrated in a central downtown-business area of a city. The objective of the project was to fill both flooded and dry coal mine voids with solids that would support overburden and pillars so as to minimize possible future damage to surface structures, streets, and public facilities. The voids to be backfilled by this project consisted of critical area 14 and about one-half of critical area 11. These areas are shown in figure 16.

FIGURE 16. - Map of a portion of Rock Springs showing location of critical area 14 and a part of critical area 11 in relation to underground mine workings in the No. 1 Seam.

To accomplish this objective, it was proposed to demonstrate that at least 150,000 tons of screened sand could be introduced into the flooded and/or otherwise inaccessible mine voids beneath the areas described above, a total planned area of about 28 acres. The project work was conducted in two phases: Phase I--Exploratory Borehole Drilling; Phase II--Hydraulic Backfilling. Execution of Phase II was contingent to a degree upon a successful Bureau evaluation of the results of the work conducted under Phase I.

The purpose of the exploratory borehole drilling was to outline and define, to the extent necessary, the size, location, and condition of the abandoned coal mines underlying the proposed project areas. Because of apparent errors in maps of the coal workings and the inaccessibility of the mines for inspection or resurveying, the proposed drill holes would help to establish the position of mine workings with relation to the surface.

Through Federal Government competitive contracting and bidding procedures, the contract to conduct both phases of the proposed work was awarded April 4, 1974, to the WHAN Engineering and Construction, Inc., Bismarck, N. Dak., the lowest bidder, in the amount of $666,667.00. Provisions were included in the contract to create a minimum of disturbance to the boom-town environment existing in downtown Rock Springs.

Phase I involved the drilling of exploratory boreholes throughout the proposed project area. Twenty-eight holes encountered mine openings, and although some caving had been noted, it was not believed to be sufficient to block effective movement of slurry. Depths to the mine voids ranged from about 30 to 160 feet, and alluvium depths ranged between 15 and 60 feet. Some of the holes intersected the mine-water pool, which helped to determine the location and depth of the pool in the project area. This information, in conjunction with a study of the mine map, gave direction on the best probable locations for water-supply wells, injection boreholes, and monitor holes. It was apparent also from the drilling that the remaining pillars in the project area were somewhat smaller than had been expected. The void space to be filled, therefore, was recalculated adding 15,000 tons of fill material to the previously estimated quantity for a new total of 165,000 tons. To accommodate handling the increased quantity, the contract was modified June 28, 1974, to increase the total contract amount to $680,167.00. An attempt to substantiate exploratory drilling findings of mine conditions by sonar surveys as in Project No. 1 was not deemed necessary in Project No. 2.

Under Phase II, two of the exploratory boreholes completed under Phase I were reamed to a larger size and converted to water-supply wells for the backfilling process. Twenty boreholes were cased with 4-inch-ID steel casing pipe and capped with removable pressure-tight covers to be used as monitoring boreholes.

Phase II also involved the installation of deep-well pumps, 1,300 feet of water-supply pipeline, the slurry injection plant, and some of the slurry distribution pipelines at sites designated by the Bureau of Mines on the right of way of the Union Pacific Railroad. At the request of the Bureau, the railroad company granted permission of the use of its property for water-supply wells

FIGURE 17. - Map of the project area showing location of the slurry plant in relation to water-supply wells and pipeline, slurry pipelines, and injection boreholes.

and much of the installation of the hydraulic backfilling facilities. The right of way was conveniently located to the project areas and provided adequate space for the operations, including room for a sand stockpile, such that the environmental impact upon the community was minimal. Use of the property for these purposes was formally arranged between the contractor and the Union Pacific Railroad. Slurrying and injections of solids into the mine voids began August 7, 1974. The location of the plant, pipelines, and injection boreholes are shown in figure 17.

Two deep-well pumps lifted the water from the mine at a minimum rate of 4,000 gpm and impelled it through about 1,300 feet of 14-inch-ID welded steel pipe to the site of the mixing plant. Two deep-well turbine pumps were used, one in each well, at points located 95 feet below the surface and 35 feet below the average mine-water pool level. The pumps were rated 4,000 gpm each at a dynamic head of 150 feet and driven by 200-hp electric motors. A part of the 1,300 feet of pipeline from the deep-well pumps to the mixing plant included crossing under the mainline of the Union Pacific Railroad. A 120-foot, 18-inch-diameter bore was used for this purpose with the permission of the railroad company.

Sand was obtained from a borrow pit on property of the Upland Industries Corp., Rock Springs, Wyo., and paid for by the Bureau at a rate of $0.10 per ton. Figure 18 is a view of equipment being operated at the borrow pit. The sand was screened to remove rock particles and other debris in excess of 5/8-inch size and loaded into 30-ton bottom-dump trucks, which hauled the sand 2.5 miles to the site of the mixing plant. The change to a larger size screen from that used in the first large-scale project reflected greater confidence in the pumped-slurry process and a saving in handling cost at the borrow pit.

The stockpiled sand was bulldozed into a hopper and fed onto a conveyor where it was weighed as it was being delivered to a mixing tank constructed of concrete. Water from the deep-well pumps entered the tank through a series of jets that created the agitation necessary to maintain the solids in suspension. During most of the time water for slurrying was used at the rate of 5,000 to 6,000 gpm. The sand, in slurry form, was taken from the tank through a suction line at the rates varying from 200 to 350 tons per hour by a horizontal centrifugal pump, capacity 8,000 gpm. The slurry, 10 to 20 percent solids by volume, was impelled by a 650-hp diesel engine through 14-inch-ID pipelines to the various injection boreholes. The engine supplied sufficient energy to overcome the friction in up to 4,000 feet of pipe and deliver the slurry to the top of the borehole at a pressure to 50 psi when required. Figures 19 and 20 are two views of the pumped-slurry mixing plant.

Pressures at the head of each injection borehole were usually in the vacuum range (less than atmospheric), a characteristic of the pumped-slurry process, because of the drop to the mine level. These pressures would sometimes rise momentarily above atmospheric pressure when the slurry pump was started at the beginning of a shift or when an occasional blockage occurred. Care was taken to stop the operation immediately when pressure increased during injection in a relatively shallow hole.

FIGURE 18. - View of earth-moving and screening equipment operations in the borrow pit.

FIGURE 19. - View of end of hopper, conveyor, slurry pump, diesel engine, pipelines and control building at mixing plant.

FIGURE 20. - View of mixing tank, suction line, sand being delivered to mixing tank, and bulldozer in the background pushing sand into hopper.

Of necessity, the slurry distribution pipelines were buried beneath the busier downtown streets; this included all intersections. On the less-traveled side streets the pipelines were laid along the curb or the side of the road where arrangements were made with property owners to permit the construction of ramps over the pipelines at entrances to driveways. To facilitate conveying slurry to injection boreholes in the downtown business district, it became necessary to lay the slurry pipeline under a spur of the Union Pacific Railroad at one point and under the mainline tracks at another. With permission of the railroad company, a 90-foot, 18-inch-diameter bore was provided for this purpose under the spur track. The crossing under the mainline tracks was accomplished by installing the slurry pipeline through the existing pedestrian underpass. (See fig. 21.)

In fiscal year 1975 and during Phase II, Congress appropriated an additional $500,000 for the Bureau of Mines to conduct further backfilling operations in Rock Springs, Wyo. This increase in funds provided an excellent opportunity for the Bureau to continue the backfilling of voids under and adjacent to critical area 11. It became apparent during Phase II that the voids under the southern and middle portions of critical area 11 were accepting far more sand than the calculations indicated. The excess was undoubtedly moving into old workings west of area 11 where surface support was also needed.

FIGURE 21. - View of the installed slurry pipeline in the pedestrian underpass beneath the mainline tracks of the Union Pacific Railroad.

The additional funds, however, were not sufficient to warrant contracting for a new backfilling project because at least $300,000 would be required to purchase and make operable the equipment comprising a new injection plant. It became expedient, therefore, to renegotiate the existing contract with WHAN Engineering and Construction, Inc., and utilize the newly available funds almost exclusively for backfilling mine voids.

On November 7, 1974, the contract was modified in the amount of $470,000.00, making the following negotiated changes and additions to the contract:

1. The contractor shall continue to pump slurry into the coal mine voids at the contract unit price of $0.90 per ton until the quantity of material placed amounts to 175,000 tons (original contract).

2. After the contractor accomplishes the work described in item 1 above, he shall then pump an additional estimated amount of 156,666 tons of slurry into the coal mine voids at a new unit price of $3.00 per ton.

3. At the conclusion of the contract, as modified, the entire pipeline system, including all piping, both surface and underground, two bypass valves,

two squeeze valves, and the connector into the tub, which would have reverted to WHAN Engineering and Construction, Inc., under the original contract shall become the property of the Bureau of Mines and all rights, title and interest in and to said pipeline shall vest in the Bureau of Mines free of all encumbrances and liens.

Acquisition of the pipeline system compensated the Bureau to some extent for the increased unit cost per ton of injected material. Moreover, it was anticipated that the newly acquired pipeline system would be used by the Bureau in implementing a future backfilling contract at Rock Springs.

This second contract modification increased the total contract amount from $680,167.00 to a new total of $1,150,167.00; increased the estimated injected fill material quantity from 165,000 tons to 331,666 tons; and increased the estimated project area from 28 acres to 54 acres to include most of area 11 (see fig. 16). On May 29, 1975, the contract was modified a third time in the amount of $50,000 to make a new contract total amount of $1,200,167.00; whereby the estimated quantity of fill material injected was increased by an additional 16,667 tons to make a new estimated total of 348,333 tons.

When the project was completed June 26, 1975, an actual total of 348,427 tons of solids had been injected into two project areas of the No. 1 Seam mined coalbed. The total surface area overlying the backfilled portion of the coalbed was estimated to be approximately 55.2 acres. Six injection boreholes were used in the backfilling. Injection boreholes W-3, W-5, and W-6 were located in inundated mine areas and boreholes W-1, W-2, and W-4 in dry areas. A total of 110,111 tons of sand was injected into dry voids.

During the backfilling operation coincidental pothole-type subsidences occurred under and near the mainline tracks of the Union Pacific Railroad. These mine areas were not included in the original project because they were thought to have been isolated by rock-wall barricades constructed underground and effectively filled earlier by the mining company. Bureau engineers designed and coordinated a backfilling subproject between the Bureau contractor and the railroad company, whereby a 6-inch spur pipeline was constructed off the main 14-inch-diameter slurry line to fill the isolated mine voids. Approximately 5,300 tons of sand was successfully injected through a 1,200-foot plastic pipeline into nine 6-inch-diameter injection boreholes, filling the isolated mine voids to refusal with the water draining away through the old barricades. The progress of backfilling is summarized in table 3.

TABLE 3. - Injection data on the second large-scale demonstration project, 1974-75

Project area and acreage	Injection borehole	Approximate injection periods	Total injections, tons
14, 8.20 acres.....	W-1.....	Aug. 7, 15-18, 21, and Sept. 17, 1974.	13,175
	W-2.....	Aug. 7-15, 18-19, 21, and Sept. 18-20, 1974.	24,802
11, 46.00 acres.....	W-3.....	Aug. 22-Sept. 14 and Sept. 21-22, 1974.	62,023
	W-4.....	Oct. 3-9, 13-26, and Nov. 15-16, 18-21, 1974.	66,834
	W-5.....	Sept. 28-Oct. 2, Oct. 26-Nov. 3, and Nov. 7-13, 1974.	97,866
	W-6.....	June 3-26, 1975.....	78,427
Union Pacific Railroad, 1 acre.	9 holes..	Oct. 13-25 and Nov. 8-9, 12, and 15-16, 1974.	5,300

Backfill material was distributed in the mine workings throughout an area of approximately 55.2 acres, according to observations from the monitoring boreholes. As in the previous project the presence of fill material in monitor holes at levels higher than the roof indicated that filling of the mine workings within the 55.2-acre area was essentially complete from floor to roof. The only place within the confines of the planned project area that the existence of fill is questionable was in dry mine voids along the north side of the railroad tracks between injection boreholes W-3 and W-4, where some suspected dead-ended updip mine rooms exist and/or mine workings are located updip from the injection boreholes in such a position that the gravity flow of the material prevented filling in the area. In addition the previously mentioned underground barricades referred to during the backfilling under the Union Pacific Railroad tracks may also have prevented filling in this small area (about 2 acres). Except for this, the estimated extent of the backfilled area is consistent with the estimated volume of void space in the mined bed and with the quantity of fill material that was injected.

The cost of the hydraulic backfill demonstration project contract, in which 348,427 tons of sandfill material was injected, was $1,200,164.07. In addition, $34,842.70 ($0.10/ton) was paid to Upland Industries Corp. for the acquisition of the sandfill material. The total cost of the project was $1,235,006.77 or $3.54 per ton. This can be compared with the $5.07 per ton cost of the previous large-scale demonstration project.

The demobilization of the operation, including cleanup and restoration of all project work areas, was conducted in accordance with the contract specifications. Although it was situated on private land, restoration of the borrow pit was completed in accordance with the stipulations set forth by the Bureau of Land Management and included seeding of the borrow area in fall 1975 and a

verification inspection of the satisfactory growth after one complete growing season in fall 1976.

PROJECT 3, WITH A MANAGEMENT SUPPORT CONTRACT

In fiscal year 1976 Congress appropriated $1,500,000 for the Bureau of Mines to continue utilizing the pumped-slurry backfilling process in areas of Rock Springs, Wyo., where surface support would be beneficial to the city.

The sites selected for this, the third large-scale subsidence control project in the city, included critical areas 2, 9, 10, 12, 13, and 15 and the remaining portion of area 11. The objective of the project was to fill flooded and dry abandoned coal mine voids in the No. 1 and No. 7 Seams with solid material to support mine overburden and alleviate potential damage to surface structures, streets, and utilities. To accomplish this objective, it was estimated that approximately 350,000 tons of sand would be needed to fill the flooded and/or otherwise inaccessible mine voids beneath the critical areas enumerated above, a total area of about 90 acres. Figure 22 is a map of a portion of Rock Springs showing the location of the critical areas included in the project.

In planning for the project, the Bureau decided that extensive drilling operations to provide monitoring could be dispensed with because the nature of the pumped-slurry filling process had been made apparent in earlier projects and was predictable in similar circumstances. Information as to the position and attitude of the coal seams was determined from the drilling of numerous boreholes. Depths to the mine voids ranged from 64 to 293 feet, with depths of alluvium ranging from 15 to 60 feet. Only seven boreholes encountered mine voids, and these were used for both injection and monitoring. The money saved in this manner was diverted to filling more of the mine voids. Appreciable savings would also accrue to the project because two water wells, two injection boreholes, and some of the pipelines, which had reverted to the Bureau of Mines from the previous project (WHAN), would be made available to the contractor for the third large-scale project. Moreover, because some of these facilities remained on property of the Union Pacific Railroad and in strategic position with respect to the project areas, the Bureau indicated in the contract specifications that the slurry injection plant, sand stockpile, and some of the slurry distribution pipelines be established in the same general area as in the previous project. In due course, the project contractor made satisfactory arrangements with the railroad company in order to comply with the specifications.

Procurement of a contractor to do the specified work was again carried out under Federal Government competitive contracting procedures. On May 28, 1976, a contract to do the work was awarded to M. J. Bober Co., Littleton, Colo., in the amount of $1,043,650.00. This was increased by $1,353.00 to pay for the lease of the plant site on railroad property. Work on the contract was started June 4, 1976. Mobilization included preparation of a borrow pit, drilling to establish injection boreholes, and repair and modification of an existing water supply system, and pipelines. A mixing plant, slurry pumps, and recording equipment were also furnished and installed by the contractor.

FIGURE 22. - Map of a large portion of Rock Springs showing location of critical area 2 in the No. 7 Seam and critical areas 9, 10, 11, 12, 13, and 15 in the No. 1 Seam in relation to underground mine workings.

The sandfill material was obtained from a Bureau of Mines designated site located about 4 miles from the mixing plant on federally owned land controlled by the Bureau of Land Management (BLM). This site was made available to the contractor through a "Free Use Permit" between the BLM and the Bureau, and had not been used as a source of material in any previous mine backfilling project. At the borrow site, the contractor furnished and installed screening and materials handling equipment adequate to remove rock and debris in excess of minus 5/8-inch size and produce sandfill at a rate of 2,800 tons per day. Shortly thereafter, under the terms of the contract, it was agreed to begin using progressively larger screen sizes (up to minus 2-inch) to assess the capability of the process in handling sand with larger size rock particles and other debris. It was transported to the stockpile adjacent to the mixing plant site in 30-ton bottom-dump trucks. Removal of the fill material and subsequent reclamation of the borrow pit site were conducted in accordance with stipulations required by the BLM.

The contractor purchased the installed deep-well pumps from the previous contractor (WHAN) and, after repairs and improvements, made operational a water-supply system using the existing wells and supply pipeline sufficient to deliver a minimum of 5,000 gpm to the designated mixing plant. The existing water well sites were located approximately 1,600 feet from the mixing plant site. The vertical distance from the top of the mine water pool in the No. 1 coal seam to the mixing plant site was approximately 75 feet. The two wells were 135 feet deep into a 7-foot mine void and cased with 18-inch-ID steel pipe to 95- and 98-foot depths.

The mixing tank, which was basically the same as those constructed for the previous projects, was fabricated of steel and arranged in such a manner that it would receive measured and weighed sand at a rate of at least 2,800 tons per day and water at a rate of at least 5,000 gpm. The water and sand were mixed to form a uniform slurry in the tank. Provisions were made to install the 12-inch-diameter intake pipe of the slurry pump in such a submerged position that it would withdraw the mixed material (slurry) from the tank. The tank was adequate in size to not only handle the designed quantities but also make sufficient allowance for surges and operational problems, thus helping to prevent overflows.

A slurry pump was installed to withdraw the slurried sand and water from the mixing tank and impel it through the existing and constructed slurry distribution pipelines to one of the two existing or one of the five new injection holes. The pump was capable of moving a minimum of 2,800 tons of sand slurried with the required amount of water per day and was equipped in such a manner that the delivery rate of the pump could be regulated by a system of valves. The pump was designed to pump the required amount of slurry through a maximum distance of approximately 6,000 feet of pipeline to the various injection boreholes at a minimum velocity of 10 feet per second. It developed 25 psi pressure at the top of each injection borehole when required.

All operations involved in the mixing and injection processes were controlled from an instrument panel located in a building supplied by the contractor adjacent to the mixing tank. Metering devices recorded the following

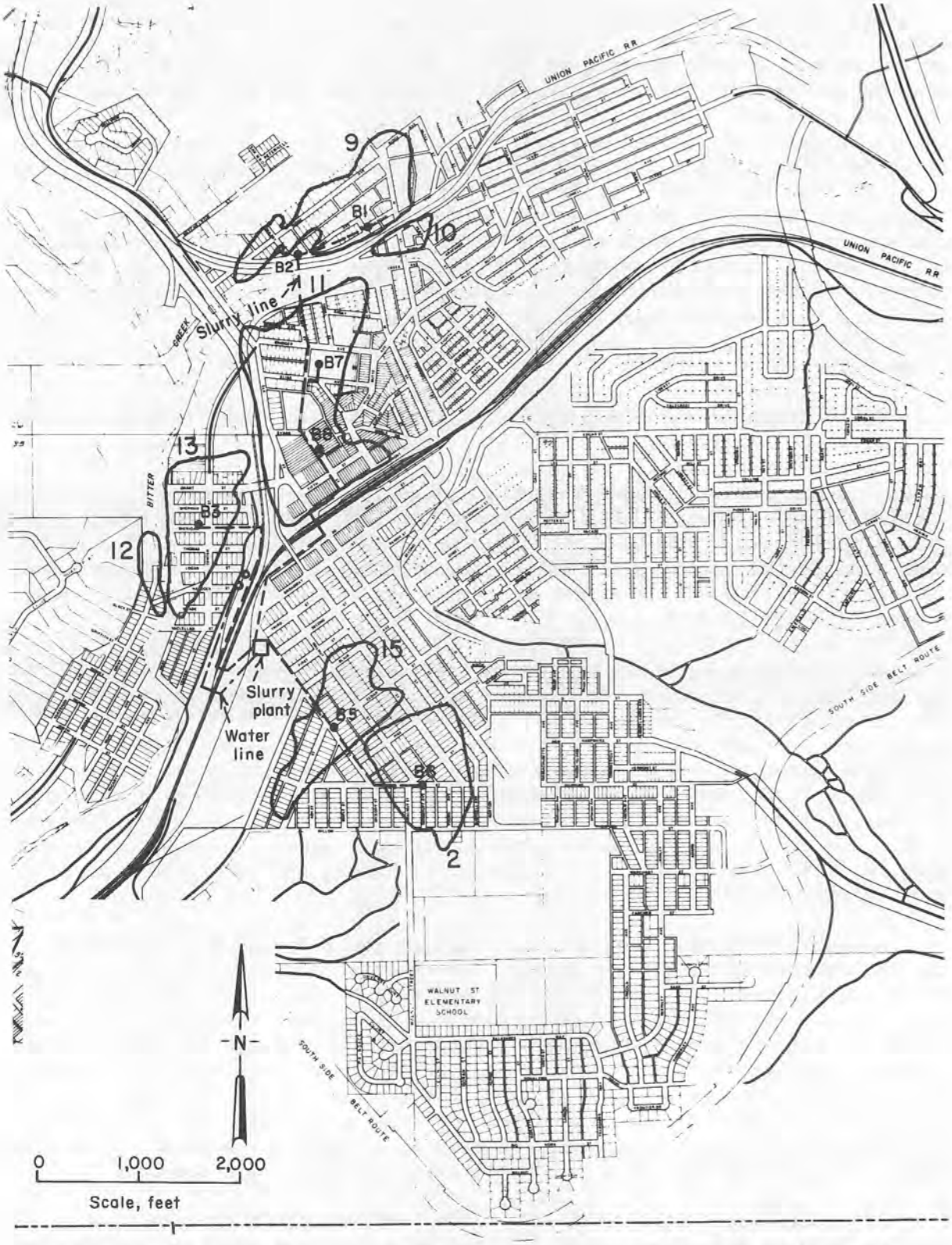

FIGURE 23. - Map of a portion of Rock Springs showing the location of Project 3 installations.

daily activity: total slurry pumping time, time to purge pipelines, gallons per minute of water and slurry pumped, tons of solid material injected, and pipeline pressures at the slurry pump and at the injection boreholes. Figure 23 is a map showing the location of the project facilities. Figures 24, 25, and 26 are views of the equipment that was being used at the borrow pit and at the slurry mixing plant.

The fiscal funding constraints imposed by the Congressional appropriation for the Rock Springs backfilling work prevented the Bureau from providing its own personnel for managing and monitoring the contract activities beyond September 30, 1976. Because of this, the Bureau awarded a $15,000 contract to Johnson-Fermelia and Crank, Inc., Rock Springs, Wyo., on July 26, 1976, to provide the necessary daily onsite project management and monitoring services as required for the backfilling contract work in order to insure that the project work was being conducted in accordance with the contract specifications.

The consulting firm of Johnson-Fermelia, and Crank, Inc., possessed a working knowledge of the Rock Springs backfilling project work. The firm was located in Rock Springs in about the center of the project work area and had an excellent rapport with the city of Rock Springs, utility companies, Wyoming State Highway Department, Union Pacific Railroad, and all other necessary Federal, State, and local organizations. In addition, Wayne Johnson, a principal of the firm, was the City Engineer for Rock Springs and had been actively associated with the backfilling work since its beginning in 1970. As City Engineer, Johnson had approved all previous and current project work plans and had maintained for the city a close watch on all previous project work.

On September 29, 1976, a second project management contract for $16,000 was awarded to Johnson-Fermelia, and Crank, Inc., to provide for necessary extended onsite project management and monitoring services to project completion.

Specific monitoring services that were performed included, but were not limited to, the following:

1. Intermittent or continuous daily inspection of the project work as directed by the Bureau representative and daily verification of completed work on the prime contractor's prepared daily work report sheets.

2. Identifying and notifying the Bureau, including the authority to stop the work, of any potential project-related public health and safety hazard and/or any work being performed in violation of Federal, State, or local law or regulation.

3. Performance of periodic borehole monitoring measurements with Bureau-provided equipment.

4. Continuous inspection during the injection pumping operations, including performing periodic calibration checks of the fill material weighing scale.

FIGURE 24. - View of earth-moving and screening equipment being used in the borrow pit.

FIGURE 25. - Photograph of sand being bulldozed into hopper and fed from conveyor into mixing tank. (Note jet action at side of tank creating turbulence.)

FIGURE 26. - View of mixing plant operation: note suction line at bottom of mixing tank leading to slurry pump with pump motor in left foreground.

5. Conveying Bureau work directives to the prime contractor as necessary and as directed by the Bureau representative.

6. Taking photographs of the project equipment and operations representative of the contract work progress.

The monitoring services were performed by an engineering technician and/or an engineer with a vehicle as deemed necessary by the Bureau representative.

The slurry operation began on July 15, 1976, and was completed on October 12, 1976. A total of 397,464 tons of solids was injected into the seven separate areas of the No. 1 and No. 7 mined coalbeds. The total surface area overlying the backfilled portions of the coalbeds was estimated to be approximately 90 acres, which included an expanded portion of area 11. Here, as in project No. 2, the sand was migrating into voids west of area 11, where it was known that surface support would also be needed. A total of seven injection boreholes, including two from the previous project, were used in the backfilling. Injection boreholes B-1, B-2, B-3, B-7, and B-8 in the No. 1 bed and B-6 in the No. 7 bed were located in inundated mined areas, and borehole B-5 in the No. 1 bed was in a dry area. A total of 378,712 tons of fill material was injected into inundated mine voids, and a total of 18,752 tons was injected into dry voids. The progress of backfilling is summarized in table 4.

TABLE 4. - Injection data on the third large-scale demonstration project, 1976

Project area and acreage	Injection borehole	Approximate injection periods, 1976	Total injections, tons
9, 17.70 acres....	B-1.....	July 15-25.....	52,807
10, 1.50 acres....	B-2.....	July 29.....	3,176
12, 3.30 acres....	B-3.....	Aug. 3-31.....	116,346
2, 22.40 acres....	B-6.....	Sept. 2-11, 13, and Oct. 8..	41,262
15, 10.50 acres...	B-5.....	Oct. 4, 7-8.....	18,752
11,[1] 20.10 acres..	B-7.....	July 30-31, Aug. 31-Sept. 2, and Sept. 13-Oct. 1.	114,194
	B-8.....	Oct. 5-6, 7-12.....	50,927

[1]Expanded area 11.

In order to provide pipeline access to areas 9 and 10, it was necessary to cross Wyoming State Highway 430 and Bitter Creek. Because of the anticipated relatively brief time of injection into boreholes B-1 and B-2 (15 days), the Wyoming Highway Department allowed the contractor to lay the slurry pipeline across Highway 430 but required construction of a ramp over the pipeline so as to maintain the flow of traffic. In crossing Bitter Creek the city

FIGURE 27. - View of slurry pipeline on the pedestrian foot bridge over Bitter Creek.

allowed the contractor to lay the pipeline on an existing pedestrian foot bridge, a less expensive and more convenient procedure than providing a supporting trestle (fig. 27). While the pipeline could have followed the ground in and out of the deep ravine, the number of vertical bends required would not only have increased the cost but also have added greatly to the friction head.

On one occasion an accumulation of entrained air in the mine cavity exhausted with high velocity through injection borehole B-6 back through the open-ended pipeline at the pumping plant site. Although no damage resulted from the air discharge, it was concluded that in future projects a pressure release valve should be installed in the pipeline at the top of all the injection boreholes to avoid similar occurrences.

When pumping through boreholes B-5 into rather shallow mine workings that were believed to be isolated from the main part of the mine, water was reported seeping to the surface near the pumping plant site. This water was determined to be the mine water that was being used for the backfilling operation, and therefore pumping was stopped in borehole B-5 prior to the planned completion.

In September, after the 350,000 tons of sand had been injected under the critical areas, it was noted that the mine voids, particularly in the north part of area 11, were accepting the sand slurry easily at less than atmospheric pressure. It was decided, therefore, to continue injection under the highly vulnerable downtown area until funds available to the Bureau were exhausted. This amounted to an additional $150,024.14, bringing the total cost of project work to $1,195,027.14. On October 12 the slurry pumping operation was completed, with a total of 397,464 tons of fill material having been injected into the old mine workings throughout an area of about 90 acres.

A total of $29,959.00 was paid to the firm of Johnson-Fermelia and Crank, Inc., for providing the necessary onsite management and monitoring services, bringing the total cost of the project to $1,224,986.14, or $3.08 per ton. This compares with the $3.54 per ton cost of the previous or second large-scale demonstration project and $5.07 per ton for the first project.

The demobilization of the operation, including cleanup and restoration of all project work areas, was conducted in accordance with the contract specifications. Restoration of the borrow pit was completed in accordance with the stipulations set forth by the Bureau of Land Management and included seeding of the borrow area in fall 1976 and a verification inspection of the satisfactory growth after one complete growing season in spring 1978.

CONCLUSIONS

In positioning injection boreholes for the critical areas in the three large-scale projects, Bureau engineers assumed that the slurried backfill material would be distributed in the flooded mine workings more or less in equal distances from the points of injection. For dry mine voids, the injection boreholes were placed at higher levels in the workings to be filled. These assumptions were apparently realistic, according to observations from

32 monitor boreholes in the first project and from 20 monitor holes in the second project. Similar phenomena were noted during this period at the initial project in Scranton (22) and in the laboratory model studies (appendix). The movement of fill material, therefore, having become predictable in the pumped-slurry process, monitoring for the third project was limited to those boreholes that were later used for injection. The estimated extent of the backfilled areas in the three projects, about 178 acres, was consistent with the estimated volume of void space in the mined beds and with the quantity of fill material that was injected.

Of the methods of hydraulic backfilling formerly used, controlled flushing (see section of "Hydraulic Backfilling Methods") also results in well-filled mine openings because confinement is provided by bulkheads, fill placement is directed by hand into designated spaces, and the daily progress can be inspected. Controlled flushing and the pumped-slurry method are not generally competitive, however, because controlled flushing is limited to accessible mine workings. The alternative method of backfilling inaccessible mine openings, known as gravity blind flushing (see section on "Hydraulic Backfilling Methods") does not involve pumping of slurry and results in incomplete filling, both laterally and vertically. Of the methods of hydraulic backfilling now known, therefore, the pumped-slurry technique (actually another form of blind flushing), provides the most complete filling of mine workings that are flooded or otherwise inaccessible.

The pumped-slurry method proved successful under the following conditions encountered in the three large-scale projects:

Depth of mine workings to be filled, between 30 and 293 feet below the surface;

Dip angle of workings, for the most part less than 6°;

Mine workings relatively unobstructed by caving of overlying strata;

Average depth of alluvium, 35 feet;

Minimum rock cover, 5 feet;

Particle size of screened sandfill material, minus 1/4-inch ranging up to minus 2-inch;

Specific gravity of sand particles, 2.6;

Bulk density (dry), 100 pcf; and

Water available in large quantities.

Although the sand emplaced by the pumped-slurry process does not totally refill the space formerly occupied by the coal, it does support the remaining pillars and reduces the amount of breakage that otherwise would occur in the overlying strata. This lessens the chance that such disturbances might

eventually reach the surface and cause subsidence. There have been no reports of subsidence affecting surface areas overlying the backfilled areas. This included the Kerback-Belmont area where subsidence incidents had been continuing.

Further use of the technique in different areas will define the range of conditions under which it is feasible. Modifications may extend the range of favorable conditions. The depth range for which the new method may be feasible has not yet been defined. At shallow depth, material injected under pressure may rise to the surface rather than being confined to the mine level, especially in areas where overlying strata are fractured. The vertical completeness of fill in mines that are well above water level needs to be determined. The optimum size range of solids for efficient transport will be defined by future experimentation.

The actual injection operations in the gravity blind flushing method require extensive drilling operations to provide injection boreholes and continues truck traffic through city streets to bring fill material to injection boreholes. Similar disturbances, but to a lesser degree, are created when the controlled method is used. At the injection borehole, personnel are working in the street to direct the solids and water down the borehole. In the pumped-slurry method, street disturbance within the project area is limited to the drilling of an occasional borehole and the installation, maintenance, and removal of the distribution pipeline. During the second and third projects, the sand stockpiles, slurry mixing plants and pumps, as well as parts of the pipelines, were situated on the right of way of the Union Pacific Railroad. While strategically located with respect to the injection boreholes, the noise and dust associated with the operation of the plants was isolated from most dwellings. In the first project the slurry mixing plant and pumps were located at the borrow pit, approximately 2 miles from the built-up area, and had minimal impact upon the environment. During the 3-year period the three projects were implemented, the slurry moved quietly through the built-up areas in pipelines, many of which were buried. Aside from the drilling of boreholes and the installation and removal of pipelines, the only work required during the injection periods was cleaning the streets after infrequent pipeline leakage and on a few occasions where slurry rose to the surface in uncapped or improperly sealed monitoring boreholes.

Cost comparisons of the different methods of backfilling are difficult to make because the total number of projects span an inflationary period of rapidly rising costs. Moreover, subsurface conditions vary in the number of coalbeds to be filled, their depth, the thickness mined, and the percentage of coal left as pillars. Of four subsidence control projects in the Anthracite region backfilled entirely or mainly by the controlled flushing method between 1963 and 1968, the cost per cubic yard or per ton of solids injected ranged from $1.84 to $2.38. On the average, a cubic yard of anthracite refuse weighs 1 short ton. For two blind flushing projects in 1965 and 1967, the costs were $2.46 per ton--a cost that is extremely high when the limited effectiveness of the gravity blind flushing method is considered. In four projects between 1966 and 1969 in which controlled and blind flushing methods were combined, the overall cost per ton ranged from $3.64 to $6.76.

The cost of the first large-scale test of the pumped-slurry method (at Scranton in 1972-73), in which about 451,000 cubic yards of crushed refuse was injected, was $2,165,915--a unit cost of $4.80 per ton. At Rock Springs, where crushing cost was not included, costs per ton varied from $5.07 in the first project (only 152,467 tons), and $3.54 in the second (348,427 tons), to $3.08 per ton in the third project (397,464 tons). Absence of monitor boreholes and reuse of some equipment undoubtedly account for part of the lower unit cost of the third project.

REFERENCES

1. Ash, S. H., and J. Westfield. Backfilling Problem in the Anthracite Region as It Relates to Conservation of Anthracite and Prevention of Subsidence. BuMines IC 7342, 1946, 18 pp.

2. Ashmead, D. C. Mining of Thin Coal Beds in the Anthracite Region of Pennsylvania. BuMines Bull. 245, 1927, 113 pp.

3. Candeub, Fleissig and Associates, Newark, N. J. Demonstration of a Technique for Limiting the Subsidence of Land Over Abandoned Mines. June 1971, 65 pp. (and appendix). Sponsored by Urban Renewal Agency, City of Rock Springs, Wyo.; available from National Technical Information Service, Springfield, Va., PB 212 708.

4. Cochran, W. Mine Subsidence--Extent and Cost of Control in a Selected Area. BuMines IC 8507, 1971, 32 pp.

5. Donner, D. L., and R. H. Whaite. Investigation of Subsidence in Rock Springs, Sweetwater County, Wyoming. Unpublished report, September 1969, 14 pp. Available for consultation from the Bureau of Mines, Division of Mining Environmental Technology, Washington, D.C.

6. Enzian, C. Hydraulic Mine Filling--Its Use in the Pennsylvania Anthracite Fields, a Preliminary Report. BuMines Bull. 60, 1913, 77 pp.

7. Griffith, W., and E. T. Connor. Mining Conditions Under the City of Scranton, Pa., Report and Maps. BuMines Bull. 25, 1912, 89 pp. (and separate case containing 25 maps).

8. Holmes, J. A. First Annual Report of Director of Bureau of Mines. 1911, pp. 42-43.

9. Johnson--Fermelia & Crank, Inc. Underground Mining Study and Evaluation. Prepared for the Rock Springs Urban Renewal Board. Mar. 1, 1972, 27 pp. and appendices.

10. Keith, R. E. Rock Springs and Blair Formations on and Adjacent to the Rock Springs Uplift, Sweetwater County, Wyoming (in Sedimentation of Late Cretaceous and Tertiary Outcrops, Rock Springs Uplift). Wyoming Geol. Assoc. Guidebook, 19th Field Conf., 1965, pp. 42-53 (geologic map and isometric diagram of Rock Springs Fm. in pocket).

11. Kennedy, T. F., and C. Hess. Report of General Investigation of Mining Conditions Under the City of Scranton, Lackawanna County, Pennsylvania. Report to Mayor and City Council of Scranton, June 14, 1960, 27 pp.

12. Lightfoot, W. E. Hydraulic Filling in Metal Mines. Calif. Dept. Conserv., Div. Mines, Spec. Dept. 12, 1951, 28 pp.

13. Morgando, F. P. Engineering Geological Investigation of a Subsided Area, "D" Street--Connecticut Avenue, Rock Springs, Wyoming. Wyo. Highway Dept. Project ARS 1435, Mar. 28, 1969, 7 pp. (8 drillhole logs, 5 figs.).

14. _____. Rock Springs Mine Backfill Project, Rock Springs, Wyoming. Wyo. Highway Dept. Project ARS 1523, Jan. 21, 1971, 2 pp. (9 drillhole logs, map).

15. Paone, J., J. L. Morning, and L. Giorgetti. Land Utilization and Reclamation in the Mining Industry, 1930-71. BuMines IC 8642, 1974, 61 pp.

16. Pennsylvania Anthracite Subsidence Commission. Final Report Submitted to the Legislature. Mar. 15, 1943, 12 pp.

17. Pennsylvania Subsidence Committee. Report of the Subsidence Committee to the General Assembly of the Commonwealth of Pennsylvania. Mar. 1, 1957, 54 pp.

18. Rice, G. S., and I. Hartmann. Coal Mining in Europe. BuMines Bull. 414, 1939, 369 pp.

19. Rock Springs, Wyo. Community Renewal Program. A Recommended Program for the City. July 1972, 32 pp. and appendices.

20. Schultz, A. R. The Southern Part of the Rock Springs Coal Field, Sweetwater County, Wyo. Paper in Contributions to Economic Geology, Part II, Mineral Fuels. U.S. Geol. Survey Bull. 381, 1910, pp. 214-281.

21. U.S. Bureau of Mines. Surface Subsidence Control in Mining Regions. Final Environmental Statement, FES 76-58, November 1976, 90 pp.

22. Whaite, R. H., and A. S. Allen. Pumped-Slurry Backfilling of Inaccessible Mine Workings for Subsidence Control. With an Appendix on Hydraulic Model Studies for Backfilling Mine Cavities by E. J. Carlson, Bureau of Reclamation, Denver, Colo. BuMines IC 8667, 1975, 83 pp.

23. Works Progress Administration. Report on Preliminary Mine Flushing Survey Under the City of Scranton, Pennsylvania. W.P.A. Work Project No. 1-35-80, Official Project No. 65-23-2614 (Proj. No. 4073), Feb. 27, 1936.

APPENDIX

REC-ERC-75-3

HYDRAULIC MODEL STUDIES FOR BACKFILLING MINE CAVITIES
(Second Series of Tests)

by
E.J. Carlson

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Hydraulics Branch
Division of General Research
Engineering and Research Center
Denver, Colorado

UNITED STATES DEPARTMENT OF THE INTERIOR * BUREAU OF RECLAMATION

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PURPOSE

This study is a continuation of the investigation reported in Report REC-ERC-73-19, "Hydraulic Model Studies for Backfilling Mine Cavities." The Bureau of Mines asked the Bureau of Reclamation to conduct additional hydraulic model tests to study different aspects of backfilling mine cavities with sand and waste material to reduce subsidence of land at the surface above the mine cavities.

SUMMARY AND CONCLUSIONS

Additional tests were made in the model of an idealized coal mine that was operated to determine the results of various conditions where mine cavities are backfilled by pumping a fine sand slurry. Fine, uniform blow sand having a median size of 0.14 millimeter obtained from the Rock Springs, Wyoming area was used to produce the sand slurry.

Eighteen tests were conducted in this second phase of hydraulic model tests. The following mine conditions with slurry injection were simulated:

1. Sloping floor with cavity submerged

2. Level floor with cavity submerged

3. Level floor with cavity dry

4. Simulated mine with and without blind entries

5. Corridors and rooms in which there were roof falls and cavities in the roof over the roof falls

Conditions under which the tests were made are summarized in table 1.

Conclusions from the first series of tests were reported in Report REC-ERC-73-19. Data from the second series consisting of 18 tests lead to the following conclusions which are in addition to conclusions made for the first series of tests. The results from the 18 tests reported here support the conclusions derived from the first series.

1. As deposited backfill material reaches the quantity and pattern to build up back pressure in the injection system, one final breakout may occur. A channel is formed down an unobstructed corridor between rows of pillars. The entire discharge from the injection pipe goes down this one channel with high enough velocity to keep the fine sand moving without causing a high back pressure in the pipeline. This final breakout channel can transport slurry material for a long time over a comparatively long distance away from the injection pipe. Deposit would occur at each cross channel junction between the pillars.

2. Fine sand backfill material injected into a submerged mine is transported over roof falls that block corridors when there is an open cavity over the top of the roof fall. In the model study, backfill material almost filled the cavities above the roof falls at the end of the tests.

3. The extent to which backfill material will be transported into and deposited in slack water areas (blind entries) will depend on the position and geometry of the entry with respect to slurry flow past the entry. Fill material will deposit in slack water areas (blind entries) if circulation of sediment-laden water occurs in and out of the slack water areas.

THE MODEL

Model Box—Pump and Slurry Sump

A watertight box made from 3/4-inch waterproof plywood, 15 feet square and 2-1/2 feet deep, was used to contain the model, figure 1. This box was used for previous tests of backfilling mine cavities as described in Report REC-ERC-73-19 prepared for the Bureau of Mines. The same slurry sump and 2-1/2-inch Kimball-Krogh sand pump as described in that report was used to pump the slurry from the sump to the model mine. A recirculating system was used in which fine sand was mixed with water in a sump that was 8 feet long by 2 feet wide by 3.5 feet deep mounted below the floor. Slurry material was pumped into the center of the model, in every case, through the injection pipe mounted in the removable mine roof. Sand material was deposited in the mine and water would flow to the simulated water table. The water level was held above the mine cavity for submerged cavity tests. For dry cavity tests, the water level control gate on the model box was lowered completely; water would leave the elevated mine floor, flow into the model box surrounding the section of simulated mine, out the 4-foot-long by 2.5-foot-wide sluice channel, and back to the slurry sump. Water level in the model box and slurry sump was maintained at the desired level by adding water as necessary. With the propeller mixer running at constant speed, concentration of fill material in the slurry sump would be varied according to the level of energy imparted to the fluid slurry by the mixer and according to the depth of fill material deposited in the slurry sump.

Piping and Measuring System

Previous tests showed that the general pattern of deposit was not dependent upon slurry concentration nor on injection pipe velocity, providing velocities were high enough to transport sediment without deposition in the injection pipe. The concentration and pipe velocities in the tests described here, therefore, were not intended to duplicate those conditions of the Rock Springs injection operations. The deposits should predict the pattern that would occur in a typical mine with a symmetrical uniform pattern of mine pillars and cavities.

For tests 1 through 3 the vertical intake to the sand pump used in previous tests was left in place. However, the pipe entrance was about 3 feet away from the vertical mixer propeller. To obtain a more uniform slurry concentration, the 2-inch nominal intake pipe was lengthened and set on a 45° angle to the vertical so the intake would be closer to the propeller mixer. The propeller mixer was used to keep the fine slurry sand in suspension. A 1/2-inch feed pipe was used for tests 1 through 6 and replaced by a 3/4-inch pipe for the remainder of the tests described in this report. The Venturi meters used for measuring discharge in previous tests were removed. A 3/4-inch Annubar flowmeter with 0.824-inch inside diameter was installed in the horizontal section of the pipe for measuring discharge in all tests, figure 1. To minimize possible plugging of the impact and low-pressure ports in the flowmeter, two purge water lines were attached to the ports, each line having a rotameter to measure the purge water.

The Annubar flowmeter was calibrated for clear water without the purge inflow at the pressure ports. Flows through the rotameters were then set to give the same discharge rating as without the purge water connections. To continuously determine the slurry discharge without getting fine sand in the meter ports and plugging them, a small amount of purge water was used.

Pressure piezometers were located about one pipe diameter from the end of the injection pipe and in the mine cavity as shown on figure 2. The pressures were read on the water manometer board and recorded at short time intervals to determine changes in the pressure as the backfill material deposited in the cavity during each test. For tests 1 through 5, seven piezometers were used including a piezometer showing the water table elevation surrounding the mine cavities. For tests 6 through 17, pressures were measured at 11 points. At points where fill material deposited up to the mine roof, the pressure taps became plugged with the fine sand.

All figures in the report showing drawings and contour maps of the mine model are oriented with north at the top of the figure for easy comparison of the deposit patterns. Contour intervals are designated in feet above the mine floor on all contour maps.

Model Scales—Mine Pillars

The model mine was constructed to represent a mine with the cavity volume equal to 60 percent and the pillar volume equal to 40 percent of the total volume. The horizontal scale for all model tests was $1^m:48^p$ (1 in the model is equal to 48 in the prototype.) The vertical scale for most tests were also $1^m:48^p$. Some early tests (1 through 5) were made with a vertical scale of $1^m:14.908^p$, a vertical distortion of 3.22, to establish deposit patterns with velocities in the model mine cavity equal to the velocities in the typical prototype cavity. Deposit patterns for undistorted and distorted scales were similar; therefore, tests 6 through 18 were performed with the model constructed to an undistorted scale of $1^m:48^p$, vertical and horizontal.

Mine pillars were constructed in the model to represent horizontal dimensions 40 feet long and 10 feet wide, with a cavity spacing of 10 feet between sides of pillars and also 10 feet between ends of pillars. This gave a mine arrangement as described above with 40-percent solid and 60-percent cavity both for the distorted and undistorted model scales. The 8-foot-square mine area in the model represented a 384-foot square or 3.39 acres in the prototype.

Backfill Material

Fine sand obtained from the Rock Springs injection project was used in the model studies. A size analysis and relative density determination for the backfill material used in the model and prototype mine is shown in figure 3. The median diameter of the fine sand was 0.14 millimeter. Standard properties and bearing capacity tests on the backfill material were made in the Soils Laboratory of the Earth Sciences Branch of the Bureau of Reclamation. These studies are reported in Report REC-ERC-73-19.

THE INVESTIGATION

Sloping Mine Floor

Distorted model tests.—Tests 1 through 5 were made to evaluate the changed piping system, the Annubar flowmeter, the seal of the roof against the mine pillars, and general operation of the pump-piping system and slurry sump. Tests 1 to 3 were conducted with the vertical intake pipe on the pump. At the end of test 3,

an inspection showed a hard crust of fine sand in the slurry sump just below the vertical pipe intake located 3 feet horizontally from the mixer propeller. The crust which apparently formed over a period of operation, was similar to hard surface crusts that form in open channels having bed material made from fine sand.

In test 1 sand was fed to the slurry sump at a rate of 1.1 pounds per minute. For test 2 the rate of sand was increased to 12 pounds per minute. On both tests 1 and 2, pressure built up in the mine cavity after fill material was deposited up to the roof level. In test 2, the pressure increased so much that the roof lifted from the pillars, and fill material was transported between the roof and the tops of the pillars, figures 4 and 5. After test 2 was completed, four bolts were installed through the pillars from the mine floor to the roof. An additional 1/4-inch layer of sponge rubber was fastened with adhesive to the roof to form a seal on the pillars as the bolts were tightened. After conducting test 2 for about 35 minutes in the model, the mine cavities were filled to cause back pressure in the injection pipe. Shortly after this pressure built up, breakouts occurred upslope first, then downslope and to the slides. The mine was set on the 5° dip.

Test 3 was operated for about half an hour. Water discharge was started at 0.030 ft^3/s. When sand was added to the slurry, the discharge dropped to 0.020 ft^3/s. The average discharge during the test was 0.025 ft^3/s. Pressures measured at the piezometer on the end of the injection pipe varied from 1.02 to 1.74 feet compared to 1.56 to 1.74 feet measured at the piezometers in the cavity. Figure 6 shows the deposit pattern at the end of test 3.

After completing test 3, the 2-inch intake pipe was lengthened 1 foot 3 inches and was reconnected to the pump intake at a 45° angle. With this arrangement, the end of the intake pipe was 0.9 foot above the floor of the slurry sump and closer to the mixer propeller. Test 4 showed that moving the intake pipe closer to the mixer propeller caused extra deposit in a cone shape around the mixer in the slurry sump. Consequently, much of the sand added at 14 pounds per minute deposited in the slurry sump and was not pumped to the model mine. The amount of fill material that was pumped and deposited in the mine was comparatively small, figures 7 and 8. Test 5 was therefore made as a continuation of test 4.

Tests 4 and 5 were made with the mine submerged and dipping 5°. The model had a horizontal scale of $1^{m}:48^{p}$ with a vertical distortion of 3.22. The breakout through the initial deposit caused clouds of slurry to come through the corridors to the edge of the mine area. Pressures on the end of the discharge pipe in test 4 remained approximately the same throughout the test, indicating that there was no back pressure; consequently, the solid fill material did not fill the cavity near the ceiling. In test 5, pressure at the end of the discharge pipe increased with continued injection. Figures 9 and 10 show the deposit pattern in the mine cavity at the end of the test.

Undistorted model tests.—For tests 6 through 18 the pillars were changed to give an undistorted geometric scale of $1^{m}:48^{p}$ in both horizontal and vertical directions. An observation test was conducted with the mine submerged and dipping at an angle of 5°. Velocity in the 1/2-inch pipe was about 9.9 ft/s. Deposits occurred and silty water that could be observed at the edge of the mine section was moving upstream in corridors 3 through 9, counting from the left side looking downslope. Additional piezometers in the mine roof were added to give a wide pattern of pressure distribution away from the injection pipe. As the backfill material deposited, pressures with the additional deposit ring were slightly higher than pressures outside the central cavity (piezometers 7 through 11). The additional piezometers 7 through 11 were added in the mine roof after test 6 was started. The test was stopped, the mine drained, the roof was raised, and the piezometers installed. No photographs were taken nor was a contour map of backfill deposit prepared for test 6 because of the changes during testing.

Test 7 was performed with the same conditions as for test 6 except the injection pipe with an inside diameter of 1/2 inch was replaced by an injection pipe with an inside diameter of 3/4 inch to get higher discharge capacity through the pump-piping system. The 1/2-inch pipe was restrictive, which caused debris to collect in the pipeline. A valve was installed on the high point to the bowl of the centrifugal pump which made it possible to bleed air and later to extract sediment samples from the pump. The valve also made it easier to prime the pump at the startup for a test. At the end of test 7, material was flowing between the pillars and the mine roof in a few places, figures 11 and 12. After test 7 was completed, two additional toggle bolts, making a total of 6, were installed to hold the roof tight against the pillars.

A water purge system for the Annubar flowmeter was installed at the end of test 7. Previous test discharges were set with only water in the piping system before the mixer was turned on. Without the purge system, when fill material was pumped in slurry form, the ports to the flowmeter would tend to plug. By using the purge system, pressure was positive at each of the two

ports of the Annubar flowmeter, which caused a small flow into the pipeline, preventing fine sand from entering and plugging the pressure tubes to the flowmeter.

Test 8 was made with a velocity of approximately 7.5 ft/s in the 3/4-inch injection pipe. The mine was dipping 5° and submerged. The deposit pattern was observed at the end of the test after the roof was raised, figures 13 and 14. No deposit on the top of the pillars indicated the roof held tight against the back pressure that occurred in the mine cavity. After the initial deposit ring was established around the injection pipe, back pressure built up and a breakout occurred downslope in corridors 8 and 9, counting from the left looking downslope. The fine sand was carried in suspension along the bed and deposited in a large mound off the edge of the mine platform. With increased pressure, a breakout occurred and high velocity flow started upslope in corridor 9, counting from the left looking downstream.

The characteristics of test 8 were typical of an injection into a submerged cavity with open corridors. After the initial deposit ring has occurred and back pressure builds up in the cavity and in the injection pipe, a breakout occurs in one or two corridors. Fill material is carried along this channel in suspension or as bedload according to basic sediment transport principles. With the full flow of the injection pipe discharging along a channel, an equilibrium condition develops for sediment transport. Fill material deposits at intersections to essentially block side corridors and confine the flow along the one channel. Deposit builds in the channel until the cross-sectional area reduces and the velocity increases to cause critical transport conditions.

Reports from field operations at Rock Springs, Wyoming, indicate flow occurs in a single channel over long distances after fill material is deposited up to roof level around the injection hole. Model tests showed that flow in a single breakout channel started when deposit sealed or nearly sealed the space adjacent to the roof around the injection hole. Pressure would build up in the cavity prior to the breakout and would lower as flow started in a single channel. Extensive deposit and lowered pressure prevented other breakout channels from forming. For test 8, after material had deposited up to the roof, slurry flowed down one corridor until the test was stopped.

Blind entries—submerged mine.—Tests 9, 10, and 11 were made as a series with the mine dipping 5° and submerged. After each test was stopped, the roof was raised and the deposit pattern observed. The mine roof was then lowered, fastened in place, and the next test in the series continued. A contour map and photographs were made of the deposit pattern for tests 9 and 11. Blind entries were simulated in the model by installing blocks at various places in corridors in the mine. Some blocks were installed to block corridors at ends of pillars and also near the middle area of pillars. Some blocks were installed to prevent communication within corridors and over considerable distances in some cases, figures 15 through 19. During test 9, initial fill material deposited up to the roof around the injection pipe, and back pressure caused a reduction in discharge. Piezometers attached to the roof showed the increase in back pressure and then the sudden decrease in back pressure when a breakout occurred. The pictures and contour map prepared at the end of test 9 are shown in figures 15 and 16.

Test 10 was a continuation of test 9, using a smaller discharge. The smaller discharge resulted in a lower intake velocity and, consequently, a lower sand concentration. A small additional amount of fill material was deposited in the mine during test 10. The pressure in the area around the injection hole was comparatively high. No photographs were taken nor was a contour map prepared at the end of test 10. Before test 11 was started, the sand deposit was carved back to the deposit pattern left at the end of test 9.

Test 11 was made with a slightly lower average discharge throughout the test. When the discharge tended to decrease because of back pressure, the control valve was opened to maintain a constant discharge. For test series 9, 10, and 11, the fill material seemed to deposit downslope first, then upslope, and then on the level out from the injection hole toward the sides. At the end of test 11, the last breakout established a comparatively high velocity flow upslope in corridor 8, counting from the left side looking downslope. The flow being confined to a single corridor caused the velocity to be comparatively high and, thus, the transport capacity continued at a comparatively high value. The flow at the end of test 8 for a mine without blind entries was similarly confined to a single corridor. At the end of the series of tests 9, 10, and 11, flow was confined to a single corridor and the slurry traveled upslope with a comparatively high velocity. Figures 17, 18, and 19 show a photograph and contour maps of deposited slurry material at the end of test 11. Figure 19 indicates that fill material will not enter and deposit where blind entries prevent flow circulation.

Level Mine Floor

Roof falls and cavities over roof falls—submerged

mine.—Tests 17 and 18 were conducted to show how fill material would be transported over simulated roof falls and through cavities above the roof falls, a condition occurring in coal mines after being abandoned for some time. Roof falls were simulated by truncated wood pyramids sloped 60° from the floor, figures 20 and 21. The top of the roof falls were 6 feet above the mine floor, the same height as the normal roof. Above the roof fall a cavity was formed by cutting the roof and constructing a box over the hole cut in the roof, figure 20. A piezometer was placed in each simulated cavity to measure pressures developed in the cavities during the backfilling operation. Two roof falls were placed at intersections of corridors; one roof fall was placed between ends of pillars and one was placed between sides of pillars, figure 21.

For tests 17 and 18, the mine was level and submerged. Before beginning test 17, sand was added to the slurry tank so that the backfill supply was sufficient to complete the test. During the 50-minute test, samples of slurry taken from the pump discharge pipe varied in concentration from 1.2 to 5.1 percent, by weight, with an average of 1.8. These samples were taken using a 1/8-inch tube with its entrance pointing upstream in the vertical pipe where flow lines were parallel. A photograph showing backfill deposit at the end of test 17 and contour maps at the end of tests 17 and 18 are shown on figures 22, 23, and 24. Test 18 duplicated test 17, except test 18 had a higher injection velocity and higher slurry concentration, table 1. At the end of both tests 17 and 18, high velocity flow moved along one corridor directly away from the injection pipe. The velocity along the breakout corridors was high enough to transport fill material without depositing.

At the end of test 17, slurry was flowing up over the roof fall, through the cavity above, and down the corridor, figure 23. The mine was submerged and the resistance offered by the roof fall was not great enough to cause slurry flow to move to another corridor. The slurry takes the flow path of least resistance.

At the end of test 18, the last breakout channel was along a corridor adjacent to a corridor having a roof fall at a corridor intersection, figure 24. This breakout channel was in the opposite direction from the last breakout channel for test 17. It is apparent that for a level mine that is submerged, there is very little difference in the resistance to flow in one direction than to the flow in the opposite direction. In tests 17 and 18, the last breakout channels were along the length of the pillars. Apparently, the abrupt expansion and contraction losses caused by the intersections of lateral channels in the short direction of the pillars may be greater than the losses in the corridors along the length of the pillars. For dry mine cavities, the flow conditions and, consequently, the patterns of resistance to flow are different from those in submerged cavities.

Dry mine cavities.—Tests 12 through 15 were conducted with the mine cavity in an unsubmerged (dry) condition. The water table is lower than the floor of the mine cavity. To provide for this condition in the model, the injection water was allowed to drain out of the model box by having the water level control gate completely lowered. The mine roof was in place for all four tests. Fill material is deposited and slurry water returns to the water table. The backfill material in a dry mine cavity develops a deposit with a surface slope that is dependent on the critical tractive force ($T_D = \gamma DS$) for the material, where T_D = tractive force, γ = specific weight of water, D = depth of water flowing over the deposit, and S - slope of the flowing water surface. A critical tractive force (tractive force that causes a given size of fill material on the bed to start moving) is related to the depth and the slope so that the product (D×S) is constant for the given size bed material.

Tests 12 through 15 were conducted as a series in which fill material was not removed at the end of each test, table 1. A photograph and contour map were made at the end of each test to compare the progress of fill deposit, figures 25 through 32. The progress of the deposit with each successive test can best be observed on the contour maps, figures 25, 27, 29, and 31. The 1-, 3-, and 5-foot contours show the deposit buildup and how the sloping face of deposited backfill material moves with time. Backfill material builds up close to the roof near the injection pipe. A breakout occurs when deposit around the injection pipe is high enough to force most of the flow in one concentrated channel, figures 27, 29, and 31. The direction of the breakout channel varies for different tests when the mine floor is level, indicating initial deposits are uniform and symmetrical for a symmetrical pillar pattern on a level floor. The difference in resistance to breaking out in one direction compared to another direction is very small.

Discharge and, consequently, injection pipe velocity for the series of four tests conducted in a dry cavity were very nearly identical, 0.013 or 0.012 ft^3/s and 3.5 or 3.2 ft^3/s, respectively. Solids concentration in the injection pipe varied from 1.0 (test 15) to 4.5 percent (test 12) by weight, table 1.

Submerged mine cavities.—Test 16 was conducted in two parts over a period of 2 days on a level submerged mine cavity. The first part had a discharge of 0.013

ft^3/s with a slurry concentration of 0.74 percent by weight, and the second part had a discharge of 0.021 ft^3/s and a slurry concentration of 6.7 percent. Photographs, figure 34, and a contour map, figure 33, were made at the end of the test on the second day. The deposit pattern, particularly the mine area that had deposited material up to and very near the roof, was extensive. This was caused by the high concentration and higher discharge and, consequently, higher injection velocity. The higher velocity caused the high concentration of backfill material to deposit at a greater depth farther from the injection pipe than could be obtained with a lower injection velocity.

The breakout channel develops whether or not there is a high injection velocity or lower injection velocity. For a higher injection velocity, pressure buildup in the mine cavity caused by pumping and deposit of backfill material takes longer than for a lower injection velocity. The deposit depth up to the roof or near the roof extends over a greater area for higher injection velocities.

In test 16, some cavities were left in corridors between pillars. Slurry material was transported past opposite ends of pillars depositing material at the same time from opposite ends of corridors. The deposit blocked the corridors, leaving a small unfilled cavity. In a prototype mine, the extent of unfilled cavities between pillars depends on the pattern of the pillars and corridors. Cavities could be left between pillars when the flow pattern was symmetrical.

A final breakout channel on test 16 formed at the end of the test in which most of the injected slurry was flowing down one corridor. This type of flow would continue if the test were not stopped. The slope of the bottom of the deposit in this last breakout channel was very flat, similar to the final breakout channels in previous tests in a submerged mine.

REFERENCES

Carlson, E. J., *"Hydraulic Model Studies for Backfilling Mine Cavities,"* Bureau of Reclamation, Report REC-ERC-73-19, prepared for U.S. Bureau of Mines, October 1973

Whaite, Ralph H., Allen, Alice S., *"Pump-slurry Backfilling of Inaccessible Mine Workings for Subsidence Control,"* U.S. Bureau of Mines, Washington, D.C., 1975.

Courtney, W. J., Singh, N. M., *"Feasibility of Pneumatic Stowing for Ground Control in Coal Mines,"* Illinois Institute of Technology Research Institute, Chicago, Illinois prepared for U.S. Bureau of Mines, January 1972

Yalin, M. Selim, *"Theory of Hydraulic Models,"* The MacMillan Press, Ltd., 1971

Yalin, M. Selim, *"Mechanics of Sediment Transport,"* Pergamon Press, 1972

Graf, Walter H., *"Hydraulics of Sediment Transport,"* McGraw-Hill Book Company, 1971

Ackenheil, Alfred C., and Dougherty, Murray P., *"Recent Developments in Grouting for Deep Mines,"* Journal of Soil Mechanics and Foundations Division, Proceedings of ASCE, Vol. 96, No. SM1, January 1970.

Table 1

SUMMARY DATA OF MODEL TESTS FOR BACKFILLING MINE CAVITIES

Test No.	Injection pipe diameter, in.	Q ft^3/s	V in pipe, ft/s	Sed. con. % by wt by imhoff cone	Mine dry or sub-merged	Time with sed. transport, min.	Fill deposited in mine, ft^3	Dip of mine floor, degrees	Center cavity: Radius to mid. contour model, ft	Center cavity: Diameter prototype ft	Comments
Model scale: H = 1^m:48P; V = 1^m:14.908P distorted											
1	0.5	0.033			Sub	59		5			
2	0.5	0.032			Sub	70	7.74	5			
3	0.5	0.025	11.8		Sub	31	1.57	5	0.76	73.0	
4	0.5	0.032	15.1		Sub	52	0.89	5	0.61	58.6	Sand deposit not removed at end of test.
5	0.5	0.035	16.5		Sub	37	2.10	5	0.16	15.4	Continuation of test 4.
Model scale 1^m:48P undistorted											
6	0.5	0.021	9.9		Sub	39		5	-	-	
1/2-inch-diameter injection pipe changed to 3/4-inch diameter											
7	0.75	0.034	9.2	3.1	Sub	35	0.44	5	0.39	37.4)	V in last break-out single channel = 3.6 ft/s.
8	0.75	0.029	7.8	11.4	Sub	46	2.34	5	0.65	62.4)	
9	0.75	0.017	4.6	5.6	Sub	15	0.60	5	0.33	31.7	Blind entries.
10	0.75	0.014	3.8	4.8	Sub	24	-	5	-	-	Blind entries.
11	0.75	0.13	3.5	4.5	Sub	17	0.33	5	0.39	37.4	Blind entries; V in last break-out channel = 3.7 ft/s

Table 1 - continued

Test No.	Injection pipe diameter, in.	Q ft^3/s	V in pipe, ft/s	Sed. con. % by wt by imhoff cone	Mine dry or sub-merged	Time with sed. transport, min.	Fill deposited in mine, ft^3	Dip of mine floor, degrees	Center cavity: Radius to mid. contour model, ft	Center cavity: Diameter prototype ft	Comments
				1/2-inch-diameter injection pipe changed to 3/4-inch diameter - continued							
12	0.75	0.013	3.5	4.5	Dry	17	0.30	0	0.38	36.5)	Accumulated time: 17 minutes
13	0.75	0.012	3.2	3.0	Dry	27	0.29	0	0.37	35.5)	44 minutes
14	0.75	0.012	3.2	1.6	Dry	51	0.16	0	0.40	38.4)	95 minutes
15	0.75	0.012	3.2	1.0	Dry	71	0.15	0	0.43	41.3)	166 minutes
16	0.75	(0.013 (0.021	3.3 5.6	0.74 6.7	Sub	17) 28)	0.87	0	0.38	36.5	
17	0.75	0.013	3.5	1.8	Sub	52	0.65	0	0.41	39.4	Roof falls and cavities.
18	0.75	0.019	5.1	3.4	Sub	50	-	0	0.48	46.1	Roof falls and cavities.

Average diameter 36.9 feet - tests 7-18. Avg 36.9

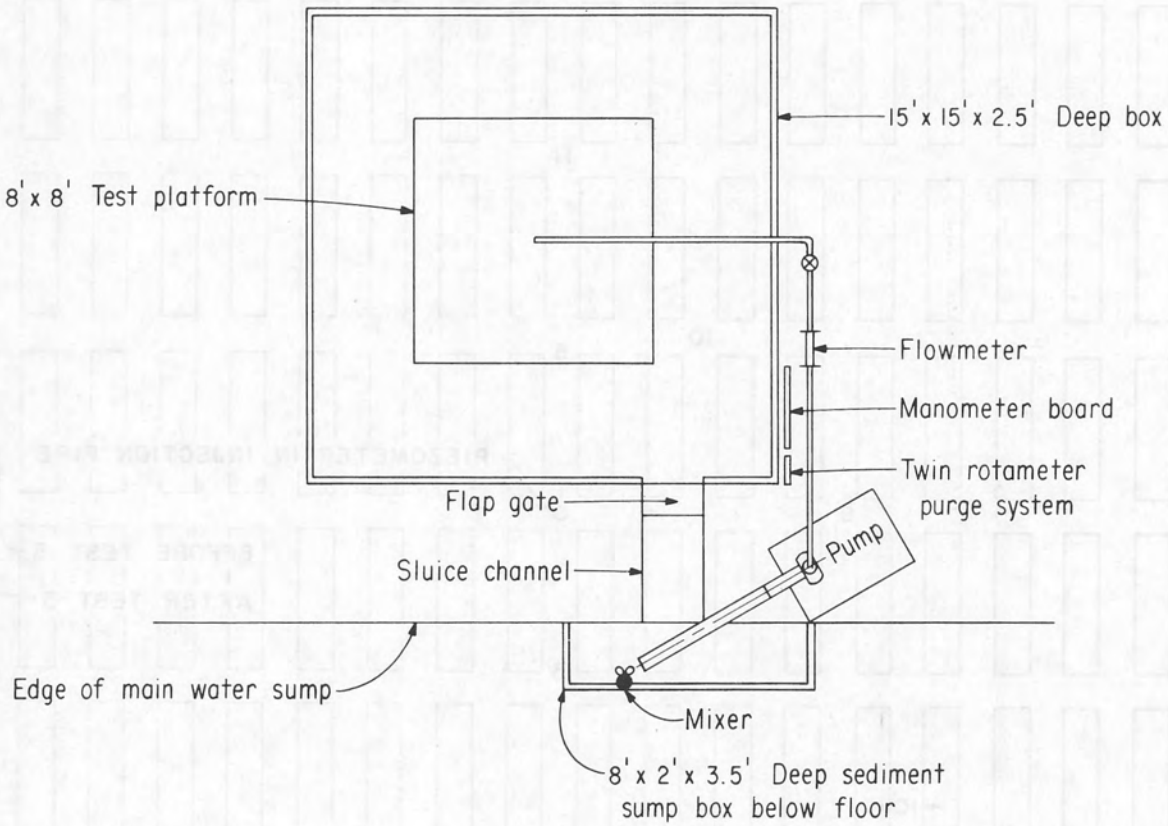


Figure 1. Model test facility.

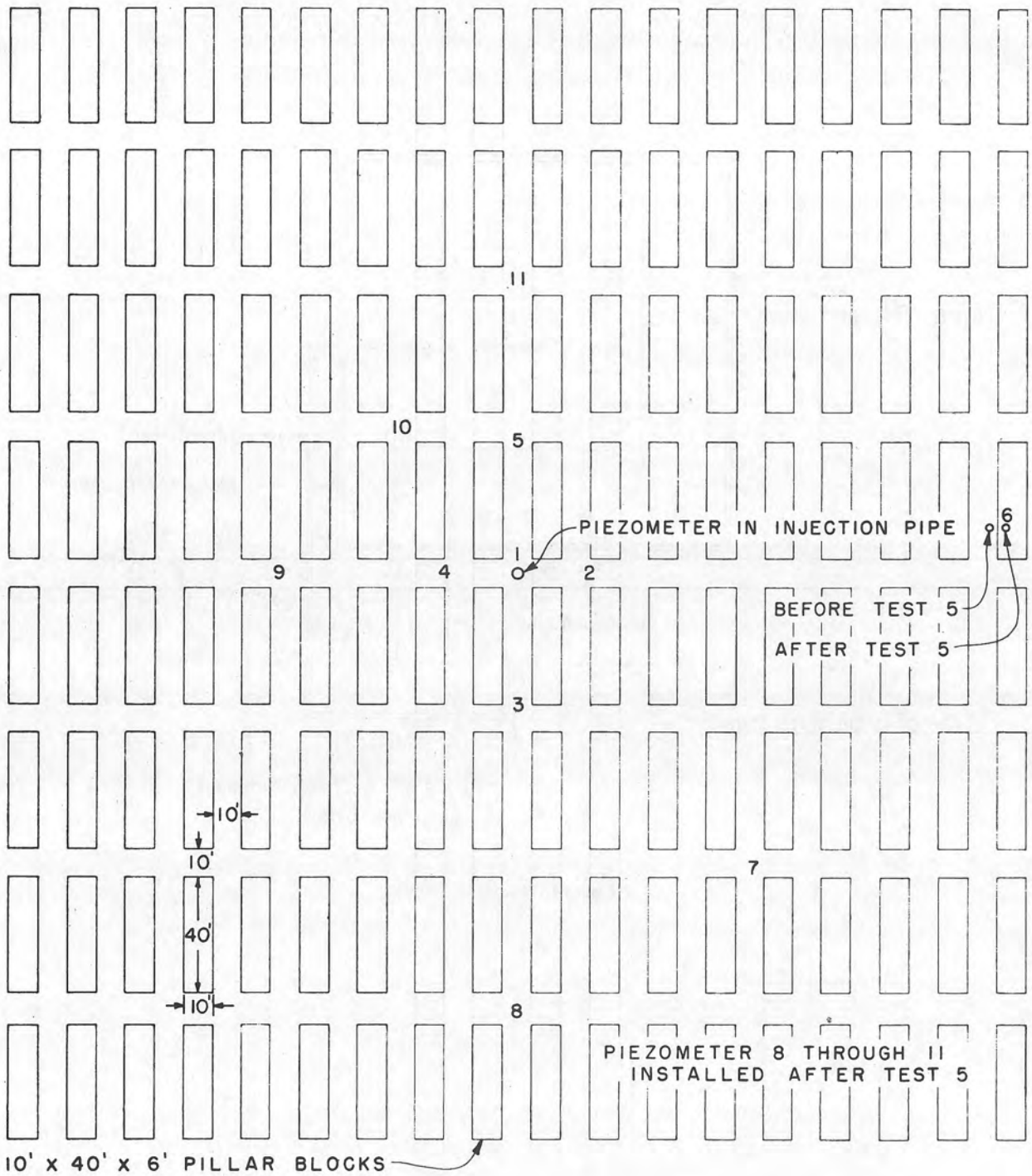
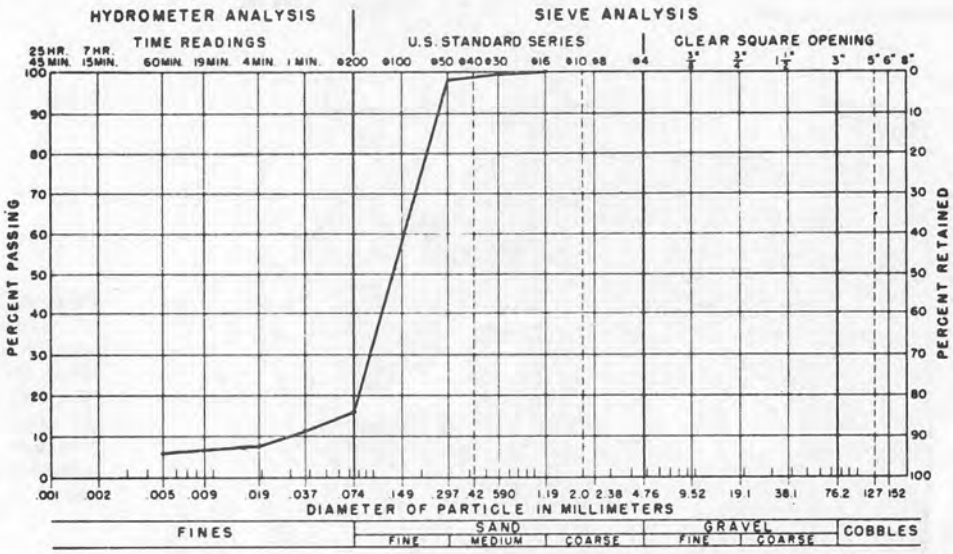
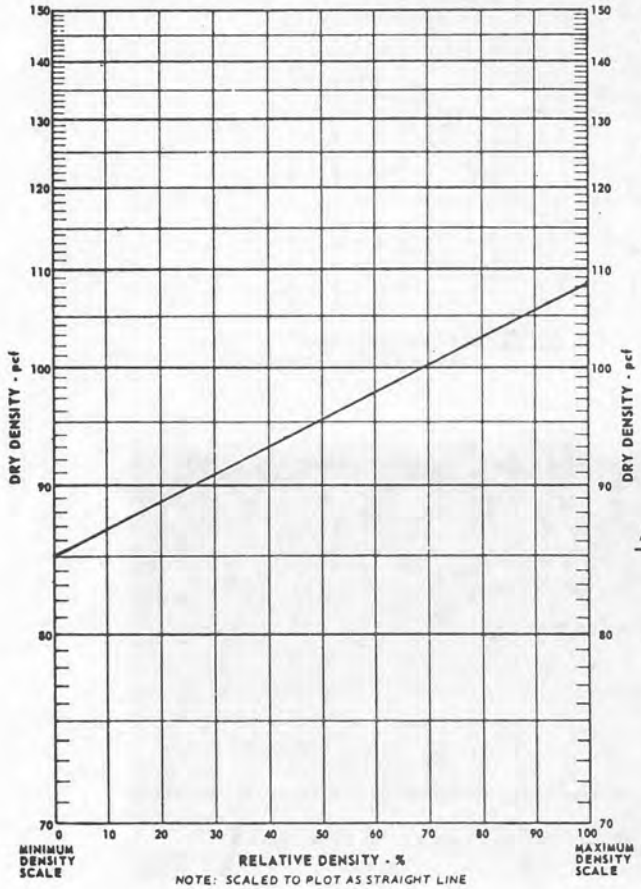


Figure 2. Location of piezometers installed in mine roof and injection pipe.

7-1742 (8-70)
Bureau of Reclamation

PHYSICAL PROPERTIES SUMMARY PLOT (Relative Density)





Classification Symbol SM

Gradation Summary
- Gravel 0 %
- Sand 84 %
- Fines 16 %

Atterberg Limits
- Liquid Limit ____ %
- Plasticity Index NP %
- Shrinkage Limit ____ %

Specific Gravity
- Minus No. 4 2.68
- Plus No. 4
- Bulk ____
- Apparent ____
- Absorption ____ %

Relative Density
- Minimum Density 85.0 PCF (1.36 gm/cm³)
- Maximum Density 108.4 PCF (1.74 gm/cm³)
- 1/ In-place Density 93.8 PCF (1.50 gm/cm³)
- Percent Relative Density 43.5

Permeability Settlement
- Placement Condition ____
- Coef of Permeability ____ ft/yr (____ cm/sec)
- Settlement Under ____ psi Load ____ % (____ kg/cm²)

Notes: 1/ as determined in laboratory model study

Sample No. 545-1 Hole No. ____ Depth ____ ft (____ m)

Figure 3. Size analysis and relative density of fine sand backfill material.

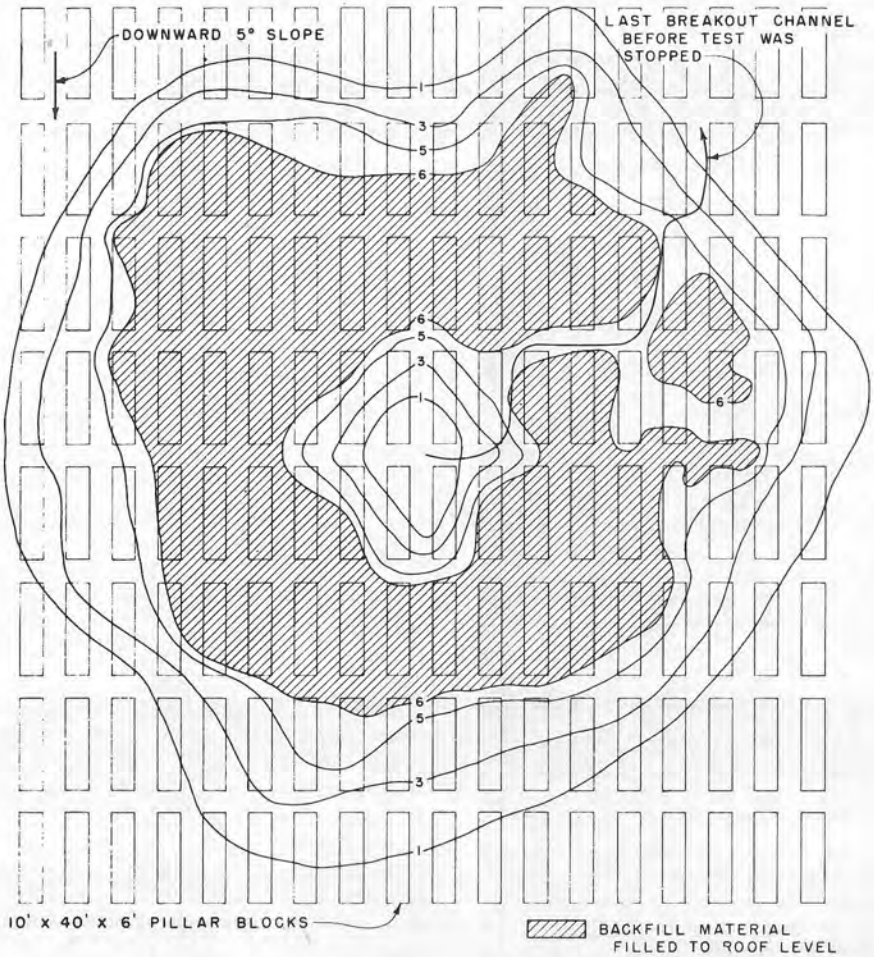


Figure 4. Test 2. Contours of deposited backfill material at end of test. Mine cavity submerged. Mine roof was lifted from the pillars by the pump pressure. (Preliminary test—distorted model scale)

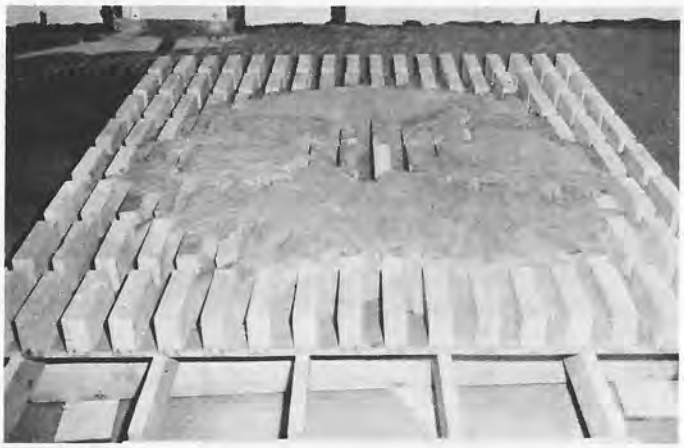


Figure 5. Test 2. Pressure in the mine cavity resulting from slurry pumping caused the roof to raise above the pillars, allowing slurry to flow over the tops of pillars. Mine submerged and dipping 5°. (Distorted model scale)

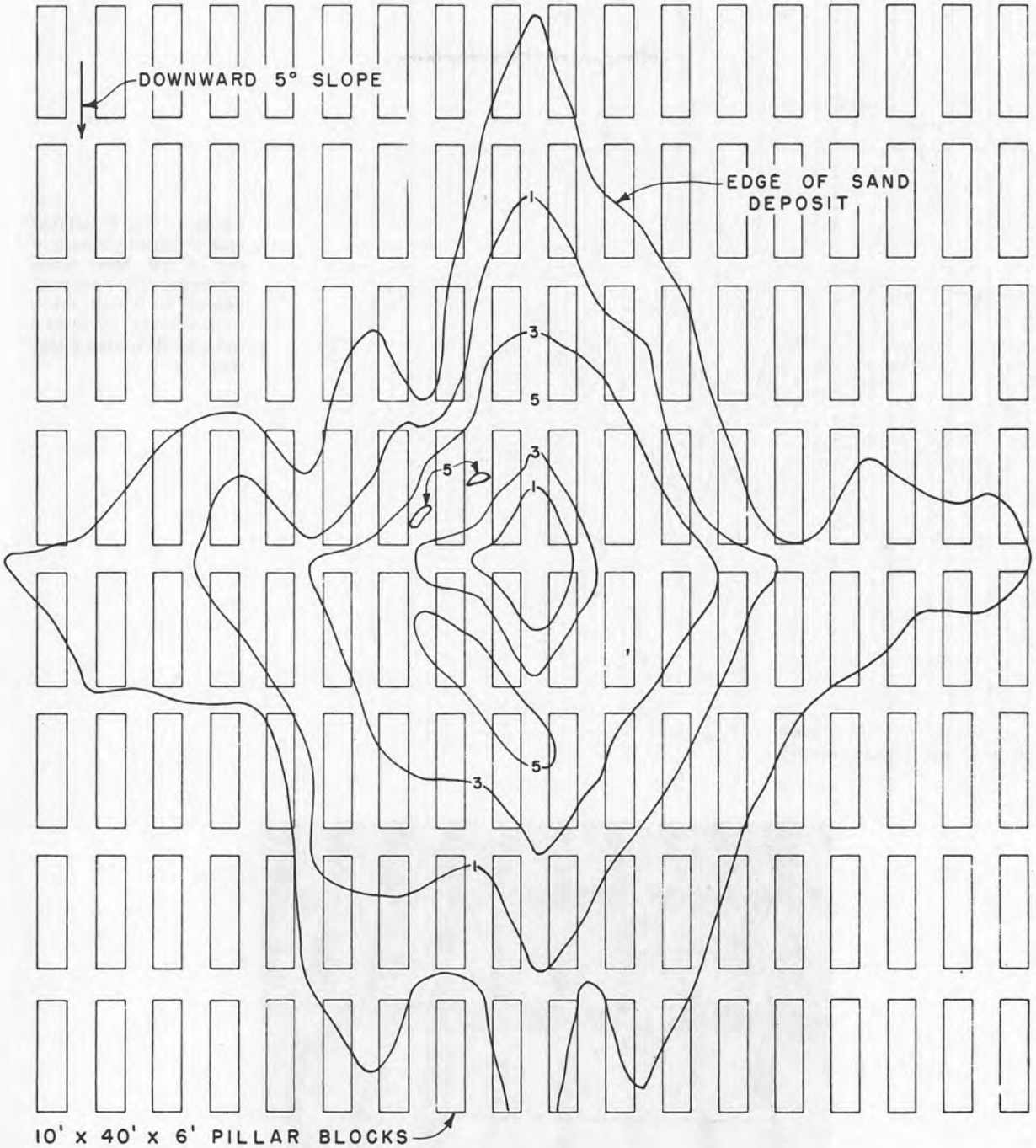


Figure 6. Test 3. Contour map of backfill deposit at end of test. Mine cavity submerged. Mine roof was bolted tight to the pillars. (Distorted model scale)

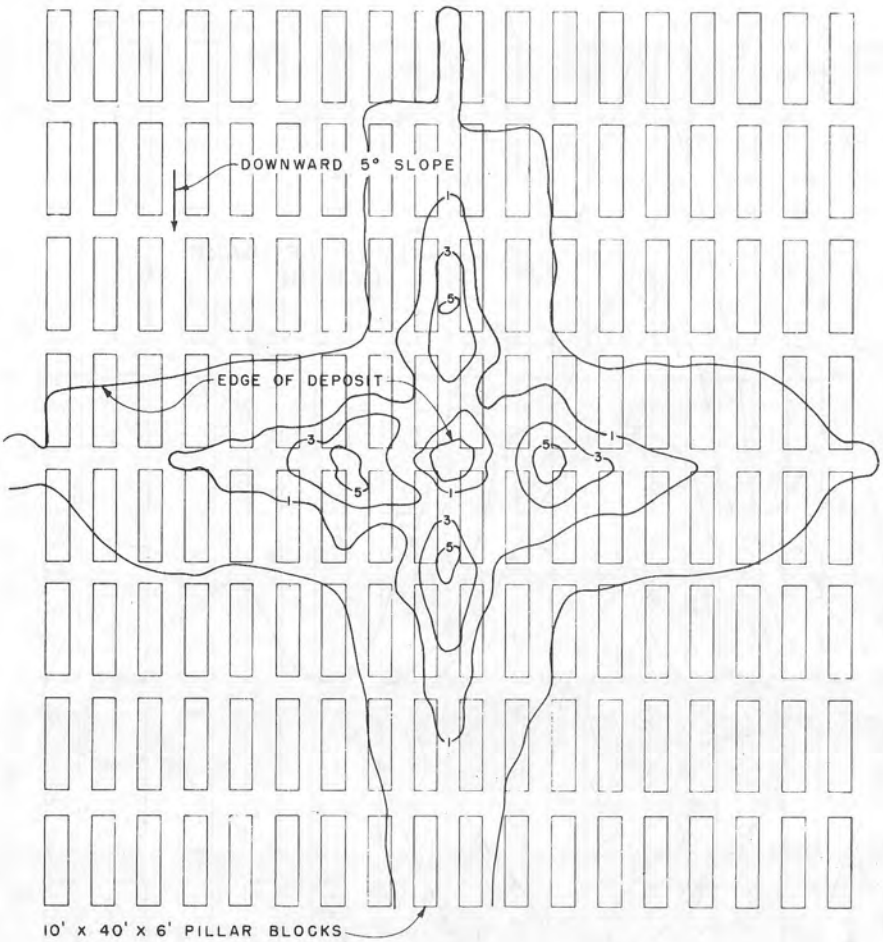


Figure 7. Test 4. Contour map of backfill deposit at end of test. Mine cavity submerged. The pattern of deposit is typical for a small amount of backfill material. (Distorted model scale)

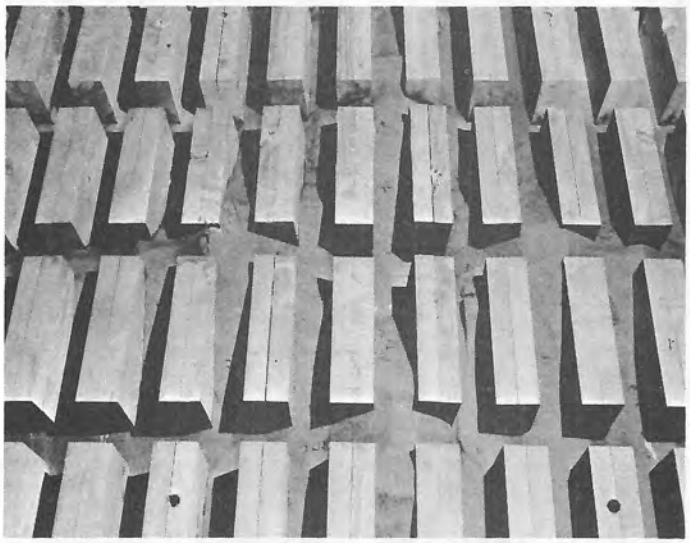


Figure 8. Test 4. The position of the pump intake was changed before test started, causing only a small deposit in the mine cavity during the test. Mine submerged and dipping 5°. (Distorted model scale)

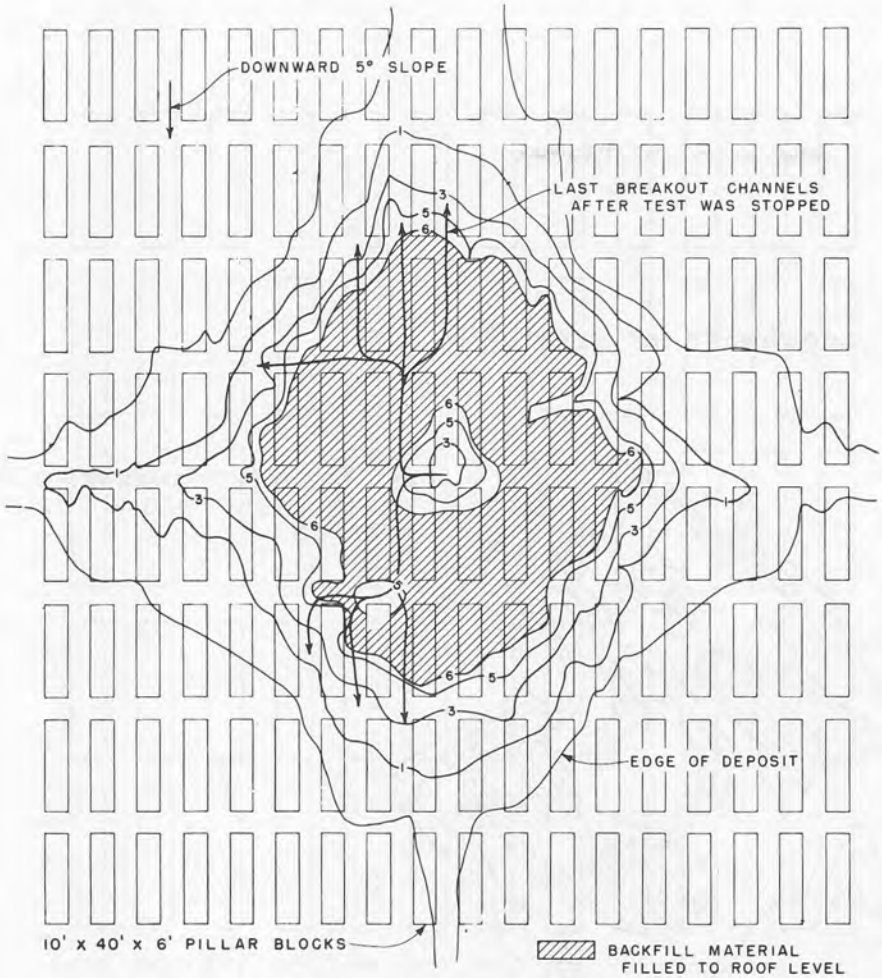


Figure 9. Test 5. Continuation of test 4. Contour map of deposit pattern at end of test. Mine cavity submerged. Compare deposit pattern with figure 7. (Distorted model scale)

Figure 10. Test 5. This test was a continuation of test 4 in which the backfill deposit was allowed to continue. Mine submerged and dipping 5°. (Distorted model scale)

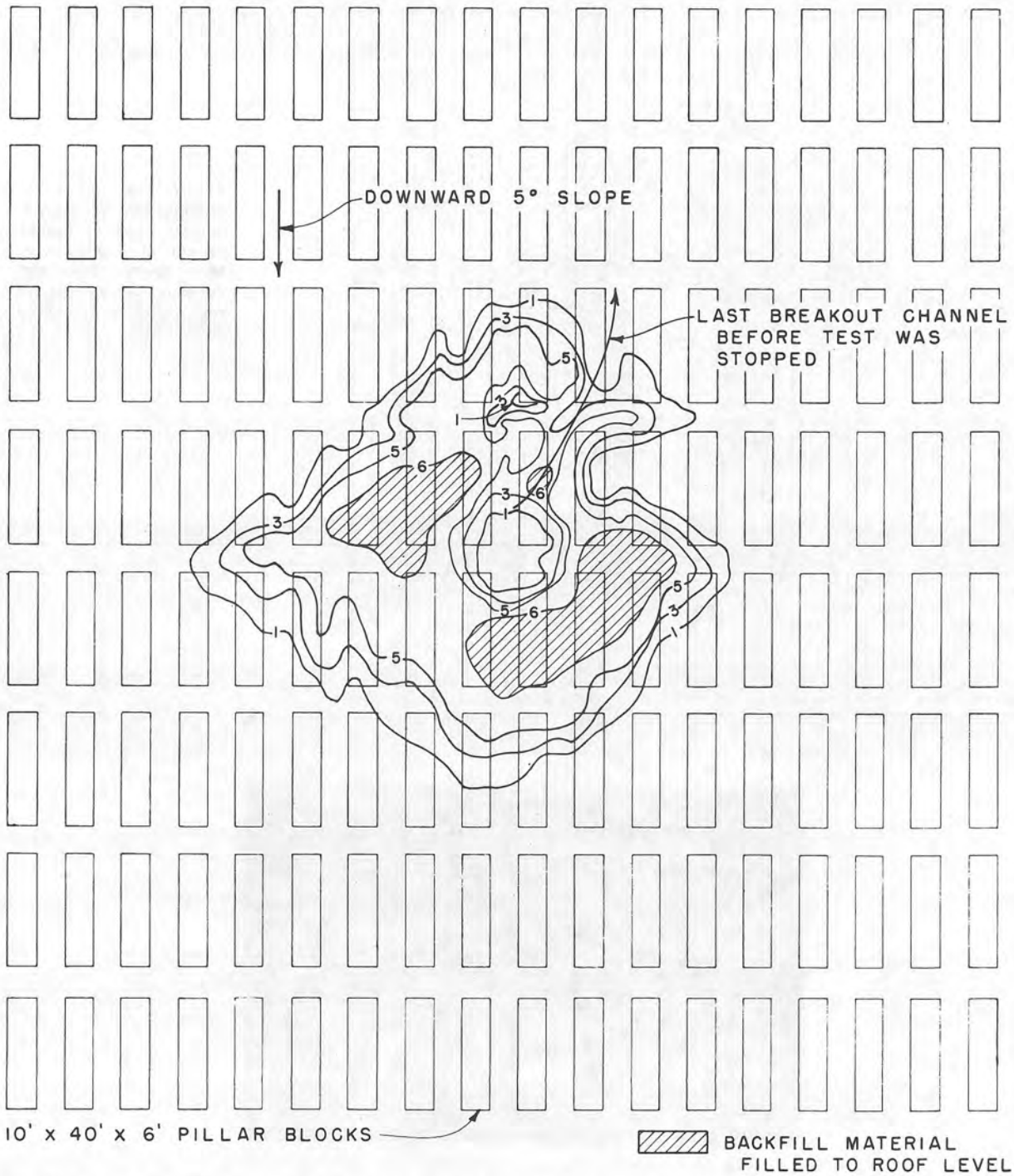


Figure 11. Test 7. Contour map of backfill deposit at end of test. Mine submerged. Inside diameter of injection pipe was increased from 1/2 to 3/4 inch for this and later tests. (Model scale was undistorted for this and all later tests)

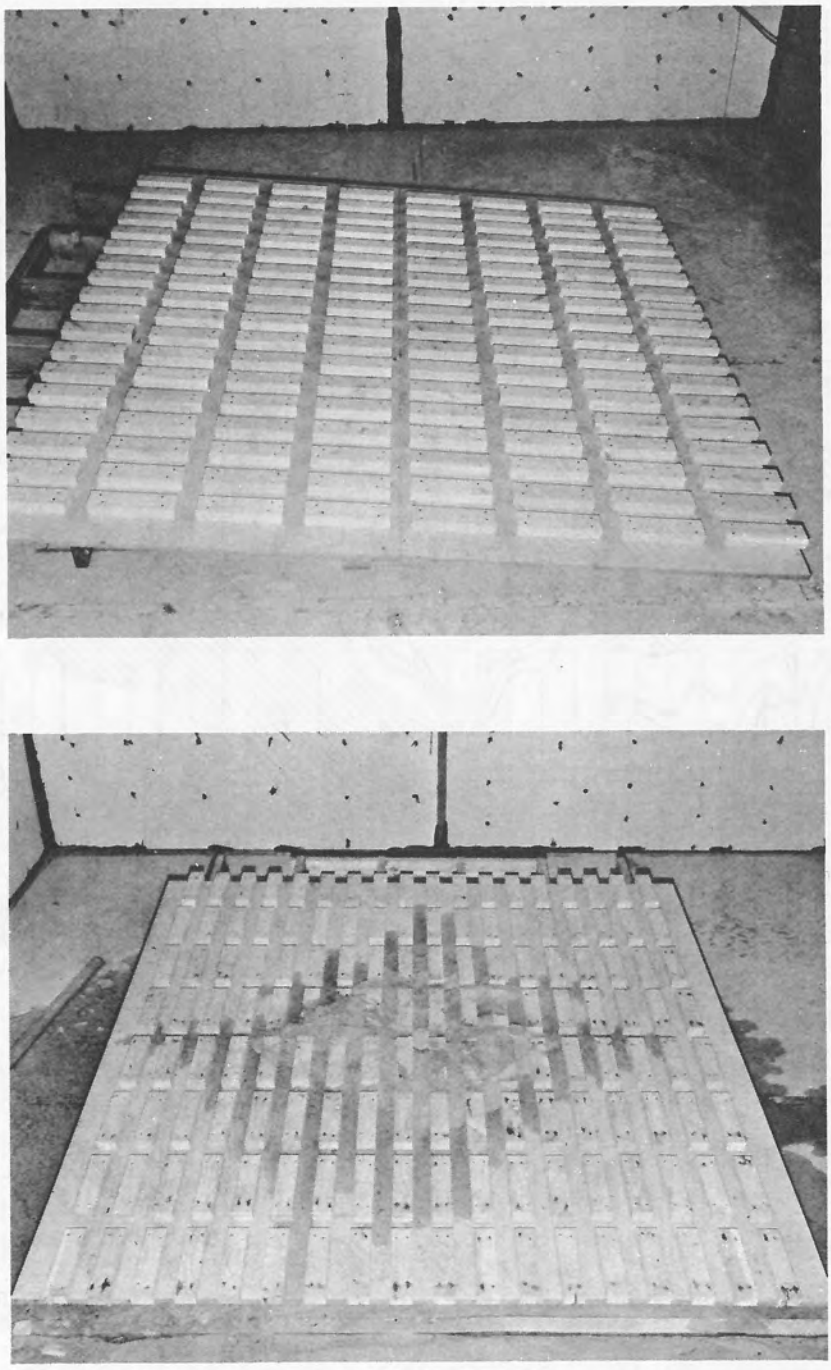


Figure 12. Test 7. Mine pillars and cavity were changed to give an undistorted scale $1^m:48^p$ for this test and all later tests. Mine submerged and dipping 5°.

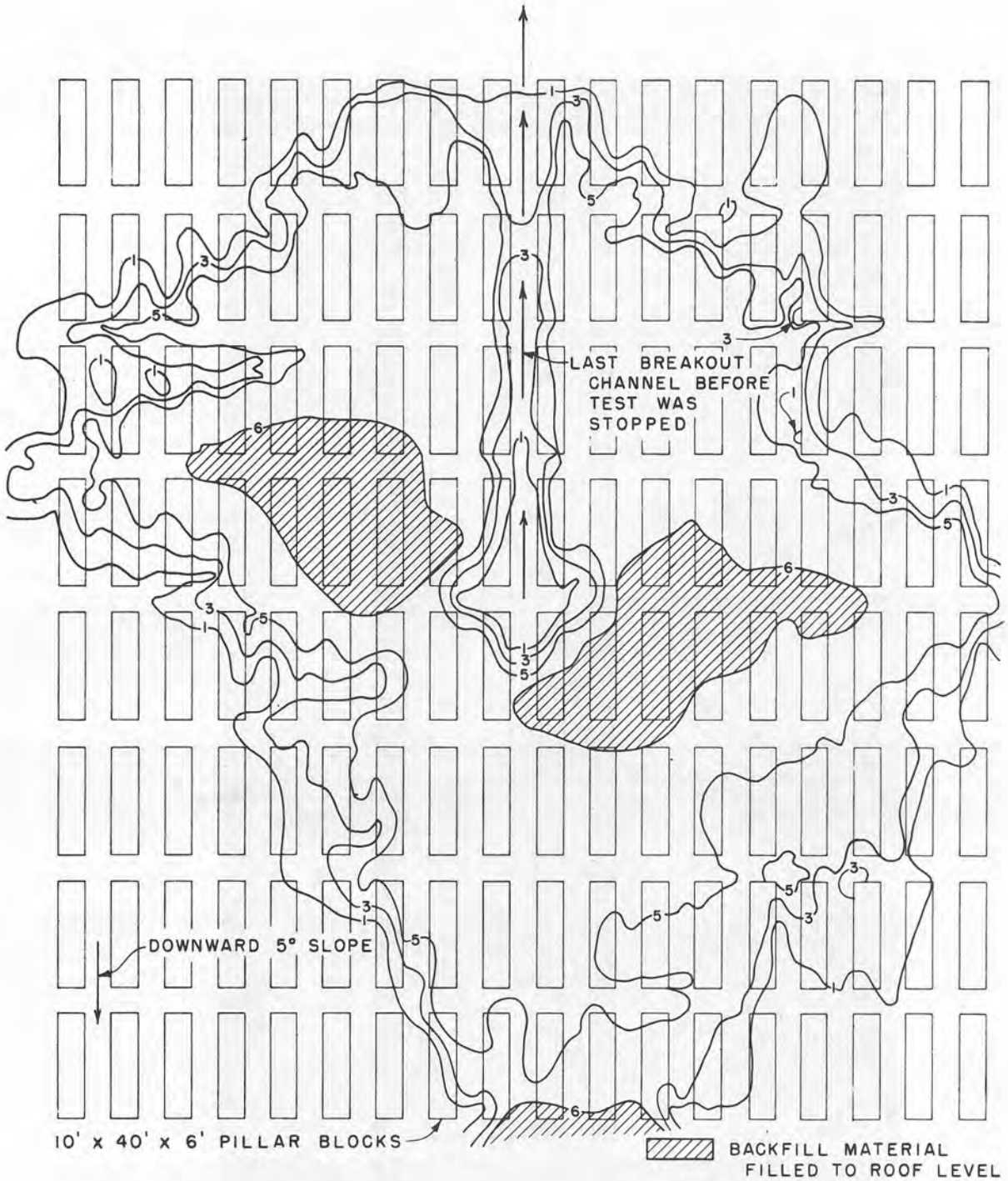


Figure 13. Test 8. Contour map at end of test. Mine submerged. Flow along the last breakout channel occurred at the end of the test. Full flow of the injection pipe down one channel caused equilibrium conditions for sediment transport in one channel to occur. Note that breakout channel flows upslope.

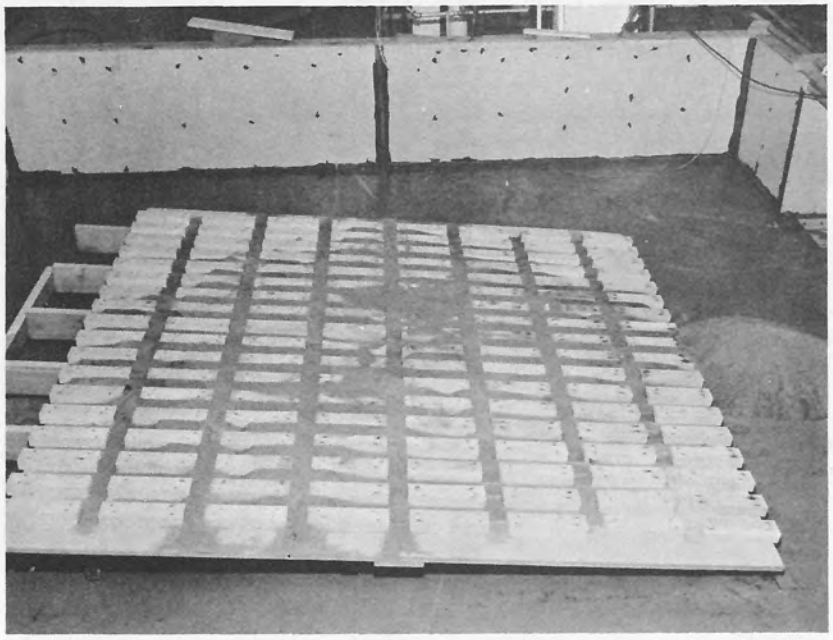


Figure 14. Test 8. Backfill material filled the mine cavity up to or near the roofline over a comparatively large area, see figure 13. The slurry concentration was 11.4 percent by weight for this test. Mine submerged and dipping 5°.

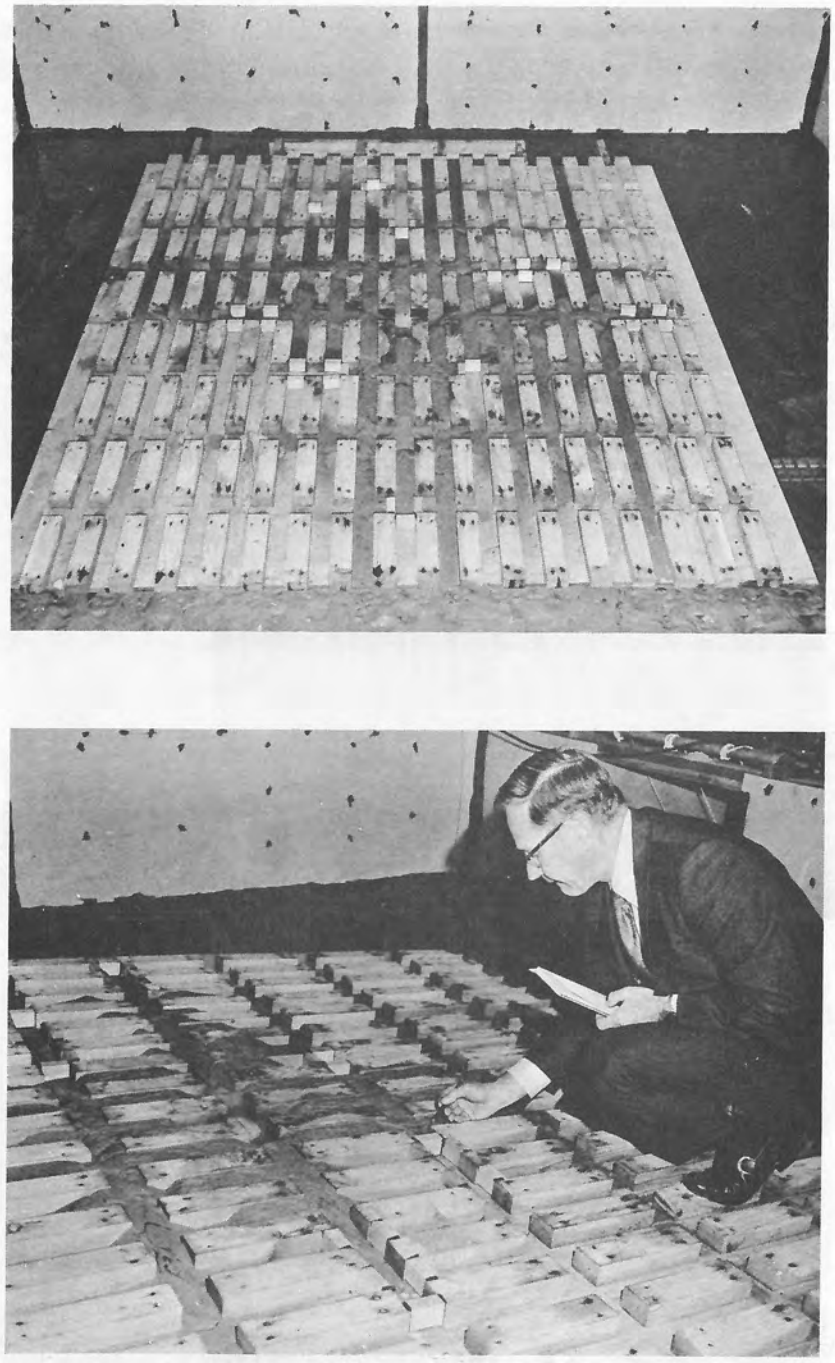


Figure 15. Test 9. First test in the series of three tests, 9–11. Blind entries were simulated by blocking corridors at various places. Mine submerged and dipping 5°.

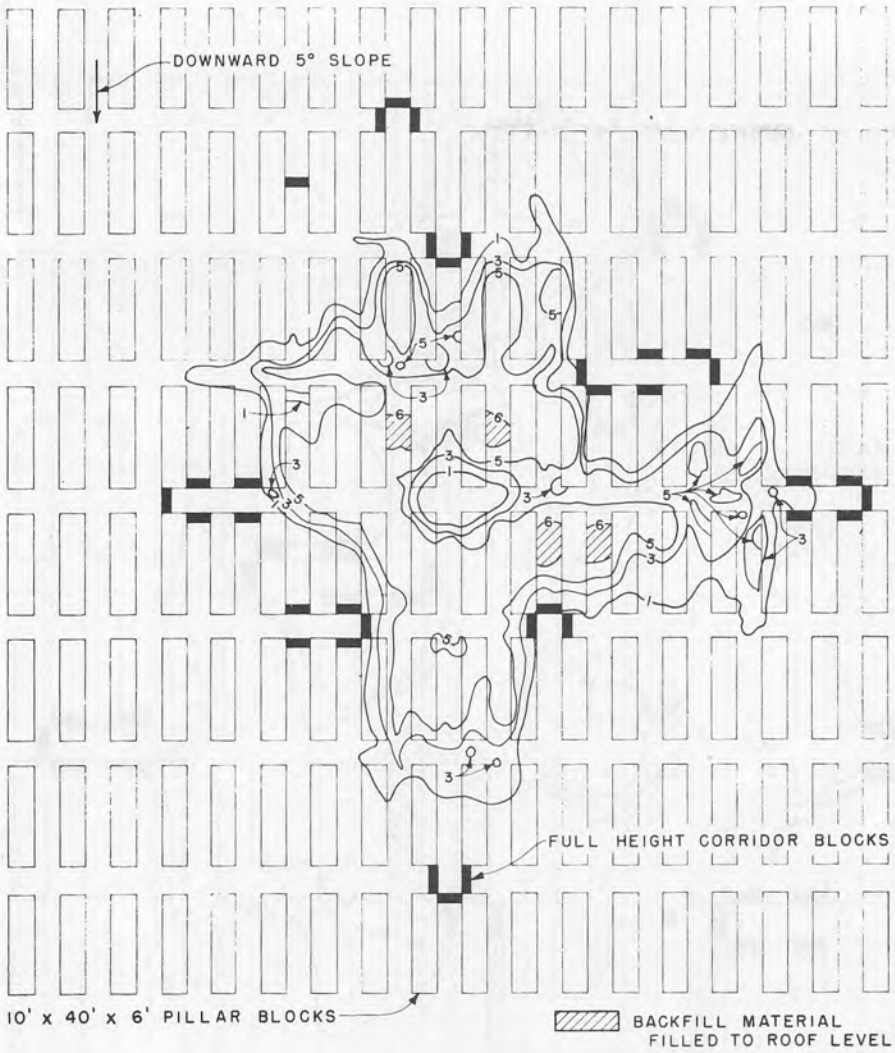


Figure 16. Test 9. Location and pattern of blind entries simulated by solid blocks in the corridors. Contour map shows deposit pattern of backfill at the end of test 9.

Figure 17. Test 11. Corridor 8 is pointed out in which the total injection flow was concentrated after backfill material filled to the roof around the injection hole. Mine submerged and dipping 5°. Last test in series of three tests, 9-11.

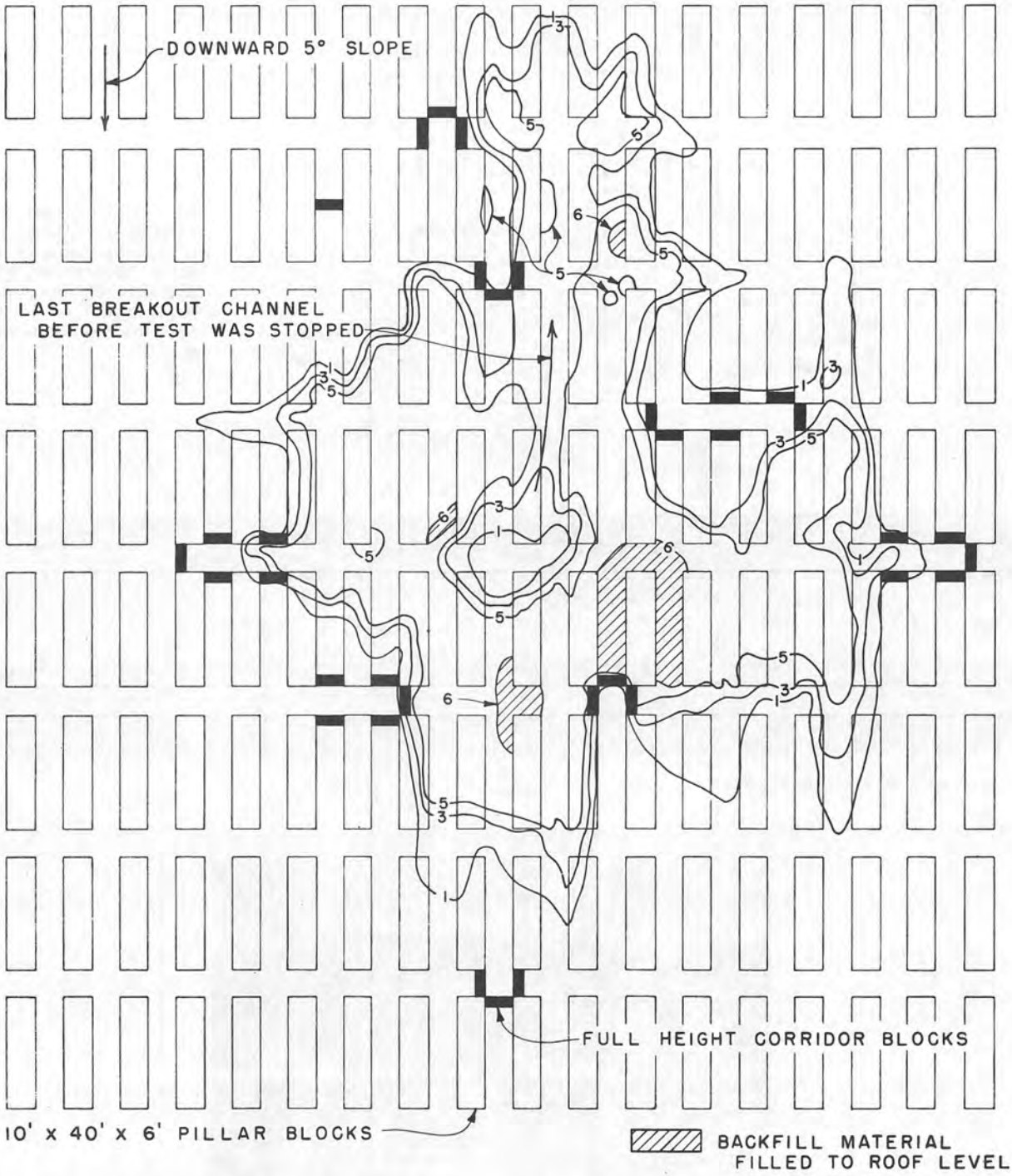


Figure 18. Test 11. Contours show deposit pattern of backfill material at the end of the test. The location and pattern of blind entries affect the general pattern of backfill deposit.

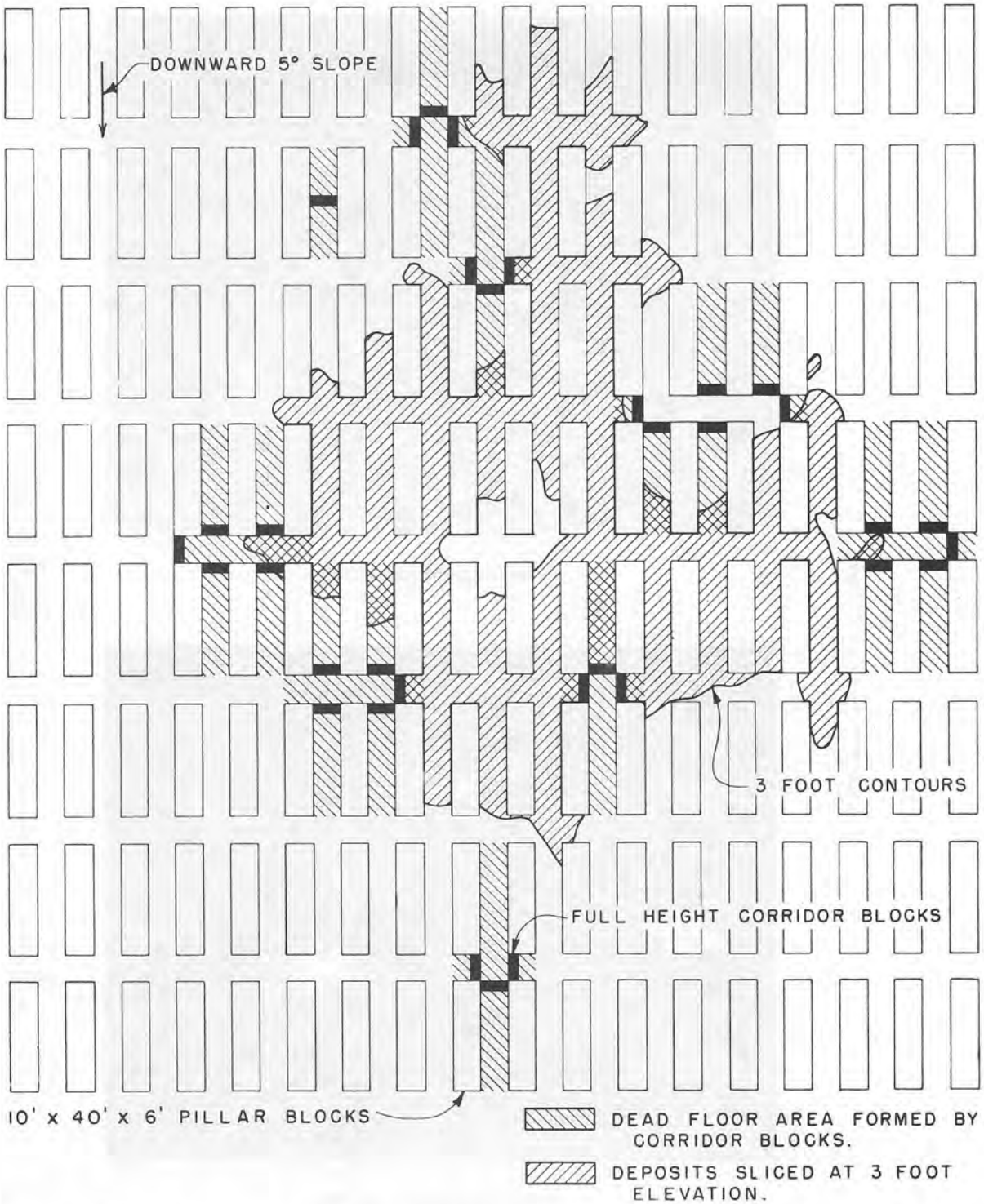
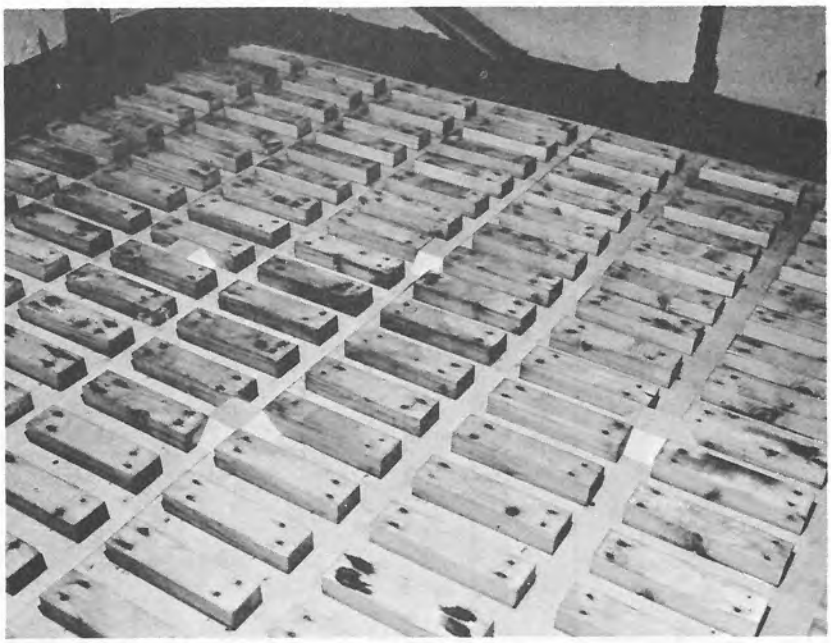
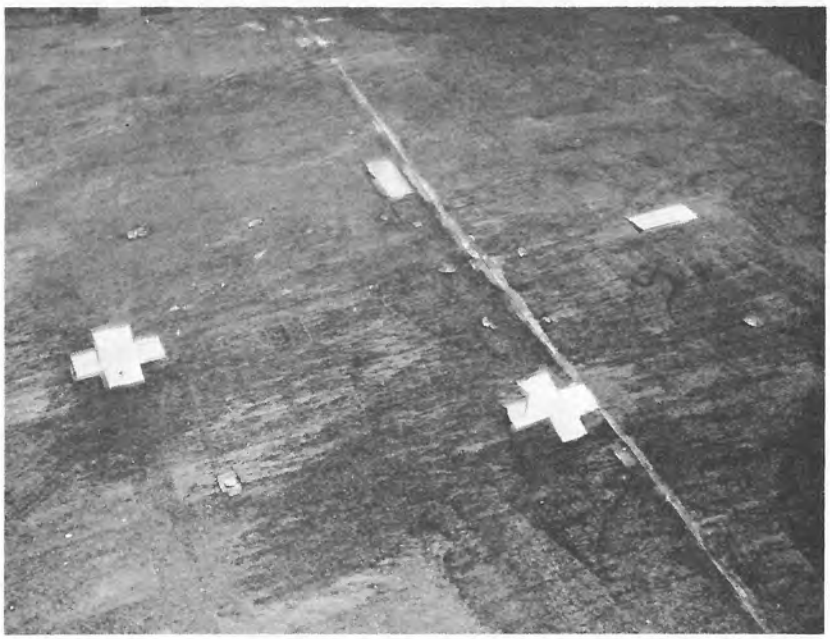


Figure 19. Test 11. Deposits above the 3-foot contour elevation cover a large area around the injection hole. Overlapping crosshatching indicates entry of fill material into blind entries.



Pillars and simulated roof falls.



Roof and openings for cavities.

Figure 20. Test 17. Mine pillars and roof before tests simulating roof falls and cavities over the roof falls.

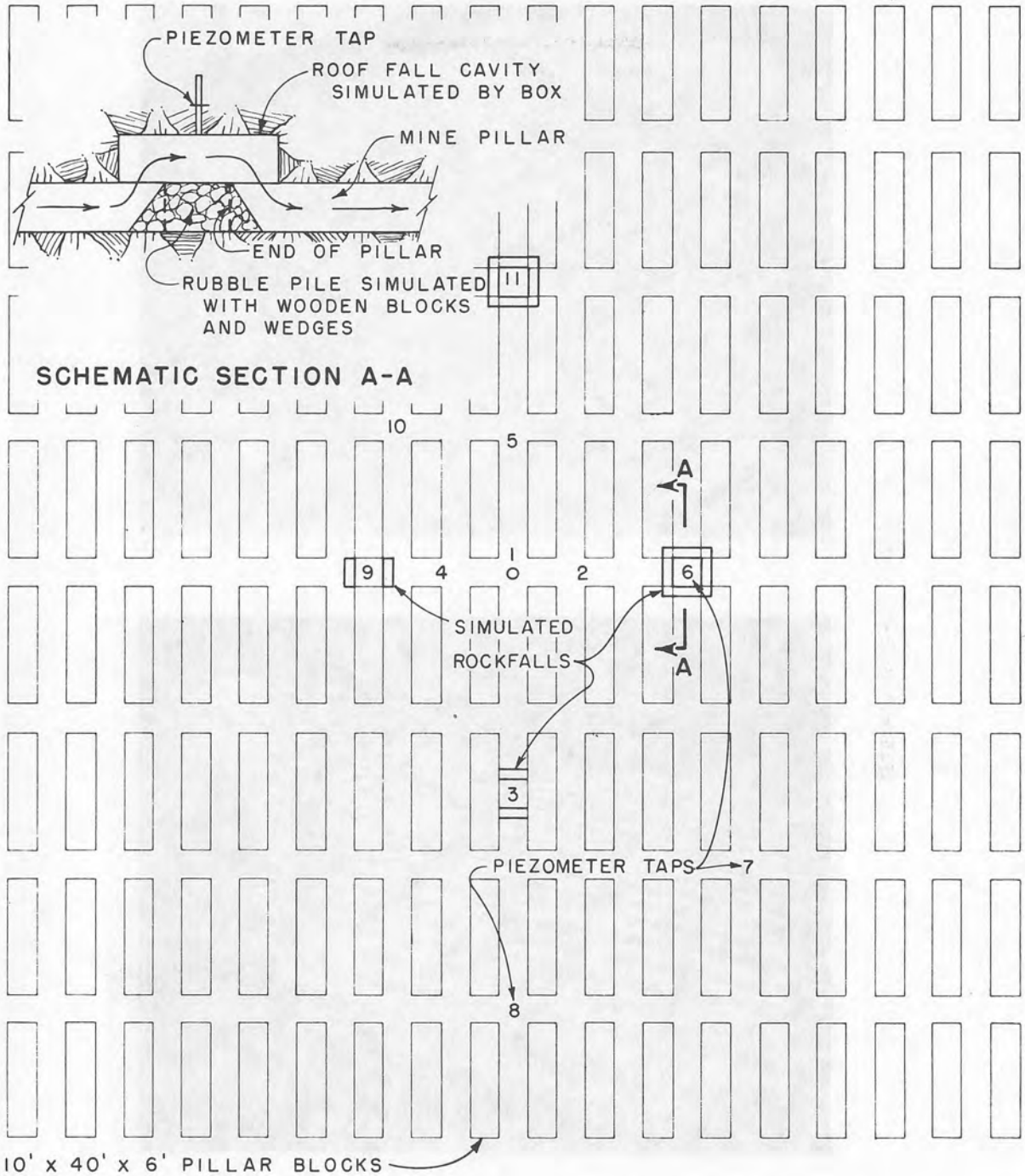


Figure 21. Test 17. Cross section and layout for tests with roof falls and roof cavities over roof falls.

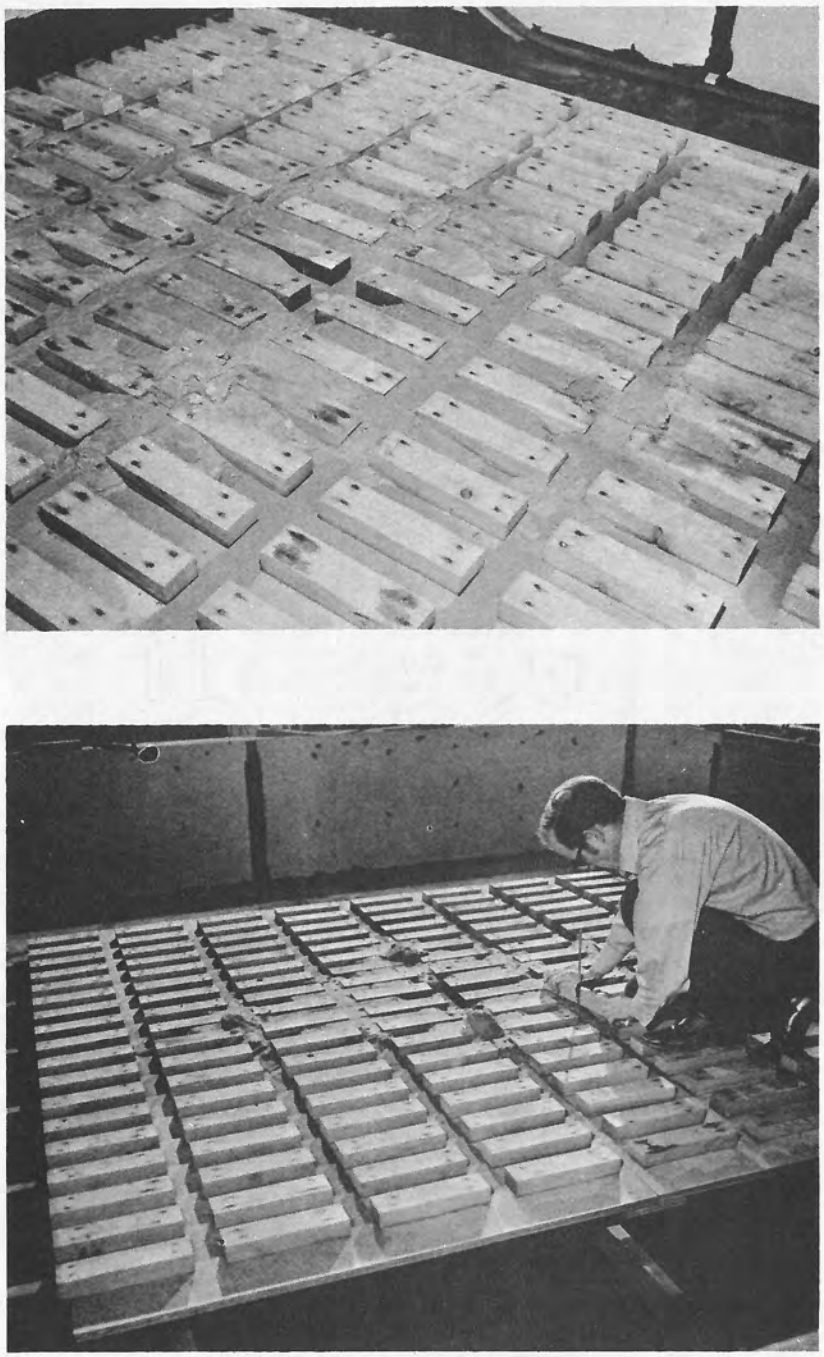


Figure 22. Test 17. Deposit pattern of backfill at the end of the test. Note backfill deposited in cavities over roof falls.

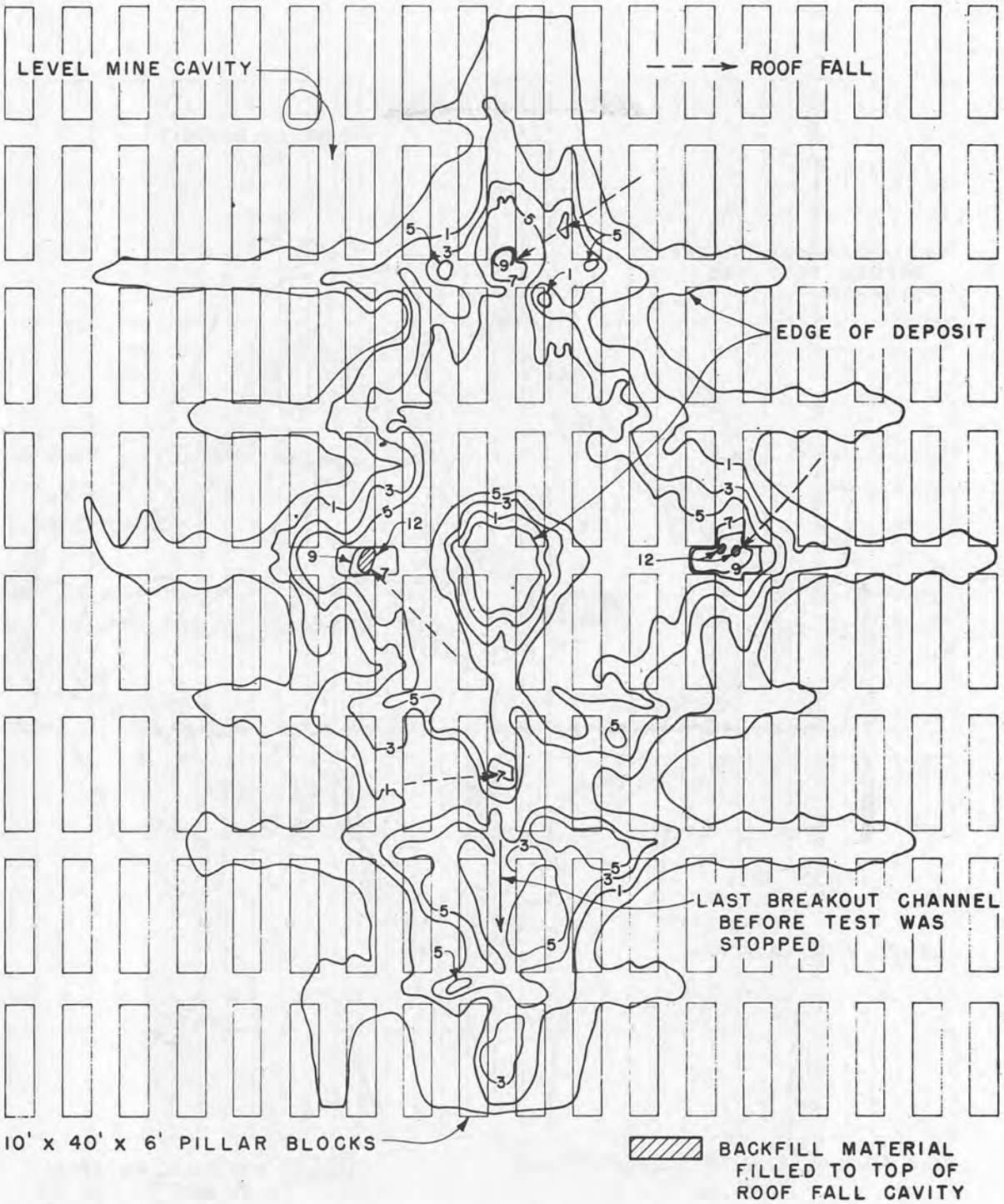


Figure 23. Test 17. Contours of deposit pattern at the end of the test. Last breakout channel flowed over a roof fall.

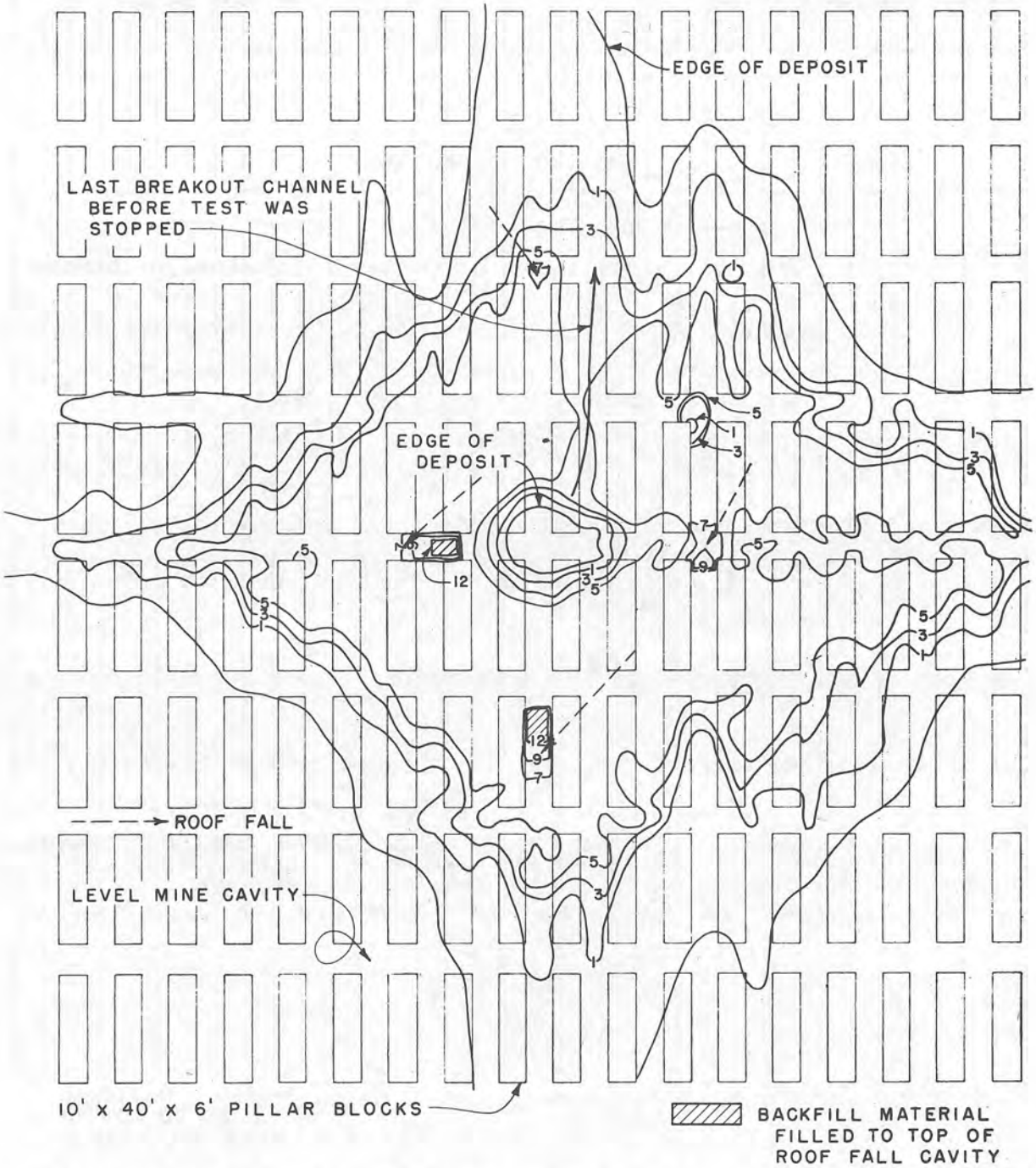


Figure 24. Test 18. This test was a duplicate of test 17 except test 18 had a higher injection velocity and a higher slurry concentration. Note breakout channel in a corridor adjacent to a rockfall.

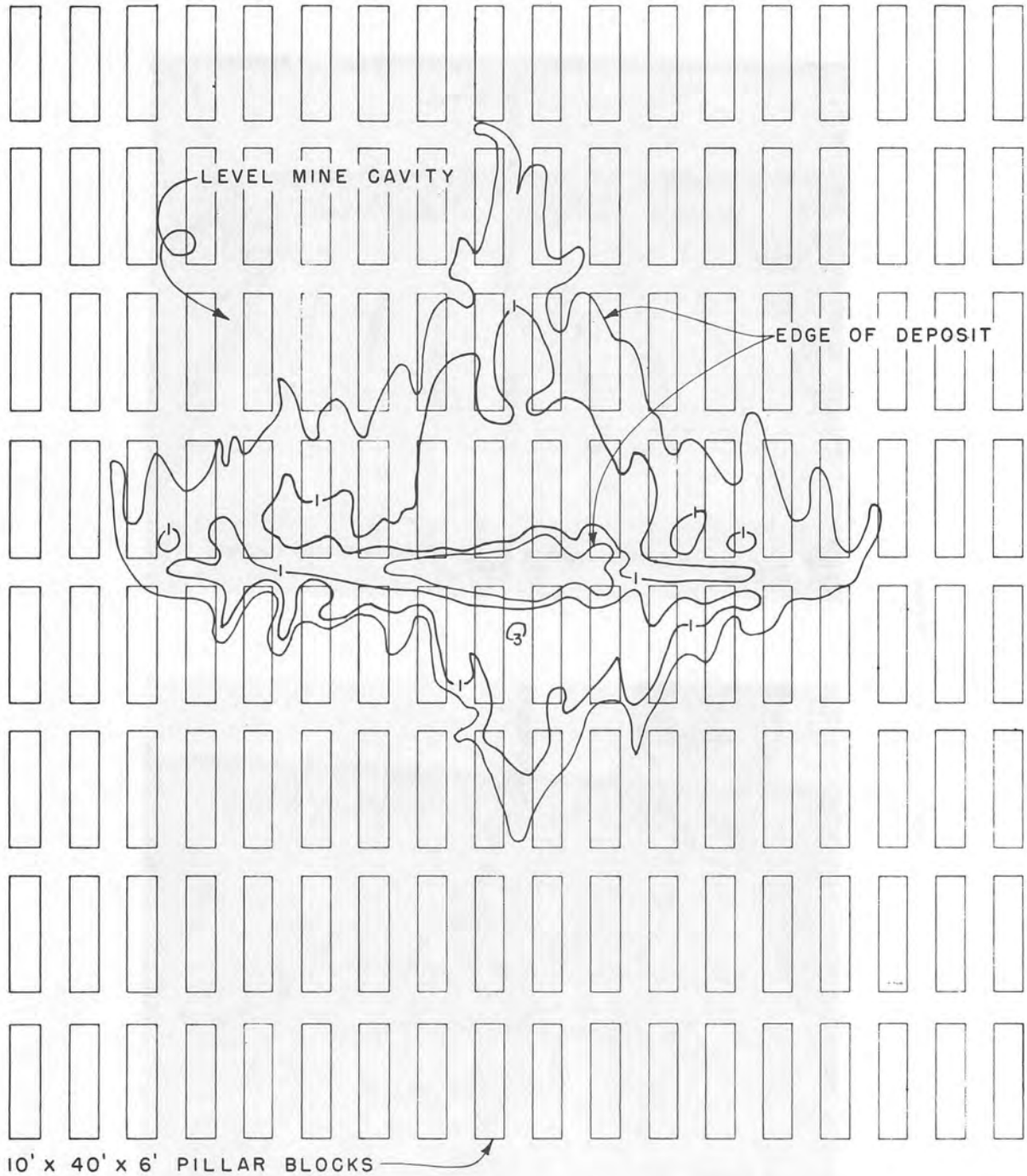


Figure 25. Test 12. The first of a series of four tests, 12-15, made in a dry cavity with a level floor. Contour map shows backfill deposit pattern at the end of the test.

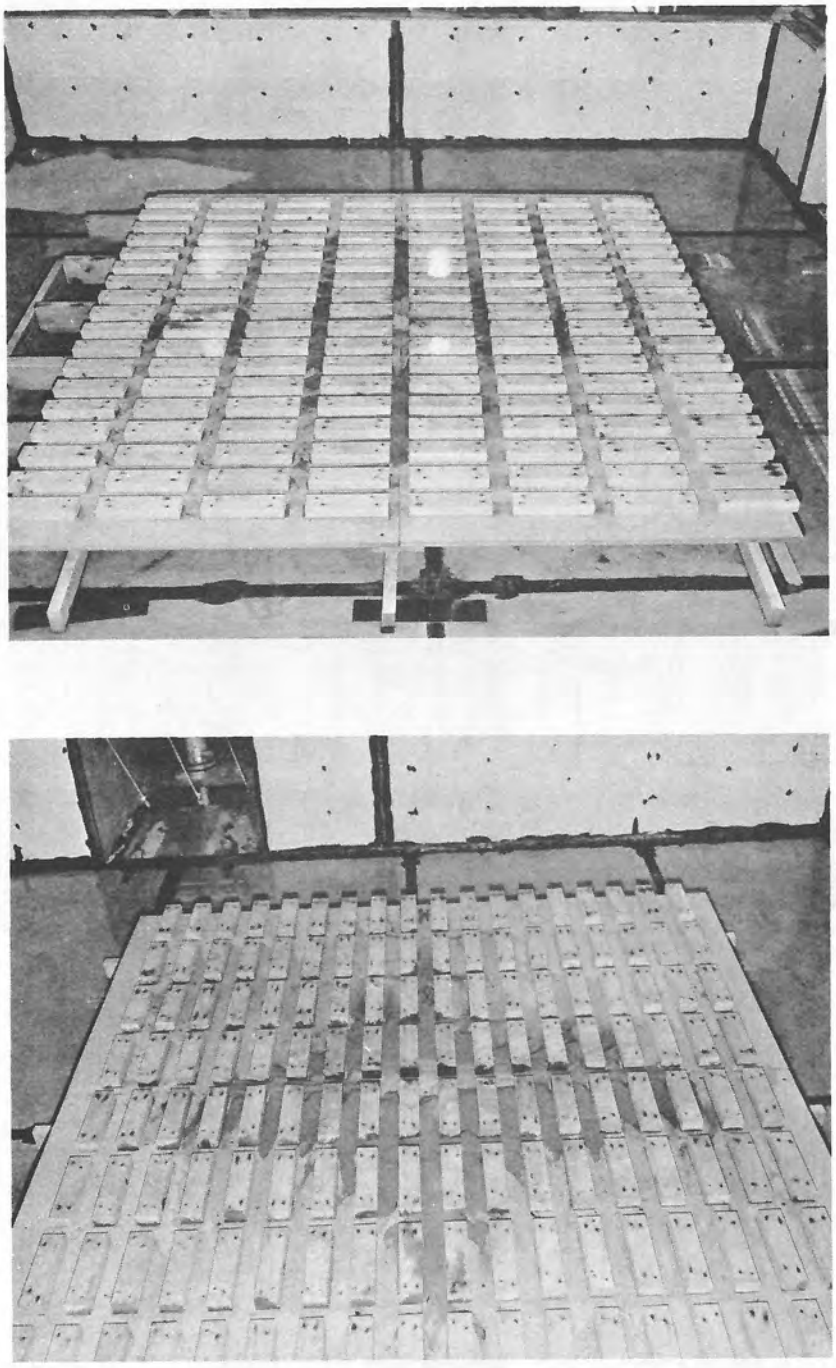


Figure 26. Test 12. The low height of deposit is a result of injection into a dry cavity with a level floor.

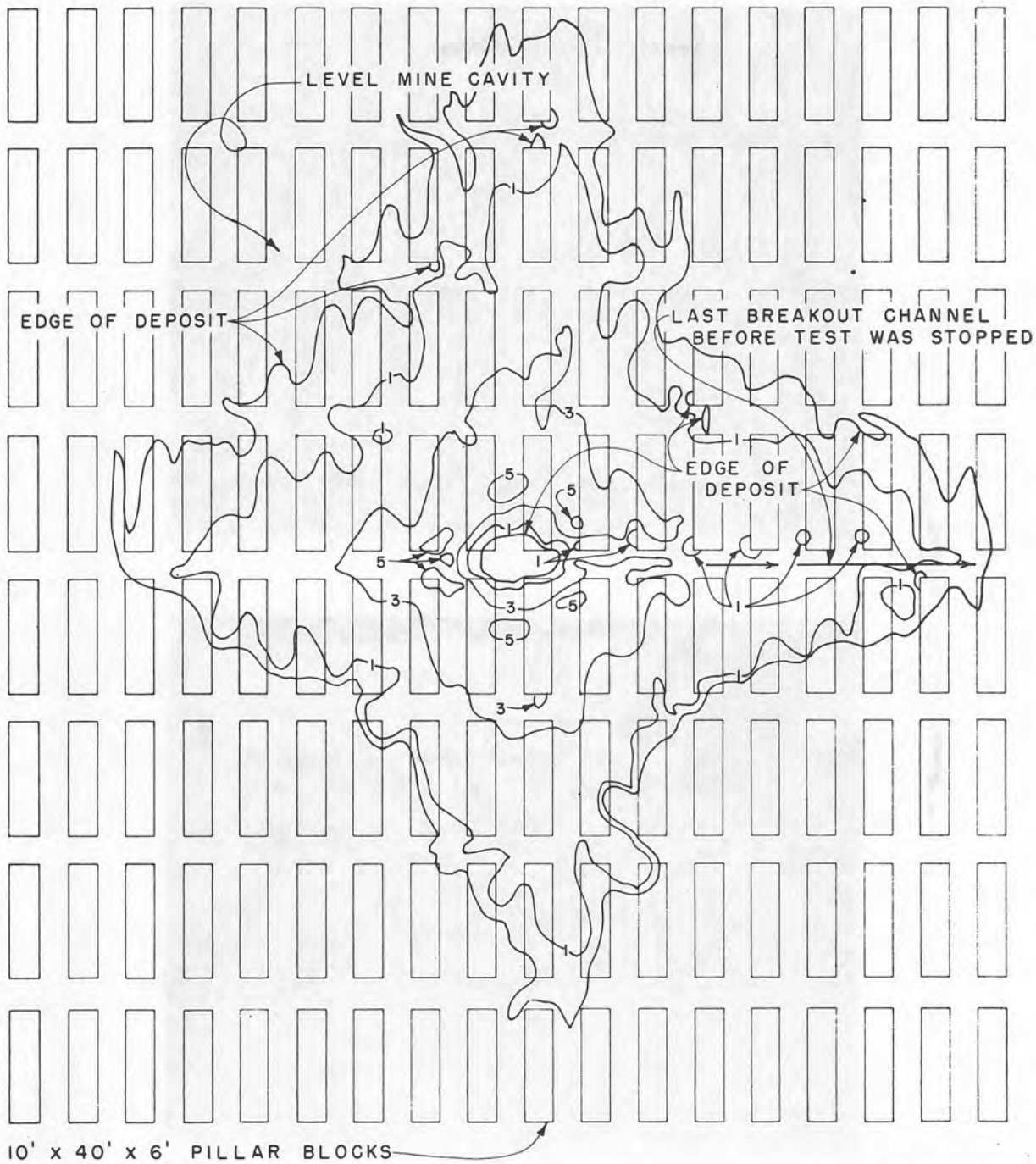


Figure 27. Test 13. The second in the series of four tests with slurry injected into a dry cavity with a level floor. Contours show accumulated deposit pattern.

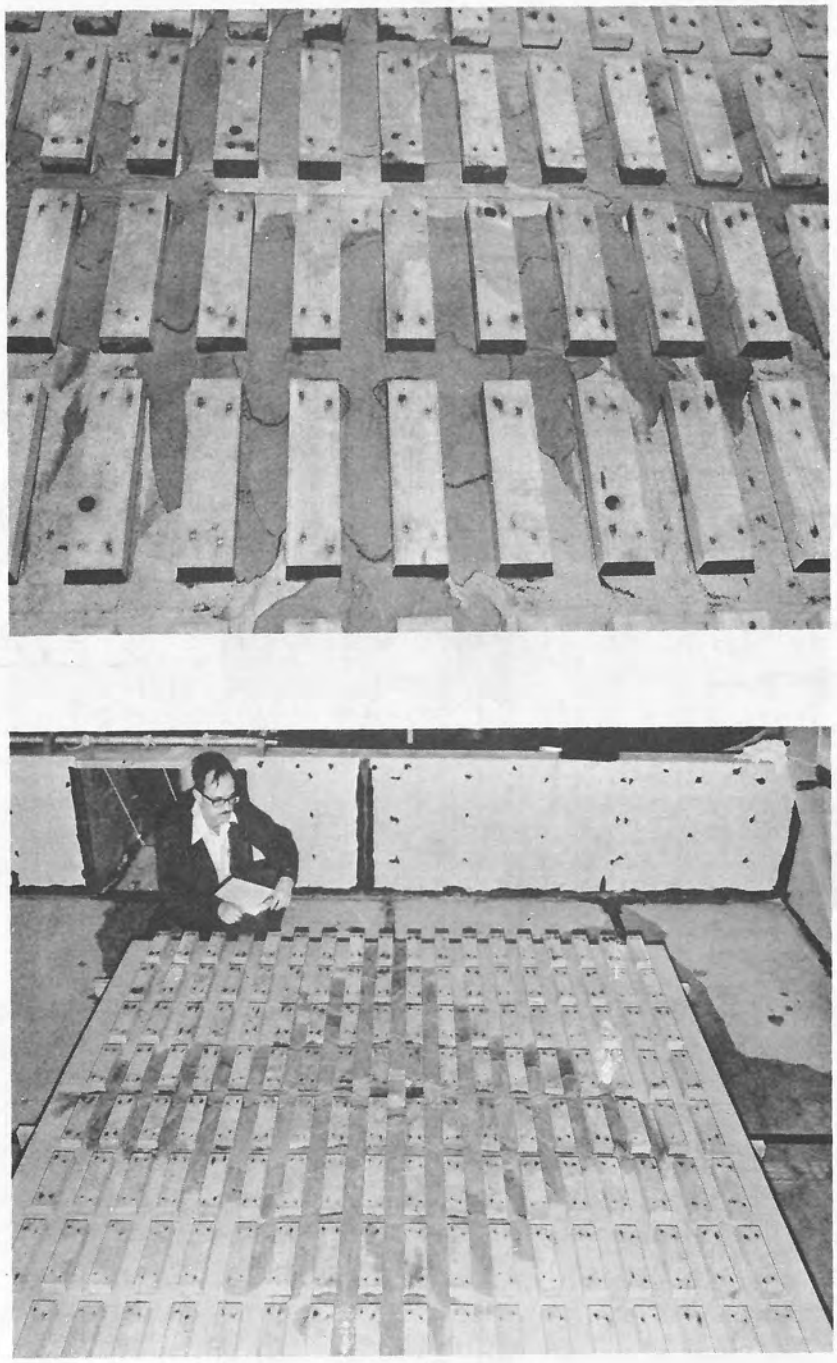


Figure 28. Test 13. Accumulated deposit after the second in the series of four tests shows shallow deposits and flat slopes on the surface of the deposited backfill material.

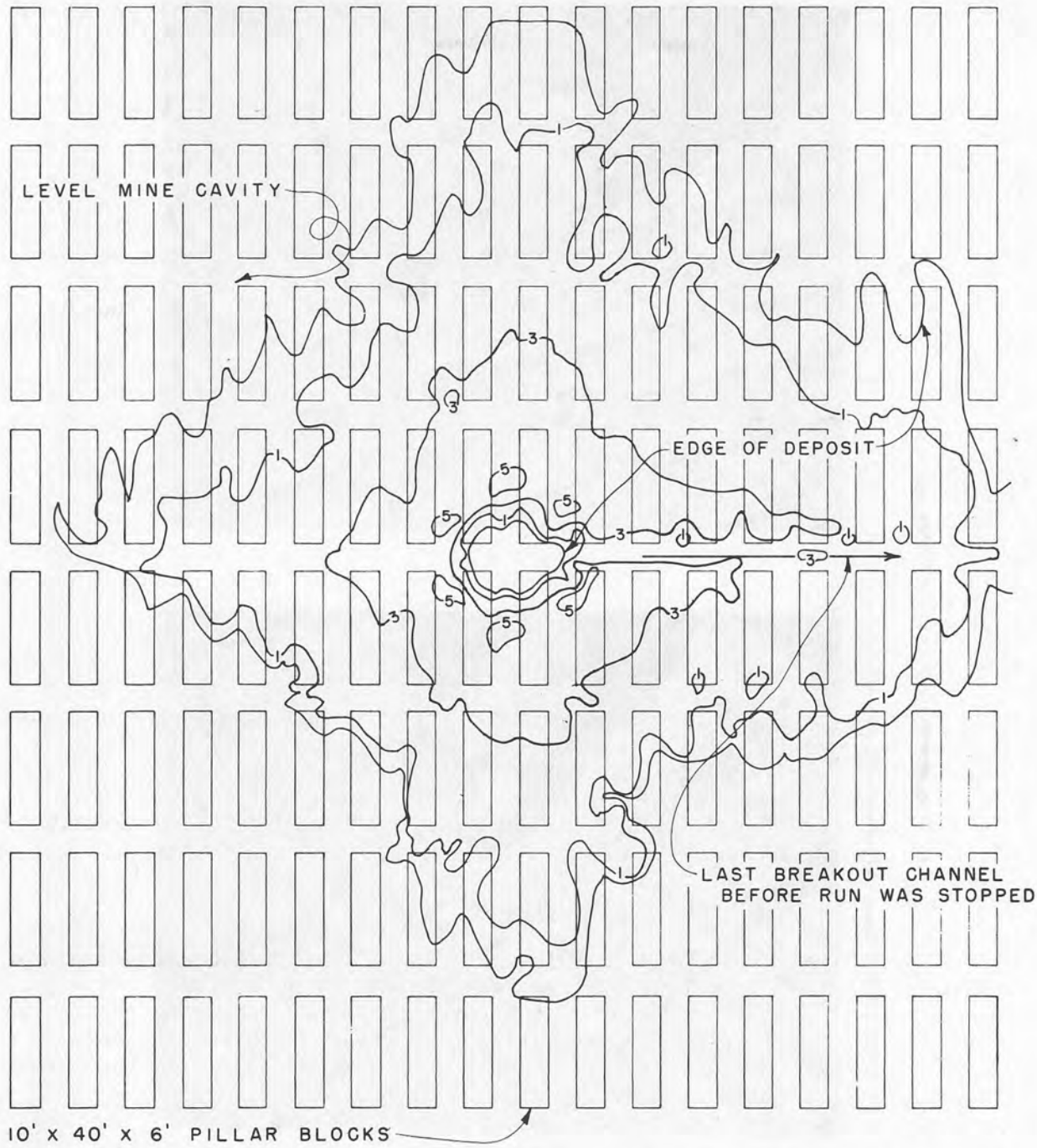


Figure 29. Test 14. The backfill material continues to build up as the third in the series of four tests is completed in a dry cavity with a level floor.

Figure 30. Test 14. Deposit around the injection hole is uniform in a dry cavity with a level floor.

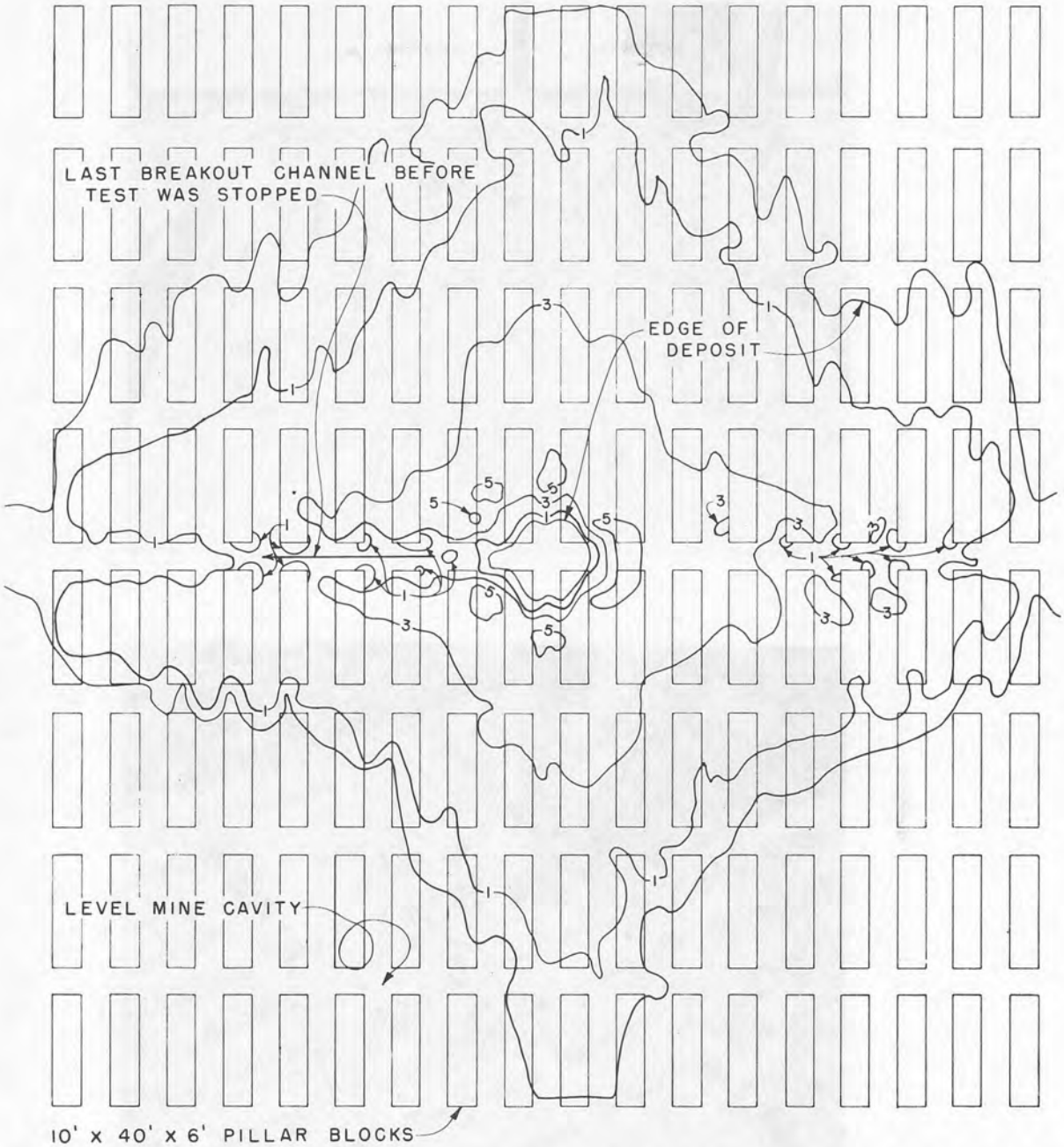


Figure 31. Test 15. Contour map showing backfill deposit at the end of the series of four tests. Note the last breakout channel before the end of the test.

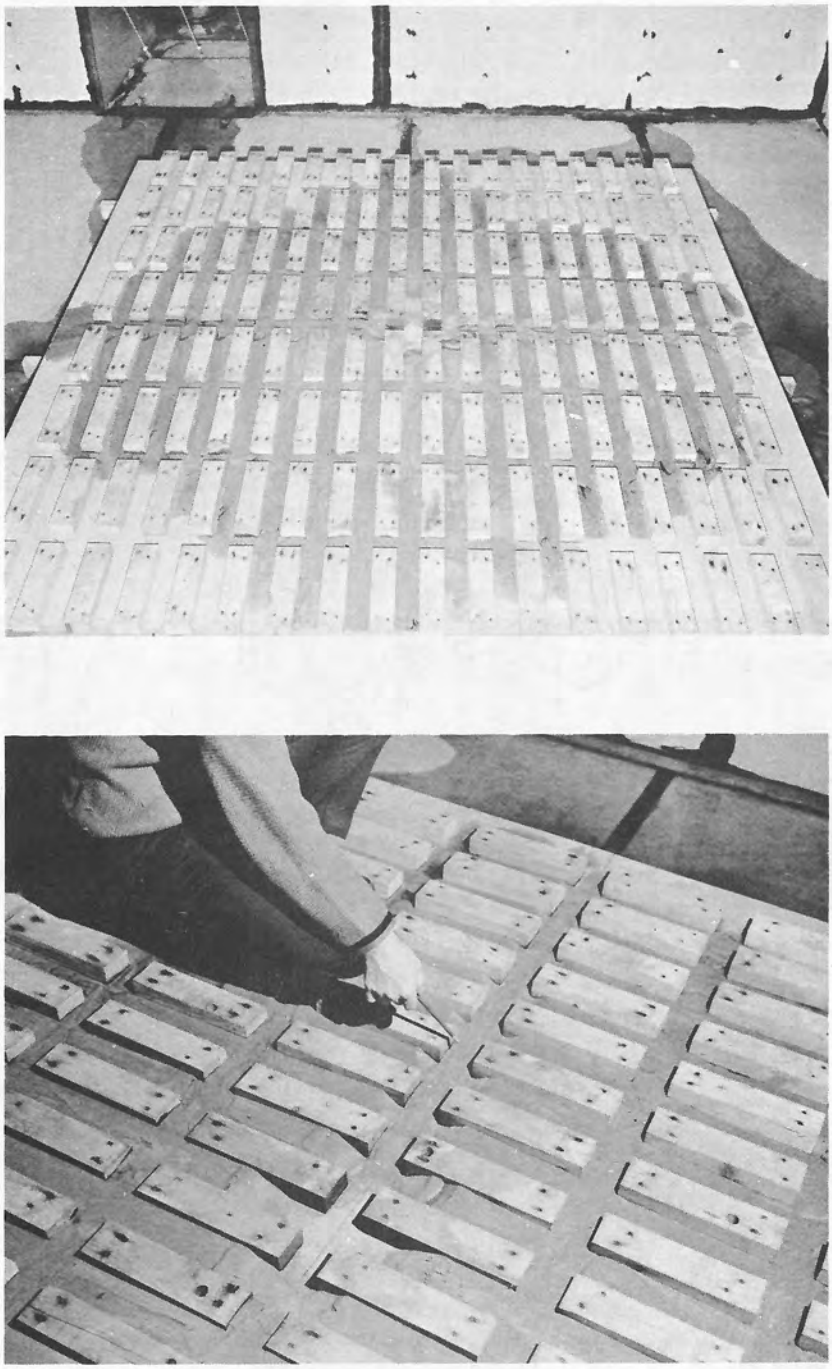


Figure 32. Test 15. Backfill material has built up near the roof close to the injection point at the end of the series of four tests in a dry mine with a level floor.

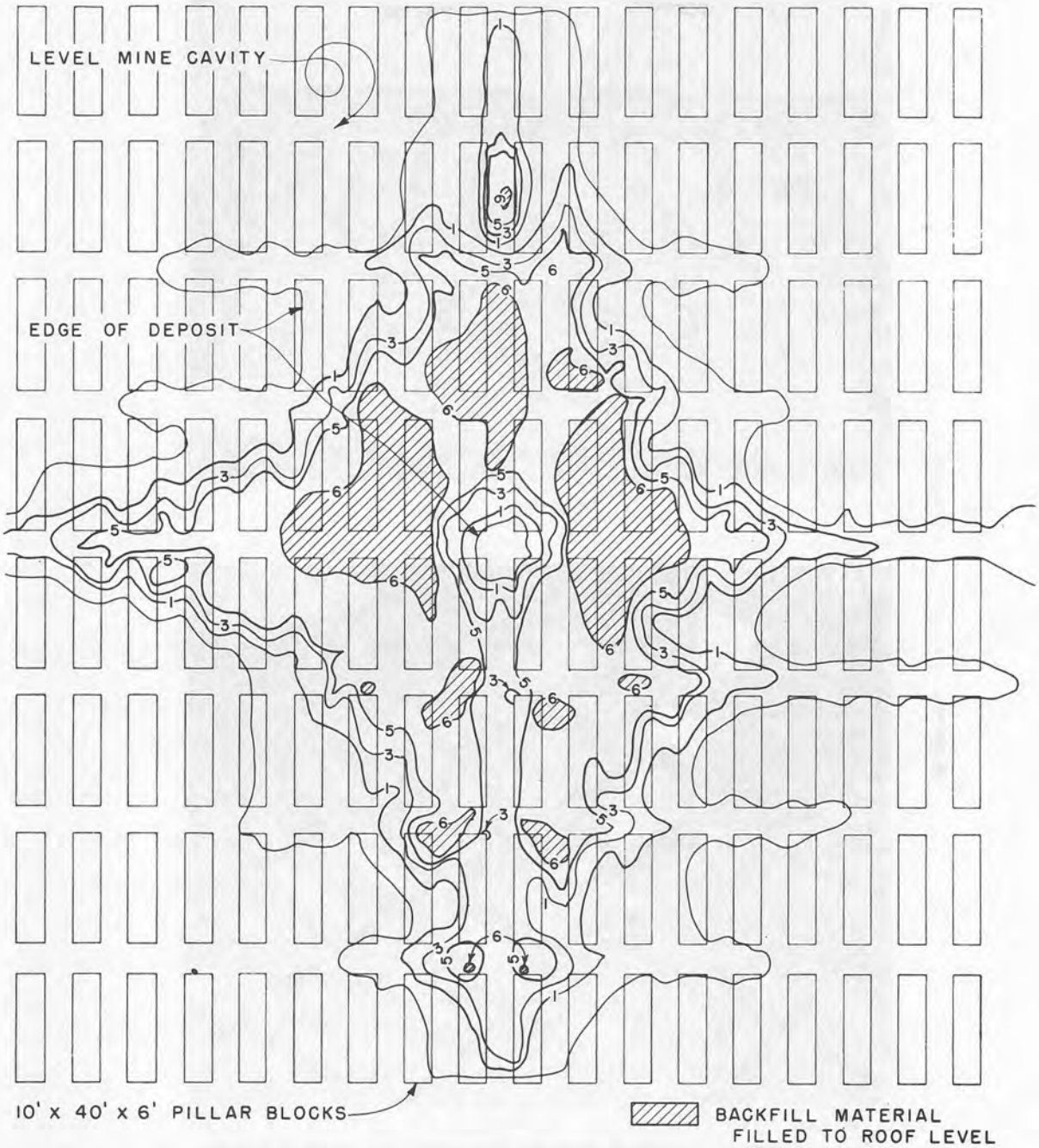


Figure 33. Test 16. Contour map of deposited fill material at end of test. Compare the deposit pattern on this level mine test with test 8 (fig. 13) in a sloping mine.

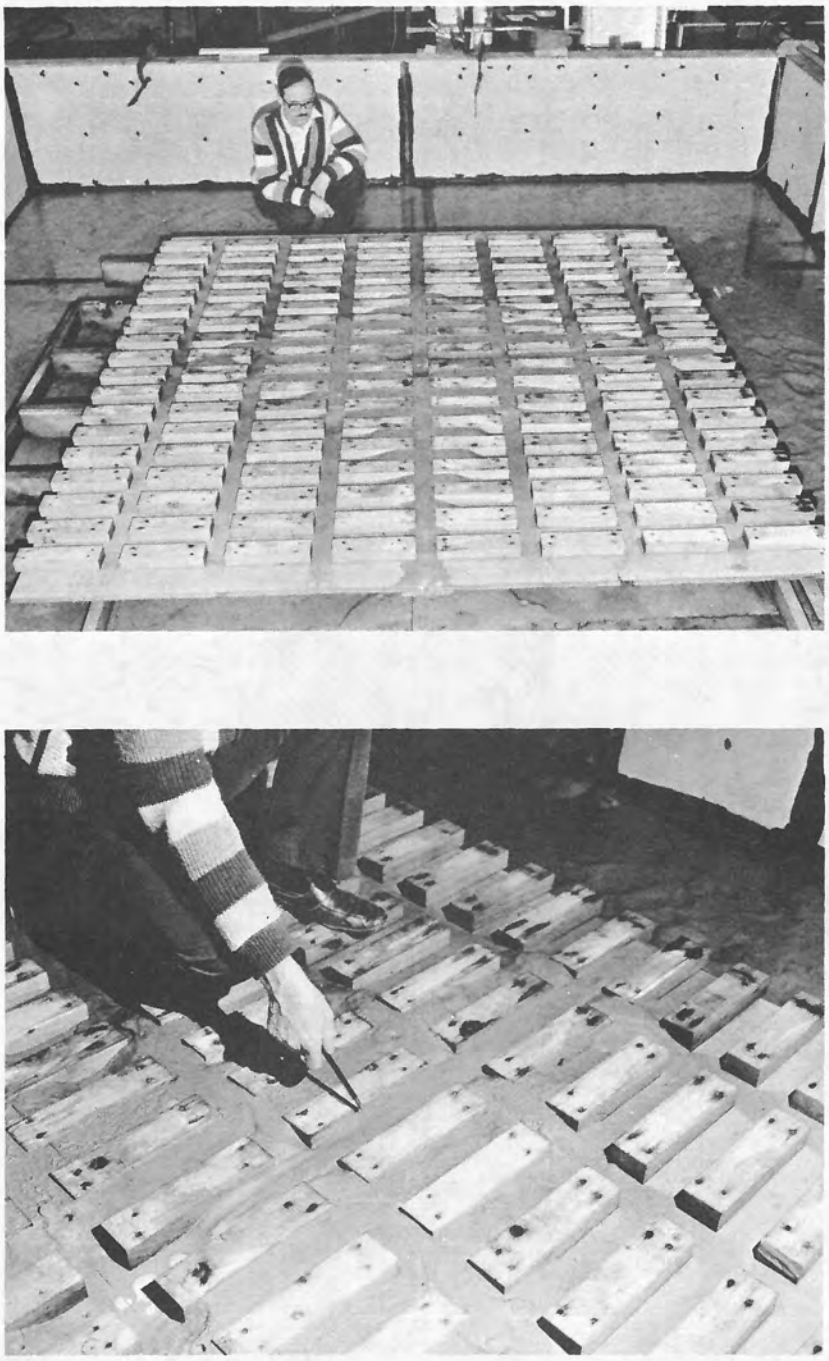


Figure 34. Test 16. the final breakout channel is pointed out on the photograph. Test was for a mine with a level floor and submerged.

7-1750 (3-71)
Bureau of Reclamation

CONVERSION FACTORS—BRITISH TO METRIC UNITS OF MEASUREMENT

The following conversion factors adopted by the Bureau of Reclamation are those published by the American Society for Testing and Materials (ASTM Metric Practice Guide, E 380-68) except that additional factors (*) commonly used in the Bureau have been added. Further discussion of definitions of quantities and units is given in the ASTM Metric Practice Guide.

The metric units and conversion factors adopted by the ASTM are based on the "International System of Units" (designated SI for Systeme International d'Unites), fixed by the International Committee for Weights and Measures; this system is also known as the Giorgi or MKSA (meter-kilogram (mass)-second-ampere) system. This system has been adopted by the International Organization for Standardization in ISO Recommendation R-31.

The metric technical unit of force is the kilogram-force; this is the force which, when applied to a body having a mass of 1 kg, gives it an acceleration of 9.80665 m/sec/sec, the standard acceleration of free fall toward the earth's center for sea level at 45 deg latitude. The metric unit of force in SI units is the newton (N), which is defined as that force which, when applied to a body having a mass of 1 kg, gives it an acceleration of 1 m/sec/sec. These units must be distinguished from the (inconstant) local weight of a body having a mass of 1 kg, that is, the weight of a body is that force with which a body is attracted to the earth and is equal to the mass of a body multiplied by the acceleration due to gravity. However, because it is general practice to use "pound" rather than the technically correct term "pound-force," the term "kilogram" (or derived mass unit) has been used in this guide instead of "kilogram-force" in expressing the conversion factors for forces. The newton unit of force will find increasing use, and is essential in SI units.

Where approximate or nominal English units are used to express a value or range of values, the converted metric units in parentheses are also approximate or nominal. Where precise English units are used, the converted metric units are expressed as equally significant values.

Table I

QUANTITIES AND UNITS OF SPACE

Multiply	By	To obtain
	LENGTH	
Mil	25.4 (exactly)	Micron
Inches	25.4 (exactly)	Millimeters
Inches	2.54 (exactly)*	Centimeters
Feet	30.48 (exactly)	Centimeters
Feet	0.3048 (exactly)*	Meters
Feet	0.0003048 (exactly)*	Kilometers
Yards	0.9144 (exactly)	Meters
Miles (statute)	1,609.344 (exactly)*	Meters
Miles	1.609344 (exactly)	Kilometers
	AREA	
Square inches	6.4516 (exactly)	Square centimeters
Square feet	*929.03	Square centimeters
Square feet	0.092903	Square meters
Square yards	0.836127	Square meters
Acres	*0.40469	Hectares
Acres	*4,046.9	Square meters
Acres	*0.0040469	Square kilometers
Square miles	2.58999	Square kilometers
	VOLUME	
Cubic inches	16.3871	Cubic centimeters
Cubic feet	0.0283168	Cubic meters
Cubic yards	0.764555	Cubic meters
	CAPACITY	
Fluid ounces (U.S.)	29.5737	Cubic centimeters
Fluid ounces (U.S.)	29.5729	Milliliters
Liquid pints (U.S.)	0.473179	Cubic decimeters
Liquid pints (U.S.)	0.473166	Liters
Quarts (U.S.)	*946.358	Cubic centimeters
Quarts (U.S.)	*0.946331	Liters
Gallons (U.S.)	*3,785.43	Cubic centimeters
Gallons (U.S.)	3.78543	Cubic decimeters
Gallons (U.S.)	3.78533	Liters
Gallons (U.S.)	*0.00378543	Cubic meters
Gallons (U.K.)	4.54609	Cubic decimeters
Gallons (U.K.)	4.54596	Liters
Cubic feet	28.3160	Liters
Cubic yards	*764.55	Liters
Acre-feet	*1,233.5	Cubic meters
Acre-feet	*1,233,500	Liters

Table II

QUANTITIES AND UNITS OF MECHANICS

Multiply	By	To obtain
MASS		
Grains (1/7,000 lb)	64.79891 (exactly)	Milligrams
Troy ounces (480 grains)	31.1035	Grams
Ounces (avdp)	28.3495	Grams
Pounds (avdp)	0.45359237 (exactly)	Kilograms
Short tons (2,000 lb)	907.185	Kilograms
Short tons (2,000 lb)	0.907185	Metric tons
Long tons (2,240 lb)	1,016.05	Kilograms
FORCE/AREA		
Pounds per square inch	0.070307	Kilograms per square centimeter
Pounds per square inch	0.689476	Newtons per square centimeter
Pounds per square foot	4.88243	Kilograms per square meter
Pounds per square foot	47.8803	Newtons per square meter
MASS/VOLUME (DENSITY)		
Ounces per cubic inch	1.72999	Grams per cubic centimeter
Pounds per cubic foot	16.0185	Kilograms per cubic meter
Pounds per cubic foot	0.0160185	Grams per cubic centimeter
Tons (long) per cubic yard	1.32894	Grams per cubic centimeter
MASS/CAPACITY		
Ounces per gallon (U.S.)	7.4893	Grams per liter
Ounces per gallon (U.K.)	6.2362	Grams per liter
Pounds per gallon (U.S.)	119.829	Grams per liter
Pounds per gallon (U.K.)	99.779	Grams per liter
BENDING MOMENT OR TORQUE		
Inch-pounds	0.011521	Meter-kilograms
Inch-pounds	1.12985×10^6	Centimeter-dynes
Foot-pounds	0.138255	Meter-kilograms
Foot-pounds	1.35582×10^7	Centimeter-dynes
Foot-pounds per inch	5.4431	Centimeter-kilograms per centimeter
Ounce-inches	72.008	Gram-centimeters
VELOCITY		
Feet per second	30.48 (exactly)	Centimeters per second
Feet per second	0.3048 (exactly)*	Meters per second
Feet per year	$*0.965873 \times 10^{-6}$	Centimeters per second
Miles per hour	1.609344 (exactly)	Kilometers per hour
Miles per hour	0.44704 (exactly)	Meters per second
ACCELERATION*		
Feet per second^2	*0.3048	Meters per second^2
FLOW		
Cubic feet per second (second-feet)	*0.028317	Cubic meters per second
Cubic feet per minute	0.4719	Liters per second
Gallons (U.S.) per minute	0.06309	Liters per second
FORCE*		
Pounds	*0.453592	Kilograms
Pounds	*4.4482	Newtons
Pounds	$*4.4482 \times 10^5$	Dynes

Table II—Continued

Multiply	By	To obtain
WORK AND ENERGY*		
British thermal units (Btu)	*0.252	Kilogram calories
British thermal units (Btu)	1,055.06	Joules
Btu per pound	2.326 (exactly)	Joules per gram
Foot-pounds	*1.35582	Joules
POWER		
Horsepower	745.700	Watts
Btu per hour	0.293071	Watts
Foot-pounds per second	1.35582	Watts
HEAT TRANSFER		
Btu in./hr ft^2 degree F (k, thermal conductivity)	1.442	Milliwatts/cm degree C
Btu in./hr ft^2 degree F (k, thermal conductivity)	0.1240	Kg cal/hr m degree C
Btu ft/hr ft^2 degree F	*1.4880	Kg cal m/hr m^2 degree C
Btu/hr ft^2 degree F (C, thermal conductance)	0.568	Milliwatts/cm^2 degree C
Btu/hr ft^2 degree F (C, thermal conductance)	4.882	Kg cal/hr m^2 degree C
Degree F hr ft^2/Btu (R, thermal resistance)	1.761	Degree C cm^2/milliwatt
Btu/lb degree F (c, heat capacity)	4.1868	J/g degree C
Btu/lb degree F	*1.000	Cal/gram degree C
Ft^2/hr (thermal diffusivity)	0.2581	Cm^2/sec
Ft^2/hr (thermal diffusivity)	*0.09290	M^2/hr
WATER VAPOR TRANSMISSION		
Grains/hr ft^2 (water vapor) transmission)	16.7	Grams/24 hr m^2
Perms (permeance)	0.659	Metric perms
Perm-inches (permeability)	1.67	Metric perm-centimeters

Table III

OTHER QUANTITIES AND UNITS

Multiply	By	To obtain
Cubic feet per square foot per day (seepage)	*304.8	Liters per square meter per day
Pound-seconds per square foot (viscosity)	*4.8824	Kilogram second per square meter
Square feet per second (viscosity)	*0.092903	Square meters per second
Fahrenheit degrees (change)*	5/9 exactly	Celsius or Kelvin degrees (change)*
Volts per mil	0.03937	Kilovolts per millimeter
Lumens per square foot (foot-candles)	10.764	Lumens per square meter
Ohm-circular mils per foot	0.001662	Ohm-square millimeters per meter
Millicuries per cubic foot	*35.3147	Millicuries per cubic meter
Milliamps per square foot	*10.7639	Milliamps per square meter
Gallons per square yard	*4.527219	Liters per square meter
Pounds per inch	*0.17858	Kilograms per centimeter

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